Preliminary Economic Assessment, NI 43-101 Technical Report

James Bay Niobium Project Ontario, Canada

Prepared for:

300-1100 avenue des Canadiens-de-Montréal
Suite 300
Montreal, Québec
Canada H3B 2S2

Prepared by:

G MINING SERVICES INC.
7900 Taschereau Blvd.
Building D, Suite 200
Brossard, Québec
Canada J4X 1C2

Louis-Pierre Gignac, P.Eng., G Mining Services Inc.
Carl Michaud, P.Eng., G Mining Services Inc.
Paul Murphy, P.Geo., G Mining Services Inc.
Antoine Champagne, P.Eng., G Mining Services Inc.
Pierre Pelletier, P.Eng., Independent Metallurgist Consultant
Dorota El Rassi, P. Eng, RPA

Effective Date: October 13, 2020
Issue Date: November 27th, 2020

Preliminary Economic Assessment, NI 43-101 Technical Report – James Bay Niobium Project

Revision #

Ontario, Canada

NIOBAY METALS INC.

300-1100, avenue des Canadiens-de-Montréal
Montréal, Québec
Canada H3B 2S2
Tel: 514-866-6500
E-mail**:** cdufresne@niobaymetals.com
Web Address: www.niobaymetals.com

G Mining Services Inc.

7900 W. Taschereau Blvd.
Suite D-200Brossard, Québec
Canada J4X 1C2
Tel: (450) 465-1950 • Fax: (450) 465-6344
E-mail: l.gignac@gmining.com
Web Address: www.gmining.com

November 27th, 2020

General Conditions and limitations

Use of the report and its contents

This report has been prepared for the exclusive use of the Client or its agents. The factual information, descriptions, interpretations, comments, recommendations and electronic files contained herein are specific to the projects described in this report and do not apply to any other project or site. Under no circumstances may this information be used for any other purposes than those specified in the scope of work unless explicitly stipulated in the text of this report of formally interpreted when taken individually or out-of-context. As well, the final version of this report and its content supersedes any other text, opinion or preliminary version produced by G Mining Services Inc.

Qualified Persons

Prepared by:

(signed and sealed) “Louis-Pierre Gignac” Date: November 27, 2020

Louis-Pierre Gignac, P. Eng.
Co-President
G Mining Services Inc.

(signed and sealed) “Carl Michaud” Date: November 27, 2020

Carl Michaud, P. Eng.
Underground Engineering Manager
G Mining Services Inc.

(signed and sealed) “Antoine Champagne” Date: November 27, 2020

Antoine Champagne, P. Eng.
Open Pit Engineering Manager
G Mining Services Inc.

(signed and sealed) “Paul Murphy” Date: November 27, 2020

Paul Murphy, P. Eng.
Project Manager
G Mining Services Inc.

(signed and sealed) “Pierre Pelletier” Date: November 27, 2020

Pierre Pelletier, P. Eng.
Independent Metallurgist Consultant

(signed and sealed) “Dorota El-Rassi” Date: November 27, 2020

Dorota El-Rassi, P. Eng.
Senior Geological Engineer
RPA

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"Preliminary Economic Assessment, NI-43-101 Technical Report – James Bay Niobium Project, Ontario, Canada", prepared for NioBay Metals Inc. effective as of October 13, 2020 (the "Technical Report").

I, Louis-Pierre Gignac, do hereby certify that:

1) I am Co-President for G Mining Services Inc. ("GMS") with an office at D-200, 7900 Taschereau Blvd, Brossard, Québec, J4X 1C2;
2) I have graduated from McGill University, Canada with a B.Sc. in Mining Engineering in 1999, and from École Polytechnique de Montréal, Canada with a M.Sc. A. in Industrial Engineering in 2002.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, licence no. 132995. I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum and I am a Chartered Financial Analyst® Charter holder;
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for over 15 years, including pit optimization, surface mine design, mineral reserve estimations and mine scheduling. This has included work at Rosebel Gold Mines NV, the Essakane Project, the Merian Project and work on other pre-feasibility and feasibility studies. I have had no prior involvement with the property which is the subject of this Report;
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
6) I am responsible for the preparation of Sections 1, 2, 3, 15, 19, 22, 25, 26, 27 of the Report;
7) I have not visited the Project;
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 5 above have been prepared in accordance with NI 43-101;
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies

Dated this 27th day of November 2020,

/signed and sealed/

_____

Louis-Pierre Gignac, P.Eng.
Co-President
G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"Preliminary Economic Assessment, NI-43-101 Technical Report – James Bay Niobium Project, Ontario, Canada", prepared for NioBay Metals Inc. effective as of October 13, 2020 (the "Technical Report").

I, Antoine Champagne, do hereby certify that:

1) I am Open Pit Engineering Director for G Mining Services Inc. ("GMS") with an office at D-200, 7900 Taschereau Blvd, Brossard, Québec, J4X 1C2;
2) I have graduated from Laval University, Canada with a B.Sc. in Mechanical Engineering in 2005.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, licence no. 137814;
4) I have been involved in mining operations and engineering for over 10 years, including pit optimization, surface mine design, mineral reserve estimations and mine scheduling. This has included work at the Essakane Project, the Merian Project and work on other pre-feasibility and feasibility studies. I have had no prior involvement with the property which is the subject of this Report;
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
6) I am responsible for the preparation of Sections 16 and 21 of the Report;
7) I did not visit the Project;
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 5 above have been prepared in accordance with NI 43-101;
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies

Dated this 27th day of November 2020,

/signed and sealed/

_____

Antoine Champagne, P.Eng.
Open-Pit Engineering Manager
G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"Preliminary Economic Assessment, NI-43-101 Technical Report – James Bay Niobium Project, Ontario, Canada", prepared for NioBay Metals Inc. effective as of October 13, 2020 (the "Technical Report").

I, Carl Michaud, do hereby certify that:

1) I am the Underground Engineering Manager for G Mining Services Inc. ("GMS") with an office at D-200, 7900 Taschereau Blvd, Brossard, Québec, J4X 1C2;
2) I have graduated from l'Université du Laval with a B.Sc. (Mine Engineering) in 1996. In addition, I obtained an M.B.A. from the Université du Québec à Chicoutimi, in 2012.
3) I am a Professional Engineer registered with the "Ordre des Ingénieurs du Québec" (OIQ-Licence: 117090).
4) I have worked as a Junior Mine Engineer from May 1996 to 1998. Following this experience, I obtained my professional registration and have worked as a Mine Engineer for a total of 20 years with Placer Dome Canada, Mines McWatters, Ressources Campbell Inc., Orica Canada Inc. Goldcorp Inc., Nyrstar and Arcelor Mittal.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I am responsible for the preparation of Sections 16 and 21 of the Report.
7) I have not visited the Project.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 5 above have been prepared in accordance with NI 43-101.
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 27th day of November 2020,

/signed and sealed/

_____

Carl Michaud, P. Eng.,
Underground Engineering Manager
G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON

To Accompany the Report entitled:

"Preliminary Economic Assessment, NI-43-101 Technical Report – James Bay Niobium Project, Ontario, Canada", prepared for NioBay Metals Inc. effective as of October 13, 2020 (the "Technical Report").

I, Paul Murphy, do hereby certify that:

1) I am Project Manager for G Mining Services Inc. ("GMS") with an office at D-200, 7900 Taschereau Blvd, Brossard, Québec, J4X 1C2;
2) I am a graduate of Université Laval with a B.Sc. (Civil Engineering) in 1986.
3) I am a Professional Engineer registered with the "Ordre des Ingénieurs du Québec" (OIQ-Licence: 43320).
4) I have practiced my profession continuously since 1988 and have been involved in project and team management, construction coordination and engineering with projects located in North and South America and North-West Africa.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
6) I am responsible for the preparation of Sections 18 and 21 of the Report;
7) I have visited the Project on August, 21, 2020;
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 5 above have been prepared in accordance with NI 43-101;
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies

Dated this 27th day of November 2020,

/signed and sealed/

_____

Paul Murphy, P.Eng.
Project Manager
G Mining Services Inc.

CERTIFICATE OF AUTHOR – PIERRE PELLETIER

I, Pierre Pelletier, a registered mining engineer in the Province of Quebec (OIQ licence No. 36825), do hereby certify that:

1. I reside at 642 Mitchell Avenue, Mont Royal, Québec, Canada, H3R 1L4;
2. I am employed as a metallurgist consultant by and carried out this assignment for Niobay metal Inc. and I am supervising the metallurgical testwork program done by an independent laboratory, SGS Canada, Lakefield, Ontario.
3. I graduated with a Bachelor of mining degree (B.Eng. Mining) from Laval University in the city of Québec in 1982.
4. I have practiced my profession of mining engineer in mineral processing continuously for the last thirty-eight (38) years in the fields of gold, base mineral, and oxide mineral processing.
5. I am a member of the Ordre des Ingénieurs du Québec (OIQ licence No. 36825) and I am a member of the Canadian Institute of Mining and Metallurgy Association.
6. I have worked at different mineral processing positions, from process metallurgist, mill manager to vice-president metallurgy and consultant for a total of 38 years since graduating from university. I acquired my mineral processing expertise with Falconbridge Copper (1983 to 1988), Audrey Resources (1988 to 1992), Barrick Gold (1992 to 2002), Cambior Inc. (2003 to 2006), and IAMGOLD corp. (2007-2015). I have been an independent consultant since 2015, working on specific mandates for engineering firms and mining companies.
7. I have read the definition of "qualified person" set out in Regulation 43-101 / National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
8. I am author and responsible for sections 13 and 17, and I am co-author of and also share responsibility for sections 1, 25, 26 and 27 of the technical report entitled “PRELIMINARY ECONOMIC ASSESSMENT STUDY, NI 43-101 TECHNICAL REPORT, James Bay Project”, prepared for Niobay Metal Inc. The effective date of the report is October 13, 2020, and the signature date is November ___, 2020.
9. I have prior and involvement on the supervision of the metallurgical test work with the project that is the subject of the Technical Report.
10. I did receive from Niobay Metal inc. participation incentive securities (‘Share purchasing Options’) and I have held company shares since 2015.
11. I have not conducted a site visit of the property.
12. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission of which would make the Technical Report misleading.
13. I am an independent of the issuer applying all the tests in section 1.5 of NI 43-101, except 1 (D) in relation to the item 10 above, but based on the opinion and conclusion from the issuer, “Independence not compromised” with regard and according to 1,5 – 2.
14. I have read NI 43-101 Respecting Standards of Disclosure for Mineral projects and Form 43-101F1, and the Technical Report has been prepared in accordance with that instrument and form.

Dated this 20th day of November 2020, in the city of Mont Royal (Québec)

(Original signed and sealed)

Pierre Pelletier, P.Eng.

Pierrepelletier555@gmail.com

DOROTA EL-RASSI

I, Dorota El-Rassi, P.Eng., as an author of this report entitled "Preliminary Economic Assessment, NI-43-101 Technical Report – James Bay Niobium Project, Ontario, Canada" prepared for NioBay Metals Inc., effective October 13, 2020, and dated November 27, 2020, do hereby certify that:

1. I am a Senior Geological Engineer with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON M5J 2H7.

2. I am a graduate of the University of Toronto in 1997 with a B.A.Sc.(Hons.) degree in Geological and Mining Engineering and in 2000 with a M.Sc. degree in Geology and Mechanical Engineering.

3. I am registered as a Professional Geological Engineer in the Province of Ontario (Reg.# 100012348). I have worked as a geologist for a total of 20 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report on exploration and mining projects for due diligence and regulatory requirements
   - Mineral Resource estimates on a variety of commodities including gold, silver, copper, nickel, zinc, PGE, and industrial mineral deposits
   - Experienced user of Gemcom, Leapfrog, Phinar's x10-Geo, and Gslib software

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

5. I visited the James Bay Niobium Project on August 21, 2020.

6. I am responsible for Sections 12, and 14, and parts of Section 1, 10, 11, 25, and 26, of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have prepared a previous Technical Report dated August 23, 2020 on the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th day of November 2020,

(Signed and Sealed) ***Dorota El-Rassi***
Dorota El-Rassi, P.Eng.

Table of Contents

1. SUMMARY 1-1
1.1 Introduction 1-1
1.2 Property Description and Location 1-1
1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography 1-2
1.4 History 1-2
1.5 Geology and Mineralization 1-3
1.6 Deposit Types 1-4
1.7 Exploration and Drilling 1-4
1.8 Metallurgical Testing and Mineral Processing 1-5
1.9 Mineral Resource Estimate 1-6
1.10 Mineral Reserve Estimate 1-7
1.11 Mining Methods 1-7
1.12 Infrastructure 1-10
1.13 Market Studies 1-11
1.14 Environmental Studies, Permitting and Social or Community Impact 1-12
1.15 Capital and Operating Cost Estimate 1-12
1.16 Economic Analysis 1-15
1.17 Interpretation and Conclusions 1-15
1.18 Recommendations 1-16
2. INTRODUCTION 2-1
2.1 Qualified Persons 2-1
2.2 Sources of Information 2-2
2.3 Site Visits and Scope of Personal Inspection 2-2
2.4 List of Abbreviations 2-2
3. RELIANCE ON OTHER EXPERTS 3-1
4. PROPERTY DESCRIPTION AND LOCATION 4-1
4.1 Land Tenure 4-1
4.2 Mineral Rights 4-2
4.3 Royalties and other Encumbrances 4-2
4.4 Permitting 4-2
4.5 First Nations Consultation 4-2
5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY 5-1
5.1 Accessibility 5-1
5.2 Climate 5-1
5.3 Local Resources 5-2
5.4 Infrastructure 5-2
5.5 Physiography 5-2

6. HISTORY 6-1
6.1 Prior Ownership 6-1
6.2 Exploration and Development History 6-1
6.3 Historical Resource Estimates 6-3
6.4 Past Production 6-4
7. GEOLOGICAL SETTING AND MINERALIZATION 7-1
7.1 Regional Geology 7-1
7.2 Local Geology 7-4
7.2.1 Early Precambrian (Archean) 7-4
7.2.2 Middle Precambrian (Protozoic) – Argor Carbonatite Complex 7-5
7.2.3 Structural Geology 7-7
7.2.4 Property Geology 7-9
7.3 Mineralization 7-15
7.3.1 Pyrochlore 7-15
7.3.2 Columbite 7-16
7.3.3 Apatite 7-16
7.3.4 Magnetite 7-16
7.3.5 Sulphides 7-17
8. DEPOSIT TYPES 8-1
9. EXPLORATION 9-1
9.1 2020 Exploration Work 9-2
10. DRILLING 10-1
10.1 Historical Drilling 10-1
10.2 Current Drilling 10-1
11. SAMPLE PREPARATION, ANALYSES, AND SECURITY 11-1
11.1 2016 Check Sampling Program 11-1
11.2 Current Drilling Program 11-1
11.3 Quality Assurance and Quality Control 11-2
11.3.1 2016 Check Sampling Program 11-2
11.3.2 Current Drilling Program 11-3
12. DATA VERIFICATION 12-1
12.1 Site Visits 12-1
12.2 Data Verification 12-1
13. MINERAL PROCESSING AND METALLURGICAL TESTING 13-1
13.1 Historical Work (1967-1969) 13-1
13.1.1 Initial Laboratory Testing 13-1
13.1.2 Mineralogy 13-1
13.1.3 Pilot Plant Test Work 13-2
13.2 Recent Work – 2017-2018 13-3
13.2.1 Composite Sample 13-4
13.2.2 Qemscam 13-7
13.2.3 Metallurgical Test Work 13-8

13.3 Recent Work - 2020 13-13
13.3.1 Master Composite 13-13
13.3.2 Gravity Separation 13-15
13.3.3 Flotation Tests 13-15
13.3.4 Ongoing Test Work 13-16
13.3.5 Conclusions and Recommendations 13-17
14. MINERAL RESOURCE ESTIMATE 14-1
14.1 Summary 14-1
14.1.1 Resource Database 14-2
14.1.2 Geological Interpretation 14-3
14.1.3 Resource Assays 14-5
14.1.4 Treatment of High-Grade Assays 14-5
14.1.5 Compositing 14-5
14.2 Trend Analysis 14-6
14.2.1 Variography 14-6
14.2.2 Grade Contouring 14-7
14.3 Search Strategy and Grade Interpolation Parameters 14-9
14.4 Bult Density 14-10
14.5 Block Model 14-10
14.6 Cut-off Grade 14-11
14.7 Classification 14-13
14.8 Block Model Validation 14-15
14.9 Mineral Resource Reporting 14-24
14.10 Comparison to the Previous Mineral Resource Estimate 14-25
15. MINERAL RESERVE ESTIMATE 15-1
16. MINING METHODS 16-1
16.1 Introduction 16-1
16.2 Open Pit Mining Method 16-1
16.2.1 Pit Optimization 16-1
16.2.2 Open Pit Mine Designs 16-7
16.2.3 Overburden and Waste Rock Storage 16-12
16.2.4 Stockpiles 16-14
16.3 Underground Mining Method 16-17
16.3.1 Underground Method Description 16-17
16.3.2 Geotechnical Considerations 16-19
16.3.3 Hydrological Considerations 16-19
16.3.4 Underground Cut-Off Grade Estimation 16-20
16.3.5 Underground Mine Design 16-22
16.3.6 Mining and Paste Dilution and Mining Recovery 16-25
16.3.7 Mine Access and Development 16-25
16.3.8 Horizontal Development 16-26
16.3.9 Vertical Development 16-28
16.3.10 Development Sequence 16-30
16.4.1 Underground Miscellaneous Infrastructure 16-35

16.5 Production Scenarios 16-36
16.5.1 Scenario 1: Open Pit Only 16-37
16.5.2 Scenario 2: Combined Open Pit and Underground 16-38
16.5.3 Scenario 3: Underground Only 16-38
16.6 Mine Operations and Equipment Selection 16-45
16.6.1 Open Pit Mining 16-45
16.6.2 Underground Mining Operations 16-54
16.7 Mine Equipment Requirements 16-64
16.8 Mine Workforce 16-67
16.8.1 Roster Schedule and Assumptions 16-67
16.8.2 Mine Workforce Requirements 16-68
17. RECOVERY METHOD 17-1
17.1 Process Selection 17-1
17.2 Process Description 17-2
17.2.1 Crushing 17-2
17.2.2 Grinding 17-2
17.2.3 Gravity Separation 17-3
17.2.4 Desliming Stage 17-3
17.2.5 Magnetic Separation 17-3
17.2.6 Sulfide Flotation 17-3
17.2.7 Mica Flotation 17-3
17.2.8 Carbonate Flotation 17-3
17.2.9 Dewatering 17-4
17.2.10 Niobium Flotation 17-4
17.2.11 Concentrate Leaching 17-4
17.2.12 Filtration, Drying and Packaging 17-4
17.2.13 Reagents 17-4
17.3 Process Design Criteria and Mass Balance 17-4
18. INFRASTRUCTURE 18-1
18.1 Existing Infrastructure and Location 18-3
18.2 Area 100 Infrastructure 18-4
18.2.1 Area 110 Roads, Bridges & Fencing 18-4
18.2.2 Area 120 – Mine Infrastructure 18-6
18.2.3 Area 130/140 – Support Infrastructure / Camp Facility 18-8
18.2.4 Area 160 – Process Plant Infrastructure 18-10
18.3 Area 200 – Power and Electrical 18-11
18.4 Area 300 Water Management 18-12
18.4.1 Effluent Treatment Plant 18-12
18.4.2 Potable and Sewage Water 18-13
18.4.3 Tailings Storage Facility 18-13
18.5 Underground Mining Infrastructure 18-13
19. MARKET STUDIES 19-1
19.1 Uses of Niobium 19-1
19.2 Niobium Supply and Demand 19-1

19.3 Marketing of Niobium 19-2
19.4 Metal Prices and Market Outlook 19-2
20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT 20-1
20.1 Environmental Studies 20-1
20.2 Project Permitting Requirements 20-1
20.2.1 Consultation Requirements 20-1
20.2.2 Provincial Requirements 20-1
20.2.3 Federal Requirements 20-2
20.3 Social or Community Impact 20-3
20.3.1 Plans, Negotiations and Agreements 20-3
21. CAPITAL AND OPERATING COSTS 21-1
21.1 Capital Expenditures 21-1
21.2 Basis of Estimate 21-2
21.2.1 Mining 21-2
21.2.2 Processing Plant and Infrastructure 21-3
21.3 Direct Costs 21-4
21.3.1 Mining 21-4
21.3.2 Process Plant and Infrastructure 21-5
21.3.3 Construction Indirect Costs and Owner's Costs 21-13
21.4 Sustaining Capital 21-16
21.4.1 Mining 21-16
21.4.2 Processing Plant and Infrastructure 21-17
21.5 Operating Costs 21-21
21.5.1 Operating Costs Summary 21-21
21.5.2 Mining Costs 21-25
21.5.3 Processing Costs 21-32
21.5.4 General & Administration 21-34
22. ECONOMIC ANALYSIS 22-1
22.1 Assumptions 22-1
22.2 Metal Production & Revenues 22-1
22.3 Royalties and Marketing Costs 22-2
22.4 Operating Cost Summary 22-2
22.5 Capital Cost Summary 22-4
22.5.1 Initial Capital Costs 22-4
22.5.2 Sustaining Capital Costs 22-4
22.6 Working Capital 22-5
22.7 Reclamation & Closure Costs 22-6
22.8 Taxation 22-6
22.8.1 Ontario Mining Taxes 22-6
22.8.2 Income Taxes 22-7
22.9 Economic Results 22-7
22.9.1 Base Case Results 22-7

22.9.2 Sensitivities 22-12
23. ADJACENT PROPERTIES 23-1
24. OTHER RELEVANT DATA AND INFORMATION 24-1
25. INTERPRETATION AND CONCLUSIONS 25-1
25.1 Exploration and Geology 25-1
25.2 Mining 25-1
25.3 Metallurgy & Processing 25-2
25.4 Infrastructure and Power Supply 25-2
25.5 OPEX, CAPEX and Economics 25-3
26. RECOMMENDATIONS 26-1
26.1 Exploration and Geology 26-1
26.2 Mining 26-1
26.3 Metallurgy and Processing 26-2
26.4 Infrastructure and Power Supply 26-2
26.5 Project Execution 26-2
26.6 Recommended Work Program 26-3
27. REFERENCES 27-1

List of Figures

Figure 1.1: Open Pit Scenario Mining Schedule 1-9
Figure 1.2: Open Pit Scenario Processing Schedule 1-9
Figure 1.3: Open Pit Scenario Niobium Production 1-10
Figure 4.1: Location Map 4-4
Figure 4.2: Mining Lease Location Map 4-5
Figure 5.1: Infrastructure Map 5-3
Figure 7.1: Regional Geology 7-3
Figure 7.2: Property Geology 7-10
Figure 7.3: Geological Section 0+00 7-14
Figure 7.4: Geological Section 8+00N 7-15
Figure 10.1: Drill Hole Locations 10-3
Figure 11.1: Performance of Blank Reference Material Nb_2O_5 – Pure Quartz (>90 µm) 11-5
Figure 11.2: Performance of Standard Reference Material OKA-1 11-6
Figure 11.3: Field Duplicate Assay Results (% Nb_2O_5) 11-7
Figure 13.1: Composite Location 13-5
Figure 13.2: Theorical Recovery Curve 13-8
Figure 13.3: Sink 3.2 – Mainly Niobium Minerals 13-11
Figure 13.4: Sink 3.1 – Mainly Pyroxene 13-11
Figure 13.5: Float Product 3.1: Mainly Silicate (perhaps feldspar) and Pyroxene 13-12
Figure 13.6: Location of Drill Holes Used for Composite Preparation 13-14
Figure 14.1: QQ Plot of the Historical and Resampled Data 14-2
Figure 14.2: Lithological Model 14-4
Figure 14.3: Histogram of James Bay Niobium Sampling Length 14-6
Figure 14.4: James Bay Niobium Bay Directional Variogram 14-7
Figure 14.5: Trend Analyses for James Bay Niobium Project Mineralization 14-8
Figure 14.6: Mineral Resource Sensitivity to Cut-off Grade 14-12
Figure 14.7: Histogram of the Classified Blocks vs. Distance to the Composites 14-13
Figure 14.8: Classified Mineral Resource Blocks 14-14
Figure 14.9: East Swath Plot 14-16
Figure 14.10: North Swath Plot 14-16
Figure 14.11: Vertical Swath Plot 14-17
Figure 14.12: Section 800N showing Estimate Block Grades vs. Composites 14-18
Figure 14.13: Section 400N showing Estimates Block Grades vs. Composites 14-19
Figure 14.14: Section 0N showing Estimated Block Grades vs. Composites 14-20
Figure 14.15: Section 200S showing Estimated Block Grades vs. Composites 14-21

Figure 14.16: Level Plan -50 M 14-22
Figure 14.17: Global Change of Support Check 14-24
Figure 16.1: 50 m Creek Buffer Zone 16-3
Figure 16.2: Pit by Pit Results for 50 m Buffer Zone 16-6
Figure 16.3: Pit by Pit Results no Buffer Zone 16-6
Figure 16.4: Open Pit Only Scenario vs. Combined Scenario 16-8
Figure 16.5: Open Pit Only Scenario vs. Combined Scenario - Pit Limits 16-10
Figure 16.6: Open Pit Only Scenario - Phasing 16-11
Figure 16.7: Combined Scenario Phasing 16-12
Figure 16.8: Waste and Overburden Dumps 16-14
Figure 16.9: Open Pit Only Scenario - Stockpile Layout 16-16
Figure 16.10: Combined Scenario - Stockpile Layout 16-16
Figure 16.11: Underground Only - Stockpile Layout 16-17
Figure 16.12: Illustration of Transverse Long Hole Stoping 16-18
Figure 16.13 :Transverse Long Hole Stoping Method 16-19
Figure 16.14: Mineralization of Deposit 16-22
Figure 16.15: DSO Generated Stopes Scenario 2 and Scenario 3 16-24
Figure 16.16: Decline Development Profile 16-26
Figure 16.17: Main Development Scenario 2 - James Bay Niobium 16-29
Figure 16.18: Main Development Scenario 3 -- James Bay Niobium 16-30
Figure 16.19: Mining Sequences for Scenario 2 16-32
Figure 16.20: Mining Sequences for Scenario 3 16-33
Figure 16.21: Final Design of Underground Mine (Scenarios 2 & 3) 16-34
Figure 16.22:Loading Point 16-36
Figure 16.23: Talpac3D Layout - Open Pit Only Scenario – End of Mine Life 16-50
Figure 16.24: Open Pit Only Scenario – Average Cycle Time per Period 16-51
Figure 16.25: Open Pit Only Scenario - Total Truck Hours 16-52
Figure 16.26: Open Pit Only Scenario - Truck Requirements 16-52
Figure 16.27: Open Pit Only - Fuel Consumptions and Rates 16-53
Figure 16.28: Typical Production Drilling Pattern 16-55
Figure 16.29: Ventilation Network for Scenario 2 16-61
Figure 16.30: Ventilation Network for Scenario 3 16-61
Figure 17.1 : Process Flowsheet Diagram 17-2
Figure 18.1: General Layout Scenario 1 18-1
Figure 18.2: General Layout Scenario 2 18-2
Figure 18.3: General Layout Scenario 3 18-2
Figure 18.4: HydroOne Renison Substation 18-4

Figure 18.5: Options for Main Access Road 18-5
Figure 18.6: Site Roads Cross-Section 18-6
Figure 18.7: Scenario 3 Mine Dry, Maintenance Shop and Warehouse 18-8
Figure 18.8: Camp/Administration Building, First Floor & 3D 18-9
Figure 18.9: Assay Laboratory Layout 18-10
Figure 18.10: Process Building – Plan View 18-11
Figure 18.11: Process Plant 3D 18-11
Figure 19.1: Historical and Forecasts Price of Ferroniobium 19-3
Figure 19.2: Ferroniobium Consumption and Forecasts 19-4
Figure 21.1: Scenario 1 General Layout 21-18
Figure 21.2: Scenario 2 General Layout 21-19
Figure 21.3: Scenario 3 General Layout 21-20
Figure 21.4: Operating Cost by Year Scenario 1 21-23
Figure 21.5: Operating Cost by Year Scenario 2 21-24
Figure 21.6: Operating Cost by Year Scenario 3 21-25
Figure 22.1: Cumulative After-Tax Cash Flow 22-12
Figure 22.2: After-Tax IRR Sensitivity of Scenario 1 22-15

List of Tables

Table 1.1: Summary of Mineral Resources – July 9, 2020 Niobay Metals Inc. – James Bay Niobium Project 1-6
Table 1.2: LOM Physical Summary 1-8
Table 1.3: Summary of LOM Capital Costs 1-13
Table 1.4: Summary of LOM Operating Costs 1-14
Table 1.5: Project Economic Results and Summary 1-15
Table 2.1: List of Abbreviations 2-3
Table 5.1: Summary of Moosonee Climate Data 5-1
Table 9.1: Core Sampling Results 9-2
Table 10.1: Summary of James Bay Drilling Programs 10-1
Table 11.1: Standards 11-3
Table 13.1: Historical Niobium Concentrates 13-3
Table 13.2: Mineral Liberation 13-7
Table 13.3: Master Composite 2017 and FS Resource Estimate 13-9
Table 13.4: Heavy Liquid on Niobium Concentrate 13-11
Table 13.5: Preliminary Flotation Test Results 13-12
Table 13.6: Master Composite 2020 13-13
Table 13.7: Gravity Separation Results 13-15
Table 13.8: Flotation Test Results 13-16
Table 13.9: Global Nb Recovery 13-16
Table 14.1: Summary of Mineral Resources – July 9, 2020 Niobay Metals Inc. – James Bay Niobium Project 14-1
Table 14.2: Statistical Summary of Nb_2O_5 Assay Data 14-5
Table 14.3: Descriptive Statistics of James Bay Nb_2O_5 Composites 14-6
Table 14.4: Variography of the Niobium Mineralization 14-7
Table 14.5: Block Model Interpolation Parameters 14-9
Table 14.6: Summary Statistics – Density 14-10
Table 14.7: Waste Rock Density 14-10
Table 14.8: Block Model Setup 14-11
Table 14.9: Mineral Resource Sensitivity to Cut-off Grade 14-12
Table 14.10: Block vs. Wireframe Volumes 14-15
Table 14.11: Basic Statistics of Nb_2O_5 Block Grades 14-23
Table 14.12: Comparison of Assay, Composite and Block Grades 14-23
Table 14.13: Summary of Mineral Resources – July 9, 2020 14-24
Table 14.14: Comparison to Previous Mineral Resource Estimate 14-25

Table 16.1: Pit Optimization Parameters 16-2
Table 16.2: Pit Shell Results with 50 m Buffer 16-4
Table 16.3: Pit Shell Results no Buffer Zone 16-5
Table 16.4: Pit Shell Selection 16-7
Table 16.5: Open Pit Design and Scenario Summary 16-8
Table 16.6: Shell Hierarchy 16-9
Table 16.7: Pit Phase Design Criteria 16-9
Table 16.8: Waste Storage Capacities 16-13
Table 16.9: Waste Pile Design Criteria 16-13
Table 16.10: Stockpile Grade Bins Cut-off-Grades 16-14
Table 16.11: Stockpile Design Criteria 16-15
Table 16.12: Stope Assumptions for the COG Estimate 16-20
Table 16.13: Cost Assumptions for COG Estimate 16-21
Table 16.14: Revenue Parameters for COG Estimate 16-21
Table 16.15: Potential Stope by Resource Classification Scenario 2 16-25
Table 16.16: Potential Stope by Resource Classification Scenario 3 16-25
Table 16.17: Excavation Dimension 16-28
Table 16.18: Meters of Development Per Excavation Type 16-31
Table 16.19: LOM Physicals Summary 16-37
Table 16.20: Scenario 1 Mine Production Schedule 16-39
Table 16.21: Scenario 1 Stockpile Schedule 16-40
Table 16.22: Scenario 1 Plant Schedule 16-41
Table 16.23: Scenario 2 Mine Production Schedule 16-42
Table 16.24: Scenario 2 Stockpile Schedule 16-43
Table 16.25: Scenario 3 Mining Schedule 16-44
Table 16.26: Scenario 3 Stockpile & Plant Schedule 16-45
Table 16.27: Loading Specifications 16-48
Table 16.28: Speed Limits 16-49
Table 16.29: Rolling Resistance 16-49
Table 16.30: Fixed Cycle Time Components 16-50
Table 16.31: LHD Specifications 16-57
Table 16.32: Truck Specifications and Parameters Summary 16-57
Table 16.33: Airflow Required For Scenarios 2 and 3 16-60
Table 16.34: Fan System 16-62
Table 16.35: Dewatering 16-62
Table 16.36: Scenario 1 Equipment Purchase Schedule 16-65
Table 16.37: Scenario 2 Equipment Purchase Schedule 16-66

Table 16.38: Scenario 3 Equipment Purchase Schedule 16-67
Table 16.39: Manpower Requirements – Scenario 1 16-69
Table 16.40: Manpower Requirements – Scenario 2 16-70
Table 16.41: Manpower Requirements – Scenario 3 16-71
Table 17.1: Major Design Criteria 17-5
Table 19.1: Annual Production Capacity 19-2
Table 21.1: Initial Capital Expenditures Summary Scenario 1 21-1
Table 21.2: WBS Level 1 21-3
Table 21.3: Mobile Equipment Capital Expenditures 21-5
Table 21.4: Mobile Equipment Capital Expenditure by Fleet Type 21-5
Table 21.5: Infrastructure Capital Expenditures 21-6
Table 21.6: Power Supply and Communications Capital Expenditures 21-7
Table 21.7: Water and Tailings Management Capital Expenditures 21-8
Table 21.8: Mobile Equipment Capital Expenditures 21-8
Table 21.9: Infrastructure Capital Expenditures 21-9
Table 21.10: Power Supply and Communications Capital Expenditures 21-10
Table 21.11: Water and Tailings Management Capital Expenditures 21-10
Table 21.12: Mobile Equipment Capital Expenditures 21-11
Table 21.13: Infrastructure Capital Expenditures 21-11
Table 21.14: Power Supply and Communications Capital Expenditures 21-12
Table 21.15: Water and Tailings Management Capital Expenditures 21-13
Table 21.16: Mobile Equipment Capital Expenditures 21-13
Table 21.17: Indirect Costs Scenario 1 21-14
Table 21.18: Indirect Costs Scenario 2 21-14
Table 21.19: Indirect Costs Scenario 3 21-14
Table 21.20: Pre-production Mining Capital Expenditures 21-15
Table 21.21: Sustaining CAPEX Scenario 21-16
Table 21.22: Scenario 1 TSF Costs 21-17
Table 21.23: Scenario 2 TSF Costs 21-18
Table 21.24: Scenario 3 TSF Costs 21-19
Table 21.25: Sustaining Expenditures Summary 21-20
Table 21.26: Operating Cost Summary Scenario 1 21-21
Table 21.27: Operating Cost Summary Scenario 2 21-21
Table 21.28: Operating Cost Summary Scenario 3 21-22
Table 21.29: Peak Operations Workforce 21-22
Table 21.30: Major Equipment Hourly Operating Cost 21-26
Table 21.31: Mining Costs 21-27

Table 21.32: Open Pit Mining Cost for Scenario 2 21-28
Table 21.33: Underground Mining Cost for Scenario 2 21-29
Table 21.34: Mining Cost for Scenario 3 21-32
Table 21.35: Power Consumption 21-33
Table 21.36: Manpower 21-33
Table 21.37: Process Operating Costs 21-34
Table 21.38: General and Administration Operating Costs 21-35
Table 22.1: Summary of LOM Metal Production & Revenues 22-2
Table 22.2: Summary of LOM Operating Costs 22-3
Table 22.3: Summary of LOM Capital Costs 22-5
Table 22.4: Closure Cost Summary 22-6
Table 22.5: Taxation Summary 22-6
Table 22.6: Project Economic Results and Summary 22-8
Table 22.7: Annual Cash Flow Model Scenario 1 22-9
Table 22.8: Annual Cash Flow Model Scenario 2 22-10
Table 22.9: Annual Cash Flow Model Scenario 3 22-11
Table 22.10: Niobium Price Sensitivity in Dollar Increments 22-13
Table 22.11: Niobium Price Sensitivity in Percentage Increments 22-13
Table 22.12: OPEX Sensitivity in Percentage Increments 22-14
Table 22.13: CAPEX Sensitivity in Percentage Increments 22-14
Table 26.1: Proposed Work Program Budget 26-3

1. SUMMARY

1.1 Introduction

G Mining Services Inc. ("GMS") was retained by Niobay Metals Inc. ("Niobay") to prepare an independent Technical Report on the James Bay Niobium Project (the Project), located in Cochrane District, northeastern Ontario, Canada. The purpose of this Technical Report is to disclose the results of a Preliminary Economic Assessment ("PEA") for the Project based on the Mineral Resource estimate prepared by Roscoe Postle Associates Inc. ("RPA"), as disclosed in a NI 43-101 Technical Report issued on August 23, 2020. This Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

Niobay is a Montreal-based company formed in January 1954 as Exploration Minière du Nord Inc. and was subsequently known as MDN Inc. ("MDN"). Niobay is a reporting issuer in British Columbia, Alberta, Ontario, and Quebec. The common shares of Niobay trade on the TSX Venture Exchange and the company is under the jurisdiction of the *Autorité des marchés financiers du Québec*.

All currency in this Technical Report is Canadian dollars ("CAD") unless otherwise noted.

1.2 Property Description and Location

The Project is located in northeastern Ontario, approximately 40 km south of the town of Moosonee. Access to the Property is by helicopter year-round and bush roads from Moosonee during the winter months. It is centred at approximately Latitude 50°50'69" N and Longitude 80°40'48" W within 1:50,000 scale NTS map sheet 42I/15 (Meengan Creek).

The Property consists of a single mining lease (Lease 19586 or Claim CLM11) covering an area of 2,585.15 ha and 306 contiguous single claims covering an additional 6,247.56 ha.

The lease includes both mining and surface rights. The Ontario Ministry of Energy, Northern Development and Mines ("MENDM") has issued a new mining lease on September 1st, 2020 and is valid until February 2028. Niobay has maintained the lease in good standing by making annual mining rental costs that total CAD 7,755.54.

The single cell claims are in good standing until their renewal date of January 30, 2022. Work required to renew all of the single cell claims on their renewal date totals approximately CAD 122,400.

On June 7, 2016, MDN, a predecessor company to Niobay, announced that it had signed a definitive property purchase agreement to acquire a 100% interest in the Project from Barrick Gold Corporation, James Bay Columbium Ltd., and Goldcorp Inc. (collectively the Vendors). In consideration for acquiring the Project, MDN agreed to make a one-time cash payment of CAD 25,000 and issue 5,000,000 common shares to the Vendors. MDN's interest in the Project is subject to a 2% net smelter return NSR royalty with MDN having the right to buy-back half the royalty (1%) at any time for CAD 2,000,000 (in constant 2016 dollars, subject to a cap of CAD 3,000,000)

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Moosonee is connected to Cochrane, Ontario by the Ontario Northland Railway which provides three day a week passenger service and twice weekly freight service. Moosonee also benefits from daily commercial flights from Timmins. The Wetum winter road links Moose Factory to the provincial road system at Otter Rapids seasonally, approximately 149 km to the south. Access to the Property is by helicopter year-round and bush roads from Moosonee during the winter months.

The Property lies within the James Bay Lowland ecoregion of the Hudson Bay Plain ecozone and is marked by brief and warm summers and cold and snowy winters.

Various services including temporary accommodations, medical services, a post office, fuel (gas diesel and propane) stations, and some heavy equipment and machinery shops are available in Moosonee.

There is no permanent infrastructure on the Property except for the historical underground excavations which consist of a 133 ft (40.54 m) shaft and a 100 ft (30.5 m) crosscut.

The area is poorly drained, flat and dominated by extensive wetlands. The elevation on the Property varies from approximately 35 MASL to 45 MASL.

1.4 History

The history of work on the Property by previous owners dates back to the 1960s with the following work programs undertaken:

- From April to June 1965, Canadian Aero Mineral Surveys Limited ("Canadian Aero") was contracted to fly a combined magnetic-electromagnetic ("EM")-radiometric survey.
- From June to August 1965, as a follow-up to the airborne survey, a helicopter-supported geological mapping program was carried out by Argor Explorations on behalf of the concession holders.

- From June to October 1965, Huntec Limited ("Huntec") was contracted to complete ground geophysical surveys over several airborne geophysical anomalies.
- In 1966, 18 holes were drilled, and niobium mineralization was traced over a strike length of 7,800 ft (2,377 m).
- An additional 67 holes were drilled in 1967, bringing the number of holes drilled to 85 for a total of 47,625 ft (14,514 m) in outlining the deposit to a maximum depth of 900 ft (274.3 m).
- In 1968, a test shaft was sunk in the central part of the mineralized body and a 250-t bulk sample of niobium-bearing carbonatite was mined for metallurgical testing.
- In 1967, Canadian Bechtel Limited ("Bechtel") completed a preliminary mining appraisal on the Project (historically referred to as the Argor Deposit). The study considered three scenarios 1) mining the entire deposit by open pit, 2) mining the entire deposit by underground methods, and 3) mining the south end of the deposit by open pit followed by underground mining of the northwest limb.
- In 1979, Bechtel updated the preliminary mining appraisal. No further work had been completed between 1979 and 2016 when Niobay acquired the Project.

1.5 Geology and Mineralization

The James Bay Niobium Deposit is hosted by the Carbonatite Complex and occurs in the northern portion of the KSZ of the Superior Province. Numerous alkalic and carbonatite intrusions occur along this structure, which extends from the east shore of Lake Superior northeast to James Bay. Rocks in the general area of the Property are characterized by granulite facies rank gneisses and a pervasive north- to northeast-trending fault pattern.

The Carbonatite Complex is overlain by approximately 10 m of overburden and 20 m of Lower Devonian rocks of the Sextant Formation consisting of poorly bedded sandstone, mudstone, siltstone, and loosely cemented conglomerate.

The Carbonatite Complex appears to be a dyke-like body with a long axis striking north. The enclosing gneisses are described as mylonitic or augen gneisses.

The principal niobium-bearing mineral is pyrochlore, but niobium also occurs, to a much lesser extent, in the mineral columbite.

According to Stockford (1972), the main pyrochlore-bearing phases of the Carbonatite Complex include lineated dolomitic carbonatite, calcite-dolomite carbonatite breccia, massive calcite-dolomite carbonatite, and “crushed” dolomitic carbonatite.

Significant pyrochlore mineralization occurs only in sodium and magnesium-rich phases of the carbonatite. The highest-grade mineralization occurs in a mixed dolomite-calcite host. The pure dolomitic host produces intermediate grades and pure calcite is normally low in grade or barren.

The columbite occurs along the eastern side of the complex over a strike length of 120 m in the crushed dolomitic carbonatite. The columbite is totally or partially pseudomorphic after pyrochlore. The columbite may occur in the core or the rim of the crystal, or it may completely replace the pyrochlore. The columbite does not appear to be restricted to any one rock unit, however, it does appear to be related to fracturing and hematitic alteration.

Pale green fluor-variety of apatite is universally present in all rock units within the carbonatite-pyroxenite complex. Usually, the highest-grade bands of pyrochlore-bearing carbonatite also contain abnormally high quantities of apatite, while the converse is not true.

1.6 Deposit Types

Carbonatite-associated deposits such as James Bay include a variety of mineral deposits that occur both within and in close spatial association with carbonatites and related alkalic silicate rocks. Carbonatite-associated deposits are mined for rare-earth elements (“REEs”), niobium, iron, copper, apatite, vermiculite, and fluorite. By-products include barite, zircon, or baddeleyite, tantalum, uranium, platinum group elements, silver, and gold.

Carbonatites may consist of a number of intrusive phases with different textural and mineralogical characteristics. Early phases typically consist mainly of calcite, do not contain peralkaline pyroxenes or amphiboles, and contain associated apatite + magnetite ± pyrochlore mineralization. Later phases, which may contain dolomite, ankerite and siderite, in addition to calcite, are commonly enriched in pyrochlore.

1.7 Exploration and Drilling

Niobay recovered all historical drill logs, collar survey data, historical assay certificates, and vertical drill sections with assays and geological interpretation from the archives of James Bay Columbium Ltd., one of the previous owners of the Property. All the existing historical drill core was stored in a secure building in Moosonee.

During the fall of 2016, Niobay undertook a program of core re-logging and check sampling to confirm the historical results on the Property.

Niobay completed seven diamond drill holes for a total of 3,090 m between February 2 and March 12, 2020. A total of 86 holes for 16,320 m have been drilled on the Property to date.

1.8 Metallurgical Testing and Mineral Processing

As part of a Feasibility Study prepared by Canadian Bechtel Ltd, preliminary metallurgical testing was carried out on drill core samples obtained during the 1967 drilling programme. The methods of concentration investigated included the following: heavy liquid separation, gravity separation (table, jig, and spiral test), magnetic and electrostatic separation and flotation (direct and dual flotation).

Samples were collected from the storage area on the property in 2017 to confirm and improve the historical work. More recently, fresh core from the 2020 drill program was used to perform additional metallurgical testing. All tests were performed by SGS Canada laboratory in Lakefield, On. The main metallurgical test work results are as follows:

- A single grinding test was performed to verify the Ball Mill work index ("Bwi"). The single test confirmed a low energy requirement at 9.6 kW-h/t.
- Heavy liquid separation in 2017 provided excellent results with 90% of Nb_2O_5 concentrated in a 5% mass pull at an SG cutting point of 3.3. These results suggest a good liberation of the niobium minerals and open the possibility of gravity concentration inside the process flowsheet.
- A gravity test was performed in 2020 using two stages of gravity spirals in series. Based on the heavy liquid separation completed in 2017 and the historical results, the spiral feed size was revised to less than 500 microns with the intent of simplifying the gravity flowsheet. The simplified flowsheet was operated in open circuit with an Nb_2O_5 recovery of 96.8% Nb_2O_5 in the concentrate; the mass rejected was at 42% with a loss of only 3.2% of Nb_2O_5.
- Ten flotation tests were completed in 2017 and the results were in line with the historical data. The concentrates produced ranged from 56% to 59% Nb_2O_5 with the Nb_2O_5 recoveries between 68% and 77% in open circuit. Fourteen flotation tests were performed in 2020. After a few reagent and parameter modifications similar results to 2017 were achieved with a niobium recovery of 80% at a concentrate grade of 64% and contaminant levels below the quality targets.
- The final Nb_2O_5 recovery achieved reached 78% for an open circuit without any recirculation. Generally, recovery might improve by closing the niobium circuit.

- The tests in 2017 identified one major issue; the silica concentration remained relatively high or close to the higher acceptable limit. Silica was the main issue addressed in the 2020 phase of test work. Introduction of the gravity concentration before the flotation combined to modification of flotation conditions delivered silica level below the target limit. It represents an improvement when compared to the historical flowsheet which included many stages of cleaner and gravity separation to address this issue.

1.9 Mineral Resource Estimate

The Mineral Resources conform to CIM (2014) definitions and are summarized in Table 1.1. RPA considers the Mineral Resources of the James Bay Project to likely be amenable to underground extraction. The Mineral Resource estimate has an effective date of July 9, 2020 and excludes a portion of the deposit designated as part of the crown pillar. RPA has excluded approximately 7.2 Mt averaging 0.5% Nb_2O_5 situated in a 46 m thick crown pillar. Part of this crown pillar was considered for the open pit mining scenarios studied.

Table 1.1: Summary of Mineral Resources – July 9, 2020
Niobay Metals Inc. – James Bay Niobium Project

Category	Tonnage (Mt)	Grade (%Nb_2O_5)	Contained Nb_2O_5 (M kg)
Indicated	29.7	0.53	158
Inferred	33.8	0.52	177

Notes:

1. *CIM (2014) definitions were followed for Mineral Resources.*
2. *Mineral Resources are reported using a cut-off grade of 0.3% Nb_2O_5 based on an Underground Mining Scenario, an operating cost of CAD 70/t, and a metallurgical recovery of 70%.*
3. *Mineral Resources are estimated using a long-term niobium price of USD 40 per kg and a USD/CAD exchange rate of 1:1.2.*
4. *A minimum mining width of approximately 7.5 m was used.*
5. *Bulk density is 2.93 t/m^3.*
6. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
7. *Resources situated in a 46 m thick crown pillar have been excluded.*
8. *Numbers may not add due to rounding.*

RPA carried out database compilation and verification, geological modelling and wireframe construction, capping and compositing data for geostatistical analysis and variography, selection of estimation strategy and estimation parameters, block modelling and grade interpolation, block model validation, classification of Mineral Resources, assessment of reasonable prospects for eventual economic extraction, and selection of reporting assumptions.

RPA used Leapfrog software to create a three-dimensional geological model from core log information. Mineralization was modelled at an approximate modelling threshold of 0.3% Nb_2O_5. RPA determined that capping was not required due to the disseminated nature of the niobium and the overall grade distribution. Prior to grade interpolation, the assay data within each of the individual mineralized grade shells were combined into three-metre long downhole composites, based on the analysis of the predominant sampling length, the style of mineralization, and continuity of grade.

Grade estimation was carried out in two passes; the first pass used full variogram ranges for the search, whereas the second pass ranges were enlarged to populate grades in the entire mineralized envelope.

The block model was validated using volumetric comparison of the blocks versus wireframes, a quantile-quantile ("QQ") plot of 2017 re-sampling versus historical data, comparison of block grades by the primary ordinary kriging estimator to results from inverse distance cubed and nearest neighbour estimates, swath plot comparisons, visual inspection of block grades versus composite grades, and statistical comparison of block grades to assay and composite grades.

1.10 Mineral Reserve Estimate

There are no current Mineral Reserves estimated for the Project.

1.11 Mining Methods

The James Bay Niobium Deposit is amenable to both open pit mining and underground mining methods given the proximity to surface and the value of the mineralization.

Three mining scenarios were studied in this Report:

- Scenario 1: Open Pit Only.
- Scenario 2: Open and Underground.
- Scenario 3: Underground Only.

The life-of-mine ("LOM") plan physicals summary is presented in Table 1.2.

Table 1.2: LOM Physical Summary

Life-of-Mine Physicals Summary	Scenario 1 (OP)	Scenario 2 (OP&UG)	Scenario 3 (UG)
Mining			
Open Pit			
Total Mined from Open Pit (kt)	198,143	24,317	
Waste Mined from Open Pit (kt)	127,299	14,553	
Strip Ratio (W:O)	1.80	1.49	
Mill Feed Mined (kt)	70,845	9,764	
Mined Grade (% Nb_2O_5)	0.44	0.51	
Contained Metal (000 kg Nb_2O_5)	311,895	49,530	
Underground			
Mill Feed Mined (kt)		43,929	53,625
Mined Grade (% Nb_2O_5)		0.51	0.51
Contained Metal (000 kg Nb_2O_5)		222,072	274,658
Total: Open Pit & Underground			
Mill Feed Mined (kt)	70,845	53,693	53,625
Mined Grade (%Nb_2O_5)	0.44	0.51	0.51
Contained Metal (000 kg Nb_2O_5)	311,895	271,602	274,658
Processing			
Total Mill Feed Processed (kt)	70,845	53,693	53,625
Head grade (% Nb_2O_5)	0.44	0.51	0.51
Contained Metal (000 kg Nb_2O_5)	311,895	271,602	274,658
Niobium Production (000 kg Nb)	164,100	142,900	144,508

The Open Pit Scenario shows the most advantages for various reasons including better nearby mining experience in open pit mining; simplicity of operation; source of borrow material for tailings storage facility construction; easier scalability of operation to future expansions. However, the Open Pit Scenario represents the largest footprint. The following information relates only to the selected Open Pit Only Scenario with the other options described in the other sections of the report.

The Open Pit Scenario generates 70.8 Mt of mill feed over the life-of-mine (“LOM”) and an average grade of 0.44% Nb_2O_5 sufficient for a 30-year life Project. The open pit is mined out by Year 24 of operations with processing continuing with stockpiled low-grade material. A full year of pre-production mining is planned where overburden and waste rock are mined to expose mineralized material and provide construction materials for the tailings storage facility embankments.

Mining of the James Bay Project consists of four nested pit phases. Waste rock will be disposed of in the primary dump adjacent to the pit. Overburden material will be dumped in a distinct overburden dump to be reused on closure. The open pit generates 21.2 Mt of overburden and waste over the LOM for an average LOM strip ratio of 1.8:1.

Mining will be carried out using conventional open pit techniques with 10 m benches. An owner mining open pit operation is planned with a peak mining rate of 10 Mtpy is targeted to feed the process plant at a rate of 2.4 Mtpy (6,600 tpd). Major equipment selected for the mine consists of two 7 m^3 hydraulic face shovels and one 8.2 m^3 front-end wheel loaders matched with a fleet of 64 t mine trucks.

Figure 1.1: Open Pit Scenario Mining Schedule

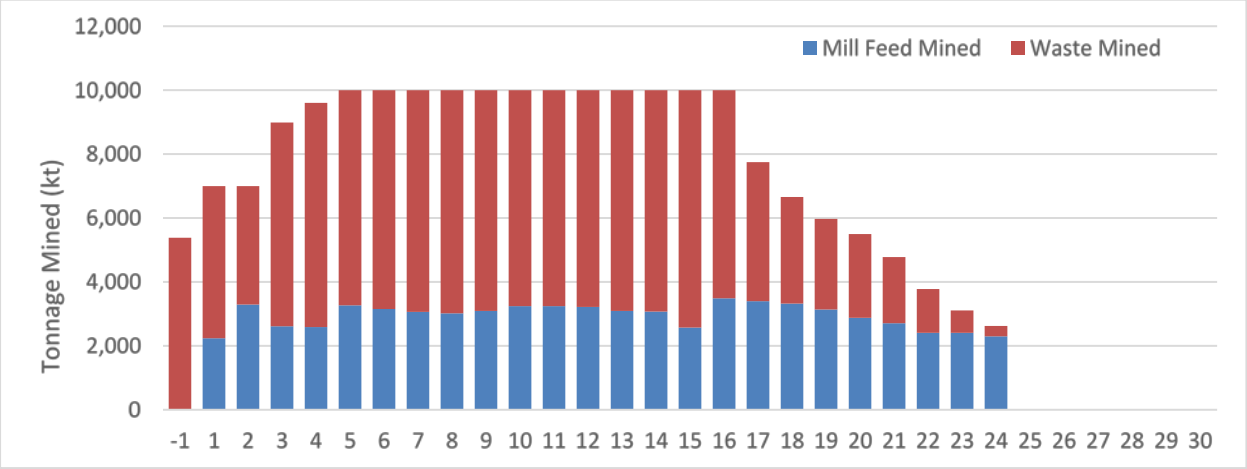


Figure 1.2: Open Pit Scenario Processing Schedule

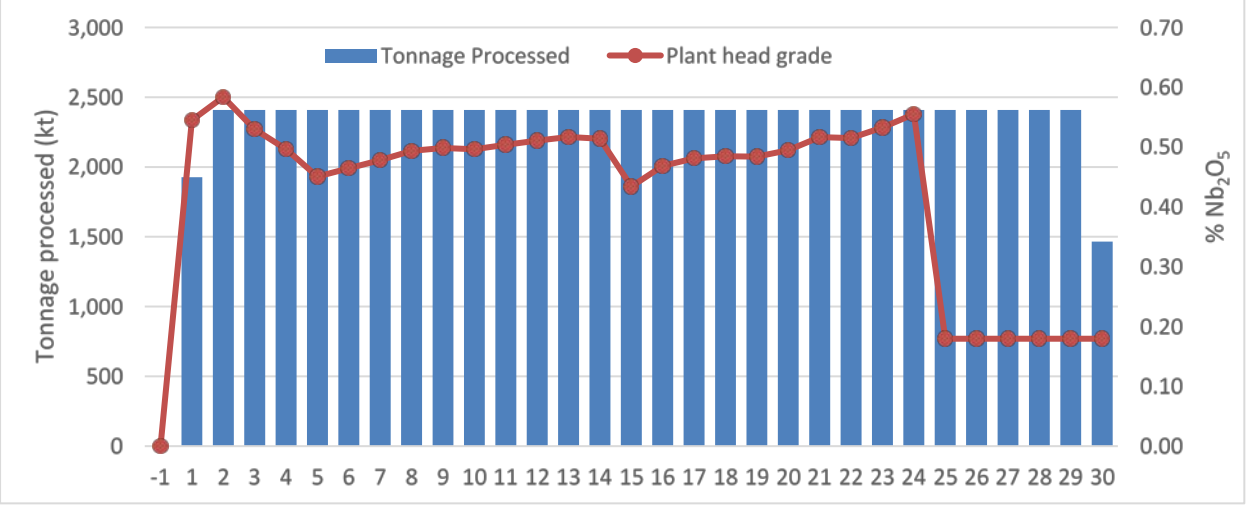
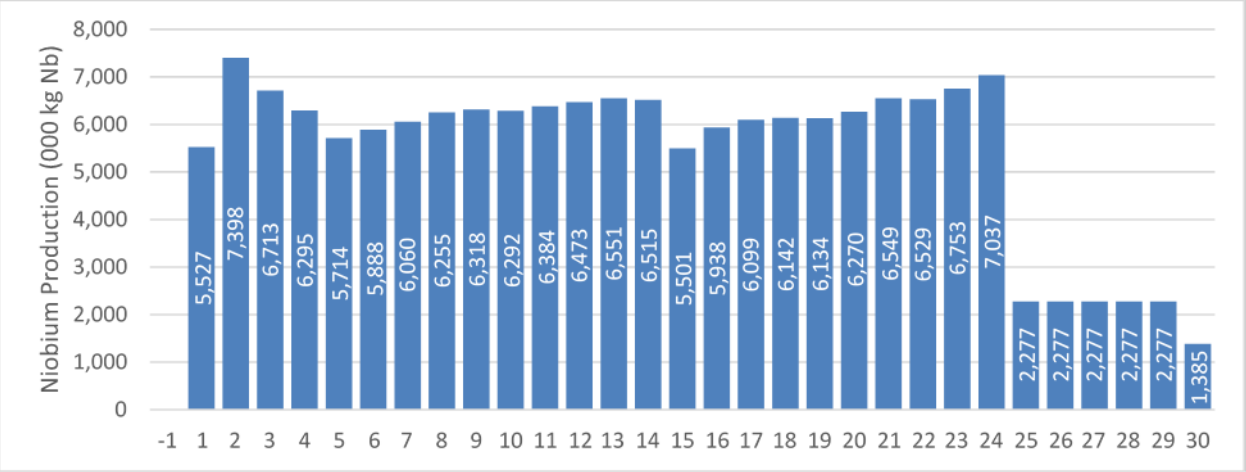


Figure 1.3: Open Pit Scenario Niobium Production



1.12 Infrastructure

The main infrastructure required to support mining and processing operations includes a permanent access road, power supply, assay laboratory, camp facility and support buildings such as a maintenance facility, warehouse, administration building and mine dry.

The site is currently inaccessible by road. The plan for accessing the site over the life-of-mine ("LOM") is to build a permanent all-season road to reach Moosonee. Three road alignment options were developed. The preferred option is an alignment along the east bank of the South Bluff Creek, crossing Moose Factory 88 Reserve to Moose River. From the east, a 6-meter-wide tunnel 4.2 km in length is planned to cross Moose River just south of Renison substation.

Five Nations Energy Inc. ("FNEI") retains the rights north of HydroOne Renison substation for power transmission/distribution. After discussions with FNEI, it is believed that the James Bay Project can obtain power from either FNEI directly or HydroOne through FNEI at the Renison Substation. This substation is approximately 45 km west of the James Bay Project. The Moose River crossing could use the planned tunnel or an aerial powerline. The powerline voltage is 115 kV. From the east bank of Moose River, the power line will follow the main access road. The power draw at site is estimated to 8 MW for the planned infrastructure.

A camp designed for 250 persons is planned. The camp facility includes kitchen, dining room and various other amenities. Most of the general services will be located in Moosonee and/or Cochrane. These services would include supply chain management, human resources, sales and shipping, accounting and other administrative functions. The assay laboratory would also be located in Moosonee.

A Reject Storage Facility is planned with embankments constructed from waste mined from the open-pit or quarried nearby. The approach is to stage the facility over the LOM.

The water balance approach for the Project will be to reclaim and re-use as much water as possible during operations. The general operating strategy assumes that the makeup water source will be used to supply water to the TSF prior to the start of operations so that there is enough water available for the mill. At this time the water supply is assumed from wells drilled on the property and will require further investigations.

1.13 Market Studies

Niobium is used in a variety of forms but the most important in terms of tonnage is standard-grade ferro-niobium, which has applications in steelmaking, most notably in high-strength low-alloy ("HSLA") and stainless steels. This market accounts for approximately 90% of niobium usage.

Niobium is a niche metal present in a variety of minerals, the most commercially important of which is pyrochlore which accounts for as much as 97% of global niobium supply. There are currently only three significant producers of pyrochlore: CBMM and China Molybdenum ("CMOC") in Brazil; and Niobec in Canada. All three convert their mine output to ferroniobium prior to sale, mostly into export markets and on a yearly contract basis.

Two factors are essentially impacting the ferroniobium demand: i) increase in overall steel production with an associated increase in production of high-value steels, and ii) a rise in the intensity of use of niobium in the steel industry.

James Bay Niobium production will be sold to markets around the world either directly or via agents/distributors. Sales in China are typically via a third party while sales in America or Europe are done directly with the consumers, each market has its particularity. Marketing fees are estimated at 1.5% of sales and includes transportation, warehousing, commissions, insurance, and related in-house staff.

Price forecasts are based on the outlook for the world economy, steel production trends (higher demand for high-value steel), exchange rates, intensity, investment by CBMM in expanding capacity. Considering the aforementioned, a ferroniobium price of USD 45/kg Nb is forecasted once the Project will begin production.

1.14 Environmental Studies, Permitting and Social or Community Impact

There have been no specific environmental studies to date. Some water and grab samples associated with exploration drilling have been undertaken to provide background data for Moose Cree First Nation. It is anticipated that baseline environmental data will be collected starting in Q2-2021 and beyond.

It is anticipated that both the federal Impact Assessment Agency and the provincial Ministry of the Environment, Conservation and Parks will be the lead at the environmental assessment phase. Significant permits will also be required to operate. The majority of permits to operate will be under provincial jurisdiction.

The future environmental baseline and engagement with the Indigenous and local communities will speak to the social and community impact. Generally, the impact will be deemed to be positive as there will be many businesses, employment and contracting opportunities generated in the region by the Project.

1.15 Capital and Operating Cost Estimate

The capital expenditures ("CAPEX") for Project construction, including processing, mine equipment purchases, pre-production activities, infrastructures and other direct and indirect costs is estimated for each Scenario as summarized in Table 1.3. The total initial Project CAPEX including a 25% contingency is estimated at CAD 510M, CAD 482M and CAD 579M respectively for Scenarios 1, 2 and 3. Sustaining CAPEX is required during operations for tailings storage facility expansion, additional equipment purchases, mine equipment capital repairs and mine civil works.

Operating costs include mining, processing, converter costs, G&A services, marketing and royalty expenses. The LOM operating cost summary is presented in Table 1.4: Summary of LOM Operating Costs

The Open Pit Scenario (Scenario 1) results in lower unit operating costs of CAD 53.22/t of mill feed compared to CAD 72.46/t of mill feed for the Underground Only Scenario with the majority of the variation due to differences in mining costs. The average LOM unit cost per kilogram of niobium is USD 17.67/kg for Scenario 1 and higher for Scenarios 2 and 3, respectively at USD 20.03/kg and USD 20.68/kg. The aforementioned operating cost includes the marketing and royalty fees.

Table 1.3: Summary of LOM Capital Costs

Capital Expenditures Summary	Scenario 1 (OP)	Scenario 2 (OP&UG)	Scenario 3 (UG)
Initial CAPEX (CAD M)			
Infrastructure	133.58	133.58	112.61
Power & Electrical	31.49	31.49	31.49
Water & Tailings Management	31.41	13.58	20.48
Mobile Equipment	35.02	35.02	5.61
Mining Infrastructure	-	-	30.00
Process Plant (including converter)	69.99	69.99	69.99
Subtotal Directs	**301.48**	**283.64**	**270.18**
Construction Indirects	35.02	32.70	34.77
Owner's Costs	40.41	37.73	40.12
Pre-production & Commissioning	31.46	31.49	117.88
Contingency (25%)	102.09	96.39	115.74
Subtotal Indirects	**208.98**	**198.31**	**308.51**
Total Initial CAPEX	**510.45**	**481.95**	**578.69**
Sustaining CAPEX (CAD M)			
Water & Tailings Management	94.81	72.91	138.65
Mine Equipment & Infrastructure	115.36	214.39	194.21
Other	16.36	-	-
Contingency (25%)	56.63	71.82	83.22
Total Sustaining CAPEX	**283.16**	**359.12**	**416.08**
Total CAPEX	**793.62**	**841.07**	**994.77**

Table 1.4: Summary of LOM Operating Costs

Operating Cost Summary	Scenario 1 (OP)	Scenario 2 (OP&UG)	Scenario 3 (UG)
LOM Operating Costs (CAD M)			
Mining Costs	878	1,421	1,578
Processing Costs	1,034	785	784
Converter Costs	813	685	691
G&A Costs	708	537	536
Sub-Total Site Costs	**3,434**	**3,428**	**3,590**
Marketing Costs	144	125	127
Royalty Costs	192	167	169
Total Operating Costs	**3,770**	**3,721**	**3,885**
LOM Unit Operating Costs (CAD/t mill feed)			
Mining Costs	12.39	26.47	29.44
Processing Costs	14.60	14.62	14.62
Converter Costs	11.48	12.76	12.89
G&A Costs	10.00	10.00	10.00
Subtotal Site Costs	**48.47**	**63.85**	**66.94**
Marketing Costs	2.03	2.34	2.36
Royalty Costs	2.71	3.11	3.15
Total Operating Costs	**53.22**	**69.30**	**72.46**
LOM Unit Operating Costs (USD/kg)			
Mining Costs	4.12	7.65	8.40
Processing Costs	4.85	4.22	4.17
Converter Costs	3.81	3.69	3.68
G&A Costs	3.32	2.89	2.85
Sub-Total Site Costs	**16.10**	**18.45**	**19.11**
Marketing Costs	0.67	0.68	0.67
Royalty Costs	0.90	0.90	0.90
Total Operating Costs	**17.67**	**20.03**	**20.68**

1.16 Economic Analysis

A summary of the Project economic results is presented in Table 1.5. The Open Pit Scenario 1 generates the best project economics with an after-tax NPV 8% of CAD 1,008M and an after-tax IRR of 27.5% with a short payback period of 3.2 years.

Table 1.5: Project Economic Results and Summary

Project Economics Summary	Scenario 1 (OP)	Scenario 2 (OP & UG)	Scenario 3 (UG)
Production Summary			
Mine Life (yrs.)	29.6	22.4	22.4
Tonnage Processed (Mt)	70.84	53.69	53.62
Head Grade (% Nb_2O_5)	0.44	0.51	0.51
Niobium Production (M Kg Nb)	164.10	142.90	144.51
Cash Flow Summary (M CAD)			
Gross Revenue	9,600	8,360	8,454
Total OPEX	-3,770	-3,721	-3,885
EBITDA	5,830	4,639	4,568
Initial Capex	-510	-482	-579
Sustaining Capex	-283	-359	-416
Closure Costs	-32	-24	-21
Taxes	-1,422	-1,078	-1,017
After-Tax Results			
After-Tax Cash Flow (NPV 0%)	3,581	2,696	2,536
After-Tax NPV 8%	1,008	856	733
After-Tax IRR	27.5%	27.0%	21.6%
After-Tax Payback (Yrs.)	3.2	3.1	4.3

1.17 Interpretation and Conclusions

- Mineral Resources were estimated and classified by RPA following CIM (2014) definitions. At a cut-off grade suitable for an underground operation of 0.3% Nb_2O_5, Indicated Mineral Resources are estimated to total 29.7 Mt grading 0.53% Nb_2O_5 containing approximately 158 million kilograms

of niobium oxide. Inferred Mineral Resources are estimated to total 33.8 Mt grading 0.52 Nb_2O_5 containing 177 million kilograms of niobium oxide.

- RPA and GMS are of the opinion that there is excellent exploration potential to increase the Mineral Resource at depth with more diamond drilling.
- There are no mining reserves estimated for the Project.
- The deposit is amenable to open pit and underground mining methods or a combination of both. The three scenarios were studied with a preference for the Open Pit Scenario for reasons including: better economics, simplicity of operation, better suitability of skills in the region, scalability of operations to future expansions.
- The gravity testwork confirms the potential application of gravity separation to reduce the mass volume to be processed and the potential benefit for contaminant rejection. The overall final Nb_2O_5 recovery reached 78% for an open circuit without any recirculation and has been the value used in the PEA.
- The CAPEX is estimated at CAD 510M, CAD 482M and CAD 579M respectively for Scenarios 1, 2 and 3.
- The average LOM unit cost per kilogram of niobium is USD 17.67/kg for Scenario 1 and higher for Scenarios 2 and 3, respectively at USD 20.03/kg and USD 20.68/kg.
- All three scenarios studied yield positive economic results with the Open Pit Scenario being the highest with an NPV 8% of CAD 1,008M and an after-tax IRR of 27.5%

1.18 Recommendations

GMS recommends that further work be undertaken to compliment the current Preliminary Economic Assessment (“PEA”) and support a future Pre-Feasibility (“PFS”) and ultimately a decision to construct the Project. The cost for this phase of the work are estimated at approximately CAD 12M.

2. INTRODUCTION

G Mining Services Inc. (“GMS”) was retained by Niobay Metals Inc. (“Niobay”) to prepare an independent Technical Report on the James Bay Niobium Project (the Project), located in Cochrane District, northeastern Ontario, Canada. The purpose of this Technical Report is to disclose the results of a Preliminary Economic Assessment (“PEA”) for the Project based on the Mineral Resource estimate prepared by Roscoe Postle Associates Inc. (“RPA”), as disclosed in a NI 43-101 Technical Report issued on August 23, 2020. This Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

Niobay is a Montreal-based company formed in January 1954 as Exploration Minière du Nord Inc. and was subsequently known as MDN Inc. (“MDN”). Niobay is a reporting issuer in British Columbia, Alberta, Ontario, and Quebec. The common shares of Niobay trade on the TSX Venture Exchange and the company is under the jurisdiction of the *Autorité des marchés financiers du Québec*.

Apart from the Project, Niobay has staked three other properties hosting carbonatites elsewhere in Ontario and controls a property in Crevier Township, Lac St. Jean area, Quebec with the potential to host niobium-tantalum mineralization.

Currently, the major asset associated with the Project is a strategic land position covering prospective lithologies and structures. The Project hosts the James Bay niobium deposit (historically referred to as the Argor Deposit), which is at the resource definition stage, as well as a large land position which merits additional exploration.

Since acquiring the Property, Niobay has completed a re-logging program of historical core, check sampling, preliminary metallurgical test work on a composite sample taken from historical drill core, and an initial diamond drilling program consisting of seven holes totaling 3,090 m to test the depth extension of a high-grade portion of the deposit.

2.1 Qualified Persons

This Report was prepared by the following Qualified Persons (QPs):

- Ms. Dorota El-Rassi, P. Eng., Senior Geological Engineer, RPA.
- Mr. Louis-Pierre Gignac, P.Eng., Co-President, G Mining Services Inc.
- Mr. Paul Murphy, P. Eng., Project Manager, GMS.

- Mr. Pierre Pelletier, P. Eng., Independent Metallurgist Consultant.
- Mr. Carl Michaud, P. Eng., Underground Engineering Manager, GMS.
- Mr. Antoine Champagne, P. Eng., Open Pit Engineering Manager, GMS.

2.2 Sources of Information

The Report is based on a preliminary economic assessment prepared for Niobay and supporting memoranda and trade-off studies. This Report is also based on internal company reports, maps, and public information, as listed in Section 27 – References of this Report.

During the preparation of this Technical Report and site visits, discussions were held with personnel from Niobay:

- Claude Dufresne, P.Eng., President and CEO.
- Jacquelin Gauthier, P.Geo., Vice-President, Geology.

2.3 Site Visits and Scope of Personal Inspection

A site visit to the Project was carried out by Dorota El-Rassi, M.Sc., P.Geo., RPA Senior Geological Engineer, on August 21, 2020. Ms. El-Rassi inspected the historical shaft site and examined the recent drill hole locations. Paul Chamois, M.Sc.(A), P.Geo., Principal Geologist with RPA, visited the site previously on September 27, 2017. During the visit, Mr. Chamois inspected the historical shaft site, examined core from historical drilling programs stored in Moosonee, confirmed the local geological setting, investigated factors that might affect the Project, and collected core samples from historical drill holes for bulk density determination.

A site visit to the Project site was carried out by Mr. Paul Murphy, P Eng., GMS Project Manager, on August 21, 2020. The general Project area was inspected as well as the routing for the site access road and electrical infrastructure in Moosonee.

2.4 List of Abbreviations

Units of measurement used in this Technical Report conform to the metric system. Imperial units are used in descriptions of historical work, provided with metric equivalents. All currency in this Technical Report is Canadian dollars (“CAD”) unless otherwise noted.

Table 2.1: List of Abbreviations

Abbreviation	Description	Abbreviation	Description
µ	micron	kVA	kilovolt-amperes
µg	microgram	kW	kilowatt
a	annum	kWh	kilowatt-hour
A	ampere	L	litre
bbl	barrels	lb	pound
Btu	British thermal units	L/s	litres per second
°C	degree Celsius	m	metre
C$	Canadian dollars	M	mega (million); molar
cal	calorie	m^2	square metre
cfm	cubic feet per minute	m^3	cubic metre
cm	centimetre	MASL	metres above sea level
cm^2	square centimetre	m^3/h	cubic metres per hour
d	day	mi	mile
dia	diameter	min	minute
dmt	dry metric tonne	µm	micrometre
dwt	dead-weight ton	mm	millimetre
°F	degree Fahrenheit	mph	miles per hour
ft	foot	MVA	megavolt-amperes
ft^2	square foot	MW	megawatt
ft^3	cubic foot	MWh	megawatt-hour
ft/s	foot per second	oz	Troy ounce (31.1035g)
g	gram	oz/st, opt	ounce per short ton
G	giga (billion)	ppb	part per billion
Gal	Imperial gallon	ppm	part per million
g/L	gram per litre	psia	pound per square inch absolute
Gpm	Imperial gallons per minute	psig	pound per square inch gauge
g/t	gram per tonne	RL	relative elevation
gr/ft^3	grain per cubic foot	s	second
gr/m^3	grain per cubic metre	st	short ton
ha	hectare	stpa	short ton per year

Abbreviation	Description	Abbreviation	Description
hp	horsepower	stpd	short ton per day
hr	hour	t	metric tonne
Hz	hertz	tpy	metric tonne per year
in.	inch	tpd	metric tonne per day
in^2	square inch	USD	United States dollar
J	joule	USg	United States gallon
k	kilo (thousand)	USgpm	US gallon per minute
kcal	kilocalorie	V	volt
kg	kilogram	W	watt
km	kilometre	wmt	wet metric tonne
km^2	square kilometre	wt%	weight percent
km/h	kilometre per hour	yd^3	cubic yard
kPa	kilopascal	y	year

3. RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by G Mining Services Inc. ("GMS") for Niobay. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to GMS at the time of preparation of this Technical Report.
- Assumptions, conditions, and qualifications as set forth in this Technical Report.

For the purpose of this Technical Report, GMS has relied on ownership information provided by Niobay. GMS has not researched property title or mineral rights for the Project and expresses no opinion as to the ownership status of the Property.

Except for the purposes legislated under provincial securities laws, any use of this Technical Report by any third party is at that party's sole risk.

4. PROPERTY DESCRIPTION AND LOCATION

The Project is located in northeastern Ontario, approximately 40 km south of the town of Moosonee and approximately 640 km north of Toronto (Figure 4.1). It is located in the West of Marberg Creek Area and West of Flinch Lake Area, District of Cochrane, within 1:50,000 scale National Topographic System ("NTS") sheet 42I/15 (Meengan Creek). The Project consists of a single, heptagonally shaped mining lease which extends over a distance of 9.36 km in a north-north-easterly direction and covers an area of approximately 2,585.1 ha. The centre of the mining lease is located at approximately Latitude 50°50'69" N and Longitude 80°40'48" W. The centre of the currently defined mineralization is located at approximately Latitude 50°43'31" N and Longitude 80°34'46" W (Berger, Singer and Orris, 2009).

4.1 Land Tenure

The Property consists of a single mining lease (LEA-19586) covering an area of 2,585.15 ha and 306 single cell claims contiguous with Lease 19586 covering an additional 6,247.56 ha (Figure 4.2). The Property covers a total area of 8,832.71 ha.

The lease includes both mining and surface rights. An application with the Ontario Ministry of Energy, Northern Development and Mines ("MENDM") to renew the mining lease for another 10-year term has been signed by the Lieutenant Governor and will be registered in the next few weeks. Niobay has maintained the lease in good standing by making annual mining rental costs that total CAD 7,755.54.

A list of the single cell claims together with relevant tenure details including their tenure number, claim type, status, surface area and issue and expiry dates is contained in Appendix 1 of this Technical Report. The work required to renew all the single cell claims on their renewal date (January 30, 2022) totals approximately CAD 122,400.

On June 7, 2016, MDN, a predecessor company to Niobay, announced that it had signed a definitive property purchase agreement to acquire a 100% interest in the Project from Barrick Gold Corporation, James Bay Columbium Ltd., and Goldcorp Inc. (collectively the Vendors). In consideration for acquiring the Project, MDN agreed to make a one-time cash payment of CAD 25,000 and issue 5,000,000 common shares to the Vendors. MDN's interest in the Project is subject to a 2% net smelter return NSR royalty with MDN having the right to buy-back half the royalty (1%) at any time for CAD 2,000,000 (in constant 2016 dollars, subject to a cap of CAD 3,000,000). The Vendors retained the right to re-acquire a 51% interest in the Project for 2.5 times MDN's expenditures should one or more deposits containing at least two million ounces of gold and/or gold equivalent ounces of resources, in aggregate, be established. The

back-in right does not apply to the niobium content. On June 28, 2016, MDN announced that the transaction had closed. On September 2, 2016, MDN announced a name change to Niobay Metals Inc.

4.2 Mineral Rights

In Canada, natural resources fall under provincial jurisdiction. In the Province of Ontario, the management of mineral resources and the granting of mining rights for mineral substances and their use are regulated by the Ontario Mining Act and administered by the MENDM. Mineral rights are owned by the Crown and are distinct from surface rights.

4.3 Royalties and other Encumbrances

Except for the NSR royalty mentioned above, GMS is not aware of any other royalties due, back-in rights, or other obligations or encumbrances by virtue of any underlying agreements. However, Niobay granted Osisko Gold Royalties an option to purchase a 1% NSR for $2 M following a private placement completed in 2018.

4.4 Permitting

The MENDM is the principal agency responsible for implementing the provincial *Mining Act* and regulating the mining industry in Ontario. It is involved in the permitting and approvals process throughout the lifecycle of a mine. Operations would include other provincial ministries including, the Ministry of Environment, Conservation, and Parks, and Ministry of Natural Resources and Forestry.

Given the Property's early stage of development, permits, approval applications, and reporting requirements for MENDM may include:

- Aboriginal Consultation Reports.
- Exploration Permits.
- Exploration Plans.

4.5 First Nations Consultation

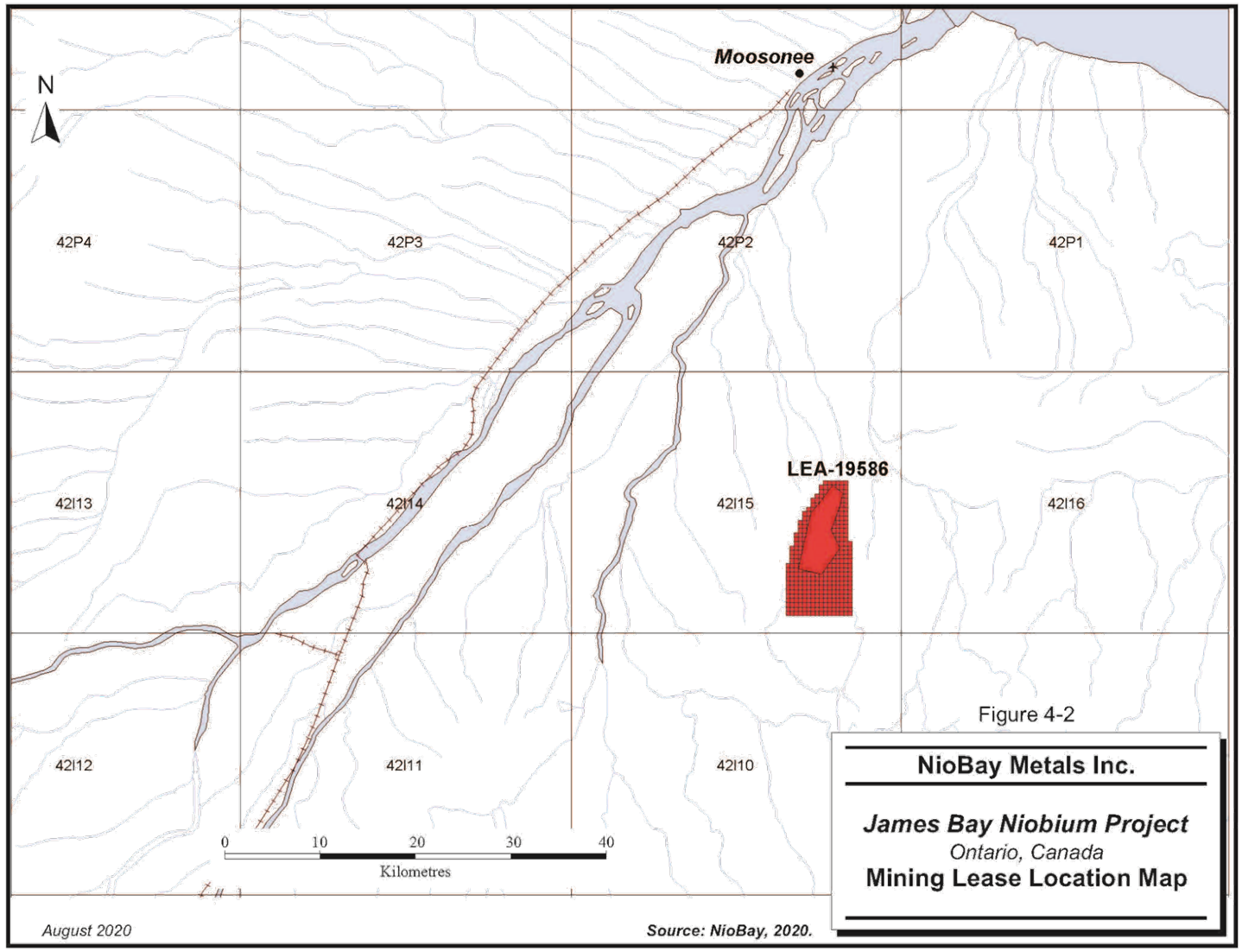
On December 24, 2019, Niobay announced that it had entered into a Protection Agreement ("PA") with the Moose Cree First Nation ("MCFN") with respect to the drilling program documented in this Technical Report.

RPA is not aware of any environmental liabilities on the Property. Niobay has all required permits to conduct the proposed work on the Property. RPA is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the Property other than the consultation process with the MCFN. Status of the consultation process is provided in Section 20 - Environmental Studies, Permitting, and Social or Community Impact.

Figure 4.1: Location Map



Figure 4.2: Mining Lease Location Map



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

5.1 Accessibility

The Project is located approximately 40 km south of the town of Moosonee which has a population of 1,725 according to the 2011 census. Moosonee is connected to Cochrane, Ontario by the Ontario Northland Railway which provides six day a week passenger service and twice weekly freight service. Moosonee also benefits from daily commercial flights from Timmins. The Wetum winter road links Moose Factory to the provincial road system at Otter Rapids seasonally, approximately 149 km to the south. Access to the Property is by helicopter year-round and bush roads from Moosonee during the winter months. Helicopters are available for charter in Timmins, Cochrane, and seasonally in Moosonee.

5.2 Climate

The Property lies within the James Bay Lowland ecoregion of the Hudson Bay Plain ecozone and is marked by brief and warm summers and cold and snowy winters. The average mean daily temperature in July ranges from 12°C to 16°C and in January is approximately -23°C to -25°C (Marshall and Schutt, 1999).

Table 5.1 illustrates the major climatic data for the closest weather station located at Moosonee, approximately 40 km to the north.

Table 5.1: Summary of Moosonee Climate Data

Description	Value
Mean January Temperature	-20.0°C
Mean July Temperature	15.8°C
Extreme Maximum Temperature	37.8°
Extreme Minimum Temperature	-48.9°C
Average Annual Precipitation	703.6 mm
Average Annual Rainfall	502.6 mm
Average Annual Snowfall	226.8 cm

Source: Environment Canada

Despite the harsh climatic conditions, geophysical surveying and diamond drilling can be performed on a year-round basis. Geological mapping and geochemical sampling are typically restricted to the months of May through to October.

5.3 Local Resources

Various services including temporary accommodations, medical services, a post office, fuel (gas diesel and propane) stations, and some heavy equipment and machinery shops are available in Moosonee. A greater range of general services are available in Cochrane and specialized services including trained manpower and contractors are available in Timmins. Moosonee is connected to the provincial power grid system by a 115 kV high tension line from the 182 MW Otter Rapids generating station located on the Abitibi River, approximately mid-way between Cochrane and Moosonee.

5.4 Infrastructure

There is no permanent infrastructure on the Property except for the historical underground excavations which consist of a 133 ft (40.54 m) shaft and a 100 ft (30.5 m) crosscut. The Property is located approximately 46 km east of the Ontario Northland Railway which links Cochrane to Moosonee. Figure 5.1 illustrates the infrastructure in the vicinity of the Project.

5.5 Physiography

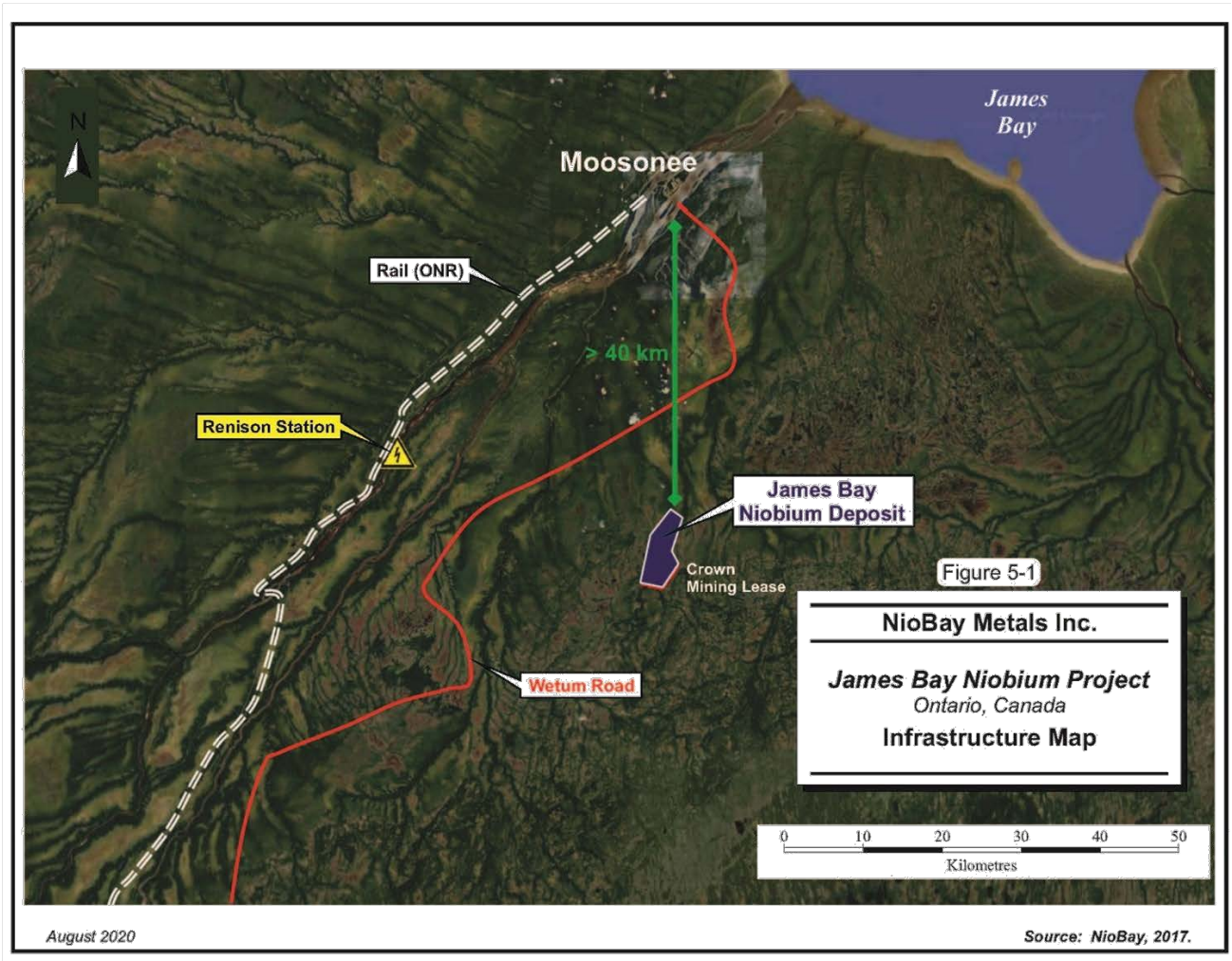
The area is poorly drained, flat and dominated by extensive wetlands. The elevation on the Property varies from approximately 35 MASL to 45 MASL.

Treed-open fen (28.6%) and treed-open bog (38.9%) comprise the primary vegetation found in the James Bay Lowland ecoregion. Coniferous forest is the predominant forest class (12.6%), with black spruce usually being the most dominant tree species, followed by sparse forest (7.6%). Well-developed forests are usually limited to alongside rivers and creeks. Open water covers 5.6% of the area (EEM Inc., 2017).

The region provides habitat for large mammals such as woodland caribou, moose, black bear, lynx, timber wolves, and small mammals including muskrat, ermine, weasel, marten, snowshoe hare, and wolverine. Migratory bird species returning to these lowlands annually to nest include Canada goose, black duck, oldsquaw, king eider, pintail, and whistling swan. Upland bird species such as willow ptarmigan, spruce grouse, snow owl, and ravens are year-round residents.

GMS is of the opinion that, to the extent relevant to the mineral project, there is a sufficiency of surface rights and water.

Figure 5.1: Infrastructure Map



6. HISTORY

6.1 Prior Ownership

Contiguous mining concessions #13758, #13759, and #13760 were held by Consolidated Morrison Explorations Ltd. ("Consolidated Morrison"), Argor Explorations Ltd. ("Argor Explorations"), and Goldray Mines Ltd. ("Goldray") as of 1965, respectively. Ownership of these concessions was eventually consolidated and held by Barrick Gold Corporation (60%), Goldcorp Inc. (9%), and James Bay Columbium/Consolidated Morrison (31%). In June 2016, Niobay entered into an agreement whereby it acquired a 100% interest in the Project, subject to a 2% NSR royalty.

The single cell claims contiguous with LEA-19586 were renewed and issued as of September 1, 2020 and are 100% owned by Niobay.

6.2 Exploration and Development History

The location of the early ground geophysical surveys and diamond drilling performed in the area of the current mining lease is not well documented. Some of the surveys and drill holes mentioned below may in fact not be located on the current mining lease.

Historically, little interest had been shown in the James Bay region of northeastern Ontario by mining companies. Some prospecting was done in the Partridge River area in 1929 and 1930. During the 1950s, a thorium prospect was discovered in Pitt Township, southwest of the Project. Later Selco Exploration Inc. ("Selco") made a reconnaissance survey along the French River for diamonds. A subsidiary of De Beers, Hard Metals, continued the work initiated by Selco, with negative results (Boyko, 1966).

The following is taken primarily from Stockford (1970) unless otherwise indicated.

In 1965, a consortium of companies including Argor Explorations, Consolidated Morrison, and Goldray were awarded three licences of occupation, each covering 64,000 acres (25,900 ha) and covering approximately 60 miles (96 km) in length by three to seven miles (5 km by 11 km) wide. The initial financial support for the Project was provided by Imperial Oil Enterprises Ltd. which earned a 60% interest in the licence areas and claim holdings in the vicinity.

From April to June 1965, Canadian Aero Mineral Surveys Limited ("Canadian Aero") was contracted to fly a combined magnetic-electromagnetic ("EM")-radiometric survey over an area including the licences of occupation held by Consolidated Morrison, Argor Explorations, and Goldray. A total of approximately

3,059 line-km were flown at an azimuth of N045°W and at a nominal flight line spacing of approximately 400 m. A total of 46 anomalous zones consisting of clusters of individual anomalies of various intensities and magnetic correlation were detected, including three high priority areas (Wagg, 1966).

From June to August 1965, as a follow-up to the airborne survey, a helicopter-supported geological mapping program was carried out by Argor Explorations on behalf of the concession holders to establish the regional geological controls and investigate the cause of specific magnetic anomalies.

From June to October 1965, Huntec Limited ("Huntec") was contracted to complete ground geophysical surveys over several airborne geophysical anomalies. Geophysical surveys consisted of vertical loop electromagnetic ("VLEM"), horizontal loop electromagnetic ("HLEM"), magnetic, and refraction seismic surveys. A total of 27 anomalies or anomaly complexes identified by these surveys were selected for diamond drill testing from 1965 to May 1969. A total of 49 holes totalling 32,041 ft (9,765.6 m) were drilled. A number of conductors were defined, two of which (the Alpha-A and Gamma targets on the Consolidated Morrison concession) were recommended for drilling. Additional geophysical surveying was recommended for several other targets prior to defining drill targets (Patterson and Lane, 1966).

One of the magnetic anomalies, the Alpha-B, was found to be caused by a niobium-bearing carbonatite complex. In 1966, 18 holes were drilled, and niobium mineralization was traced over a strike length of 7,800 ft (2,377 m). An additional 67 holes were drilled in 1967, bringing the number of holes drilled to 85 for a total of 47,625 ft (14,514 m) in outlining the deposit to a maximum depth of 900 ft (274.3 m). Niobium mineralization was encountered between sections 8+00S and 40+00N but the mineralized carbonatite remained open to the north (Stockford, 1972).

Fifteen soil test holes were drilled during this period to investigate the stability of the glacial overburden and Paleozoic sedimentary cover, which together average approximately 100 ft (30.5 m), over the Precambrian basement.

In 1968, a test shaft was sunk in the central part of the mineralized body and a 225-tonne bulk sample of niobium-bearing carbonatite was mined for metallurgical testing. The shaft was sunk to 133 ft (40.54 m) and a 100 ft (30.5 m) crosscut was driven.

In 1967, Canadian Bechtel Limited (Bechtel) completed a preliminary mining appraisal on the Argor deposit. The study considered three scenarios 1) mining the entire deposit by open pit, 2) mining the entire deposit by underground methods, and 3) mining the south end of the deposit by open pit followed by underground mining of the northwest limb. The assessment of the three scenarios was based on annual tonnage sufficient to produce 7.5 million pounds of Nb_2O_5 for 20 years with an overall mill recovery of 75%.

From June to October 1968, Huntec completed approximately 87.5 line-km of reconnaissance HLEM and ground magnetic surveys in the area of the Goldray #2 anomaly. Two distinct conductors were defined and recommended for diamond drilling (Patterson, 1969).

In 1969, McPhar Geophysics Ltd. ("McPhar") completed ground magnetic and EM surveys over five airborne anomalies on behalf of Argor Explorations. Two strong conductors corresponding to Anomalies "P" and "G" were defined and recommended for drilling.

In early 1970, Bergmann (1970) reported on ground magnetic and EM surveys completed by Prospecting Geophysics Ltd. ("Prospecting Geophysics") to recover airborne geophysical anomalies on behalf of Argor Explorations. Several conductors were identified and two (Anomalies "A" and "B") were recommended for drilling.

In late 1970, Questor Surveys Limited ("Questor") flew a combined magnetic and EM (Mark IV INPUT) survey totalling approximately 264 line-km on behalf of Argor Explorations over an area northwest of Kesagami Lake. A total of 13 conductors were detected, most of which were recommended for either VLEM or HLEM follow-up (De Carle, 1970).

In 1979, Bechtel updated the preliminary mining appraisal. No further work had been completed between 1979 and 2016 when Niobay acquired the Project.

6.3 Historical Resource Estimates

In July 1967, Bechtel estimated "geological ore reserves" on behalf of Argor Explorations. Between sections 6+00S and 18+00N, Bechtel estimated 61,560,000 short tons grading 0.52% Nb_2O_5 for the pyrochlore and columbite mineralized zones (Canadian Bechtel Limited, 1967).

This estimate was prepared prior to the implementation of NI 43-101 and is considered to be historical in nature and should not be relied upon. A qualified person has not completed sufficient work to classify the historical estimate as a current Mineral Resource or Mineral Reserve and Niobay is not treating the historical estimate as current Mineral Resources or Mineral Reserves. This historical estimate is a good indication of significant mineralization on the Property.

In 2018, RPA was retained to prepare an initial Mineral Resource estimate and a NI 43-101 Technical Report for the Project. The 2018 estimate is superseded by the current Mineral Resource estimate summarized in Section 14 – Mineral Resource Estimate of this Technical Report.

6.4 Past Production

With the exception of the bulk sample taken in 1968, there has been no past production from the Project.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

According to Sage (1991), the Project lies at the northern portion of the Kapuskasing Structural Zone ("KSZ"). The following description of the regional geology is taken from Sage (1991).

The KSZ extends from the east shore of Lake Superior northeast to James Bay. The KSZ is poorly defined along the east shore of Lake Superior but becomes better defined towards James Bay. The KSZ crosscuts an east-trending fabric within Archean rocks of the Superior Province and is sub-parallel to the Trans-Superior Tectonic Zone ("TSTZ").

The KSZ is characterized by a north-northeast-striking linear aeromagnetic pattern (400 to 600 gammas above regional background) and positive gravity highs (up to 20 mgal) (Innis, 1960; ODM-GSC, 1970; GSC, 1984). Numerous alkalic and carbonatite intrusions occur along this structure.

The KSZ has been interpreted as an upwarp of the Conrad Discontinuity (Wilson and Brisbin, 1965; Bennett et al., 1967; Thurston et al., 1977), a product of collision of the Churchill and Superior cratons in Paleoproterozoic time (Gibb, 1978), and as a deep transcurrent shear (Watson, 1980). More recently, Percival and Card (1983) have proposed that the KSZ is an east-verging thrust fault which has exposed an oblique section through 20 km of uplifted Archean crust. Granulite-facies rocks of the KSZ are juxtaposed against greenschist-facies rocks of the Abitibi Sub-province along the Ivanhoe Lake cataclastic zone. The KSZ is characterized by a high-grade gneiss terrain and grades westward into a central gneiss terrain and then into low-grade terrain of east-west-striking linear belts composed of supracrustal rocks.

In addition to the major fault which forms the east boundary of the KSZ, three major northeast-striking faults dip 60° to 70° northwest and are present within the uplift (Percival and McGrath 1986). These internal faults are west-side-down with displacement of 7 km to 10 km and result from a late tensional event that followed the compressional uplift (Percival and McGrath, 1986; Percival, 1987).

Northey and West (1986) have interpreted seismic refraction data to indicate a crustal thickness of 48 km below the KSZ, which thins slightly to the west and the east. The interpretation of seismic data is consistent with the upthrust of mid-crustal rocks along a west-dipping listric fault (Northey and West, 1986; Boland et al., 1988).

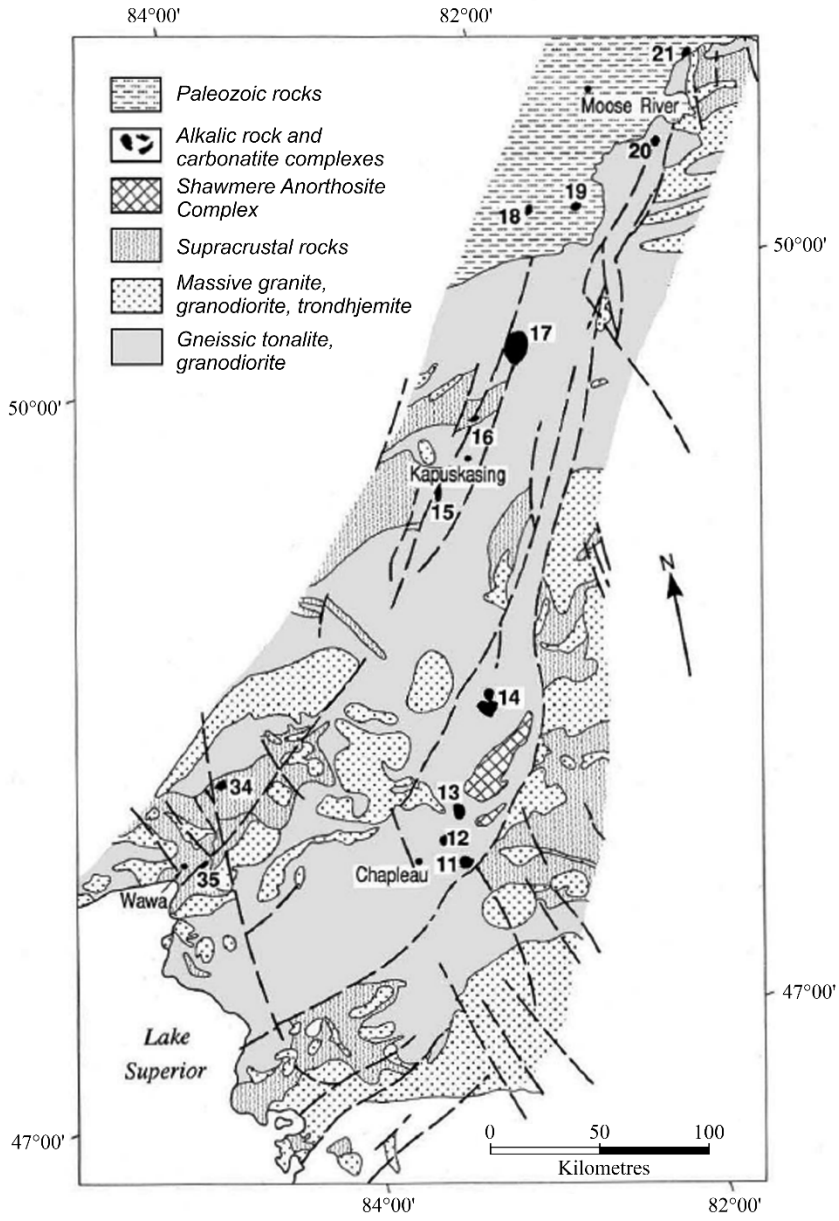
The precise age of the KSZ is uncertain (Watson, 1980; Percival and Card, 1983). The 1,800 to 1,900 Ma carbonatite intrusions are exposed at a high structural level and are enclosed within high-grade deep level

gneisses. The KSZ has therefore formed prior to the 1,800 to 1,900 Ma carbonatite intrusion event, however, the KSZ remains seismically active (Forsyth and Morel, 1982).

Alkaline magmatism along the KSZ took place during the Penokean and Grenville-Keweenawan orogenic events and in the Neoproterozoic to Early Cambrian and Jurassic periods. Wooley (1989) has documented the worldwide spatial and temporal distribution of carbonatite with doming, linear structures, and orogenesis.

Figure 7.1 illustrates the regional geology of the KSZ and indicates the location of the Project (indicated as the Argor Carbonatite Complex #21).

Figure 7.1: Regional Geology



NOTE: *Kapuskasing Structural Zone (KSZ) and location of alkalic rock and carbonatite intrusions (geology modified from Card, 1982).* ***11****-Lackner Lake Alkalic Complex;* ***12****-Borden Township Carbonatite Complex;* ***13****-Nemegosenda Lake Alkalic Complex;* ***14****-Shenango Township alkalic rock;* ***15****-Cargill Township Carbonatite Complex;* ***16*** *Teetzel Township Carbonatite;* ***17****-Clay-Howells Alkalic Complex;* ***18****-Hecla-Kilmer Alkalic Complex;* ***19****-Valentine Township Carbonatite Complex;* ***20****-Goldray Carbonatite Complex;* ***21****-Argor Carbonatite Complex;* ***34****-Herman Lake Alkalic Complex;* ***35****- Firesand River Carbonatite Complex.*

Figure 7-1

NioBay Metals Inc.

James Bay Niobium Project
Ontario, Canada
Regional Geology

August 2020

Source: Ontario Geology Survey, Special Volume 4 Part 1, 1991.

7.2 Local Geology

The following description of the local geology is abridged from Sage (1988).

The James Bay niobium deposit is hosted by the Argor Carbonatite Complex ("the Carbonatite Complex") and occurs in the northern portion of the KSZ of the Superior Province. Rocks in the general area are characterized by granulite facies rank gneisses and a pervasive north- to northeast-trending fault pattern (Bennett et al., 1967).

The Carbonatite Complex is overlain by approximately 9 m of overburden and 21 m of Lower Devonian rocks of the Sextant Formation consisting of poorly bedded sandstone, mudstone, siltstone, and loosely cemented conglomerate (Stockford, 1972).

7.2.1 Early Precambrian (Archean)

Gneiss

In proximity to the Carbonatite Complex, Stockford (1972) reported the presence of garnet-hornblende-feldspar gneiss and augen gneiss. The gneisses contain up to 20% quartz, up to 50% plagioclase, 10% to 15% biotite, 10% to 20% hornblende, 5% sericite, and 5% carbonate (Stockford, 1970). Stockford observed that nepheline-bearing rocks and evidence of fenitization are absent. The reported presence of carbonate and possibly sericite in the gneisses suggests some weakly developed metasomatic activity.

Twyman (1983) reported the presence of fine to medium grained, light grey syenitic dykes with a phaneritic to porphyritic-phaneritic texture. The feldspars are microcline perthite or orthoclase and albite. Calcite occurs interstitially to the feldspars. The mafic minerals reported by Twyman include biotite, pyroxene (salite to sodic salite), and local arfvedsonite. The relationship of the syenite to the carbonatite is unknown, but Twyman (1983) considered it to be older than the complex.

Gabbro Dykes

Stockford (1972) reported that gabbroic dykes cutting the gneisses east of the Carbonatite Complex appear to have been crushed and metamorphosed along with the gneisses. He reported that the dykes consist of plagioclase, feldspar, hornblende, augite and garnet, with accessory apatite, sphene, quartz, chlorite, calcite, pyrite, and magnetite. Stockford (1970) reported that the mode of the gabbro dykes is 30% to 35% plagioclase, 30% to 40% hornblende, 10% to 15% augite, and up to 25% garnet. A lamprophyre dyke was reported in drill core by Stockford (1972), although he stated that lamprophyres are uncommon in the area.

7.2.2 Middle Precambrian (Protozoic) – Argor Carbonatite Complex

Pyroxenite-Hornblendite

This phase of the Carbonatite Complex is not well represented in the drill core examined. Hornblende is dominant over clinopyroxene and the pyroxenite and hornblendite appear to grade into each other. While some hornblende is undoubtedly primary, other hornblende clearly formed as a replacement of the clinopyroxene. The pyroxenite and hornblendite are likely part of the same lithologic unit and part of the same magmatic phase. Because of the intimate relationship between the two rock types, they are discussed as one unit. Stockford (1972) described the unit as pyroxenite and he also interpreted the hornblendite to have formed by alteration of the pyroxene. Pyroxenite-hornblendite drill core is black to dark green in colour.

In thin section the rock is fine to coarse grained, massive, inequigranular seriate, allotriomorphic, with straight to curved grain boundaries. Several specimens have a weakly developed granoblastic texture and several appear to have been weakly deformed.

Silicocarbonatite

The limited number of silicocarbonatite specimens examined appear to be possible cumulate phases of the carbonatite magma, however, some may be reaction products between sovite and other rocks. Drill core representative of this lithology is grey in colour or mottled grey and black.

Samples from the western side of the complex were described by Twyman (1983) as hybrid rocks with mylonitic fabric and are likely silicocarbonatite. Twyman investigated the possibility of a genetic link between carbonate and silicate magmas. On the basis of observed mineral disequilibrium, Twyman concluded the silicocarbonatite rocks examined by him were the result of assimilation or reaction of carbonatite magma with the enclosing rocks. Microprobe analyses of biotite indicated Al content intermediate between silicate and carbonatite rocks; microprobe analyses of amphibole indicated high Fe content which would have resulted from reaction with a carbonatite magma.

Sovite and Minor Rauhaugite

Sovite contains in excess of 50% carbonate. The vast majority of the diamond drilling completed on this complex intersected this lithology since it is host to the niobium mineralization. Rocks of this grouping are grey to white in colour. No attempt was made to estimate the dolomite-calcite ratios in these rocks.

Twyman (1983) reported dolomite (rauhaugite) interstitial to calcite (sovite), implying that both carbonates crystallized simultaneously or that dolomite formed somewhat later. Two types of dolomite were reported by Twyman (1983): (1) a lineated variety composed of clear, anhedral to subhedral, elongated, interlocking tabular crystals; and (2) a fine-grained recrystallized variety with irregular grain boundaries. The lineated variety is interlayered with the sovite and the fine-grained variety occurs along intrusive contacts.

Microprobe analyses of carbonate, presented by Twyman (1983), indicate that dolomite at the Carbonatite Complex has a Mn enrichment trend while dolomite at the Cargill Complex displays a constant Mn content. At both complexes, the carbonate contains minor Mg, Fe, Mn and Sr, while the Argor rocks are richer in Fe. Twyman reported that the Fe enrichment trends in dolomite are compatible with chemical variations in associated amphiboles and biotites.

The minor element trends in dolomite within sovites and rauhaugites are similar, indicating that the sovites and rauhaugites formed at the same time (Twyman, 1983).

Stockford (1972) sub-divided rocks in this group into five sub-units based on texture, colour, and calcite-dolomite ratios. Stockford's (1972) efforts were directed to close identification of the pyrochlore-bearing carbonatite. While the pyrochlore is the essential economic mineral, it is nonetheless a minor accessory phase within the rock. Stockford (1972) places the pyrochlore content at 1.0% for the potentially economic portion of the carbonatite. Texturally or mineralogically, there is no difference between sovite with and without pyrochlore.

Twyman (1983) reported the presence of olivine sovite and provided a chemical analysis of this rock type. The olivine sovite occurs only on the western side of the complex, does not occur with dolomite, and has been interpreted to be an early phase of the carbonatite magma.

Pyrochlore occurs as anhedral to sharply euhedral grains that commonly contain rounded, irregular grains of carbonate. The mineral is red-brown to grey in colour. Its high relief and isotropic nature make it relatively easy to distinguish from other minor mineral phases. The mineral often appears crudely zoned from red-brown cores to grey rims, however, the reverse zoning pattern has also been observed. The zonation implies compositional variation.

7.2.2.1 Geochronology

The Carbonatite Complex has been dated at 1,655 Ma by K-Ar isotopic techniques on biotite (Gittins et al., 1967). Most Middle Precambrian (Proterozoic) complexes have isotopic ages of 1,800 Ma.

7.2.2.2 Metamorphism

The Carbonatite Complex displays all variations, from a well-developed granoblastic texture with curved grain boundaries typical of carbonatite rocks, to one that is very fine to fine-grained recrystallized with serrate to lobate grain boundaries. The Carbonatite Complex is relatively undeformed.

Stockford (1972) reported the absence of fenitization presumably on the basis of the absence of green to blue-green soda-iron amphiboles and pyroxenes within the enclosing gneissic wall rocks. The presence of carbonate, the alteration of pyroxene to amphibole and amphibole to biotite-phlogopite in the pyroxenite rocks, is interpreted to represent metasomatism of the pyroxenite by alkali-rich, aqueous, carbonate-rich fluids similar, if not identical, to a fenitizing liquid. The pyroxenite is, however, an early crystallizing phase of the Carbonatite Complex and not part of the much older wall rock assemblage.

Stockford (1972) reported the presence of sodic amphiboles in a mylonitic gneiss at the pyroxenite-gneiss contact on the east side of the complex. These sodic amphiboles may be indicative of more typical fenitization developed within restricted zones of greater permeability. Carbonate along fractures in the gneisses was reported by Stockford (1972) to occur for distances of up to a few hundred metres from the carbonatite contact. This is considered by Sage (1988) to be due to either the effect of a fenitizing liquid in a distal setting, or to a fenitizing fluid at relatively low temperatures, in essence, under hydrothermal conditions.

During the drilling and relogging campaigns, Joly (2020), suggests that the carbonatite is mostly undeformed body, with magmatic layering. Most brecciated zones mentioned in Stockford (1972) appear to be magmatic breccias showing magmatic foliation.

7.2.3 Structural Geology

7.2.3.1 Regional Setting

The Carbonatite Complex is located in the north portion of the KSZ of the Superior Province of the Canadian Shield. This KSZ is characterized geophysically by a north northeast-trending zone of gravity highs and pronounced linear aeromagnetic trends (Innes 1960; ODM-GSC 1970). This anomalous gravity zone has been interpreted to be due to an upwarp in the Conrad discontinuity caused by major regional faulting and the formation of a complex horst structure (Wilson and Brisbin, 1965; Bennett et al., 1967). This upthrown block is characterized by rocks that have locally reached the granulite facies rank of metamorphism (Bennett et al., 1967; Thurston et al., 1977).

The KSZ has been interpreted by Percival and Card (1983) to be an oblique section through 20 km of the Archean crust. The structural zone has been uplifted along a northeast-striking, northwest-dipping thrust fault. Along the east side of the structural zone, high-grade rocks are thrust against low-grade rocks. To the west, the high-grade rocks grade into low-grade rocks over a distance exceeding 100 km. Thrusting has exposed the Conrad Discontinuity within the KSZ (Percival, 1986).

7.2.3.2 Local Structures

Stockford (1972) indicated that the Carbonatite Complex was emplaced into a series of northeast- and north-trending faults. The enclosing gneisses are described as mylonitic or augen gneisses and minor faulting along the east contact has been reported (Stockford, 1972). Stockford (1972) indicated the Carbonatite Complex to be a dyke-like body with a long axis striking north. More recently, Niobay geologists have noted that there are no significant faults at the contact or inside the Carbonatite Complex.

The intrusion has not been totally delineated in strike length. It is known to be approximately 1,440 m long and remains open to the north. The complex appears to have a minimum width of approximately 360 m in the area of pyrochlore mineralization. Pyrochlore mineralization has been identified to a maximum depth of 270 m. The zone of pyrochlore mineralization occurs in the southern portion of the complex and, in plan view, it covers a maximum area approximately 900 m long and 180 m wide.

The carbonatite displays a mineralogical variation which imparts a banding to the rock. In addition, the elongate minerals, amphibole and apatite, have their long axes subparallel to each other and define a lineation in the plane of the magmatic layering. The attitude of the banding is steep to vertical and contacts appear conformable (Stockford, 1972).

Stockford (1972) reported a zone of crushed dolomitic carbonatite (Zone 8), which is now interpreted to be a magmatic breccia located in the approximate centre of the carbonate phase of the complex. The zone consists of crushed and recrystallized carbonate in a turbid groundmass of carbonate, chlorite, and hematite. Stockford (1972) reported that accessory apatite, feldspar, pyrite, and pyrochlore-columbite are also present. Amphibole crystals within this rock unit are contorted and minor molybdenite has been recognized (Stockford, 1972).

Stockford (1972) has interpreted the deformation textures to be the result of the pulsating emplacement of the carbonatite magma as a crystal mush, now interpreted as a magmatic breccia.

Based on its re-logging of the historical core, Niobay is of the opinion that the carbonatite is undeformed and that the deformation textures reported in the literature are related to magmatic processes and events

(Gauthier, 2017, pers. comm.). According to Niobay, the carbonite complex is shaped as a sigmoid body most likely intruding the gneiss during a late state extensional period of the KSZ formation (Gauthier, 2020, pers. comm).

7.2.4 Property Geology

A more detailed description of the lithologies found in the immediate vicinity of the deposit are taken from Stockford (1972). A property geology map, showing the lithologies, is provided in Figure 7.2.

7.2.4.1 Pyroxenite (Unit No 1)

Pyroxenite forms metres wide dykes intruded into Carbonatite Complex and locally in its paragneiss/gneiss host rocks (Joly, 2020). The field terms, “pyroxenite” or “hornblendite”, have generally been used, although the original composition was probably pyroxenitic.

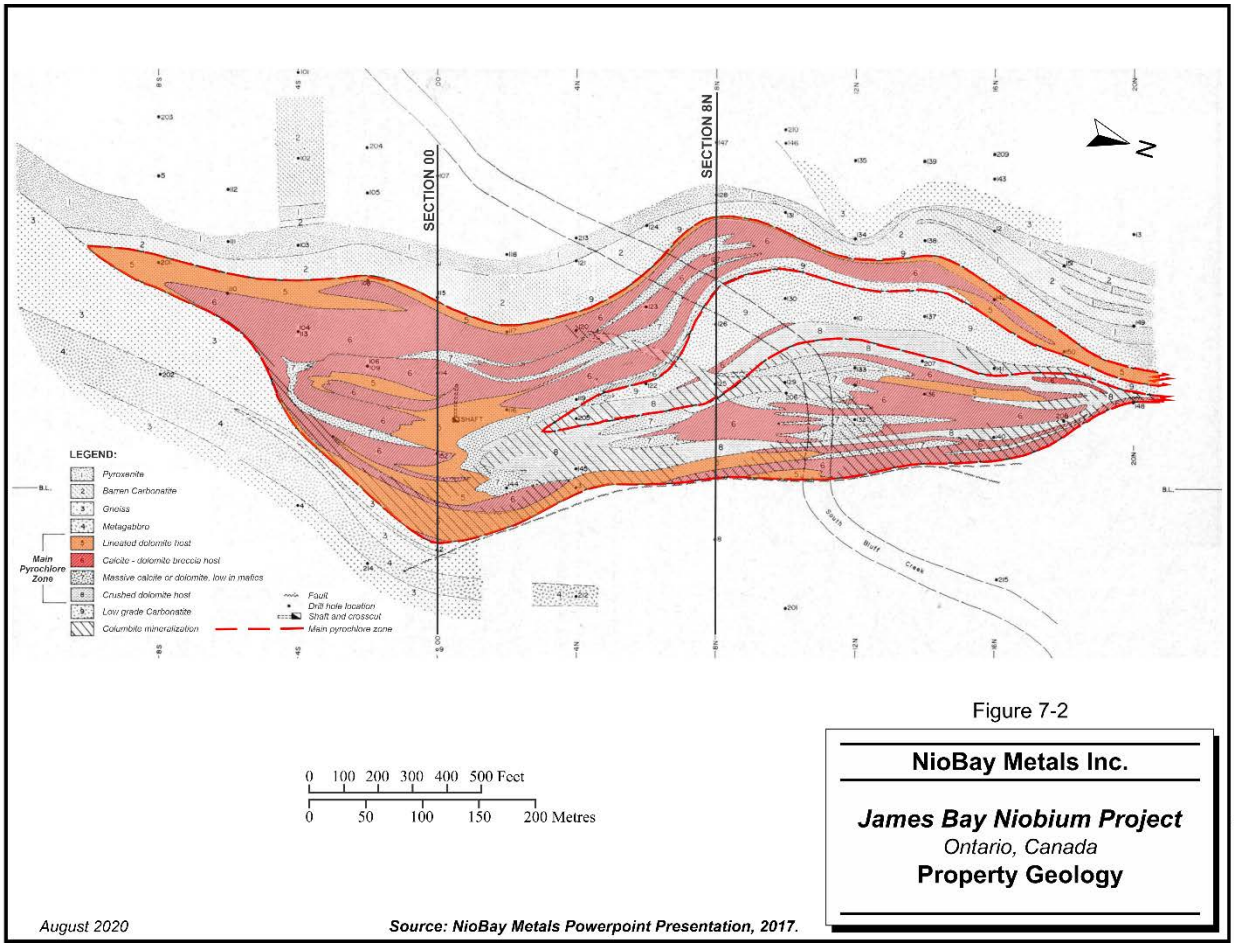
Mineralogically, it consists of hornblende, diopside and augite, coarse biotite after hornblende, and variable calcite. Accessory minerals include sphene, apatite, coarse titaniferous magnetite, and minor sulphides (1% to 2% pyrite and pyrrhotite with occasional chalcopyrite). Zircon is fairly common, particularly where the carbonate content increases. Thin section studies indicate that hornblende is often secondary after pyroxene, occurring as uralitic rims and cleavage replacements, although much of the amphibole is clearly primary as well.

Pyroxenite is interbanded with barren carbonatite along the west side of the complex. A narrow band also extends between mineralized carbonatite and mylonitic gneiss along the southeast boundary from 8+00S to 00+00. Boundaries of the pyroxenite have not yet been delimited to the north, south, and west of the niobium-bearing carbonatite.

At least one age of barren, calcitic carbonatite (Unit No. 2) has invaded the pyroxenite and consequently all combinations between massive pyroxenite and massive carbonatite and their mixed counterparts are seen in the complex.

Frequent inclusions and bands of hybrid material are apparent within the pyroxenite and barren carbonatite phases. These hybrid zones, which consist of pink or grey feldspar, biotite mica and carbonate, are presumed to be xenoliths of gneissic country rock, partially digested by the intrusive magma. Additional petrographic work is required to determine the detailed mineralogy of this type of material.

Figure 7.2: Property Geology



7.2.4.2 Amphibole-Biotite Carbonatite (Units No. 2 and No. 9)

Intrusion of the first carbonatite phase followed the pyroxenite. Brecciation and veining of the pyroxenite by carbonate support this theory. The primary constituent of the rock is calcite (generally 60% to 80%) with variable quantities of the following minerals: 5% to 10% hornblende, 5% to 15% pale green apatite, 2% to 10% titaniferous magnetite, 2% to 15% biotite, 1% to 2% pyrrhotite, up to 1% zircon, and occasional olivine.

The amphibole-biotite carbonatite is white to grey, coarsely crystalline and massive to well banded. Rolled crystal aggregates of carbonatite enclosed in a carbonate groundmass are thought to be indicative of pulsating movement during periods of intrusion and cooling.

Rock Unit No. 9 is interpreted as being a contact phase between Units No. 2 and No. 5 along the western boundary of the pyrochlore zone and in the centre of the intrusive north of section 4+00N. This phase appears to have been subjected to soda metasomatism and contains soda amphibole instead of hornblende

and phlogopite in place of biotite. The unit is distinctly pinkish or mauve-grey in colour, coarser grained and contains scattered grains of dark red-brown pyrochlore. This unit, which rarely grades more than 0.20% Nb_2O_5, could alternatively be a separate intrusive carbonatite phase.

Partial sideritization of the carbonate and chloritization of ferromagnesian minerals has occurred near the Precambrian surface.

7.2.4.3 Gneiss (Unit No. 3)

Only the eastern gneiss-carbonatite contact has been outlined to the south of section l0+00N; both east and west contacts have been intersected to the north. The gneiss is generally dark grey to black, consisting of quartz, plagioclase feldspar, hornblende and biotite-chlorite, with accessory amounts of sericite, carbonate, magnetite, zircon, sphene, apatite, epidote, and garnet. Minor pyrite and pyrrhotite have also been noted.

A narrow zone of chloritized gneiss extends along the eastern contact of the carbonatite from 00+00 to 16+00N. Minor faulting along this contact appears to have caused some fracturing of the eastern limb of the carbonatite in this area. There is local evidence of intense crushing and mylonitization of the gneiss, with local well-developed augen textures. It is unlikely that the augen textures would have been produced by intrusion of the carbonatite; it seems more probable that these structures were already in existence before intrusion.

Most of the carbonate in the gneisses occupies minute fractures extending for several hundred feet away from the carbonatite contact.

7.2.4.4 Metagabbro (Unit No. 4)

Dykes of gabbroic composition are found to the east of the carbonatite intrusion within the gneisses. Knowledge of these dykes is limited due to lack of detailed drilling; however, they generally appear to have been metamorphosed and crushed along with the gneisses.

The main constituents are plagioclase feldspar, hornblende, augite, and 5% to 2·5% garnet. Accessory minerals include apatite, sphene, quartz, chlorite, calcite, pyrite, and magnetite. Notably absent in the country rocks are lamprophyre dykes which are commonly associated with carbonatites, although one isolated occurrence was noted in a drill hole 2.4 km north of the main anomaly.

7.2.4.5 Pyrochlore Carbonatite (Units No. 5 to No. 8)

Rock Units No. 5 through No. 8, described in the following paragraphs, constitute the main pyrochlore-bearing carbonatite. The various rock units reflect the different textures and compositions of the carbonate hosts encountered in drill core. As such, they indicate mappable units containing fairly predictable niobium grades and possibly represent several different ages of carbonatite intrusion.

Thin sections indicate that the carbonate in Unit No. 5 is clear and unstrained, although distinctly lineated parallel to the dip and strike of the contacts. It is most likely that this lineated, dolomitic unit is the contact zone between Units No. 2 and No. 6, along the east and west margins of the pyrochlore zone and between Units No. 6 and No. 8 in the centre of the mass. The mineral constituents of the rock are 60% to 80% pale grey, medium- to coarse-grained dolomite, 5% to 10% sodic amphibole (riebeckite), 5% to 15% apatite, 5% to 10% phlogopite, 2% to 15% magnetite, 1% to 2% pyrrhotite, and 0.5% to 1% pyrochlore, with occasional epidote and zircon.

The host dolomite occurs as elongated crystals which, together with parallel orientation of riebeckite, apatite, and mica, impart the distinct wavy lineation to the rock.

Sideritization has taken place along the surface of the carbonatite. It is visualized as a layer of varying depth - up to three metres in the test shaft - which has been affected by groundwater and oxidation. The source of secondary iron appears to have been the ferruginous sediments above and/or magnetite in the dolomitic host rock. The dolomite has presumably been stained and partially replaced by siderite, particularly along fracture planes. There is no evidence in underground exposures to suggest a separate primary mass of siderite.

The pyrochlore content in this rock is usually uniform, with grades averaging between 0.3% Nb_2O_5 and 0.5% Nb_2O_5, and is generally the pale honey-coloured variety occurring as octahedra 0.1 mm to 4.0 mm in size. The heart of the intrusion appears to be a solidified mush of medium- to coarse-grained brecciated carbonatite (Unit No. 6), so called because of the frequent occurrence of rolled crystalline aggregates of carbonate rock within a carbonate groundmass. The unit consists of 50% to 70% grey to pinkish, mixed calcite and dolomite, 5% to 15% riebeckite, 5% to 15% phlogopite, 2% to 10% magnetite (hematite along east side of deposit), 5% to 20% apatite, 1% to 2% pyrrhotite, 0.5% to 3% pyrochlore, and up to 3% zircon. This rock type is the host for the best grade of columbium mineralization, with assays ranging up to a maximum of nearly 3% Nb_2O_5 over three metres. Segregation of accessory minerals into wavy bands is a feature of the rock, although the lineation is less distinct than in the dolomitic type (Unit No. 5). Pyrochlore is generally light to dark brown in colour and individual crystals frequently exhibit internal zoning. Pyrochlore is generally accompanied by riebeckite, although many exceptions to this rule have been found. Crystals

range in size from 0.2 cm to 2.0 cm and are easily identified by their resinous lustre and octahedral crystal form, sometimes modified by cube faces.

Almost massive concentrations of mica appear locally as rolled blocks or inclusions up to a few metres across. Where feldspar is present, these have been termed "hybrid zones" and are possibly remnant xenoliths of country rock, as seen in rock Unit No. 2.

Bands of medium to coarse-grained sovite (Unit No. 7), which are almost lacking in riebeckite and magnetite, have been identified as distinct units within rock Unit No. 6. The pyrochlore content is erratic and normally low in tenor. The massive sovite contains between 5% and 10% apatite and 0 to 15% coarse phlogopite.

A disturbed and crushed dolomitic zone (Unit No. 8), interpreted as being the final phase of intrusion, is the most difficult rock unit to correlate between drill holes. However, it could represent the central core of the original intrusive which has been squeezed, recrystallized, and hydrothermally altered. The mineral assemblage in the unit is quite different from other phases within the complex. It consists mainly of crushed, coarsely recrystallized dolomite enclosed in a dirty groundmass of fine carbonate, chlorite and, locally, hematite and is generally low in sodic amphibole content. Accessory minerals include apatite, feldspar, and pyrite, with erratic pyrochlore and/or columbite and scattered molybdenite.

Figure 7.3 and Figure 7.4 illustrate geological sections with composite intersection grades for Sections 0+00 and 8+00N, respectively.

Figure 7.3: Geological Section 0+00

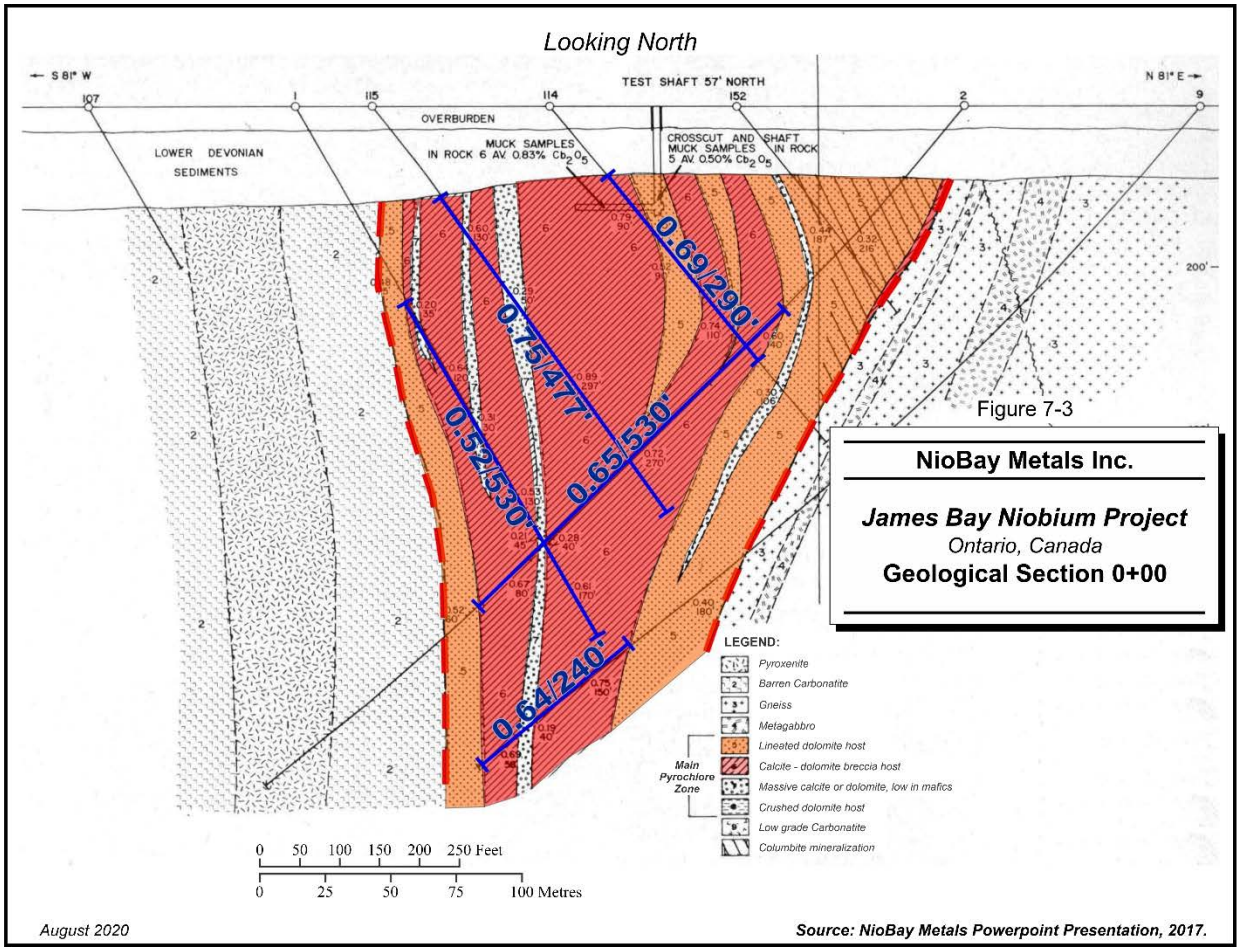
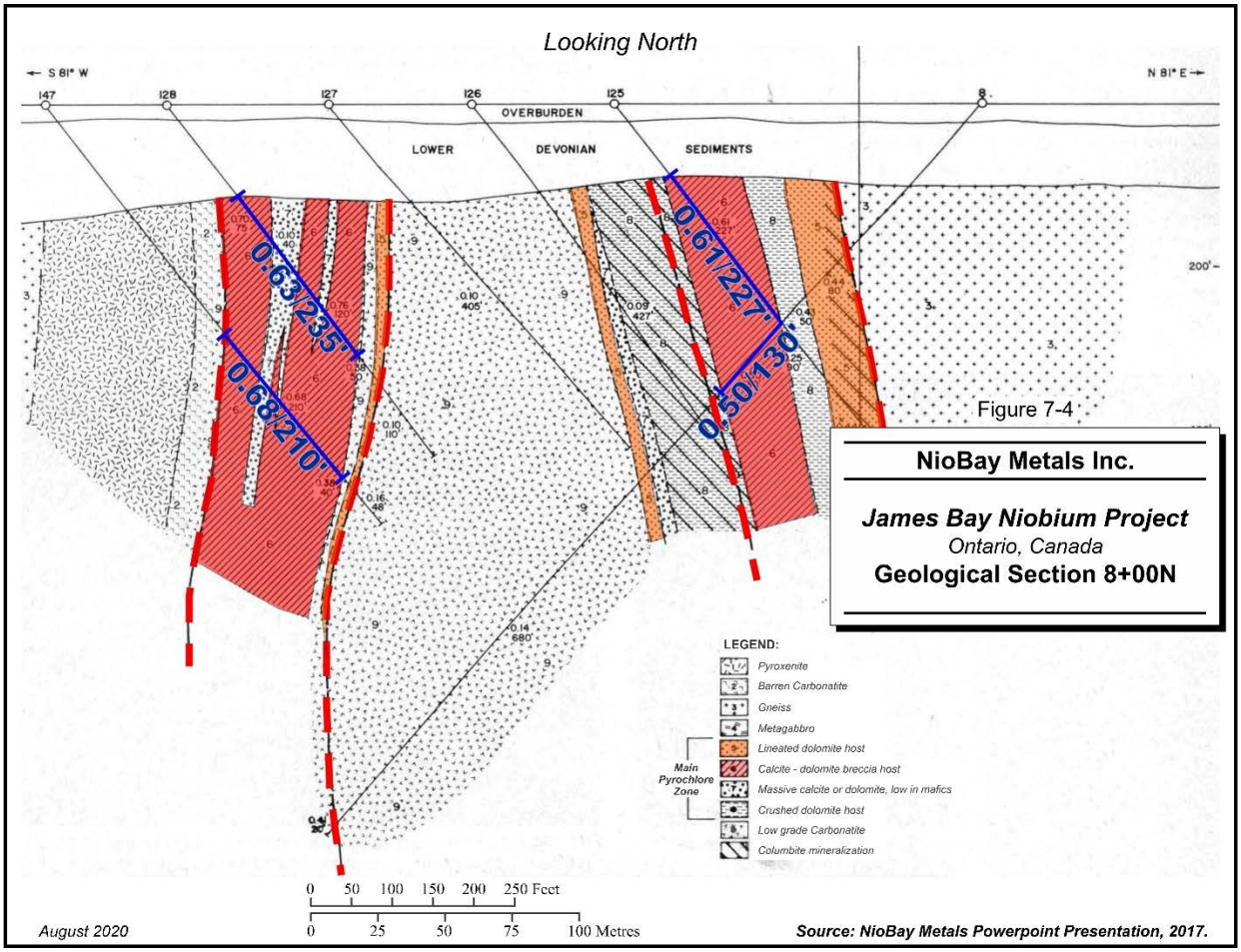


Figure 7.4: Geological Section 8+00N



7.3 Mineralization

The following is taken from Stockford (1972).

7.3.1 Pyrochlore

Pyrochlore, the main mineral of economic interest, occurs as discrete octahedral crystals in carbonatite. It is usually accompanied by, and sometimes enclosed in, apatite, riebeckite, magnetite, and phlogopite. Although these accompanying minerals seem to be essential to pyrochlore occurrences when considering the carbonatite as a whole, the columbium mineral does occur locally on its own, within a pure carbonate host.

Pyrochlore varies in colour from a pale honey to dark brown or reddish, and frequently displays spectacular zoning. Preliminary electron-microprobe studies by Dr. J. Gittins at the University of Toronto were carried

out on seven pyrochlore crystals to determine the difference in composition of various zones, if any. He found that the most striking features were the compositional simplicity, homogeneity despite the colour variation and the absence of tantalum. Some of the darker areas were assumed to be iron-rich, however, further work is required to verify this fact.

The microprobe studies indicate that the pyrochlore ranges in composition from 57% to 65% Nb_2O_5 and contains minor quantities of the following compounds: 0.8% to 2.0% TiO_2; 0 to 0.4% FeO; 0.01% to 4.0% Al_2O_3; 13% to 16% CaO; 6% to 14% Na_2O. Most of the crystals were found to contain Ce (0.5% Ce_2O_3) and half contained La (0.04% La_2O_3). Other elements searched for but not detected were U, Th, Zr, Ta, Ba, Eu, and Sr. Historical ore dressing tests on the core and bulk sample indicated that a concentrate averaging 64% $FeNb_2O_5$ could be produced from the deposit.

7.3.2 Columbite

Columbite, $FeNb_2O_6$, occurs along the eastern side of the deposit and in the transgressive crushed dolomite zone between sections 4+00N and 8+00N. The mineral totally or partially pseudomorphs pyrochlore and is typically earthy and dark brown to black in colour. Individual crystals of pyrochlore have been replaced by columbite in various ways - central cores, inner zones, outer rims, or total crystals. Columbite is not restricted to any one rock unit within the complex but appears to be associated with fracturing and hematitic alteration.

7.3.3 Apatite

The pale green, fluor-variety of apatite is universally present in all rock units within the carbonatite-pyroxenite complex. Usually, the best-grade bands of pyrochlore carbonatite also contain abnormally high quantities of apatite, while the converse is not true. The mineral occurs as fine- to coarse-grained, hexagonal prisms aligned parallel to the wavy foliation. Assays for phosphate content from crosscut samples indicated an apatite range of 6% to 10%.

7.3.4 Magnetite

Titaniferous magnetite, like apatite, is present in all units within the intrusive, and also in the gneissic country rocks to a lesser degree. It is most abundant in the pyroxenitic phase (Unit No. 1) and in the barren carbonatite (Unit No. 2); the magnetite content of the pyrochlore zone rarely exceeds 5%. Hematite or martite are dominant over magnetite along the east side of the deposit and in brecciated sections of the crushed dolomite zone in association with minor, late-stage quartz.

Submicroscopic hydrothermal hematite could also account for the reddish colouring observed in some of the mylonitic gneisses.

7.3.5 Sulphides

Up to 2% primary pyrrhotite is evenly distributed throughout all intrusive units with the exception of the crushed dolomite zone, where pyrite is dominant. Molybdenite occurs only in the crushed dolomite (Unit No. 8), as discrete, scaley grains, in sub-economic quantities. Both molybdenite and pyrite may have been primary constituents of the last intrusive phase, or they could represent a later stage of hydrothermal mineralization.

Chalcopyrite, of minor importance, occurs in the pyroxenite in association with pyrite and pyrrhotite.

8. DEPOSIT TYPES

The following is taken from Richardson and Birkett (1996).

Carbonatite-associated deposits include a variety of mineral deposits that occur both within and in close spatial association with carbonatites and related alkalic silicate rocks. Carbonatite-associated deposits are mined for rare-earth elements ("REEs"), niobium, iron, copper, apatite, vermiculite, and fluorite. By-products include barite, zircon, or baddeleyite, tantalum, uranium, platinum group elements, silver, and gold. In some complexes, calcite-rich carbonatite is mined as a source of lime to produce Portland cement.

Carbonatites are igneous rocks which contain at least 50% modal carbonate minerals, mainly calcite, dolomite, ankerite, or sodium- and potassium-bearing carbonates. Other minerals commonly present include diopside, sodic pyroxenes or amphiboles, phlogopite, apatite, and olivine. A large number of rare or exotic minerals also occur in carbonatites.

Carbonatites occur mainly as intrusive bodies of generally modest dimensions (as much as a few tens of square kilometres) and to a lesser extent as volcanic rocks which are associated with a wide range of alkalic silicate rocks (syenites, nepheline syenites, nephelinites, ijolites, urtites, pyroxenites, etc.). Carbonatites are generally surrounded by an aureole of metasomatically altered rocks called fenites produced by the reaction of country rocks with peralkaline fluids released from the carbonatite complex.

Carbonatite-associated deposits can be sub-divided into magmatic and metasomatic types. Magmatic deposits are formed through processes associated with the crystallization of carbonatites, whereas metasomatic deposits form by the reaction of fluids released during crystallization with pre-existing carbonatite or country rocks.

Many carbonatite-associated deposits are relatively small, in the order of tens of thousands to hundreds of thousands of tonnes, however, significant production of phosphate, niobium, and rare earth oxides is derived from larger, higher grade deposits in Brazil, Canada and South Africa, which vary greatly in size and grade.

Most carbonatite complexes occur in relatively stable, intra-plate areas. The regional distribution of these complexes is controlled by major tectonic features. About half the known carbonatites are located in topographic highs or domes and are bounded by zones of crustal-scale faulting (Wooley, 1989). Other major controls on carbonatite emplacement are major faults, anorogenic rifts, and the intersection of major faults. A few carbonatites are found near plate margins and may be linked with orogenic activity or plate separation. Because carbonatites generally occur in clusters or in provinces that display episodic magmatic

activity, physical and/or chemical properties of lithospheric plates may exert some control on their location and genesis (Wooley, 1989).

There appears to have been a gradual increase in carbonatite magmatism with time. Dates fall into groups that generally correspond to major orogenic and tectonic events (Wooley, 1989). Identified age groups include: 1) a mid-Proterozoic group (1,800-1,550 Ma) that corresponds to the Hudsonian and Svecokarelian orogenies in North America and Europe, respectively; 2) a mid- to late Proterozoic group corresponding to the Grenville orogeny (peak at ca. 1,100 Ma); 3) a group between 750 Ma and 500 Ma, and 4) a major period starting at 200 Ma. A few carbonatites of Archean age are known.

Carbonatites may consist of a number of intrusive phases with different textural and mineralogical characteristics. Early phases typically consist mainly of calcite, do not contain peralkaline pyroxenes or amphiboles, and contain associated apatite + magnetite ± pyrochlore mineralization. Later phases, which may contain dolomite, ankerite and siderite, in addition to calcite, are commonly enriched in pyrochlore. Many very late stage carbonatites contain only trace, or no, Nb mineralization and are enriched in primary REE-bearing minerals.

Economic mineralization in carbonatites is generally associated with plutonic carbonatite phases, not with lavas. Magmatic carbonatite deposits generally occur in small (3 km to 5 km) plug-like and cressentic bodies in composite plutons with coeval silica-undersaturated mafic and/or ultramafic rocks. Mineralization is commonly related to magmatic layering, and flow structures with the host rocks. The deposits are commonly groups of lenses or irregular ore shoots that, in plan, have crescent-shaped or annular forms. In section, these deposits generally have steep dips, parallel to the walls of the intrusive complex, and may extend to great depths.

Metasomatic carbonatite deposits typically have the form of 1) dykes and dilatant veins of ankerite or dolomite, with or without calcite; 2) thin hydrothermal veins; 3) stockworks, and 4) replacement bodies rich in calcite and dolomite or ankerite. Veins and dykes which locally form radial or annular patterns, commonly crosscut consanguineous fenitized alkaline lithologies and adjacent country rocks.

9. EXPLORATION

Niobay recovered all historical drill logs, collar survey data, historical assay certificates, and vertical drill sections with assays and geological interpretation from the archives of James Bay Columbium Ltd., one of the previous owners of the Property. All the existing historical drill core was stored in a secure building in Moosonee.

During the fall of 2016, Niobay undertook a program of core re-logging and check sampling to confirm the historical results on the Property.

Niobay selected two holes on each of six vertical sections located at 400 ft (~122 m) spacing for a total of twelve holes for the re-logging and check sampling study. The sections involved are Sections 400S, 00, 400N, 800N, 1200N, and 1600N. Niobay re-analyzed all of the remaining core, with the exception of a three-inch character sample for each sampled interval. Samples were typically 10 ft in length with some samples measuring not less than 5 ft. In some instances, previous sampling for metallurgical testing during the 1960s left only 25% of the split core available for re-assaying.

A total of 629 samples, exclusive of blanks and Certified Reference Material (CRM or standards), were taken for analysis

Table 9.1 illustrates the composite results of the original sampling and the results of the check sampling for the corresponding intervals.

Table 9.1: Core Sampling Results

Drill Hole	From (ft)	To (ft)	Length (ft)	1967 (% Nb_2O_5)	2017 (% Nb_2O_5)
66-2	117.0	915.0	798.0	0.55	0.56
incl.	333.0	915.0	581.5	0.64	0.65
incl.	395.0	695.0	300.0	0.74	0.79
66-10	140.0	800.0	660.0	0.44	0.41
incl.	540.0	580.0	40.0	0.85	0.93
incl.	690.0	740.0	50.0	0.66	0.65
66-12	255.0	393.2	138.2	0.56	0.50
	665.0	735.0	70.0	0.49	0.47
	945.0	115.0	205.0	0.41	0.36
67-101	820.0	1,120.0	300.0	0.57	0.51
67-103	160.0	600.0	440.0	0.54	0.53
incl.	260.0	400.0	140.0	0.77	0.74
67-115	150.0	617.0	467.0	0.76	0.74
incl.	320.0	617.0	297.0	0.89	0.85
67-119	150.0	586.0	436.0	0.39	0.38
incl.	300.0	380.0	80.0	0.55	0.54
67-121	250.0	570.0	320.0	0.50	0.39
incl.	358.0	560.0	202.0	0.64	0.61
67-125	113.0	548.8	435.8	0.49	0.48
incl.	113.0	350.0	237.0	0.60	0.59
67-135	450.0	560.0	110.0	0.49	0.46
67-142	400.0	564.0	164.0	0.38	0.38
67-147	350.0	610.0	260.0	0.62	0.60
incl.	370.0	480.0	110.0	0.80	0.74

The results of the check sampling program suggest that, while there is moderate variation over individual sample intervals, the overall average grades over larger intervals are comparable.

9.1 2020 Exploration Work

Except the drilling campaign no additional exploration work was carried on the Project.

10. DRILLING

10.1 Historical Drilling

The historical drilling is described in Section 6 - History.

10.2 Current Drilling

Niobay completed seven diamond drill holes for a total of 3,089.6 m between February 2 and March 12, 2020. Holes are oriented 081º with a dip of between -45° and -63° to the east. The drill holes ranged in length from 349 m to 504 m. Table 10.1 summarizes all the drilling completed on the Property to date.

Table 10.1: Summary of James Bay Drilling Programs

Year	Number	Metreage
1966	12	4,029.5
1967	67	9,200.4
2020	7	3,089.6
Total	**86**	**16,319.5**

Diamond drilling was contracted to Rouillier Drilling from Amos, QC. A joint venture between the Moose Cree First Nation ("MCFN") and Expedition Helicopters Inc. ("Expedition") from Cochrane, Ontario was chartered for staff and drilling equipment transport between Expedition's base in Moosonee and the drilling area, and between drill sites. All drilling used NQ (47.6 mm) core size. A 9/15-hour schedule was planned due to limited daylight, however, extended shifts, up to 22 hours, were occasionally required due to bad weather hampering the helicopter operation. A safety shack was installed near the drilling site.

MCFN required the drilling site to be thoroughly cleaned, with the mud collected and removed. Mud was collected in decanting tanks and water recycled for drilling operations.

Holes were partly cemented over the main carbonatite interval with VanRhoute plugs at the beginning and the end of each zone. The cement was hand mixed and pumped into holes. Twenty to twenty-six bags were required for each hole. Casings were removed or cut at the ground level, and a 4' peg was inserted in the hole.

Deviation measurements were taken by the drillers with a Reflex EZ-track device at 50 m intervals (single shots) down hole, and in continuous, from bottom to top (multi-shots) at every 15 m for holes NBY-20-E1 to NBY-20-E5 and every three metres for holes NBY-20-E6 and NBY-20-E7. Significant deviation errors, from 5° to 20°, in azimuth were caused by locally massive magnetite, mainly in carbonatite and pyroxenite units. Downhole azimuth measurements are consequently not dependable, and a gyroscope device should have been used. The issue did not affect the dip of the holes, which varies less than 1° between readings.

IOS Services Geoscientifiques ("IOS") was mandated to supervise drilling operations, spot and survey the hole location, log the core, and supervise the sampling process conducted by local workers (Joly, 2020).

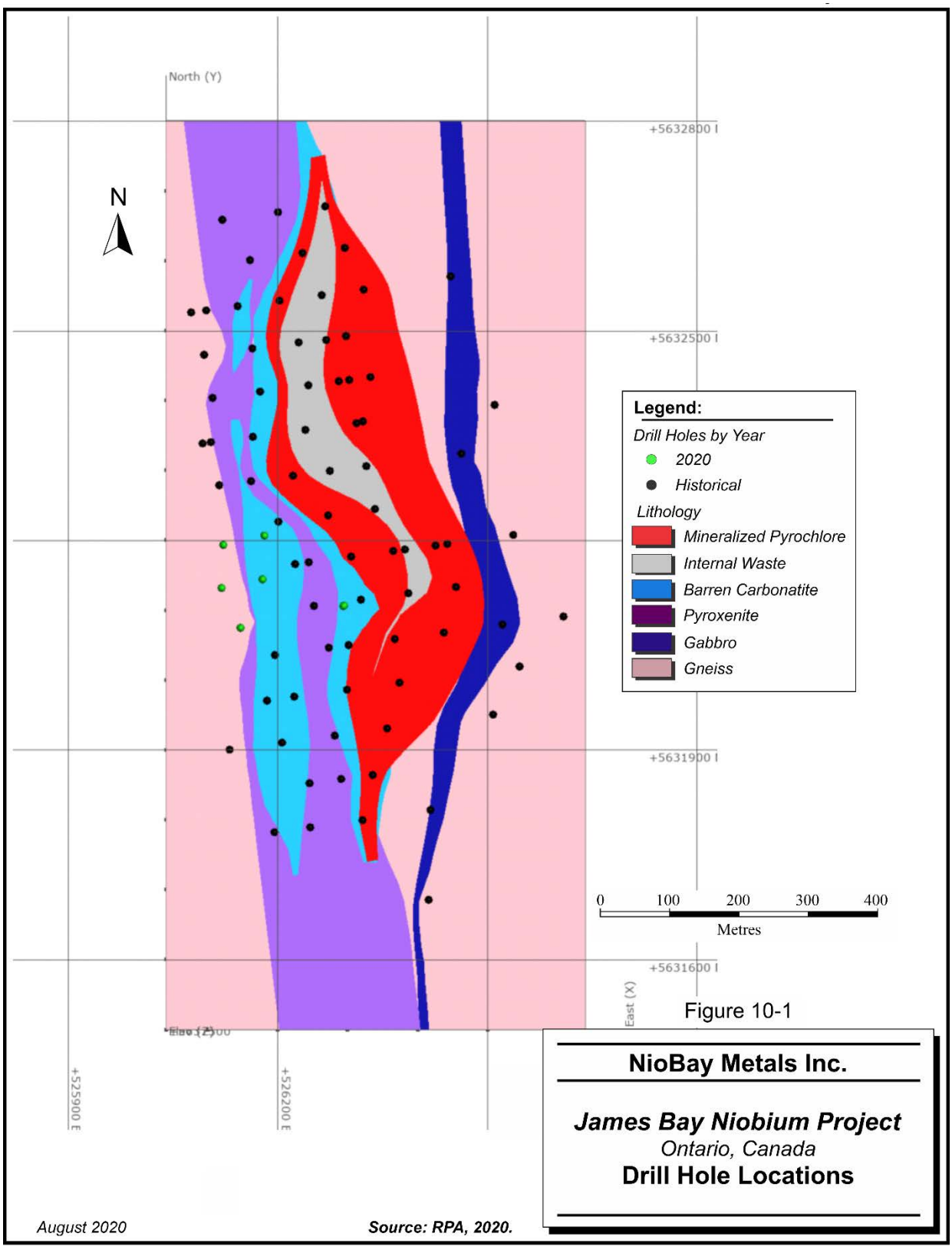
A geological and mineralogical description was completed according to Niobay's nomenclature established in 2016 from 1966 to 1967 holes relogging. GeoticLog8 software was used for data entry of core logging and technical measurements in MS Access database. Collar coordinates are in UTM WGS 84 Zone 17.

Entire core marked as carbonatite, and approximately three metres of the host rock units were sampled. The sample length was kept typically to three metres with the shortest length set at 0.5 m with respect to geological contacts, mineralization, and significant alteration.

Core boxes are stored in core racks in a shed near Niobay's core shack, in Moosonee. Hole numbers and footages are indicated on core boxes with black felt pen and on sides with aluminum tags. A slip with sample number was stapled in the core boxes at the beginning of each sample.

Figure 10.1 shows the location of historical and current drilling used in Mineral Resource estimation.

Figure 10.1: Drill Hole Locations



11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 2016 Check Sampling Program

The core from the historical drilling on the Property was continuously stored in a secure building in Moosonee.

Samples taken during the 2016 check sampling program were bagged on site and were assigned a unique sample number. The samples were first transported on two stretch wrapped pallets by Niobay personnel to Niobay's warehouse in Montreal where standards were inserted into the sample sequence. The samples were then bagged, stretch wrapped on pallets, and sent directly to SGS Minerals in Lakefield, Ontario, ("SGS Lakefield") by courier for processing and analysis. SGS Lakefield is Accredited Laboratory No. 184 and conforms with requirements of CAN-P-1579, CAN-P-4E (ISO/TEC 17025:2005).

At SGS Lakefield, core samples were dried and weighed, then crushed to 90% passing 2 mm, split into representative sub-samples using a riffle (or rotary) splitter, dry screened to -180 mesh and a 500 g sample pulverized to 85% passing 75 µm according to SGS Lakefield sample preparation code PRP91.

Samples were routinely analyzed for a suite of 12 major oxides including Al_2O_3, CaO, Cr_2O_3, K_2O, MgO, MnO, Na_2O, P_2O_5, Fe_2O_3, SiO_2, TiO_2, and V_2O_5 and loss on ignition ("LOI") as well as Nb_2O_5. Analyses were done by the borate fusion/X-ray fluorescence (XRF) method (SGS Lakefield lab code GO XRF76V).

11.2 Current Drilling Program

During the 2020 winter drilling campaign, samples were selected and marked by the IOS geologist. Samples were split using a hydraulic splitter, bagged on site, and assigned a unique sample number. Standards, blanks, and core duplicates were inserted into the sample sequence.

Samples bags were sealed and placed in rice bags and sent directly to the SGS Lakefield warehouse in Cochrane, Ontario. Samples were then shipped by IOS personnel or Niobay staff to SGS Lakefield, the same laboratory as in 2016, for preparation and assaying. As previously noted, SGS Lakefield is Accredited Laboratory No. 184 and conforms with requirements of CAN-P-1579, CAN-P-4E (ISO/TEC 17025:2005).

Samples were assayed using XRF76V protocol for a suit of 13 major oxides including SiO_2, Al_2O_3, Fe_2O_3, MgO, CaO, K_2O, Na_2O, TiO_2, MnO, P_2O_5, Cr_2O_3, V_2O_5, Nb_2O_5. and LOI. Trace elements were analysed using package GE_ICM90A, for 56 traces elements analysis by inductively coupled plasma mass

spectrometry (“ICP-MS”) or inductively coupled plasma atomic emission spectroscopy (“ICP-AES”) after sodium peroxide fusion.

Niobay, IOS, and RPA are independent of SGS Lakefield.

RPA is of the opinion that the sample preparation, analysis, and security procedures at the Project are adequate for use in the estimation of Mineral Resources.

11.3 Quality Assurance and Quality Control

11.3.1 2016 Check Sampling Program

In 2016, Niobay initiated a Quality Assurance and Quality Control (“QA/QC”) program that includes the systematic use of CRMs and blanks. No field, coarse reject, or pulp duplicate samples were collected during the 2016 check sampling program. RPA is unaware of QA/QC results collected by previous operators.

Blanks

The regular submission of blank material is used to assess contamination during sample preparation and to identify sample numbering errors. Niobay’s QA/QC protocol calls for blanks to be inserted into the sample stream at a rate of approximately 1 in 20.

The blank material used by Niobay consisted of decorative calcite acquired from a local hardware store in Moosonee which was expected to have a very low Nb_2O_5 content.

RPA received results for 28 analyses of blanks. All the blanks from 11 of the 12 holes analyzed returned values of less than 0.01% Nb_2O_5. Four of the five blanks from hole 66-2 returned values ranging from 0.05% Nb_2O_5 to 0.94% Nb_2O_5 suggesting either contamination at the laboratory or, more likely, sample misnumbering. RPA recommends that Niobay investigate the associated sample batches from hole 66-2 and re-analyze if necessary.

Certified Reference Material (Standards)

Results for the regular submission of CRMs are used to identify issues with specific sample batches and long-term biases associated with the regular assay laboratory.

Niobay acquired two non-certified standards from GéoMéga Resources Inc. ("GéoMéga"). GéoMéga is in the process of developing the Montviel REE-Nb deposit located approximately 125 km north-northeast of Lebel-sur-Quévillon, Québec and developed in-house standard material for its internal purposes.

Table 11.1 documents the composition of the standards acquired from GéoMéga.

Table 11.1: Standards

	Low Standard (S4) Nb (ppm)	High Standard (S5) Nb (%)
Mean	75.36	0.845
Variance	2.43	0.000025
Std. Dev. (S)	1.56	0.005
Rel. Std. Dev. (%)	2.07	0.59

The results for 24 of the 27 standards were reasonable, however, three standards show significant differences from the expected value. RPA recommends that Niobay investigate the associated sample batches and reanalyze if necessary.

11.3.2 Current Drilling Program

Similar to the 2016 check sampling program, Niobay conducted a QA/QC program that includes the systematic use of CRMs and blanks and field duplicates. No coarse reject or pulp duplicate samples were collected during the 2020 check sampling program. In total, 128 samples of control material (approximately 15% of the total samples) were inserted into the sample sequence every 10 samples, alternating between CRMs, blanks, and duplicates.

Blanks

Three types of blank material were used at the Project to assess contamination during sample preparation and to identify sample numbering errors. Niobay inserted blank material at the rate of approximately 1 in 20.

The blank materials included:

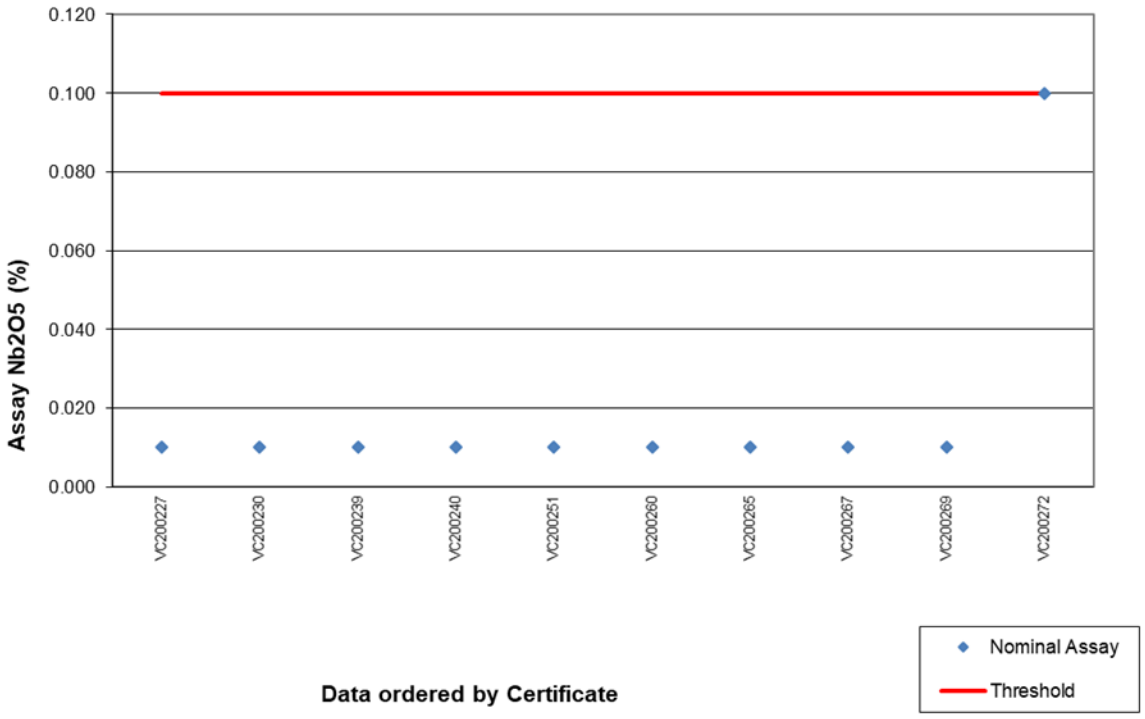
- Block blanks made of pure quartzite obtained from La Galette quartzite, Sitec Amérique du Nord Inc.

- Crushed blanks made of pure quartz from Lac Bouchette deposit crushed to >90 µm with jaw crusher and ceramic disk pulverizer.
- Pulverized blanks made of pure quartz from Lac Bouchette deposit pulverized with carbon steel ringmill to <90 µm.

All the blanks from six of the seven holes submitted for analysis returned values of less than 0.01% Nb_2O_5. One blank made from pure quartz crushed to >90 µm, inserted after a sequence of several higher grade field samples, returned a value of 0.1% Nb_2O_5 (Figure 11.1), indicating that contamination can occur and cleaning procedures should be more carefully followed. Due to the low grades RPA considers it to be non-material.

RPA noticed that the failure criteria were applied to either the content of metal (Nb ppm - GE_IC90A, sodium peroxide fusion/ICP-AES finish) or oxide (%Nb_2O_5 - GO_XRF76V, Ore grade Borate fusion, XRF), depending on the assaying method reported by the lab and used for comparison with the failing criteria. The oxide percent threshold (0.01%) is much higher than the ppm threshold value (1 ppm = 0.0001%), leading to a much larger number of potential failures when the Nb ppm value is considered. A consistent approach should be used in order to reduce the potential of identifying false failures. The Nb ppm value used as threshold should be equal to the Nb_2O_5 percent value, converted to ppm. The occurrence of the blank apparent failures in the Nb ppm data reflects trace level contamination from previous higher grade samples, however, due to the very low values detected, RPA considers it to be non-material.

Figure 11.1: Performance of Blank Reference Material Nb_2O_5 – Pure Quartz (>90 µm)



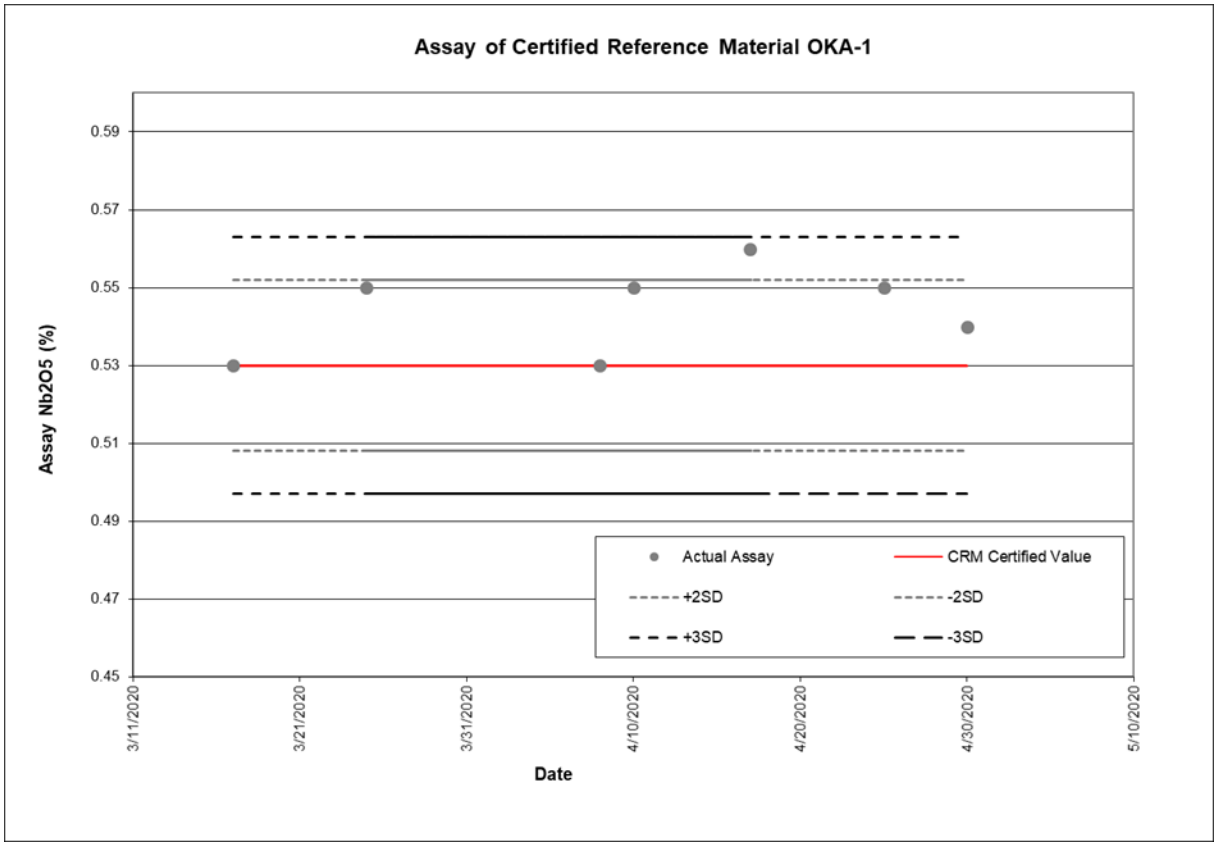
Certified Reference Material (Standards)

In order to identify a potential issue with a single sample batch or biases associated with the regular assay laboratory, Niobay used four types of commercial CRMs. One CRM, OKA-1, is certified for niobium and was acquired from CANMET (Canada Centre for Mineral and Energy Technology). The other three were certified for tantalum, suite of oxides (SiO_2, Al_2O_3, Fe_2O_3, MgO, CaO, K_2O, Na_2O, TiO_2, MnO, P_2O_5, Cr_2O_3, V_2O_5, and LOI), and rare earth elements.

Additionally, Niobay used an internal reference material made of niobium and tantalum bearing nepheline syenitic pegmatite, which was obtained from IOS facilities in 2009.

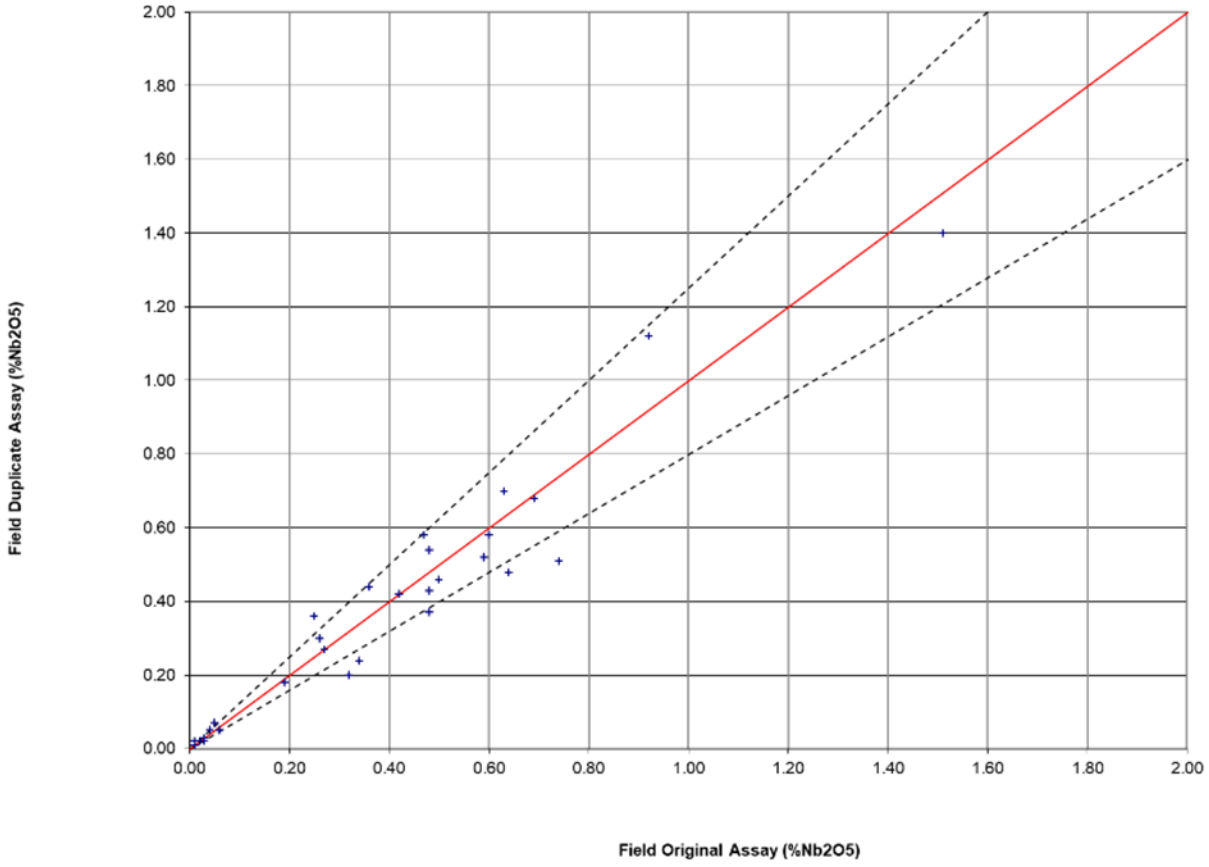
Performance of the OKA-1 CRM was acceptable with slight positive bias (Figure 11.2), with an average assayed value of 0.54% Nb_2O_5, which is 1.9% higher than the CRM's nominal certified value of 0.53% Nb_2O_5.

Figure 11.2: Performance of Standard Reference Material OKA-1



The purpose of field sample duplicates is to test the quality of preparation and repeatability of samples along the analysis pathway. Niobay inserted field duplicates obtained from the quarter split core submitted in different sample shipment. Figure 11.3 is a scatter plot of the pairs of original sample and field duplicate. Most of the pairs plot within the ±20% difference lines, which indicates that the mineralization is reasonably well-distributed in the drill core. RPA considers the overall field sample duplicate results to be acceptable.

Figure 11.3: Field Duplicate Assay Results (% Nb_2O_5)



Enhancements to QA/QC Program

RPA recommends several enhancements to the QA/QC protocol for use during the drilling programs recommended in this Technical Report including the regular submission of pulp duplicates to an alternative laboratory and a temporary coarse reject duplicate analysis program. Niobay should also implement a QA data monitoring system used to detect failed batches, and in turn, identify sample batches for reanalysis.

Pulp duplicates are submitted to a second laboratory to make an additional assessment of laboratory bias. The primary laboratory should be instructed to prepare one pulp duplicate for every 50 samples. These should be forwarded to an alternative laboratory for analysis using similar digestion and analysis methods as used by primary laboratory.

Reject duplicates consist of a second split of the crushed sample and should be prepared and analyzed at the primary laboratory. The split should be taken using the same method and have the same weight as the original sample. RPA recommends an initial test program of 50 reject duplicates of samples over a range of grades. Results from the reject duplicate QC program will determine if the splitting procedures are

applied consistently and are appropriate. Niobay should then continue to submit one coarse reject duplicate every 50 samples.

In RPA's opinion, the QA/QC program as designed and implemented by Niobay is adequate and the assay results within the database are suitable for use in a Mineral Resource estimate.

12. DATA VERIFICATION

12.1 Site Visits

A site visit to the Project was carried out by Dorota El-Rassi, M.Sc., P.Geo., RPA Senior Geological Engineer, on August 21, 2020. Ms. El-Rassi inspected the historical shaft site and examined the recent drill hole locations.

Paul Chamois, P. Geo., Principal Geologist with RPA and an independent QP, visited the Project on September 27, 2017. During the visit, Mr. Chamois examined the historical shaft site on the Property, examined core from historical drilling programs stored in Moosonee, identified pyrochlore mineralization in the historical core, confirmed the local geological setting, and investigated factors that might affect the Project.

A site visit to the Project site was carried out by Mr. Paul Murphy, P Eng., GMS Project Manager, on August 21, 2020. The general Project area was inspected as well as the routing for the site access road and electrical infrastructure in Moosonee.

12.2 Data Verification

RPA visually inspected the drill hole traces, performed basic database validation procedures, and reviewed the drill hole traces in 3D, level plan, and in vertical sections, and found no unreasonable geometries. RPA also confirmed that there are no duplicate sample numbers and that samples numbers are available for every assayed interval.

RPA extracted 422 samples from the 2016 check sampling results, representing approximately 67% of the total samples, and compared them to the original assay certificates. For the 2020 drilling data, RPA compared 20 SGS Lakefield laboratory assay certificates, representing essentially all of the 2020 drill hole samples, to the assay values in the database. No significant discrepancies were identified.

In addition, a number of standard data integrity checks were performed by the software program on the drill hole database such as:

1. Intervals exceeding the total hole length (from–to issue).
2. Negative length intervals (from–to issue).
3. Inconsistent downhole survey records.

4. Out-of-sequence and overlapping intervals (from–to issue; additional sampling /QA/QC / check sampling included in table).
5. No interval defined within analyzed sequences (not sampled or missing samples/results).
6. Inconsistent drill hole labelling between tables.
7. Invalid data formats and out-of-range values.

RPA is of the opinion that the drill hole database is acceptable for the purposes of Mineral Resource estimation.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

This section provides a summary of the historical and recent metallurgical test work completed on the James Bay niobium deposit. Samples collected from the storage area on the property in 2017 were tested to confirm and improve over the historical work. More recently, a drilling program was executed in 2020 to generate fresh core to perform additional metallurgical testing.

13.1 Historical Work (1967-1969)

13.1.1 Initial Laboratory Testing

As part of a Feasibility Study prepared by Canadian Bechtel Ltd, preliminary metallurgical testing was carried out on drill core samples obtained during the 1967 drilling programme. The methods of concentration investigated included the following:

- Heavy liquid separation.
- Gravity separation (table, jig, and spiral test).
- Magnetic and electrostatic separation.
- Flotation (direct and dual flotation).

The results of the preliminary work suggested that part of the gangue in the ore material could be removed before the flotation process with a positive impact to the process plant size and costs. Two flotation techniques were considered to be feasible for the recovery of the niobium minerals:

a. Direct flotation in which a rougher pyrochlore concentrate is floated leaving the bulk of the gangue, mainly carbonates, as a tailing.
b. Dual flotation in which a primary anionic float removes the carbonate, leaving the pyrochlore to be floated subsequently by cationic flotation.

The second method provided better results than direct flotation.

13.1.2 Mineralogy

The mineralogy study indicated that only two niobium-bearing minerals were present: sodium pyrochlore (a fluocalciopyrochlore salt (NaCa)2 (NbTi)2 (O,F)7) and columbite ($FeNb_2O_6$).

13.1.3 Pilot Plant Test Work

Based on the initial laboratory results, a pilot plant test was carried out on a bulk sample from underground generated during shaft sinking (40.5 m) and a crosscut (30.5 m) at 37 m below the surface. The sample assayed 0.73% Nb_2O_5, which is higher than the average grade reported for the historical resource estimate (0.55% Nb_2O_5). The total amount of sample sent to SGS Lakefield for the pilot plant was 227 t and another 16 t were sent to the Ontario Research Centre ("ORC").

The initial grinding test investigation was conducted at the ORC and included wet autogenous and dry autogenous grinding. A wet rod mill was suggested but not investigated. Results suggested that wet autogenous grinding produced a desirable particle size distribution and niobium content in the minus 48 mesh fractions.

Subsequently, a detailed pilot plant program was conducted at Lakefield Research Ltd. (presently SGS Lakefield) to assess the final plant flowsheet and pilot plant design criteria. The pilot plant consisted of the following stages:

- Grinding and gravity separation.
- Magnetic separation.
- Calcite flotation.
- Pyrochlore flotation.
- Reverse sulfide flotation.
- Acid leach.
- Gravity separation on concentrate.

Results from the Lakefield pilot plant were:

- 45% of the initial mass was removed with niobium losses of only 4% to 5%.
- A final niobium concentrates of 64% Nb_2O_5 was produced at a niobium recovery of 79%.
- Two types of concentrate were produced: a lower and higher silica content (Grade 1 with max 1% SiO_2 and Grade 2 with max 4% SiO_2). Table 13.1 presents the quality of the niobium concentrates produced from the historical test work.

Despite the positive results achieved, the final process flowsheet was complicated and included several cleaning stages, a regrinding, and a gravity separation stage on the niobium concentrate. It may be

challenging to operate and control the niobium losses. Some of the reagents are hazardous ("HF") or are obsolete. With the evolution in niobium ore processing during the past four decades, it appeared that this process flowsheet needed to be revisited and possibly simplified. It was the main goal of the 2017 and, more recently, of the 2020 test work program.

Table 13.1: Historical Niobium Concentrates

	High Grade	Low Grade
Distribution	41%	59%
Nb_2O_5	67.7%	62.3%
Ta_2O_5	0.49%	0.50%
Fe_2O_5	1.24%	3.45%
SrO	1.11%	1.01%
Pb	0.04%	0.03%
SnO_2	0.02%	0.01%
TiO_2	1.76%	2.73%
SiO_2	0.68%	2.90%
CaO	15.7%	13.6%
MgO	0.26%	0.69%
Na_2O	7.31%	7.28%
Al_2O_3	0.04%	0.13%
S	0.05%	0.06
P_2O_5	0.08	0.23
MnO	0.01	0.03

13.2 Recent Work – 2017-2018

The recent metallurgical work was performed by SGS Mineral Services - Lakefield Laboratory.

The test work included:

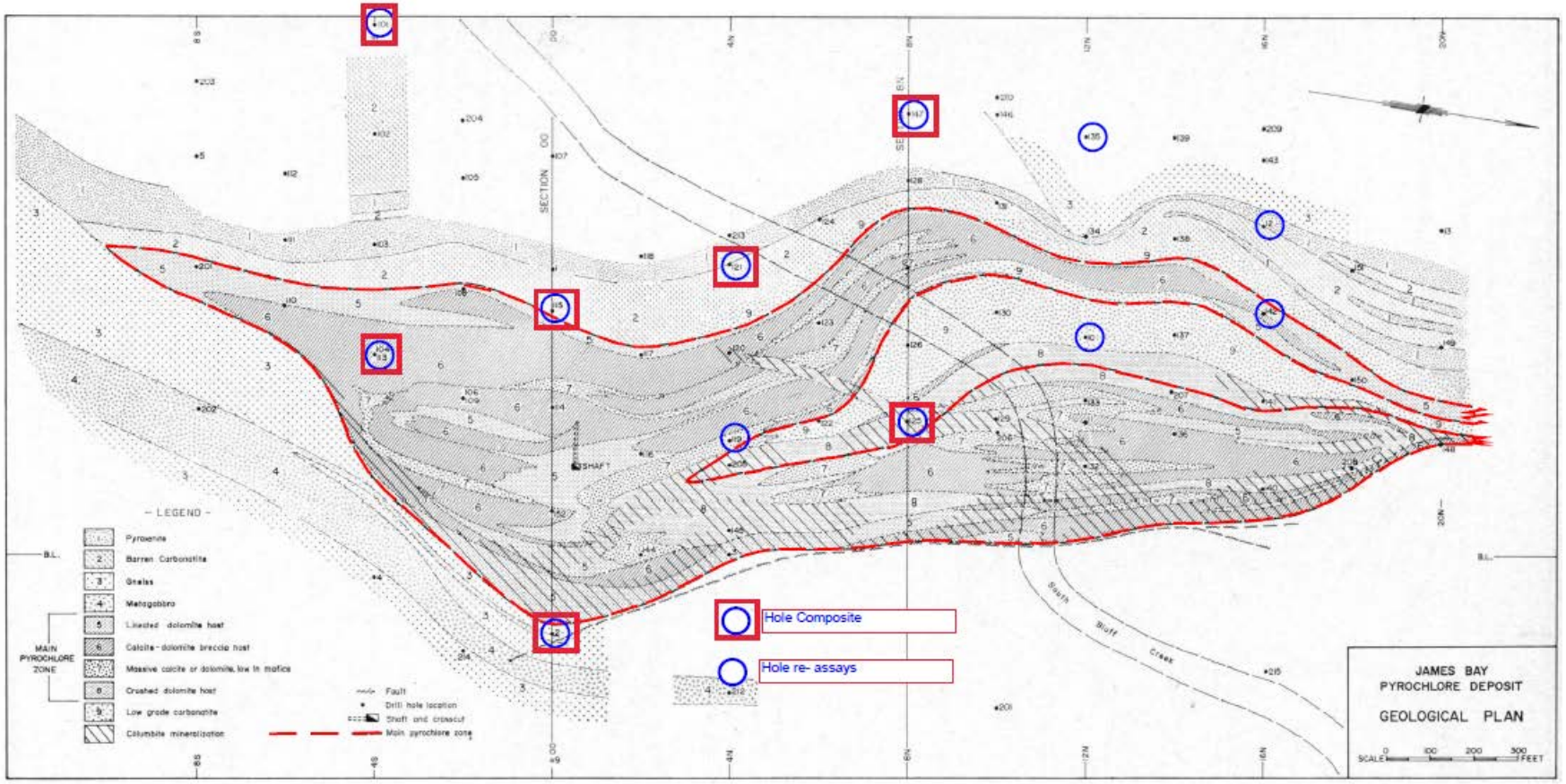
- Completion of detailed assays for 12 holes to understand the variability of the principal elements/contaminants.
- QEMSCAN mineralogy.
- Grinding test.

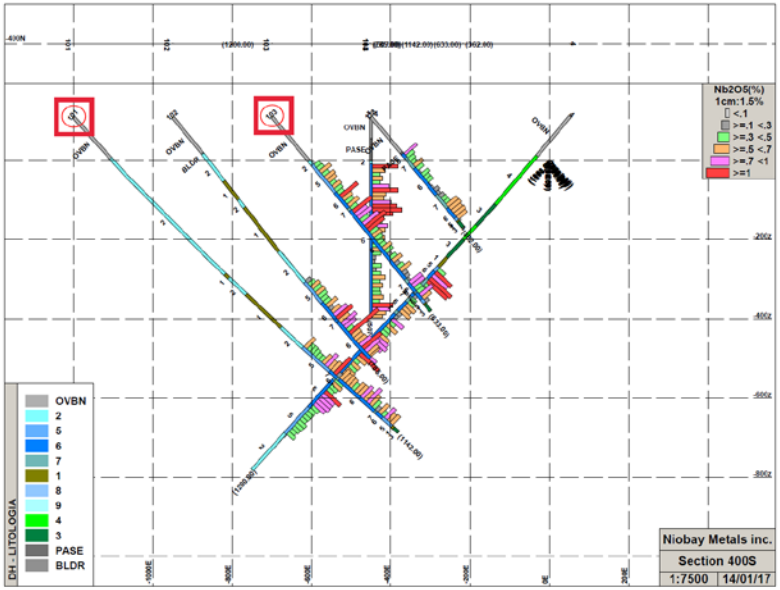
- Heavy liquid separation.
- Gravity separation (2).
- Flotation tests (10).

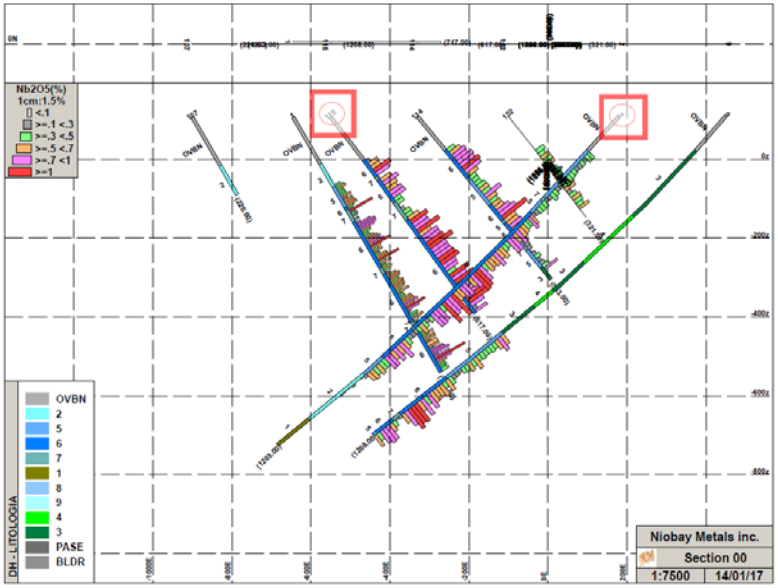
13.2.1 Composite Sample

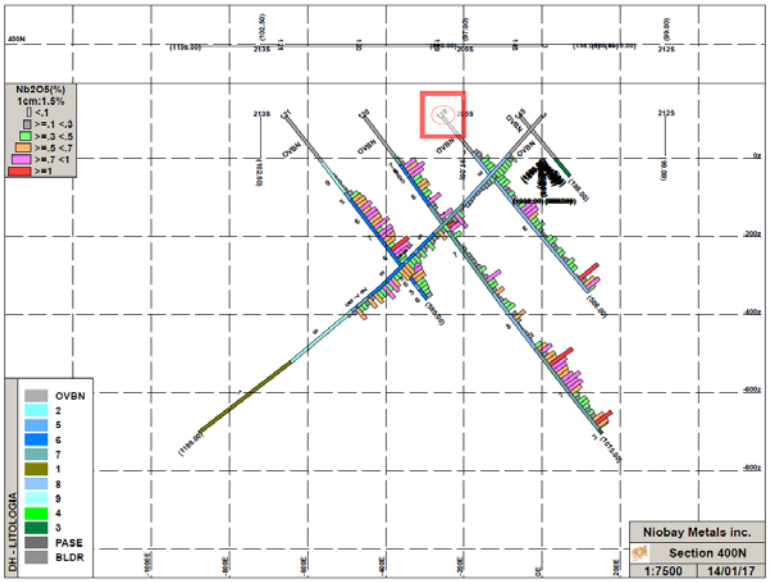
The 2017 test work was performed on a composite sample generated by a sampling program from the core stored onsite from the 1967-1969 Feasibility Study Work. These were the only samples available at the time and, considering the nature of the deposit with mainly oxide minerals, it was judged satisfactory for the next phase of metallurgical testwork. All core samples were prepared to be optimal for the flotation program. During the core preparation, each sample core was stage-crushed to minimize fine production and a sub sample was collected for assays. Each individual sample was assayed for Nb_2O_5, whole rock analysis ("WRO"), a complete ICP ES/MS (multi-element analysis) and Sulfur (S^{-2}) determination. The intent was to track and provide a mapping of the potential niobium concentrate contaminant (SiO_2, TiO_2, P_2O_5, S^{-2}). Three hundred eight (308) individual samples were combined from seven selected holes to prepare the composite at an estimated weight of 378 kg. The intent of this program was to initiate preliminary flowsheet review and design with the intent to validate the results in the future with fresh samples made available from future drilling program. Figure 13.1 presents the location of the seven holes used to generate the composite.

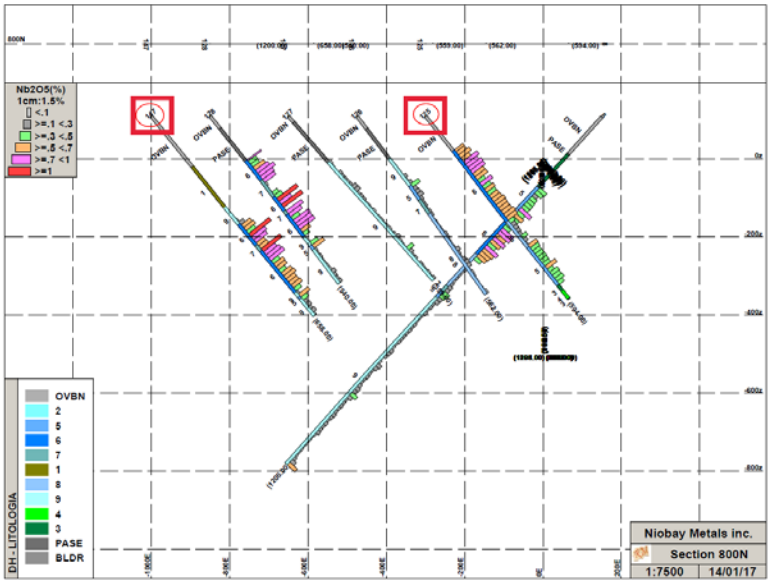
Figure 13.1: Composite Location




Nb2O5(%)
1cm:1.5%
<.1
>=.1 <.3
>=.3 <.5
>=.5 <.7
>=.7 <1
>=1
OVBN
2
5
6
7
1
8
9
4
3
PASE
BLDR
DH - LITOLOGIA
Niobay Metals inc.
Section 400S
1:7500
14/01/17


Nb2O5(%)
1cm:1.5%
<.1
>=.1 <.3
>=.3 <.5
>=.5 <.7
>=.7 <1
>=1
OVBN
2
5
6
7
1
8
9
4
3
PASE
BLDR
DH - LITOLOGIA
Niobay Metals inc.
Section 00
1:7500
14/01/17


Nb2O5(%)
1cm:1.5%
<.1
>=.1 <.3
>=.3 <.5
>=.5 <.7
>=.7 <1
>=1
OVBN
2
5
6
7
1
8
9
4
3
PASE
BLDR
DH - LITOLOGIA
Niobay Metals inc.
Section 400N
1:7500
14/01/17


Nb2O5(%)
1cm:1.5%
<.1
>=.1 <.3
>=.3 <.5
>=.5 <.7
>=.7 <1
>=1
OVBN
2
5
6
7
1
8
9
4
3
PASE
BLDR
DH - LITOLOGIA
Niobay Metals inc.
Section 800N
1:7500
14/01/17

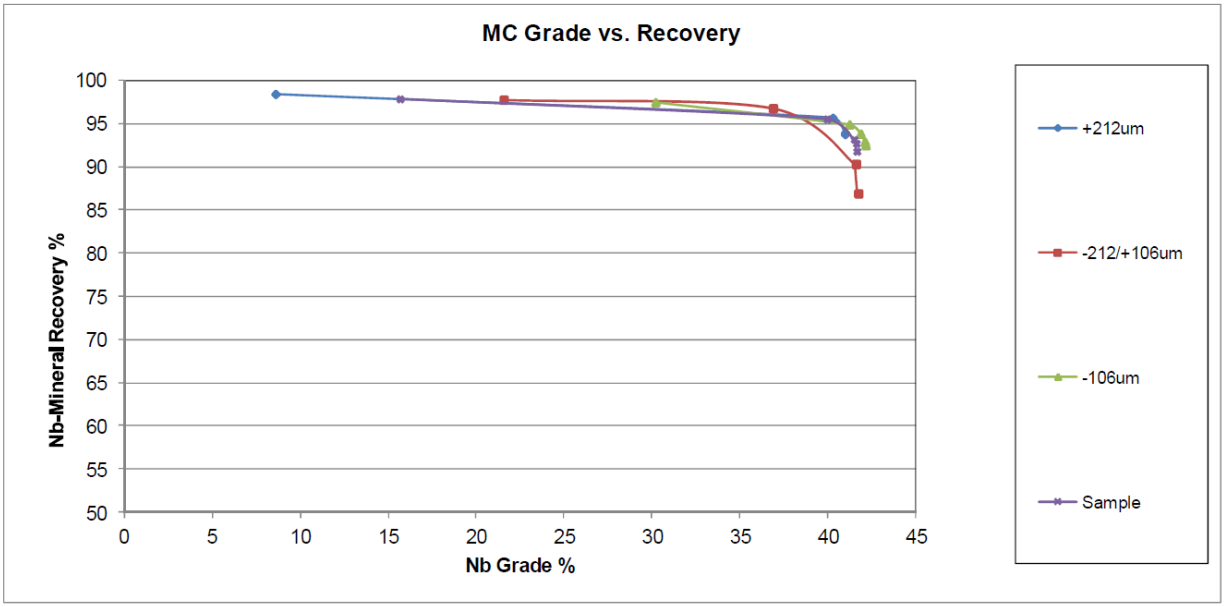
13.2.2 Qemscam

Advanced mineralogy revealed a good liberation of most of the minerals in the samples. QEMSCAN results combined with the heavy liquid separation results indicated that a better niobium recovery can be achieved with the James Bay ore compared to other niobium deposits. The main niobium minerals are pyrochlore (87%), and columbite (9.6%). The niobium-bearing pyrochlore is easier to recover than columbite. Good niobium recovery can be anticipated based on the niobium liberation, with approximately 95% of the niobium minerals free or liberated, as shown in Table 13.2. The other critical minerals in the niobium ore are also well liberated. The theoretical grade recovery curve shown in Figure 13.2 indicate a high recovery at a relatively coarse grind.

Table 13.2: Mineral Liberation

	Free	Liberated	Total Free and Liberated
Niobium	71.1%	23.8%	94.9%
Silica	89.5%	5.6%	95.1%
Apatite	97.2%	1.4%	98.6%
Sulphide	87.5%	4.1%	91.6%
Fe Oxide	68.7%	26.5%	95.2%

Figure 13.2: Theorical Recovery Curve



13.2.3 Metallurgical Test Work

A master composite was prepared based on the assay results of historical drill holes. The grade of the master composite approximated the historical grade of 0.56% Nb_2O_5 reported in the Feasibility Study. Direct assay returned 0.59% Nb_2O_5. All the test works were performed on this master composite. Table 13.3 provides the detailed assays of the master composite compared to the Feasibility Study resource estimate.

Table 13.3: Master Composite 2017 and FS Resource Estimate

	FS Resource Estimate (%)	Direct Master Composite (%)
Nb_2O_5	0.56	0.59
SiO_2	4.64	4.74
Al_2O_3	0.55	0.54
Fe_2O_3	8.96	8.79
MgO	6.54	6.37
CaO	38.20	38.40
Na_2O	0.53	0.53
K_2O	0.46	0.44
TiO_2	0.22	0.25
P_2O_5	3.29	3.33
MnO	0.75	0.75
S	0.49	0.47

A single grinding test was performed to verify the Ball Mill work index ("Bwi"). The single test confirmed a low energy requirement at 9.6 kW-h/t.

Heavy liquid separation provided excellent results with 90% of Nb_2O_5 concentrated in a 5% mass pull at an SG cutting point of 3.3. These results suggest a good liberation of the niobium minerals and open the possibility of gravity concentration inside the process flowsheet. Historical process flowsheets used gravity separation to remove up to 40-45% of the mass prior to flotation with limited niobium losses.

The flotation test work used the dual flotation sequence. A typical reagents scheme was used; the pH regulation was performed using fluosilicic and oxalic acids, which are less corrosive and hazardous than hydrofluoric acid ("HF"). Ten flotation tests were completed, and the results were in line with the historical data. The concentrates produced ranged from 56% to 59% Nb_2O_5 with the Nb_2O_5 recoveries between 68% and 77% in open circuit. The grade of the main contaminant met the specifications except for the silica. TiO_2 ranged from 3.9% to 6.0%, P_2O_5 was low at 0.12% to 0.14% (the HCl leaching cost will be low), and the sulfur (S) ranged from 0.04% to 0.8%. After adjustment to the pyrite flotation, sulfur (S) range from 0.06% to 0.12% S. For now, no secondary sulfide mineral flotation appears necessary on the final concentrate. The tests were performed in open circuit and the results will need to be confirmed by locked cycle and possibly pilot plant tests.

The tests identified one major issue; the silica concentration remained relatively high or close to the higher acceptable limit. For the silica, the lowest grade achieved was 3.21% SiO_2 and 4.0% SiO_2 for the next best flotation test as shown in Table 13.4. Characterization of the concentrate including heavy liquid separation and Qemscan were carried out to find the best alternative to resolve this issue; results are presented in Table 13.5. Amphibole mineral appears to be the main source of the silica problem.

Silica will be the main issue to address in the next phase of test work. The historical flowsheet included many stages of cleaner and gravity separation to address this issue. However, in the pilot plant test performed historically, the silica was successfully removed by gravity. In this current program, gravity on the concentrate was not tested because of the small quantity of niobium concentrate produced.

The results achieved are higher in terms of Nb_2O_5 recovery than other niobium operations. The current flowsheet is simplified compared to the historical one. For the current testing, the decision was taken to complete the flotation test on the entire sample without any mass removal by gravity. Considering its impact on the process plant, capital costs, as well as the flotation, it was decided to review gravity separation in the future test work.

Table 13.4: Heavy Liquid on Niobium Concentrate

Feed Fractions	HL SG	Weight		Assays%					Distribution%				
	g/cm^3	g	%	Nb_2O_5	SiO_2	TiO_2	S	P_2O_5	Nb_2O_5	SiO_2	TiO_2	S	P_2O_5
HLS Test: 3.20 Sink	3.2	71.1	82.1	55.3	4.15	6.17	0.17	0.14	96.0	29.0	95.4	96.7	92.3
HLS Test: 3.10 Sink	3.1	10.7	12.3	12.7	42.9	1.64	0.03	0.10	3.3	45.1	3.8	2.6	5.9
HLS Test: 3.10 Float		4.9	5.6	5.75	54.1	0.78	0.02	0.04	0.7	25.9	0.8	0.8	1.8
Feed (Calc)		87.2	100	47.3	11.7	5.31	0.14	0.12	100	100	100	100	100
Feed (Dir)				47.5	11.9	5.1	0.12	0.12					

Figure 13.3: Sink 3.2 – Mainly Niobium Minerals

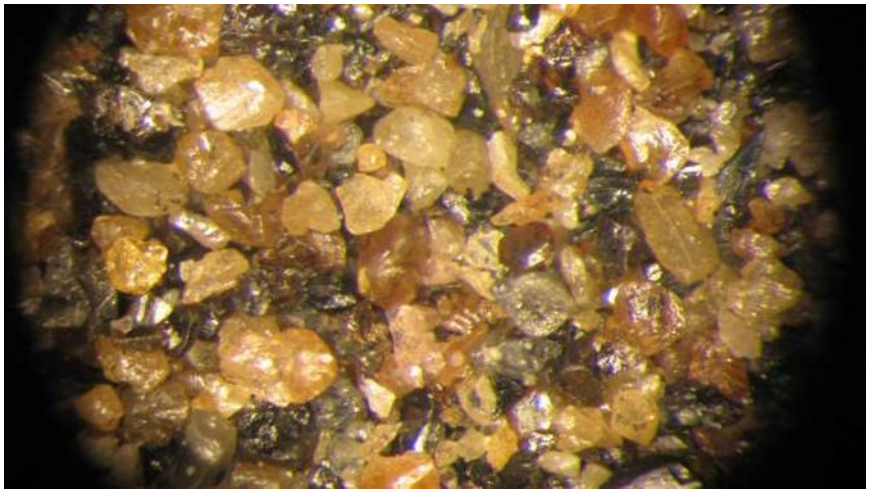


Figure 13.4: Sink 3.1 – Mainly Pyroxene

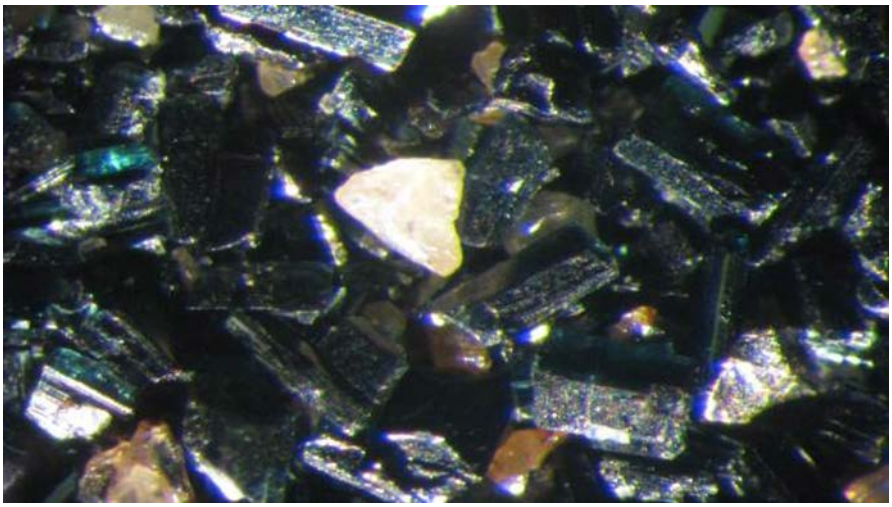


Figure 13.5: Float Product 3.1: Mainly Silicate (perhaps feldspar) and Pyroxene

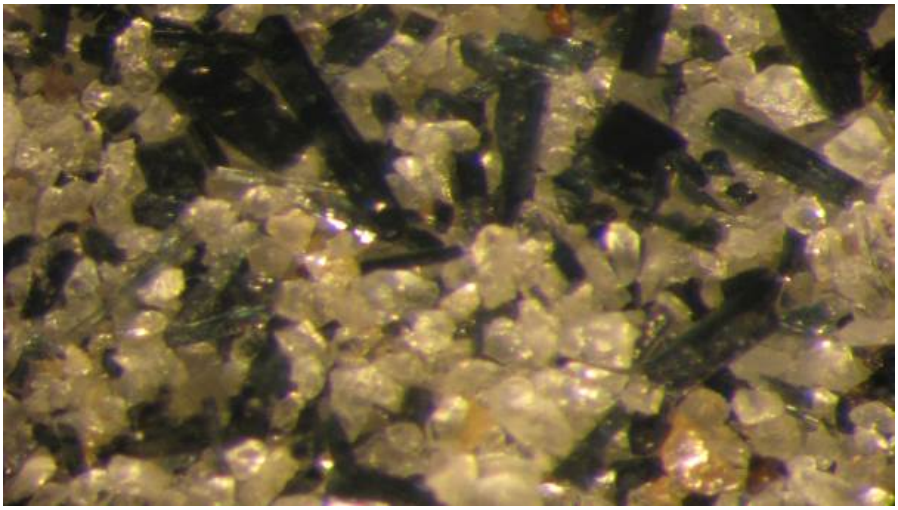


Table 13.5: Preliminary Flotation Test Results

	F1	F2	F6
Distribution	%	%	%
Nb_2O_5 Concentrate %	58.9	55.7	56.3
Nb_2O_5 Recovery (final conc)	67.7	68.3	73.9
Nb_2O_5 Recovery (rougher conc)	86.6	87.8	85.0
Fe_2O_3	6.81	8.33	7.74
P_2O_5	0.14	0.13	0.13
TiO_2	3.91	6.56	6.01
SiO_2	3.21	4.00	4.45
S	-	0.06	0.04

Despite limited test work in the 2017 program, good results have been achieved. Silica remains the main area of concern and will require attention in the next phase of test work. The sulphur issue appears to be well controlled in the current test work program by using a different flotation approach than in the past. The sulphur grade in the concentrate is well below the target limit. Additional testing on fresh ore samples was recommended to verify the 2017 test work results and to include gravity concentration.

13.3 Recent Work - 2020

The objective of this program was to revisit the gravity separation and to verify on fresh ore samples the results from the preliminary flowsheet developed in 2017. The preliminary cleaning stage on the final concentrate includes gravity and leaching.

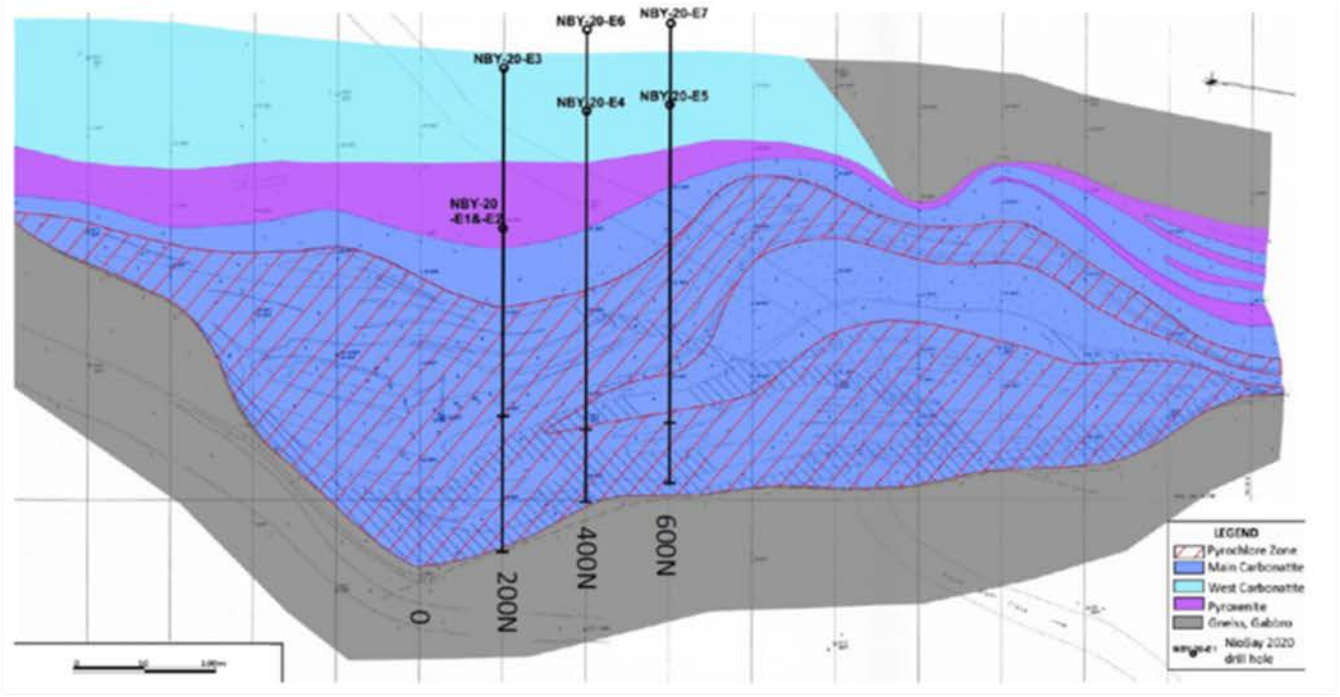
13.3.1 Master Composite

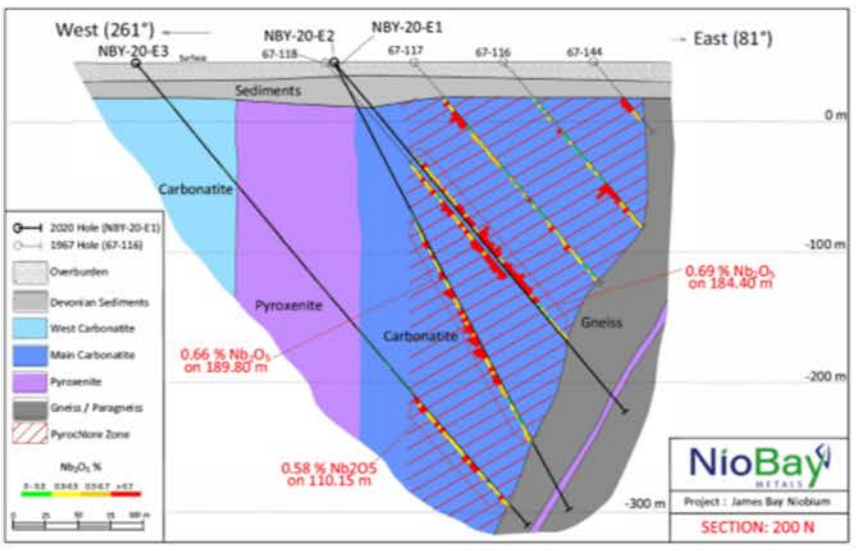
The 2020 metallurgical test work was performed on a fresh composite sample from the drilling program performed on site. Eight primary hole composites were done from the core of seven hole, each composite representing an individual hole, with the exception of one hole which had two composites; Figure 13.6 presents the location of the holes used for the composite preparation. A master composite was prepared taking half of the weight of the primary hole composites. Three hundred sixty-four (364) individual samples were used to prepare the composites. The hole composites were split and half of the weight from each individual primary hole composite was stored for future use. The other halves were mixed and a total weight of approximately 580 kg was obtained for the master composite. As shown in Table 13.6, the Nb_2O_5 grade of the 2020 master composite is higher than the 2017 composite; however, the other minerals were relatively at the same level. Each individual core sample was assayed for Nb_2O_5, WRO and a complete ICP AES/MS (multi-element analysis) was performed. The intent was to track the potential contaminants (SiO_2, TiO_2, P_2O) in the steps leading to the final Nb_2O_5 concentrate.

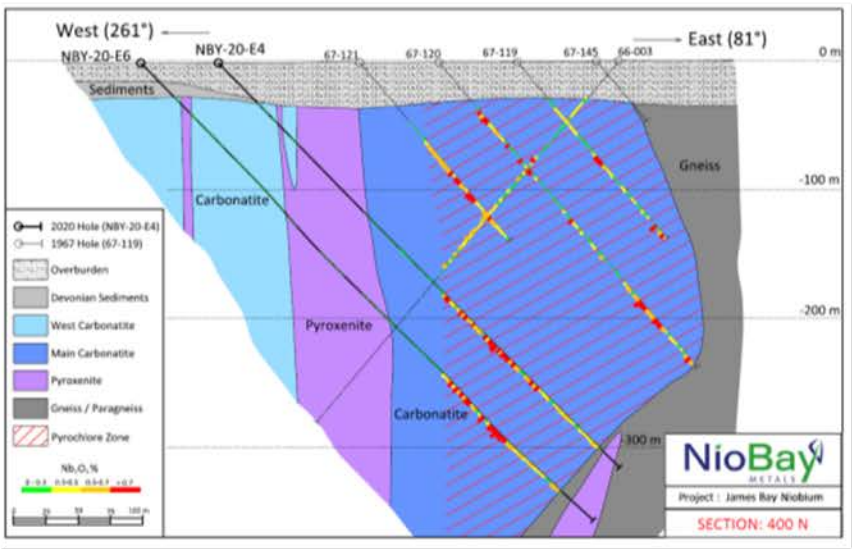
Table 13.6: Master Composite 2020

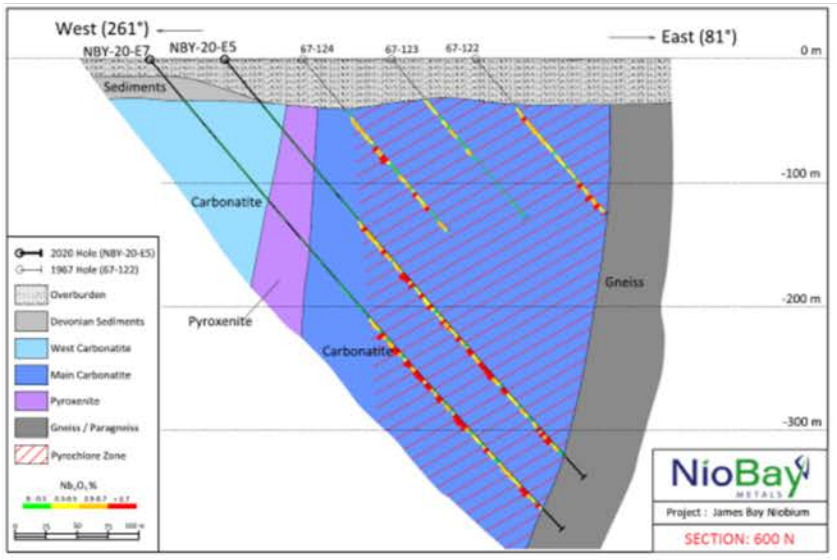
	Master Composite 2020	Master Composite 2017
Nb_2O_5	0.68	0.59
SiO_2	4.83	4.74
Al_2O_3	0.51	0.54
Fe_2O_3	9.47	8.79
MgO	5.71	6.37
CaO	38.75	38.40
Na_2O	0.63	0.53
K_2O	0.43	0.44
TiO_2	0.23	0.25
P_2O_5	3.56	3.33
MnO	0.74	0.75
S	0.51	0.47

Figure 13.6: Location of Drill Holes Used for Composite Preparation









13.3.2 Gravity Separation

A sub sample of 345 kg was collected from the Master composite for the gravity test. The gravity test was performed using two stages of gravity spirals in series. Based on the heavy liquid separation completed in 2017 and the historical results, the spiral feed size was revised to less than 500 microns with the intent of simplifying the gravity flowsheet; the past gravity test were realized at a coarser size. The sample was crushed and screened at 500 microns, and the screen overflow was fed to a rod mill for additional grinding. The product at less than 500 microns was screened to remove the 74 microns prior to the spiral feed. The simplified flowsheet was operated in open circuit with a Nb_2O_5 recovery of 96.8% Nb_2O_5 in the concentrate; the mass rejected was at 42% with only 3.2 % of Nb_2O_5 losses. The results, as shown in Table 13.7, confirm the historical results and the potential application of gravity separation to reduce the mass volume to be processed and the potential benefit for contaminant rejection. The gravity test achieved similar results as those performed in the past with a simplified flowsheet.

Table 13.7: Gravity Separation Results

Product	Weight %	Nb_2O_5 %	Nb_2O_5 % Rec	SiO_2 % Rec	CaO % Rec
Spiral Feed	100	0.68	100	100	100
Spiral Circuit Concentrate	58.4	1.15	96.8	51	56.9
Spiral Circuit Reject	41.6	0.05	3.2	49	43.1

13.3.3 Flotation Tests

A total of 14 batch tests were performed to verify the results on fresh mineralized samples and optimize niobium recovery. Two different approaches were investigated applying the same flotation flowsheet and parameters on the master composite and the gravity concentrate to compare the two alternatives. The initial results (Test F1-F2) were comparable to each other but failed to reproduce the 2017 testwork results. After a few reagent and parameter modifications (Test F5), a niobium recovery of 80% was achieved with a concentrate grade of 64% Nb_2O_5 and contaminant levels below the quality targets as shown in Table 13.8. The gravity circuit appears to reject part of the silica reporting to the concentrate when compared to direct ore flotation.

Additional optimization tests did not deliver better results. A final test (F14) was performed with the best flowsheet (F5) to duplicate the results and prepare for a locked cycle test. The test results closely duplicate F5 (Table 13.8). However, some adjustments will need to be completed to control the sulfur grade. Based on the flowsheet and parameters selected, a 10 kg flotation test was performed to produce sufficient

concentrate for the gravity and concentrate leaching test. The test also reproduced the F5 results prior to the concentrate leaching stage (Table 13.8).

Table 13.8: Flotation Test Results

	F5	F14	F15
Distribution	Standard	Duplicate	Bulk Product
Nb of Cleaner Stage	7	4	5
Nb_2O_5 Concentrate	63.8%	63.8%	64.5%
Nb_2O_5 Recovery (final conc)	80.6%	79.4%	81.3%
Nb_2O_5 Recovery (rougher conc)	86.9%	85.9%	90.7%
Fe_2O_3	3.03%	3.50%	3.4%
P_2O_5	0.18%	0.11%	0.11%
TiO_2	2.04%	2.01%	1.69%
SiO_2	2.40%	2.8%	2.70%
S	0.13%	0.27%	0.58%

The final Nb_2O_5 recovery achieved reached 78% (Table 13.9); this result was for an open circuit without any recirculation. Generally, recovery might improve by closing the niobium circuit.

Table 13.9: Global Nb Recovery

Product	Distribution	% Nb_2O_5	Nb_2O_5 % rec	SiO_2 % rec	CaO % rec
Spiral Feed	100	0.68	100	100	100
Spiral Circuit Concentrate	58.4	1.15	96.8	51	56.9
Spiral Circuit Reject	41.6	0.05	3.2	49	43.1
Flotation Concentrate	1.5	63.8	80.6	N/A	N/A
Final Recovery			**78.0**		

13.3.4 Ongoing Test Work

Testwork continues to verify niobium concentrate treatments to develop alternative treatment scenarios on the gravity concentrate and to optimize the final niobium recovery and concentrate quality. These alternatives include HCl concentrate leaching, gravity separation and caustic leach.

13.3.5 Conclusions and Recommendations

- The 2020 program succeeded in reproducing the historical results using an initial gravity separation stage and a simplified process flowsheet. Approximately 42% of the mass was rejected with positive impacts on capital and operating costs.
- Compared to historical testwork, fewer cleaner stages were required to produce a flotation concentrate inside quality specifications.
- Silica challenges were resolved reaching specification level in the flotation concentrates which were better than the historical pilot plant results.
- Considering the higher grade of the composites compared to the grade of the resources, specially for the Open Pit Scenario, further testing will be needed to validate results with different ore grades and different location in the deposit
- The next testing phase should include alternative reagents utilization and optimization. In parallel, locked cycle tests need to be performed to adjust reagents addition and consumption, reproduce the results achieved and attempt to improve niobium recovery.
- Based on the locked cycle test results, a pilot plant test is recommended to validate and increase the level of confidence in the flowsheet and the results. The pilot plant results will also provide data for plant sizing and should produce sufficient concentrate for conversion tests to ferroniobium

14. MINERAL RESOURCE ESTIMATE

14.1 Summary

The Mineral Resources conform to Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM (2014) definitions) and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 ("NI-43-101"). Mineral Resources are not Mineral Reserves and have not demonstrated economic viability. RPA considers the Mineral Resources of the Project to be amenable to underground extraction. A summary of the Mineral Resources is presented in Table 14.1. RPA has excluded approximately 7.2 Mt averaging 0.5% Nb_2O_5 situated in a 46 m thick crown pillar.

Table 14.1: Summary of Mineral Resources – July 9, 2020
Niobay Metals Inc. – James Bay Niobium Project

Category	Tonnage (Mt)	Grade (%Nb_2O_5)	Contained Nb_2O_5 (M kg)
Indicated	29.7	0.53	158
Inferred	33.8	0.52	177

Notes:

1. *CIM (2014) definitions were followed for Mineral Resources.*
2. *Mineral Resources are reported using a cut-off grade of 0.3% Nb_2O_5 based on an Underground Mining Scenario, an operating cost of CAD 70/t, and a metallurgical recovery of 70%.*
3. *Mineral Resources are estimated using a long-term niobium price of USD 40 per kg and a USD/CAD exchange rate of 1:1.2.*
4. *A minimum mining width of approximately 7.5 m was used.*
5. *Bulk density is 2.93 t/m^3.*
6. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
7. *Resources situated in a 46 m thick crown pillar have been excluded.*
8. *Numbers may not add due to rounding.*

Leapfrog Geo software (version 5.0.4) was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, estimate niobium oxide grades, and tabulate Mineral Resources.

RPA is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

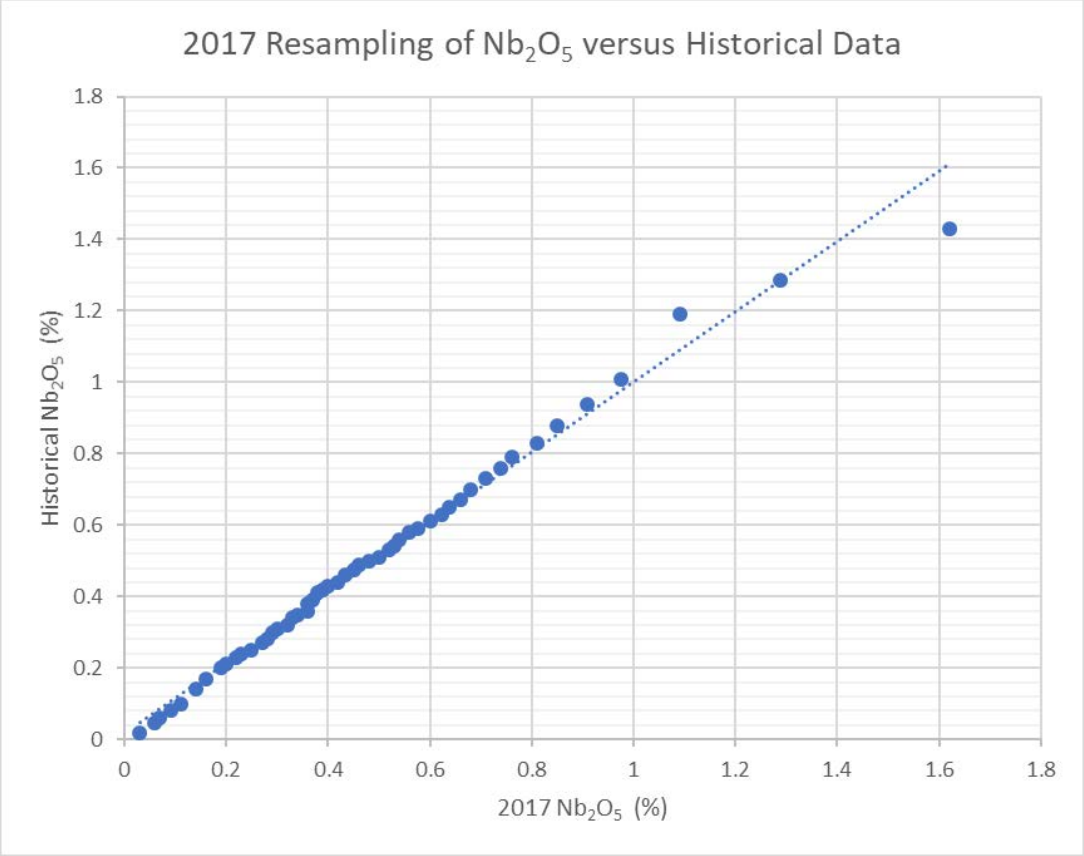
14.1.1 Resource Database

The previous Mineral Resource estimate was generated using a local mine grid in Imperial units. RPA converted the geological and mineralization model created in Leapfrog from feet to UTM coordinates (Imperial units to metric units).

NioBay provided the Project data as separate MS Excel files containing location, survey, lithology, mineralogy, and assays. The data used for grade interpolation contains records for 86 holes (16,320 m), including seven holes (3,090 m) drilled in the 2020 winter program. All drilling is located within the resource model area. The total number of samples within the database is 3,449, with 64 containing a value of zero for Nb_2O_5. There are 2,956 samples located within the mineralized pyrochlore.

NioBay re-logged and re-assayed 12 boreholes completed in the 1960s to confirm that the historical borehole data can be used to support a new Mineral Resource estimate without drilling any new holes. In RPA's opinion, there is a good correlation of the historical and re-sampled Nb_2O_5 values (Figure 14.1).

Figure 14.1: QQ Plot of the Historical and Resampled Data



RPA conducted a number of checks on the resource database, including a search for unique, missing, and overlapping intervals, a total depth comparison, and a visual search for extreme or deviant survey values. As part of the data review, RPA compared the MS Excel files with the scan copy of the original paper log. No errors were encountered.

The resource database is considered by RPA to be reliable for grade modelling and Mineral Resource estimation.

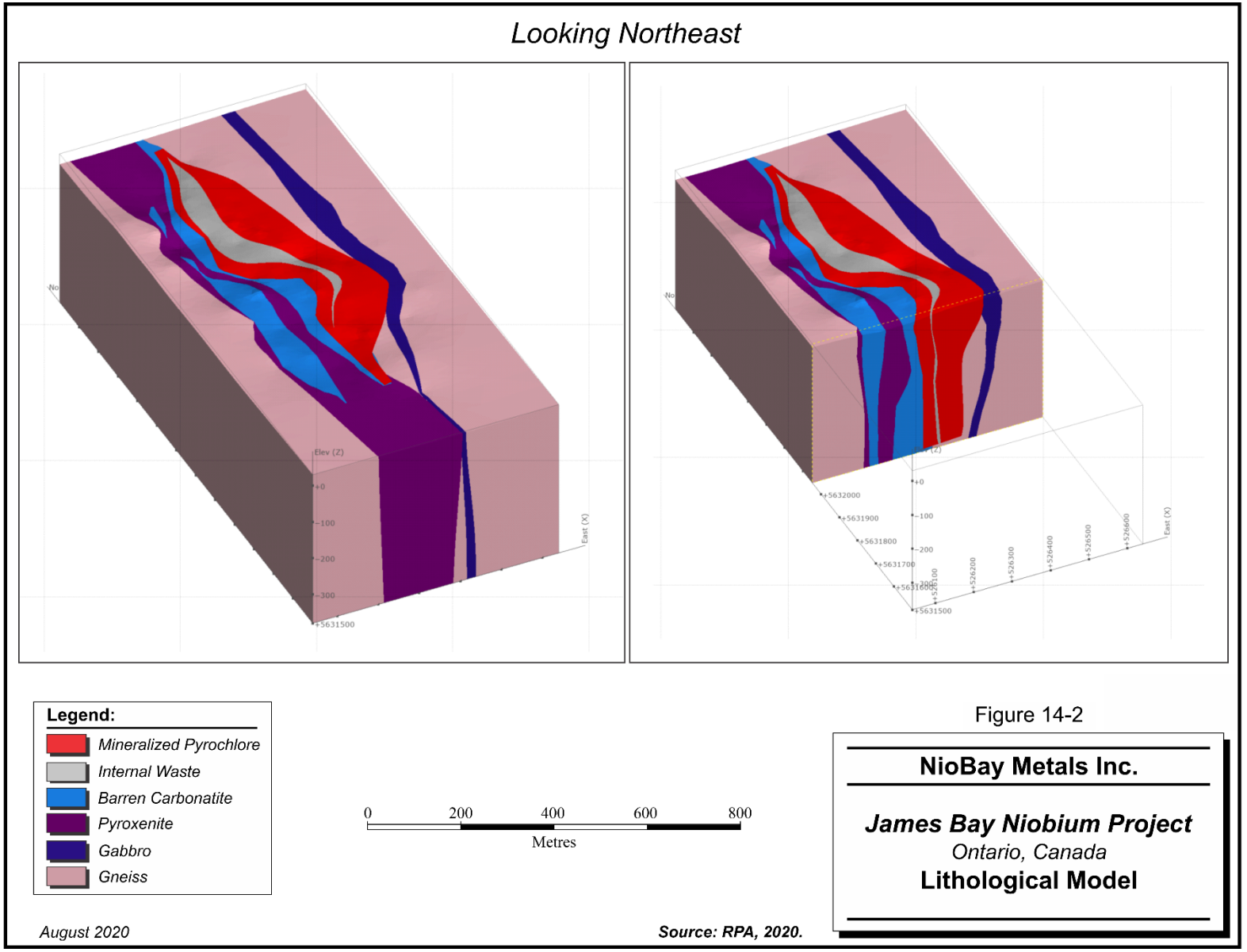
14.1.2 Geological Interpretation

RPA used Leapfrog software to create a three-dimensional geological model from core log information. As part of the modelling routine, RPA simplified original logging codes into seven main geological groups:

1. Overburden
2. Sediments
3. Gneiss
4. Gabbro
5. Pyroxenite
6. Carbonatite
7. Pyrochlore

Niobium mineralization occurs only in the pyrochlore, in which RPA defined two zones of enriched mineralization separated by internal waste. The two domains join at the south end to form a single body. RPA retained the two-domain differentiation for modelling purposes. Mineralization was modelled at an approximate modelling threshold of 0.3% Nb_2O_5 (Figure 14.2).

Figure 14.2: Lithological Model



14.1.3 Resource Assays

Niobium assays located inside the wireframe models were tagged with domain identifiers for statistical analysis. Results were used to help verify the modelling process. Descriptive statistics by domain are summarized in Table 14.2.

Table 14.2: Statistical Summary of Nb_2O_5 Assay Data

Domain	Count	Min (%Nb_2O_5)	Max (%Nb_2O_5)	Mean (%Nb_2O_5)	Variance	Std. Dev (%Nb_2O_5)	CV
East Zone	1,714	0.001	2.66	0.54	0.08	0.28	0.53
West Zone	697	0.001	1.77	0.51	0.07	0.27	0.53
Internal Waste	545	0.001	1.00	0.14	0.01	0.12	0.86
Total	2,956	0.001	2.66	0.46	0.09	0.30	0.66

14.1.4 Treatment of High-Grade Assays

Capping analysis was performed using decile analysis, histograms, log probability plots, and cutting curves. Considering the disseminated nature of the niobium distribution, RPA is of the opinion that the capping is not necessary for this deposit.

14.1.5 Compositing

The composite lengths used during interpolation were chosen considering the predominant sampling length, style of mineralization, and continuity of grade. The raw assay data contains samples with irregular sample lengths. Sample lengths range from 0.61 m to 6.1 m within the wireframe models, with 98% of the samples taken at three metre intervals (Figure 14.3). Given the width of the mineralization and considering the block size, RPA chose to composite to three metre lengths within domain boundaries. Assays within the wireframe domains were composited using Leapfrog, starting at the first mineralized wireframe boundary from the collar and resetting at each new wireframe boundary. Since assay intervals cross lithological contacts, the created composites contain a small number of residuals with lengths less than three metres. Composites less than 1.5 m were added to the previous interval. The descriptive statistics for composites used for resource estimation are presented in Table 14.3.

Figure 14.3: Histogram of James Bay Niobium Sampling Length

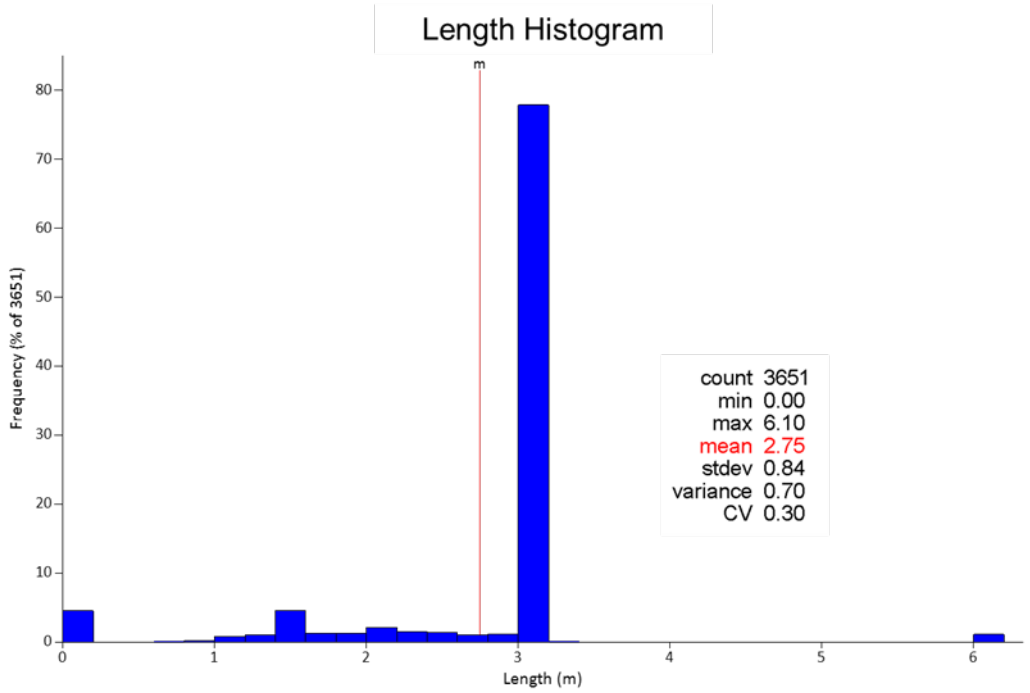


Table 14.3: Descriptive Statistics of James Bay Nb_2O_5 Composites

Domain	Count	Min (%Nb_2O_5)	Max (%Nb_2O_5)	Mean (%Nb_2O_5)	Variance	Std. Dev (%Nb_2O_5)	CV
East Zone	1,665	0.00	2.16	0.54	0.07	0.26	0.48
West Zone	638	0.00	1.45	0.51	0.06	0.24	0.48
Internal Waste	557	0.00	0.96	0.14	0.01	0.11	0.80
Total	**2,860**	**0.00**	**2.16**	**0.45**	**0.08**	**0.28**	**0.62**

14.2 Trend Analysis

14.2.1 Variography

RPA generated downhole, omni-directional, and directional variograms and correlograms using the three metre composite %Nb_2O_5 values located within the mineralized wireframes. The variograms were used to support search ellipsoid anisotropy and plunges observed in the data, and Mineral Resource classification decisions. RPA modelled the variograms using two spherical structures with a nugget effect of between 0% and 0.1% of the sill (Figure 14.4). A summary of the variogram parameters is shown in Table 14.4.

Figure 14.4: James Bay Niobium Bay Directional Variogram

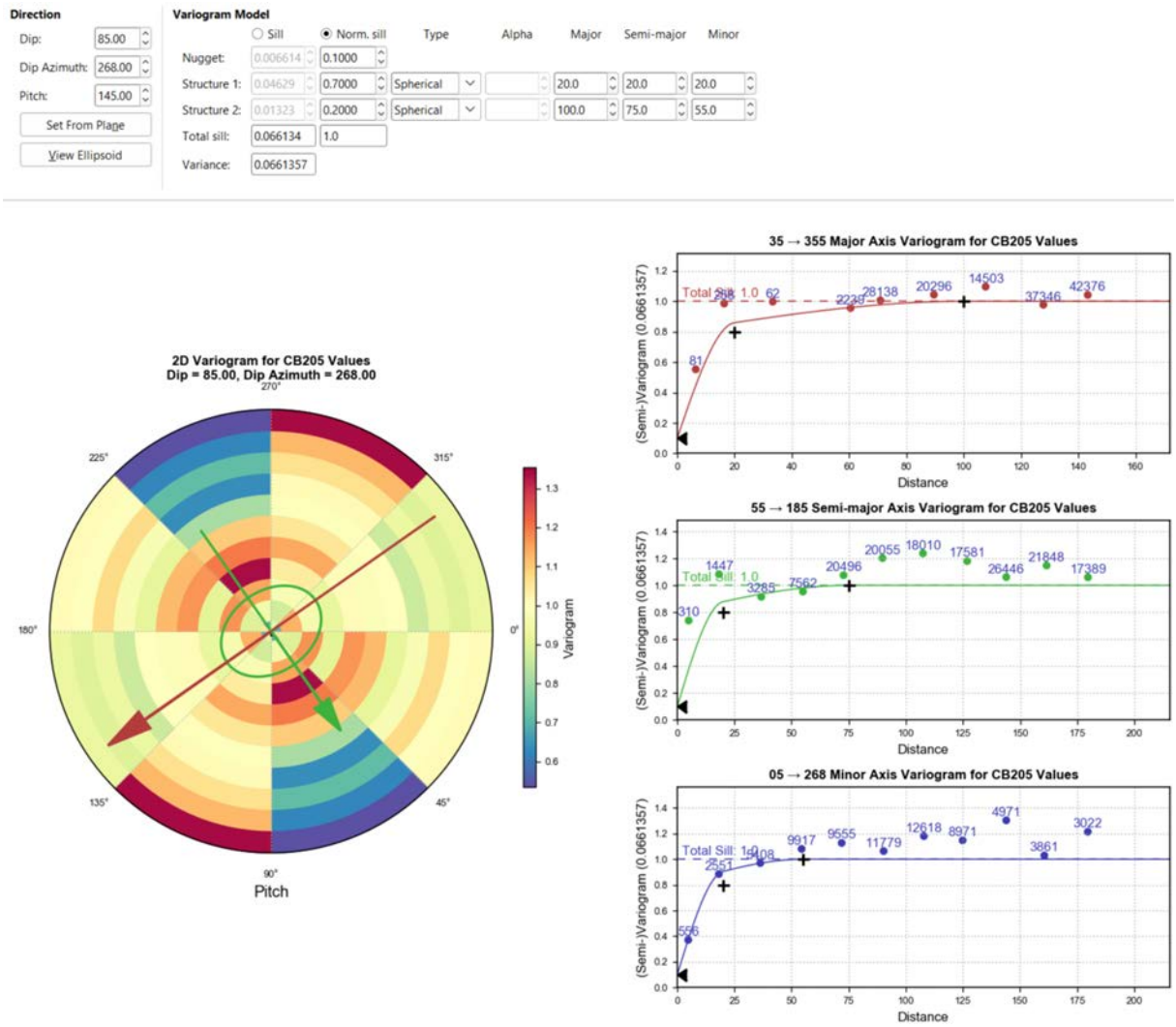


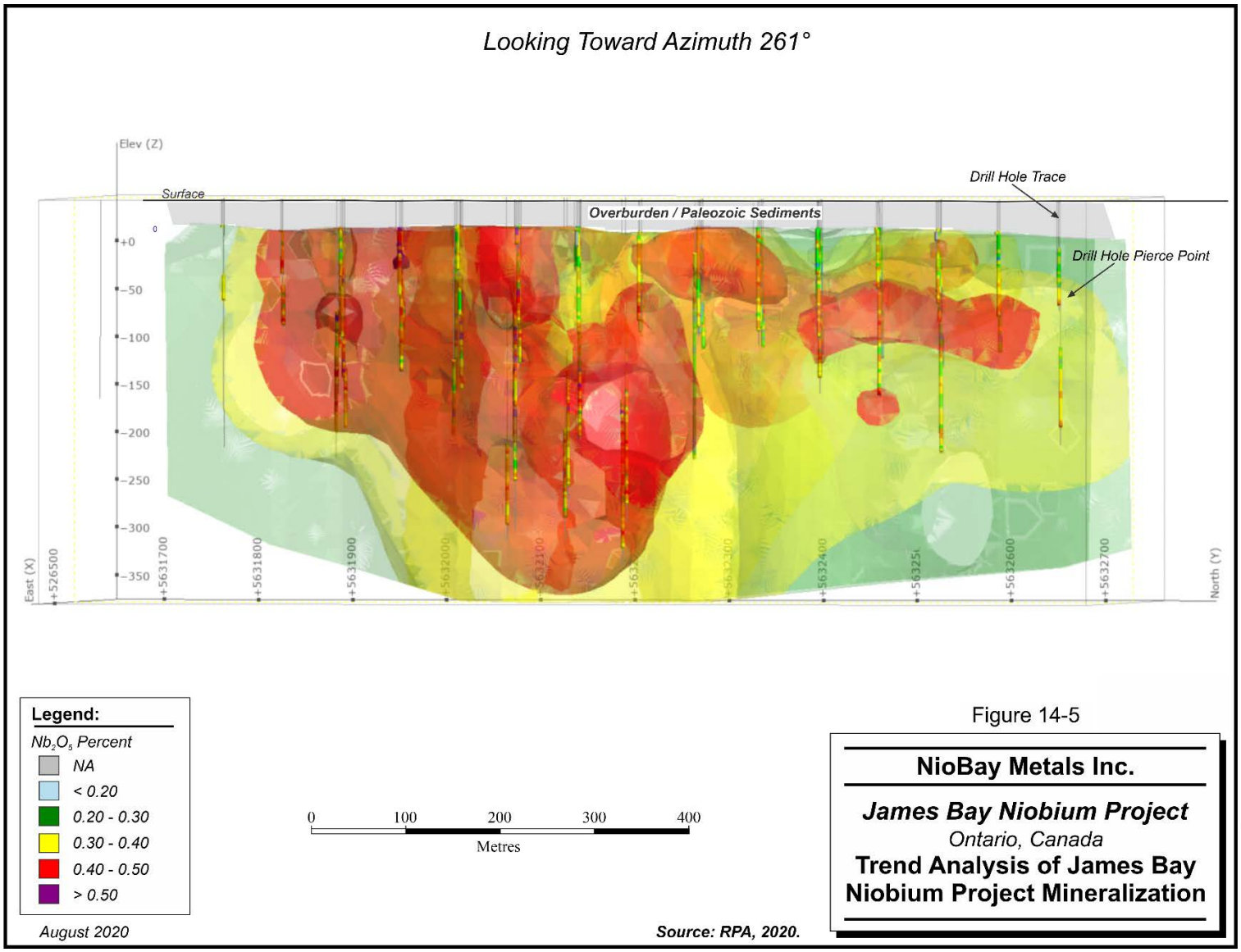
Table 14.4: Variography of the Niobium Mineralization

Nugget				First Structure					Second Structure				
	Azimuth	Dip	Plunge	Sill	X	Y	Z	Type	Sill	X	Y	Z	Type
0.1	98	85	145	0.7	20	20	20	Sph	0.20	100	75	55	Sph

14.2.2 Grade Contouring

The niobium grade continuity for the Project was investigated by generating a set of grade shells in Leapfrog for the mineralized envelopes. The identified major trend is plunging approximately 55° as shown in Figure 14.5.

Figure 14.5: Trend Analyses for James Bay Niobium Project Mineralization



14.3 Search Strategy and Grade Interpolation Parameters

Grades were interpolated by ordinary kriging ("OK") with a minimum of five to a maximum of nine composites per block estimate for the first pass, and a minimum of two to a maximum of ten composites per block estimate in the second pass. A minimum of two drill holes and a maximum of four composites per drill hole were applied during the first pass (Table 14.5). Search ellipse dimensions were chosen following a review of variography and interpolation efficiency. The first pass used the full variogram ranges for the search, whereas the second pass ranges were enlarged to populate grades in the entire mineralized envelope.

Soft boundaries were applied between the two mineralized domains limited to ten metres away from their boundary. The two domains locally share hanging wall and footwall surfaces where they form a combined single domain and exhibit similar grade distribution.

Overall, the mineralization at the Project shows a strike of 98°, with varying orientation due to folding. In order to reproduce the direction of the trends, RPA employed a Variable Orientation tool in Leapfrog. The tool allows the search to be locally adjusted to the orientation of the mineralization, which results in improved local grade estimates. RPA used hanging wall, footwall, as well as the centreline of each domain to guide the variable direction search. In addition, the variogram directions were employed as global trends to improve the variable search.

Table 14.5: Block Model Interpolation Parameters

Parameter	Pass 1	Pass 2
Min. No. Composites	3/5*	2
Max. No. Composites	9	10
Max. Composites per Drill Hole	2/4*	0
Major (m)	100	450
Semi-Major (m)	75	450
Minor (m)	15	20

** Search parameters for 10 m x 10 m x 10 m block model*

RPA has also estimated grades in the Internal Waste domain between the two mineralized domains using the same interpolation parameters as those used in the East and West domains.

Grade interpolation was carried out using OK, with inverse distance cubed ("ID^3") and nearest neighbour ("NN") used as check estimates in the mineralized domains.

14.4 Bult Density

During the 2017 site visit, RPA collected 21 independent samples to determine the density of the niobium mineralization. RPA assigned an average value of 2.93 g/cm³ to material in the pyrochlore domain. Table 14.6 shows summary statistics for density data provided by NioBay.

Table 14.6: Summary Statistics – Density

Parameter	Value (g/cm³)
Count	21
Max	3.21
Min	2.67
Average	2.93
Variance	0.01
Std. Dev.	0.11

Table 14.7: Waste Rock Density

Rock Unit	Density (g/cm³)
Gneiss	2.7
Gabbro	2.9
Pyroxenite	3.1
Barren Carbonatite	2.87
Sediments	2.7
Overburden	2.2

14.5 Block Model

One single block model covering the entire deposit was constructed in Leapfrog EDGE software to estimate Mineral Resources at the Project. Each block is 10.0 m long by 5.0 m wide by 10.0 m high, sub-blocked to 2.5 m long by 2.5 m wide by 2.5 m high. A summary of the definition data for the block model is provided in Table 14.8: Block Model Setup

Table 14.8: Block Model Setup

Description	Easting	Northing	Elevation
Origin (m)	526,000	5,631,550	49
Block Size (m)	10	5	10
Sub Block Size (m)	2.5	2.5	2.5
Number of Parent Blocks	140	133	45
Boundary Size (m)	700	1,330	450
Rotation	0°		

14.6 Cut-off Grade

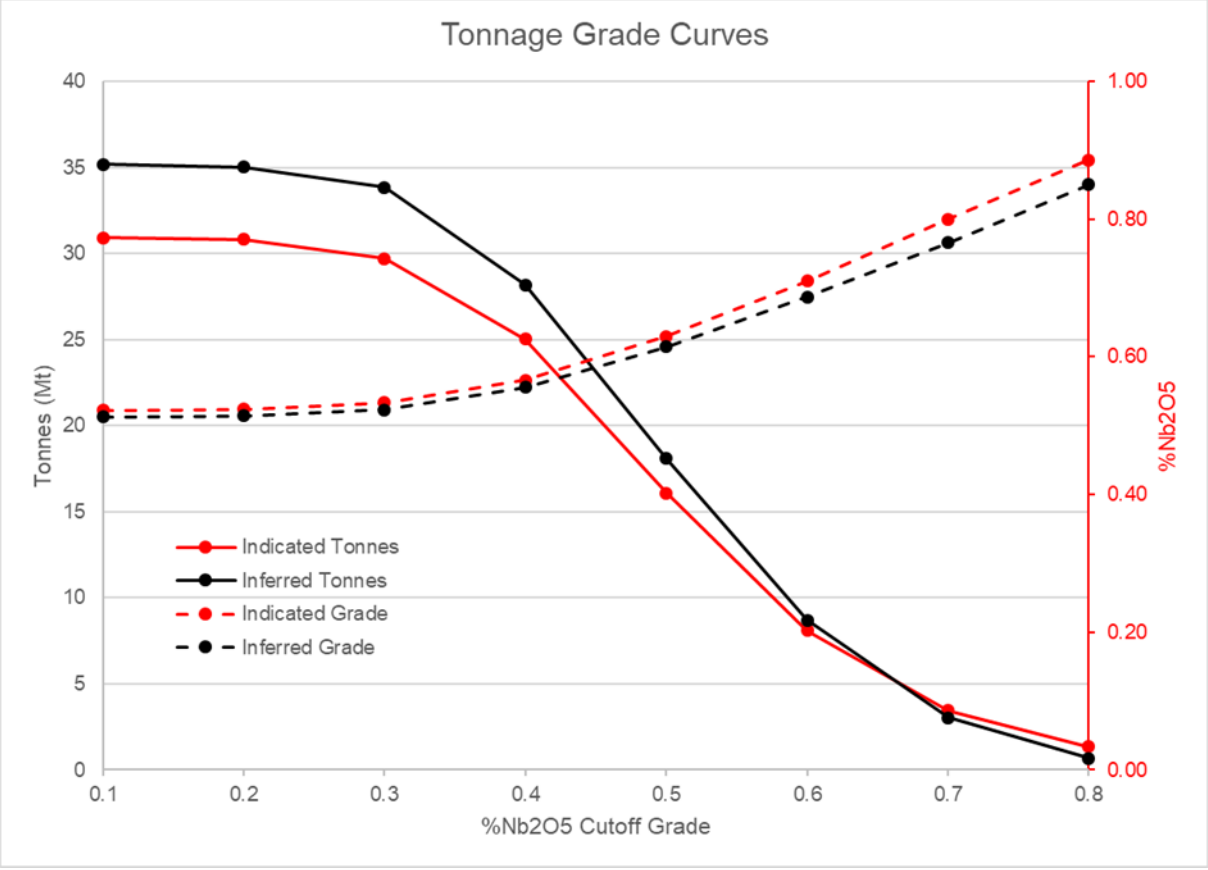
The cut-off grade calculation was based on an average underground operating cost of CAD 70/t, metallurgical recovery of 70%, and niobium price of USD 40/kg with a USD/CAD exchange rate of 1:1.2.

Table 14.9 and Figure 14.6 show the sensitivity of the block grade estimates to a range of cut-off grades both above and below the 0.3% Nb_2O_5 resource cut-off grade. The Indicated Mineral Resource grade increases from 0.53% Nb_2O_5 to 0.63% Nb_2O_5 when the cut-off grade is raised to 0.5% Nb_2O_5. At a 0.1% Nb_2O_5 cut-off grade, the Indicated mineralization grade is essentially the same and the tonnage increase is very minor compared to the grade and tonnage of the Indicated Mineral Resource estimated by RPA at a 0.3% Nb_2O_5 cut-off grade, because the mineralization wireframes were built based on an approximately 0.3% Nb_2O_5 cut-off grade. This means that there is only a small amount of low grade mineralization present in the mineralization wireframes and that reporting the resources at a zero cut-off grade or a 0.3% Nb_2O_5 cut-off grade does not make much difference. Approximately 1.4 Mt of low grade material within the wireframes are below the 0.3% Nb_2O_5 resource cut-off grade and were not reported as part of the Mineral Resource. RPA reviewed the location of this material and determined that it was relatively contiguous and could be avoided in underground mining.

Table 14.9: Mineral Resource Sensitivity to Cut-off Grade

Cut-off ($\%Nb_2O_5$)	Indicated			Inferred		
	Tonnage (Mt)	Grade ($\%Nb_2O_5$)	Contained Nb_2O_5 (000 t)	Tonnage (Mt)	Grade ($\%Nb_2O_5$)	Contained Nb_2O_5 (M kg)
0.1	30.9	0.52	162	35.2	0.51	180
0.2	30.8	0.52	161	35.0	0.51	180
0.3	29.7	0.53	158	33.8	0.52	177
0.4	25.0	0.57	142	28.2	0.56	157
0.5	16.1	0.63	101	18.1	0.61	111
0.6	8.1	0.71	58	8.7	0.69	60
0.7	3.5	0.80	28	3.0	0.77	23
0.8	1.4	0.89	12	0.7	0.85	6

Figure 14.6: Mineral Resource Sensitivity to Cut-off Grade



14.7 Classification

Definitions for resource categories used in this Technical Report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction". Mineral Resources are classified into Measured, Indicated, and Inferred categories. No Mineral Reserves have been estimated for the James Bay niobium deposit.

The basis for the classification is a distance-based scheme using the relative confidence expressed by the range of variograms, distance to nearest neighbor, and apparent continuity of mineralization. RPA manually defined the portions of the deposit supported by drill hole spacing up to approximately 70 m and 120 m and classified these areas as Indicated and Inferred Mineral Resources, respectively (Figure 14.7 and Figure 14.8).

Figure 14.7: Histogram of the Classified Blocks vs. Distance to the Composites

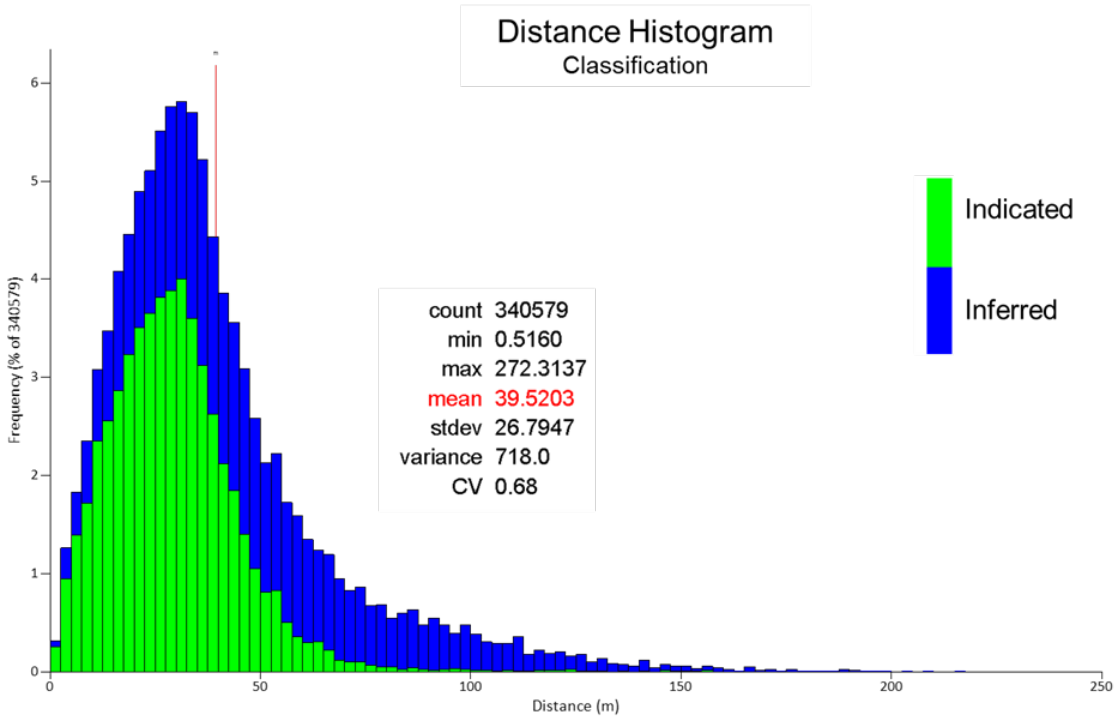
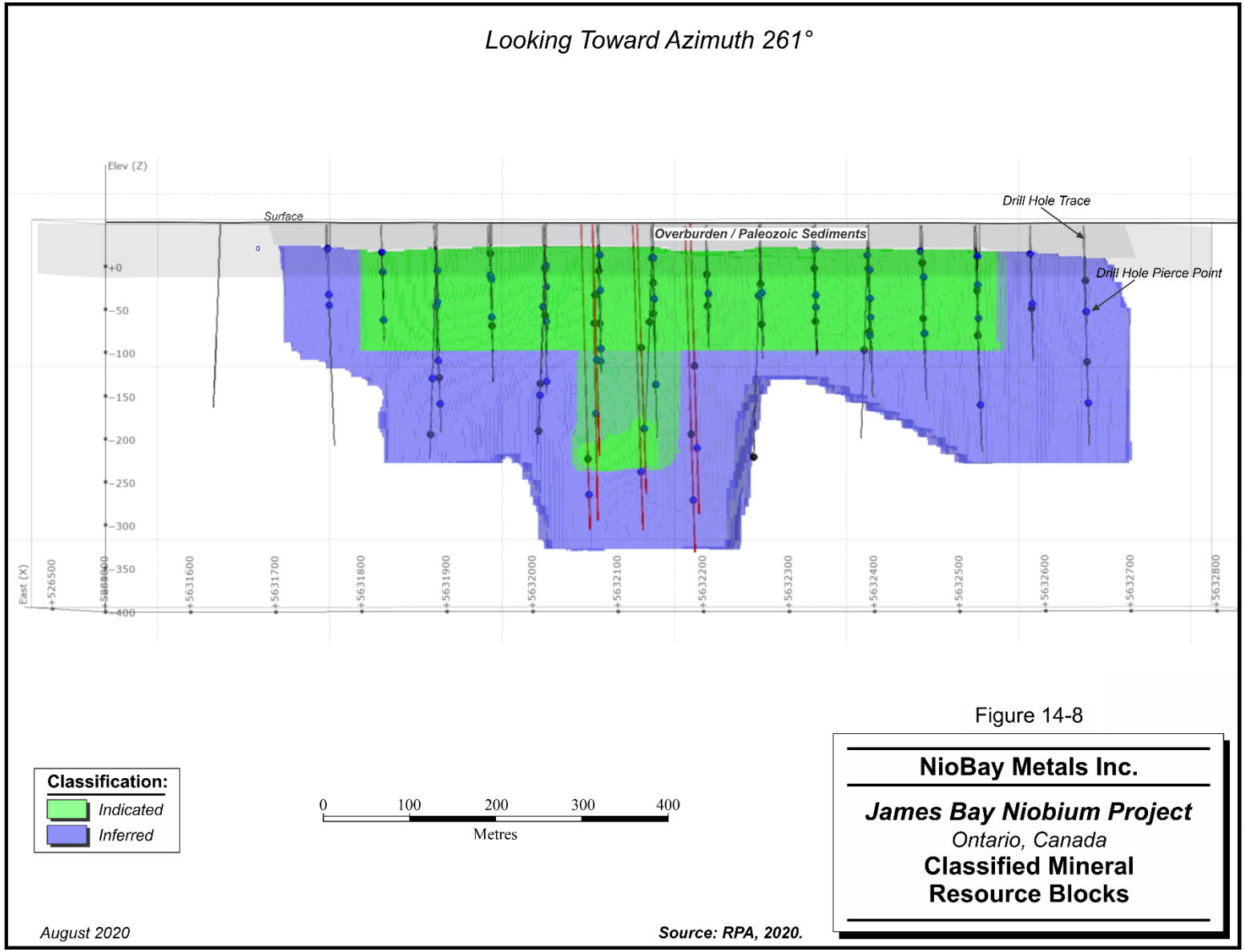


Figure 14.8: Classified Mineral Resource Blocks



14.8 Block Model Validation

RPA validated the block model by visual inspection, volumetric comparison, and scatterplots. Visual comparison on vertical sections and plan views, and a series of swath plots found good overall correlation between the block grade estimates and supporting composite grades.

The estimated total volume of the wireframe models is 50,793,000 m^3 while the volume of the block model at a zero-grade cut-off is 50,696,578 m^3, showing a 0.19% difference. Results are listed by domain in Table 14.10.

Table 14.10: Block vs. Wireframe Volumes

Domain	Wireframe Volume (000 m^3)	Block Volume (000 m^3)	Volume Difference (%)
East	28,036	28,039	0.01%
West	10,288	10,273	0.15%
Internal Waste	12,469	12,385	0.67%
Total	**50,793**	**50,697**	**0.19%**

Comparing the swath plots of the block grades estimated using OK, ID^3, and NN (Figure 14.9, to Figure 14.11) shows that there is slight smoothing in the OK estimate as expected. Since the OK results are very similar to the ID^3 results, RPA decided to use the blocks estimated by the OK method for the Mineral Resource reporting. The swath plots compare the mean block grades estimated by the three interpolation methods to the composite grades in the X, Y, and Z directions.

The visual inspection of the block grades versus the composite data on sections and level plans indicates that the OK interpolation performed well (Figure 14.12 to Figure 14.16).

Figure 14.9: East Swath Plot

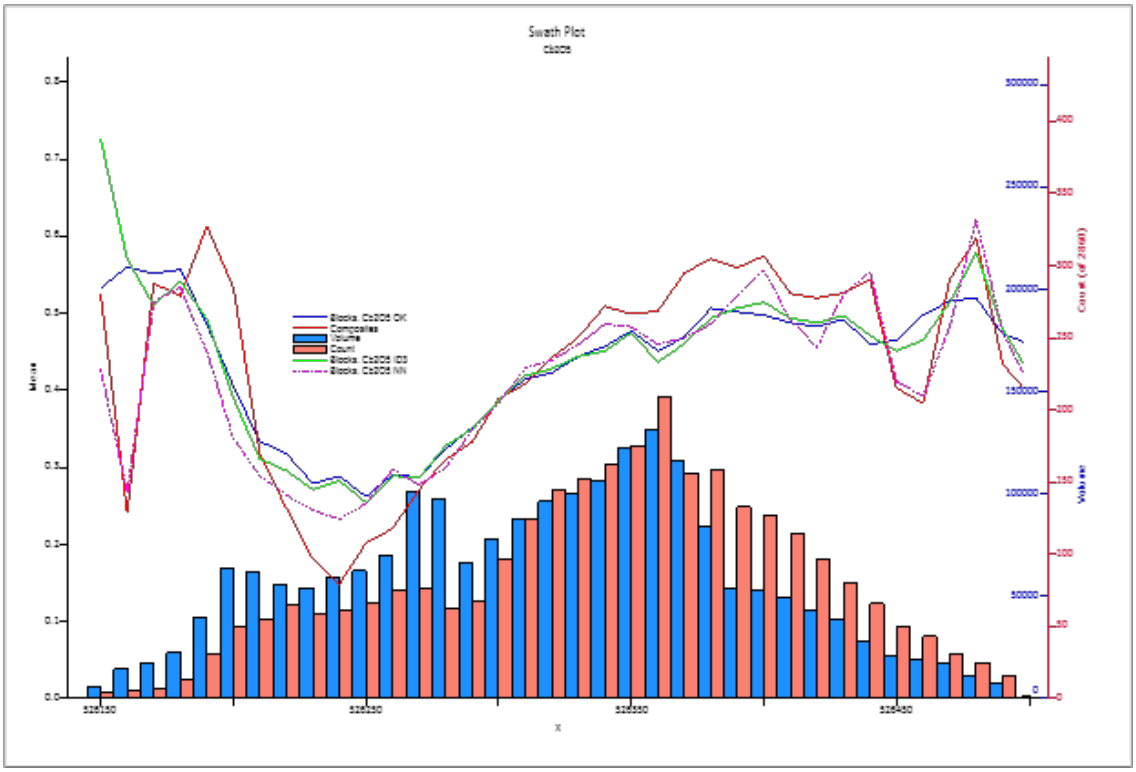


Figure 14.10: North Swath Plot

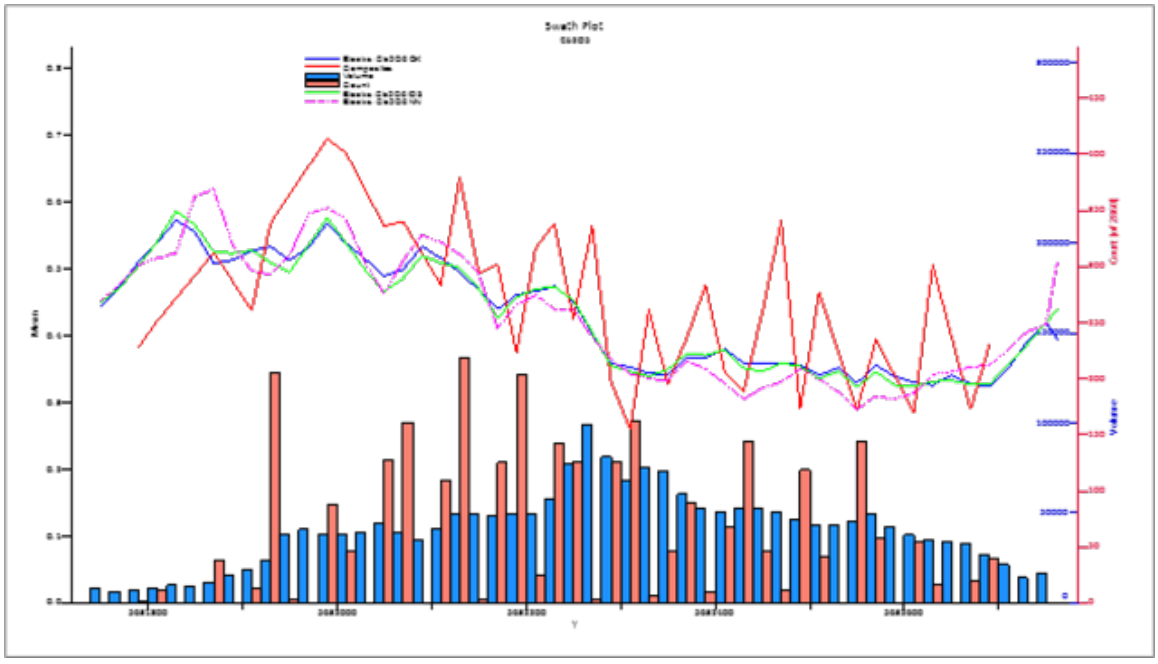


Figure 14.11: Vertical Swath Plot

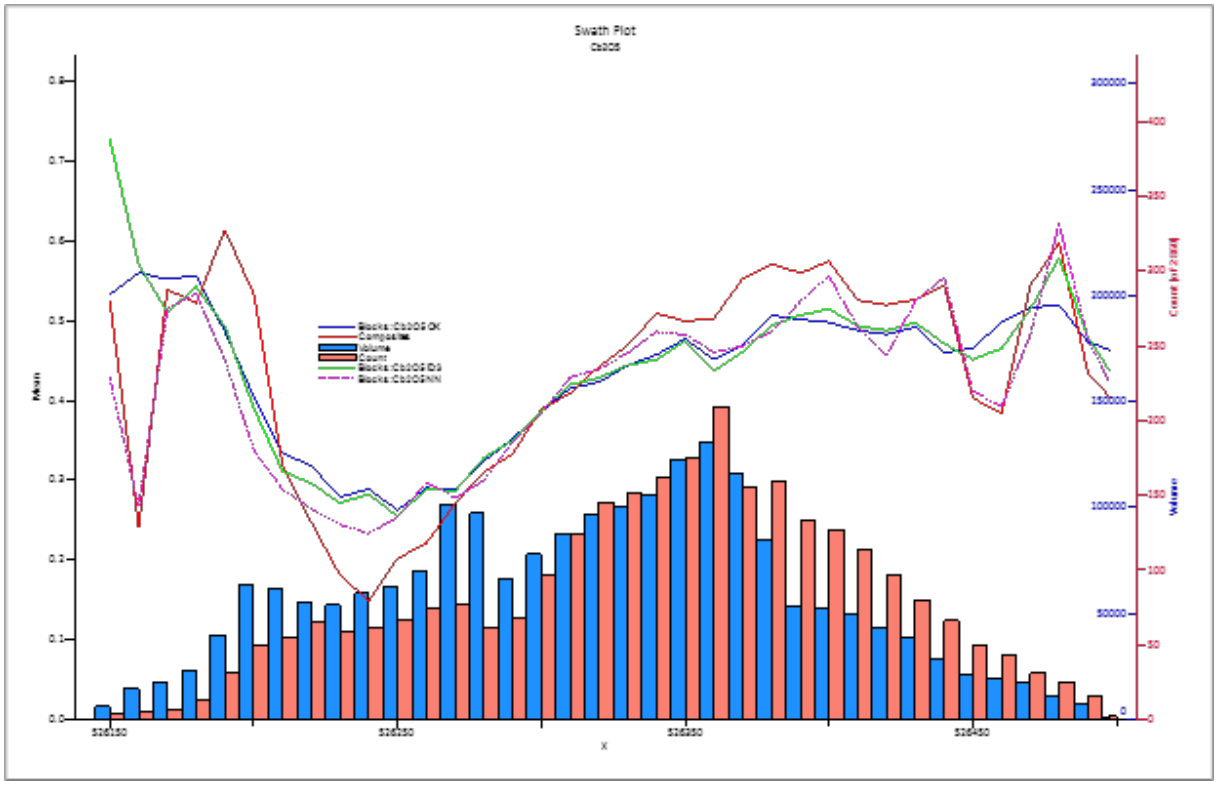


Figure 14.12: Section 800N showing Estimate Block Grades vs. Composites

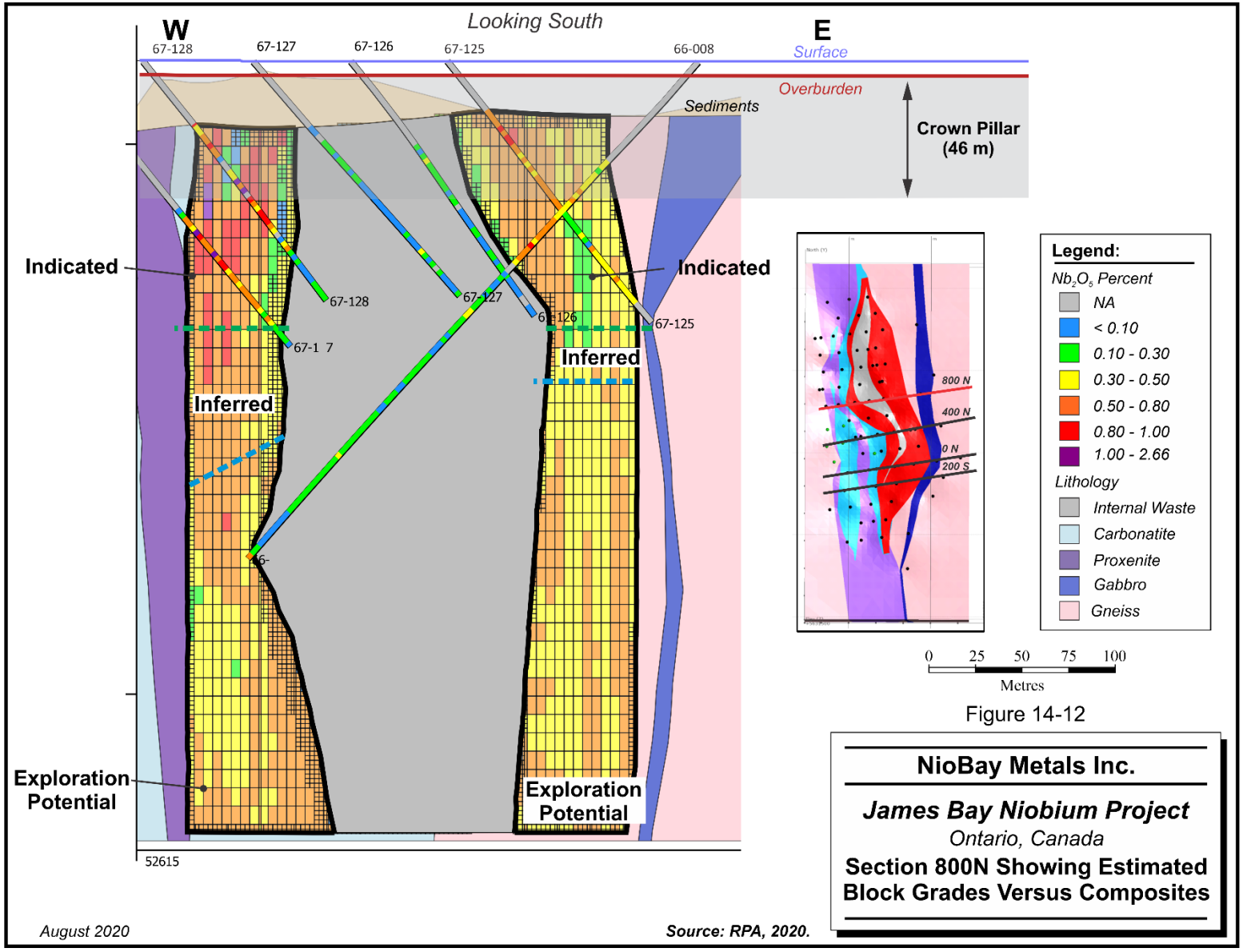


Figure 14.13: Section 400N showing Estimates Block Grades vs. Composites

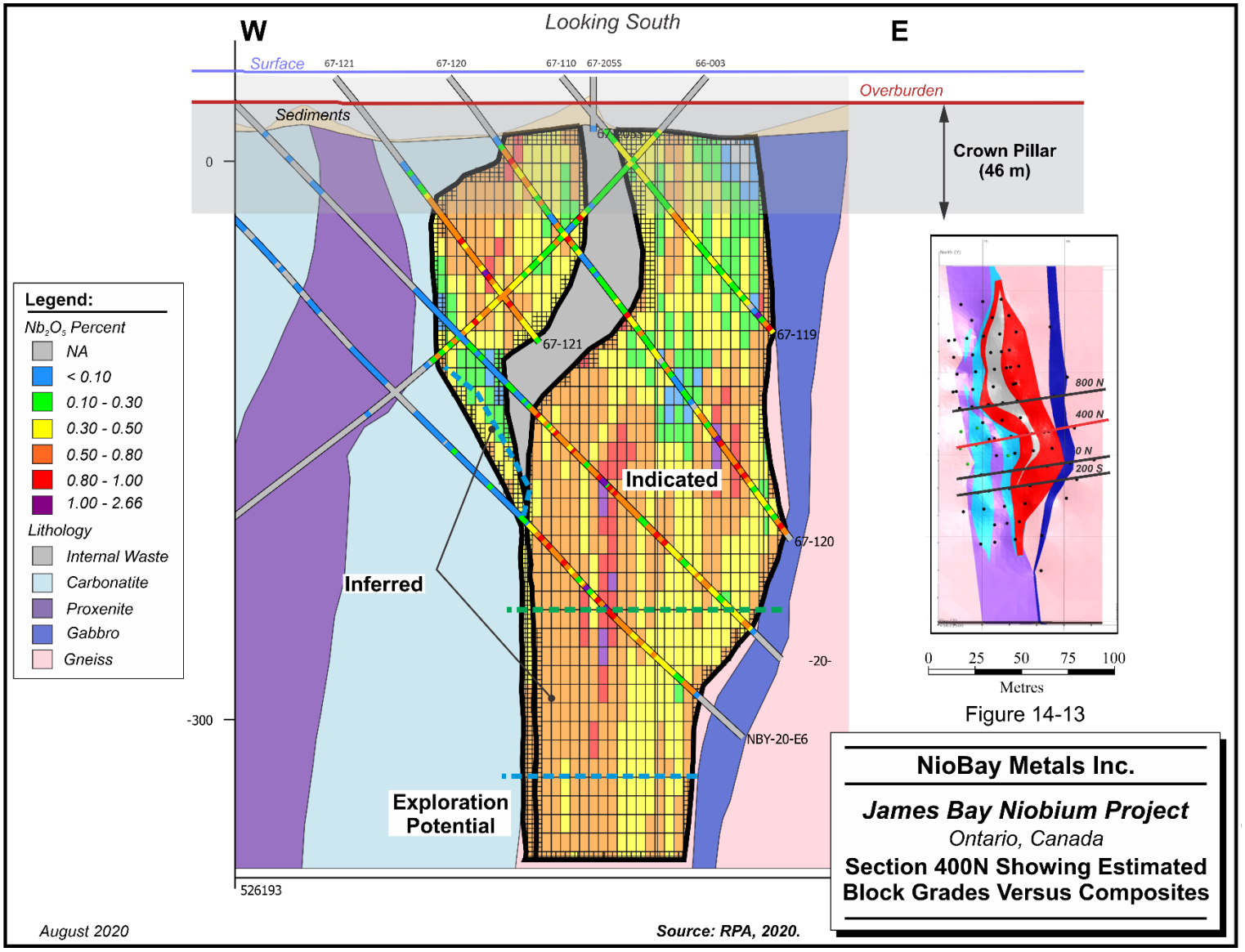


Figure 14.14: Section 0N showing Estimated Block Grades vs. Composites

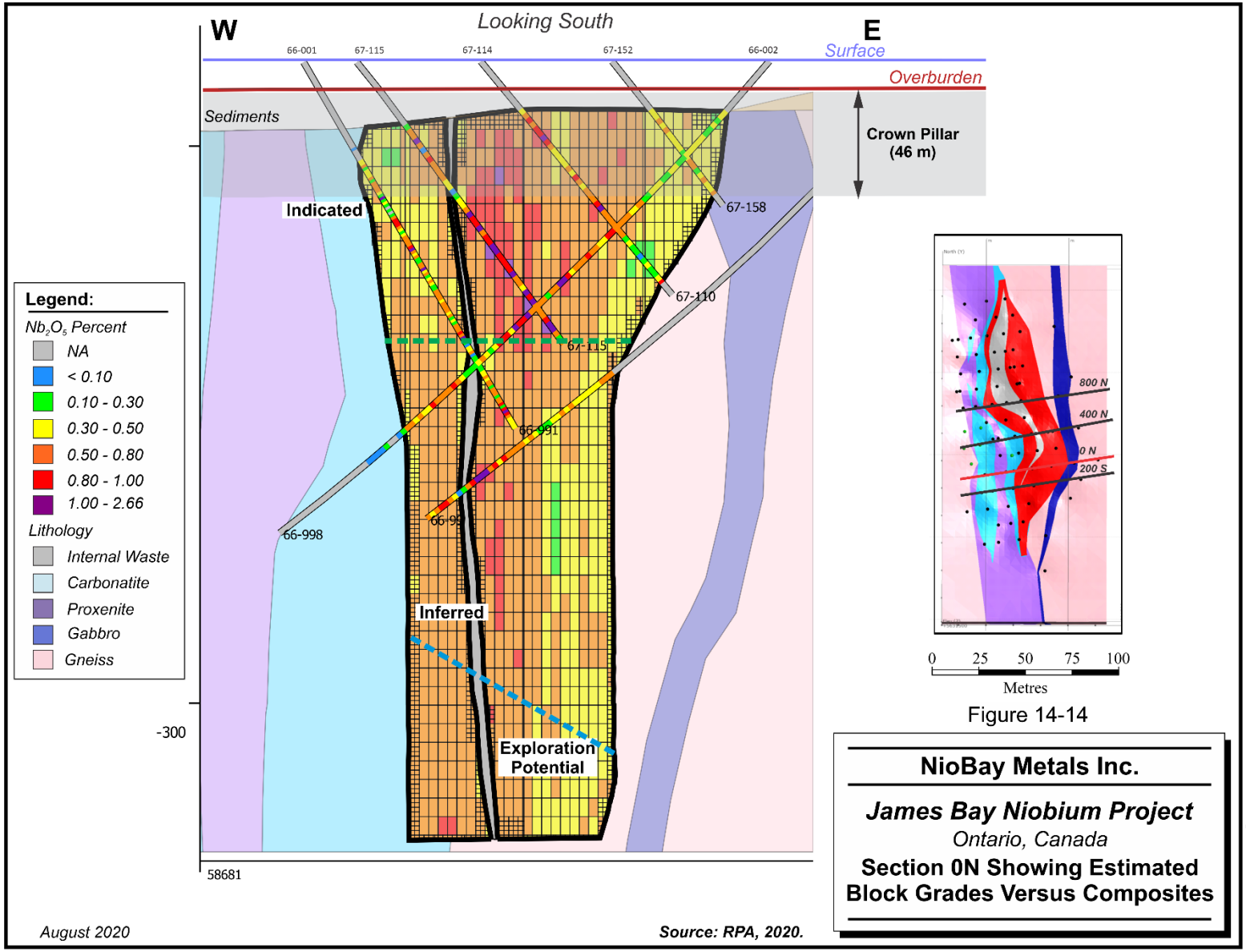


Figure 14.15: Section 200S showing Estimated Block Grades vs. Composites

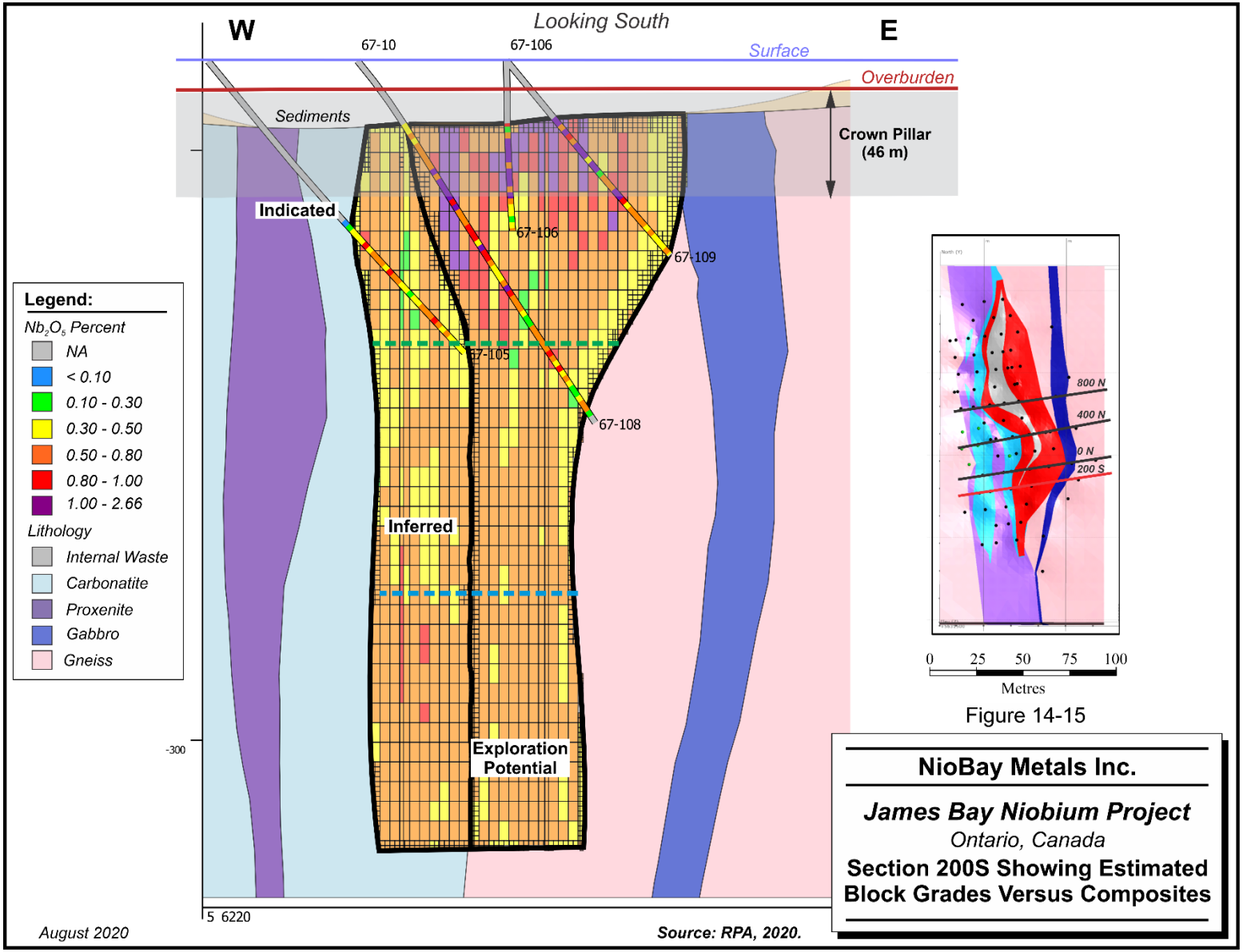
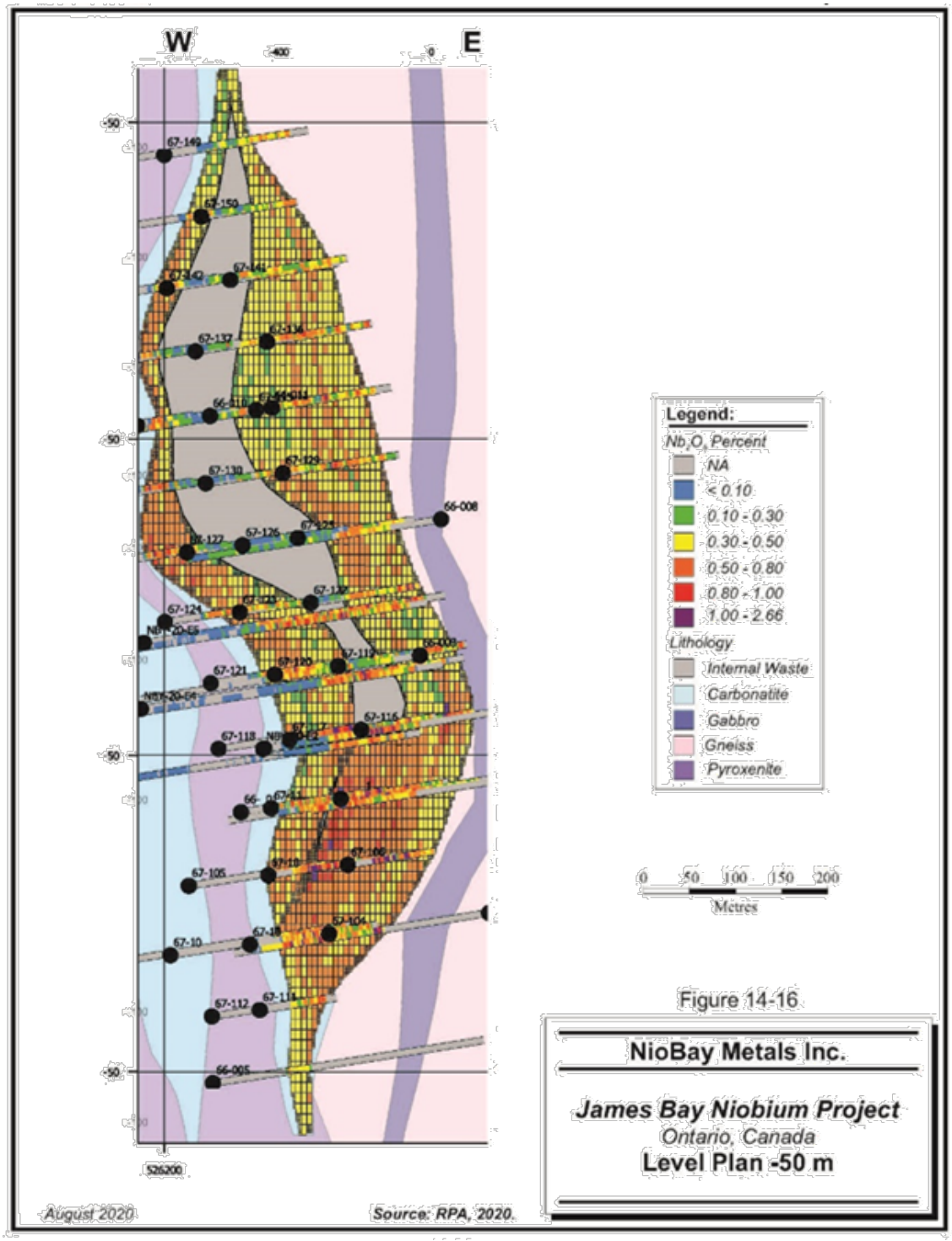


Figure 14.16: Level Plan -50 M



Descriptive statistics of the block model are illustrated in Table 14.11, and a comparison of the minimum, maximum, and mean of the assay, composite, and block grades is presented in Table 14.12.

Table 14.11: Basic Statistics of Nb_2O_5 Block Grades

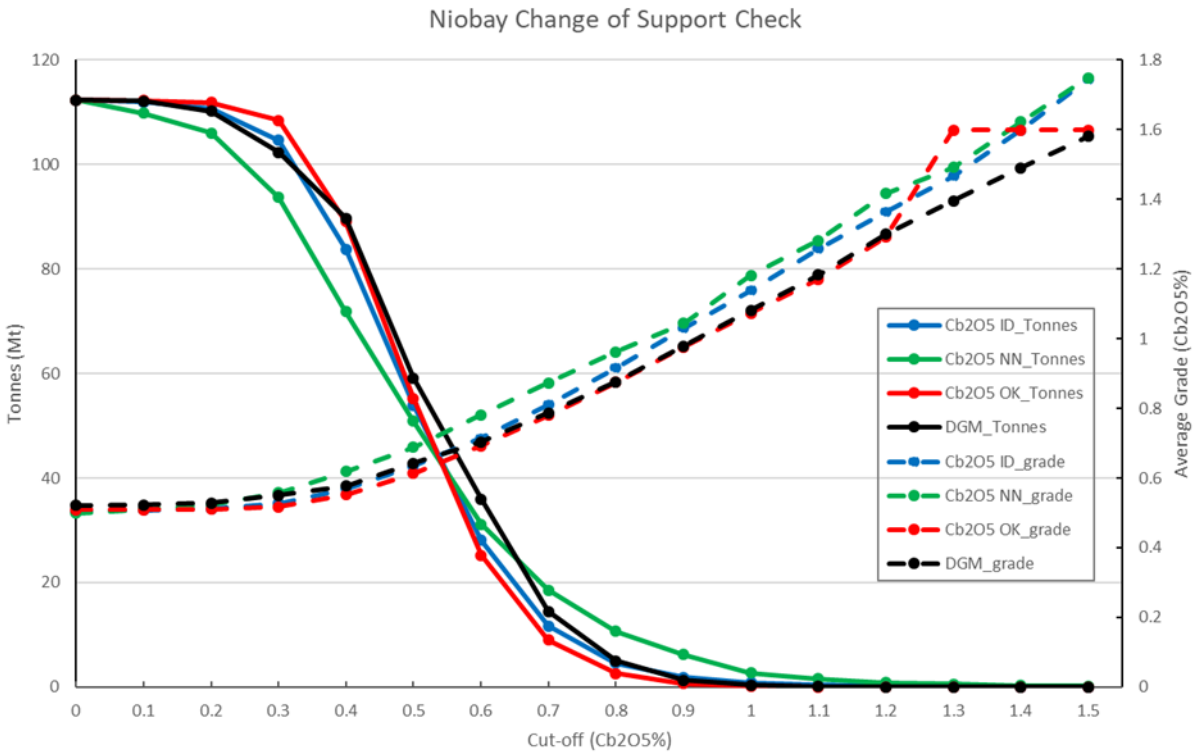
Domain	Count	Min ($\%Nb_2O_5$)	Max ($\%Nb_2O_5$)	Mean ($\%Nb_2O_5$)	Variance	Std. Dev. ($\%Nb_2O_5$)	CV
East	244,246	0.00	2.01	0.50	0.03	0.16	0.32
West	154,137	0.03	1.36	0.52	0.02	0.14	0.28

Table 14.12: Comparison of Assay, Composite and Block Grades

	Assay			Composites			Block Model		
Domain	Min ($\%Nb_2O_5$)	Max ($\%Nb_2O_5$)	Mean ($\%Nb_2O_5$)	Min ($\%Nb_2O_5$)	Max ($\%Nb_2O_5$)	Mean ($\%Nb_2O_5$)	Min ($\%Nb_2O_5$)	Max ($\%Nb_2O_5$)	Mean ($\%Nb_2O_5$)
East	0.00	2.66	0.54	0.00	2.16	0.54	0.00	2.01	0.50
West	0.00	1.77	0.51	0.00	1.45	0.51	0.03	1.36	0.52

RPA performed a Global Change of Support Check (“GCOS”) using the composite variogram and the NN model. The Discrete Gaussian Model (“DGM”) was used to correct the NN grades and then compared to the OK and ID estimates. Above the cut-off grade, the DGM reports slightly less tonnes at a similar grade to both the ID and OK estimates. RPA noted that the Inferred tonnes showed the largest variance while the Indicated ID and OK tonnes were very close to the DGM tonnes. The results suggest that the block model grade and tonnes are reasonable for the proposed scale of mining (Figure 14.17).

Figure 14.17: Global Change of Support Check



14.9 Mineral Resource Reporting

The Mineral Resource estimate has an effective date of July 9, 2020 (Table 14.13).

Table 14.13: Summary of Mineral Resources – July 9, 2020

Category	Tonnage (Mt)	Grade (%Nb_2O_5)	Contained Nb_2O_5 (M kg)
Indicated	29.7	0.53	158
Inferred	33.8	0.52	177

Notes:

1. *CIM (2014) definitions were followed for Mineral Resources.*
2. *Mineral Resources are reported using a cut-off grade of 0.3% Nb_2O_5 based on an Underground Mining Scenario, an operating cost of CAD 70/t, and a metallurgical recovery of 70%.*
3. *Mineral Resources are estimated using a long-term niobium price of USD 40 per kg and a USD/CAD exchange rate of 1:1.2.*
4. *A minimum mining width of approximately 7.5 m was used.*
5. *Bulk density is 2.93 t/m^3.*
6. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
7. *Resources situated in a 46 m thick crown pillar have been excluded*
8. *Numbers may not add due to rounding.*

14.10 Comparison to the Previous Mineral Resource Estimate

The current Mineral Resource estimate, dated July 9, 2020, supersedes the previous Mineral Resource estimate dated October 31, 2018. The current model incorporates additional drilling (3,090 m) completed in 2020. All the price and recovery assumptions remain unchanged. Table 14.14 compares the two estimates.

Table 14.14: Comparison to Previous Mineral Resource Estimate

Category	Tonnage (Mt)	Grade (%Nb_2O_5)	Contained Nb_2O_5 (M kg)
2018			
Indicated	26.1	0.53	139
Inferred	25.3	0.51	129
2020			
Indicated	29.7	0.53	158
Inferred	33.8	0.52	177
Differences			
Indicated	13.8%	0.0%	13.7%
Inferred	33.6%	2.0%	37.2%

The seven boreholes completed during the 2020 drilling campaign confirmed niobium mineralization at depth, which extended resource envelope by approximately 80 m downdip. All the new drilling intersected significant mineralization (above 0.5% Nb_2O_5) which added confidence in the continuity of the pyrochlore zone and thus improved classification of the Mineral Resources at depth.

15. MINERAL RESERVE ESTIMATE

There are no current Mineral Reserves estimated for the Project.

16. MINING METHODS

16.1 Introduction

The James Bay Niobium Deposit is amenable to both open pit mining and underground mining methods given the proximity to surface and the value of the mineralization. The orebody sub-crops in the basement formation overlain by sediments and overburden ranging from 10 to 20 m in thickness.

Three scenarios were investigated and are presented for the James Bay deposit:

- Scenario 1: Open Pit Only.
- Scenario 2: Open and Underground.
- Scenario 3: Underground Only.

16.2 Open Pit Mining Method

16.2.1 Pit Optimization

Open pit optimization was undertaken using Whittle software to determine the optimal economic shape of the open pit to guide practical pit designs. For this report the indicated and inferred mineralization was considered for optimization purposes. The pit optimization parameters are summarized in Table 16.1. The open pit marginal cut-off grade is estimated at about 0.10% Nb_2O_5. However, a raised cut-off grade ("COG") of 0.12% Nb_2O_5 was applied in the optimization process and mineable resource estimates.

Table 16.1: Pit Optimization Parameters

Open Pit Optimization Parameters		Value
Niobium Price	USD/kg Nb	45.00
Marketing & Converter Costs	USD/kg Nb	5.00
Royalty Costs (2%)	USD/kg Nb	0.90
Exchange Rate	CAD/USD	1.30
Concentrator Nb_2O_5 Recovery	%	78.0%
Converter Recovery	%	96.5%
Yield	Nb_2O_5 kg/t	75.3%
Niobium Production Ratio	kg Nb/t mill feed	52.6%
Total Ore-Based Cost	CAD/t mill feed	23.00
Mining Dilution	%	4.0
Mining Operating Cost	CAD/t mined	4.75
Overburden Slope	Degrees	26
Rock Slope	Degrees	45

A small creek passes through the middle of the deposit as presented in Figure 16.1. For pit optimization, a scenario was evaluated where a buffer of 50 m on each side of the centerline of the creek was imposed in order to leave the creek intact with results presented in Table 16.2 and Figure 16.2, and another where it is removed and would require a deviation of the creek around the open pit (Table 16.3 and Figure 16.3).

The Open Pit Only Scenario requires a relocation of the creek whereas the Combined Open Pit and Underground Scenarios do not.

Figure 16.1: 50 m Creek Buffer Zone

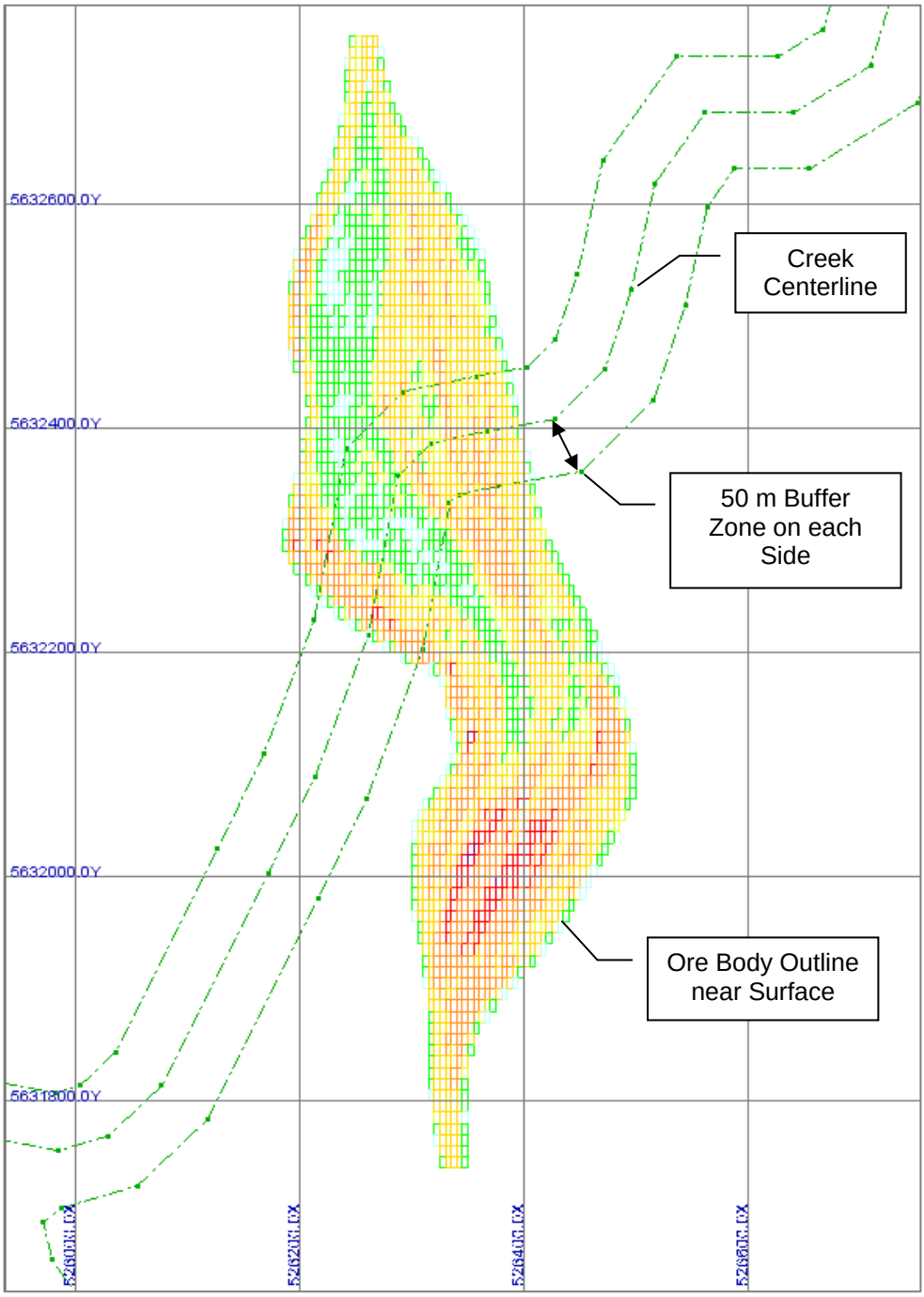


Table 16.2: Pit Shell Results with 50 m Buffer

Pit Shell	Rev. Factor	Nb Price (USD/kg)	Total (kt)	Mill Feed (kt)	Strip Ratio	Grade % Nb_2O_5	Contained M kg Nb_2O_5
1	0.30	13.50	3,721	1,318	1.82	0.72	9.5
2	0.32	14.40	7,348	3,114	1.36	0.64	20.0
3	0.34	15.30	10,126	4,626	1.19	0.60	27.8
4	0.36	16.20	12,500	5,800	1.16	0.58	33.5
5	0.38	17.10	14,163	6,549	1.16	0.56	36.9
6	0.40	18.00	16,584	7,517	1.21	0.55	41.1
7	0.42	18.90	18,302	8,156	1.24	0.54	43.8
8	0.44	19.80	19,916	8,694	1.29	0.53	46.0
9	0.46	20.70	23,738	9,895	1.40	0.51	50.7
10	0.48	21.60	25,406	10,428	1.44	0.50	52.6
11	0.50	22.50	32,021	12,128	1.64	0.49	59.0
12	0.52	23.40	34,626	12,816	1.70	0.48	61.4
13	0.54	24.30	36,172	13,253	1.73	0.47	62.8
14	0.56	25.20	38,591	13,890	1.78	0.47	64.8
15	0.58	26.10	39,775	14,271	1.79	0.46	65.9
16	0.60	27.00	40,514	14,511	1.79	0.46	66.5
17	0.62	27.90	42,044	14,902	1.82	0.45	67.6
18	0.64	28.80	43,708	15,272	1.86	0.45	68.7
19	0.66	29.70	44,089	15,411	1.86	0.45	69.0
20	0.68	30.60	45,170	15,661	1.88	0.44	69.7
21	0.70	31.50	46,049	15,873	1.90	0.44	70.2
22	0.72	32.40	46,670	16,010	1.92	0.44	70.5
23	0.74	33.30	46,829	16,096	1.91	0.44	70.7
24	0.76	34.20	47,542	16,264	1.92	0.44	71.1
25	0.78	35.10	48,410	16,415	1.95	0.44	71.5
26	0.80	36.00	48,532	16,427	1.95	0.44	71.5
27	0.82	36.90	49,764	16,567	2.00	0.43	71.9
28	0.84	37.80	51,160	16,727	2.06	0.43	72.4
29	0.86	38.70	51,397	16,748	2.07	0.43	72.5
30	0.88	39.60	51,430	16,756	2.07	0.43	72.5
31	0.90	40.50	51,453	16,759	2.07	0.43	72.5
32	0.92	41.40	52,736	16,869	2.13	0.43	72.9
33	0.94	42.30	52,808	16,878	2.13	0.43	72.9
34	0.96	43.20	53,600	16,946	2.16	0.43	73.1
35	0.98	44.10	53,915	16,969	2.18	0.43	73.2
36	1.00	45.00	55,045	17,065	2.23	0.43	73.5

Table 16.3: Pit Shell Results no Buffer Zone

Pit Shell	Rev. Factor	Nb Price (USD/kg)	Total (kt)	Mill Feed (kt)	Strip Ratio	Grade % Nb_2O_5	Contained M kg Nb_2O_5
1	0.30	13.50	4,429	1,635	1.71	0.71	11.6
2	0.32	14.40	9,101	4,064	1.24	0.63	25.8
3	0.34	15.30	15,452	7,318	1.11	0.59	43.1
4	0.36	16.20	44,766	19,504	1.30	0.54	105.1
5	0.38	17.10	68,951	28,961	1.38	0.52	150.3
6	0.40	18.00	90,319	36,937	1.45	0.50	186.5
7	0.42	18.90	114,767	44,504	1.58	0.50	220.7
8	0.44	19.80	130,499	49,026	1.66	0.49	240.7
9	0.46	20.70	148,387	53,815	1.76	0.49	261.6
10	0.48	21.60	162,303	57,472	1.82	0.48	276.8
11	0.50	22.50	170,368	59,895	1.84	0.48	285.9
12	0.52	23.40	176,287	61,915	1.85	0.47	292.7
13	0.54	24.30	182,442	63,857	1.86	0.47	299.3
14	0.56	25.20	187,928	65,659	1.86	0.46	305.1
15	0.58	26.10	191,164	67,009	1.85	0.46	309.1
16	0.60	27.00	196,256	68,709	1.86	0.46	314.1
17	0.62	27.90	197,971	69,692	1.84	0.45	316.3
18	0.64	28.80	203,267	71,272	1.85	0.45	321.1
19	0.66	29.70	208,109	72,646	1.86	0.45	325.7
20	0.68	30.60	208,397	73,313	1.84	0.45	326.7
21	0.70	31.50	210,671	74,314	1.83	0.44	328.9
22	0.72	32.40	211,628	75,101	1.82	0.44	330.4
23	0.74	33.30	212,268	75,692	1.80	0.44	331.4
24	0.76	34.20	212,502	76,240	1.79	0.44	332.2
25	0.78	35.10	214,597	77,090	1.78	0.43	334.0
26	0.80	36.00	214,762	77,136	1.78	0.43	334.1
27	0.82	36.90	215,347	77,281	1.79	0.43	334.5
28	0.84	37.80	216,896	77,558	1.80	0.43	335.7
29	0.86	38.70	217,166	77,633	1.80	0.43	335.9
30	0.88	39.60	217,490	77,706	1.80	0.43	336.1
31	0.90	40.50	217,898	77,785	1.80	0.43	336.3
32	0.92	41.40	217,903	77,788	1.80	0.43	336.4
33	0.94	42.30	217,932	77,799	1.80	0.43	336.3
34	0.96	43.20	219,813	78,122	1.81	0.43	337.3
35	0.98	44.10	221,338	78,371	1.82	0.43	338.5
36	1.00	45.00	221,411	78,397	1.82	0.43	338.5

Figure 16.2: Pit by Pit Results for 50 m Buffer Zone

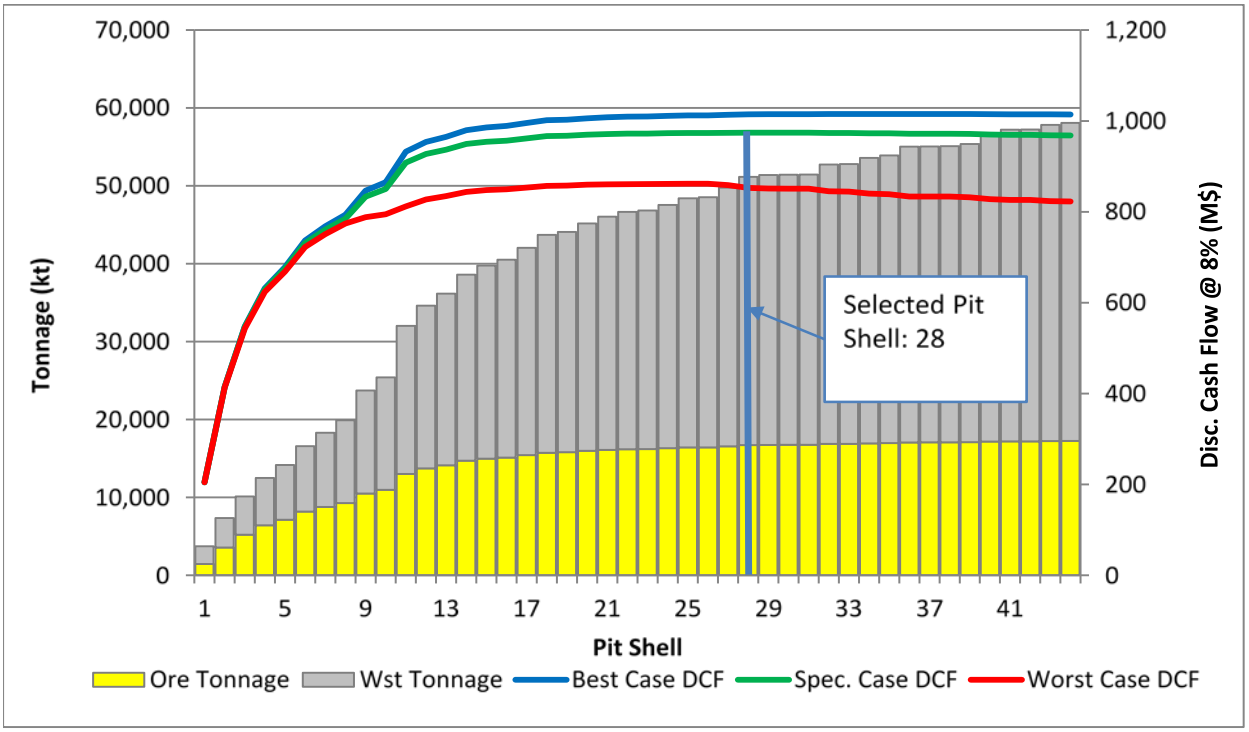
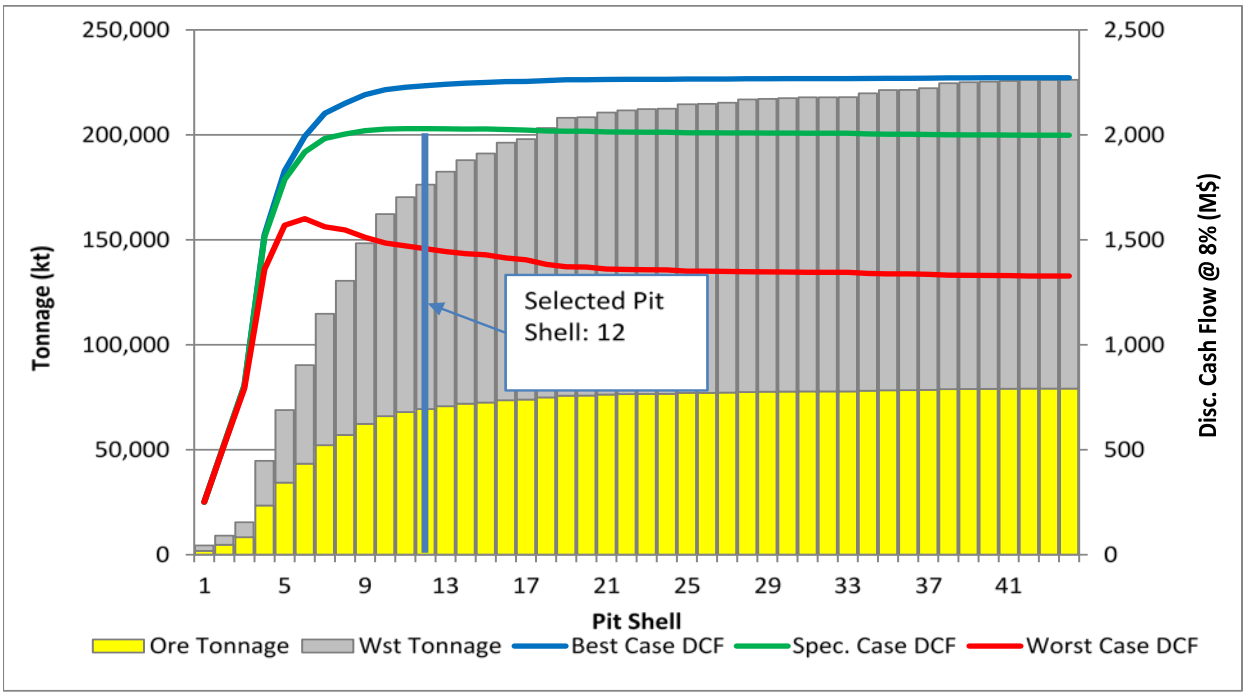


Figure 16.3: Pit by Pit Results no Buffer Zone



The selected pit shells to guide pit designs are summarized for both scenarios in Table 16.4. Essentially, the no buffer zone selected shell represents the final shell for the Open Pit Only Scenario. The shell selection with 50 m buffer zone represents the pits that would be mined under the Combined Open Pit and Underground Scenarios and would also be mining phases in the Open Pit Only Scenario which would allow for the deferral of the creek diversion.

Table 16.4: Pit Shell Selection

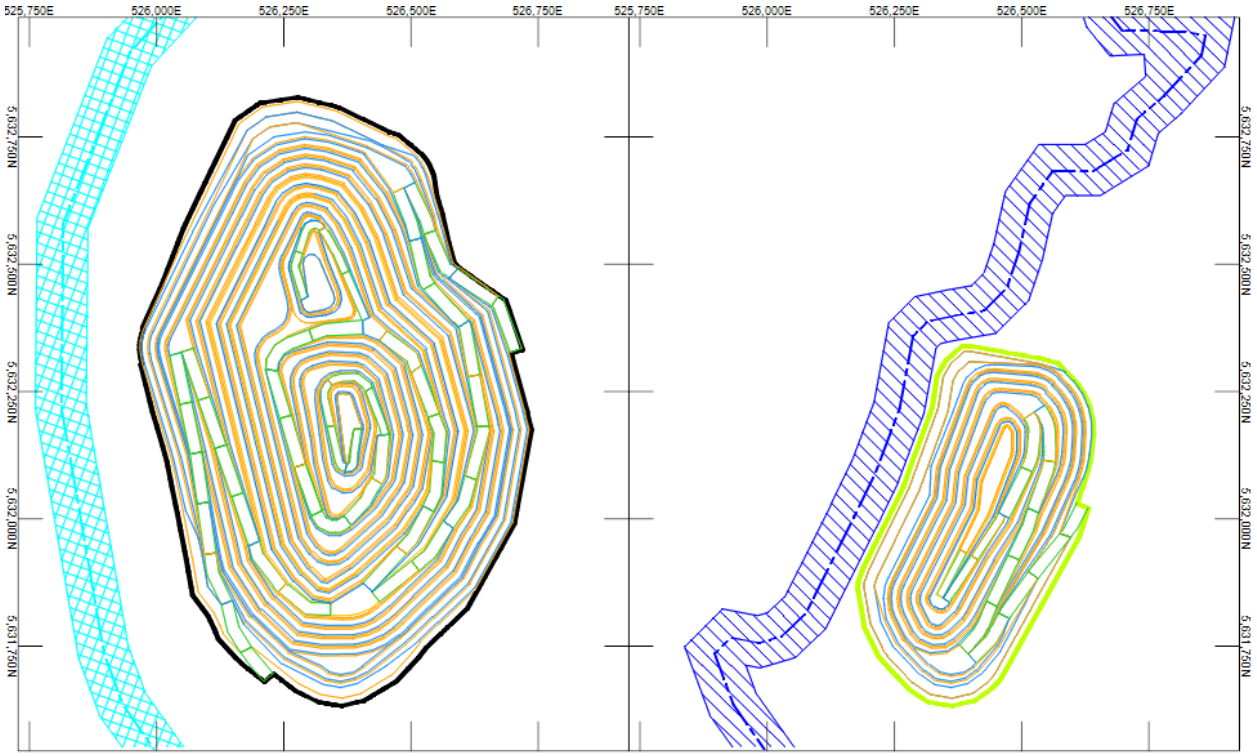
Shell Selection	50 m Buffer Zone	No Buffer Zone
Shell Number	28	12
Shell RF	0.840	0.520
Shell Price	38	23
Total Tonnage (kt)	51,160	176,287
Waste Tonnage (kt)	34,433	106,914
Strip Ratio (W:O)	2.1	1.5
Mill Feed Tonnage (kt)	16,727	69,373
Grade (% Nb_2O_5)	0.43	0.44
Contained Metal (M kgs Nb_2O_5)	72	304
$NVP_{8\%}$ Operating Cash Flow (M$)	974	2,030

With the creek buffer zone applied, the pit shells contain 16.7 Mt of mill feed at 0.43% Nb_2O_5 with a strip ratio of 2.1. With the buffer zone removed, the larger pit shell contains 69.7 Mt of mill feed at 0.44% Nb_2O_5 with a strip ratio of 1.5.

16.2.2 Open Pit Mine Designs

Specific open pit designs were generated for Scenario 1 and Scenario 2. In the Open Pit Only Scenario there will be no underground option, allowing the final pit to be much bigger and deeper than otherwise possible. In the Combined Scenario there is both an open pit and underground mine. The open pit size is limited in the Combined Scenario and the river will not be rerouted to increase pit size as the quantity of mill feed that would be gained from that will be taken by the underground portion albeit with a crown pillar sterilizing some mineralization. Open Pit Only Scenario contains four (4) phases nested within one another. The Combined Scenario contains only the first two (2) phases of the Open Pit Only Scenario. Figure 16.4 depicts the end of life pits of both scenarios side by side. The lighter blue river represents the rerouted river and the darker blue hatch is the original untouched river, both with a 50 m buffer on both sides.

Figure 16.4: Open Pit Only Scenario vs. Combined Scenario



The tonnage differences between the two (2) scenarios is significant, a comparative summary of the two scenarios are depicted in Table 16.5. The Large Pit Only Scenario is approximately 3.5 times larger in total tonnage producing 81.6 Mt more mill feed at a similar overall stripping ratio. Note that the numbers presented in this section refer only to open pit mill feed.

Table 16.5: Open Pit Design and Scenario Summary

		Open Pit Only Scenario					Combined Scenario		
Phase Design Content		**S_Ph1**	**S_Ph2**	**C_Ph1**	**C_Ph 2**	**Total**	**S_Ph1**	**S_Ph2**	**Total**
Total Tonnage	*kt*	15,139	11,553	50,916	120,535	**198,143**	15,139	11,553	**26,692**
Waste Rock	*kt*	8,299	8,147	25,105	78,074	**106,104**	8,299	8,147	**16,446**
Sediments	*kt*	-	-	-	-	**15,264**	-	-	**0**
Overburden	*kt*	-	-	-	7,674	**21,194**	-	-	**0**
Total Waste	*kt*	8,299	8,147	25,105	85,748	**127,299**	8,299	8,147	**16,446**
Strip Rate	*W:O*	1.21	2.39	0.97	2.46	**1.8**	1.21	2.39	**1.60**
Mill Feed Tonnage	*kt*	6,840	3,407	25,811	34,787	**70,845**	6,840	3,407	**10,247**
Grade	*%Nb_2O_5*	0.54	0.42	0.44	0.42	**0.44**	0.54	0.42	**0.50**

The shell hierarchy and the corresponding Whittle Shells that guided the design for the two scenarios are depicted in Table 16.6.

Table 16.6: Shell Hierarchy

Open Pit Scenario	
Phasing	**Whittle Shell**
→ Combined Final (C Ph2)	No Offset Shell #10
→ Combined Phase 1 (C_Ph1)	No Offset Shell #3
→ South Pit Final (S_Ph1)	50 m Offset Shell #28
→ South Pit Phase 1 (S_Ph2)	50 m Offset Shell #5
Combined Scenario	
Phasing	**Whittle Shell**
→ South Pit Final (S_Ph2)	50 m Offset Shell #28
→ South Pit Phase 1 (S Ph1)	50 m Offset Shell #5

Both mining scenarios use the same geotechnical and design parameters outlined below. All phases are created with the geotechnical parameters outlined in Table 16.7. There are no berms in the overburden zones except at the fresh rock contact zone which establishes a 6 m berm. Ramps are sized according to the specifications and requirements of the CAT 775 with single lane ramp segments consisting of 15 m widths and double lane 24 m. Single lane segments are employed at the bottom four benches to reduce stripping and increase access to mineralization. Box cuts of 10 m are employed in each phase to further reduce the stripping ratio. The processing plant, waste dump and stockpiles are planned for west of the pit and all pit exits are planned accordingly to reduce haulage distance.

Table 16.7: Pit Phase Design Criteria

Slope Parameters	Fresh Rock	Overburden
Final Bench Height (m)	20.0	20.0
Bench Face Angle (º)	70	26
Avg. Design Catch Bench Width (m)	11.4	0
Overall Slope Angle (º)	47	22.2
Geotechnical Benches (m)	0	6[t]

Note: Geotechnical catch berm appear at the Overburden and Fresh contact.

Combined Phase 1 and Combined Final pits are built without the 50 m offset buffer from the river and the river is rerouted north of the pits. Combined Phase 1 shares the east ramp from South Pit Final to access the bottom and as an additional ramp to the north-east to mine surface material.

The Combined Final Pit has an additional ramp to the east that exits near the south to lessen the haulage distances for material on the east side of the pit and adds additional flexibility. The two (2) ramps meet on the southern point of the pit at the halfway point to converge into on primary ramp.

Figure 16.5: Open Pit Only Scenario vs. Combined Scenario - Pit Limits

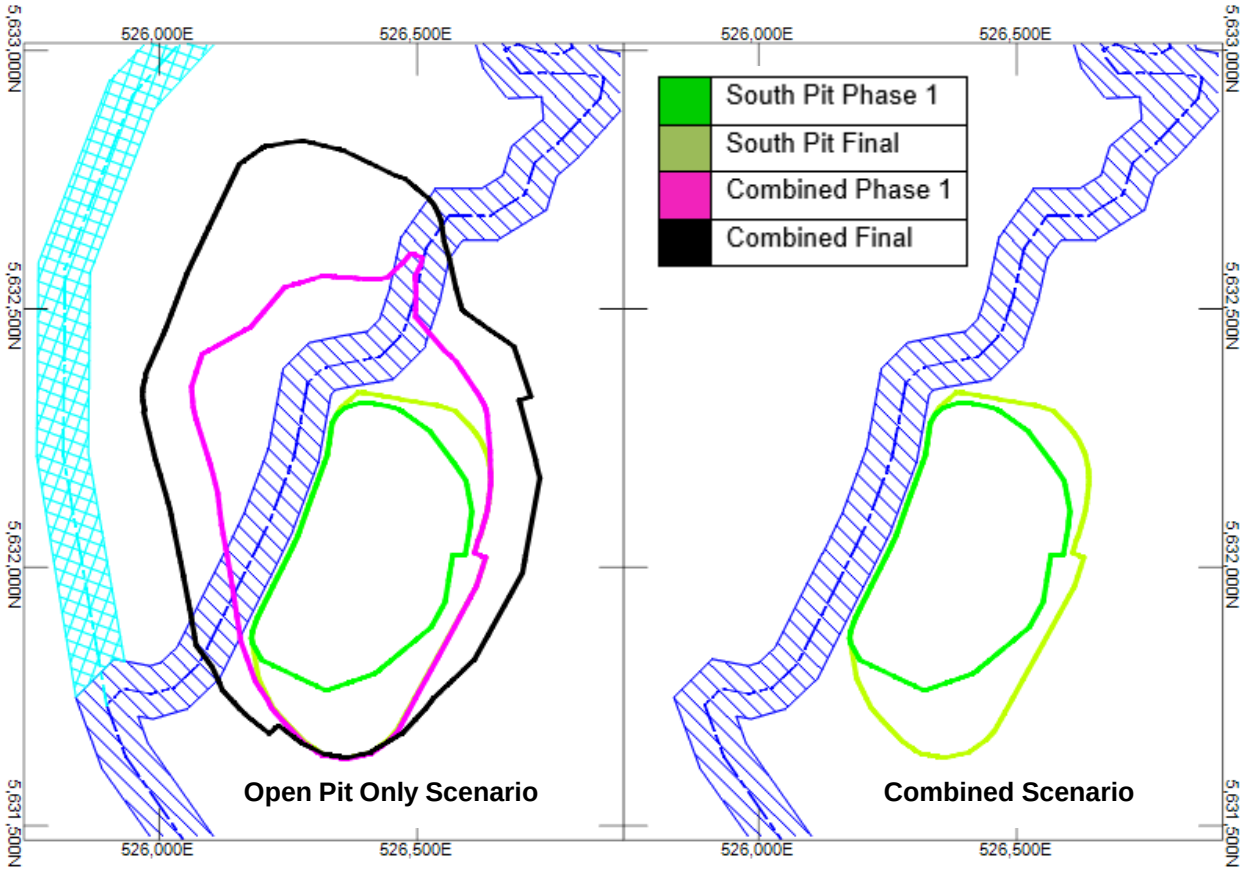


Figure 16.6: Open Pit Only Scenario - Phasing

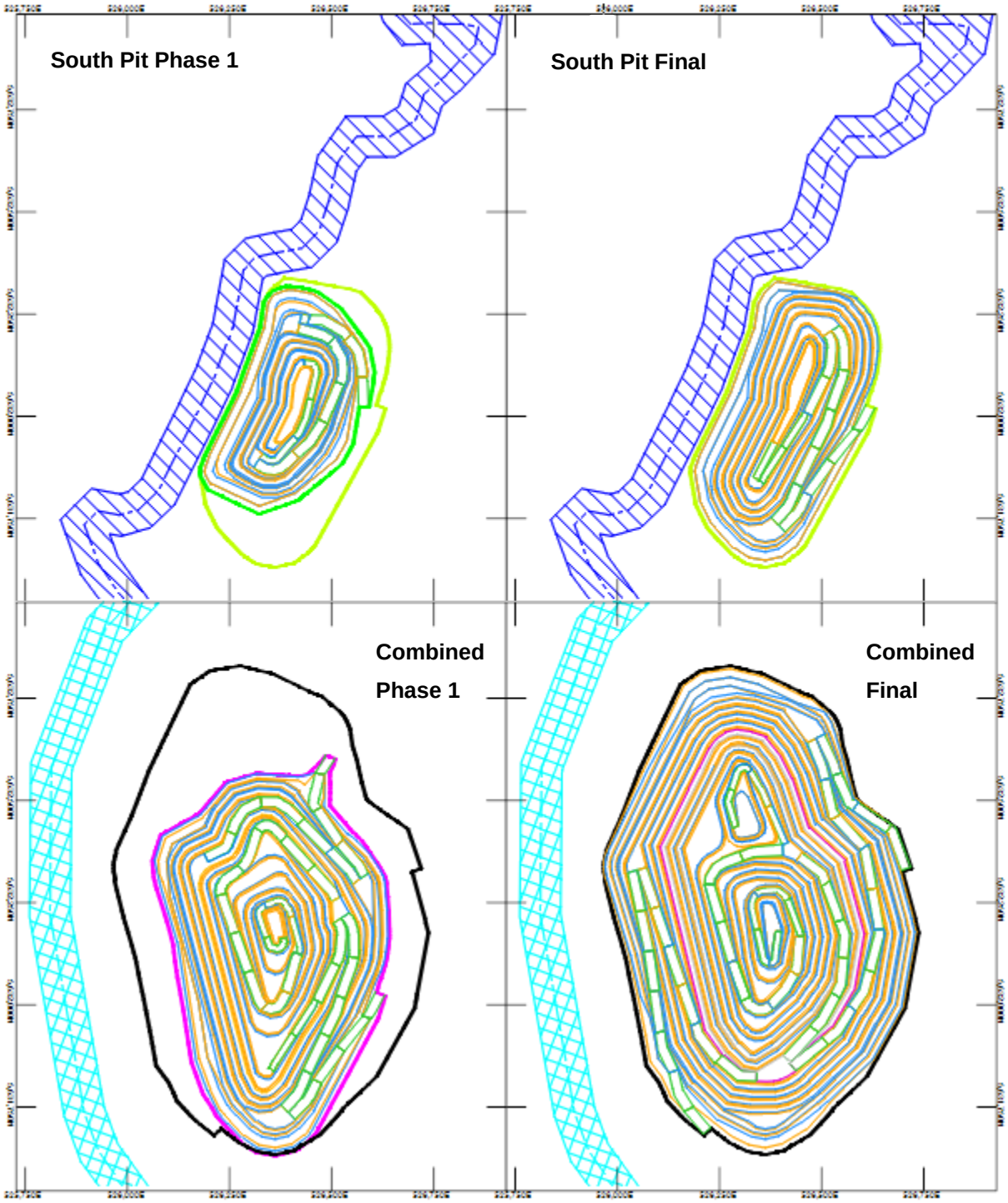
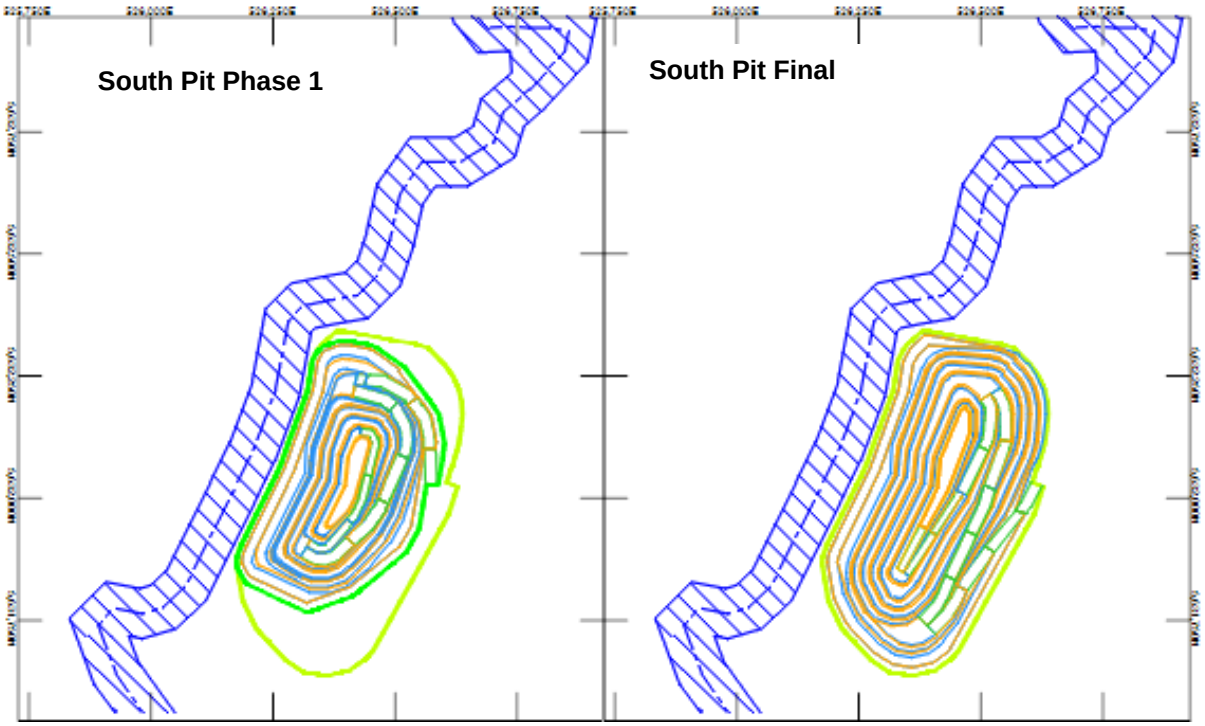


Figure 16.7: Combined Scenario Phasing



16.2.3 Overburden and Waste Rock Storage

In both scenarios the waste rock will be disposed of in a waste rock storage area (“WRSA”) of which there is only one, located south-east of the pit. The primary dump is accessible from both sides to reduce haulage requirements. The ore pad will also require a waste pad beneath it to give a staging area for the ore stockpiles and to reduce dilution from rehandling. The tailings material produced from the processing plant will be transported for disposal within the Tailings Management Facility (“TMF”).

The design criteria of the waste dumps are preliminary and will require a reassessment when further foundation stability assessments and waste rock testing has been done. All rock waste dumps have 20 m high lifts to allow for wider catch benches to facilitate reclamation. All waste dump capacities are shown in Table 16.8.

Table 16.8: Waste Storage Capacities

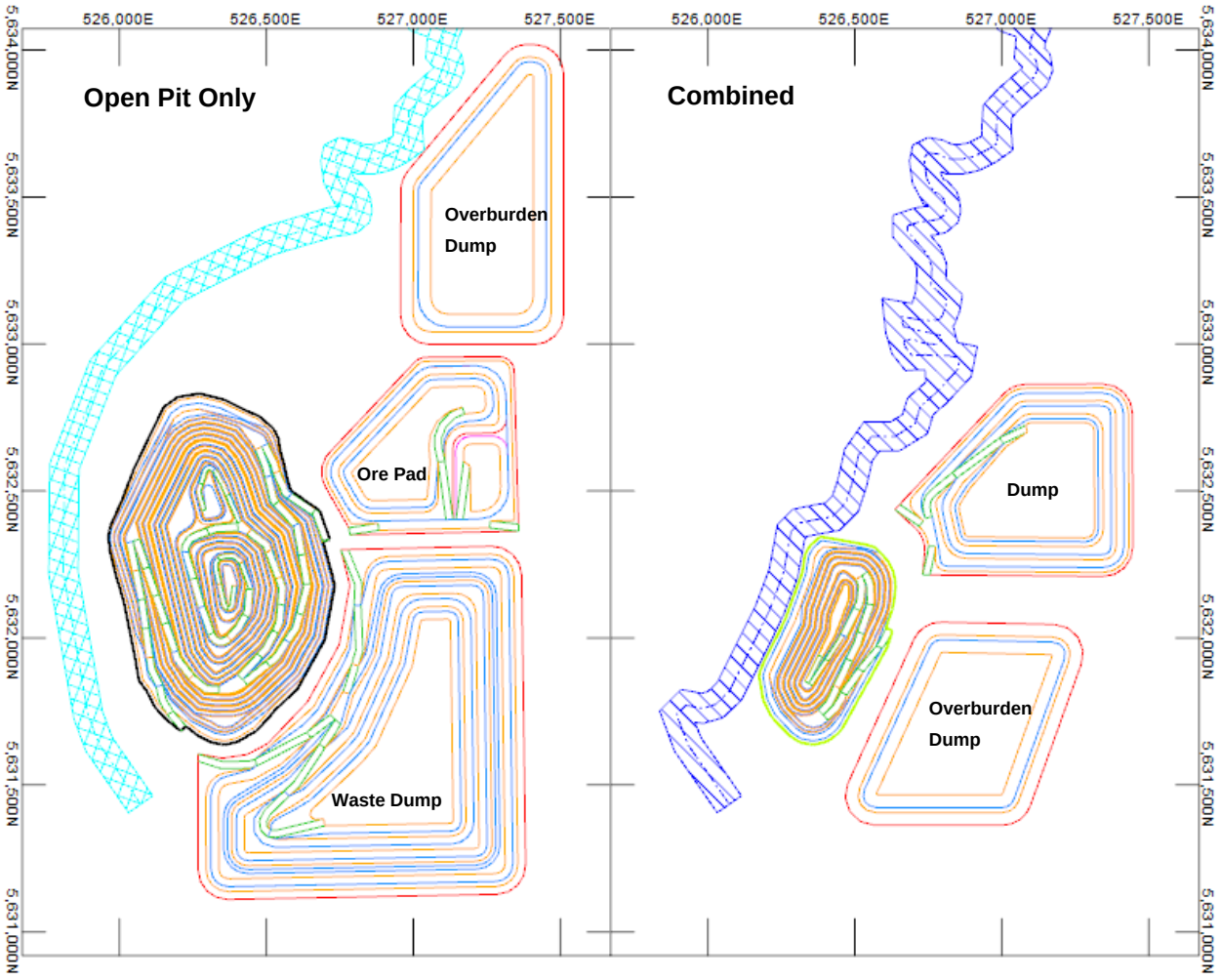
		Open Pit Only Scenario			Combined Scenario		
Dump Designs		**Waste Dump**	**Ore Pad**	**Overburden Dump**	**Waste Dump**	**Ore Pad**	**Overburden Dump**
Capacity	*Mt*	142.87	6.72	21.95	34.04	-	19.57
Capacity	*Mm^3*	66.14	3.11	13.55	15.76	-	12.08
Surface Area	*hc*	101.04	32.79	47.06	40.02	-	41.86
% Filled	*%*	80%	100%	45%	97.30%	-	75.62%

Table 16.9: Waste Pile Design Criteria

Waste Dump	Avg. Catch Bench Width (m)	Pile Face Angle (deg)	Overall Slope Angle (H:V)	Maximum Elevation (m)	Approximate Height (m)
Waste Dump	17	37	2:1	350*	120
Overburden Pile	17	25	3:1	-	40

(*) Height is from the bottom of the pit

Figure 16.8: Waste and Overburden Dumps



16.2.4 Stockpiles

Two (2) grade bins are used to stockpile extra material, the cut off grades and designations depicted in Table 16.10.

Table 16.10: Stockpile Grade Bins Cut-off-Grades

Grade Bins	COG ($\%Nb_2O_5$)
Low Grade	≥0.12 and <0.30
High Grade	≥0.30

The maximum stockpile inventory in the Combined Scenario is 213 kt of low-grade mineralization which occurs in 2024. Due to the minimal amount of stockpile material a flexible stockpile can be placed immediately adjacent to the crusher and can be serviced by a front-end loader.

For the Open Pit Only Scenario two (2) stockpiles are required with a maximum combined inventory of 14.8 Mt in 2048. The low-grade stockpile is significantly larger with a maximum inventory of 13.8 Mt and 2.4 Mt for the low-grade stockpile peaking at different years. The two (2) stockpiles must be separated and placed on a flat waste pad to reduce dilution between different stockpiles and waste material. The ore pad is located between the pits and the crusher allowing a minimal rehandle time and minimal surface hauling from the pit. The current plan has additional stockpile capacity to account for potential changes to the mine plan.

For the Underground Only Scenario a singular stockpile is used between the underground entrance and the processing plant.

The stockpile design criteria are presented in Table 16.11. The capacity found below is for both stockpiled designations.

Figure 16.9 depicts the site layout with the two stockpiles on the waste pad. The dump and the waste pad are made with waste rock and are depicted in orange. The high-grade stockpile is in dark blue and low grade in green.

Table 16.11: Stockpile Design Criteria

Stockpiles	Catch Bench Width (m)	Overall Slope Angle (H:V)	Maximum Elevation (m)	Approximate Height (m)	Max Capacity (Mt)
Low Grade Stockpile	17	2:1	280	40	20.3
High Grade Stockpile	17	2:1	260	20	3.5

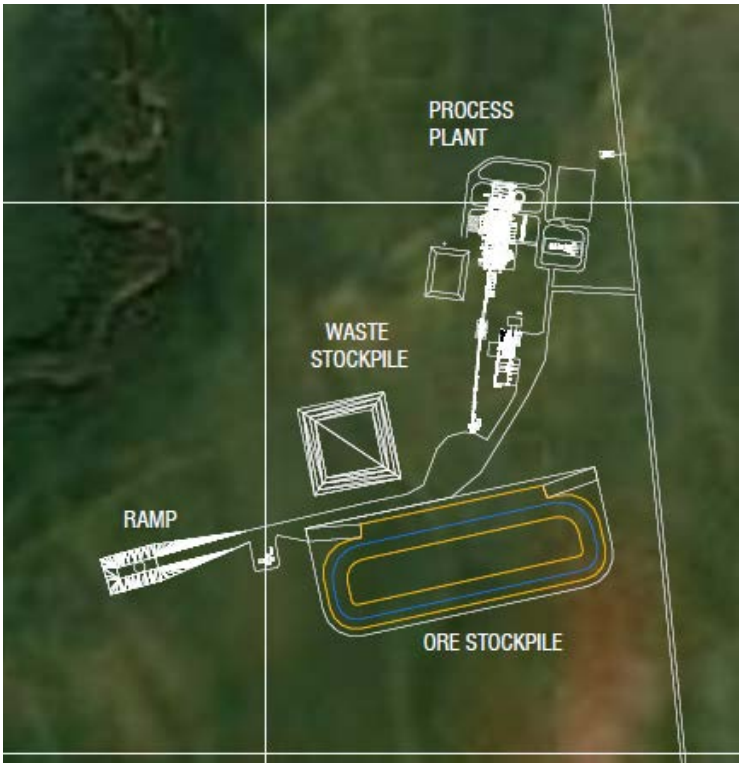
Figure 16.9: Open Pit Only Scenario - Stockpile Layout



Figure 16.10: Combined Scenario - Stockpile Layout



Figure 16.11: Underground Only - Stockpile Layout

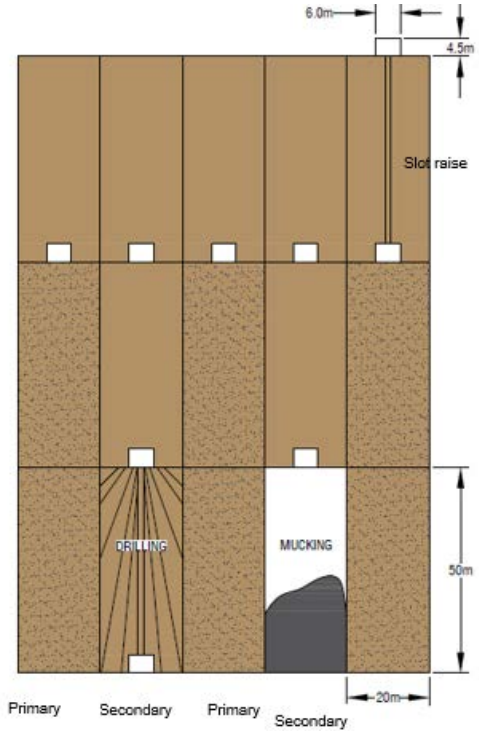


16.3 Underground Mining Method

16.3.1 Underground Method Description

The mining methods used for Scenarios 2 and 3 are similar, with the difference being the tonnage of mineable stopes due to the presence of an open pit for Scenario 2. All the stopes will use transverse long hole with a primary-secondary mining sequence. Transverse long hole stoping (“TLHS”) is a bulk mining method in which the long axis of the stope and access drifts are perpendicular to the strike of the orebody. The draw-points from the undercut access drifts extend from the footwall, and the free face is mined in a horizontal retreat from the hanging wall to the footwall. Transverse stoping is used because the quality of the rock limits the length of the open span but allows good flexibility with regards to sequencing and scheduling. This method also allows for a high production rate. The main disadvantage of this method is the increased amount of development requirements in comparison to a longitudinal approach. Paste backfill will be used in the primary and secondary stopes. To avoid hauling of waste rock to surface, secondary stopes will be filled with development waste rock and paste backfill in a controlled manner. Figure 16.12 illustrates a typical transverse long hole stoping configuration.

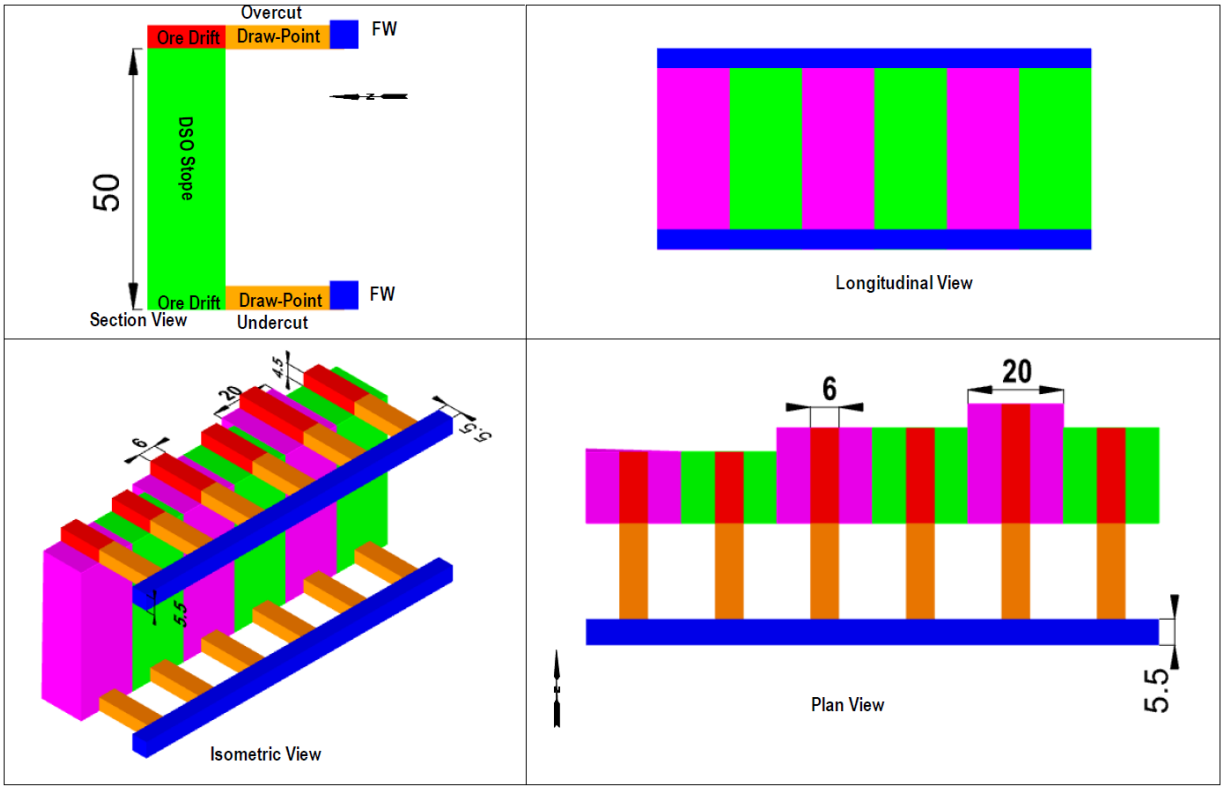
Figure 16.12: Illustration of Transverse Long Hole Stoping



The mining stopes will be accessed by a decline at every 50 m vertical interval to the level access (5.5m h x 5.5m W) then connected to the footwall drift (5.5m H x 5.5m W) which then splits into East and West to cover the whole orebody. Multiple draw-point drifts (4.5m H x 6m W) will typically be excavated at every 20 m apart from center line to center line and as well as the ore drifts (4.5m H x 6m W).The stope dimensions are designed at 50m H x 20m L x 20m W . Stopes at the bottom of the mine are reduced to 30 m in height.

Once all the necessary developments are completed, the mining stopes will be drilled with a long hole drill. The stope is blasted and mucked out by using a Load Haul Dump scoop tram (“LHD”) then loaded into trucks and hauled to the surface. Backfill will start by using paste backfill in all stopes except when development waste is available for secondary stopes. Figure 16.13 illustrates the transverse mining method configuration.

Figure 16.13 :Transverse Long Hole Stoping Method



16.3.2 Geotechnical Considerations

No geotechnical investigation was carried out for this PEA. Additional geotechnical investigation is recommended for the next phase of project development. In the period preceding 1970, an exploration shaft was sunk into the ore to assess the potential of the deposit. During the excavation of this shaft no problems were encountered. The rock type present in the mine is known to be good quality rock. For this Study a crown pillar of 50 m is considered. Probability is high that this crown pillar could be reduced with additional investigations and geotechnical studies.

16.3.3 Hydrological Considerations

For this Study, the assumption was made that water seepage into the mine was assessed at 2,800 m³/d during the peak period. However, it is highly recommended to do further studies for the next phases of project development. The river must be considered in future hydrological studies.

16.3.4 Underground Cut-Off Grade Estimation

The Project cut-off grade was evaluated to be able to design the minable stope and define which portion of the geological resource can be included in a mining plan. The stope break-even cut-off grade is defined as the grade which will make the material value cover the operating costs.

Table 16.11, Table 16.13 and Table 16.14 summarize the different assumptions used for calculating the cut-off grade. The cut-off grade was estimated at 0.3% Nb_2O_5. It was decided to elevate the COG above the break-even to 0.3% Nb_2O_5 to increase the operating margin albeit slightly decreasing the mine life.

Table 16.12: Stope Assumptions for the COG Estimate

Stope Dimensions	Niobay	Units
Block Height	50	M
Width	20	M
Strike Length	20	M
Density	2.92	T/M^3
Tonnage Block	58,400	T
Estimated Dilution	2.50	%
Estimated Dilution (Paste Backfill)	6.00	%

Table 16.13: Cost Assumptions for COG Estimate

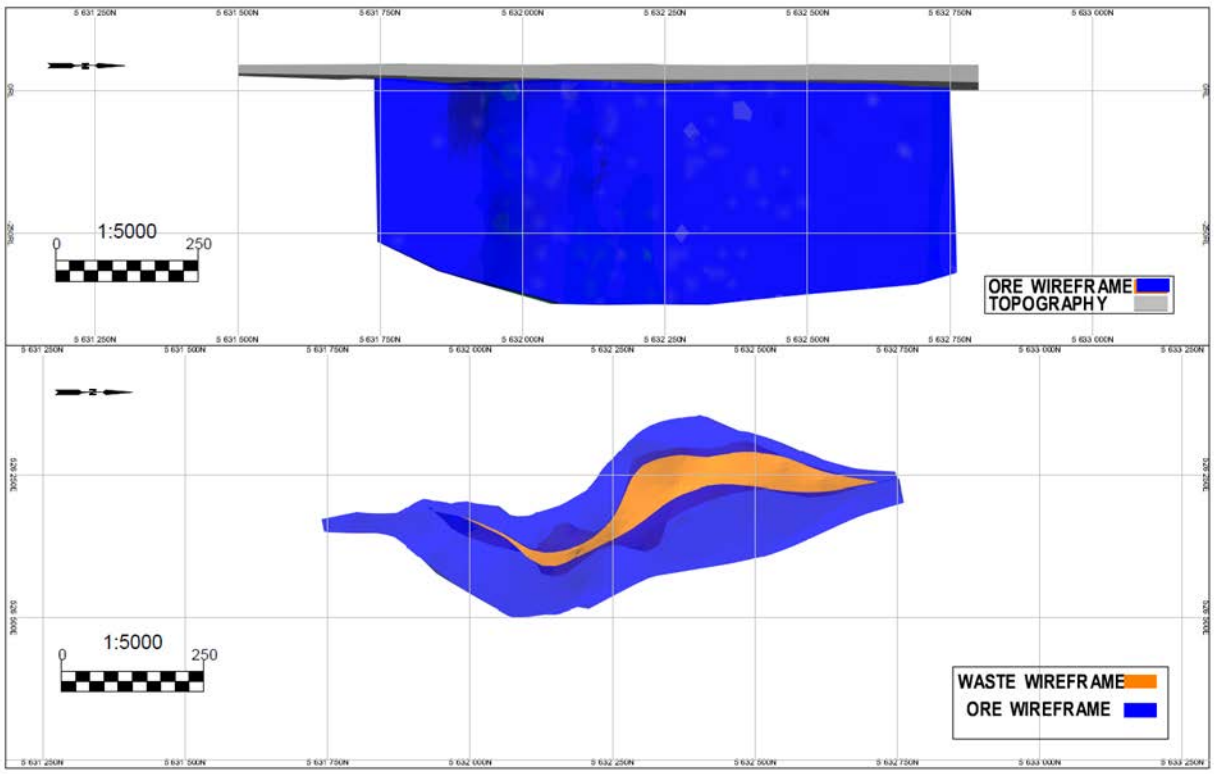
Cost Center	Niobay	Units
Diamond Drilling	0.20	CAD/t
Stope Preparation	2.75	CAD/t
Mining	16.00	CAD/t
Mine Services	5.00	CAD/t
Surface Surfaces	2.00	CAD/t
Mechanical Services	2.15	CAD/t
Electrical Services	2.50	CAD/t
Technical Services	2.50	CAD/t
G&A	10.00	CAD/t
Processing & Environment	15.30	CAD/t
Converter	12.00	CAD/t
Total Costs	**70.40**	**CAD/t**

Table 16.14: Revenue Parameters for COG Estimate

Revenue	Niobay	Units
Niobium Price	45.00	USD/Kg
Exchange Rate	1.30	CAD / USD
Metallurgical Recovery	70	%
Recovery Converter	95	%
Marketing	1.5	%
Value per Kilogram	0.88	CAD/Kg

16.3.5 Underground Mine Design

Figure 16.14: Mineralization of Deposit



The software Deswik Stope Optimizer. (''DSO'') was used to determine the mineable portion of mineral resources. The DSO is a strategic mine planning tool that automates the design of stope shapes for a range of stoping methods for underground mines. The goal is to maximize the value of an orebody within existing constraints using a specified mining method and design parameters. The outputs (stope wireframes, section strings, and reports) are suitable for use in strategic and tactical planning. To generate stope shapes and optimize the value of the orebody we used the datamine format block model Niobay2020 10x5x10.dm, received on July 3rd, 2020 from Niobay personnel modelized by RPA containing the following: Central coordinates of the blocks, Lithology, Mineralization, Density, Grade (%Nb_2O_5), Resources Class, Resources located in the crown pillar, Overburden and Sediments. The Mineralization of the zone is shown Figure 16.14.

Multiple DSO scenarios were run to obtain the best results in term of tonnage and grade. The optimized stopes were built from DSO CAD 2019.4 with Stope Optimized V3 plugin. Below are the final parameters used for the optimization scenario:

- Maximum height, length and width of stope blocks are 50 m x 20 m x 20 m.

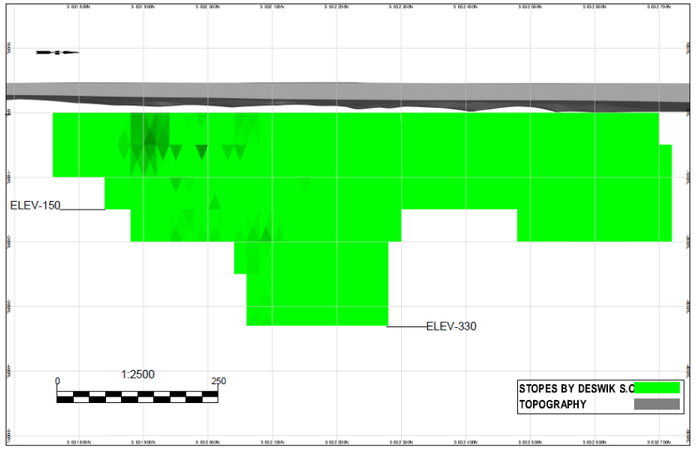
- Waste pillar width to determine the minimum distance between parallel stopes is 5 m.
- Dip minimum and maximum were selected to have blasted rock flowing to the undercut; they are respectively 50 and 130.
- Dilution from the hanging wall and footwall over break was not set; most of stopes are adjacent, the dilution comes from stopes around. Thus, an overall average dilution of 2.5% from the hanging wall and footwall was applied
- Cut-off grades were selected based on the operating costs. COG is 0.3% Nb_2O_5 (refer to Section 16.3.4).

Deswik DSO software generated stopes shape that met the parameters shown in the previous list, however not every stope will be kept for the life of mine plan. Some criteria were used to eliminate the undesired stopes in crown pillar region, isolated, un-mineable stopes or lower than the production cut off grade. The following figures below illustrate the results from the DSO software to the selection of stopes used in the life of mine plan.

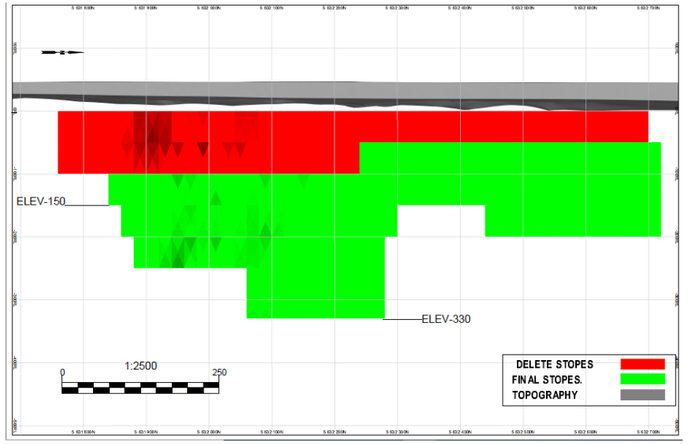
The production plan includes Inferred Mineral Resource for this PEA. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration. The stopes generated by DSO are the same for Scenarios 2 and 3. It is important to note that the lowest level of the mine is 30 m height to allow a better recovery of mineral resources.

Figure 16.15: DSO Generated Stopes Scenario 2 and Scenario 3

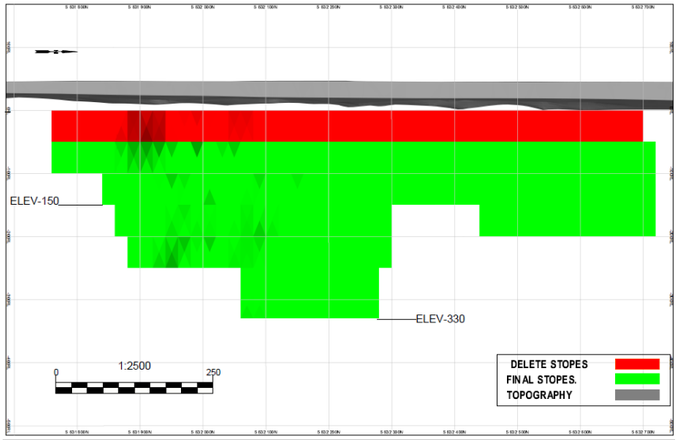
DSO Generated Stopes Scenarios 2 and 3



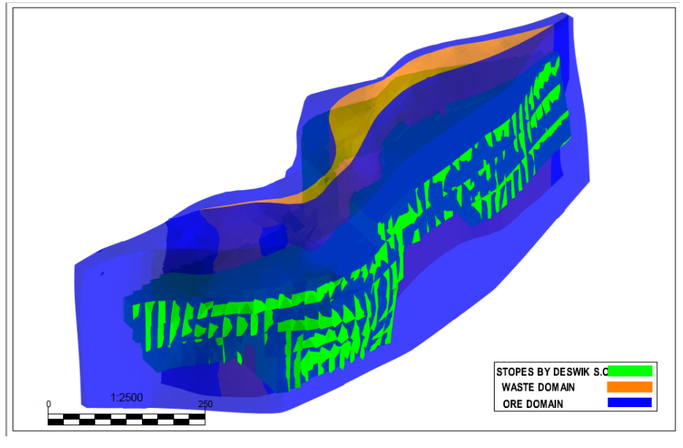
Sc 2 - Stopes, in Red Removed Stope
(in Crown Pillar, in Open Pit or Unmineable)



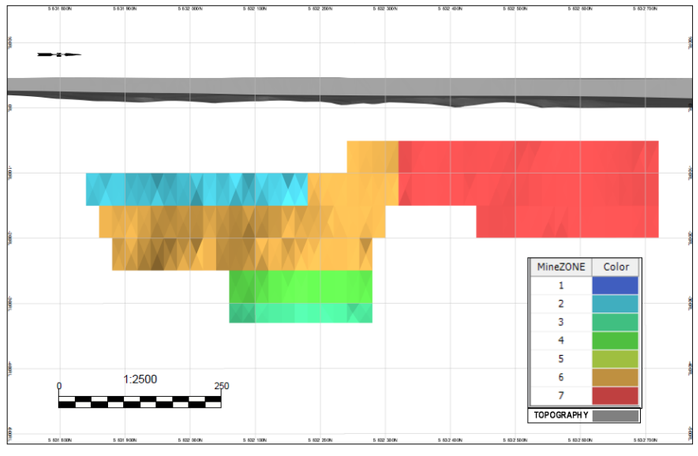
Sc 3 - Stopes, in Red Removed Stope
(in Crown Pillar, in Open Pit or Unmineable)



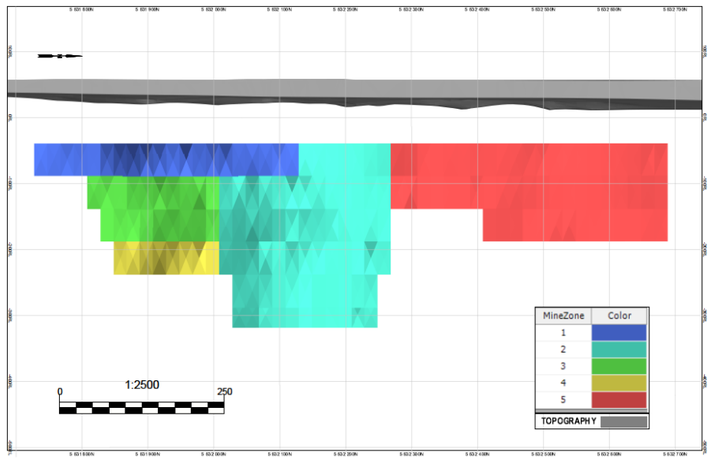
James Bay Niobium Mineralization vs. DSO Blocks



Sc 2 - Stopes Retained (After Clean-Up)



Sc 3 - Stopes Retained (After Clean-Up)



16.3.6 Mining and Paste Dilution and Mining Recovery

Mining dilution was not applied through the DSO process. Therefore, dilution expected from the hanging wall, footwall and backfill was added to the undiluted DSO estimates. An overall average dilution of 2.5% from the hanging wall and footwall was applied and a 6% dilution from the paste-fill. In addition, a 95% mine recovery factor was applied.

Block model contains about 93.2 Mt at 0.44% Nb_2O_5 for a cut-off grade at 0.3% Nb_2O_5. The DSO process and development design generated about 43.92 Mt at 0.50% Nb_2O_5 for Scenario 2. The DSO process and development design generated about 53.62 Mt at 0.51% Nb_2O_5 for Scenario 3.

Table 16.15: Potential Stope by Resource Classification Scenario 2

Class	Tonnage (Mt)	Grade Nb_2O_5	Contained Nb (Mt)
Indicated	30.8	0.52	11.2
Inferred	13.097	0.50	7.3
Total	43.92	0.51	18.5

Table 16.16: Potential Stope by Resource Classification Scenario 3

Class	Tonnage (Mt)	Grade Nb_2O_5	Contained Nb (Mt)
Indicated	**20.86**	**0.51**	**7.5**
Inferred	**32.76**	**0.51**	**11.7**
Total	**53.62**	**0.51**	**19.2**

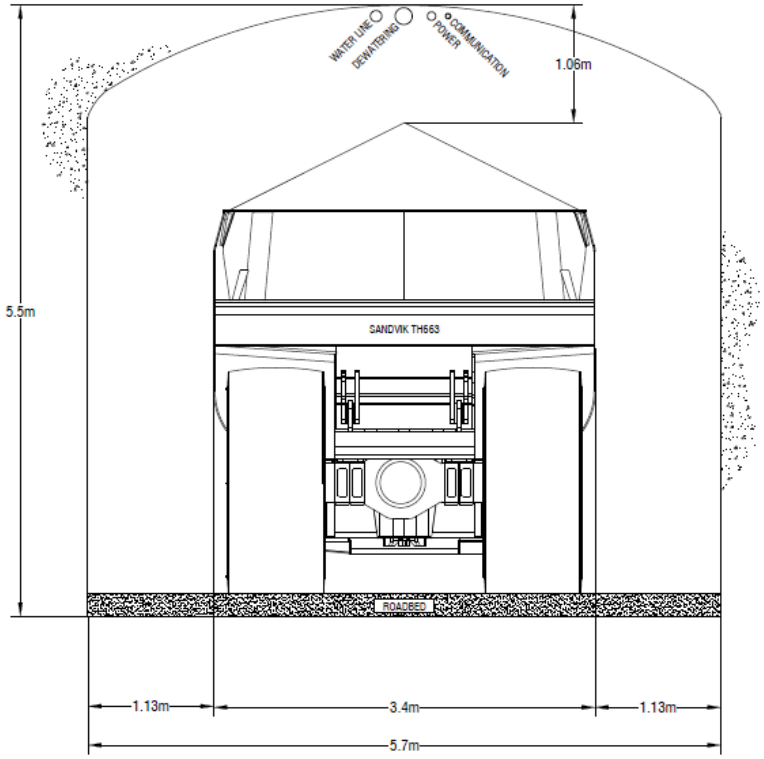
16.3.7 Mine Access and Development

The ore zones will be accessed via a portal located at the east side of the mineralized zone to avoid the river on the west side. The portal will open onto an underground decline and the ore will be transported by trucks from the underground to the surface via the same decline. The possibility of sinking a mine shaft to access the deposit was eliminated at the beginning of the Study. The high capital cost of a mine shaft and the proximity of the ore body to surface meant that a ramp access would be most cost effective. In case, mineral resources are deeper, a trade-off should be done to justify a production shaft. As a lot of overburden is present on the surface, an excavation with sheet and pile will be used to solidify the overburden during the portal excavation .The decline dimensions are 5.7 m W x 5.5 m H to allow use of high capacity truck like 60 t-truck.

16.3.8 Horizontal Development

A single decline will be excavated from surface down to approximately 93 m in elevation from surface where it will enter the first mining level. The decline will provide access to the mining levels every 50 m vertical and 30 m for last level (from -300 to -330 in elevation). Re-muck bays and safety bays will be excavated in the declines and will be used throughout the life of the mine. The re-muck bays are designed to be used as a mucking bay when developing the decline. The priority will be to access the fresh air ventilation raise as soon as possible. The principal objective is to establish a first ventilation circuit and thus reduce the need for ventilation ducts. The second objective of the decline is to reach the bottom of the first mining zone so that it may be brought into production as quickly as possible. Figure 16.16 illustrates the proposed decline dimensions.

Figure 16.16: Decline Development Profile



Each production level will be connected to the main decline via a level access drift. A footwall drift is located 20 m from the deposit on all levels. From the footwall drift, draw-points will be excavated perpendicularly to reach the ore zone. The services such as refuge stations, lunchrooms, paste backfill stations, electric sub-stations, etc. will be located directly in the footwall drifts. The level sumps, however, will be in the level access.

The strike length of the stopes is constant at 20 m, so draw-points are located at 20 m intervals, in the center of the stope block. Drifting across the mineralization will enable to position precisely the ore contacts for better stope delineation and to collect all relevant information related to rock mechanics, ground support and grades.

The dimensions of the excavations have been optimized to accommodate equipment and mine service installations such as ventilation ducts, pipes, electrical cables and the communication network, paste backfill pipe, etc. Figure 16.17 shows a section view of the decline. The slope of the decline is of -15% and it flattens on arrival at the level to facilitate equipment maneuvering. For design simplification reasons, it was decided not to apply any slopes to the other drifts such as footwall drifts.

The horizontal development drilling will be performed with a two-boom hydraulic jumbo drills and the ground support with scissor lift bolters. The decline will be excavated at an average rate of 5.1 m/day (single face) or 151 m/month. Once several working faces are available (outside decline excavations), the average advance rate can reach 9 m/day or 270 m/month. These rates are based on performances obtained in different mines in the Abitibi and northern Ontario. The development mucking during the production period will be done with 20-tonne capacity LHD's and the hauling with 60-tonne trucks whereas during the pre-production period different mucking and hauling equipment could be used by the contractor. The estimated cycle times, capital and operating expenditures are based on the Sandvik TH663i model.

The start of development is planned to be done with a contractor. Normally, it becomes more practical that the entire decline, level access and a certain part of footwall drifts be excavated by a contractor. Gradually, the owner's development crew will take over all the remaining development work except for specialized excavations such as raise-boring work. The breakpoint to switch to full owner development crew will be at first (1st) year of production. However, the decline costs used reflect a higher cost assuming contractor work. For the start of the pre-production period, only one (1) owner development team will be required.

The dimensions, intervals and slope for the main ramp, level accesses, footwall drifts, draw-points, and all parts of lateral and vertical development, are summarized in Table 16.17.

Table 16.17: Excavation Dimension

Excavation	H (m)	W (m)	L (m)
Ramp	5.50	5.70	N/a
Level Access	5.50	5.50	50
Haulage Drift	5.50	5.50	N/a
Draw Point	4.50	6.00	20
Ore Drift	4.50	6.00	N/a
Remuck Bay	5.00	6.00	50
Electrical Station	4.00	6.00	10
Safety Bay	2.00	2.00	
Access Ventilation Raise	5.00	5.00	20
Sump	4.00	6.00	10
Backfill Station	5.00	5.00	5
Pump Station	5.00	5.00	20
Refuge& Lunchroom	5.00	5.00	15
Cap Magazine	4.00	8.00	8
Powder Magazine	4.00	6.00	23
Storage	5.00	5.00	15
Escape Way	2 meters diameter		
Fresh Air Raise	5 meters diameter		
Exhaust Raise	5 meters diameter		

16.3.9 Vertical Development

Only one (1) fresh air ventilation raise, one (1) escape way and four (4) exhaust ventilation raises for both scenarios will be needed during the life of mine and it is planned that the ramp will be used as well as exhaust air. It is planned to excavate and establish a concrete collar in the overburden extending to the bedrock / overburden interface for the raise. Once it is established, the excavation will be performed through raise-boring by a specialized contractor. The raises vary in size from 2 m in diameter for the escapeway raise to 5 m in diameter for the fresh air raise. Considering the good ground condition, fresh air raise, escapeway and exhaust raise between the levels will be excavated using the drop raise method in short intervals as soon as the access to a given level is available.

This Study does not to install ground support in these raises. A prefabricated modular manway system will be installed in the emergency exits. This system is proven to save time and cost to establish the manway.

Figure 16.17 and Figure 16.18 show the main development configuration for Scenarios 2 and 3 respectively.

Figure 16.17: Main Development Scenario 2 - James Bay Niobium

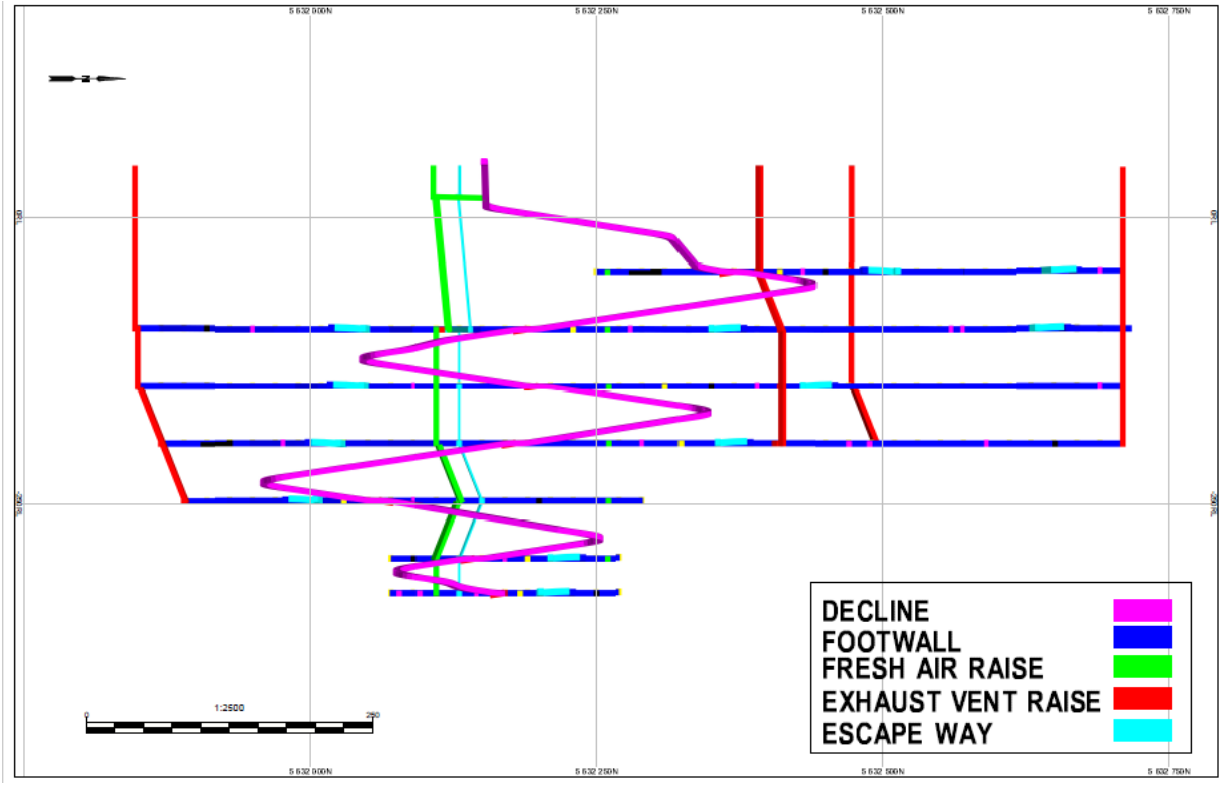
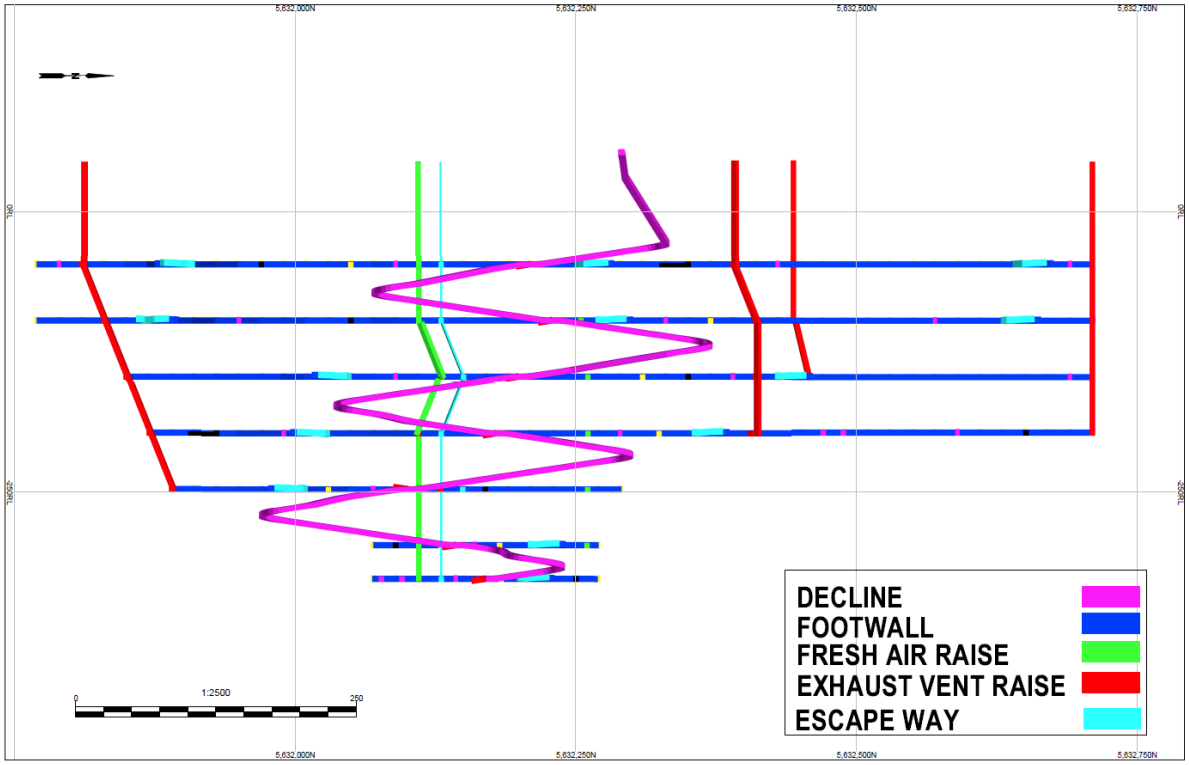


Figure 16.18: Main Development Scenario 3 -- James Bay Niobium



16.3.10 Development Sequence

The development schedule for Scenarios 2 and 3 are practically similar, with the only difference being the number of sequences. There are seven (7) sequences for Scenario 2 and five (5) sequences for Scenario 3. At the start of the pre-production period, the focus will be in the development decline to reach the first vent access which will allows for the completion of the ventilation raise and escape way as quickly as possible. As the development decline continues, the ventilation raises, and the escape way will be connected from one level to another. Once the decline arrives at level -150, priorities will be given to develop both levels -100 and -150 where the first mining sequence (Mine Horizon 1) will start. Only developments related to Mine Horizon 1 will be excavated. Developments on level -150 will be mined as the Mine Horizon 1 stopes located from the bottom to the top of the orebody. Once the developments required for Mine Horizon 1 are completed, developments related to Mine Horizon 2 will start and then Mine Horizon 3 and so forth until the last mine horizon (or sequence). Table 16.18 presents development requirements by excavation type for Scenario 2 and Scenario 3.

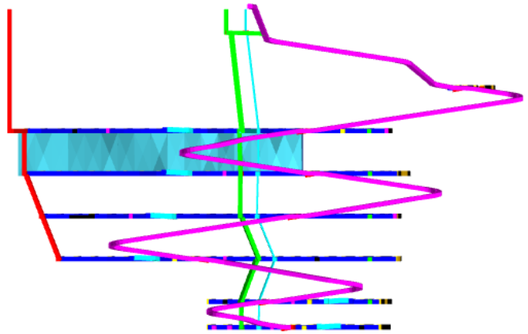
Table 16.18: Meters of Development Per Excavation Type

Excavation Type	Meters of Development (m)	
	Scenario 2	Scenario 3
Ramp	2,936	2,687
Level Access	350	387
Haulage Drift	5,738	6,050
Draw Point	7,699	8,226
Remuck Bay	853	925
Electrical Station	185	196
Vent Raise Acc	1,353	648
Sump	89	89
Backfill Station	44	44
Pump Station	91	91
Refuge	121	121
Powder Mag	51	51
Storage	121	121
Ore Drift	20,808	24,183
Fresh Air Raise	350	391
Exhaust Raise	1,102	756
Escape Way	405	391
Total	**42,295**	**45,356**

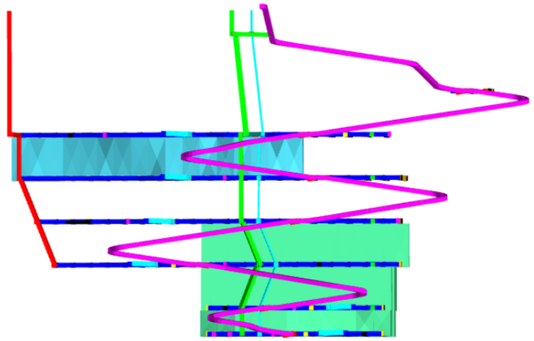
Following figures illustrate the mine sequences for Scenario 2 and Scenario 3.

Figure 16.19: Mining Sequences for Scenario 2

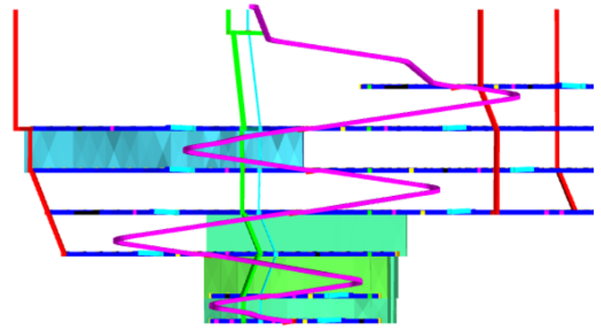
Sequences 1 & 2



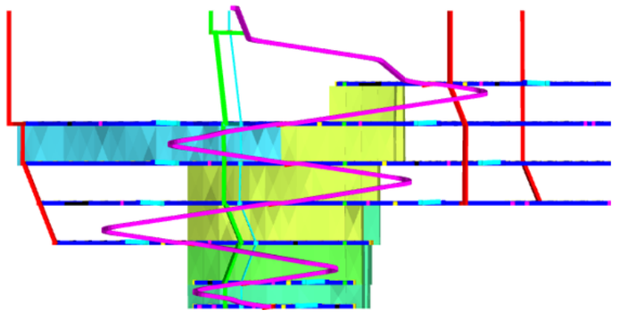
Sequence 3



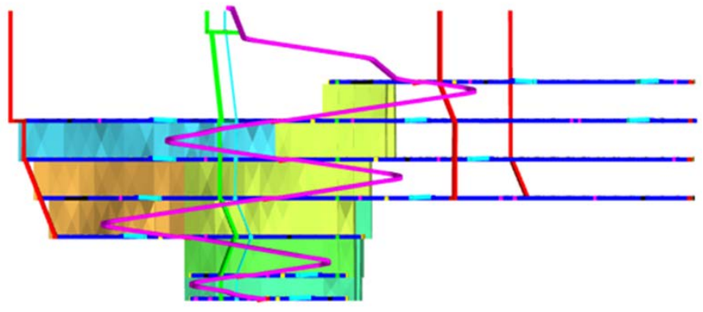
Sequence 4



Sequence 5



Sequence 6



Sequence 7

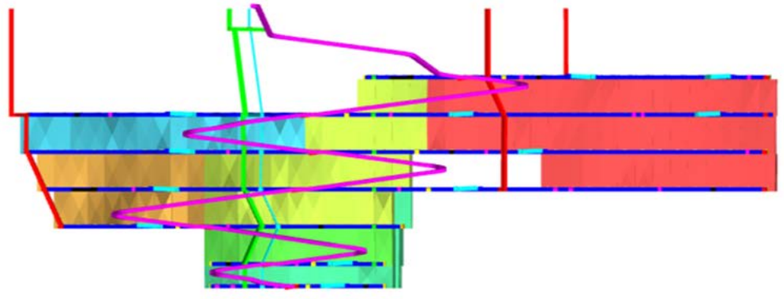
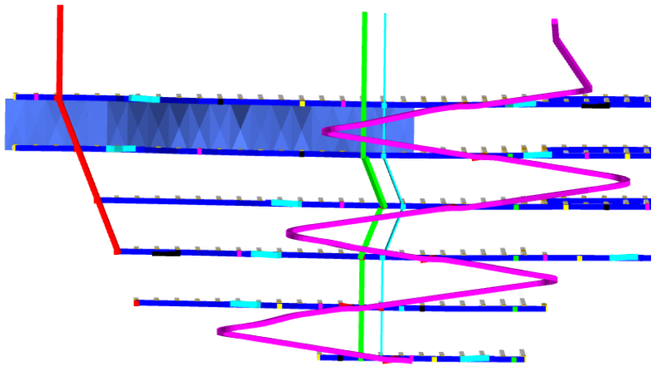
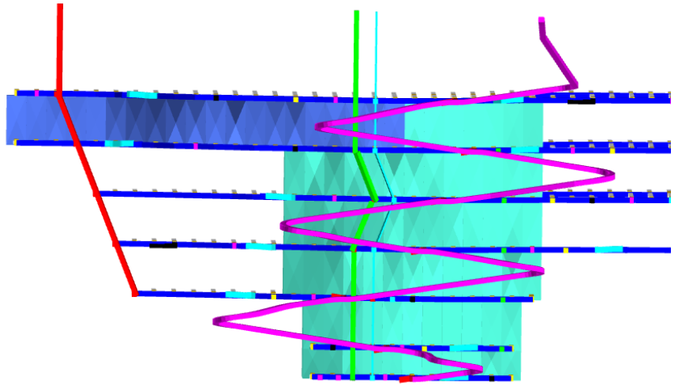


Figure 16.20: Mining Sequences for Scenario 3

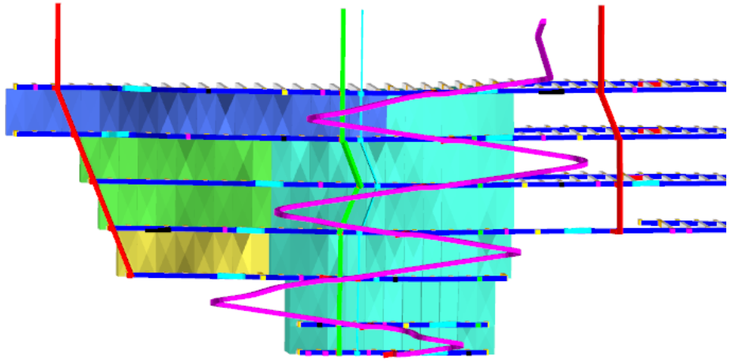
Sequence 1



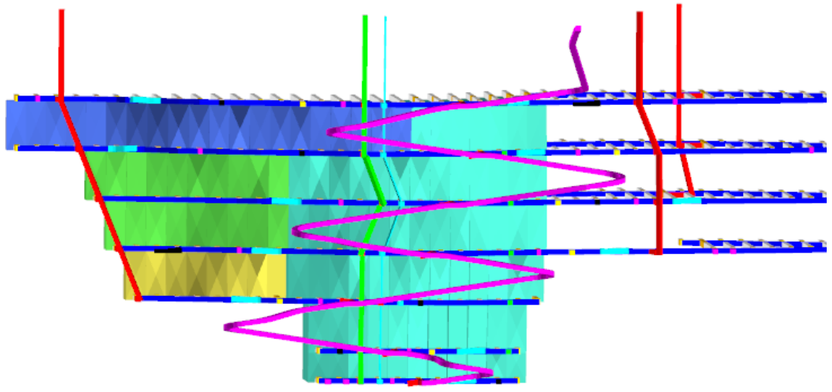
Sequence 2



Sequence 3



Sequence 4



Sequence 5

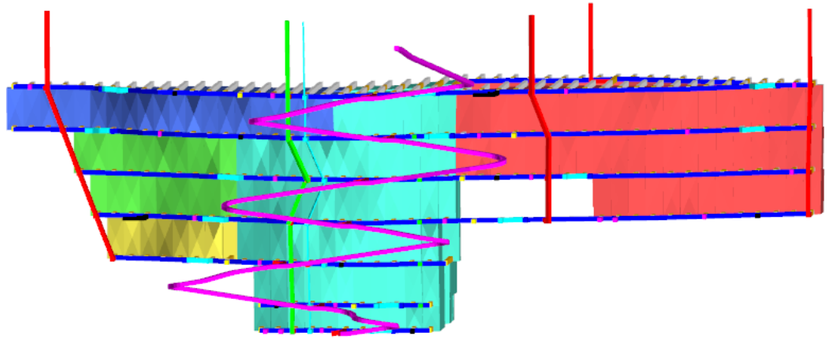
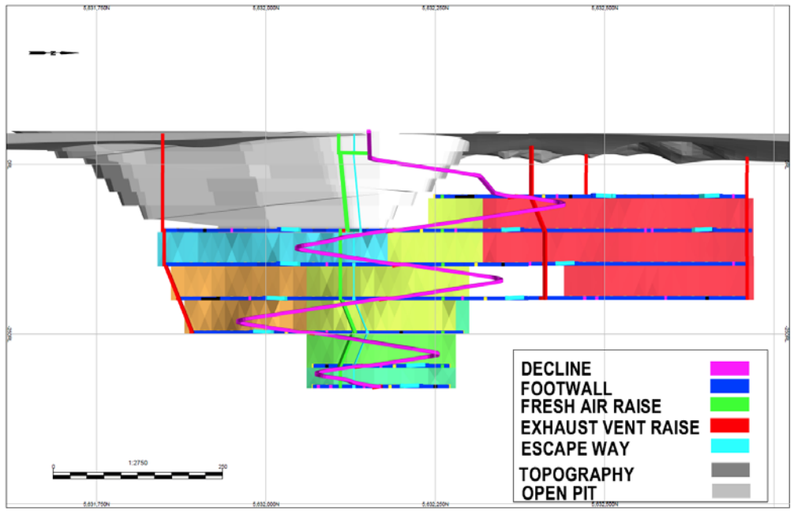
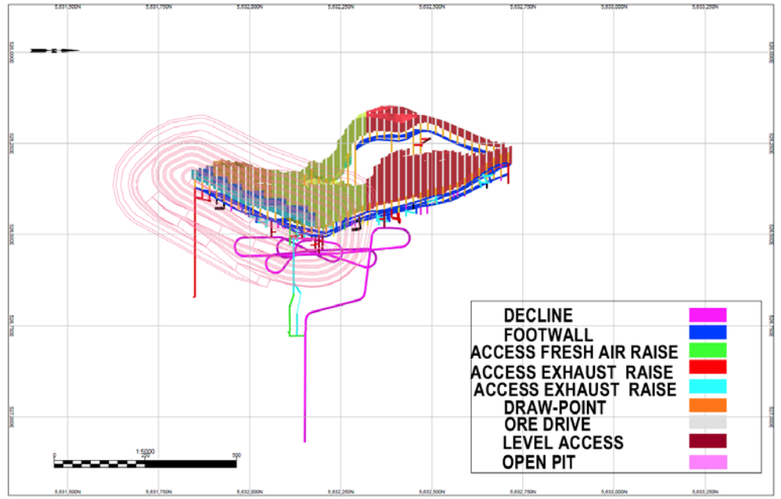


Figure 16.21: Final Design of Underground Mine (Scenarios 2 & 3)

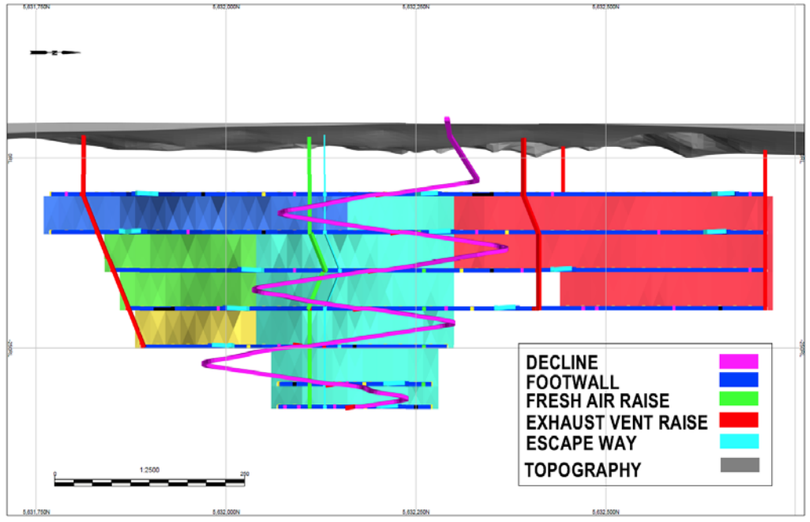
Scenario 2 Longitudinal View of Final Design



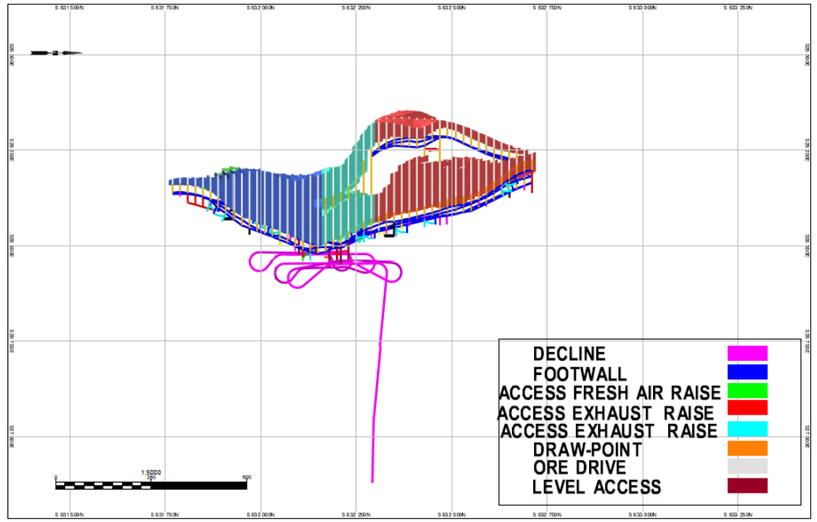
Scenario 2 Plan View of Final Design



Scenario 3 Longitudinal View of Final Design



Scenario 3 Plan View of Final Design



16.4.1 Underground Miscellaneous Infrastructure

16.4.1.1 Lunchrooms and Refuges

A refuge will be excavated on each mining level. The refuges on certain mining levels will also serve as a lunchroom for the workers. The dimensions of the refuge and lunchroom will be of 5m H x 5m W x 15m L. The refuge will include an excavation for an airlock and equipped in accordance with current Ontario rules and regulations. The refuge serving as a lunchroom will be equipped with tables and all the usual amenities found in an underground mine. Seven (7) refuges are planned and two (2) of them will serve of lunchrooms.

16.4.1.2 Powder and Cap Magazines

It is currently expected to have two (2) powder and cap magazines. The magazines will be located at levels -50 and -200. The configuration of magazines is composed of two rooms separated by an 8m rock wall as required by Ontario legislation (R.R.O. 1990, Reg. 854, s. 128 (6).

16.4.1.3 Electrical Substations

Several excavations are required for the electrical sub-stations. The recommended maximum distance between the excavations of the electrical substations is 400 m; beyond this distance the loss of voltage is too important to sustain mining equipment power requirements. The electrical substation dimensions are 4m H x 6m W x 10m L to accommodate all the electrical equipment required.

16.4.1.4 Sump and Pump Stations

A sump will be excavated at every level access to collect water from mine seepage water and production equipment. Seven sumps with a slope of -15% will be excavated for all mine level access to perform a first sedimentation of the particles in the water. Locating the level sumps at approximately the same coordinates will allow all sumps to be interconnected to each other with a borehole in the rock, enabling the transfer of water to the lower levels and ultimately to the pumping station. Pump stations will be installed at lowest and middle levels at elevation -330 m and level -200 m.

16.4.1.5 Paste-fill Station

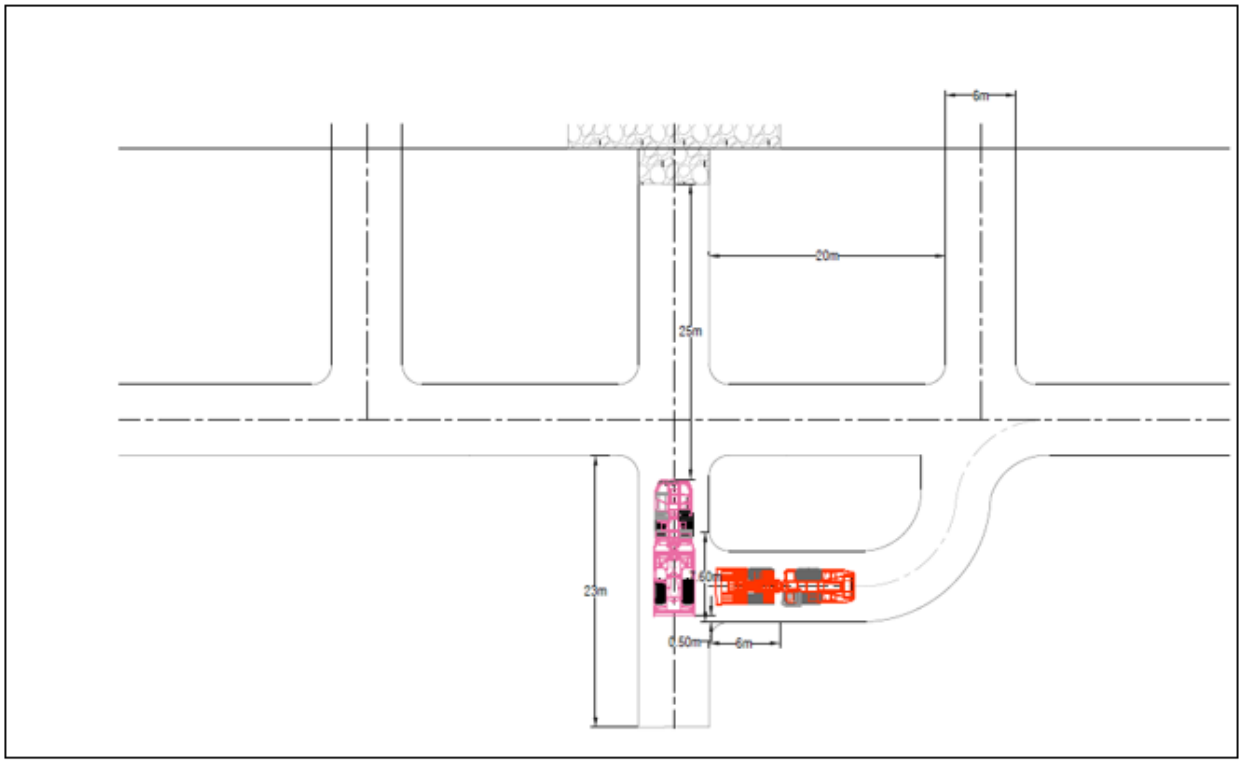
A paste-fill station will be located on each level except for the bottom level. The paste backfill station dimensions are 5m H x 5m W x 5m L. The 5 m height is required to allow the installation of large curve radius piping. A transfer between the main line and the secondary level line will be installed in this paste-

fill station. Currently the design is created so that the backfill line is located near the center of gravity of the deposit and the entrance of the ramp. This configuration will improve the backfill distribution and accelerate the initial construction of the paste backfill line.

16.4.1.6 Loading Points

The loading points are designed to allow for the maximum performance of the LHD when loading trucks. Loading bays are excavated every 400m per level. The loading points are designed with two (2) drifts. One (1) will be used to position the truck for loading and the second one (1) will be excavated perpendicularly and with slope of -3% and 7%. This will allow the LHD to load the trucks. A backslash of 2.2 m height is taken to the drift back to allow the LHD to load the truck thus to facilitate the dumping. The floors of both drifts are at the same elevation. The loading bay will be excavated every 200 m along the ramp. These excavations will be perpendicular to the ramp.

Figure 16.22:Loading Point



16.5 Production Scenarios

The life-of-mine (“LOM”) plan for the James Bay Niobium Project was based on the current potential mineable tonnage which includes Inferred resources. The LOM schedules for the three scenarios studied

were developed to feed the mill at a rate of 6,600 tpd (2,409 kt/y). Mill feed material from the underground is composed of mineralized material from development and from stoping activities.

Table 16.19: LOM Physicals Summary

LOM Physicals Summary		Scenario 1	Scenario 2	Scenario 3
Open Pit				
Total Mined	Mt	198.14	24.32	-
Strip Ratio	W:O	1.80	1.49	-
Waste Mined	Mt	127.30	14.55	-
Mill Feed Tonnage Mined	Mt	70.84	9.76	-
Mill Feed Grade	% Nb_2O_5	0.44	0.51	-
Underground				
Mill Feed Tonnage Mined	Mt	-	43.93	53.62
Mill Feed Grade	% Nb_2O_5	-	0.51	0.51
Total				
Mill Feed Tonnage Mined	Mt	70.84	53.69	53.62
Mill Feed Grade	% Nb_2O_5	0.44	0.51	0.51

16.5.1 Scenario 1: Open Pit Only

The LOM scheduling for the Open Pit Only Scenario was optimized using Minemax software to maximize discounted operating cash flow of the Project.

The mine production schedule is completed on an annual basis throughout the mine life. There is only one year of pre-development where no ore is processed. The process plant is not scheduled at 100% capacity the first year to allow for a continued ramp-up to nameplate. Mining production ramps up to the peak mining rate of 10 Mt/y by Year 5. The processing plant starts at the maximum capacity of 6,000 tpd in Year 2 and maintains that until end of the Project life.

The total mining life of the Project is 24 years. The processing life of the Project is 30 years as there is an approximate 6 years or 13.6 Mt of low-grade material in the stockpile to be processed after the end of open pit mining. High grade ore is typically sent directly to the mill to increase early cash flow and surplus low-grade ore is stockpiled. There is a massive drop off in grade as the low-grade is being milled without any high-grade mixed in. The mining, stockpiling and plant schedules are presented in Table 16.20, Table 16.21 and Table 16.22 respectively.

16.5.2 Scenario 2: Combined Open Pit and Underground

The mine production schedule is completed on a yearly basis throughout the mine life. There is only one (1) year of pre-development in which no mill feed material is processed. Any mill feed material generated from the pit in this period is stockpiled. The open pit mine lasts 5 years including the pre-production year where the peak mining rate of 7.3 Mt is reached in Year 1. The underground mine starts in Year 3 and delivers sufficient ore to continue feeding the plant from Year 5 onwards.

The total mine life of the Project is 22 years. The processing life of the Project is 23 years as there is a small stockpile to process at the end. The mining, stockpiling and plant schedules are presented in Table 16.23 and Table 16.24.

16.5.3 Scenario 3: Underground Only

For Scenario 3 with only an underground mine there is two years of pre-development planned to establish the mine and prepare for a sustainable production rate of 6,600 tpd. This scenario similar to Scenario 2 generates 23 years of mill feed as presented in Table 16.25 and Table 16.26.

Table 16.20: Scenario 1 Mine Production Schedule

Period	Total Mined (kt)	Waste Rock (kt)	Sediments (kt)	Ovb. (kt)	Total Waste (kt)	Strip Ratio (W:O)	Mined Mill Feed (kt)	Mined Grade (% Nb_2O_5)
-1	5,386	11	393	4,982	5,386			
1	7,000	2,020	1,100	1,643	4,763	2.13	2,237	0.50
2	7,000	3,707	0	0	3,707	1.13	3,293	0.51
3	8,987	2,093	572	3,710	6,375	2.44	2,612	0.50
4	9,601	1,181	2,873	2,957	7,012	2.71	2,590	0.47
5	10,000	3,981	940	1,810	6,731	2.06	3,269	0.38
6	10,000	2,921	1,763	2,163	6,847	2.17	3,153	0.40
7	10,000	2,331	2,606	1,999	6,936	2.26	3,064	0.41
8	10,000	2,384	3,355	1,241	6,980	2.31	3,020	0.43
9	10,000	4,871	1,431	602	6,904	2.23	3,096	0.43
10	10,000	6,441	230	87	6,758	2.08	3,242	0.42
11	10,000	6,754			6,754	2.08	3,246	0.43
12	10,000	6,780			6,780	2.11	3,220	0.43
13	10,000	6,907			6,907	2.23	3,093	0.44
14	10,000	6,923			6,923	2.25	3,077	0.44
15	10,000	7,424			7,424	2.88	2,576	0.36
16	10,000	6,513			6,513	1.87	3,487	0.38
17	7,748	4,351			4,351	1.28	3,397	0.39
18	6,660	3,334			3,334	1.00	3,326	0.40
19	5,973	2,833			2,833	0.90	3,140	0.41
20	5,501	2,621			2,621	0.91	2,880	0.44
21	4,777	2,067			2,067	0.76	2,710	0.48
22	3,779	1,370			1,370	0.57	2,409	0.52
23	3,107	698			698	0.29	2,409	0.53
24	2,623	326			326	0.14	2,297	0.56
25								
26								
27								
28								
29								
30								
Total	**198,143**	**90,840**	**15,264**	**21,194**	**127,299**	**1.80**	**70,845**	**0.44**

Table 16.21: Scenario 1 Stockpile Schedule

Period	Additions (kt)	Withdrawal (kt)	Low Grade SP Inv. (kt)	High Grade SP Inv. (kt)	Total Stockpile Inv. (kt)
-1					
1	310		250	60	310
2	884		620	574	1,194
3	203		823	574	1,397
4	242	61	1,065	513	1,578
5	860		1,925	513	2,438
6	744		2,669	513	3,182
7	655		3,324	513	3,837
8	611		3,922	526	4,448
9	687		4,515	620	5,135
10	833		5,225	744	5,968
11	837		5,991	814	6,805
12	811		6,709	908	7,616
13	684		7,393	908	8,301
14	730	62	8,123	846	8,968
15	889	722	9,012	124	9,135
16	1,108	30	10,119	94	10,213
17	988		11,107	94	11,201
18	917		12,025	94	12,119
19	731		12,756	94	12,850
20	471		13,227	94	13,321
21	301		13,528	94	13,622
22			13,528	94	13,622
23			13,528	94	13,622
24		112	13,510		13,510
25		2,409	11,101		11,101
26		2,409	8,692		8,692
27		2,409	6,283		6,283
28		2,409	3,874		3,874
29		2,409	1,465		1,465
30		1,465	0		0
Total	**14,497**	**14,497**			

Table 16.22: Scenario 1 Plant Schedule

Period	Mill Feed (kt)	Head Grade (% Nb_2O_5)	Contained Metal (000 kg Nb_2O_5)	Recovered Niobium (000 kg Nb)
-1				
1	1,927	0.55	10,504	5,527
2	2,409	0.58	14,062	7,398
3	2,409	0.53	12,758	6,713
4	2,409	0.50	11,964	6,295
5	2,409	0.45	10,859	5,714
6	2,409	0.46	11,191	5,888
7	2,409	0.48	11,517	6,060
8	2,409	0.49	11,889	6,255
9	2,409	0.50	12,008	6,318
10	2,409	0.50	11,958	6,292
11	2,409	0.50	12,134	6,384
12	2,409	0.51	12,302	6,473
13	2,409	0.52	12,451	6,551
14	2,409	0.51	12,382	6,515
15	2,409	0.43	10,456	5,501
16	2,409	0.47	11,286	5,938
17	2,409	0.48	11,591	6,099
18	2,409	0.48	11,673	6,142
19	2,409	0.48	11,658	6,134
20	2,409	0.49	11,917	6,270
21	2,409	0.52	12,447	6,549
22	2,409	0.52	12,409	6,529
23	2,409	0.53	12,834	6,753
24	2,409	0.56	13,374	7,037
25	2,409	0.18	4,327	2,277
26	2,409	0.18	4,327	2,277
27	2,409	0.18	4,327	2,277
28	2,409	0.18	4,327	2,277
29	2,409	0.18	4,327	2,277
30	1,465	0.18	2,632	1,385
Total	**70,845**	**0.44**	**311,895**	**164,100**

Table 16.23: Scenario 2 Mine Production Schedule

Period	Open Pit					Underground			
	Total Mined	Total Waste	Strip Ratio	Mined Mill Feed	OP Mined Grade	Devel.	Stoping	Total UG	UG Mined Grade
	(kt)	(kt)	(W:O)	(kt)	(% Nb_2O_5)	(kt)	(kt)	(kt)	(%Nb_2O_5)
-2									
-1	4,999	4,999							
1	7,290	5,119	2.36	2,171	0.50				
2	5,408	2,699	1.00	2,709	0.52				
3	3,449	823	0.31	2,626	0.53	2	0	2	0.50
4	3,172	913	0.40	2,258	0.47	245	361	606	0.56
5						318	2,091	2,409	0.54
6						105	2,304	2,409	0.58
7						105	2,304	2,409	0.58
8						105	2,304	2,409	0.52
9						82	2,327	2,409	0.54
10						82	2,327	2,409	0.54
11						82	2,327	2,409	0.54
12						82	2,327	2,409	0.54
13						82	2,327	2,409	0.54
14						82	2,327	2,409	0.53
15						82	2,327	2,409	0.51
16						65	2,344	2,409	0.50
17						36	2,373	2,409	0.44
18						36	2,373	2,409	0.44
19						36	2,373	2,409	0.44
20							2,409	2,409	0.44
21							2,409	2,409	0.44
22							2,368	2,368	0.44
23									
Total	**24,317**	**14,553**	**1.49**	**9,764**	**0.51**	**1,627**	**42,302**	**43,929**	**0.51**

Table 16.24: Scenario 2 Stockpile Schedule

Period	Stockpile			Process Plant			
	IN	OUT	Total Stockpile Inv.	Mill Feed	Head Grade	Contained metal	Recovered Niobium
	(kt)	(kt)	(kt)	(kt)	(% Nb_2O_5)	(000 kg Nb_2O_5)	(000 kg Nb)
-2							
-1							
1	244		244	1,927	0.54	10,360	5,451
2	300		544	2,409	0.56	13,395	7,048
3	219		763	2,409	0.56	13,408	7,054
4	455		1,218	2,409	0.52	12,414	6,532
5			1,218	2,409	0.54	12,916	6,796
6			1,218	2,409	0.58	13,876	7,301
7			1,218	2,409	0.58	13,876	7,301
8			1,218	2,409	0.52	12,547	6,602
9			1,218	2,409	0.54	12,898	6,786
10			1,218	2,409	0.54	13,028	6,855
11			1,218	2,409	0.54	13,028	6,855
12			1,218	2,409	0.54	13,028	6,855
13			1,218	2,409	0.54	13,028	6,855
14			1,218	2,409	0.53	12,862	6,767
15			1,218	2,409	0.51	12,242	6,441
16			1,218	2,409	0.50	12,040	6,335
17			1,219	2,409	0.45	10,848	5,707
18			1,219	2,409	0.44	10,579	5,566
19			1,219	2,409	0.44	10,579	5,566
20			1,219	2,409	0.44	10,557	5,555
21			1,219	2,409	0.44	10,557	5,555
22		41	1,177	2,409	0.44	10,558	5,555
23		1,177	0	1,177	0.25	2,975	1,565
Total	**1,218**	**1,219**		**53,693**	**0.51**	**271,602**	**142,900**

Table 16.25: Scenario 3 Mining Schedule

Period	Underground			
	Devel. (kt)	Stoping (kt)	Total UG (kt)	UG Mined Grade (% Nb_2O_5)
-2				
-1	228	405	633	0.53
1	282	2,127	2,409	0.54
2	93	2,316	2,409	0.54
3	86	2,323	2,409	0.54
4	77	2,332	2,409	0.55
5	61	2,348	2,409	0.55
6	78	2,331	2,409	0.55
7	79	2,330	2,409	0.55
8	79	2,330	2,409	0.55
9	68	2,341	2,409	0.55
10	68	2,341	2,409	0.55
11	68	2,341	2,409	0.55
12	68	2,341	2,409	0.55
13	68	2,341	2,409	0.55
14	68	2,341	2,409	0.55
15	68	2,341	2,409	0.51
16	68	2,341	2,409	0.43
17	68	2,341	2,409	0.45
18	54	2,355	2,409	0.45
19	54	2,355	2,409	0.45
20	54	2,355	2,409	0.45
21	54	2,355	2,409	0.45
22		2,403	2,403	0.45
Total	**1,893**	**51,731**	**53,625**	**0.51**

Table 16.26: Scenario 3 Stockpile & Plant Schedule

eriod	Stockpile			Process Plant			
	IN	OUT	Total Stockpile Inv.	Mill Feed	Head Grade	Contained Metal	Recovered Niobium
	(kt)	(kt)	(kt)	(kt)	(% Nb_2O_5)	(000 kg Nb_2O_5)	(000 kg Nb)
-2							
-1	633		633				
1	482		1,115	1,927	0.54	10,360	5,451
2			1,115	2,409	0.54	13,031	6,856
3			1,115	2,409	0.54	13,113	6,899
4			1,115	2,409	0.55	13,161	6,924
5			1,115	2,409	0.55	13,169	6,929
6			1,115	2,409	0.55	13,161	6,924
7			1,115	2,409	0.55	13,160	6,924
8			1,115	2,409	0.55	13,160	6,924
9			1,115	2,409	0.55	13,165	6,927
10			1,115	2,409	0.55	13,165	6,927
11			1,115	2,409	0.55	13,165	6,927
12			1,115	2,409	0.55	13,165	6,927
13			1,115	2,409	0.55	13,193	6,941
14			1,115	2,409	0.55	13,353	7,025
15			1,115	2,409	0.51	12,327	6,486
16			1,115	2,409	0.43	10,399	5,471
17			1,115	2,409	0.45	10,787	5,675
18			1,115	2,409	0.45	10,779	5,671
19			1,115	2,409	0.45	10,779	5,671
20			1,115	2,409	0.45	10,779	5,671
21			1,115	2,409	0.45	10,779	5,671
22			1,115	2,403	0.45	10,722	5,641
23		1,115	0	1,115	0.52	5,784	3,043
Total	**1,115**	**1,115**		**53,625**	**0.51**	**274,658**	**144,508**

16.6 Mine Operations and Equipment Selection

16.6.1 Open Pit Mining

Mining is to be carried out using conventional open pit techniques with hydraulic shovels, wheel loaders and mining trucks in a bulk mining approach with 10 m benches. An owner mining open pit operation is

planned with the outsourcing of certain support activities such as explosives manufacturing and blasting activities.

The open pit mining operations include production drilling and blasting, grade control, loading, hauling, and various support operations as described in the following sections.

16.6.1.1 Production Drilling and Blasting

Drill and blast specifications are established to effectively single pass drill and blast a 10 m bench. For this bench height, a 200 mm blast hole size is proposed with a 5.5m x 5.5m pattern with 1.6 m of sub-drill. These drill parameters combined with a high energy bulk emulsion with a density of 1.2 kg/m^3 result in a powder factor of 0.32 kg/t. Blast holes are initiated with NONEL detonators and primed with 450 g boosters. The bulk emulsion product is a gas sensitized pumped emulsion blend specifically designed for use in wet blasting applications.

Estimates on the pure penetration rate for the 200 mm diameter holes is 49.9 m/h. The overall drilling rate is 28.7 m/h. The overall drilling rate includes factors such as: moving the drill, changing drill bits, repositioning within patter, etc. Resulting in a drill productivity is 2,192 t/h.

The blast hole rig selected for production drilling will have a hole size range of 152 mm to 270 mm with a single pass drill depth of 12.2 m with a 40 ft tower configuration. This rig will have both rotary and down-the-hole (“DTH”) drilling capability. It is expected that DTH drilling mode will be most efficient.

For the first two (2) years, the blasting activities will be outsourced to an explosives provider who will be responsible for supplying and delivering explosives in the hole through a shot service contract. Starting in Year 3, the mine will operate an explosive plant facility and have its own blasting crew. The mine engineering department will be responsible for designing blast patterns and relaying hole information to the drills.

Along the design excavation limit of interim and final walls, the holes are loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a crack that forms the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves. As a best practice, it is recommended that operations restrict production blasts to within 50 m of an unblasted pre-shear line. Once the pre-split is shot, production blasts will be taken to within 10 m of the pre-shear and then a trim shot used to clean the face. Pre-split holes spaced 1.5 m apart will be 20 m in length and drilled with a smaller diameter of 127 mm (5 in.). Pre-split drill and blast is planned to maximize stable bench faces and to maximize inter-ramp angles along pit walls.

The drill selected for this application is a more flexible type of rig capable of drilling angled holes for probe drilling and pit wall drain holes. The hole size range of this rig is between 110 mm and 203 mm with a maximum hole depth of 31.5 m.

16.6.1.2 Grade Control

The ore control program will consist of establishing dig limits for the ore and waste in the field to guide loading unit operators.

Grade control will rely primarily on blast hole sampling. The program will encompass 100% of all ore material and capture around 50% of the total waste in the pit (mainly the contact zones with ore).

The ore control boundaries will be established by the technical services department based on grade control information obtained through the blast hole sampling with post-blast boundaries adjusted for blast movement measurements made using a BMM® system. A blast movement monitoring system has been included in the blasting cost.

The samples collected will be sent to a nearby off-site laboratory for sample preparation and assaying for the LOM Samples will be collected on the bench and properly tagged by grade control samplers on each shift.

16.6.1.3 Loading

Most of the loading in the pit will be performed by two 7 m^3 face shovels. The shovels will be matched with a fleet of 70t payload capacity mine trucks. The hydraulic shovels will be complemented by one production front-end wheel loaders (“FEL”) with an 8.2 m^3 bucket.

Although interchangeable, the hydraulic shovels will primarily be operating in ore and overburden while the wheel loaders will primarily be operating in waste.

The two 7 m^3 face shovels are expected to achieve a productivity of 827 t/h based on a 3-pass match with the mine trucks. The productivity in overburden will decrease at 440 t/h due to lower density of material and extra pass and a half required. The 8.2 m^3 FEL, used for better selectivity, will have a 821 t/h production rate in ore and waste rock, whereas in overburden the FEL will reach a productivity of 515 t/h.

Table 16.27: Loading Specifications

Loading Unit		Shovel (7 m³)	Shovel (7 m³)	Shovel (7 m³)	FEL (8.2 m³)	FEL (8.2 m³)	FEL (8.2 m³)
Haulage Unit		**Truck (60 t)**	**Truck (60 t)**	**Truck (60 t)**	**Truck (60 t)**	**Truck (60 t)**	**Truck (60 t)**
Material		**Rock**	**Waste**	**OVB**	**Rock**	**Waste**	**OVB**
Rated Payload	t	64	64	64	64	64	64
Heaped Volume	m³	42	42	42	42	42	42
Bucket Capacity	m³	7.0	7.0	7.0	8.2	8.2	8.2
Bucket Fill Factor	%	85%	85%	88%	85%	85%	88%
In-Situ Dry Density	t/bcm	2.93	2.93	2.00	2.93	2.93	2.00
Moisture	%	3%	3%	3%	3%	3%	3%
Swell	%	40%	40%	40%	40%	40%	40%
Wet Loose Density	t/lcm	2.16	2.16	1.47	2.16	2.16	1.47
Actual Load per Bucket	t	12.83	12.83	9.06	15.02	15.02	10.62
Passes (Decimal)	#	4.99	4.99	7.23	4.02	4.02	6.03
Passes (Whole)	#	5.00	5.00	7.00	4.00	4.00	6.00
Actual Truck Wet Payload	t	64	64	62	64	64	64
Actual Truck Dry Payload	t	62	62	60	62	62	62
Actual Heaped Volume	m³	30	30	42	30	30	43
Payload Capacity		100%	100%	97%	99%	99%	100%
Heaped Capacity		71%	71%	100%	70%	70%	103%
Cycle Time							
Truck Exchange	min	0.60	0.60	0.80	0.70	0.70	0.80
First Bucket Dump	min	0.10	0.10	0.10	0.10	0.10	0.10
Average Cycle Time	min	0.70	0.67	0.90	0.80	0.80	0.90
Load Time	min	3.50	3.37	6.30	3.20	3.20	5.40
Cycle Efficiency	%	75%	75%	75%	75%	75%	75%
Number of Trucks Loaded/h	#	12.86	13.37	7.14	14.06	14.06	8.33
Production / Productivity							
Avg. Prod. Dry Tonnes/h	t/h	801	832	430	869	869	515

16.6.1.4 Hauling

Haulage will be performed with a fleet of 70 t class mine trucks. The truck fleet productivity was simulated in Talpac software. Several haulage profiles were digitized and simulated in Talpac3D. Haulage times were estimated for all levels of every phase in each scenario to both the waste dumps and the processing plant. Fuel consumption was also calculated by bench.

The assumptions and input factors for the Talpac simulations in Table 16.28, Table 16.29 and Table 16.30.

Two speed limits were applied in the simulation. On production benches, a speed limitation of 30 km/h was imposed to reflect the lack of proper road and less favorable rolling conditions in addition to having stopes in the pit floor. For all downhill ramps with an incline greater than 5%, the speed is limited to 30 km/h. Otherwise, there is a site wide 50 km/h limit on all surface traffic for haulage trucks.

Rolling resistances applied are typical values for main haulage road, ramp and pit floor/dump rolling resistances. This reflects the quality of the road and how well the trucks can navigate it. Figure 16.23 depicts a layout for the last year of mining for Scenario 1. The haulage routes are depicted to going to the processing plant and the dump. The routes are coloured in accordance to the rolling resistances set in those segments.

Table 16.28: Speed Limits

Site Location	Speed Limit (km/h)
Pit on Working Bench, Near Dump Face	30
Downhill Ramp < -5%	30
Mine Road and Ramps	50

Table 16.29: Rolling Resistance

Road Type	Rolling Resistance (%)
Main Road	2.5
Ramp	3.0
Pit Floor and Near Dump Face	3.5

Figure 16.23: Talpac3D Layout - Open Pit Only Scenario – End of Mine Life

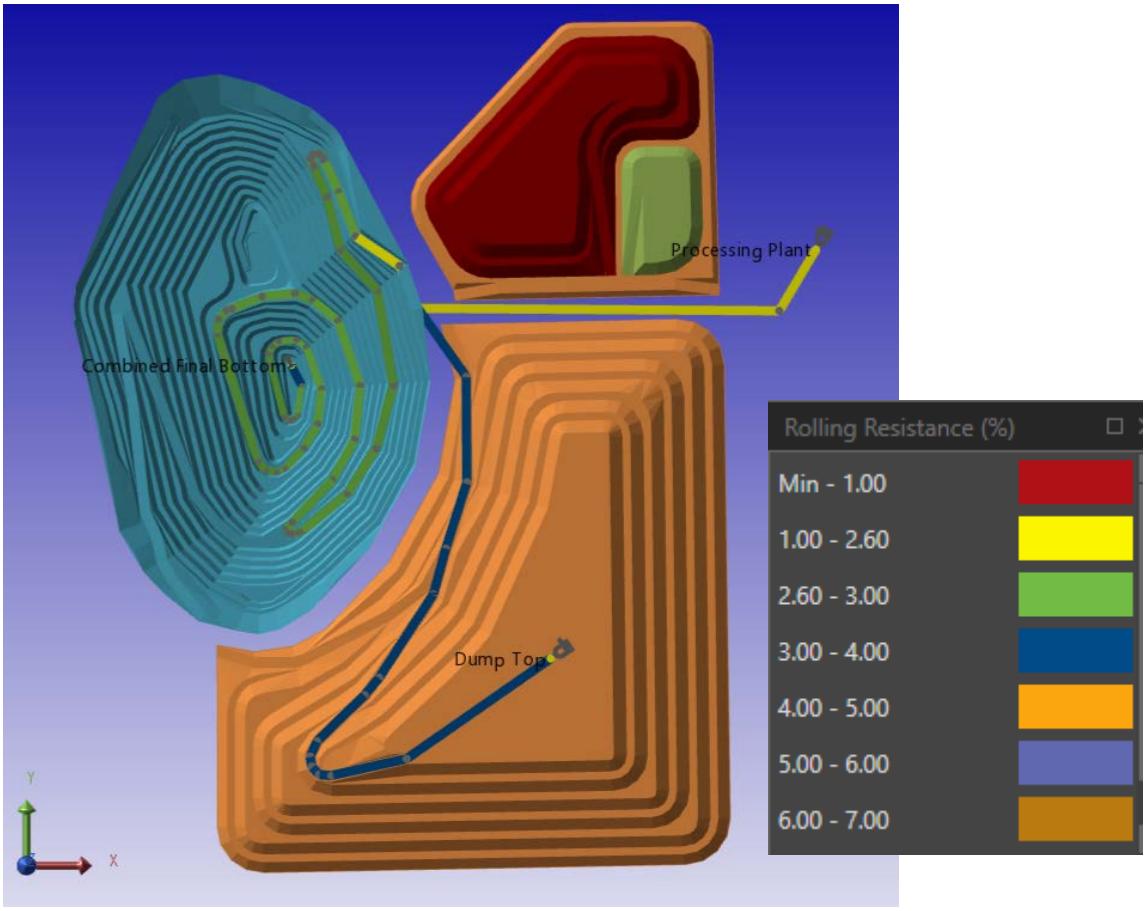


Table 16.30: Fixed Cycle Time Components

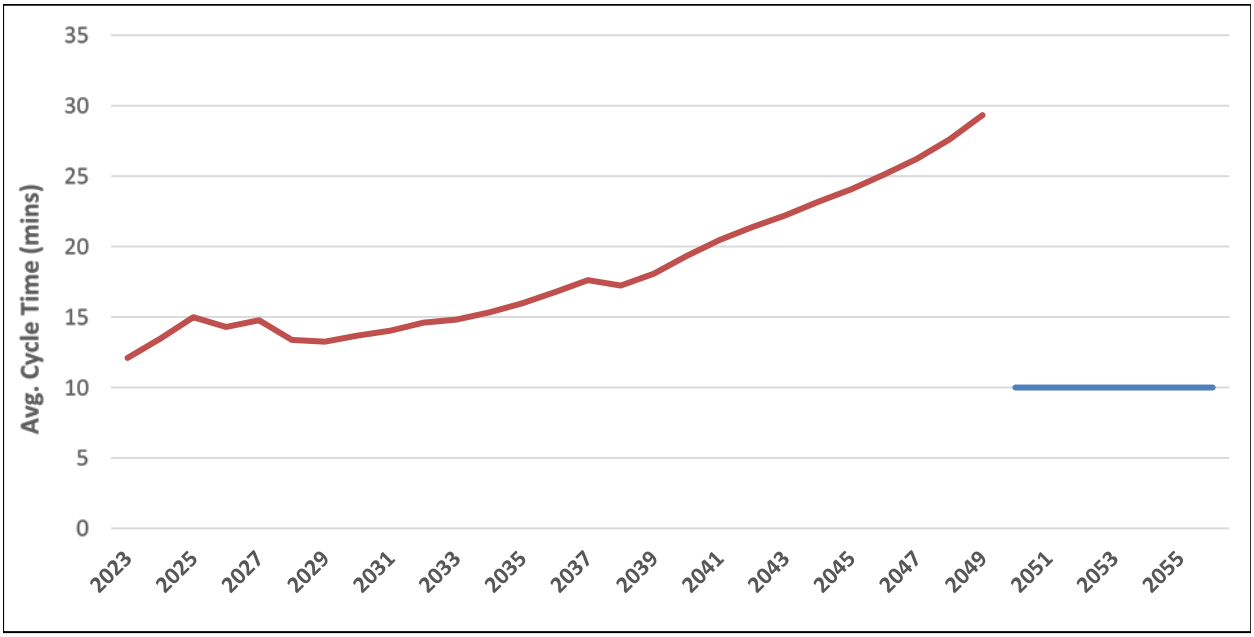
Cycle Time Component	Duration (min)
Truck Average Load Queue Time [1]	1.03
Truck Average Spot Time at Loader	0.60
Truck Average Loading Time	1.47
Truck Average Dump Queue Time	0.5
Truck Average Spot Time at Dump	0.5
Truck Average Dumping Time	0.75
Total Fixed Times	**4.85**

Note 1: The Average Load Queue Time was set to 40% of the Loading Time and Spot Time

The fixed time cycle time components are added to the travel time values simulated from Talpac. Each phase has a cycle time per bench to both the waste dump and the processing plant. Depending on the tonnages from a phase and the respective level, an overall average cycle time per year can be created as

depicted in Figure 16.24. Note an increase and decrease in the average yearly cycle times in early years. As new phases are started, and shallower benches are mined, there is a reduction in haulage distance and subsequently cycle time. The trend is in a generally upward as mining becomes deeper in later mining periods. The section in blue represents the period after the end of mining where only rehandle from the stockpile and mill is taking place. A fixed cycle time is used for this period.

Figure 16.24: Open Pit Only Scenario – Average Cycle Time per Period



The average cycle times by phase and bench combined with the productivity and capacities of the trucks were used to estimate the number of truck hours required.

The total hours an individual truck works in a given year is calculated to be 5,547 h/unit. This is a determined via availability, utilization and effectiveness. The required number of trucks required to meet truck hour demands from Figure 16.25 can be converted to truck units required. A maximum fleet of nine (9) trucks are required to maintain peak production. Fuel consumption is simulated through Talpac alongside the cycle time calculations. The total fuel required and the average fuel consumption and fuel consumption rates by period are shown in Figure 16.27.

Figure 16.25: Open Pit Only Scenario - Total Truck Hours

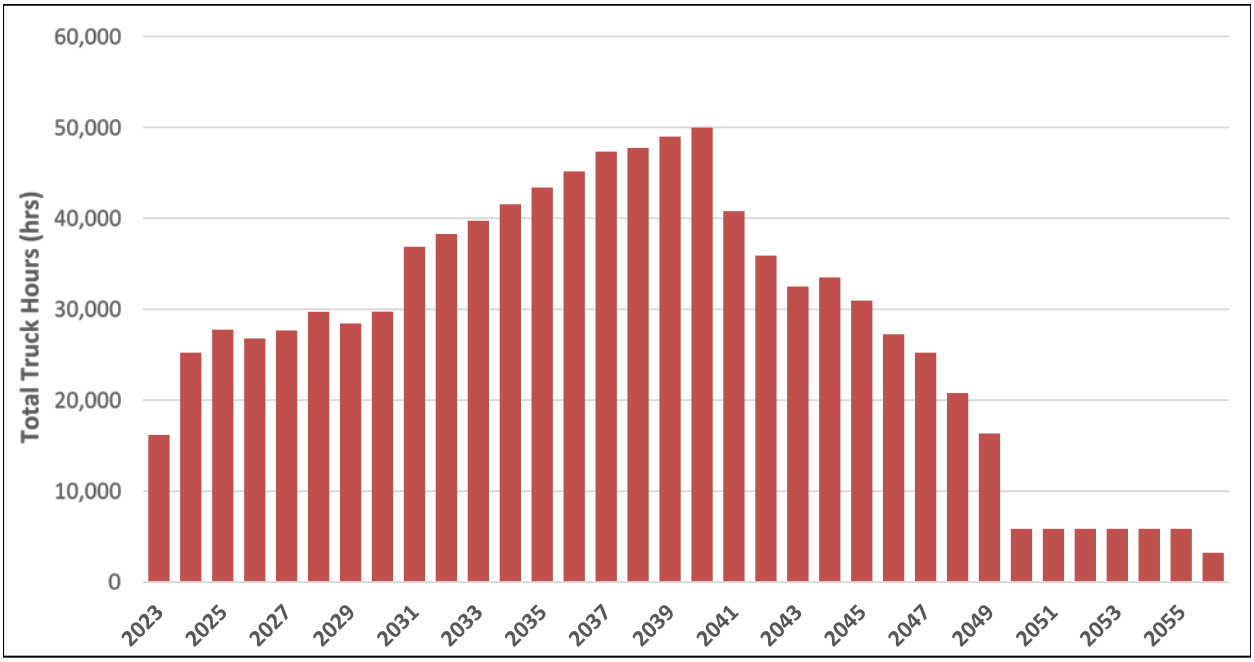


Figure 16.26: Open Pit Only Scenario - Truck Requirements

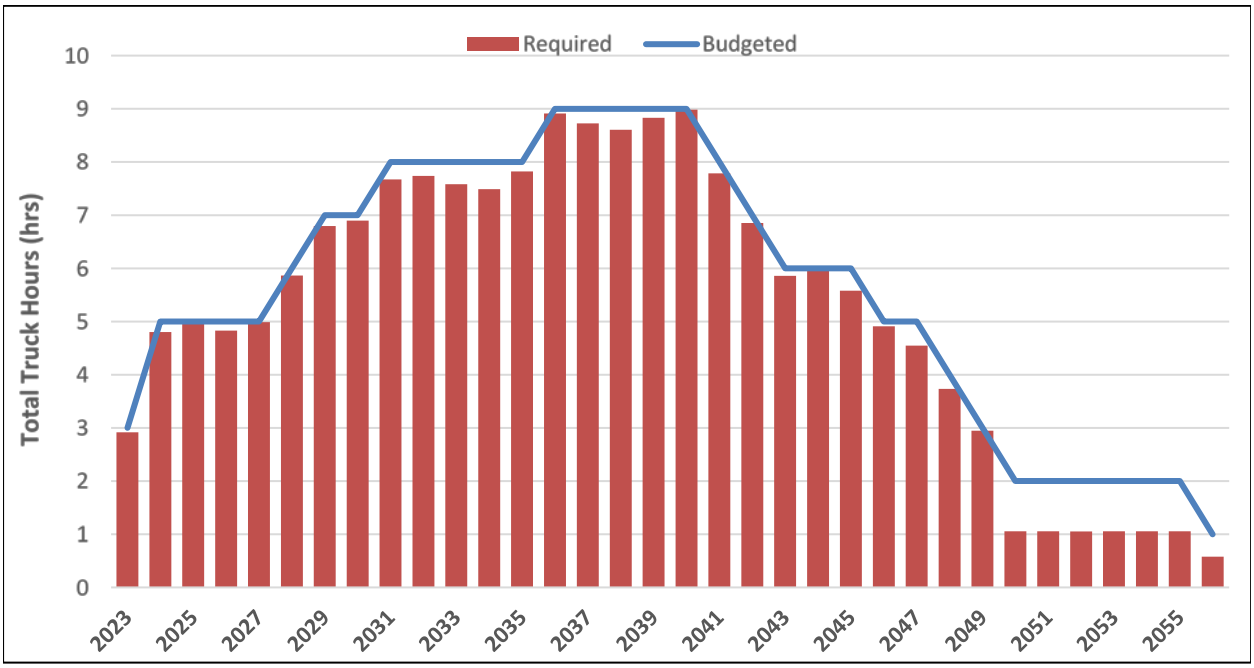
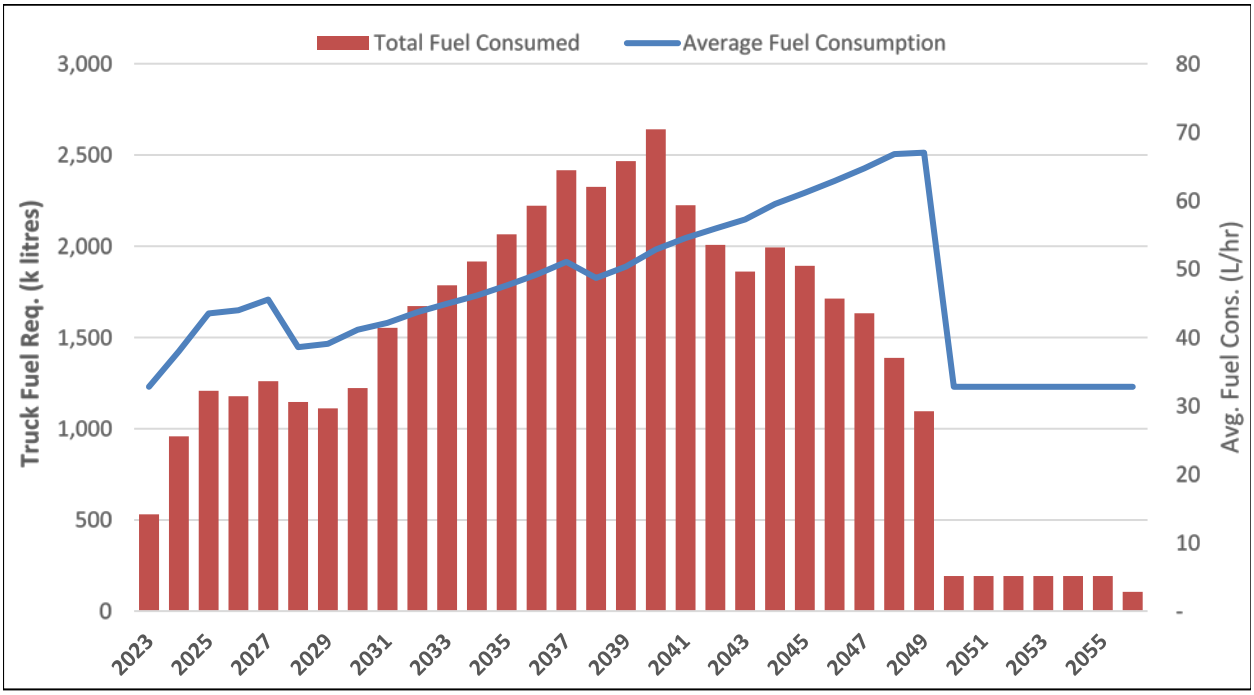


Figure 16.27: Open Pit Only - Fuel Consumptions and Rates



16.6.1.5 Road and Dump Maintenance

Waste and ore storage areas will be maintained by a fleet of three 317 HP track-type dozers. Mine roads will be maintained by one 14-ft blade motor grader. A water / sand articulated truck will be used to spray roads to suppress dust or spread road aggregate during winter months.

16.6.1.6 Support Equipment

All construction related work, such as berm construction, water ditch cleaning will be done by one 90t excavator. The same unit will take care of the pit wall scaling activities. Two (2) articulated dump trucks will also be available.

One (1) pit bus will transport workers to their assigned workplace and a total of 10 pick-ups will be purchased for the mining departments.

Several other equipment purchases are planned to support the mining activities. Part of this list of equipment are one 250 HP utility wheel loader for smaller work (and tire changes), one (1) boom trucks (28 t crane), one (1)telehandler, and a 100 t low-boy trailer and tractor for moving some of the tracked equipment.

16.6.2 Underground Mining Operations

The underground mining operations include production drilling and blasting, mucking, hauling, backfilling, and various support functions as described in the following sections.

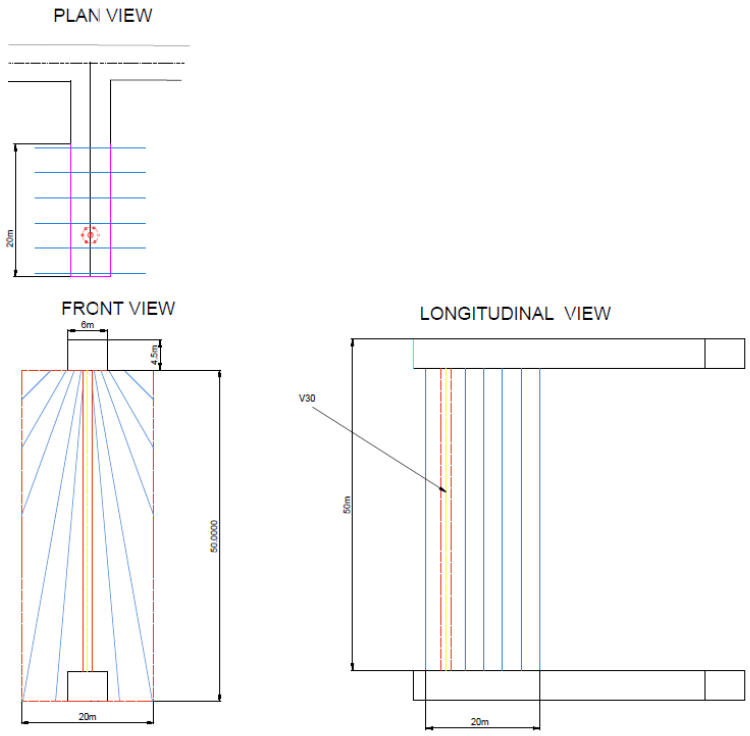
16.6.2.1 Production Drilling and Blasting

The primary stope opening (slot raise) will be done with a V-30 raise boring machine. To do this raise bore, a 6.5 in (165 mm) pilot hole is drilled to the target on the lower level. This pilot hole needs to be reamed to 10 in (250 mm). After this a machine Roger V-30 drilling head is then attached and is pulled to the upper lever. The drill rig is an ITH (Cubex type) drill adapted to the use of the V-30. This study intends to excavate the V-30 raises with a contractor. Niobay Metals may purchase the drill rig and the V-30 drill head to carry out the V-30 raises excavation. This could be beneficial but should be part of a more detailed assessment, a trade-off for use of a V-30 or of other type of raise bore could be done.

A drilling pattern has been designed for the James Bay Niobium Project. The production holes will 6.25 in diameter and fan drilled downward. Considering stope height, it was decided to use ITH tracked production drill rigs to perform the drilling. The drill holes are up to 50 m long; this requires us to reject the use of a top hammer drill, the ITH drill being much more accurate than the top hammer drill. The drilling pattern is designed with an average spacing of 3.8 m and a burden of 3.8 m. Typically, the drill rings are composed of 14 production boreholes and a stope is composed of 6 drill rings, but this can vary depending on the actual depth of the stope. A 10% provision is included to account for re-drilling and a 10% provision for additional drilling around the V-30 lift. The drilling factors are around 21.34 t/m drilled and the equivalent tonnage per ring is 10,682 t/ring. The estimated drilling performance for the ITH drill is 80 m/shift.

Pumpable emulsion type explosives will be used in stope blastholes. A powder factor of 0.8 kg/t was used with an explosive density of 1.15 g/cc Generally, the explosive emulsion used in the underground mine is sensitized with micro balloons to avoid desensitization in deep holes, this will be the case in this operation because the stopes are 50 m high.

Figure 16.28: Typical Production Drilling Pattern



16.6.2.2 Mucking

Considering the important volumes of rock to be moved, large capacity LHDs were considered. LHD loaders of 20 t were selected. To reduce airflow required for the mine ventilation, the selected LHD loader's engine must be Tier 4 or 5.

Numerous mucking situations can be anticipated in this Project. The following assumptions were taken to estimate the number of LHD units required and their costs.

LHD units will remove the ore from the stope and transport it to a muck bay or it can directly load the haul trucks. It is assumed that 50% of the ore mucking would be performed with remote control and 50% without remote control. For the cycle time calculation, it was fixed that 50% of the tonnes would transit via a muck bay. The remaining 50% would be directly loaded into the trucks.

The assumption is that 100% of the development ore will transit to a muck bay and will have to be re-handled to load the trucks. As for waste development, it is assumed that part of the waste rock will be used as rockfill and a part of it will be transported to surface.

Hauling distances were evaluated by mining sequence from the tonnage center of gravity on each level to the muck bays and loading stations. Combined with LHD performances, time cycles were evaluated. The LHD fleet requirement was estimated according to the planned tonnage associated to each situation, the planned hours and the specific performances assumed.

Table 16.31 presents the LHD specifications, speed assumptions and fixed time components of the cycle. It is estimated that remote mucking time will take three (3) times as much time as regular mucking in a stope.

16.6.2.3 Hauling

Diesel truck haulage in the ramp was selected to carry rock to the surface. The current depth of the mineralization makes trucking economically acceptable. With the high tonnage to be hauled, 60 t trucks are selected. It is important that truck engines be tier 4 or 5 to reduce mine ventilation requirements. The trucks will be loaded by the LHD at the loading point; each sector of the mine has loading points per production level. The trucks will then travel to surface where broken (not crushed) ore or waste will be unloaded on a surface transfer pad. From this pad, the ore will be transported by surface mining trucks to the process plant. The waste will be transported by surface mining trucks to the appropriate waste dump.

To estimate the required truck fleet size, haulage cycle time were evaluated for each mine zone and each level based on the development and production plan. Specifications of the 63-t haulage truck were established according to speed, fixed time and truck capacity. Table 16.32 summarizes the truck specifications and parameters.

A standard work schedule was established to determine the number of operating hours per year. It takes into account weather and other work stoppages, scheduled delays during a shift (travelling, workplace and equipment inspection, supervision, lunch). Typically, a total of 3,756 hours per year are available for effective trucking. According to those, the maximum trucks needed per year over the lifetime of the mine is of nine (9) for both scenarios.

Table 16.31: LHD Specifications

Equipment Specifications							
Equipment Capacity			Loading	45	sec	**Remote Control Mucking**	
Bucket Volume	9.8	m^3	Loading rmt	135	sec	add:	300%
Fill Capacity	90%		Unload.MB	15	sec	to normal mucking time	
Material Density	2.09	t/m3	Unload. Tr.	30	sec	**Speed Loaded (km/hr)**	**Speed Empty (km/hr)**
Tonnes/Load	18.40	t	Delays stope	90	sec	6.1	10.9
			Delay Changing dev face	90	sec		
			Waiting for truck (stope) (300 sec every 4 buckets)	**300**	sec		

Table 16.32: Truck Specifications and Parameters Summary

LHD Bucket Size:	9.8	m^3
Fill Capacity:	0.95	
Effective Volume:	9.31	m^3
Number of Passes:	3	
Effective Load/Truck:	27.93	m^3
Effective Weight/Truck:	58.25	t

EQUIPMENT CAPACITY		**Sandvik TH663**	
Box Volume	==>>	32	m^3
Fill Capacity		90%	
Material Density		2.09	t/m^3
Truck Capacity		60.07	t
Effective Load Based on LHD Capacity		58.25	t
Loading		300	sec
Unloading		45	sec
Delays		180	sec
Speed up & Loaded	2nd gear	7.5	km/h
Speed Down & Empty	3rd gear	10.6	km/h
Speed on Flat Ground	4th gear	15.9	km/h
Mechanical Availability		85%	
Use of Availability			
Surface Distance		250	m

16.6.2.4 Backfill

Paste backfill was selected as the main backfilling means for the James Bay Niobium Project. The availability of tailings, its suitable properties for the type of mining method selected, the high fill volume requirements and the good knowledge of the technology by the Owners makes paste a backfill of choice for this Project.

It is recommended to conduct testing and to initiate the engineering of the design to the prefeasibility level, obtaining equipment provider filtration test results, performing pressure loss flow loop tests, improving UCS testing with low binder content (below 3%).

The paste backfill system will allow pumping paste to two distinct networks feeding the north and south sectors. There will be one (1) active borehole for underground distribution. However, one (1) adjacent replacement hole will be drilled in case of loss of the main hole and piping will be installed in all the holes prior to the start of the operation. These holes will be drilled sufficiently large to accommodate the installation and grouting of a high pressure welded 8 in diameter schedule 80 pipes. The paste backfill will then be sent to the different levels where they will be distributed by an 8 in diameter schedule 40 horizontal pipes network installed along the footwall drifts of each mine zone.

To allow backfilling of the stopes, a barricade is built at the bottom of the stope. The barricade is made of shotcrete and reinforced with steel mesh. This type of barricade resists hydrostatic pressures developed when pouring the plug. The waste rock could also be used as a barricade.

Backfilling of the first metres of the stopes will be performed with a binder percentage of 5%. This backfill plug is expected to be poured to a height of 1 to 2 m above the brow of the draw-point access. This will allow the sealing of the barricade and resist the pressure created by 50 m of backfill poured quickly afterwards into the stope. The percentage of binder for the primary stope will 3% and 1.5% for the secondary stope. When possible, some backfill will include rock fill in the secondary slope.

It is expected that over time, up to 1,084,264 t of waste rock will be mined in Scenario 2 and 1,246,177 t in Scenario 3. 25% will be hauled to the surface (mostly at the beginning of the operation as no stopes are available to receive backfill) while 75% of waste material will be left underground, mixed with paste fill mostly as part of the secondary stope backfilling strategy. In total, the mine will receive 13,499,577 m³ or 26,999,154 t of paste fill necessitating 19,169,399 t of dry tailings for the Scenario 2. For Scenario 3, the mine will receive 18,208,868 m³ or 36,417,737 t of paste fill necessitating 25,856,593 t of dry tailings.

16.6.2.5 Ventilation

16.6.2.5.1 Fresh Air Required

Fresh air requirement has been established in order to satisfy the Ontario. R.O. 1990, Reg. 854: MINES AND MINING PLANTS under Occupational Health and Safety Act, R.S.O. 1990, c. O.1 article 183.1.

As per Ontario regulations, a dilution rate of 0.06 m³/s per kW must be supplied to each working area where diesel equipment is operating. The dilution rate for each equipment is listed in Table 16.33. To estimate the cumulative rate of fresh airflow for the mine, a percentage of utilization has been used to estimate the real use time of equipment. A contingency of 10% has been added on the total airflow required to prevent additional equipment that could be added during the life of mine. It also allows a quantity of air for potential leaks in the system. The number of equipment at the peak for Scenarios 2 and 3 are the same. According to following table, the airflow required at the peak is estimated at 830 kcfm (or 391 m³/s).

Table 16.33: Airflow Required For Scenarios 2 and 3

Equipment		Power		Req/Equip*		#Equip.	Total Airflow Required		
		(kW)	(HP)	(m^3/s)	(CFM)		(% util)	(m^3/s)	(CFM)
2-Boom Jumbo	Sandvik	122	163	7.3	15,511	2	0.35	5.1	10,858
ITH	Sandvik	122	163	7.3	15,511	3	0.50	11.0	23,267
Cable Bolter	Sandvik	119	159	7.1	15,130	1	0.25	1.8	3,782
U/G Loader 21 Mt	Sandvik	352	472	21.1	44,753	3	**0.75**	47.5	100,695
U/G Truck 63t	Sandvik	565	757	33.9	71,834	9	**0.75**	228.8	484,880
U/G Grader	CAT	104	139	6.2	13,188	1	**0.75**	4.7	9,891
Lube Truck	Maclean	110	147	6.6	13,985	1	0.50	3.3	6,993
Tractor Lift	Kubota	57	76	3.4	7,211	6	0.50	10.2	21,633
Tractor	Kubota	46	62	2.8	5,883	20	0.25	13.9	29,413
Backhoe	CAT	206	276	12.4	26,187	1	0.15	1.9	3,928
Explosives Loader	Getman	127	170	7.6	16,147	2	0.50	7.6	16,147
Personnel Carrier	Getman	127	170	7.6	16,147	2	0.25	3.8	8,073
Scissor Lift	Getman	127	170	7.6	16,147	4	0.15	4.6	9,688
Boom Truck	Getman	127	170	7.6	16,147	2	0.25	3.8	8,073
Shotcrete Sprayer	Getman			4.7	10,000	1	0.25	1.2	2,500
Bolter	Maclean	110	147	6.6	13,985	2	0.50	6.6	13,985
Subtotal								**355.7**	**753,807**
Contingency	10%							**35.6**	**75,381**
Total								**391**	**829,187**

16.6.2.5.2 Ventilation Infrastructure and Network

The mine ventilation system consists of one air intake. Each level is supplied with fresh air with this raise breaking through each of the active levels at level access.

The exhaust system consists of the ramp and four raises located on haulage drift. Figure 16.29 and Figure 16.30 respectively present section views of the mine ventilation network for Scenarios 2 and 3

The fresh air raise breaks through all active levels and airflows on levels will be controlled by automated regulators. Contaminated air will be evacuated by the ramp and the exhaust raise system.

Figure 16.29: Ventilation Network for Scenario 2

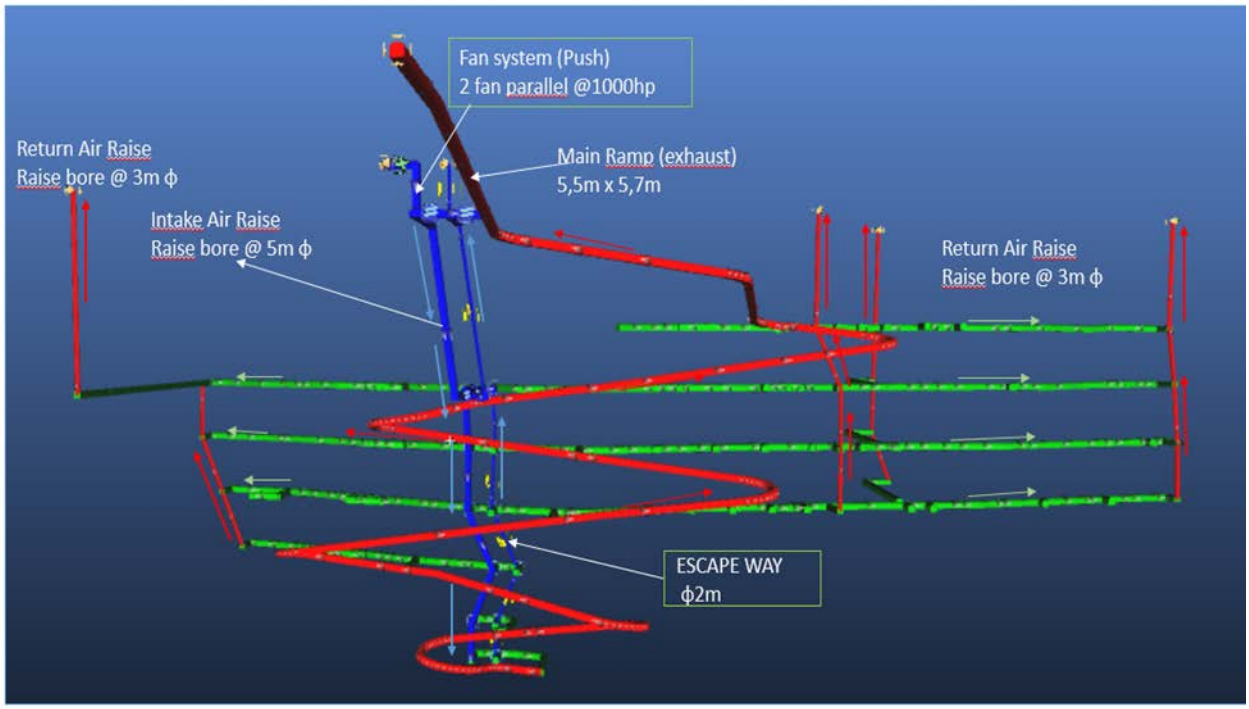
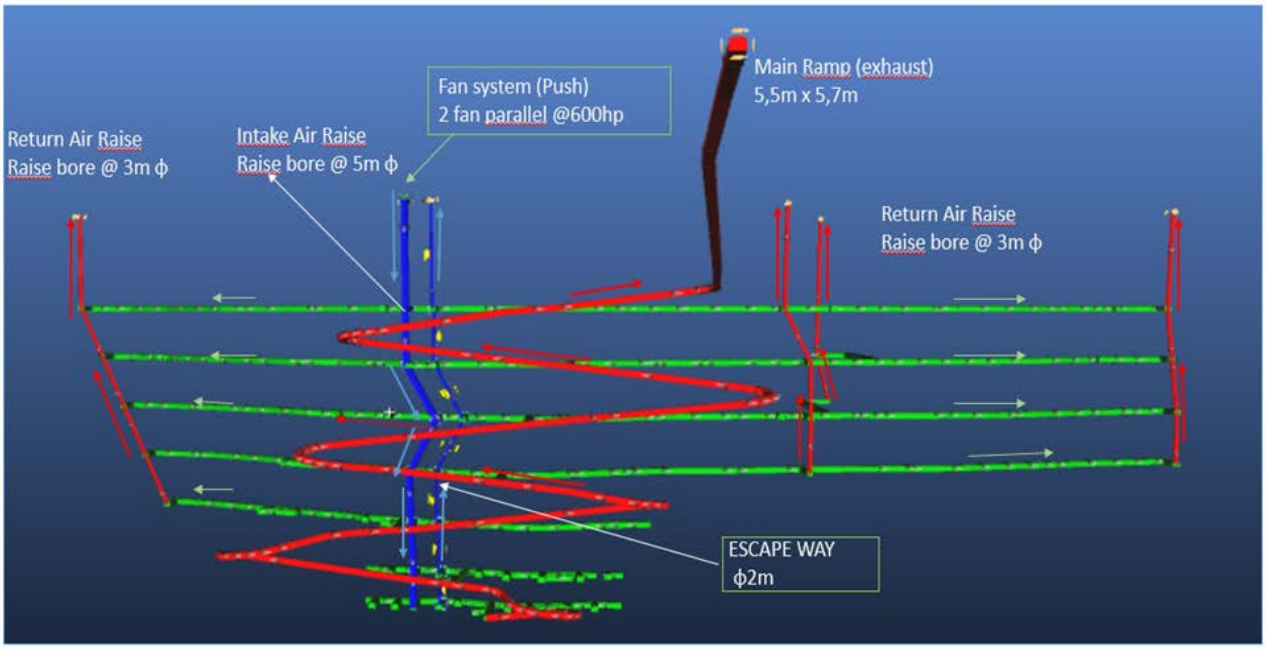


Figure 16.30: Ventilation Network for Scenario 3



As mentioned, ventilation network will be supplied with fresh air through an independent fresh air raise. This raise will be equipped with two (2) fans in parallel. Depending on the scenario, the estimated fan operating point simulated in *Ventsim*™ varies slightly.

Table 16.34: Fan System

Scenario	Airflow		Pressure		Total Power		Fan Arrangement
	(m^3/s)	(k CFM)	(Pa)	(in H_2O)	(kW)	(HP)	
Sc 2	391	830	2 600	10.2	1 356	1,800	2 x fan parallel @ 1000 HP
Sc 3	391	830	1 600	6.5	815	1,100	2 x fan parallel @ 600 HP

16.6.2.6 Dewatering

The amount of water pumped is contingent upon two separate water supplies. The first is groundwater seepage in the underground mine excavation; the second is water consumption for mining equipment. Water seepage into the mine was assessed at 2,800 m^3/d during the peak period. The estimation of water infiltration in the mine should be performed in more detail as this is only a preliminary assumption. The water consumption of the equipment is evaluated according to the equipment used in the study and considers a utilization factor. The different pump capacity calculations are calculated for the development period and the production period. In the development period the pumping system will only be temporary and replaced by a permanent system during operations where the peak pumping requirement was estimated to reach up to 2,854 l/min (754 USGM).

Table 16.35: Dewatering

Type	USGM
Natural Ground Water	514
Wastewater	240
Total	**754**
Safety Factor	1.45
Pumping Capacity	1,100

16.6.2.7 Pumping System

A mobile pumping system with a steel bin will be used during the development phase of the Project. The system will be equipped with two 50 HP (37 KW) pumps with a total capacity greater than 335 USGPM given that groundwater infiltrations and operation mine water. One (1) system will be required for the

development from the surface to level -100 and two (2) systems in series will be required to develop from level -100 to level -330.

There will be permanent pumping station for the James Bay Niobium Project, which will be located on the last level -330. The pumping station on level -330 will direct water to the surface water treatment facility. The system has a capacity of 1,100 USGPM compared to groundwater infiltrations estimated at 514 USGPM and mining equipment water use estimated at 240 USGPM. The pumps will therefore operate 60% of the time, which allows for margin in the system. The station at level -330 requires a 500 HP pump capable of overcoming 264 m of static head. An 8 in pipe will be required to dewater the mine. Drifts driven for the pumping stations will be 5 m H x 5m W x 20m L to include enough space for one pump and a 74 m^3 reservoir.

A Mudwizard system will be installed on level -330 to manage mud and sediments in the dewatering system. All the dirty water in the mine will be sent to this system to decant water from the solids. A drift measuring 5m H x 5m W x 20m L will need to be driven to house the Mudwizard system.

16.6.2.8 Power

Major electrical power consumption in the mine will be required for the main and auxiliary ventilation fans, jumbo and bolter equipment and the main dewatering pumps.

Power consumption was based on the required mine equipment derived from the development plan and the production plan estimated at just under 4 MW of connected load.

A high voltage cable (4.16 kV) will be installed in the ramp and in drill holes to electrify the different areas of the mine. During the initial development of the ramp, junction boxes will be installed at every 230 m along the cable. A mobile electrical substation converting high voltage to 600 V will be used to electrify mining equipment. Permanent electrical substations converting high voltage to 600 V will be installed at the entrance of each production level and in other areas outside of the 230 m reach. Drill holes between levels will also be used to enhance the reach of certain permanent electrical substations to other levels. The two (2) pump stations will also be electrified by the main high voltage cable and will require electrical substations as well. For the case of the ventilation fans, another high voltage cable will run directly from surface to an electrical substation to electrify the VFD.

16.7 Mine Equipment Requirements

The main factors which influenced the selection of the major mine equipment included the annual production requirements and optimization of the fleet size based on an owner mining approach.

The peak equipment requirements with the units purchased as part of initial capital and sustaining capital over the LOM are presented in Table 16.36, Table 16.37 and Table 16.38 for Scenario 1, Scenario 2 and Scenario 3 respectively.

Table 16.36: Scenario 1 Equipment Purchase Schedule

Equipment Requirements & Purchase Schedule	Peak Requirements	Units Purchased		
		Initial	Sustaining	Total
Major Equipment				
Mining Haul Truck (70 t)	9	4	16	20
Diesel Hydraulic Excavator (7 m^3)	2	2	3	5
Wheel Loader (8.2 m^3)	1	1	1	2
Production Drill (6-10")	1	1	1	2
Track Dozer (317 HP)	3	3	11	14
Motor Grader (14 ft)	1	1	4	5
Articulated Dump Truck (34 kL tank)	1	1	4	5
Auxiliary Pre-split Drill (4.5-8")	1		2	2
Support Equipment				
Articulated Dump Truck 45 t	2	2	2	4
Excavator (90 t)	1	1	1	2
Hydraulic Hammers for Excavator	1	1	4	5
Emulsion Truck	1		3	3
Stemming Loader	1		2	2
Utility Wheel Loader (271 HP)	1	1	2	3
Boom Truck (28 t crane)	1	1	3	4
Telehandler	1	1	2	3
Forklift 4 t	1	1	1	2
Mechanic Service Truck	2	1	7	8
Tire Handler Attachment	1	1	1	2
Tire Handler Tooling & Equipment	1	1	0	1
Fuel & Lube truck (7.5 kL) Conventional	1	1	3	4
Lowboy Tractor	1	1	2	3
Low Boy Trailer 50 t	1	1	1	2
Excavator Moving System - Sleipner	1		2	2
Pick-up Truck	10	10	40	50
Pit Bus	1	1	4	5
Electric Welding Machines	2	2	4	6
Diesel Welding Machines	2	2	6	8
Tower Lights	4	4	20	24
Genset 6 kW	2	2	6	8
Genset 60 kw	1	1	5	6
Diesel Dewatering Pump 10 in	1	1	1	2
Diesel Dewatering Pump 6 in	1	1	4	5
Trash Pump 3 in	4	4	28	32
Diesel Powered Air Heaters	2	2	8	10

Table 16.37: Scenario 2 Equipment Purchase Schedule

Equipment Requirements & Purchase Schedule	Peak Requirements	Units Purchased		
		Initial	Sustaining	Total
Major Equipment Open Pit				
Mining Haul Truck (70 t)	9	4	16	20
Diesel Hydraulic Excavator (7 m^3)	2	2	3	5
Wheel Loader (8.2 m^3)	1	1	1	2
Production Drill (6-10")	1	1	1	2
Track Dozer (317 HP)	3	3	11	14
Motor Grader (14 ft)	1	1	4	5
Articulated Dump Truck (34 kL tank)	1	1	4	5
Auxiliary Pre-split Drill (4.5-8")	1		2	2
Major Equipment Underground				
2 Boom -Jumbo	3	1	1	2
ITH	4	1	3	4
U/G Truck 60t	9	3	11	14
U/G loader 20t	7	0	11	11
Bolter	2	1	6	7
Support Equipment Open Pit				
Articulated Dump Truck 45 t	2	2	2	4
Excavator (90 t)	1	1	1	2
Hydraulic Hammers for Excavator	1	1	4	5
Emulsion Truck	1		3	3
Stemming Loader	1		2	2
Utility Wheel Loader (271 HP)	1	1	2	3
Boom Truck (28 t crane)	1	1	3	4
Telehandler	1	1	2	3
Forklift 4 t	1	1	1	2
Mechanic Service Truck	2	1	7	8
Tire Handler Attachment	1	1	1	2
Tire Handler Tooling & Equipment	1	1	0	1
Fuel & Lube Truck (7.5 kL) Conventional	1	1	3	4
Lowboy Tractor	1	1	2	3
Low Boy Trailer 50 t	1	1	1	2
Excavator Moving System - Sleipner	1		2	2
Pick-up Truck	10	10	40	50
Pit Bus	1	1	4	5
Electric Welding Machines	2	2	0	2
Diesel Welding Machines	2	2	0	2
Tower Lights	4	4	0	4
Genset 6 kW	2	2	0	2
Genset 60 kw	1	1	0	1
Diesel Dewatering Pump 10 in	1	1	1	2
Diesel Dewatering Pump 6 in	1	1	4	5
Trash Pump 3 in	4	4	28	32
Diesel Powered Air Heaters	2	2	8	10
Support Equipment Underground				
Explosives Truck	2	1	2	3
U/G Grader	1		2	2
Pick-up Truck	3	1	8	9
Lube Truck	2	1	3	4
Scissor Lift	5	1	4	5
Tractor	20	4	76	80
Tractor - Services	6	1	17	18
Cable Bolter	1	0	1	1
Shortcrete Sprayer	1	1	1	2
Boom Truck	2	1	3	4
Wheel Backhoe	2	1	2	3
Personnel Carrier	2	1	3	4

Table 16.38: Scenario 3 Equipment Purchase Schedule

Equipment Requirements & Purchase Schedule	Peak Requirements	Units Purchased		
		Initial	Sustaining	Total
Major Equipment				
2 Boom -Jumbo	3	1	1	2
ITH	4	1	3	4
U/G Truck 60 t	9	3	11	14
U/G loader 20 t	7	0	11	11
Bolter	2	1	6	7
Support Equipment				
Explosives Truck	2	1	2	3
U/G Grader	1		2	2
Pick-up Truck	3	1	8	9
Lube Truck	2	1	3	4
Scissor Lift	5	1	4	5
Tractor	20	4	76	80
Tractor - Services	6	1	17	18
Cable Bolter	1	0	1	1
Shortcrete Sprayer	1	1	1	2
Boom Truck	2	1	3	4
Wheel Backhoe	2	1	2	3
Personnel Carrier	2	1	3	4

16.8 Mine Workforce

16.8.1 Roster Schedule and Assumptions

The mine will be planned with a typical northern Canadian operations schedule consisting of 14 days on, 14 days off rotating schedule with twelve-hour shifts. Four (4) crews are required to operate on a continuous basis 24 hours per day 365 days per year.

16.8.2 Mine Workforce Requirements

For Scenario 1 the mine workforce reaches a peak of 149 people at Year 8 (Table 16.39Table 16.39: Manpower Requirements – Scenario 1

). The mine workforce requirements over the mine life vary with the tonnage mined and required operators. with a reduction occurring when the tonnage decreases starting at Year 14 of operation. Workforce numbers start declining in Year 17 as the strip ratio decrease substantially.

For Scenarios 2 and 3, the mine will operate seven (7) days per week with two (2) 12-hour shifts per day. A local mining contractor is originally planned to undertake all the lateral development, the raises and for specific surface work during the pre-production period.

Niobay Metals will gradually hire mine workers, to be largely selected from the contractor's team, as the mine development progresses until it reaches the full employee complement. Except for the raise and definition drilling, Niobay Metals will hire its own mine crews to fill all the positions related to mine development, production, construction, and mine services by the end of pre-production.

Development and production crews will be on a schedule of 14 days working / 14 days off, for 12 h/shift, night and day. The maintenance crew including mine services, mechanical services, electrical services, surface services crew and technicians will also be on a schedule of 7 working days / 7 days off, for 12 h/shift, night and day. Technical services crew except technicians and superintendent for electrical and mechanical services will be on a schedule of 7 working days / 7 days off, for 10 h/shift, night and day.

Table 16.40 and Table 16.41 list the total Project manpower requirements for each department and over the LOM respectively for Scenarios 2 and 3.

Table 16.39: Manpower Requirements – Scenario 1

Year	Open Pit					Total
	Mine Ops.	Maint.	Geology	Eng.	Sub-Total Open Pit	
-2					**0**	**0**
-1	66	27	7	9	**109**	**109**
1	81	34	13	10	**138**	**138**
2	81	34	13	10	**138**	**138**
3	85	38	13	10	**146**	**146**
4	85	38	13	10	**146**	**146**
5	84	38	13	10	**145**	**145**
6	84	38	13	10	**145**	**145**
7	84	38	13	10	**145**	**145**
8	88	38	13	10	**149**	**149**
9	88	38	13	10	**149**	**149**
10	88	38	13	10	**149**	**149**
11	88	38	13	10	**149**	**149**
12	80	38	13	10	**141**	**141**
13	84	38	13	10	**145**	**145**
14	84	38	13	10	**145**	**145**
15	84	38	13	10	**145**	**145**
16	84	38	13	10	**145**	**145**
17	72	38	13	7	**130**	**130**
18	72	34	13	6	**125**	**125**
19	68	34	13	10	**125**	**125**
20	64	34	13	10	**121**	**121**
21	64	34	13	10	**121**	**121**
22	64	34	13	10	**121**	**121**
23	60	34	13	10	**117**	**117**
24	60	34	13	4	**111**	**111**

Table 16.40: Manpower Requirements – Scenario 2

Year	Open Pit Mine					Underground Mine							Total
	Mine Ops.	Maint.	Geology	Eng.	Sub-Total Open Pit	Mine Ops.	Mine Services	Mech. Services	Elect. Services	Surface Services	Tech. Services	Sub-Total UG	
-2													**0**
-1	66	27	7	9	**109**								**109**
1	81	38	13	10	**142**								**142**
2	81	34	13	10	**138**								**138**
3	65	30	13	10	**118**	11	7	2	5	9	2	**36**	**154**
4	56	30	13	10	**109**	48	10	6	7	9	15	**95**	**204**
5						91	20	24	17	14	32	**198**	**198**
6						90	20	31	22	14	32	**209**	**209**
7						98	20	31	22	14	32	**217**	**217**
8						98	20	35	26	14	32	**225**	**225**
9						86	20	35	26	14	32	**213**	**213**
10						86	20	31	22	14	32	**205**	**205**
11						87	20	31	22	14	32	**206**	**206**
12						86	20	31	22	14	32	**205**	**205**
13						86	20	31	22	14	32	**205**	**205**
14						82	20	31	22	14	32	**201**	**201**
15						84	20	31	22	14	32	**203**	**203**
16						84	20	31	22	14	32	**203**	**203**
17						75	20	31	22	14	32	**194**	**194**
18						75	20	31	22	14	32	**194**	**194**
19						75	20	31	22	14	32	**194**	**194**
20						70	20	31	22	14	32	**189**	**189**
21						71	20	31	22	14	32	**190**	**190**
22						65	20	31	22	14	32	**184**	**184**

Table 16.41: Manpower Requirements – Scenario 3

Year	Underground Mine							Total
	Mine Ops.	Mine Services	Mech. Services	Elect. Services	Surface Services	Tech. Services	Sub-Total UG Mine	
-2	6	4	2	2	5	1	**20**	**20**
-1	39	10	7	7	9	15	**87**	**87**
1	91	20	17	17	14	32	**191**	**191**
2	86	20	26	26	14	32	**204**	**204**
3	88	20	26	26	14	32	**206**	**206**
4	92	20	26	26	14	32	**210**	**210**
5	93	20	26	26	14	32	**211**	**211**
6	92	20	26	26	14	32	**210**	**210**
7	92	20	26	26	14	32	**210**	**210**
8	92	20	26	26	14	32	**210**	**210**
9	92	20	26	26	14	32	**210**	**210**
10	92	20	26	26	14	32	**210**	**210**
11	92	20	26	26	14	32	**210**	**210**
12	92	20	26	26	14	32	**210**	**210**
13	91	20	26	26	14	32	**209**	**209**
14	84	20	26	26	14	32	**202**	**202**
15	85	20	26	26	14	32	**203**	**203**
16	85	20	26	26	14	32	**203**	**203**
17	84	20	26	26	14	32	**202**	**202**
18	84	20	26	26	14	32	**202**	**202**
19	84	20	26	26	14	32	**202**	**202**
20	84	20	22	22	14	32	**194**	**194**
21	84	20	22	22	14	32	**194**	**194**
22	73	20	22	22	14	32	**183**	**183**

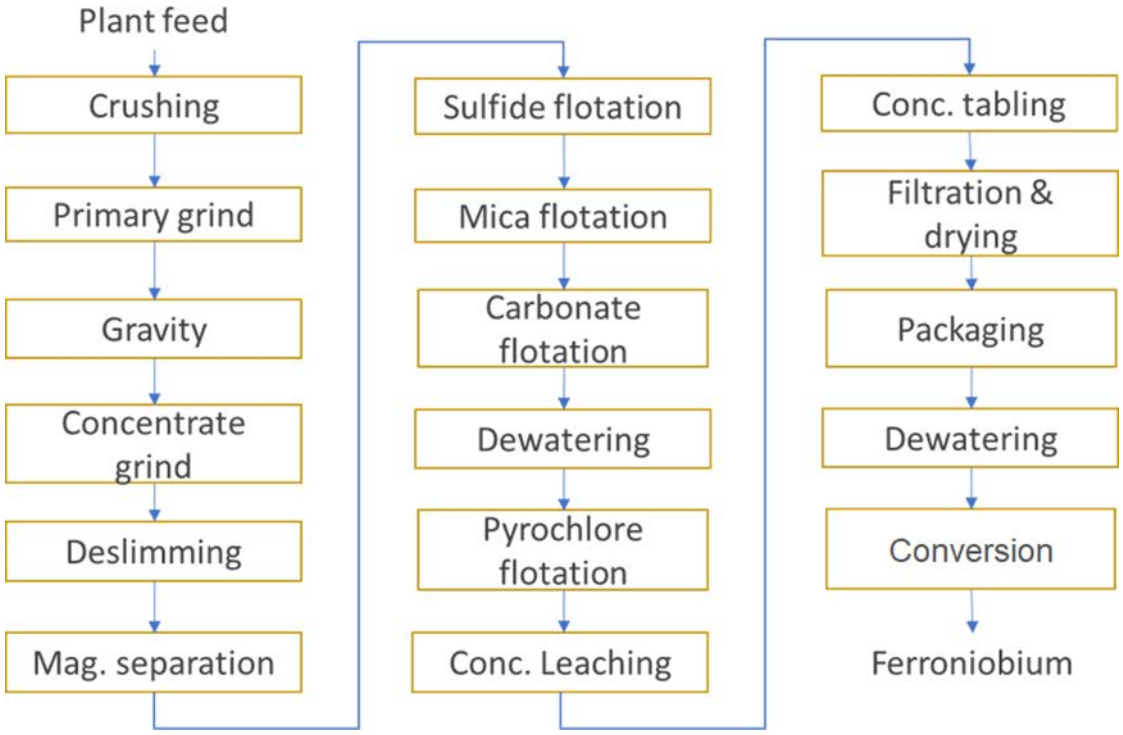
17. RECOVERY METHOD

17.1 Process Selection

The selection of the recovery method was based on the metallurgical test work described in Section 13. This selection also reflects the technology available and in use today. The process flowsheet and design criteria are based on the metallurgical test work results and current industry practice. The process flowsheet description is for a tonnage of 2,400,000 tpy at a process plant availability of 93%. The flowsheet retained includes a gravity circuit removing close to 42% of the mass with limited niobium losses. This aspect of the process minimizes the energy required, considerably reduces the volume of pulp and has a positive impact on reagent costs. The reagents consumption was estimated from metallurgical test results.

The low grinding index of the ore and the coarse grind required for a good liberation of the niobium mineral minimize the power requirement for grinding. A total of 1,900 kW was estimated for the entire grinding stage to prepare ore for processing; the flowsheet will include two stages of grinding with a gravity interstage. Grinding and gravity concentration will be followed by pulp desliming, magnetic separation, and three steps of specific mineral removal prior to niobium flotation. The niobium concentrate will be leached, filtered and a final gravity separation will be conducted to generate two different concentrate grades. The final concentrates will be dried and bagged to generate the feed for the conversion process. The selected process flowsheet sequence is presented in Figure 17.1.

Figure 17.1 : Process Flowsheet Diagram



17.2 Process Description

17.2.1 Crushing

The ore will be crushed at less than 200 mm using a primary jaw crusher installed close to the process plant for the scenario based on an open pit operation. When an underground operation becomes the ore source, crushing may be done underground before being hoisted to surface.

17.2.2 Grinding

The grinding circuit will consist of two grinding steps in series with an interstage gravity circuit. The primary grinding will be done using a semi-autogenous (“SAG”) mill in closed circuit with cyclones with a final product passing 500 microns. The cyclone product will be further classified to remove the less 74 microns fraction before feeding the gravity circuit. The gravity concentrate will be reground in a ball mill in closed circuit to less than 250 microns prior to feeding the flotation circuit.

17.2.3 Gravity Separation

The specific characteristics of the James Bay ore provide the opportunity to remove a significant part of the gangue mineral with limited niobium mineral losses. The gravity separation is done in two-steps with gravity spiral in series. The current test work shows that 42% of the mass is removed at this stage with a positive impact on the downstream flowsheet. The gangue removed is sent to tailings.

17.2.4 Desliming Stage

To prepare the ore for the flotation step, the ore is deslimed. The trash is removed by using a linear screen and desliming occurs in a two-step cyclone classification. The cyclone overflow is sent to tailings.

17.2.5 Magnetic Separation

The magnetic iron mineral is removed from the ore using a two-step magnetic separator; the concentrate from the first step is cleaned in the second step and the final iron concentrate is sent to tailings.

17.2.6 Sulfide Flotation

The sulfides minerals are removed by flotation in three rougher flotation cells in series, and the sulphides concentrate is cleaned in a single stage of cleaner. Prior to the sulfide feed, the reagents are conditioned in two steps in conditioner tanks. The final sulphide concentrate is also sent to tailings.

17.2.7 Mica Flotation

Mica minerals are removed from the ore in a single step using three flotation cells in series. The mica minerals concentrate is sent to tailings.

17.2.8 Carbonate Flotation

Prior to carbonate flotation, the reagents are conditioned in two tanks in series. The carbonate flotation occurs in three flotation cells in series, and the concentrate is cleaned twice. The carbonate concentrate is sent to tailings.

17.2.9 Dewatering

Combined rejects from the carbonates flotation step are sent to the dewatering stage The pulp is dewatered in three steps of cycloning to remove extra water and fine particles and increase the pulp density. The cyclone overflow from the third step is sent to the tailings and the final cyclone underflow from the third step is fed to the niobium circuit.

17.2.10 Niobium Flotation

Prior to the niobium flotation, the pulp is scrubbed in an attrition tank followed by reagent conditioning in a second tank in series. The niobium rougher flotation is done in eight flotation cells followed by a six-step concentrate cleaning. Interstage conditioning and magnetic separation are included in the cleaning circuit.

17.2.11 Concentrate Leaching

The niobium concentrate is thickened and leached with hydrochloric acid (HCL) to remove residual phosphate and carbonate minerals. The leached concentrate is washed and cleaned on a gravity table to further remove silicate minerals and produce two niobium concentrate grade, a low silica and a standard silica grade.

17.2.12 Filtration, Drying and Packaging

The two niobium concentrates are filtered and dried in a rotative kiln before being stored in designated bins depending on the concentrate quality. The concentrates are mixed and bagged to provide adequate charges for the conversion to FeNb.

17.2.13 Reagents

Based on the current metallurgical test work, sixteen (16) different reagents will be used in the processing flowsheet. The type and consumption of reagents will be refined in future metallurgical test work. Depending on the properties of each reagent, adequate storage facility will be built according to the safety data sheet and recommendations for storage, handling, and mixing.

17.3 Process Design Criteria and Mass Balance

The process design criteria are based on the metallurgical test work and current practice in the industry. The proposed design criteria are preliminary and will need to be refined with future metallurgical test work.

The major design criteria are presented in Table 17.1

Table 17.1: Major Design Criteria

Description	Unit	
Process Ore Feed		
Annual Ore Production	tpy	2,400,000
Mill Throughput, Operation	tph	294.5
Mill Availability	%	93
Niobium Feed Grade	% Nb_2O_5	0.53
Ore Properties		
Moisture	%	3%
Specific Gravity	g/cm^3	2.8
Crushing (Jaw Crusher)		
Operation Hours	h/d	16
Crushing Index (CWI)	kwh/t	12
Power	kW	375
Feed Size (F80)	mm	619
Product Size (P80)	mm	150
Ore Bin	Number	2
Live Load	t	2,000
Grinding		
SAG		
SAG Mill Feed	tph	294.5
SAG Mill Power	**kW**	**1,500**
Abrasion Index		0.07
SAG Mill Feed Size (F80)	µm	150,000
SAG Mill Product (P80)	µm	500
SAG Mill Circulating Load	%	50
SAG Mill Ball Size	mm	125
SAG Mill Cyclone Size	cm	60
Sag Mill Cyclone Product (Max Size)	µm	500
Number of Cyclones	Number	4
Spiral Cyclone Size	cm	30
Number of Cyclones	Number	10
Number of Spirals	Number	160
Ball Mill		
Ball Mill Power	kW	400

Description	Unit	
Ball Mill Feed (Max Size)	µm	500
Ball Mill Product (Max Size)	µm	250
Ball Mill Ball Size	mm	50
Ball Mill Circulating Load	%	250
Total Cyclones	Number	10
Ball Mill Cyclone Size	cm	50
Gravity Recovery	%	58
Gravity Concentrate	tpy	1,276,000
Desliming Stage		
Trash Screen Size Opening	µm	800
Primary Cyclones Size	mm	250
Number of Primary Cyclones	Number	12
Secondary Cyclone Size	mm	40
Number of Secondary Cyclones	Number	480
Magnetic Separation		
Primary Separator Size Diameter	mm	1,200
Secondary Separator Size Diameter	mm	900
Pyrite Flotation		
Conditioning	min	5+5
Rougher Flotation	min	12.5
Cleaner Flotation	min	10
Mica Flotation		
Rougher Flotation	min	15
Carbonate Flotation		
Conditioning	min	5+5
Rougher Flotation	min	15
First Cleaner Flotation	min	12.5
Second Cleaner Flotation	min	12.5
Secondary Desliming		
Primary Cyclones - Size	mm	250
Number of Primary Cyclones	Number	6
Secondary Cyclones – Size	mm	100
Number of Secondary Cyclones	Number	18
Tertiary Cyclones – Size	mm	40
Number of Tertiary Cyclones	Number	240
Niobium Flotation		
Scrubbing	min	5

Description	Unit	
Conditioning	min	5
Rougher Flotation	min	10
First Cleaner Flotation	min	5
Second Cleaner Flotation	min	5
Cleaner Conditioning	min	5
Cleaners 3 & 4	min	4
Magnetic Separation Cleaner Diameter	mm	900
Cleaners 5 & 6	min	3.5
Concentrate Leaching		
Retention Time	hour	12
Niobium Concentrate and Recovery		
Final Concentrate Grade	% Nb_2O_5	60
Nb_2O_5 Recovery in Final Concentrate	%	78
Concentrate Tonnage	tpy	16,412
Niobium Production		
Nb_2O_5 Production	kg	9,846,720
Converter Nb Recovery	%	96.5
Ferroniobium Grade	% Nb	66
Ferroniobium Production	kg FeNb	10,063,571

18. INFRASTRUCTURE

This section discusses the required infrastructure to support the mining and processing operations and transportation for three (3) mining scenarios, Open-Pit, a combination of Open-Pit followed by an Underground and Underground Only, respectively known as Scenarios 1, 2 and 3. The following areas will be discussed:

- Existing infrastructure.
- Area 110 – Roads, Bridges and Fencing.
- Area 120 – Mine Infrastructure.
- Area 130/140 – Support Infrastructure / Camp Facility.
- Area 160 – Process Plant Infrastructure.
- Area 200 – Power & Electrical.
- Area 300 – Water Management.

Figure 18.1, Figure 18.2 and Figure 18.3 show the general layout for each scenario.

Figure 18.1: General Layout Scenario 1



Figure 18.2: General Layout Scenario 2



Figure 18.3: General Layout Scenario 3



18.1 Existing Infrastructure and Location

Due to the remote location of the Project, there is no infrastructure or public services of any sort in the vicinity of James Bay Project. Moosonee and Moose Factory are the closest communities located approximately 40 km northwest of James Bay Project.

The Project is located in northeastern Ontario, approximately 40 km south of the town of Moosonee. Access to the Property is by helicopter year-round. From the James Bay Project site, only winter bush roads are available. Since February 2013, the “Wetum” winter road, built, owned and maintained by Moose Cree First Nation connects Moose Factory to the Ontario highway network and provides approximately 3 months of access during the winter to the region. The total length of Wetum Road is 170 km.

A ferry operating from the end of April until mid-October connects Moosonee to Mouse Factory. A new ferry shall be commissioned in 2019. The new ferry will be able to carry two tractor-trailer combos or 8 passenger cars.

The Ontario Northland Company services Moosonee to Cochrane, Ontario daily. The railroad company provides yearlong service connecting Cochrane to Moosonee and back the same day. The Ontario Northland is mainly for passengers and light freight, however with an additional locomotive, one train can carry up to 1,400 t payload.

Hydro One Remote reaches Moosonee through the Ontario transmission network (115 kV). Hydro One is responsible for distributing the power in Moosonee and Moose Factory from a main substation (Renison Sub Station) approximately 10 km south of Moosonee. A transmission line leaves the same substation to service north of Moosonee region. Five Nations Energy Inc. (“FNEI”) is the company responsible for the operation of the transmission line. At this point of the Study, it is assumed that the existing substation is designed to provide power to the James Bay Niobium Project without any major modifications to the existing powerline.

Figure 18.4: HydroOne Renison Substation

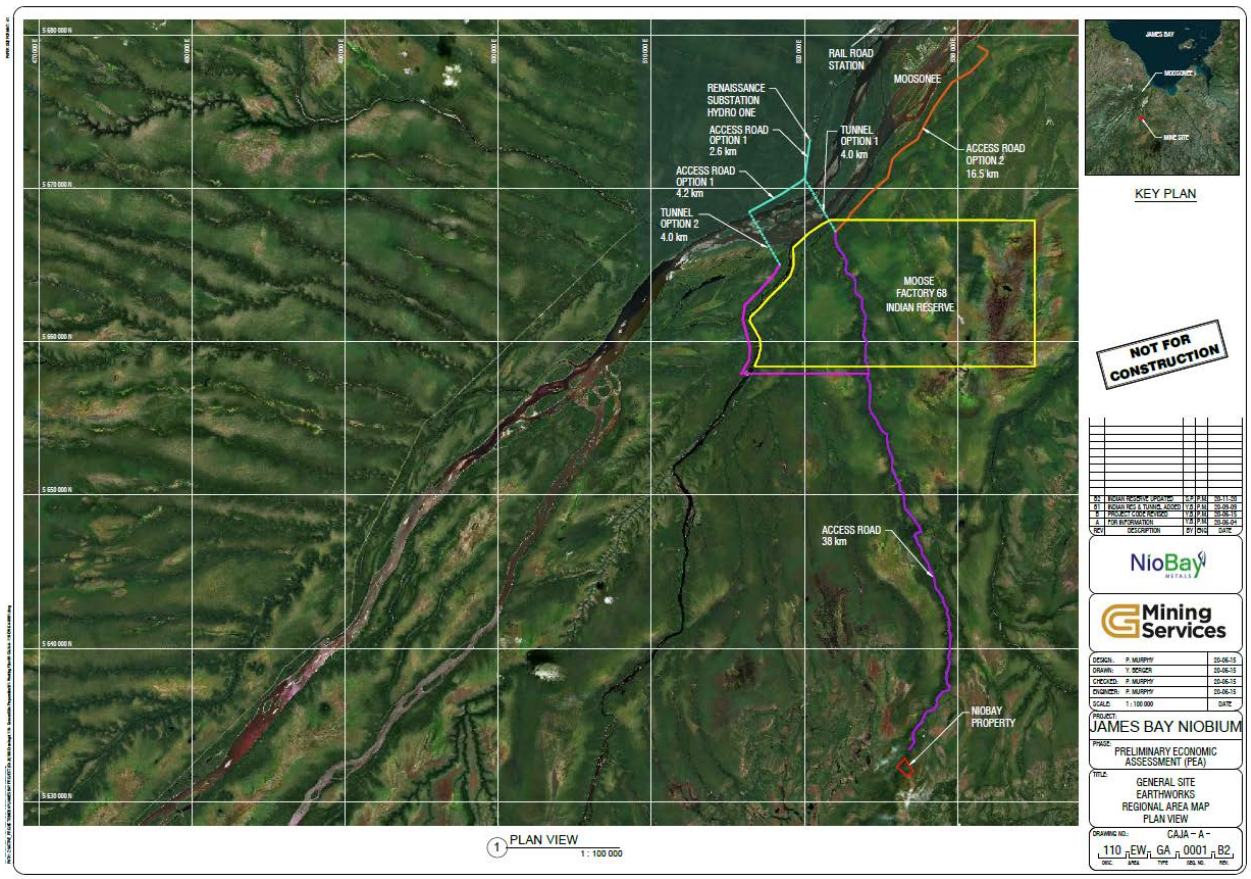


18.2 Area 100 Infrastructure

18.2.1 Area 110 Roads, Bridges & Fencing

The site is undeveloped. The plan for accessing the site over the life-of-mine ("LOM") is to build a permanent all-season road to reach Moosonee. There are three options developed, which are shown in Figure 18.5.

Figure 18.5: Options for Main Access Road



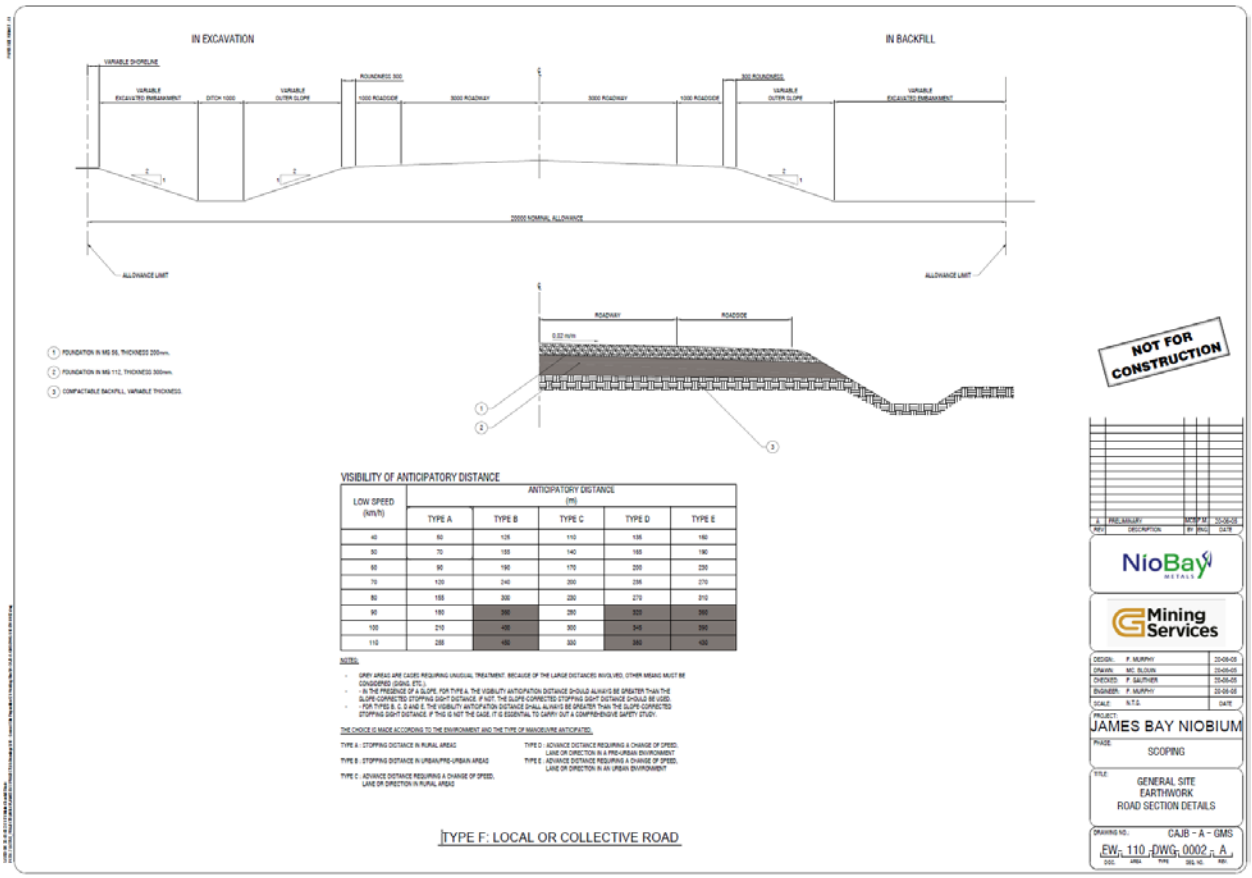
The preferred access road option is an alignment along the east bank of the South Bluff Creek, crossing Moose Factory 88 Reserve to Moose River. From the east, a tunnel of 6-meter-wide, 4.2 km long crossing Moose River, just south of Renison substation. The shotcrete and ground support tunnel is wide enough for one vehicle at a time to cross, has a specific protected cable bank for the 115 kV powerline and optic fiber. Ventilation and lighting are provided.

Two other options are considered. Option 2 would avoid crossing Moose Factory 88 Reserve. Instead, two bridges need to cross the South Bluff Creek and another one is required to cross the French River. From the east bank of Moose River, a 4 km tunnel reaches the west bank having the same preliminary design as the other option.

Finally, the third option is extending the road up to Moose Factory and allows Moose River crossing to be done in Summer by barge and in winter on iced bridge as Wetum winter road is doing now. This option is does not provide an all year long access to site. Therefore, this option could not be considered to fulfill Project requirements.

Site roads will interconnect the site infrastructure and will be usable and maintained year long. The site roads begin at the gatehouse. The site roads will provide access from the gatehouse, the camp/office building, maintenance shop, dry house and warehouse buildings, process plant, open-pit/underground mine access, ore stockpile area and ventilation raises when necessary. The total length of the upgraded trails is approximately 5 km. The newly constructed site roads will have the same cross-section with a 6 m wide aggregate driving surface placed directly on the subgrade along with ditches on both sides.

Figure 18.6: Site Roads Cross-Section



There is no plan to build an airstrip for the Project.

18.2.2 Area 120 – Mine Infrastructure

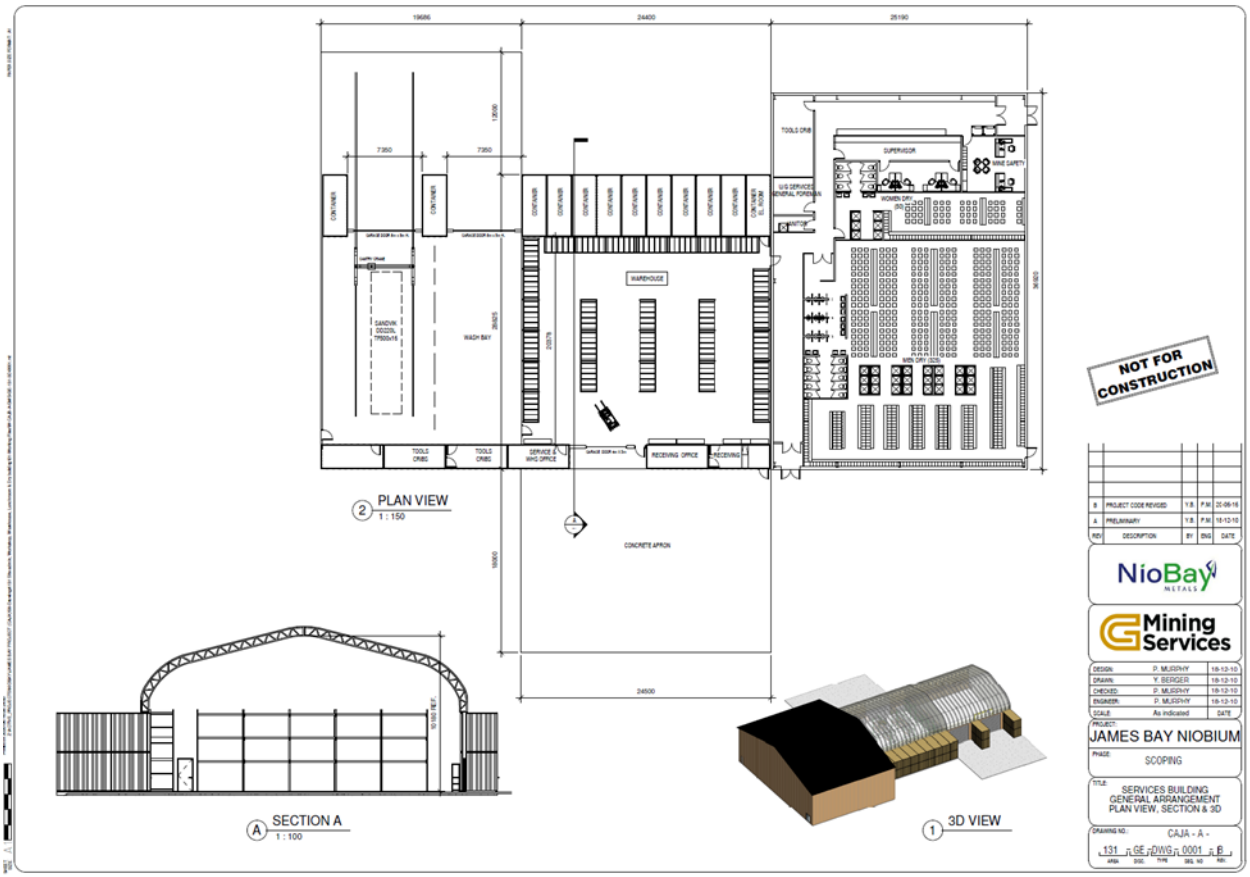
Figure 18.7 shows the main mining facility for Option 3 underground mining. For open-pit mining, Options 1 & 2, three (3) additional bays will be required with a larger width A 7.5 m wide garage door is required as opposed to a 6 m wide garage door. The warehouse and dry will be designed similarly.

In general, the truck shop and warehouse are connected. The truck shop and warehouse will share a single insulated tension fabric roof set on top of the containers but are divided with a fabric wall. The truck shop will be used primarily for heavy-duty vehicle maintenance. The truck shop will have two separate bays (4 for open pit operations) each equipped with a 6 m wide by 5 m high roll-up door. One bay will have its own railed gantry crane with a 15-t capacity which can be moved inside or outside of the building. One bay may be used for equipment washing purposes or may be used for welding and general maintenance. Some of the containers supporting the dome will be insulated and converted into offices. Water used to clean vehicles in the truck shop is considered contact water and will be collected and sent to the Water Treatment Plant ("WTP").

The warehouse will include racking to store spare parts and consumables. The warehouse's interior dimensions will be 20 m x 25 m. The warehouse and truck shop will both have concrete aprons to better handle heavy vehicle traffic.

The mine dry house will be adjacent to the truck shop and warehouse. It will serve as a locker room for the mine workers between shifts and contain mine rescue equipment, medical offices and a few offices for management personnel. The dry house has enough locker space for 375 workers. The men's section accommodates 225 workers and includes showers, toilets, urinals, lockers and baskets. The women's section accommodates 25 workers and includes showers, toilets, lockers, and baskets. The dry house consists of a pre-engineered steel-clad building.

Figure 18.7: Scenario 3 Mine Dry, Maintenance Shop and Warehouse



Explosives magazine and oil storage facilities are included in all the options. The diesel fuel storage facility will contain 1 M liters representing a reasonable assumption for consumption. This quantity represents the annual consumption of diesel on site.

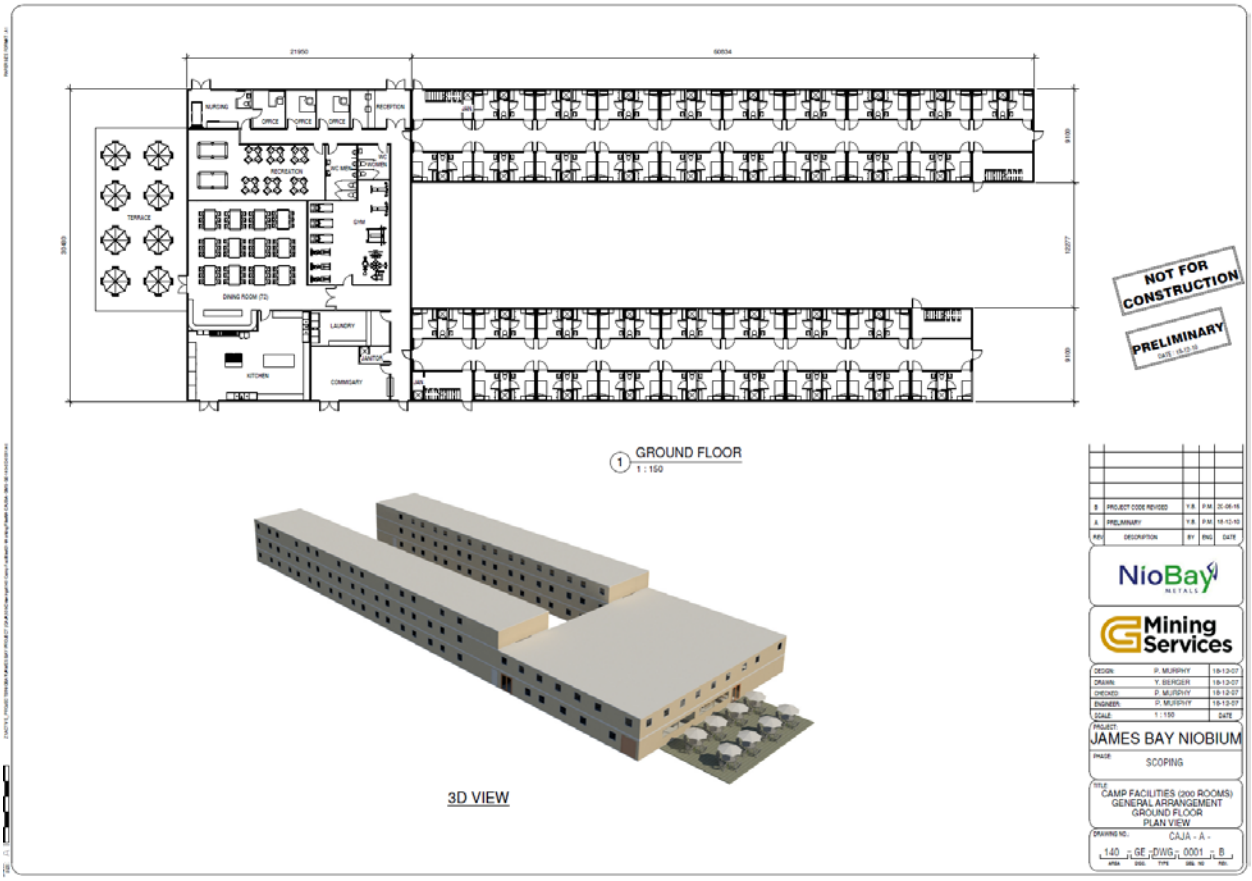
18.2.3 Area 130/140 – Support Infrastructure / Camp Facility

Most of the general services will be located in Moosonee. These services would include supply chain management, human resources, sales and shipping, accounting and other administrative functions.

All other services will remain on site, such as general services representatives, engineering, surface services and geology.

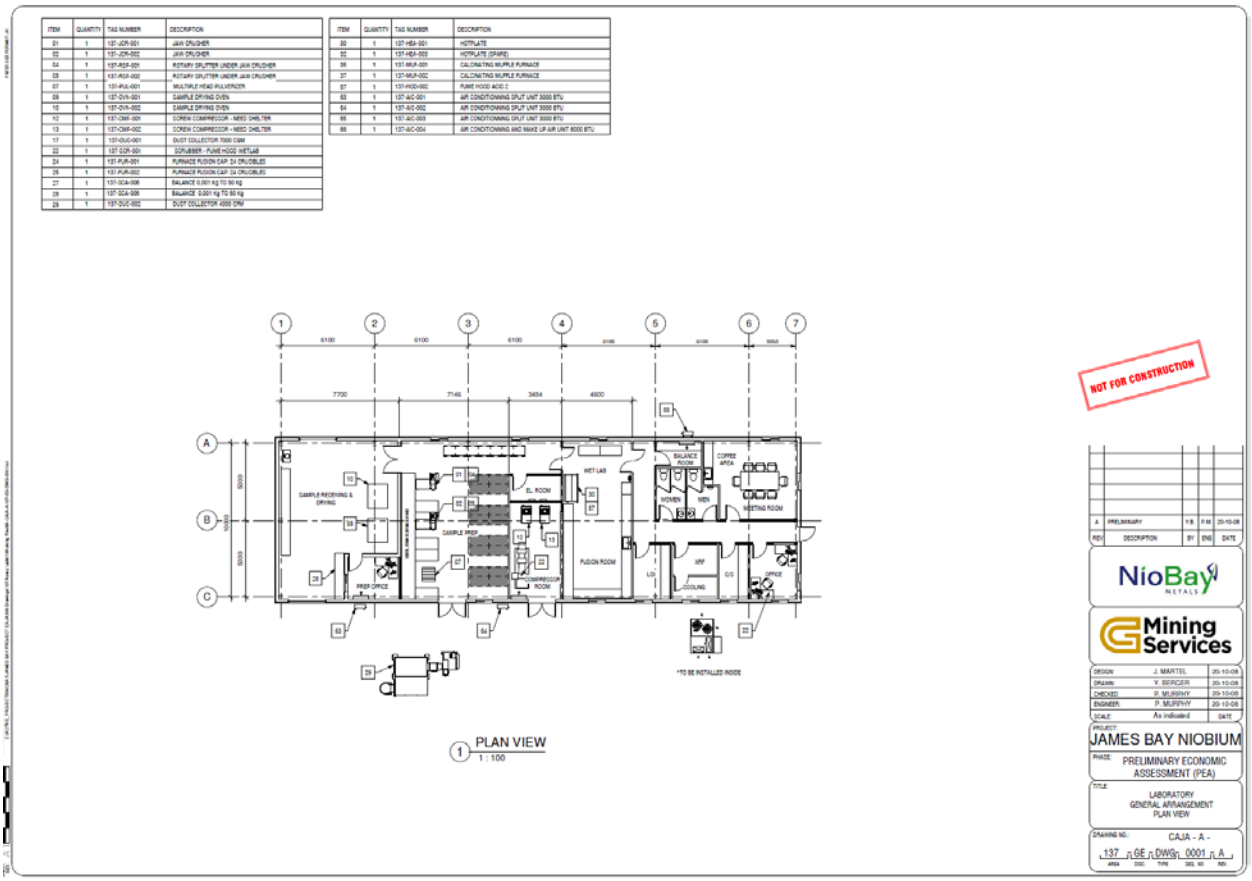
The administration is connected to the camp facility. The camp can accommodate 250 persons. The camp-related services such as camp reception, kitchen and dining room, fitness room, laundry, medical center and commissary will be located on the first floor while the above mentioned services are located on the second floor. Figure 18.8 shows a preliminary of the camp/administrative facility.

Figure 18.8: Camp/Administration Building, First Floor & 3D



The assay laboratory for this Study is located in Moosonee. However, the Project team developed a floor plan of the laboratory. Figure 18.8 shows the Assay Laboratory layout.

Figure 18.9: Assay Laboratory Layout



At the time of writing this Preliminary Economic Assessment, the site guardhouse is not located, but the related costs and works are planned. All traffic coming to or leaving the process area will pass through the gate house. The rooms in the gate house will include visitor registration, security office, induction room, vehicle control room, and bathrooms. The interior area of the gate house will be 150 m^2 with two covered 50 m^2 concrete aprons. Vehicle access will be controlled by a boom gate.

18.2.4 Area 160 – Process Plant Infrastructure

The process plant building is designed based on the process flow diagram; the major equipment sizes as provided by the Project metallurgist. The layout is based on G Mining Services Inc. (“GMS”) experience of similar arrangements. The layout is shown in Figure 18.10 where all process areas are identified and the 3D version is presented in Figure 18.11. The structural steel is a pre-engineered building 36 m wide by 114 m long.

Figure 18.10: Process Building – Plan View

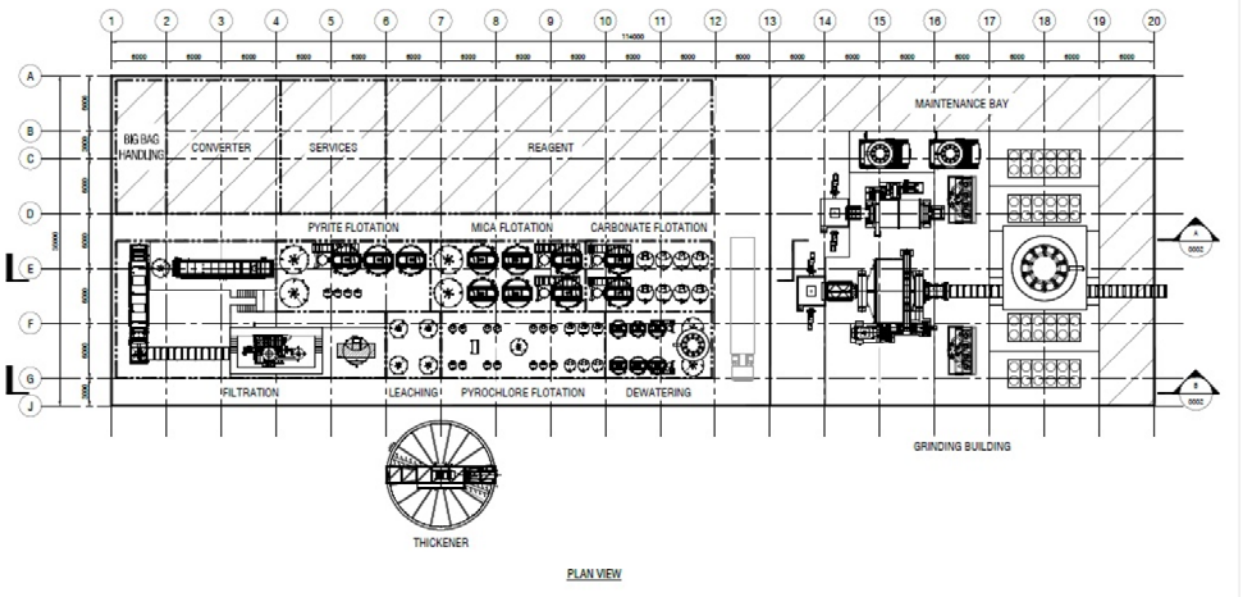
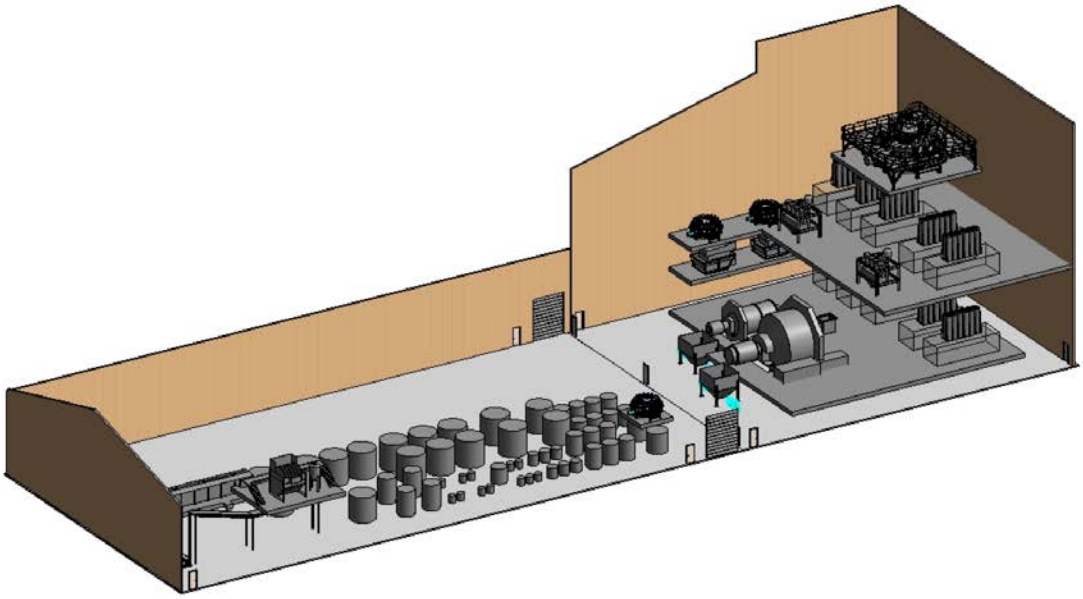


Figure 18.11: Process Plant 3D



18.3 Area 200 – Power and Electrical

Five Nations Energy Inc. (“FNEI”) retains the rights north of HydroOne Renison substation for power transmission/distribution. After discussions with FNEI, it is believed that the James Bay Project can obtain power from either FNEI directly or HydroOne through FNEI at the Renison Substation.

The plan considered for this Study is to obtain the power supply from the Ontario grid network through FNEI. This substation is approximately 45 km west of the James Bay site. The Moose River crossing could use the planned tunnel or an aerial powerline. The powerline voltage is 115 kV. From the east bank of Moose River the power line will follow the main access road regardless of the road route chosen. The preferred route for this Study remains along the South Bluff Creek access option.

The power draw at site is estimated to 8 MW for the planned infrastructure.

The power distribution on site would be at 5 kV with the main substation located as close as possible to the process plant and connects all the project components including: the process plant, the underground mine, camp and offices as well as the ancillary buildings.

The mine's communication network will be connected to the national network through optic fiber spread along the powerline up to Moosonee. Very little information has been gathered at this time. Radio systems will be implemented for the site and underground mine.

18.4 Area 300 Water Management

18.4.1 Effluent Treatment Plant

The water balance approach for the Project will be to reclaim and re-use as much water as possible during operations. The general operating strategy assumes that the makeup water source will be used to supply water to the Tailings Storage Facility ("TSF") prior to the start of operations so that there is enough water available for the mill. At this time the water supply is assumed from wells drilled on the property and will require further investigations.

The total volume of water is obtained from direct precipitation, run-off from the TSF liner area, water pumped from the event pond at the mill site, and makeup water from a well field. After the start of mill operations, the makeup water source will continue to be used as needed.

The inputs required to establish the water balance model include the following:

- Production schedule.
- Climate data.
- Underground mine flow.
- TSF operation approach.

- ETP start date and capacity.
- Potable water treatment.
- Event pond and contact area runoff to the TSF.

The Effluent Treatment Plant ("ETP") design requires influent water quality by source and for the blended water. The TSF contaminant concentration can be managed at the source before directing water to the TSF.

Final water discharge will be identified in subsequent studies and will need to comply with all provincial and national regulations. For costing purposes, the final discharge point is assumed to be located at less than 1 km of the effluent treatment plant and the design is based on GMS previous experience.

18.4.2 Potable and Sewage Water

The potable and sewage treatment plants are pre-containerized. The capacity of the plant will be designed to treat used water from the 250 workers camp, dry house and offices for operations and up to 500 workers for construction. For cost purposes, the discharge is planned to be in a creek located less than 1 km from the plant.

18.4.3 Tailings Storage Facility

The TSF concept accounts for the subsurface conditions, the anticipated embankment fill materials, the water and tailings storage requirements, and the physical characteristics of the tailings. The principles for optimizing the TSF design include:

- Balanced cut and fill within the footprint of the TSF.
- Construction of the facility in stages depending on mine waste material from open-pit and quarry operations for underground.
- Storage capacity for the planned mine production and estimated TSF water balance for each scenario.

The TSF capacities are 70.8 Mt, 26.8 Mt and 31.4 Mt for Scenarios 1, 2 and 3, respectively.

18.5 Underground Mining Infrastructure

The underground mine access ramp is located approximately 500 m north-west of the mill area and provides access to the mine dry, maintenance shop and warehouse. The ore and waste stockpiles are

located along the surface access road to the mine on each side. It is assumed that the mine portal design will consist of a 250 m long ramp at -15% grade that connects to the main decline. It is excavated at a minimum of 15 m into the fresh rock, where tunnel multi-plate liners are built and backfilled. The ramp uses 6 m diameter fully rounded steel tunnels. The culvert of the steel tunnel is backfilled to create a driving surface for the mine equipment.

The ore stockpile pad is located close to the crushing building approximately 200 m east of the ramp. The ore stockpile is designed with a capacity of 80,000 t at a maximum height of 15 m. Over the pre-production period, the ore will be hauled with mining trucks to the stockpile pad. The maximum ore stockpile is at the end of the process plant commissioning period.

The pad is approximately 20,000 m^2 in area and will consist of at least 300 mm of low permeability fill placed on top of the existing ground. The fill is consisting of a low permeability backfill or if no availability of such material a geomembrane will be used. For the purpose of this Study the low permeability option is chosen. Water that contacts ore on the pad is considered contact water and must be directed to the TSF. The stockpile has a cross-slope that directs all runoff water into low permeability material ditches. The water will eventually drain to a collection point where it will be pumped to the TSF or the Water Treatment Plant ("WTF") later in the LOM.

The waste dump stockpile is designed similarly to the ore stockpile. The plan over the LOM is to recycle the waste and keep it underground in such a way that the maximum capacity of 135,000 t will not be reached. Otherwise at the beginning of the mining period, all the waste will be used to build platforms, pads and roads.

The design principles of the stockpiles pads for protecting the native soil and the control of contact water run-off is replicated for Scenarios 1 & 2.

19. MARKET STUDIES

19.1 Uses of Niobium

Niobium is used in a variety of forms but the most important in terms of tonnage is standard-grade ferro-niobium, which has applications in steelmaking, most notably in high-strength low-alloy ("HSLA") and stainless steels. This market accounts for approximately 90% of niobium usage. Niobium is added in very small amounts (usually in the order of 0.02%) as a grain refiner to produce high-quality steels for use in gas pipelines, automobiles, construction steels, stainless steels, and other applications. Only 8% of overall steels production contains niobium. This percentage will likely raise overtime as seen in China and other developing countries.

19.2 Niobium Supply and Demand

Niobium is a niche metal present in a variety of minerals, the most commercially important of which is pyrochlore which accounts for as much as 97% of global niobium supply. There are currently only three significant producers of pyrochlore: CBMM and China Molybdenum ("CMOC") in Brazil; and Niobec in Canada. All three convert their mine output to ferroniobium prior to sale, mostly into export markets and on a yearly contract basis. CBMM, a privately-owned company, is by far the largest producer of ferro-niobium in the world and, unlike CMOC and Niobec, CBMM also supplies a range of other downstream products like niobium oxide and niobium metals. Its share of the global niobium products market has peaked at +80%, and its market share will continue to grow considering demand is expanding while the other two producers are not increasing their capacity.

Two factors are essentially impacting the ferroniobium demands: i) increase in overall steel production with an associated increase in production of high-value steels, and ii) a rise in the intensity of use of niobium in the steel industry. Traditionally steel production growth has always been the greatest contributor to niobium consumption growth, while with moderate steel production forecasts, the greatest potential for niobium consumption growth lies in an increase of intensity of use, which is currently approximately 52 grams per tonne of steel. China, which represents half of the world steel output, is a key region for future ferroniobium consumption. Chinese GDP growth is forecasted to slow down in the upcoming years, which could negatively affect the rate of increase for steel consumption however the country is increasing the percentage of high-value steels which will increase ferroniobium consumption. In addition, the importance of construction and pipeline steels produced in China combined with new regulations in construction material quality, as rebar, could push Chinese niobium requirements to new heights.

In terms of ferroniobium supply, Table 19.1 provides the reported annual production capacity of ferroniobium from the three largest mine operations along with the Project's estimates. Note the capacity/production are reported in niobium contained rather than ferroniobium (66% of niobium in ferroniobium).

Table 19.1: Annual Production Capacity

Asset	Ownership	Location	Resources	Annual Capacity (t Nb)
Araxa	CBMM	Brazil	+400 Mt @ 2.5% Nb_2O_5 / + 200 years	100,000 (2021)
Catalao	CMOC	Brazil	25 Mt 0.93% Nb_2O_5 / 15 years	9,000
Niobec	Magris Resources	Canada	75 Mt @ 0.55 % Nb_2O_5 / + 20 years	7,000
James Bay Niobium	NioBay Metals	Canada	63.5 Mt @ 0.53% Nb_2O_5 / 23 to 30 years	6,250

19.3 Marketing of Niobium

James Bay Niobium production will be sold to markets around the world either directly or via agents/distributors. Sales in China are typically via a third party while sales in America or Europe are done directly with the consumers, each market has its particularity. Marketing fees are estimated at 1.5% of sales and includes transportation, warehousing, commissions, insurance, and related in-house personals.

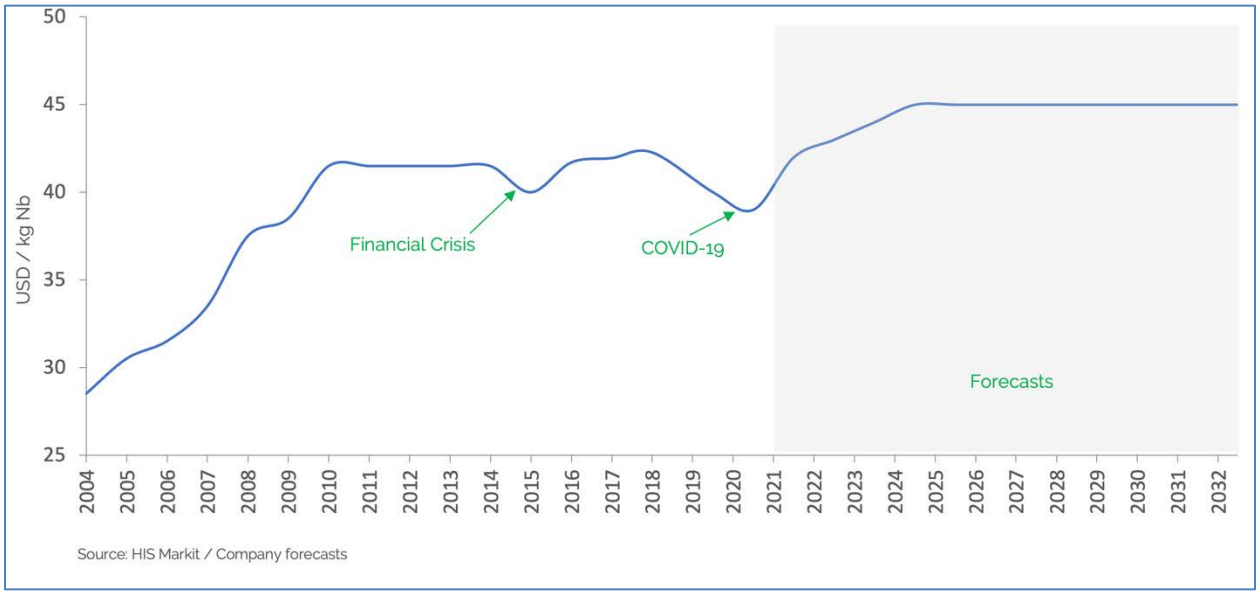
Ferroniobium contracts a typically valid for a year or two with end-users while distributors. The company may sign off-take agreements with third party.

19.4 Metal Prices and Market Outlook

Between 2006 and 2008, CBMM doubled the price of ferroniobium. At first, the movement appeared to be a spike, but it later became clear that it was a permanent increase to address what was a structural under-valuing of niobium. Up to that point, the strong growth in demand for ferroniobium had not been reflected in its price. Relative stability returned at the higher price level and there have been no dramatic price increases since 2008. Overall, the price of ferroniobium is fairly demand-inelastic. The global economic downturn in 2009 caused demand for ferroniobium to fall sharply. That had minimal effect on prices, however. Similarly, the upturn in demand for ferroniobium in 2014 coincided with what seemed to have been a general fall in prices.

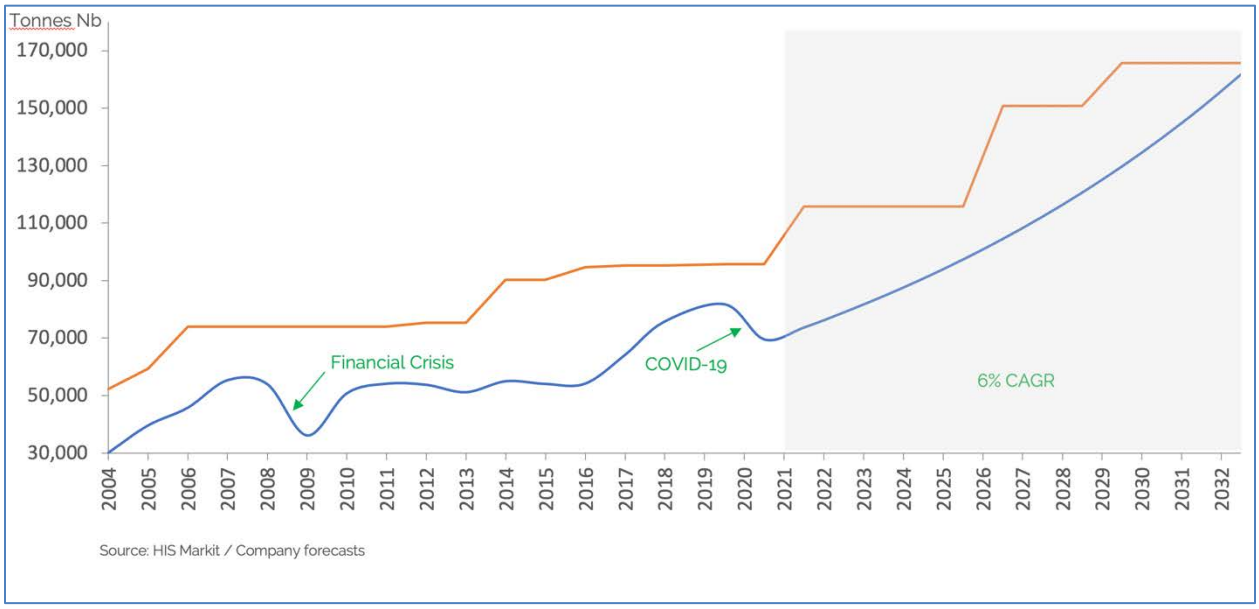
That was not strictly the case. International trade in ferroniobium is reported in USD. It is actually purchased in a variety of currencies: Euros in Europe, the CIS and the Middle East; Yen in Japan; Renminbi in China; and USD elsewhere. The strengthening of the USD, notably against the Euro, has thus resulted in an apparent fall in unit values when converted to Euros. In reality, the average unit value of exports from Brazil and Canada has moved little. The strong USD has contributed to an absence of price increases in 2015 (not possible to increase prices under current conditions). Another influence has been the appreciation of the USD against the CAD and Real (production costs are incurred mostly in local currency).

Figure 19.1: Historical and Forecasts Price of Ferroniobium



Price forecasts are based on the outlook for the world economy, steel production trends (higher demand for high-value steel), exchange rates, intensity, investment by CBMM in expanding capacity. Considering the aforementioned, a ferroniobium price of USD 45/kg Nb is forecasted once the Project will begin production. In addition, the ferroniobium price forecasts were also the same, as projected by Roskill Consulting Group Limited, in the most recent feasibility study on the Elk Creek Niobium Project.

Figure 19.2: Ferroniobium Consumption and Forecasts



Ferroniobium consumption grew 20% in 2018, 15% in 2019, and for 2020 demand should fall by 15-20% due the COVID pandemic. Since the beginning of 2000, ferroniobium consumption grew on average by 6%. Market is expected to maintain the same trend and should grow by 5% to 7% CAGR due to the continuous progress in micro alloyed steels and the evolution of ferroniobium addition in all steel market segments in which ferroniobium is used.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

There have been no specific environmental studies to date. Some water and grab samples associated with exploration drilling have been undertaken to provide background data for Moose Cree First Nation. It is anticipated that baseline environmental data will be collected as from Q2-2021 and beyond.

20.2 Project Permitting Requirements

It is anticipated that both the federal Impact Assessment Agency and the provincial Ministry of the Environment, Conservation and Parks will be the lead at the environmental assessment phase. Significant permits will also be required to operate. The majority of permits to operate will be under provincial jurisdiction.

20.2.1 Consultation Requirements

The federal and provincial Crown has a constitutional duty to consult, and where appropriate, accommodate Indigenous groups when it considers resource development projects that might adversely impact potential or established Indigenous or Treaty rights. While third parties such as project proponents do not have a legal obligation to consult Indigenous communities in practice, the Crown may delegate some responsibility for consultation to third parties to collect information about how the impacts of project development may affect potential or established Indigenous or Treaty rights.

20.2.2 Provincial Requirements

At the provincial level, the Ministry of Energy Northern Development and Mines ("MENDM") has established requirements to ensure that project proponents consider Indigenous consultation specifically for exploration and mine closure. As a project develops other ministries such as Environment Conservation and Parks, along with Natural Resources and Forestry will have to consult as part of the *Environmental Protection Act*. At the exploration stage, MENDM has identified Moose Cree First Nation ("MCFN") as the Indigenous community that must be consulted. Niobay believes that at a minimum, Taykwa Tagamou and the Grand Council of the Cree will also have to be consulted. As part of the consultation process, Niobay is required to:

- Provide further details to the communities about Niobay's proposed exploration activities.

- Gather information from the communities about whether or how the proposed activities have the potential to adversely affect the communities' Aboriginal or treaty rights.
- Discuss with the communities, and the provincial ministries as appropriate, ways to avoid, eliminate, or minimize any concerns raised.
- Seek further direction or advice from the provincial ministries, if needed.
- Document the process and decisions made and report to MENDM.

If the Project progresses to the advanced exploration and production stages of the its lifecycle, Niobay will be required to participate in a provincial Environmental Assessment. This may include an Individual Environmental Assessment for the mine, road and transmission line. Or, there may be Class Environmental Assessments for each of the components. In either case, consultation will be required. Lastly, consultation will be required prior to submitting the mine Closure Plan. The Closure Plan and financial assurance must be finalized prior to construction. There will also be a variety of provincial permits required:

- Permits to take water (camp wells, underground or open pit dewatering).
- Environmental Compliance Approvals (air and noise).
- Land and Rivers Improvement Act (discharge).
- Forestry Permits.
- Potentially Species at Risk Permit (caribou).

At this point, Niobay has concluded a Protection Agreement ("PA") with MCFN to guide the relationship and allow the exploration program to proceed. The PA also details how further exploration, including advanced exploration would proceed. Niobay has hosted community open houses and regular meetings with the Chief and Council to familiarize them with the mining cycle, niobium as a commodity, and the proposed environmental approach.

20.2.3 Federal Requirements

At the federal level, the *Impact Assessment Act* maybe triggered when the Project Description submitted to Impact Assessment Agency ("IAA") includes one or more physical activities defined in the *Inclusion List Regulations or Regulations Designating Physical Activities*. The federal IAA process requires proponents to undertake consultation with Indigenous communities, the public, and other stakeholders to promote meaningful public participation before and during the environmental assessment process. Like the provincial process, the IAA requires project proponents to collect and/or facilitate the collection of traditional, archeological and community knowledge (mainly related to land use and ecological values) relative to the

proposed project area. Indigenous communities that are included within the scope of the federal environmental assessment are determined by project-specific IAA guidelines.

20.3 Social or Community Impact

The future environmental baseline and engagement with the Indigenous and local communities will speak to the social and community impact. Generally, the impact will be deemed to be positive as there will be many businesses, employment and contracting opportunities the Project will bring to the region.

20.3.1 Plans, Negotiations and Agreements

Above and beyond its legal obligations, Niobay is committed to building lasting relationships with Project partners and stakeholders based on the principles of transparency, trust, and mutual respect to ensure the maximum possible local benefit from the Project. Niobay recognizes that the Project is situated in the Traditional Homelands of the MCFN, with an overlap of territory with Taykwa Tagamou and in an area that has asserted aboriginal and treaty rights by the Grand Council of the Quebec Cree and is committed to fully respect traditional land, culture, environment, and water. As the Project proceeds, engagement with impacted communities will increase.

Niobay has held information session(s) in Moosonee and Moose Factory. Two (2) recent information sessions were scheduled; however, one was canceled due to the death of an Elder. An information session was held on March 5, 2020 after the completion of the drill program. Niobay wishes to emphasize that the Project would likely be a mine located outside the North French River watershed and that it will have a minimal impact on traditional harvesting activities and to the environment. The Project would create important economic benefits to potentially impacted communities in the short, medium, and long term. Due to the proximity of the Project to Moose Factory and Moosonee, daily transport services would allow all local workers to return to their homes and families at the end of the workday.

Niobay is interested in entering into a partnership agreement with demonstrably impacted Indigenous communities with emphasis on MCFN. As part of such a partnership agreement, some of the aspects of the Project that Niobay would like to discuss with the impacted communities will include:

- Project monitoring and mitigation.
- Bird and wildlife conservation.
- Support for the community, including traditional concerns.
- Employment.

- Contracting and procurement opportunities.
- Apprenticeship, training, and career development opportunities.
- Any other issues of importance to MCFN, as identified during consultation and negotiation.

21. CAPITAL AND OPERATING COSTS

21.1 Capital Expenditures

The CAPEX estimate is summarized in Table 21.1. Work Breakdown Structure ("WBS") Areas 100 to 600 include the Project's direct costs, while WBS Areas 700 to 900 cover indirect costs, Owner's costs and mine-preproduction. The CAPEX for Project construction, equipment purchases, and pre-production activities for each scenario is estimated to be respectively CAD 510M, CAD 482M and CAD 578M. The CAPEX includes a contingency of 25% of the total before contingency.

Table 21.1: Initial Capital Expenditures Summary Scenario 1

Capital Expenditures	Scenario 1 CAD (000)	Scenario 2 CAD (000)	Scenario 3 CAD (000)
100 – Infrastructure	133,575	133,575	112,615
200 - Power and Electrical	31,485	31,485	31,485
300 – Water & tailings	31,413	13,575	20,482
400 – Surface Mobile Equipment	5,613	5,614	5,612
500 – Mining Mobile Equipment	29,405	29,405	-*
600 - Process Plant	69,985	69,985	99,985
700 - Construction Indirects	35,018	32,699	34,772
800 - General Services	40,406	37,730	40,122
900 - Pre-production, Start-up, Commissioning	31,462	31,488	117,729
990 – Contingency	102,090	96,389	115,738
Total	**510,452**	**481,945**	**578,691**

**Note: Mining Mobile equipment for Scenario 3 has been included in the Mining pre-production costs.*

The paste backfill plant for Scenario 3 was added in the initial capital, while the U/G mining equipment was included within the pre-production costs.

21.2 Basis of Estimate

21.2.1 Mining

The CAPEX estimate is aligned with an owner managed project delivery model.

A budgetary price RFP process was undertaken for most of the mine equipment fleet (in both the open pit and underground scenarios), and a preliminary technical and commercial evaluation process has been completed. The pricing is the basis for the CAPEX estimate.

All the mining equipment purchase costs are captured in WBS Area 500.The equipment pricing includes the base machine with several required options, tires, and fire suppression systems in most cases. It also includes additional costs for freight to the Project, assembly, and commissioning.

All mining pre-production costs are captured in WBS Area 900. The costs are estimated in a similar manner to the operating costs and correspond to those costs incurred during the pre-production period until commercial production is achieved. The direct costs include all operating costs for equipment such as fuel, maintenance parts, operators, and consumables (tires, explosives, etc.). The labour costs for mine supervision, management, and technical support are also captured in the same WBS area.

The pre-production costs are estimated according to the pre-production schedule which gradually ramps up to full production rate by the end of the pre-production period.

The following items were assumed:

- All equipment and materials will be new.
- The labour rate build-up will be based on the statutory laws governing benefits to workers.
- No cost escalation (or de-escalation) is assumed.
- No costs relating to certain agreements with third parties.
- No cost relating to financing and interest.
- No cost for pre start-up operations and maintenance training.
- Transport and handling of niobium from the converter by train from Moosonee to Cochrane and by road to an export destination.
- Power Cost: CAD 0.08/kWh.

- Fuel Cost: CAD 0.75/l.
- Exchange rate: CAD 1.30/USD.
- Average daily LOM process plant throughput is 6,600 t/d.
- The mine ground condition is assumed to be excellent.

21.2.2 Processing Plant and Infrastructure

The physical conceptual design is prepared in accordance with the WBS where all the tasks and areas were developed in enough details to establish an estimate of +/-35%, general contingency of 25% in accordance with Pincock Perspectives for Preliminary Economic Assessments.

Table 21.2 presents WBS Level 1.

Table 21.2: WBS Level 1

Document No.: CAJB-A-001-PM-WBS-0001

WBS L2	WBS L3	Item
	001	All site General
100	100	INFRASTRUCTURE
200		POWER & ELECTRICAL
300		WATER
400		SURFACE OPERATIONS
500		MINING (Open Pit and/or U/G)
600		PROCESS PLANT
700		CONSTRUCTION INDIRECT
800		General Services - Owner's Costs
900		Pre-Prod, Start-up, Commissioning
910		Mining Pre-Prod
950		Process Plant Pre-Prod
960		First Fill, Spares & Consumables
990		Contingency

The process plant equipment costs are defined using high level budgetary quotations for major processing equipment. Allowances are provided for all the disciplines where possible. A plant general layout draft was designed according with the process flowsheet diagrams. The layout is shown in Figure 18.10. The plant

building estimate is based on previous projects having the same footprint and similar technologies for processing. The converter estimate is based on historical data.

The material take-offs for earthworks, included the tailings dams design and sequences for each scenario are based on physical material take-offs from preliminary design estimates for their respective area. The material planned to build the surface infrastructure for Scenarios 1 & 2 will come from the pre-stripping excavation assuming it will be appropriate for construction. The assumption for Scenario 3 is to develop several quarries within 20 km of the vicinity of the Project. Unit costs are estimated using Quebec / Ontario equipment / labour hourly costs as defined in Quebec and Ontario from different recent projects, collective agreements and historical data.

Estimates for G Mining Services Inc. ("GMS") experience and data room from similar projects were completed for most of the temporary and permanent effluent treatment plants, the sewage treatment plant and the mine mobile equipment fleet, other ancillary buildings and temporary / construction infrastructure. The remaining equipment and material costs were based on budgetary bid processes, quotes, consultant's historical data and in-house databases, or benchmarked from a previous project.

The tunnel under Moose River estimate is based on underground productivity, per meter costs and design. The tunnel can be assimilated to a 6 m x 6 m underground mine development drift. Services / ventilation have not been developed; however, allowances are shown by increasing the per meter costs.

The power supply costs are based on historical data for a 115 kV powerline as the alignment is along the future main access road.

21.3 Direct Costs

21.3.1 Mining

The CAPEX estimate for WBS Area 500 - Mobile Equipment for all three scenarios is summarized in Table 21.3.

Table 21.3: Mobile Equipment Capital Expenditures

Work Breakdown Structure	Scenario 1 Total (CAD M)	Scenario 2 Total (CAD M	Scenario 3* Total (CAD M
500 - Mobile Equipment			
550 - Mine Equipment	29.4	29.4	-*
Grand Total	**29.4**	**29.4**	

**The mobile mining equipment capital expenditure for Scenario 3, which represents CAD 14.33M, has been included in the mine pre-prod initial CAPEX*

Mobile mining equipment capital expenditures are detailed in Table 21.4.

Table 21.4: Mobile Equipment Capital Expenditure by Fleet Type

Equipment Capital Schedule	Scenario 1		Scenario 2		Scenario 3	
	Pre-Prod CAPEX	Sustaining CAPEX	Pre-Prod CAPEX	Sustaining CAPEX	Pre-Prod CAPEX	Sustaining CAPEX
Open Pit						
Major Equipment	21,225	47,214	21,225	-	-	-
Support Equipment	8,180	19,402	8,180	1,021	-	-
Underground						
Major Equipment	-	-	-	47,789	10,117	54,014
Support Equipment	-	-	-	57,374	4,214	73,238
Total Equipment CAPEX	**29,405**	**66,616**	**29,405**	**105,163**	**14,331**	**127,252**

21.3.2 Process Plant and Infrastructure

21.3.2.1 Process Plant Initial Capital Expenditures

GMS has been overseeing the engineering with contributions from Pierre Pelletier, ing., Metallurgist Consultant from Montreal. Mr. Pelletier, ing, managed the metallurgical test works, developed the process flow diagram ("PFD") and established the major equipment sizes. The initial capital costs for the process plant was based on the PFD. The PFD does not change with the scenarios. The throughput tonnage remains regardless of the scenarios. The major equipment costs are based on quotations from Metso-Outotec.

21.3.2.2 Scenario 1

21.3.2.2.1 Infrastructure

The infrastructure costs are presented as a major item, WBS Area 110 - Roads. These costs represent the access road from Moosonee to the mine site, which includes the tunnel under Moose River.

Table 21.5: Infrastructure Capital Expenditures

Work Breakdown Structure	Total (CAD M)
100 - Infrastructure	
110 - Roads	75.59
120 – Process Plant Building, Workshops / Storage	48.55
130 - Support Buildings	4.88
140 – Camp Facilities	2.32
160 - Laboratories	1.98
170 - Fuel Systems	0.19
190 - Other Facilities	0.05
Grand Total	**133.56**

21.3.2.2.2 Power and Electrical

The CAPEX estimate for WBS Area 200 - Power Supply and Communications is summarized in Table 21.6. The power plant presented in WBA Area 210, includes all material, equipment and construction costs related to the power plant.

Table 21.6: Power Supply and Communications Capital Expenditures

Work Breakdown Structure	Total (CAD M)
200 – Power & Electrical	
210 – Power Supply	22.80
220 – Process Plant Electrical	3.06
230 – Temporary / Back-up Power Generation	0.50
240 – Power Distribution	0.13
260 – IT and Communications	5.00
Grand Total	**31.49**

The CAPEX estimate for WBS Area 300 - Water and Tailings Storage Facility ("TSF") is presented in Table 21.7

. The TSF estimate includes the scope that will be completed during the pre-production phase. Mine pre-production will be responsible for the transport and placement of mine rock fill for the TSF main dams and TSF access road.

Potable water supply is from wells. Effluent and surface water management consists primarily of the mine water effluent treatment plant, collection ditches and ponds.

Table 21.7: Water and Tailings Management Capital Expenditures

Work Breakdown Structure	Total (CAD M)
300 - Water & Tailings Management	
310 – Raw & Potable Water	0.29
320 - Reclaim Water	0.98
340 - Tailings Storage Facility (TSF)	26.70
350 – Mine Water Management	0.08
360 - Effluent Water Management	2.90
370 - Fire Water	0.21
380 - Sewage	0.25
Grand Total	**31.41**

The CAPEX estimate for WBS Area 400 - Mobile Equipment is summarized in Table 21.8 . A quotation price RFP process was completed for the mine mobile equipment fleet. The equipment pricing includes tires, fire suppression, transport to the Project site, assembly and commissioning.

Table 21.8: Mobile Equipment Capital Expenditures

Work Breakdown Structure	Total (CAD M)
400 - Mobile Equipment	
410 - Mine Equipment	29.41
430 - Plant and Surface Mobile Equipment	5.61
Grand Total	**35.02**

21.3.2.3 Scenario 2

21.3.2.3.1 Infrastructure

The infrastructure costs remain from Scenario 1 to Scenario 2.

Table 21.9: Infrastructure Capital Expenditures

Work Breakdown Structure	Total (CAD M)
100 - Infrastructure	
110 - Roads	75.59
120 – Process Plant Building, workshops / Storage	48.55
130 - Support Buildings	4.88
140 – Camp Facilities	2.32
160 - Laboratories	1.98
170 - Fuel Systems	0.19
190 - Other Facilities	0.05
Grand Total	**133.56**

21.3.2.3.2 Power and Electrical

The CAPEX estimate for WBS Area 200 - Power Supply and Communications remains from Scenario 1 to Scenario 2.

Table 21.10: Power Supply and Communications Capital Expenditures

Work Breakdown Structure	Total (CAD M)
200 – Power & Electrical	
210 – Power Supply	22.80
220 – Process Plant Electrical	3.06
230 – Temporary / Back-up Power Generation	0.50
240 – Power Distribution	0.13
260 – IT and Communications	5.00
Grand Total	**31.49**

21.3.2.3.3 Water and Tailings Storage Facility

The CAPEX estimate for WBS Area 300 - Water and Tailings Storage Facility is presented in Table 21.11Table 21.11: Water and Tailings Management Capital Expenditures

. The TSF estimate is reduced mainly due to the change in sequence and mining waste availability.

The potable water supply is provided from the wells. Effluent and surface water management primarily consists of the mine water effluent treatment plant, collection ditches and ponds.

Table 21.11: Water and Tailings Management Capital Expenditures

Work Breakdown Structure	Total (CAD M)
300 - Water & Tailings Management	
310 – Raw & Potable Water	0.29
320 - Reclaim Water	0.98
340 - Tailings Storage Facility (TSF)	8.87
350 – Mine Water Management	0.08
360 - Effluent Water Management	2.90
370 - Fire Water	0.21
380 - Sewage	0.25
Grand Total	**13.58**

21.3.2.3.4 Mobile Equipment

The CAPEX estimate for WBS Area 400 - Mobile Equipment is summarized in Table 21.12. A quotation price RFP process was completed for the mine mobile equipment fleet. The equipment pricing includes tires, fire suppression, transport to the Project site, assembly and commissioning.

Table 21.12: Mobile Equipment Capital Expenditures

Work Breakdown Structure	Total (CAD M)
400 - Mobile Equipment	
410 - Mine Equipment	29.41
430 - Plant and Surface Mobile Equipment	5.61
Grand Total	**35.02**

21.3.2.4 Scenario 3

21.3.2.4.1 Infrastructure

The difference between the infrastructure costs in Scenario 3 and the other scenarios relies on the size of the mining fleet maintenance shop.

Table 21.13: Infrastructure Capital Expenditures

Work Breakdown Structure	Total (CAD M)
100 - Infrastructure	
110 - Roads	75.59
120 – Process Plant Building, Workshops / Storage	27.59
130 - Support Buildings	4.88
140 – Camp Facilities	2.33
160 - Laboratories	1.98
170 - Fuel Systems	0.19
190 - Other Facilities	0.05
Grand Total	**112.61**

21.3.2.4.2 Power and Electrical

The CAPEX estimate for WBS Area 200 - Power Supply and Communications remains from Scenario 1 to Scenario 2.

Table 21.14: Power Supply and Communications Capital Expenditures

Work Breakdown Structure	Total (CAD M)
200 – Power & Electrical	
210 – Power Supply	22.80
220 – Process Plant Electrical	3.06
230 – Temporary / Back-up Power Generation	0.50
240 – Power Distribution	0.13
260 – IT and Communications	5.00
Grand Total	**31.49**

21.3.2.4.3 Water and Tailings Storage Facility

The CAPEX estimate for WBS |Area 300 - Water and Tailings Storage Facility is presented in Table 21.15. The TSF estimate changes as the sequence and size of the TSF changes. It should be mentioned that the dams are planned to be built with quarry material.

Potable water supply is from the wells. Effluent and surface water management consists primarily of the mine water effluent treatment plant, collection ditches and ponds.

Table 21.15: Water and Tailings Management Capital Expenditures

Work Breakdown Structure	Total (CAD M)
300 - Water & Tailings Management	
310 – Raw & Potable Water	0.29
320 - Reclaim Water	0.98
340 - Tailings Storage Facility (TSF)	15.77
350 – Mine Water Management	0.08
360 - Effluent Water Management	2.90
370 - Fire Water	0.21
380 - Sewage	0.25
Grand Total	**20.48**

21.3.2.4.4 Mobile Equipment

The CAPEX estimate for WBS Area 400 - Mobile Equipment is summarized in Table 21.16. The costs of the underground mining fleet and support equipment are included in the pre-production section.

Table 21.16: Mobile Equipment Capital Expenditures

Work Breakdown Structure	Total (CAD M)
400 - Mobile Equipment	
410 - Mine Equipment	-
430 - Plant and Surface Mobile Equipment	5.61
Grand Total	**5.61**

21.3.3 Construction Indirect Costs and Owner's Costs

Indirect costs have been developed primarily from detailed estimates:

- Construction indirect costs were developed by GMS based on the historical data for similar projects.
- Freight and logistics were estimated based on the estimate volume of freight from historical data and unit costs as well.
- General & Administration Owner's costs were developed on GMS data base estimate.

- Pre-production mining costs were developed by GMS and are consistent with the basis of the OPEX costs.
- Commissioning costs were based on a ramp-up schedule and OPEX as estimated.

The CAPEX estimate for indirect costs for Scenarios 1, 2 and 3 respectively are summarized in Table 21.17, Table 21.18 and Table 21.19

Table 21.17: Indirect Costs Scenario 1

Work Breakdown Structure	Total (CAD M)
700 - Construction Indirect Costs	35.02
800 - General Services - Owner's Cost	40.41
900 - Preproduction, Start-up, Commissioning	31.46
990 - Contingency	102.09
Grand Total	**208.98**

Table 21.18: Indirect Costs Scenario 2

Work Breakdown Structure	Total (CAD M)
700 - Construction Indirect Costs	32.70
800 - General Services - Owner's Cost	37.73
900 - Preproduction, Start-up, Commissioning	31.49
990 - Contingency	96.39
Grand Total	**198.31**

Table 21.19: Indirect Costs Scenario 3

Work Breakdown Structure	Total (CAD M)
700 - Construction Indirect Costs	34.77
800 - General Services - Owner's Cost	40.12
900 - Preproduction, Start-up, Commissioning	117.88
990 - Contingency	115.74
Grand Total	**308.51**

CAPEX estimate for WBS Area 910 – Mining Operation Pre-production Costs

The CAPEX includes pre-production mining prior to commercial production All three scenarios are presented in Table 21.1.

Table 21.20: Pre-production Mining Capital Expenditures

Activity	Scenario 1 Pre-Prod CAPEX (k$)	Scenario 2 Pre-Prod CAPEX (k$)	Scenario 3 Pre-Prod CAPEX (k$)
OP - Mine Operations	1,591	1,591	-
OP - Mine Maintenance Admin.	2,298	2,298	-
OP - Mine Geology	402	402	-
OP - Mine Engineering	1,475	1,475	-
OP - Grade Control	630	630	-
OP - Drilling	1,367	1,193	-
OP - Blasting	6,117	5,877	-
OP - Pre-Split Drilling and Blasting	47	1,608	-
OP - Loading	4,639	4,155	-
OP - Hauling	4,234	3,598	-
OP - Dump Maintenance	3,699	3,699	-
OP - Road Maintenance	1,09	1,09	-
OP - Dewatering	-	-	-
OP - Equipment	3,721	3,721	-
UG - Mine Development			24,930
UG - Construction	-	-	8,024
UG - Mobile Equipment	-	-	14,331
UG - Fixed Equipment	-	-	1,325
UG - Misc. Equipment	-	-	4,771
UG - Mine Surface Constructions	-	-	6,730
UG - Diamond Drilling	-	-	124
UG - Stope Preparation	-	-	16,435
UG - Mining	-	-	22,856
UG - Mines Services	-	-	6,609
UG - Mechanical Services	-	-	3,407
UG - Electrical Services	-	-	1,818
UG-Technical Services	-	-	3,113
UG-Surface service	-	-	3,254
Sub-Total In-Situ Mining	**31,312**	**31,338**	**117,729**
Rehandling	-		
Total Mining Cost	**31,312**	**31,338**	**117,729**

21.3.3.1 Allowances, Contingency and Escalation

Both the CAPEX and sustaining capital costs include a contingency of 25% of the total cost before contingency. There is no provision for escalation in the CAPEX.

21.4 Sustaining Capital

21.4.1 Mining

Sustaining capital is presented in Table 21.21.

Sustaining capital for the mine includes additional equipment purchases. In the case of the scenarios involving open pit equipment, the major equipment repairs are capitalized over the LOM. The sustaining capital estimate also includes the remaining mining construction civil works for ditches, ponds and dump shear.

The tailings storage expansion over the LOM are accounted for in the sustaining capital. They represent a cost of CAD 57M for Scenario 1, CAD 73 M for Scenario 2 and CAD 139 M for Scenario 3.

Table 21.21: Sustaining CAPEX Scenario

Capital Expenditures	Scenario 1 (CAD M)	Scenario 2 (CAD M)	Scenario 3 (CAD M)
Open Pit Equipment	66.61	1.02	
Open Pit Major Components	48.75	5.19	
Mine surface Constructions / Civil Works	16.36	10.98	5.50
Underground Development		54.12	31.27
Underground Construction		29.94	27.70
Underground Mobile Equipment		105.15	127.25
Underground Fixed Equipment		2.30	1.10
Underground Misc.		5.99	1.39
Tailings Storage Facility Expansion	94.81	72.91	138.65
Contingency	56.63	71.52	83.22
Total	**283.16**	**359.12**	**416.08**

The sustaining capital costs for Scenarios 1 and 2 consist of the purchase and replacement of the open pit equipment, together with the operating costs to attain the mine rate while following the mining schedule.

The sustaining capital costs for Scenarios 2 and 3 with regard to underground mine, account for the development and maintenance of a development rate that is sufficiently advanced to open enough working faces to support the planned mining rate. The second highest sustaining capital requirement is the renewal and purchase of mining equipment (CAD 105M for Scenario 2 and CAD 127M for Scenario 3).

21.4.2 Processing Plant and Infrastructure

21.4.2.1 Introduction

The major sustaining capital costs estimated for each scenario are from the extension of the tailings storage facility. There is no plan in this Study to increase the capacity of process plant. The sustaining capital for Scenario 2 will include the full implementation of the underground mine and infrastructure. At this moment, the plan is to purchase the mining equipment, the construction of the underground ramp access, the ventilation raises and the projected paste backfill plant. As for the initial capital, it has been agreed that a contingency of 25% will be applied to sustaining capital investments.

21.4.2.2 Sustaining CAPEX Scenario 1

Figure 21.1 shows the general layout at maturity for Scenario 1. It demonstrates how the TSF will be developed with two (2) cells built in sequence to minimize the initial capital. As mentioned, the initial capital is CAD 26.71M and Table 21.22 presents the sustaining capital and timing related to the TSF. In addition to the TSF and the mining equipment as discussed in Subsection 21.4.1, South Bluff Creek will be diverted. The estimated costs for this works is CAD 16.36M.

Table 21.22: Scenario 1 TSF Costs

Niobay TSF Costing

Scenario 1

TSF Cell	Tailings Capacity			Embankment	Peat Moss	Liner	Aggregate	Construction
	M m3	Mt		Volume (M m3)	Volume (M m3)	Area (Msq.m.)	(K cu.m)	Cost (M$)
3	16,792	23,509	1 192 510	1,193	3,755	1,9	119	$ 56,755
12	8,197	11,475	902 440	0,902	0,077	0,3	45	$ 6,931
16	8,972	12,561	1 438 150	1,438	0,099	0,3	72	$ 8,711
21	8,023	11,233	1 742 216	1,742	0,099	0,3	87	$ 9,624
26	7,863	11,008	2 794 120	2,794	0,099	0,3	140	$ 12,785
			8 069 436	8	4	3	463	$ 94,806

Figure 21.1: Scenario 1 General Layout



21.4.2.3 Sustaining CAPEX Scenarios 2 and 3

As with Scenario 1, Scenario 2 and Scenario 3, sustaining CAPEX remains TSF and mining equipment / infrastructure. The infrastructure for Scenario 2 is described in Subsection 21.3.2.3.1

The following tables and figures present the TSF costs and general layouts for Scenarios 2 and 3 respectively.

Table 21.23: Scenario 2 TSF Costs

Niobay TSF Costing Scenario 2								
TSF Cell	Tailings Capacity			Embankment	Peat Moss	Liner	Aggregate	Construction
	M m3	Mt		Volume (M m3)	Volume (M m3)	Area (Msq.m.)	(K cu.m)	Cost (M$)
1	5,837	8,172	564 200	0,564	1,600	0,8	169	$ **25,520**
2	6,641	9,298	427 800	0,428	0,074	0,3	128	$ **6,561**
3	8,077	11,308	696 000	0,696	0,096	0,3	209	$ **8,217**
5	8,961	12,545	840 000	0,840	0,096	0,3	252	$ **9,013**
12	7,254	10,156	1 344 000	1,344	0,096	0,3	403	$ **11,800**
17	6,064	8,490	1 344 000	1,344	0,096	0,3	403	$ **11,800**
			696 000	1	0	0	209	$ 72,912

Figure 21.2: Scenario 2 General Layout

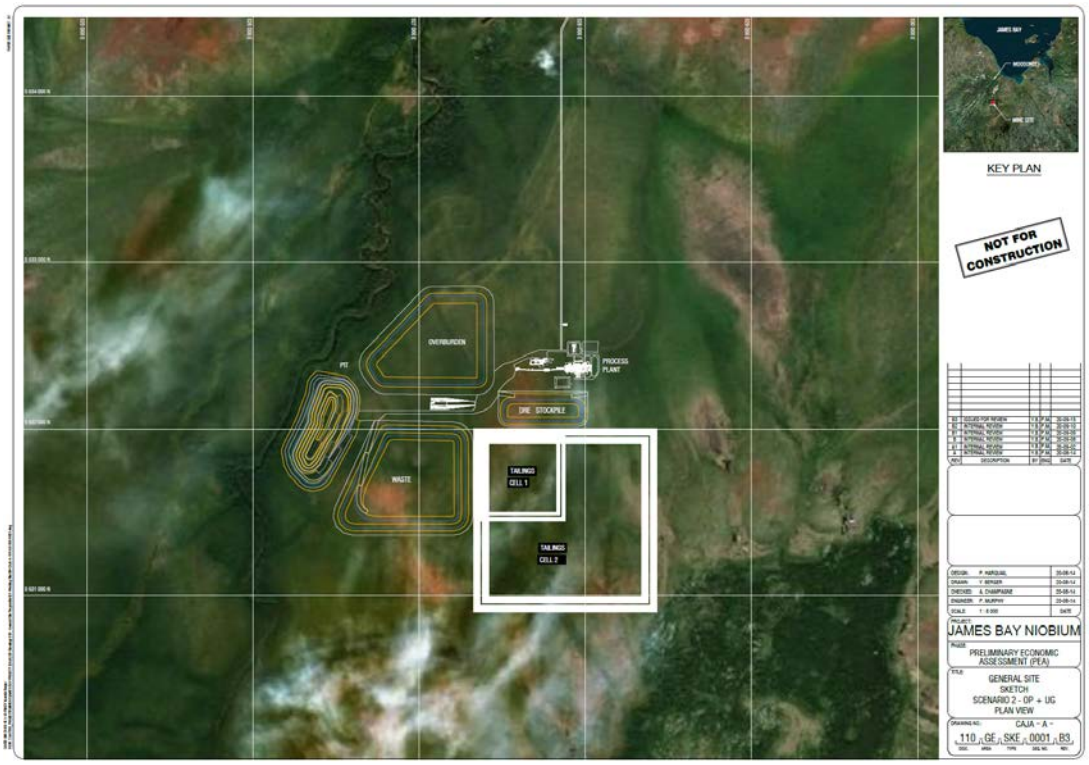


Table 21.24: Scenario 3 TSF Costs

Niobay TSF Costing

Scenario 3

TSF Cell	Tailings Capacity			Embankment	Peat Moss	Liner	Aggregate	Construction
	M m3	Mt		Volume (M m3)	Volume (M m3)	Area (Msq.m.)	(K cu.m)	Cost (M$)
3	5,744	8,041	564 200	0,564	1,600	0,8	169	**$ 32,110**
9	2,250	3,150	427 800	0,428	0,074	0,3	128	**$ 11,558**
12	1,748	2,448	696 000	0,696	0,096	0,3	209	**$ 16,346**
14	2,498	3,497	840 000	0,840	0,096	0,3	252	**$ 18,824**
17	2,792	3,909	1 344 000	1,344	0,096	0,3	403	**$ 27,498**
19	4,505	6,307	1 624 000	1,624	0,096	0,3	487	**$ 32,317**
			1 260 200	1	2	1	378	$ 138,653

Figure 21.3: Scenario 3 General Layout



The following tables present the sustaining capital for each scenario.

Table 21.25: Sustaining Expenditures Summary

Sustaining Expenditures Summary	Scenario 1 (OP)	Scenario 2 OP & UG	Scenario 3 (UG
Sustaining CAPEX (CAD M)			
Water & Tailings Management	94.81	72.91	138.65
Mine Equipment & Infrastructure	115.36	214.39	194.21
Other	16.36	-	-
Contingency (25%)	56.63	71.82	83.22
Total Sustaining CAPEX	**283.16**	**359.12**	**416.08**

21.5 Operating Costs

21.5.1 Operating Costs Summary

Operating costs are summarized in Table 21.26. The operating costs include mining, rehandling, processing, G&A, and converted costs. Operating costs were estimated from first principles and validated in comparable operating mines in the Abitibi region and northern Ontario. The LOM operating costs for each scenario are summarized in following tables.

Table 21.26: Operating Cost Summary Scenario 1

Description	OPEX (CAD M)	Milled (CAD/t)	Produced (CAD/kg-Nb)	Produced (USD/kg-Nb)
Mining Costs	859	12.13	5.24	4.03
Stockpile Rehandling	18.9	0.27	0.11	0.09
Processing	1,034	14.60	6.30	4.85
G&A	708	10.00	4.32	3.32
Converter Costs	813	11.48	4.96	3.81
Total	**3,434**	**48.48**	**20.93**	**16.10**

Table 21.27: Operating Cost Summary Scenario 2

Description	OPEX (CAD M)	Milled (CAD/t)	Produced (CAD/kg-Nb)	Produced (USD/kg-Nb)
Mining Costs Open Pit	133	2.48	0.93	0.72
Mining Costs UG	1,285	23.94	8.99	6.92
Stockpile Rehandling	2.8	0.05	0.02	0.02
Processing	785	14.62	5.49	4.22
G&A	537	10.00	3.76	2.89
Converter Costs	685	12.76	4.80	3.69
Total	**3,428**	**63.85**	**23.99**	**18.45**

Table 21.28: Operating Cost Summary Scenario 3

Description	OPEX (CAD M)	Milled (CAD/t)	Produced (CAD/kg-Nb)	Produced (USD/kg-Nb)
Mining Costs	1,576	29.39	10.906	8.389
Stockpile Rehandling	2.5	0.05	0.017	0.013
Processing	784	14.62	5.424	4.172
G&A	536	10.00	3.711	2.855
Converter Costs	691	12.89	4.782	3.678
Total	**3,590**	**66.94**	**24.84**	**19.11**

The operating organization consists of three departments: mine, including mine operations, geology, engineering and maintenance; process and power plant; and G&A including human resources, environment, health and safety, site services and accounting. Table 21.29presents the labor peak per scenario and per department.

Table 21.29: Peak Operations Workforce

Operations Department	Scenario 1 Peak Workforce	Scenario 2 Peak Workforce	Scenario 3 Peak Workforce
Mine (OP and/or UG)	149	367(1)	211
Process Plant	120	120	120
G&A	30	30	30
Number of Employees	**299**	**517**	**361**

Note (1) There is a peak requirement caused by the overlap of the open pit and underground mine operations

A summary of the total operating costs, by year, is presented in Figure 21.4, Figure 21.5 and Figure 21.6.

Figure 21.4: Operating Cost by Year Scenario 1

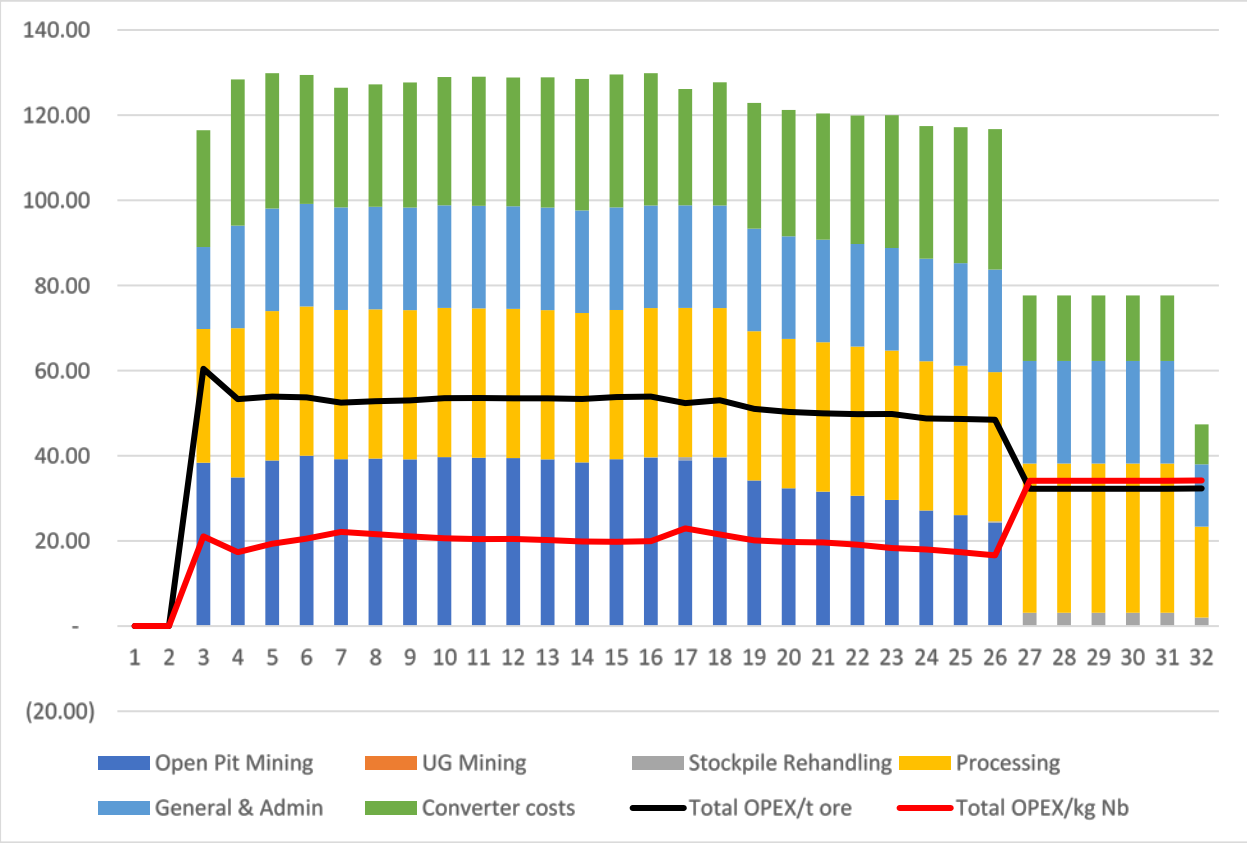


Figure 21.5: Operating Cost by Year Scenario 2

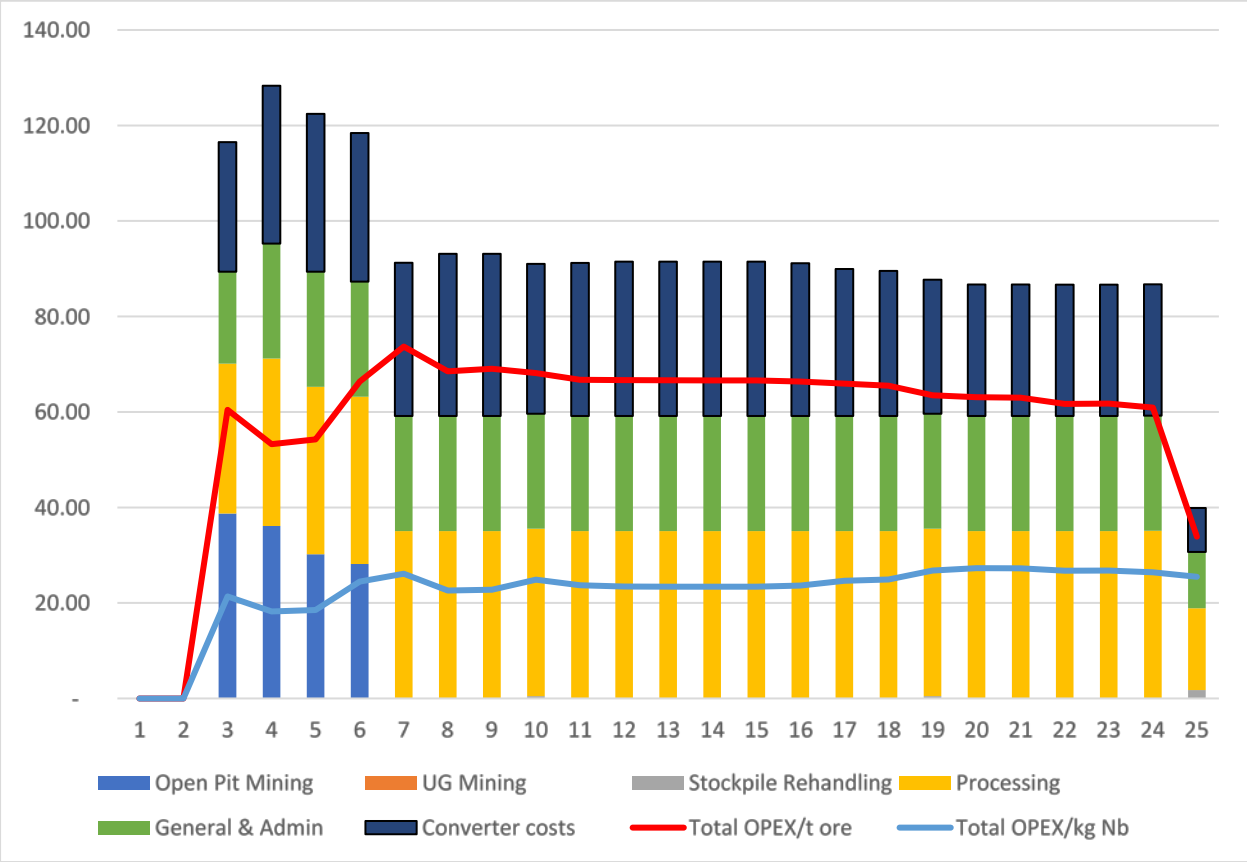
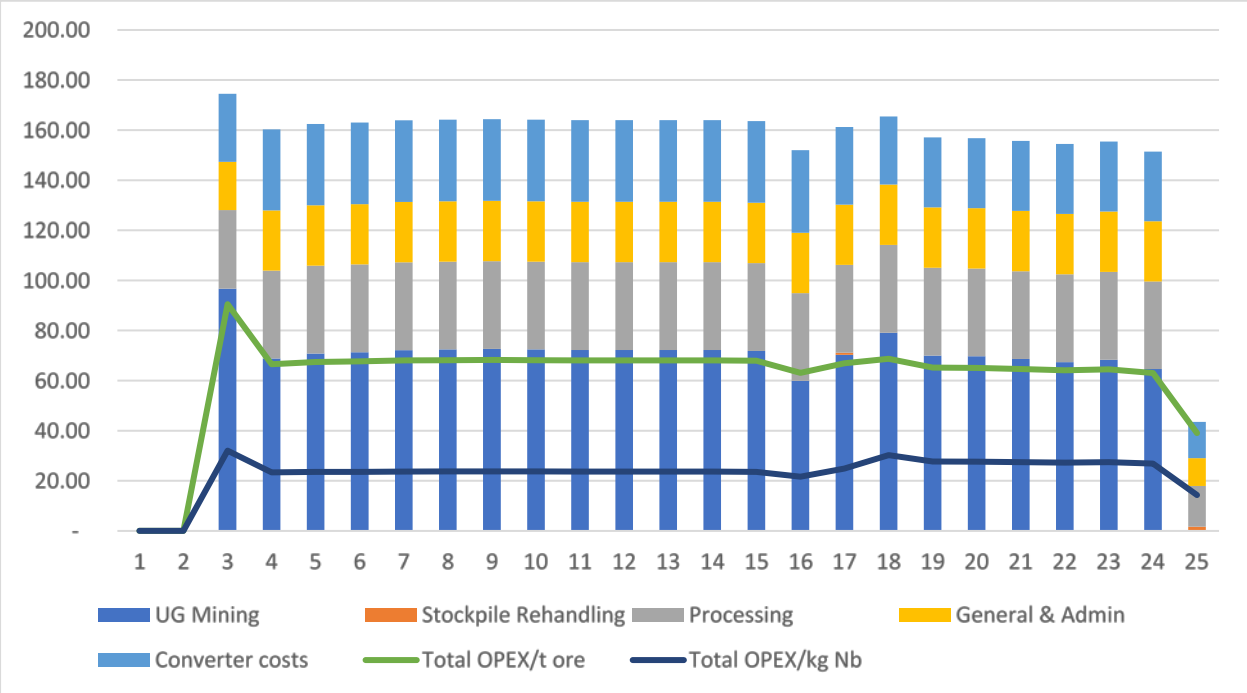


Figure 21.6: Operating Cost by Year Scenario 3



21.5.2 Mining Costs

21.5.2.1 Scenario 1

Table 21.31 presents the breakdown of mining costs, by department, while Table 21.30 presents the major cost drivers for the mine department.

The mine operating costs are estimated from first principles for all mine activities. Equipment hours required to meet the production requirements of the LOM plan are based on productivity factors or equipment simulations. Each piece of equipment has an hourly operating cost which includes operating and maintenance labour, fuel and lube, maintenance parts, tires (if required) and ground engaging tools (if required). A formal budgetary RFP process has been completed for the mine equipment and associated operating costs, fuel, tires, explosives and accessories, etc.

Equipment hours required to meet the life-of-mine (“LOM”) plan are based on productivity factors or equipment simulations. The delivered fuel price to site used in estimating mining costs is CAD 1.12/L. The mine wage scale established for the Project has operators making between CAD 54.94/h and CAD 64.56/h, including benefits and bonus allowances. The hourly operating costs of a few of the major equipment’s are presented for reference in Table 21.30.

Table 21.30: Major Equipment Hourly Operating Cost

Machines	Maint. Labour (CAD/h)	Fuel (CAD/h)	DEF (CAD/h)	Power Costs (CAD/h)	Parts & Repairs (CAD/h)*	Lube (CAD/h)	GET (CAD/h)	Tires (CAD/h)	Total (CAD/h)
Open Pit Production Drill (6-10")	28.11	115.89	2.59		34.08	3.56	0.00	0.00	184.23
Open Pit Diesel Hydraulic Excavator (7 m^3)	12.49	71.32	1.27		20.79	1.54	28.66	0.00	136.08
Open Pit Wheel Loader (8.2 m^3)	18.74	88.73	-		80.46	6.99	72.30	18.88	286.10
Open Pit Haul Truck (70 t)	12.49	64.27	-		28.92	3.87	6.41	14.01	129.97
Open Pit Track Dozer (317 HP)	12.49	40.37	0.72		30.67	2.32	26.23	0.00	112.80
Open Pit Motor Grader (14 ft)	15.62	24.16	0.43		34.86	1.81	9.51	8.99	95.39
U/G 2-Boom Jumbo	42.23	13.77	-	9.60	115	2.18	-	1.32	184.10
U/G ITH Drill	41.20	18.75	-	15.20	73.00	3.30	-	1.70	153.15
U/G Production Loader (LHD Scooptram)	47.68	52.64	-	-	80.00	1.46	20.68	22.00	224.16
U/G Mine Haul Truck	28.33	85.53	-	-	90.00	1.46	10.00	22.00	237.32

**Parts & Repairs excludes Major Components as they are capitalized*

The average mining cost during operations is estimated at CAD 4.56/t mined including re-handling costs. This operating cost estimate excludes capital repairs which treated as sustaining capital.

Haulage is the major mining cost activity representing 22% of total costs followed by loading (11%) and Blasting (9%) Loading and haulage for stockpile re-handling is also captured as a separate activity cost.

Salaries are the dominant cost, by element, representing 45% of total costs, followed by Fuel (19%), and maintenance parts (14%).

Table 21.31: Mining Costs

Activity	Total Cost (CAD M)	Cost/t Milled ($/t)	% of OPEX
Mine Operations	35.00	0.65	4%
Mine Maintenance Admin.	75.08	1.40	9%
Mine Geology	20.71	0.39	2%
Mine Engineering	36.49	0.68	4%
Grade Control	27.47	0.51	3%
Drilling	41.24	0.77	5%
Blasting	74.59	1.39	9%
Pre-Split Drilling and Blasting	40.83	0.76	5%
Loading	91.39	1.70	11%
Hauling	192.84	3.60	22%
Dump Maintenance	74.18	1.38	9%
Road Maintenance	48.44	0.90	6%
Dewatering	2.88	0.05	0%
Support Equipment	98.07	1.83	11%
Total Mining Cost (exl Rehandling)	**859.21**	**16.02**	**100%**

21.5.2.2 Scenario 2

21.5.2.2.1 Open Pit

Open pit mining will follow the same principles as those used in Scenario 1. OPEX estimation was considered. Open pit mining costs are estimated at CAD 138M during operations. The following tables summarize the costs for open pit mining.

Table 21.32: Open Pit Mining Cost for Scenario 2

Activity	Total Cost (CAD M)	Cost/t Milled (CAD/t)	% of OPEX
Mine Operations	6.21	0.12	5%
Mine Maintenance Admin.	12.55	0.23	9%
Mine Geology	3.45	0.06	3%
Mine Engineering	6.45	0.12	5%
Grade Control	4.40	0.08	3%
Drilling	4.06	0.08	3%
Blasting	23.32	0.43	17%
Pre-Split Drilling and Blasting	5.42	0.10	4%
Loading	11.40	0.21	9%
Hauling	19.23	0.36	14%
Dump Maintenance	11.91	0.22	9%
Road Maintenance	8.07	0.15	6%
Dewatering	0.48	0.01	0%
Support Equipment	16.53	0.31	12%
Total Mining Cost (exl Rehandling)	**133.49**	**2.49**	**100%**

21.5.2.2.2 Underground Mining

The mining cost for the open pit is estimated at CAD 1,285M during operation. Table 21.33 summarizes mining costs in Scenario 2 for the underground mine.

Table 21.33: Underground Mining Cost for Scenario 2

Activity	Total Costs (CAD M)	Cost/t Milled (CAD/t)	% of OPEX
Diamond Drilling	9.00	0.20	1%
Stope Preparation	98.45	2.24	8%
Mining	665.85	15.16	52%
Mines Services	178.41	5.03	14%
Mechanic Services	83.62	1.90	7%
Electric Services	100.10	2.28	8%
Technical Services	91.68	2.09	7%
Surface Service	58.50	1.33	5%
Total Mining Cost	**1,285.25**	**30.23**	100%

21.5.2.2.3 Diamond Drilling Costs

Diamond drilling in this evaluation is entirely referred to stope definition prior to final stope design and extraction. This task will be contracted at an estimated global cost of CAD 150/m including cost per meter for the drilling contract, the assays, a provision for grouting holes once they are completed and various drilling supplies.

The meters drilled estimated at 3,156 m/y is based on having at least one hole passed through each stope. Based on a drill performance of 1,800 m drilled per month per drill, it is assumed that only one drill would be required.

In their lifetime a total of 44,978 m will be drilled for Scenario 2. Costs per year are estimated at CAD 0.5M/y or CAD 0.20/t processed.

21.5.2.2.4 Stope Preparation Costs

Stope preparation costs includes the following items:

- Excavation of conventional draw-points to transport ore from transverse long-hole stopes.
- Slot raises (initial hole only).
- All excavations in the ore (O/C, U/C) to provide access for production drilling and blasting activities and for mucking purposes.

Costs are built from first principles and are expressed in unit costs per meter for each category of excavation. These unit costs are multiplied by the number of meters per category and per year to obtain costs per year.

21.5.2.2.5 Mining Costs

Mining costs expenses for Scenario 2 are estimated at CAD 665M. The mining costs include the following items:

- Mine supervision (includes superintendent, assistant superintendent, mine clerks, captains, trainers and mine supervisors).
- Drill and blast.
- Mucking (including rockfill transport).
- Truck hauling.
- Backfilling.

Backfill costs include paste fill fabrication (slag, cement, flocculent, part and lube, plant maintenance) and distribution underground (OPEX piping and barricades). All waste development rock expected to be used as backfill (mixed in stope with pastefill) are costed in the re-handling estimate as part of the LHD requirement calculations and accounted for in the development LHD costs and manpower.

The paste fill is built from first principles and are estimated at CAD 10 per paste fill tonne. Paste fill tonnage requirements consider the waste tonnage from the development work left underground when openings are available. The principal paste fill distribution network costs are all capitalized. The backfill requirement and material balance for Scenario 2 are estimated at CAD 282M.

21.5.2.2.6 Mining Services Costs

The mine service represents an expense of CAD 221 M over the LOM or CAD 5.03/t milled. The mine service cost includes the following items:

- Mine service manpower.
- Underground road maintenance.
- Underground construction.
- Service equipment.

- Ventilation (heating).
- Material handling.
- Drift rehabilitation.
- Lamps and dry.
- Construction repair.

Surface Services

Surface services include mostly mines consumables handling, waste and ore handling and site general maintenance associated to roads and buildings, including the operation and maintenance costs of the surface equipment such as loader, lift and pick-ups. It is estimated at CAD 58.5M for Scenario 2. Table 21.33: Underground Mining Cost for Scenario 2

shows the services cost.

Mechanical and Electrical Services

Mechanical mobile equipment maintenance is included in development unit costs and production costs. Services equipment maintenance costs are however kept in mechanical services costs and represent besides supervision, the largest component of the costs. Remaining maintenance costs related to repair equipment is also included in the mechanical services cost center.

All mechanical maintenance supervision is included in this service and is not redistributed as part of unit costs for production and development purposes.

Electrical mobile equipment maintenance is included in development unit costs and production costs redistributed to according cost centers. Major remaining items included in electrical services are supervision, underground power costs, services vehicles electrical maintenance costs, fans maintenance and other fix equipment costs. Table 21.33 shows the services cost.

Technical Services

Technical services include mine engineering and geology department costs. Specialized design software and license fees and consultants are also part of this cost center. These services are only for underground mining; other technical services associated to other general purposes are not considered in this Study.

21.5.2.3 Scenario 3

The same principles used for the underground mine in Scenario 2 OPEX estimation were considered. Table 21.34 summarizes the costs for Scenario 3.

Table 21.34: Mining Cost for Scenario 3

	Total Costs (CAD M)	(CAD/t Milled)	% of OPEX
Diamond Drilling	10	0.19	1%
Stope Preparation	98	1.83	6%
Mining	846	15.77	54%
Mines Services	216	4.03	14%
Mechanical Services	107	1.99	7%
Electrical Services	125	2.32	8%
Technical Services	108	2.02	7%
Surface Service	66	1.23	4%
Total Mining Cost (excl. Rehandle)	**1,576**	**29.39**	**100%**

21.5.3 Processing Costs

21.5.3.1 Power

Power was estimated from crushing and grinding requirements and the positive impact of the gravity circuit resulting in gangue elimination and reduced tonnage in the subsequent processing steps.

Table 21.35: Power Consumption

Area	Power (kWh/t)
Crushing	1,5
Grinding	7
Flotation	14
Total Concentrator	22.5
Converter	4
Backfill	1
Total Process Facility	**27.5**

21.5.3.2 Reagents

The reagents consumption was estimated from the metallurgical test work program in 2020. Those consumption are preliminary and will be optimized in future metallurgical work.

21.5.3.3 Manpower

The manpower estimate is taking into account the number of stages and complexity of the process plant.

Table 21.36: Manpower

Area	Number
Concentrator	72
Convertor	29
Backfill	5
Total Process Facility	106
Assay Lab.	14
Grand Total	**120**

21.5.3.4 Total Process & Converter Operating Costs

Table 21.37 summarizes the operating costs of the Process Facility which includes the concentrator and converter. The converter operating costs are best expressed and estimated per kilogram of niobium produced. The conversion of the niobium concentrate into ferroniobium (FeNb) is performed by way of thermite reaction with aluminum granules being the main raw material required to generate the reaction.

Table 21.37: Process Operating Costs

Name / Type	
Concentrator	**CAD/t of mill feed**
Manpower	4.05
Power	1.80
Reagents	5.80
Material	2.50
Other	0.45
Total Concentrator	**14.60**
Converter	**CAD/kg Nb**
Manpower	0.74
Power	0.12
Reagents	3.59
Material	0.49
Other	0.01
Total Convertor	**4.95**

21.5.4 General & Administration

The general and administration costs that includes all services departments as well as insurance, camp catering and employee transportation costs are estimated at an annual cost of CAD 2M per year or CAD 10/t of mill feed as summarized in Table 21.38.

Table 21.38: General and Administration Operating Costs

Description	Annual Cost (CAD M)	Unit Cost (CAD/t mill feed)
General Management	1.50	0.63
Insurance	0.60	0.25
Security	0.50	0.21
Camp Costs	4.50	1.88
Flights	4.20	1.75
Accounting & Finance	0.80	0.33
Information Technology	1.10	0.46
Human Resources	1.70	0.71
Health & Safety	0.70	0.29
Surface Support	2.00	0.83
Environment	2.40	1.00
Logistics & Supply Chain	2.60	1.08
Other	1.40	0.58
Total G&A	**24.00**	**10.00**

22. ECONOMIC ANALYSIS

This section presents all elements of the economic model which principally consist of metal production and revenues, royalty payments, operating costs, capital costs, sustaining capital, salvage value, closure and reclamation costs, taxation and net Project cash flow. The three scenarios studies are presented:

- Scenario 1: Open Pit Only.
- Scenario 2: Open Pit & Underground.
- Scenario 3: Underground Only.

The economic analysis is carried out in real terms (i.e. without inflation factors) in Q2 2020 Canadian dollars without any project or equipment financing assumptions. The economic results are calculated from the start of construction which corresponds to a 24-months construction period which excludes permitting, additional engineering studies, and procurement, with all prior costs treated as sunk or excluded from the evaluation. The economic results such as the net present value ("NPV") and internal rate of return ("IRR") are calculated on an annual basis.

22.1 Assumptions

The economic analysis is based on a long-term constant price for niobium, sold as ferroniobium (FeNb), of USD 45/kg. Further information on supply and demand and price forecasts are presented in Section 20. The niobium revenues are converted to Canadian dollars based on an exchange rate of CAD 1.30/USD.

22.2 Metal Production & Revenues

Niobium metal production and therefore revenue varies for the three scenarios as summarized for the life of mine ("LOM") in Table 22.1. More tonnage at a lower grade is processed under Scenario 1 resulting in a LOM production of 164.1 M kg for gross revenues of CAD 9.6 billion.

Table 22.1: Summary of LOM Metal Production & Revenues

Revenue Summary	Scenario 1 (OP)	Scenario 2 (OP&UG)	Scenario 3 (UG)
Total Tonnage Processed (kt)	70,845	53,693	53,625
Niobium Production (000 kg Nb)	164,100	142,900	144,508
Sales Price (USD/kg)	45.00	45.00	45.00
Gross Revenues from Nb Sales (M USD)	7,384	6,431	6,503
Gross Revenues from Nb Sales (M CAD)	9,600	8,360	8,454
Gross Revenue per Tonne of Mill Feed (CAD/t)	135.51	155.69	157.65

22.3 Royalties and Marketing Costs

A 2% royalty is payable to Barrick Gold Corporation, James Bay Columbium Ltd. And Goldcorp Inc. ("collectively the Vendors"). Niobay has the right to buyback half the royalty (1%) at any time for CAD 2M (in 2016 constant dollars subject to a cap of CAD 3M).

Osisko Gold Royalties ("OGR") was granted an option to purchase a 1% royalty on all metals to be produced by the James Bay Niobium Project. The option is exercisable by paying CAD 2M to Niobay at any time within 90 days following the delivery of a bankable feasibility study.

For the purposes of the PEA it was considered that the OGR option payment offsets the buyback cost of Niobay with a resulting 1% royalty payable to the vendor group and 1% to OGR.

Realization costs for marketing the niobium, packaging and freight costs to customers is estimated at 1.5% of sales. Product is typically packaged in big bags or drums.

22.4 Operating Cost Summary

Operating costs include mining, processing, converter costs, G&A services, marketing and royalty expenses. The LOM operating cost summary is presented in Table 22.2. The Open Pit Scenario (Scenario 1) results in lower unit operating costs of CAD 53.22/t of mill feed compared to CAD 72.46/t of mill feed for the Underground Only Scenario with the majority of the variation due to differences in mining costs. The average LOM unit cost per kilogram of niobium is USD 17.67/kg for Scenario 1 and higher for Scenarios 2 and 3, respectively at USD 20.03/kg and USD 20.68/kg.

Table 22.2: Summary of LOM Operating Costs

Operating Cost Summary	Scenario 1 (OP)	Scenario 2 (OP&UG)	Scenario 3 (UG)
LOM Operating Costs (CAD M)			
Mining Costs	878	1,421	1,578
Processing Costs	1,034	785	784
Converter Costs	813	685	691
G&A Costs	708	537	536
Sub-Total Site Costs	**3,434**	**3,428**	**3,590**
Marketing Costs	144	125	127
Royalty Costs	192	167	169
Total Operating Costs	**3,770**	**3,721**	**3,885**
LOM Unit Operating Costs (CAD/t mill feed)			
Mining Costs	12.39	26.47	29.44
Processing Costs	14.60	14.62	14.62
Converter Costs	11.48	12.76	12.89
G&A Costs	10.00	10.00	10.00
Subtotal Site Costs	**48.47**	**63.85**	**66.94**
Marketing Costs	2.03	2.34	2.36
Royalty Costs	2.71	3.11	3.15
Total Operating Costs	**53.22**	**69.30**	**72.46**
LOM Unit Operating Costs (USD/kg)			
Mining Costs	4.12	7.65	8.40
Processing Costs	4.85	4.22	4.17
Converter Costs	3.81	3.69	3.68
G&A Costs	3.32	2.89	2.85
Sub-Total Site Costs	**16.10**	**18.45**	**19.11**
Marketing Costs	0.67	0.68	0.67
Royalty Costs	0.90	0.90	0.90
Total Operating Costs	**17.67**	**20.03**	**20.68**

22.5 Capital Cost Summary

The capital expenditures include initial capital ("CAPEX") as well as sustaining capital to be spent after commencement of commercial operations. Excluded from the initial CAPEX are costs related to further engineering studies, ESIA and permitting costs.

22.5.1 Initial Capital Costs

The CAPEX for Project construction, including processing, mine equipment purchases, pre-production activities, infrastructures and other direct and indirect costs is estimated for each Scenario as summarized in Table 22.3. The total initial Project capital including a 25% contingency is estimated at CAD 510M, CAD 482M and CAD 579M respectively for Scenarios 1, 2 and 3.

22.5.2 Sustaining Capital Costs

Sustaining capital is required during operations for tailings storage facility expansion, additional equipment purchases, mine equipment capital repairs and mine civil works.

Table 22.3: Summary of LOM Capital Costs

Capital Expenditures Summary	Scenario 1 (OP)	Scenario 2 (OP&UG)	Scenario 3 (UG)
Initial CAPEX (CAD M)			
Infrastructure	133.58	133.58	112.61
Power & Electrical	31.49	31.49	31.49
Water & tailings management	31.41	13.58	20.48
Mobile Equipment	35.02	35.02	5.61
Mining Infrastructure	-	-	30.00
Process Plant (including converter)	69.99	69.99	69.99
Subtotal Directs	**301.48**	**283.64**	**270.18**
Construction Indirects	35.02	32.70	34.77
Owner's Costs	40.41	37.73	40.12
Pre-production & Commissioning	31.46	31.49	117.88
Contingency (25%)	102.09	96.39	115.74
Subtotal Indirects	**208.98**	**198.31**	**308.51**
Total Initial CAPEX	**510.45**	**481.95**	**578.69**
Sustaining CAPEX (CAD M)			
Water & Tailings Management	94.81	72.91	138.65
Mine Equipment & Infrastructure	115.36	214.39	194.21
Other	16.36	-	-
Contingency (25%)	56.63	71.82	83.22
Total Sustaining CAPEX	**283.16**	**359.12**	**416.08**
Total CAPEX	**793.62**	**841.07**	**994.77**

22.6 Working Capital

Working capital is required to finance supplies in inventory. A conservative 120 days of operating costs was estimated resulting in an initial working capital requirement of approximately CAD 38M for Scenarios 1 and 2 and higher at CAD 57M for Scenario 3.

22.7 Reclamation & Closure Costs

Reclamation and closure costs include infrastructure decommissioning, site preparation and revegetation, maintenance and post closure monitoring. A specific closure cost was estimated for each scenario based on the surface footprint impact related to mine waste rock dumps and tailings storage facility footprint which is differs.

Table 22.4: Closure Cost Summary

Closure Cost Summary	Scenario 1 (OP)	Scenario 2 (OP&UG)	Scenario 3 (UG)
Closure & Rehabilitation	16.44	10.54	8.94
Water Management	3.01	2.41	1.50
Mine Hazards	0.12	0.14	0.10
Monitoring & Studies	5.37	5.37	5.37
Engineering & Project Management	2.49	1.85	1.59
Contingency	4.99	3.69	3.18
Total Closure Cost	**32.42**	**23.99**	**20.69**

22.8 Taxation

Taxation includes Ontario mining taxes and provincial and federal income taxes with the summary presented in Table 22.5.

Table 22.5: Taxation Summary

Taxation Summary (CAD M)	Scenario 1 (OP)	Scenario 2 (OP&UG)	Scenario 3 (UG)
Ontario Mining Taxes	225.63	174.06	157.24
Provincial Income Taxes	478.64	361.52	343.95
Federal Income Taxes	717.95	542.27	515.92
Total Taxes	**1,422.22**	**1,077.85**	**1,017.10**

22.8.1 Ontario Mining Taxes

Ontario mining tax is levied at a rate of 10% on taxable profit in excess of CAD 0.5M derived from a non-remote mining operation in Ontario and reduced to 5% for remote mines. For the purposes of this PEA it

was assumed that the James Bay Niobium Project would qualify as a remote mine as there is at least 30 km between the mine and the nearest all-weather road or railway suitable to meet the transportation requirements of the mine.

There are specific guidelines for the calculation of profit and depreciation for the purpose of the Ontario mining tax. A mining tax exemption on up to CAD 10M of profit during a 3-year period is available to each new non-remote mine, of which James Bay does qualify.

22.8.2 Income Taxes

The federal and provincial income taxes have both been estimated from an identical taxable income which is arrived at by deducting the Ontario mining tax and various tax depreciations allowances. The federal income tax rate is 15% while the Ontario income tax rate is 10%.

22.9 Economic Results

The main economic metrics used to evaluate the Project consist of net undiscounted after-tax cash flow, net discounted after-tax cash flow or NPV, IRR and payback period. A discount rate of 8% was used to evaluate the present value of the Project which is meant to reflect the cost of capital and is considered an acceptable discount rate for industrial minerals.

22.9.1 Base Case Results

A summary of the Project economic results is presented in Table 22.6. The annual cash flow model for each scenario is presented in Table 22.7.

The open pit Scenario 1 generates the best project economics with an after-tax NPV 8% of CAD 1,008M and an after-tax IRR of 27.5% with a short payback period of 3.2 years.

Table 22.6: Project Economic Results and Summary

Project Economics Summary	Scenario 1 (OP)	Scenario 2 (OP & UG)	Scenario 3 (UG)
Production Summary			
Mine Life (yrs.)	29.6	22.4	22.4
Tonnage Processed (Mt)	70.84	53.69	53.62
Head Grade (% Nb_2O_5)	0.44	0.51	0.51
Niobium Production (M Kg Nb)	164.10	142.90	144.51
Cash Flow Summary (M CAD)			
Gross Revenue	9,600	8,360	8,454
Mining Costs	-878	-1,421	-1,578
Processing Costs	-1,034	-785	-784
Converter Costs	-813	-685	-691
G&A Costs	-708	-537	-536
Marketing Costs	-144	-125	-127
Royalty Costs	-192	-167	-169
Total Opex	-3,770	-3,721	-3,885
EBITDA	5,830	4,639	4,568
Initial Capex	-510	-482	-579
Sustaining Capex	-283	-359	-416
Closure Costs	-32	-24	-21
Taxes	-1,422	-1,078	-1,017
Before-Tax Results			
Before-Tax Cash Flow (NPV 0%)	5,004	3,774	3,553
Before-Tax NPV 8%	1,475	1,268	1,104
Before-Tax IRR	33.6%	33.4%	26.0%
Before-Tax Payback (yrs.)	2.6	2.5	3.8
After-Tax Results			
After-Tax Cash Flow (NPV 0%)	3,581	2,696	2,536
After-Tax NPV 8%	1,008	856	733
After-Tax IRR	27.5%	27.0%	21.6%
After-Tax Payback (yrs.)	3.2	3.1	4.3

Table 22.7: Annual Cash Flow Model Scenario 1

Project Economics Summary	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
Production Summary																																			
Mine Life (yrs.)	29.6	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.6	-	-
Tonnage Processed (Mt)	70.8	-	-	1.93	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	1.47	-	-
Head Grade (% Nb_2O_5)	0.44	-	-	0.55	0.58	0.53	0.50	0.45	0.46	0.48	0.49	0.50	0.50	0.50	0.51	0.52	0.51	0.43	0.47	0.48	0.48	0.48	0.49	0.52	0.52	0.53	0.56	0.18	0.18	0.18	0.18	0.18	0.18	-	-
Nb Production (M kg)	164.1	-	-	5.53	7.40	6.71	6.29	5.71	5.89	6.06	6.26	6.32	6.29	6.38	6.47	6.55	6.51	5.50	5.94	6.10	6.14	6.13	6.27	6.55	6.53	6.75	7.04	2.28	2.28	2.28	2.28	2.28	1.38	-	-
Cash Flow Summary (M CAD)																																			
Gross Revenue	9,600	-	-	323	433	393	368	334	344	354	366	370	368	373	379	383	381	322	347	357	359	359	367	383	382	395	412	133	133	133	133	133	81	-	-
Mining Costs	(878)	-	-	(38)	(35)	(39)	(40)	(39)	(39)	(39)	(40)	(40)	(39)	(39)	(38)	(39)	(40)	(40)	(40)	(34)	(32)	(32)	(31)	(30)	(27)	(26)	(25)	(3)	(3)	(3)	(3)	(3)	(2)	-	-
Processing Costs	(1,034)	-	-	(31)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(21)	-	-
Converter Costs	(813)	-	-	(27)	(34)	(32)	(30)	(28)	(29)	(29)	(30)	(30)	(30)	(31)	(31)	(31)	(31)	(27)	(29)	(30)	(30)	(30)	(30)	(31)	(31)	(32)	(33)	(15)	(15)	(15)	(15)	(15)	(9)	-	-
G&A Costs	(708)	-	-	(19)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(15)	-	-
Marketing Costs	(144)	-	-	(5)	(6)	(6)	(6)	(5)	(5)	(5)	(5)	(6)	(6)	(6)	(6)	(6)	(6)	(5)	(5)	(5)	(5)	(5)	(6)	(6)	(6)	(6)	(6)	(2)	(2)	(2)	(2)	(2)	(1)	-	-
Royalty Costs	(192)	-	-	(6)	(9)	(8)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(7)	(8)	(8)	(8)	(6)	(7)	(7)	(7)	(7)	(7)	(8)	(8)	(8)	(8)	(3)	(3)	(3)	(3)	(3)	(2)	-	-
Total OPEX	(3,770)	-	-	(128)	(144)	(144)	(142)	(138)	(139)	(140)	(142)	(142)	(142)	(142)	(142)	(143)	(143)	(137)	(140)	(135)	(134)	(133)	(133)	(133)	(131)	(131)	(131)	(82)	(82)	(82)	(82)	(82)	(50)	-	-
EBITDA	5,830	-	-	196	289	249	226	196	205	214	224	228	226	232	237	240	238	184	207	221	225	226	234	250	251	264	281	51	51	51	51	51	31	-	-
Initial CAPEX	(510)	(207)	(304)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	(283)	-	-	(15)	(14)	(78)	(6)	(8)	(5)	(11)	(4)	(13)	(11)	(6)	(19)	(4)	(8)	(7)	(14)	(7)	(4)	(6)	(8)	(15)	(0)	(2)	(1)	(0)	(16)	-	(0)	(1)	-	-	-
Working Capital	(0)	-	(38)	(4)	(0)	0	1	(0)	(0)	(0)	(0)	0	(0)	0	(0)	(0)	1	(1)	2	1	0	0	(0)	1	0	0	13	-	-	-	-	10	16	-	-
Closure Costs	(32)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(6)	(13)	(13)
Before-Tax CF	5,004	(207)	(342)	177	274	172	221	188	200	203	220	215	216	225	218	236	231	176	195	215	222	220	226	236	251	262	293	50	34	51	51	60	40	(13)	(13)
Taxes	(1,422)	-	-	(15)	(53)	(43)	(43)	(39)	(45)	(51)	(57)	(59)	(59)	(61)	(63)	(65)	(64)	(49)	(56)	(60)	(62)	(62)	(64)	(68)	(69)	(73)	(79)	(13)	(12)	(12)	(9)	(10)	(6)	-	-
After-Tax CF	3,581	(207)	(342)	162	222	129	178	149	155	152	163	156	157	164	155	171	167	127	139	155	160	158	162	167	181	189	214	38	22	39	41	51	34	(13)	(13)
Metrics																																			
EBITDA Margin (%)	155%			153%	202%	173%	159%	142%	147%	153%	158%	160%	160%	163%	167%	168%	166%	134%	148%	164%	169%	170%	176%	187%	192%	202%	214%	62%	62%	62%	62%	62%	61%		
Unit Cost (CAD/t ore)	53.22			66.29	59.59	59.61	59.07	57.34	57.82	58.15	58.84	58.93	58.83	58.93	58.85	59.34	59.43	57.04	58.06	56.19	55.53	55.19	55.10	55.37	54.29	54.37	54.44	34.18	34.18	34.18	34.18	34.18	34.25		
Unit Cost (USD/kg Nb)	17.67			17.78	14.92	16.45	17.39	18.60	18.20	17.78	17.43	17.28	17.33	17.10	16.85	16.79	16.91	19.21	18.12	17.07	16.76	16.67	16.28	15.67	15.41	14.92	14.34	27.82	27.82	27.82	27.82	27.82	27.88		

Table 22.8: Annual Cash Flow Model Scenario 2

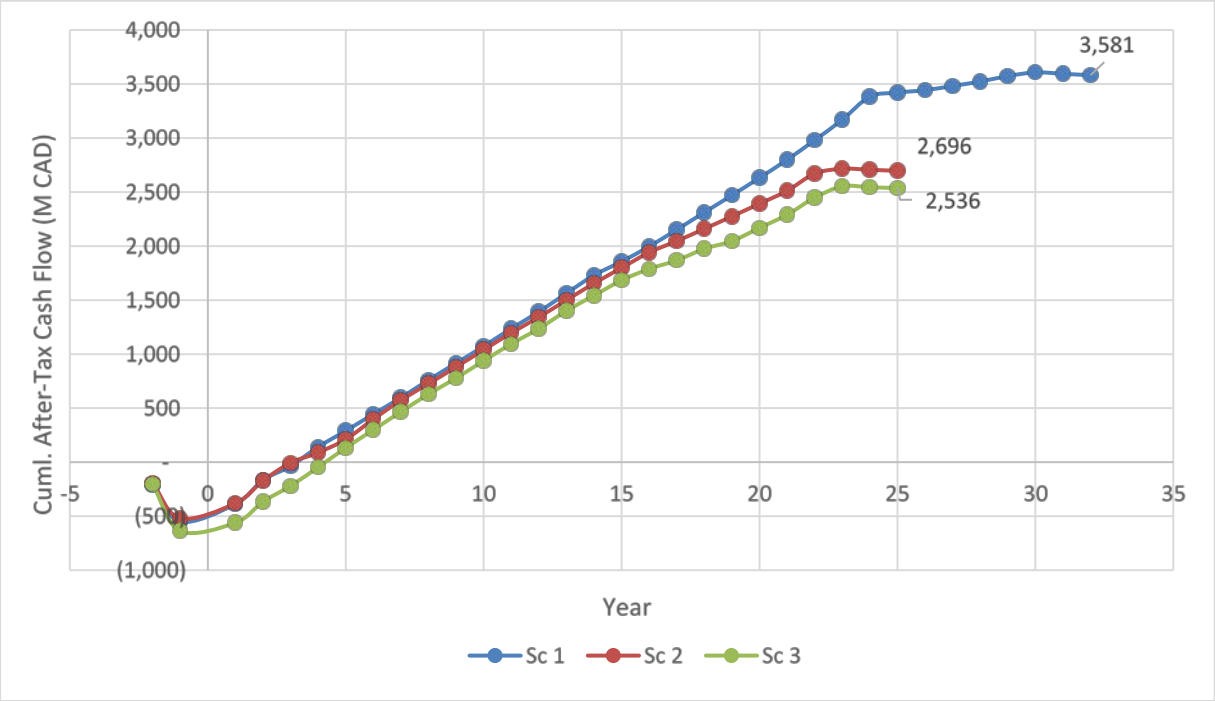
Project Economics Summary	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
Production Summary																																			
Mine Life (yrs.)	22.4	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.4	-	-	-	-	-	-	-	-	-
Tonnage Processed (Mt)	53.7	-	-	1.93	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	1.18	-	-	-	-	-	-	-	-	-
Head Grade (% Nb_2O_5)	0.51	-	-	0.54	0.56	0.56	0.52	0.54	0.58	0.58	0.52	0.54	0.54	0.54	0.54	0.54	0.53	0.51	0.50	0.45	0.44	0.44	0.44	0.44	0.44	0.25	-	-	-	-	-	-	-	-	-
Nb Production (M kg)	142.9	-	-	5.45	7.05	7.05	6.53	6.80	7.30	7.30	6.60	6.79	6.85	6.85	6.85	6.85	6.77	6.44	6.33	5.71	5.57	5.57	5.55	5.55	5.56	1.57	-	-	-	-	-	-	-	-	-
Cash Flow Summary (M CAD)																																			
Gross Revenue	8,360	-	-	319	412	413	382	398	427	427	386	397	401	401	401	401	396	377	371	334	326	326	325	325	325	92	-	-	-	-	-	-	-	-	-
Mining Costs	(1,421)		-	(39)	(36)	(38)	(70)	(86)	(72)	(73)	(74)	(70)	(69)	(69)	(69)	(69)	(69)	(69)	(68)	(66)	(65)	(65)	(62)	(62)	(60)	(2)	-	-	-	-	-	-	-	-	-
Processing Costs	(785)		-	(31)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(17)	-	-	-	-	-	-	-	-	-
Converter Costs	(685)		-	(27)	(33)	(33)	(31)	(32)	(34)	(34)	(31)	(32)	(32)	(32)	(32)	(32)	(32)	(31)	(30)	(28)	(28)	(28)	(28)	(28)	(28)	(9)	-	-	-	-	-	-	-	-	-
G&A Costs	(537)		-	(19)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(12)	-	-	-	-	-	-	-	-	-
Marketing Costs	(125)		-	(5)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(5)	(5)	(5)	(5)	(5)	(5)	(1)	-	-	-	-	-	-	-	-	-
Royalty Costs	(167)		-	(6)	(8)	(8)	(8)	(8)	(9)	(9)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(7)	(7)	(7)	(7)	(6)	(6)	(6)	(2)	-	-	-	-	-	-	-	-	-
Total OPEX	(3,721)		-	(128)	(143)	(145)	(173)	(191)	(180)	(181)	(178)	(175)	(175)	(175)	(175)	(174)	(174)	(172)	(171)	(165)	(163)	(163)	(160)	(160)	(158)	(43)	-	-	-	-	-	-	-	-	-
EBITDA	4,639		-	191	270	268	209	206	247	246	209	222	226	226	226	227	222	205	200	169	162	162	165	165	167	48	-	-	-	-	-	-	-	-	-
Initial CAPEX	(482)	(194)	(288)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	(359)	-	-	(34)	(10)	(42)	(70)	(50)	(8)	(13)	(9)	(11)	(11)	(10)	(22)	(10)	(9)	(7)	(5)	(19)	(7)	(5)	(3)	(4)	-	-	-	-	-	-	-	-	-	-	-
Working Capital	(0)		(38)	(4)	(1)	(10)	(6)	4	(0)	1	1	0	0	0	0	0	0	0	2	0	0	1	(0)	1	35	13	-	-	-	-	-	-	-	-	-
Closure Costs	(24)		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(5)	(10)	(10)	-	-	-	-	-	-	-
Before-Tax CF	3,774	(194)	(326)	153	258	216	133	160	239	234	201	211	215	216	205	217	214	198	197	150	155	159	161	162	202	57	(10)	(10)	-	-	-	-	-	-	-
Taxes	(1,078)	-	-	(14)	(48)	(52)	(37)	(38)	(53)	(57)	(50)	(55)	(58)	(59)	(59)	(59)	(59)	(54)	(54)	(45)	(43)	(43)	(44)	(42)	(43)	(11)	-	-	-	-	-	-	-	-	-
After-Tax CF	2,696	(194)	(326)	139	210	165	95	122	186	177	151	156	158	157	146	157	155	144	143	106	113	116	117	119	159	45	(10)	(10)	-	-	-	-	-	-	-
Metrics																																			
EBITDA Margin (%)	125%			150%	189%	184%	121%	108%	137%	136%	117%	127%	130%	130%	130%	130%	128%	119%	117%	103%	99%	100%	103%	103%	105%	113%									
Unit Cost (CAD/t ore)	69.30			66.25	59.25	60.25	71.90	79.45	74.72	75.24	73.74	72.52	72.48	72.46	72.44	72.42	72.12	71.39	70.87	68.36	67.81	67.72	66.41	66.48	65.65	36.58									
Unit Cost (USD/kg Nb)	20.03			18.02	15.58	15.83	20.40	21.67	18.96	19.10	20.70	19.80	19.60	19.59	19.58	19.58	19.75	20.54	20.73	22.19	22.58	22.54	22.15	22.18	21.90	21.17									

Table 22.9: Annual Cash Flow Model Scenario 3

Project Economics Summary	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
Production Summary																																			
Mine life (yrs.)	22.4	-	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.4	-	-							
Tonnage Processed (Mt)	53.6	-	-	1.93	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.41	2.40	1.12	-	-							
Head Grade (% Nb_2O_5)	0.51	-	-	0.54	0.54	0.54	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.51	0.43	0.45	0.45	0.45	0.45	0.45	0.45	0.52	-	-							
Nb Production (M kg)	144.5	-	-	5.45	6.86	6.90	6.92	6.93	6.92	6.92	6.92	6.93	6.93	6.93	6.93	6.94	7.03	6.49	5.47	5.68	5.67	5.67	5.67	5.67	5.64	3.04	-	-							
Cash Flow Summary (M CAD)																																			
Gross Revenue	8,454	-	-	319	401	404	405	405	405	405	405	405	405	405	405	406	411	379	320	332	332	332	332	332	330	178	-	-							
Mining Costs	(1,578)	-	-	(97)	(69)	(71)	(71)	(72)	(72)	(73)	(72)	(72)	(72)	(72)	(72)	(72)	(60)	(71)	(79)	(70)	(70)	(69)	(67)	(68)	(65)	(2)	-	-							
Processing Costs	(784)	-	-	(31)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(35)	(16)	-	-							
Converter Costs	(691)	-	-	(27)	(32)	(32)	(33)	(33)	(33)	(33)	(33)	(33)	(33)	(33)	(33)	(33)	(33)	(31)	(27)	(28)	(28)	(28)	(28)	(28)	(28)	(14)	-	-							
G&A Costs	(536)	-	-	(19)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(11)	-	-							
Marketing Costs	(127)	-	-	(5)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(6)	(5)	(5)	(5)	(5)	(5)	(5)	(5)	(3)	-	-							
Royalty Costs	(169)	-	-	(6)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(6)	(7)	(7)	(7)	(7)	(7)	(7)	(4)	-	-							
Total OPEX	(3,885)	-	-	(186)	(174)	(177)	(177)	(178)	(178)	(179)	(178)	(178)	(178)	(178)	(178)	(178)	(166)	(175)	(177)	(169)	(168)	(167)	(166)	(167)	(163)	(50)	-	-							
EBITDA	4,568	-	-	133	227	227	228	227	227	227	227	227	227	227	227	228	245	205	143	163	163	164	166	165	167	128	-	-							
Initial CAPEX	(579)	(203)	(376)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-							
Sustaining CAPEX	(416)	-	-	(62)	(12)	(48)	(12)	(3)	(9)	(6)	(6)	(22)	(8)	(12)	(27)	(8)	(36)	(10)	(6)	(44)	(18)	(54)	(5)	(5)	(5)	-	-	-							
Working Capital	(0)	-	(57)	5	(1)	(0)	(0)	(0)	(0)	0	0	(0)	-	0	0	4	(3)	(1)	3	0	0	0	(0)	1	35	14	-	-							
Closure Costs	(21)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(4)	(8)	(8)							
Before-Tax CF	3,553	(203)	(433)	76	214	179	215	224	218	220	221	205	219	216	201	224	206	193	140	120	146	111	161	161	198	138	(8)	(8)							
Taxes	(1,017)	-	-	-	(19)	(33)	(41)	(47)	(50)	(53)	(57)	(58)	(58)	(59)	(59)	(60)	(64)	(52)	(36)	(40)	(40)	(38)	(40)	(39)	(41)	(33)	-	-							
After-Tax CF	2,536	(203)	(433)	76	195	146	174	178	168	167	164	148	160	156	142	164	142	141	105	80	106	72	121	123	157	105	(8)	(8)							
Metrics																																			
EBITDA Margin (%)	118%			72%	130%	129%	129%	128%	127%	127%	127%	127%	127%	127%	127%	128%	147%	117%	81%	97%	97%	98%	100%	99%	103%	258%									
Unit Cost (CAD/t ore)	72.46			96.33	72.37	73.30	73.57	73.92	74.03	74.11	74.04	73.95	73.96	73.96	73.95	73.82	69.06	72.44	73.34	70.04	69.91	69.46	68.94	69.34	67.82	44.63									
Unit Cost (USD/kg Nb)	20.68			26.20	19.56	19.69	19.69	19.77	19.81	19.83	19.81	19.78	19.79	19.79	19.78	19.71	18.22	20.70	24.84	22.87	22.84	22.70	22.53	22.66	22.22	12.58									

The cumulative after-tax cash flow for each scenario is presented in Figure 22.1. Scenario 1 generates additional after-tax cash flow due to lower overall capital costs and the increased tonnage due to the lower cut-off grade resulting in a longer mine life.

Figure 22.1: Cumulative After-Tax Cash Flow



22.9.2 Sensitivities

A sensitivity analysis was performed for ±10% and ±15% variations to OPEX and CAPEX and for various prices of niobium in the following tables. Each parameter was calculated independently.

Table 22.10: Niobium Price Sensitivity in Dollar Increments

Scenarios	Niobium Price (USD/kg)				
	35	40	45*	50	55
Scenario 1 (OP)					
After-Tax NPV 8% (CAD M)	490	749	**1,008**	1,268	1,527
After-Tax IRR (%)	18.2%	22.9%	**27.5%**	31.8%	36.0%
After-Tax Payback (yrs.)	4.9	3.8	**3.2**	2.7	2.3
Scenario 2 (OP+UG)					
After-Tax NPV 8% (CAD M)	341	599	**856**	1,114	1,372
After-Tax IRR (%)	16.5%	21.9%	**27.0%**	31.8%	36.3%
After-Tax Payback (yrs.)	5.7	4.3	**3.1**	2.6	2.3
Scenario 3 (UG)					
After-Tax NPV 8% (CAD M)	210	473	**733**	992	1,251
After-Tax IRR (%)	12.4%	17.2%	**21.6%**	25.7%	29.5%
After-Tax Payback (yrs.)	6.6	5.12	**4.3**	3.7	3.2

Table 22.11: Niobium Price Sensitivity in Percentage Increments

Scenarios	Niobium Price Sensitivity				
	-15%	-10%	0%	10%	15%
Scenario 1 (OP)					
After-Tax NPV 8% (CAD M)	658	775	**1,008**	1,242	1,359
After-Tax IRR (%)	21.3%	23.4%	**27.5%**	31.4%	33.3%
After-Tax Payback (yrs.)	4.1	3.8	**3.2**	2.7	2.5
Scenario 2 (OP+UG)					
After-Tax NPV 8% (CAD M)	509	625	**856**	1,089	1,205
After-Tax IRR (%)	20.1%	22.5%	**27.0%**	31.3%	33.4%
After-Tax Payback (yrs.)	4.8	4.2	**3.1**	2.6	2.5
Scenario 3 (UG)					
After-Tax NPV 8% (CAD M)	381	499	**733**	966	1,083
After-Tax IRR (%)	15.6%	17.7%	**21.6%**	25.3%	27.0%
After-Tax Payback (yrs.)	5.6	5.0	**4.3**	3.7	3.5

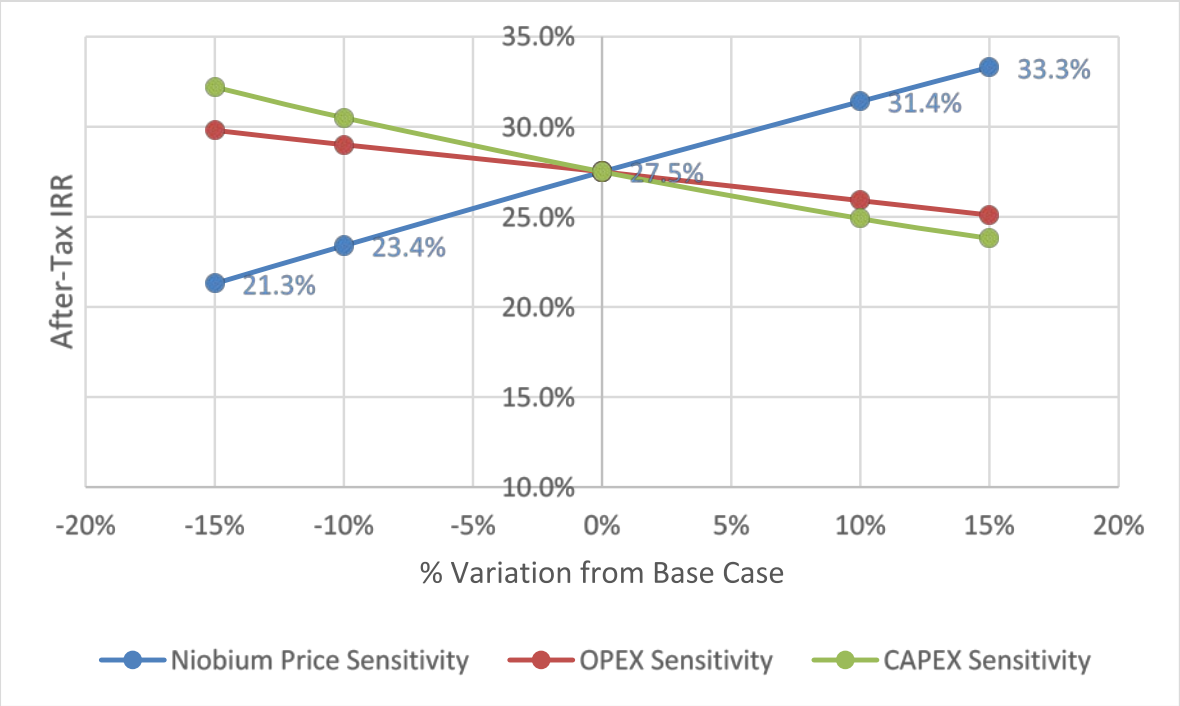
Table 22.12: OPEX Sensitivity in Percentage Increments

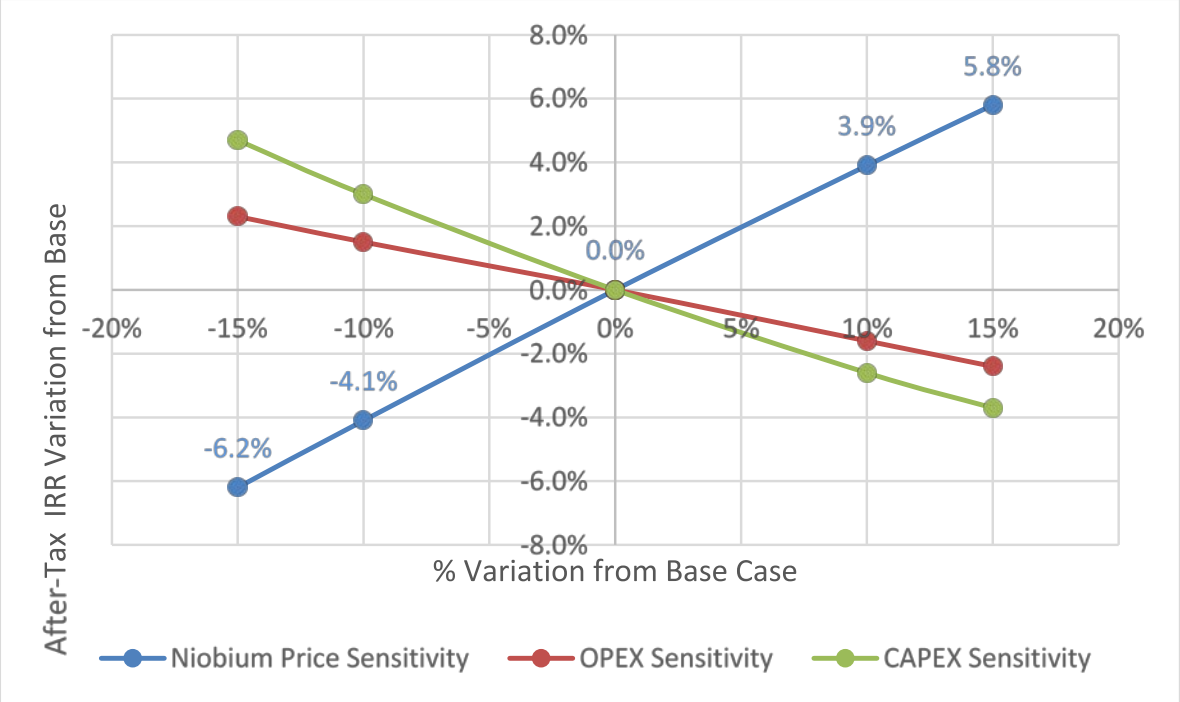
Scenarios	OPEX Sensitivity				
	-15%	-10%	0%	10%	15%
Scenario 1 (OP)					
After-Tax NPV 8% (CAD M)	1,140	1,096	**1,008**	921	877
After-Tax IRR (%)	29.8%	29.0%	**27.5%**	25.9%	25.1%
After-Tax Payback (yrs.)	2.9	3.0	**3.2**	3.4	3.2
Scenario 2 (OP+UG)					
After-Tax NPV 8% (CAD M)	1,004	955	**856**	758	709
After-Tax IRR (%)	29.8%	28.9%	**27.0%**	25.1%	24.1%
After-Tax Payback (yrs.)	2.8	2.9	**3.1**	3.5	3.7
Scenario 3 (UG)					
After-Tax NPV 8% (CAD M)	837	802	**733**	664	629
After-Tax IRR (%)	25.4%	24.0%	**21.6%**	19.5%	18.5%
After-Tax Payback (yrs.)	3.7	3.9	**4.3**	4.7	4.9

Table 22.13: CAPEX Sensitivity in Percentage Increments

Scenarios	CAPEX Sensitivity				
	-15%	-10%	0%	10%	15%
Scenario 1 (OP)					
After-Tax NPV 8% (CAD M)	1,096	1,067	**1,008**	950	921
After-Tax IRR (%)	32.2%	30.5%	**27.5%**	24.9%	23.8%
After-Tax Payback (yrs.)	2.6	2.8	**3.2**	3.6	3.7
Scenario 2 (OP+UG)					
After-Tax NPV 8% (CAD M)	949	918	**856**	795	764
After-Tax IRR (%)	32.0%	30.1%	**27.0%**	24.3%	23.1%
After-Tax Payback (yrs.)	2.5	2.7	**3.1**	3.7	4.1
Scenario 3 (UG)					
After-Tax NPV 8% (CAD M)	892	839	**733**	627	573
After-Tax IRR (%)	24.3%	23.4%	**21.6%**	19.7%	18.8%
After-Tax Payback (yrs.)	3.8	4.0	**4.3**	4.6	4.8

Figure 22.2: After-Tax IRR Sensitivity of Scenario 1





23. ADJACENT PROPERTIES

There are no properties adjacent to the Project.

24. OTHER RELEVANT DATA AND INFORMATION

The reader is cautioned that this Preliminary Economic Assessment ("PEA") is preliminary in nature as it includes Inferred Mineral Resources that are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

This Report has several cut-off dates for information:

- The effective date of the Current Mineral Resource is July 9, 2020
- The effective date of this Report is October 13, 2020
- The amounts are in Canadian dollars of Q2 2020

Additional Information is not required to make the Technical Report understandable and not misleading.

25. INTERPRETATION AND CONCLUSIONS

25.1 Exploration and Geology

- The Project is hosted by the Carbonatite Complex which occurs within the northern portion of the KSZ.
- Pyrochlore and, to a lesser extent, columbite, are the economic minerals of interest and are hosted predominantly by the 25-1oviet phase of the carbonatite.
- Extensive work during the mid to late 1960s resulted in a historical Mineral Resource estimate and feasibility study. The Property has been dormant since 1972. Re-sampling of the historical diamond drill core by NioBay has confirmed that, despite some variation at the individual sample level, the overall grade over wide intervals is similar to historical values. RPA reviewed the analytical quality control results and did not find any material issues. In RPA's opinion, the resource database is acceptable to estimate Mineral Resources
- Mineral Resources were estimated and classified by RPA following CIM (2014) definitions. At a cut-off grade suitable for an underground operation of 0.3% Nb_2O_5, Indicated Mineral Resources are estimated to total 29.7 Mt grading 0.53% Nb_2O_5 containing approximately 158 million kilograms of niobium oxide. Inferred Mineral Resources are estimated to total 33.8 Mt grading 0.52 Nb_2O_5 containing 177 million kilograms of niobium oxide.
- RPA is of the opinion that there is excellent exploration potential to increase the Mineral Resource at depth with more diamond drilling.

25.2 Mining

- The deposit can be mined by either open pit, underground or a combination of both. The Open Pit Scenario is more economical and is a simpler operation compared to that of an underground or a Combined Open Pit and Underground Scenario. However, all scenarios yield attractive economic results.
- Mineral Reserve estimates for the Project were determined solely on an Open Pit Only Scenario to a level appropriate for a PEA Study.
- The river located along the proposed mine will have to be redirected halfway through the mine life.
- At a cut-off grade of 0.12% Nb_2O_5, a mineral reserve is estimated to be 70.8 Mt at an average grade of 0.44% Nb_2O_5.

- Mining is to be carried out using conventional open pit techniques with hydraulic shovels and mining trucks. An Owner mining open pit operation is planned with the outsourcing of certain support activities such as explosives manufacturing and blasting.
- Mining operations will occur 365 days a year on the basis of two (2) twelve hour shifts per day, seven (7) days a week.
- The LOM plan details 25 years of mining including a 12-month ramp-up and commissioning period.
- The open pit generations 21.2 Mt of overburden and 121.3 Mt waste rock for strip ratio 1.8:1.

25.3 Metallurgy & Processing

- The 2020 program succeeded in reproducing the historical results using an initial gravity separation stage and a simplified process flowsheet. Approximately 42% of the mass was rejected with positive impacts on capital and operating costs.
- Compared to historical test work, fewer cleaner stages were required to produce a flotation concentrate inside quality specifications.
- Silica challenges were resolved reaching specification level in the flotation concentrates which were better than the historical pilot plant results.
- Niobium concentrate reached quality target level for the contaminants.
- Considering the higher grade of the composites compared to the grade of the resources, especially for the Open Pit Scenario, further testing will be needed to validate results with different ore grades.

25.4 Infrastructure and Power Supply

- The PEA contemplates a permanent all-season road to reach Moosonee. Three road alignment options were developed. The preferred option is an alignment along the east bank of the South Bluff Creek, crossing Moose Factory 88 Reserve to Moose River. From the east, a 6-meter-wide tunnel 4.2 km in length is planned to cross Moose River just south of Renison substation.
- Five Nations Energy Inc. (“FNEI”) retains the rights north of HydroOne Renison substation for power transmission/distribution. This substation is approximately 45 km west of the James Bay Project. The Moose River crossing could use the planned tunnel or an aerial powerline. The powerline voltage is 115 kV. From the east bank of Moose River, the power line will follow the main access road. The power draw at site is estimated to 8 MW for the planned infrastructure.

- The main infrastructure required to support mining and processing operations includes a permanent access road, power supply, assay laboratory, camp facility and support buildings such as a maintenance facility, warehouse, administration building and mine dry.

25.5 OPEX, CAPEX and Economics

- The capital expenditures ("CAPEX") for Project construction, including processing, mine equipment purchases, pre-production activities, infrastructures and other direct and indirect costs is estimated for each Scenario. The total initial Project CAPEX including a 25% contingency is estimated at CAD 510M, CAD 482M and CAD 579M respectively for Scenarios 1, 2 and 3. Sustaining CAPEX is required during operations for tailings storage facility expansion, additional equipment purchases, mine equipment capital repairs and mine civil works.
- Operating costs include mining, processing, converter costs, G&A services, marketing and royalty expenses. The Open Pit Scenario (Scenario 1) results in lower unit operating costs of CAD 53.22/t of mill feed compared to CAD 72.46/t of mill feed for the Underground Only Scenario with the majority of the variation due to differences in mining costs. The average LOM unit cost per kilogram of niobium is USD 17.67/kg for Scenario 1 and higher for Scenarios 2 and 3, respectively at USD 20.03/kg and USD 20.68/kg.
- The economic results are as follows:
    - Scenario 1: after-tax NPV 8% of CAD 1,008M and an after-tax IRR of 27.5%.
    - Scenario 2: after-tax NPV 8% of CAD 856M and after-tax IRR of 27.0%.
    - Scenario 3: after-tax NPV 8% of CAD 733M and after-tax IRR of 21.6%.

26. RECOMMENDATIONS

G Mining Services Inc. (“GMS”) recommends that further work be undertaken to compliment the current Preliminary Economic Assessment (“PEA”) and support a future Pre-Feasibility (“PFS”) and ultimately a decision to construct the Project.

Niobay Metals Inc. plans the following principal tasks in the next phase of development:

- Complete a prefeasibility study (PFS).
- Continue or start the process of an EIS/EA.
- Initiating all required environmental and construction permits.
- Managing and mitigating key risks and pursuing opportunities to improve project economics.

The cost for this phase of the work are estimated at approximately CAD 7.9M. The list of specific recommendations that follow applies to this and successive phases of work.

26.1 Exploration and Geology

- RPA and GMS are of the opinion that the Project hosts a significant niobium mineralized system, there is good potential to increase the resource base, and additional exploration and technical studies are warranted.
- RPA has reviewed and concurs with Niobay’s proposed exploration programs and budgets. Phase II of the recommended work will include 12,000 m of drilling focused on continuing to upgrade portions of the Inferred Resources to Indicated Resources and extending the Mineral Resources at depth.

26.2 Mining

- It is recommended to weigh the advantages and disadvantages of each mining scenario and commit to one for the nest stage as this will influence the supporting technical work required in particular geotechnical studies.
- It is recommended to initiate geotechnical studies with supporting field work and rock characterization programs to better support mine design criteria depending on the scenario (i.e. pit slope criteria, stope sizing, pillar dimensions, etc.)

- Use of waste rock for construction for should be further investigated and characterized (i.e. suitability for aggregate, fill material for platforms and embankment construction).
- Additional mine planning optimization opportunities should be investigated such as a better positioning and design of infrastructure locations to reduce haulage cycle times, optimizing grade bins and stockpiling strategy.

26.3 Metallurgy and Processing

- Additional metallurgical test work.
- Considering the higher grade of the composites compared to the grade of the resources, especially for the Open Pit Scenario, further testing will be needed to validate results with different ore grades and different location inside the deposit.
- The next testing phase should include alternative reagents utilization and optimization. In parallel, locked cycle tests need to be performed to adjust reagents addition and consumption, reproduce the results achieved and attempt to improve niobium recovery.
- Based on the locked cycle test results, a pilot plant test is recommended to validate and increase the level of confidence in the flowsheet and the results. The pilot plant results will also provide data for plant sizing and should produce sufficient concentrate for conversion tests to ferroniobium.
- Niobium recovery and contaminants mapping over the deposit still continues.

26.4 Infrastructure and Power Supply

- A tailings management facility design and geotechnical study is recommended with an evaluation of suitability of waste rock from the open pit.
- A water balance model and water management strategy for the complete site is recommended which includes the creek diversion around the open pit.
- The tunnel contemplated underneath Moose River requires further investigation on its feasibility and cost.

26.5 Project Execution

- Preliminary project execution to define construction requirements.
- Trade-off study for the construction/operations strategy.

26.6 Recommended Work Program

Details of the recommended work program estimated at CAD 7.9M is detailed in Table 26.1.

Table 26.1: Proposed Work Program Budget

Work Program	Cost (k CAD)
Exploration	
Drilling Costs (12,000M)	3,900
Pre-Feasibility Study	
Metallurgical Test Work Program	400
Geotechnical Investigations	250
Geological Modelling & Resource Estimation	100
Mining Engineering & Reserve Estimation	300
Process Plant Engineering	200
Balance of Plant Engineering	350
Tailings & Water Management Study	400
Other	100
Sub-Total	**2,100**
Environmental, Social & Permitting	
EIS Baseline Study	1,400
First Nations Engagement	500
Sub-Total	**1,900**
Total	**7,900**

27. REFERENCES

Bennett, G., Brown, D.O., George, P.T., and Leahy, E.J., 1967: Operation Kapuskasing; Ontario Department of Mines, Miscellaneous Paper 10, 98 p.

Berger, V.I., Singer, D.A., and Orris, G.J., 2009: Carbonatites of the World. Explored Deposits of Nb and REE – Databases and Grade and Tonnage Models. USGS Open File 2009-1139.

Bergmann, H.J., 1970: Report on Geophysical Surveys, Kesagami Lake area, Ontario. A report prepared for Argor Explorations Limited by Prospecting Geophysics Ltd., 9 p.

Boland, A.V., Ellis, R.M., Northey, D.J., West, G.F., Green, A.G., Forsyth, D.A., Mereu, R.F., Meyer, R.P., Morel-a-l'Huissier, Buchbinder, G.G.R., Asudeh, I., and Haddon, R.W., 1988: Seismic delineation of upthrust Archean crust in Kapuskasing, northern Ontario; Nature, v.335, pp. 711-713.

Boyko, W.P., 1966: Project Terrane Report on Geological Survey of Consolidated Morrison, Argor and Goldray Concessions Area, James Bay Region. A report prepared by Argor Explorations Limited.

Camet Metallurgy Incorporated, 2017: Ferro-Niobium Market Review. A report prepared for NioBay Metals Inc., 15 p.

Canadian Bechtel Limited, 1969: Project Terrane Feasibility Report and Cost Estimate, James Bay Lowlands Area. A report prepared for Argor Explorations Limited by Canadian Bechtel Limited.

Canadian Bechtel Limited, 1967: Project Terrane Interim Report. A report prepared for Argor Explorations Limited by Canadian Bechtel Limited, 31 p.

De Carle, R.J., 1970: Airborne Electromagnetic and Magnetic Report on Area 1970-43, Kesagami Lake area, Ontario. A report prepared for Argor Explorations Limited by Questor Surveys Limited.

EEM Inc., 2017: Environmental and Social Information for James Bay Niobium Project NI43-101 Report. A report prepared NioBay Metals Inc., 22 p.

Forsyth, D.A., and Morel, P., 1982: Comparative study of the geophysical and geological information in the Timiskaming-Kapuskasing area; *in* Program and Abstracts, Geological Association of Canada-Mineralogical Association Canada, v.7, p. 49.

Geological Survey of Canada, 1984: Magnetic anomaly map, Timmins, Map NM-17-M, scale 1:1,000,000.

Gibb, R.A., 1978: A gravity survey of James Bay and its bearing on the Kapuskasing Gneiss Belt, Ontario. Tectonophysics, v.45, pp. T-7-T-13.

Innis, M.J.S., 1960: Gravity and isostacy in northern Ontario and Manitoba; Canadian Department of Mines and Technical Surveys, Dominion Observatories, v.21, no.6, pp. 27-46.

Marshall, I.B. and Schutt, P.H., 1999: A national ecological framework for Canada – Overview. A co-operative product by Ecosystems Science Directorate, Environment Canada and Research Branch, Agriculture and Agri-Food Canada.

Nothey, D.J., and West, G.F., 1986: A crustal scale seismic refraction experiment over the Kapuskasing structural zone, i*n* Program with Abstracts, Geological Association of Canada-Mineralogical Association of Canada-Canadian Geophysical Union, v.11, p.108.

Ontario Department of Mines-Geological Survey of Canada, 1970: Albany River, Algoma, Cochrane, Kenora, Sudbury, Thunder Bay and Timiskaming; Ontario Department of Mines, Miscellaneous Paper 134, pp. 219-232.

Joly, M, 2020: 2020 Winter Exploration Drilling Program, James Bay Niobium Project Moosonee, Ontario. A report prepared by IOS Services Geoscientifiques Inc., 2020 for NioBay Metals Inc.

Patterson, N.R., 1969: Report on Ground E.M., Moose River Area, Cochrane District for Argor Explorations Limited. A report prepared by Huntec Limited.

Patterson, N.R., and Lane, P.E., 1966: Report on Combined Ground Geophysical Surveys, Moosonee, Ontario for Argor Explorations Limited. A report prepared by Huntec Limited.

Pelletier, P., 2017: Review of the metallurgical data from the James Bay project. A memorandum prepared for NioBay Metals Inc., 10 p.

Pelton, W.H., and Bell, R.A., 1969: Report on the Electromagnetic and Magnetic Surveys, Kesagami Lake Project, Porcupine Mining District, Ontario. A report prepared for Argor Explorations Limited by McPhar Geophysics.

Percival, J.A., 1987: The Kapuskasing uplift: Archean greenstones and granulites; Institute of Lake Superior Geology, 33rd Annual Meeting, v.33, pt.5, 54 p.

Percival, J.A., and Card, K.D., 1983: Archean crust as revealed in the Kapuskasing uplift, Superior Province, Canada. Geology v.11, pp. 323-326.

Percival, J.D., and McGrath, P.H., 1986: Deep crustal structure and tectonic history of the northern Kapuskasing uplift of Ontario, an integrated petrological-geophysical study. Tectonics, v.5, no.4, pp.553-572.

Richardson, D.G., and Birkett, T.C., 1996: Carbonatite-associated deposits *in* Geology of Canadian Mineral Deposit Types, (ed.) O.R. Eckstrand, W.D. Sinclair, and R.T. Thorpe; Geological Survey of Canada, Geology of Canada no. 8, pp.541-558.

Roscoe Postle Associates Inc. (RPA)/SRL Consulting (Canada) Ltd, D. El-Rassi, M. Sc., P.Eng., August 2020: Technical Report on the James Bay Niobium Project, Cochrane District, Northeastern Ontario, Canada, Niobay Metal Inc. Project #3263, 129 p.

Sage, R.P., 1991: Alkalic Rock, Carbonatite and Kimberlite Complexes of Ontario, Superior Province *in* Geology of Ontario, Ontario Geological Survey, Special Volume 4, Part 1, pp. 683-709.

Sage, R.P., 1988: Geology of Carbonatite – Alkalic Rock Complexes in Ontario: Argor Carbonatite Complex, District of Cochrane; Ontario Geological Survey, Study 41, 90 p.

Sage, R.P., and Watkinson, D.H., 1991: Alkalic rock – Carbonatite Complexes of the Superior Structural Province, northern Ontario, Canada. Chronique de la Recherche Minière, Vol. No. 504, pp. 5-19.

SGS Minerals, February 27, 2018, Analysis of drill croe samples and flotation of niobium mineral from ore from the James Bay property, Niobay Metal Inc., Project 16075-001- Final Report

Stockford, H.R., 1972: The James Bay Pyrochlore Deposit. CIM Bulletin, June 1972, pp. 61-69.

Stockford, H.R., 1970: Project Terrane, the Alpha-B Columbium Deposit, James Bay Lowlands, N.E. Ontario. An unpublished report prepared for Argor Explorations Ltd., 40 p.

Thurston, P.C., Siragusa, G.M., and Sage, R.P., 1077: Geology of the Chapleau area, districts of Algoma, Sudbury and Cochrane, Ontario Division of Mines, Geological Report 157, 293 p.

Twyman, J.D., 1983: The generation and crystallization and differentiation of carbonatite magmas: Evidence from the Argor and Cargill complexes. Unpublished Ph. D. thesis, University of Toronto, Toronto, Ontario, 248 p.

Wagg, D.M., 1966: Report on Airborne Geophysical Survey of the Morrison, Argor and Goldray Concessions for Argor Explorations Ltd. A report prepared by Canadian Aero Mineral Surveys Limited.

Watson, J., 1980: The origin and history of the Kapuskasing structural zone, Ontario, Canada; Canadian Journal of Earth Sciences, v.17, pp. 866-876.

Wilson, H.D.B., and Brisbin, W.C., 1965: The mid-North American ridge structure; *in* Geological Society of America Abstracts for 1965, Special Paper no.87, pp. 180-187.

Wooley, A.R., 1989: The spatial and temporal distribution of carbonatites, *in* Carbonatites, Genesis and Evolution, Unwin Hyman, London, pp. 15-37.