MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA
PRELIMINARY ECONOMIC ASSESSMENT
NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT

PREPARED FOR: FOX RIVER RESOURCES CORPORATION

REPORT DATE: JUNE 1, 2022

EFFECTIVE DATE: APRIL 21, 2022

REPORT PREPARED BY:

Hatch Limited
Ausenco PSI
Chemetics Inc.
DMT (UK) Limited
JESA Technologies

QUALIFIED PERSONS:

Ken Armstrong (Chemetics Inc)
Steve Ball (Hatch Limited)
Michael Bobotis (Hatch Limited)
Rafael Dávila (Hatch Limited)
Tim Horner (DMT (UK) Limited)
David Ivell (JESA Technologies)
Mike Kelahan (JESA Technologies)
Richard May (Hatch Limited)
Kelly Snyder (Ausenco PSI)

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QUALIFIED PERSON STATEMENTS

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **FOX RIVER RESOURCES CORPORATION, MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA, PRELIMINARY ECONOMIC ASSESSMENT REPORT**, **NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT** effective date **21 APRIL 2022** (hereinafter the "Technical Report")

I, **KEN ARMSTRONG, P.Eng** do hereby certify that:

1. I am a **PROFESSIONAL ENGINEER** with **CHEMETICS INC**.at **SUITE 200, 2930 VIRTUAL WAY, VANCOUVER, BC, CANADA V5M 0A5**.
2. I graduated from the **UNIVERSITY OF BRITISH COLUMBIA** with a **BSC in CHEMISTRY** in **1987**, a **BACHELOR'S DEGREE IN CHEMICAL ENGINEERING** in **1988** and **a MASTER'S OF BUSINESS ADMINISTRATION** in **1997**.
3. I am a registered member of the **ASSOCIATION OF PROFESSIONAL ENGINEER'S AND GEOSCIENTISTS OF BRITISH COLUMBIA**, under Registration No. 17500. My principal experience is in the areas of **CHEMICAL ENGINEERING AND COST ESTIMATION**.
4. I have practiced as a Professional Engineer for 32 years.
5. I have **NOT** personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for **ALL SECTIONS LISTED UNDER MY NAME IN TABLE i BELOW** and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. Neither I, nor any affiliated entity of mine, own directly or indirectly, nor expect to receive any interest in the properties of securities of Fox River Resources Corporation, or any associated or affiliated companies.
11. I have read National Instrument 43-101 and the sections of the Technical Report referenced in Paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
12. As of the effective date of this Certificate, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to ensure the Technical Report not misleading.

Signed and sealed this 1st day of June 2022

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Ken Armstrong, P.Eng
Principal Sulfuric Acid Plant Engineer

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **FOX RIVER RESOURCES CORPORATION, MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA, PRELIMINARY ECONOMIC ASSESSMENT REPORT**, **NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT**, effective date **21 APRIL 2022** (hereinafter the "Technical Report")

I, **STEPHEN DAVID BALL, P.Eng** do hereby certify that:

1. I am a **PROJECT MANAGER - MINING** with the firm of **HATCH LIMITED** with an office located at **2800 SPEAKMAN DRIVE, MISSISSAUGA, ONTARIO, CANADA L5K 2R7**.
2. I am a graduate of the **UNIVERSITY OF WALES (CARDIFF, WALES, UK)**, where, in **1982** I obtained a **BSc** in **MINING GEOLOGY** through the **MINERAL EXPLORATION DEPARTMENT**.
3. I have practiced my current profession continuously since **1984**. My principal experience is in the areas of **MINE GEOLOGY, MINE ENGINEERING, MINE OPERATIONS, PROJECT MANAGEMENT**.
4. I am a professional **ENGINEER** registered with the **PROFESSIONAL ENGINEERS OF ONTARIO, CANADA**.
5. I have **NOT** personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for **ALL SECTIONS LISTED UNDER MY NAME IN TABLE i BELOW** and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in Paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 1st day of June 2022

STEPHEN BALL
PROJECT MANAGER – MINING

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **FOX RIVER RESOURCES CORPORATION, MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA, PRELIMINARY ECONOMIC ASSESSMENT REPORT**, **NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT**, effective date **21 APRIL 2022** (hereinafter the "Technical Report")

I, **MICHAEL BOBOTIS, Eng.** do hereby certify that:

1. I am a **SENIOR MINING ENGINEER** with the firm of **HATCH LIMITED** with an office located at **5 PLACE VILLE MARIE SUITE 1400, MONTRÉAL, QC H3B 2G2**.
2. I am a graduate of **MCGILL UNIVERSITY (MONTREAL, QC, CANADA)**, where, in **2012** I obtained a **B.ENG.** in **MINING ENGINEERING** through the **MINING & MATERIALS ENGINEERING DEPARTMENT**.
3. I have practiced my current profession continuously since **2012**. My principal experience is in the areas of **MINE ENGINEERING, MINE DESIGN & PLANNING, MINE DEVELOPMENT & PROJECT MANAGEMENT**.
4. I am a professional **ENGINEER** registered with the **ORDER OF ENGINEERS OF QUEBEC, CANADA (OIQ REGISTRATION #5039888)**.
5. I have **NOT** personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for **ALL SECTIONS LISTED UNDER MY NAME IN TABLE i BELOW** and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in Paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 1st day of June 2022

MICHAEL BOBOTIS, ENG.
SENIOR MINING ENGINEER

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **FOX RIVER RESOURCES CORPORATION, MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA, PRELIMINARY ECONOMIC ASSESSMENT REPORT**, **NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT**, effective date **21 APRIL 2022** (hereinafter the "Technical Report")

I, **RAFAEL S. DAVILA** do hereby certify that:

1. I am a **PRINCIPAL GEOTECHNICAL ENGINEER AND GLOBAL PRACTICE DIRECTOR FOR MINE WASTE MANAGEMENT** with the firm of **HATCH LIMITED** with an office located at **2800 SPEAKMAN DRIVE, MISSISSAUGA, ONTARIO, CANADA L5K 2R7**.
2. I am a graduate of:
   - **a. University of Alberta, where, in 2000 I obtained an MBA degree in Business Administration (Energy and Natural Resources) through the Faculties of Business Administration and Graduate Studies.**
   - **b. University of Alberta, where, in 1992 I obtained an M.Sc. degree in Civil Engineering (Geotechnique) through the Civil Engineering Dept. and Faculty of Graduate Studies.**
   - **c. Universidade Federal de Ouro Preto, where, in 1988 I obtained an Ingenieur degree in Civil Engineering through the Dept of Civil Engineering at the associated School of Mines of Ouro Preto**.
3. I have practiced my current profession continuously since **1989**. My principal experience is in the areas of **GEOTECHNICAL ENGINEERING APPLIED TO MINE WASTE MANAGEMENT AND MINE CLOSURE**.
4. I am a professional **ENGINEER** registered with the **PROFESSIONAL ENGINEERS OF ONTARIO, CANADA**.
5. I have **NOT** personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for **ALL SECTIONS LISTED UNDER MY NAME IN TABLE i BELOW** and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.

10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in Paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 1st day of June 2022

Rafael S. Dávila, P.Eng.
Principal Geotechnical Engineer and GPD Tailings

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CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **FOX RIVER RESOURCES CORPORATION, MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA, PRELIMINARY ECONOMIC ASSESSMENT REPORT**, **NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT**, effective date **21 APRIL 2022** (hereinafter the "Technical Report")

I, **TIMOTHY HORNER** do hereby certify that:

1. I am an **ASSOCIATE CONSULTANT** with the firm of **DMT (UK) LIMITED** with an office located at **SHERWOOD BUSINESS PARK, ANNESLEY, NOTTINGHAMSHIRE, ENGLAND, UK**.
2. I am a graduate of the **UNIVERSITY OF WALES (CARDIFF, WALES, UK)**, where, in **1977 I** obtained a **BSc** in **GEOLOGY** through the **GEOLOGY DEPARTMENT**.
3. I am a graduate of **CAMBORNE SCHOOL OF MINES, (CORNWALL, ENGLAND, UK**) where, in **1984** I obtained a **MSc** in **MINING GEOLOGY** through the **MINING DEPARTMENT**.
4. I have practiced my current profession continuously since **1984**. My principal experience is in the areas of **MINE GEOLOGY, MINERAL EXPLORATION, ENGINEERING GEOLOGY & PROJECT MANAGEMENT**.
5. I am a professional **GEOSCIENTIST** registered with the **PROFESSIONAL GEOSCIENCE ASSOCIATION OF ONTARIO, CANADA**.
6. I have personally inspected the subject property. on **OCTOBER 21 TO OCTOBER 23, 2014**.
7. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
8. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
9. I am a contributing author of the Technical Report and am responsible for **ALL SECTIONS LISTED UNDER MY NAME IN TABLE i BELOW** and accept professional responsibility for these sections of the Technical Report.
10. I have had prior involvement with the subject property.

    SITE GEOLOGIST & PROJECT MANAGER FOR DRILL PROGRAM IN 2008.

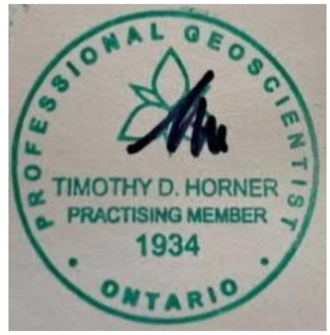
    PROJECT MANAGER FOR DRILL PROGRAM IN 2012.

    QUALIFIED PERSON FOR TECHNICAL REPORT AND MINERAL RESOURCE ESTIMATE ISSUED ON BEHALF OF PHOSCAN CHEMICAL CORP MARCH 2015 AND ALSO FOR THE TECHNICAL REPORT SUBSEQUENTLY RE-ISSUED ON BEHALF OF FOX RIVER RESOURCES CORPORATION IN APRIL 2016.

    I have read National Instrument 43-101 and the sections of the Technical Report referenced in Paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 1st day of June 2022



TIMOTHY HORNER
ASSOCIATE CONSULTANT – GEOLOGY

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **FOX RIVER RESOURCES CORPORATION, MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA, PRELIMINARY ECONOMIC ASSESSMENT REPORT**, **NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT**, effective date **21 APRIL 2022** (hereinafter the "Technical Report")

I, **DAVID MARK IVELL** do hereby certify that:

1. I am a **CHEMICAL PROCESS ENGINEER** with the firm of **JESA TECHNOLOGIES LLC,** with an office located at **3149 WINTERLAKE ROAD, LAKELAND, FL 33803**.
2. I am a graduate of the **IMPERIAL COLLEGE, LONDON, UK**, where, in **1975** I obtained a **BSc**. in **CHEMICAL ENGINEERING**.
3. I have practiced my current profession continuously since **1975**. My principal experience is in the areas of **PHOSPHATE FERTILIZERS**.
4. I am a **CHARTERED ENGINEER** registered with the **UK INSTITUTE OF CHEMICAL ENGINEERS.** My membership number is 221899.
5. I have **NOT** personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for **ALL SECTIONS LISTED UNDER MY NAME IN TABLE i BELOW** and accept professional responsibility for these sections of the Technical Report.
9. As an employee of Jacobs Engineering, I was previously involved in the **2007/2008 PHOSCAN PFS** related to granulation.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in Paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed this 1st day of June 2022

DAVID IVELL

QP FERTILIZER CONVERSION COMPLEX

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **FOX RIVER RESOURCES CORPORATION, MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA, PRELIMINARY ECONOMIC ASSESSMENT REPORT**, **NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT**, effective date **21 APRIL 2022** (hereinafter the "Technical Report")

I, **MICHAEL KELAHAN** do hereby certify that:

1. I am a **METALLURGICAL ENGINEER - BENEFICIATION** with the firm of **JESA TECHNOLOGIES LLC**, with an office located at **3149 WINTER LAKE ROAD, LAKELAND, FLORIDA, USA, 33803**.
2. I am a graduate of the **UNIVERSITY OF MISSOURI, ROLLA, MISSOURI**, where, in 1969, I obtained a BSc in **METALLURICAL ENGINEERING**. I also am a graduate of the **UNIVERSITY OF UTAH, SALT LAKE CITY, UTAH,** where, in 1971, I obtained a MSc. in **METALLURIGICAL ENGINEERING**.
3. I have practiced my current profession continuously since 1971. My principal experience is in the areas of **PHOSHPATE BENEFICIATION AND MINERAL PROCESSING**.
4. I am a **QUALIFIED PROFESSIONAL (QP) MEMBER**, registered with the **MINING AND METALLURGICAL SOCIETY OF AMERICA (MMSA)**. My member number is **01446QP**.
5. I have personally inspected the subject property in **JUNE 2007**.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, I am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for **ALL SECTIONS LISTED UNDER MY NAME IN TABLE i BELOW** and accept professional responsibility for these sections of the Technical Report.
9. As an employee of Jacobs Engineering, I was previously involved in the 2007/2008 PhosCan PFS relating to beneficiation.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in Paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed this 1st day of June 2022

MICHAEL E. KELAHAN
METALLURGICAL ENGINEER

PHOSPHATE BENEFICIATION

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **FOX RIVER RESOURCES CORPORATION, MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA, PRELIMINARY ECONOMIC ASSESSMENT REPORT**, **NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT**, effective date **21 APRIL 2022** (hereinafter the “Technical Report”)

I, **RICHARD ANTHONY JOHN MAY** do hereby certify that:

1. I am a **SENIOR PROCESS CONSULTANT** with the firm of **HATCH LIMITED** with an office located at **291-121 RESEARCH DRIVE, SASKATOON, SASKATCHEWAN, CANADA S7N 1K2**.
2. I am a graduate of the **ROYAL SCHOOL OF MINES, IMPERIAL COLLEGE, UNIVERSITY OF LONDON (LONDON, ENGLAND, UK)** where, in **1973** I obtained a **BSc** in **MINERALTECHNOLOGY** through the **MINING DEPARTMENT**.
3. I have practiced my current profession continuously since **1973**. My principal experience is in the areas of **MINERAL PROCESSING, PROCESS ENGINEERING, PLANT OPERATIONS, PROJECT MANAGEMENT**.
4. I am a professional **ENGINEER** registered with the ASSOCIATION OF **PROFESSIONAL ENGINEERS AND GEOSCIENTISTS OF SASKATCHEWAN, CANADA**.
5. I have **NOT** personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for **ALL SECTIONS LISTED UNDER MY NAME IN TABLE i BELOW** and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in Paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 1st day of June 2022

RICHARD MAY

SENIOR PROCESS CONSULTANT

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **FOX RIVER RESOURCES CORPORATION, MARTISON PHOSPHATE PROJECT, ONTARIO, CANADA, PRELIMINARY ECONOMIC ASSESSMENT REPORT**, **NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT**, effective date **21 APRIL 2022** (hereinafter the "Technical Report")

I, **KELLY SNYDER** do hereby certify that:

1. I am a **PRINCIPAL PIPELINE ENGINEER** with the firm of **AUSENCO PSI** with an office located at **4071 PORT CHICAGO HWY, STE 120, CONCORD, CA 94520, USA**.
2. I am a graduate of the **COLORADO SCHOOL OF MINES**, where, in **1997** I obtained a **BS** in **METALLURGY AND MATERIALS ENGINEERING**.

   I am a graduate of the **NEW YORK INSTITUTE OF TECHNOLOGY**, where, in **2009** I obtained a **MS** in **COMPUTER SCIENCE**.
3. I have practiced my current profession since **2000**. My principal experience is in the areas of **SLURRY PIPELINE TRANSPORT**.
4. I am a professional **ENGINEER** registered with the **BOARD OF PROFESSIONAL ENGINEERS, CALIFORNIA**.
5. I have **NOT** personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for **ALL SECTIONS LISTED UNDER MY NAME IN TABLE i BELOW** and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in Paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 1st day of June 2022

KELLY SNYDER

PRINCIPAL PIPELINE ENGINEER

IMPORTANT NOTICE TO READER

This report was prepared by the qualified persons (QPs) listed in "Table i" below. Each QP assumes responsibility for those sections or areas of this report that are referenced opposite their name. None of the QPs, however, accept any responsibility or liability for the sections or areas of this report that were prepared by other QPs. This report was prepared to allow Fox River Resources Corporation (the "Owner") to reach informed decisions respecting the development of the Martison Phosphate Project. Except for the purposes legislated under provincial securities law and without limiting any statutory right afforded thereunder, any use of this report by any third party is at that party's sole risk, and except pursuant to provincial securities laws, none of the contributors shall have any liability to any third party for any such use for any reason whatsoever, including negligence. This report is intended to be read as a whole, and sections should not be read or relied upon out of context. This report contains estimates, projections and conclusions that are forward-looking information within the meaning of applicable securities laws. Forward-looking statements are based upon the responsible QP's opinion at the time that they are made but, in most cases, involve significant risk and uncertainty. Although each of the responsible QPs has attempted to identify factors that could cause actual events or results to differ materially from those described in this report, there may be other factors that cause events or results to not be as anticipated, estimated or projected. None of the QPs undertake any obligation to update any information contained in this report, including, without limitation, any forward-looking information.

Table i: Qualified Persons who contributed to this Technical Report

Section	Section Title	Contributing Company	Qualified Person
1	**SUMMARY**		
1.1	Introduction	Hatch	Stephen Ball
1.2	Project Location	DMT	Tim Horner
1.3	History	DMT	Tim Horner
1.4	Geology & Mineralization	DMT	Tim Horner
1.5	Mineral Resource Estimate	DMT	Tim Horner
1.6	Mining Methods	Hatch	Michael Bobotis
1.7	Metallurgical Testing	JT	Michael Kelahan
1.8	Recovery Methods	JT	Michael Kelahan
1.9	Infrastructure	See below	See below
1.9.1	Mine Site Infrastructure	Hatch	Stephen Ball
1.9.2	Fertilizer Conversion Complex	JT	David Ivell
1.10	Market Studies & Contracts	Michael R Rahm Consulting LLC	Richard May
1.11	Environmental Considerations	Hatch & DMT	Stephen Ball
1.12	Capital & Operating Costs	Hatch & JT	Stephen Ball & David Ivell
1.13	Economic Analysis	JT	David Ivell
1.14	Conclusions and Recommendations	Multiple contributors	Stephen Ball

Section	Section Title	Contributing Company	Qualified Person
2.	**INTRODUCTION**		
2.0	Introduction	Hatch	Stephen Ball
3	**RELIANCE ON OTHER EXPERTS**		
3.0	Reliance On Other Experts	Hatch	Stephen Ball
4	**PROPERTY DESCRIPTION & LOCATION**		
4.0	Property Description & Location	DMT	Tim Horner
5	**ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY**		
5.0	Accessibility, Climate, Local Resources, Infrastructure & Physiography	DMT	Tim Horner
6	**HISTORY**		
6.0	History	DMT	Tim Horner
7	**GEOLOGICAL SETTING & MINERALIZATON**		
7.0	Geological Setting & Mineralization	DMT	Tim Horner
8	**DEPOSIT TYPES**		
8.0	Deposit Types	DMT	Tim Horner
9	**EXPLORATION**		
9.0	Exploration	DMT	Tim Horner
10	**DRILLING**		
10.0	Drilling	DMT	Tim Horner
11	**SAMPLE PREPARATION, ANALYSIS & SECURITY**		
11.0	Sample Preparation, Analysis & Security	DMT	Tim Horner
12	**DATA VERIFICATION**		
12.0	Data Verification	DMT	Tim Horner
13	**MINERAL PROCESSING & METALLURGICAL TESTING**		
13.0	Mineral Processing & Metallurgical Testing	JT	Michael Kelahan
14	**MINERAL RESOURCE ESTIMATES**		
14.0	Mineral Resource Estimates	DMT	Tim Horner
15	**MINERAL RESERVE ESTIMATES**		
15.0	Mineral Reserve Estimates	Not applicable	Not applicable
16	**MINING METHODS**		
16.0	Mining Methods	Hatch	Michael Bobotis
17	**RECOVERY METHODS**		
17.0	Recovery Methods	JT	Michael Kelahan
18	**PROJECT INFRASTRUCTURE**		
18.1	**Mine Site Infrastructure**		
18.1.1	Access Roads	Hatch	Stephen Ball
18.1.2.	Electrical Transmission Line & Site Power	Hatch	Stephen Ball
18.1.3	Beneficiation Plant	JT	Mike Kelahan
18.1.4	Utilities	Hatch	Stephen Ball
18.1.5	Ancillary Buildings & Services	Hatch	Stephen Ball
18.1.6	Site Preparation	Hatch	Stephen Ball
18.1.7	Tailings Impoundment	Hatch	Rafael Dávila
18.1.8	Site Water Management	Hatch	Rafael Dávila
18.1.9	Concentrate Slurry Pipeline	Ausenco PSI	Kelly Snyder
18.2	**Fertilizer Conversion Complex**		
18.2.1	Sulfuric Acid Plant	Chemetics	Ken Armstrong
18.2.2	Phosphoric Acid Plant	JT	David Ivell
18.2.3	Super Phosphoric Acid Plant	JT	David Ivell
18.2.4	Granulation Plant	JT	David Ivell
18.2.5	Infrastructure – Process Support	JT	David Ivell
18.2.6	Access Roads	JT	David Ivell
18.2.7	Rail Yard	AECOM	David Ivell
18.2.8	Cogeneration Plant and FCC Distribution	JT	David Ivell

Section	Section Title	Contributing Company	Qualified Person
18.2.9	Natural Gas Supply	JT	David Ivell
18.2.10	Utilities	JT	David Ivell
18.2.11	Ancillary Buildings and Services	JT	David Ivell
18.2.12	Site Preparation	JT	David Ivell
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Acronyms and Abbreviations

Term	Description
AACE	Association for the Advancement of Cost Engineering (AACE International)
AAPFCO	Association of American Plant Food Control Officials
AMSL	Above Mean Sea Level
ANFO	Ammonium Nitrate / Fuel Oil
aP_2O_5	Apatite Phosphorous Pentoxide
APP	Ammonia Polyphosphate Solution (10-34-0 or 11-37-0)
ASF	African Swine Fever
Baltic	Baltic Resources Inc. A wholly owned subsidiary of Fox River Resources Corporation
b/d	Barrels Per Day
BFW	Boiler Feed Water
BPL	Bone Phosphate of Lime. By convention, 2.185 x % P_2O_5 also known as Tricalcium Phosphate (TCP)
B/L	Battery Limit
bt	Billion Tonnes
CAD	Canadian Dollar
CaO	Calcium Oxide
CAGR	Compound Annual Growth Rate
CAPEX	Capital Cost Estimate – initial required investment in the construction and pre-production phase
CAGR	Compound Annual Growth Rate
cfr	Cost and Freight
CI	Citrate-Insoluble P_2O_5
CLFN	Constance Lake First Nation
cP	Centipoise (Unit of Viscosity)
CRU	CRU Group or CRU International Ltd. – privately owned business intelligence company
CS	Citrate-Soluble P_2O_5
CVD	Countervailing Duties
Cw	Weight Concentration
DAP	Diammonium Phosphate $(NH_4)_2HPO_4$ (18-46-00)
dBA	Decibel
DFO	Department Of Fisheries and Oceans Canada
DMT	DMT Consulting Limited. An internal mining consultant engaged previously to perform work for Fox River Resources Corporation
dmth	Dry Metric Tonnes Per Hour
ea	Each
EA	Environmental Assessment
EDF	Environmental Design Flood
EPCM	Engineering Procurement and Construction Management service provider

Term	Description
ESR	Environmental Study Report
EV	Electric Vehicle
FCC	Fertilizer Conversion Complex
fob	Free On Board
FRP	Fiber Reinforced Plastic
FSA	Fluosilicic Acid
G&A	General and Administrative
GHG	Greenhouse Gases
GPS	Global Positioning System
h (or hr)	Hour
HDPE	High Density Polyethylene
HMI	Human-Machine Interface
HP	High Pressure
h:v	Horizontal to Vertical ratio
HxGN	Hexagon AB
h/y	Hours per year
IFA	International Fertilizer Association
IP	Induced Polarization
IRR	Internal Rate of Return
ISBL	Inside Scope Battery Limit
km	Kilometre
KP	Kilometre Point
kPa	Kilopascal
kt	Kilo Metric Tonnes
ktpa	Kilo Metric Tonnes Per Annum
ktpy	Kilo Metric Tonnes Per Year (same as ktpa)
L	Litres
LFP	Lithium Iron Phosphate
LIMS	Low Intensity Magnetic Separation
LOM	Life Of Mine
LP	Low Pressure
L/s	Litres per second
M-m^3	Millions of cubic metres
Ma	Millions Of Years in The Past
MAP	Monoammonium Phosphate ($NH_4H_2PO_4$) (11-52-00)
Mg/Nm3	Milligrams Per Normal Cubic Meters
MCC	Motor Control Centre

Term	Description
MECP	Ministry Of Environment, Conservation and Parks
MER	Minor Element Ratio = (%Fe_2O_3 + % Al_2O_3 + % MgO)/% P_2O_5. Used as a predictor of phosphoric acid quality.
MGA	Merchant Grade Acid
MHSCTI	Ministry of Heritage, Sport, Culture and Tourism Industries
MLAS	Lands Administration System managed by the NDMNRF
MP	Medium Pressure
MPP	Martison Phosphate Project
MRE	Mineral Resource Estimate
MRRC	Michael. R. Rahm. Consulting
m/s	Metres Per Second
Mtpa	Million Metric Tonnes Per Annum
Nb_2O_5	Niobium Pentoxide
NDMNRF	Ministry of Northern Development, Mines, Natural Resources and Forestry
NPK	Nitrogen Phosphorous Potassium
NPV	Net Present Value
NPS	Nitrogen Phosphate Sulfur Based Fertilizers (12-40-0-S)
NRDC	National Development & Reform Commission
NUE	Nutrient Use Efficiency
ONR	Ontario Northland Railway
OPEX	Operating Cost Estimate
P_2O_5	Phosphorous Pentoxide
PAP	Phosphoric Acid Plant
P&ID	Piping And Instrumentation Diagram
PD	Positive Displacement
PEA	Preliminary Economic Assessment
PFD	Process Flow Diagram
PFS	Prefeasibility Study specifically the Martison Phosphate Project Report from 2008
ppm	Parts Per Million
PSD	Particle Size Distribution
PVC	PolyVinyl Chloride
QP	Qualified Person
RC	Reverse Circulation
RD	Renewable Diesel
REE	Rare Earth Elements
RLCS	Rubber Lined Carbon Steel
RO	Reverse Osmosis

Term	Description
ROM	Run Of Mine
ROW	Right-Of-Way. The area cleared to the tree or bush line for access road.
SAF	Sustainable Aviation Fuel
SAP	Sulfuric Acid Plant
SCADA	Supervisory Control and Data Acquisition
SG&A	Selling, General & Administrative
SMU	Selective Mining Unit
SPA	Super Phosphoric Acid (68-72% P_2O_5)
SSP	Single Superphosphate
SUSEX	Sustaining Capital Cost Estimate – required capital expenditures in the production phase
t	Metric Tonne = 1000 kg
TAM	Total Addressable Market
t/d	Metric Tonnes Per Day
TFI	The Fertilizer Institute
t/h	Metric Tonnes Per Hour
t/m^3	Metric Tonnes Per Cubic Metre
tMAP	Technical MAP
TMF	Tailings Management Facility
t/y	Metric Tonnes Per Year
TSP	Triple Superphosphate
UPS	Uninterruptable Power Supply
VFD	Variable Frequency Drive
VoIP	Voice Over Internet Protocol
WHIMS	Wet High Intensity Magnetic Separation
WS	Water soluble P_2O_5 which is not recovered during filtration and washing of the gypsum cake. This type of loss will increase as “free sulfate” in the reaction system is either higher or lower than the optimum and/or if P_2O_5 concentration is too high.
YTD	Year To Date
µS/cm	Micro-Siemens per centi-meter (Unit of Conductivity)

Definitions

Term	Description
Acidulation	The process whereby phosphate rock is reacted with a mineral acid to produce phosphoric acid and gypsum. Hemihydrate and dihydrate processes use sulfuric acid and produce gypsum with the formula $CaSO_4.1/2H_2O$ and $CaSO_4.2H_2O$ respectively.
Attack	The reaction or digestion of phosphate rock by acid.
CI or Carbon Insoluble	This type of loss occurs as phosphate rock passes through the attack system but does not react with sulfuric acid. The cause for such losses is usually too high “free sulfate” in the reaction medium. This is compounded by inadequate rock grind and depends also on rock reactivity. In chemical analysis, such losses report as “citrate-insoluble P_2O_5”.
CS or Carbon Soluble	This type of loss occurs as dicalcium phosphate ($CaHPO_4.2H_2O$) co-crystallizes with gypsum. This is also known as “lattice” losses, as substitution of HPO_4 for SO_4 in the gypsum lattice occurs. Co-crystallized losses tend to occur as “free sulfate” is low. In the chemical analysis of the gypsum cake, such losses report as “citrate-soluble P_2O_5” (i.e., insoluble in water but soluble in a neutral ammonium citrate solution).
Dry Tonnes	Metric tonnage calculated without considering in-situ moisture content.
Fertilizer Conversion Complex (FCC)	Mine concentrate processing plant with proposed location near Hearst, Ontario including the phosphate plant, granulation plant, sulfuric acid plant and associated power cogeneration facility.
Filter Acid	The phosphoric acid filtrate product prior to washing the filter cake.
Filter Cake	The gypsum crystals retained on the filter cloth after filtration and washing.
Filtration	The separation of phosphoric acid from gypsum using a porous medium (filter cloth) to retain the gypsum crystals and pass the liquid acid.
Gangue	Typically, non-valuable and unwanted minerals in a mineral deposit that dilutes the grade of the valuable material. For phosphate deposits, typical gangue minerals includes quartz, calcite, and dolomite in addition to minerals containing iron, aluminum, and silicon.
Mill Feed	Mineralized rock material feed to the beneficiation plant
Mine Site	Martison mine site including all facilities and infrastructure
O&M	Operations and Maintenance
Project	The Martison Phosphate Project referring to the project as a whole, including the “Mine site” and the “Fertilizer Conversion Complex” and all other information relating to these locations as contributions to this report.
Reactor	The vessel in which acidulation takes place (also known as attack tank).
Recovery	% of the P_2O_5 in the rock that reports to the product acid.
Residuum	An apatite rich paleo – soil and the main source of the phosphatic material
Reverse Osmosis	A technology that is used to remove a large majority of contaminants from water by pushing the water under pressure through a semi-permeable membrane.
Rock	Phosphate rock (concentrate)
Sludge	Region of liquid phase wherein fine particles are in suspension.

Term	Description
Solids	Sedimented and precipitate solids occupying a defined region in bottom of a clarification process.
Slurry	A mixture of solid and liquid components. The components fed to the reactor are phosphate rock, sulfuric acid, and return acid. The reactor discharge contains gypsum crystals and phosphoric acid.
Storativity	Otherwise known as storage coefficient, is the volume of water released from storage per unit decline in hydraulic head in the aquifer, per unit area of the aquifer.
Sulfate	In this report sulfate is referring to free excess sulfate expressed as SO_3 in g/dm^3.
Tonnes	Metric tonnage (1000 kg)
Waste	All non-economic residue including overburden, other mine open pit material and tailings discharge from the beneficiation plant.
Wet Tonnes	Metric tonnage calculated while considering in-situ moisture content
X Compounds	The iron precipitating as iron (potassium or sodium) phosphates. These complex phosphates precipitate at any concentration of phosphoric acid, but much faster at the higher phosphoric acid concentrations.

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1. Summary

1.1 Introduction

This report has been compiled by Hatch Ltd. (“Hatch”) for Fox River Resources Corporation (the “Company” or “Fox River”) (CSE: FOX) with input from the following independent consultants:

- Ausenco PSI. (“Ausenco”)
- Chemetics Inc.
- DMT Group Consulting Ltd. (“DMT”)
- Fox River Resources Corporation (“Fox River”) – owner contributions
- Hatch Ltd (“Hatch”)
- JESA Technologies (“JT”).

Fox River has commissioned this Preliminary Economic Assessment (“PEA”) and updated Mineral Resource Estimate (“MRE”) for its 100% owned Martison Phosphate Project (the Project”), located near Hearst, Ontario. All currency figures are shown in United States dollars, unless otherwise noted.

The proposed Martison Phosphate Project is a vertically integrated mining and fertilizer complex utilizing an igneous phosphate deposit. The PEA examined the types and quantities of fertilizers which will be produced, the process technology deployed, and the sulfur technology utilized in making fertilizer products from the phosphate concentrate.

The Project design entails an open pit mine, a phosphate beneficiation plant (located at the mine site), a slurry pipeline, a road corridor, and a Fertilizer Conversion Complex (FCC) located west of Hearst, Ontario, and 86 km south of the mine site. The FCC location is in close proximity to existing rail, power, and natural gas infrastructure. This facility includes a phosphoric acid plant, a super phosphoric acid plant, a granulation plant, a sulfur conversion plant with cogeneration capacity, a warehouse and loadout facility, and a railyard.

Based on the current Indicated and Inferred resources, the Project has a 26 year mine life with the potential for extension should additional resources be identified. The PEA examined the economics of producing 221,000 solution tonnes per year of super phosphoric acid (SPA), 474,000 tonnes of granular monoammonium phosphate (MAP) and 247,000 tonnes of granular nitrogen, phosphate + sulfur (NPS) at the proposed FCC.

The target market includes the Eastern Canadian provinces, Canadian Prairie provinces and US northern tier states. The Martison facility will capture a freight advantage relative to US. and offshore producers in its target market, and especially in nearby Canadian provinces where demand is projected to grow and where a larger share of Martison output is forecast to ship over time.

The current total addressable markets (TAMs) for MAP, NPS and SPA are estimated to total 4.0 mt, 2.0 mt and 0.7 mt, respectively.

This report is compliant with National Instrument 43-101 (NI 43-101) Standards of Disclosure for Mineral Projects.

1.2 Project Location

The Martison Carbonatite Complex ("the Carbonatite Complex"), which contains the Project's phosphate and niobium resources, and proposed mine site area are located approximately 70 km northeast of the town of Hearst, Ontario, in the James Bay Lowlands. The mine site area is located in the "South of Ridge Lake" area (township) and centred about 50º18' 52" N., 83º24' 52" W. The site currently remains a "winter access only" site for the purposes of further site works and advancement of the Project.

Figure 1-1: Location of Martison Phosphate Project

The property consists of three Fox River mining leases, numbers LEA108639 (expires 29 April 2032), LEA 108638 (expires 29 April 2032) and LEA 107438 (expires 30 July 2023), and 19 unpatented contiguous mineral claims totalling 265 units, which together comprise approximately 8,256 hectares.

The Project is situated in the large expanse of “low ground” southwest of Hudson/James Bay, referred to as the James Bay Lowlands. The property comprises very gently rolling terrain dominated by muskeg and black spruce swamp.

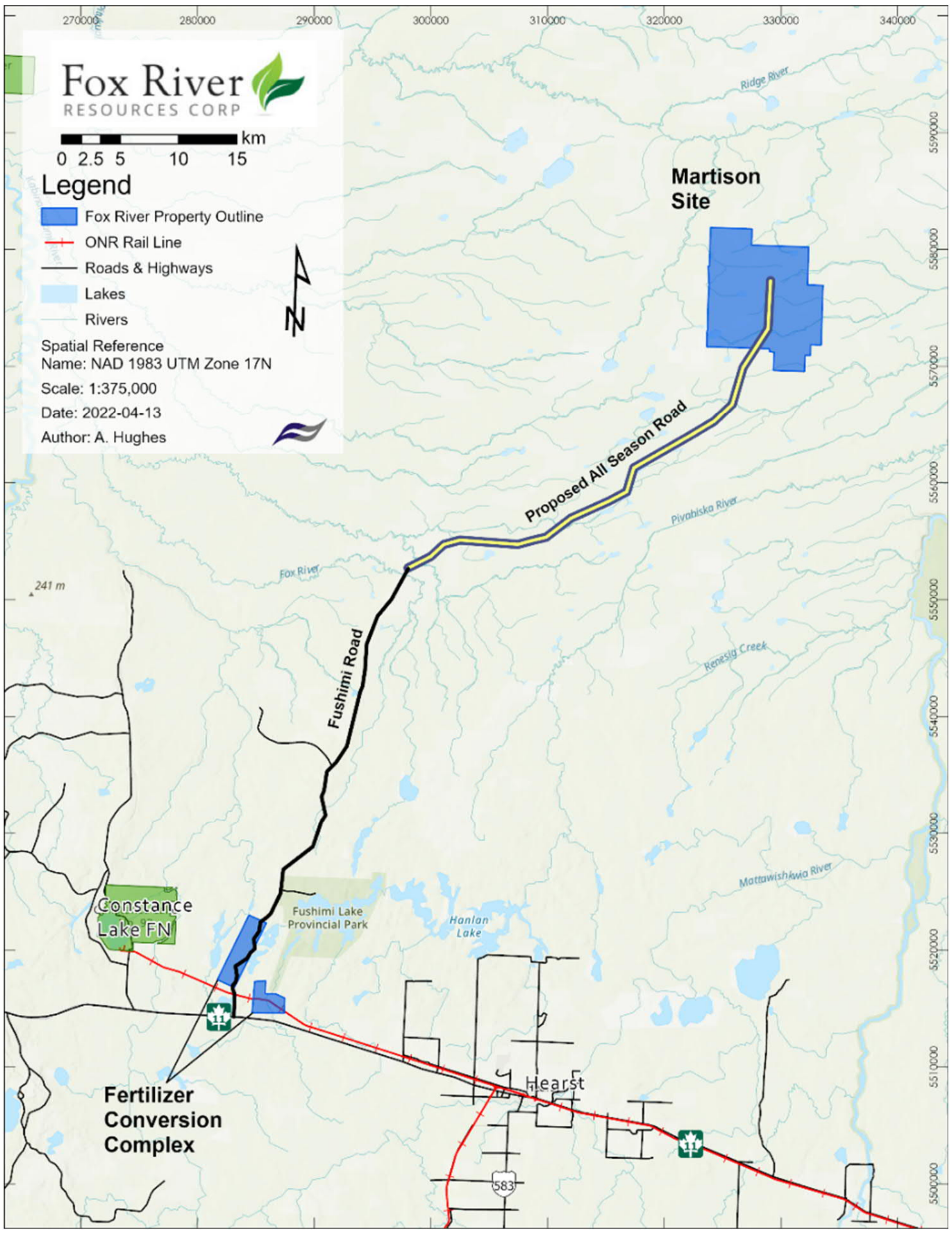


Figure 1-2: Martison Project Location and Access

1.3 History

The Carbonatite Complex was initially identified during a Canada wide airborne magnetic-electromagnetic survey carried out in 1965. In 1967, the large northern magnetic anomaly (Anomaly A) was covered by 98 claims staked by an unknown party though most probably Goldray Mines Ltd. The existence of the Carbonatite Complex was first formally interpreted in 1970 by the Ontario Department of Mines and Northern Affairs.

Between April 1980 and June 1981, Shell Canada Resources Limited staked 222 mining claims in a single contiguous block over the interpreted Carbonatite Complex. Major drilling programs were conducted in 1981, 1982, 1983, and 1984 by various parties/owners.

Drilling in the 1980s tested much of the property with a 200 m grid of holes and included a 50 m grid of holes over two zones of significant phosphate mineralisation.

As part of a wider drill program in 1984, a bulk sample generated approximately 65 t of the phosphate bearing residuum recovered from the central part of Anomaly A for analysis and pilot plant testing. During 1993, Sherritt Gordon Ltd. (the then controlling party) allowed the Martison claim block to expire. In the same year, McKinnon Prospecting of Timmins, Ontario, established a new claim block covering the Carbonatite Complex.

In early 1997, PhosCan (formerly “MCK Mining Corporation” and, before that, “Hendricks Minerals Canada Limited”) entered into an agreement with Baltic Resources Inc. whereby each would earn 50% in the Martison property from Donald McKinnon, principal and owner of McKinnon Prospecting.

In 1999, 2001, 2002, 2008 and 2012, the Carbonatite Complex area was further explored by drilling. With the exception of the 2001 Anomaly B program and one single hole in Anomaly C, all the other drilling (20,408 m) was focussed on Anomaly A. The drill programs of 2008 and 2012 included over 2000 m of drilling for hydrogeological studies and geotechnical site investigations.

In total, close to 22,000 m of drilling has been completed on the Project and which has been summarized in Table 1-1 and Table 1-2 for Anomaly A and Anomaly B respectively.

Table 1-1: Anomaly A Drilling Program Summary

Year	Company	No. of Holes	Meterage	Type and Comment
1981	Shell	3	278.8	DD*
1982	Shell	37	2,919.1	DD
1983	CAMCHIB	29	2,782.0	DD
1984	CAMCHIB	35	2758.5	DD (Includes three redrills) & 2x Churn drillholes for metallurgical bulk sample
1999	MCK Mining	14**	1,698	DD (includes one redrill)
2002	MCK Mining	6	943.2	DD
2008	PhosCan	34	4,888.3	Sonic (Cluster) - Metallurgical
2008	PhosCan	12	178.3	Auger / DD (Geotechnical)
2008	PhosCan	8	691.3	DD (Geotechnical – Includes two redrills)
2008	PhosCan	4	465.0	Auger (Hydrogeological)
2012	PhosCan	15	1,947.1	Sonic
2012	PhosCan	10	858.1	Auger (Hydrogeological)
Total		**207**	**20,407.7**	

* DD – Diamond Drill.

** Cargill funded six of the 14 holes drilled.

Table 1-2: Anomaly B Drilling Program Summary

Year	Company	No. of Holes	Meterage	Type and Comment
1981	Shell	2	275	DD
2001	Baltic	12	1,296	DD. (Includes one re-drill)
Total		**14**	**1,571**	

Significant phosphate beneficiation studies were completed pre-2008, using samples collected from the drill programmes in 1982, 1983, 1984, 1999, and 2002.

1.3.1 2008 Pre-Feasibility Study

In May 2008, the PFS was completed on the Project focussing on the options for mining the phosphate resources from Anomaly A as an open pit, fully integrated, fertilizer producing operation.

The 2008 PFS examined two scenarios for the Project. The scenarios were differentiated by the types and quantities of fertilizers produced, the process technology for producing fertilizer solutions, and the sources of sulfuric acid used to make fertilizer solutions from the phosphate concentrate.

The 2008 PFS was completed prior to the 2008-2009 global economic recession. There have been significant changes to the global economies since the filing of the 2008 PFS, and specifically to the economics related to the fertilizer market.

1.3.2 Post – 2008 PFS Work

Since the filing of the 2007 Technical Report and the 2008 PFS, there has been a significant amount of additional site-based exploration and project investigative work undertaken at the proposed mine site by the former owner, PhosCan. This has included resource definition drilling and sampling, hydrogeological, geotechnical, geophysical and environmental studies, and laboratory based test work.

Significant phosphate beneficiation studies were completed during 2010 and 2011 using samples collected from a drill program conducted in 2008. The 2008 drill program was designed primarily to generate a large bulk sample in excess of 40 t of the phosphate bearing residuum for pilot scale testing of the phosphate processing.

Sampling, and metallurgical testing, primarily targeting the niobium rich lateritic horizon, commenced in 2011 using samples from existing pre-2008 and the 2008 drill program. The 2012 sonic drill program added definition and confidence to several parts of the phosphate and niobium bearing zones.

1.4 Geology & Mineralization

The Carbonatite Complex is situated in the large expanse of "low ground" southwest of Hudson/James Bay, referred to as the James Bay Lowlands. The property comprises very gently rolling terrain dominated by muskeg and black spruce swamp. There are no exposures of the carbonatite or the surrounding country rock and all geological data has resulted from drilling information and interpretations of geophysical surveys.

Most carbonatites in Ontario are of Precambrian age and belong to two age groupings: 1,800 to 1,900 Ma (Paleoproterozoic) and 1,000 to 1,100 Ma (Mesoproterozoic). It has not been established to which grouping the Carbonatite Complex belongs, if either.

Differential weathering of the Carbonatite Complex has resulted in an irregular weathered 'karst' type surface of carbonatite, the depth of which varies greatly over short distances. Depressions in this carbonate rich surface are filled with the weathered breakdown product of the carbonatite, a 'residuum', which is effectively a paleosoil profile. This apatite rich residuum represents the bulk of the phosphatic material of economic interest. Above the residuum lies a less consistent layer of iron rich, pseudolateritic material, containing niobium and Rare Earth Elements (REE) mineralisation at levels of economic interest.

More recent glacial deposits, typical of the James Bay Lowlands, form a blanket of glacial till over the residuum and lateritic material sub–outcrop reaching up to 80 m in depth.

Initial geophysical exploration of the Carbonatite Complex identified three aeromagnetic anomalies. The mineral resources at Anomaly A are the bulk of the subject of this Technical Report. Anomaly B is approximately 5 km to the southeast. Anomaly C is approximately 3 km east southeast of Anomaly A. Only Anomaly A is currently under consideration for the production of phosphate concentrate.

Within Anomaly A, the residuum material has typically been divided into two sub-units based on lithology and grade of the contained phosphate (“P_2O_5”) mineralisation

- Unit 2A is unconsolidated.
- Unit 2B is consolidated (recemented) residuum material.
- Unit 2C is a third and minor low grade type of material of partially weathered carbonatite is referred to as. It occurs as lenses within the residuum, typically towards the basal contact with the relatively unweathered Carbonatite basement.

Almost all mineralogy studies, pre-2009, have been focused on the residuum and the components for the various flow streams resulting from beneficiation study programs. The minerals of the residuum fall into three classifications: primary, secondary, and detrital. The chief primary minerals are apatite, magnetite, pyrochlore, calcite, dolomite, barite, columbite, and occasional quartz.

1.5 Mineral Resource Estimate

The Mineral Resource Estimate (MRE) in this Technical Report encompasses all of the historical and recent resource drilling and includes those relevant and analysed intersections from the other site investigation drilling (hydrogeological and geotechnical holes).

DMT visited the property and core storage facility in Hearst on October 22 - 23, 2014. The core storage facility, now relocated to Timmins, was visited by QP – Tim Horner again in November 2019, where reinspection of some of the 2012 drill hole material took place, and additional reference samples were collected for analyses of P_2O_5, Nb, Sc, Cd, Cl and REEs.

Exploration work, limited to the winter months, has collectively accumulated close to 22,000 m of drill core for a total of approximately 8,170 samples analysed.

Although no additional site investigative work at the mine site has occurred since the 2012 drilling program and, overall, the drillhole database is of a satisfactory industry standard for resource estimation work. During 2021, there has been a closer review of the database applied in the previous MRE and Technical Report reissued on behalf of Fox River in 2016.

A review and remodelling of the geological data, and the input parameters for the block model, have been modified in the light of the changed technical and economic configuration of the Project. Specifically, greater emphasis has been placed on the deposit geochemistry and the downstream effects this may have on the modified mineral processing flowsheet.

The reinterpretation of the geological model and input parameters has resulted in an adjustment to the global mineral resources. Additional Inferred category resources have been added to the mineral inventory estimated in 2015 and, as a result, has highlighted potential targets for additional resource drilling to the north of the deposit.

The revised model was interpreted and generated using industry standard Surpac 3D software. The mineralized envelope constrained within a (Whittle) pit shell. HxGN's MinePlan software was used to convert the resource block model for ease of mine planning work. The MRE for Anomaly A is summarized in Table 1-3 below.

Table 1-3: Anomaly A Mineral Resource

Deposit	Classification	Tonnes Mt	Phosphate Grade % P_2O_5	Niobium Grade % Nb_2O_5
Anomaly A Residuum	Indicated Resources	53.8	22.99	0.42
	Inferred Resources	128.3	17.09	0.42
Anomaly A Lateritic material	Indicated Resources	6.2	7.97	1.13
	Inferred Resources	5.3	6.40	0.69

Notes:

1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a cut-off grade of 6% P_2O_5 in the Residuum or 0.2% Nb_2O_5 in the Lateritic material.
3. Mineral Resources are estimated at a dry Bulk Density of 1.89 t/m^3, 1.70 t/m^3, 1.90 t/m^3, 2.12 t/m^3 for till, lateritic material, residuum and carbonatite, respectively.
4. Mineral Resources are constrained by a Whittle open pit shell.
5. A minimum mineralisation width of 5 m was used for Indicated Resources and 2 m for Inferred.
6. Values for tonnage and grade may not add up due to rounding.

Cautionary Note: Mineral resources that are not mineral reserves do not have demonstrated economic viability. The PEA includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. The foregoing mineral resource estimates are as of December 31, 2021.

1.6 Mining Methods

The Anomaly A deposit is amenable to a large scale, conventional open pit mining method. The open pit mining operation was designed to provide sufficient beneficiation plant mill feed tonnages and produce a steady state of phosphate products containing 500 ktpa of P_2O_5 (MAP, NPS and SPA) through the FCC. The ultimate pit designed for this PEA mine plan is scheduled to be partitioned into six (6) phases, divided into one (1) starter pit phase for pre-stripping during the pre-production phase of the Project, and five (5) intermediate pit phases to be mined during the production phase of the Project.

Contractor labour and equipment will be employed during the pre-production phase, both for mine site preparation activities as well as for the pre-stripping activities to establish the first 20 m box cut for the starter pit phase. The mine will transition to an owner operation during the first year of production, utilizing conventional truck and shovel mining equipment throughout the remainder of the LOM.

As illustrated in Figure 1-3, materials mined from Anomaly A will be directed to the following destinations:

- Beneficiation plant – Mill feed materials.
- Waste facility – Waste materials.
- Pit backfill facility – Waste materials backfilled into the pit.
- Niobium stockpile – Niobium-rich lateritic materials, which will be rehandled should a viable means of processing this material be implemented during or after production mining.
- Tailings management facility – Waste materials, where glacial till will be used for berm construction.

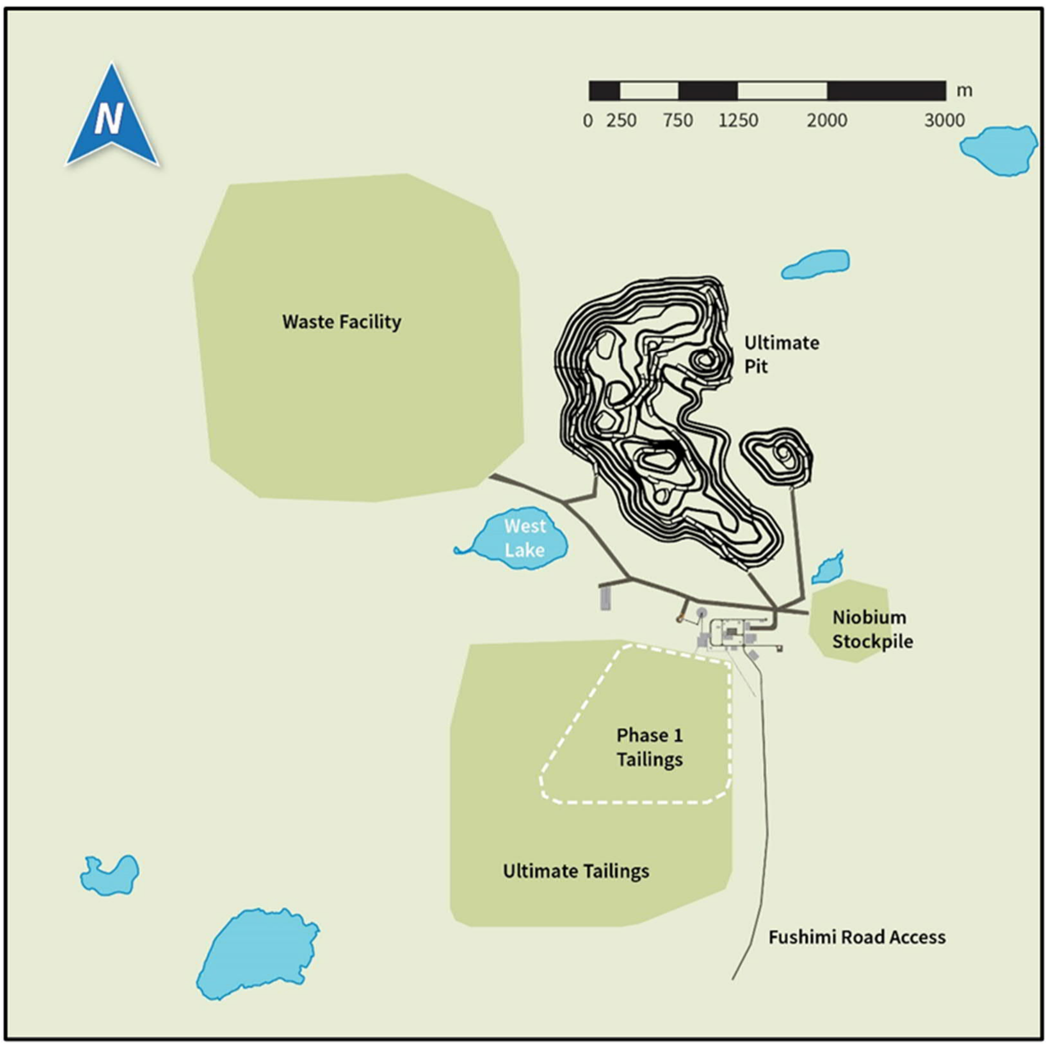


Figure 1-3: Martison Site – General Layout

The mine production schedule (see Table 1-4) is governed by pit phasing, haulage truck cycle times and the beneficiation plant's production ramp up. The mine life is estimated at 26 years, where Year 1 of production mining will produce 69% of 500 kt (or 345.0 kt) of P_2O_5 and Year 2 of production mining will produce 95% of 500 kt (or 473.5 kt) of P_2O_5.

The mine production schedule was developed annually and achieves a steady state production with final phosphate product tonnages containing 500 ktpa of P_2O_5 from Year 3 and onwards. The stockpiled lateritic material is designated as a separate material type from waste and is included as part of the strip ratio calculation.

Table 1-4: Mine Production Schedule - Summary

Mine Year	Total Mined	Waste Mined	Lateritic Material Stockpiled	Beneficiation Plant Feed			Strip Ratio	Total Phosphate Product Tonnage [1]
				Total Feed	Average Feed Grades			
(years)	kt (dry)	kt (dry)	kt (dry)	kt (dry)	% aP_2O_5	% CaO	t:t	kt
-1	11,500	11,500	-	-	-	-	N/A	-
1	19,308	16,650	530	2,128	23.03	32.91	8.07	345.0
2	18,849	16,277	40	2,533	25.78	35.77	6.44	473.5
3	20,995	18,074	95	2,827	24.70	36.45	6.43	500.0
4	19,318	16,336	129	2,853	24.50	34.10	5.77	500.0
5	18,109	14,991	463	2,655	25.81	36.10	5.82	500.0
6	18,284	14,660	500	3,124	22.81	31.61	4.85	500.0
7	17,151	13,892	289	2,970	23.75	35.15	4.77	500.0
8	15,014	11,995	83	2,936	23.81	33.73	4.11	500.0
9	14,011	9,812	283	3,916	18.87	26.59	2.58	500.0
10	16,530	12,334	559	3,637	20.10	28.34	3.55	500.0
11	17,378	14,610	22	2,746	25.27	33.52	5.33	500.0
12	15,881	13,131	69	2,680	25.56	35.75	4.93	500.0
13	15,540	11,702	322	3,516	20.58	31.38	3.42	500.0
14	15,984	12,588	69	3,327	21.65	30.22	3.80	500.0
15	15,666	12,904	0	2,762	25.21	35.88	4.67	500.0
16	14,183	10,698	134	3,351	21.50	30.28	3.23	500.0
17	14,576	10,594	527	3,454	21.05	28.75	3.22	500.0
18	14,571	10,017	701	3,853	19.24	28.30	2.78	500.0
19	13,692	10,512	204	2,976	23.72	32.36	3.60	500.0
20	12,959	9,323	7	3,629	20.25	27.68	2.57	500.0
21	13,049	9,189	26	3,834	19.25	27.56	2.40	500.0
22	12,906	9,223	37	3,646	19.97	26.74	2.54	500.0

Mine Year	Total Mined	Waste Mined	Lateritic Material Stockpiled	Beneficiation Plant Feed		Strip Ratio	Total Phosphate Product Tonnage (1)
				Total Feed	Average Feed Grades			
(years)	kt (dry)	kt (dry)	kt (dry)	kt (dry)	% aP_2O_5	% CaO	t:t	kt
23	16,626	12,071	46	4,508	16.92	23.32	2.69	500.0
24	15,033	10,266	512	4,255	17.76	25.99	2.53	500.0
25	10,186	5,573	944	3,668	19.99	27.88	1.78	500.0
26	2,178	354	0	1,823	15.33	25.09	0.19	178.5
Total[2]	**409,476**	**319,277**	**6,593**	**83,606**	**21.48**	**30.35**	**3.90**	**12,497.0**

Notes:
1. kt P_2O_5 contained in phosphoric acid at the Fertilizer Conversion Complex.
2. Total Mined (kt) = Mill Feed + Waste + Lateritic Material.

1.7 Metallurgical Testing

Metallurgical test work has been carried out on samples extracted from the Martison phosphate deposit since the 1980s, with the objective of developing methods to recover phosphate and niobium. Initially, Lakefield Research of Canada Ltd. developed flowsheets for niobium and phosphate recovery. The phosphate flowsheet was later investigated further by lab and pilot scale tests led by Jacobs Engineering, USA. Jacobs work also addressed production of phosphate fertilizers. Subsequent column cell testing of Martison flotation feed samples by Eriez Magnetics Laboratories, USA, contributed significantly to the flowsheet development studies.

The PEA summarizes the extensive history of these studies and some of the important results in section 13. Commencing in 1982, these studies successively investigated processing schemes to recover the phosphate and niobium in samples from the Martison site. Since 2007, most of the work has addressed recovery of phosphate and the potential to produce phosphatic fertilizer products.

1.8 Recovery Methods

The project's beneficiation process includes a number of modifications, relative to the 2008 PFS design, which have been incorporated in the PEA. These changes are designed to increase plant metallurgical efficiency and P_2O_5 recovery, and to improve operability.

The PEA design incorporates seven main processing stages to recover P_2O_5 from the mined residuum.

1.8.1 Crushing, Storage, and Blending of Mill Feed

The crushing system includes three stages, with the primary and secondary crushing operated in open circuit, and the tertiary stage consisting of a scalping screen, surge bin, feeder, and a cone crusher with a closed side setting of nominally 10 mm. The cone crusher discharge feeds a circular stacker/reclaimer system. This system enables blending of ROM grades to manage short term mill feed variability. Stacking is achieved by a fan shaped sprinkling action in an arc to ensure homogenization. Reclaiming is accomplished by a bridge reclaimer.

1.8.2 Grinding and Desliming

The reclaimed material is fed to the rod mill grinding circuit. Feed to the rod mills is controlled via a single feed bin equipped with two belt feeders, each feeding a rod mill. The grinding and desliming circuit is configured into two parallel trains. Discharge from each rod mill is screened, recycling oversize material to the mill, and passing material smaller than 425µm to the primary cyclones. The overflow from the primary cyclones, which contain fines that are problematic in flotation, will be dispatched tailings, while the underflow will be diluted with process water and pumped to the low intensity wet magnetic separation in the Magnetic Separation area.

1.8.3 Magnetic Separation

The Magnetic Separation process removes magnetic gangue, which reduces reagent consumption in the downstream flotation process, and makes the flotation separation easier. Both low intensity and high intensity magnetic separation are configured as two parallel operating trains. Material from grinding and desliming is distributed to Low-Intensity Magnetic Separators (LIMS) to remove ferromagnetic gangue, which will be discarded to tailings. The nonmagnetic product will then be treated by Wet Hi-Intensity Magnetic Separators (WHIMS) to remove paramagnetic gangue. The WHIMS nonmagnetic product, which is too dilute to be efficiently conditioned with flotation reagents, will be pumped to dewatering cyclones located upstream of the pre-conditioning step in the Flotation area.

1.8.4 Flotation

The flotation circuit, also configured into two parallel operating trains, includes pre-conditioning and conditioning with flotation reagents, followed by three stages of direct flotation. Reagents for pre-conditioning and conditioning are introduced into series connected vertical stirred tanks which provide the necessary mixing and residence time. The three stage flotation process includes rougher flotation column cells followed by two stages of cleaner flotation, utilizing column cells instead of mechanical flotation cells.

The underflow from the rougher columns is sent to tailings. The froth product (concentrate) from the rougher columns is collected and pumped to the first cleaner circuit. This circuit features stirred tank conditioners and flotation column cells. The first cleaner concentrate will be pumped to the second cleaner circuit, while the underflow is routed to tailings. Froth product from the second cleaner is final concentrate, which will be pumped to product dewatering and grinding. The underflow from the second cleaner circuit (cleaner 2 tails) will be pumped to a cyclone cluster to reject low grade fine particles, while the coarser fraction of the dewatered material will contain some phosphate which is partially conditioned with reagent, and which can be recovered recycling through the flotation stages. This process will maximize the P_2O_5 recovery of flotation.

1.8.5 Tailings Disposal and Water Recycle

The fine tailings are combined with the magnetic rejects from LIMS and WHIMS and are pumped to the tailings management facility (TMF) where natural sedimentation will consolidate them. The flotation tailings, which will have an elevated niobium content, will be pumped in a second pipeline for storage in a different cell within the TMF. This will enable the potential for future reclamation in order to recover niobium, should an economic means of doing so be found. Reclaimed water from the TMF will be returned to the plant into the fire water tank, which overflows to the process water tank.

1.8.6 Reagent Storage, Preparation and Dosing

The beneficiation plant features a collection of tanks to receive reagents and prepare and store stock solutions of the reagents ready to feed to flotation.

1.8.7 Concentrate Grinding

An additional grinding unit prepares the concentrate for efficient transport via the slurry pipeline to the FCC. Here the particle size and density of the concentrate slurry are adjusted and pumped to agitated storage tanks. Water removed from the concentrated slurry will be fed to the process water tank.

1.9 Infrastructure

Project infrastructure consists of two distinct and separate locations where operating facilities and associated infrastructures will take place:

- An open pit phosphate mine and beneficiation plant, located 72 km north of Hearst, Ontario. This operation will produce concentrated phosphate rock to be transported from the mine site to Fertilizer Conversion Complex via slurry pipeline.
- A Fertilizer Conversion Complex accessed by the Fushimi Road located to the west of Hearst. From concentrate supplied from the mine, this operation will produce super phosphoric acid (SPA), granular monoammonium phosphate (MAP) and granular nitrogen, phosphate + sulfur (NPS).

These two sites will be connected by an access road which will also provide a corridor for an electric transmission line to provide power to the mine site and a slurry pipeline to transport concentrate from the mine site to the FCC. Infrastructure construction and on-site provisions at both locations during production mining are accounted for in this section, including the preliminary sourcing and handling of raw materials, consumables and supplies for access roads, site preparation, plant and non-process building construction, utilities and power.

1.9.1 Mine Site Infrastructure

The mine site, being situated on the Hudson Bay lowlands, is currently only accessible by air or winter road. To support the mine site operation an access road will be constructed as an all-weather access road as an extension to the existing Fushimi Road.

1.9.1.1 Access Road

An access corridor will extend to the mine site and beneficiation plant from Highway 11, west of the town of Hearst. This will require the upgrading of the existing Fushimi Road and an additional northeast trending extension to provide an all-weather access road to the mine site. The FCC will be accessed on the section of upgraded road close to the intersection with Highway 11. The remaining upgraded section and road extension will be capable of handling the delivery of all materials and equipment to the mine site.

Road construction and upgrade activities identified in this study phase are the following:

- **Fushimi Road South Section**. The first section is approximately 13 km from Highway 11 and, with upgrades only to the existing road a surface course upgrade with crushed granular material will be required.
- **Fushimi Road North Section**. This section is approximately 38 km to the north of Fushimi Park Access Road and crosses four (4) culverts. This section is narrower and will require clearing to the full ROW of 30m and road widening from 6 m to design width of 9 m.
- **Extension to Mine site**. This section is approximately 37 km and will require the full construction of a new road. This will include tree and bush clearing, removal of stumps and topsoil and excavation of side ditches, installation of a geotextile layer overlain by compacted material with engineered aggregates on the top.

Fox River has secured a permitted aggregate pit, approximately 25 km north on the Fushimi Road from the junction at Highway 11, which will supply construction fill materials for the initial construction of the access road.

This access road will need to be established prior to major construction operations taking place at the mine site, allowing equipment and materials access for site preparation activities, construction of the beneficiation plant, surrounding mine site infrastructure, and for pre-stripping mining activities.

1.9.1.2 *Electric Transmission Line and Site Power*

Electrical power from the main grid will be connected to the FCC and supplied to the mine site by a newly constructed overhead line which will connect to the Hydro One Networks Inc (HONI) primary 115kV supply following Highway 11 west of Hearst. The primary demand for electrical power at the FCC will be provided through the cogeneration process from the combustion of sulfur which will offset the operating cost of power as well as provide benefits in the application of carbon taxation.

The power line to the mine site will be constructed in a cleared easement running parallel to the existing Fushimi Road and the required extension to the mine site. At the FCC and mine site, power will be stepped down from transformer stations at the required voltage for further distribution. Backup generators will also be provided at both locations to provide emergency power in the event of a main supply disruption.

Electrical grid power at the mine site (surplus from the FCC or otherwise purchased) will be used for all mining and processing demands and will also be used for heating in all buildings.

It is presently anticipated that the primary power supply will consist of the following main components:

- Equipment and works for connection to HONI 115kV line.
- Short 115kV line tap to the FCC main substation.
- Main 115kV switching station at the FCC.
- Step down substation 115/4.16kV 15MVA to supply Fertilizer Conversion Complex loads.
- 115kV 88km line to mine site.
- Main mine site substation 115/4.16kV 20MVA to supply mine site loads.
- Mine standby diesel generation plant, presently assumed to be 3x2MW.

1.9.1.3 *Beneficiation Plant*

Run of mine mill feed is transported from the mine to the beneficiation plant crushing circuit by haul trucks having a 181 tonne payload. The crushed feed is blended on a storage pile to reduce grade variation and allow the beneficiation plant to operate at a higher operating factor than the crushing circuit. The blended feed is reclaimed, ground, and processed to produce a slurry of phosphate rock and to reject phosphate-lean material to the tailings management facility. The slurry pipeline delivers 526,000 t/y P_2O_5 of phosphate rock concentrate to the FCC.

This PEA study also incorporates column flotation, a third crushing stage upstream of mill feed storage, modifications to the desliming circuits, and additional minor changes to produce 1,412 ktpy of concentrate over a 26 year LOM. The PEA mill feed grade has been slightly reduced and the concentrate MER specification has been relaxed from a ratio of 0.06 to 0.09 to improve P_2O_5 recovery. In addition, the flotation tailings containing high grade Nb will be stored separately from other tailings, in the event that an economical means of recovering the Nb can be achieved.

1.9.1.4 *Utilities*

The following primary utilities will be established to support the mine operations and beneficiation plant.

Potable water will be provided from a constructed well and will be tested and treated for safe use for site ablutions, sanitary facilities, and emergency showers.

All drinking water will be provided from an external supplier in exchangeable and reusable containers. A supply will be maintained at the mine site including an inventory for emergency use.

Sewage and waste water treatment will be managed by a centrally located, vendor supplied (off the shelf), fit-for-purpose, system comprising a self-contained biofilter tertiary wastewater unit. This system will operate year round using a low energy, foam bacterial medium to break down and remove contaminants.

Protection of all site infrastructure against the risk of fire across the site will be analyzed in greater detail in the next phase of study as part of a full risk assessment during design. As a minimum the beneficiation plant will be serviced by a fire water tank sufficient to supply up to three hours of operation of the fire water pump. Other buildings will be protected by standard fire extinguishers unless also deemed to be of sufficient risk and size to be serviced by the primary fire water system.

1.9.1.5 *Ancillary Buildings and Services*

The Mine infrastructure will consist of the following buildings. Where possible these will be pre-engineered buildings or modular. Larger buildings will be stick built.

- Mine administration offices and changing facilities. This will include first aid facilities. Mine and beneficiation plant operations offices will also be incorporated into this building.
- Warehouse. This will be stocked with everyday consumables required by all personnel as well as smaller supplies required for the mine and beneficiation plant. Smaller office and workshop areas for electricians and millwrights will also be part of the warehouse infrastructure.
- Hot and cold storage areas. These will be separately located and will be primarily dedicated to the safe storage of critical spares for all areas in an environment which will protect larger equipment from exposure to all weather conditions.

- Mobile equipment maintenance facilities. This pre-engineered maintenance facility will be designed for an overhead clearance of at least 16 m to allow room for the largest equipment (haulage trucks) and overhead cranes. The facility will provide preventative maintenance and repair, tire handling, machine welding, and washing services for the mobile equipment, as well as office space, clean and dry areas and equipment parts storage.
- Fuel storage. A self-contained and spill protected above ground diesel tank will be provided in a strategically located area, separate from other infrastructure and for ease of access for primary users. This refueling facility is intended to be used mostly by fuel trucks supplying the main fleet of production mining equipment in the pit.
- Laboratory facility. The details of the equipment required in the laboratory facility are yet to be finalized though an allowance has been made to both provide for the facility and equipment. The laboratory will serve as an analytical check on samples as part of process controls from the beneficiation plant and to process core samples from site drilling within the mining area. It will also serve to analyze additional exploration drill cores from Anomaly B and Anomaly C subject to quality and data integrity requirements.
- Site security and site entrance. This fit for purpose trailer unit will be staffed 24 hours a day and 7 days per week and will be equipped to monitor communications and vehicle movement on the Fushimi access road and key site infrastructure.
- Explosives magazine. Explosives used during mine drilling and blasting operations will be stored in a secure (fenced) facility segregated from all other mine infrastructure in accordance with the Explosives Act RSC.' 1985, c.E-17, Explosives Regulations, 2013 (SOR/2013-211), Explosives – Quantity Distances (CAN/BNQ 2910-510/2015) and R.R.O 1990, Reg 854: Mines and Mining Plants. All explosives will be stored in approved bunkers.
- Core logging and storage. A separate facility will be provided for the processing of drill core from the mill feed definition drilling and further site exploration. This will be supplied as one or more custom built trailer units mounted on a rough, compacted foundation.
- Communications tower. A wireless communications tower will be installed at the site as one of the first pieces of infrastructure at the start of construction, to enable both verbal communications and internet access. Further study will need to determine if a second relay tower is required to connect to existing communications infrastructure in the Hearst area.
- Helipad. The mine site will provide for an all-season helipad which will allow a medivac helicopter to land and take off in the event of a site emergency.

There will be no accommodation facilities and all employees will be bussed to and from the mine site on a daily basis. The detailed design of each building will be determined in a future phase of study.

A preliminary layout of the mine site infrastructure is shown below in Figure 1-4.

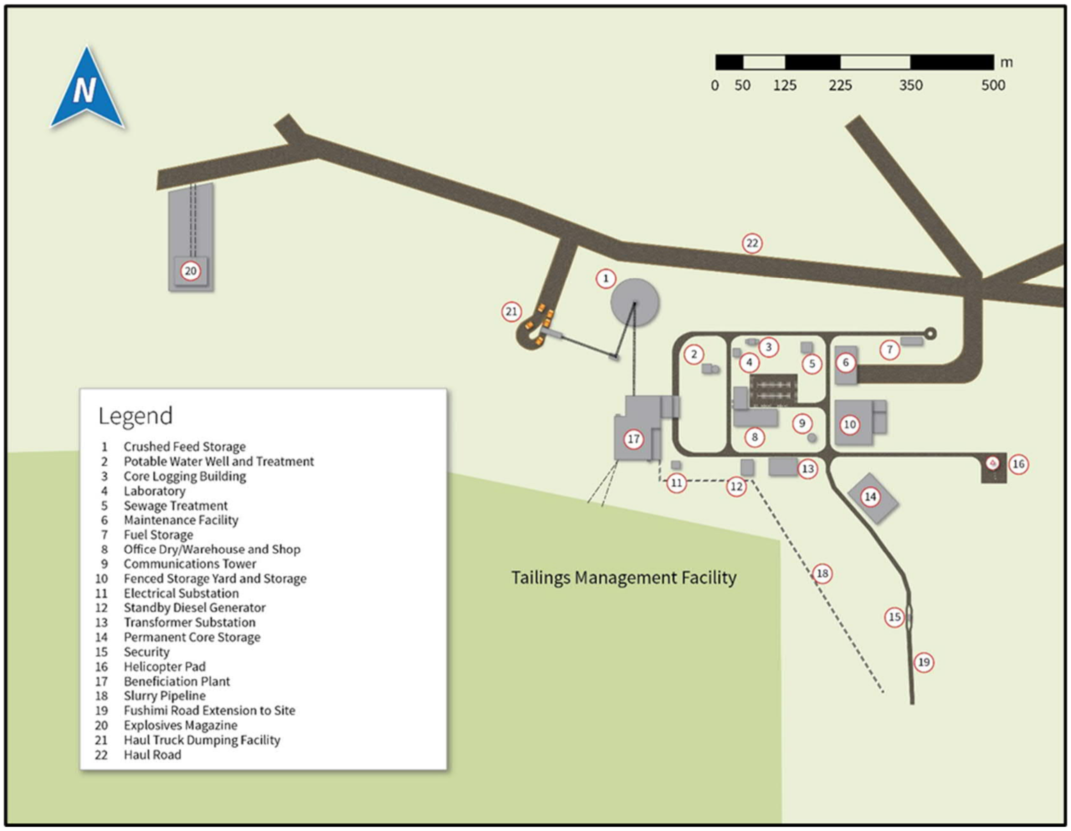


Figure 1-4: Conceptual Mine Site Infrastructure Layout

1.9.1.6 Site Preparation

The approach to the initial excavation to prepare the site for construction is summarized in this section.

All site infrastructure will be located on a low relief terrain consisting of saturated shallow muskeg overlaying deep and impermeable glacial till. A black spruce forest covers much of the mining lease terrain. The initial site preparation will require a period of activity to clear trees and drain water from the muskeg and establish berms and collection channels to prevent ingress both during construction and then during mine operations.

Once sufficient water has been removed, the muskeg will be removed down to the glacial till layer to enable construction activities to begin.

Site preparation for infrastructure will focus on the following activities:

- Diversion of impacted creeks.
- Preparation of site roads.
- Preparation of site infrastructure foundation locations.

- Preparation of initial waste facility area (initial draining though muskeg will remain in place).
- Preparation of starter pit area.
- Preparation of tailings facility (initial berm locations only).
- Establishment of pipeline corridors.
- Protective berms to prevent flooding (around site infrastructure area and other locations as needed).

1.9.1.7 Tailings Impoundment

The ringed tailings management facility (TMF) has been sited to the south of the mine site with the northwest edge approximately 50 m away from the beneficiation plant and has a footprint which lies over generally flat ground. The subsurface condition at the TMF is consistent with the rest of the mine site and generally comprises of muskeg underlain by dense to very dense silt and a basement below of glacial till deposits. Beyond storing the tailings solids generated by the concentrating process in the beneficiation plant, the TMF is considered in the Project as the ultimate management facility for excess water from all areas and allowing residence time, if required, to comply with water quality limits prior to discharge to the environment.

Two streams of slurry tailings, bulk tailings (from the primary mining process) and niobium (Nb) rich tailings will be transported to the TMF via separate pipelines from the beneficiation plant and managed at the TMF within separate adjacent cells.

The tailings characteristics are not known at this time though it is assumed to eventually consolidate to a solids content (Cw) of 66%. Decant water from the TMF will be returned to the beneficiation plant for reuse via a common reclaim water pipeline.

The TMF containment facility will be housed within zoned embankments which will be up to 20 m high and a constructed as a typical cross section comprising of compacted till with an internal drainage system for seepage control. The embankments will have an upstream face lined with HDPE geomembrane along with a compacted upstream shell of till fill keyed into the native glacial till for further seepage control within the foundation.

Native muskeg will only be removed down to the underlying glacial till for construction of the ringed embankment (berm) footprint to ensure structural integrity and to prevent seepage through to the natural environment.

The TMF Is designed to contain the LOM tailings with the initial stage to provide five years of tailings storage capacity equating to 5.9 M-m^3 of bulk tailings and 1.4 M-m^3 of Nb rich tailings for a total of 7.3 M-m^3. Disposal tailings cells can be developed in three stages while Nb tailings cells can be developed in two stages. Total capacity of the TMF at its ultimate configuration will be 37 M-m^3 for bulk tailings and 8.7 M-m^3 for Nb rich tailings for a total of 45.7 M-m^3 of solid particles.

Closure of the TMF may be staged as each of the cells reach capacity. As the geochemical characteristics of the tailings are defined, it is currently assumed that acid rock drainage will not be a factor in the cover design and a dry cover will be installed for closure which will comprise of drainage layer, topsoil and vegetation. Installation of the dry cover will take place once supernatant water is drained and the surface of tailings deposit is sufficiently dry and easily accessible to construction equipment.

1.9.1.8 Site Water Management

Water will be managed at the mine site by water quality designation. This will effectively divide water quality into two discrete categories:

- Water that has been in contact with mining and mineral processing activities, which will be considered “contact” water and will be managed in dedicated facilities to prevent unintended release to the environment.
- Water that has not been in contact with mining and mineral processing activities, which will be considered “non-contact” and will remain segregated from these processes and treated as clean surface run-off.

To the extent possible, non-contact water should be diverted away from the site to avoid any risk of contamination (and thus becoming contact water) and, as a result, minimize water management requirements. Additionally, site water will be managed in such a way that minimizes the impact on the existing regime for natural streams and creeks.

The mine site lies within two watersheds: the Ridge River (to the west) and the Soweska River watersheds (to the east). As a result, site water will be managed to the extent possible by this southwest to northeast trending watershed divide so that if discharge is required, water is returned to the watershed in which it originated. The site draining plan and ultimate facility footprint is shown in Figure 1-5 below.

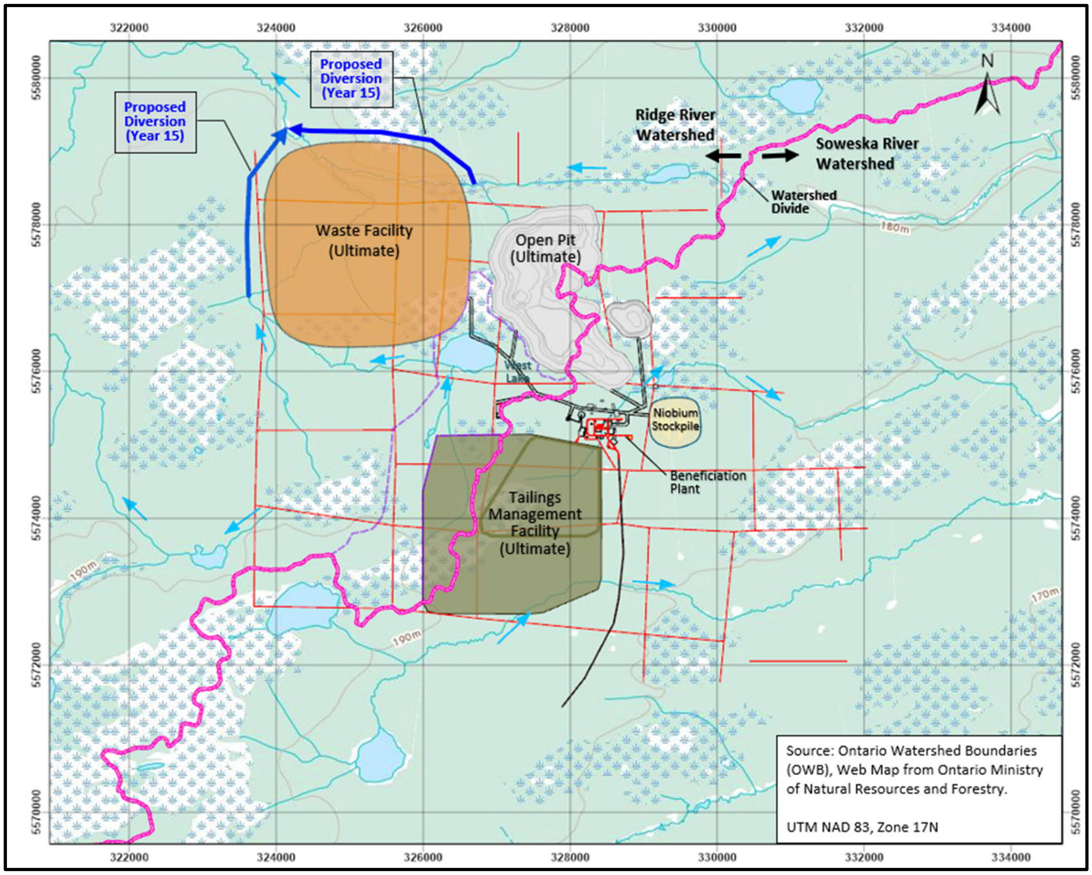


Figure 1-5: Mine Site Drainage Plan - End of LOM Footprint

1.9.1.9 Concentrate Slurry Pipeline

Phosphate rock slurry is transported from the beneficiation plant at the mine site to receiving facilities at the FCC by a buried pipeline which follows the route of the site access road. The slurry is concentrated in a thickener within the beneficiation plant, stored in agitated tanks, and pumped 86 km to the FCC. The pipeline facilities include additional storage at the FCC, cathodic protection, monitoring and telecommunication.

The pipeline is identical in length and pipe diameter to what was proposed in the 2008 PFS nevertheless an increase in capacity from 1.16 Mtpa to 1.41 Mtpa has been accomplished by increasing the slurry flow rate and selecting a pump capable of operating at higher pressures, up to 250 bar.

All of the upstream and downstream pipeline facilities, including the thickener, slurry tanks, and pump station will be inside shared facility buildings. Two mainline PD piston diaphragm pumps (one operating and one standby) provide the required discharge pressure to deliver the slurry though the pipeline to the terminal station at the FCC.

Slurry tanks will be located at the pump station and terminal facilities. For the pump station, there will be two storage tanks. Each tank has a dimension of 15 m in diameter by 15 m high to provide a combined total slurry storage time of 22 hours. The terminal station at the FCC will include identical 15 m by 15 m tanks.

Flushing water is required prior to a normal shutdown and the process water at the mine site via the beneficiation plant water system will be used for this purpose. Normal start-up occurs with water in the pipeline and then a transition to slurry is made once stable flows and pressures are sustained.

The mainline pump station includes one operating charge pump to supply the required suction pressure to the mainline pump. A test loop will also be installed between the charge pumps and the mainline pumps, which can be used on an as-needed basis for assessing the slurry performance in a controlled manner.

Two pressure monitoring stations are located along the pipeline. The first station will be located near the 1 km marker on the route. This will be used primarily for tracking the pressure losses of the slurry passing through the pipeline. The second station will be installed near the middle of the pipeline between the mine site and the FCC. This second station will provide critical pressure data to the Pipeline Adviser™ and the leak detection systems.

A permanent impressed voltage cathodic protection system will be installed. The cathodic protection system inhibits the onset and advancement of pipe external corrosion by changing the potential difference that naturally exists between the pipe and the earth by means of impressed voltage. During pipeline construction, test leads will be installed approximately every kilometre and at river crossings, cased crossings, and foreign pipeline crossings.

In the unlikely event of a pipeline rupture, the leak detection system, Pipeline Adviser™, and SCADA system will warn the operator, prompting the activation of an emergency shutdown sequence if the data appears valid. The leak location can be determined by the leak detection system and confirmed by field inspection.

Pipeline monitoring instruments such as flow meters and density meters will be installed at the terminal to verify the slurry properties as it leaves the pipe. The terminal station is monitored and controlled from the beneficiation plant control room.

1.9.2 *Fertilizer Conversion Complex*

The processing facilities at the Fertilizer Conversion Complex includes a 1,500 t/d dihydrate phosphoric acid plant with concentration units for production of both Merchant Grade Acid (MGA) a Super Phosphoric Acid (SPA), a 4,200 t/d sulfuric acid plant with a turbo generator and a 2,310 t/d granulation plant producing both Monoammonium Phosphate (MAP) and Nitrogen Phosphate Sulfur (NPS) based fertilizers. The FCC also includes:

- Sulfur receiving and storage
- Sulfuric acid storage
- Concentrate thickening and storage
- Gypsum wet stacking
- Ammonia receiving and storage
- Raw materials receiving
- Process utilities
- Granular product storage and shipping
- SPA shipping
- Fully equipped rail yard for the inbound and outbound shipments.

The main feedstocks received at the FCC are phosphate rock slurry, sulfur, and ammonia. The rail yard is designed to receive approximately 540,000 t of inbound raw materials and approximately 950,000 t of outbound finished product. The Facilities production capacity is designed to produce SPA (150 ktpy of P_2O_5), MAP (247 ktpy P_2O_5) and NPS (99 ktpy P_2O_5).

1.9.2.1 *Sulfuric Acid Plant (SAP)*

The Sulfuric Acid Plant (SAP) provides sulfuric acid, steam, and cogenerated electrical power to the FCC. The plant is designed to produce up to 4,200 t/d as 100 wt% sulfuric acid. The plant processes molten sulphur delivered by rail. The SAP plant features a turbo generator that provides heat recovery options to boost steam production for use in the FCC plants and sufficient electricity is produced to supply the entire FCC facility and export additional and variable excess power to the mine site.

1.9.2.2 *Phosphoric Acid Plant (PAP)*

The dihydrate process reacts phosphate rock with sulfuric acid to produce gypsum crystals and phosphoric acid containing 28% P_2O_5. The acid will be separated from the gypsum by three belt filters. The filter cake will be sluiced with pond water and pumped to the wet gypsum stack. The gypsum will be consolidated, and the resultant water decanted and returned to the plant as pond water for sluicing and process cooling. The 28% acid is clarified and pumped to storage tanks. Sludge from clarification is recycled to the reactor. Clarified 28% acid will be distributed between the concentration area (PAP and SPA) and the granulation plant as granular product mix demands. Clarified acid to concentration will be concentrated to 54% in five parallel evaporators using low pressure steam. The 54% acid is clarified and pumped to storage as the feedstock for the granulation and the SPA plants.

1.9.2.3 *Super Phosphoric Acid Plant (SPA)*

The SPA feedstock (54% P_2O_5) will pass through three heat exchangers before entering the SPA flash chamber where it will be concentrated to 68.5% P_2O_5 using high pressure steam. The SPA will then be oxidized with an oxidizing reagent, reduced with ferrous iron, filtered to remove magnesium, and pumped to SPA storage before loading into tank cars.

1.9.2.4 *Granulation Plant (MAP & NPS)*

Solids recovered from the SPA filtration mixed with 54% acid will be reacted with anhydrous ammonia in the Preneutralizer forming a slurry that will be sprayed onto a stream of recycled undersize granular material and additional anhydrous ammonia in the rotary granulator. The MAP flows from the granulator to the rotary dryer where the moisture is reduced to approximately 1.5 percent free water.

The product size will be controlled using double deck oversize screens and crushers followed by single deck product screens to separate the on-size material before routing to the fluidized bed product cooler. The cooled product will be screened one final time to remove any fines that have been generated in the cooler then routed to the product coater. All fines will be recycled to the granulator. The product coater is a ribbon blender where the product is covered with a thin coat of inert oil to control product dusting.

The NPS product will be produced in a similar method with the introduction of molten sulfur and sulfuric acid in the Preneutralizer to produce NPS 12-40-0-10S. When making NPS fertilizer with zinc, the additives can be introduced into the recycle system to target a 1% zinc concentration in the final product (12-40-0-10S-1Zn).

The fertilizer products will be conveyed to a single warehouse with a capacity of 120,000 metric tonnes. NPS and MAP are stored within segregated areas and have dedicated product loading systems. Product is then loaded into trucks and/or rail cars at 640 t/h.

1.10 Market Studies & Contracts

Long term supply and demand fundamentals are positive, and prices over the life of the project likely will exceed the values required to generate a threshold or better IRR for the Martison Project.

The positive outlook is based on the need for this sector to build the capacity needed to meet projected demand during the next two decades. It is expected that the demand for phosphate products produced from phosphoric acid will increase about 18 mt P_2O_5 between 2020 and 2040, which is in line with recent long term forecasts from the International Fertilizer Association (IFA).

Phosphate demand forecasts are underpinned by steady increases in food production to meet the needs of a growing and more affluent global population. In addition, developments such as the exponential growth of the renewable diesel and sustainable aviation fuel production as well as increases in Chinese feed grain imports to supply a restructuring hog industry are expected to be important drivers for North American and Global phosphate demand. The expected growth in the lithium iron phosphate battery production within China could further focus the Chinese phosphate industry on the domestic rather than the international export markets.

The supply side assumes 15.7 mt P_2O_5 of new phosphoric acid capacity will come online between 2020 and 2040. This implies that the global operating rate will need to increase from about 87% today to 90%-92% throughout the forecast period, and that is constructive for the price outlook.

The addition of specific projects is speculative, but Morocco and Saudi Arabia are expected to account for roughly 40% and 30%, respectively. It is also assumed that at least two world-scale greenfield projects will move forward in Brazil and Algeria. The analysis also includes the Martison project.

Two US facilities are expected to close during the forecast period. Furthermore, the world has become dependent on Chinese phosphate exports during the last two decades, but Chinese supplies are uncertain if not highly suspect in the long term. Also, more mine closures than are indicated are possible.

Based on PEA operating cost estimates and current freight rates, the Martison facility is expected to rank at the low end of delivered cost curves for nearly all its target markets.

Martison cash operating costs are expected to rival the lowest cost offshore producers that benefit from either low cost phosphate rock (Morocco) or low cost natural gas and integrated ammonia production (Saudi Arabia and Russia). The cost advantage of the Martison operation is based on the cost and quality of its phosphate rock, lower sulfur procurement and transportation costs, and competitive ammonia costs. The Martison fob plant costs are less than those estimated for its four US competitors. Finally, lower transportation and logistics costs cement the project's cost advantage to its target markets.

1.11 Environmental Considerations

Environmental baseline studies were undertaken in 2008 and 2009 by Golder Associates Inc. and AMEC Earth and Environmental Ltd. These studies included a variety of subject areas with a particular emphasis on identifying winter habitat for caribou. An Environmental Study Report (ESR) was prepared by AMEC in 2008 for the all-season access road in accordance with requirements of a Category “C” Class Environmental Assessment (EA), pursuant to the Northern Development, Mines, Natural Resources and Forestry (NDMNRF) Class EA for Resource Stewardship and Facility Development Projects. A Statement of Completion was issued by the NDMNRF in January 2009.

As more than five years have passed since the Statement of Completion was issued, a Notice of Intent to Proceed with the project must be issued. This process allows agencies, Indigenous communities, members of the public and other interested stakeholders to review and comment on the project and any proposed changes. If the project has changed and according to the NDMNRF will result in additional impacts to the environment and significant public concern, or if there are significant concerns raised with regards to Indigenous peoples and Treaty rights, there will be a requirement to complete further consultation, additional studies, and a revised ESR.

To support additional permits and approvals needed by the project such as approval under Ontario’s Endangered Species Act, additional baseline information, related to caribou and bats will be required. Additional consultation and engagement with Indigenous and local communities will also be required to facilitate the permit and approval processes.

The information summarized in Table 1-5 below identifies the recognized gaps in environmental considerations for the Project based on the available information and the changes in legislation at the Provincial and Federal levels in recent years.

Table 1-5: Summary of Work Required to Fill Gaps

Data to be updated	Supporting Permit / Approval
Wildlife habitat and use (caribou and bats)	• Approval under the Endangered Species Act from the MECP • EA Notice of Intent to Proceed with all season access road • EA for other project components e.g., transmission line
Fish and fish habitat, Hydrology	• Authorization or letter of advice from DFO • Crown Land work permits from the NDMNRF
Consultation / Engagement	• EA notice of Intent to proceed with access road • EA for other project components (e.g., transmission line) • Authorization or letter of advice from DFO • Approval under the Endangered Species Act from the MECP
Cultural heritage and archaeology if required	• Clearance letter from the MHSTCI

1.12 Capital & Operating Costs

The total estimated capital cost for the base case is USD2,404M of which USD1,859M is the initial capital cost (CAPEX) required during the project period and the remaining USD545M is sustaining capital (SUSEX) required for the life of mine. A contingency of 15.5% was considered for all CAPEX costs. The capital and sustaining cost summaries are detailed in Table 1-6 below.

Table 1-6: Capital Cost Estimate Summary

Area	Capital Cost (USD MM)	Sustaining Cost (USD MM)	Total Cost (USDMM)
Mine Site	766.07	222.75	988.82
FCC Site	1,093.40	322.42	1,425.82
Total	**1,859.47**	**545.17**	**2,404.64**

The average operating cost for each area of the mine site and FCC are detailed in the operating cost estimate summary in Table 1-7 below.

Table 1-7: Operating Cost Estimate Summary

Area	Operating Cost Per Unit Reported	Unit OPEX Reported As
Mine Site		
Mine Operations(1)	$13.36	USD/t of Mill Feed
Beneficiation Plant	$6.44	USD/t of Mill Feed
Mine Infrastructure	$3.46	USD/t of Mill Feed
Pipeline	$0.48	USD/t of Mill Feed
FCC Site		
Sulfuric Acid Plant(3)	$99.55	USD/t of H_2SO_4
Phosphoric Acid Plant(5)	$423.02	USD/t of P_2O_5
FCC Infrastructure(4)	$21.05	USD/t of P_2O_5
Super Phosphoric Acid Plant(2)	$395.16	USD/t of SPA Solution
Granulation Plant (MAP)(3)	$319.10	USD/t MAP
Granulation Plant (NPS)(3)	$321.34	USD/t NPS

Notes:
(1)Includes dewatering and site preparation.
(2)SPA solution (68% P_2O_5).
(3)Base case prices: sulfur price of USD274/t delivered, and ammonia price of USD602/t delivered.
(4)Excludes: Electrical usage for FCC Heating is 12.92 kWh/t of P_2O_5 over the LOM.
(5) Phosphoric acid operating cost reported on 500,000 t/yr basis.

Operating costs expressed in the above table take into consideration all variable costs such as mine explosives, fuel, reagents, purchase power, and operating and maintenance supplies, as well as fixed costs such as staffing, local fees and insurance, general supplies, and SG&A costs.

1.13 Economic Analysis

The economic model was developed specifically for evaluation of this project scope on a fully funded project basis. The economic analysis uses a cash flow model at a base sulfur price of 274 USD/t, a base ammonia price of 602 USD/t and an 8% discount rate.

The financial assessment was carried out on a 100% equity basis without the inclusion of debt or other funding. Inflationary effects were not applied in the assessment. Canadian Federal tax and Ontario Provincial regulations were applied to assess the tax liabilities.

The economic model was used to examine variations of a specific Base Case as defined in Section 19. Results were produced on a pre-tax and after-tax basis as well as providing information on IRR, Payback, and LOP Cash Flow Sensitivity Analysis and presented in Section 22 in detail.

The key assumptions and parameters used in the PEA are tabulated below in Table 1-8.

Table 1-8: Key Assumptions and Parameters

Description	Units	Amount
Product Prices / Input Costs / FX		**Base Case[1]**
Product Prices		
Mono Ammonium Phosphate (MAP)[2]	USD/t DEL	$800
Super Phosphoric Acid 68% P_2O_5 (SPA)[3]	USD/t DEL	$1,060
Nitrogen, Phosphate, Sulfur (NPS)[4]	USD/t DEL	$810
Input Costs		
Sulfur[5]	USD/t DEL	$274
Ammonia[6]	USD/t DEL	$602
Currency Exchange Rate	USD/CAD	0.79365
Mine Site Production Inputs		
Total Tonnes Mined, Life of Mine Plan	Mt/Dry	409.48
Beneficiation Mill Feed, Life of Mine Plan	Mt/Dry	83.61
Concentrate Grade	% P_2O_5	37.28
Mine Life	Years	26
Average Mill Feed (Years 3-25)	Mt/y	3.35
Phosphate Concentrate Production (Years 3-25)	Mt/y	1.41
Fertilizer Conversion Complex (FCC) Production Inputs		
Phosphoric Acid Plant Capacity	P_2O_5 t per annum	500,000
SPA Plant Capacity	P_2O_5 t per annum	150,000
Granulation Plant Capacity	P_2O_5 t per annum	346,000
Sulfuric Acid Plant Production Rate	H_2SO_4 t per annum	1,276,000

Description	Units	Amount
Average Annual Product Tonnes (Years 3-25)		
MAP	t	474,000
NPS	t	247,000
SPA	t	221,000
Average Annual Consumption (Years 3-25)		
Sulfur	t	433,000
Ammonia for MAP	t	63,000
Ammonia for NPS	t	36,100

Notes:

1. The “Base Case” is a weighted average of three market forecast scenarios for the years 2022 to 2047.
2. Reference prices (CAD/tonne MAP delivered Western Canada) for Base Case is $1,060.
3. Reference prices (USD/tonne P_2O_5 delivered Corn Belt) for Base Case is $1,570.
4. Reference prices (CAD/tonne NPS delivered Western Canada) for Base Case is $1,065.
5. Reference prices (USD/long ton S CIF Tampa) for Base Case is $320.
6. Reference prices (USD/tonne NH_3 CIF Tampa) for Base Case is $630.

Key results from the economic model in the PEA are tabulated below in Table 1-9.

Table 1-9: Key Financial Base Case Results[1,2]

MAP Price	$800	
SPA (68% P_2O_5) Price	$1,060	
NPS Price	$810	
Sulfur Price	$274	
Ammonia Price	$602	
	Pre-Tax	**After-Tax**
$NPV_{8\%}$ (USD MM)	**$2,144**	**$1,467**
IRR	**20.2%**	**17.4%**
After-Tax Payback (years)	**5.2**	
Cumulative Cash Flow (USD MM)	**$6,460**	

Notes:

1. All results developed at an exchange rate of 0.79365 USD/CAD for the CAPEX and OPEX calculations. Product prices are in US Dollars per metric tonne.
2. The “Base Case” is a weighted average of three market forecast scenarios for the years 2022 to 2047.

1.14 Conclusions & Recommendations

1.14.1 Geology and Mineralization

There has been a closer review, by DMT, of the database applied in the previous 2015 MRE and Technical Report issued on behalf of Fox River in 2016. This has resulted in the re-interpretation and remodelling of the geological data, and subsequently the input parameters for the block model, which have been modified in the light of the changed technical and economic configuration of the Project. Specifically, greater emphasis has been placed on the deposit geochemistry, and the downstream effects this may have on the modified mineral processing flowsheet.

It is the opinion of DMT that reasonable opportunity exists to add additional resources to the mineral resource estimate (MRE) with further drilling, in that the Anomaly A deposit still remains open at depth in several areas, particularly in the northern part of the trough or valley feature running northwest – southeast.

Based on current drillhole intersections and supportive evidence from recently applied ground geophysics, the Anomaly A deposit also appears to have potential for further extension to the east and northeast of the proposed open pit area.

Anomaly B remains unchanged from the 2015 MRE and represents a target for further exploration. This is currently estimated at between 35 Mt and 70 Mt of residuum containing 14% - 30% P_2O_5.

Anomaly C remains an early stage exploration target.

No resource estimate has been established for the Rare Earth Elements (REEs) due to the paucity of sample data, particularly from the historical drillholes and the limited metallurgical test work carried out.

To improve the confidence levels in the MRE and geological model, additional drilling and testing is required. There is a paucity of analytical data from the historical drill holes (early 80s), particularly the MER (oxides), and the limited data for both bulk density and moisture content for the various lithotypes requires to be addressed.

There remains uncertainty too for the lower contact of the deposit residuum with the basement source rock (the carbonatite). This basal contact has only been identified in a limited number of drill holes. This key boundary is important to establish the lower limits of the deposit which remains open at depth, particularly in the north. Closer spaced ground geophysical surveys will add to the definition of the basal contact, although ultimately this contact will require ‘ground truthing’ by way of additional drilling to target this boundary and improve the confidence in the geological model.

1.14.2 Other Site Based Technical Studies

The significant hydrogeological test work undertaken in 2008 and 2012 has demonstrated that the deposit aquifer conditions have measurable boundaries and a relatively slow recharge rate, which will allow for the design of a dewatering solution for a proposed open pit operation. It is anticipated that further hydrogeological investigations will be required to reinforce the preliminary observations and dewatering model proposed by AMEC in 2012.

More geotechnical site and laboratory testing will be required to determine detailed in-pit engineering properties of the various lithotypes. Specifically, this relates to slope stability characteristics, foundation engineering properties for the mine site infrastructure, at the overburden (waste) collection areas and, at the locations of the FCC located at the south end of the Fushimi Road. In addition, construction materials for the all-season access road, re-engineering of mine site foundations, berms and site roads should be identified and appropriately tested for purpose.

Currently the mine site remains winter access only which constrains future drilling programs, and other site investigative work (which would include further geophysical, hydrogeological and geotechnical work), and this would have to be phased over several winter seasons as described in Table 1-10 below.

Table 1-10: Proposed Drilling And Mine Site Investigative Work Program

Phase	Activity	Cost CAD
I	Combined Sonic and Diamond Drill program: 60 drillholes (8,500 m) to probe deposit at depth and north (Resource generation) and twinning of historic holes.	6,800,000
	Ground Geophysics of Anomalies B & C (High Sensitivity Resistivity). 20 profiles @ average length 1250 m (25,000 m)	250,000
	Hydro-Geological Pump Testing and Monitoring.	350,000
	Bulk Density & Moisture Content Testing.	15,000
	Geotechnical site and lab based testing.	300,000
	EIA review and revision.	250,000
	Phase I Sub-Total	**7,740,000**
IIA	Revise/Update Geological Resource model, including Core Re-logging and Variography studies.	150,000
	Combined Sonic and Diamond Drill program: 60 drillholes (8,500 m) to probe deposit at depth, north and east (Resource generation and infill), further historic hole twinning.	6,800,000
	Bulk Density & Moisture Content Testing.	15,000
	Phase IIA Sub-Total	**6,965,000**
IIB	Revise/Update Geological Resource model.	85,000
	Combined Sonic and Diamond Drill program: 60 drillholes (8,500 m) (Resource generation and infill).	6,800,000
	Bulk Density & Moisture Content Testing.	15,000
	Phase IIB Sub-Total	**6,900,000**
	Grand Total	**21,605,000**

1.14.3 *Mining*

The use of large mobile mining equipment will facilitate high levels of operational flexibility, high mining rates and lower unit operating costs. Mine design and scheduling analysis indicate that the deposit can be developed in a logical progression, both in the delivery of plant feed material and equipment requirements over the LOM.

However, the site's topography is dominated by saturated muskeg and black spruce swamp and consequently soft ground conditions can result in rutting or sinking of large, wheeled mobile mining equipment if not adequately prepared for. Therefore, ex-pit haul roads will need to be constructed and maintained with sufficient quantities and quality of locally sourced granular fill. It is assumed for this study that sufficient fill will be sourced from nearby aggregate quarry sites, or from Anomaly A's bedrock once exposed.

In-pit haulage is also expected to be impacted by soft rock conditions, therefore additional efforts are anticipated for road maintenance within the pit over the LOM. The management of subsurface geotechnical and hydrogeological conditions are predicted to be a priority for the application of efficient mining operations, haulage ramp maintenance and pit wall slope control.

An aquifer is present mainly within a zone of weathered bedrock at the base of the residuum. Active dewatering of the mining area, utilizing dewatering wells, is required to minimize impact on the mining operations, notably during seasonal thaw when mining at depth. Pending further geotechnical and hydrogeological work to improve site knowledge and model subsurface conditions, the assumption has been made that pit wall slope stability can be improved by active depressurization via dewatering wells, as well as the placement of a toe buttress made of granular fill to support the pit wall and promote drainage.

The subsequent study phase should aim to achieve the following milestones necessary towards improving the economic viability of the Project's mining operation:

- Build an updated geological block model utilizing both historical and new assay data to be collected and tested from the proposed geological drilling campaigns. Infill and step-out drilling data will improve the geological confidence of the model, increasing measured and indicated mineral resource estimates and providing a basis for establishing a future mineral reserve.
- Conduct a geotechnical and hydrogeological gap analysis of all available historical data and studies completed to date, ahead of the proposed geological drilling campaigns. This will establish any additional field work or testing programs that can be integrated into the geological campaigns and ensure sufficiency of data collected prior to any future geotechnical and hydrogeological modelling and analyses.
- Perform both condemnation drilling and geotechnical assessments of waste facility and stockpile locations. Review and reassess potential locations for waste deposition, as well as the geotechnical parameters for the facility designs to assess whether footprints can be reduced using higher ultimate lift height limits for the respective piles.

- Assess the material suitability for the Anomaly A deposit bedrock for both haulage road and in-pit ramp construction and subsequent maintenance.
- Investigate known quarry (borrow pit) sites to determine the available quantities and quality of locally sourced rockfill required during the construction phase of the Project.
- Conduct mine design and planning work, incorporating the updated geological model as well as the latest geotechnical and hydrogeological assessments and plant processing parameters. For the selection of a preferred mining method, it is highly recommended that a focused mining methodology and equipment selection trade-off study be performed, centering on equipment sizing and the evaluation of mine decarbonization and electrification opportunities.
- Investigate mill feed blending opportunities, ensuring that mine grade control practices and ROM blending provides the best possible mill feed quality on a daily basis with a goal to maximize P_2O_5 grade while mitigating the presence of contaminant grades.
- Conduct detailed pit optimization and design exercises, investigating the potential to minimize pre-stripping tonnages, to determine the best pit phasing regimes and to maximize both overall Project economics and in-pit dumping opportunities.
- For the next phases of engineering, it is recommended that mine production plans consider seasonal conditions to optimize material extraction from pit bottoms and pit overburden during the winter months. Although this period is challenging at most Canadian surface operations, the mine site appears to be particularly susceptible due to the wet and soft rock insitu conditions.

1.14.4 Mine Site Infrastructure

The infrastructure proposed to support the mine site has been identified at a conceptual level in order to provide the minimum services required to support the mining operation and the beneficiation plant. In addition, the most optimum layout to best service the operation will also need to be determined. The following infrastructure is proposed:

- Access road.
- Electrical transmission line and site power.
- Beneficiation plant.
- Utilities.
- Ancillary buildings and services.
- Site preparation.
- Tailings impoundment.
- Site water management.
- Concentrate slurry pipeline.

The upgrading of the existing access road and the proposed extension to the mine site has been designed as an all-season, unpaved road which will consist of a foundation of coarse material overlain by compacted engineered fill. For the extension, the route will be cleared of trees though the muskeg layer will be compressed and topped with a geotextile layer prior to the addition of fill. This right of way will be sufficient to also house the transmission line and slurry pipeline. The most optimal route for the extension will have to be confirmed and the road will require all year maintenance.

Power to the site will be supplied through an electrical transmission line which will connect to primary power and the FCC in order to provide the surplus power generated at this location to the mine site. The design has provided for all step down transformers at both the FCC and mine site. The line capacity is sufficient to meet the demand at the mine site. The arrangement for the extent of operation and ownership will still need to be confirmed with the local provider.

The beneficiation plant is located in close proximity to the open pit mining operations where mill feed will be ground and processed to produce a slurry of phosphate rock and rejecting phosphate-lean material to a tailings management facility. This PEA study increases the phosphate rock production from 421,000 t/y (PFS) to 526,000 t/y P_2O_5 and uses the same slurry pipeline concept to deliver the concentrate to the FCC. The updated design incorporates column flotation, a third crushing stage upstream of mill feed storage and modifications to the desliming circuits. This plant is based on proven technology and will produce 1,412 ktpy of concentrate over a 26 year LOM.

Site utilities have been designed to meet the basic requirements for the mine site. The sewage treatment plant will be an off-the-shelf, sufficiently sized to meet the expected site resourcing, with a design which will enable treated water to be sent to the tailings facility. Potable water to supply basic needs will be drawn from a drilled well and treated to a quality sufficient for domestic use with the exception of drinking water which will be supplied externally. Initial fire protection has been allowed for though will require a more thorough risk assessment to determine the extent of protection for each building in the next phase of design.

The size of each building and the most effective means of construction (modular, pre-engineered or stick build) will be determined in future stages of study. The proposed site layout will still require confirmation of the most optimal layout which will be as compact as possible to minimize the extent of clearing for construction and for water retention. The current layout provides for placement of the primary maintenance facilities in proximity to the mining operations. Other buildings are in close proximity for ease of access, especially in poor weather conditions. A traffic flow analysis will also be required to ensure safe interaction between site vehicles and pedestrians. The location of the explosives magazine will also be revisited.

It is recognized that the initial preparation of the site will be most effectively performed in the winter months to enable the water laden muskeg to be removed with minimal run-off. The approach to initial site dewatering through the establishment of drainage channels and water collection areas is a logical step prior to muskeg removal. Subsequent site preparation will focus only on the areas required to establish the mine infrastructure and operations requirements and the extent of which will need to be detailed further in a subsequent phase.

The tailings impoundment area has been identified as the large footprint to the south of the beneficiation plant and contained within lined embankments built from glacial till removed during the starter open pit preparation. This containment area will be developed in phases with additional cells being constructed as required to an ultimate size for a 26 year mine life. Within the cells, the site will be clear cut although the muskeg will remain in place. The muskeg will be removed from the footprint of the lined contained embankments. The design is compliant with storage requirements for tailings in a low relief environment and based on the expectations for tailings content and settling.

Site water management has been approached from addressing diversion of non-contact water courses as the site expands and collecting all contact (site impacted) water for collection and storage in the tailings management facility. The study has completed an analysis of the natural watershed to enable understanding of water management as well as development of the water balance based on best available information. Further detail will be required in a future stage of study.

The slurry pipeline is a buried pipeline which follows the route of the site access road enabling ease of construction, maintenance and monitoring. The design for the pipeline facilities includes additional storage at the FCC, cathodic protection, monitoring (using Pipeline Adviser™, and SCADA systems) and telecommunication. The PEA pipeline is identical in length and pipe diameter to what was proposed in the 2008 PFS nevertheless an increase in capacity from 1.16 Mtpa to 1.41 Mtpa has been accomplished by increasing the slurry flow rate and selecting a pump capable of safely operating at higher pressures, up to 250 bar. A leak detection system will monitor for pipeline integrity.

It is envisaged that there will be no accommodation facilities on-site and all personnel will be transported to and from the Hearst area daily along the all-season road. Long daily commutes may eventually impact employee retention and this current base case will need to be examined in a future phase.

1.14.5 Metallurgical Testing and Recovery Methods

The metallurgical test programs described in this PEA provide the basis for developing the Martison beneficiation process. The results currently available demonstrate a viable process to proceed with additional exploration and metallurgical testing.

Fox River PEA beneficiation plant design incorporates several process modifications to the plant design utilized in the 2008 PFS study. The process modifications, which resulted from bench scale and pilot plant tests performed after the PFS, are designed to increase plant metallurgical efficiency and P_2O_5 recovery, and to improve operability. This PEA details these modifications at length.

Future test work should include testing to confirm that the coarser mesh of grind and incorporation of a WHIMS magnetic product regrind circuit will provide the intended increase in P_2O_5 recovery. In addition, trade off studies that may result in capital cost and operating costs savings should be considered by Fox River.

It is also recommended to conduct locked cycle flotation testing using Martison site water to determine if recycle of process water has an impact on flotation performance utilizing the optimized beneficiation process.

The Martison deposit is also a potential source of niobium (Nb), therefore Fox River should continue with the research and development leading to an economically feasible process to recover Nb_2O_5. Depending on the niobium R&D results, incorporation of niobium recovery into a single phosphate/niobium process flowsheet may be warranted in future studies.

1.14.6 Fertilizer Conversion Complex

The FCC proposed in this PEA is an integrated grouping of infrastructure that processes the phosphate concentrate received via pipeline to produce intermediate and final products. This infrastructure includes the following to meet the Fox River planned production schedule:

- A 1,500 t/d dihydrate phosphoric acid plant that will produce the required 500 ktpy P_2O_5.
- A 545 t/d super phosphoric acid plant will produce the required 150 ktpy P_2O_5 SPA product.
- A 4,200 t/d sulfuric acid plant that will produce the required 1,212 ktpy H_2SO_4.
- A 2,310 t/d granulation plant that will produce the required granular product mix of 474 ktpy MAP and 247 ktpy NPS.

The 1,500 t/d Dihydrate phosphoric acid plant is a proven design and capacity found in facilities throughout the world with various fertilizer companies. The design accommodates wet gypsum stacking for maximum P_2O_5 recovery. Recommended areas for design enhancement includes:

- Further study to optimize the gypsum handling.
- Engagement of gypsum stack design specialists to advance the gypsum testing and wet stack design.

The 545 t/d Super-phosphoric Acid Plant is a proven design albeit the number of super phosphoric acid manufacturers is a select group of companies. The Super phosphoric Acid Plant makes use of available steam from the sulfuric acid unit to produce a high strength liquid fertilizer product. Recommended areas for design enhancement includes:

- Further study of the Martinson rock having the target MER and MER^+ values.
- Corrosion studies for optimal selection for the process equipment.
- Optimal equipment sizing and configuration.

The 4,200 t/d Sulfuric Acid Plant is a proven design and capacity found in facilities throughout the world with various fertilizer companies. The Sulfuric Acid Plant will produce process steam for the other FCC units with enough excess to produce electrical power for the entire facility. Power inputs into the local grid will supply approximately 83% of the electrical power requirements for the slurry pipeline and beneficiation plant at the mine site. Recommended areas for design enhancement includes:

- Further investigation into blower selection and dump condenser design for possible CAPEX savings.
- Opportunities to increase the cogeneration capacity with a goal to be able to provide all electrical power requirements for both sites and minimize GHG emissions.

The 2,310 t/d Granulation Plant is a proven design and capacity found in facilities throughout the world with various fertilizer companies. The granulation plant makes use of steam, sulfuric acid, and molten sulfur to produce NPS which otherwise would be cost prohibitive in the absence of a sulfuric acid plant. Warehouse sizing and dual load-out provide year round availability of both MAP and NPS products. Recommended areas for design enhancement includes:

- Further study and pilot plant testing on the actual feed acid from the Martinson phosphate rock.
- Ammonia storage configurations.
- Sulfur addition technology.
- Filler requirements.
- Optimize warehouse capacity.

An FCC process area that meets the requirements for material unloading and handling, water treatment, wastewater treatment, and air systems is necessary to operate in an efficient and cost effective manner. The rail yard is designed to receive approximately 540,000 tonnes of inbound raw materials and approximately 950,000 tonnes of outbound finished product. Recommended areas for design enhancement includes:

- Optimization of equipment and infrastructure layout for the FCC facility (including rail layout and design).
- Alternative wastewater treatment and reuse systems to minimize waste.
- Geotechnical studies for optimal site preparations.

1.14.7 Environmental Baseline Studies, Permitting & Social or Community Impact

In the past the former owners, PhosCan, and most recently Fox River, had begun consultations with the local impacted communities and with the respective legislative departments in regard to permitting. This effort had previously established a solid foundation for the Project. In the years since the completion of the PFS, there have been changes to the legislation which require future work to update the previous work to the current requirements. The fundamental environmental findings in the EIA will continue intact or unchanged and fauna and flora, habitats, heritage, archaeological and the socioeconomic environment will remain largely as observed in 2008.

Current legislation requires all site work to be permitted well in advance of the proposed start date and involves an application of the proposed site work to MNDMF, MECP, MHSTCI and DFO, as well as consultation with impacted First Nations communities. Additional environmental baseline information will need to be collected to secure the necessary permits and approvals. To maintain desired project schedules, this baseline work needs to commence early in the next phase of study.

2. Introduction

This Preliminary Economic Assessment (PEA) report has been compiled by Hatch Ltd. (“Hatch”) from contributions from multiple sources for Fox River Resources Corporation (“Fox River”). In addition to Hatch, input has been provided from the following additional independent consultants:

- AECOM Canada Ltd
- Ausenco Ltd. (“Ausenco”).
- Chemetics.
- DMT Group Consulting Ltd. (“DMT”).
- JESA Technologies (“JT”).
- Logistic Marketing Services Inc.
- Michael R Rahm Consulting LLC.

This report is compliant with National Instrument 43-101 (NI 43-101) Standards of Disclosure for Mineral Projects.

This report provides an updated Mineral Resource Estimate (MRE) arising from the re-interpolation of all drilling, sample and metallurgical testing data gathered todate. This report also includes all preliminary mining and processing engineering study work, and related costs, revenues and economic analyses associated with the vertically integrated Project. The results of the economic analysis of the PEA are summarized in Section 22 of this report.

2.1 Site Visits

The following table identifies the site visits paid by qualified persons. Mr. Horner is the only QP who has visited the site having overseen two drilling campaigns. At the time of compilation of this PEA report, there was no additional site work in progress and the site was inaccessible as a result.

Table 2-1: Site Visits

Qualified Person	Employer	Signature	Date of Signature	Date Of Site Visit
Ken Armstrong	Chemetics	n/a	n/a	n/a
Steve Ball	Hatch	n/a	n/a	n/a
Michael Bobotis	Hatch	n/a	n/a	n/a
Rafael Davila	Hatch	n/a	n/a	n/a
Tim Horner	DMT (UK) Limited		30th of May	2008, 2012, 2014
David Ivell	JESA Technologies	n/a	n/a	n/a
Michael Kelahan	JESA Technologies	Michael E. Kelahan	5-30-2022	2007
Richard May	Hatch	n/a	n/a	n/a

2.2 Information Sources and References

The documentation listed in Section 27 was used to support this PEA Report. Excerpts or summaries from documents authored by other consultants are indicated in the text. The assessment by QPs of the Property has been based on published and unpublished material, and the data and opinions provided by other professionals.

The QPs reviewed and appraised the information used to prepare this PEA Report, including the conclusions and recommendations, and believe that such information is valid for the purpose of which this PEA Report has been prepared.

The QPs have thoroughly researched and documented the conclusions and recommendations made in this PEA Report.

2.3 Previous Technical Reports

The following NI 43-101 technical reports have been submitted in prior years for this project. These documents have been used as reference material in the preparation of this report:

- Martison Phosphate Project Technical Report, issued by James Spalding, Registered Professional Geologist dated 30 May 2007.
- Martison Phosphate Project Preliminary Feasibility Study issued by Jacobs Engineering Inc. dated May 2008.
- Technical Report On The Martison Phosphate Project, Ontario, Canada issued by DMT Consulting Limited dated 11 April 2016.

3. Reliance on Other Experts

The opinions, conclusions, recommendations, and cost estimates included in this Technical Report are based data and reports made available during the preparation of this study, with all assumptions and qualifications set forth in the subsequent Report sections.

For the purposes of this Technical Report, Richard May and Hatch Limited relied on Michael Rahm of Michael R Rahm Consulting LLC and Dave Spearin of Logistic Marketing Services Inc. for the entirety of Section 19 (Market Studies and Contracts). Richard May of Hatch Limited has reviewed the work undertaken by Michael Rahm and confirms that the contents of Section 19 is in compliance with NI 43-101 reporting standards.

In addition, for the purposes of this Technical Report, JESA Technologies relied on AECOM Canada Inc for Section 18.2.7 (Rail Yard). David Ivell of JESA Technologies has reviewed the work undertaken by AECOM and confirms that the contents of Section 18.2.7 is in compliance with NI 43-101 reporting standards.

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

4. Property Description & Location

4.1 Location

The Carbonatite Complex, which contains the Project's phosphate and niobium resources and the proposed mine site area, is located approximately 70 km northeast of the town of Hearst, Ontario, in the James Bay Lowlands. The mine site itself is located in the "South of Ridge Lake" area (township) and centered on Long / Lat 50° 18' 52" N., 83° 24' 52" W. (UTM NAD 87/WGS 84 328300; 5576400) and is currently accessible only in the winter months for the purposes of further site works and advancement of the Project. The location is illustrated in Figure 4-1 below.

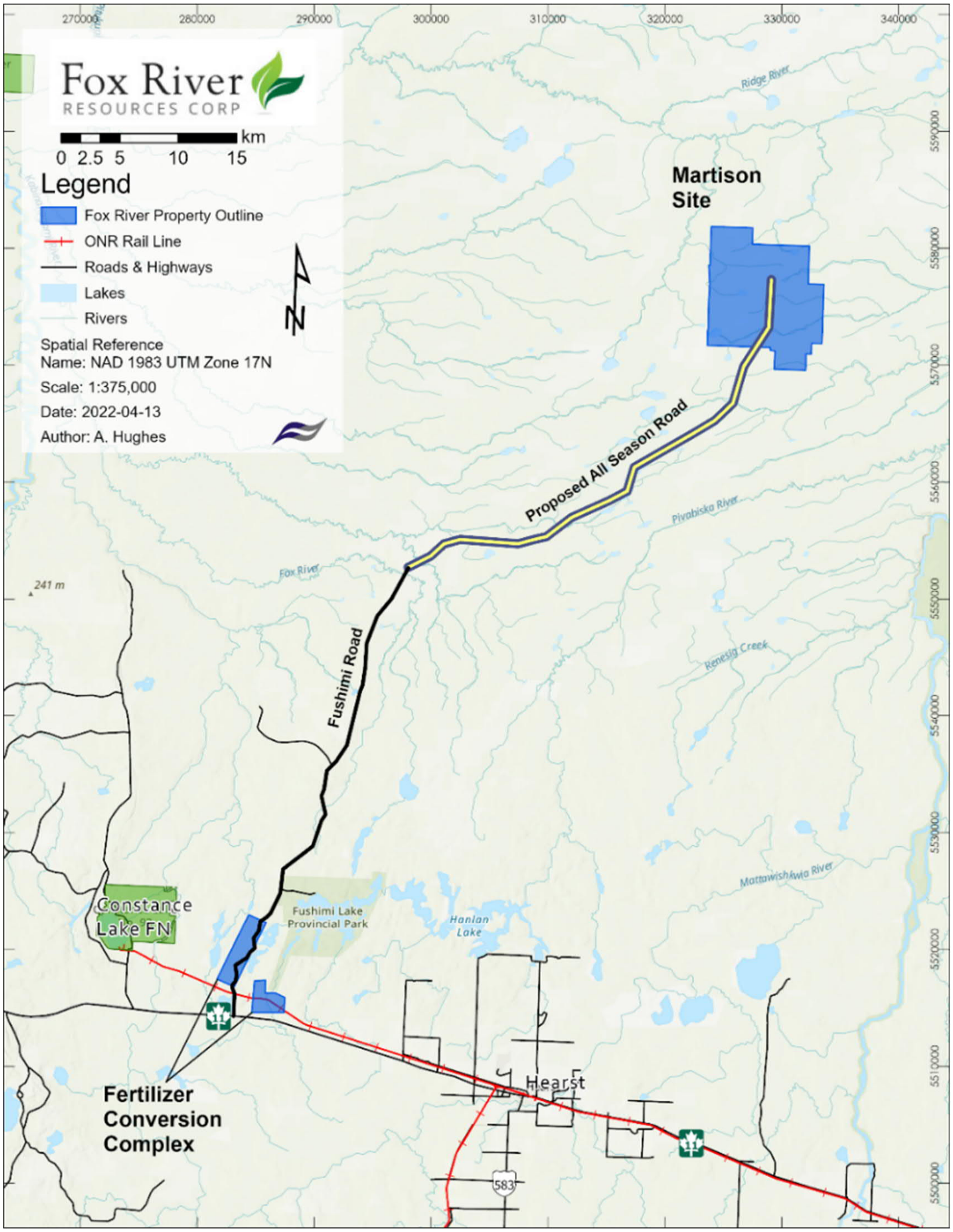


Figure 4-1: Project Location (Martison Mine Site)

The preferred means of access to the proposed mine site, and the route which has been used during past exploration programs, is via the Fushimi Road that intersects the primary Highway 11 west of Hearst. The total length of this route from Hearst to the mine site is approximately 112 km, comprised of 26 km of Highway 11, 51 km on the Fushimi gravel road, and 38 km of a seasonally constructed snow and ice route which has been used for winter access to the deposit.

This route is favored as the long term primary access since it has the approval of the Constance Lake First Nations (CLFN) on whose tribal lands the Project is located. The CLFN reserve is located on Highway 663 approximately 80 km southwest of the Project area and 10 km west of the Fushimi Road (Figure 4-1).

4.2 Mineral Claims

The property consists of three mining leases numbers LEA108639 (expires 29 April 2032), LEA 108638 (expires 29 April 2032) and LEA 107438 (expires 30 July 2023) and contiguous mineral claims totaling 265 units (expiring variously between March and August 2024), which together comprise approximately 8,256 hectares.

The mineral leases and all claims are located within the "South of Ridge Lake" area, Porcupine Mining Division, Cochrane Land Titles & Registry Division, Province of Ontario.

The claims are registered in the name of Fox River Resources Corporation and Baltic. Each company owns title to 50% of such lease and claims. Baltic is a wholly owned subsidiary of Fox River, such that Fox River owns directly or indirectly 100% of the Martison Phosphate Project.

In connection with the Agreement registered May 5, 2016, Fox River have now all of the rights, title and interest to the mining claims and leases comprising the Martison Project.

A complete mining lease and mineral claim listing is presented in Table 4-1 and Table 4-2 respectively. The details of which have been confirmed by DMT by reference to the Ontario Government Ministry of Northern Development, Mines, Natural Resources and Forestry website, MLAS map viewer.

Table 4-1: Martison Mining Leases as Of October 2021[1]

Number	Type	Status	Expiry Date	Claim Units	Hectares
Surveyed					
LEA 107438	Lease	Active	30/07/2023	14	226.305
LEA 108638	Lease	Active	29/04/2032	123	1,950.97
LEA 108639	Lease	Active	29/04/2032	134	2,078.55

Table 4-2: Martison Mineral Claims as Of October 2021[1]

Number	Type	Status	Expiry Date
Estimated			
100203	Claim	Active	27/06/2024
100444	Claim	Active	27/06/2024
100445	Claim	Active	27/06/2024
100446	Claim	Active	27/06/2024
100447	Claim	Active	27/06/2024
100455	Claim	Active	10/04/2024
101151	Claim	Active	10/04/2024
101152	Claim	Active	10/04/2024
101153	Claim	Active	10/04/2024
102496	Claim	Active	15/03/2024
102497	Claim	Active	15/03/2024
103820	Claim	Active	16/04/2024
104205	Claim	Active	15/03/2024
104783	Claim	Active	16/04/2024
105113	Claim	Active	11/08/2024
115000	Claim	Active	15/03/2024
115001	Claim	Active	15/03/2024
116440	Claim	Active	10/04/2024
116689	Claim	Active	15/03/2024
116690	Claim	Active	10/04/2024
116769	Claim	Active	16/04/2024
117087	Claim	Active	15/03/2024
117631	Claim	Active	10/04/2024
117632	Claim	Active	10/04/2024
119315	Claim	Active	15/03/2024
119316	Claim	Active	15/03/2024
119448	Claim	Active	15/03/2024
119449	Claim	Active	15/03/2024
119450	Claim	Active	10/04/2024
121043	Claim	Active	16/04/2024
121206	Claim	Active	15/03/2024
121207	Claim	Active	15/03/2024
121860	Claim	Active	15/03/2024
121862	Claim	Active	15/03/2024
123166	Claim	Active	10/04/2024

Number	Type	Status	Expiry Date
123167	Claim	Active	10/04/2024
123168	Claim	Active	10/04/2024
123911	Claim	Active	27/06/2024
123912	Claim	Active	27/06/2024
124015	Claim	Active	11/08/2024
124375	Claim	Active	15/03/2024
125972	Claim	Active	10/04/2024
126977	Claim	Active	15/03/2024
127917	Claim	Active	16/04/2024
128255	Claim	Active	27/06/2024
128256	Claim	Active	27/06/2024
135366	Claim	Active	16/04/2024
135404	Claim	Active	27/06/2024
135949	Claim	Active	27/06/2024
135950	Claim	Active	27/06/2024
136003	Claim	Active	11/08/2024
137960	Claim	Active	10/04/2024
139239	Claim	Active	10/04/2024
143449	Claim	Active	27/06/2024
143451	Claim	Active	15/03/2024
143452	Claim	Active	15/03/2024
144150	Claim	Active	15/03/2024
144151	Claim	Active	15/03/2024
151687	Claim	Active	10/04/2024
151688	Claim	Active	10/04/2024
151689	Claim	Active	10/04/2024
156772	Claim	Active	15/03/2024
157551	Claim	Active	27/06/2024
159662	Claim	Active	15/03/2024
160195	Claim	Active	16/04/2024
160340	Claim	Active	15/03/2024
160343	Claim	Active	15/03/2024
163570	Claim	Active	27/06/2024
163571	Claim	Active	27/06/2024
163572	Claim	Active	27/06/2024
163573	Claim	Active	27/06/2024
163574	Claim	Active	27/06/2024
166408	Claim	Active	15/03/2024

Number	Type	Status	Expiry Date
166409	Claim	Active	15/03/2024
166410	Claim	Active	11/08/2024
168397	Claim	Active	16/04/2024
168434	Claim	Active	27/06/2024
168975	Claim	Active	15/03/2024
168976	Claim	Active	15/03/2024
168977	Claim	Active	15/03/2024
170516	Claim	Active	10/04/2024
172409	Claim	Active	16/04/2024
179049	Claim	Active	16/04/2024
179190	Claim	Active	15/03/2024
179191	Claim	Active	15/03/2024
179837	Claim	Active	15/03/2024
179838	Claim	Active	15/03/2024
179839	Claim	Active	10/04/2024
180462	Claim	Active	27/06/2024
181068	Claim	Active	11/08/2024
181094	Claim	Active	10/04/2024
181787	Claim	Active	15/03/2024
181788	Claim	Active	15/03/2024
181869	Claim	Active	27/06/2024
181870	Claim	Active	27/06/2024
181871	Claim	Active	27/06/2024
185223	Claim	Active	15/03/2024
185224	Claim	Active	15/03/2024
185869	Claim	Active	11/08/2024
185870	Claim	Active	15/03/2024
185871	Claim	Active	10/04/2024
185872	Claim	Active	11/08/2024
187375	Claim	Active	16/04/2024
187376	Claim	Active	16/04/2024
188008	Claim	Active	11/08/2024
188009	Claim	Active	11/08/2024
191394	Claim	Active	16/04/2024
194910	Claim	Active	10/04/2024
194974	Claim	Active	16/04/2024
202115	Claim	Active	15/03/2024
202854	Claim	Active	10/04/2024

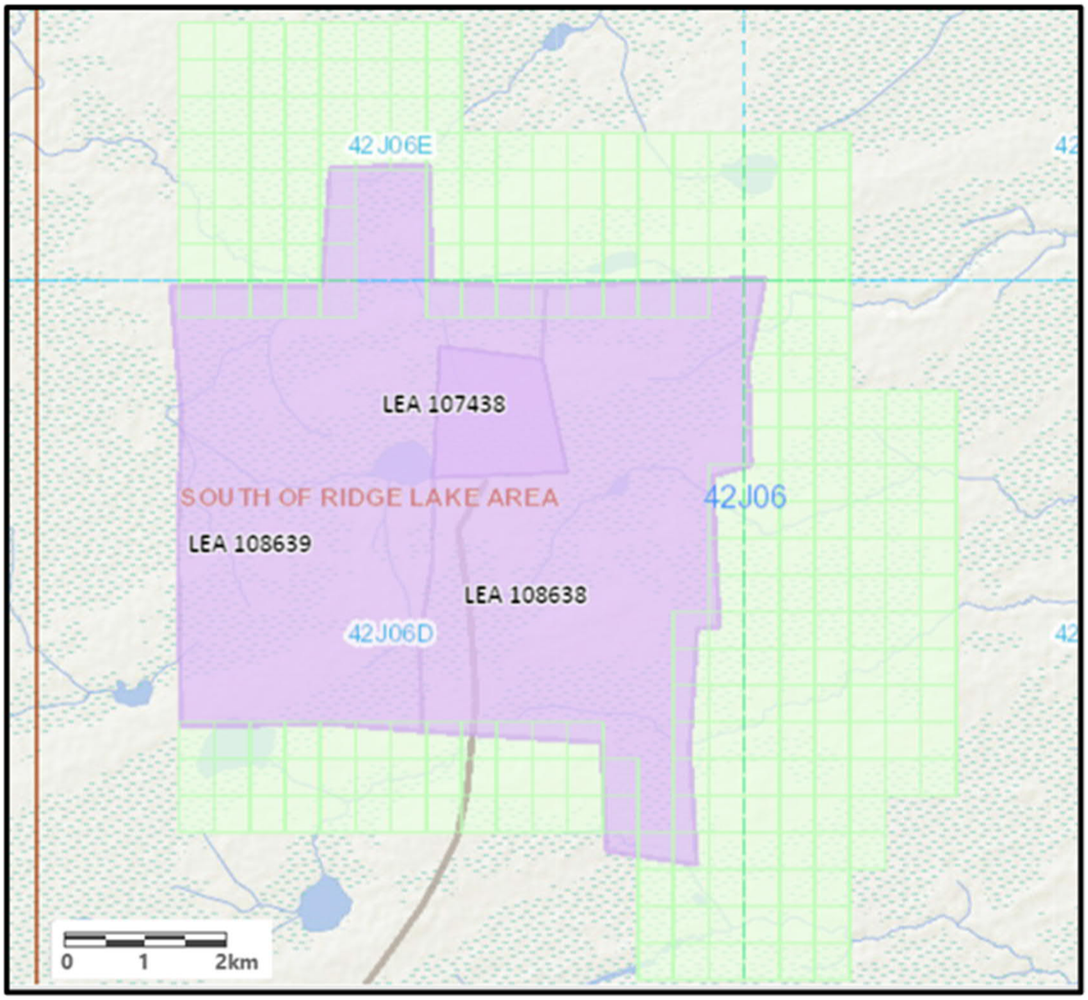
Number	Type	Status	Expiry Date
205629	Claim	Active	10/04/2024
205630	Claim	Active	10/04/2024
205631	Claim	Active	10/04/2024
205632	Claim	Active	10/04/2024
206288	Claim	Active	15/03/2024
210150	Claim	Active	15/03/2024
210151	Claim	Active	15/03/2024
210152	Claim	Active	15/03/2024
210174	Claim	Active	15/03/2024
210175	Claim	Active	15/03/2024
211448	Claim	Active	27/06/2024
211449	Claim	Active	27/06/2024
211452	Claim	Active	10/04/2024
211913	Claim	Active	10/04/2024
214151	Claim	Active	16/04/2024
214152	Claim	Active	16/04/2024
215148	Claim	Active	15/03/2024
215149	Claim	Active	15/03/2024
216450	Claim	Active	10/04/2024
216451	Claim	Active	10/04/2024
216452	Claim	Active	11/08/2024
216988	Claim	Active	10/04/2024
217196	Claim	Active	16/04/2024
217240	Claim	Active	27/06/2024
217669	Claim	Active	15/03/2024
217822	Claim	Active	11/08/2024
217823	Claim	Active	11/08/2024
222197	Claim	Active	15/03/2024
222198	Claim	Active	15/03/2024
222199	Claim	Active	15/03/2024
222983	Claim	Active	27/06/2024
222984	Claim	Active	27/06/2024
223469	Claim	Active	10/04/2024
225047	Claim	Active	15/03/2024
225643	Claim	Active	16/04/2024
227281	Claim	Active	10/04/2024
228653	Claim	Active	16/04/2024
228978	Claim	Active	15/03/2024

Number	Type	Status	Expiry Date
230269	Claim	Active	27/06/2024
230272	Claim	Active	15/03/2024
232347	Claim	Active	15/03/2024
232921	Claim	Active	16/04/2024
232922	Claim	Active	16/04/2024
233185	Claim	Active	15/03/2024
233186	Claim	Active	15/03/2024
233187	Claim	Active	15/03/2024
233188	Claim	Active	15/03/2024
233189	Claim	Active	10/04/2024
234352	Claim	Active	27/06/2024
235002	Claim	Active	10/04/2024
235700	Claim	Active	15/03/2024
235758	Claim	Active	27/06/2024
235759	Claim	Active	27/06/2024
237275	Claim	Active	16/04/2024
237398	Claim	Active	11/08/2024
237399	Claim	Active	11/08/2024
237400	Claim	Active	11/08/2024
237401	Claim	Active	11/08/2024
240799	Claim	Active	16/04/2024
245887	Claim	Active	15/03/2024
252016	Claim	Active	15/03/2024
252017	Claim	Active	15/03/2024
254030	Claim	Active	16/04/2024
254031	Claim	Active	16/04/2024
254032	Claim	Active	16/04/2024
254066	Claim	Active	27/06/2024
254099	Claim	Active	27/06/2024
256627	Claim	Active	10/04/2024
256628	Claim	Active	10/04/2024
258170	Claim	Active	15/03/2024
258171	Claim	Active	15/03/2024
258192	Claim	Active	15/03/2024
258925	Claim	Active	27/06/2024
258928	Claim	Active	11/08/2024
259413	Claim	Active	10/04/2024
260407	Claim	Active	15/03/2024

Number	Type	Status	Expiry Date
260408	Claim	Active	15/03/2024
261596	Claim	Active	16/04/2024
263538	Claim	Active	27/06/2024
264180	Claim	Active	10/04/2024
264181	Claim	Active	10/04/2024
269000	Claim	Active	15/03/2024
269567	Claim	Active	16/04/2024
269568	Claim	Active	16/04/2024
269569	Claim	Active	16/04/2024
269570	Claim	Active	16/04/2024
271091	Claim	Active	15/03/2024
271092	Claim	Active	15/03/2024
271095	Claim	Active	15/03/2024
271096	Claim	Active	15/03/2024
271661	Claim	Active	10/04/2024
271662	Claim	Active	10/04/2024
272523	Claim	Active	11/08/2024
276157	Claim	Active	15/03/2024
277378	Claim	Active	10/04/2024
277379	Claim	Active	10/04/2024
278983	Claim	Active	27/06/2024
278988	Claim	Active	10/04/2024
280504	Claim	Active	15/03/2024
281165	Claim	Active	11/08/2024
281166	Claim	Active	15/03/2024
281167	Claim	Active	15/03/2024
281171	Claim	Active	15/03/2024
283183	Claim	Active	16/04/2024
283184	Claim	Active	16/04/2024
283217	Claim	Active	27/06/2024
283616	Claim	Active	27/06/2024
283733	Claim	Active	10/04/2024
283734	Claim	Active	10/04/2024
283735	Claim	Active	10/04/2024
283807	Claim	Active	11/08/2024
283808	Claim	Active	11/08/2024
284952	Claim	Active	15/03/2024
287218	Claim	Active	16/04/2024

Number	Type	Status	Expiry Date
288275	Claim	Active	15/03/2024
289107	Claim	Active	15/03/2024
289108	Claim	Active	15/03/2024
289257	Claim	Active	15/03/2024
289258	Claim	Active	15/03/2024
290428	Claim	Active	27/06/2024
291077	Claim	Active	10/04/2024
291273	Claim	Active	27/06/2024
291763	Claim	Active	15/03/2024
291764	Claim	Active	15/03/2024
295515	Claim	Active	15/03/2024
295856	Claim	Active	16/04/2024
296314	Claim	Active	27/06/2024
298869	Claim	Active	15/03/2024
298870	Claim	Active	15/03/2024
298961	Claim	Active	16/04/2024
313973	Claim	Active	10/04/2024
324849	Claim	Active	15/03/2024
325131	Claim	Active	16/04/2024
326088	Claim	Active	27/06/2024
328230	Claim	Active	16/04/2024
328275	Claim	Active	15/03/2024
328934	Claim	Active	15/03/2024
328936	Claim	Active	10/04/2024
330201	Claim	Active	27/06/2024
330815	Claim	Active	11/08/2024
330997	Claim	Active	27/06/2024
331088	Claim	Active	11/08/2024
340840	Claim	Active	15/03/2024
342040	Claim	Active	27/06/2024
342041	Claim	Active	27/06/2024
342042	Claim	Active	27/06/2024
342043	Claim	Active	27/06/2024
342044	Claim	Active	27/06/2024
342693	Claim	Active	10/04/2024
342844	Claim	Active	27/06/2024

Number	Type	Status	Expiry Date
343384	Claim	Active	15/03/2024
343385	Claim	Active	15/03/2024
343439	Claim	Active	11/08/2024
343440	Claim	Active	11/08/2024



Source: NDMNRF (MLAS) Green – Mineral claims only. Purple – Mining lease and mineral claims

Figure 4-2: Fox River Mining Leases and Claims 'South of Ridge Lake Area', Porcupine District

4.3 Permits & Environmental Liabilities

Fox River also holds an aggregate permit to the south of the Project area approximately 33 km west of Hearst and 27 km north of Highway 11 on the west side of the Fushimi Road. This location is a former aggregate pit and a source of sand and gravel, which is expected to provide suitable construction materials for the access road, mine site roads and mine site infrastructure. Fox River aggregate pit permit (# 624977) was approved by the NDMNRF in 2009.

Within the mining leases, up to 10% of the surface rights are withheld for future public transportation routes. Also withheld are unspecified areas for the future development of hydropower infrastructure, power transmission and hydrocarbon pipeline corridors, as well as free use and passage upon all navigable waterways including access to these locations.

The mineral claims withhold surface rights up to 122 m around all lakes and rivers, including land under water, as well as reserving all sand, gravel and peat deposits.

The Project property is located on lands which Constance Lake First Nations asserts are their traditional lands and to which the First Nation asserts it holds as constitutionally protected rights. The former owners, PhosCan, entered into Exploration Agreements with CLFN regarding exploration and development of the Project. It is predicted that Fox River will establish similar agreements with CLFN ahead of any future exploration work at the mine site.

4.4 Royalties

The mine site leases and claims controlled by Fox River and its subsidiary, Baltic, are subject to a Net Sales Returns ("NSR") royalty. The royalty amount payable is 1% of NSR on phosphate concentrate. Additionally, a Production Royalty is to be paid which varies with the price of phosphoric acid and is payable on each tonne of phosphate concentrate produced.

Prior to the commencement of commercial production, Fox River may elect to acquire the 1% NSR royalty for a payment of CAD 3,000,000.

Further, a NSR for Special Products of 2% of all special products sold is in place. "Special products" does not include any "ores" sold on the basis of their phosphate content, phosphate concentrate, any and all products manufactured downstream of the phosphate beneficiation plant, or any aggregate used for the purposes of the Martison Phosphate Project.

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

5.1 Accessibility

The terrain overlying the Martison Phosphate deposit consists of spruce forest, wet muskeg, and numerous small lakes, rivers and creeks. The maximum reported depth for the lakes in the area is 4 m. Local relief is minimal with variations of only a few metres, resulting in ground which is very poorly drained and consequently varying between very wet and semi-wet terrain. The maximum relief over the Anomaly A deposit itself is reported to be only 2.5 m, although relief varies from a maximum height of 199 m AMSL in the southwest extent to below 180 m in the northwest extent of the mine site area. These wet and challenging conditions currently limit access to the site for field activities to only the winter season, necessitating the use of local construction contractors to maintain winter roads.

To permanently access the site, construction of an all-season road from Highway 11 to the mine site will be required and consisting of approximately 90 km of new build and upgrades to the existing Fushimi Road. The access road would need to be constructed prior to bringing equipment and materials to the Project site for Mine initial dewatering, pre-mining and site infrastructure construction activities. The section for road upgrade and the proposed extension to the Project site is illustrated in Figure 4-1.

Preliminary construction work on the all–season road construction commenced in the summer of 2008. This work focused on upgrading of sections of the Fushimi Road, notably with a significant upgrade of the wooden single track, axle weight limited, Fox River bridge, which was dismantled and a wider, twin track, dual culvert pipe crossing installed to facilitate the transport of the required heavy equipment for the construction of the new access road. The new crossing with culverts is shown in Figure 5-1. Subsequently, in the years since that time, no further construction work has been carried out on the existing road upgrade or any construction commenced on the northern (new build) section of the all–season access road.

Figure 5-1: Fushimi Road – Fox River Culvert Construction

After construction of the culvert crossing, the water flow rates were monitored periodically by Blue Heron Solutions of Sudbury, Ontario, and in September 2009 they reported that the culverts were adequately sized and that their installation does not impede either migration or spawning of local fish species (Brook Trout) because of low water velocities (less than 0.7 m/s).

5.2 Climate

The climate in the Project area is typical of a Canadian mid-continental climate with long cold winters and short warm summers. Temperatures vary dramatically over short time intervals. The region experiences five months of often very cold winter and four months of warm summer, with milder and wetter weather in the spring and autumn transition periods between.

The nearest national weather station, with substantial historical records, is located at Kapuskasing, 125 km southeast of the Project site. The period of records kept here is from 1971 through 2000. At this location, the average annual temperature is 0.8°C, ranging from an average daily temperature of -18.7°C in January to 17.2°C in July. The average annual precipitation is 83.2 cm with the most occurring in July and the least in February. The average annual wind velocity is 12.6 km/h from the southwest. Sea level atmospheric pressure averages 101.6 kPa.

The mine site location in winter is more exposed due to low relief and low stunted tree cover and can therefore be subject to very low temperatures with the associated wind chill. During the 2008 drill campaign, temperatures in January and February routinely fell below -30°C at night, daytime temperatures were typically mid -20°C. In spring, thaw water typically ponds, particularly on the drill pad access roads due to the lack of topography and gradient around the site Figure 5-2.

5.3 Hydrology

On site surface waters consist of numerous first or second order headwater streams.

The proposed open pit at the mine site sits astride a watershed. This drains on the east side towards the headwaters and catchment of the Soweska River and Moose River system and surface water drains slowly on the west side into the Upper Ridge River and downstream into the Albany River system.

The hydrogeological test work undertaken in 2008 and 2012 has demonstrated that the deposit aquifer conditions have measurable boundaries and a relatively slow recharge rate, which will allow for the design of a dewatering solution for a proposed open pit operation.

5.4 Local Resources

The Project mine site is located 70 km by air from Hearst and the FCC is in close proximity to rail, power, highway and other industrial infrastructure in the area. It is maintained that, pending further study, sufficient water is available nearby the site for mining and industrial use, which will, in part, be provided by the pumping of the significant aquifer sources available at the mine site.

Local socioeconomic resources are limited due to the paucity of population in the region. Basic food, lumber, exploration supplies and fuel can be purchased in Hearst, while supplies required to support the needs of the Project can be obtained in Timmins, Ontario (which is a long established primary mining centre). The residents of Hearst are favourably committed to the responsible development of the natural resources of the region and are eager for the new employment opportunities such an undertaking would bring.

The CLFN community has played a significant role in supporting the development of the Project through the provision of their approval for necessary access permits and also with the deployment of skilled and unskilled labour to assist during site related work activities. The continued support and cooperation of the CLFN is considered an essential requirement for advancing the Martison Project.

5.5 Existing Infrastructure

No infrastructure of value to the Project mine site currently exists other than a seasonal winter trail that is used, after seasonal construction of a snow and ice road, to connect the north end of the Fushimi Road with the Project site where a series of drill pads, other cleared areas and a network of interconnecting trails have been established from exploration activity. These are visible in Figure 5-2.

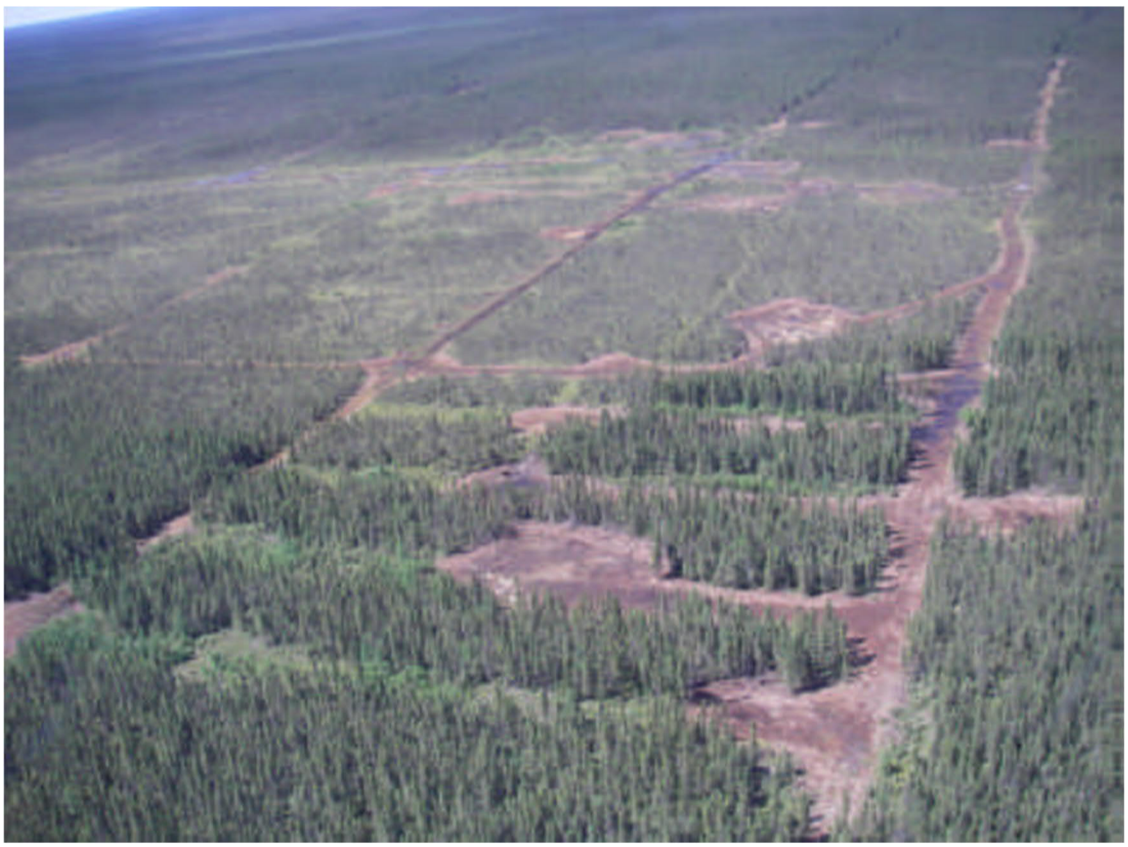


Figure 5-2: View looking North West across the central part of Anomaly A

As a result of the Project mine site remaining winter access only, the requirements for eventual advancement of the Project would, in the first instance, include the construction of all season access road to enable site preparation, construction of mine based infrastructure and for the long term through the life of the mine. This access road will connect Highway 11 to the mine site and will require approximately 90 km of combined new build and upgrading of the existing Fushimi Road. The new all-season build (approximately 45 km), will follow a different route to that currently used for winter access traversing through thin boreal forest and saturated shallow muskeg.

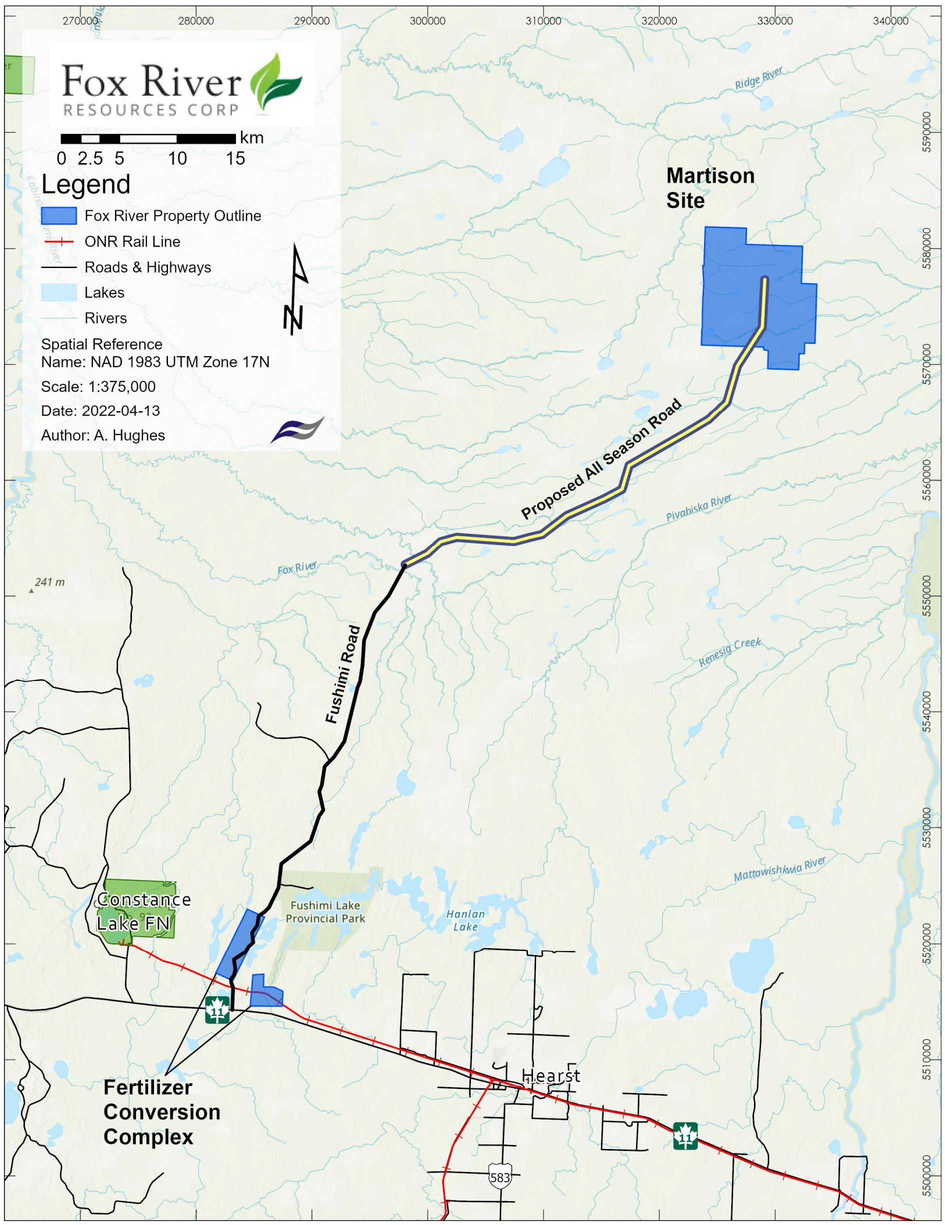


Figure 5-3: Proposed All Season Access Road Route

An all-season road access to the Project site, as shown in Figure 5-3, would remove the current uncertainty associated with seasonal winter route construction caused by the annual fluctuations in climatic conditions. Climatic trends over the last number of decades have identified that warmer winters are occurring with increased frequency which make winter road construction much more difficult and reducing the period of time for which these temporary roads are functional.

The underlying areas at the proposed Project mine site, open pit and required mine infrastructure (the waste facility, mine buildings, beneficiation plant and tailings impoundment area) are saturated shallow muskeg perched on a thick layer of impermeable glacial till. A coordinated dewatering operation will be required for the development of the site infrastructure and operations. Site preparation activities for the open pit mining area and beneficiation plant would necessarily include, excavating the muskeg to the glacial till layer for the footprint and construction of impermeable perimeter berms which will be built with material to an elevation above grade to prevent water inflow.

An access ‘corridor’ leading from the mine site to the rail road at the south end of the Fushimi Road and Highway 11 junction will also carry an overhead mine power line and a slurry pipeline as a means to transport phosphate concentrate from the mine site to a load out and processing facility near the Fushimi Road railhead (running parallel to Highway 11 and shown in Figure 5-3).

Separate from the mine site, Fox River holds aggregate claims to an area of known sand and gravel bearing ground, which was used first by the local logging companies as a former quarry, providing material for the construction of the Fushimi Road and would be a source of material for the all-season road construction and subsequent road maintenance.

5.6 Physiography

The Project mine site is situated in the large expanse of “low ground” southwest of Hudson/James Bay, referred to as the James Bay Lowlands. The property comprises very gently rolling terrain dominated by muskeg and black spruce swamp. There are no exposures of the source rock carbonatite or the surrounding country rock, and all geological data result from drilling information and interpretations from geophysical surveys.

The underlying areas at the proposed project mine site are of saturated shallow muskeg perched on impermeable glacial till. A coordinated dewatering operation will be required for the development of the site infrastructure and operations.

The Martison deposit is located astride a major drainage divide with generally the western portion draining into the Albany River System and the eastern portion draining into the Moose River System.

6. History

6.1 Historical Ownership

Carbonatite complexes occur in several parts of Northern Ontario and some of them have been explored for minerals for many years. The Carbonatite Complex was located by an airborne magnetic-electromagnetic survey in 1965 carried out by the Ontario Geological Survey and the Geological Survey of Canada. Also in 1965, ground surveys indicated a conductive zone approximately 500 metres long with a coincident magnetic anomaly. This work, along with a hole drilled in the anomaly, was conducted by a consortium that included Falconbridge Nickel Mines, Uranium Ridge Mines Limited, and Matachewan Consolidated Mines Limited.

The Carbonatite Complex was originally, and incorrectly, referred to as the Martison Lake Carbonatite Complex and is actually named after N.W. Martison, a Shell geologist who explored the area for petroleum in 1946.

In 1967, the large northern magnetic anomaly (Anomaly A) was covered by 98 claims staked by an unknown party, probably Goldray Mines Ltd. An airborne magnetometer survey was completed, and the resulting anomaly was recommended for testing by drilling. The drilling program was never carried out and the claims were allowed to lapse. The existence of the Carbonatite Complex was first formally interpreted in 1970 by the Ontario Department of Mines and Northern Affairs partly on the basis of the 1965 drillhole.

In 1980, Shell Canada Resources Limited staked 222 mining claims in a single contiguous block over the interpreted Carbonatite Complex. In order to more precisely map the Carbonatite Complex, which is completely buried by overburden and contains no rock outcrops, an airborne geophysical survey was completed in February 1981. In March and April of 1981, five drillholes were completed, three were centred on Anomaly A and two on Anomaly B. Based on the interpreted results from this work, a large field campaign was planned for the 1982 winter season.

An additional 124 contiguous claims were staked in 1981 by Shell Canada. In late 1981, seismic and DC resistivity test surveys were completed on Anomaly A (between drillholes 81-03 and 81-04) to evaluate the methods for determining the thickness of the residuum. The tests were successful in outlining the carbonatite but unsuccessful in determining residuum thickness.

In February 1982, Shell Canada made the decision to sell the Martison property to Eastern Petroleum Corporation and Camchib Mines Incorporated, with Camchib being the operator for the joint venture. However, pending the completion of the sale, the field program was conducted under the direction of Shell Canada. The program consisted of 38 drillholes (including one redrill) completed between January 19 and April 5 using hole spacings of 200 m to 400 m. A total of 32 holes were completed using RC drilling methods and six using sonic drilling techniques. Lakefield Research of Canada Limited conducted beneficiation tests for the production of phosphate and niobium concentrates using sonic core from drillholes 82-32, 82-34, and 82-36.

The divestiture of the Martison property by Shell Canada was completed in December 1982.

The 1983 field program, under the direction of Camchib, began on February 9 and drilling operations were complete by March 29. A total of 29 drillholes were completed using a mixture of sonic drilling techniques and RC techniques where drilling conditions dictated. The sonic drilling methods permitted the collection of cores for use in lithological descriptions and beneficiation testing. Additionally, geological, geochemical, geophysical, and geotechnical studies were completed in 1983. Comprehensive beneficiation batch and closed-cycle bench tests for phosphate and niobium concentrate production and additional residuum microscopic studies were completed.

In January 1984, Kilborn Limited completed a “Preliminary Capital and Operating Cost Estimate for an Open Pit Mine/Mill Complex” at the Martison deposit. This work was completed on behalf of Camchib.

In 1984, from January 13 through March 29, a total of 37 drillholes were completed (including four redrills) by two drilling contractors. Of this total, 15 holes were completed using a combination of a standard diamond drill penetrating through the glacial till and Cretaceous sediments using a tricone bit and NQ coring through the residuum. Sonic drilling techniques were used to recover core from 17 holes. Five holes were completed using RC methods. Unfortunately, the program generally called for drilling to a predetermined depth of 76.2 m, regardless of the geology, and 22 of the drillholes were “stopped” prematurely in the residuum for this reason. Additionally, drilling problems and/or equipment capacities forced the stoppage of another three holes in the residuum. During the drilling program, a test was completed comparing drill cuttings recovered from the circulating medium with the chemical analyses of the core recovered over the same interval. This test indicated that the cutting’s analyses and the core analyses generally compared favourably.

Also in 1984, Camchib drilled two large, 48-inch (1.22 m) diameter churn drillholes to collect a bulk residuum sample for beneficiation pilot plant studies at Lakefield Research, Florida. The location for this bulk sample was selected adjacent to the drillhole 83-60. Technical problems forced the early abandonment of the first attempt at 32 m, a second hole was stopped short of the target depth at approximately 70 m, again due to technical problems. Approximately 65 t of residuum sample were received at Lakefield for sampling and testing.

During June to July of 1984, a sample of concentrate from Lakefield's work was evaluated at the International Fertilizer Development Centre in Muscle Shoals, Alabama. The study tested the viability of producing phosphoric acid from the Martison concentrate by acidulation with sulfuric acid.

During the period from 1985 to 1987, no further field work was completed on the property. Camchib continued to study the merits of various production plans but was unable to conclude that it could penetrate the fertilizer market without a partner already engaged in the business. Thus, in 1987, Camchib formed a partnership with Sherritt Gordon Limited whereby Camchib contributed the Martison property and Sherritt the Kapuskasing deposit to a new entity in which each company held 50%.

In 1987, under contract to the Ontario Ministry of Northern Development and Mines, Jacobs Engineering and Blue, Johnson & Associates completed a summary evaluation of the prospects for development of the weathered carbonatite phosphate deposits in Ontario. Although the detailed study included both the Kapuskasing and Martison deposits, the study recommended that the Kapuskasing deposit be advanced as it was further developed.

In 1989, Camchib sold its 50% interest in Kapuskasing and Martison to Newphos Ltd., a wholly owned subsidiary of Central Capital Corporation (CCC). Work began in earnest on the Kapuskasing deposit following the sale. Due to the pre-occupation with Kapuskasing, interest in the Martison deposit diminished.

During 1993, Sherritt allowed the Martison claim block to expire through lack of timely filing of assessment work. In the same year, McKinnon Prospecting of Timmins, Ontario, established a new claim block covering the Carbonatite Complex.

The historical Martison drilling programs are summarised in Table 6-1 and Table 6-2 below.

Table 6-1: Anomaly A – Historical Drilling Program Summary

Year	Company	No. Holes	Total Metres	Type and Comment
1981	Shell	3	278.8	DD
1982	Shell	37	2,919.1	DD
1983	CAMCHIB	29	2,782.0	DD
1984	CAMCHIB	35	2758.5	DD (Includes 3 redrills) 2 x churn drill for metallurgical bulk sample

Table 6-2: Anomaly B – Historical Drilling Program Summary

Year	Company	No. Holes	Total Metres	Type and Comment
1981	Shell	2	275	DD
Total		**2**	**275**	

6.2 MCK Mining (PhosCan) 1997-2012

In early 1997, MCK Mining Corporation, formerly named Hendricks Minerals Canada Limited, was reorganised to more aggressively pursue advanced mining projects. MCK entered an agreement with Baltic Resources Inc. whereby each would earn 50% in the Martison property from Donald McKinnon, principal and owner of McKinnon Prospecting, by completing work and issuing shares pursuant to an option and joint venture agreement. After having met all of the requirements under the option agreement, both MCK and Baltic earned their respective 50% ownership interest in Martison. Both parties signed the Martison Joint Venture Agreement which governed their relationships with respect to Martison and provided for production royalties to McKinnon (Section 4.4).

J.H. Reedman & Associates Ltd. completed a computer model for the Martison deposit and an “open pit resource” estimate for a mining period of 10 years. This work was completed in early 1997 for McKinnon Prospecting.

Also, in 1997, MCK Mining Corporation engaged MRDI to re-evaluate the previously collected data and to complete a Scoping Study for the Martison Project. The Scoping Study evaluated the geology, constructed a computer resource model, presented a “reserves” statement, and completed a project level estimate of capital and operating costs for the development and operation of a mine and beneficiation plant. The final report was issued in May 1998.

Using several contractors in 1997, MCK and Baltic examined fertilizer markets, regional sulfuric acid production and forecasts, regional freight rates, fertilizer manufacturing plant capital and operating costs, and alternative financing and tax handling schemes.

In January 1998, a brief field programme by MCK Mining Corporation evaluated the use of lake sediment samples as a carbonatite exploration tool. Although the lake sediment samples were collected and analysed, the programme was never completed to the point where definitive conclusions were published.

In late 1998, an agreement was reached between MCK Mining/Baltic Resources and Cargill Fertilizer, Inc. whereby Cargill would “purchase” six of the 13 drillholes scheduled for drilling in January 1999 (CG series holes). Cargill would use the data generated from beneficiation tests on these six holes as well as other MCK/Baltic data to complete its own evaluation of the Martison deposit.

In 1999, from February 22 to March 27, a total of 14 drillholes (including one redrill) were completed under the field supervision of MCK (CG series & M-99 series). All holes were continuously cored from the surface to total depth using triple tube HQ coring technology. The locations of the holes were along the previously defined “economic axis” of Anomaly A and provided some infill drilling as well as corroboration of earlier work. Cargill’s report issued in October 1999 indicated a favourable result and held out the possibility of a simplified beneficiation process flowsheet as compared to earlier work. The report also generally confirmed earlier MCK resource estimates and recommended a slurry pipeline for concentrate transport to a rail siding at Hearst for drying and load-out prior to transport.

Also in 1999, an aeromagnetic survey was conducted over the Carbonatite Complex (Figure 6-1).

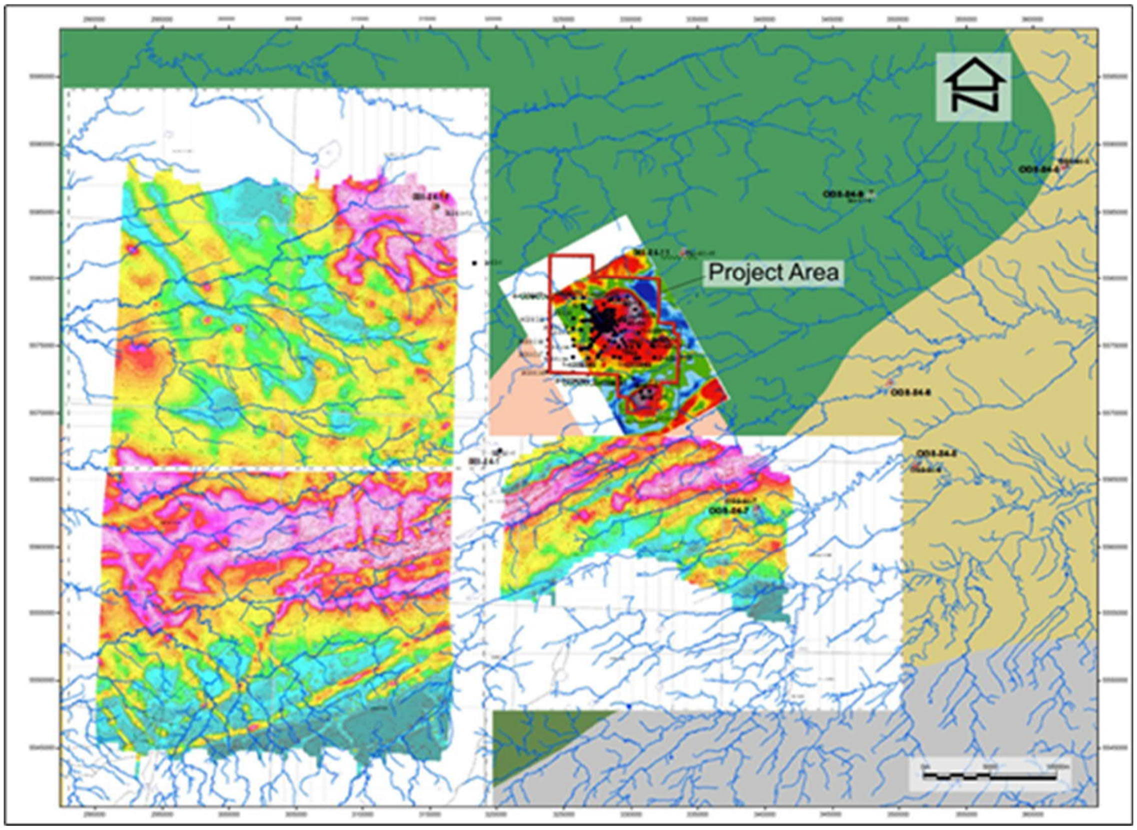


Figure 6-1: Aeromagnetic Survey – Martison Project & Region. (SEM 1999)

From February 17 to April 3, 2001, a total of 12 drillholes (one redrill) were completed on Anomaly B of the Martison Carbonatite Deposit. This was the first drilling program on this anomaly since the very first two holes (drillhole numbers 81-01 and 81-02) were drilled at Martison in 1981. All holes were continuously cored from the surface to total depth using triple tube HQ coring technology. The drilling centres on this anomaly remain at about 200 m. Initial interpretations of this program show geologic conditions and analytical results similar to Anomaly A.

In February 2002, a revised block model was prepared, and a resource re-estimate were completed and reported by J.H. Reedman & Associates for the Martison Project, Anomaly A. This block model and resource estimate included the first use of re-interpreted lithological units from all previous drilling campaigns and the establishment of the nomenclature used for the 2008 Preliminary Feasibility Study. In 2002, from March 18 through April 2, a total of six drillholes were completed on the northwest fringes of Anomaly A ('M' series). All holes were continuously cored from the surface to total depth using triple tube HQ coring technology.

The objectives of this program were to test the residuum in this sparsely drilled area and to examine the REEs and niobium-rich lateritic sediments in this location.

A re-computation of resources issued in November 2002 includes the results of this programme and highlights the significant tonnage of niobium-rich material in the northwest sector of Anomaly A in the pseudo lateritic material.

In June 2002, Falconbridge Limited formed an alliance with MCK Mining and Baltic Resources to promote the development of Martison. Falconbridge's interest was solely in the supply of sulfuric acid to the Project from its smelters in the Timmins and Sudbury areas.

MCK Mining reorganized its Board of Directors in January 2006, to facilitate the development of the company and of the Project and in July 2006, MCK Mining changed its name to PhosCan Chemical Corporation.

The following October (2006), PhosCan announced the initiation of a prefeasibility study for the Project which was completed in June 2008.

In March 2008, PhosCan acquired all of the issued shares of Baltic, such that PhosCan, directly and indirectly, owned 100% of the Martison Phosphate Project.

From January to April 2008, PhosCan conducted a major field campaign to collect a bulk sample, to gather preliminary geotechnical information, to complete initial hydrological tests, and to begin the preparation of topographic maps of the Martison site and access corridor. Over 42 tonnes of residuum material were collected from seven drill sites and shipped to Jacobs Engineering in Lakeland, Florida, for beneficiation process analysis and pilot scale beneficiation testing.

From mid-January until the end of February 2012, PhosCan conducted additional winter drilling and other site investigations. The sonic drilling program consisted of 15 drillholes spread across Anomaly A from south to north along the northwest – southeast trend. Almost 2,000 m were drilled with the aim to provide some infill drilling of the residuum and recover additional intersections of the lateritic horizon.

Additionally, five test wells were drilled to pump test and monitor the drawdown characteristics of the deposit aquifer. The sample data from some of these holes were also incorporated into the 2015 MRE (for PhosCan), and which was resubmitted in March 2016 in favour of Fox River Resources Corporation, the new owners of the Martison Project.

The results of the drilling programs completed by PhosCan for Anomaly A and Anomaly B between 1999 and 2012 are shown below in Table 6-3 and Table 6-4 respectively.

Table 6-3: PhosCan Anomaly A Drilling Program Summary (including 2012)

Year	Company	No. Holes	Total Metres	Type and Comment
1999	MCK Mining (Baltic)	14*	1,698	DD (Includes one redrill)
2002	MCK Mining (Baltic)	6	943.2	Metallurgical
2008	PhosCan	34	4,888.3	Sonic (Cluster) - Metallurgical
2008	PhosCan	12	178.3	Auger / DD (Geotechnical)
2008	PhosCan	8	691.3	DD (Geotechnical- Includes two redrills
2008	PhosCan	4	465	Auger (Hydrogeological)
2012	PhosCan	15	1945	Sonic – metallurgical
2012	PhosCan	9	1003.6	Auger (Hydrogeological)
Total		**102**	**11,813.**	

*Includes six holes funded by Cargill.

Table 6-4: PhosCan Anomaly B Drilling Program Summary

Year	Company	No. Holes	Total Metres	Type and Comment
2001	Baltic	12	1296	DD. Includes 1 redrill
Total		**12**	**1,296**	

6.3 Historical Mineral Resource Estimates

The information presented in this section is historical in nature and is presented for information purposes only. This information has been largely reproduced from the 2008 resource estimate submitted by J. Spalding. Other than the 2015 MRE for PhosCan, which was subsequently resubmitted in April 2016 for Fox River (Table 6-5), none of these studies have been validated or verified by DMT and it is understood that none of these historical resource estimates as a result of these studies are in compliance with NI 43-101 guidelines. Consequently, the results summarised below in Table 6-5 should not be interpreted as an endorsement of the study data or the estimates generated therefrom.

In addition, the terms used in these discussions are historical and do not necessarily comply with the currently accepted definitions employed by the CIM or NI 43-101 guiding instruments.

Historically, the contained rare earth elements as “Total Rare Earth Oxides”, or as individual elements, did not form part of a resource estimate due to a lack of interest or limitation of analytical information in the majority of the historical drillholes. However, subsequent to and including the drill program of 2008, whole rock analyses have been carried out to include rare earth element analysis.

Anomaly B has not been included in a resource estimate to date.

Table 6-5: Historical Resource Estimates for the Martison Project

Year	Consultant	P_2O_5 Cut Off Grade %	Nb_2O_5 Cut Off Grade %	'Mineable' Resource (Mt)	'Global' Resource (Mt)	Bulk Density (t/m3)	P_2O_5 Avg Grade %	Nb_2O_5 Avg Grade %
1974	IMC			62.5			19.6	
1984	Camchib	14	0.62		145		20	0.35
				56.0			24.2	0.44
1997	Reedman & Assoc.	10	-		127.25	2.0 (dry)	20.8	0.39
				53.4			20.9	0.35
1998	MRDI	10	-		155.6	2.3 (wet)	17.2	-
				54.8			22.9	-
1999	Bete Inc. (Cargill/MCK -Baltic)	10		68.2		1.85 (dry)	25.0	-
2002	Reedman & Assoc.	10	-		64.8	1.85 (dry)	23.4	0.36
		12 (Residuum)	-	45.4			25.4	0.35
		10 (Laterite)	-	3.7			12.4	0.83

Note: None of the estimates in Table 6-5 conform to CIM Standards of Disclosure, are not NI43-101 compliant and are presented for information only.

6.4 Previous NI 43-101 Compliant Resource Estimates

In 2007, PhosCan released a phosphate resource estimate for the Carbonatite Complex which was produced by James Spalding P.Geo. based on a 3D block model. No grade cut-offs based on P_2O_5 content were used in any of the estimating methods employed for the mineral resource estimate. Using only a Litho-Unit approach and consideration of sub-units 2A (Unconsolidated Residuum) and 2B (Consolidated or Recemented Residuum), which effectively isolated lower grade material to other sub-litho units (2C, for instance), this essentially imparts a P_2O_5 cut-off of approximately 10% for the resource and is summarized below in Table 6-6. This estimate was used as the basis of mine planning for the 2008 PFS where only the Measured and Indicated Resources (M&I) were considered in the mine planning evaluation.

In 2007/2008, no estimate was made of the niobium bearing lateritic resource.

In 2014, DMT (UK) Ltd. Added the data generated from the 2012 drilling program, and from a resampling exercise in 2011 to the database with which to generate an updated geological model and mineral resource estimate. Also included in the 2014 database were several holes drilled in 2012 as part of the geotechnical and hydrogeological studies.

The 2015 DMT mineral resource estimate closely agreed with the Spalding 2008 MRE with the exception that no Measured resources were classified. However, the 2015 estimate for the first time was able to generate a niobium resource, primarily from the pseudo-lateritic horizon lying directly above the residuum.

The DMT MRE, originally submitted in March 2015, was resubmitted in April 2016 in favour of Fox River Resources Corporation as the new Martison Project owners by that time.

Table 6-6: Previous NI 43-101 Compliant Resource Estimates

Year	Consultant	P_2O_5 Cut Off Grade (CoG) %	Measured & Indicated Resource Mt	Indicated Resource Mt	Inferred Resource Mt	Bulk Density t/m^3	P_2O_5 Avg Grade %	Nb_2O_5 Avg Grade %
2008	J. Spalding (LaFleur)	Nominal 10% CoG using only 'Low' Grade (2A Unconsolidated Residuum) & 'High' Grade (2B Consolidated Residuum)	62.3			1.91*	23.5	0.34
					55.7	1.91*	22.7	0.34
2015**	DMT(UK) Ltd	Residuum constrained at 6% (Whittle pit shell)		54.3		1.90*	23.4	
					83.5	1.90*	19.1	
		Laterite		9.3		1.90*	7.2	1.21
					6.3		4.8	0.94

*Estimated Global Bulk (Dry) Density (2A & 2B Lithotypes) from Spalding et al 2008.
**Resubmitted in April 2016 in favour of Fox River Resources Corporation.

7. Geological Setting & Mineralization

7.1 Regional Geology

The Martison phosphate deposit lies in a geological province referred to as Precambrian volcanic and metamorphic rock sequences which are over one billion years in age. It is an exceptional region where crustal stability has occurred over extended periods of geological time. The occurrence of carbonatite deposits is the result of late magmatic injections of carbon dioxide gases, calcium and magnesium carbonate solutions, including associated crystalline apatite, magnetite and mica minerals, through conduits into volcanic vents. The subsequent exposure of the carbonatite rock for long periods of time to erosion and chemical weathering has resulted in the thick accumulation of a paleo–soil residue called a "residuum" which has concentrations of relatively insoluble minerals such as phosphate bearing apatite, lying on top of the competent and largely unweathered surface of the carbonatite. The regional geology is depicted in Figure 7-1 below.

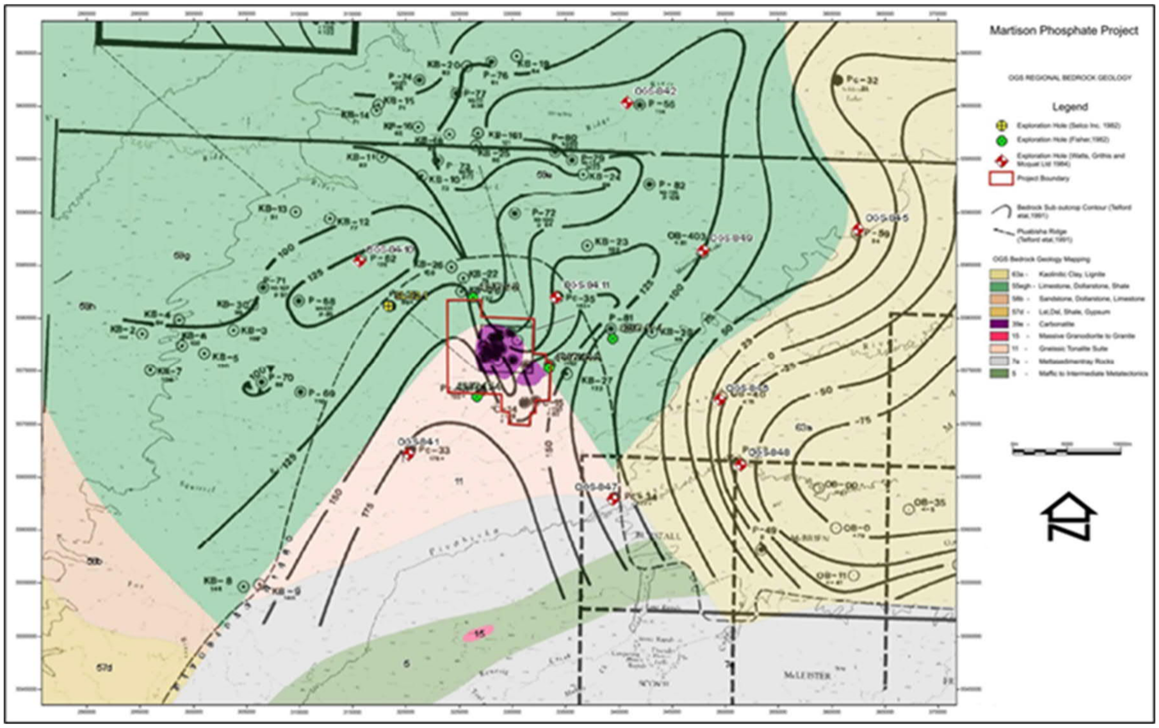


Figure 7-1: Regional Geology

The Martison carbonatite is one of several known locations of the Central Ontario Carbonatite Complex found on the Kapuskasing structural high (located 110 km east of Martison) to the Albany Forks structural high, (located 260 km west of Martison). Almost all of the carbonatite bodies occur along recognisable major tectonic features. According to their ages, the carbonatite bodies belong to four groups. The two younger groups, dated 120 Ma and 570 Ma, are restricted to the Ottawa graben, whereas the two older groups, dated 1,100 Ma and 1,700 Ma, are situated along the Kapuskasing High and the Albany Forks and Carb structures.

A number of complexes have been examined for their mineral potential. They all contain apatite in the carbonatite phase between 5% to 25%, and some contain significant enrichments of apatite through leaching out of carbonates. Such enrichment occurs on the Cargill Complex, located on a branch structure off the Kapuskasing High and at Martison.

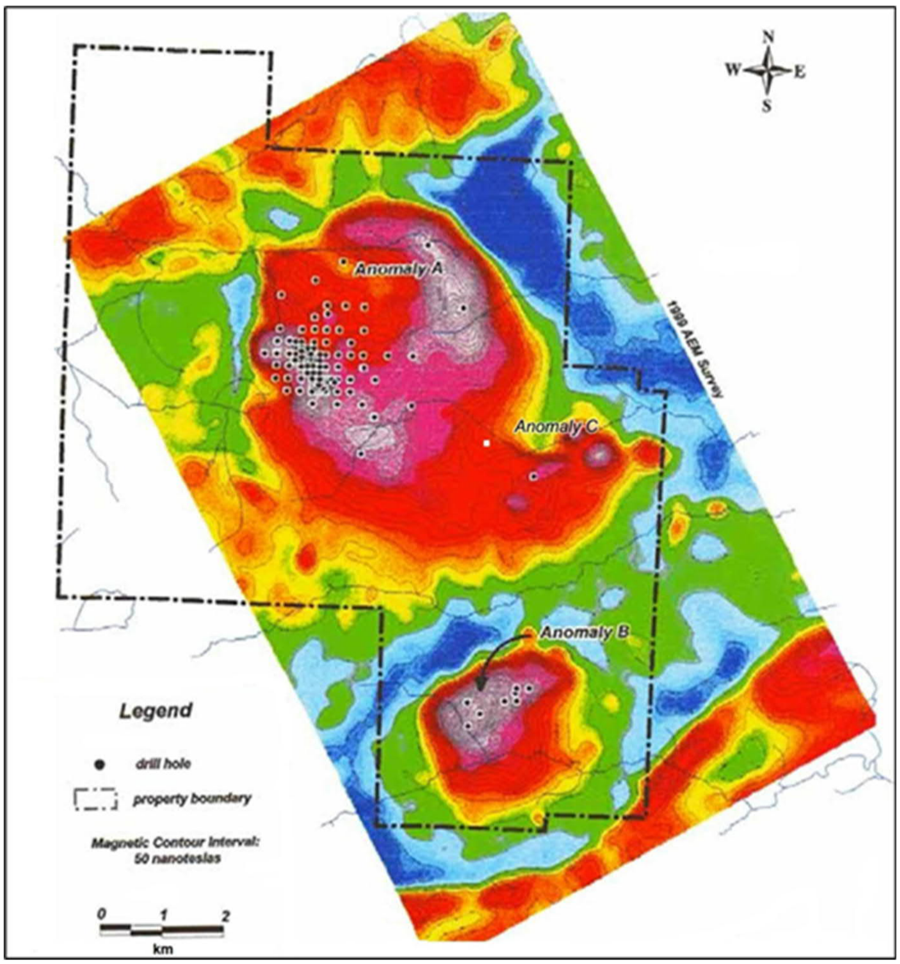
The Kapuskasing Complex, located on Cargill Township, contains a very high grade residual phosphate deposit associated with a well developed karst topography now buried under glacial lake clays. During karst development carbonates were dissolved from the carbonatite, and residual minerals, mainly apatite, were concentrated in sink holes and troughs. Sorting and reworking of the apatite-rich residuum by surface and subsurface water formed concentrations of nearly pure apatite sand, locally several tens of metres thick.

Concentrations of rare earth minerals are present in a discontinuous, iron rich, thin blanket of secondary weathering products on top of the residuum. The Martison Complex is considered to have formed in very similar geological conditions to the Kapuskasing occurrence.

Agrium, a major North American fertilizer producer, mined the high grade Kapuskasing phosphate occurrence (which lies approximately 120 km southeast of Martison) from 1999 until 2013 to produce a phosphate rich concentrate for their Redwater fertilizer operations.

7.2 Property Geology

There are no exposures of the carbonatite source rock or the surrounding country rock, and therefore all geological data result from drilling information and interpretations from geophysical surveys. The aeromagnetic geophysical surveys, conducted in 1965, identified several anomalies at the Martison site. Subsequently, exploration work has concentrated on the largest of three anomalies, referred to as Anomaly A, the two other smaller anomalies are referred to as Anomaly B and Anomaly C. These three anomalies are illustrated in Figure 7-2 below.



Source: AEM Survey (1999)

Figure 7-2: Magnetic Anomalies A, B & C

Differential weathering of the Carbonatite Complex has resulted in an irregular weathered, ‘karstic – type’, bedrock sub – outcrop of carbonatite, the depth of which varies significantly over short distances. Depressions in this carbonate rich igneous rock are filled with the weathered breakdown product of the carbonatite, which is effectively a paleo soil, referred to as a ‘Residuum’. The Residuum is enriched in apatite and represents the bulk of the phosphatic material of economic interest.

Above the Residuum, but not as widely distributed, lies a further sub-outcrop of iron oxide rich lateritic material which is similarly enriched in niobium and other Rare Earth minerals.

The geology of the Martison deposit has been defined by exploration drilling, drillhole bulk samples and airborne and ground geophysics. It can be summarised as follows and schematically in the section through Anomaly A in Figure 7-4.

- Muskeg deposits varying between 0.5 m and 4 m thick, averaging 2 m across the deposit.
- Glacial till composed of calcareous clay to coarse gravel (which is competent in a dry condition). This varies in thickness from 30 m to 90 m and averages approximately 50 m across the deposit.
- Local occurrences of several metres of thick, black, carbonaceous paleo soil.
- Pseudo-lateritic material variously composed of lignitic peat, non-calcareous clays of a variety of colours and silica sands, which have correlative properties to the Mattagami Formation (Spalding et al., 2008; Sage, 1991), are often identified at the base of the till. They are typically iron rich (Fe_2O_3 often > 40% and predominantly magnetite and haematite). The thickness of these sediments corresponds well to areas where the depth to the residuum is greatest, which may be valley fill within the trough created by the postulated fault (Spalding et al., 2008). (Note: This unit was referred to as Unit 3 in historical project literature).
- Weathered carbonatite residuum, which is a silty, sandy paleo soil, enriched by insoluble minerals (apatite) (lithotype 2A). Recementation by circulating phosphate rich fluids have typically formed higher grade zones of phosphate referred to as “Recemented or Consolidated Residuum” (lithotype 2B).
- Competent/fresh carbonatite. The contact between the fresh carbonatite and the residuum is gradational (lithotype 2C).

The numbers and types of drillholes are discussed in Section 10 “Drilling” and are shown in Table 10-1 and Table 10-2.

It has been postulated that the Carbonatite Complex has been cut by a northwest trending fault in the vicinity of Anomaly A. Evidence of this includes deformation and recrystallization of the carbonatite minerals, aeromagnetic interpretations, as wells as spatial analysis of the depth to the top of bedrock (Spalding et al 2008; Sage 1991). This trend was extended to the northwest to the Moose River Basin by Telford et al., (1991), who interpreted the trough (bedrock sub-outcrop contours).

Along the alignment of the conjectural northwest to southeast striking fault, a trough has formed in the top of the bedrock that deepens towards the northwest. This interpretation is supported both by drillhole data and ground geophysical evidence for Anomaly A (illustrated in Figure 7-3 and Figure 7-4).

The Martison Phosphate Project, as currently defined by past drilling campaigns, is composed of three magnetic anomalies of which Anomaly A is by far the most intensely explored and studied of these deposits.

Anomaly A strikes approximately N 30° W and is without a definable dip. The current strike length is approximately 1,700 m with a width varying between 300 m and 600 m. The northwest and southwest edges of this anomaly zone are sharp due to the effects of the possible postulated faults and the resulting intensive weathering of the carbonatite in this fractured zone. However, the residuum resource in Anomaly A remains open to the northwest, northeast and east and at depth in its central and northern areas.

Anomaly B is located approximately 5 km south of Anomaly A. An initial two holes were drilled in Anomaly B in 1981, no further work was carried out until 2001 when an additional 12 holes were drilled at approximately 200 m spacing. Although not fully explored, Anomaly B is considered to have been developed by the same geological processes as Anomaly A. Several of the drillholes have intersected phosphate mineralization of a similar level as Anomaly A. The aeromagnetic anomaly is as strong as the Anomaly A response but does not appear to have the aerial extent. Geologically Anomaly B is very similar to Anomaly A. Residuum thickness in Anomaly B varies in the holes drilled from 4.5 m to 90 m, with P_2O_5 grades up to 30%.

An approximate average thickness of 18 m of residuum is identified in the borehole logging, though the phosphate distribution appears to be more irregular.

Anomaly C occurs as a significantly smaller magnetic anomaly approximately 2 km to the east southeast of Anomaly A. Only one hole appears to have been drilled in Anomaly C in 1981 (Drillhole number 81-13), which apparently did not intersect any of the mineralised residuum. Since that time no further drilling has been carried out on this anomaly.

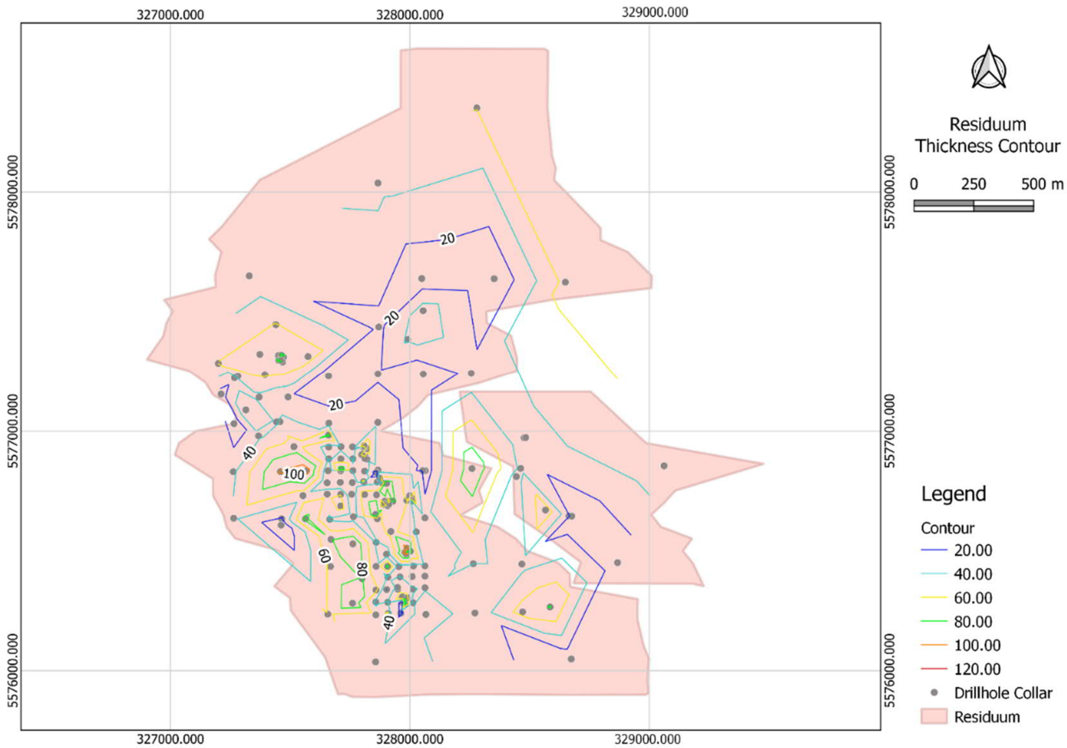


Figure 7-3: Anomaly A Residuum thickness

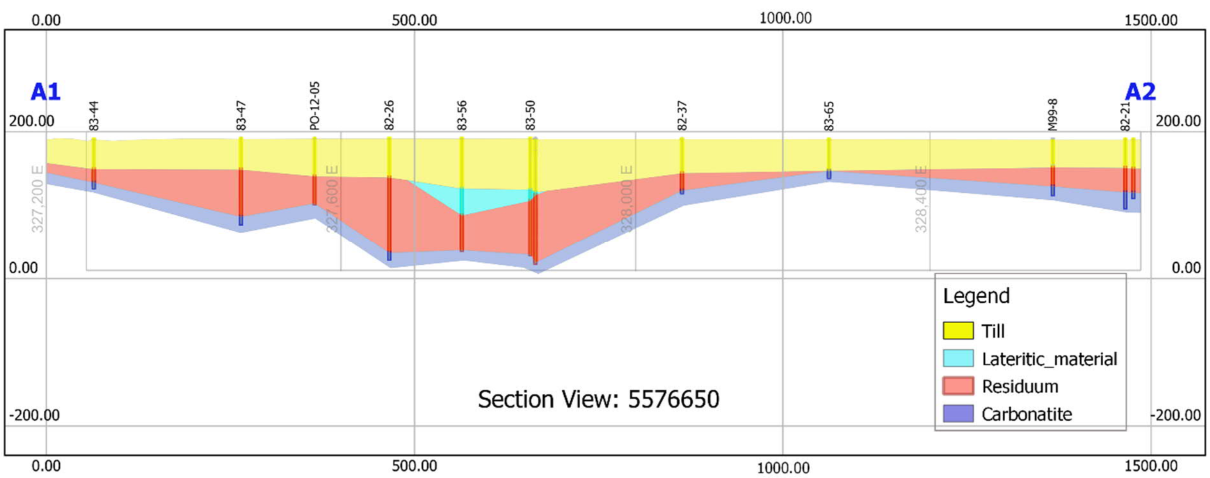


Figure 7-4: Property Geology – Interpretive Section Anomaly A

7.3 Mineralization

Apatite is the principal phosphate bearing mineral of economic interest within the residuum. All minerals identified in the deposit are presented in Table 7-1.

The pseudo lateritic material is enriched in niobium typically found in the form of pyrochlore. Its occurrence is of economic significance and has been the subject of significant metallurgical test work and study in recent years to establish if it may be extracted economically.

Table 7-1: Principal Residuum Minerals

Phosphate Minerals		**Iron Minerals**	
Apatite	$Ca_5(PO_4)_3(F,Cl)$	Magnetite	Fe_3O_4
Hydroxyl apatite	$Ca_5(PO_4)_3(F,Cl,OH)$	Goethite	FeO(OH)
Florencite	Sr, Ce, Ca, $(PO_4)_2$, $(SO)_4F(OH)_5$, H_2 O	Llmenite	$FeTiO_3$
Crandallite	Ca, Sr, Pb, Al, PO_4, OH, H_2O		
Pyrochlore	Na, Ca, Nb, Ta, O_6 F, OH		

8. Deposit Types

It is postulated that the Martison phosphate-rich residuum deposit was formed by karstic weathering of the underlying carbonatite basement rock.

Karstic weathering of the carbonatite at depth would require the water table to have been lower in the area at some time in the past and likely prior to glaciation.

One theory proposed to lower the water table would require the presence of a deep channel through the site in the location of the trough that exited through the surrounding gneissic rock to the north (Fisher, 1981). The presence of this deep trough is supported by the geological interpretations of the drillhole data and the geophysics model generated as a result of the 2009 resistivity ground survey. Water table fluctuations in response to presumed changes in river level in such a channel would then allow periods of recementation of the apatite rich residuum during high water table periods.

9. Exploration

As part of the advancement of the Project, technical investigations other than drilling (which is summarised in Section 10) have been undertaken and are presented below.

9.1 Ground and Air Survey (2008)

In March 1983, a survey of 24 drillholes was carried out by T E Rody Limited. The coordinate system used was a local system that was later verified to be the “Shell Oil Grid System”. Of the 24 drillholes surveyed, 22 had coordinates identical to the coordinates using this system, while the remaining two holes (numbered 83-45 & 83-61) had coordinates which were slightly different and were considered by the surveyor T.E Rody to be transcription errors for the same system.

A comprehensive ground survey in 2008 undertook a transformation of the Shell Oil Grid System to NAD 83 / WGS 84 UTM. The survey was undertaken by Sutcliffe Rody Quesnel Inc. (SRQ) surveyors and engineers of Cochrane, Ontario (now expGeomatics). A total of 117 pre-existing drillholes (from 1981 to 1999) were surveyed in the field. Of these, direct evidence (pickets, flagging, PVC pipe etc.) was found for 37 holes. Precise positions for an additional 24 drillholes were derived from the 1983 survey. The transformed Shell system coordinates were used to navigate to the remaining drillhole locations by handheld GPS. Most of these hole locations were consistent with visible drill pads or intersections of drill roads.

The 2008 survey addressed these following main key features:

- Coordinate system and elevation datum.
- On-site control monuments.
- GPS survey operations.
- Reconciliation with "Shell Oil Grid" system.
- Survey of existing drillholes (pre-2008).
- New drillholes.
- Other surveyed features.
- Airborne Light Detection and Ranging ("**LiDAR**") topographic mapping (carried out by sub-consultant Terrapoint Canada Inc.).

The LiDAR survey covered PhosCan's mining leases and claims, for a total area of approximately 35 km^2. In addition, a proposed access road corridor was flown and mapped.

The corridor width was approximately one kilometre and length of approximately 40 km.

The LiDAR generated map included 0.5 m topographic contours, streams, lakes/ponds and roads/trails. A mean discrepancy of 20 cm was established in the licence area after 'ground truthing'. 'Ground truthing' of the access corridor was not possible due to the paucity of control points (*Report of Survey Operations,* Sutcliffe Rody Quesnel Inc. Feb.-Mar. 2008).

9.2 Geotechnical Site Investigations (2008)

The following is summarised from the report *"Preliminary Geotechnical Investigation Proposed Martison Phosphate Mine, Hearst, Ontario",* September 2008 by AMEC Earth & Environmental.

AMEC Earth & Environmental, a division of AMEC Americas Limited ("AMEC"), was retained by PhosCan to provide engineering services for a preliminary geotechnical investigation for a proposed mine site development. The fieldwork for the investigation consisted of 12 shallow drillholes outside the proposed open pit footprint, six deeper drillholes within the open pit footprint, and 37 test pits. The drillholes were advanced to depths of up to 115.5 m below existing grade and are summarized in Table 9-1 below. The drillholes were complete by truck and track mounted drills between January 31 and March 11, 2008. The test pits were constructed with an excavator between February 10-15, 2008.

Table 9-1: 2008 Geotechnical Site Investigation Drill Summary

Boreholes	Total Holes	Total Metres Depth (m)	Average Depth (m)	Range (m)
GT08-01- GT08-12	12	178.3	14.8	12.8 -16.6
PT08-01-PT08-6	8 (including 2 redrills)	691.3	86.0	40.1 – 115.5
Total	**20**	**869.6**		

The deep drillholes (PT08-01 to PT08-06) were located around the perimeter of, and within what was at the time, the proposed outline of the open pit. The shallow drillholes (GT08-01 to GT08-12) were located on the northwest, west and southwest areas outside the open pit area and were designed to probe the glacial till only.

In addition to these drillholes, 37 test pits (TP08-01 to TP08-27), from 4.3 m to 4.9 m deep, were excavated. The drillhole and test pit locations were determined by the SRQ 2008 ground survey and are shown in Figure 9-1.

The location of these drillholes and test pits were designed to explore the soil conditions and engineering properties over a wide area of the Project site and, in particular, those areas which had been provisionally designated as potential locations for mine infrastructure, such as tailings dams, waste rock facility and the beneficiation plant. The area to the east of the proposed open pit was not studied within this program as it has been indicated (in earlier drilling programs and subsequently from the interpretation of the 2008-09 ground geophysics profiles) that the potential exists in this area for a further extension of the resource towards the northeast. Provisionally therefore, no mine infrastructure would be located in this area as a result.

The preliminary geotechnical site investigation established basic soil engineering parameters of the main soil types, the glacial till and residuum. This information would be used to inform construction designs, foundations for infrastructure buildings and the primary soils exposed during the open pit mining operations.

The use of imported engineered fill materials of specified geotechnical characteristics will likely be required for specific applications (such as drainage, road, and slab-on-grade foundations), where the on-site materials may not be of appropriate quality. Additional geotechnical investigation, including field insitu and laboratory testing, will be required to further identify the soil and rock engineering properties required to carry out detailed engineering and geotechnical designs.

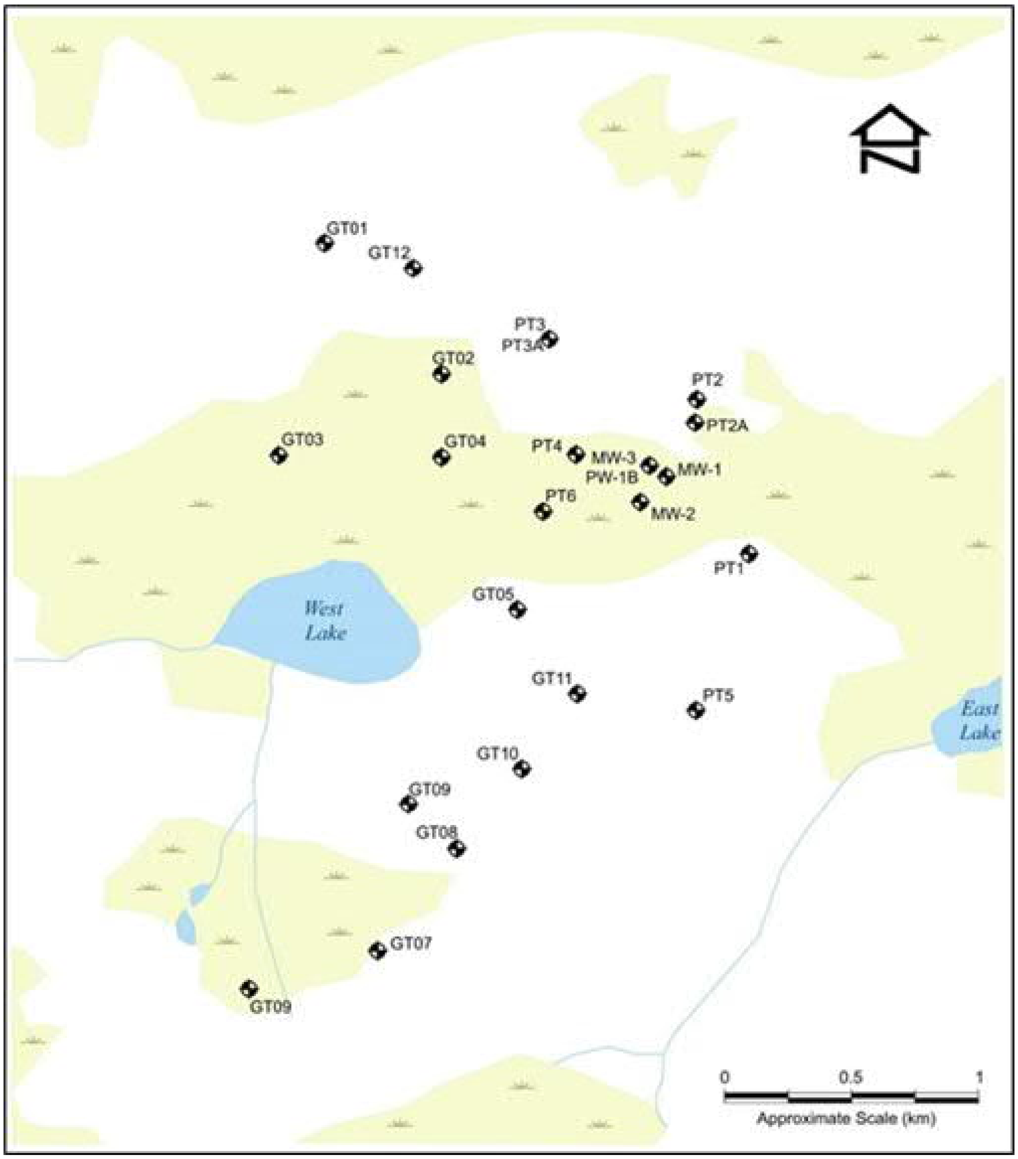


Figure 9-1: Hydrogeological and Geotechnical Site Investigation Borehole Locations 2008

9.3 Hydrogeological Investigations (2008)

In the winter of 2008, AMEC was also requested to complete a preliminary hydrogeological study at the proposed Martison mine site, focussing on the proposed pit area. This evaluation included supervision of a pumping well installation, completion of hydraulic conductivity testing at selected intervals throughout the stratigraphic sequence, and a pump test program to collect data to complete a preliminary evaluation of dewatering activities, both during construction and long term production. A drill summary of this program is shown in Table 9-2 below.

The program consisted of these main elements:

- Drilling, installation and development of eight monitoring wells and one pumping well, which included the use of geotechnical (PT series) holes drilled earlier during the same winter program. The geotechnical holes were used as monitoring wells and holes PT08-1/02A/03A/04/05 and /06 were all installed with piezometers, and along with holes MW 01 and MW 02. (MW 03 was not installed).
- The elevations of the new wells were surveyed to facilitate the interpretation of ground water flow.
- A 72 hour aquifer response (the pump test) and a variable rate (step drawdown) test was carried out to determine efficiencies of the pumping well at 5, 10, 15 and 20 L/s pumping rates.
- Pump test and slug test analysis to estimate transmissivity, storativity and hydraulic conductivity of the high permeability aquifer zone encountered at a depth of approximately 100 m.
- Representative ground water samples taken from the pumping well at 24 hour intervals during the 72 hour test to observe changes in water quality with respect to time during the pump test. This was also to establish the suitability of the ground water quality for use in the planned beneficiation process. Samples were also collected from the 50 mm diameter monitoring wells.

Transmissivity values were estimated to be in the range of 300 m^2/day – 1,000 m^2/day. In addition, the monitoring wells 100 m and 125 m away recorded water level drawdowns of one metre and 0.5 m, respectively.

Table 9-2: 2008 Hydrogeological Site Investigation Drill Summary

Borehole ID	Total Holes	Total Depth m	Average Depth m	Range m
MW 01, 02, 03*	3	352	117	90 – 140
PW08- 01b	1	113		
Total	**4**	**465**		

* Drill steel in hole, hole sealed with bentonite.

9.4 Hydrogeological Investigations (2012)

A multipurpose, nine hole drilling program was carried out between January and March 2012 which was designed to test the profile of the residuum geology to bedrock, conduct borehole geophysical surveys to further refine the subsurface geological units and to install well screen and casing for future hydrogeological studies. A drill summary of this program is shown in Table 9-3 below.

Table 9-3: 2012 Hydrogeological Site Investigation Drill Summary

Borehole ID	Total Holes	Total Depth m	Average Depth m	Range m
PW -12 – 01/	3	269.8	87	78.0-96.5
TW-01 /06	6	733.8	122.3	71.9 – 160.3
	9	**1003.6**		

This program was also conducted by AMEC. The array of test wells was located generally along the margin of the main phosphate deposit (Anomaly A). The results are reported in an AMEC report entitled “Martison Phosphate Project – 2012 Critical Issues Study”, December 2012.

AMEC observed that the main aquifer appears to be restricted to the weathered bedrock below the areas of thicker residuum which comprise the deposit. This deposit forms a northwest-southeast trough that deepens to the northwest. Pumping test information from wells installed as part of the 2012 program, outside the zone of thicker residuum, indicate that the carbonatite rock that surrounds the deposit is of low to moderate permeability with the hydraulic conductivity of the upper rock appearing to increase with proximity to the trough.

The hydrogeologic properties of the bedrock aquifer were investigated through a seven day pumping test. Following the completion of the pumping activity, the aquifer was noted to recover slowly, indicating that there is no source of recharge or significant supply of water to the aquifer.

The properties of the bedrock surrounding the carbonatite deposit were investigated through a background review of historical exploration hole data filed by other companies and Ontario Geologic Survey reference hole drilling. The dataset was sparse; however, it indicated that the carbonatite is surrounded by granitic rock which generally has a much lower hydraulic conductivity than measured in the weathered carbonatite as part of this program.

Following the analysis of the pumping tests and other field data, a groundwater model was developed and calibrated to field observations of water levels and the response in the aquifer to the pumping test. This model was then modified to enable predictive simulations of dewatering scenarios.

The results of the predictive modelling indicate that the mine can be dewatered by pumping from five pumping centres located around and within the deposit at combined pumping rates of between 23,000 m^3/day and 32,000 m^3/day.

The dewatering of the bedrock will also not effectively drain the overburden and, during the initial phases of excavation through the glacial till, sumps will need to be employed to remove water during overburden stripping.

It was assumed that all the water from the dewatering wells will be directed to the beneficiation plant after this facility becomes operational. During the first year, however, dewatering will begin prior to plant construction when the overburden is being stripped. During this period, discharge from the dewatering wells and trenches is assumed to be discharge directly into the local environment as discussed in Section 18.1.6 “Site Preparation”.

Analysis of groundwater samples collected during the pumping tests indicates that the groundwater is generally of good quality and, with the exception of phosphate and ammonia, meets the Provincial Water Quality Objectives which are the standard criteria for discharging to surface water into the environment. The dataset of existing surface water quality data is poor but generally indicates that the surface water features are also high in phosphate and ammonia. These appear to be naturally occurring in the area and it may be possible to discharge the drainage water to local creeks without treatment.

9.5 Ground Geophysics (2008/2009)

In late 2008 and early 2009, Geophysics GPR International Inc., Mississauga, Ontario, were retained by PhosCan to undertake a ground geophysical survey of Anomaly A.

Six resistivity profiles were collected for a total length of approximately 12.45 km and over 39,000 data points. Induced Polarization data was collected along segments of Profile #2 and Profile #4. The results of this survey are shown in Figure 9-2 below. Ground surface elevation is relatively level over the survey area and the position of the profile lines was predetermined by GPS coordinates. The coordinates along the profiles were recorded with a handheld “wide area augmentation system” enabled GPS unit and should be accurate to within -/+ 10 m.

Typically, the contact between the overburden and bedrock is well defined as bedrock tends to be more resistive.

At this site, the resistivity model has been interpreted in terms of three geological contacts as follows:

1. Base of Overburden. The overburden material was expected to comprise of 2 m thick peat and muskeg with varying degrees of water saturation, underlain by 35 m to 50 m of silt to sand glacial till.
2. Clay. Isolated pockets of assumed Cretaceous age clay are known to exist in some areas. The areas of low resistivity have been identified as potential clay deposits.
3. Top of Carbonatite (Bedrock). The contact between overburden and bedrock material is typically characterized by a relatively well defined increase in resistivity.

These profiles were interpreted in conjunction with each other and the boreholes and, in general, there was excellent agreement between both types of data.

The resistivity profiles appear to show less irregularity than indicated in the drilling results. Some of the irregularity may also be attributed to mistaken identification of re-cemented residuum as carbonatite basement, which may have resulted in the mistaken reporting of the end of hole depth of a borehole as bedrock rather than the residuum.

The profiles support the interpretation of the deep valley feature which appears to run northwest-southeast along the Anomaly A (see intersection between Profile Line # 1 & 4 in Figure 9-2).

Profile #4 was extended (from the intersection of Line #6) to the east and shows that potential exists for additional residuum resources to be explored in this direction. Very little exploration drilling has been undertaken east of Line #6.

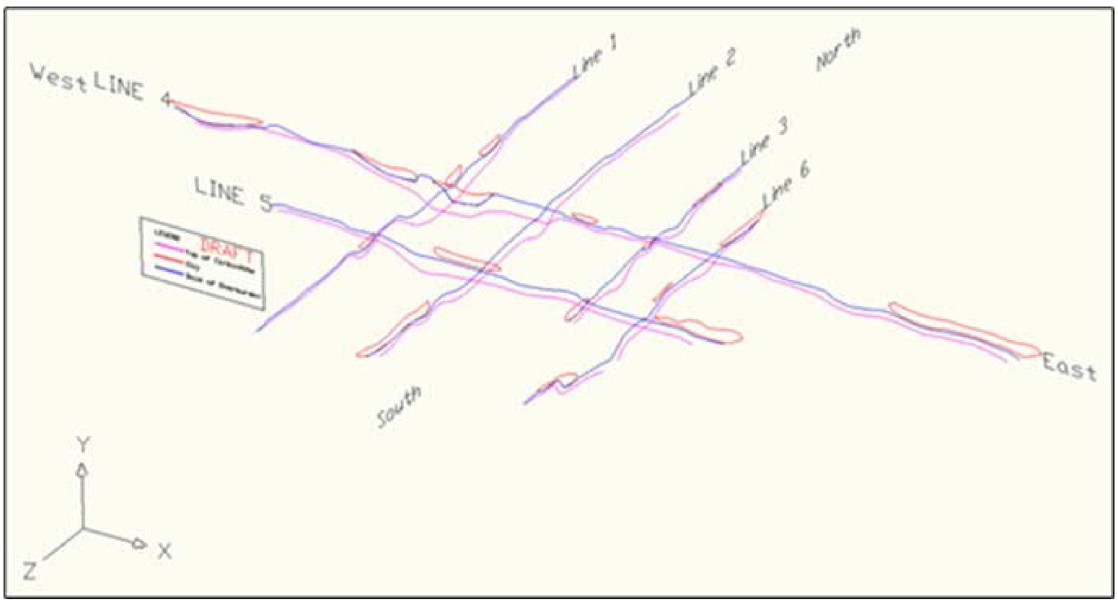


Figure 9-2: 3 D Interpreted Resistivity Profiles

10. Drilling

Fox River, thus far, have not conducted any drilling programs since acquiring the property. As a result, the 2012 site investigations remain the most current of drilling programs at the Project site.

PhosCan (and its predecessor companies) undertook drilling programs in the winters of 1999, 2001, 2002, 2008 and 2012 at the Martison Phosphate deposit.

It is understood that in all the drill programs the holes were drilled vertically and have utilized several different drilling methods over the years. This has included large diameter diamond drill coring (HQ size), reverse circulation, return air blast method, auger and larger diameter (6”) Sonic drill rigs.

In view of the nature of the formation (paleo soils) and relatively shallow depth of the drillholes, it is reasonable to assume that the holes did not significantly deviate and the drill intersection lengths represent the true thicknesses of the stratigraphy.

Early diamond drilling, from the 1980s, had recoveries in the 50% range using an NQ core tube size. In 1999 and 2002, recoveries increased to an average of 70% with an increased core tube size to HQ and typically using triple tube core barrels. After a study was conducted in 1984 comparing these results to core and drill cuttings, the similar grades suggest that the residuum material is relatively homogeneous.

Table 10-1 below provides a summary of the PhosCan resource drilling programs for Anomaly A. This Table 10-1 does not include the 2008 hydrogeological and geotechnical drill programs and the 2012 hydrogeological program (summarised in Table 9-1 and Table 9-2 respectively Table 9-3). Although most of these site investigation holes were drilled purposely on the edge, or outside, of the anticipated mining envelope, some of these holes intersected significant residuum and were subsequently sampled to establish the level of mineralization in each. As a result, several of these holes were included in the resource estimate database. The Anomaly B drilling program is presented in Table 10-2.

Only one hole (81-13, in 1981) has been drilled to date in Anomaly C.

Table 10-1: Anomaly A Resource Drilling Program Summary

Year	Company	No. Holes	Total Length m	Type and Comment
1999	PhosCan (MCK Mining)	14*	1,698	DD (Includes one re-drill)
2002	PhosCan (MCK Mining)	6	943.2	DD
2008	PhosCan	34	4,888.3	Sonic (Cluster)- Metallurgical
2012	PhosCan	15	1,947.1	Sonic
Total		**63**	**9476.6**	

* Includes six drillholes funded by Cargill.

Table 10-2: Anomaly B Exploration Drilling Program Summary

Year	Company	No. Holes	Total Length m	Type and Comment
2001	Baltic	12	1296	DD. (Includes one re-drill)

The historical holes drilled in Anomaly A were largely drilled in the 1980s on a 200 m grid, with two smaller infill grids on a 50 m spacing. Those drilled in the 1982 program were to a predetermined depth of 78 m (37 holes for 2,919 m total).

Current drillholes include those drilled in 1999 in a series of 100 m spaced drillholes within the main deposit area. In 2002, six holes tested the northern part of the deposit.

In 2008, seven clusters of sonic drilling (106 mm core diameter) collected over 42 t of material for metallurgical test work and pilot scale testing of beneficiation; each cluster was comprised of four to six holes over a +/- 30 m radius area.

In early 2012, a further program of 15 holes totalling just under 2,000 m was completed. And provided additional material for metallurgical test work and better delineation of the main part of the deposit.

The historical and recent drillhole locations are shown, by year date drilled, in Figure 10-1.

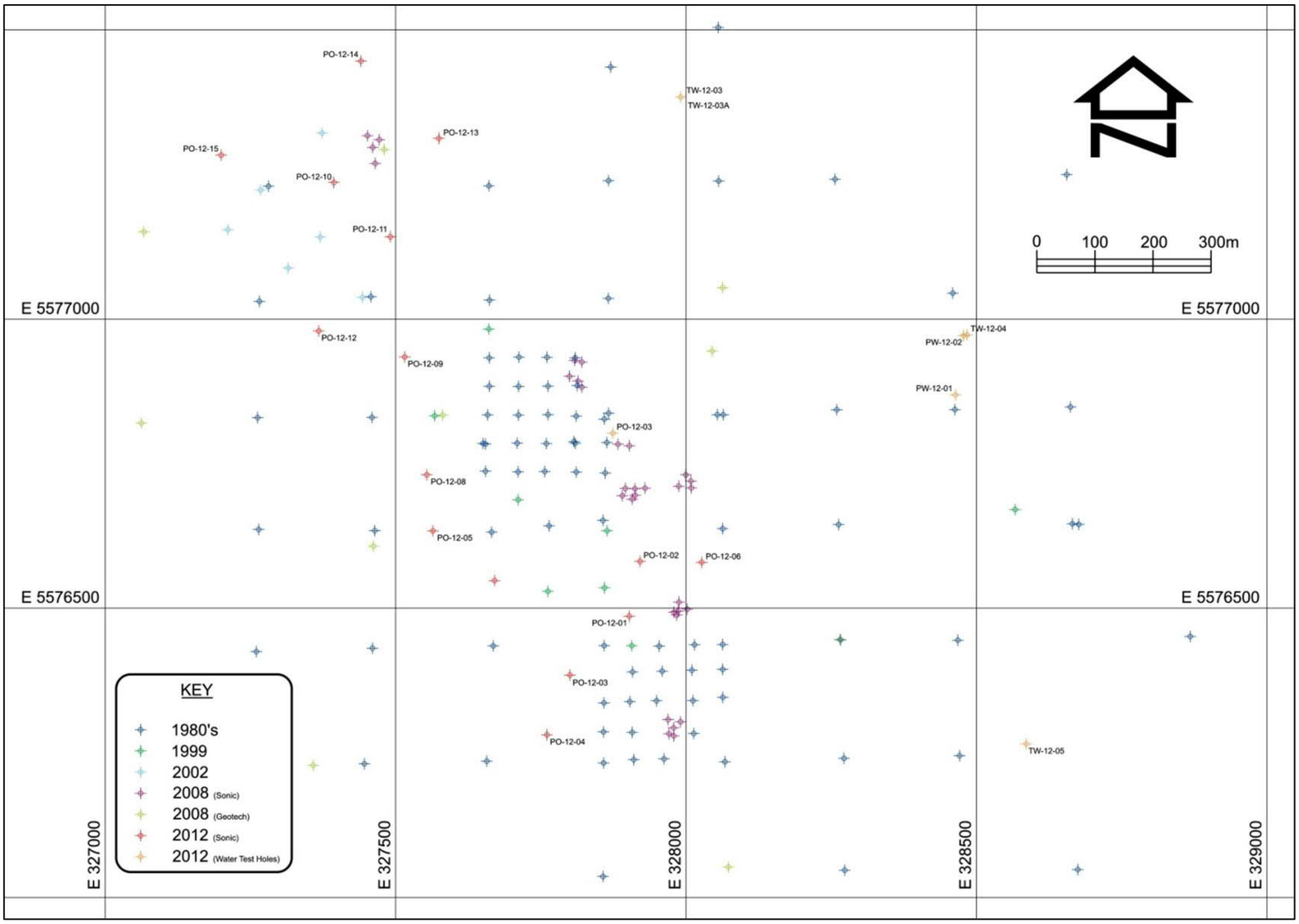


Figure 10-1: Martison Anomaly A – Historical and Recent Drilling Programs

10.1 Reverse Circulation (RC)

Sample recovery, utilising the RC method, generally exceeded an estimated 90% with the sample loss mostly confined to the high clay fractions which remain suspended in the circulating medium. No systematic records of sample recovery were kept as part of the program records.

At appropriate intervals, typically 1.5 m, the material collected on the screens and from the decantation process was placed into a plastic bag and identified with a unique sample identification number. In the decantation process, two pails were used with the second pail allowing for the settling of fine particles from the overflow of the first pail. Samples were stored for shipment in dual wrapped plastic bags which were identified with unique codes.

10.2 Sonic Drilling

The sonic drill technique utilises a winterised 300C ATV drilling rig using 5' and 10' standard core barrels, drilling a 6" diameter hole to recover a 4" diameter core. The sonic drilling technique uses a mechanical oscillation to the drill head generating high frequency vibration to the cutting edge. These machines drill two to three times faster than conventional machines and provide continuous coring typically to 150 m, although some holes have achieved drill depths of over 200 m on this Project. In addition, the sonic method does not require the use of any traditional drilling additive (such as mud) and, in comparison, uses only small quantities of water.

The sonic drill tube encapsulates the core in a cylindrical clear plastic bag liner which is inserted in the core barrel before drilling commences. The core is extracted from the core barrel effectively wrapped in the plastic sheath, the upper and lower ends of the plastic sheath are tied off to prevent loss of sample.

In response to what is largely a paleo soil profile and, the type of deposit and mineralisation encountered, the sonic drilling has proved to be a very effective method of sample recovery for the local conditions. Core recoveries for sonic drilling in the residuum are typically in excess of 90%, although drill penetration rates and core recovery decline rapidly where the in-hole conditions become more variable with increasing levels of weakly weathered basement material. This method however cannot penetrate any distance into relatively unweathered basement carbonatite material and therefore may be problematic for a geologist to decide if true basement has been reached or if intact large blocks or boulders within the soil to rock transition zone have been encountered.

10.3 Diamond Drilling

Typically, coring was accomplished using standard diamond drilling equipment with a tricone bit and NW casing penetrating the glacial till and sediments immediately below before switching to wire line coring near the residuum contact. Early diamond drilling programs used twin tube core barrels for sample recovery. Larger diameter casing and HQ size, triple tube, barrels were implemented in later programs (1999 and thereafter) which improved sample recoveries.

Samples were prepared for shipment in properly identified wooden core boxes. While all core runs were 1.5 m length, samples submitted for analyses and testing were composites of these individual core runs.

10.4 Auger (Hydrogeological) Drilling

A series of 10 auger holes were drilled in 2012 primarily for the purpose of hydrogeological investigations providing follow up studies from the initial pump testing in 2008. The auger holes were logged and sampled and provided additional geological information on the deposit. Test well hole TW-12-02 in the northern extension of the mineralised footprint returned over 30 m of residuum averaging 24% P_2O_5 and over 40 m lateritic material averaging > 0.5% Nb. Pump well hole PW-12-02 and test well hole TW-12-04 both intersected lower grade material to the east on the fringe of the main mineralized zone.

11. Sample Preparation, Analyses & Security

11.1 Introduction

In the 2008 NI 43-101 Technical Report, J. Spalding reviewed the history of the sample preparation, analysis and security relating to the Martison Project prior to PhosCan's involvement.

In Spalding's opinion, the Anomaly A sampling and the pre-laboratory sample preparation was carried out to industry standard practice, using procedures adopted widely within the phosphate minerals industry.

Routine methodology, including 'round robin' checks between the laboratories, indicate that based on the data reviewed by Spalding, no statistically significant analytical biases are evident. Spalding pointed out, however, that prior to 1999, when PhosCan effectively took over management of the drill programs, no routine analysis for deleterious elements had taken place. These elements included Fe_2O_3, Al_2O_3, CaO, MgO, and acid insoluables ("AI"), referred to as the Minor Element Ratio ("MER"). The MER is a measure of the quality of phosphoric acid that can be produced and its suitability for making fertilizer products which is represented as follows:

$$MER = (\%Fe_2O_3 + \%\ Al_2O_3 + \%\ MgO) / \%\ P_2O_5.$$

Since 1999, this deficit in the analytical record has been corrected and subsequent sample programs have analysed for the MER components.

11.2 Sample Preparation

A number of sample collection and preparation methods were adopted by PhosCan depending on the type of sampling method used. The sample collection depended on what type of drilling method was used (core, reverse circulation, auger or sonic type). Drill core, crushed rock and pulps have all been used as sample mediums and the sample preparation adopted accordingly. Samples from the sonic core drill programs were usually taken by large spoon or hand trowel depending on the sample material or required size of the bulk. In 2008, the entire core of selected intersections from the 'cluster' drilling program (after a smaller analytical sample had been removed), were bagged and crated for the intended bulk (42 t) sample.

In 2011, several of the 2008 and pre-2008 holes were resampled primarily to generate a larger bulk sample for analysis and metallurgical test work for the recovery of niobium, mainly from the iron rich, pseudo-lateritic layers lying above the main phosphate bearing residuum.

This resampling program also provided another useful cross check on the analyses already carried out on these holes. Samples of up to half core bulk were removed over a selected sample length, (1.0-1.5 m), depending on the geology. The samples were then 'coned and quartered' to further composite the selected interval and to generate a duplicate backup of each sample should it be required. Finally, the samples, weighing up to 10 kg were double bagged in large heavy gauge clear plastic bags, with individual 'bar code' identification tickets inserted between the dual bag layers.

In 2012, the sampling procedure followed a similar methodology to that adopted in 2008 and 2011. The samples were selected based on visual lithology and/or drill interval and removed with hand trowel sampling tools. Since bulk sampling was not a prerequisite for this program, one quarter to one half of the core was removed. In the intersections or parts of sample runs where intact boulders were drilled, the rock core was broken using a hammer to provide a representative sample. In most cases these intact pieces of rock were rarely more than 10 cm to 20 cm in length unless the hole was stopped in relatively unweathered basement material.

The samples were bagged and tagged with a unique identification number using pre-printed, barcoded tickets supplied from the contracted analytical laboratories. Ticket stubs were retained in the sample books with basic information of the bagged sample, such as date sampled, core box number, sample interval (from – to) and a very brief description of the sample material lithology.

The individual bagged samples were then weighed, and a summary preprinted laboratory 'chain of custody' dispatch sheet was completed to accompany the samples, which were crated in batches and sent to the assigned laboratory.

11.3 Sample Analysis

Summaries of all analytical work are part of the current project data. Very detailed sample descriptions and analytical summaries are contained in the current Project records but not all certificates of analysis are available. Refer to Table 11-1 and Table 11-2 below for the summary of the Project sample analysis prior to PFS, and post PFS, respectively.

Table 11-1: Historical Sample Analysis Programmes

Year	Total Samples Analysed	Laboratory	Analytical Methods	Elements Analysed	Analytical Certificates in Project Record
1981	238	X-Ray Assay Labs, Toronto	XRF	P_2O_5,La,Ce, Nb & Minor Elements	Yes
1982	708	Bondar-Clegg & Co., Ottawa, Ontario	XRF Gravimetric Colourmetric/XRF Peroxide Fusion/Dichromate	Nb, La P_2O_5 Nb_2O_5 Fe_2O_3	No
1983	900	Lakefield Research, Lakefield, Ontario	XRF	P_2O_5, Nb_2O_5	Yes
	14 (Check samples)	X-Ray Assay Labs, Toronto	-	P_2O_5, Nb_2O_5	Yes
	38 (Composites)	Lakefield, Ontario	XRF	La	Yes
	8*(Composite check samples)	Atomic Energy, Canada	Neutron Activation	REEs	Yes
	21*(Composite check samples	Neutron Activation Services, Hamilton, Ontario & Hazen Research, Golden, Colorado	Neutron Activation	REEs	Yes
1984	1036	Lakefield, Ontario	XRF	P_2O_5, Nb_2O_5	No
1999	356	Thornton, Tampa, Florida	AR**/AA*** AR/SP**** AR/Ceric Sulphate HCl/AA	MgO, Al_2O_3 P_2O_5 CaO Fe_2O_3	Yes
2002	149	Swastika Labs, Swastika, Ontario	Li Metaborate/HNO3 ICP	Whole Rock Whole Rock	Yes

* Check samples of the 38 original composites sent to Lakefield (1983).
** Aqua Regia.
*** Atomic Absorption.
**** Spectrophotometric.

Table 11-2: Post 2008 (PFS) Sample Analysis Programs

Year	Total Samples Analysed	Laboratory	Analytical Methods	Elements Analysed	Analytical Certificates in Project Record
2008	2204	ACME, Vancouver	4 Acid ICP-MS Analysis Phosphoric Acid Leach, ICP-ES analysis	Whole Rock	Yes
2011	934*	ALS Timmins/ Vancouver	ICP-AES	Whole Rock (& REEs)	Yes
	738	ALS Timmins/ Vancouver	XRF – 10	Nb	Yes
2012	908*	ALS Timmins /Vancouver	ICP-AES	Whole Rock	Yes
2019	9 (nine)	ALS Timmins /Vancouver	ICP-MS Cl-KOH Fusion and IC	Whole Rock & Cd, Sc & Cl	Yes

* Includes QA/QC blanks and standards inserted by PhosCan.

In 2011 a sampling exercise of pre-existing core was carried out to complete the sample analysis which was not completed as part of the original bulk sample drilling program in 2008. This included taking samples of the ‘overburden’ (overlying residuum), ‘interburden’ (waste within the residuum) and ‘basement’ (below the residuum).

To reduce sample costs, the initial semi-quantitative assay run of 2008 identified samples was used to target those intersections with anomalous niobium for a more accurate quantitative analysis.

This sampling program was designed to identify all material by chemical parameters rather than rely on subjective interpretation through visual logging in material which proved difficult to separate in core samples. Much of this material was lateritic in appearance and, excluded the glacial till, which was easily identified in the field.

The 2011 sample set of 934 samples included drill core, crushed rock and pulps and was initially analysed for whole rock and Rare Earth Elements at ALS laboratories in Timmins and Vancouver. Later in 2011 part of the sample set (738 samples) was re-analyzed only for niobium using XRF-10 analytical methods. ALS had established that the niobium content reported higher using ME XRF-10 compared to the mass spectrometry methods, largely due to the highly resistive nature of the niobium element.

In 2019, as a separate exercise to establish other deleterious element levels, nine samples were recovered from the core of several 2012 drill holes. These holes were specifically selected for their representation of the deposit from north to south. In addition, whole rock analyses were carried out for cadmium, scandium and chlorine. Cadmium was recorded at an average of less than 6.1 ppm, and chlorine an average (below detection limits) at less than 50 ppm.

11.4 Sample Security

In the opinion of DMT, the handling, and the subsequent batching and crating of the samples to the assigned laboratories, was carried out to an acceptable industry standard.

Core has been carefully handled and secured in wooden boxes for the drill programs since and including 2008. Boxes were purpose built for both diamond core and the sonic drill programs using lidded boxes which could be secured for transportation. The sample core boxes, rejects and pulps were observed to be in secure, clean and well lit conditions at the Project storage facility in Hearst. These are now removed and restored in a similar facility in Timmins, Ontario, which has been visited in November 2019 by QP, Tim Horner.

The individual core boxes are clearly numbered and batched according to drill hole identification. The core is safely stacked and strapped on palettes for ease of re-access to the retained sample.

11.5 QA/QC

In 2008 the goal of the sample program was to generate a large tonnage bulk samples for bench and pilot scale testing at Jacobs Engineering (now JESA), Florida. Typical spoon or trowel channel samples were taken from the selected bulk sample intersections before dispatch to Jacobs. A total of 2,204 samples were analysed for whole rock at the ACME Laboratories in Vancouver. PhosCan did not insert any additional QA/QC samples in the form of blanks and standards into these sample runs. PhosCan, did, however, carry out a series of check samples which from five of the seven holes drilled labelled with the suffix ‘A’, (83-61A for example). In total, 85 check samples were analyzed. The samples were tested for P_2O_5 and the MER compounds (Fe_2O_3, Al_2O_3, MgO, CaO), and were found to be well within acceptable analytical variance and error.

In March 2011, the resampling program, referred to in Section 11.3, incorporated either a blank (clear ground glass), a standard, or a sample duplicate which was inserted in the sample stream at a regular (usually every tenth sample) interval.

Three types of standards were created for typical representative material for 2A, 2B (residuum) and CA (laterite) using 30 pulp samples of the three material types. CDN Resource Laboratories Ltd of Langley B.C, coordinated the process, which included a ‘round robin’ of analyses by six separate laboratories. Smee and Associates (Geochemists), Vancouver, issued the Certificates of Analysis in November 2011. These standards were used in the 2012 for insertion into the sample streams resulting from the 2012 winter drill sampling program.

Blanks were also obtained from an external laboratory source consisting of clean fine sand. In addition, the laboratories used also ran their own standards and duplicates within the sample batch. Copies of the analytical certificates are held in the project records.

12. Data Verification

Independent Qualified Person (QP) Verification

In early 2012, during the drill program that year and the subsequent sampling exercise, all of the Timmins based Project core (from historical and 2008 drill programs), drillhole logs, analytical certificates and sample returns were transferred to a Hearst storage facility for reference purposes.

After 2014, all the boxed core (which included the drillholes from the 2012 program), sample duplicates and related sample materials were subsequently relocated to Timmins, where the Project core is currently stored.

A number of drill hole collar positions were visited during the October 2014 site visit and the coordinates checked with a handheld GPS. These were found to be reasonably accurate (within a 10 m radius of error) given the check method used.

Using the 2008 ground survey data it has been possible to plot the drillhole positions against satellite imagery of the site. The drilling grid is very evident on satellite imagery and the drillhole positions as surveyed in 2008 line up closely with the known locations Figure 12-1.

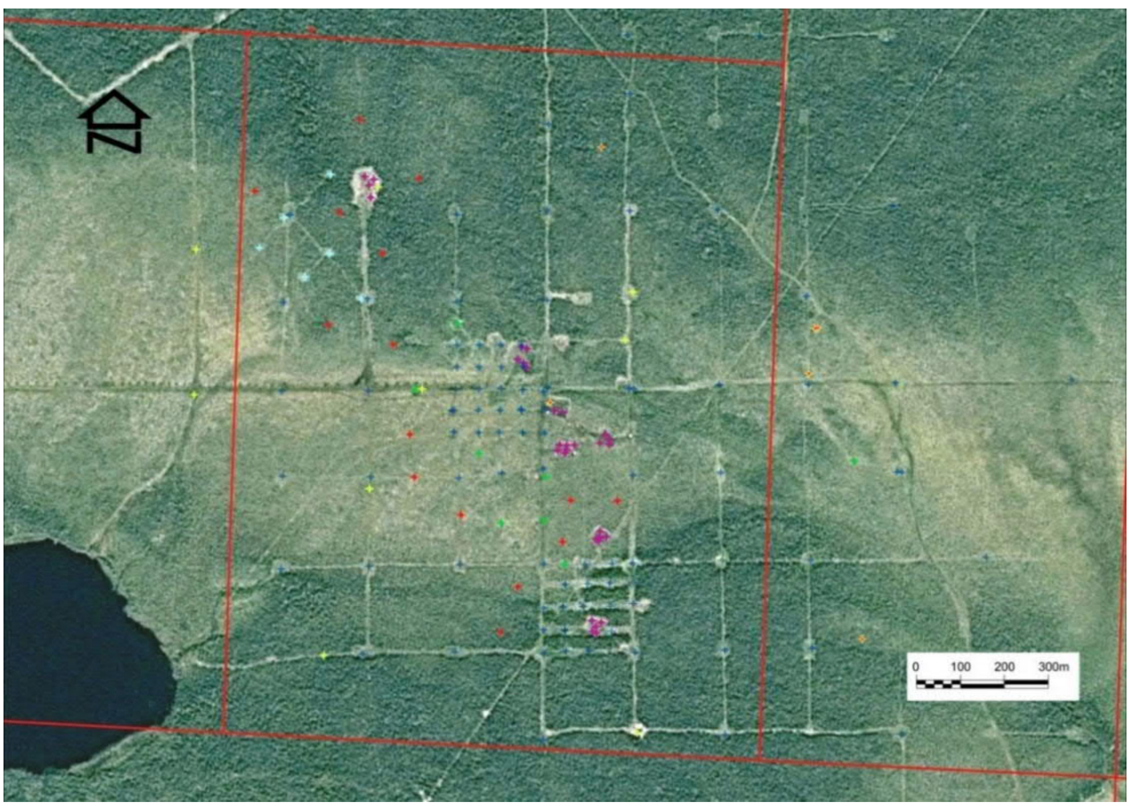


Figure 12-1: Martison Drillhole Surveyed Positions geo-referenced on a Project Air Photo Image

During the 2014 site visit, the QP checked a number of drillhole logs for the recorded sample intervals against sample book entry (Drillhole Number; From/To depths) and also checked the analytical data entries into the drillhole database against the data contained on the original analytical certificates. These were found to be in accordance with industry standard practice and no errors in data transfer were observed.

A number of drillhole logs and sample intervals were examined against the original analytical certificates at the core at the storage facility in Timmins in November 2019. Additional verification samples were taken from selected holes from the 2012 drill program, and the QP visited the ALS sample preparation laboratories in Timmins as part of the ongoing QA/QC process.

As a result of the ongoing data verification, the QP is of the opinion that the drillhole and sample database comply with industry standards and are adequate for the purposes of Mineral Resource estimation.

13. Mineral Processing & Metallurgical Testing

13.1 Metallurgical Testing History

The minerals of economic interest in the Martison deposit are apatite (42.2% P_2O_5) and pyrochlore (75.1% Nb_2O_5). The deposit formed as a result of severe weathering of an igneous rock (carbonatite) and the residuum contains chlorite, iron oxides such as magnetite, hematite, limonite, and goethite as well as the phosphate and niobium minerals and trace quantities of rare earths.

Significant metallurgical test work has been carried out on samples extracted from the Martison phosphate deposit. Initial testing took place at Lakefield Research of Canada Ltd, during the 1980s with the objective of developing flowsheets for recovering both phosphate and niobium concentrates. Later test work was largely conducted by Jacobs Engineering, Florida, US, who performed bench and pilot scale testing to examine the phosphate flowsheet recommended by Lakefield and subsequent processing of the phosphate concentrate to produce phosphatic fertilizers. ERIEZ Magnetics laboratories, Pennsylvania, US, (ERIEZ) also contributed to the detailed engineering studies with column cell and semi pilot plant scale testing.

13.2 History Prior to the 2008 PFS

Prior to the completion of the 2008 Martison Phosphate Project Prefeasibility Study (PFS), the following test work and studies were conducted:

1982[1]: 31 comprehensive beneficiation batch and closed circuit bench tests to address preliminary mineralogy, primary grinding, desliming, scrubbing and high intensity magnetic separation. A concentrate grade of 32.0% P_2O_5 was obtained with a P_2O_5 recovery of 73.4%. The tests were carried out on samples from drill cores. Niobium flotation was also tested.

[1] Fox River Reference ME1. An Investigation of THE RECOVERY OF PHOSPHATE AND PYROCHLORE from Martison Lake samples submitted by SHELL CANADA RESOURCES LIMITED, Progress Report No.1 by LAKEFIELD RESEARCH OF CANADA LIMITED, June 11, 1982

1983[2]: Mineralogical work was performed on flotation products and core samples to identify minerals of economic interest.

Additional tests were conducted on a higher grade material (25.1% P_2O_5) using samples of residuum from 13 sonic drill holes.[3]

At the conclusion of this program locked cycle testing produced the following concentrates:

- A phosphate concentrate containing 35.9% P_2O_5, and a P_2O_5 recovery of 80.4%.
- A niobium concentrate containing 36.0% Nb_2O_5, and a Nb_2O_5 recovery of 49.9%.

1984[4],[5]: Seven grinding and eight phosphate flotation pilot plant scale tests were conducted to finalize the beneficiation flowsheet. A concentrate grade of 36.3% P_2O_5 was obtained with a P_2O_5 recovery of 80.7%. The tests were performed on a 65 t ore sample from a single 48 inch churn hole designed to recover 110 t of residuum in the center of the deposit, including 45 t that were used in the pilot plant.

1999[6]: Bench scale metallurgical tests were performed by Cargill on 105 residuum samples from six core holes drilled along the center axis of the Martison deposit. Cargill concluded that about 29% of the residuum was acceptable as product without beneficiation and that the remaining 71% could be treated by desliming, magnetic separation, and flotation to produce acceptable concentrate. In 2004, Cargill merged with IMCF to form the Mosaic Company.

2007-2008: Preceding the 2008 PFS, Jacobs Engineering developed and managed a metallurgical test program that was carried out by Lakefield Research. The purpose of the metallurgical test program was to obtain enough phosphate concentrate to conduct pilot scale hemihydrate acidulation tests. The metallurgical testing followed the flotation flowsheet previously recommended by Lakefield to process two composites prepared using selected retained core samples from the 1999 and 2002 drilling programs. The flotation concentrate produced by Lakefield was sent to Eriez (Eriez Manufacturing Co.) to remove excess Fe_2O_3 by wet high intensity magnetic separation (WHIMS). Jacobs successfully converted the upgraded concentrate to phosphoric acid using the hemihydrate process and concentrated the resultant filter acid to merchant grade acid (MGA) and super phosphoric acid (SPA)[7].

[2] Fox River Reference ME1. MINERALOGICAL EXAMINATION of Phosphate Samples submitted by CAMCHIB RESOURCES LIMITED, Progress Report by LAKEFIELD RESEARCH OF CANADA LIMITED, July 22, 1983
[3] Fox River Reference ME 3. THE RECOVERY OF PHOSPHATE AND NIOBIUM from samples submitted by CAMCHIB RESOURCES LIMITED, Progress Report No. 1 by LAKEFIELD RESEARCH OF CANADA LIMITED, November 22, 1983
[4] Fox River Reference ME 8. A Laboratory and Pilot Plant Investigation of THE RECOVERY OF PHOSPHATE AND NIOBIUM from Hole 60B samples submitted by CAMCHIB RESOURCES LIMITED, Preliminary Report by LAKEFIELD RESEARCH, August 10, 1984
[5] Fox River Reference ME 8 pilot plant appendix. A Laboratory and Pilot Plant Investigation of THE RECOVERY OF PHOSPHATE AND NIOBIUM from pilot plant samples submitted by CAMCHIB RESOURCES LIMITED, Progress Report No. 3 by LAKEFIELD RESEARCH, November 9, 1984
[6] Martison Lake 1998-1999 Phosphate Drilling, Beneficiation report and Resource Estimation Study by Cargill Fertilizer, Inc., October 1999
[7] Evaluation of Martison Concentrate, by Jacobs Engineering SA, September 2007

The PFS beneficiation plant (mill) was designed with four main processing areas (crushing, grinding, magnetic separation, and flotation) as well as tailings disposal, water recycle, and reagent storage/use. The plant was designed for 3.13 Mtpa (dry) mill feed throughput to produce an annual concentrate production of 1.16 Mtpa.

The PFS mill flowsheet was based largely on Lakefield's preferred flowsheet with several modifications to address differences in quality between the pilot plant sample and the average grade for mill feed. The Jacobs' modifications consisted of:

- Added feed blending.
- Added wet high intensity magnetic separation ("WHIMS") to accommodate lower grade mill feed.
- Eliminated rougher/scavenger flotation circuits due to ineffective conditioning with dilute pulps.
- Replaced the cleaner/scavenger circuit with a classification step to recover coarse phosphate and reject fine gangue minerals.
- Replaced semi-autogenous grinding ("SAG") mill with less expensive equipment.

Jacobs recommended further batch and pilot scale studies using core samples from the 2008 bulk sample drilling program.

13.3 Post 2008 Studies[8],[9]

Following the completion of the 2008 PFS, additional test work was carried out by various parties as discussed in the paragraphs below. Phosphate beneficiation testing was carried out at Jacobs Engineering and consisted of locked cycle tests and continuous pilot plant testing. ERIEZ conducted bench scale column cell flotation tests as well as semi-pilot plant testing of drill core samples collected from seven locations within the Martison Anomaly A (the 2008 drill program).

Bench scale beneficiation tests by Jacobs were conducted to supplement the 2008 beneficiation program. The tests were performed on the same two feed samples that were treated in the pilot plant.

- Unconsolidated residuum or 2A material (from lower grade material assaying approximately 20% P_2O_5).
- Consolidated residuum or 2B material containing >25% P_2O_5.

These two samples were prepared using pilot scale equipment. A third sample was freshly prepared from the composite sample feed (2A+2B sub-lithotypes) to be used for the 2009 pilot plant program. The 2A+2B flotation feed was also prepared using Jacobs' pilot plant. The pilot plant generated feed was used to evaluate flotation reagents using bench scale tests and to evaluate column cell flotation performance.

[8] 2010 Program Beneficiation Test Report, by Jacobs Engineering SA, September 2011
[9] 2010 – 2011 Beneficiation Program Supplemental Report, by Jacobs Engineering SA, October 2011

13.4 2009 Jacobs Engineering Bench Scale Tests

Two samples, categorized according to chemical criteria as 2A and 2B lithotypes types, were prepared from cores drilled at seven locations. The 2A material ranged from 12% to 25% P_2O_5 and the 2B material exceeded 25% P_2O_5. The pilot plant tests examined a 45% / 55% blend of 2A and 2B material types as a representative average of the PFS mine plan. The test programs and summary findings from this work are listed below.

- The 2A+2B flotation feed prepared in Jacobs' pilot plant was used to evaluate flotation reagents using bench scale laboratory tests. Some of this material was shipped to Eriez to evaluate column cell flotation. Two fatty acids (JL7 and AP140) were found to be more selective than FA1, which was used previously. These reagents gave 2% to 4% higher P_2O_5 recovery for a given grade of concentrate than either FA1 or JL5.
- Two different amphoteric collectors were tested but did no better than the sarcosine previously tested. Reducing the level of sarcosine adversely impacted the flotation performance when used in conjunction with the standard reagent conditioning scheme because sarcosine also acts as a froth modifier. Adding sarcosine when slimes are present reduced frothing and prevented mechanical entrainment, while reducing sarcosine increased frothing which promotes mechanical entrainment and thereby reduces selectivity.
- Locked cycle testing of 2A feed was conducted to simulate continuous recycling of process water and coarse cleaner tailing and demonstrated that high quality concentrate could be reliably reproduced in bench scale testing. The P_2O_5 recovery from cycle to cycle fluctuated slightly, but the concentrate grade remained consistent. On average, a concentrate MER of 0.06 was achieved at 79% P_2O_5 recovery from flotation feed, equivalent to 62% P_2O_5 recovery from ore.
- Decantation tests of the 2A flotation feed indicated that desliming the feed at 20 µm to 10 µm would reject more than 40% of the problematic Fe_2O_3 and lose less phosphate than flotation. Removing the slimes would be expected to improve flotation performance and reduce reagent usage.
- The 2A+2B material was deslimed using standard procedures for scrubbing and hydrocycloning test work. The resultant feed was ground to two fractions, -300 µm and -425 µm, which were treated by two stages of magnetic separation to prepare the material for laboratory flotation tests. P_2O_5 recovery across desliming and magnetic separation was nominally 4% higher than achieved in the pilot plant; however, Fe_2O_3 rejection was 4% to 5% lower.

- Sieve and chemical analysis of the feed and concentrate indicated that liberation was achieved at the -300 µm and -425 µm grinds, thus concluding that capital and operating costs of grinding could be significantly reduced relative to the PFS. The pilot plant ground the test samples to pass either 300 µm or 220 µm after the hydraulic sizer was replaced by a Derrick vibrating screen. The WHIMS magnetic product had to be ground to pass 150 µm and then was reprocessed in the WHIMS to improve P_2O_5 recovery. Conclusions from this test program are:
  - Grinding mill feed to pass 425 µm is acceptable for desliming, low intensity magnetic separation (LIMS), and flotation.
  - Some of the phosphate is not liberated from the paramagnetic minerals until they are ground to pass 150 µm.

The overriding conclusion is to grind the mill feed to pass 425 µm and then regrind the WHIMS magnetic product to pass 150 µm in order to minimize slimes losses and maximize P_2O_5 recovery.

- Coarser (-425 µm) feed was easier to float than the -300 µm feed. Changing to a -425 µm grind is expected to reduce the capital investment associated with the grinding circuit. However, -300 µm feed exhibited excess frothing when the collector (sarcosine) dosage was lowered. Ultimately, a minimum of 0.6 kg/t of sarcosine was required to control frothing. This result prompts the question whether other, less costly, defoamers should be evaluated.
- The -300 µm feed was deslimed by elutriation (26% deslimed) and by wet screening (100% deslimed) and then floated. The 100% deslimed feed behaved more normally and acceptable quality concentrate was obtained without any sarcosine and frothing was not a problem. Preliminary economic comparison of desliming indicated that 100% desliming would lower cash costs; however, P_2O_5 recovery from ore to concentrate was reduced by 10% (from 74% to 64%).
- Settling tests were conducted using 2A feed to evaluate thickening parameters. Two tests were performed without flocculent using feeds ground to -212 µm and -150 µm. Test results indicated that thickening without flocculent was possible, however, flocculent may be required if the clarity of the thickener overflow becomes problematic.
- Two flotation tests were performed on 2A+2B feed to investigate whether flocculent would impact flotation performance. Without flocculent, the flotation P_2O_5 recovery from feed was 89.2% and the concentrate contained 38.53% P_2O_5. Repeating the test with water containing 5 g/t flocculent resulted in no change in P_2O_5 recovery and the concentrate contained 37.92% P_2O_5. Jacobs concluded that process water containing residual flocculent is not problematic in flotation.

- Eight flotation tests were performed on 2A+2B feed to investigate the impact of solids content and pH in the conditioning step. Lakefield's earlier work concluded that higher conditioning solids and higher pH were beneficial. Jacobs' experimental design did not confirm the Lakefield results. Furthermore, the test program revealed that concentrate P_2O_5 and MER were not significantly influenced by changes in conditioning solids content. As a result, conditioning solids for the cleaner flotation circuits for the PEA was specified at 45% to 50%.

13.5 Jacobs 2009/2010 Pilot Plant Test Program[10]

Twenty three (23) formal pilot plant runs were conducted by Jacobs in 2009/2010. The first 12 runs used the grinding flowsheet presented in the 2008 PFS report and showed that the grinding circuit was not well suited to the Martison material due to excessive generation of grinding slimes. An alternate grinding flowsheet, comprised of a rod mill in closed circuit with a Derrick screen, was utilized and resulted in lowering the grinding slimes and better flotation performance. Between runs 12 and 13, the pilot plant was modified to improve desliming performance. The modifications were successful to the extent that test results mimicked results obtained from the batch scale tests.

Pilot plant runs 17, 18 and 22-1 gave the best overall performance. A comparison of the flotation performance for these tests and compared to the performance from the locked cycle test are given in Table 13-1. The same comparison is presented graphically in Figure 13-1.

General conclusions derived from the pilot plant test program were:

- The Lakefield flowsheet rejected natural slimes by desliming prior to grinding but retained the slimes produced by grinding (milling slimes) in the flotation feed. The pilot plant results demonstrated that grinding slimes interfered with flotation and contaminated the concentrate with Fe_2O_3, Al_2O_3, and MgO.
- Continuous pilot plant conditioning and mechanical cell flotation performance was inferior to that obtained from bench scale batch testing.
- The variability of the pilot plant performance data and sensitivity to grinding slimes indicated that additional testing was required to develop and optimize a beneficiation flowsheet that will produce acceptable grade concentrates on a reliable basis.

[10] 2009 Beneficiation Program Report, by Jacobs Engineering SA, September, 2010

Table 13-1: Pilot Plant Test Results Comparison

Products	% Distribution		% P_2O_5	Chemical Analyses			MER
	Weight	P_2O_5		Fe_2O_3	Al_2O_3	MgO	
Locked Cycle Bench Tests for Combined 2A+2B Material							
Slimes Waste	20	10	13.5	22.8	6.6	4.7	2.53
LIMS Magnetic Waste	10	3	8.8	42.7	2.0	3.3	5.43
WHIMS Magnetic Waste	9	4	13.0	26.6	3.5	4.0	2.62
Flotation Tail	13	8	15.2	12.4	7.9	7.2	1.81
Concentrate	49	74	38.2	1.7	0.6	0.4	0.07
Combined 2A+2B	100	100	25.3	13.4	3.1	2.7	0.76
Pilot Plant (Best Three Runs)							
Slimes Waste	18	10	15.3	21.5	8.3	5.0	2.27
LIMS Magnetic Waste	6	2	8.8	42.7	2.0	3.3	5.47
WHIMS Magnetic Waste	10	5	15	24.4	4.3	3.1	2.12
Flotation Tail	15	11	18.6	12.0	6.1	6.1	1.31
Concentrate	51	72	36.9	2.2	0.6	0.4	0.09
Combined 2A+2B	100	100	26.4	11.7	3.3	2.5	0.66

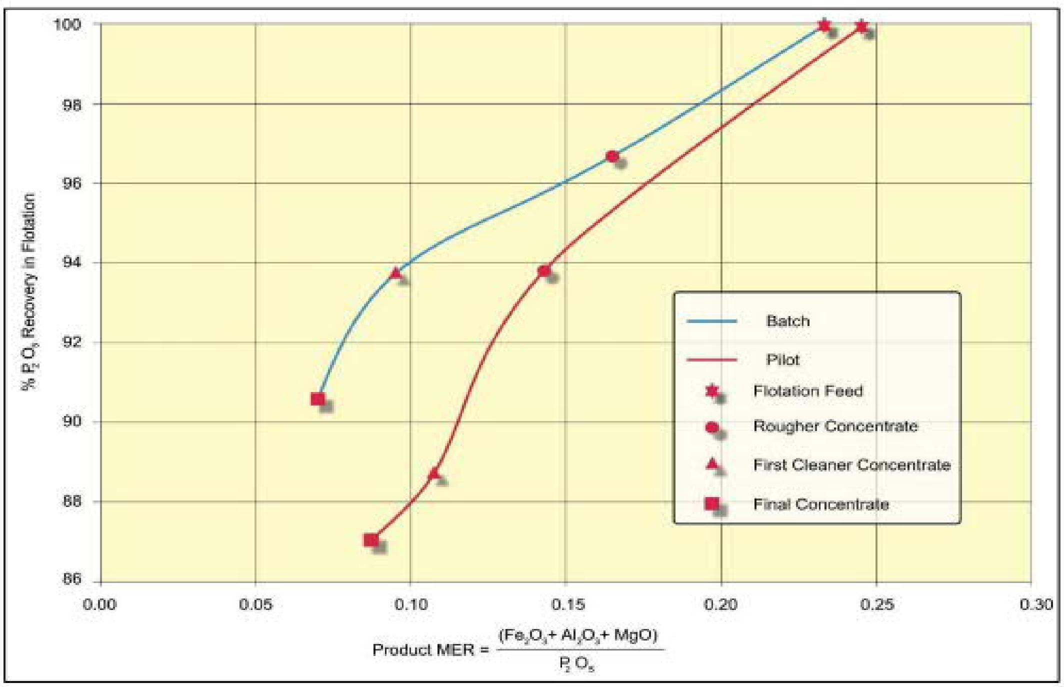


Figure 13-1: Bench Scale versus Pilot Plant Performance (2A Material)

13.6 Eriez 2011 Column Cell Test Program[11]

In 2011, Eriez conducted additional tests to supplement the 2009 beneficiation pilot plant program. These tests included column cell flotation on fully deslimed 2A, 2B, and 2A+2B samples, and column cell flotation on partially deslimed 2A+2B material.

The 2A sample was upgraded by desliming and flotation using bench scale column technology to simulate an open circuit rougher cleaner arrangement (without recycle). The objective was to maximize phosphate recovery and grade while minimizing the MER (Minor Element Ratio). A 3-inch diameter laboratory scale CPT (Canadian Process Technologies) cavitation tube column flotation cell and 2-inch diameter Krebs gMax cyclone were used. As summarized in Table 13-2 below, the P_2O_5 recovery after two cleaning stages was 80.2% with a MER of 0.061. If a second recleaning stage is utilized, the P_2O_5 recovery dropped to 76.1% with a MER of 0.053. Ultimately, a concentrate containing +39% P_2O_5 was produced at a fatty acid dosage of 0.5 kg/t. Sarcosine addition was set at 0.25 kg/t with an additional 0.05 kg/t added prior to the recleaning stage. The Eriez reagent dosages are per tonne of flotation feed.

Table 13-2: Mass Balance, 2A Mill Feed

	Mass	*Dist. (%)*	*Grade (%)*					
	%	**P_2O_5**	**P_2O_5**	**Insol**	**Fe_2O_3**	**Al_2O_3**	**MgO**	**MER**
Feed	100.0	100.0	30.8	4.8	4.9	2.2	2.6	0.316
Cyclone (Desliming)								
1st Stage Cyclone Overflow	11.11	7.5	20.8	9.6	15.6	5.3	3.9	1.196
1st Stage Cyclone Underflow	88.89	92.5	32.0	4.2	3.6	1.8	2.4	0.244
2nd Stage Overflow	2.97	2.2	23.0	10.2	11.0	5.2	4.6	0.905
2nd Stage Underflow	85.92	90.3	32.3	4.0	3.3	1.7	2.3	0.228
Rougher Flotation								
Concentrate	75.00	88.8	36.4	1.1	2.2	0.8	1.0	0.190
Tailings	10.90	1.5	4.2	24.2	11.1	8.2	11.6	7.278
Cleaner Flotation								
Concentrate	72.94	87.9	37.1	0.5	2.0	0.6	0.8	0.092
Tailings	2.06	0.9	13.4	20.0	8.4	6.2	8.6	1.725
Recleaner Flotation								
Concentrate	**63.93**	**80.2**	**38.6**	**0.3**	**1.5**	**0.4**	**0.5**	**0.061**
Tailings	9.01	7.7	26.3	2.2	5.8	2.2	2.9	0.413
2nd Recleaner Flotation								
Concentrate	**59.71**	**76.1**	**39.2**	**0.2**	**1.3**	**0.3**	**0.4**	**0.053**
Tailings	4.22	4.1	30.0	1.5	3.8	1.3	1.4	0.217

[11] Please refer to footnote 9.

The column cell testing on fully deslimed 2A material concluded that:

- Compared with locked cycle testing, the column cell gave 3% higher P_2O_5 recovery with a concentrate MER of 0.08.
- Column cell flotation of deslimed material required less collector to achieve equivalent metallurgical performance.

Eriez conducted additional column flotation testing on deslimed 2B ore after magnetic separation. The objective was to upgrade deslimed phosphate ore by utilizing both LIMS and WHIMS and using a series of bench-scale column flotation cells configured to simulate an open circuit rougher cleaner arrangement (without recycle).

LIMS test results show that approximately 95% of the material reported to the nonmagnetic stream. The magnetic material assayed 5.4% P_2O_5 yielding a unit phosphate recovery of over 99%. There was modest upgrading of the sample in terms of iron content as the Fe_2O_3 content dropped from over 8.1% to 5.0%. Due to this degree of iron capture, the MER was reduced from 0.314 to 0.206. The WHIMS test results show that the first pass through the WHIMS gave 82.7% mass recovery to the nonmagnetic stream which assayed 36.7% P_2O_5 and 2.3% Fe_2O_3. The MER for this fraction was only 0.107, which represented a significant reduction in minor element content. The first stage magnetic material, which contained some middling particles, was then reground and fed a second time through the WHIMS. The first pass and second pass nonmagnetic streams, when combined represent a rougher flotation feed having an Fe_2O_3 content of 2.6% and a P_2O_5 content of 36.4%.

The combined WHIMS testing provided a unit mass yield of 89.7% and P_2O_5 recovery of 95.0%. The MER was reduced from 0.206 to 0.118.

The 2011 Eriez test program results for 2B material are given in Table 13-3 and are summarized as follows:

- The concentrate grade was increased from 32.9% to 38.8% P_2O_5 using a rougher and two cleaner flotation stages.
- The MER was decreased from 0.314 to 0.044.
- LIMS was successful in selectively removing ferromagnetic material as demonstrated by the 5% mass recovery and Fe_2O_3 content of the material concentrated in the LIMS magnetic fraction. This material assayed at 65.6% Fe_2O_3.
- A significant upgrade to the LIMS nonmagnetic fraction was achieved by passing this material through two stages of WHIMS separation. This step included regrinding to improve liberation of phosphate. In this case, approximately 10%, of the circuit feed mass was lost to the WHIMS magnetic fraction as high iron content tailings.
- The rougher cleaner flotation circuit further upgraded the material resulting in a final concentrate containing 38.8% P_2O_5 with a MER of 0.044. Ultimately 69% of the available P_2O_5 was recovered at a mass yield of 58.7% (with respect to the LIMS feed).

Table 13-3: Summary of Eriez Test Results for 2B Mill Feed

	Mass	*Dist. (%)*	*Grade (%)*						
	%	P_2O_5	P_2O_5	Insol	Fe_2O_3	Al_2O_3	MgO	CaO	MER
Feed	100.0	100.0	32.89	2.73	8.11	1.29	0.92	47.20	0.314
LIMS									
Mag	5.07	0.8	5.36	17.64	65.66	2.04	3.64	8.54	13.310
Non-Mag	94.93	99.2	34.36	1.94	5.04	1.25	0.78	49.27	0.206
WHIMS (Pass 1)									
Mag	16.45	11.5	23.01	6.26	18.02	2.30	1.72	36.27	0.958
Non-Mag	78.48	87.7	36.74	1.03	2.32	1.02	0.58	51.99	0.107
WHIMS (Pass 2)									
Mag	9.80	5.0	16.64	8.81	26.51	2.67	1.88	43.93	1.866
Non-Mag	6.65	6.6	32.40	2.50	5.52	1.76	1.48	24.99	0.270
Flotation Feed									
Combined WHIMS N-Mag	85.13	94.2	36.40	1.15	2.57	1.08	0.65	49.88	0.118
Rougher Flotation									
Concentrate	82.45	92.8	37.00	0.81	2.40	0.83	0.41	50.73	0.098
Tailings	2.70	1.5	17.92	11.41	7.62	8.75	8.23	23.97	1.373
Cleaner Flotation									
Concentrate	80.73	91.6	37.33	0.68	2.17	0.71	0.34	51.22	0.086
Tailings	1.71	1.1	21.49	7.09	13.56	6.71	3.67	27.58	1.114
2nd Cleaner Flotation									
Final Concentrate	**58.69**	**69.3**	**38.84**	**0.42**	**1.13**	**0.38**	**0.19**	**52.51**	**0.044**
Tailings	22.04	22.3	33.31	1.36	4.93	1.58	0.73	47.78	0.217

In summary, Eriez concluded from separate testing of deslimed Martison 2A and 2B materials, that LIMS and WHIMS removal of magnetic and paramagnetic material followed by froth flotation using column flotation cells resulted in a robust metallurgical performance and that column cells are superior to mechanical cells for the samples tested.

The recovery versus grade test data from fully deslimed 2A and 2B samples using column flotation were used to establish the following equation to predict P_2O_5 recovery to flotation concentrate as a function of mill feed % P_2O_5.

$$\% \ P_2O_5 \ \text{Recovery} = 53.8 + 0.84(\text{Mill Feed} \ \% \ P_2O_5)$$

This equation is based on producing a concentrate with a MER of 0.08. The PEA concentrate assumes a MER of 0.09 and therefore the above recovery equation used for PEA mine planning purposes to calculate the tonnes of mill feed from each mining block required to satisfy the P_2O_5 demand of the downstream FCC is slightly conservative.

Jacobs evaluated both high grade and low grade flotation concentrate from the beneficiation pilot plant to determine suitability for producing phosphoric acid using the dihydrate process and for converting the filter acid to merchant grade acid (MGA), granular fertilizer, and super phosphoric acid (SPA). The high grade concentrate tests successfully produced MGA, DAP, and SPA. Liquid fertilizer (10:34:0) was produced using the SPA. The MER of the filter acid was 0.032. The filter acid produced from the low grade concentrate had a MER of 0.102 and consequently the MAP produced did not meet the market specification of 11:52:0 for commercial MAP.

No testing of rock concentrate slurry for the design of the pipeline has been performed and the pipeline design is based on Ausenco PSI's experience with phosphate slurry pipelines.

13.7 Post 2009 Niobium Testing[12]

Various samples streams from the Jacobs phosphate pilot plant were examined by COREM, SGS, and Eriez to evaluate Niobium recovery. The testing indicated that concentrates containing greater than 50% Nb_2O_5 could be produced, but recovery was low for all tests. The tailings from phosphate flotation had lower iron content than other waste streams and were identified as a preferred material for further testing of niobium recovery process development.

14. Mineral Resource Estimates

14.1 Summary

As part of the ongoing validation process of the Martison Phosphate Project database, and in response to the changed technical and economic configuration of the Project, a review of the geological database has been carried out.

Increased knowledge of the chemical relationships of key elements, and compounds within the deposit, have enabled a reanalysis of the deposit geochemistry.

These two activities were undertaken in response to the ongoing metallurgical studies and the projected downstream effects the deposit geochemistry may have on the proposed Project mineral processing flowsheet.

The database review has also resulted in the reinterpretation of some of the lithological descriptions and minor revision of some litho-codes and litho-contacts used in the geological remodelling of the deposit and, subsequently, the reconfiguration of the input parameters for the block model.

In alignment with these adjustments, the database has been given a revised cut-off date as of December 31, 2021.

A 3D geological model was constructed based on lithotype wireframes which was built using all available validated and verified data, up to and including the latest drill and sampling program of 2012.

[12] Technical Report on the Martison Phosphate Project, Ontario, Canada pages 90 – 105, (Document Ref.:D57 – R – 178), Reissued April 11, 2016

From the 3D geological wireframe an open pit shell was constructed to constrain the two litho-types of interest, the residuum and lateritic material, to arrive at an estimate of Mineral Resources for both phosphate (P_2O_5) and niobium (Nb) for the Martison Project.

The Mineral Resource Estimate (MRE) is presented in Table 14-1 and has an effective date of December 31, 2021.

Table 14-1: Martison Mineral Resource Estimate as of December 31, 2021

Deposit	Classification	Tonnes Mt	Phosphate Grade % P_2O_5	Niobium Grade % Nb_2O_5
Anomaly A Residuum	Indicated Resources	53.8	22.99	0.42
	Inferred Resources	128.3	17.09	0.42
Anomaly A Lateritic material	Indicated Resources	6.2	7.97	1.13
	Inferred Resources	5.3	6.40	0.69

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a cut-off grade of 6% P_2O_5 in the Residuum or 0.2% Nb_2O_5 in the Lateritic material.
3. Mineral Resources are estimated at a dry Bulk Density of 1.89 t/m3, 1.70 t/m³, 1.90 t/m³, 2.12 t/m³ for till, lateritic material, residuum and carbonatite respectively.
4. Mineral Resources are constrained by a Whittle open pit shell.
5. A minimum mineralisation width of five metres was used for Indicated Resources and 2 m for Inferred.
6. Values for tonnage and grade may not add up due to rounding.

14.2 Geological Model

Three dimensional models of the geology in the Martison Project were constructed in Geovia Surpac v7. A topographic digital terrain model ("DTM") was generated from LiDAR survey data provided by Fox River and from satellite imagery.

Two wireframe models of the phosphate mineralized residuum in Anomaly A (based on a 6% P_2O_5 threshold) and a number of niobium rich lateritic material wireframes (based on a 0.2% Nb_2O_5 threshold) were used in geological and grade continuity studies to constrain the block model interpolation.

Capping (also known as cutting) was not used on the phosphate grades in the residuum on account of the presence of minimal high-grade outliers. Capping was used in the lateritic material due to the disproportionate number of P_2O_5 high grade outliers. The niobium was capped in the both the residuum and the lateritic material. Samples were composited to 2 m intervals. The Mineral Resource Estimate was estimated using the Inverse Distance Cubed (ID^3) method.

The definitions for Mineral Resource Categories used in this estimate are consistent with those set out in the Canadian Institute of Mining, Metallurgy and Petroleum 2014 Definition Standards on Mineral Resources and Mineral Reserves ("CIM Definitions").

Anomaly A residuum contains an estimated 53.8 Mt of Indicated Mineral Resources at a grade of 22.99% P_2O_5 and 0.42% Nb_2O_5, and 128.3 Mt of Inferred Mineral Resources at a grade of 17.09% P_2O_5 and 0.42% Nb_2O_5. The residuum resources are reported at a cut-off grade of 6% P_2O_5.

The lateritic material contains an estimated Indicated Mineral Resource of 6.2 Mt at 1.13% Nb_2O_5 and 7.97% P_2O_5 and an Inferred Mineral Resource of 5.3 Mt at 0.69% Nb_2O_5 and 6.40% P_2O_5 at a cut-off grade of 0.2% Nb_2O_5.

The Mineral Resources are reported within a conceptual (Whittle) open pit at a cut-off grade of 6% P_2O_5.

Anomaly B remains unchanged from the 2015 MRE and represents a target for further exploration currently estimated at between 35 Mt and 70 Mt of residuum containing 14% - 30% P_2O_5. The quantity and grade are conceptual in nature and there has been insufficient exploration to define a Mineral Resource and it is uncertain if further exploration will result in the target being classified in this manner in the future. Currently this anomaly does not have demonstrated economic viability and is disclosed with a potential quantity and grade, expressed as ranges. Further definition will be the target of future exploration.

Anomaly C is an early exploration target only.

14.3 Wireframe Modelling

14.3.1 Lithological Wireframe Modelling

Lithological wireframe models were created to facilitate geological understanding and controls on the geochemistry and mineralisation for Anomalies A and B.

Wireframe models were generated for the glacial till, lateritic material, residuum and bedrock (carbonatite) in Anomaly A (Figure 14-1 and Figure 14-2).

A distinction was not made at this stage of the resource estimation between the different lithotypes of residuum noted during logging (e.g., 2A unconsolidated residuum; 2B re-cemented residuum and 2C transitional (waste) residuum).

An additional parameter was introduced for the block model to further establish sub-lithotype definition, principally for the purposes of identifying variability in residuum geochemistry for downstream processing studies. This parameter is based on the ratio $CaO:P_2O_5$. (Refer to Section 14.5 Block Model).

Cross sections looking North, at intervals of 50 m, were drawn through Anomaly A. Typically, drillholes within 25 m either side of the section line were incorporated into the section. The geological model was extrapolated typically 250 m and interpreted between drillholes and between sections. The digitized interpretations were “snapped” to drillholes.

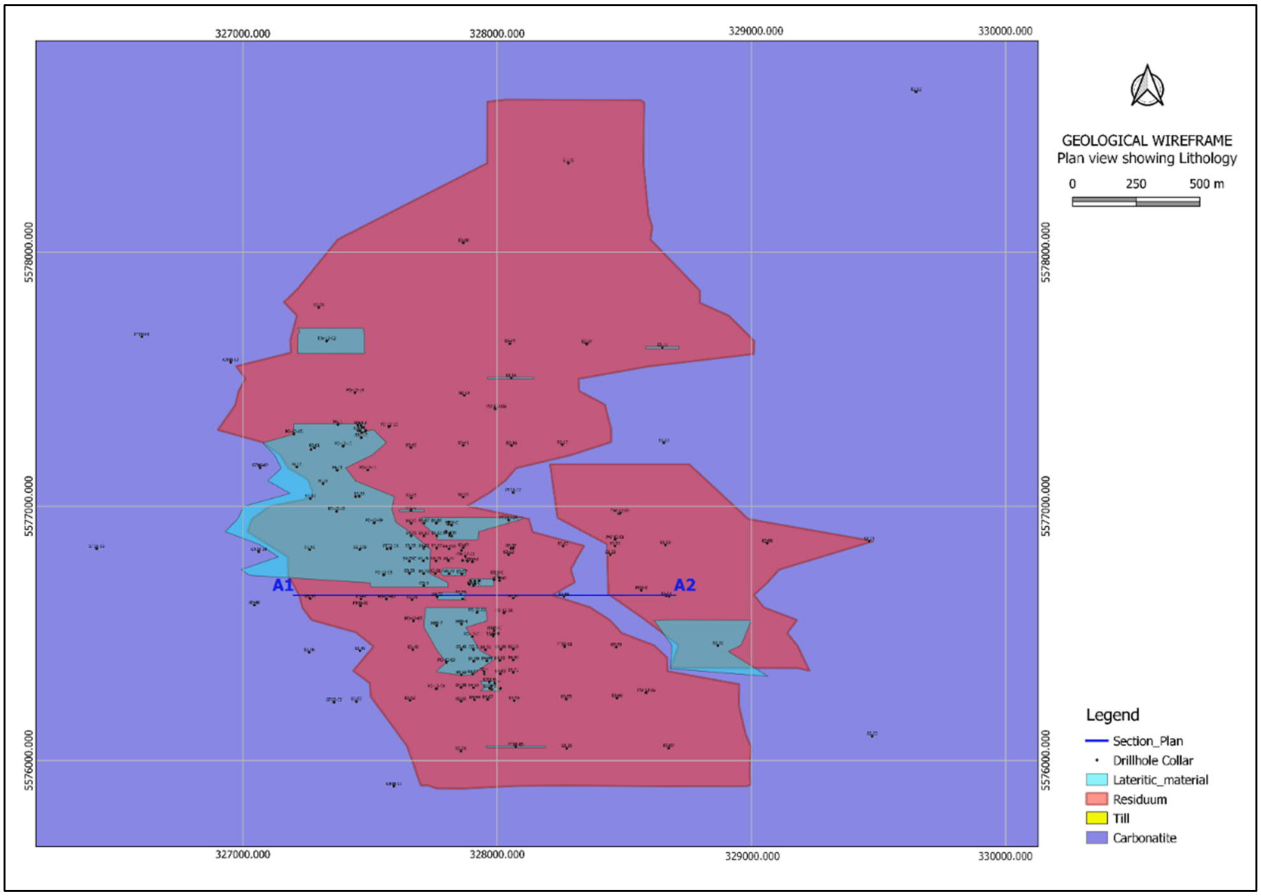


Figure 14-1: Geological Wireframes – Plan View showing DH locations with respect to Residuum, Lateritic Material and Carbonatite Domains

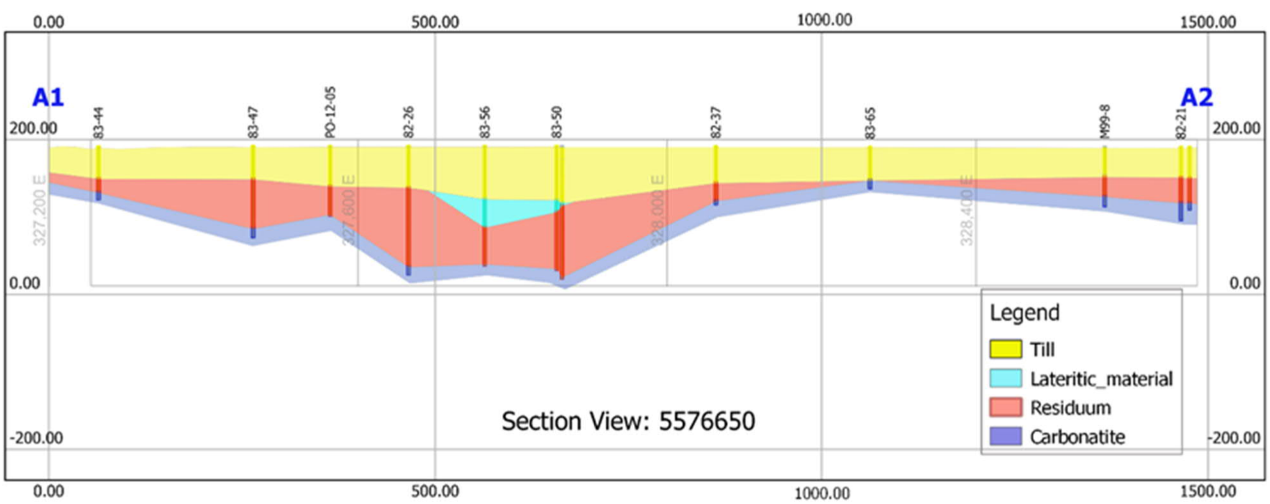


Figure 14-2: Geological Wireframes – Section View (5576650N)

Figure 14-3 is a histogram plot of the phosphate assays within Anomaly A residuum and indicates a threshold of 6% P_2O_5 between the anomalous and background populations, and hence the pit shell constraint at 6% P_2O_5.

The wireframes were checked and validated for intersections, inconsistencies, and closure.

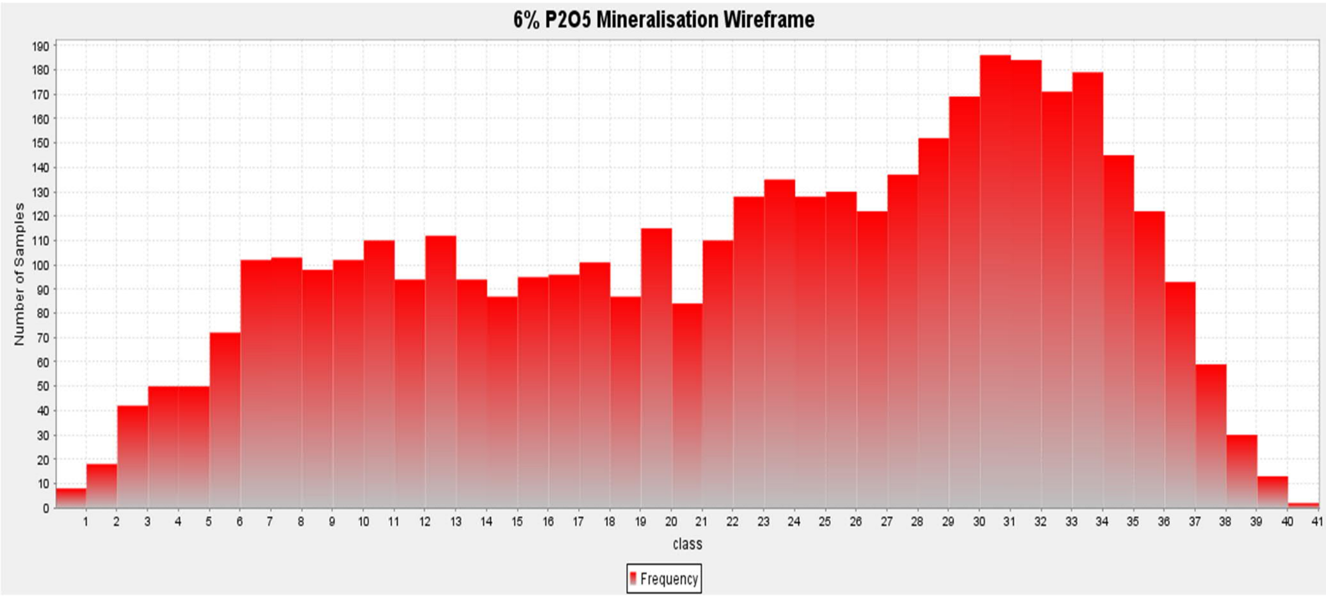


Figure 14-3: Phosphate (P_2O_5) Sample Frequency Histogram

14.3.2 Mineralized Residuum Wireframe Modelling

Wireframe models of the Anomaly A phosphate mineralised residuum were utilized in geological and grade continuity studies to constrain the block model interpolation. A threshold of 6% P_2O_5 was taken as the break between mineralized (anomalous) and non-mineralized (background) material. Mineralization was interpreted between drillholes and between sections.

Two mineralized lenses at a threshold grade of 6% P_2O_5 were wireframed, east and west. The western lens is significantly larger and contains 92% of the total estimated Anomaly A resource and all of the estimated Indicated resource. The eastern lens contains only 8% of the total estimated resource and all the tonnage is in the Inferred category. For this reason, the technical study of the resource database in Section 14.6 through to Section 14.7 refers to observations made within the larger western lens of Anomaly A only. A plan and section of the mineralized wireframes for the phosphate are depicted in Figure 14-4 and Figure 14-5 respectively. The mineralized lenses contain some material below 6% P_2O_5 to preserve interpretation continuity.

A minimum thickness of 5 m was employed to constrain the mineralisation within the Indicated wireframe, and 2 m within the Inferred wireframe.

Along trend and perpendicular to trend (strike and dip), continuity was generally limited to 200 m (Indicated) – 250 m (Inferred). (Note: extrapolation was extended in selected areas for reasons of geological continuity).

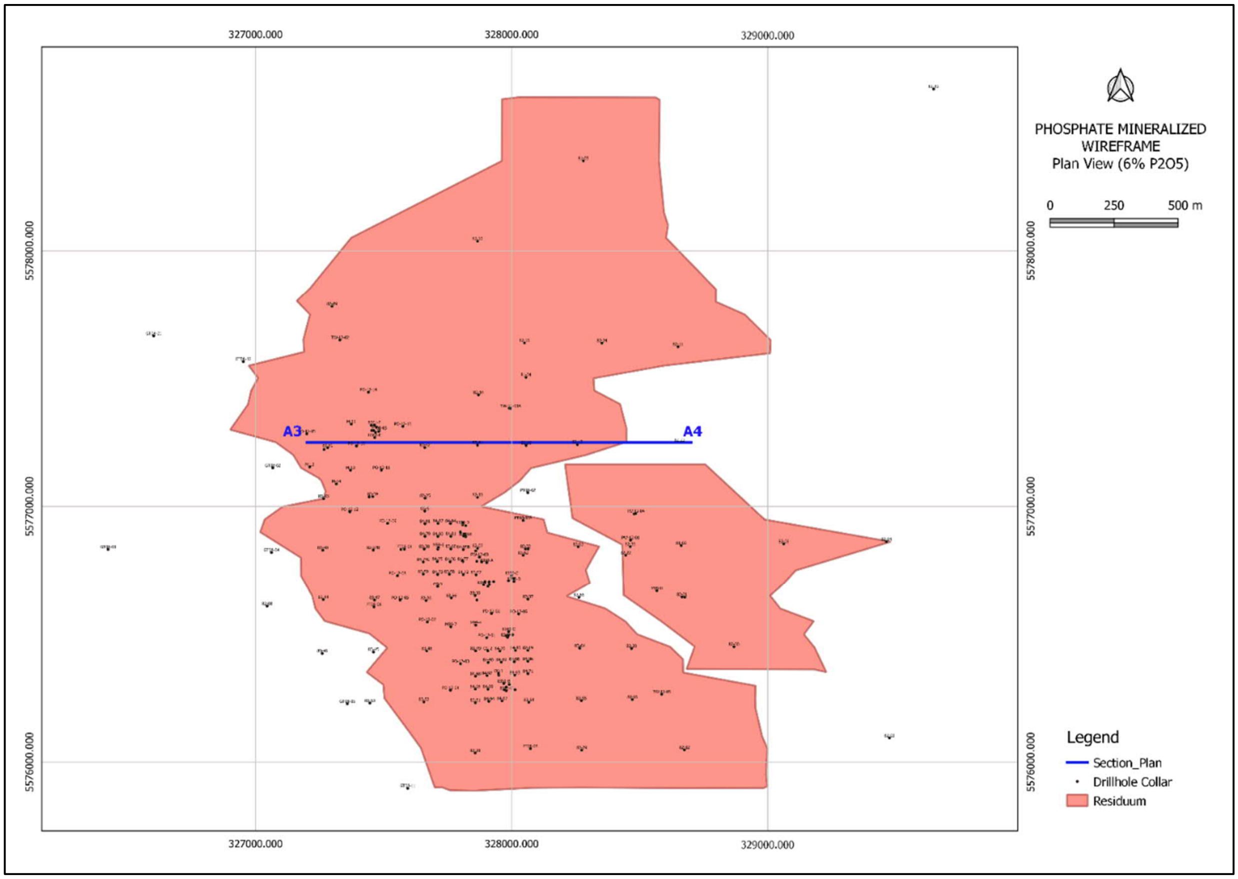


Figure 14-4: Phosphate Mineralized Wireframes (=6% P_2O_5) – Plan View

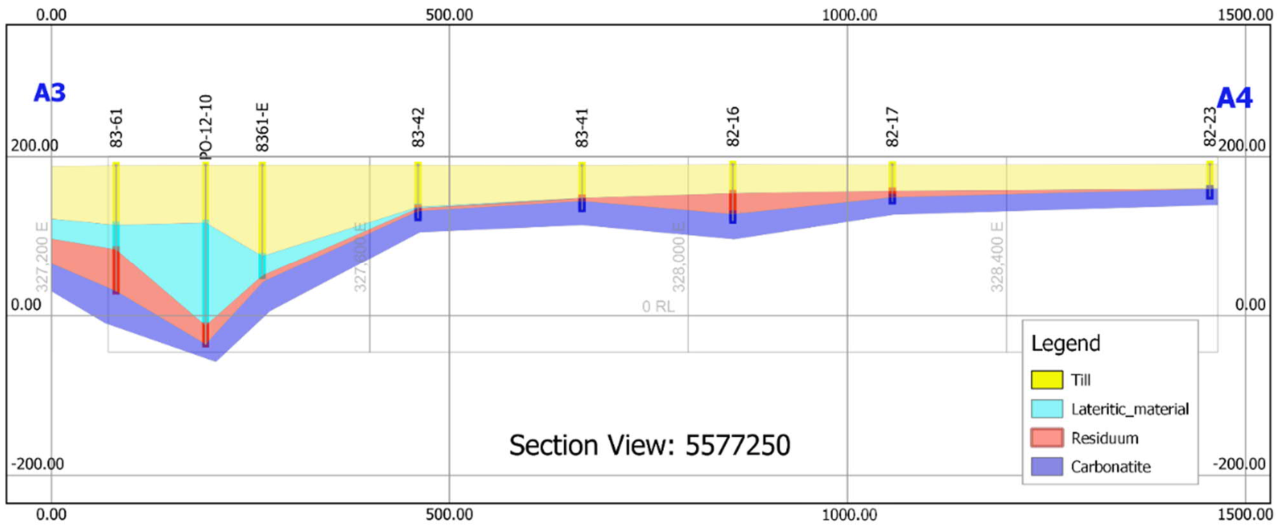


Figure 14-5: Phosphate Mineralized Wireframes (=6% P_2O_5) Section View (5577250N)

Anomaly B phosphate mineralization was not modelled.

14.3.3 Topography (LiDAR)

A digital terrain model (DTM) was created to represent the topographic surface from the LiDAR survey. 78 drillhole collar locations were assigned within the Anomaly A resource area of interest and the elevations were estimated from the LiDAR survey as they had not been accurately surveyed with regard to elevation. An additional 13 drillholes outside the area of interest have an assumed elevation of 190 m.

The drillholes in Anomaly B have been assigned an assumed elevation of 190 m.

14.4 Resource Database

200 drillholes for 19,840.8 m exist in the digital drillhole database (totalling 214 holes for 21,412 m overall) pertain to Anomaly A and were lithologically modelled to include 499 intercepts over 18,009.92 m, as shown in Table 14-2.

87 of the Anomaly A drillholes intersected residuum. The average residuum intersected was 46.5 m thick, approximately 50 m to 60 m deep below the glacial till or a combination of glacial till and lateritic material. 58 drillholes intersected the lateritic material with an average thickness of 15.46 m.

Table 14-2: Summary of Anomaly A Drill Hole Intersections Used for Lithological Modelling

Lithology	No. of Drillhole Intercepts	Average Depth from – to m	Average Thickness m	Total Length of Intercepts m	% of Total Intercepts by Length
Till	185	0-88.5	45.26	8,373.80	45.27
Lateritic material	58	31.7-142.3	15.46	896.70	4.85
Residuum	166	30.5-223.08	48.53	8,056.34	43.55
Bedrock (Carbonatite)	92	30.48-213	30.48	1,171.60	6.33
Total	**501**			**18,498.44**	**100**

The Anomaly A residuum lithological model captured 4,732 samples of which 4,509 had phosphate assays and 4,384 niobium assays. The assays in the residuum have an average assay grade of 20.68% P_2O_5 and 0.42% Nb_2O_5 as referenced in Table 14-3 below for the Anomaly A Lithological Wireframe Assay Statistics.

The lateritic material model has an average assay grade of 1.02% Nb_2O_5 and 6.73%P_2O_5.

Table 14-3: Anomaly A Lithological Wireframe Assay Statistics

Parameter	Lithological Domain				Total
	Till	Lateritic material	Residuum	Bedrock	
No. Samples	724	543	4,732	632	6,631
Mean Sample Length (m)	11.57	1.70	1.70	1.85	2.79
P_2O_5					
No. P_2O_5% Assays	386	510	4,509	535	5,940
Mean P_2O_5% Grade	2.13	6.64	20.68	6.75	16.70
Median P_2O_5% Grade	0.14	5.81	22.10	4.13	15.20
Min. P_2O_5% Grade	0.00	0.02	0.02	0.02	0.00
Max. P_2O_5% Grade	33.40	36.60	40.20	36.00	40.20
P_2O_5% Standard Deviation	6.42	5.62	10.67	7.35	11.85
P_2O_5% Coeff. Of Variation	3.02	0.85	0.51	1.09	0.70

Parameter	Lithological Domain				Total
	Till	Lateritic material	Residuum	Bedrock	
Nb_2O_5					
No. Nb_2O_5% Assays	384	510	4,384	500	5,778
Mean Nb_2O_5% Grade	0.05	0.99	0.42	0.25	0.44
Median Nb_2O_5% Grade	0.00	0.67	0.28	0.12	0.26
Min. Nb_2O_5% Grade	0.00	0.03	0.00	0.00	0.01
Max. Nb_2O_5% Grade	1.19	6.24	6.22	5.16	6.24
Nb_2O_5% Standard Deviation	0.17	0.99	0.44	0.44	0.54
Nb_2O_5% Coeff. Of Variation	3.26	1.01	1.04	1.77	1.23

14.4.1 Phosphate Mineralized Residuum Database

164 of the drillholes were further utilized to model the Anomaly A mineralized residuum at a threshold of 6% P_2O_5, which incorporated 7,567.82 m of drill intercepts, as shown in Table 14-4.

Table 14-4: Summary of Anomaly A Drillhole Intersections used for Phosphate Modelling

Area	No. of Drillhole Intercepts	Average Depth from – to m	Average Thickness m	Total Length of Intercepts m
Total	164	30.5-224.63	46.15	7,567.82

The Anomaly A mineralized residuum wireframes captured 4,442 samples with 4,248 phosphate and 4,141 niobium assays, as shown in Table 14-5. The average sample length within the mineralized residuum wireframes was 1.70 m, the average phosphate sample grade was 21.62% P_2O_5 and the average niobium sample grade was 0.43% Nb_2O_5.

Table 14-5: Anomaly A Mineralised Residuum Wireframe Assay Statistics

Parameter	Mineralised Residuum Domain		Total
	West (No. 8)	East (No. 9)	
No. Samples	4,328	114	4,141
Mean Sample Length (m)	1.70	1.82	1.70
P_2O_5			
No. P_2O_5% Assays	4,137	111	4,248
Mean P_2O_5% Grade	21.70	18.63	21.62
Median P_2O_5% Grade	22.90	16.80	22.80
Min. P_2O_5% Grade	0.02	1.54	0.02
Max. P_2O_5% Grade	40.20	38.60	40.20
P_2O_5% Standard Deviation	10.30	11.87	10.35
P_2O_5% Coeff. Of Variation	0.47	0.64	0.48
Nb_2O_5			
No. Nb_2O_5% Assays	4,030	111	4,141
Mean Nb_2O_5% Grade	0.44	0.23	0.43
Median Nb_2O_5% Grade	0.29	0.12	0.28
Min. Nb_2O_5% Grade	0.00	0.04	0.00
Max. Nb_2O_5% Grade	6.22	1.89	6.22
Nb_2O_5% Standard Deviation	0.45	0.23	0.43
Nb_2O_5% Coeff. Of Variation	1.04	0.99	1.00

For reference only, Table 14-6 below summarizes the drillhole intersections for Anomaly B. The 157 phosphate assays in Anomaly B range between 2.0% and 37.5% P_2O_5, with a median grade of 18.6% P_2O_5 and a mean of 18.7% P_2O_5.

Table 14-6: Summary of Anomaly B Residuum Drill Hole Intersections

Lithology	No. of Drill Hole Intercepts	Typical Depth m	Average Thickness m	Total Length of Intercepts m
Residuum	14	64	39	466

14.4.2 *Niobium Mineralized Lateritic Material Database*

Fifty-eight drillholes were also utilized to model the mineralised lateritic material at a threshold of 0.2% Nb_2O_5, which incorporated 896.7 m of drill intercepts, as shown in Table 14-7 below.

Table 14-7: Summary of Anomaly A Drillhole Intersections used for Niobium Modelling

Lithology	No. of Drillhole Intercepts	Average Depth from – to m	Average Thickness m	Total Length of Intercepts m
Lateritic Material	505	31.7-142.3	15.46	896.7

The niobium mineralized lateritic material domain wireframes captured 530 samples with 505 having phosphate and niobium assays as shown in Table 14-8 below.

The average sample length within the mineralized lateritic material wireframes was 1.7 m, the average phosphate sample grade was 6.73% P_2O_5 and the average niobium sample grade was 1.02% Nb_2O_5.

Table 14-8: Summary of Anomaly A Drillhole Intersections used for Niobium Modelling

Laterite Domain	No. Of Drillhole Intercepts	Average Depth From-To (m)	Average Thickness (m)	Total Length of Intercepts (m)	% of Total Intercepts by Length
1	5	32-68	26	105	11.71
2	6	35-60	8	47	5.24
3	2	72-84	8	16	1.78
4	29	36-142	27	557	62.12
5	1	81-108	37	27	3.01
6	2	54-63	5	11	1.23
7	10	33-54	18	117	13.05
8	1	40-43	3	3	0.33
9	1	43-46	3	3	0.33
10	1	66-76	10	10.7	1.19
Total	**58**		**15**	**897**	**100.00**

14.4.3 *Data Verification*

Verification of the drillhole database was conducted by checking for:

- Drillholes omitted from the 2015 MRE. These were subsequently included in the database (82-31, 82-22 ,82-08, 82-10 and 81-05). DH 82-10 and 81-05 were included in the residuum domain.
- Capture of drillholes from 2001 in Anomaly B.
- Missing and overlapping sample intervals (lithological logs and assay results). These were captured and corrected respectively.
- Visual plotting of drillhole locations to determine any obvious errors Figure 12-1. A GPS comparison of a number of drillholes, with corresponding survey coordinates, were checked during the 2014 site visit.

14.4.4 *Bulk Density*

Bulk densities for the Martison Project are based on empirical data from the Agrium Kapuskasing phosphate deposit, and a series of tests conducted in 1984 and 1999 on Martison samples. The 1984 tests were primarily series of wet volume displacement methods on competent materials. The 1999 tests were a series of manual packed-mould tests on unconsolidated residuum (which is particularly subject to “over packing” and often does not reflect actual in-situ porosity conditions).

The principal difficulty in estimating the bulk density for the residuum material is directly related to its geological history, in-situ porosity, and resulting moisture content. Many drill logs and most core-sample descriptions portray the residuum, particularly the ‘recemented’ (2B) sub-lithology as being “vuggy” to “extremely vuggy”. This, by necessity, must reduce the estimated bulk density of the material from what might be expected of the same material in a less weathered state.

Based on the testing results and anecdotal notes, conservative bulk densities and estimated moisture contents are listed in Table 14-9 and were determined by previous work carried by PhosCan and have been utilised in the 2021 estimate. All tonnages are reported as dry tonnes unless otherwise stated.

Table 14-9: Bulk Density

Lithology	Moisture Content %	Dry Bulk Density (t/m^3)
Glacial Till	10	1.89
Lateritic material	15	1.70
Residuum	15 – 11	1.90
Bedrock (Carbonatite)	10	2.12

14.4.5 *Assay Capping (Cutting)*

To avoid any disproportionate influence of random, anomalously high-grade assays on the resource average grade, the data was assessed for grade capping requirements.

Histogram, probability and cumulative frequency plots of the phosphate assays within the mineralised residuum wireframes are presented in Figure 14-6, Figure 14-7 and Figure 14-8 respectively. The probability plot shows the 99th percentile of the assays is at 38% P_2O_5 and the cumulative frequency curve flattens at approximately 38% P_2O_5, indicating a possible capping level. Of the 4,248 phosphate assays, 46 (1.08%) have a grade of 38% P_2O_5 or above, which is not considered a significant proportion.

The very high-grade assays (≥ 38% P_2O_5) can be observed throughout the domain (laterally and vertically) and show good continuity. Hence, the high values are not regarded as anomalous, and therefore no capping of the assays for P_2O_5 in the Residuum was considered necessary.

The histogram of phosphate assays encapsulated in Anomaly A indicates the presence of three sub-domains (6% - 20% P_2O_5, 20% - 26% P_2O_5 and 26% - 40% P_2O_5). These sub-domains correspond approximately with the residuum sub-types of 2C, 2A and 2B respectively.

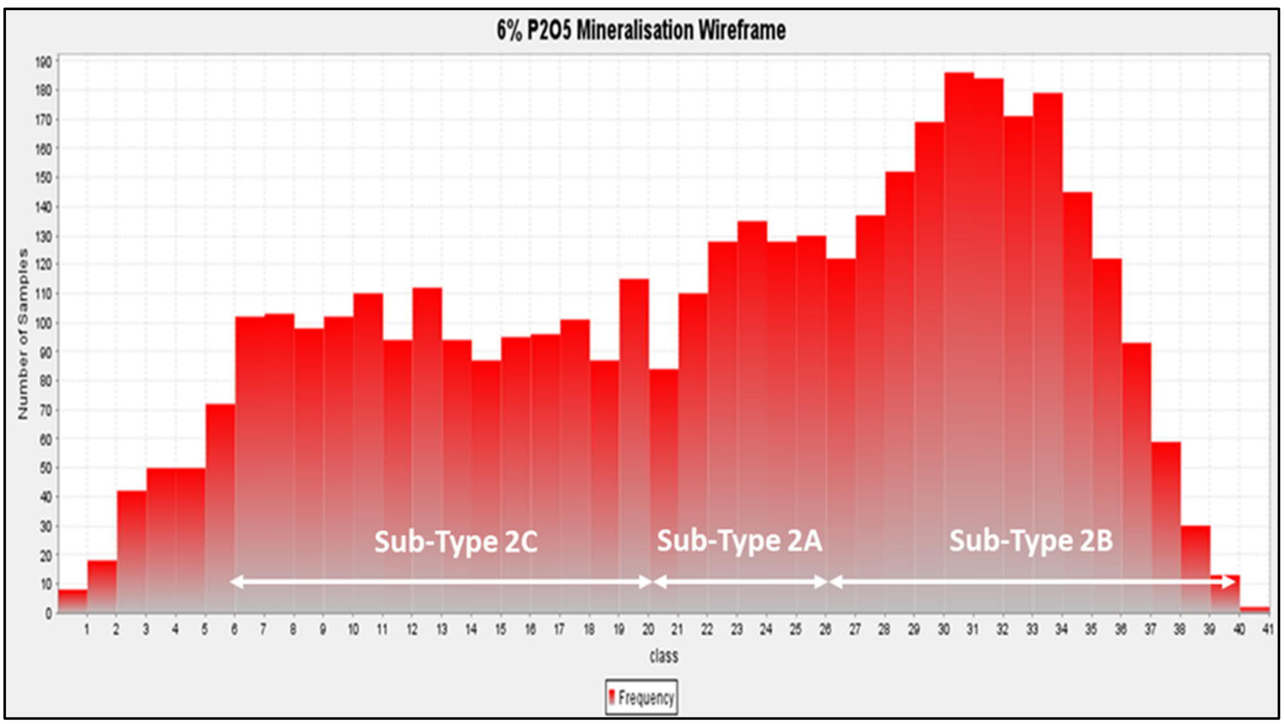


Figure 14-6: **Mineralized Residuum P_2O_5 Sample Histogram**

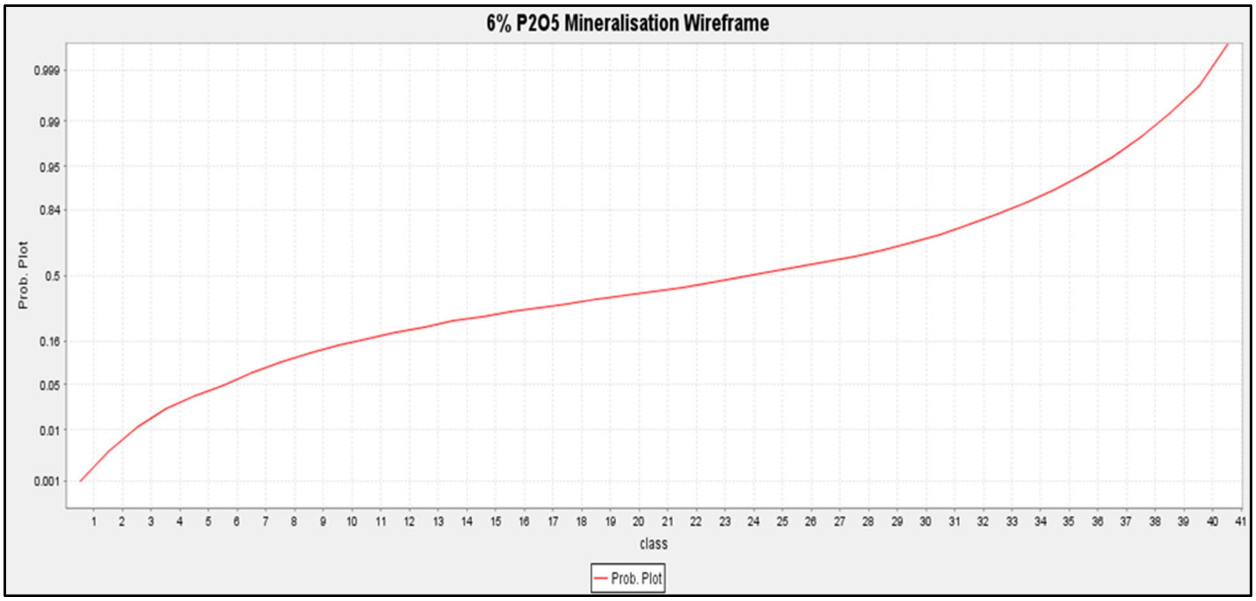


Figure 14-7: Mineralized Residuum P_2O_5 Sample Probability Plot

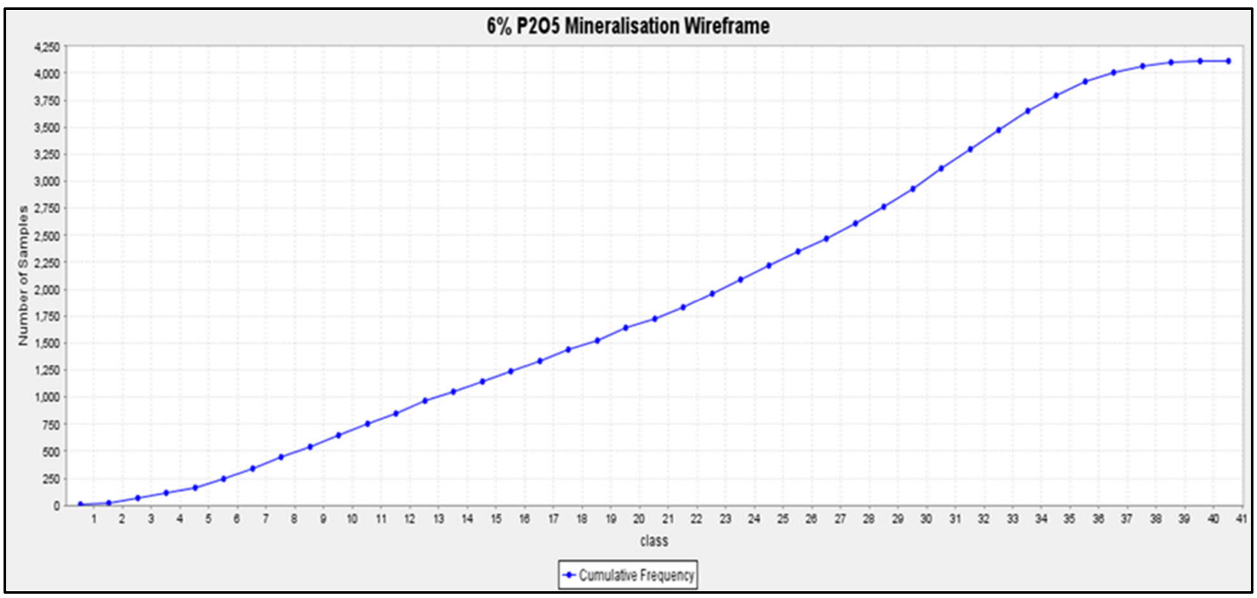


Figure 14-8: Mineralized Residuum P_2O_5 Sample Cumulative Frequency

Histogram, probability and cumulative frequency plots of the niobium assays within the mineralized residuum wireframes are presented in Figure 14-9 and Figure 14-10 and Figure 14-11.

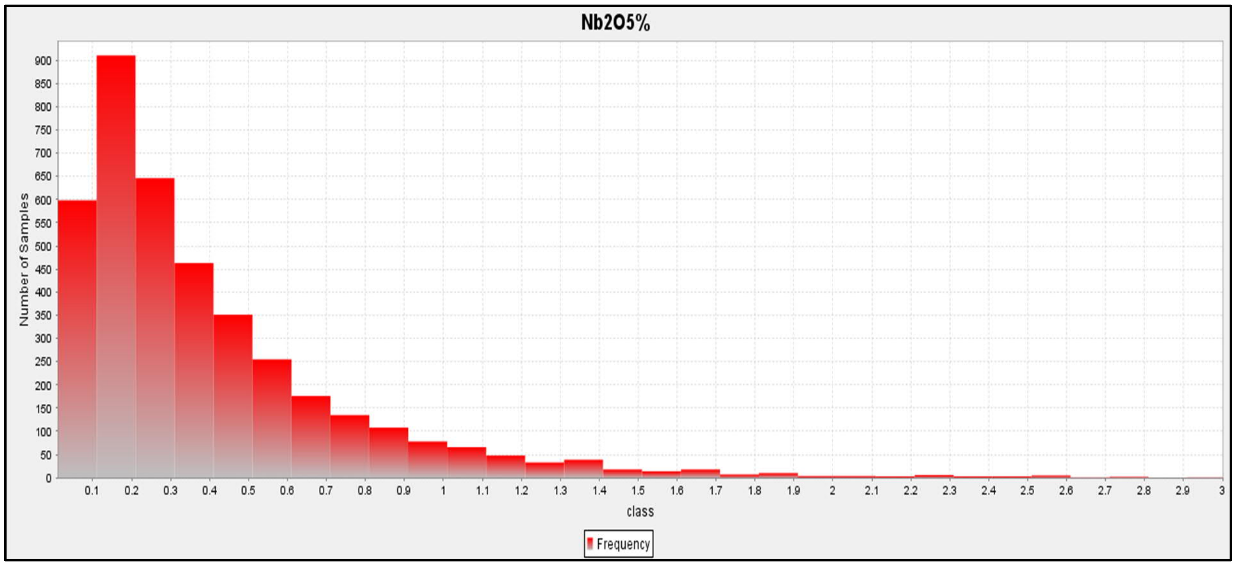


Figure 14-9: Mineralized Residuum Niobium Sample Histogram

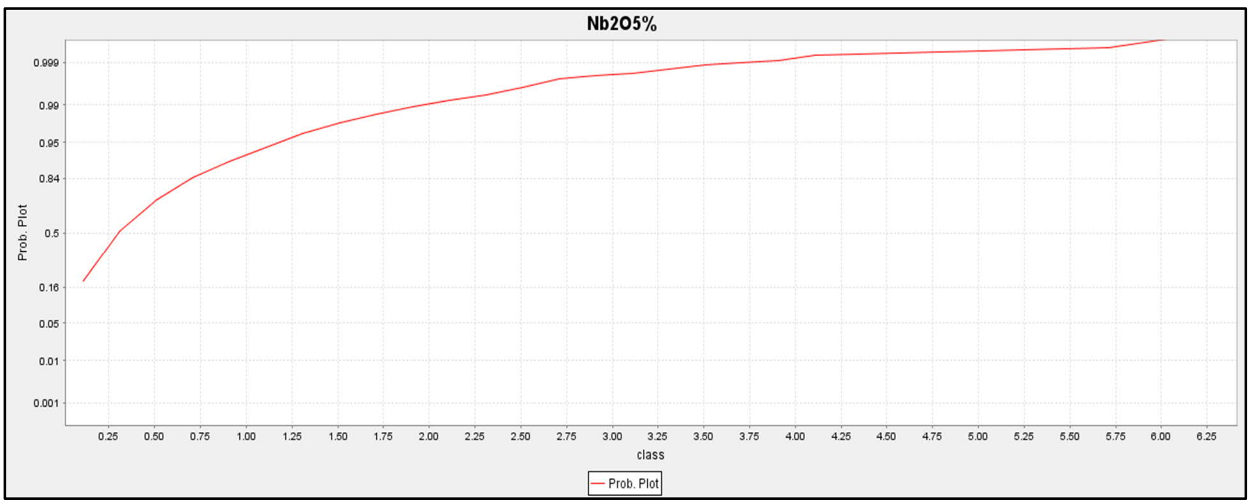


Figure 14-10: Mineralized Residuum Niobium Probability Plot

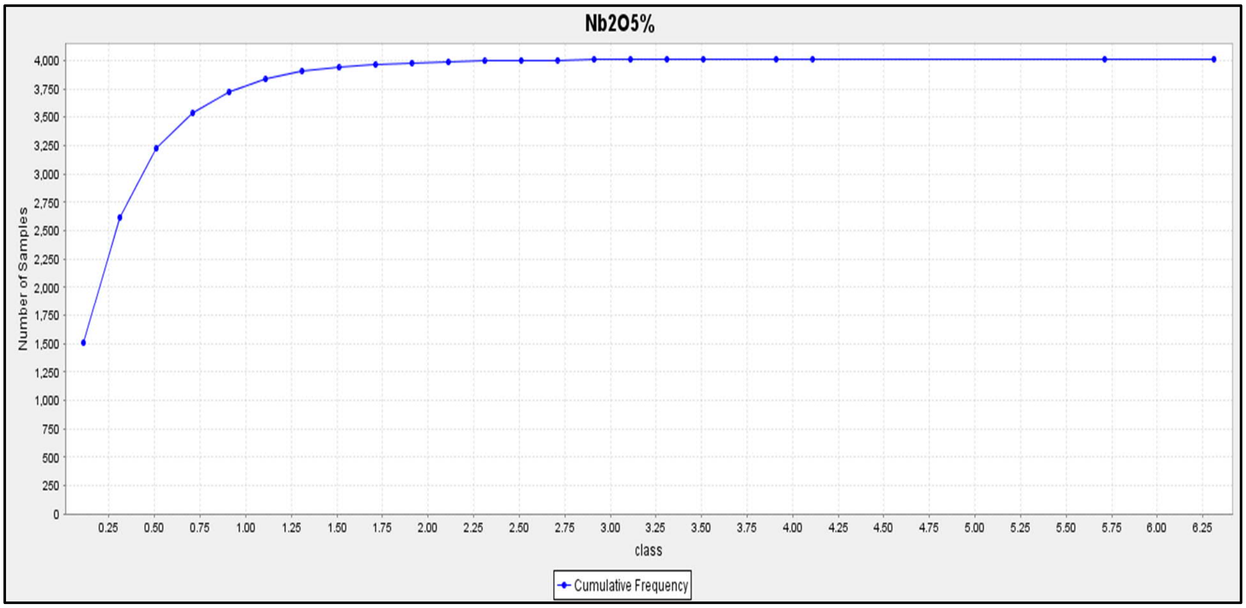


Figure 14-11: Mineralized Residuum Niobium Cumulative Frequency

Histogram, probability and cumulative frequency plots of the phosphate assays within the mineralised residuum wireframes are presented in Figure 14-12, Figure 14-13 and Figure 14-14 respectively.

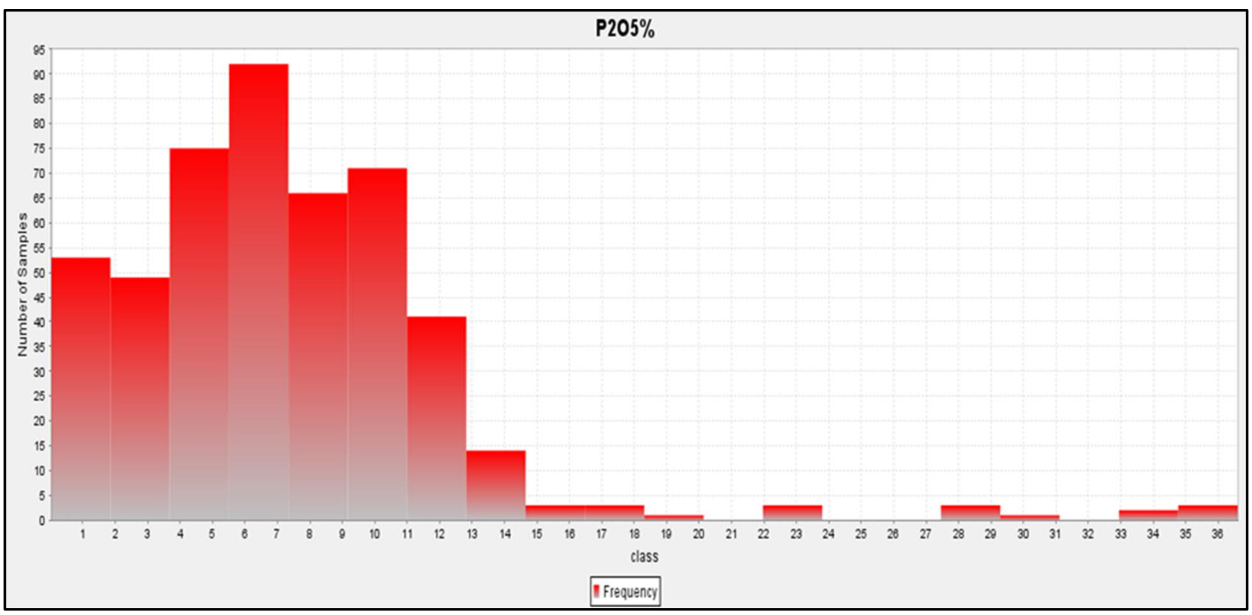


Figure 14-12: Mineralized Lateritic Material – P_2O_5 Sample Histogram

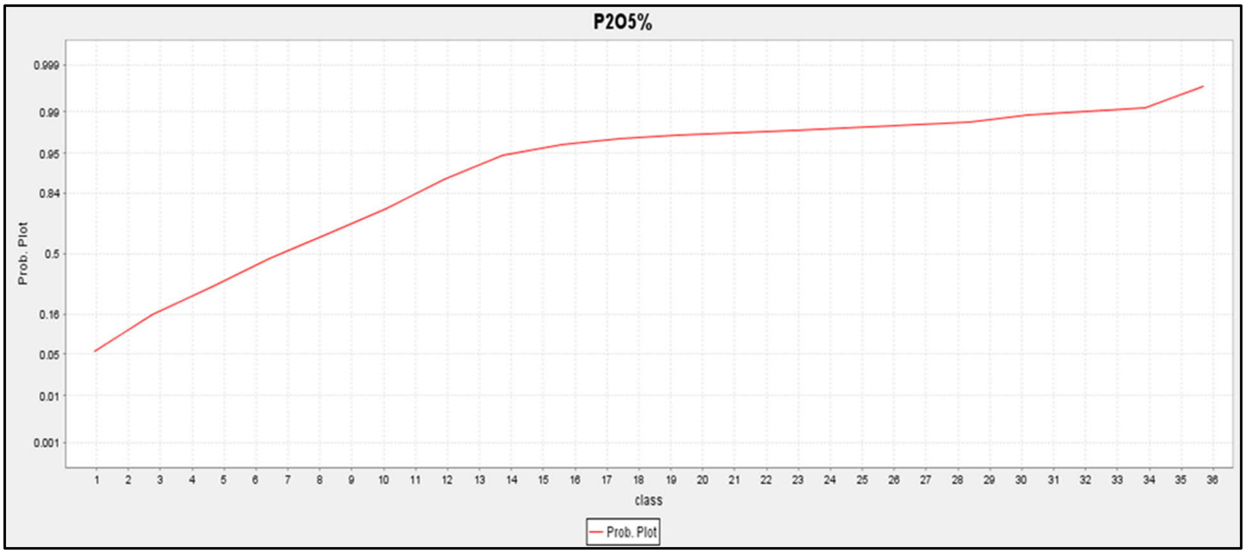


Figure 14-13: Mineralized Lateritic material – P_2O_5 Sample Probability Plot

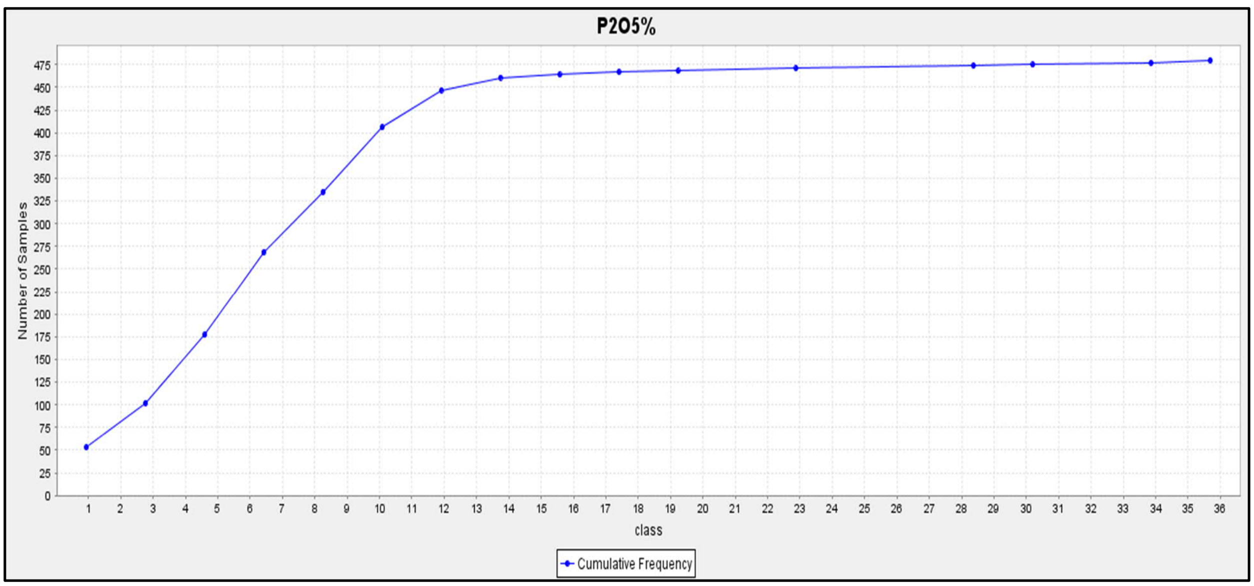


Figure 14-14: Mineralized Lateritic material – P_2O_5 Sample Cumulative Frequency

Table 14-10 summarises the population distribution characteristics and capping levels selected.

Table 14-10: Population Distribution Characteristics and Capping Levels

	Mineralised Lateritic material		Mineralised Residuum	
	P_2O_5%	**Nb_2O_5%**	**P_2O_5%**	**Nb_2O_5%**
Break in Histogram	15%	3.2, 3.4 or 3.8%	None	2.0
Cumulative Frequency Curve Flatten	14%	3.1%	38%	1.7
95% Probability	14%	3.3%	36%	1.2
99% Probability	31%	5.0%	38%	2.0
Capping Level	**15%**	**3.2%**	**None**	**2.0**
No. Capped Assays	200 (42%)	24 (5%)	0	127 (3%)

No bias was observed between phosphate grade and sample length in the Anomaly A Residuum mineralized lens, see Figure 14-15 and Figure 14-16 below.

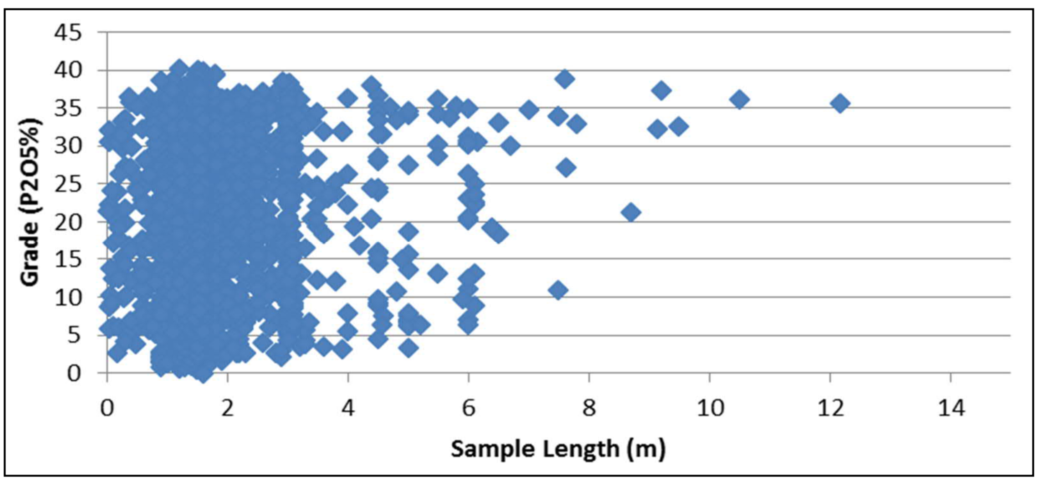


Figure 14-15: Anomaly A Phosphate Mineralized Lens Grade v Length (6% P_2O_5)

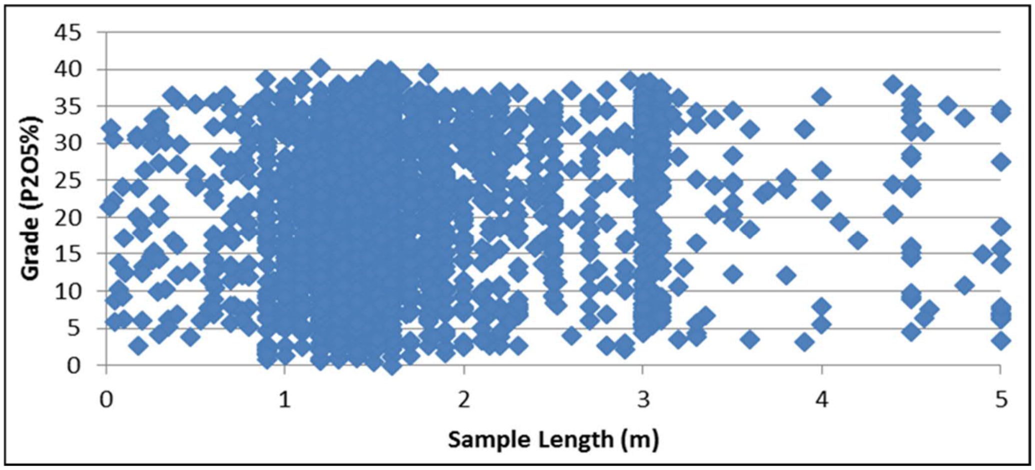


Figure 14-16: Anomaly A Phosphate Mineralized Lens Grade v Length ($6\%\ P_2O_5$) – enlarged

14.4.6 Assay Interval Compositing

A 2 m down-the-hole ("DTH") sample composite length was selected to reduce the variability of the data and to maintain geological definition of the mineralization. The minimum composite size used in the resource estimate was 1.5 m (75%).

Orphan, short composites less than 1.5 m long were removed from the database. The average grade of these discarded short composites was examined in comparison to the average grade of the interpolated composites (minus those removed) to ensure that removing these samples did not introduce a grade bias.

Most of the sample lengths range between 1.4 m and 1.7 m, with the majority taken at 1.5 m. Given this distribution and the thickness of the mineralization, 2 m composite lengths were chosen. Assays within the mineralization domains were composited starting at the first mineralized wireframe boundary from the drillhole collar and resetting at each new wireframe boundary. The mineralized residuum and lateritic material composite statistics are presented in Table 14-11.

Table 14-11: Anomaly A – Two metre (2 m) Composite Descriptive Statistics by Domain

Parameter	Mineralised Residuum	
	Total	
	P_2O_5	Nb_2O_5
No. of Composites	3,708	3,532
Mean P_2O_5% Grade	21.73	0.42
Median P_2O_5% Grade	23.10	0.31
Min. P_2O_5% Grade	0.30	0.01
Max. P_2O_5% Grade	39.70	4.74
P_2O_5% Standard Deviation	9.87	0.39
P_2O_5% Coeff. Of Variation	0.45	0.92
	Mineralized Lateritic Material	
No. of Composites	444	444
Mean P_2O_5% Grade	7.12	1.00
Median P_2O_5% Grade	5.92	0.69
Min. P_2O_5% Grade	0.15	0.08
Max. P_2O_5% Grade	36.60	6.19
P_2O_5% Standard Deviation	6.18	0.99
P_2O_5% Coeff. Of Variation	0.87	0.98

14.4.7 Grade Trend Analysis

Phosphate grade and thickness contours were developed in Anomaly A in order to evaluate any trends as a geological composite such that all of the samples from one drillhole as intersected by the wireframe are composited into one value for that intersection. A minor high-grade trend was observed in the direction of bedrock trough/thickness as depicted in Figure 14-17. A minor trend was also observed in the thickness, corresponding with a depression/trough in the bedrock (which may be related to a regional structure in the basement), see Figure 14-18.

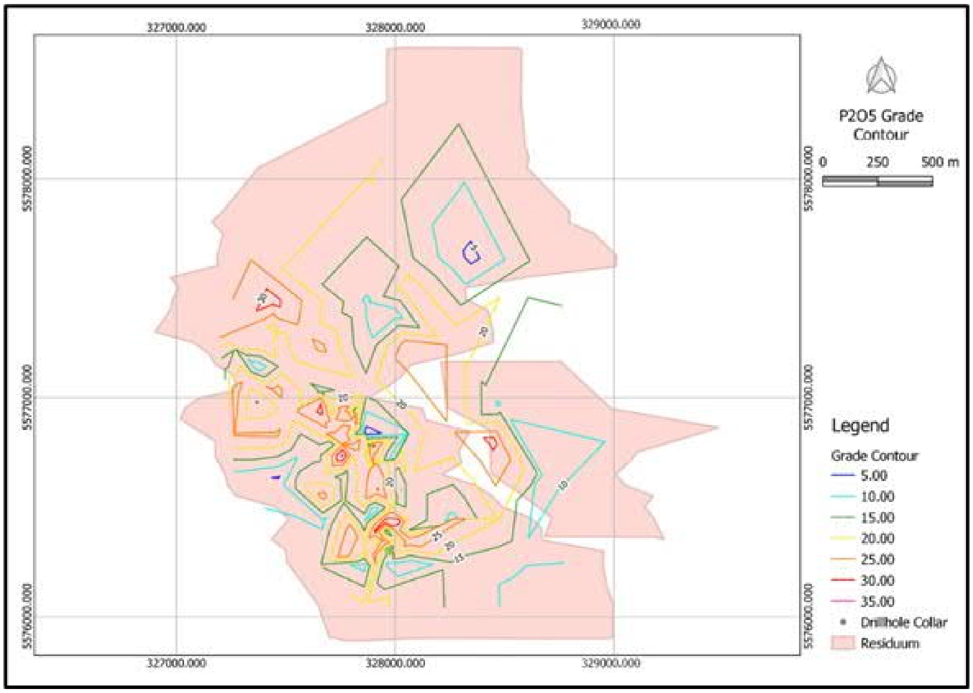


Figure 14-17: Martison Anomaly A – P_2O_5 Grade Contours

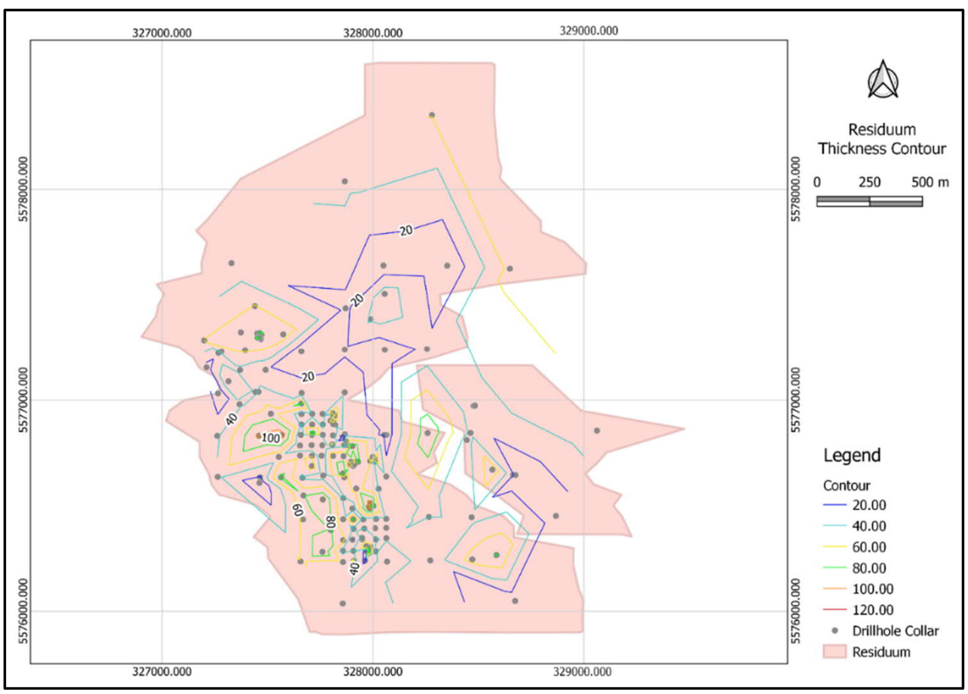


Figure 14-18: Martison Anomaly A Residuum Thickness Contours

14.4.8 *Mineralization Continuity & Variography*

Phosphate mineralization continuity and variography for Anomaly A residuum and niobium mineralization in the lateritic material was analysed using Geovia Surpac and was based on the 2 m DTH composite data. Table 14-12 summarizes the parameters used and derived. The experimental directional variograms were modelled using a spherical model.

Table 14-12: Variography Summary

Parameter	Anomaly A Residuum Phosphate Domain	Lateritic material Niobium Domain
No. Composites	3,708	308
Bearing	120º	130º
Plunge	0º	0º
Dip	0º	0º
Nugget	0.22	0.15
Sill	0.63	0.43
Range	196	120
Major: Semi-Major Ratio	2	1
Major: Minor Ratio	2	2

The phosphate in the residuum has an anisotropic (ellipsoid) search radius with a major/semi-major ratio of 2, a semi-major/minor ratio of 2, and a search radius of 200 m for the Indicated category resources and 250 m for the Inferred category resources. The phosphate variography parameters were assumed to be indicative of the other elements (including niobium) in the residuum.

The Nb_2O_5 in the lateritic material has an anisotropic (ellipsoid) search radius with a major/semi-major ratio of 2, a semi-major/minor ratio of 2 and a search radius of 200 m for the Indicated category resources, and 250 m for the Inferred category resources. This has a nugget effect of 0.22 which has been taken as indicative of the deposit as a whole.

14.5 Block Model Definition

Distinction was not made at the geological wireframe stage between the different lithotypes of residuum noted during logging (2A unconsolidated residuum, 2B re-cemented residuum and 2C transitional residuum).

An additional parameter was introduced as part of the revision of the block model in this 2021 report to establish sub-lithotype definition further, principally for the purposes of identifying variability in residuum geochemistry for downstream processing studies. These parameters are based on the $CaO:P_2O_5$ (CPRAT); as shown in the Table 14-13.

Table 14-13: Material Description (CPRAT)

Material Description
"Unconsolidated" Residuum, (<2.0 or >0.9 CPRAT, or 12%-22% P_2O_5 % grade)
"Consolidated" Residuum (<2.0 or >0.9 CPRAT, or >22% P_2O_5 % grade)
Residuum (<0.9, CPRAT, or 12%-22% P_2O_5 % grade)
Residuum (<0.9 CPRAT, or >22% P_2O_5 % grade)
Residuum (>2.0 CPRAT, or 12%-22% P_2O_5 % grade)
Residuum (>2.0 CPRAT, or >22% P_2O_5 % grade)
Residuum (<12% P_2O_5 % grade)

In areas where CaO assays are missing and, consequently blocks are not populated with CaO values, estimated CaO values were used to calculate CPRAT. The corresponding formulas, derived from the existing assay data (3885 core samples with CaO assays in total) as a function of P_2O_5 content are shown in Table 14-14.

Table 14-14: Calculated CPRAT formulae

Cut-off grade	Equation
none	$\%CaO = 12.2 + 0.83(\%P_2O_5)$
6.0% P_2O_5	$\%CaO = 1.29(\%P_2O_5)$
14.0% P_2O_5	$\%CaO = 1.28(\%P_2O_5)$

The Martison Anomaly A block model incorporates an area of 7,000 m (easting) by 6,000 m (northing) by 320 m (height) in 10 m x 10 m x 10 m blocks and sub-block at 5 m x 5 m x 5 m. The model was not inclined or rotated. Table 14-15 summarizes the block model parameters, extents, and dimensions. The block model extents have increased compared to the 2015 MRE to allow for the inclusion of additional data not applied in 2015.

Table 14-15: Block Model Parameters

Parameter		Value
Block Model Origin	Easting	325,000
	Northing	5,574,000
	Z	-100
Parent Block Size (m)	X	10
	Y	10
	Z	10
Minimum Block Size (m)	X	5
	Y	5
	Z	5
Bearing	0	
Plunge	0	
Dip	0	

14.5.1 Interpolation Search Parameters and Grade Interpolation

Grade interpolation was performed on a parent block basis by using ID^3. The search orientations and ellipsoids are presented in Table 14-16. A minimum number of three samples and a maximum number of 15 samples were used to estimate block grades for the Indicated resource. A minimum number of two samples and a maximum number of 15 samples were used to estimate block grades for the Inferred resource.

A search radius of 200 m for Indicated, and 250 m for Inferred, category resources was used. A maximum vertical search distance of 16 m (derived from the variogram range) was applied in a search ellipse bearing 120º, not dipping or plunging with a major/semi-major ratio of 2, a semi-major/minor ratio of 2 for the first pass. The search radius and vertical search distance was increased to 400 m and 32 m in the second pass and 600 m and 48 m in the third pass to ensure all blocks within the mineralized wireframes were populated.

In the lateritic material, a search radius of 120 m (200 m for Inferred category resources) and a maximum vertical search distance of 42 m were applied in a search ellipse bearing 130º, not dipping or plunging with a major/semi-major ratio of 1, a semi-major/minor ratio of 2 for the first pass. The search radius was increased to 400 m in a second pass to ensure all blocks within the mineralized wireframes were populated. The vertical search distance was not increased in the second pass as 42 m was deemed to be sufficient.

The grade interpolation respected individual lens boundaries as “hard boundaries” to prevent the influence of assays in different domains and possible subsequent smearing.

Table 14-16: Block Model Search Passes

Parameter	Mineralised Residuum		Mineralised Lateritic material	
	Search Radius m	Vertical Search Distance m	Search Radius m	Vertical Search Distance m
1st Pass	200 (250)	16	120 (200)	42
2nd Pass	400	32	240	42
3rd Pass	600	48	360	42

14.6 Mineral Resource Classification

The definitions for resource categories used in this report are consistent with CIM definitions. Under the CIM classification system, a Mineral Resource is defined as:

"…a concentration or occurrence of natural, solid, inorganic or fossilised organic material in or on the Earth's crust in such form and quantity and of such grade or quality that it has reasonable prospects for economic extraction.

"The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

Resources are classified into Measured, Indicated and Inferred categories based upon increasing geological confidence.

Resource classification within mineralization envelopes is generally based on drillhole spacing, grade continuity, and overall geological continuity. The distance to the nearest composite and the number of drillholes are also considered in the classification.

For this revised geological model and mineral resource estimate, the same protocols as were used in the 2015 MRE have been followed, with some modifications to the search and input parameters in response to the geochemical criteria for the downstream processing requirements.

The following sets out the broad criteria used for Mineral Resource classification:

- A block must fall within the Whittle pit shell, below the topography:
  - within the 6% P_2O_5 mineralised wireframes.
  - over the 6% P_2O_5 cut–off grade.
  - within the lateritic material wireframe and over the 0.2% Nb_2O_5 cut–off grade.
  - an Indicated Mineral Resource block must be within 200 m of an informing sample.
  - in a continuous area of blocks within 200 m of a sample.
  - an Inferred Mineral Resource must be within 250 m of an informing sample.
- No part of the Anomaly A Mineral Resource has been classified as Measured due to:
  - the uncertainty of the assumed bulk densities used.
  - the preliminary level of data regarding moisture content; and possible over-compression of the bulk test samples, as they will directly affect volume/tonnage conversion factors and will have implications for mine planning.
  - the paucity of drillhole intersections into the carbonatite basement bedrock, which further defines the deposit limits.

Anomaly B is classed as a target for exploration as the quantity and grade is conceptual in nature and there has been insufficient exploration to define a Mineral Resource.

14.6.1 Conceptual Pit Shell Constraints

A set of technical (geochemical) and economic input constraints were applied to the block model.

A preliminary open pit shell was used with a cut-off grade of 6% P_2O_5 in order to constrain the Mineral Resources (mineralized from unmineralized material) that demonstrate reasonable prospects for eventual economic extraction.

The preliminary technical and economic input assumptions for the block model are summarized in Table 14-17. These were applied to create a preliminary open pit shell, using Hexagons MinePlan software, in order to constrain the estimated Mineral Resources to input parameters that demonstrate reasonable prospects for eventual economic extraction.

The resulting economic pit shell is shown in Figure 14-19.

Table 14-17: Preliminary Technical & Economic Assumptions

Input Parameter	Units	P_2O_5	Nb_2O_5
Pit Wall Slopes	Degrees	23	23
Mining Cost	USD/t	2.5	2.5
Mining Extraction*	%	100	100
Mining Dilution*	%	0	0
Processing and G&A Costs	USD/t	9	9
Processing Recovery	%	70	20***
Final Product Concentration	%	100	50
Price	USD/t	475**	30,000

*Mining dilution and mining extraction were used as assumptions but **not** as inputs to calculate the cut-off grade or to report Mineral Resources.

**A price of USD 475/t P_2O_5 is based on a 100% final product concentrate using a base price of USD153 for a 32% P_2O_5 product.

***Nb recovery based on reported metallurgical test data.

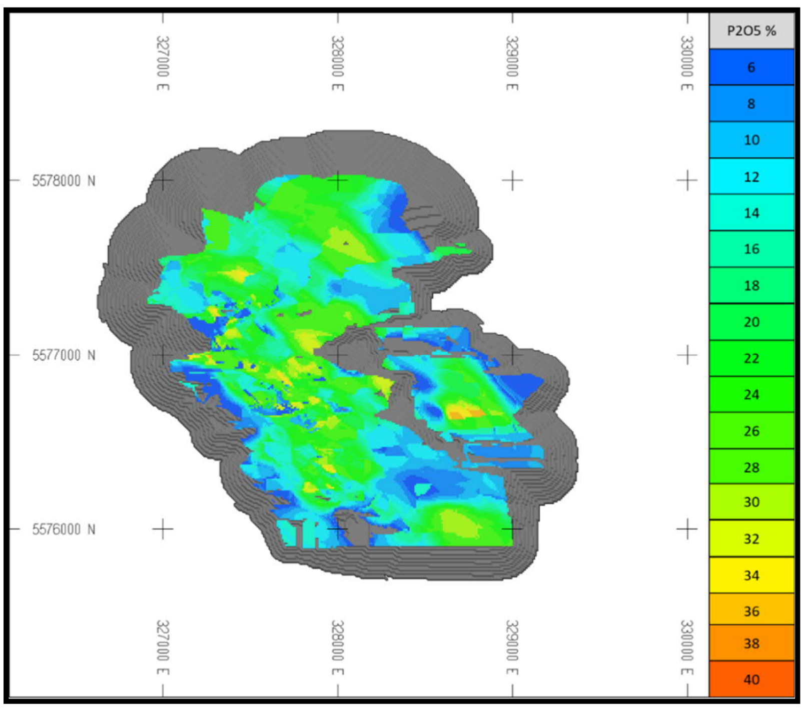


Figure 14-19: P_2O_5 Variations Within Economic Pit Shell

14.6.2 *Cut-off Grade Assumptions*

The preliminary open pit shell provides a constraint for the reported open pit resources based on the CIM definition of Mineral Resources for "reasonable prospects for economic extraction" and resulted in an open pit cut-off grade of 6% P_2O_5 being estimated.

The portions of the wireframes that fell within the open pit shell and below overburden ('Glacial Till' lithotype) were considered to have reasonable prospects for economic extraction, thereby qualifying as a Mineral Resource.

Table 14-18 and Table 14-19 illustrate the Mineral Resource sensitivity by cut-off grade (as constrained by the pit shell) for the mineralized residuum and lateritic material respectively.

Table 14-18: Mineralized Residuum Sensitivity By Cut-Off Grade

Cut-Off Grade (% P_2O_5)	Tonnage (Mt)	Phosphate Grade (% P_2O_5)	Niobium Grade (% Nb_2O_5)
6	182	18.83	0.42
8	173	19.45	0.42
10	158	20.41	0.43
12	140	21.65	0.44
14	122	22.91	0.45

Table 14-19: Mineralized Lateritic Material Sensitivity By Cut-Off Grade

Cut-Off Grade (% Nb_2O_5)	Tonnage (Mt)	Niobium Grade (% Nb_2O_5)	Phosphate Grade (% P_2O_5)
0.2	12	0.93	7.25
0.4	10	1.06	7.63
0.6	7	1.28	7.69
0.8	5	1.50	7.51
1.0	4	1.67	7.28

14.6.3 *Model Validation*

A number of block model validation procedures were carried out including:

- Volume comparisons.
- Statistical comparisons.
- Visual comparisons of block grades versus composite grades.
- Comparing individual block and composite grades.

Differences between the wireframe and block model volumes were small enough to be considered insignificant (Table 14-20). The small differences in wireframe and block model volumes may be due to the minimum sub-blocking size.

Table 14-20: Volume Comparison

	Anomaly A Mineralized Residuum	Mineralized Lateritic Material
Wireframe Volume (m^3)	131,691,098	6,995,575
Block Model Volume (m^3)	131,662,000	7,000,625
% Difference	0.03%	0.08%

A comparison of the descriptive statistics for the raw assays, composites samples, and blocks for the lenses modelled are summarized in Table 14-21. It can be seen that the mean grades between the raw assays, composites, and block grades match reasonably well, suggesting that there is little bias present in the estimate.

Model validation has also visually compared drillhole sample grades, supporting composite grades and block model grades Figure 14-20 and Figure 14-21. Overall, good correlation was found in both vertical sections and plans. All blocks within the domains have been filled with grade, bulk density, number of informing samples and distance to nearest informing sample. The blocks are also coded as to whether they are reported within the Whittle Pit and whether they lie above or below the topography and rock type.

On the basis of its review and validation procedures, the opinion of the QP is that the block model is valid and acceptable for estimating the Mineral Resources.

Table 14-21: Composite and Block Model Comparison

Parameter	Assay	Composite	Block Model
Anomaly A Mineralised Residuum Domain			
Mean P_2O_5% Grade	21.62	21.73	22.34
Median P_2O_5% Grade	22.80	23.10	22.94
Min. Grade P_2O_5% Grade	0.02	0.30	1.60
Max. Grade P_2O_5% Grade	40.20	39.70	38.30
P_2O_5% Standard Deviation	10.35	9.87	7.32
P_2O_5% Co-eff. Of Variation	0.48	0.45	0.33
Anomaly A Mineralised Lateritic Material Domain			
Mean Nb_2O_5% Grade	1.02	1.00	1.10
Median Nb_2O_5% Grade	0.67	0.69	0.94
Min. Nb_2O_5% Grade	0.01	0.08	0.04
Max. Nb_2O_5% Grade	6.24	6.19	2.85
Nb_2O_5% Standard Deviation	1.02	0.99	0.59
Nb_2O_5% Co-eff. Of Variation	1.00	0.98	0.53

14.6.4 Surpac v MinePlan; Reconciliation of MRE

The generated Surpac 3D model, and the block model input parameters were transferred to MinePlan block modelling software.

The grades and tonnages for both P_2O_5 and Nb were confirmed in MinePlan to have no significant variation from the resource estimate generated in Surpac (Table 14-22).

Table 14-22: Surpac v MinePlan Block Model Reconciliation

		Surpac Model			**MinePlan Model**			**Deviation**		
Deposit	Classification	Tonnes (Mt)	Phosphate P_2O_5 (%)	Niobium Nb_2O_5 (%)	Tonnes (Mt)	Phosphate P_2O_5 (%)	Niobium Nb_2O_5 (%)	Tonnes (Mt)	Phosphate P_2O_5 (%)	Niobium Nb_2O_5 (%)
Anomaly A Residuum	Indicated	53.8	22.99	0.42	53.7	22.99	0.42	0.1	0.00%	0.00%
	Inferred	128.3	17.09	0.42	127.7	17.1	0.42	0.3	0.01%	0.00%
Anomaly A Lateritic Material	Indicated	6.2	7.97	1.13	6.3	8.04	1.13	0.1	0.07%	0.00%
	Inferred	5.3	6.4	0.69	5.6	6.08	0.66	0.3	0.32%	0.03%

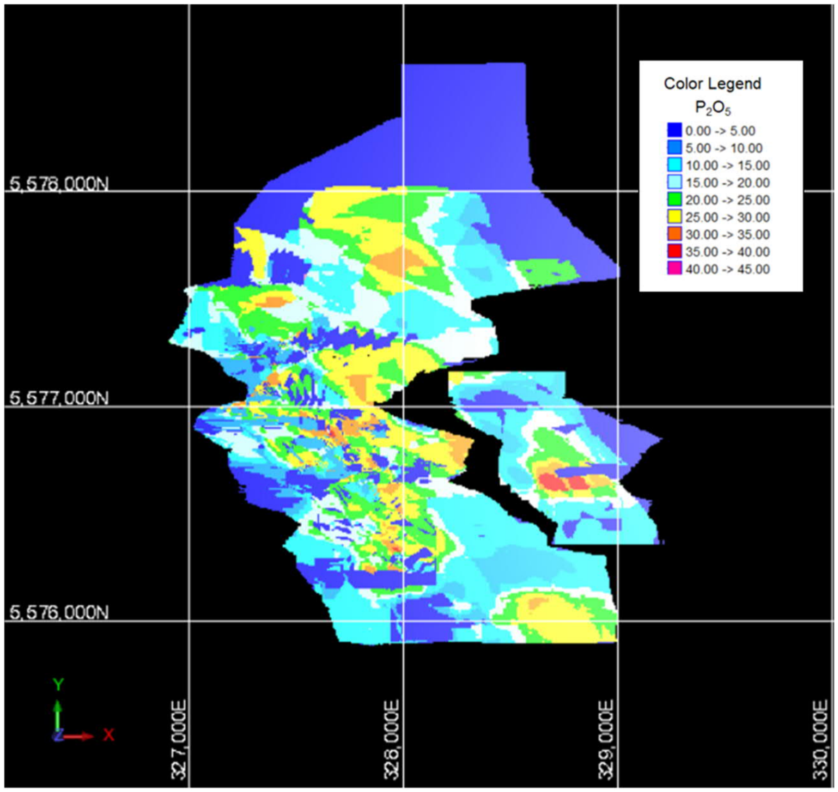


Figure 14-20: Anomaly A P_2O_5 Grade Model

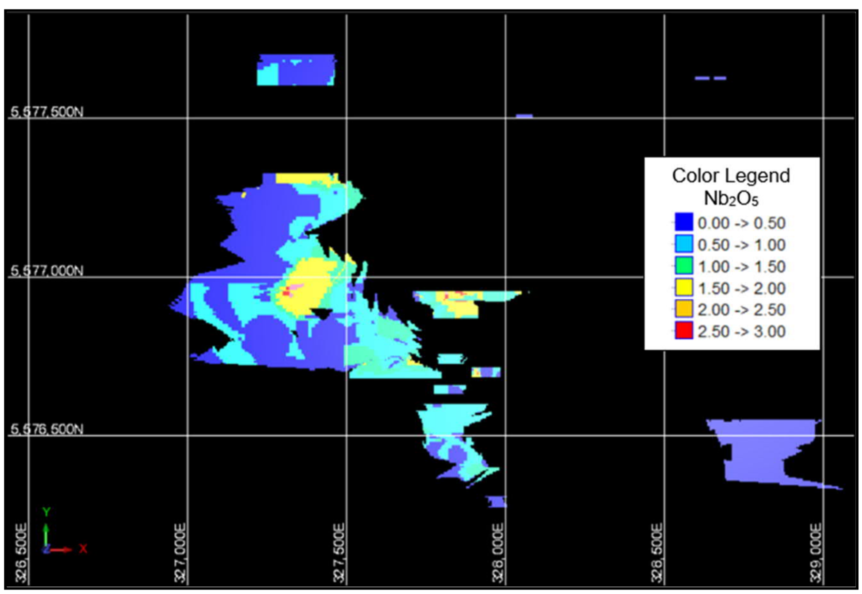


Figure 14-21: Anomaly A Nb_2O_5 Grade Model

14.7 Mineral Resource Statement

The Mineral Resource estimate for the Martison Project is with a drillhole database cut-off date of December 31, 2021.

The Mineral Resource estimate has an effective date of December 31, 2021 and has an issue date of January 31, 2022.

Table 14-23 summarises the Mineral Resource Estimate for the Martison Project as of January 31, 2022.

Anomaly A Residuum contains as estimated 53.8 Mt of Indicated Mineral Resources at a grade of 22.99.4% P_2O_5 and 0.42% Nb_2O_5, and 128.3 Mt of Inferred Mineral Resources at a grade of 17.09% P_2O_5 and 0.42% Nb_2O_5. The residuum resources are reported at a cut-off grade of 6% P_2O_5.

The lateritic material contains 6.2 Mt of Indicated Resources at a grade of 7.97% P_2O_5 and 1.13% Nb_2O_5 and 5.3 Mt of Inferred Resources at a grade of 6.4% P_2O_5 and 0.69% Nb_2O_5.

The lateritic material resources are reported at a cut-off grade of 0.2% Nb_2O_5.

Table 14-23: Martison Mineral Resource Estimate as of December 31, 2021

Deposit	Resource Classification	Tonnes (Mt)	Dry Bulk Density (t/m3)	Phosphate Grade (P_2O_5 %)	Niobium Grade (Nb_2O_5 %)
Anomaly A Residuum	Indicated	53.7	1.90	22.99	0.42
	Inferred	128.3		17.09	0.42
Anomaly A Lateritic material	Indicated	6.2	1.70	7.97	1.13
	Inferred	5.3		6.40	0.69

Notes:

1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a cut-off grade of 6% P_2O_5 in the Residuum or 0.2% Nb_2O_5 in the Lateritic material.
3. Phosphate Mineral Resources are estimated using a price of USD475 per tonne (basis 100% P_2O_5).
4. Niobium Mineral Resources are estimated using a price of USD30 per kilogramme (65% Nb_2O_5 concentrate).
5. Mineral Resources are constrained by an economic pit shell.
6. A minimum mineralisation width of five metres was used for Indicated resources and two metres for Inferred resources.
7. Numbers may not add up due to rounding.

Cautionary Note: Mineral resources that are not mineral reserves do not have demonstrated economic viability. The PEA includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. The foregoing mineral resource estimates are as of December 31, 2021.

The ID^3 method, paired with the 5 m sub-block size at the boundaries gave a high level of resolution in the Resource estimate and ensured that the internal waste remained distinct.

The Martison Mineral Resource Estimate is based on the following criteria:

- Mineralized residuum blocks above a 6% phosphate cut-off grade.

- Mineralized lateritic material blocks above a 0.2% Nb_2O_5 cut-off grade.
- Above the Mineral Resource pit shell, below the topography.
- Three informing samples are applied for Indicated resources and two informing samples for Inferred resources.
- Within 200 (Indicated) – 250 m (Inferred) of an informing sample.

Anomaly B, as an exploration target, has been estimated from the volume of the residuum lithological wireframe of the residuum at a bulk density of 1.9 t/m^3 and an average grade of 18% P_2O_5 (derived from the assays within the residuum). It represents an exploration potential of 35 Mt to 70 Mt containing 16% - 20% P_2O_5 and is summarized in Table 14-24 below.

Table 14-24: Martison Exploration Potential as of November 30, 2014

Deposit	Classification	Tonnes (Mt)	Bulk Density t/m^3	Grade (P_2O_5 %)
Anomaly B	Exploration Target	35 – 70	1.91 (dry)	16 – 20

The quantity and grade are conceptual in nature and there has been insufficient exploration to define a Mineral Resource and it is uncertain if further exploration will result in a change to in classification status for this anomaly. It does not represent a Mineral Resource at present on account of a lack of information to demonstrate economic viability and is disclosed with a potential quantity and grade, expressed as ranges, that is to be the target of future exploration.

15. Mineral Reserve Estimates

This section is not applicable to this report. There are no Mineral Reserves to be declared at this stage, per CIM definitions.

16. Mining Methods

16.1 Summary

This section details this Technical Report's proposed open pit mining operation, focusing exclusively on the future mining of the Anomaly A deposit. Based on information currently available, Anomaly A is the only of the three known aeromagnetic anomalies defined to the extent where a mining plan can be developed.

Both Indicated and Inferred resources have been used in the mine planning, though the latter are considered too speculative geologically to have the economic considerations applied to them. Consequently, further exploration will be required to confirm the results of this Technical Report based on the combination of these two resource classifications.

Conventional open pit mining methods are planned for Anomaly A to produce a phosphate concentrate for further processing. The open pit mining operation was designed to provide sufficient feed to the beneficiation plant (subsequently referred to in this section as “mill feed”) and will produce a steady state containing 500 ktpa of P_2O_5 products (i.e., MAP, NPS and SPA).

Mine planning for the Project is based on a conventional truck and shovel mining method, which aims to strip overburden, waste, lateritic and mill feed materials from Anomaly A. Materials mined from Anomaly A will be directed to the following destinations for deposition and potential rehandling:

- Beneficiation plant – Mill feed materials.
- Waste facility – Waste materials with overlying muskeg and mine organics stored separately and within the footprint of the waste facility for ease of future reclamation.
- Pit backfill facility – Waste materials (to be located within the pit shell).
- Niobium stockpile – Niobium-rich lateritic materials.
- Tailings management facility – Waste materials (i.e., glacial till for berm construction).

The general site plan illustrating the location of all these areas is shown in Figure 16-1.

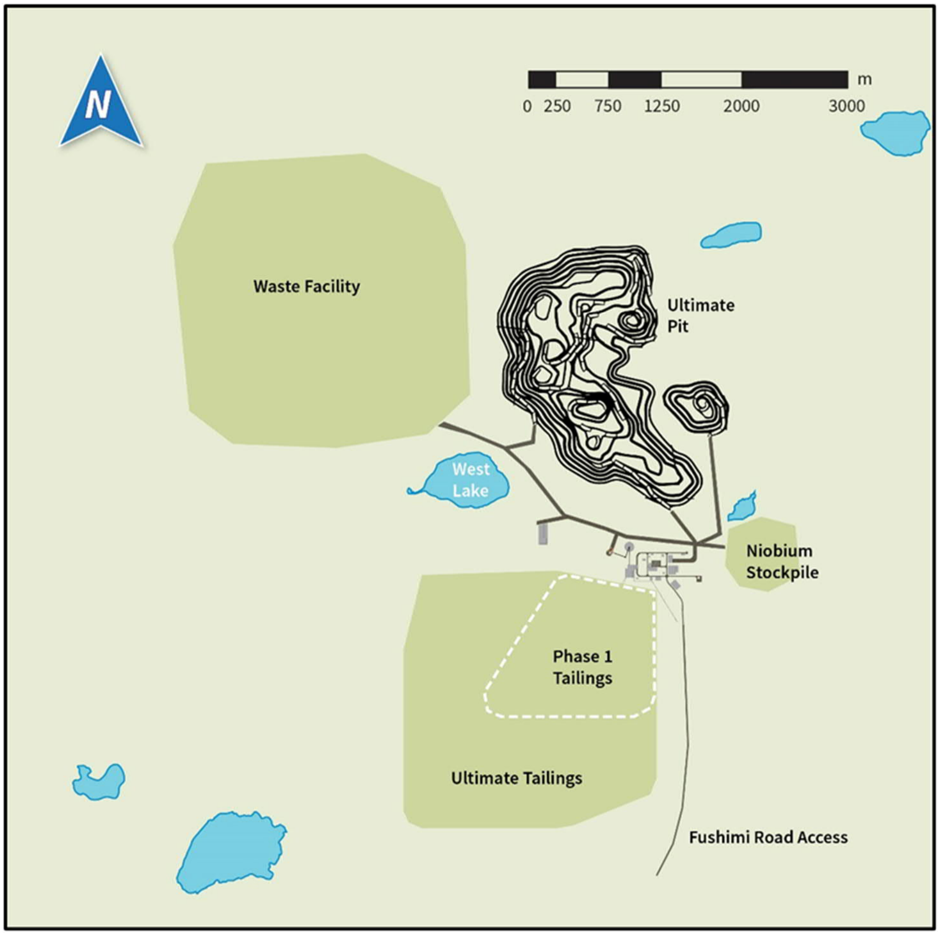


Figure 16-1: Martison Mine Site – General Layout

Initial site clearing and grubbing for the open pit and all infrastructure facilities will be performed by a qualified contractor capable of determining the most effective stripping methodology for this type of environment. Site preparation activities will focus on the removal of trees, the installation of perimeter berms to prevent water ingress into the working areas, the excavation of muskeg and backfilling with additional structural fill where necessary.

In addition to site preparation, this pre-production period will include the excavation of the first 20 m box cut of the starter pit. A transition to an owner-operated site is planned to take place during the first year of production mining once all site preparation and pre-production activities have been completed.

The Carbonatite Complex materials classified as overburden, bedrock, and lateritic material will be deposited in the designated Waste Facility. Furthermore, residuum materials which are low grade (sub-economic) in P_2O_5 and Nb_2O_5 will also be treated as mine waste and will be deposited at the same location. Waste material will also be backfilled into the pit towards the end of the LOM. All residuum material high in P_2O_5 grade, as well as satisfying all processing criteria, will be classified as mill feed, and be sent to the beneficiation plant for processing into a concentrate slurry.

The lateritic material within Anomaly A, which contains Nb_2O_5 grade above 0.5%, will be mined selectively and stockpiled in the Niobium Stockpile near the beneficiation plant. This will allow for ease of rehandling, should a viable means of economically processing this material be implemented in the future.

The summary of all material within the modeling of the open pit operation is shown below in Table 16-1.

Table 16-1: Martison Ultimate Pit – Diluted Material Inventory

Material / Resource Categories	Cut-Off Grades	Dry Tonnage	Moisture Content	aP_2O_5 Grade	CaO Grade
	(%)	Mt	%	%	%
Mill Feed Material (Indicated Resource)	aP_2O_5 > 11.0	45.52	14.06	21.86	32.04
Mill Feed Material (Inferred Resource)	aP_2O_5 > 11.0	38.09	14.15	21.02	28.33
Total Mill Feed Material	aP_2O_5 > 11.0	83.61	14.10	21.48	30.35
Niobium-Rich Lateritic Material	Nb_2O_5 > 0.5	6.59	15.00	0.00	5.36
Waste Material	-	319.28	10.22	0.66	2.38

Notes:
1. Material inventory reported from re-blocked, diluted mining model.
2. Mill feed refers to mineralized material that will be sent to the beneficiation plant for processing and includes inferred resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves.
3. Moisture content represents 100 x moisture mass / dry mass.
4. aP_2O_5 grade is the % P_2O_5 derived from the primary phosphate mineral apatite and excludes P_2O_5 derived from secondary phosphate minerals.

16.2 Mine Design Criteria

16.2.1 Geological Resource Model

A three-dimensional resource block model was provided by DMT and converted to a HxGN's MinePlan format for the purposes of interrogating and re-blocking the model for mine planning purposes. The coordinate system, extents and attributes criteria used in the modeling of the open pit are shown below in Table 16-2 and Table 16-3 respectively.

Table 16-2: Geological Block Model Extents (UTM NAD 83)

Parameter	X (Easting)	Y (Northing)	Z (Elevation)
Parent Block Size (m)	10	10	10
Minimum Coordinates (m)	326,600	5,575,700	-100
Maximum Coordinates (m)	330,500	5,579,000	220
Number of Parent Blocks (ea)	3,900	3,300	320

Table 16-3: Geological Block Model Attributes

Model Item	Min	Max	Precision	Description
AL_2O_3	0	100	0.001	Decimal item indicating a block's Aluminum Oxide grade value
CAO%	0	100	0.001	Decimal item indicating a block's Calcium Oxide grade value, interpolated from assay data
CE%	0	100	0.001	Decimal item indicating a block's Cerium grade value
EU%	0	100	0.001	Decimal item indicating a block's Europium grade value
FE_2O_3	0	100	0.001	Decimal item indicating a block's Ferric Oxide grade value
LA%	0	100	0.001	Decimal item indicating a block's Lanthanum grade value
MGO%	0	100	0.001	Decimal item indicating a block's Magnesium Oxide grade value
NB_2O_5	0	100	0.001	Decimal item indicating a block's Niobium Pentoxide grade value
ND%	0	100	0.001	Decimal item indicating a block's Neodymium grade value
P_2O_5%	0	100	0.001	Decimal item indicating a block's Phosphorus Pentoxide grade value
PR%	0	100	0.001	Decimal item indicating a block's Praseodymium grade value
ROCK	0	100	1	Integer item denoting a block's material categorization: 1 = Carbonatite/Bedrock 2 = Residuum 3 = Lateritic Material 4 = Overburden

Model Item	Min	Max	Precision	Description
CLASS	0	100	1	Integer item denoting a block's mineral resource classification: 0 = Unclassified 1 = Inferred 2 = Indicated 3 = Measured
CPRAT	0	100	0.01	Decimal item calculating the ratio of Calcium Oxide (CAO%) over Phosphorus Pentoxide
TOPO	0	100	1	Integer item denoting whether a block is above (0) or below the site's topographical surface
BULKD	0	100	0.001	Decimal item indicating a block's dry density value
CAO%I	0	100	0.01	Decimal item indicating a block's Calcium Oxide grade value, derived from a correlation equation (see Section 14.5)
CPRAI	0	100	0.01	Decimal item calculating the ratio of Calcium Oxide (CAO%I) over Phosphorus Pentoxide

16.2.2 Dilution and Mining Recovery

To establish the mining dilution and recovery losses for the mine production schedule, the geological resource (sub-blocked) model provided by DMT was re-blocked or "regularized" to its parent block model dimensions of 10 m x 10 m x 10 m. This converted the resource model into a re-blocked mining model, which was subsequently used for mine planning purposes. The parent block dimensions are considered reasonable Selective Mining Unit (SMU) sizing, adequately simulating mining losses and dilution factors for the purposes of a PEA-level Technical Report.

The model interrogation detail of the entire geological resource (sub-blocked) and the regularized mining models are summarized in Table 16-4 and Table 16-5 respectively. As a check against the re-blocked mining model, and to summarize mill feed mine dilution and recovery/loss factors, a comparison between the two models was completed and is summarized in Table 16-6 below.

Table 16-4: Model Interrogation – Geological Resource (Sub-Blocked Model)

Material Type	Resource Class	Dry Tonnage (Mt)	Volume (M-m^3)	P_2O_5 (%)	Nb_2O_5 (%)
Residuum	Indicated	54.57	28.72	22.73	0.43
	Inferred	147.35	77.55	15.25	0.37
	Unclassified	50.91	26.79	0.19	0.00
Residuum Total		252.83	133.07	13.83	0.31
Overburden Total		671.08	355.07	0.29	0.01
Lateritic Material Total		13.85	8.14	6.34	0.79
Bedrock Total		3,930.23	1,853.88	0.13	0.00
Grand Total		4,867.98	2,350.16	0.88	0.02

Table 16-5: Model Interrogation – Mining (Re-Blocked Model)

Material Type	Resource Class	Dry Tonnage (Mt)	Volume (M-m^3)	P_2O_5 (%)	Nb_2O_5 (%)
Residuum	Indicated	53.11	27.98	22.17	0.43
	Inferred	139.16	73.19	15.29	0.38
	Unclassified	62.87	33.05	2.50	0.05
Residuum Total		255.13	134.23	13.57	0.31
Overburden Total		639.76	338.40	0.20	0.01
Lateritic Material Total		14.29	8.25	5.41	0.68
Bedrock Total		3,958.81	1,869.29	0.16	0.00
Grand Total		4,867.99	2,350.16	0.88	0.02

Table 16-6: Geological Model to Mining Model Comparison

Material Type	Resource Class	Dry Tonnage (%)	Volume (%)	P_2O_5 (%)	Nb_2O_5 (%)
Residuum	Indicated	-2.7%	-2.6%	-2.5%	0.0%
	Inferred	-5.6%	-5.6%	0.3%	2.7%
	Unclassified	23.5%	23.4%	1216%	0.0%
Residuum Total		0.9%	0.9%	-1.9%	0.0%
Overburden Total		-4.7%	-4.7%	-31.0%	0.0%
Lateritic Material Total		3.2%	1.3%	-14.7%	-13.9%
Bedrock Total		0.7%	0.8%	23.1%	0.0%
Grand Total		0.0%	0.0%	0.0%	0.0%

For the purposes of reporting mine production schedule tonnage, volume and grade values, no additional mining dilution and recovery factors are applied to the mining model. A more in-depth evaluation of mining recoveries and dilution factors will need to be conducted in a subsequent study phase.

This future evaluation is to be inclusive of items such as higher-resolution resource modelling with additional geological data, refined mill feed and waste classifications, sources of internal/external waste dilution, optimal equipment sizing, grade control practices and prediction of blast heave, where applicable.

16.2.3 Material Categorization and Cut-off Grades

For mine planning purposes, two types of mill feed categories are delineated for residuum material handling for the beneficiation plant:

- Mill Feed 2A or “Unconsolidated residuum”.
- Mill Feed 2B or “Consolidated residuum”.

Residuum mill feed categorizations approximate historical lithological unit categorizations within Anomaly A. For example, lithological unit 2A is unconsolidated residuum (0.0 to 58.5 m thick) and lithological unit 2B is consolidated or "re-cemented" residuum material (0.0 to 91.6 m thick). Currently, both mill feed categories are based on differing cut-off grades of Apatite Phosphorous Pentoxide (aP_2O_5) grade to distinguish 2A from 2B materials, with all remaining residuum below the lowest aP_2O_5 cut-off grade classified as 2C or waste material (see Table 16-7).

Table 16-7: Residuum Material Categorization

Residuum Material	Description
2A Mill Feed	"Unconsolidated" Residuum (≥ 11% aP_2O_5 % grade, < 25% aP_2O_5 % grade)
2B Mil Feed	"Consolidated" Residuum (≥ 25% aP_2O_5 % grade)
2C Waste	Waste Residuum (< 11% aP_2O_5 % grade)

The major phosphate mineral in Anomaly A is Apatite $Ca_5F(PO_4)_3$, but small quantities of secondary phosphate minerals such as Florencite $CeAl_3(PO_4)_2(OH)_6$ do also exist. Metallurgical testing to date has focused on recovering Apatite and no data exists for recovering secondary phosphate minerals from this deposit. Consequently, the phosphate resources considered in the mine plan are based on Apatite P_2O_5 (aP_2O_5) rather than total P_2O_5, which includes Apatite P_2O_5 and secondary phosphate P_2O_5.

Apatite P_2O_5 grade was selected as the sole means of classifying whether a residuum block could be categorized as mill feed. A cut-off grade value of 11% aP_2O_5 was chosen to satisfy processing criteria for the beneficiation plant and from historical metallurgical test work. The economics do not inform the selection of the aP_2O_5 cut-off grade, however an economic calculation of marginal or break-even cut-off grades will likely lead to lower cut-off grade values than the chosen 11% aP_2O_5, depending on the sensitivity analysis of future phosphate product prices.

Apatite P_2O_5 grade is calculated based on the ratio of CaO over P_2O_5 grades:

- If CaO/ $P_2O_5 \geq 1.315$, $aP_2O_5 = P_2O_5$.
- If CaO/ $P_2O_5 < 1.315$, $aP_2O_5 = CaO / 1.315$.

Where the CaO / P_2O_5 ratio value is 1.315 or greater, this indicates that the phosphate is from Apatite. A CaO / P_2O_5 ratio of less than 1.315 indicates that secondary phosphates are present.

Table 16-8 details the material categories used for mine planning purposes and includes moisture content values, which were used to calculate wet tonnes for volumetric calculations when estimating mining equipment requirements and are derived from historical laboratory testing results.

Table 16-8: Material Categories and Assigned Moisture Content

Material Categories	"ROCK2" Code	Moisture Content (%)
FW Carbonatite / Bedrock	100	10
"Unconsolidated" 2A Mill Feed Residuum	210	15
"Consolidated" 2B Mill Feed Residuum	220	12
2C Waste Residuum	290	11
Lateritic < 0.5 Nb_2O_5 (High Grade Niobium)	310	15
Lateritic > 0.5 Nb_2O_5 (Low Grade Niobium)	320	15
Overburden	400	10

16.2.4 P_2O_5 Price and Recovery Formulation

To run pit optimization exercises and produce economic pits shells, an average selling price of USD1,300 per tonne of P_2O_5 (recovered) was used for all phosphate products. Recovered P_2O_5 tonnes represents the total product tonnages. A concentrate grade of 37.5% P_2O_5 is assumed as a result from processing in the beneficiation plant.

The following details all the recovery assumptions, inputs and formulae used for both the pit optimization and the mine scheduling exercises to estimate sufficient mill feed tonnages and produce 500 ktpa of phosphate product tonnages:

These recovery factors were applied to individual blocks categorized as mill feed within the mining model, therefore applying these factors to blended mill feed tonnages and grades on a per year basis will yield marginally different total P_2O_5 recovered tonnes than those presented in the PEA mine production schedule (see Table 16-21):

- Beneficiation Plant (metallurgical) recovery (%) = 53.8% + (0.84 * aP_2O_5 grade).
- Phosphoric Acid Plant recovery = 95.0%.

The following is an example of the total recovery formulation for the final phosphate product tonnes to be generated:

- Assumed mill feed diluted aP_2O_5 Grade = 23.0% P_2O_5.
- Mill feed production rate per annum = 3,129,554 dry tonnes.
- Beneficiation / metallurgical recovery = 53.8% + (0.84 * 23.0% aP_2O_5) = 73.1%.
- Total P_2O_5 Recovered Tonnes =
  - Mill Feed Dry Tonnage * Mill Feed Diluted P_2O_5 Grade * Beneficiation / Metallurgical Recovery * Phosphoric Acid Recovery.
  - This equates total recovered tonnes as: 3,129,554 dry tonnes * 23.0% P_2O_5 * 73.1% * 95.0% = 500,000 total P_2O_5 recovered tonnes.

16.2.5 *Geotechnical Design Criteria*

The pit slope parameters used in the preparation of the starter and intermediate pit phases and the ultimate pit design are consistent with the 2007 "Technical Memorandum – Preliminary Pit Slope Design Criteria" prepared by Golder Associates ("Golder"). This technical memorandum was reviewed and considered adequate for the purposes of creating interim pit shells and designing the ultimate pit for inclusion within the context of a PEA-level study.

Table 16-9: Pit Slope Design Criteria

Parameters	Units	Major Material Categories			
		Residuum	Overburden	Lateritic Material	Bedrock/ Carbonatite
Bench Height	m	20	20	20	20
Catch Bench Width	m	20	20	20	8
Bench Face Angle	degrees	45	45	18	52
Inter-Ramp Angle	degrees	27	27	N/A	40

A summary of Golder's 2007 proposed pit slope design criteria, delineated per general type of material category or stratigraphic unit, is shown in Table 16-9 and presumes that:

- Adequate slope depressurization is achieved.
- Localized instability may be mitigated by placing granular fill, such as sand and gravel followed by coarser rock fill, on the slope and in the toe region as a buttress to provide toe support and drainage.

Table 16-10 summarizes the waste and stockpile facility design criteria, consistent with the Golder 2007 "Technical Memorandum – Preliminary Pit Slope Design Criteria".

Table 16-10: Waste Facility and Stockpile Design Criteria

Parameters	Units	Values
Lift Height	m	15
Catch Bench Width	m	15
Lift Face Angle	degrees	27
Inter-Ramp Angle	degrees	19

16.3 Pit Optimization

16.3.1 Mining Model Attributes

To conduct the pit optimization and mine production scheduling exercises, the model items identified in Table 16-11 were coded into the regularized mining model.

Table 16-11: Mining Model Items

Model Items	Description
AP_2O_5	Apatite P_2O_5 grade. If CPRAT > 1.315, AP_2O_5 = P_2O_5%. If CPRAT < 1.315, A P_2O_5 = CaO%/1.315
ROCK2	Mine Planning Material Codes, see details in Table 16-8
MOIST	Moisture of material based on ROCK2 material codes: (100+MOIST)/100 * BULKD for Wet Density
RECP	Beneficiation/Metallurgical Recovery = 53.8 + (aP_2O_5 * 0.84). This formula is based on results of metallurgical tests performed on composite samples of 2A and 2B residuum.
RECF	Total P_2O_5 Recovery (Beneficiation to Acid Recoveries): P_2O_5 * RECP * 95 (Phosphoric Acid Recovery) / 10000 – only applies to Mill Feed material
P_2O_5T	Total P_2O_5 Tonnage Recovered = RECF/100 * 1000 * BULKD
REVB	Revenue of Mill Feed Blocks = P_2O_5T * 1300

16.3.2 Pit Optimization Parameters

Table 16-12 details the pit optimization parameters used to generate economic pit shells for the PEA study.

Table 16-12: Pit Optimization Parameters

Pit Optimization Parameters	Units	Values
2A Mill Feed Mining Costs	USD/dry tonne	2.66
2B Mill Feed Mining Costs	USD/dry tonne	3.26
2C Waste Mining Costs	USD/dry tonne	2.66
Bedrock/Carbonatite Mining Costs	USD/dry tonne	3.26
Lateritic Material Mining Costs	USD/dry tonne	2.66
Overburden Mining Costs	USD/dry tonne	2.35
Incremental Bench Mining Costs	USD/dry tonne (per 10m bench)	0.01
Processing and G&A Costs	USD/dry tonne of Mill Feed	59.97
Transport (Freight Railcar) Costs	USD/dry tonne of P_2O_5 recovered	142.56
Base Case Phosphate Product(s) Price	USD/dry tonne of P_2O_5 recovered	1300.00
Exchange Rate	USD/CAD	1.20
Discount Rate	%	10.0
Bedrock (1) Slope Angle	degrees	36.0
“Unconsolidated” Residuum (2A) Slope Angle	degrees	23.0
“Consolidated” Residuum (2B) Slope Angle	degrees	36.0
Waste Residuum (2C) Slope Angle	degrees	23.0
Lateritic Material (3) Slope Angle	degrees	18.0
Overburden (4) Slope Angle	degrees	23.0

Notes:

1. Mining costs were estimated based on Hatch’s internal benchmarking database for similarly sized, Canadian and owner-operated open pit mining operations – it is assumed that for unconsolidated residuum, lateritic and overburden materials, minimal blasting (only during winter months for frozen materials) will be required.
2. Indicative processing costs were provided by JESA, which approximate the total processing operating cost for a steady state operation aiming to achieve 500 ktpa of recovered P_2O_5.
3. Transport (Freight Railcar) costs were provided by Fox River Resources Corporation (CN Rail via Logistics Marketing Services Inc.) and represent an average transportation cost on a per metric tonne basis for the delivery of the three products produced at the Fertilizer Conversion Complex (MAP, NPS and SPA).
4. Both Indicative and Inferred Mineral Resources are included in the mill feed categorizations (the geological block model provided by DMT presents no Measured Resources).
5. Pit shell slope angles were modified from the pit design inter-ramp slope angles to accommodate the presence of at least one (1) haulage ramp along the pit walls. No sectorization of pit wall slope angles were applied.

16.3.3 *Pit Optimization Results*

The pit optimization exercise was completed using HxGN's MinePlan 3D software and has considered Indicated and Inferred Mineral Resources as potential cash flow generating materials. The Pseudoflow algorithm was deployed to generate optimal pit shells using the regularized mining model.

The optimal pit shells summarized in Table 16-13 were generated using a range of revenue factors applied to the final price of the phosphate products, allowing the selection of both the ultimate pit limits and intermediate pit phases. Pit phase selection assumes a minimum mining width and pushback distance of 50 m, or an optimal pushback distance of 100 m, where possible.

Based on the Discounted Cash Flow analysis (see Figure 16-2), Pit Shell No. 10 (highlighted in Table 16-13), using a revenue factor of 0.70, was selected to represent the ultimate pit limits, the basis for designing the ultimate pit design and final pushback. Figure 16-3 and Figure 16-4 further illustrate the basis for selecting Pit Shell No.10 as the basis is for targeting the ultimate pit limits.

Table 16-13: Pit Optimization Results

Pit Shell No.	Revenue Factor	Total Dry Tonnage (kt)	Waste Dry Tonnage (kt)	Mill Feed Dry Tonnage (kt)	Strip Ratio (t:t)	Phosphate Product(s) Tonnage (kt)	Mine Life (Years)
1	0.43	47,631	37,801	9,830	3.8	1,864	3.7
2	0.44	55,110	43,967	11,143	3.9	2,052	4.1
3	0.45	57,992	45,855	12,136	3.8	2,215	4.4
4	0.46	88,395	72,117	16,277	4.4	2,998	6.0
5	0.47	95,086	77,029	18,057	4.3	3,281	6.6
6	0.48	126,427	102,397	24,030	4.3	4,292	8.6
7	0.49	200,841	153,530	47,312	3.2	7,888	15.8
8	0.50	208,991	159,540	49,451	3.2	8,212	16.4
9	0.60	316,470	245,209	71,261	3.4	11,312	22.6
10	0.70	381,483	296,498	84,984	3.5	12,898	25.8
11	0.80	426,198	333,820	92,379	3.6	13,667	27.3
12	0.90	450,866	354,674	96,193	3.7	14,004	28.0
13	1.0	470,498	372,358	98,140	3.8	14,180	28.4
14	1.10	481,742	382,549	99,193	3.9	14,266	28.5
15	1.20	492,077	392,013	100,064	3.9	14,338	28.7
16	1.30	509,185	408,017	101,167	4.0	14,427	28.9

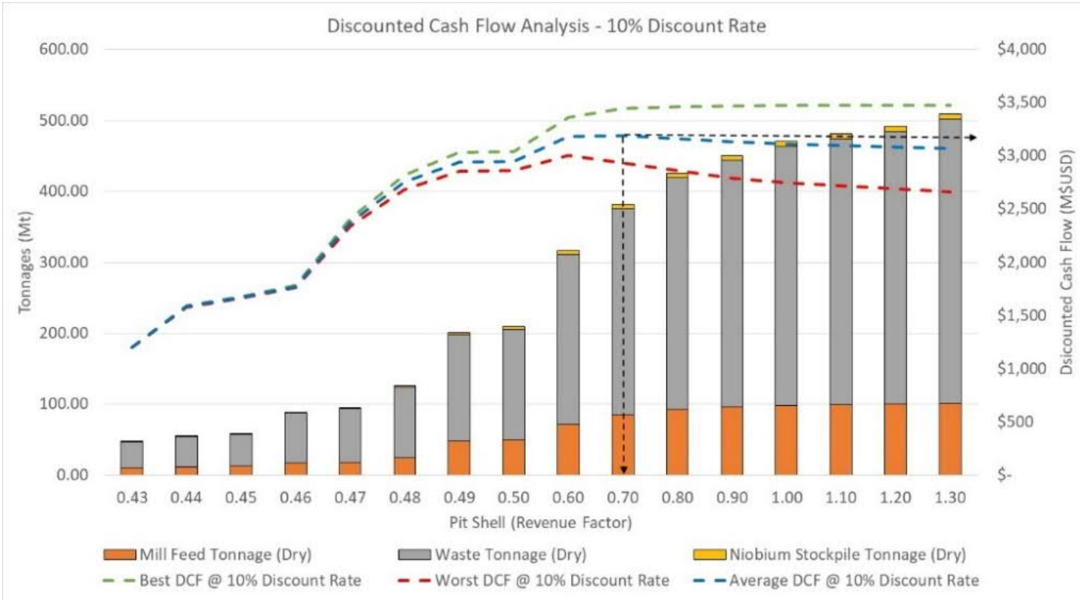


Figure 16-2: Pit Optimization Results – Discounted Cas Flow Analysis

The discounted profit values returned by the optimizer, and represented in the graphs below, do not include initial (CAPEX) or sustaining (SUSEX) capital costs and are only intended to comparatively evaluate the sensitivity of the deposit to phosphate product prices and static operating costs.

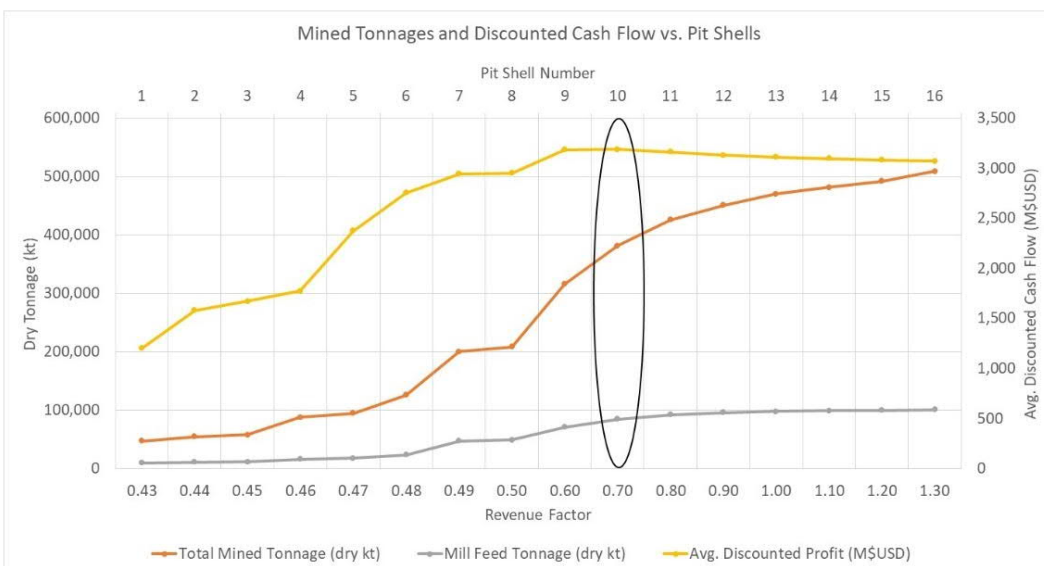


Figure 16-3: Mined Tonnages and Discounted Cash Flow vs. Pit Shells

Figure 16-4: Mine Life, Strip Ratio and Discounted Cash Flow vs. Pit Shells

16.4 Mine Design

16.4.1 Haul Road Design

The widths of the haulage roads on the surface and within the open pit and adjoining ramps are based on both two-way and one-way traffic (as applicable) assuming 227 t haulage trucks. These trucks are one size bigger than the 181 t haulage trucks selected as the base case for mine haul trucks in the mine plan described in this report.

The design road widths were calculated based on the requirements of the Ontario Mining Regulations with details listed in Table 16-14.The one-way ramp design width is included only in the final two benches of the ultimate pit design.

Table 16-14: Haul Road / Ramp Design Criteria

Parameters	Unit	Value
Truck Operating Width	m	8.3
Truck Tire Diameter	m	3.6
Safety Berm Slope Angle	h:v	1.33:1
Safety Berm Height	m	2.7
Ditch Width	m	1.5
Two-Way Running Surface	m	24.9
One-Way Running Surface	m	16.6
Two-Way Design Width	m	34.0
One-Way Design Width	m	25.0

16.4.2 *Ultimate Pit Design*

A plan view of the ultimate pit design is shown in Figure 16-5 below. The ultimate pit was designed using pit and haulage ramp design criteria, incorporating a minimum mining width of 50 m and double benching, 10 m bench height configuration.

Current staging of pit phases offers limited to no waste material backfilling opportunities prior to the final pit phase. Further assessment of the opportunity to permanently store waste material in the mine earlier in the mine is recommended for the subsequent study phase, both to reduce waste haulage cycle times and minimize site disturbance.

Multiple ramp exits are incorporated into the ultimate pit design, accounting for destination points as well as pit phasing. Both two-way and one-way ramps include switchbacks, and an operating turning radius of 12.5 m appropriate for 181 tonne haulage trucks.

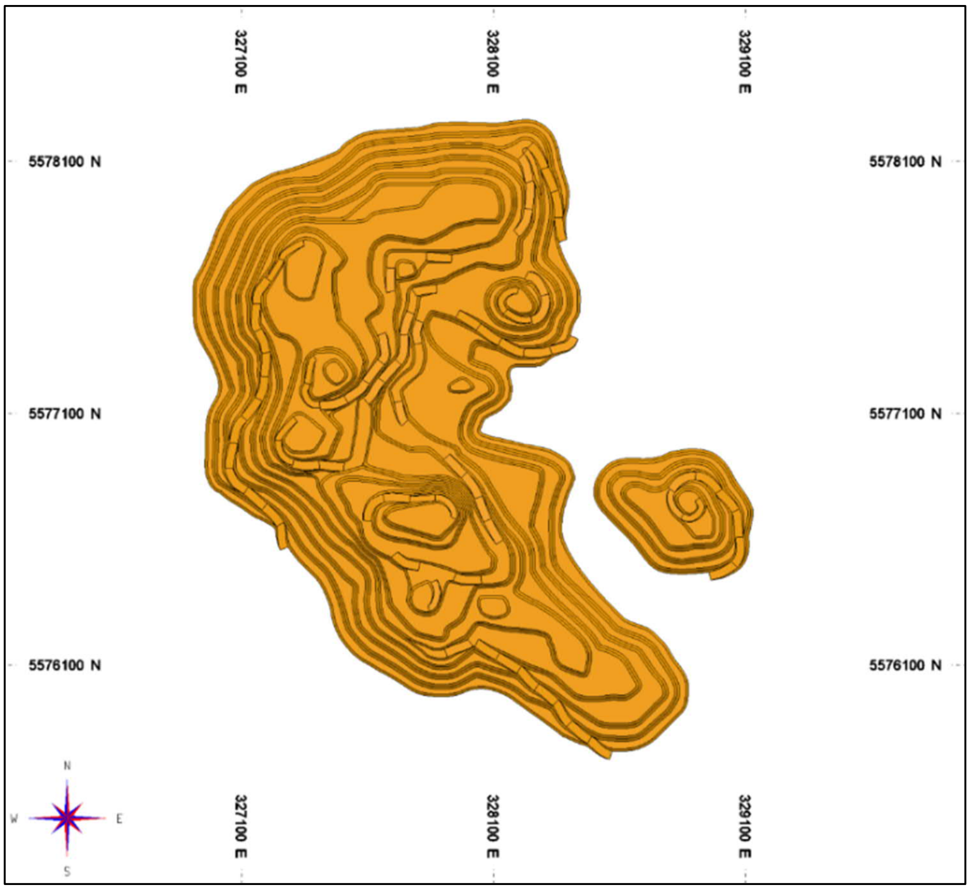


Figure 16-5: Ultimate Pit Design – Plan View

Table 16-15 and Table 16-16 provide summaries of the diluted mineral inventory and scheduled inventory by resource classification respectively for the ultimate pit design.

Table 16-15: Ultimate Pit Design – Diluted Material Inventory

Material	Dry Tonnage (Mt)	Volume (M-m^3)	Moisture Content (%)	aP_2O_5 (%)	Nb_2O_5 (%)	CaO (%)	P_2O_5 Tonnage (kt)
Overburden	265.67	140.57	10.00	0.00	0.02	0.62	0
Bedrock/Carbonatite	11.28	5.41	10.00	0.00	0.15	10.95	0
Lateritic (< 0.5 Nb_2O_5)	4.01	2.29	15.00	0.00	0.27	5.94	0
Lateritic (> 0.5 Nb_2O_5)	6.59	3.84	15.00	0.00	1.06	5.36	0
Residuum – Mill Feed 2A	58.45	30.78	15.00	18.34	0.43	26.69	7,118
Residuum – Mill Feed 2B	25.16	13.24	12.00	28.78	0.48	38.85	5,379
Residuum – Waste 2C	38.32	20.20	11.34	5.49	0.31	11.66	0
Grand Total	**409.48**	**216.33**	**11.09**	**4.90**	**0.16**	**8.14**	**12,497**

Table 16-16: Ultimate Pit Design – Scheduled Inventory by Resource Classification

Material / Resource Categories	Cut-Off Grades	Dry Tonnage	Moisture Content	aP_2O_5 Grade	CaO Grade
	(%)	Mt	%	%	%
Mill Feed Material (Indicated Resource)	aP_2O_5 > 11.0	45.52	14.06	21.86	32.04
Mill Feed Material (Inferred Resource)	aP_2O_5 > 11.0	38.09	14.15	21.02	28.33
Total Mill Feed Material	aP_2O_5 > 11.0	83.61	14.10	21.48	30.35
Niobium-Rich Lateritic Material	Nb_2O_5 > 0.5	6.59	15.00	0.00	5.36
Waste Material	-	319.28	10.22	0.66	2.38

Notes:
1. Material inventory reported from re-blocked, diluted mining model.
2. Mill feed refers to mineralized material that will be sent to the beneficiation plant for processing and includes inferred resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves.
3. Moisture content represents 100 x moisture mass / dry mass.
4. aP_2O_5 grade is the % P_2O_5 derived from the primary phosphate mineral apatite and excludes P_2O_5 derived from secondary phosphate minerals.

16.4.3 *Pit Phases*

Table 16-17 lists the intermediate pit phases generated during the pit optimization exercise and selected for mine production sequencing and planning. These are depicted in plan view in Figure 16-6.

Table 16-17: Selected Pit Phases based on Pit Optimization Exercise

Pit Phase No.	Pit Shell No.	Revenue Factor	Total Dry Tonnage (kt)	Waste Dry Tonnage (kt)	Mill Feed Dry Tonnage (kt)	Strip Ratio (t:t)	Phosphate Product(s) Tonnage (kt)	Mine Life (Years)
2	3	0.45	57,992	45,855	12,136	3.8	2,215	4.4
3	6	0.48	126,427	102,397	24,030	4.3	4,292	8.6
4	7	0.49	200,841	153,530	47,312	3.2	7,888	15.8
5	9	0.6	316,470	245,209	71,261	3.4	11,312	22.6

The pit shells were generated using slope angles that assume the presence of at least one ramp on all sides of the pit, and are therefore carried forward to inform the development of the pit throughout the LOM plan. All pit shells were selected with a minimum 50-100 m pushback distance between phases, wherever possible.

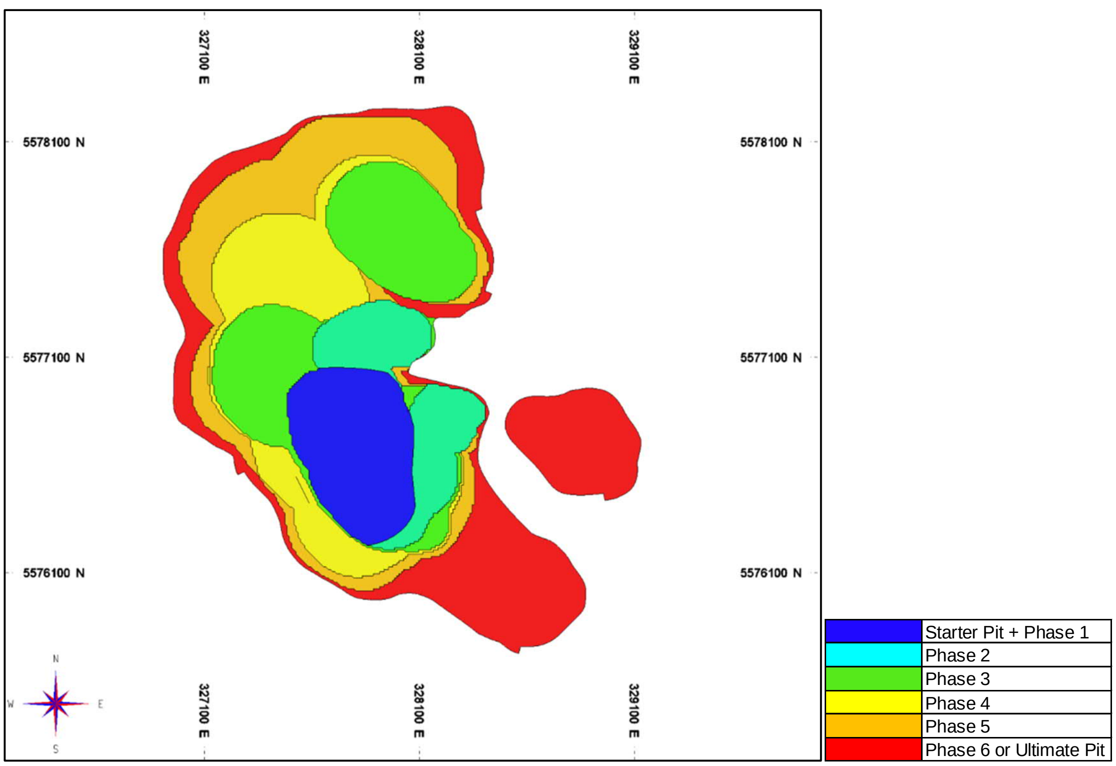


Figure 16-6: Selected Open Pit Phases – Plan View

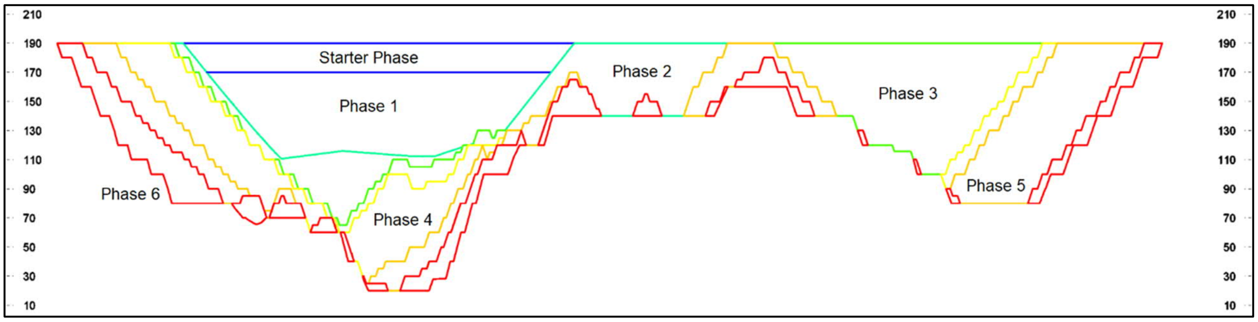


Figure 16-7: Selected Open Pit Phases – Cross Section View (Easting 327,890)

The pit optimization exercise did not generate a pit shell small enough to act as a viable starter pit, therefore Pit Shell No.3 was split into the following three stages:

- **Starter Pit**. This 20 m box cut was created to minimize site preparation and slated for pre-stripping during the pre-production year. The location of the starter pit (and the Phase 1 pit immediately below) was selected based on the highest concentration of drillhole data, therefore encompassing the highest confidence levels of resource classification available in the block model (>99% Indicated Resource).
- **Phase 1**. This comprises the portion of Pit Shell No.3 immediately below the 20 m box cut of the starter pit. This stage provides approximately two years of mill feed.
- **Phase 2**. This which comprises the remainder of Pit Shell No.3 and provides approximately another two to three years of mill feed.

The material inventories within each pit phase are summarized below in Table 16-18.

Table 16-18: Pit Phases – Material Inventories

Item	Unit	Values						
		Starter	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	Phase 6
Mill Feed Tonnage	Dry Mt	0.00	5.35	6.78	10.42	22.66	20.73	17.67
Mill Feed aP_2O_5 Grade	%	0.00	23.86	26.11	23.88	21.97	20.45	18.14
2C Residuum (Waste) Tonnage	Dry Mt	0.00	0.73	1.71	2.18	6.04	8.14	19.52
Bedrock/Carbonatite Tonnage	Dry Mt	0.00	0.01	0.03	0.07	0.52	0.72	9.94
Lateritic (< 0.5 NbO_2) Tonnage	Dry Mt	0.00	0.49	0.05	0.43	0.44	1.32	1.27
Lateritic (> 0.5 NbO_2) Tonnage	Dry Mt	0.00	0.66	0.01	1.44	1.20	1.70	1.58
Overburden Tonnage	Dry Mt	12.52	11.64	18.29	44.68	42.57	61.99	73.98
Waste Tonnage	Dry Mt	12.52	13.53	20.08	48.80	50.77	73.86	106.30
Stripping Ratio	t:t	-	2.53	2.96	4.68	2.24	3.56	6.02

16.5 Waste Facility and Niobium Stockpile

16.5.1 Waste Facility

To minimize haulage cycle times, waste materials will be deposited on the west side of the Anomaly A deposit. Sufficient geological evidence suggests that minimal mineralization is present underneath the proposed location of the Waste Facility and is therefore considered very low risk to establish this area for the permanent disposal of waste material (see Figure 16-8).

Sufficient evidence of mineralization of interest is present towards the east of Anomaly A deposit, therefore excludes this location from consideration for waste deposition. It is recommended that the areas to the west and east of the Anomaly A deposit be subject to step-out and condemnation drilling, both to confirm the proposed location of the current Waste Facility and further define any additional potential waste deposition areas to the east of the Anomaly A deposit, which may improve truck haulage times on the east side of the pit.

A swell factor of 30% and an additional design capacity of 10% was factored to estimate the footprint of the Waste Facility. This facility will have an ultimate height of 30 m, or two 15 m height lifts.

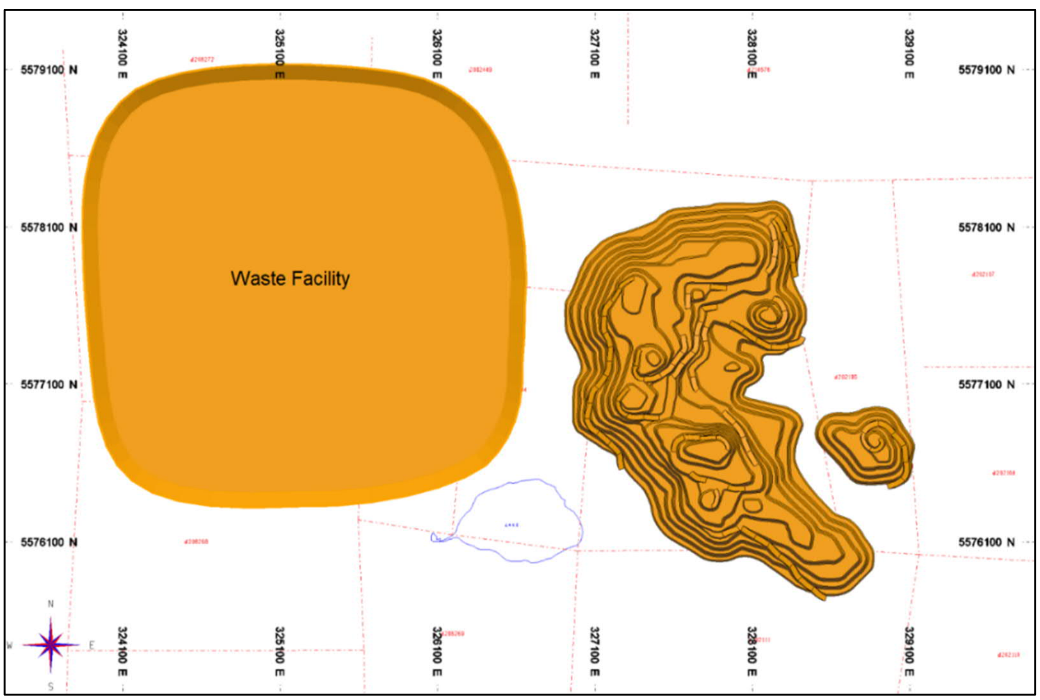


Figure 16-8: Waste Facility Location

The ultimate pit will be backfilled towards the end of the LOM, as shown in Figure 16-9. Backfilling of the pit will mitigate total land disturbance, as well as reduce waste haulage cycle times and promote progressive/pre-closure site rehabilitation.

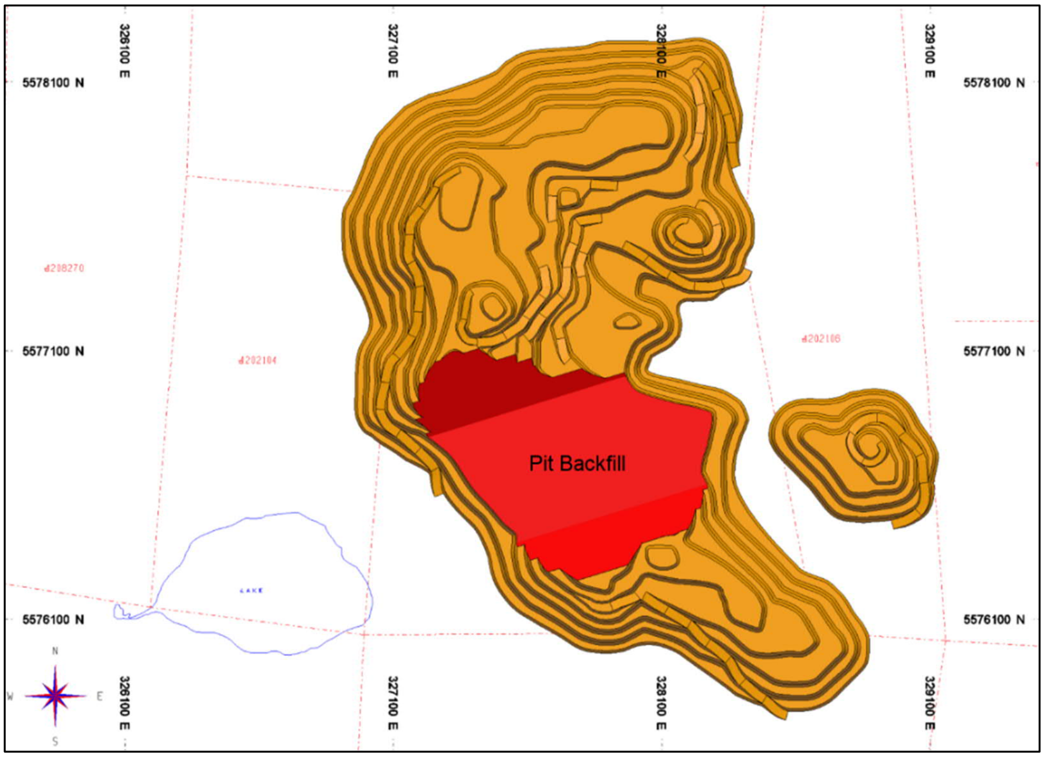


Figure 16-9: Ultimate Pit Design with Pit Backfill

The design capacities of the final storage areas for waste material both on surface and in the open pit are summarized below in Table 16-19.

Table 16-19: Waste Facilities Capacity Summary

Facilities	In-Situ Volume	Swell Volume	Design Capacity
Units	**M-m^3**	**M-m^3**	**M-m^3**
Waste Facility	145.38	188.99	207.89
Pit Backfill	23.10	30.03	33.03

As clearing and grubbing activities are carried out, muskeg and other surficial organic materials will be stored within the footprint of the Waste Facility, segregated from the remainder of waste materials to facilitate future rehandling and site rehabilitation activities.

16.5.2 *Niobium Stockpile*

A location approximately 2.5 km from the centroid of the ultimate pit design is proposed to stockpile niobium-rich lateritic material, for ease of rehandling to the beneficiation plant. Although there is currently no defined means to economically recover niobium from this material, it is recognized that the potential exists and that a separate stockpile should be planned for.

A swell factor of 30% and an additional design capacity of 10% was factored to estimate the footprint of the niobium stockpile, comprising a single 15 m lift height. The planned location of this stockpile is shown in Figure 16-10 and capacity details in Table 16-20.

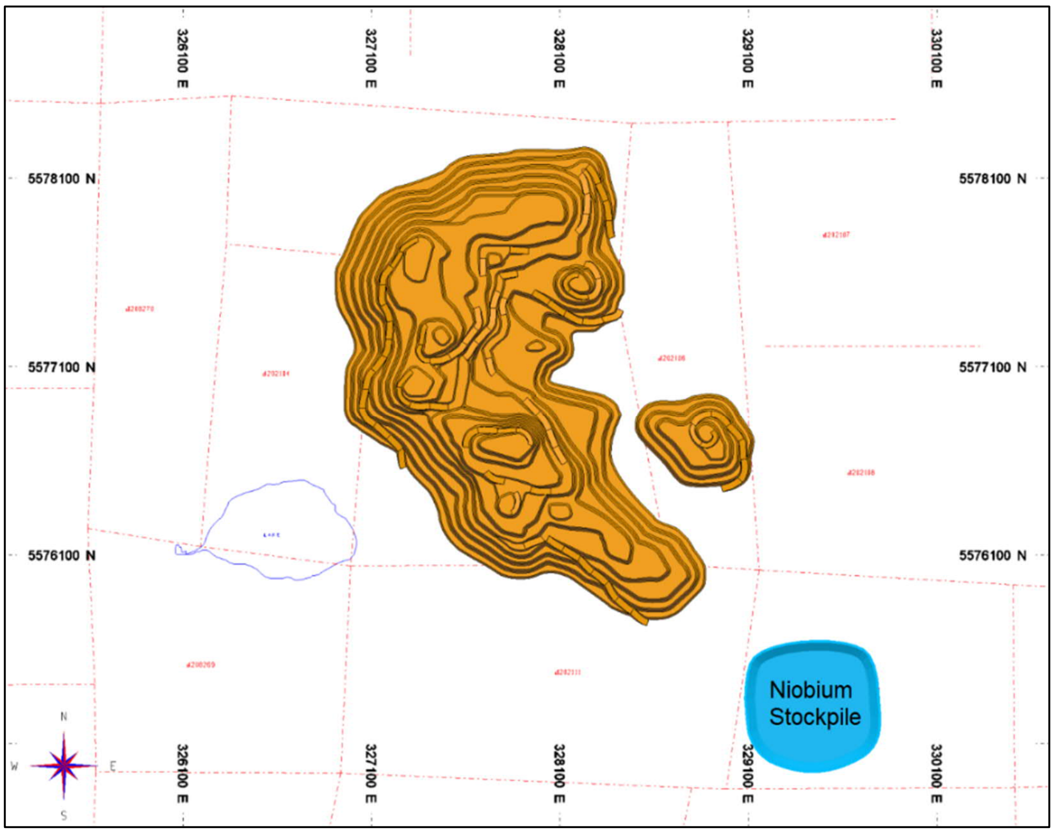


Figure 16-10: Niobium Stockpile Location

Table 16-20: Niobium Stockpile Capacity Summary

Facility	In-Situ Volume	Swell Volume	Design Capacity
Units	**M-m³**	**M-m³**	**M-m³**
Niobium Stockpile	3.84	4.99	5.48

16.6 Mine Production Schedule

The mine production schedule was developed using HxGN's MinePlan Scheduling Optimizer tool (MPSO) to optimize mill feed tonnages and grades while mitigating waste stripping, where possible, and optimizing the material handling activities from within the pit to the following destinations:

- Beneficiation plant – Mill feed materials.
- Waste Facility – Waste materials.
- Ultimate Pit Backfill – Waste materials.
- Niobium Stockpile – Lateritic material (above 0.5 % Nb_2O_5) materials.

The mine production schedule is governed by pit phasing, haulage truck cycle times and the production criteria for mill feed to be sent to the beneficiation plant. The production ramp up factors applied are:

- Year 1 – 69% Final Phosphate Product Tonnage (345 kt).
- Year 2 – 95% Final Phosphate Product Tonnage (473.5 kt).

The mine production schedule was developed on annual increments and is summarized in Table 16-21. Steady state is achieved from Year 3 and onwards at an annual rate of final phosphate production tonnage of 500 ktpa.

Table 16-21: Mine Production Schedule – Summary

Mine Year	Total Mined	Waste Mined	Lateritic Material Stockpiled	Beneficiation Plant Feed			Strip Ratio	Final Phosphate Product(s) Tonnages (1)
				Total Feed	Average Feed Grades			
(years)	kt (dry)	kt (dry)	kt (dry)	kt (dry)	% aP_2O_5	% CaO	t:t	kt
-1	11,500	11,500	-	-	-	-	N/A	-
1	19,308	16,650	530	2,128	23.03	32.91	8.07	345.0
2	18,849	16,277	40	2,533	25.78	35.77	6.44	473.5
3	20,995	18,074	95	2,827	24.70	36.45	6.43	500.0
4	19,318	16,336	129	2,853	24.50	34.10	5.77	500.0
5	18,109	14,991	463	2,655	25.81	36.10	5.82	500.0
6	18,284	14,660	500	3,124	22.81	31.61	4.85	500.0
7	17,151	13,892	289	2,970	23.75	35.15	4.77	500.0
8	15,014	11,995	83	2,936	23.81	33.73	4.11	500.0
9	14,011	9,812	283	3,916	18.87	26.59	2.58	500.0
10	16,530	12,334	559	3,637	20.10	28.34	3.55	500.0
11	17,378	14,610	22	2,746	25.27	33.52	5.33	500.0

Mine Year	Total Mined	Waste Mined	Lateritic Material Stockpiled	Beneficiation Plant Feed			Strip Ratio	Final Phosphate Product(s) Tonnages [1]
				Total Feed	Average Feed Grades			
(years)	kt (dry)	kt (dry)	kt (dry)	kt (dry)	% aP_2O_5	% CaO	t:t	kt
12	15,881	13,131	69	2,680	25.56	35.75	4.93	500.0
13	15,540	11,702	322	3,516	20.58	31.38	3.42	500.0
14	15,984	12,588	69	3,327	21.65	30.22	3.80	500.0
15	15,666	12,904	0	2,762	25.21	35.88	4.67	500.0
16	14,183	10,698	134	3,351	21.50	30.28	3.23	500.0
17	14,576	10,594	527	3,454	21.05	28.75	3.22	500.0
18	14,571	10,017	701	3,853	19.24	28.30	2.78	500.0
19	13,692	10,512	204	2,976	23.72	32.36	3.60	500.0
20	12,959	9,323	7	3,629	20.25	27.68	2.57	500.0
21	13,049	9,189	26	3,834	19.25	27.56	2.40	500.0
22	12,906	9,223	37	3,646	19.97	26.74	2.54	500.0
23	16,626	12,071	46	4,508	16.92	23.32	2.69	500.0
24	15,033	10,266	512	4,255	17.76	25.99	2.53	500.0
25	10,186	5,573	944	3,668	19.99	27.88	1.78	500.0
26	2,178	354	0	1,823	15.33	25.09	0.19	178.5
Total	**409,476**	**319,277**	**6,593**	**83,606**	**21.48**	**30.35**	**3.90**	**12,497.0**

Note:
1. kt P_2O_5 contained in products produced at the Fertilizer Conversion Complex.

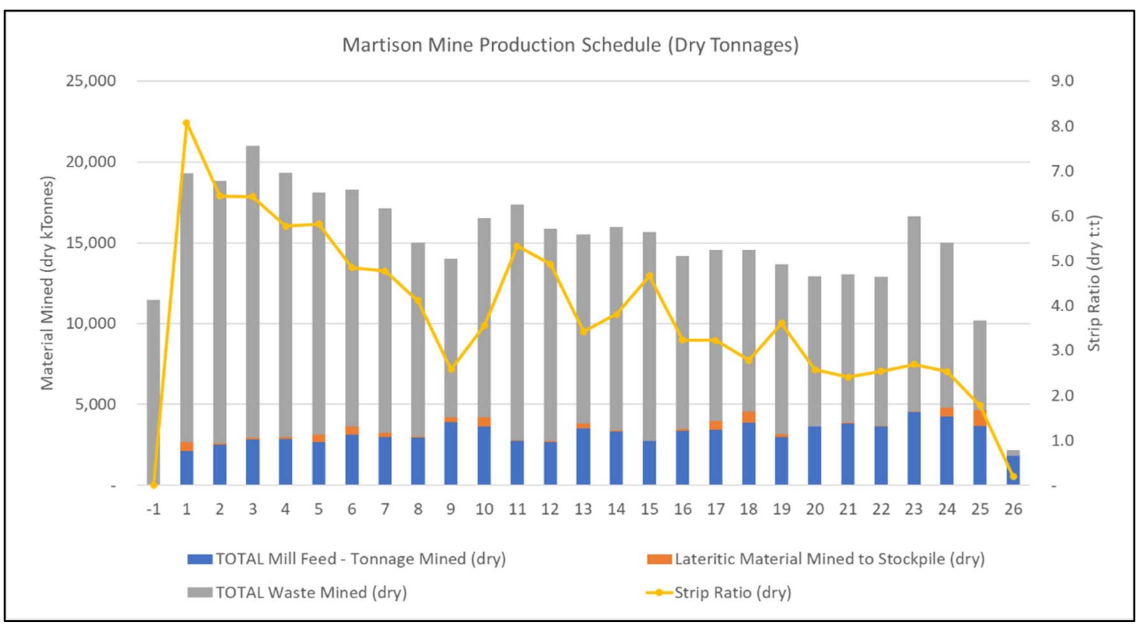


Figure 16-11: Life of Mine (Dry) Tonnages

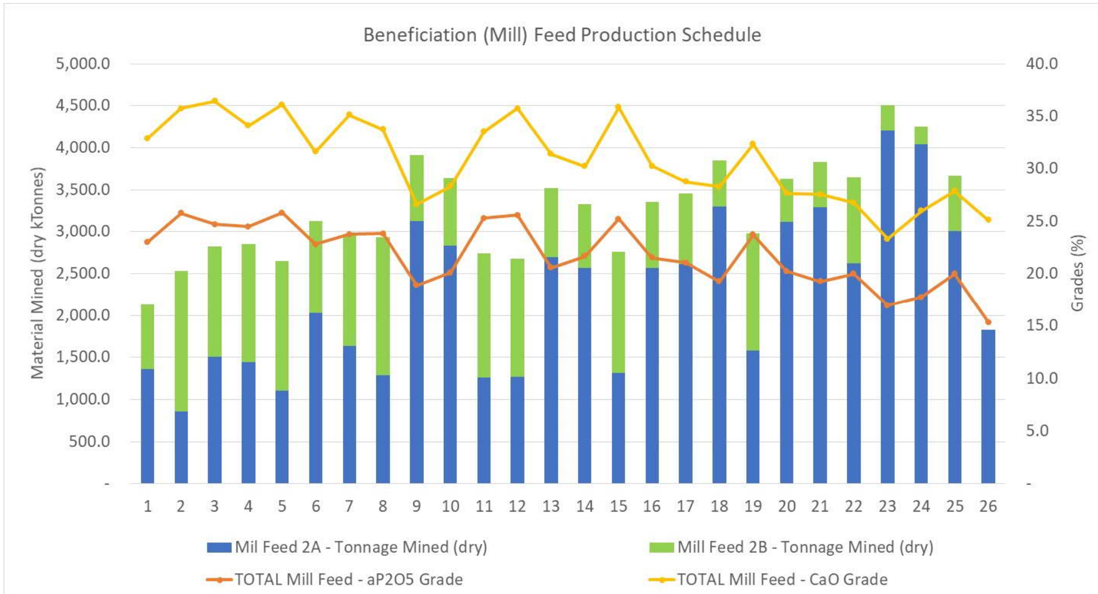


Figure 16-12: Life of Mine Mill Feed (Dry) Tonnages and Grades

16.7 Mine Equipment

All mining equipment was selected based on the overall mine production schedule and with consideration of related best practices in similar operations with similar site conditions. Equipment operating hours are based on two (2) 12-hour shifts, seven days a week, 360 days per year. It has been assumed that five (5) days are lost on an annual basis due to weather and other major disruptions.

16.7.1 Haulage Trucks

Diesel-powered, 181 tonne payload off-highway trucks were selected as the primary units for handling all mill feed and waste materials to and from the pit. Trucks of this size and class were selected for the purposes of this report to optimize the fleet size in relation to material tonnages and cycle times.

Relatively soft base ground conditions are expected to be present throughout much of the mine site. In consideration of truck unit sizing, most rigid-frame trucks with a payload of 100 short tons or above have comparable ground bearing pressure values and therefore was not a factor in the truck class selection. Soft ground issues are to be mitigated with the placement of locally sourced rockfill on haul roads and work areas, and the use of oversized tires to reduce ground bearing pressure.

Table 16-22 summarizes the haulage truck productivity and efficiency factors used to estimate truck requirements. Utilizing a nominal box capacity of 107 cubic meters with an average loose wet density of 1.62 t/m^3 (assuming a 30% average swell factor), the calculated average payload per haul trip was estimated to be 172.8 tonnes (an effective 89% fill factor).

Based on these haulage and operating parameters, determination was made for the number of haulage trucks required during the LOM. The requirement of the number of units per year is illustrated in Figure 16-13 below.

Truck replacements are scheduled to occur at 80,000 hours.

Table 16-22: Haulage Truck Overall Equipment Efficiency

Truck Efficiency Factors	Unit	Value
Fixed Delays per day	*hrs/day*	2.0
Variable Delays per operating hours	*mins/hr*	3.9
Available Operating Hours per day	*hrs/day*	20.6
Operational Efficiency	%	86%
Mechanical Availability	%	85%
Mechanical Utilization	%	85%
Overall Equipment Efficiency	**%**	**62%**

The following nominal spotting, loading and dumping times were applied when pairing the selected trucks with production shovels and loaders:

- Spot time: 90 seconds
- Load time: 180 seconds
- Dump time: 60 seconds.

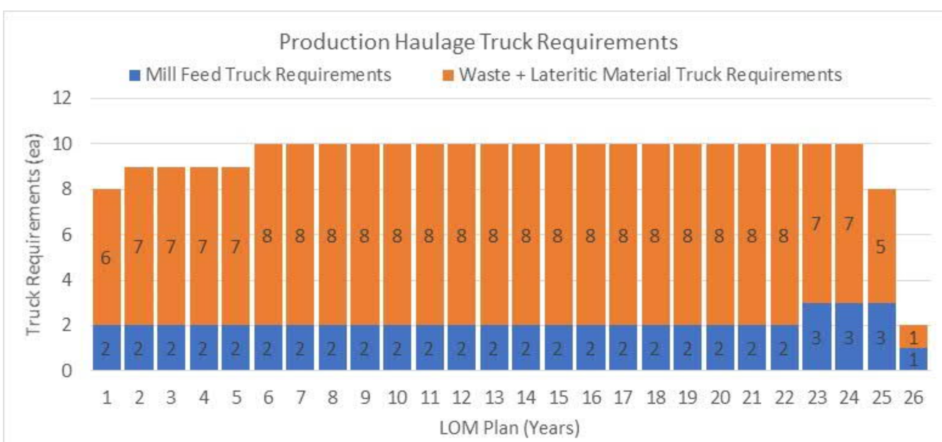


Figure 16-13: Production Haulage Truck Requirements

As part of the next phase of study, it is recommended to evaluate mine road conditions, and to prepare construction specifications and designs that best address the relatively poor base ground quality anticipated both on-surface and in-pit. The assessment should include evaluation of ground bearing capacities, construction and road dressing material sources, as well as more detailed/accurate cost estimations for available rockfill. The findings from these assessments should incorporated into a dedicated equipment sizing trade-off study.

This trade-off study, accounting for operational precedence in similar conditions, may result in the introduction of lighter, articulated haulage trucks tackling specific material types and/or areas in the pit, or conceivably the use of semi-mobile bulk material handling systems during peak years of waste removal and handling.

16.7.2 Loading Units

The following equipment were selected to load the 181 tonne payload, off-highway haulage trucks:

- Production shovel: 22.0 m^3 capacity diesel-powered, hydraulic front shovel.
- Production loader: 19.1 m^3 capacity diesel-powered, hydraulic wheel loader.

For the purposes of estimating production shovel and loader unit requirements, it is assumed that 15% of production excavating and loading for the production haulage trucks will be conducted by the wheel loader units, with the remaining tonnages handled by the front shovel units. An additional 1,500 hours of activity per year was assigned to the wheel loader unit for road building, sump excavation, snow removal activities and other general mine earthworks construction.

For projecting costing estimates both loading unit classes are assigned a unit life of equipment of 60,000 hours. Estimated productivity details of these units are shown in Table 16-23.

Table 16-23: Loading Units – Estimated Productivities

Loading Units	Unit	Front Shovel	Wheel Loader
Loading Unit Rated Capacity	m^3	22.0	19.1
Number of Passes (181 tonne truck)	ea	5	6
Daily Estimated Production per unit	t	53,796	39,939
Annual Estimated Production per unit	Mt	19.1	14.2
Pre-Production Unit Requirements	ea	1	1
Peak Production Unit Requirements	ea	2	1

It is assumed that most of the material mined from the pit is “free dug” with drilling and blasting operations conducted only on more consolidated materials within the Anomaly A deposit.

16.7.3 *Drilling and Blasting*

Production rotary blasthole drilling units, capable of drilling hole diameters of 152 mm and conducting single pass drilling to a depth of 13.6 m, were selected to perform drilling and blasting operations on consolidated and/or frozen materials which cannot be free dug.

For the purposes of calculating drilling requirements and blasting consumables: only 60% of the materials classified in the mining model as consolidated residuum (Mill Feed 2B) and bedrock/carbonatite materials, as well as 10% of the overburden, unconsolidated residuum, and lateritic materials, are assumed to require drilling and blasting.

Table 16-24 illustrates the drilling unit productivity factors and drilling patterns used to estimate drilling unit requirements.

These assumptions are consistent with the Golder 2007 “Technical Memorandum – Preliminary Pit Slope Design Criteria”. Additional drilling allowances were incorporated into the estimation of drilling units required, including an increase in required drilling of 15% to account for redrilling, and an additional increase of 5% for wall control. The drilling units are assumed to have a life of equipment of 45,000 hours.

Table 16-24: Production Drilling Parameters and Requirements

Parameters	Unit	Value
Overall Equipment Efficiency	%	50%
Drill Pattern – Burden	m	5
Drill Pattern – Spacing	m	6
Drill Pattern – Sub-Drill	m	1.6
Drill Penetration Rate	m/hr	20.0
Pre-Production Unit Requirements	ea	1
Peak Production Unit Requirements	ea	2

For blasting consumables, a waterproof, heavy ammonium nitrate product (ANFO) was selected to improve the explosive’s resistance to water infiltration, increasing the material’s detonation performance and preventing this blasting agent from dissolving. To further reduce the risk of water contamination, the use of plastic sleeves (preferably made of a non-combustible material) in the drillholes can be used to effectively keep the ANFO dry during loading and blasting operations.

16.7.4 *Support Equipment*

Based on the number of loading units estimated and the production schedule’s peak tonnages, three track dozers, two of which are 600 hp class and one of 400 hp class, are required to support shovel activities. These will also be used for waste/stockpile facility contouring and ramp development support. One additional wheel dozer of 650 hp class is required for general pit work. Two road graders estimated as required to conduct general haulage road and ramp work at peak production. A portable crusher with a dedicated wheel loader and two 45 t articulated trucks will be used for road construction, maintenance and dressing.

A Reverse Circulation (RC) drill rig is included for grade control activities. Other typical support units, such as fuel trucks, water/sander trucks, support crawlers and support blasting equipment for general mine operation use, are included in the mine cost estimates and are listed in Table 16-25.

Table 16-25: Mine Support Equipment Fleet Requirements

Equipment Type	Equipment Description	Annual Operating Hours (hrs)	Peak Production Requirement (ea)
Tracked Dozer	600 hp (447 kW)	4,500	2
Tracked Dozer	400 hp (298 kW)	4,500	1
Wheel Dozer	650 hp (485 kW)	3,500	1
Road Grader	16’ (4.9 m) blade width	4,000	2
Utility Excavator	4 m capacity	3,000	1
Reverse Circulation Drill Rig	6 m rod length	3,000	1
Support Percussion Crawler	1.75-2.50” (44-63.5 mm)	1,000	1
Water/Sander Truck	870 hp (649 kW)	3,000	2
Fuel Truck	496 HP (370 kW)	2,500	2
45t Rough Terrain Crane	45t mobile crane	1,500	1
IT class loader (stemmer, utility)	70 hp (52 kW)	1,500	1
Blasters Truck	4x4 Crew Cab Flatdeck	3,500	2
Pickup Truck	4x4 pickups	3,000	10
Crew Bus	30 seat bus	2,000	3
Portable Crushing Plant	-	1,500	1
Wheel Loader	555 hp (414 kW)	1,250	1
Articulated Truck	45t payload	1,250	2

16.8 Mine Labour

16.8.1 Work Schedule

Four mine production crews (including supervision) each working 12-hr shifts, two shifts per day, 365 days per year are planned and costed accordingly. Support technical staff, administrative staff and senior management workforce counts are based on 7-day dayshift coverage (in essential roles) and one shift per day on weekdays (for administration and other management roles).

16.8.2 Operator and Maintenance Labour Requirements

The workforce requirements for mine operations hourly labour requirements, including operators, maintenance personnel and general labourers were estimated and are summarized in Table 16-26 below.

Table 16-26: Hourly Operator and Maintenance Labour Summary

Hourly Labour Categories	Peak Requirements (ea)
Truck Operators	40
Shovel Operators	8
Loader Operators	4
Blasters & Surface Crew	8
Dozer Operators	10
Grader Operators	5
Heavy Equipment Operators	13
Electrical and Instrumentation Technician	2
Shop Mechanic	10
Millwright	6
Maintenance Shop Labourer	8
Security	4
Total Operator and Maintenance Labour	**118**

16.8.3 Staff Labour Requirements

Mine operations staff labour is subdivided into three categories:

- Operations technical/supervision staff
- Maintenance supervision and planning staff
- Other general staff.

Table 16-27 has a summary of the mine staffing requirements. Several roles are considered as being shared between the FCC and mine Site (General Manager, Controller, Environmental Engineer for example) and these personnel have been captured separately in the economic model prepared by JT.

Table 16-27: Mine Staffing Summary

Operations Technical/Supervision Position	Peak Requirements
Mine Site Manager	1
Administrative Assistant	1
Environmental Engineer	1
Environmental Technician	2
Mine Superintendent	1
Mine Clerk	2
Mine Operations Pit Supervisor	4
Senior Mine Engineer	1
Senior Geologist	1
Mining Engineer	1
Geologist	1
Dispatch Technician	4
Surveyor	1
Mine Technician	1
Total Operations Technical/Supervision Staff	**22**
Maintenance and General Staff Position	**Peak Requirements**
Maintenance Superintendent	1
Maintenance Supervisor	1
Maintenance Planner	1
Maintenance Clerk	1
Warehouse Lead	1
Stores Clerk	1
Buyer	1
Total Maintenance Supervision and General Staff	**7**
Total Staffing Requirements	**29**

17. Recovery Methods

17.1 Overview

The beneficiation facilities to recover P_2O_5 from mined residuum are comprised of seven main processing areas or modules as listed below.

- Crushing, storage, blending, and reclaim of mill feed.
- Grinding and desliming.
- Magnetic separation.
- Flotation.
- Tailings disposal and water recycle.
- Reagent storage, preparation and dosing.
- Concentrate grinding.

The PEA design concentrate rate of 1,412 ktpy is achieved in years three through 25, while the LOP average rate over the 26-year mine life was reduced to 1,357 ktpy due to start up and mine out. The LOP average grade of mill feed was 21.5% P_2O_5.

The beneficiation plant design incorporates several process modifications to the previous plant design utilized in the 2008 PFS study. The process modifications (which have resulted from bench scale and pilot plant tests performed since the PFS) are listed below. The process modifications are designed to increase plant metallurgical efficiency and P_2O_5 recovery, and to improve operability.

17.2 Crushing, Storage, Blending and Reclaim

Relative to the previous design, a third crushing stage has been added upstream of mill feed storage to facilitate a more reasonable reduction ratio for the downstream rod mills. Run-of-mine (ROM) material will be delivered by truck and dumped through a grizzly into a nominal 270 t surge hopper. Material will be removed from the surge hopper by an apron feeder to an open circuit toothed double roll crusher (primary crusher).

The primary crusher product will be conveyed to the secondary crusher, also configured in open circuit. The secondary crusher contains two synchronized toothed rolls to shear and break down the mill feed prior to conveying to the tertiary crusher circuit. This final crushing phase consists of a scalping screen, surge bin, feeder, and a cone crusher with a closed side setting of nominally 10 mm. A scalping screen will also remove -10 mm fines before tertiary crushing. The combined cone crusher discharge and the scalping screen fines are then conveyed to the circular stacker/reclaimer package system.

Meeting the required concentrate specification and maintaining acceptable P_2O_5 recovery requires blending of ROM grades to mitigate short-term mill feed variability. Accordingly, the flowsheet includes a homogenizing stacker and reclaimer package system consisting of a slewing stacker and a rail-mounted bridge type reclaimer enclosed within a circular building. This type of stacker/reclaimer is designed for continuous chevron stacking in one ring-shaped pile. Stacking is achieved by a fan-shaped sprinkling action in an arc to ensure homogenization. Reclaiming is accomplished at the opposite end of the pile by a bridge reclaimer working parallel to a radius line.

17.3 Grinding and Desliming

The Feed Preparation area contains two rod mills operating in parallel and configured in closed circuit with Derrick vibrating screens. The Derrick screens are designed to make a separation at 425 µm. The -425 µm Derrick screen fines are then deslimed at 20 µm and the resulting -425+20 µm material is pumped to the magnetic separation circuit. A simplified block flow diagram of the proposed flotation feed preparation circuit is shown in Figure 17-1 below.

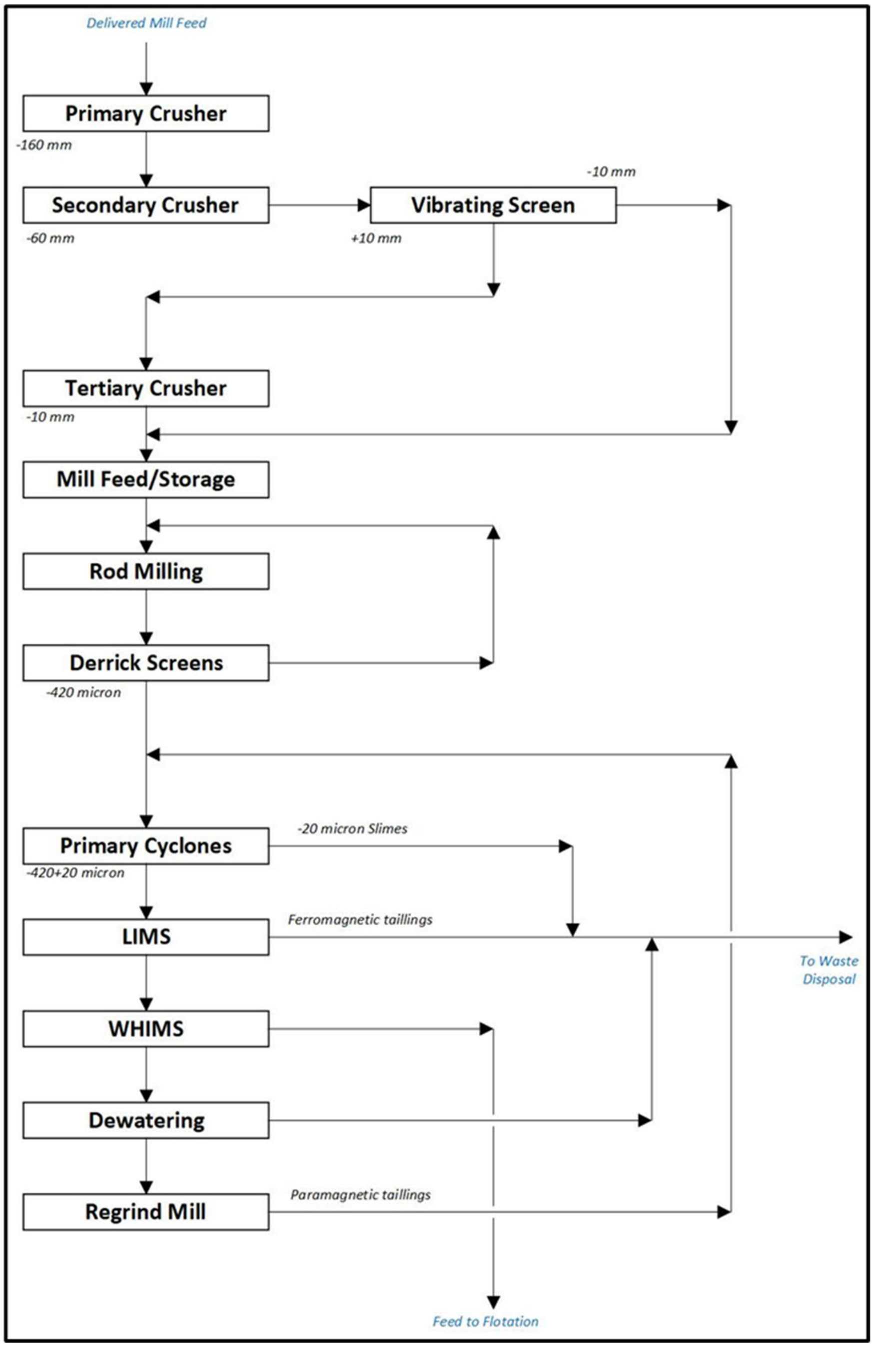


Figure 17-1: Flotation Feed Preparation Circuit

Reclaimed material will be conveyed to the rod mill grinding circuit. Feed to the rod mills is controlled via a single feed bin equipped with two belt feeders, each feeding a rod mill. The grinding and desliming circuit are configured into two parallel trains.

There are two parallel circuits, and the following description applies to both of these. Rod mill discharge, at 65% solids, will be pumped to a distributor feeding three parallel Derrick vibrating screens equipped with screen cloth making a 425 µm size separation and utilizing water sprays to wash the oversize. The screen oversize is returned to the rod mill while the screen fines flow to a pump box that feeds the primary cyclones. The primary cyclones are fitted with cyclo-wash attachments to displace primary slimes from the underflow into the overflow.

The overflow (fine tailings) from the primary cyclones will be dispatched to the mill tailings pump box. The underflow from the primary cyclones will be diluted with process water and pumped to the low intensity wet magnetic separation feed distributor located in the Magnetic Separation area.

The dewatering step, following the WHIMS (shown previously in Figure 17-1) is actually a dewatering and classification device that removes the WHIMS reject for tailings disposal.

17.4 Magnetic Separation

Magnetic Separation is also configured primarily into two parallel trains. A simplified block flow diagram of the proposed magnetic separation circuit is also shown in Figure 17-1.

The material from Grinding and Desliming is distributed to two Eriez type SL LIMS to remove ferromagnetic gangue, which will be discarded with the fine tailing from desliming. The nonmagnetic product will then be treated by two Eriez Type SSS-1-3000 WHIMS to remove paramagnetic gangue. WHIMS paramagnetics are collected in a pump tank and pumped to a cyclone cluster designed to classify and densify the slurry.

The cyclone coarse product is reground in a ball mill. Ball mill discharge, at nominally 65% solids, is combined with process water and returned to the primary cyclones to recover P_2O_5 liberated during regrinding. The cyclone fine product is rejected as waste. The removal of primary slimes and magnetic gangue reduces flotation reagent consumption and makes the flotation separation easier.

The WHIMS nonmagnetic product, which is too dilute to be efficiently conditioned with flotation reagents, will be pumped to dewatering cyclones located upstream of the pre-conditioning step in the Flotation area.

17.5 Flotation

The flotation area is also configured into two parallel operating trains, each capable of independent operation. The flotation circuit includes pre-conditioning and conditioning with flotation reagents followed by three stages of direct flotation.

The pre-conditioning step adjusts the pH of the 65% solids slurry with caustic soda and soda ash and disperses caustic starch, which acts as a depressant for iron oxides. Conditioning with anionic collectors while maintaining the pH using caustic soda follows pre-conditioning. All pre-conditioner and conditioner tanks are series-connected vertical stirred tanks. Slurry exiting the conditioners will be diluted with process water and fed to the rougher flotation circuit.

Each flotation circuit has two parallel rougher flotation column cells followed by two stages of cleaner flotation utilizing two parallel column cells instead of mechanical flotation cells. The cell underflow (tailings) from the rougher columns is collected in a single pump box and then pumped to the TMF. The froth product (concentrate) from the rougher columns is collected and pumped the first cleaner circuit. This circuit consists of two series-connected stirred tank conditioners and two parallel flotation column cells. The first cleaner concentrate will be pumped to the second cleaner circuit that is configured similarly to the first cleaner circuit. The froth product from the second cleaner is final concentrate, which will be pumped to product dewatering and grinding upstream of the concentrate pumping station.

The underflow from the second cleaner circuit (cleaner 2 tails) will be pumped to a cleaner dewatering cyclone cluster to reject low grade fine particles to the tailing. The coarser fraction of the dewatered material will contain some phosphate which is partially conditioned with reagent, and can be recovered by adding more reagents and repeating the flotation stages. This process will maximize the P_2O_5 recovery of flotation.

A simplified block flow diagram of the proposed phosphate flotation circuit is shown below in Figure 17-2.

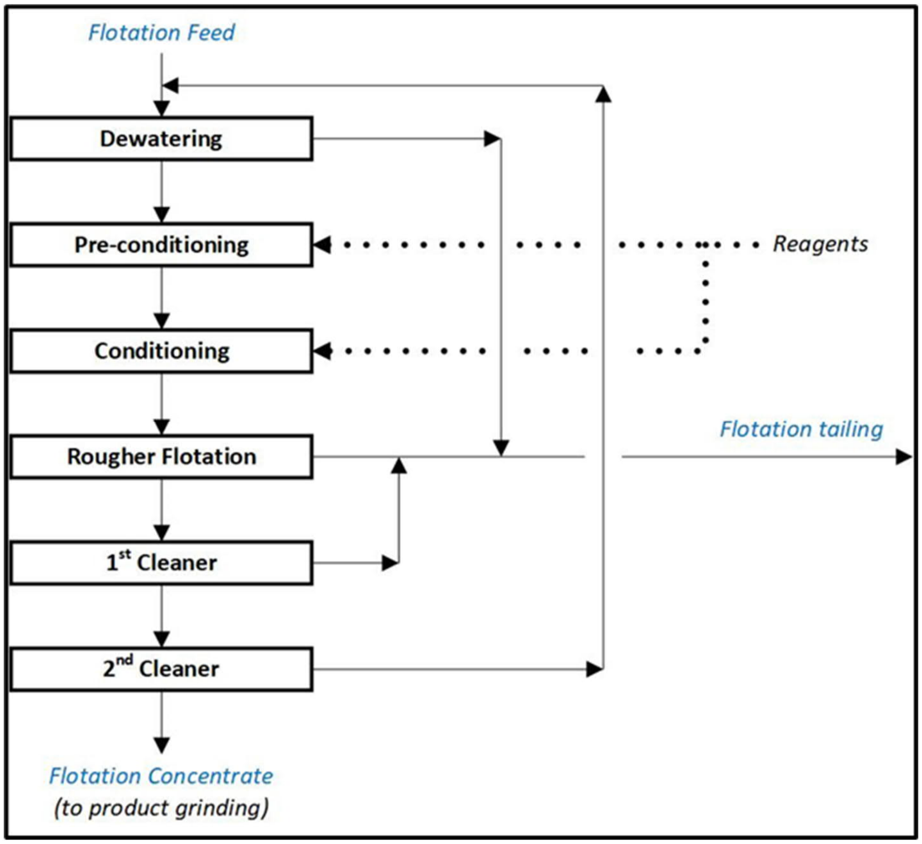


Figure 17-2: Flotation Block Flow Diagram

17.6 Tailings Disposal and Water Recycle

The fine tailings are combined with the magnetic rejects from LIMS and WHIMS and are pumped to the TMF where natural sedimentation will consolidate them. The flotation tailings, which will have concentrated elevated niobium content will be pumped in a second pipeline for storage in a different cell within the TMF to enable future reclamation in order to recover niobium should an economic means of doing so be found.

The mine dewatering pumps will also discharge into the TMF for clarification and to provide make-up water for the beneficiation process. The TMF reclaim water will be returned to the plant by barge-mounted, centrifugal pumps. These pumps will feed a HDPE pipeline, which discharges the reclaimed water into the fire water tank and which overflows to the process water tank. The process water tank will also receive the overflow from the concentrate thickener.

17.7 Reagent Storage, Preparation and Dosing

The beneficiation plant is designed with a reagent tank farm to receive reagents and prepare, store, and meter stock solutions of the reagents required for flotation.

17.8 Concentrate Grinding

In order to achieve the target concentrate particle size distribution (required for efficient transport using a slurry pipeline), the concentrate from the flotation area is dewatered by hydrocyclones and then ground. The dense hydrocyclone underflows are fed to the open circuit ball mill. The dilute hydrocyclone overflows and the ball mill discharge are fed to the concentrate thickener. The thickener underflow, at 55% solids, is pumped to agitated storage tanks, while the clarified overflow water is returned to the beneficiation plant recycle water system.

18. Project Infrastructure

18.1 Mine Site Infrastructure

18.1.1 Access Road

An access corridor will extend to the mine site and beneficiation plant from Highway 11 and Fushimi Road 22 km, west of the town of Hearst. This will require the upgrading of the existing Fushimi Road and an additional northeast trending extension to provide an all-weather access road to the mine site. The FCC will be accessed on the section of upgraded road close to the intersection with Highway 11. The remaining upgraded section and road extension will be capable of handling the delivery of all materials and equipment to the mine site.

The road will be unpaved for the full length and will require regular grading to remove potholes and “washboarding” and winter plowing and gritting to maintain suitability for safe operation of large equipment including frequent delivery of aggregates, fuel, reagents and other consumables, and the transportation of site personnel in buses and other smaller vehicles daily.

The full width of the cleared road ROW will be 30 m in straight sections. An assessment of natural undulations and short changes in road direction will be conducted as part of the design to determine the width of the ROW in these areas to ensure that the risk to traffic from restricted sight lines or road curves are mitigated in order to enable safe all-season access and egress. The access road will be built to a width of 9 m which is sufficient width to enable traffic to safely pass in both directions without the need for passing lanes.

Road construction and upgrade activities identified in this study phase are the following:

- Fushimi Road South Section. The first section is approximately 13 km from Highway 11 and is already being use regularly by the general public to reach Fushimi Lake Provincial Park and area cottages and camp. The cleared road ROW and the road width are already within the design criteria, so no work is expected in terms of road geometry. Only a surface course upgrade with crushed granular material will be required to reinforce the existing road structure and reach the road proper serviceability level for the additional FCC and mine traffic.

- Fushimi Road North Section. This section is approximately 38 km to the north of Fushimi Park Access Road and crosses four (4) culverts. The current cleared road ROW is 10 m and the road width is 6 m. Vegetation will be cleared to reach a 30 m ROW and the road itself will be widened from 6 m to 9 m on one side to align with the required design criteria. The work includes:
  - Partial excavation of the existing road structure to create a stable transition.
  - Cleaning and backfilling the existing ditch.
  - Excavation of a new ditch at the bottom of the projected road slope.
  - Building a new road foundation for the section to be widened.
  - Regrading of the whole road surface to provide a proper crown and 150 mm of 0-20 mm crushed granular will be added.
  - Widening of culverts to suit the width of the new road.
- Extension to mine site. The extension to the mine site is approximately 37 km. This section will require the full construction of a new road. In general, the work to be performed will consist of ROW tree and bush clearing, removal of stumps and topsoil and excavation of side ditches. For the road structure, different types of subgrade are predicted along this section and these different structures have been assumed:
  - For subgrade of granular soils, the road structure will be composed of 300 mm of general backfill, 200 mm of 0-80 mm crushed granular and 150 mm of 0-20 mm crushed granular.
  - For subgrade of freezing soils, a 450 mm of 0-150 mm granular material will be added under the road structure described for granular soils.
  - For subgrade of peat, a 1000 mm of 0-150 mm granular material will be first placed to create gravity displacement and 200 mm of 0-80 mm crushed granular and 150 mm of 0-20 mm crushed granular will be placed after.

For the extension, a geotextile will also be installed on the subgrade to reinforce the native soils below and thus reducing the overall road structure and facilitate construction works directly on top of this organic material.

Fox River has secured a permitted aggregate pit, approximately 25 km north on the Fushimi Road from the junction at Highway 11, which will supply construction fill materials for the initial construction of the access road. The aggregate pit is easily accessible from the Fushimi Road, which itself is in good condition.

This access road will need to be established prior to major construction operations taking place at the mine site, allowing equipment and materials access for site preparation activities, construction of the beneficiation plant, surrounding mine site infrastructure, and for pre-stripping mining activities.

18.1.2 Electrical Transmission Line & Site Power

18.1.2.1 Overview

Electrical power from the main grid will be supplied to the FCC and mine site by a newly constructed overhead line which will connect to the Hydro One Networks Inc (HONI) primary 115kV supply following Highway 11 west of Hearst. Demand for electrical power will be supplemented through the cogeneration process at the FCC which will offset the operating cost of power as well as provide benefits in the application of carbon taxation.

The power line will be constructed in a cleared easement parallel to the Fushimi Road and the extension to the mine site. At the FCC and mine site power will be stepped down from transformer stations at the required voltage for further distribution. Backup generators will also be provided at both locations to provide emergency power in the event of a main supply disruption.

Electrical grid power (surplus from the FCC or otherwise purchased) will be used for all mining and processing demands (except for the use of natural gas for the granulation process) and will also be used for heating in all building locations.

A simplified block diagram of the power system is illustrated in Figure 18-1 and it is presently anticipated that this will consist of the following main components:

- Equipment and works for connection to HONI 115kV line.
- Short 115kV line tap to the FCC main substation.
- Main 115kV switching station at the FCC.
- Step down substation 115/4.16kV 15MVA to supply fertilizer conversion complex loads.
- 115kV 88km line to mine site.
- Main mine site substation 115/4.16kV 20MVA to supply mine site loads.
- Mine standby diesel generation plant, presently assumed to be 3x2MW.

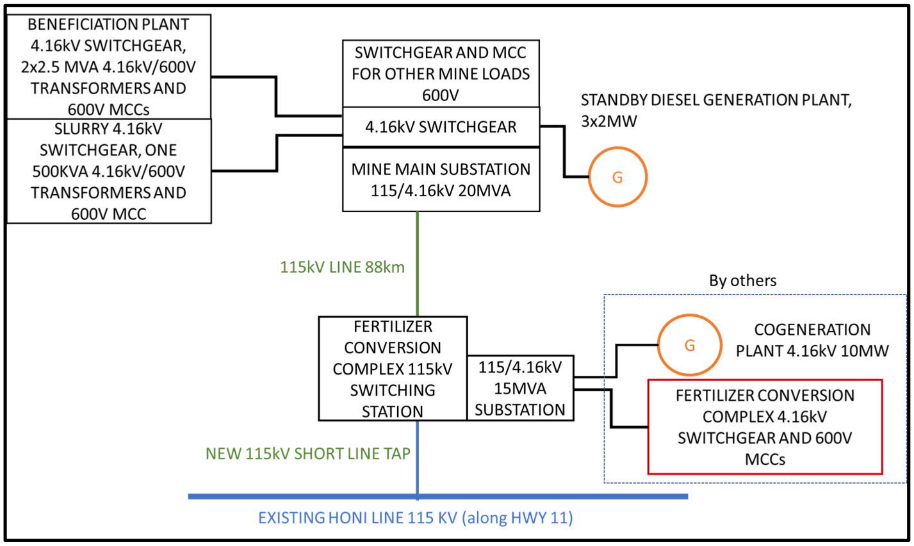


Figure 18-1: Block Diagram – Overall Electrical Power Supply

All transformer sizes shown on the block diagram are provisional based on presently available limited information. Actual ratings will be possible when a more detailed load list and operational philosophy are developed in future phases of study.

18.1.2.2 Connection to Hydro One Networks Inc. ("HONI") transmission Grid

The Transmission System Code (TSC) issued by the Ontario Energy Board (OEB) provides load customers with three options for constructing and owning new load transmission level connected facilities (in Ontario transmission level is defined as operating voltages above 50kV):

- **Option 1:** The proponent can elect to have HONI construct and own the new load connection facilities.
- **Option 2.** The proponent can elect to construct the new load connection facilities and transfer ownership to HONI. This option only applies to new load connection facilities that have been identified by HONI as contestable.
- **Option 3.** The proponent can elect to construct and own the new load connection facilities. This option also only applies to new dedicated load connection facilities that have been identified by HONI as contestable (work the customer can carry out on its own or through a third party).

18.1.2.3 115kV line tap to Fertilizer Conversion Complex switching station

It is assumed that this short line tap Boiler171. 1km long will be designed, permitted, built and owned by HONI.

18.1.2.4 *Fertilizer Conversion Complex 115kV Switching Station*

This facility will consist of the following main equipment:

- Incoming motorized switch.
- Surge arresters.
- Potential transformers.
- Revenue metering system consisting of, a115kV CT/PT combo.
- Main incoming 115kV breaker.
- 115kV breaker + switch for FCC transformer.
- 115kV breaker + switch for transmission line to the mine.
- An electrical building for 115kV P&C. The same building can also accommodate 4.16kV switchgear and LV MCCs for the FCC as well as switchgear for cogeneration plant connection.

18.1.2.5 *FCC Substation (115/4.16kV 15MVA)*

It is presently assumed (subject to peak load confirmation) that the step-down transformer to 4.16kV will be sized 15MVA. The transformer will have an onload tap changer to regulate output voltage due to load changes and primary voltage fluctuations. An oil containment pit will also likely be required.

Transformer protection and controls will be located in the 115kV protection room. Part of this facility will be the main 4.16kV switchgear which will accommodate all FCC loads as well as cogeneration plant grid connection.

18.1.2.6 *Cogeneration Plant and FCC Distribution*

This information is provided elsewhere in this report by JT in Section 18.2.8 (Cogeneration Plant and FCC Distribution).

18.1.2.7 *115kV 88km Line to Mine Site*

The 115kV line to the mine site will be constructed using mostly wooden poles with aluminum conductors sized for local weather conditions and, for the most part, the line will be running parallel to the access road. As current will be small, 75Amp for load of 15MVA, conductor size will not be dictated by electrical but mostly by mechanical parameters to withstand weather elements, snow, ice and wind. To enable effective communications between the mine site and FCC a fiber optic cable will also run along the line.

This line may have to be included in the overall project environmental assessment.

18.1.2.8 *Main Mine Site Substation (115/4.16kV 20MVA)*

This facility will consist of the following main equipment:

- Incoming motorized switch.
- Surge arresters.
- Potential transformers.
- Main incoming 115kV breaker.
- One 115/4.16kV 20MVA transformer.
- One 4.16kV/600V 2 MVA transformer.
- An electrical building for 115kV P&C, main 4.16kV switchgear and LV MCCs for nearby loads.

With the exclusion of beneficiation plant and slurry pumping system, it is assumed that the 2MVA transformer at the main substation will be sufficient to cover all other smaller 600V site loads as illustrated on the block diagram below (Figure 18-2).

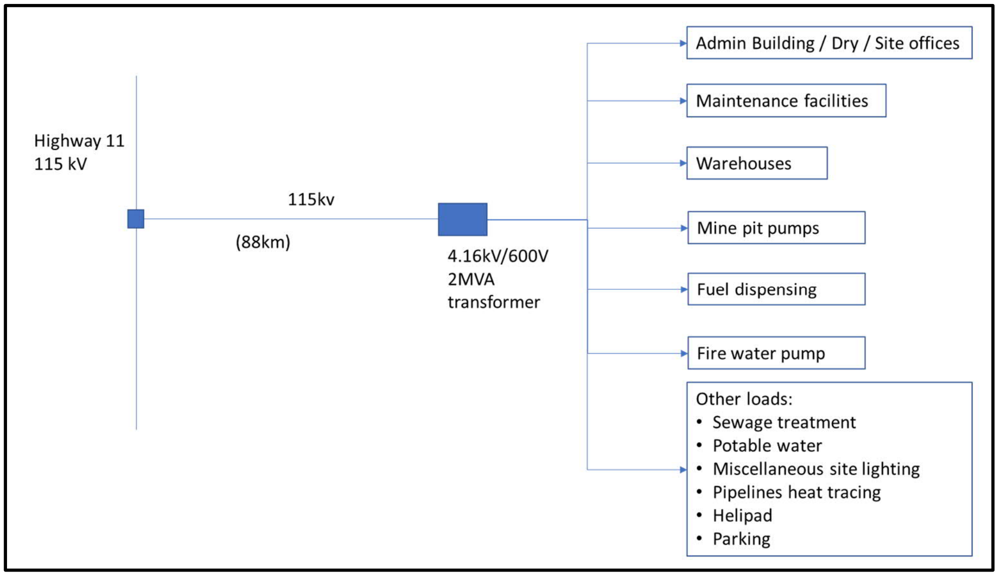


Figure 18-2: 2MVA Transformer At The Main Substation

18.1.2.9 Mine Standby Diesel Generation Plant

It is presently assumed that the mine standby emergency power load will be approximately 4MW. This is based on 2.5MW emergency loads for the beneficiation plant and 1MW for the slurry pumps. To achieve N+1 redundancy, the proposed concept includes three 2MW units, with the assumption that only two units will be running at any time.

The plant will operate at 4.16kV, will be located adjacent to the mine main substation and connected to the main mine site switchgear. The standby power plant will only operate off grid. Therefore, an interlocking system which prevents paralleling the plant to the grid will be required.

18.1.2.10 Beneficiation Plant Substation

The total connected load for the beneficiation plant will be approximately 13MW, with an operating load of 12MW. Based on current assumptions, the split for continuously operating load is 7MW operating at 4.16kV and 5MW operating at 600V.

18.1.2.11 Slurry substation

The total connected load for the slurry handling facility will be approximately 4MW, with an operating load of 2.2MW. Based on current assumptions the split for continuously operating load is 1.8MW operating at 4.16kV and 400kW operating at 600V.

A separate 4.16kV feeder from the main substation will feed the slurry switchgear and that one 4.16/600V 500kVA transformer will be required to satisfy a 600V load.

Tailings pumps, and other requirements, will be fed from the slurry switchgear or MCC depending on load requirements.

18.1.3 Beneficiation Plant

18.1.3.1 Summary

In the 2008 Martison PFS Study, the beneficiation plant was designed to produce concentrated phosphate rock containing 421,000 t/y P_2O_5 which was delivered to the FCC via an 86 km slurry pipeline following the route of the Fushimi Road and the extension of this road to the mine site. The proposed location of the beneficiation plant was located in close proximity to the open pit mining operations where mill feed was ground and processed to produce a slurry of phosphate rock and rejecting phosphate-lean material to a tailings management facility. This PEA study increases the phosphate rock production to 526,000 t/y P_2O_5 and uses the same slurry pipeline concept to deliver the concentrate to the FCC.

This updated design also incorporates column flotation, a third crushing stage upstream of mill feed storage and modifications to the desliming circuits, and additional minor changes to produce 1,412 ktpy of concentrate over a 26-year LOM. Relative to the PFS, the PEA mill feed grade has been slightly reduced and the concentrate MER specification has been relaxed from 0.06 to 0.09 to improve P_2O_5 recovery. In addition, flotation tailings are stored separately from other tailings in anticipation of treating them to recover concentrated niobium from this part of the process at an unspecified future date.

18.1.3.2 *Process Description*

The beneficiation plant is comprised of seven main processing areas or modules as listed below and described in Sections 18.1.3.2.1 through 18.1.3.2.7.

- Crushing, storage, blending, and reclaim of mill feed.
- Grinding and desliming.
- Magnetic separation.
- Flotation.
- Tailings disposal and water recycling.
- Reagent storage, preparation and dosing.
- Concentrate grinding.

The 86 km slurry pipeline which transports phosphate concentrate from the mine site to the FCC is described in Section 18.1.9.

Relative to the 2008 PFS, the PEA has an increased annual concentrate production rate (from 1,161 ktpy to 1,412 ktpy), increased plant life (from 20 years to 26 years), and reduced mill feed grade (from 22.5% P_2O_5 to 21.5 % P_2O_5).

The beneficiation plant design incorporates several process modifications to the plant design utilized in the 2008 PFS study. The process modifications, which resulted from bench scale and pilot plant tests performed after the PFS are listed below. The process modifications are designed to increase plant metallurgical efficiency and P_2O_5 recovery, and to improve operability.

- Addition of a third crushing stage upstream of mill feed storage to facilitate a more reasonable reduction ratio for the downstream rod mills.
- Increasing the mesh of grind from 212 µm to 425 µm by utilizing two rod mills (in parallel) to replace the PFS rod mill/ball mill combination (in series). The rod mills will operate in closed circuit with Derrick vibrating screens. This current closed-circuit grinding flowsheet resulted from pilot plant testing conducted by Jacobs in 2009.
- Modification of the desliming circuits to remove both natural slimes and grinding slimes after grinding. The grinding slimes were problematic for flotation. The increased P_2O_5 losses from rejecting grinding slimes were offset by improved flotation performance and reduced reagent consumption.
- Incorporation of a WHIMS magnetic product regrind mill in closed circuit to liberate and recover P_2O_5 from middling particles.
- Replacement of the mechanical flotation cells with column flotation cells. The Eriez column flotation cells improved flotation performance over the mechanical cells. The flotation circuit is configured as a rougher and two cleaners.

The beneficiation process was reconfigured to operate as two parallel trains instead of the single train design used in the 2008 study.

18.1.3.2.1 Mill Feed Crushing, Storage, Blending and Reclaim

Run-of-mine material (mill feed) will be delivered by truck and dumped through a grizzly with 500 x 500 mm grids into a nominal 270 t surge hopper. Crushing will occur in three steps:

1. Material will be fed from the surge hopper by an apron feeder to an open circuit, toothed, double roll crusher (primary crusher). The primary crusher product (<150 mm) will be conveyed to the secondary crusher, also configured in open circuit.
2. The secondary contains two synchronized toothed rolls to shear/break the mill feed. Secondary crusher product (<50 mm) will be conveyed to the tertiary crusher circuit.
3. The tertiary crusher consists of a scalping screen, surge bin, feeder, and a cone crusher with a closed side setting of nominally 10 mm. The scalping screen is designed to remove <10 mm fines before tertiary crushing. The combined cone crusher discharge and the scalping screen fines are then conveyed to the circular stacker/reclaimer package system.

In order to meet the required concentrate specification, and maintain acceptable P_2O_5 recovery, it will be necessary to blend mill feed to prevent short term feed grade variability. Accordingly, the flowsheet includes a homogenizing stacker/reclaimer package system consisting of a slewing stacker and a rail-mounted bridge type reclaimer enclosed within a circular building. This type of stacker/reclaimer is designed for continuous chevron stacking in one ring-shaped pile. Stacking is achieved by a fan-shaped sprinkling action in an arc to ensure homogenization. Reclaiming is accomplished at the opposite end of the pile by a bridge reclaimer working parallel to a radius line.

The stacker storage capacities are:

- Mixed capacity: 28,000 t.
- Ramp capacity: 14,000 t.
- Total capacity: 42,000 t.

This capacity is equivalent to a nominal 80 hours average storage.

18.1.3.2.2 Grinding and Desliming

The Grinding and Desliming (feed preparation) area contains two rod mills operating in parallel and configured in closed circuit with Derrick vibrating screens. The Derrick screens are designed to make a separation at 425 µm. The <425 µm Derrick screen fines are then deslimed at 20 µm and the resulting 425>20 µm material is pumped to the magnetic separation circuit. A simplified block flow diagram of the proposed flotation feed preparation circuit is shown in Figure 18-3 below.

Reclaimed material will be conveyed to the rod mill grinding circuit. Feed to the rod mills is controlled via a single feed bin equipped with two belt feeders, each feeding a rod mill. The grinding and desliming circuit consist of the following major equipment items configured into two parallel trains each equipped with:

- One 4.6 m long by 6.2 m diameter overflow rod mill equipped with a 1600 kW motor.
- Three Derrick Superstak 2W56-60R vibrating screens.
- One radial cluster of ten desliming hydrocyclones, each 0.25 m-diameter, and complete with underflow tub and overflow collection launder.

Although there are two parallel circuits, the following description applies to each individual circuit. Rod mill discharge, at about 65% solids, will be pumped to a distributor feeding three parallel Derrick vibrating screens equipped with screen cloth making a 425 µm size separation and utilizing water sprays to wash the oversize. The screen oversize is returned to the rod mill while the screen fines flow to a pump box that feeds the primary cyclones. The primary cyclones are fitted with cyclo-wash attachments to displace primary slimes from the underflow into the overflow.

The overflow (fine tailings) from the primary cyclones will be dispatched to the mill tailings pump box. The underflow from the primary cyclones will be diluted with process water and pumped to the LIMS (low intensity wet magnetic separation) feed distributor located in the Magnetic Separation area.

It should be noted that the dewatering box following the WHIMS (refer to Figure 18-3 below) is really a dewatering/classification device that removes the WHIMS reject.

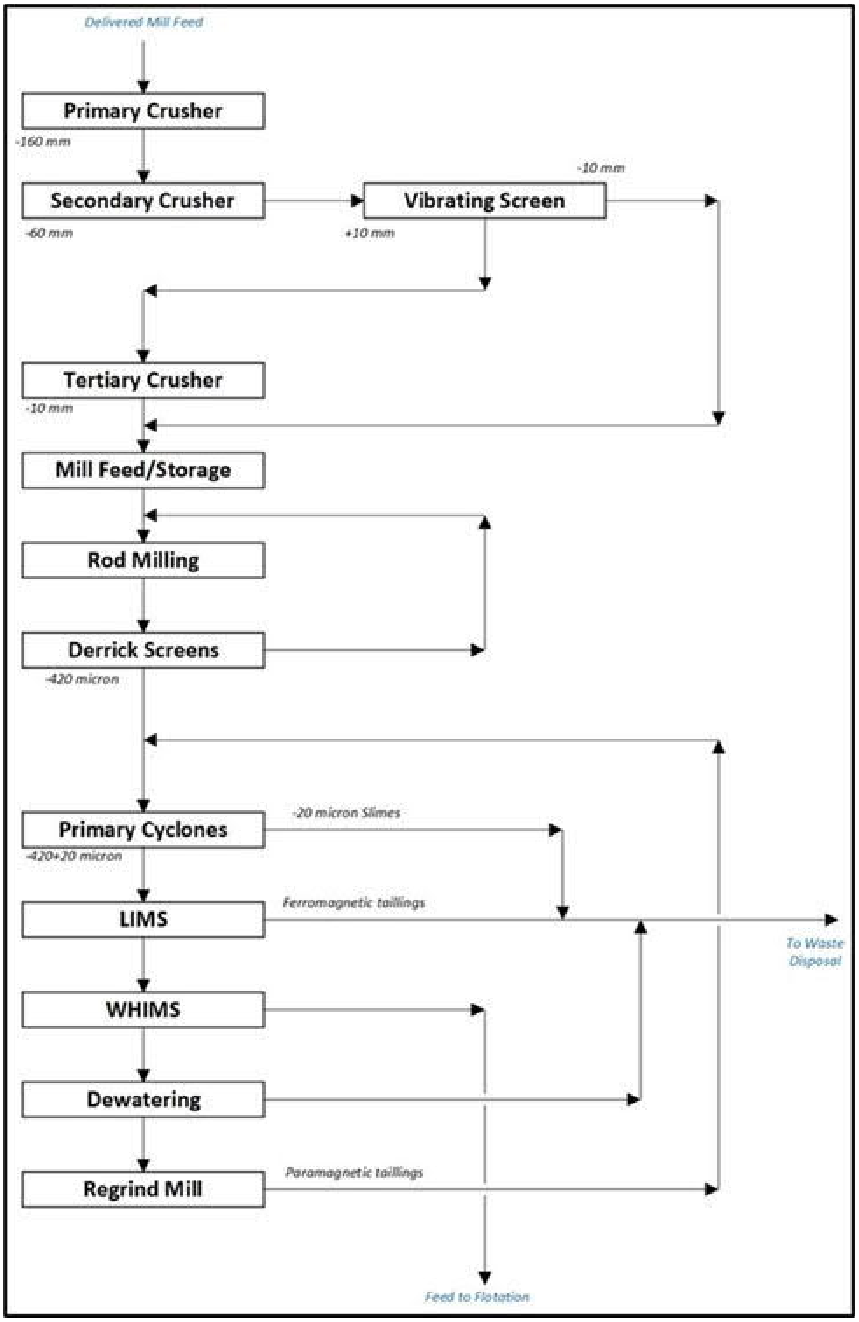


Figure 18-3: Feed Preparation and Magnetic Separation Block Flow Diagram

18.1.3.2.3 Magnetic Separation

Magnetic separation is also configured primarily into two parallel trains. A simplified block flow diagram of the proposed magnetic separation circuit is also shown in Figure 18-3.

The material from Grinding and Desliming is distributed to two Eriez type SL LIMS to remove ferromagnetic gangue, which will be discarded with the fine tailing from desliming. The nonmagnetic product will then be treated by two Eriez Type SSS-1-3000 WHIMS to remove paramagnetic gangue. WHIMS paramagnetics are collected in a pump tank and pumped to a cyclone cluster designed to classify and densify the slurry. The cyclone coarse product is reground in a 2.4 m diameter by 3.6 m long 250 kW ball mill. Ball mill discharge, at nominally 65% solids, is converted to a slurry with process water and returned to the primary cyclones to recover P_2O_5 liberated during regrinding. The cyclone fine product is rejected as waste. The removal of primary slimes and magnetic gangue reduces flotation reagent consumption and makes the flotation separation easier.

The WHIMS nonmagnetic product, which is too dilute to be efficiently conditioned with flotation reagents, will be pumped to dewatering cyclones located upstream of the pre-conditioning step in the Flotation area.

18.1.3.2.4 Flotation

The flotation area is also configured into two parallel operating trains, each capable of independent operation. The flotation circuit includes pre-conditioning and conditioning with flotation reagents followed by three stages of direct flotation.

The pre-conditioning step adjusts the pH of the 65% solids slurry with caustic soda and soda ash and disperses caustic starch, which acts as a depressant for iron oxides. Conditioning with anionic collectors while maintaining the pH using caustic soda follows pre-conditioning. All pre-conditioner and conditioner tanks are series-connected vertical stirred tanks. Slurry exiting the conditioners will be diluted with process water and fed to the rougher flotation circuit.

Each flotation circuit has two parallel rougher flotation column cells followed by two stages of cleaner flotation utilizing two parallel column cells instead of mechanical flotation cells. The cell underflow (tailings) from the rougher columns is collected in a single pump box and then pumped to the TMF. The froth product (concentrate) from the rougher columns is collected and pumped the first cleaner circuit. This circuit consists of two series-connected stirred tank conditioners and two parallel flotation column cells. The first cleaner concentrate will be pumped to the second cleaner circuit that is configured similarly to the first cleaner circuit. The froth product from the second cleaner is final concentrate, which will be pumped to product dewatering and grinding upstream of the concentrate pumping station.

The underflow from the second cleaner circuit (cleaner 2 tails) will be pumped to a cleaner dewatering cyclone cluster to reject low grade fine particles to the tailing. The coarser fraction of the dewatered material will contain some partially reagentized phosphate that can be recovered by adding more reagents and repeating the flotation stages; thus, maximizing the P_2O_5 recovery of flotation.

A simplified block flow diagram of the proposed phosphate flotation circuit is shown in Figure 18-4 below.

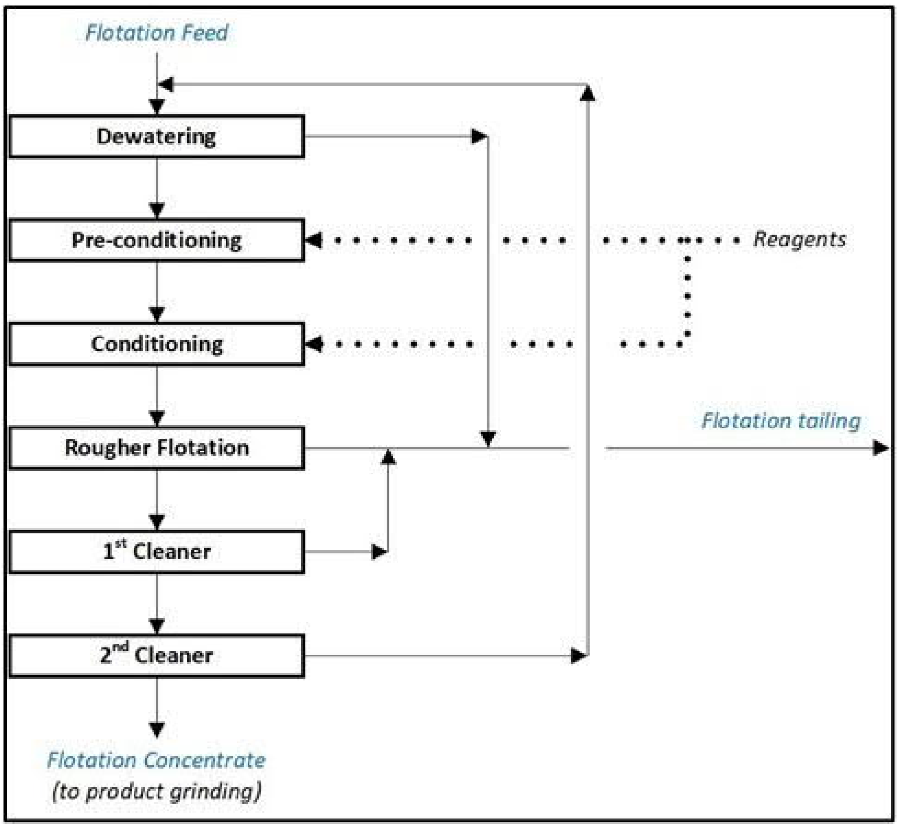


Figure 18-4: Flotation Block Flow Diagram

18.1.3.2.5 Water Recycle

Water from settled beneficiation tailings and also from the open pit dewatering pumps will be discharged into the TMF for clarification and to provide make-up water for the beneficiation process and to offset the water exported to the FCC via the slurry pipeline. The TMF reclaim water will be returned to the plant by barge-mounted, centrifugal pumps. These pumps will feed a HDPE pipeline, which discharges the reclaimed water into the fire water tank which overflows to the process water tank. The process water tank will also receive the overflow from the concentrate thickener.

The fire water tank is sized for four hours of storage. An electrically driven fire water pump, a diesel driven spare, and an electric jockey pump will be required to supply fire water via an underground (sub-surface) loop.

Management of the mine site water in the TMF is designed to recycle and reuse virtually all of the beneficiation plant process water. Snow melt in April and May distorts the water balance in the impoundment area, but the excess water is not expected to require release.

Table 18-1 shows the annual average water balance within the beneficiation process, as derived from the mass balances on the process flow diagrams.

Table 18-1: Annual Average Water Balance for Beneficiation

Process Stream	PFD Stream Number	Annual Average Quantities				
		Mass			Water Volume	
		t/hour	ktpy year	Pct.	m^3/hour	M m^3/year
Mill Feed	8	462	3,603		75	0.587
Recycle Process Water	105	0	0		3,884	30.299
Pipeline Thickener Recycle	94	0	0		114	0.892
Total Input		**462**	**3,603**		**4,074**	**31.777**
Concentrate to Thickener	83 + 88	181	1,412	39	215	1.679
Process Water to Thickener	90 + 93 + 95	0	0	0	47	0.368
Disposable Tailings	101	227	1,772	49	2,285	17.826
High Nb Tailings	61	54	419	12	1,526	11.904
Total Output		**462**	**3,603**	**100**	**4,074**	**31.777**

18.1.3.2.6 Reagent Storage, Preparation and Dosing

The function of the reagent tank farm is to receive reagents and prepare, store, and metre stock solutions of the reagents required for flotation. Most reagents will be received at the mine site in tank trucks. Fatty acid and sarcosine will be delivered in rail cars which will be off-loaded at the FCC and loaded into trucks for transport to the mine site. Basic data for reagents are given in Table 18-2 below.

The reagents used in the flotation process are as follows:

- Fatty acid soap. This is a phosphate collector used in the rougher conditioners.
- Sarcosine. This is a collector for phosphate as well as a depressant for dolomite and calcite. Sarcosine also has froth depressing characteristics. It is used in the rougher and cleaner conditioners.
- Starch. This is an iron depressant used in the pre-conditioners and cleaner conditioners. The starch is prepared with a solution of water and caustic soda.
- Soda ash. This is a pH modifier used in the pre-conditioners.
- Caustic soda (sodium hydroxide). This is also a pH modifier and is also used to saponify the fatty acid and to causticize the starch.

Table 18-2: Beneficiation Plant Reagents

Reagent		As-Received		As-Used
Name	Description	Vehicle	State	
Soda Ash	pH Modifier	By Truck	Dry Powder	10% solution w/Water
Starch	Iron Depressant	By Truck	Dry Powder	Causticized Solution w/Water
Caustic Soda	pH Modifier	By Truck	50% Solution	10% Solution w/Water
Fatty Acid	Phosphate Collector	By Truck	Liquid	Causticized Solution w/Water
Sarcosine	Phosphate Collector	By Truck	Liquid	10% solution w/Water

Soda ash and starch will be stored as solids in 53 m^3 storage silos. As required, each of these reagents is transferred to a mix tank where the solids are dissolved in process water to make a stock solution and then pumped to a use tank for metering into the process.

Caustic soda will be received as a nominal 50% solution and stored in a dedicated storage tank. As required, this will be transferred to a mix tank and combined with process water to make a stock solution and then pumped to a use tank for metering into the process.

Fatty acid and sarcosine are also received as liquids. Fatty acid will be pumped into a storage tank and then transferred to a mix tank where caustic and process water are added to make a soap solution. The fatty acid soap is then pumped to a use tank for metering into the process. Sarcosine is pumped into a storage tank and then transferred to a mix tank where process water is added make a solution. The sarcosine solution is then pumped to a use tank for metering into the process.

18.1.3.2.7 Concentrate Grinding

There are two separate and limiting conditions for slurry transport:

- Excess fines give the slurry a paste-like consistency and limit the maximum concentrations that can be achieved with a thickener and subsequently transported.
- Coarse oversize can lead to abrasive wear at the pipe bottom if the line velocities cannot be kept high enough.

The solid line in Figure 18-5 below represents the particle size distribution (PSD) of the as-produced concentrate from beneficiation. Key comments by Ausenco-PSI concerning the PSD are:

- The top size of nominally 5% >300 µm fines is manageable without the need for grinding.
- The quantity of <40 µm fines in the as-produced concentrate (nominally <10%) is insufficient to support coarser particles and prevent laminar flow.

Ausenco-PSI recommends that the feed to the slurry pipeline contain 30-40% passing 40 µm. This is a sufficient quantity of fines to create a homogeneous slurry for pumping, and low enough not to impact slurry rheology. As a result, a ball mill was added to grind the concentrate to nominally 35% passing 425 µm. The projected PSD of the ground concentrate is represented by the dotted line in Figure 18-5.

To achieve the requisite concentrate PSD, concentrate from the flotation area is pumped to densifying hydrocyclones mounted above a 3.5 m diameter by 3.5 m long 800 kW ball mill operating in open circuit. Hydrocyclone overflow solids and ball mill product are fed to the concentrate thickener. Thickener underflow, at 55% solids, is pumped to the PSI agitated storage tanks, while the clarified overflow water is returned to the recycle water system.

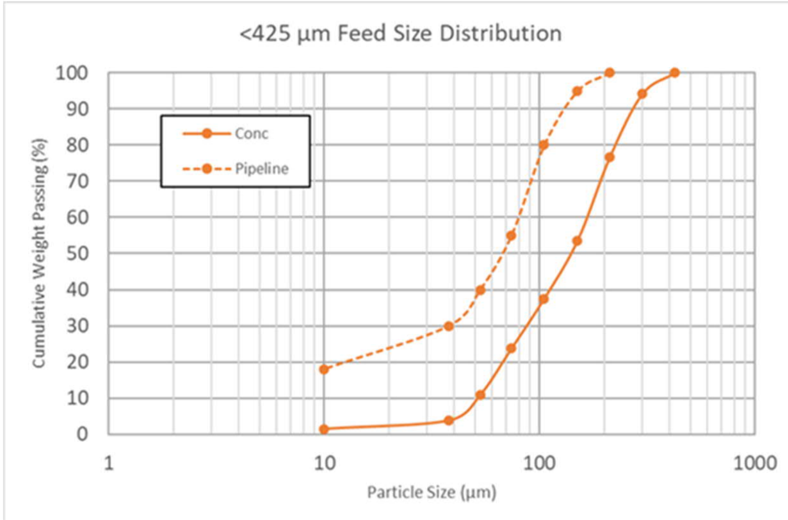


Figure 18-5: Concentrate PSD (as-produced and ground for pipeline slurry transport)

18.1.3.2.8 Pollution Controls and Waste Disposal

Beneficiation plant wastes will be stored in the TMF, and the decanted water will be returned for use as process water.

Beneficiation processes nominally 3.5 mtpa of residuum to produce 1.4 mtpa concentrate product (equivalent to nominally 526,000 P_2O_5 tpy) while generating approximately 2.2 mtpa of waste material. The annual quantities of the individual waste streams exiting the beneficiation plant are given in Table 18-3 below.

Table 18-3: Beneficiation Plant Waste Streams

`Process Stream	**Annual Average Quantities**				
	Mass			**Volume**	
	t/hour	**t/year**	**Pct.**	**m³/hour**	**M m³ /year**
High Nb Tails(1)	**53.7**	**418,860**	**19.1**	**1,543.5**	**12.039**
LIMS Magnetic Rejects	67.6	527,280	24.1	104.5	0.815
WHIMS Rejects (Regrind Mill O/F)	23.7	184,860	8.4	81.9	0.639
Pri. Cyclone O/F (Fine Tailings)	135.9	1,060,020	48.4	1,433.9	11.184
Scavenger Cyclone O/F(1)	0.0	0	0.0	738.5	5.760
Subtotal (Disposable Tailings)	**227.2**	**1,772,160**	**80.9**	**2,358.8**	**18.399**
Total Combined Tailings Waste	**280.9**	**2,191,020**	**100.0**	**3,902.3**	**30.438**

(1) High Nb tailings are comprised of rougher tails and cleaner 1 tails.

The fine tailings are combined with the magnetic rejects from LIMS and WHIMS and are pumped to the TMF where natural sedimentation will consolidate them. Over the life of the mine, wastes from the beneficiation process and tailings will be stored in retention cells contained by engineered embankments (berms). An initial tailings impoundment area or cell will be constructed that has a perimeter of approximately 4,400 m and a maximum height of 11 m. Ultimately, five tailing cells will be required with a combined perimeter of 8,200 m with a maximum embankment height of 19 m. Of the five cells required, two will be dedicated for high Nb tails storage. It is envisioned that the Nb rich tailings will be stored in the eastern cells of the TMF so that they may be processed later to recover this element should an economical means of doing so be found.

The main crushing areas are enclosed, and the mill feed storage area is covered therefore, there are no air emissions except for fugitive dust at conveyor transfer points. The moisture content of the mill feed exceeds 10% and consequently dust is not expected to be problematic.

18.1.3.3 Production Schedule

A summary of the life-of-mine (LOM) production schedule giving the mill feed tonnage (dry basis) and grade, concentrate tonnage, and total tailings tonnage (plus high niobium option) are presented in Table 18-4.

Table 18-4: Beneficiation Plant Output

Year	Mill Feed		Concentrate		Tailings (kt)	
	kt	% P_2O_5	kt	% P_2O_5	High Nb	Total
1	2,128	23.03	974.1	37.28	220.7	1,154.03
2	2,533	25.78	1,337.0	37.28	228.6	1,195.56
3	2,827	24.70	1,411.8	37.28	270.5	1,414.93
4	2,853	24.50	1,411.8	37.28	275.6	1,441.43
5	2,655	25.81	1,411.8	37.28	237.7	1,243.34
6	3,124	22.81	1,411.8	37.28	327.3	1,711.76
7	2,970	23.75	1,411.8	37.28	297.9	1,558.12
8	2,936	23.81	1,411.8	37.28	291.4	1,524.12
9	3,916	18.87	1,411.8	37.28	478.8	2,504.35
10	3,637	20.10	1,411.8	37.28	425.4	2,224.81
11	2,746	25.27	1,411.8	37.28	255.2	1,334.51
12	2,680	25.56	1,411.8	37.28	242.5	1,268.22
13	3,516	20.58	1,411.8	37.28	402.3	2,104.06
14	3,327	21.65	1,411.8	37.28	366.2	1,914.99
15	2,762	25.21	1,411.8	37.28	258.2	1,350.24
16	3,351	21.50	1,411.8	37.28	370.8	1,939.06
17	3,454	21.05	1,411.8	37.28	390.5	2,042.43

Year	Mill Feed		Concentrate		Tailings (kt)	
	kt	% P_2O_5	kt	% P_2O_5	High Nb	Total
18	3,853	19.24	1,411.8	37.28	466.7	2,440.86
19	2,976	23.72	1,411.8	37.28	299.1	1,564.45
20	3,629	20.25	1,411.8	37.28	423.9	2,217.06
21	3,834	19.25	1,411.8	37.28	463.1	2,422.10
22	3,646	19.97	1,411.8	37.28	427.1	2,233.97
23	4,508	16.92	1,411.8	37.28	592.0	3,096.33
24	4,255	17.76	1,411.8	37.28	543.6	2,843.25
25	3,668	19.99	1,411.8	37.28	431.4	2,256.27
26	1,823	15.33	504.0	37.28	252.3	1,319.47
LOM	**83,607**	**21.48**	**35,286.3**	**37.28**	**9,238.7**	**48,319.7**

18.1.3.3.1 Consumptive Use Estimate

The estimated unit consumptions of reagents utilized in beneficiation is given in Table 18-5.

Table 18-5: Beneficiation Plant Reagent Consumption

Reagent	Consumption
	kg/t feed[1]
Soda Ash	3.60
Starch	0.80
Caustic Soda	0.60
Fatty Acid	0.50
Sarcosine	0.30
[1] Flotation Feed	

The estimated electric power consumption of the beneficiation plant is approximately 23.3 kWh/t mill feed.

An estimate of the workforce required for the operation of the beneficiation plant is shown in. Table 18-6.

Table 18-6: Beneficiation Plant Workforce Estimate

Workforce Category	No. of Personnel
Management/Supervision	6
Operations	21
Maintenance[(1)]	5.5
Technical	4
Other	7
Total (Rounded up)	**44**

[(1)] Personnel to be split with Mining.

18.1.4 Utilities

18.1.4.1 Potable water

Potable water will be provided to the site from a contractor constructed well, to be drilled at a location strategic to the final confirmation of the mine site infrastructure. The well will be constructed in compliance with Regulation 903 for Well under the Ontario Water Resources Act, R.S.O. 1990, c. O.40. The water obtained from the well will be tested and treated for safe use for site ablutions, sanitary facilities, and emergency showers.

All drinking water will be provided from an external supplier in exchangeable and reusable containers. A supply will be maintained at the mine site including an inventory for emergency use.

The well will be capped with a distribution pump, protected from extreme weather conditions and freezing. Water will be circulated to the site buildings through buried, and heat traced, distribution lines.

18.1.4.2 Sewage treatment

Sewage and waste water treatment will be managed by a centrally located, vendor supplied (off the shelf), fit-for-purpose, system comprising a self-contained biofilter tertiary wastewater unit. This system will operate year round using a low energy, foam bacterial medium to break down and remove contaminants. Treated water will be compliant for quality for discharge to the mine tailings management facility and for reuse. Periodic pumping out of residual waste will be conducted by a qualified contractor for off-site disposal in a sewage treatment facility.

18.1.4.3 Fire Protection

Protection of all site infrastructure against the risk of fire across the site will be analyzed in greater detail in the next phase of study as part of a full risk assessment during design. As a minimum the beneficiation plant is equipped with a diesel-driven fire water package system that is supplied from the fire water tank located in the beneficiation building. Other buildings will be protected by standard fire extinguishers unless also deemed to be of sufficient risk and size to be serviced by the primary fire water system.

18.1.5 Ancillary Buildings & Services

18.1.5.1 Mine And Administration Offices

The mine and administration offices will consist of the following combined infrastructure:

- Administration offices.
- Mine operations offices.
- Mine changing and washroom facilities (dry) segregated for male and female personnel.
- Canteen and kitchen facilities.
- First aid treatment room.

This building will be a modular assembly mounted on blocks laying on pre-prepared compacted fill and is envisioned as a single building to reduce construction requirements as well as improve overall energy usage. It will also reduce the requirement for travel between buildings in inclement weather conditions. The total office space and number of changing and washroom facilities required will be determined from an assessment of the total on-site workforce in later stages of study.

Offices for maintenance personnel, beneficiation plant operators and others will also be included in the design of the other infrastructure as needed.

18.1.5.2 Warehousing

The site will provide for warehousing facilities which will be divided as follows:

- A warehouse, managed part time, which will be in close proximity (or attached) to the administration-mine offices building and will be stocked with everyday consumables required by all personnel. This will include personal protective equipment, office consumables and smaller items required in the processing, maintenance and open pit operations. Inventory will be monitored and issue controlled. This building will also house smaller office-workshop areas for electricians and millwrights.
- A larger, separate building divided into hot and cold storage areas primarily dedicated to the safe storage of critical spares for all areas. The heated section will house large equipment, which cannot be frozen and will include pumps, major electrical equipment and any critical items identified in the beneficiation plant. The cold section will house large equipment to protect from exposure to all weather conditions, though where freezing conditions will not cause damage. This building will also store spares for mobile equipment.

An approximation of the footprint for these buildings has been made to support cost estimation at this stage. The final sizes of these buildings, together with the requirements for critical spares and the degree of protection required, will be determined by advancing designs in a future study phase.

Other supplies (mostly bulk) which are not susceptible to being exposed to all weather conditions will be stored in the warehouse yard which will be maintained frequently to ensure safe access. This will include pipes and reels of electrical cable.

18.1.5.3 Mobile Equipment Maintenance Facilities

A pre-engineered, maintenance facilities building will be present on-site with an overhead clearance of at least 16 m to allow room for the largest equipment (haulage trucks) and overhead cranes. The building will be erected in a convenient location in proximity of the other mine infrastructure with ease of access from the open pit. The facility will provide preventative maintenance and repair, tire handling, machine welding, and washing services for the mobile equipment, as well as office space, clean and dry areas and equipment parts storage.

The maximum size of this building will be comprised of two (2) truck bays servicing haulage trucks and production wheel loaders, two (2) heavy duty bays servicing support mining equipment such as dozers, graders, and support wheel loaders, one (1) wash bay, one (1) tire handling and machine welding bay, an office area and a parts storage area. Production drills and shovels will be serviced on the field as required.

Determination in a future phase of study will be made for the most suitable location for the maintenance of smaller, lighter mobile equipment as well as the main mining fleet. At this time, it is assumed that all mine site mobile equipment will be serviced and maintained in a single facility which may require additional bays specific for the smaller equipment.

18.1.5.4 Fuel storage

Only diesel fuel will be stored at the mine site. A self-contained and spill protected above ground tank will be provided in a strategically located area, separate from other infrastructure and for ease of access for primary users. This refueling facility is intended to be used by fuel trucks supplying the main fleet of production mining equipment in the pit. Other miscellaneous mobile equipment will use the same refueling facility in periods of lower demand. This facility will be equipped with a regular dispensing nozzle for smaller equipment and a fast-fueling nozzle (Wiggins-type) system for the site fuel trucks.

Resupply will be contracted through a local fuel provider and will be a frequent traveler on the mine access road as a result. Any vehicles which are approved for off-site highway use and require gasoline will be refilled in Hearst prior to departing this location.

A 125,000 L storage tank has been included for diesel fuel with the provisional expectation that this will need refilling at least twice per week.

18.1.5.5 Trade Shops

Two small trade shops are planned, one for electricians and one for millwrights. These have not been described in detail at this phase though will have the following features:

- Located within the main warehouse building.
- Workbenches.

- Sufficient lighting.
- Storage racks and shelving for smaller consumables.

18.1.5.6 *Laboratory Facility*

The details of the equipment required in the laboratory facility is yet to be finalized though an allowance has been made to both provide for the facility and equipment. This facility will be located in close proximity to both the beneficiation plant and the drillhole core logging building. The laboratory will serve as an analytical check on samples as part of process controls from the beneficiation plant and to process core samples from site drilling within the mining area. It will also serve to analyze additional exploration drill cores from Anomaly B and Anomaly C subject to quality and data integrity requirements.

As a minimum the laboratory will contain the following equipment:

- Twenty to 30 linear feet of bench space to work upon.
- One to two standard/conventional microscopes.
- Ro-tap sieve stack.
- Muffle furnace.
- Two ovens for drying.
- Pressure filters – bench type.
- Float machine – bench type.
- Access to plant compressed air.

18.1.5.7 *Site Security and Site Entrance*

The site entrance will be established on the access road south of all mine infrastructure. This facility will be staffed 24 hours a day and 7 days per week. Personnel have been included in the staffing plan, though it will be determined in future if this service will be provided through an impartial and experienced third-party contractor.

A site security building (currently envisioned as a fit for purpose trailer unit) will be erected in the center of the road with gated entrance and exit either side. The duties of security will include monitoring communications and vehicle movement on the Fushimi access road, monitoring key site infrastructure, logging of equipment in and out of the mine site, emergency incident response, site orientations and inspections of vehicles entering and exiting the site.

Some third-party services also provide medical services from advanced first responder training for their staff.

Future studies will determine the need and extent of site perimeter fencing both to manage site security as well as a deterrent to wildlife encroachment.

18.1.5.8 *Explosives Magazine*

Explosives used during mine drilling and blasting operations will be stored in a secure (fenced) facility segregated from all other mine infrastructure in accordance with the Explosives Act RSC.' 1985, c.E-17, Explosives Regulations, 2013 (SOR/2013-211), Explosives – Quantity Distances (CAN/BNQ 2910-510/2015) and R.R.O 1990, Reg 854: Mines and Mining Plants. All explosives will be stored in approved bunkers.

This facility will have locked and restricted access and will be lighted and outfitted with an alarm system and a dedicated access road. Explosives will be delivered on an agreed schedule by an approved vendor via trucks on the access road. The storage facility will be sized according to the requirement to maintain an inventory that meets mine operational demand.

18.1.5.9 *Core Logging and Storage*

A separate facility will be provided for the processing of drill core from the mill feed definition drilling and further site exploration. This will be supplied as one or more custom built trailer units mounted on a rough, compacted foundation. The facility will consist of work benches for the placement of core boxes, storage areas for materials and a separate office. A room will also be provided for cutting of core using a diamond saw. Power, potable water (for core cutting) and internet access will be provided.

Sufficient room outside of the core logging room will be provided for the temporary storage of core in racks. A permanent, larger, core storage area will be designated in another part of the Mie site and core to be stored in this area will be moved using a site forklift or wheel loader with forks.

18.1.5.10 *Communications Tower*

A wireless communications tower will be installed at the site as one of the first pieces of infrastructure at the start of construction, to enable both verbal communications and internet access. If anchoring into bedrock at a suitable location is not achievable, the tower will sit on a concrete foundation and will be self-powered by solar panels with backup energy cells.

Connections to key site infrastructure will be via receivers mounted on the outside of the buildings. Phone system will be VoIP. Dependent upon the arrangement with the vendor, this tower will be managed on a monthly, “supply as a service” agreement after initial erection and site setup costs.

The tower will enable external communications. Further study will need to determine if a second relay tower is required to connect to existing communications infrastructure in the Hearst area.

A separate, local, mine site radio system will also have to be established for communications between personnel.

18.1.5.11 *Helipad*

The mine site will provide for an all-season helipad which will allow a medivac helicopter to land and take off in the event of a site emergency. The pad will be located sufficiently separate from, though in close proximity to, the main administration and office building. Construction will be in accordance with Transport Canada Standard 325 and the access road to the helipad will be maintained as a priority year round.

18.1.6 Site Preparation

The process for initial site preparation has been well described in previous reports and the approach to the initial excavation to prepare the site for construction has not substantially changed in this update.

All site infrastructure will be located on a low relief terrain consisting of saturated shallow muskeg overlaying deep and impermeable glacial till. A black spruce forest covers much of the mining lease terrain.

Site preparation for infrastructure will focus on the following activities:

- Diversion of impacted creeks.
- Preparation of site roads.
- Preparation of site infrastructure foundation locations.
- Preparation of initial waste facility area (initial draining though muskeg will remain in place).
- Preparation of starter pit area.
- Preparation of tailings facility (initial berm locations only).
- Establishment of pipeline corridors.
- Protective berms to prevent flooding (around site infrastructure area and other locations as needed).

The preparation of the site will require removal of the forest and muskeg and water ingress prevention management through the construction of impermeable berms.

The following assumptions have been made with respect to site preparation:

- Site preparation work will not be possible until the all-weather road is completed.
- Tree cutting can occur without drainage of the muskeg.
- The muskeg horizon will not freeze completely in the winter months due to snow cover and organic heating from the muskeg itself.
- Muskeg cannot be effectively excavated until most of the contained water has been drained.

- The most effective means to manage the dewatering process is to utilize the natural gradient of the land to collect water in lower lying areas.
- The glacial till is an impermeable layer and all water drained from the muskeg will flow across it.
- The drained water will be contaminated with sediments and organic matter though can be directed to naturally lower lying areas for disposal.
- The period to establish the drainage system will commence in the winter months when the muskeg layer is more manageable with lower water flows.
- The requirement for material for berms and other barriers to water inflow is available from site stripping activities or, initially, from a nearby borrow pit as pit-run material.
- The battery limit for site preparation is limited up to the glacial till layer. The supply and installation of engineered fill, and other work to prepare building foundations, is part of construction.
- The duration of the initial site dewatering is no longer than one year (source: PhosCan PFS), though will be dependent upon the seasonal conditions when this work starts.

The sequence will follow these steps upon site access completion of the all-weather road and tree cutting:

1. The equipment to be used for dewatering activities will be determined by the contractor employed to do the work and mobilized to the mine site.
2. Creation of trenches within the muskeg and excavated to the glacial till horizon channel direct water to collection sumps. The location and size of the trench network will be determined in a future phase of study.
3. Concurrent establishment of collection sumps to receive water directly from the drainage trenches in lower lying areas. The design, location and size of the sumps will be determined in a future phase of study.
4. Water collected in the sumps will be retained for a short time to enable heavy sediments to settle and then pumped through pipelines to locations which will not drain back to the excavated areas. Management of organics will be required to prevent clogging of pumps. The type and size of the pumps will be determined in a future phase of study, although each pump will have a self-contained in-built diesel fuel supply to allow independent operation and strategic placement of sumps.

Perimeter berms will be constructed around drained areas to prevent flow of water from areas which are not being excavated. The berm locations and extent will be determined in the field dependent upon the conditions which arise during excavation.

Although there are no major water courses cutting through the site which will be impacted by mining activities the location of trenches will also need consideration of smaller creeks traversing the site as well as a natural, low-gradient, watershed which cross the site diagonally from southwest to northeast (refer to Section 18.1.8 “Site Water Management”).

Further site preparation and draining of mining areas is addressed in other sections of this report.

18.1.7 Tailings Impoundment

The ringed tailings management facility (TMF) has been sited to the south of the mine site with the northwest edge approximately 50 m away from the beneficiation plant and has a footprint which lies over generally flat ground. The subsurface conditions at the TMF are consistent with the rest of the mine site and generally comprises of one to two metres of muskeg underlain by dense to very dense silt and a bedrock below of glacial till deposits. Sparse pockets of silty clay to clayey-silt may also be present in the foundation within the footprint of the TMF. The groundwater level appears to be generally close to the terrain surface. Beyond storing the tailings solids generated by the concentrating process in the beneficiation plant, the TMF is considered in the Project as the ultimate management facility for excess water from all areas and allowing residence time, if required, to comply with water quality limits prior to discharge to the environment.

Two streams of slurry tailings, bulk tailings (from the primary mining process) and niobium (Nb) rich tailings and a separate output, will be transported to the TMF via slurry pipelines from the beneficiation plant and managed at the TMF within separate adjacent cells. This will enable the future potential reprocessing of Nb rich tailings in the event that an economical means to do so is found. Tailings slurry will have approximately 9% wt. solids at discharge.

The tailings characteristics are not known at this time though it is assumed to eventually consolidate to a solids content (Cw) of 66%. Decant water from the TMF will be returned to the beneficiation plant for reuse via a common reclaim water pipeline. An independent pumping system will be installed at the TMF cells for bulk and Nb rich tailings to transport reclaim water to the beneficiation plant via a single pipeline.

The TMF containment facility will be housed within zoned embankments which will be up to 20 m high and a constructed as a typical cross section comprising of compacted till with an internal drainage system for seepage control. The internal drainage system will connect a chimney drain to a downstream basal drain underlying the full extent of the downstream glacial till shell. The embankments will have an upstream face lined with HDPE geomembrane along with a compacted upstream shell of till fill keyed into the native glacial till for further seepage control within the foundation.

Native muskeg will only be removed down to the underlying glacial till for construction of the ringed embankment (berm) footprint to ensure structural integrity and to prevent seepage through to the natural environment. Within the TMF containment cells, tree cutting will take place although the muskeg will be left in place with no requirement to be removed and which will be a considerable construction costs and schedule saving. This muskeg is expected to compress over the life of the TMF under the weight of the tailings slurry. Construction dewatering for the containment embankments will be required as perched water within muskeg is expected, which is considered a constructability issue as well.

The TMF Is designed to contain the LOM tailings with the initial stage to provide five years of tailings storage capacity equating to 5.9 M-m^3 of bulk tailings and 1.4 M-m^3 of Nb rich tailings for a total of 7.3 M-m^3. Disposal tailings cells can be developed in three stages while Nb tailings cells can be developed in two stages. Total capacity of the TMF at its ultimate configuration will be 37 M-m^3 for bulk tailings and 8.7 M-m^3 for Nb rich tailings for a total of 45.7 M-m^3 of solid particles. The TMF is expected to maintain an operating water pond of approximately 0.5 M-m^3 through its operating life. The TMF is designed to contain an environmental design flood (EDF) which is considered as the 100-yr, 30-day snowmelt and rainfall. Internal transfer spillways will be constructed to connect the tailings cell with flood water routed to pass via a final main emergency spillway.

Closure of the TMF may be staged as each of the cells reach capacity. As the geochemical characteristics of the tailings are defined, it is currently assumed that acid rock drainage will not be a factor in the cover design and a dry cover will be installed for closure (after draining the water) which will comprise of drainage layer, topsoil and vegetation. Installation of the dry cover will take place once supernatant water is drained and the surface of tailings deposit is sufficiently dry and easily accessible to construction equipment.

18.1.8 Site Water Management

In general, water will be managed at the mine site by water quality designation. This will effectively divide water quality into two discreet categories:

- Water that has been in contact with mining and mineral processing activities, which will be considered “contact” water and will be managed in dedicated facilities to prevent unintended release to the environment.
- Water that has not been in contact with mining and mineral processing activities, which will be considered “non-contact” and will remain segregated from these processes and treated as clean surface run-off.

To the extent possible, non-contact water should be diverted away from the site to avoid any risk of contamination (and thus becoming contact water) and, as a result, minimize water management requirements. Additionally, site water will be managed in such a way that minimizes the impact on the existing regime for natural streams and creeks, (i.e., the variability of flows and depths).

The mine site lies within two watersheds: the Ridge River (to the west) and the Soweska River watersheds (to the east). As a result, site water will be managed to the extent possible by this southwest to northeast trending watershed divide so that if discharge is required, water is returned to the watershed in which it originated. The site draining plan and ultimate facility footprint is shown in Figure 18-6 below and the following features are shown:

- Ultimate footprints of the major facilities (open pit, waste facility, niobium stockpile, plant site, and tailings management facility).
- Watershed divide between the Ridge River watershed and the Soweska River watershed.
- Existing drainage works and flow directions.
- Proposed location of diversion channels north and west of the waste facility.

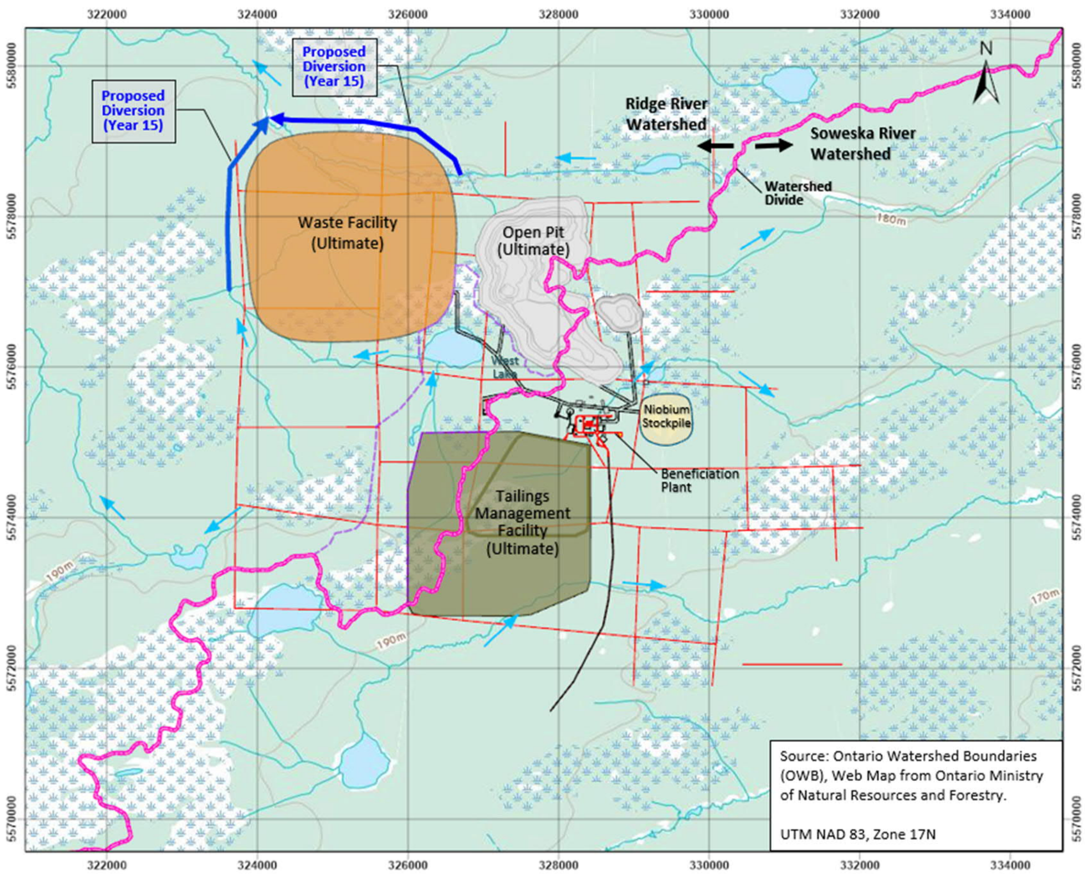


Figure 18-6: Mine Site Drainage Plan – End of LOM Footprint

Figure 18-7 illustrates a simplified site-wide water balance diagram at steady state mine production during an average climatic year. The catchment areas, as well as the annual average inflows and outflows from the major facilities, are shown on this diagram. The annual climatic data is based on Canadian Climate Normals data (1981 to 2010) from the Kapuskasing CDA climatic station (Station ID: 6073960).

The following sections summarize key assumptions and considerations for site water management.

18.1.8.1 Contact Water Collection and Effluent Discharge

Contact water collected from the major facilities (open pit, waste facility, niobium stockpile, plant site, and tailings management facility) will be reused on site for process or mining use (dust suppression, drill water, beneficiation) to every extent possible to minimize freshwater takings and to reduce the volume of water to be discharged to the environment.

Seepage losses and run-off from the contaminated sources will be collected and returned to the site contact water management facilities.

The water balance indicates that the site has a positive water balance, and consequently there will be excess water from the site operation needing to be discharged off the site in the environment. The limited water quality data currently available from work performed thus far at the mine site is not sufficient to determine if the part, or all, of the contact water is suitable for direct environmental discharge. Contact water from the potential contaminated areas must therefore be collected, monitored and treated (if required) prior to discharging to the local creeks via ditches or pipelines. Discharge from the site will need to meet all applicable environmental discharge criteria. Further analysis and testing of mine and facility contact water quality and chemistry should be carried out during the future studies.

18.1.8.2 Open Pit Dewatering

Open pit groundwater inflows are estimated by Hatch based on Report – 2012 Hydrogeological Critical Issues Study (AMEC, December 2012). The maximum groundwater inflow collected in the dewatering wells during the initial pit development is estimated at 38,400 m^3/d and the pumping requirements will decrease over time during pit operation given that the aquifer has no significant recharge source.

Pit water will be collected in the dewatering wells and the sumps at the bottom of the pit and then pumped to the surface from where water can be pumped to the TMF or the beneficiation plant based on process water demand or discharged to the local creeks if water quality is suitable for release.

As stated in AMEC's report (AMEC, December 2012), "Analysis of groundwater samples collected during the pumping tests, indicates that the groundwater is generally of good quality, and, with the exception of phosphate and ammonia, meets the Provincial Water Quality Objectives which are the standard criteria for discharging to surface water. The dataset of existing surface water quality data is poor, but generally indicates that the surface water features are also high in phosphate and ammonia, which appear to be naturally occurring in the area, and it may be possible to discharge the water to local creeks without treatment. Should treatment be required, pH adjustment to reduce the toxicity of ammonia and flocculation of phosphate using iron or aluminum would likely be sufficient. A pond with a minimum of a three-day retention time (approximately 120,000 m^3 capacity) was estimated to be sufficient to allow settling of the flocculated phosphate. This pond is estimated to cost approximately $2.5 million to develop, with an additional $0.35 million for the treatment plant."

For this PEA study it is assumed that the pit water volume resulting from a flood event (such as a 100-year rainfall or rain-plus-snowmelt event) will be transferred to, and collected within, the TMF. Further assessment on pit water discharge location should be carried out during the future studies based on the reviews of water quality data, permitted discharge criteria, risk and cost.

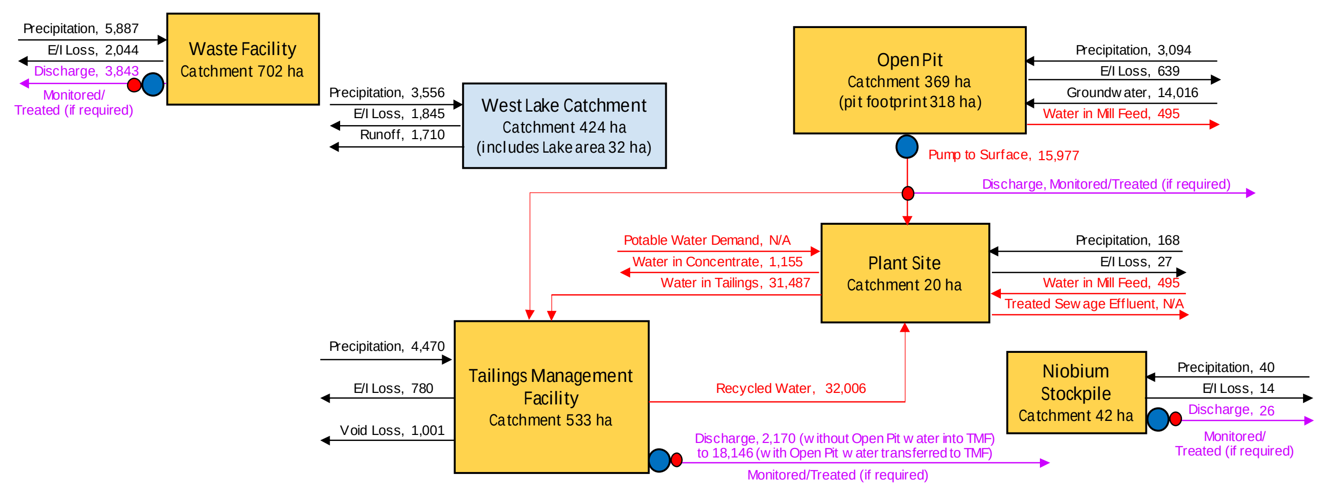
18.1.8.3 Natural Creek Diversion

Two diversion channels will be required to divert the clean (non-contact) surface runoff located in natural creeks away from the proposed waste facility footprint to avoid any chance of contamination. This includes one 2.5 km long diversion channel to the west of the facility and one 3 km long diversion channel to the north of the facility. To account for the waste facility footprint expansion and encroachment on the path of these creeks, both diversion channels will be required at a later stage of the facility development (estimated to be constructed in Year 15 of operation). The existing West Lake catchment is upstream of the west diversion channel and contributes clean surface runoff to this diversion channel.

18.1.8.4 West Lake

The West Lake can be considered as the potential source for site raw water and potable water supply. Further analysis of site freshwater demand and testing of West Lake water quality should be carried during the future studies.

Conceptual Site Wide Water Balance Diagram - Ultimate Footprint, Average Annual Water Volumes (in Km^3/yr)



Notes:

1. Unit: in Km^3/yr (thousand cubic meter per year), representing the average annual water volume at ultimate footprint during Year 3 to Year 25 operations.
2. E/I Loss: evaporation/infiltration loss.
3. Annual climatic data is based on the Climate Normals data (1981 to 2010) from the Kapuskasing CDA climatic station (Station ID: 6073960).
4. The facility layout and catchment areas are for ultimate phase development.
5. The process flows in and out the Plant site are provided by JESA.
6. Open pit groundwater inflows were estimated by Hatch based on Report - 2012 Hydrogeological Critical Issues Study (AMEC, December 2012). The 14,016 Km^3/yr annual volume represents the maximum groundwater inflow collected in the dewatering wells during the initial pit development; the pumping requirements will decrease during pit operation given that the aquifer has no significant recharge source.
7. Pit water will be collected in the dewatering wells and sumps at the bottom of the pit and then pumped to the surface; from where water can be sent to the TMF or the Plant based on process water demand, or discharge to the local creeks if water quality is suitable for release.
8. Seepage losses from the contaminated sources will be collected and returned back to the site contact water management facilities.
9. Contact water effluents from the site should be collected, monitored and treated (if required) prior to discharging to the local creeks via ditches or pipelines. Discharge from the site should meet all applicable environmental discharge criteria. Further analysis and testing of mine and facility contact water quality/chemistry should be carried out during the future studies.

LEGEND

- Clean Water Catchment
- Contact Water Catchment
- Natural inflow or outflow/losses
- Pipeline/Pumping
- Discharge from Contact Area
- Contact Water Management Pond
- Effluent Monitoring /Treatment (if required)

Figure 18-7: Conceptual Site Wide Water Balance Diagram

18.1.9 Concentrate Slurry Pipeline

18.1.9.1 Summary

Phosphate rock slurry is transported from the beneficiation plant at the mine site to receiving facilities at the FCC by a buried pipeline which follows the route of the site access road. The slurry is concentrated in a thickener within the beneficiation plant, stored in agitated tanks, and pumped approximately 86 km to the FCC. The pipeline facilities include additional storage at the FCC, cathodic protection, monitoring and telecommunication.

The PEA pipeline is identical in length and pipe diameter to what was proposed in the 2008 PFS nevertheless an increase in capacity from 1.16 Mtpa to 1.41 Mtpa has been accomplished by increasing the slurry flow rate and selecting a pump capable of operating at higher pressures, up to 250 bar.

18.1.9.2 Process Description

All of the upstream and downstream pipeline facilities, including the thickener, slurry tanks, and pump station will be inside shared facility buildings.

The concentrate slurry is thickened in a rock slurry thickener located in the beneficiation plant adjacent to the pump station. The thickener accommodates a maximum of 199 dmth of solids of concentrate and the design underflow slurry concentration will be 55% by weight. The variable speed underflow pumps (one operating and one standby) control slurry density and flow to the agitated slurry tanks.

Two centrifugal charge pumps (one operating and one standby) draw slurry from the agitated slurry tanks and provide the required positive suction pressure to the mainline pumps. These pumps can also circulate the slurry back to the storage tanks. Two mainline PD piston diaphragm pumps (one operating and one standby) provide the required discharge pressure to deliver the slurry though the pipeline to the terminal station at the FCC.

Two remote pressure monitoring stations will be included along the pipeline route. The terminal station will include two agitated slurry tanks.

18.1.9.2.1 Oversize Control

Control of slurry top size (the quantity of particles exceeding 200 µm) is critical to maintaining homogeneous slurry flowing in the pipeline. With heterogeneous slurry there is a significant concentration gradient between the top of the pipe and the bottom, which creates wear rates in the bottom of the pipe higher than has been allowed for.

The upstream grinding process enables the feed to pass 425 µm and then regrinds the concentrate to produce ground solids suitable for long distance pipeline transport. Additionally, a strainer is provided in the mainline pump suction line to prevent tramp oversize (6+ mm) from entering the pumps and pipeline. This is the mechanical limit of the piston diaphragm pumps in terms of the size of particle that will pass the internal check valves without preventing them from closing completely.

The purpose of limiting oversize is to protect the pipeline from pebbles which will ultimately cause bottom wear and early failure of the pipeline.

18.1.9.2.2 Slurry Thickener

The thickener tank and mechanism will be fabricated of steel. Up to four drive motors will be required for the rotation and lift functions on the thickener rake mechanism. The thickener underflow pumps (one operating and one standby) will have 65 kW motors powered by VFDs. The underflow concentration will primarily be controlled through a SCADA system that receives a signal from a radiation density gauge and controls the speed of the operating underflow pump to maintain a set-point concentration value.

18.1.9.2.3 Agitated Slurry Tanks

Slurry tanks will be located at the pump station and terminal facilities. For the pump station, there will be two storage tanks. Each tank has a dimension of 15 m in diameter by 15 m high to provide a combined total slurry storage time of 22 hours. The terminal station at the FCC will include identical 15 m by 15 m tanks. All tanks will be made of steel (field erected), and each is equipped with an agitator unit with a 200 kW motor.

18.1.9.2.4 Flush Water Supply System

Flushing water is required prior to a normal shut-down and the process water at the mine site via the beneficiation plant water system will be used for this purpose. Normal start-up occurs with water in the pipeline and then a transition to slurry is made once stable flows and pressures are sustained.

18.1.9.2.5 Gland Seal Water

The thickener underflow pumps and mainline charge pumps will require clean water for sealing and cooling the motor shaft gland. This water will be supplied from the beneficiation process water. In most cases, some treatment, such as a wye strainer, will be required to assure that the gland water is clean. A dedicated tank is included for the gland seal water in the pump station design to maintain a buffer supply of this clean water. Each centrifugal slurry pump will be installed with a dedicated gland seal water pump.

18.1.9.2.6 Pump Station

The mainline pump station includes one operating charge pump to supply the required suction pressure to the mainline pump. A test loop will also be installed between the charge pumps and the mainline pumps, which can be used on an as-needed basis for assessing the slurry performance in a controlled manner.

Each mainline pump is provided with an AC motor supplied by a VFD.

Ball valves are provided for pump station isolation and switching between operating pumps. A "pig" launcher will be included as an integral part of the station piping.

18.1.9.2.7 Pressure Monitoring Stations

Two pressure monitoring stations are located along the pipeline. The first station will be located near the 1 km marker on the route. This will be used primarily for tracking the pressure losses of the slurry passing through the pipeline. The second station will be installed near the middle of the pipeline between the mine site and the FCC. This second station will provide critical pressure data to the Pipeline Adviser™ and the leak detection systems.

18.1.9.2.8 Pipeline Overpressure Protection

The pipeline and equipment will be protected from over-pressurization by several levels of protection as follows:

1. Implementation of proven operating procedures.
2. SCADA system software deployment.
3. Electrical or hardware interlocks or control loops.
4. Mechanical pressure relieving devices.

The locations, sizing, and set points of the mechanical pressure relieving devices will be established during future detailed design hydraulic studies through the use of transient analyses and operational simulation. The operational procedures, such as the sequence of normal station start up and shutdowns, timing of valve operations, and so on will also be determined at that time.

The pressure relief devices will be located at the discharge of the mainline pumps and upstream of the terminal valves.

18.1.9.2.9 Cathodic Protection

A permanent impressed voltage cathodic protection system will be installed. The cathodic protection system inhibits the onset and advancement of pipe external corrosion by changing the potential difference that naturally exists between the pipe and the earth by means of impressed voltage. During pipeline construction, test leads will be installed approximately every kilometre and at river crossings, cased crossings, and foreign pipeline crossings. These leads will permit pipe-to-soil potential and other tests to determine the effectiveness of the operating cathodic protection system after the initial installation and during the operating life of the pipeline.

18.1.9.2.10 Leak Detection System

In the unlikely event of a pipeline rupture, the leak detection system, Pipeline Adviser™, and SCADA system will warn the operator, prompting the activation of an emergency shutdown sequence if the data appears valid. The leak location can be determined by the leak detection system and confirmed by field inspection.

18.1.9.2.11 Terminal Station

The terminal station has two agitated slurry tanks and will utilize metal-seated slurry ball valves to stop the flow during the shutdown. The present profile shows the high point at the terminal. If this configuration prevails, the pipeline flow will stop automatically when the mainline pump is stopped. A pig receiver will be included as an integral part of the station piping. A station isolation ball valve will be used for maintenance purposes.

Pipeline monitoring instruments such as flow meters and density meters will be installed at the terminal to verify the slurry properties as it leaves the pipe.

The pipeline scope ends at the outlet flange of the storage tanks.

The terminal station is monitored and controlled from the beneficiation plant control room.

18.1.9.3 Production Schedule

The production schedule is referenced in the beneficiation plant (Section 18.1.3.3).

18.1.9.4 Consumptive Use Estimate

Detailed OPEX information can be found in Appendix 1 of the Ausenco Report. The total power consumed by the pipeline system is 14.7 kWh/t concentrate, of which 9.2 kWh/t is for pumping and 5.5 kWh/t is for tank agitators and other minor loads.

18.2 Fertilizer Conversion Complex

18.2.1 Sulfuric Acid Plant

18.2.1.1 Summary

The Sulfuric Acid Plant (SAP) provides sulfuric acid, steam and electrical power to the FCC. The SAP is designed to export 4,200 t/d as 100 wt% sulfuric acid assuming 350 operating days per year with the acid strength varied between summer at 98% and winter at 93% to avoid freezing.

In addition to the sulfuric acid plant and attendant cooling tower, the SAP battery limits includes the molten sulfur storage and the 50 MW turbogenerator.

Steam production is 60 bar(g), 485°C in the sulfuric acid plant and received at the turbo generator to produce power. Additional steam services are available at 32, 7, and 3.5 bar (g) respectively for consumption by various units within the FCC.

Cooling water for the sulfuric acid process and the turbo generator is supplied internally via a cooling tower package.

18.2.1.2 *Process Description*

This section provides an overview and summary of the SAP process. It has been extracted, for the purpose of this report, from a more comprehensive description of the process produced in support of this document. This plant is designed to export 4,200 t/d as 100 wt% sulfuric acid assuming 350 operating days per year.

The SAP is a sulfur burning plant and produces strong sulfuric acid from molten sulfur supplied as a bulk commodity.

The overall process consists of the following major steps:

1. Molten sulfur is delivered by rail car and unloaded into a sulfur distribution pit.
2. After unloading, the sulfur is filtered prior to sending to a storage tank and from there it is pumped to the sulfur furnace for combustion.
3. Clean molten sulfur is combusted with dry air to produce gaseous sulfur dioxide:

   $S + O_2 \rightarrow SO_2$

4. Sulfur dioxide is converted in several stages over catalyst beds into sulfur trioxide:

   $SO_2 + ½ O_2 \rightarrow SO_3$

5. Energy released is recovered to produce steam.
6. Sulfur trioxide is absorbed in strong sulfuric acid with dilution water added to control acid strength:

   $SO_3 + H_2O \rightarrow H_2SO_4$

7. Strong acid is cooled to maintain the heat balance in the acid circuit.

Rotating pieces of equipment in the plant are motor driven, however steam turbine-driven equipment can also be considered for major equipment (main blower and boiler feed water (BFW) pumps) because it is more energy-efficient to produce power using a dedicated steam turbine.

18.2.1.2.1 Sulfur System

Molten sulfur received from rail cars is collected by a steam heated trench to a sulfur pit which is provided with steam heating coils to maintain temperature. The liquid sulfur unloading pump sends the molten sulfur to a polishing filter and the cake collected off the filter will be periodically discharged to the filter sludge bin for disposal.

Filtered sulfur is sent to the clean sulfur tanks which are provided with steam coils to maintain temperature. The capacity of the two storage tanks is equivalent to 11 days of hold up at the SAP design capacity.

The clean sulfur tank pump feeds molten sulfur to the sulfur furnace for combustion using steam coils around the piping to keeps the sulfur at 130-140°C in order to maintain minimum viscosity conditions.

18.2.1.2.2 Gas Contact System

The schematic for the sulfur burning sulfuric acid plant with an ALPHA™ ("alpha") heat recovery system is shown below in Figure 18-8.

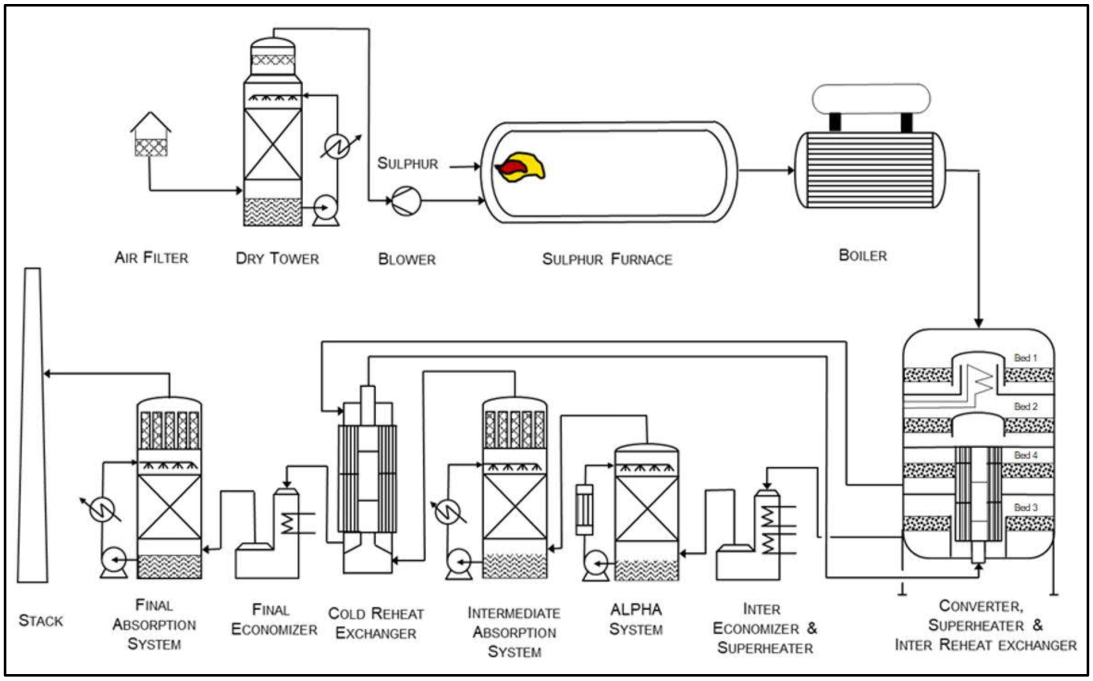


Figure 18-8: Typical Chemetics Sulfur Burning Sulfuric Acid Plant with ALPHA

Ambient air is drawn through the air filter and then pulled through the dry tower into the suction of the blower. The air is dried in the dry tower by counter current contact with a stream of 98.5% acid. Acid mist carryover from the dry tower is reduced by means of a mesh-pad type mist eliminator.

Air leaving the dry tower is pulled through the blower and into the sulfur furnace which is a horizontal cylindrical carbon steel vessel lined with refractory and insulating firebrick. The lining and insulation protect the shell from high temperatures generated when burning the sulfur.

At the front of the sulfur furnace there are steam jacketed sulfur atomizers which pressure atomize the molten sulfur as it is sprayed into the furnace body. At elevated temperatures, the atomized sulfur instantly combusts in the presence of the air from the dry tower.

The combustion in the sulfur furnace produces a hot gas mixture of about 11.5% SO_2. This hot gas is cooled in the boiler producing high-pressure steam at about 65 bar(g). The SO_2 gas from the furnace proceeds to the converter to produce SO_3. this unit consists of four catalyst beds arranged vertically in the order of bed 1, bed 2, bed 4, bed 3 at the bottom. The Chemetics converter design incorporates internal heat exchangers leading to the unique bed numbering arrangement.

The gas enters at bed 1 and an exothermic conversion of SO_2 to SO_3 raises the gas stream temperature. The gas that exits bed 1 is cooled in the superheater, located inside the converter, before entry to bed 2. The additional conversion of SO_2 to SO_3 in bed 2 again raises the temperature requiring the gas to be cooled in the inter reheat exchanger (also located inside the converter) before entering bed 3. The gas when it leaves bed 3 has about 95% of the SO_2 converted to SO_3.

The hot gas exiting from bed 3 is cooled to the desired temperature in the inter economizer and superheater which are connected in series. Gas then enters the alpha tower where approximately 95% of the SO_3 is absorbed in strong acid at elevated temperatures which enables production of low-pressure steam.

The gas exiting this tower is sent to the intermediate absorption tower system where the remaining SO_3 is absorbed by counter-current contact with circulating 98.5% H_2SO_4. Acid mist carryover from the intermediate absorption tower system is reduced by means of Brownian-diffusion candle mist eliminators located at the top of the tower.

The gas then exits the intermediate tower absorption system and is reheated in two stages in the reheat exchangers before being returned to bed 4 in the converter. Residual SO_2 is converted to SO_3 in bed 4 to achieve an overall conversion of more than 99.8% of SO_2. After passing through bed 4, the gas is cooled to the desired temperature in the cold reheat exchanger and the final economizer before entering the final absorption system. Here the SO_3 formed in bed 4 is absorbed by counter-current contact with circulating 98.5% H_2SO_4. Acid mist carryover in the tail gas is reduced by means of Brownian diffusion candle mist eliminators before being discharged to atmosphere via the plant stack. The gas leaving the acid plant will have a low concentration of SO_2 and is within acceptable Ontario regulatory standards for emission to the atmosphere.

18.2.1.2.3 Acid Circulation System

The main purpose of the acid circulation system is to remove moisture and SO_3 from the gas flowing through the SAP. The removal of moisture occurs in the dry tower and the removal of SO_3 occurs in the alpha, intermediate, and final absorption systems. As a result of performing these functions, sulfuric acid is produced, and heat is generated. The acid circulation and energy recovery system is designed to control the acid flow rate, acid strength and acid temperature to each tower for optimal operation. The heat recovered will be used to heat the boiler feed water in the steam system and production of low-pressure steam in the alpha boiler. The remaining heat load will be removed in the main acid cooler and product acid cooler.

18.2.1.2.4 ALPHA™ System

The alpha system is designed to produce steam at ~5 bar(g) from the energy released during he absorption of SO_3 into the strong acid in the alpha tower.

The overall process consists of the following major steps:

1. SO_3 is absorbed in strong sulfuric acid with dilution water added to control acid strength.

$$SO_3 + H_2O \rightarrow H_2SO_4$$

2. Energy released into the strong acid is recovered to produce low pressure steam and maintains the heat balance in the acid circuit.

The system is installed upstream of the inter tower. The hot gas from bed 3 in the converter is first cooled to ~170°C in the inter economizer and superheater arranged in series. It then enters the alpha tower where approximately 95% of the SO_3 is absorbed by counter-current contact with hot circulating 99% H_2SO_4.

The gas exiting the tower, which has some remaining SO_3, is sent to the intermediate absorption tower system where the remaining SO_3 is absorbed by counter-current contact with circulating 98.5% H_2SO_4. The entire system can be placed offline (with no acid circulation) or bypassed to maintain high availability of the acid production.

The alpha acid circulation system is separate from the main acid circulation system of the SAP and is operated at a nominal 99% H_2SO_4 strength. Acid is circulated by vertically oriented, submerged, centrifugal pumps from the pump tank, through the alpha boiler and then to the alpha tower. In this boiler, the acid is cooled to ~185°C and low-pressure steam is produced. The boiler is a kettle type operating at ~5 bar(g). The saturated steam leaving this boiler passes through a mesh pad to remove entrained droplets before it is combined with the turbine exhaust steam in the low-pressure steam header.

18.2.1.2.5 Condensate System and HP Steam System

Condensate from other parts of the complex (SPA, phosphoric acid and granulation plants), sulfur system and the dump condenser is returned to the condensate tank. Additional make-up water is added to the tank to replace losses. All condensate streams can be sampled and tested individually.

Heat from the combustion of sulfur in the furnace is transferred into the steam system in the two fire-tube boilers arranged in parallel. Hot BFW is fed to the boilers through close coupled gravity down-comers from the steam drum. A mixture of steam and water flows from the boilers through the risers back into the steam drum. A continuous blowdown from the steam drum controls the concentration of impurities in the boiler water. The blowdown is directed to the blowdown drum where it is flashed. The flash steam is vented to the low-pressure steam header and the liquid effluent is sent to the battery limits. The boiler system is equipped with a boiler chemical system.

Superheated high-pressure steam produced in the sulfuric acid plant is sent to the turbine generator at 60 bar(g) for power production.

18.2.1.2.6 Turbine-Generator System and Power

High pressure steam from the sulfuric acid plant is delivered to the turbine-generator (TG) system for generation of electrical power. The TG is a multi-stage condensing unit with extraction capabilities.

The first extraction from the turbine will supply steam to the SPA plant and a small amount of this steam is let down to supply intermittent steam to the granulation plant.

A second extraction from the turbine will supply low pressure steam to the LP steam header. The remaining steam will pass through the final turbine section at very low pressure and is condensed in the turbine condenser.

A letdown system with desuperheater is provided to bypass the turbine and maintain HP/MP & LP steam production in case the turbine must be taken off-line.

The operating and design parameters for the TG system are shown below in Table 18-7.

Table 18-7: TG Power Production and Export Power Availability

Parameter	Value	Unit
TG Production:	49.5	MW (Maximum, Note 1)
	42.9	MW (Design, Note 2)
	31	MW (Average operation, Note 3)
Sulfuric Acid Plant Consumption (Note 4):	10.5	MW (Design, Note 2)
	8.1	MW (Average operation, Note 3)
Power Export	32.4	MW (Design, Note 2)
	22.9	MW (Average operation, Note 3)
Supply/Output level:	TBD	kV, 3 pH, 60 Hz

Notes:
1. Maximum power at 4200 MTPD capacity during periods with no steam extraction from the turbine.
2. Design power production at 4200 MTPD capacity with 32 barg and 3.5 barg steam extraction at design flowrate.
3. Average operation at 3650 MTPD plant capacity and design steam export to B/L.
4. SAP electrical consumption includes all items identified on the process equipment list that are in continuous operation at the design conditions and production rate.
5. Power generated is a preliminary estimate and subject to the turbine efficiency of the selected vendor.

18.2.1.2.7 LP Steam System

Low pressure steam from the turbine extraction, the alpha system and the flash blowdown drum is collected in the LP steam header. The low-pressure steam is used within the SAP (deaerator and sulfur heat tracing) and the remainder is exported to the PAP.

A dump condenser is provided to condense excess LP steam in the event the complex does not require all the steam produced and this condensate is returned to the condensate collection system for use as boiler feed water.

18.2.1.2.8 Cooling Water System

Cooling water will be supplied in an evaporative cooling tower circuit to the strong acid coolers, blower lube oil cooler, turbine-generator set and sample coolers.

Cooling water will be supplied to the dump condenser after passing through the acid coolers to reduce the required cooling water circulation rate.

18.2.1.2.9 Pollution Controls and Waste Disposal

Stack Gases

Stack emissions are a function of design, catalyst loading, and conversion of SO_2 into SO_3 (absorbed into the sulfuric acid). Continuous discharge rates given, correspond to SAP operation at design production rate (4200 t/d).

Table 18-8: Stack Emission Discharge Rates

Parameter	Value	Unit
Tail Gas:		
SO_2 (expressed as SO_2)	<140	Ppm
Acid Mist (expressed as H_2SO_4)	<35	Mg/ Nm^3
Opacity	<10	%
Flow Rate	282,620	Nm^3/hr
Pressure:	Ambient	bar (g)
Temperature:	75	°C
Battery Limit:	Stack tip	

Plant Liquid Effluents

Boiler blowdown and streams with the potential to become contaminated are directed internally to the boiler blowdown tank. The flash steam is sent to the Deaerator and remaining effluent sent to battery limits for processing and recycle in the water treatment unit.

The cooling tower blowdown is sent to the battery limits for processing and recycle in the water treatment unit.

Miscellaneous sumps are discharged to the battery limits and directed to wastewater treatment.

Solid Effluents

Solids from the sulfur filter, consisting of particulates removed from the molten sulfur are discharged and dumped below the sulfur polishing filters into a bin. Flow rate will depend on the sulfur quality.

- Flow Rate: kg/day (to be confirmed)
- Sulfur Content: 40 wt%.

18.2.1.3 *Production Schedule*

Years 1 and 2 of the production schedule allow for a reduced throughput during start-up of both the mine site and the fertilizer complex. Full production is achieved from the start of Year 3 through to the end of Year 25. Year 26 has reduced production because the current mine plan ceases to provide additional feedstock to the FCC.

This also applies to the production schedules for the PAP, SPA and granulation plant.

The planned annual production of 100% sulfuric acid is shown below in Table 18-9.

Table 18-9: Annual Production Of 100% Sulfuric Acid

Production Schedule				
Year	1	2	3 - 25	26
SAP production as 100% Sulfuric Acid ktpy	883	1,212	1,280	457

18.2.1.4 *Consumptive Use Estimate*

Raw material consumption per tonne of product and annually is quantified in Table 18-10 below.

Table 18-10: SAP Raw Material Consumption

Parameter	Units	Specific Consumption
Sulfur	t/t H_2SO_4	0.33
Reverse Osmosis (RO) Water(1)	t/t H_2SO_4	0.71
Process Water(2)	t/t H_2SO_4	0.18

Notes:

1. Make-up for boiler feedwater system and dilution water.
2. Normalized for cooling tower make-up only.

The estimated electric power consumption of the SAP is 53.2 kWh/t of P_2O_5. Electric power is supplied by a turbo-generator within this facility and requires approximately 8.1 MW per day.

An estimate of the workforce required for the operation of this facility is shown below in Table 18-11.

Table 18-11: Estimated SAP Workforce

Workforce Category	No. Of Personnel
Management/Supervision[(1,2)]	2.2
Operations	8
Maintenance[(3)]	4.5
Technical	1
Other	2
Total (rounded up)	18

Notes:

1. Superintendent covers the three areas of SPA, PAP, and SAP.
2. Shift supervision and day supervisors cover the three areas of SPA, PAP, and SAP.
3. Maintenance personnel allocated across the three areas of SPA, PAP, and SAP.

18.2.2 Phosphoric Acid Plant

18.2.2.1 Summary

In the 2008 PhosCan PFS Study, the design was based on a phosphoric acid complex producing 400,000 t/p P_2O_5 in which two (2) scenarios were examined (both hemihydrate[13] and dihydrate technology[14]). Pilot scale production of phosphoric acid, using the wet hemihydrate process, has been successfully performed by Jacobs, and pilot scale production of phosphoric acid using the wet dihydrate process has been successfully performed by both IFDC and Jacobs. The current study increases the phosphoric acid production to 500,000 t/y P_2O_5 using dihydrate technology. The 2008 PhosCan PFS Study design employed dry gypsum stacking with a cooling tower located in the phosphoric acid area. This PEA study employs wet gypsum stacking and an integrated cooling pond, eliminating the cooling tower in the phosphoric acid area.

In the dihydrate phosphoric acid plant (PAP), rock slurry concentrate is reacted with 93-98% sulfuric acid to produce filter acid containing 28% P_2O_5. The sulfuric acid in this reaction is produced in the sulfuric acid plant. The 500,000 t/y P_2O_5 produced is allocated 350,000 t/y for granular fertilizer products (NPS/MAP), 150,000 t/y for Super-Phosphoric Acid (SPA). Facilities to ship MGA are included for future use. SPA and MGA product storage and shipping are covered in the SPA Scope.

Phosphate rock slurry is reacted with the sulfuric acid and recycle acid in the JT annular reactor and a slurry containing calcium sulfate dihydrate and phosphoric acid is the primary result. The reactor slurry is pumped to one of three belt filters where the gypsum and phosphoric acid are separated. The 28% phosphoric acid from the filters is pumped to a clarification system and then into storage tanks. The 28% phosphoric acid is pumped to the granulation plant as a feed acid, to the SPA's concentration area, or to the PAP's concentration area to one of three evaporators. In PAP's concentration area, the acid is increased to 54% and then the acid is transferred to SPA and granulation as unclarified 54% phosphoric feed acid. Dihydrate gypsum ($CaSO_4 \cdot 2H_2O$) is a solid by-product of the reaction of phosphate rock and sulfuric acid. Gypsum is conveyed from the filters to the gypsum tank.

[13] Evaluation of Martison Concentrate, by Jacobs Engineering SA. September, 2007

[14] Fox River Reference ME 9. BENCH-SCALE WET-PROCESS PHOSPHORIC ACID PRODUCTION USING MARTISON, CANADA, PHOSPHATE CONCENTRATE by International Fertilizer Development Center (IFDC). August, 1984

The gypsum is sluiced with pond water forming a gypsum slurry that is pumped to the gypsum wet stack.

The gypsum wet stack is lined with impervious HDPE and will eventually reach a height of 50 m. Drainage water from the gypsum wet stack will be collected in an adjacent retention pond (decant pond). The decant pond overflows to a cooling pond and water is pumped back to the phosphate plant for use as a cooling medium for reaction and evaporation heat loads.

Emissions will be controlled by a multistage cross flow fume scrubber designed to comply with current US EPA regulations for fluoride emissions from the phosphoric acid plant.

18.2.2.2 Process Description

18.2.2.2.1 Phosphate Rock Thickening

The rock phosphate thickening process is to increases the percentage solids in the slurry by sedimentation and decantation. The slurry received from the rock receiving terminal is pumped to the slurry thickener. Flocculant, which is added via a flocculant addition package, is used to enhance the thickening process. The thickened slurry underflowing the thickener is pumped to the reactor feed tanks via thickener transfer pumps. The rock slurry is then fed to the JT dihydrate reactor.

The thickener overflow with minimal fine solids from the weir travels by gravity to the thickener overflow tank which is then pumped to water treatment.

18.2.2.2.2 Dihydrate Phosphoric Acid Plant

In an integrated fertilizer complex, the dihydrate phosphoric acid plant (Figure 18-9) is located central to the site operations on account of all other areas supporting it or processing the products from it.

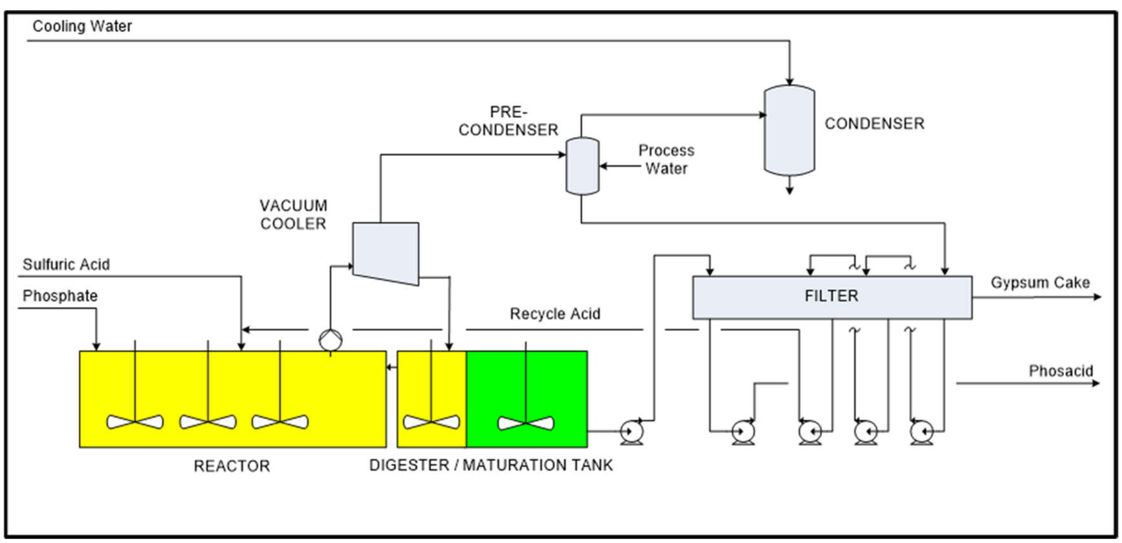


Figure 18-9: Typical JT Dihydrate Phosphoric Acid Plant

Rock slurry is fed to the reactor, where the phosphate rock is dissolved in phosphoric acid to form monocalcium phosphate as shown below.

$Ca_3(PO_4)_2$	+	$4 H_3PO_4$	$\rightarrow$	$3 Ca(H_2PO_4)_2$
Tricalcium Phosphate		Phosphoric Acid		Monocalcium Phosphate

The temperature is maintained by a low level vacuum cooler at about 82°C. Slurry passes through the outer annulus of the reactor until it enters a high sulfate zone where sulfuric acid and filter recycle acid are added to the reactor and calcium sulfate dihydrate and phosphoric acid are formed according to the following reaction:

$Ca(H_2PO_4)_2$	+	H_2SO_4	+	$2 H_2O$	$\rightarrow$	$CaSO_4 \cdot 2H_2O$	+	$2H_3PO_4$
Monocalcium Phosphate		Sulfuric Acid				Gypsum		Phosphoric Acid

Slurry containing 28% P_2O_5 phosphoric acid and gypsum passes from the reactor to the maturation tank to allow gypsum crystals to grow before being pumped to the filters. The recovery of P_2O_5 from the phosphate concentrate in the reaction and filtration system is approximately 95%.

Evaporated water vapor from the vacuum cooler passes through a precondenser, to preheat filter wash water, and then a barometric condenser. A vacuum pump is used to maintain the necessary vacuum for this system.

Exhaust gas from the reactor passes to the fume scrubber, which is a multistage, crossflow design using Kimre packing. Regulations for fluoride from phosphoric acid plants are 10g F/t P_2O_5 feed / day. Cooling water for the reactor and evaporators is provided by a separate cooling pond.

The filter type chosen for this process is a belt filter, which is expected to have the minimum capital cost. Based on test work, three (3) belt filters are required. A three stage wash system is used to remove entrained phosphoric acid from the gypsum cake on the filters. A large amount of acid is recycled from the filters to the reactor to maintain the optimal solids concentration. Product acid from the filters is pumped to a clarification system followed by storage tanks. Product acid from storage is pumped to the concentration area. Sludge from the clarifiers is recycled to the reactor.

Filter product acid (28%) from storage is pumped to the concentration area where three (3) evaporators in parallel are installed. The evaporators are used to increase the P_2O_5 concentration to 54% for merchant grade acid (MGA).

Filter product acid (28%) from storage is also pumped to the granulation plant and super phosphoric acid (SPA) plant.

A wash water tank receives wash water from filters and evaporators during periodic wash cycles. Wash water is used as make up water to the process. Provision is made to transfer excess wash water to the wastewater treatment plant if needed.

18.2.2.2.3 Gypsum Disposal

Wet stacking is considered for gypsum storage. Gypsum cake from the filters is transported to the gypsum slurry tank via the gypsum conveyor. Pond water return from the cooler condenser is used to sluice the gypsum cake as well as to maintain the solids content in the gypsum slurry. Gypsum slurry is pumped to the wet gypsum stack.

In wet gypsum stacking, the gypsum slurry is pumped to the settling compartment of the stack, where solid gypsum is settled. Some water will be retained in the wet gypsum stack and remaining decanted water is collected in a separate decant pond which overflows to the cooling pond.

Pond water from the cooling pond is recycled to the condensers and scrubbers as a cooling medium. Used water is received back to the cooling pond from the hotwell.

Part of the pond water from the cooling pond is used back in the process. This enables water soluble P_2O_5, lost along with gypsum, to be recovered. This also improves global P_2O_5 recovery. Water evaporated from the gypsum stack and cooling pond needs to be made up and this can be from snow melt, rain, or recycled water of good quality from the plant.

18.2.2.2.4 Concentration (54% Acid)

The concentration process (Figure 18-10) is accomplished using low pressure steam in a forced circulation vacuum evaporator system. Each evaporator system includes a flash chamber, an axial flow circulation pump and a heater with special graphite tubes. Typically, the evaporated vapor passes through a P_2O_5 entrainment separator, fluorine scrubbers, FSA seal tanks, and a barometric condenser. The vacuum needed for evaporation at low temperature is provided by a vacuum pump. Provisions have been kept in the layout for future product grade FSA recovery. In this phase of the project no recovery units have been included in the evaluation.

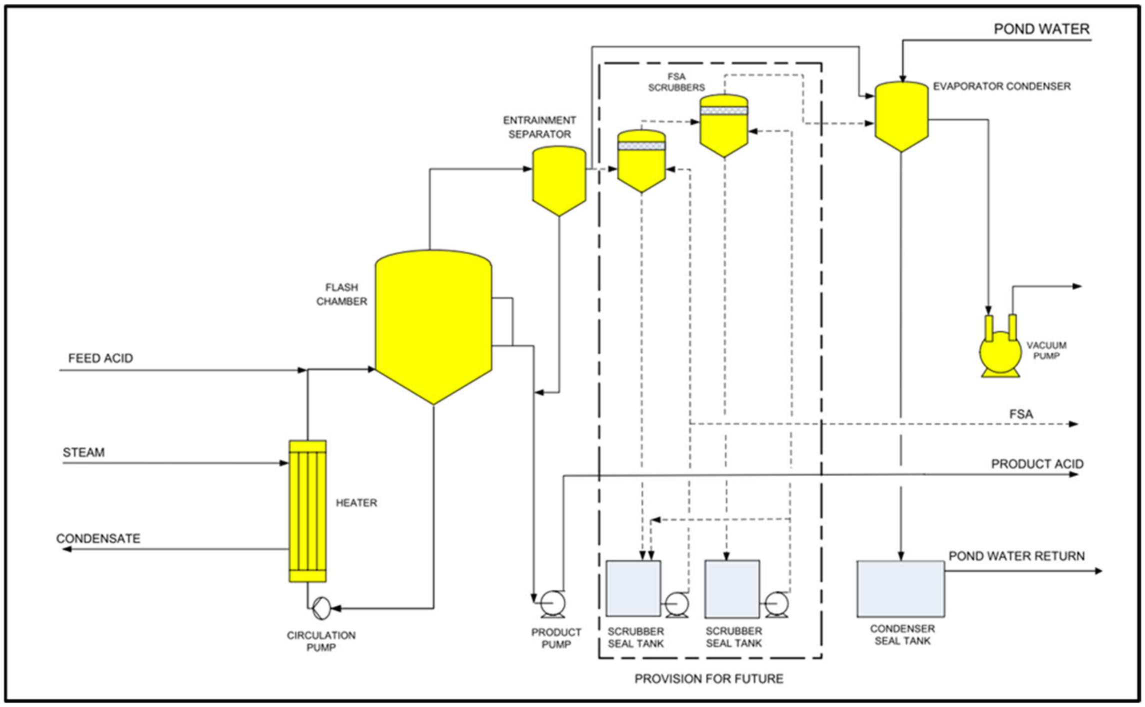


Figure 18-10: Concentration Unit (54% Acid)

Steam condensate is collected and pumped back to the condensate return tank located in the offsites area.

18.2.2.2.5 Acid Storage (54%)

Acid, at 54% concentration, is pumped from the evaporators to one of two storage tanks (942 m^3 each). From storage, it is pumped by acid transfer pumps to a clarifier in the SPA area or to phosphoric acid storage tanks in granulation area.

18.2.2.2.6 Pollution Controls and Waste Disposal

Air

Fluorine fumes are released in the reaction from phosphate rock to phosphoric acid. Regulations for fluorine from phosphoric acid plants are 10g F / t P_2O_5 feed / day. These fumes are contained by various scrubbers and absorbers to meet regulatory limits, as follows:

- The fume scrubber is used for exhaust fumes from the main phosphoric acid reactor and the three (3) belt filter hoods. This scrubber is a multistage spray system with Kimre packing.
- Vapors from the 28% and 54% storage tanks pass to the storage area scrubber.

Liquid

To minimize the possibility of chemical spills, all large chemical storage tanks are contained within a lined diked area.

All process liquid effluent is recovered and reused or treated as follows:

- Effluent from the fume scrubber is collected in the process water sump and reused in the process.
- Contaminated process water from filter and evaporator washing is collected in the wash tank and reused in the process.
- All liquid effluent generated will be reused in the process. However, provision in the design allows for pumping excess liquid effluent to the wastewater treatment plant.
- Cooling pond water used in the process is returned to the cooling pond via hotwell.
- Water during rainfalls or snow melt off, is accommodated in the decant pond.

Solids

By-product gypsum cake from the filters is transported to a gypsum wet stack. The gypsum wet stack will have a maximum height of approximately 50 m and will be lined by impervious HDPE. Drainage water from the gypsum wet stack will be collected in an adjacent decant pond and pumped back to the phosphoric acid plant for reuse in the process.

18.2.2.3 Production Schedule

The planned annual production of phosphoric acid as P_2O_5 is shown below in Table 18-12.

Table 18-12: PAP Production Schedule

Production Schedule				
Year	**1**	**2**	**3 – 25**	**26**
Total P_2O_5 Production ktpy	345	474	500	178
P_2O_5 Transfer to SPA Plant ktpy (1)	126	172	177	65
P_2O_5 Transfer to Granulation Plant ktpy (1)	219	302	323	113
Total P_2O_5 transfer as Weak Acid ktpy (1)	175	241	254	91
Total P_2O_5 transfer as Strong Acid ktpy (1)	170	233	246	87

Note:
1. P_2O_5 to SPA Plant includes:
Approx. 27 ktpy transferred from SPA as sludge to Granulation for years 3-25.

18.2.2.4 Consumptive Use Estimate

Raw material consumption per ton of product and annually is quantified in the Table 18-13 below.

Table 18-13: PAP Raw Material Consumption

Parameter	Units	Specific Consumption
Rock (Dry Basis)	t/t P_2O_5	2.81
Sulfuric Acid Consumption (100% Sulfuric Acid Basis)	t/t P_2O_5	2.5
Low Pressure Steam	t/t P_2O_5	2.2
Process Water[(1)]	t/t P_2O_5	6
Defoamer	kg/t P_2O_5	2
Flocculant	kg/t P_2O_5	0.4

Note:
1. All sources of water including concentrate slurry water content.

The estimated electric power consumption of the PAP is 101 kWh/t P_2O_5. Electric power is supplied by a turbo generator located in the SAP. This facility requires approximately 9 MW per day.

An estimate of the workforce required for the operation of the PAP is shown below in Table 18-14.

Table 18-14: Estimated PAP Workforce

Workforce Category	No. of Personnel
Management/Supervision[(1,2)]	2.2
Operations	14
Maintenance[(3)]	4.5
Technical	1
Other	7
Total (rounded up)	29

Notes:
1. Superintendent covers the three areas of SPA, PAP, and SAP.
2. Shift supervision and day supervisors cover the three areas of SPA, PAP, and SAP.
3. Maintenance personnel allocated across the three areas of SPA, PAP, and SAP.

18.2.3 Super Phosphoric Acid Plant

18.2.3.1 Summary

The Super Phosphoric Acid (SPA) Plant has a nameplate output capacity of 150 ktpy of P_2O_5. The SPA plant is designed to produce acid containing a minimum of 68% P_2O_5 in the form of ortho and poly phosphoric acid. During the SPA process impurities are removed from the PAP unit acid feed in the form of pyrophosphates which are diverted to granulation for incorporation into dry fertilizer products. The SPA product is typically delivered to area distributors who convert it to liquid Ammonium Poly Phosphate (APP) fertilizer, which is sold to local farmers.

The Martison Phosphate Project PFS[15] from 2008 also included a 150 ktpy of P_2O_5 SPA plant. The feedstock to the PFS SPA plant was phosphoric acid produced by a hemihydrate phosphoric acid plant treating phosphate rock with an MER of 0.06. The feedstock to the PEA SPA plant is phosphoric acid produced by a dihydrate phosphoric acid plant treating phosphate rock with an MER of 0.09. The changes in feedstock required an increase in the SPA plant evaporation, clarification and MgO removal capacity. This higher MER will also require the granulation plant to absorb a higher quantity of SPA sludge (clarifier underflow and hydrolyzed pyro-phosphates).

In the granulation section of this report (Section 18.2.4) a sludge balance is displayed which indicates that all of the impurities from the SPA plant can be consumed in dry fertilizer products without compromising the fertilizer grade.

Prior to the PFS, International Fertilizer Development (IFDC) successfully produced phosphoric acid from Martison phosphate rock[16]. As part of the PFS, Jacobs successfully pilot plant tested the production of phosphoric acid from Martison phosphate rock using the hemihydrate process and bench scale tested the production of SPA[17]. In 2011, Jacobs performed pilot tests to produce phosphoric acid from Martison phosphate rock using the dihydrate process and bench scale tested the production of SPA and 10:34:00 ammonium polyphosphate[18].

The SPA plant includes Merchant Grade Acid (MGA) evaporators, MGA clarifier, SPA evaporator, SPA oxidation and MgO removal treatment, MGA/SPA storage and shipping.

The raw material for MGA production is clarified weak (28%) phosphoric acid. This requires 177,000 t/y of P_2O_5 in the form of weak phosphoric acid fed to the MGA evaporators followed by the MGA clarifier.

The raw material for SPA production is MGA. The SPA plant requires 163,500 t/y of P_2O_5 in the form of MGA. The ability to export a limited amount of MGA is included in the SPA plant design.

The solids removed from the MGA clarification and MgO filtration and accompanying P_2O_5 report to the fertilizer mix tank where the pyrophosphate solids are dissolved (hydrolyzed) before reporting to granulation for incorporation into granular fertilizer.

The MGA evaporator structures are designed to have sufficient space for future Fluosilicic Acid (FSA) recovery equipment. The FSA recovery equipment is not provided in this evaluation.

[15] N.I. 43-101 TECHNICAL REPORT: MARTISON PHOSPHATE PROJECT PRELIMINARY FEASIBILITY STUDY, Report Date: May 16, 2008
[16] BENCH-SCALE WET-PROCESS PHOSPHORIC ACID PRODUCTION USING MARTISON, CANADA, PHOSPHATE CONCENTRATE by International Fertilizer Development (IFDC)
[17] EVALUATION OF MARTISON CONCENTRATE by Jacobs Engineering, August 2007
[18] ACIDULATION OF MARTISON CONCENTRATE BY DIHYDRATE PROCESS by Jacobs Engineering, May 2011

18.2.3.2 *Process Description*

The following description applies to a SPA plant to produce MGA and SPA. The SPA design capacity is 150,000 t/y of P_2O_5. The provision to export a limited amount of MGA is included in the design. Raw material is 177,500 t/y of P_2O_5 weak clarified phosphoric acid at a concentration of 28% P_2O_5. Additional chemicals and utilities required include defoamer, 50% ammonium nitrate solution for SPA oxidation, iron balls for SPA reduction, process water, steam, and instrument air.

The process consists of MGA evaporators, MGA clarification, SPA evaporator, SPA treatment and filtration, and MGA/SPA storage and shipping.

18.2.3.2.1 MGA Evaporators

The SPA plant includes two single stage evaporators. The purpose of the evaporators is to remove water from the acid to increase the concentration of the P_2O_5 in the acid from 28% to 54-56% P_2O_5. The evaporators use a noncontact low pressure steam heater to heat the acid under vacuum. The water vapor passes through an entrainment separator to recover any acid entrained in the water vapor. The water vapour removed from the acid is condensed and recovered in a contact cooling system using a barometric condenser. See the flow diagram below (Figure 18-11) for MGA Evaporators.

The weak phosphoric acid contains contaminants such as sulfate, fluoride, gypsum, and metal cations. A significant portion of the fluoride is stripped from the acid as SiF_4 or HF gas. The fluoride gas is removed from the vapor stream with an optional FSA scrubber or scrubbed from the gas stream in the barometric condenser. The gypsum and metal cations (primarily iron) precipitate to form solids in the strong phosphoric acid. The production of MGA for SPA uses the same equipment as used for producing strong phosphoric acid for fertilizer production.

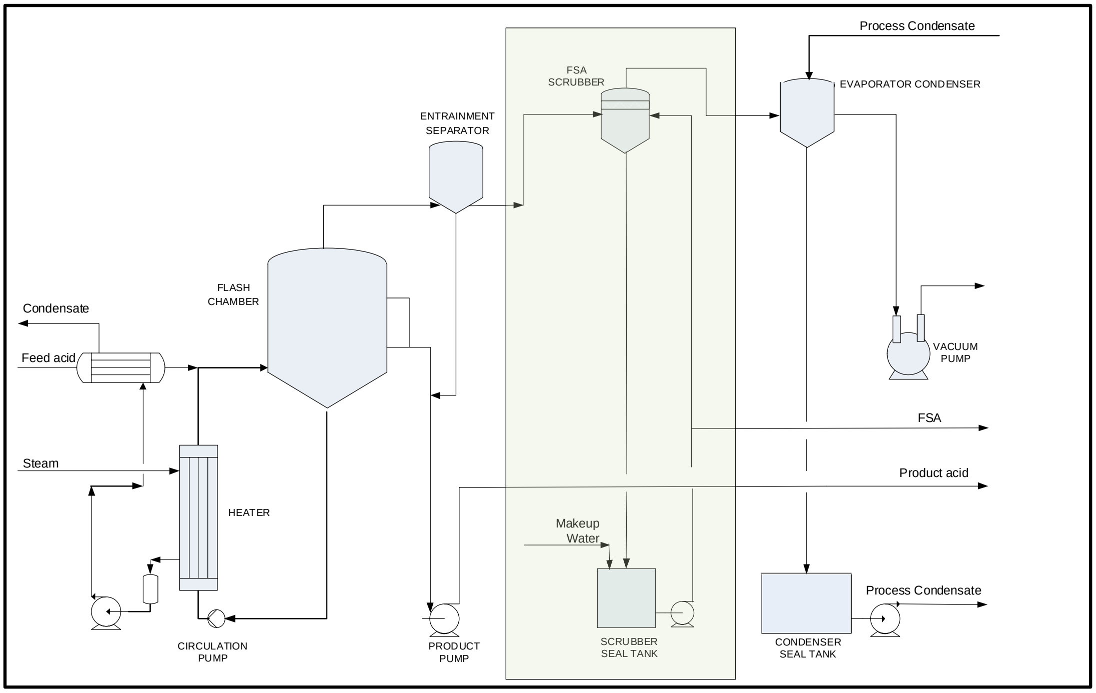


Figure 18-11: Block Flow Diagram of The MGA Evaporator with Optional FSA Recovery Section Noted in Highlight

18.2.3.2.2 SPA Concentration

The SPA concentration area consists of an MGA clarifier and SPA evaporator. MGA is an intermediate product required to produce SPA. MGA is made by clarifying strong phosphoric acid and reducing the solids content of the acid to less than 1% by weight in the MGA clarifier. The optimum strength for removing the solids is 54-56% P_2O_5. The strong acid is aged several hours prior to desaturate the acid and maximize the quantity of solid impurities removed from the acid. The underflow from the clarifier is pumped to the acid mix tank feeding the granular fertilizer plant. The solids in the underflow act as filler to prevent over formulation of the P_2O_5 content in the dry fertilizer. 177,500 t/y P_2O_5 of strong acid is required to produce 163,500 t/y P_2O_5 of unfiltered SPA.

The SPA evaporator is a single stage evaporator. The purpose of the evaporator is to remove water from the acid and produce poly phosphoric acid (polys) to increase the concentration of the P_2O_5 in the acid from 54-56% to 68-70%, whereby, 20-25% of the total P_2O_5 is poly phosphoric acid. The evaporators use a noncontact high pressure steam heater to heat the acid under vacuum. The water vapour passes through an entrainment separator to recover any acid entrained in the water vapour. The water vapour removed from the acid is condensed and recovered in a contact cooling system using a barometric condenser.

The MGA feed acid contains contaminants such as fluoride, gypsum, and metal cations. A significant portion of the fluoride is stripped from the acid as HF gas. The fluoride gas is scrubbed from the gas stream in the barometric condenser. Almost all the calcium precipitates as anhydrite gypsum and some of the metal cations (primarily iron, aluminum and magnesium) precipitate to form solids in the unfiltered SPA. In the case of magnesium, the solid formed is magnesium pyrophosphate ($Mg_2P_2O_7$). Pyrophosphate solids when combined with free water will hydrolyze into a high magnesium phosphoric acid which can be combined with additional phosphoric acid in the acid mix tank and sent to granulation to produce dry phosphate fertilizer.

After the SPA is evaporated, the colour of the acid is black due to the presence of organics. The organics are removed by oxidation with ammonium nitrate solution. A side effect of the oxidation reaction changes the valence state of the iron from +3 to +2. Elemental iron is added to the acid to change the valance state of the iron back to + 3 improving the stability of the SPA. See flow diagram below (Figure 18-12) for SPA concentration.

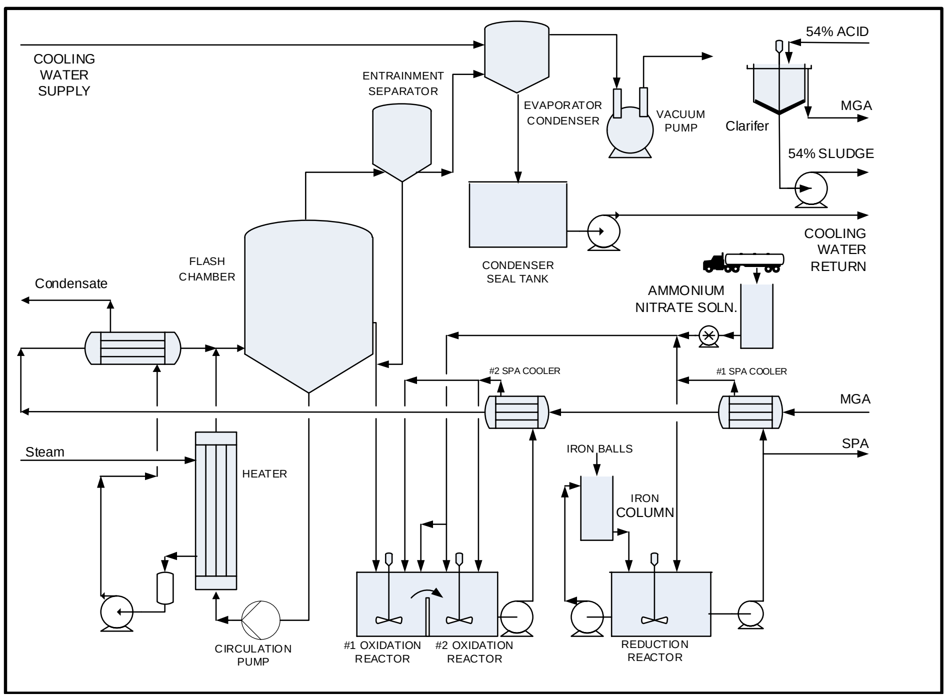


Figure 18-12: Block Flow Diagram of The SPA Evaporator, Oxidation/Reduction Reactors and MGA Clarifier

18.2.3.2.3 MgO Removal

Figure 18-13 below illustrates the MgO removal process. This area consists of agitated tanks, pumps, and a “plate and frame” filter complete with filter aid system. The unfiltered SPA acid is high in pyro phosphate solids and these solids are removed using a plate and frame filter press. Filter aid is used to improve the operation of the plate and frame filter. The removal of the solids increases the quality and strength of the SPA filtrate.

The filter cake is discharged into a tank, where it is converted to a slurry with strong acid. It is then pumped to the fertilizer mix tank in the fertilizer area to provide impurities to prevent over formulation of P_2O_5 content in the dry fertilizer product. 163,500 t/y P_2O_5 of unfiltered SPA is required to produce 150,000 t/y P_2O_5 of SPA.

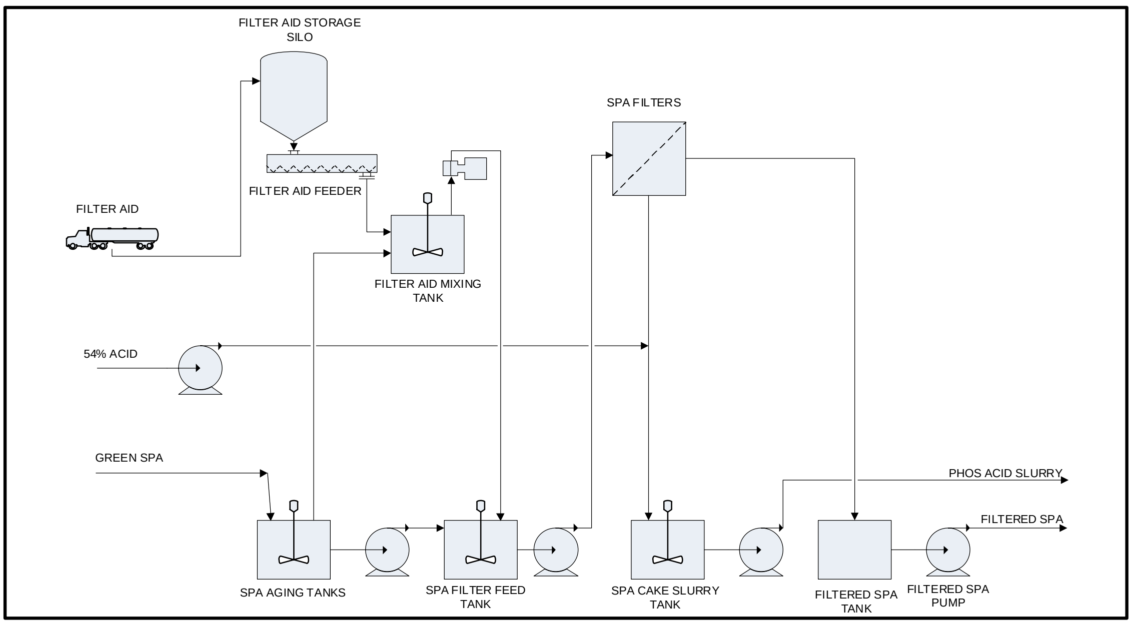


Figure 18-13: Block Flow Diagram of the MgO Removal

18.2.3.3 MGA/SPA Storage and Shipping

This area includes two agitated 1,260 m^3 SPA storage tanks, one agitated 1909 m^3 MGA storage tank, loading stations for SPA/MGA and a heater system for the SPA storage tanks. SPA is a very viscous material and heat is required to make the SPA pumpable. The temperature of the SPA storage tanks is maintained at no less than 80°C using a dedicated shell and tube heat exchanger and low pressure steam for each tank. Steam condensate is collected and pumped back to the condensate return tank located in the off-sites area.

See the flow diagram below (Figure 18-14) for the MGA/SPA storage and shipping area.

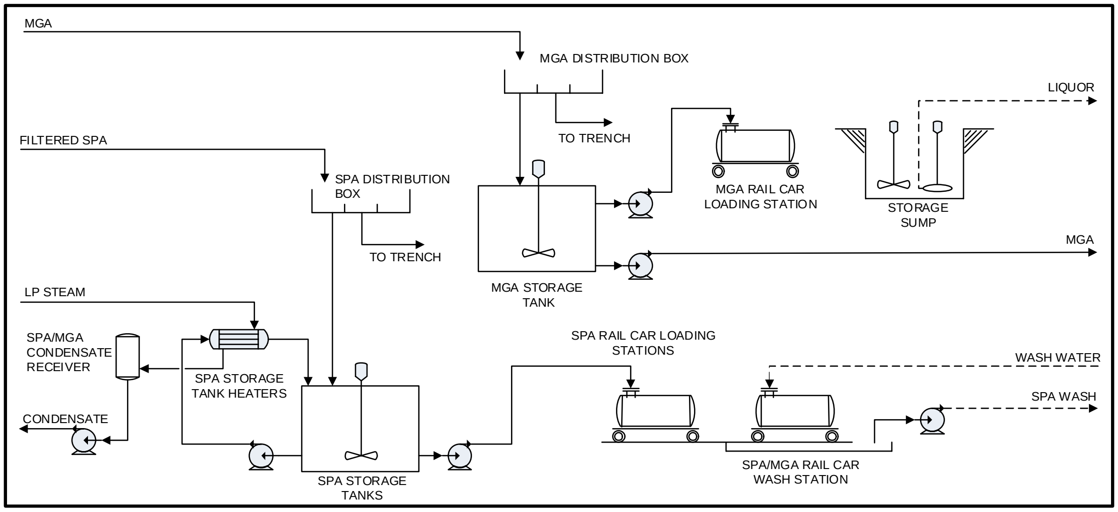


Figure 18-14: Block Flow Diagram of the Area 4901 MGA/SPA Storage and Shipping

18.2.3.4 Production Schedule

The planned annual SPA100% P_20_5 production is shown below in Table 18-15.

Table 18-15: Annual SPA100% P_20_5 Production

Production Schedule				
Year	**1**	**2**	**3 – 25**	**26**
SPA production as P_2O_5 ktpy	104	142	150	54
SPA production ktpy of solution	152	209	221	79

18.2.3.5 Consumptive Use Estimate

Raw material consumption per ton of product and annually is quantified in the Table 18-16 below.

Table 18-16: SPA Raw Material Consumption

Product	Unit	MGA	SPA	Total
Production	(ktpy)	0	150	150
50% Solution Ammonium Nitrate	(t solution/ t product)	-	0.00363	-
	(ktpy)	-	0.545	0.545
Iron Balls	(t balls/t product)	-	0.00131	-
	(ktpy)	-	0.197	0.197
Filter Aid	(t/t product)	-	0.0535	-
	(ktpy)	-	0.802	0.802
Low Pressure Steam	(l/t product)	2.23	2.43	
	(ktpy)	-	365	365
High Pressure Steam (60 Bar)	(m^3/y product)	-	1.02	
	(ktpy)	-	153	153

The estimated electric power consumption of the SPA facility is 75 kWh/t SPA. Electric power will be supplied by a turbo generator located in the SAP. The SPA facility will require approximately 2.5 MW per day.

An estimate of the workforce required for the operation of this facility is shown below in Table 18-17.

Table 18-17: Estimated SPA Workforce

Item	Value
Management/Supervision(1,2)	2.2
Operations	8
Maintenance(3)	4.4
Technical	1
Other	6
Total (rounded up)	22

Notes:
1. Superintendent covers the three areas of SPA, PAP, and SAP.
2. Shift supervision and day supervisors cover the three areas of SPA, PAP, and SAP.
3. Maintenance personnel allocated across the three areas of SPA, PAP, and SAP.

18.2.3.6 *Pollution Controls and Waste Disposal*

Air

Fluorine fumes are present in the phosphoric acid tank vapor space. Current regulations for fluorine from phosphoric acid plants are 5g F/t P_2O_5 fed/day. These fumes are contained by various scrubbers and absorbers to meet domestic limits, as follows:

- The storage area scrubber is used for exhaust fumes from the strong phosphoric acid tanks, clarifiers and various equipment in the SPA treatment area. This scrubber is a venturi type scrubber with a cyclonic separator.
- The scrubber exhaust gas has a maximum outlet fluorine concentration of 5.0 mg/Nm^3.

Liquid

To minimize the possibility of chemical spills, all large chemical storage tanks are contained within a lined diked area.

All process liquid effluent is recovered and reused or treated as follows:

- Effluent from the storage area fume scrubber is pumped to the phosphoric area, collected in the process water sump and reused in the process.
- All liquid effluent generated will be reused in the process. However, provision in the design allows for pumping excess liquid effluent to the wastewater treatment plant.
- Cooling pond water and condensate from the process barometric condensers is returned to the cooling pond via hotwell (a tank that receives hot discharge liquid).
- Water during rainfalls or snow melt off is accommodated in the decant pond.

Solids

- Solids removed from the strong phosphoric acid in the MGA clarifier are collected in the underflow from the clarifier and pumped to the granulation unit mix tank.
- Solids removed from the super phosphoric acid in the plate and frame filter are mixed with strong acid from the mix tank and pumped back to the mix tank.
- The strong acid with solids in the mix tank is consumed by the granular fertilizer plant.

18.2.4 Granulation Plant

18.2.4.1 Granulation Plant Summary

In the 2008 PhosCan PFS Study, the design was based on MAP products[19,20] with the granular fertilizer production located in Brandon, Manitoba, and the phosphoric acid production in Ontario. In mid-2021, an analysis was conducted and determined that a combined facility near Hearst, Ontario, was more economically appealing. Due to the increased demand for sulfur enhanced fertilizers, NPS fertilizer (12-40-0-10S and 12-40-0-10S-1Zn) were included for the present PEA study.

In 2011, Jacobs (now JESA Technologies) performed pilot tests to produce phosphoric acid from Martison phosphate rock using the dihydrate process and bench scale tested the production of MAP chemical grade was achieved and it granulated well.

The granulation plant converts 350,000 t/y of P_2O_5 to finished fertilizer products, where approximately 250,000 t/y of P_2O_5 is converted to MAP (11-52-0) and the remainder converted to NPS (12-40-0-10S or 12-40-0-10S-1Zn) fertilizer. The granulation unit is designed for 105 t/h of either product at an 80% operating factor (7,000 h/y). This equates to approximately 721,000 t/y, where 474,000 t are MAP, and 247,000 t are NPS fertilizer. Plant recovery is estimated at 98.9% for P_2O_5 and 98% for ammonia.

MAP is produced by reacting phosphoric acid solution and ammonia. NPS is produced similarly to MAP, with zinc, elemental sulfur, and additional sulfuric acid being added to the process as well as using phosphoric acid and ammonia. Elemental sulfur addition involves third party licensed technology. In the next stage of project development, a licensor will be selected, and complete details of the system can be developed. For the present study, sufficient information is available to develop a preliminary estimate of capital costs and raw material consumption for NPS, which is included in the analysis.

All equipment, hydraulics and controls are designed to meet all the operating cases with the appropriate design margin.

18.2.4.2 Process Description

The following description applies to a granulation train to produce of MAP and NPS fertilizers. The capacity when producing MAP and NPS fertilizers is 105 t/h at an operating factor of 80% (based on 7000 h/y). Raw materials are phosphoric acid at concentrations of 54% (strong) and 28% (weak) P_2O_5, SPA sludge, liquid anhydrous ammonia, 93% or 98% sulfuric acid, sulfur, zinc additives (when making NPS) and filler as required depending on formulation. Additional chemicals and utilities required includes defoamer, coating oil, process water, fuel gas and steam.

[19] EVALUATION OF MARTISON CONCENTRATE by Jacobs Engineering, August 2007
[20] ACIDULATION OF MARTISON CONCENTRATE BY DIHYDRATE PROCESS by Jacobs Engineering, May 2011

The reaction between ammonia and phosphoric acid begins in the preneutralizer. Sulfuric acid is also added to the preneutralizer, which has a greater affinity for ammonia than phosphoric acid and results in ammonium sulfate compounds and which is used for NPS formulations and for grade control.

The preneutralizer is operated based on the forward titration process, where it operates at a mole ratio of approximately 0.7 and a specific gravity of 1.625 for grades with high levels of sulfuric acid or high impurities. The final mole ratio is approximately 1.0 thus it is transitioning “forward” from 0.7 to a mole ratio of 1.0.

The slurry from the preneutralizer is sprayed onto a bed of recycled undersize granular material in the rotary granulator. The design recycle ratio for this facility is 4 to 1. The MAP/NPS flows from the granulator to the rotary dryer where the moisture is reduced to approximately 1.5%free water.

The product size is controlled using double deck oversize screens and crushers followed by single deck product screens to separate the on-size material before routing to the fluidized bed cooler. The cooled product is screened one final time to remove any fines that have been generated in the cooler and then routed for coating in a ribbon blender where the product is covered with a thin layer of coating oil to control product dusting.

When making NPS fertilizer, zinc additives can be introduced into the recycle system to target a 1% zinc concentration in the final product (12-40-0-10S-1Zn).

Figure 18-15 below provides an overview of the process systems in the granulation plant. MAP does not require sulfur or sulfuric acid for product formulation, whereas the NPS product does.

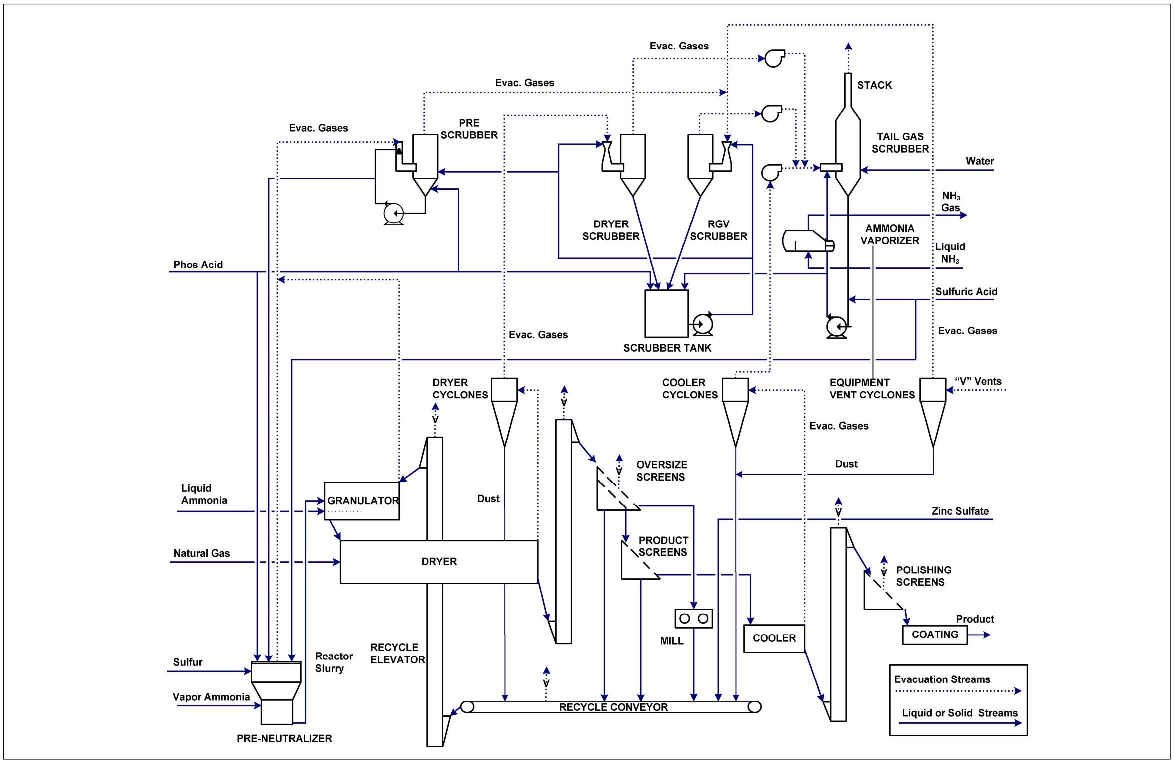


Figure 18-15: Granulation Plant Block Flow Diagram

18.2.4.3 *Product Storage and Shipping*

The product storage area will consist of an A-frame warehouse sized to hold 120,000 metric tonnes of product. The storage building will be fed via the product conveyor from granulation. A traveling tripper will be used to direct the product onto the storage piles.

The product will be reclaimed using payloaders that will deposit the product onto the shipping conveyor. The shipping conveyor directs material to the shipping elevator where it is lifted up to the shipping screens for a final polishing step prior to loading into either rail cars or trucks. Limited dual product loading capability will be included in the design by installing two weigh and dump hopper systems. All equipment in the shipping area (conveyors, elevator, screens and weighing system) have been designed for 640 t/h.

18.2.4.4 *Production Schedule*

The planned annual production of the granulation plant is shown below in Table 18-18.

Table 18-18: Granulation Plant Production Schedule

Production Schedule				
Year	**1**	**2**	**3 – 25**	**26**
MAP production as P_2O_5 ktpy	171	234	247	88
Total MAP production ktpy	327	449	474	169
NPS production as P_2O_5 ktpy	68	94	99	35
Total NPS production ktpy	171	234	247	88
Total production as P_2O_5 ktpy	239	328	346	123
Total fertilizer production ktpy	498	683	721	257

18.2.4.5 Consumptive Use Estimate

Raw Material consumption per ton of product and annually is quantified in the Table 18-19 below.

Table 18-19: Granulation Raw Material Consumption

Product	Units	MAP	NPS	Total
Production	(ktpy)	474	247	721
Ammonia	(t ammonia/t product)	0.136	0.149	-
	(ktpy)	64.5	36.8	101.3
Phosphoric Acid	(t phos acid/t product)	0.528	0.404	
	(ktpy)	247	99	346
98% Sulfuric Acid(1,2)	(t SA/t product)	-	0.12	-
	(ktpy)	-	38.6	38.6
Sulfur	(t S/t product)	-	0.05	-
	(ktpy)	-	12.4	12.4
$ZnSO_4$	(t $ZnSO_4$/t product)	-	0.030	-
	(ktpy)	-	7.42	7.42
Defoamer	(liter/t product)	4.3	4.3	-
	(m3/y)	2,038	1,062	3,100
Natural Gas	(Gj/t P_2O_5)	0.38	0.54	-
	(t/y)	1,968	1,121	3,089

Notes:
1. For product grade consideration only.
2. Excludes sulfuric acid content already present in the phosphoric acid feed.

The estimated electric power consumption of the granulation facility is 37 kWh/t product. Electric power is supplied by a turbo generator within this facility and requires approximately 5.3 MW per day.

An estimate of the workforce required for the operation of the granulation facility is shown below in Table 18-20.

Table 18-20: Estimated Granulation Workforce

Workforce Category	No. of Personnel
Management/Supervision(1,2)	3.5
Operations	12
Maintenance(3)	4.5
Technical	1
Other	6
Total (rounded up)	27

Notes:
1. Superintendent covers the two areas of GRAN and INFRA.
2. Shift supervision and day supervisors cover the two areas of GRAN and INFRA.
3. Maintenance personnel split between the two areas of GRAN and INFRA.

18.2.4.6 *Pollution Controls and Waste Disposal*

Air

Figure 18-16 provides an overview of the scrubbing system.

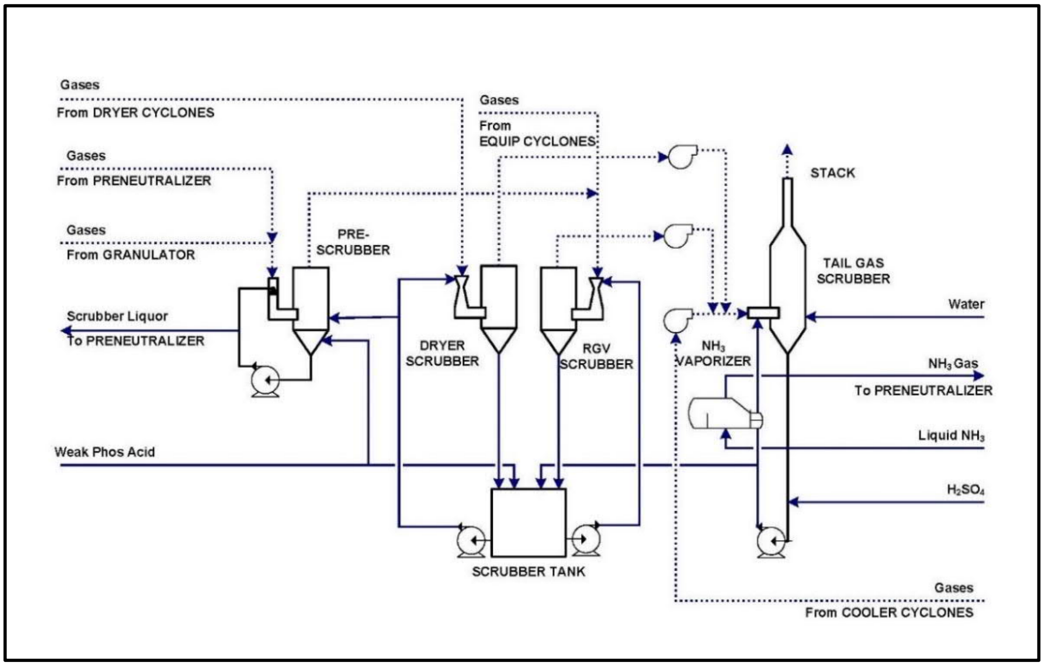


Figure 18-16: Scrubbing Area Block Flow Diagram

Two primary scrubbers are provided in the scrubbing section. Each of these scrubbers, the dryer scrubber and the dust and fume scrubber (RG Scrubber) is of highly efficient design, ensuring maximum removal of both dust and volatiles, which are mainly ammonia, and fluorides.

The flowsheet includes a “dual mole” scrubbing system which gives maximum ammonia and fluorine removal from the gases.

The gases most heavily laden with ammonia are first scrubbed in a prescrubber, which is irrigated by a solution of monoammonium phosphate and diammonium phosphate at a mole ratio of about 1.5 $N:P_2O_5$ (pH 6.0). These ammonium phosphates are formed from the ammonia escaping the preneutralizer and the granulator, reacting with phosphoric acid, which is fed to the prescrubber and recirculated by the prescrubber pumps. 50% to 70% of the NH_3 in the gas stream is captured in the prescrubber.

The gases from the prescrubber is then scrubbed again, this time by a solution of dilute monoammonium phosphate and phosphoric acid which is pumped out of the primary scrubber tank by the primary scrubber pumps. The same solution also scrubs the fumes from the dryer in the dryer scrubber. The second scrubber solution has a mole ratio of about 0.7. This two stage scrubbing system is known as the “dual mole” system, where the prescrubber operates at an ammonia to phosphoric acid mole ratio of about 1.5, while the second stage operates at a mole ratio of about 0.7. The advantage of this system is that less phosphoric acid is used in the scrubbing system than in other systems where all primary scrubbing is done at a mole ratio of about 0.8. The resultant fluorine emission from the plant is well below the US EPA emission limits.

After passing through the primary scrubbers, all exhaust gases including those from the product cooler are finally passed through the tail gas scrubber, where they are contacted with a recirculated dilute scrubber solution, to remove final traces of dust, ammonia, and fluoride emissions. The scrubber solution can be acidified as needed with sulfuric acid to maintain a pH of 3.0 to 4.0, which is the optimum for ammonia and fluoride recovery.

Plant Liquid and Solid Effluents

Liquid drains, spills, and washdowns are collected in the plant sump and transferred to the scrubbing system by a sump pump. A wash tank is also provided for temporary storage of recoverable liquor.

Solid spillage within the plant and temporary storage of the in-process solids when equipment must be emptied are accumulated in a dump area. When this material is reclaimed by a front end loader, it is deposited in the intake hopper and fed to the filler/reclaim conveyor to the intake elevator, through a diverter and to the reclaim hopper. The reclaimed material is fed by the filler weigh feeder to the raw material transfer conveyor and then onto the fines conveyor for return to the process.

18.2.5 Infrastructure – Process Support

18.2.5.1 Infrastructure – Process Support Summary

In the 2008 PhosCan PFS Study, the design was based on MAP products with the granular fertilizer production located in Brandon, Manitoba, separate from the phosphoric acid production in Ontario. In mid-2021, an analysis was conducted and determined that a combined facility near Hearst, Ontario, was more economically appealing. The combined location eliminates duplicate services such as air and water treatment, albeit slightly larger, does provide economy of scale. Additionally, administration, maintenance, and warehousing structures are now in a single location.

The offsites area supplies utilities to all process plants. This includes steam supply from SAP ISBL, condensate return, water, fire water, instrument air, and plant air.

In the 2008 PhosCan PFS study, molten sulfur storage, distribution, and a boiler package were included in the offsites scope of work. See Section 18.2.1.

For sulfuric acid, offsites is responsible for receiving sulfur from railcars. Each sulfur car is steamed at one of forty eight steaming stations, melting the sulfur which is then drained into a sulfur pit and pumped to sulfur storage within the SAP. Sulfuric acid storage tanks and supply pumps are designed to provide the required quantity to both PAP and granulation units.

Raw materials (fatty acid, sarcosine, and soda ash) for the beneficiation plant (located at the mine site) are received by railcars into storage tanks and a silo and then loaded into trucks for periodic delivery via the main access road.

Waste treatment is provided by a neutralization system to treat any excess wastewater from the FCC. This includes a lime slaker, treatment tanks, and settling ponds.

18.2.5.2 Process Description

18.2.5.2.1 Off Sites – Sulfuric Acid

Molten sulfur is received by railcars which are first heated to ensure sulfur is no longer in the solid phase. The cars are heated with steam at dedicated steaming stations for up to 72 hours in winter conditions. The cars are then emptied (by gravity) to a launder system where the molten sulfur gravity flows to a concrete lined in-ground pit. The sulfur pit is equipped with vertical pumps to pump the sulfur to the molten sulfur storage tank (transfer pumps and storage are within sulfuric acid plant ISBL. From storage, the sulfur is pumped to the sulfuric acid plant where it is converted to sulfuric acid. Heat recovery in the sulfuric acid plant produces steam that is converted by a turbogenerator to electric power for internal usage and possible sale. Cooling water for the sulfuric acid plant is provided by a separate cooling tower system designed by CHEMETICS.

Product sulfuric acid is transferred to one of two 3,989 m^3 storage tanks before being pumped to various users across the facility.

18.2.5.2.2 Off Sites – Process Support

The offsites area provides utilities for the sulfuric acid plant, phosphoric acid plant, granulation and SPA plant areas.

Raw water from a nearby lake provides makeup for the fire water, pump seals, process water single users, and water treatment feed. Recovered water from the slurry thickener and pipeline flushes supplement the raw water demand. The fire water system includes a large storage tank (raw water tank) and multiple pumps.

The boiler feed water system consists of ultrafiltration and reverse osmosis systems. Steam is produced from heat recovery system in the sulfuric acid plant. The steam is used to produce power in a turbo generator system as well as provide heat for the evaporators, heaters, and heat tracing. Potable water is to be provided by well and drinking water by truck. A double liming and neutralization system consisting of mixing tanks and settling ponds is provided to treat excess wastewater from the plant. A diesel system includes a fuel storage tank and a metering pump for fire water pumps and an emergency generator.

The instrument and plant air system will be a vendor package and provides air at a -40°C dew point for both instrument and plant air services.

Electrical infrastructure includes a step up transformer for exported electrical power, various MCC rooms, and an emergency generator.

The fatty acid system consists of an unloading pump, storage tank, and recirculation/loading pump. A tank heater is used to maintain the temperature of the acid. Condensate generated is collected in a condensate receiver which is pumped to the condensate collection tank.

A similar system is reserved for sarcosine as well.

The soda ash unloading/loading and storage is a package unit. Soda ash is received via hopper railcar and transfer to a silo. Trucks are loaded from the silo via a rotary valve at the bottom of the hopper before delivery to the beneficiation plant.

Solid lime is received by trucks into a package unit. The package produces lime slurry for neutralization in the wastewater treatment unit.

18.2.5.2.3 Pollution Controls and Waste Disposal

Air

Vapors from the sulfur pit are vented to the scrubbers in the sulfuric acid plant.

Liquid

To minimize the possibility of chemical spills, the sulfuric acid tanks require concrete diking. All railcar and truck unloading and loading stations will have collection pans, pits, and sumps to recover any chemical spills during transfer operations.

All process liquid effluent is recovered and reused or treated as follows:

- Any spills from sulfuric acid tanks and pumps during transfer are collected in the sulfuric acid sump.
- All steam distribution and condensate return lines are insulated.
- In wastewater treatment, water is treated by a double-liming and neutralization system consisting of mixing tanks and settling ponds. Double liming will remove phosphate and fluorine to low levels. The neutralization equipment will discharge water with a controlled pH of 6.0 to 6.5.

Solids

- Lime is received as a solid from trucks into the lime slaker package. Slurry is made as required for wastewater treatment.

18.2.5.3 *Consumptive Use Estimate*

Raw material consumption per tonne of wastewater is quantified in the Table 18-21 details the below.

Table 18-21: Process Support Raw Material Consumption

Raw Material	(kg/t of Wastewater Feed)
Lime	9.80

The estimated electric power consumption of the process support area is 15.4 kWh/t of P_2O_5 as measured against the daily P_2O_5 production at the PAP. Electric power is supplied by a turbo generator at the SAP and requires approximately 1.41 MW per day.

An estimate of the workforce required for the operation of this facility is shown below in Table 18-22.

Table 18-22: Estimated Process Support Workforce

Workforce Category	Value
Management/Supervision (1,2,3)	6.5
Operations(6)	22
Maintenance(3)	7.5
Technical(4,5)	17
Other(6)	22
Total	75

Notes:

1. Superintendent covers the two areas of GRAN and INFRA.
2. Shift supervision and day supervisors cover the two areas of GRAN and INFRA.
3. Maintenance personnel split between the two areas of GRAN and INFRA.
4. Overall FCC Site, technical, and maintenance management included under INFRA.
5. Laboratory included under INFRA.
6. Railroad operations included under INFRA.

18.2.6 Access Roads

The FCC is located near the Highway 11 intersection and will require a 1.5 km access road constructed of asphalt to be built on the east side of Fushimi Road. The access road will initially be gravel during the construction phase and then will be paved for use by personnel and receipt of supplies prior to the commencement of regular operations.

- Access road construction activities identified in this study phase are the following:
  - Tree and bush clearing, removal of stumps and topsoil and excavation of side ditches.
  - Partial excavation depending on subgrade types encountered in route from Fushimi Road to FCC plant site.
  - Backfilling and building a road foundation with crushed granular material for the construction phase of the FCC.
  - Post FCC construction, grading and compacting the road surface and applying asphalt for heavy truck use.

A 3,500 m^2 parking and laydown area will be gravel and located within the plant site area.

18.2.7 Rail Yard

The rail yard will consist of 10 tracks and 2 pullbacks. Three arrival and departure tracks will assist in supporting train movement. In total, over 17 km of track, 8 km of adjacent parallel roads, and 20 turnouts will be required to accommodate the anticipated volume of product at the site.

The rail terminal track is required to accommodate up to 381 cars a week through multiple train deliveries (based on anticipated railcar volumes) and has an ultimate capacity of 508 cars while maintaining the access to the arrival, and pullback tracks.

CN (Canadian National Railway) design standards were followed when determining track and materials for the conceptual design. All rail terminal track turnout sizes are #10 and the Ontario Northland Railway connection turnout is a #12.

The track structure will consist of new 115 rails assembled as jointed track from 12.2 m lengths of rail, hardwood ties at 0.53 m centres, 0.15 m ballast shoulder with 0.2 m of ballast below the tie. A minimum of 0.31 m of sub-ballast will be installed below the ballast. A geotechnical investigation will be required to validate the total granular depth required.

Four parallel access roads provide access along the rail yard for maintenance and accommodate crew and trackmobile flexibility during operations.

Operations at the rail terminal will be supported by a used GP30 locomotive, new Rail King RK 330 trackmobile, as well as two railcar indexers.

The locomotive will be the primary means to switch cars onsite with the trackmobile serving as a backup as well as a primarily focusing on granular product loading and moving smaller cuts of railcars around the FCC site given the limited power of the trackmobile which can only accommodate movement of up to 10 cars at a time.

Railcar indexers were assumed to be cylinder driven progressioners. They will be installed on each granular product loading track and can accommodate 10 cars at a time.

The serving railway will distribute inbound railcars onto the south end of the designated track. Railcars will be loaded and unloaded moving south to north through respective loadouts with non railway equipment. Railcars will then be staged on the departure track with non railway equipment. Railcars will be assembled on the departure track and awaiting pickup from the serving railway.

It is assumed inbound trains to the site will be a mixture of various car types assembled as a manifest train with a limit of 90 railcars per train. Should larger trains be inbound, staging them in the yard at Hearst, Ontario, would be required prior to arriving onsite.

18.2.8 Cogeneration Plant and FCC Distribution

Electrical power is produced at the FCC by routing high pressure steam from the SAP to the Turbine-Generator (TG) system. The TG is a multistage condensing unit with extraction capabilities described in further detail in other sections of this report. Nominal production is 31 MW which provides power for the FCC and, additionally, export capacity to the local power grid (HONI) for distribution to the mine site. The HONI connection is noted in Section 18.1.2 for further description of the external power connection. Distribution within the FCC is accomplished by multiple MCC located adjacent and sometimes within the various operating units.

FCC standby emergency power load is currently provided by a single 1.7 MW diesel generator to cover emergency loads in the various units.

The standby power plant will only operate off grid. Therefore, an interlocking system which prevents paralleling the plant to the grid will be required.

18.2.9 Natural Gas Supply

Natural gas is utilized at the FCC site continuously with granulation production and infrequently as a heat source for a sulfuric acid plant cold start-up. It is envisioned that the local gas distribution company will install the feeder pipeline between the nearby gas supply line and the FCC. Any isolation, reducing station, and feeder pipeline protection (pressure relief station) would be located inside the FCC property line but fall under the gas company responsibility. The installation cost will be recovered through gas rates.

The distribution system within the FCC will be installed in accordance with Ontario governing regulations as applicable. Unit isolation valves, additional regulators and relief devices will be designed and specified as deemed necessary in the next phase.

18.2.10 Utilities

18.2.10.1 *Potable water*

Potable water will be provided to the FCC from a contractor constructed well to be drilled at a location strategic to the final arrangement of the plant infrastructure. The well will be constructed in compliance with Regulation 903 for Well under the Ontario Water Resources Act, R.S.O. 1990, c. O.40. The water obtained from the well will be tested and treated for safe use for site ablutions, sanitary facilities, and emergency showers.

The well will be capped with a distribution pump, protected from extreme weather conditions, and freezing. Water will be circulated to the site buildings through buried, and heat traced, distribution lines.

All drinking water will be provided from an external supplier and received by truck into a segregated storage tank. Drinking water is distributed to end users via pump and a drinking water network to all operation and maintenance buildings to be designed and specified in the next phase.

18.2.10.2 *Raw Water*

Subject to necessary permitting approvals, raw water will be drawn from a nearby lake to provide makeup for the process water, fire water and boiler feed systems. A buried and wrapped 300 mm diameter carbon steel water line of an estimated length of 2,500 m will be installed to provide 1600 m^3/hr of raw water (peak).

The system is designed to operate with two pumps running and a spare available for maintenance. The intake supply will be strained with inline filters before discharge into the 2,561 m^3 raw water tank (dual service as noted in following section). Raw water is then distributed throughout the FCC for various uses.

18.2.10.3 Fire Protection System

The protection of all FCC infrastructure against the risk of fire across the site will be analyzed in greater detail in the next phase of study as part of a full risk assessment during design. For the PEA, the fire water system includes a 2,561 m^3 storage tank with a 680 m^3/hr electric pump which feeds the fire loop. The system includes a diesel backup pump and a jockey pump to maintain system pressure in the fire loop.

All FCC administration, operations, and maintenance buildings will be protected by fire extinguishers with the number and type as specified during the risk assessment.

Additionally, MCC and control system rooms may be equipped with automatic fire suppression systems as required by local regulations and insurance underwriters.

18.2.10.4 Sewage Treatment Package

Sewage treatment will be managed by a vendor supplied (off the shelf), fit-for-purpose, system comprising a self-contained biofilter tertiary wastewater unit. The sewage treatment will be near the wastewater treatment for discharge. This system will operate year round using a low energy, foam bacterial medium to break down and remove contaminants. Treated water will be compliant for quality for discharge to the wastewater treatment unit for disposal. Periodic pumping out of residual waste will be conducted by a qualified contractor for off site disposal in a sewage treatment facility.

Sewer routing, collection, and lift stations will be assessed during the next phase.

18.2.10.5 Wastewater Treatment

FCC wastewater is treated by a double liming and neutralization system consisting of mixing tanks and settling ponds. It is estimated at treating 100 m^3/hr (design rate) of wastewater from the plant. The treated wastewater will be discharged into the same lake that supplies the raw water makeup. A buried and wrapped 300 mm diameter carbon steel pipeline of an estimated length of 3,500 m will be used to handle the treated wastewater. Removal of residual waste from the ponds will be conducted by a qualified contractor for off-site disposal.

18.2.10.6 Water Treatment

Water treatment for the FCC consists of two stages. In the first stage, raw water and reclaimed water from the rock slurry thickener are combined in a 330 m^3 surge tank before feeding two ultrafiltration modules running in parallel. Reject from the ultrafiltration is recycled back for use in the PAP unit. Filtered water discharges to a 240 m^3 surge tank. In the second stage, filtered water is fed to a single reverse osmosis (RO) module. The RO discharge (permeate) is sufficient for boiler feed water system make-up necessary for sulfuric acid plant operations. Reject from the RO module is routed to wastewater treatment for disposal.

Condensate from all operating units is collected and monitored for quality and returned to the sulfuric acid unit. Provisions for monitoring the quality of the condensate, and disposition should the quality become off spec, are not considered in this phase of the project.

18.2.10.7 *Fuel Storage*

Diesel fuel for mobile equipment is received by truck and stored in a 44 m^3 storage tank protected by secondary containment. It is envisioned that the fuel storage will be equipped with a fire protection system with automatic and remote activation. The risk of fire at the fuel storage will be analyzed in greater detail in the next phase of study as part of a full risk assessment during design.

18.2.10.8 *Compressed Air Systems*

The FCC instrument and plant air are provided as a pre-engineered package. Air is produced at a -40°C dew point for both instrument and plant air services. Compressed air is produced at 7.0 barg and 25°C (plant and instrument).

18.2.11 Ancillary Buildings and Services

The FCC offices, buildings, and enclosures are shown in Table 18-23 below. For this PEA, building sizes and construction are based on the current design and expected to be further optimized in the next phase. Building construction is primarily steel frame on foundation covered with insulated panels. Other structures employ cinder blocks. Minimized distances and interconnecting walk corridors will also reduce the requirement for travel between buildings in inclement weather conditions.

Table 18-23: FCC Ancillary Buildings and Services

Building	Number	Unit
Administration Offices and Parts Warehouse[(1)]	7,500	m^2
Maintenance and Warehouse Facilities	7,500	m^2
Water Treatment Building (includes air compressors)	5,810	m^2
Main MCC Building	1,250	m^2
Reagent Storage Building	375	m^2
Soda Ash Building	375	m^2
Sanitary Sewer Building	120	m^2
Wastewater Treatment Building (settling ponds)	7,700	m^2
Rock Slurry Receiving	5,625	m^2
Railcar Covering – Sulfur, MGA, Reagent	1,806	m^2
Railcar Covering – Phosphoric and Reagent	576	m^2
Railcar covering – NH_3 Unloading	576	m^2

Note:

1. Includes Operational/Technical Offices, lunchroom, and Change Room.

18.2.12 Site Preparation

Site preparation at the FCC is expected to be a conventional operation where a thin surface organic layer of muskeg (assumed to be an average of 300 mm thick) is stripped off to a more competent layer or bedrock and replaced with a structural backfill. New storm water features will be added to the site. The FCC covers an area of 700 m x 650 m (excluding railyard) and the initial gypsum stack is 2,160 m x 1,000 m which includes the pond for process cooling. Both the FCC and gypsum stack plots have patchy tree cover.

Gypsum stacking will be a wet system with an associated cooling pond. Preparation of the initial gypsum stack area is similar in that the surface organic layer is stripped off and replaced with a structural backfill. It is assumed that the exposed glacial material or bedrock will support a stack height of 50 m. These assumptions need to be confirmed by future geotechnical testing.

Initial construction for the gypsum stack area will require access to be established on the west side of Fushimi Road, north of the rail line (Ontario Northland Railway).

During the construction, a shallow perimeter ditch will be dug around the stack area and the excavated material will be used to construct a low berm on the outside of the ditch. The purpose of the perimeter ditch and berm are to collect stack runoff and prevent runoff from leaving the site and to provide the circulation circuit for the cooling pond. The cooling pond is currently envisioned on the south side of the gypsum stack. Returning cooling water (hot) from the FCC will discharge into the channel with the direction of flow around the gypsum stack. This allows for maximum heat loss before the pond water is returned to the FCC for process cooling.

The gypsum stack, perimeter ditch, and the cooling pond are all built upon a thick HDPE liner preventing seepage into the soil below. Gypsum stack would be designed and specified in the next phase.

Site preparation for infrastructure will focus on the follow activities for both areas:

- Diversion of impacted creeks (if present).
- Preparation of site roads.
- Preparation of site infrastructure foundation locations.
- Establishment of pipeline corridors.
- The need for protective berms to prevent flooding (around site infrastructure area and other locations as needed) will be determined from additional geotechnical investigation in a future phase.

The following assumptions have been made with respect to site preparation:

- FCC site preparation work will not be possible until the initial phase of the FCC access road is completed.
- Gypsum stack preparation will not be possible until access from Fushimi Road is established to the gypsum stack area. Sequence planning should allow for the FCC work to proceed first.
- Tree cutting can occur without drainage of the muskeg.
- Muskeg cannot be effectively excavated until most of the contained water, if present, has been drained from the areas to be removed.
- The most effective means to manage the dewatering process is to utilize the natural gradient of the land to collect water in lower lying areas.
- The drained water will be contaminated with sediments and organic matter though can be directed to naturally lower lying areas for disposal.
- The period to establish the drainage system will commence in the winter months when the muskeg layer is more manageable with lower water flows.

- The requirement for material for berms and other barriers to water inflow is available from site stripping activities or, initially, from a nearby borrow pit as pit run material.
- Battery limit for site preparation is to the glacial till layer or bedrock, whichever is first exposed at shallow depth. The supply and installation of engineered fill, and other work to prepare building foundations, is part of construction.
- The duration of the initial site dewatering should be no longer than one year though it will be dependent upon the seasonal conditions when this work starts.

The sequence will follow these steps upon site access completion of the FCC access road and tree cutting:

1. The equipment to be used for dewatering activities will be determined by the contractor employed to do the work and mobilized to the FCC.
2. Creation of trenches within the muskeg and excavated to the glacial till horizon channel direct water to collection sumps. The location and size of the trench network will be determined in a future phase of study as the amount of muskeg at the FCC is expected to be less than at the mine site.
3. Concurrent establishment of collection sumps to receive water directly from the drainage trenches in lower lying areas. The design, location and size of the sumps will be determined in a future phase of study.
4. Water collected in the sumps will be retained for a short time to enable heavy sediments to settle and then pumped through pipelines to locations which will not drain back to the excavated areas. Management of organics will be required to prevent clogging of pumps. The type and size of the pumps will be determined in a future phase of study although each pump will have a self-contained and built in diesel fuel supply to allow independent operation and strategic placement of sumps.

If necessary, limited perimeter berms will be constructed around drained areas to prevent flow of water from areas which are not being excavated. This will enable removal of muskeg inside the berms. The berm locations and extent will be determined in the field dependent upon the conditions which arise during excavation. Any muskeg removed in the clearing process can be stockpiled for later uses as determined in the next phase.

The degree of geotechnical information for the FCC site is limited and will require further investigation in a future phase.

18.2.13 *Mobile Equipment*

FCC mobile equipment listed in Table 18-24 encompass the current design of both the FCC and the railyard.

Table 18-24: FCC Mobile Equipment

Equipment	Number
Pickup Trucks	3
Bobcat	1
Front End Loader	3
Fork truck	2
Snow Removal (Grader)	1
GP30 locomotive	1
Track Mobile (RK330)	1

19. Market Studies & Contracts

19.1 Introduction

A market, transportation and logistics study was conducted to determine the preliminary economic analysis of the Project. The results demonstrate that the phosphate market and industry is attractive, and that this project is competitive with other facilities from around the world which currently supply the Project's target markets. The first part of this study examines the highly dynamic current situation and near term outlook as well as long term prospects for the global and North American phosphate markets. The second part of the study includes estimates of the total addressable market (TAM) and detailed delivered cost estimates to target market destinations from the FCC.

19.2 Summary

Long term supply and demand fundamentals are positive, and prices over the life of the Project will exceed the values required to generate a threshold or better IRR.

The positive outlook is based on the need for this sector to build the capacity needed to meet projected global demand during the next two decades. It is expected that the demand for phosphate products produced from phosphoric acid will increase by close to 18 mt P_2O_5 between 2020 and 2040, which is in line with recent long term forecasts from the International Fertilizer Association (IFA).

Phosphate demand forecasts are underpinned by steady increases in food production to meet the needs of a growing and more affluent global population. In addition, developments such as the exponential growth of renewable diesel and sustainable aviation fuel production as well as increases in Chinese feed grain imports to supply a restructuring hog industry are expected to be important drivers for North American and global phosphate demand. The predicted growth in lithium iron phosphate battery production within China could further focus the Chinese phosphate industry on the domestic rather than the international export markets.

The supply side assumes 15.7 mt P_2O_5 of new phosphoric acid capacity will come online between 2020 and 2040. This implies that the global operating rate will need to increase from 87% today to 90%-92% throughout the forecast period, and that is constructive for the price outlook.

The addition of specific projects is speculative, and Morocco and Saudi Arabia are expected to account for roughly 40% and 30%, respectively. It is also assumed that at least two world scale greenfield projects will move forward in Brazil and Algeria. The analysis also includes the forecasted contribution from the Project.

Two US facilities are expected to close during the forecast period. Furthermore, the world has become dependent on Chinese phosphate exports during the last two decades, however Chinese supplies are uncertain if not highly suspect in the long term. Furthermore, additional mine closures than have been indicated are possible.

Based on the operating cost estimates and current freight rates identified in the PEA, the Martison facility is expected to rank at the low end of delivered cost curves for nearly all of its target markets.

Martison cash operating costs are forecasted to rival the lowest cost offshore producers which benefit from either low cost phosphate rock (Morocco) or low cost natural gas and integrated ammonia production (Saudi Arabia and Russia). The cost advantage of the Martison operation is based on the cost and quality of its phosphate rock, lower sulphur procurement and transportation costs, and competitive ammonia costs. The Martison fob plant costs are less than those estimated for its four US competitors.

Finally, lower transportation and logistics costs to target markets cement the Project's cost advantage to its target markets. Charts later in this section show that Martison sits at the far left of the delivered cost curve to the Canadian provinces and the US northern tier states.

19.3 Market Overview

19.3.1 Global Market Overview

19.3.1.1 Phosphate Rock Production and Trade

Global phosphate rock production totaled 207.9 mt in 2020. CRU estimates that output increased to 211.3 mt in 2021. Production has increased from 130 mt in 2000 to a peak of 213 mt in 2015. Since then, global rock output has stabilized at 210 mt per year give or take three mt in any given year. Global phosphoric acid production has increased moderately since 2015, implying a drawdown of rock inventories worldwide.

The Table 19-1 below lists the top 10 rock producing countries during the last three years and Figure 19-1 identifies the growing world rock production supply since 2000. The top five producing countries accounted for more than three quarters of global production and the top 10 claimed almost 90% of the total during the last three years.

Recent statistics mask a few important trends. The table shows production by key country. After increasing sharply from 2000 to 2015, annual Chinese production dropped 17.1 mt between 2015 and 2020.

Annual output in Morocco and Saudi Arabia increased 10.7 and 4.2 mt, respectively, during the same period. US output continued to trend down, dropping 4.4 mt since 2015.

Table 19-1: Top 10 Producing Countries (Past 3 Years)

Rank	1000 Tonnes	Production 2018-20 Average	Cumulative Production	Share	Cumulative Share	CAGR 2000-21
1	China	77,695	77,695	37%	37%	6.6%
2	Morocco	35,530	113,225	17%	54%	2.5%
3	United States	24,237	137,462	12%	66%	-2.4%
4	Russia	13,721	151,183	7%	73%	1.0%
5	Jordan	8,731	159,914	4%	77%	2.0%
6	Saudi Arabia	7,710	167,624	4%	81%	na
7	Egypt	5,523	173,147	3%	83%	8.6%
8	Brazil	4,935	178,081	2%	86%	0.6%
9	Peru	3,651	181,733	2%	87%	36.2%
10	Vietnam	3,253	184,986	2%	89%	7.3%
	Other	22,938	207,923	11%	100%	-5.9%
	Total	207,923		100%		2.2%

Source: CRU August 2021

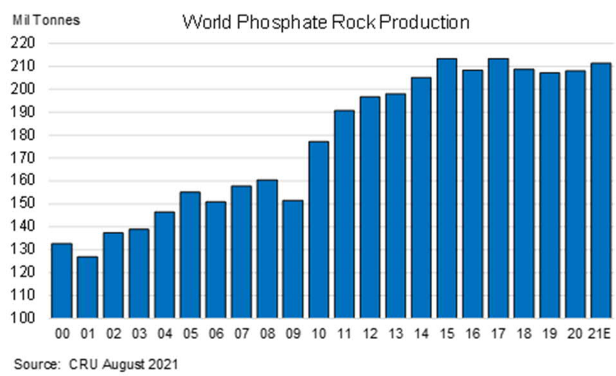


Figure 19-1: World Phosphate Rock Production

The production of finished phosphate products takes place in chemical complexes that range from fully integrated to non-integrated operations. Fully integrated operations mine phosphate rock, burn sulphur and may even manufacture ammonia to produce finished phosphate products. Non-integrated producers purchase rock or phosphoric acid as well as other inputs to fabricate final products. Figure 19-2 identifies the primary world rock production by country from 2000-2021.

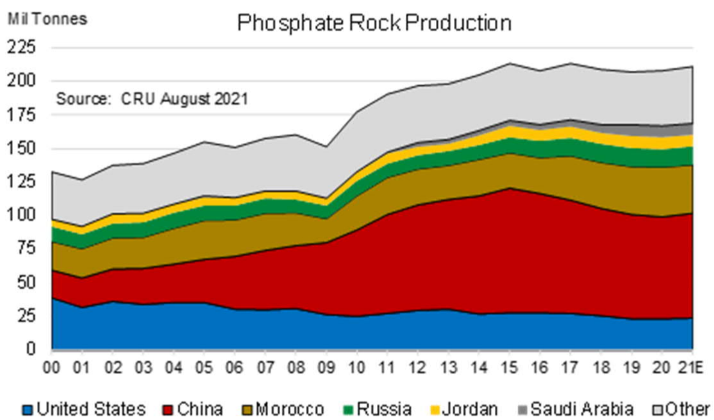


Figure 19-2: Primary Phosphate Rock Producers (2000-2021)

India is a good example. The second largest phosphate consuming country possesses only small deposits of phosphate rock. As a consequence, India relies heavily on imports for nearly all of its phosphate needs. The country has diversified sources by stage of processing, importing raw materials (rock and sulphur), intermediate products (phosphoric acid and ammonia) as well as finished products (DAP/NP/NPS/NPKs).

There is significant trade in the raw material (rock) as well as the intermediate product (phosphoric acid). Figure 19-3 shows that phosphate rock trade has stayed relatively stable at roughly 30 mt per year which makes sense in that non-integrated complexes typically operate and have consistent demand over time.

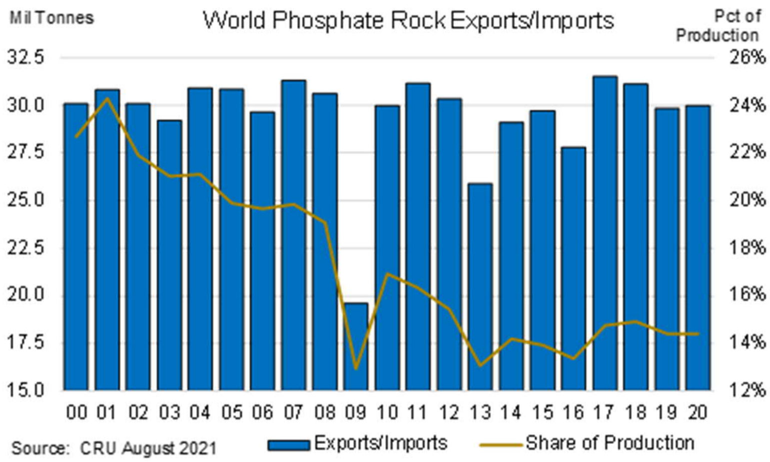


Figure 19-3: Annual Trade In Phosphate Rock

There have been some changes such as the closure of a few non-integrated operations (MissPhos Pascagoula) and switching from domestic to mostly imported rock by others (Mosaic Uncle Sam). Nevertheless trade, as a percentage of production, has trended down about 10 percentage points since 2000 (Figure 19-3). Table 19-2 and Table 19-3 shows the top 10 rock importing and exporting countries, respectively. These highlight the dominance of Morocco and India. The USA ranks third among importers.

Table 19-2: Top 10 Importing Countries

Rank	1000 Tonnes	Imports 2018-20 Average	Cumulative Imports	Share	Cumulative Share
1	India	7,295	7,295	24%	24%
2	Indonesia	2,584	9,879	9%	33%
3	United States	2,412	12,291	8%	41%
4	Brazil	2,094	14,385	7%	47%
5	Mexico	1,829	16,214	6%	53%
6	Lithuania	1,352	17,566	4%	58%
7	Poland	1,142	18,708	4%	62%
8	Turkey	948	19,657	3%	65%
9	Belgium	795	20,451	3%	67%
10	Norway	716	21,167	2%	70%
	Other	9,140	30,307	30%	100%
	Total	30,307		100%	

Source: CRU August 2021

Table 19-3: Top 10 Exporting Countries

Rank	1000 Tonnes	Exports 2018-20 Average	Cumulative Exports	Share	Cumulative Share
1	Morocco	10,331	10,331	34%	34%
2	Jordan	4,630	14,962	15%	49%
3	Peru	3,645	18,607	12%	61%
4	Egypt	3,092	21,699	10%	72%
5	Russia	2,408	24,107	8%	80%
6	Algeria	1,323	25,429	4%	84%
7	Togo	995	26,424	3%	87%
8	Israel	651	27,075	2%	89%
9	Kazakhstan	613	27,689	2%	91%
10	Syria	413	28,102	1%	93%
	Other	2,205	30,307	7%	100%
	Total	30,307		100%	

Source: CRU August 2021

19.3.1.2 Phosphoric Acid Production

Phosphoric acid is the intermediate product that is used to produce most high analysis final products. Phosphoric acid provides a comprehensive measure of phosphate capacity and production, and it is used in the supply/demand analysis later in this section. The increase in world phosphoric acid production from 2000-2021 is illustrated in Figure 19-4.

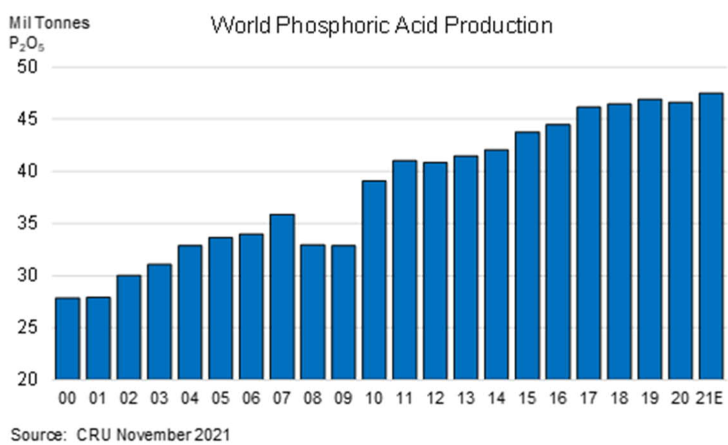


Figure 19-4: World Phosphoric Acid Production

Global phosphoric acid production totaled 46.6 mt P_2O_5 in 2020. CRU estimates that production increased to 47.5 million in 2021. Annual production has increased at a CAGR of 2.5% since 2000 or from 28 mt at the beginning of this century to an estimated 47.5 million in 2021. Output flattened at about 46.5 mt give or take about one half mt from 2017 to 2020. Recent demand gains, in response to higher crop prices, are stressing supply and fueling the price rally that began in mid-2020.

Table 19-4 lists the top 10 phosphoric acid producing countries during the last three years. The list is similar to the rock producing countries with a few exceptions. China also ranks as the largest acid producer with annual output of 17.2 mt P_2O_5, accounting for 37% of global output during the last three years. Morocco ranks second with annual output of 6.6 mt and claiming 14% of the total. The top five countries accounted for 77% of global production and the top 10 claimed 88% of the total during the last three years. While India ranked as the sixth largest producer, production is from non-integrated operations that rely on imported rock.

Acid production trends are similar to rock trends as shown in Table 19-4 and Figure 19-5. After peaking at 18.0 mt P_2O_5 in 2015, Chinese output declined to 16.6 mt in 2020. US acid production also continues to trend lower despite the recent price fly-up. These declines were offset by gains mostly from Morocco, Saudi Arabia and Russia.

Table 19-4: Top P_2O_5 Producers (2018-2020)

Rank	1000 Tonnes P_2O_5	Production 2018-20 Average	Cumulative Production	Share	Cumulative Share	CAGR 2000-21
1	China	17,217	17,217	37%	37%	10.3%
2	Morocco	6,644	23,861	14%	51%	4.6%
3	United States	6,435	30,296	14%	65%	-2.6%
4	Russia	3,630	33,926	8%	73%	3.2%
5	Saudi Arabia	2,064	35,990	4%	77%	na
6	India	1,729	37,719	4%	81%	2.3%
7	Brazil	1,174	38,893	3%	83%	1.4%
8	Jordan	996	39,889	2%	85%	3.6%
9	Mexico	610	40,499	1%	87%	-1.0%
10	Israel	574	41,073	1%	88%	0.9%
	Other	5,599	46,672	12%	100%	2.5%
	Total	46,672		100%		2.6%

Source: CRU November 2021

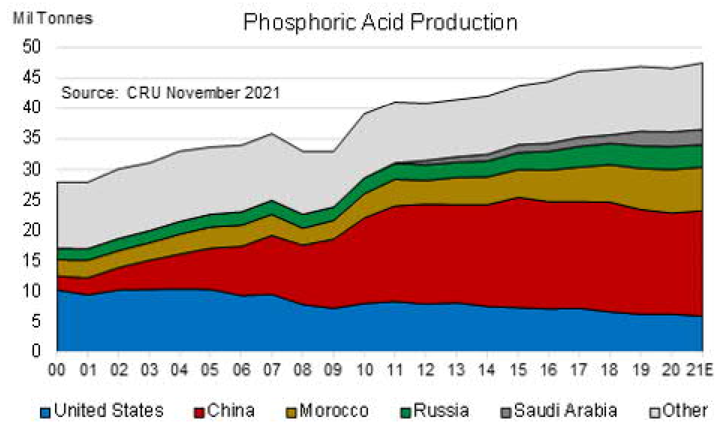


Figure 19-5: Phosphoric Acid Production (2000-2021)

19.3.1.3 Phosphoric Acid Trade

Phosphoric acid trade has averaged 4.1 mt P_2O_5 during the last three years (Figure 19-6). Since 2000 trade has trended down slightly though several non-integrated facilities especially in India and Pakistan operate at consistent rates from year to year.

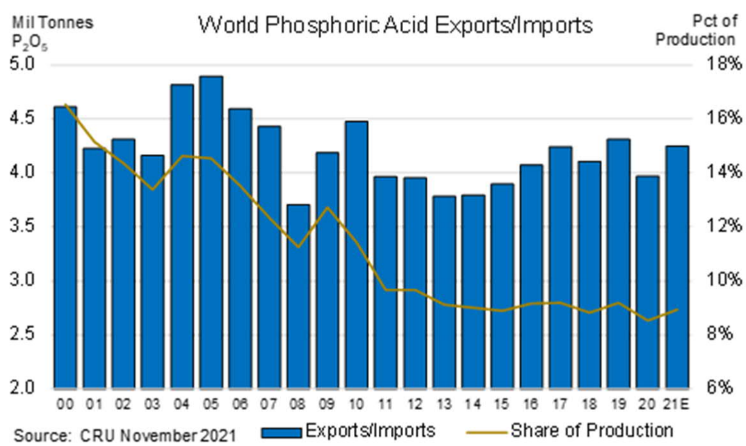


Figure 19-6: World Phosphoric Acid Exports/Imports (2000-2021)

Nearly all new phosphate capacity in China, Morocco and Saudi Arabia is integrated from mining to granulation. As a result, trade as a percentage of production has dropped in half from 16% in 2000 to 8% in 2020.

Phosphoric acid trade is dominated by Morocco and India (Table 19-6). Morocco is, by far the largest exporter, shipping about 1.9 mt P_2O_5 of acid each year and accounting for 46% of global trade. Jordan (JPMC) is the next largest supplier, exporting 645,000 t P_2O_5 per year and claiming 16% of the total. Both Moroccan (OCP SA) and Jordanian (JPMC) producers have joint venture acid operations with Indian partners. The top three acid exporters accounted for nearly three quarters of total exports during the last three years.

On the demand side of the ledger, India imports 2.4 mt P_2O_5 of acid each year. Granulation facilities dot the Indian coast where fabricators import acid and ammonia and produce DAP/NP/NPS/NPK products. India alone accounts for 58% of total imports. Morocco is the largest Indian supplier, but Indian fabricators also source acid from most of the top exporters including the United States.

Pakistan ranks a distant second importing about 365,000 t P_2O_5 per year mostly from Morocco (Table 19-5). Several European countries import acid to make NP and NPK products.

Table 19-5: Top P_2O_5 Importers (2018-2020)

Rank	1000 Tonnes P_2O_5	Imports 2018-20 Average	Cumulative Imports	Share	Cumulative Share
1	India	2,379	2,379	58%	58%
2	Pakistan	367	2,745	9%	66%
3	France	192	2,937	5%	71%
4	Turkey	174	3,111	4%	75%
5	Netherlands	141	3,253	3%	79%
6	Belgium	131	3,383	3%	82%
7	Spain	124	3,508	3%	85%
8	Brazil	101	3,609	2%	87%
9	Mexico	71	3,680	2%	89%
10	Bangladesh	43	3,722	1%	90%
	Other	406	4,128	10%	100%
	Total	4,128		100%	

Source: CRU November 2021

Table 19-6: Top P_2O_5 Exporters (2018-2020)

Rank	1000 Tonnes P_2O_5	Exports 2018-20 Average	Cumulative Exports	Share	Cumulative Share
1	Morocco	1,888	1,888	46%	46%
2	Jordan	645	2,533	16%	61%
3	Senegal	472	3,005	11%	73%
4	Tunisia	288	3,292	7%	80%
5	United States	244	3,536	6%	86%
6	Israel	156	3,692	4%	89%
7	Belgium	117	3,809	3%	92%
8	South Africa	105	3,914	3%	95%
9	Mexico	50	3,965	1%	96%
10	Lebanon	42	4,006	1%	97%
	Other	122	4,128	3%	100%
	Total	4,128		100%	

Source: CRU November 2021

19.3.2 North America Market Overview

Canada, the key target market for the Martison project, is a growth story that few analysts have noticed. Phosphate deliveries increased 7.5% per year or 1.1 mt during the last decade. Demand has climbed as a result of significant increases in crop production, which increased 44% or 30 mt from ~70 mt in 2010 to ~100 mt in 2020 due to increases in both planted area and yields.

Canadian deliveries (mostly MAP and NPS) are estimated to grow from 2.2 mt in 2020 to 2.6 mt in 2022. CRU forecasts that deliveries will increase to 3.3 mt by 2025. Further growth is the result of small increases in wheat and canola area and continued yield increases that require higher phosphate application rates.

19.3.2.1 Canada Phosphate Production Ends in 2019

Canada no longer produces phosphate. The Nutrien (formerly Agrium) Redwater facility ceased phosphate operations in early 2019 and was repurposed to produce ammonium sulphate. As a result, Canada now relies on imports of finished products for all its phosphate needs.

The Redwater facility had produced on average 550,000 t of MAP per year during the 10 years prior to its closure The facility ran on imported rock from Morocco following the mine-out of the Agrium Kapuskasing rock mine in 2013.

The announcement of the closure of the Redwater facility opened the Canadian market to U.S.A., Morocco and Russia. The combination of strong demand growth and the loss of the only domestic supply has resulted in extraordinary growth in finished phosphate imports since 2014 as illustrated in Figure 19-7.

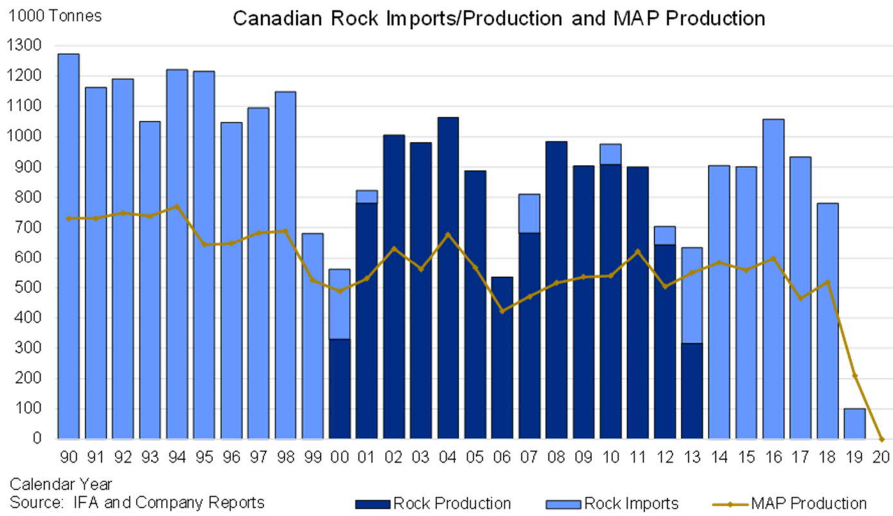


Figure 19-7: Canadian Rock Imports/Production & MAP Production

19.3.2.2 *Canadian Imports by Product and Province of Entry*

As noted above, Canada now relies on imports of finished products for all of its phosphate needs. Canadian MAP/NPS/DAP imports have increased from less than 700,000 t at the beginning of the last decade to 2.43 mt for the fertilizer year that ended on June 30, 2021, representing an increase of 1.7 mt and a CAGR of 12% during this period. This growth is illustrated by product type and by region in Figure 19-8 and Figure 19-9 respectively.

MAP imports totaled 1.69 mt or 69% of phosphate imports in 2020/21. NPS products registered 607,000 t or 25% of the total, up from less than 5,000 t in 2009/10 (but importers may have declared some NPS as MAP during this period and before HS codes were established for NPS). MAP and NPS captured 94% of the growth in Canadian phosphate imports from 2010 to 2021. DAP imports totaled 136,000 t in 2020/21 or 6% of the total. DAP imports increased in 2020/21 but remain far below MAP/NPS imports.

The Stats Canada statistics provide imports by province of entry. The import statistics consolidated into the Prairies, Big East and Other provide reasonably reliable estimates of where the products went down on farm fields.

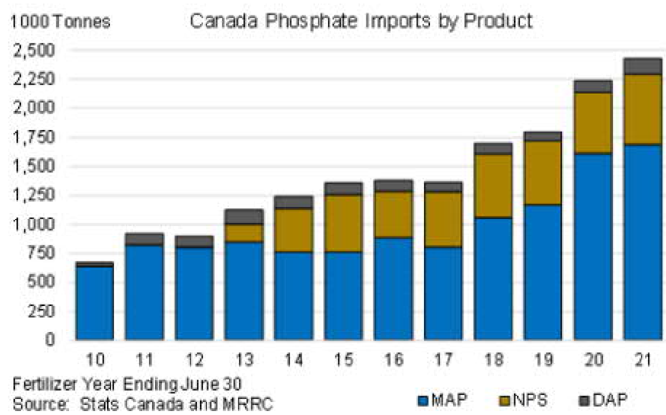


Figure 19-8: Canada Phosphate Imports By Product Type

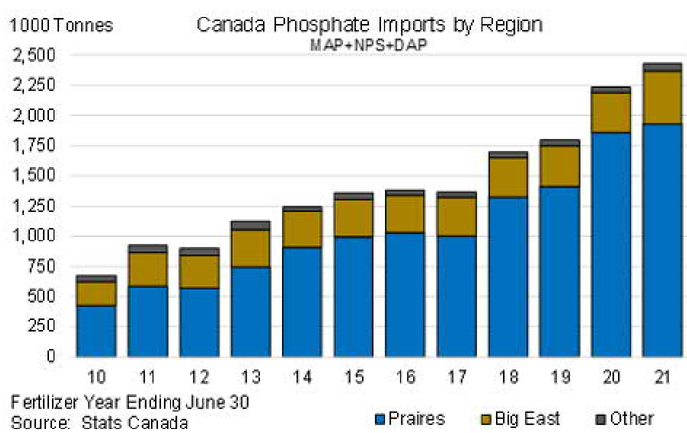


Figure 19-9: Canada Phosphate Imports By Region

19.3.2.3 Canadian Imports by Origin

The United States dominates phosphate trade with Canada (Figure 19-10 and Figure 19-11) and has captured most of the large growth in Canadian demand during the last decade. According to Stats Canada statistics, the United States exported 2.00 mt of MAP/NPS/DAP to Canada 2020/21, accounting for 82% of the total. US exports climbed 1.4 mt from 2010 to 2021, capturing 77% of the increase during this period. Morocco shipped 295,000 tonnes or 12% of the 2020/21 total while Russia exported 112,000 tonnes or 5% of the total. Much of the increase in imports from Morocco and Russia came after the US Counter Vailing Duties (CVD) petition was filed on June 26, 2020.

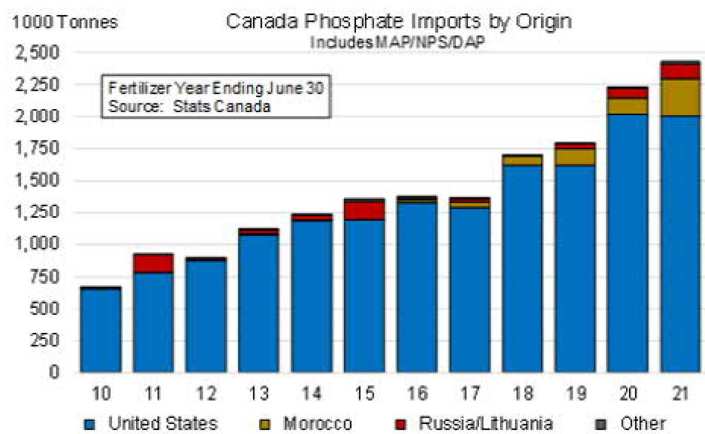


Figure 19-10: Canada Phosphate Imports By Origin

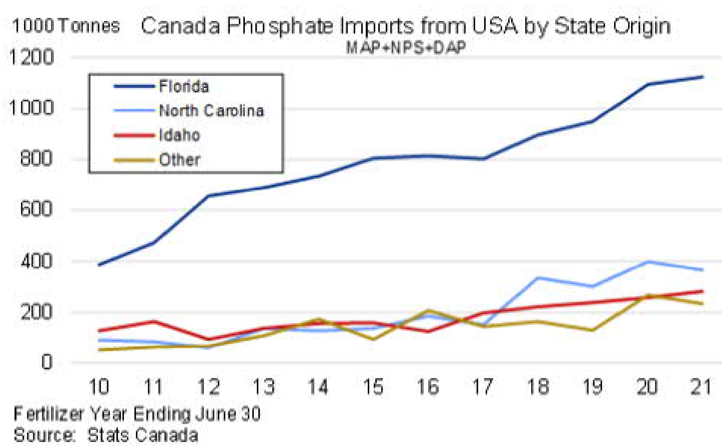


Figure 19-11: Canada Phosphate Imports By USA State Origin

19.3.2.4 Canadian MAP Imports

Canadian MAP imports surged following the announcement by Nutrien in January 2017 that it planned to close its Redwater facility (operated by Agrium) and import product from its US operations in North Carolina and Florida (operated by PotashCorp). Other US and offshore suppliers immediately set out to claim a share of the lost tonnage from the closure. Canadian MAP imports prior to 2017/18 were stable at about 800,000 t per year. The MAP imports by region and by origin are shown in Figure 19-12 and Figure 19-13 respectively.

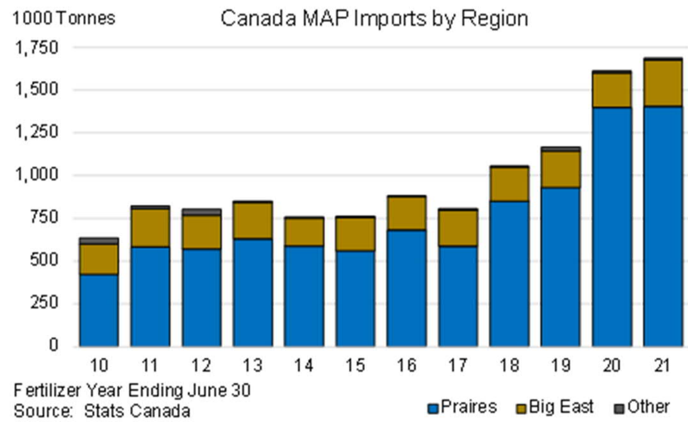


Figure 19-12: Canada MAP Imports By Region

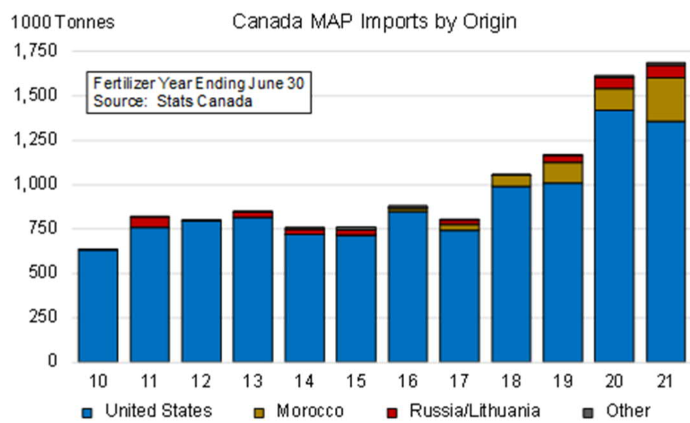


Figure 19-13: Canada MAP Imports By Origin

Canada imported 1.69 mt of MAP in 2020/21, more than double the level prior to 2017/18. MAP imports replaced the lost Redwater tonnage and captured a large share of the growth in phosphate demand. The Prairies dominate MAP shipments and imports. MAP imports into the region totaled 1.40 mt in 2020/21 or 83% of the total. The Prairies accounted for 93% of the growth in MAP imports from 2010 to 2021. Imports into the two Big East provinces totaled 274,000 tonnes in 2020/21 and claimed 16% of the total. Imports cleared in the other provinces registered just 8,000 t and accounted for less than 1% of the total. The United States exported 1.36 mt of MAP to Canada in 2020/21, accounting for 80% of the total. However, that is down from an average share of 95% from 2010 to 2017. Shipments from all three states where MAP is produced have increased since 2017.

Morocco shipped 243,000 t of MAP and accounted for 14% of the total, up from virtually nothing five years ago. Russia accounted for the remaining 4%. Suppliers from both countries have Canada in their crosshairs in response to US CVD.

19.3.2.5 *Canadian NPS Imports*

Gains in NPS imports have captured a big chunk of the recent growth in phosphate demand. NPS imports climbed to a record 607,000 t in 2020/21 versus an average of 542,000 t during the previous three years and nearly nothing in 2010.

The Prairies also account for the lion's share of Canadian NPS imports. Imports into this region totaled 523,000 t in 2020/21 or 86% of the total. The Prairies accounted for 87% of the growth in NPS imports from 2010 to 2021. Imports cleared into the two Big East provinces totaled 77,000 mt or 12% of the total.

The United States dominates the Canadian NPS market today. Imports from the United States totaled 604,000 tonnes and accounted for 99% of Canadian NPS imports in 2020/21. Imports originating from Florida accounted for the bulk of US imports. Shipments from the Sunshine state totaled 530,000 t and accounted for 88% of US exports to Canada. In addition, many of the imports originating from warehouses in other states such as Minnesota were likely produced in Florida. The Nutrien facility in North Carolina does not produce NPS, and the two producers in Idaho (Simplot and Itafos) accounted for only 4% of US NPS exports to Canada in 2020/21. The NPS imports by region and by origin are shown in Figure 19-14 and Figure 19-15 respectively.

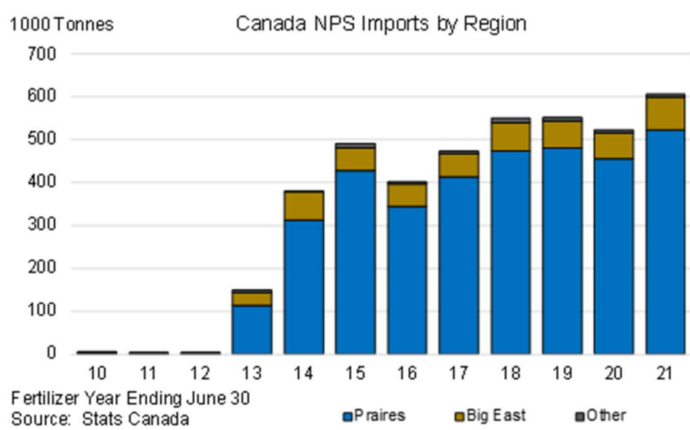


Figure 19-14: Canada NPS Imports By Region

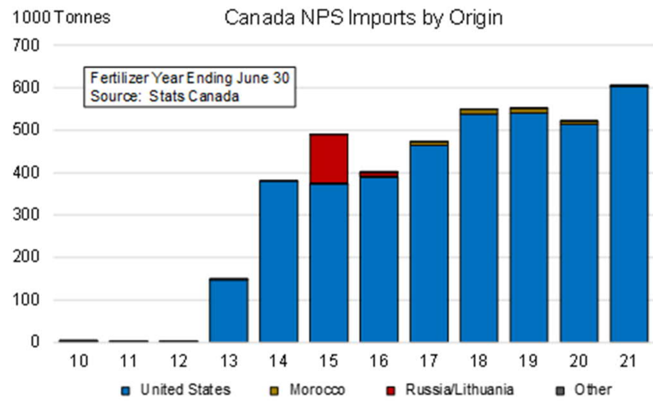


Figure 19-15: Canada NPS Imports By Origin

19.3.2.6 Canadian DAP Imports

Canadian DAP imports are relatively small and concentrated in the eastern provinces. After trending down for several years, DAP imports rebounded during the last two fertilizer years. Imports totaled 136,000 t in 2020/21, up from a five year average of 87,000 t. Recent gains are probably the result of higher corn, wheat and soybean prices for row crop producers in Ontario and Quebec.

The United States had dominated exports until 2020/21, accounting for 96% of Canadian DAP imports during the previous five years. Nearly all US product was shipped by Mosaic from Florida and via vessel from Tampa into the Montreal/Contrecour. Following the United States levying countervailing duties on phosphate imports from Morocco and Russia, producers in both countries increased DAP exports to Canada. Morocco and Russia shipped 51,000 t and 40,000 t of DAP to Eastern Canada in 2020/21, accounting for 38% and 29% of the total, respectively. The DAP imports by region and by origin are shown in Figure 19-16 and Figure 19-17 respectively.

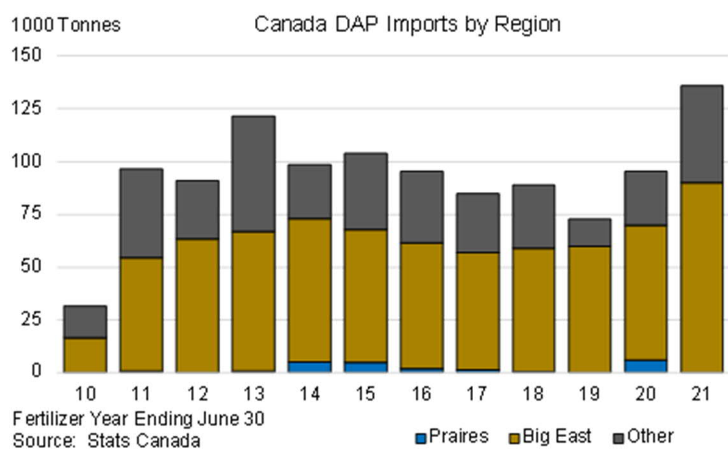


Figure 19-16: Canada DAP Imports By Region (2010-2021)

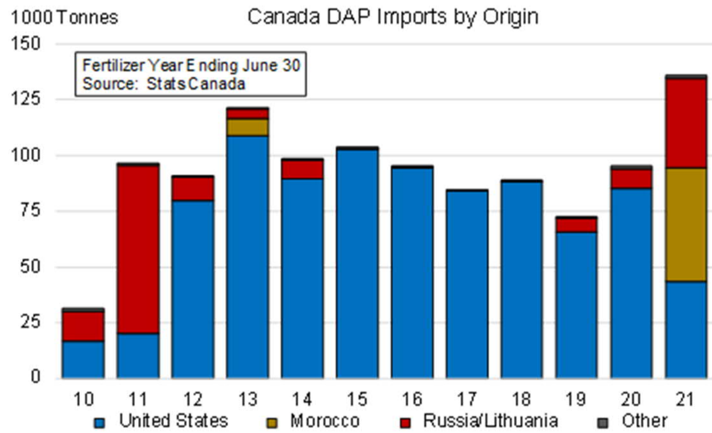


Figure 19-17: Canada DAP Imports By Origin (2010-2021)

19.3.2.7 Canadian Imports by Origin

Table 19-7 below summarizes Canadian phosphate imports by origin.

Table 19-7: Canada Phosphate Imports by Origin

Canada Phosphate Imports by Origin

	Fertilizer Year Ending June 30											
1000 Tonnes	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
MAP	**636.6**	**822.8**	**803.5**	**850.6**	**758.9**	**761.8**	**881.8**	**805.8**	**1058.4**	**1168.2**	**1613.9**	**1687.2**
United States	634.0	760.6	796.1	814.7	719.7	715.2	848.6	742.9	989.5	1009.4	1418.4	1356.6
Florida	*371.2*	*454.9*	*576.8*	*454.5*	*324.0*	*370.7*	*391.9*	*301.1*	*325.1*	*386.6*	*548.7*	*569.2*
Idaho	*126.8*	*163.0*	*94.1*	*130.2*	*143.4*	*126.6*	*89.8*	*163.6*	*185.1*	*201.8*	*232.3*	*259.4*
Minnesota	*18.2*	*19.9*	*36.0*	*43.4*	*51.1*	*42.7*	*80.6*	*69.7*	*94.5*	*64.9*	*151.8*	*124.3*
North Carolina	*90.6*	*84.0*	*60.7*	*136.6*	*126.4*	*136.7*	*186.1*	*151.1*	*334.8*	*300.4*	*398.8*	*366.2*
Other	*27.2*	*38.8*	*28.5*	*50.1*	*74.7*	*38.4*	*100.3*	*57.4*	*50.0*	*55.8*	*86.8*	*37.6*
Morocco	0.0	0.0	0.0	0.0	1.1	2.2	19.4	33.4	65.0	116.0	122.5	243.3
Russia/Lithuania	0.0	59.8	4.6	30.3	30.1	29.0	0.1	27.5	0.0	39.9	63.5	72.5
Other	2.6	2.5	2.8	5.6	8.0	15.4	13.6	2.0	3.9	2.9	9.6	14.8
NPS	**4.6**	**3.6**	**3.5**	**149.5**	**380.6**	**491.0**	**402.8**	**474.2**	**549.9**	**552.3**	**523.5**	**607.0**
United States	4.1	2.5	2.7	148.6	379.9	374.4	389.5	465.6	538.7	541.7	515.2	603.7
Florida	*0.0*	*0.0*	*0.0*	*129.0*	*333.5*	*335.9*	*333.3*	*418.2*	*483.8*	*496.5*	*463.0*	*530.5*
Idaho	*0.1*	*0.1*	*0.1*	*6.9*	*13.0*	*31.2*	*35.9*	*32.6*	*36.8*	*36.1*	*25.8*	*22.3*
Minnesota	*0.1*	*0.0*	*0.1*	*1.1*	*3.4*	*1.7*	*15.1*	*10.1*	*11.9*	*6.2*	*17.9*	*41.6*
Other	*3.9*	*2.4*	*2.5*	*11.6*	*30.0*	*5.7*	*5.1*	*4.7*	*6.1*	*2.9*	*8.5*	*9.3*
Morocco	0.0	0.0	0.0	0.2	0.0	0.8	2.1	7.8	10.4	10.2	7.0	0.4
Russia/Lithuania	0.0	0.0	0.0	0.0	0.0	115.0	9.6	0.0	0.0	0.0	0.0	0.0
Other	0.5	1.1	0.8	0.7	0.7	0.7	1.6	0.8	0.8	0.5	1.3	2.9
DAP	**31.5**	**96.6**	**90.9**	**121.5**	**98.6**	**103.7**	**95.2**	**84.6**	**89.0**	**72.6**	**95.2**	**135.8**
United States	16.8	20.3	79.8	108.9	89.6	102.9	94.6	84.1	88.3	65.8	85.2	43.4
Florida	*14.6*	*19.3*	*78.2*	*106.7*	*76.3*	*98.1*	*89.3*	*82.6*	*88.0*	*65.6*	*84.1*	*23.6*
Other	*2.2*	*1.0*	*1.5*	*2.1*	*13.3*	*4.9*	*5.2*	*1.5*	*0.3*	*0.2*	*1.1*	*19.8*
Morocco	0.0	0.0	0.0	7.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	51.2
Russia/Lithuania	13.2	75.1	10.6	4.3	8.3	0.0	0.1	0.0	0.1	6.0	8.7	39.8
Other	1.5	1.2	0.5	0.6	0.7	0.8	0.6	0.5	0.6	0.9	1.4	1.3
Total	**672.7**	**923.0**	**897.8**	**1121.5**	**1238.2**	**1356.5**	**1379.8**	**1364.5**	**1697.3**	**1793.1**	**2232.6**	**2429.9**
United States	654.9	783.4	878.6	1072.1	1189.2	1192.6	1332.7	1292.5	1616.4	1616.9	2018.7	2003.6
Florida	*385.8*	*474.2*	*655.1*	*690.3*	*733.9*	*804.6*	*814.5*	*801.9*	*896.8*	*948.7*	*1095.8*	*1123.3*
Idaho	*126.9*	*163.0*	*94.3*	*137.1*	*156.4*	*157.7*	*125.7*	*196.2*	*221.9*	*237.9*	*258.1*	*281.7*
North Carolina	*90.6*	*84.0*	*60.7*	*136.6*	*126.4*	*136.7*	*186.1*	*151.1*	*334.8*	*300.4*	*398.8*	*366.2*
Other	*51.6*	*62.1*	*68.5*	*108.2*	*172.5*	*93.5*	*206.4*	*143.3*	*162.9*	*130.0*	*266.0*	*232.5*
Morocco	0.0	0.0	0.0	7.9	1.1	3.0	21.6	41.2	75.4	126.2	129.5	294.9
Russia/Lithuania	13.2	134.9	15.1	34.7	38.4	144.1	9.7	27.5	0.2	45.9	72.2	112.4
Other	4.6	4.7	4.1	6.8	9.5	16.9	15.8	3.3	5.3	4.2	12.2	19.0

Source: Stats Canada

Note: MAP includes HS Code 310540, NPS includes HS Code 310559 and DAP includes HS Code 310530

19.3.2.8 *Canadian MAP Shipments by Province*

Stats Canada publishes fertilizer shipment statistics by province based on surveys of producers and distributors. In the case of MAP, the Prairie Provinces accounted for 86% of shipments (Figure 19-18) during the last three years. Saskatchewan is the largest MAP consuming province with shipments averaging more than 585,000 t per year during the last three years. Stats Canada import statistics by province of entry for the last two fertilizer years (or since the closure of the Redwater facility) show that the Prairie Provinces accounted for 85% of imports so the statistics look consistent. MAP usage of more than 200,000 t in the Big East Provinces is noteworthy given this region is in the backyard of the Martison facility.

The NPS shipment statistics are not broken out separately but are included in another category, and much of DAP shipment information is suppressed due to disclosure rules. As a result, only MAP shipment statistics by province are available. Nevertheless, import statistics by province of entry appears to provide a reliable estimate for usage by province.

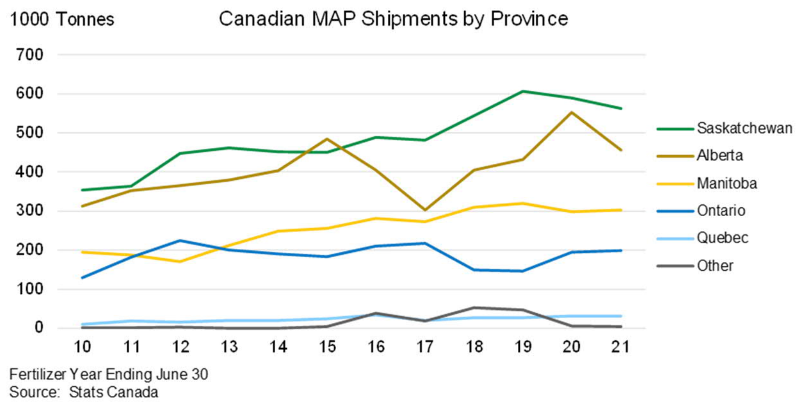


Figure 19-18: Canadian MAP Shipments by Province (2010-2021)

Table 19-8: Canadian Phosphate Shipments by Province (2010-2021)

Canada Phosphate Shipments by Province

Fertilizer Year Ending June 30

1000 Tonnes	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2019-20 Average	Pct of Total
MAP	1,002.0	1,106.0	1,227.0	1,263.0	1,306.0	1,402.0	1,457.0	1,312.0	1,488.0	1,578.0	1,671.0	1,557.0	1,602.0	100%
Quebec	*10.0*	*19.0*	*16.0*	*20.0*	*20.0*	*24.0*	*34.0*	*20.0*	*26.7*	*26.7*	*31.0*	*31.0*	29.6	2%
Ontario	*130.0*	*182.0*	*224.0*	*200.0*	*190.0*	*184.0*	*210.0*	*218.0*	*149.0*	*146.0*	*194.0*	*199.0*	179.7	11%
Big East	*140.0*	*201.0*	*240.0*	*220.0*	*210.0*	*208.0*	*244.0*	*238.0*	*175.7*	*172.7*	*225.0*	*230.0*	209.2	13%
Manitoba	*194.0*	*188.0*	*171.0*	*212.0*	*249.0*	*256.0*	*281.0*	*273.0*	*310.0*	*319.0*	*298.0*	*303.0*	306.7	19%
Saskatchewan	*354.0*	*363.0*	*448.0*	*462.0*	*452.0*	*450.0*	*488.0*	*481.0*	*544.0*	*607.0*	*590.0*	*563.0*	586.7	37%
Alberta	*312.0*	*353.0*	*365.0*	*379.0*	*403.0*	*484.0*	*405.0*	*302.0*	*405.0*	*432.0*	*552.0*	*456.0*	480.0	30%
Prairies	*860.0*	*904.0*	*984.0*	*1,053.0*	*1,104.0*	*1,190.0*	*1,174.0*	*1,056.0*	*1,259.0*	*1,358.0*	*1,440.0*	*1,322.0*	1,373.3	86%
Other	*2.0*	*1.0*	*3.0*	*0.0*	*0.0*	*4.0*	*39.0*	*18.0*	*53.3*	*47.3*	*6.0*	*5.0*	19.4	1%

19.3.2.9 *The Contraction and Consolidation of the US Phosphate Industry*

The US industry has contracted and consolidated during the last 30 years due to the depletion of phosphate rock reserves as well as other factors such as the rapid development of a large Chinese phosphate industry in the 2000s.

The United States was the dominant exporter of phosphate (DAP) to China prior to the building of its domestic industry so this development accelerated changes in the US phosphate industry.

Today, four companies, The Mosaic Company, Nutrien, Simplot and Itafos, operate nine phosphate facilities in the United States. That is down from 17 companies operating 22 facilities in 1990. This contraction and consolidation of companies is illustrated in Figure 19-19 below.

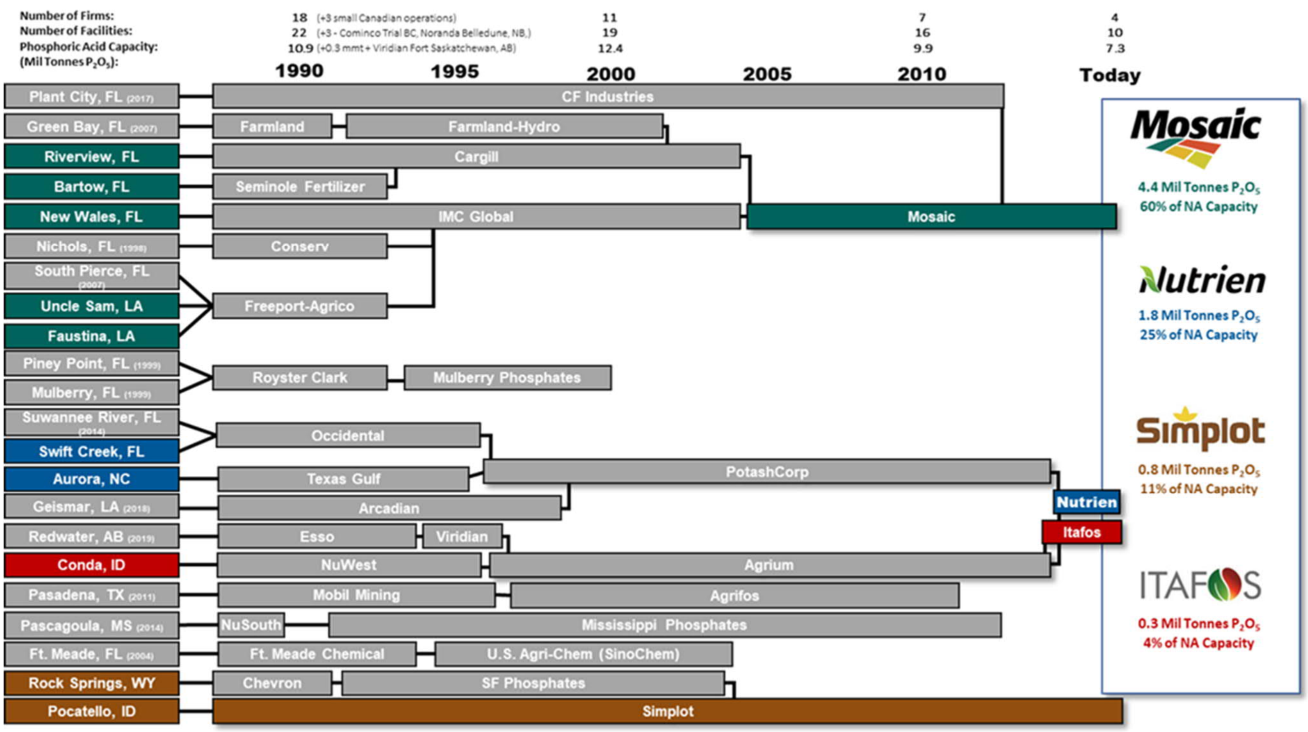


Figure 19-19: Change In Phosphate Producing Companies Since 1990

19.3.2.10 US Phosphate Production and Mine/Plant Closures 1990-2020

Statistics from the International Fertilizer Association (IFA) over the past three decades show that US phosphate production peaked during the last half of the 1990s and has trended steadily downward since then (Figure 19-20). Annual phosphate rock output averaged 43.5 mt per year from 1995 to 1999 compared to 23.7 mt in 2020, a drop of 45% or 19.8 mt. US phosphate companies closed or idled 11 mines with combined annual capacity of 28.3 mt during this period (Table 19-9).

Phosphoric acid statistics, the most comprehensive measure of phosphate output, also show that the US industry reached its pinnacle during the five year run from 1995 to 1999 (Figure 19-21) when annual production averaged 11.3 mt P_2O_5. Production dropped 45% or 5.1 mt to 6.3 mt by 2020. Companies closed 13 chemical plants with combined annual capacity of 5.7 mt P_2O_5 during this period (Table 19-10).

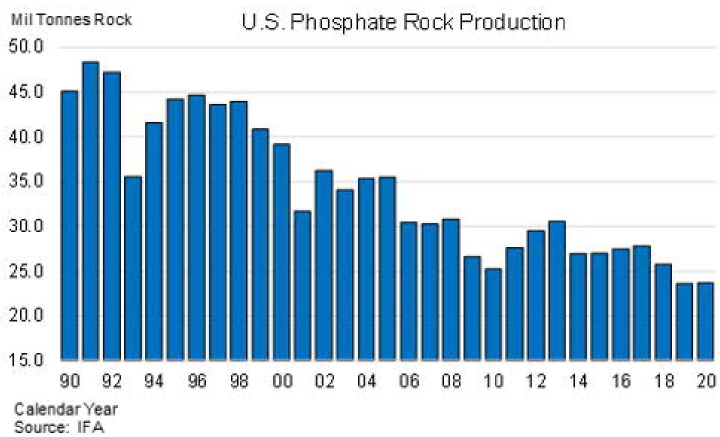


Figure 19-20: US Phosphate Rock Production

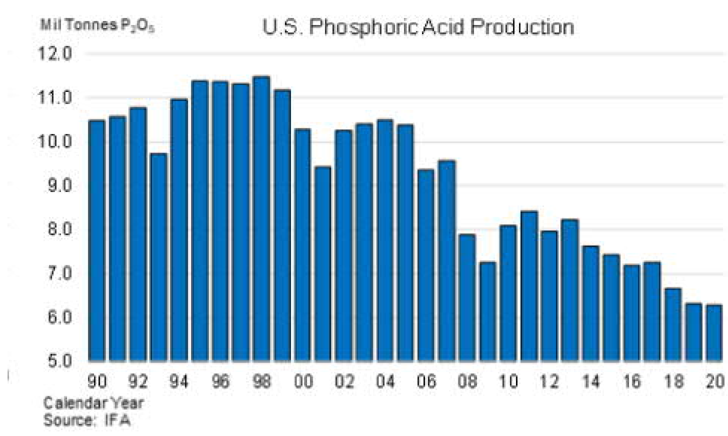


Figure 19-21: US Phosphoric Acid Production

Table 19-9: US Phosphoric Rock Mine Closures

U.S. Phosphate Rock Mine Closures
Unit: 1000 Tonnes Rock

Company	Mine	Capacity	Idled/Closed
IMC-Agrico	Haynesworth	2,270	1995
IMC-Agrico	Lonesome, FL	2,720	1995
Cargill	Ft Meade, FL	2,720	1996
IMC-Agrico	Noralyn-Phosphoria, FL	2,720	1999
IMC-Agrico	Payne Creek, FL	2,720	1999
Agrifos	Nichols, FL	1,400	2000
Mosaic	Kingsford, FL	3,200	2005
Mosaic	Ft Green, FL	4,900	2006
Mosaic	Hopewell, FL	500	2011
Mosaic	Hookers Prairie, FL	2,000	2014
Mosaic	South Pasture, FL (Idled)	3,170	2018
Total		28,320	

Source: USGS. IFDC and CRU

Table 19-10: US Phosphoric Acid Plant Closures

U.S. Phosphoric Acid Plant Closures
Unit: 1000 Tonnes P_2O_5

Company	Mine	Capacity	Idled/Closed
CF Industries	Bonnie, FL	450	1990
IMC-Agrico	Nichols, FL	235	1998
Mulberry Phosphates	Mulberry, FL	300	1999
Mulberry Phosphates	Pinney Point, FL	170	1999
IMC-Agrico	Faustina, LA	590	2000
USAC	Ft Meade, FL	515	2004
Mosaic	Green Bay, FL	640	2007
Mosaic	South Pierce, FL	540	2007
Afrifos	Pasadena, TX	340	2011
PotashCorp	Suwannee River, FL	420	2014
MissPhos	Pascacoula, MS	385	2014
Mosaic	Plant City, FL	950	2017
Nutrien	Geismar, LA	200	2018
Total		5,735	

Source: IFA, IFDC and CRU

19.3.2.11 Possible Closures

US phosphate production is predicted to stabilize at current levels over the next several years. However, another round of plant closures will begin later this decade due to mine closures and other developments such as reaching phosphogypsum storage limits. This section highlights one potential development that is expected to continue the 30 year trend.

Nutrien has indicated in its 2020 annual report that its White Springs, FL mine will exhaust proven and probable reserves by 2029 (refer to the impairment disclosure in Table 19-13 below). Annual capacity of the mine is 2.0 mt and average annual production during the last five years has been 1.69 mt.

It is unlikely that the company will import phosphate rock or phosphoric acid in order to continue to run the chemical plants. As a result, the chemical complex is also expected to close when the mine closes. Annual phosphoric acid capacity is 500,000 t P_2O_5 and average annual output during the last five years was 462,000 t. The company reported that the facility produced 400,000 t of MAP and MAP MST (using an estimated 193,000 t P_2O_5 of acid) in 2021. This translates into SPA production of 277,000 t of 100% P_2O_5 solution or more than 400,000 t of 68% P_2O_5 solution in 2021 (Nutrien reported 440,000 t of liquid product production at the site last year. The scale and product mix of this complex is similar to that of the Project, and the White Springs facility is projected to close at about the same time Martison could possibly be ramping up.

Recent production results for White Springs for phosphate rock and phosphoric acid are shown in Table 19-11 and Table 19-12 respectively.

Table 19-11: Nutrien White Springs Phosphate Rock Production (2017-2021)

Nutrien White Springs, FL Operations

Million Tonnes	2017	2018	2019	2020	2021
Phosphate Rock (29% P_2O_5)					
Reserves on January 1	23.80	22.25	20.40	18.79	16.98
Capacity	2.00	2.00	2.00	2.00	2.00
Operating Rate	78%	93%	81%	91%	81%
Production	1.55	1.85	1.61	1.81	1.62
For Phosphoric Acid	1.56	1.74	1.82	1.71	1.74
Other/Inventory Change	-0.01	0.11	-0.21	0.10	-0.12

Note: PotashCorp reported reserves of 23.8 million tonnes as of January 1, 2017.

Source: PotashCorp and Nutrien Annual Reports

Table 19-12: Nutrien White Springs Phosphoric Acid Production (2017-2021)

Nutrien White Springs, FL Operations

1000 Tonnes	2017	2018	2019	2020	2021
Phosphoric Acid (P_2O_5)					
Capacity	500	500	500	500	500
Operating Rate	84%	94%	98%	92%	94%
Production	420	470	490	460	470
for SPA	*420*	*387*	*381*	*314*	*277*
for MAP/MAP MST	*0.06*	*83*	*110*	*146*	*193*

Source: PotashCorp and Nutrien Annual Reports

Table 19-13: White Springs Impairment Disclosure Statement

Impairment

In 2020, we recorded the following impairments:

Cash-generating units ("CGUs")	Aurora	White Springs
Segment	Phosphate	
Impairment indicator	Lower long-term forecasted global phosphate prices	
Pre-tax impairment loss ($)	545	215
Recoverable amount ($)	995 (post-tax)	160 (pre-tax)
Valuation technique	FVLCD a Level 3 measurement	Value in use ("VIU")
Key assumptions		
End of mine life (proven and probable reserves) (year)	2050	2029
Post-tax discount rate (%)	10.5	12.0 (pre-tax – 16.0)

19.3.2.12 Recent US Supply and Demand Trends

This section takes a closer look at US supply and demand trends during the last dozen years. The analysis uses reliable production and inventory data from The Fertilizer Institute (TFI) and trade statistics from the US Department of Commerce to put together annual supply and demand balance sheets for solid high analysis products (DAP/MAP/NPS/TSP). Table 19-14 below shows the balance sheet for the four products combined for the fertilizer year that runs from July 1 to June 30.

US Solid High Analysis Product Shipments

US shipments are calculated in the balance sheet as beginning producer inventories plus production and imports and less exports and ending producer inventories. This is the quantity that moves from production facilities and import ports into the vast US distribution channel and is summarized in Table 19-14. Shipments can differ significantly from the amount of product that ends up on farm fields, yet this statistic clearly tracks and identifies the demand trend over time. The swings in channel inventories often are the key driver of near term price movements.

Table 19-14: US DAP+MAP+NPS+TSP Balance Sheet (2010-2021)

U.S. DAP+MAP+NPS+TSP

	Fertilizer Year Ending June 30											
1000 Tonnes	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Producer Beginning On-Site Stocks	419	453	352	303	363	286	367	451	423	431	404	412
Production	12,470	12,749	12,632	12,506	11,230	10,964	10,894	11,199	11,035	10,531	10,715	10,252
Pct Change Year Before	*18%*	*2%*	*-1%*	*-1%*	*-10%*	*-2%*	*-1%*	*3%*	*-1%*	*-5%*	*2%*	*-4%*
Imports	273	1,317	627	1,099	882	2,139	1,790	2,079	2,788	3,651	2,962	2,742
Pct Change Year Before	*26%*	*383%*	*-52%*	*75%*	*-20%*	*143%*	*-16%*	*16%*	*34%*	*31%*	*-19%*	*-7%*
Domestic Deliveries	5,690	7,275	6,066	7,430	6,186	7,022	7,415	7,012	8,654	8,948	8,249	8,378
Pct Change Year Before	*78%*	*28%*	*-17%*	*22%*	*-17%*	*14%*	*6%*	*-5%*	*23%*	*3%*	*-8%*	*2%*
Exports	7,019	6,892	7,242	6,115	6,003	6,001	5,186	6,293	5,161	5,262	5,420	4,655
Pct Change Year Before	*-6%*	*-2%*	*5%*	*-16%*	*-2%*	*0%*	*-14%*	*21%*	*-18%*	*2%*	*3%*	*-14%*
Producer Ending On-Site Stocks	453	352	303	363	286	367	451	423	431	404	412	373
Pct Change Year Before	*8%*	*-22%*	*-14%*	*20%*	*-21%*	*28%*	*23%*	*-6%*	*2%*	*-6%*	*2%*	*-9%*
Producer Ending Off-Site Stocks	570	370	452	671	586	574	840	662	392	665	404	278
Pct Change Year Before	*13%*	*-35%*	*22%*	*49%*	*-13%*	*-2%*	*46%*	*-21%*	*-41%*	*70%*	*-39%*	*-31%*
Producer Ending Total Stocks	973	672	679	960	797	866	1,191	985	698	943	690	521
Pct Change Year Before	*9%*	*-31%*	*1%*	*41%*	*-17%*	*9%*	*37%*	*-17%*	*-29%*	*35%*	*-27%*	*-25%*

Figure 19-22 shows that US shipments of solid high analysis products have increased from 6.0 mt in 2009/10 to more than 8.6 mt in 2020/21. Canadian MAP/NPS/DAP shipments totaled 2.4 mt in 2020/21, so the North American market is a 11.0 mt market today. That is up from less than 9.0 mt at the beginning of the last decade. A common perception is that North America is a mature market with little to no to even negative growth potential, but the numbers tell a different story and show that shipments have increased significantly on both sides of the US-Canadian border during the last 12 years.

The increase in US shipments is the result of at least three factors. First, US phosphate demand has increased during this period due to gains in corn and soybean acreage as well as steady yield increases that remove more phosphorus from farm fields and require larger replacement applications. Second, solid high analysis products have captured share from other products, particularly liquid products. Finally, there is a little tonnage creep due to the growth of NPS products that contain less P_2O_5 than MAP or DAP.

Figure 19-22 shows the breakdown of shipments by product. MAP is the most widely used phosphate product in the United States. Shipments totaled 3.3 mt in 2020/21. MAP accounted for 40% of total product shipments and 44% of the total phosphate shipped in these products (due to its 52% P_2O_5 content). MAP shipments have trended upward driven by factors noted earlier through increases in soybean area and acreage increases in western states that have alkaline soils. MAP shipments peaked at 4.29 mt in 2018/19 but have dropped significantly due to US countervailing duties on phosphate imports from Morocco and Russia. Both countries are large MAP producers and accounted for the bulk of US MAP imports. US importers have not sourced enough MAP from other origins to replace Moroccan and Russian tonnage, so distributors and farmers have had to use more of the other three products.

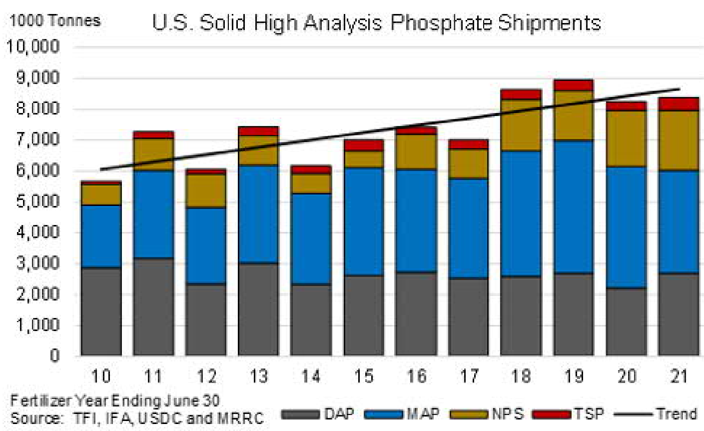


Figure 19-22: US Shipments Of Solid High Analysis Phosphate

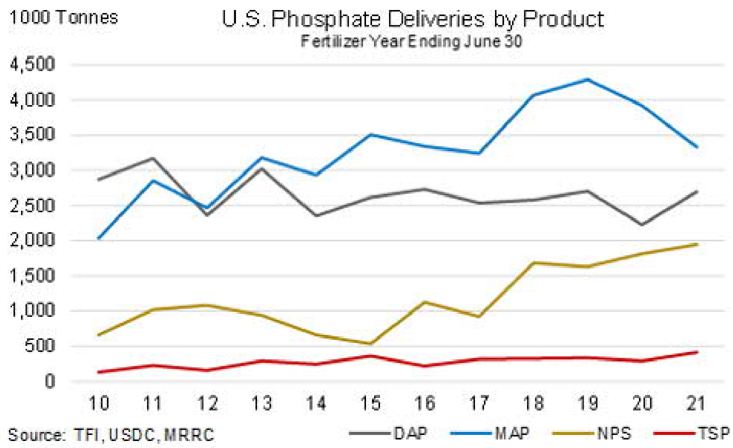


Figure 19-23: US Phosphate Deliveries By Product

Figure 19-23 identifies that DAP is the second most widely used product. Shipments totaled 2.70 mt in 2020/21 or 32% of total product shipments and 32% of the phosphate shipped in these products. DAP shipments have remained steady at 2.65 mt per year during this period.

NPS shipments have increased rapidly from an average of just more than 800,000 t during the first half of the last decade to 1.95 mt in 2020/21. NPS products accounted for 23% of total product shipments and 19% of the phosphate shipped in these products. NPS shipments will eclipse DAP shipments within the next few years.

Each US producer markets a line of branded NPS products. Mosaic, however, was the first mover and is the clear market leader with its Micro Essentials brand. The rapid adoption of NPS products is a testament to the significant value they create throughout the entire supply chain.

NPS products deliver value to farmers by correcting sulfur and micronutrient deficiencies, providing uniform distribution of nutrients, and enhancing phosphate uptake in plants. NPS products also add to the bottom line of distributors. Producers often market NPS products through select channels and work with key distributors to develop and support marketing programs for these higher margin products.

TSP (Triple Superphosphate) shipments are small relative to the three other solid high analysis products, but volume is trending upward. US shipments totaled 410,000 t in 2020/21 or just 5% of total product shipments and 6% of the phosphate shipped in these products which is more than double the average volume during the first three years of the last decade. No TSP is produced in the United States, so all shipments are from imports. TSP shipments are trending up as a result of the increase in soybean area, fallout from US CVD on phosphate imports from Morocco and Russia, restrictions on fall nitrogen use in some regions, and effective marketing programs by a few key importers/distributors.

19.3.2.13 *US Solid High Analysis Product Production*

US phosphate production has continued to trend downward since 2009/10. Phosphoric acid production, which is the best comprehensive gauge of output, declined 23% or 1.8 mt P_2O_5 from 7.9 mt in 2009/10 to 6.1 mt in 2020/21 (Figure 19-24). Five facilities with combined capacity of 2.3 mt P_2O_5 capacity were permanently closed during this period.

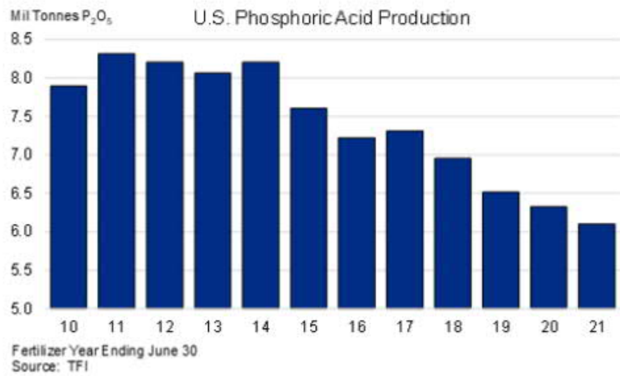


Figure 19-24: US Phosphoric Acid Production (2010-2021)

TFI collects DAP and MAP production statistics from all US producers, although the association has published only the combined output in P_2O_5 tonnes since 2013/14 due to disclosure issues. These statistics show that DAP+MAP production has declined 42% or 2.37 mt P_2O_5 during this period (Figure 19-25). The larger drop in DAP+MAP production coupled with declines in output of other downstream products such as MGA exports, SPA and feed phosphate implies that a larger share of acid is getting upgraded to NPS products.

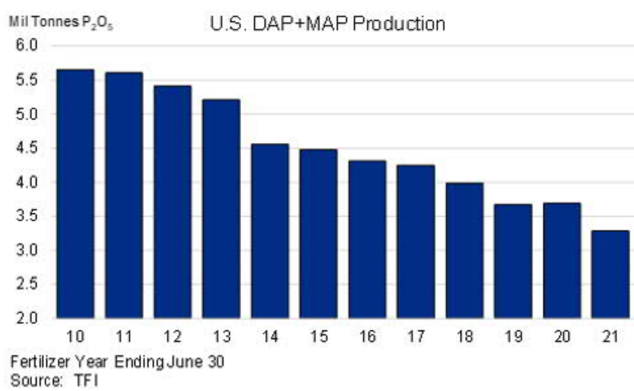


Figure 19-25: US DAP+MAP Production (2010-2021)

MRRC has estimated production by product based on IFA statistics and company reports. These estimates indicate that US DAP+MAP+NPS production has declined 18% or 2.2 mt from 12.5 mt in 2009/10 to 10.3 mt in 2020/21 (Figure 19-26 and Figure 19-27).

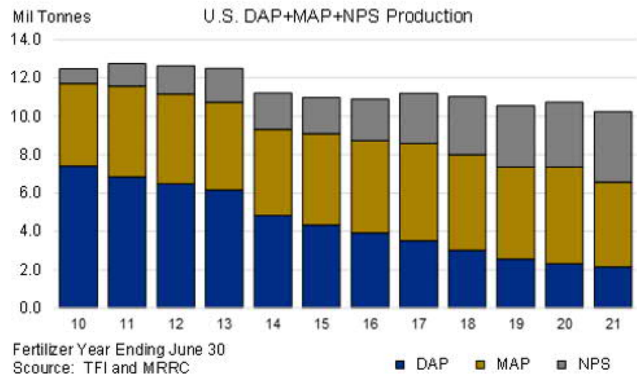


Figure 19-26: US DAP+MAP+NPS Production (2010-2021)

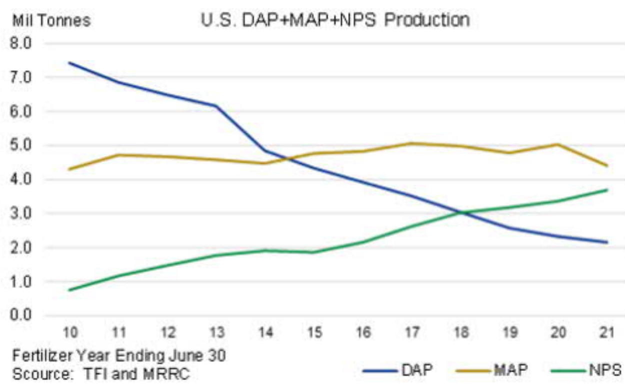


Figure 19-27: US DAP+MAP+NPS Production Trends (2010-2021)

Figure 19-27 clearly identifies how DAP production has plummeted 71% or 5.3 mt from 7.4 mt in 2009/10 to 2.1 mt in 2020/21. One granulation plant, the large Mosaic Bartow #4 DAP only plant, now accounts for more than 70% of US DAP production. The drop in US DAP output coincides with the fall in exports to large DAP-using countries in Asia (primarily India).

US MAP production has remained steady during this period. Output increased just 2% or 110,000 t from 4.3 mt in 2009/10 to 4.4 mt in 2020/21. MAP demand throughout the Americas has increased significantly during this period, so the US industry has lost share to other producers such as those in Morocco, Russia and Saudi Arabia.

No official NPS statistics are available, but some companies have published sales volumes and touted the success of these branded products. Table 19-15 shows MRRC NPS production estimates by firm based on this information.

NPS was the big winner during this period. By this count, US NPS production surged 280% or 2.8 mt from 1.0 mt in 2010 to 3.8 mt in 2020/21. Most of US NPS goes down on farmer's fields in the Americas with the Canada, United States, and Brazil accounting for the majority of the use.

Mosaic still is the dominant US NPS producer (Table 19-15). By this count, Mosaic accounts for almost 90% of US NPS output however other companies, both in the US and abroad, have developed similar NPS products.

US NPS Production Estimates

Table 19-15: Primary US NPS Producers

	Calendar Year											
1000 Tonnes	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Itafos (MAP+)	0	0	0	0	0	0	0	0	0	9	20	70
Mosaic (MicroEssentials)	1,000	1,350	1,600	1,950	1,850	1,800	2,350	2,650	3,050	2,850	3,150	3,325
Nutrien (MAP MST)	0	0	0	0	0	0	0	0	50	75	100	150
Simplot (40 Rock)	0	0	0	0	25	50	100	150	200	250	275	275
Total	1,000	1,350	1,600	1,950	1,875	1,850	2,450	2,800	3,300	3,184	3,545	3,820

Source: Itafos annual reports, Mosaic performance data and investor day reports, and MRRC estimates for Nutrien and Simplot

19.3.2.14 *Solid High Analysis Imports*

Diverging US demand and production trends have opened a big gap that producers and importers have filled by reducing exports and increasing imports, respectively. US imports of DAP+MAP+NPS+TSP increased more than 270% or 2.00 mt from a three year average of 740,000 t at the beginning of the last decade to 2.74 mt in 2020/21 however, US imports are off 900,000 t from a peak of 3.65 mt in 2018/19.

DAP has accounted for one half of the increase during this period. US DAP imports jumped nearly 550% or 1.00 mt from the three year average of just 180,000 t at the beginning of the last decade to 1.18 mt in 2020/21 (Figure 19-28).

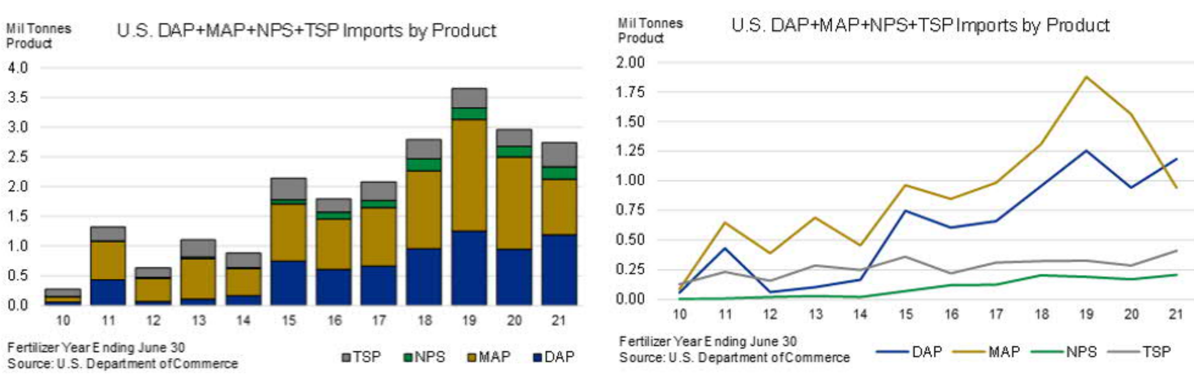


Figure 19-28: US DAP+MAP+NPS+TSP Cumulative and Trend Imports by Product

MAP imports increased more than 150% or 560,000 t from the three year average of 380,000 t to 940,000 t in 2020/21. However, MAP imports were off 933,000 t from the peak in 2018/19 due mostly to high CVD on the large Moroccan and Russian MAP producers and exporters. TSP imports also have trended up during this period. Imports climbed 140% from the three year average of 170,000 t to 410,000 t in 2020/21. US NPS imports also have trended up but were less than 200,000 t in 2020/21 (Figure 19-29).

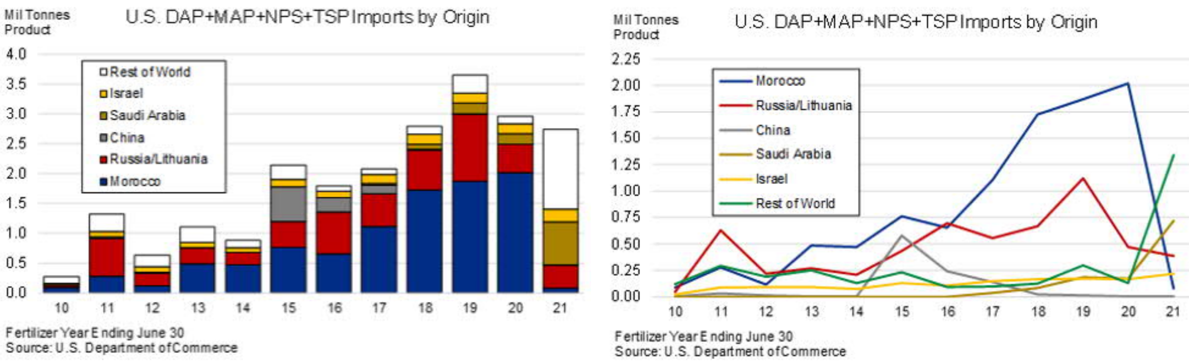
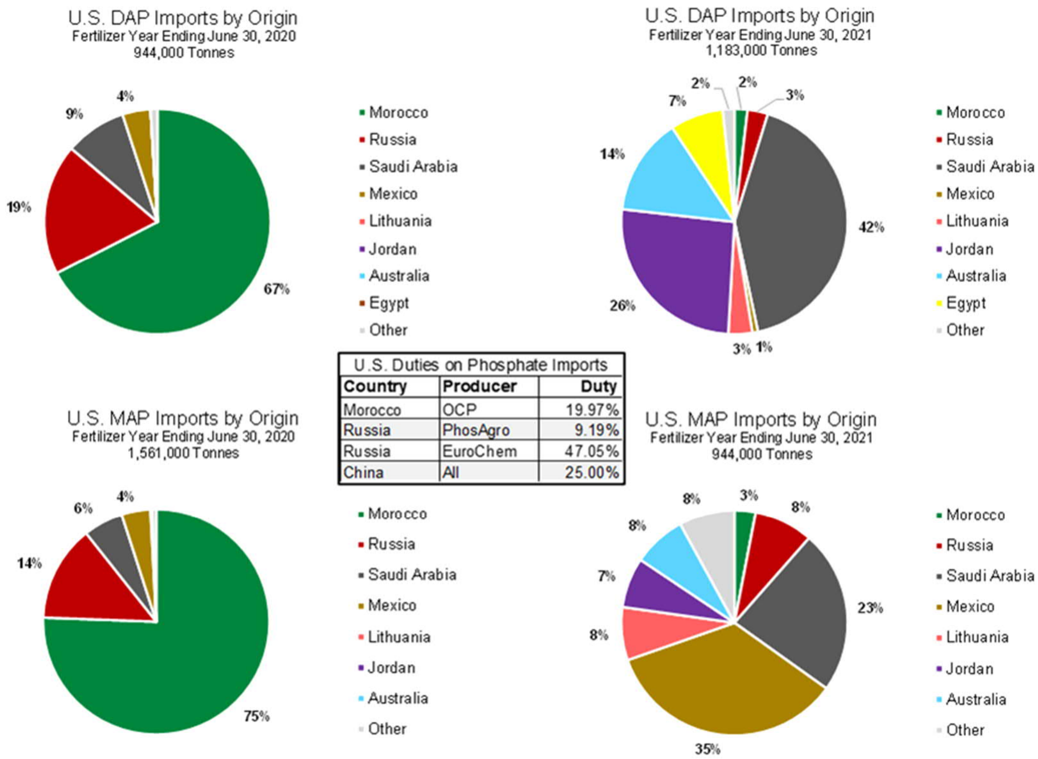


Figure 19-29: US DAP+MAP+NPS+TSP Cumulative and Trend Imports by Origin

US CVDs on phosphate imports from Morocco and Russia have scrambled trade flows. Figure 19-30 highlights the significant changes in product mix and origins. For example, Morocco and Russia combined accounted for 76% and 89% of US DAP and MAP imports, respectively, in 2019/20. Imports from the two largest origins ceased after Mosaic filed the CVD petition on 26 June 2020.

US DAP imports increased between 2019/20 and 2020/21. The pie charts show that increased imports from Saudi Arabia, Jordan, Australia and Egypt more than made up the lost tonnage from Morocco and Russia. In the case of MAP, however, imports declined more than 600,000 t in 2020/21. Gains from Mexico, Saudi Arabia, Lithuania, Jordan and Australia were not sufficient to replace the declines from Morocco and Russia.

Imports from China still are subject to the 25% tariff levied by the Trump administration. So, the first, second and fifth largest phosphate exporters are subject to high US tariffs.



Country	Producer	Duty
Morocco	OCP	19.97%
Russia	PhosAgro	9.19%
Russia	EuroChem	47.05%
China	All	25.00%

Figure 19-30: Changes In US DAP & MAP Imports 2020 To 2021

19.3.2.15 US High Analysis Exports

The decline in US production not surprisingly has also resulted in a decline in US exports. DAP+MAP+NPS exports declined 34% or 2.36 mt from 7.02 mt in 2009/10 to 4.66 mt in 2020/21 (Figure 19-31).

DAP exports to the large Asian market plummeted during this period. US DAP exports plunged 86% or 3.98 mt from 4.62 mt in 2009/10 to just 640,000 t in 2020/21.

MAP exports declined moderately from 2.32 mt in 2009/10 to 2.07 mt in 2020/21, a drop of 11% or 251,000 t. However, US exports dropped 600,000 t in 2020/21 as more production stayed home to meet domestic demand following the filing of the US CVD petition against phosphate imports from Morocco and Russia and the drop in MAP imports from these two dominant suppliers.

NPS was the big winner during this period. Exports climbed from less than 80,000 t in 2009/10 to 1.95 mt in 2020/21, an increase of more than 2400% or 1.87 mt. Mosaic is the dominant producer, and most exports were to Brazil and Canada where it has developed strong NPS demand.

The geographic shift in US exports is nothing short of remarkable. Exports to Asia plummeted 91% or 3.56 mt from 3.93 mt in 2009/10 to just 370,000 t in 2020/21. Asia accounted for just 8% of US exports in 2020/21, down from 56% in 2009/10. China and Saudi Arabia have captured most of Asian demand.

US exports to the Americas increased 50% or 1.42 mt during the same period and now account for 92% of total exports, up from 41% in 2009/10 (Figure 19-32).

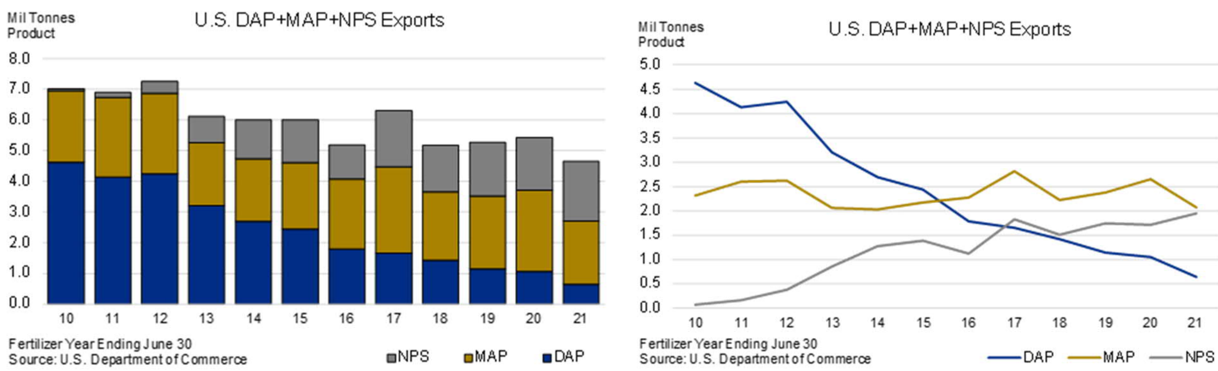


Figure 19-31: US DAP+MAP+NPS Cumulative and Exports Trends (2010-2021)

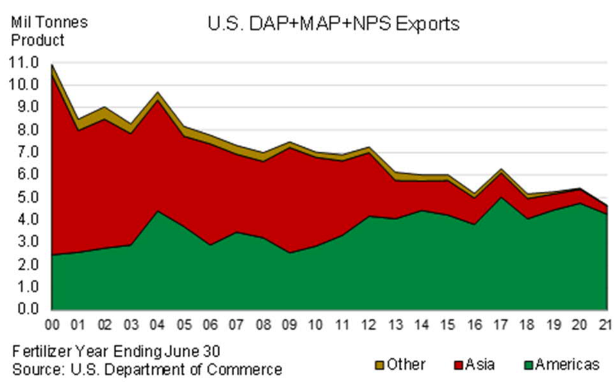


Figure 19-32: US DAP+MAP+NPS Export Destinations

19.3.2.16 The North American SPA Market

Super phosphoric acid (SPA) is a gel-like liquid product with a high phosphate content of 68%-72% P_2O_5. It is produced by concentrating and clarifying phosphoric acid through a series of evaporators and clarifiers. SPA is a small and almost uniquely US product.

Nearly all US SPA is used to produce ammonium polyphosphate (APP) fertilizer. APP is a liquid product that is formed by combining SPA, ammonia and water in a cross pipe reactor. These liquid versions of DAP/MAP have analyses of 10-34-0 or 11-37-0. 10-34-0 is the dominant product. APP is often referred to simply by this analysis and is mostly used as a starter fertilizer on corn.

SPA trade is immaterial. US SPA exports have totaled 7,000 to 10,000 t P_2O_5 per year since 2018, with Mexico and Canada accounting for roughly 75% and 25% of the total, respectively.

In the US, only three companies, Nutrien, Simplot and Itafos, produce SPA at five facilities. Nutrien was forced to divest the Conda facility when the company was formed in 2018 because the combination of Agrium (Conda, ID) and PotashCorp (Aurora, NC and White Springs, FL) reduced the number of US producers from three to two and set off alarms at the US Department of Justice. The Conda facility was sold to Itafos.

Table 19-16 lists capacity and production estimates by company based on publicly available reports, CRU statistics, industry information, and MMRC estimates for the last three calendar years for both SPA and APP.

Nutrien's White Springs, FL plant accounts for more than 30% of US SPA production. The facility is expected to close at the end of this decade because of an exhausted rock supply.

Eroding US phosphate rock quality also creates challenges for making high quality SPA.

Table 19-16: SPA Production By US Company

SPA for Sale and for On-Site APP Production

1000 Tonnes P_2O_5	2019	2020	2021
Nutrien			
SPA Production	446	406	429
for Sale	300	236	233
for On-Site APP	146	170	196
Simplot			
SPA Production	230	230	230
for Sale	193	193	193
for On-Site APP	38	38	38
Itafos			
SPA Production	146	138	140
for Sale	131	124	127
for On-Site APP	14	13	13
Total			
SPA Production	822	774	799
for Sale	623	553	552
Share of Production	*76%*	*71%*	*69%*
for On-Site APP	198	221	247
Share of Production	*24%*	*29%*	*31%*
APP Production Estimate	***2,197***	***2,056***	***2,131***
(1000 Tonnes Product)			

Source: Company reports, CRU, TFI/AAFPCO and MRRC

19.4 Current Situation and Near Term Outlook

19.4.1 The Current Price Increase

After bottoming in December 2019 at the lowest levels since 2006, phosphate prices have surged to the highest levels since 2008. Prices have climbed 300% or $750 per tonne at three of the most liquid and transparent price discovery points, and the high correlation between these prices demonstrates the global dimension of the phosphate market. Figure 19-33 shows phosphate prices, recorded weekly, from January 2005 through March 2022, and Table 19-17 lists the Pearson correlation coefficients for the 898 weekly prices.

In the US, the price of DAP fob New Orleans (NOLA) barge increased from $262 per tonne ($238 per ton) in December 2019 to $1036 per tonne ($940 per ton) in March 2022, a 295% or $774 per tonne ($702 per ton) increase.

In Brazil, the price of MAP cfr port increased from $281 per tonne in December 2019 to $1,176 per tonne in March 2022, a 319%, or $895 per tonne, increase.

In India, the price of DAP cfr port increased from $301 per tonne in December 2019 to $974 per tonne in March 2022, a 224%, or $673 per tonne, increase.

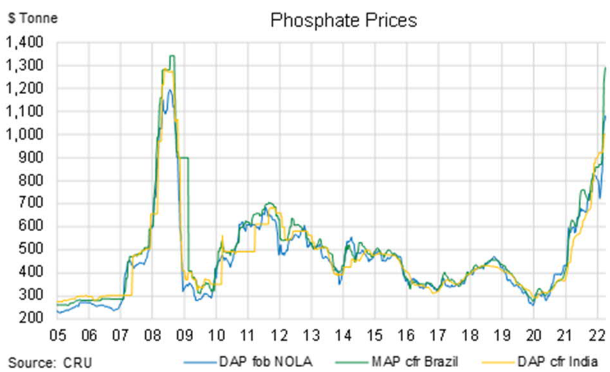


Figure 19-33: Phosphate Prices Recorded Weekly (2005-2022 YTD)

Table 19-17: Pearson Correlation Points For Phosphate Price Discovery Points (2005-2022 YTD)

Pearson Correlation Coefficients

	USA	Brazil	India
USA	1.00	0.94	0.95
Brazil	0.94	1.00	0.94
India	0.95	0.94	1.00

19.4.2 Fundamental Drivers

19.4.2.1 Short Term Outlook

As is the case with such extraordinary price moves, a combination of several bullish fundamental developments is driving phosphate prices higher:

1. **Strong demand underpinned by record or near record crop prices.**

Higher crop prices have fueled strong phosphate demand growth in 2020 and 2021, despite the sharp rise in prices. CRU estimates that global shipments of solid high analysis products increased 2.9% or 2.4 mt to 84.9 mt in 2020 (Figure 19-34). Shipments are estimated to have climbed another 2.1% or 1.8 mt to 86.7 mt in 2021.

The 4.2 mt increase in global shipments during the last two years has stressed supply capabilities of global producers.

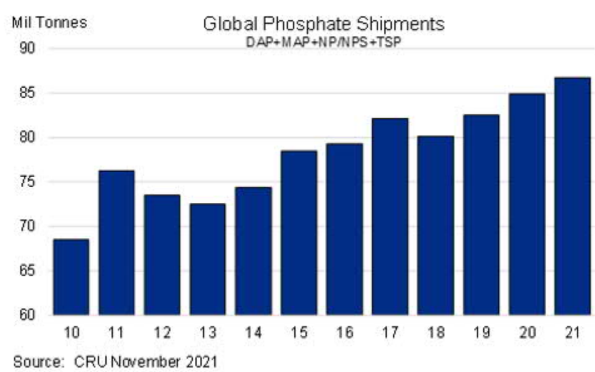


Figure 19-34: Global Phosphate Shipments (2010-2021)

Recent gains are broad-based and South America, led by massive increases in Brazil, is the leader for demand growth. CRU statistics show that South American shipments increased 15% or 1.7 mt to 13.3 mt in 2020. CRU estimates for 2021 indicate that shipments increased another 6% or 0.9 mt to 14.2 mt (Figure 19-35). However, customs statistics show that Brazil imported record shattering quantities of solid high analysis products last year (Figure 19-36), implying that shipment estimates for 2021 are quite possibly too low. Brazil imports of DAP/MAP/NPS/TSP surged 30% or 2.4 mt to 10.4 mt in 2021. The combination of a high crop prices and an extremely weak Brazilian real continues to super charge the export oriented agricultural sector (Figure 19-37).

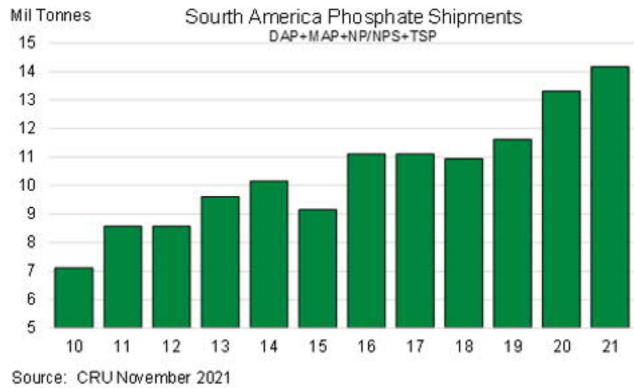


Figure 19-35: South America Phosphate Shipments (2010-2021)

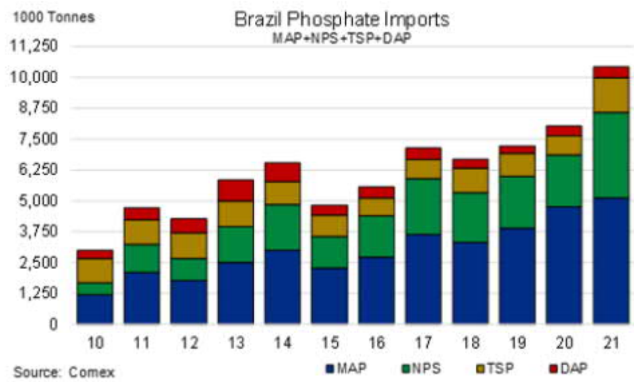


Figure 19-36: Brazil Phosphate Imports (2010-2021)

Figure 19-37: Brazilian Real -USD Exchange Rate (2010-2022 YTD)

Much higher crop prices underpin strong global demand growth. Figure 19-38 shows that price indexes for corn, soybeans and wheat at the end of March 2022 were up 95%, 81% and 135% respectively from the averages for 2019.

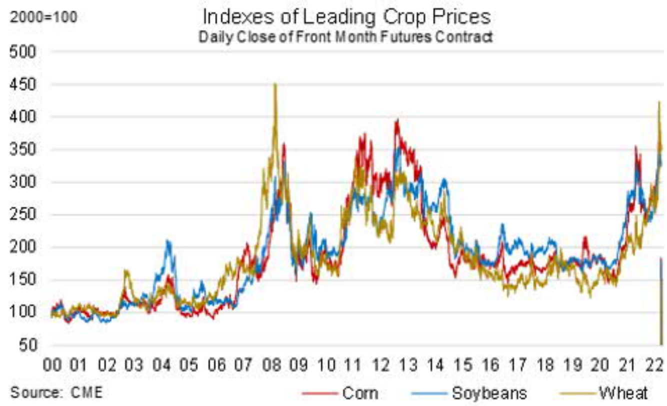


Figure 19-38: Indices Of Leading Crop Prices (2000-2022 YTD)

Crop prices have surged in response to the drawdown in grain and oilseed inventories outside China during the last few years. Figure 19-39 shows that the stocks-to-use percentage has ranged between 16% and 19% during the last 20 years. Grain and oilseed prices spiked when this percentage dropped to the low end of the range in 2003/04, 2007/08, 2011/12 and 2021/22.

USDA projections for the 2021/22 crop year released on 8 April 2022 indicate that stocks outside China will drop 9 mt to 403 mt or 15.5% of estimated use by the end of the 2021/22 crop year, the lowest percentage since 14.9% at the end of 2003/04. Much of the drop this year is due to lower South American soybean and corn crops in 2021/22.

If the 2021/22 projections are on target, global grain and oilseed inventories will have declined 57 mt during the last three crop years and the stocks-to-use percentage will have dropped nearly 300 basis points from 18.3% at the end of the 2018/19 crop year.

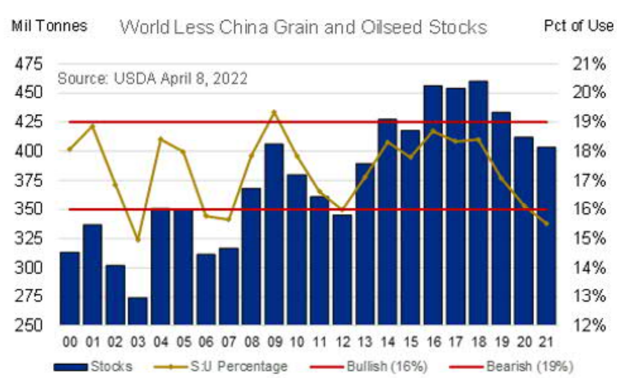


Figure 19-39: World Grain And Oilseed Stocks Less China (2000-2021)

2. **Thin channel inventories worldwide.**

The combination of strong farm demand, scrambled trade flows, and a muted supply response has resulted in drawdowns of channel inventories in many countries. The US was the leader for a bare distribution channel in 2020, although India was the clear leader in 2021.

In the US, estimated channel inventories jumped in 2018/19. Distributors positioned large tonnages in anticipation of strong demand, however farmers were unable to plant nearly 20 million acres due to extremely wet weather in 2019. The channel was destocked in 2019/20 and 2020/21 owing to strong farm demand and a large drop in imports during the last half of 2020. Imports collapsed while traders scrambled to find alternative origins following the filing of a US countervailing duty petition on phosphate imports from Morocco and Russia on June 26, 2020. It appeared that the US distribution channel was empty at the end of 2020 following an outstanding fall application season and limited supplies (Figure 19-40).

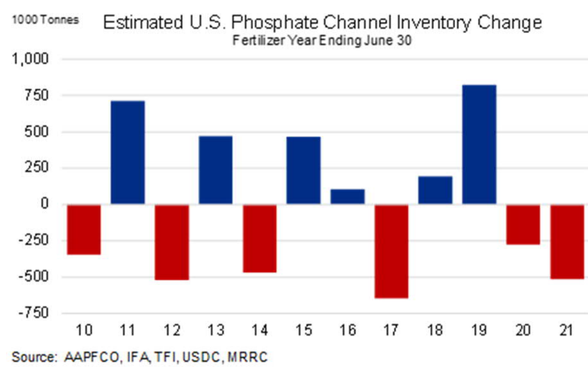


Figure 19-40: Estimated US Phosphate Channel Inventory Change

In India, DAP stocks throughout the entire distribution channel totaled less than 1.2 mt at the end of 2021, down from 2.7 mt one year ago and off from 4.0 mt two years ago (Figure 19-41). Indian farm demand remains strong given the relatively low DAP MRP. With no inventories to draw on this year, imports are projected to jump from just 4.6 mt in 2021 to a more normal level of 6.0-6.5 mt in 2022.

India bid aggressively for product tonnes earlier this year. The NOLA India DAP spread plunged from $150 per tonne in early 2021 to a record negative $200 per tonne in January 2022. India had purchased about 2.0 mt of DAP by the end of January.

Figure 19-41: India DAP Inventories (April 2019 – December 2021)

3. **A muted supply response.**

   As noted earlier, the top five phosphoric acid producing countries accounted for more than 75% of global output in 2020 (Table 19-18). Other than China, supply responses to the increase in phosphate prices by the other four top producing countries have been muted at best. Some producers have experienced supply shocks such as outages caused by Hurricane Ida in the US. Some have experienced production losses due to equipment malfunctions, supply chain disruptions and extended turnarounds as a result of Covid-19 complications. Others simply are running at capacity.

   The US, the third largest producer, is a prime example for muted supply responses. Statistics for the fertilizer year that ended on June 30, 2021, showed that US phosphoric acid production declined 4% or 230,000 t P_2O_5 from 2019/20 (Figure 19-42). DAP prices more than doubled during this period and the 2021/22 fertilizer year got off to a poor start due to outages caused by Hurricane Ida, mechanical problems and supply chain disruptions. Phosphoric acid output during the last half of 2021 was off 3% or 72,000 t P_2O_5 from a year earlier.

Table 19-18: Phosphoric Acid Production (Top Producers)

Phosphoric Acid Production

Rank	1000 Tonnes P_2O_5	Production 2020	Cumulative Production	Share	Cumulative Share
1	China	16,572	16,572	36%	36%
2	Morocco	7,149	23,721	15%	51%
3	United States	6,290	30,011	13%	64%
4	Russia	3,719	33,730	8%	72%
5	Saudi Arabia	2,375	36,105	5%	77%
	Other	10,516	46,621	23%	100%
	Total	46,621		100%	

Source: CRU November 2021

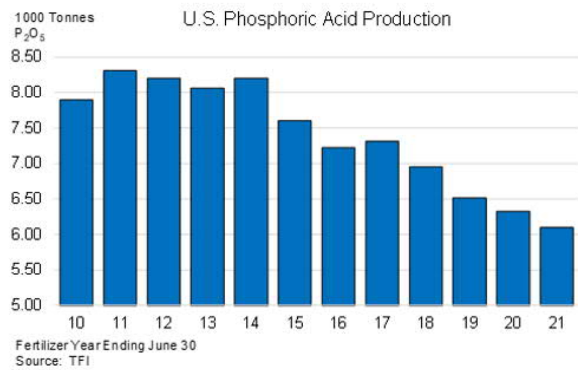


Figure 19-42: US Phosphoric Acid Production (2010-2021)

Figure 19-43 and Figure 19-44 show exports of the leading phosphate products from Morocco and Russia respectively for the last four fertilizer years ending on 30 June 2021 (consistent with the timeframe for the US statistics above). Neither show a significant supply response to the doubling of prices in 2020/21. Moroccan exports had jumped in 2019/20 or before the surge in prices due to the commissioning of new capacity as well as lower phosphoric acid sales. Exports of solid high analysis products increased just 3% from this new base to 9.77 mt in 2020/21.

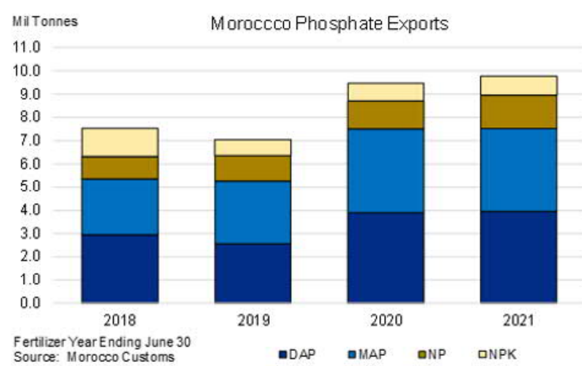


Figure 19-43: Morocco Phosphate Exports

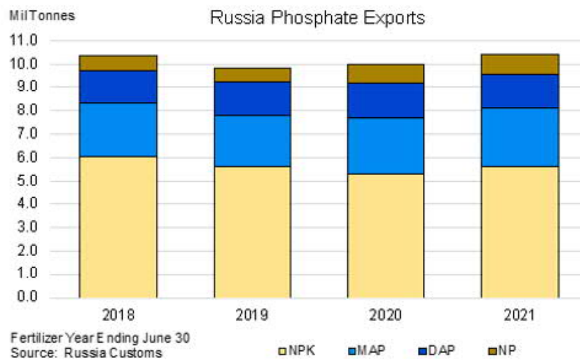


Figure 19-44: Russia Phosphate Exports

Russian exports of solid high analysis phosphate products including large quantities of NPK compounds totaled 10.45 mt in 2020/21, up 5% from 2019/20 but roughly the same quantity that was exported in 2017/18.

4. **Reduction in Chinese phosphate exports.**

Chinese producers responded to higher prices. DAP/MAP exports in 2021 totaled 10.0 mt, up 22% or 1.8 mt from a year earlier and the second highest total ever. Exports through October 2021 were up 42% or 2.9 mt from a year earlier and up 29% or 2.2 mt from a three year average for this period (Figure 19-45 and Figure 19-46.

Government officials noticed this surge and in mid-October the Chinese National Development and Reform Commission (NDRC) implemented an export inspection and approval directive aimed at restricting phosphate (as well as nitrogen) exports, citing the need to ensure Chinese farmers had adequate supplies of reasonably priced fertilizer.

As a result, DAP/MAP exports from November 2021 through March 2022 totaled just 1.2 mt, down 58% or 1.7 mt from the same period a year earlier and down 51% or 1.3 mt from a three year average for this period. The government has indicated that restrictions will remain in place until at least June 2022.

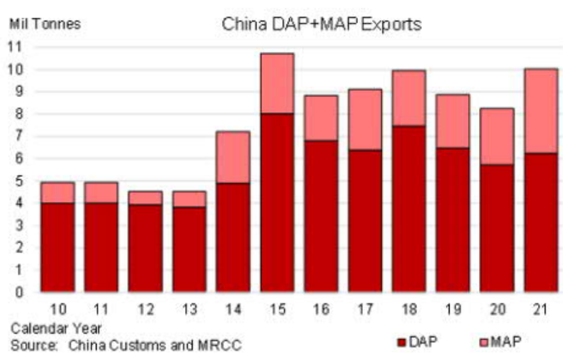


Figure 19-45: China DAP/MAP Exports (2010-2021)

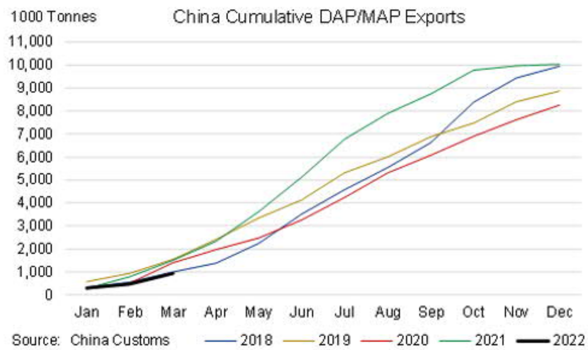


Figure 19-46: China Cumulative DAP/MAP Exports (2010-2021)

China has an exportable surplus about 10 mt of DAP/MAP, so export restrictions are about more than security of local supplies. An energy shortage, efforts to improve environmental quality, and preservation of scarce energy and mineral resources are also the drivers of export controls.

This development, implemented at a time when India has an enormous import requirement, has fueled the latest surge in prices and swung Asian prices from a large discount to a large premium to prices in the Americas earlier this year. As a consequence, near term fundamentals continue to look tight, and the price outlook remains positive.

The key question is whether this is just a temporary policy or a permanent shift in Chinese industrial and trade policy.

5. **Elevated raw materials costs driven in part by a rapid transition to low carbon energy.**

Phosphate producers are operating under intense cost pressure. For most producers, sulfur and ammonia are the two key purchased raw materials. Increases in sulfur and ammonia costs since January 2021 have added to the cost of producing one tonne of MAP, NPS and DAP in Florida (Figure 19-47 and Figure 19-48).

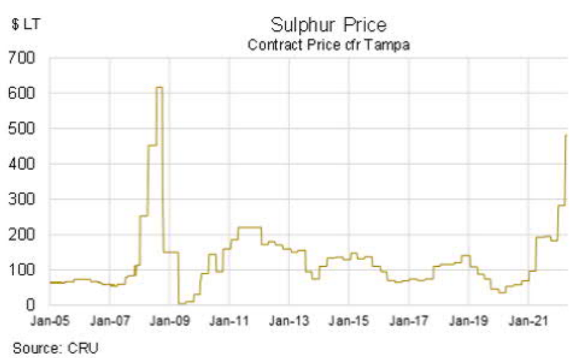


Figure 19-47: Sulfur Price (2005-2021)

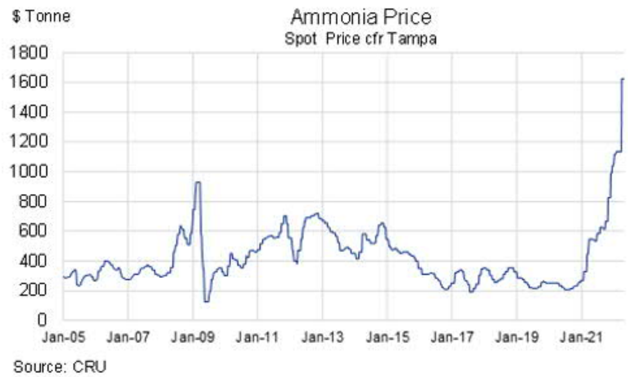


Figure 19-48: Ammonia Price (2005-2021)

6. **Disruptions caused by the Russia-Ukraine war.**

Russia's unprovoked invasion of Ukraine on February 24, 2022, thrust grain and fertilizer markets into turmoil. As a result, it is likely that the impact on these markets is in the early stages. Much of the world has sanctioned the Russian central bank, leading commercial banks and oligarchs. These are impacting the ability of financial institutions and companies to conduct business. In addition, the war has curtailed Black Sea and Baltic port operations, and freight and insurance rates have skyrocketed for exports from this region.

Sanctions are expected to have significant impacts on agricultural commodity and fertilizer markets. Russia and Ukraine account for 29% of global wheat exports, 18% of global corn exports, and 77% of global sunflower oil exports (and sunflower oil trade is only slightly less than soybean oil trade) (Figure 19-49). So, the direct impacts on grain and oilseed markets are significant. There is much speculation about the size of Ukrainian crops this year. Most estimates range from about 50% to 75% of a normal harvest.

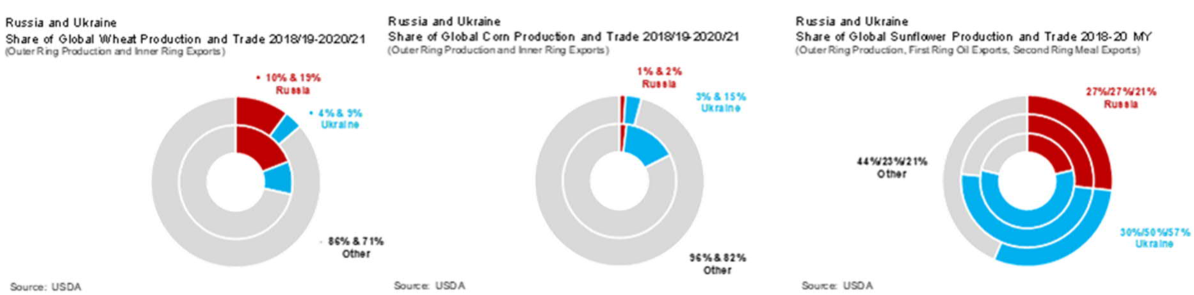


Figure 19-49: Russia and Ukraine Wheat, Corn and Sunflower Oil Export Share

Finally, there are significant direct impacts on global nitrogen, phosphate and potash markets (Figure 19-50). Russia produces about 6.0 mt of DAP/MAP/NP per year, accounting for 8% of global production, and exports around 4.4 mt per year or 12% of global exports. Russia also produces and exports large quantities of NPK compounds. Phosphate exports including NPK compounds exceed 10 mt per year.

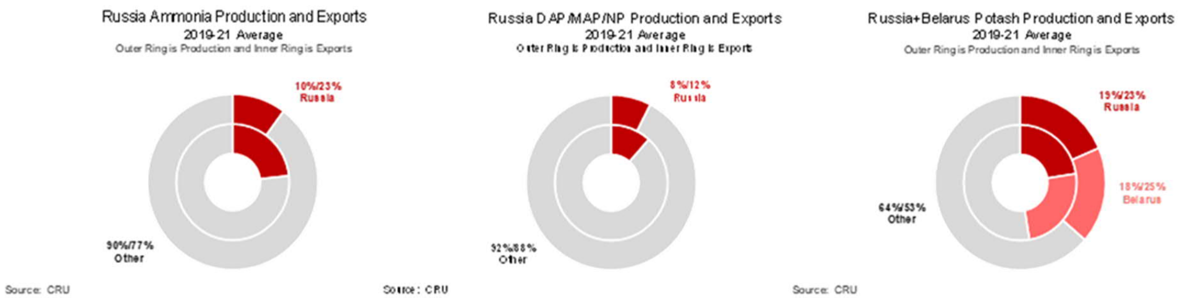


Figure 19-50: Russia Ammonia, DAP/MAP/NP and Potash Production and Exports

Russia is a leading producer and exporter of ammonia, a critical input for phosphate producers. Production totals about 19.0 mt of ammonia per year, accounting for 10% of global output. Russia exports about 4.5 mt of ammonia and accounts for 23% of total exports. The top destinations are Europe for downstream N and NPK production as well as industrial uses and North Africa (Morocco) for ammonium phosphate production.

19.4.2.1.1 Summary

The result of all these factors is that the near term outlook continues to look positive. Phosphate demand prospects still look good despite high phosphate prices. Farmers in some countries, particularly in India and China, are insulated to some extent from higher fertilizer prices. Fertilizer demand becomes a government decision. In other regions, much higher agricultural commodity prices more than compensate for the large increases in input costs resulting in still profitable farm economics. However, demand is no doubt at risk on crops which have not had a big run-up in prices (rice for example) or in regions where farm prices remain low, or the crop is for household consumption.

Based on import demand and export supply projections for this year, the global market will most probably need more than 9.0 mt of DAP/MAP exports from China to meet projected demand in 2022. If export restrictions stay in place well into the second quarter, and if China again imposes restrictions later this year, it increases the likelihood that production, transportation and logistics constraints may make it impossible for China to export more than 9.0 mt in 2022.

The combination of a significant decline in Chinese export supply, a drop in Russian exports, and satisfactory demand prospects underpinned by government policies and high crop prices is expected to keep phosphate prices at elevated levels in the near term.

19.5 Long Term Outlook

19.5.1 Methodology

This long term analysis forecasts the demand for phosphoric acid which is used to produce downstream fertilizer and non-fertilizer products. These include solid high analysis fertilizers (DAP/MAP/NP/NPS/TSP) as well as other solid and liquid fertilizer products (NPK/PK/SPA). It also includes non-fertilizer products, feed phosphate and purified acid used for a variety of industrial and some fertilizer uses.

On the supply side of the ledger, this analysis estimates current effective phosphoric acid capacity and also adds new capacity which is predicted to come online during the forecast period. It then estimates the effective global operating rate.

19.5.2 Summary

A more detailed description of demand and supply forecasts is included below. The long-term global phosphate outlook is positive based on conservative demand forecasts and aggressive supply projections. Much of this conclusion relies on there being increasingly scarce deposits of decent quality phosphate ore to develop and current producers exhausting some reserves during this forecast period.

Global phosphoric acid demand is expected to increase in the order of 18 mt P_2O_5 between 2020 and 2040. At the same time, new phosphoric acid capacity is projected to increase 15.7 mt P_2O_5 with 70% of new capacity from aggressive expansion by leading producers in Morocco and Saudi Arabia as well as the development of world scale greenfield projects in Algeria and Brazil. The analysis concludes that the effective global operating rate will increase from about 87% today to 93% in 2040 (Figure 19-51 and Figure 19-52). Moreover, the rate climbs and stays in the 90+% range for the last 15 years of the forecast period.

The analysis assumes that two US plants close due to the mine-out of rock reserves or phosphogypsum storage limits. It also assumes that Chinese capacity will remain unchanged during this period, and that a sizeable chunk of additional output from higher operating rates will come from China.

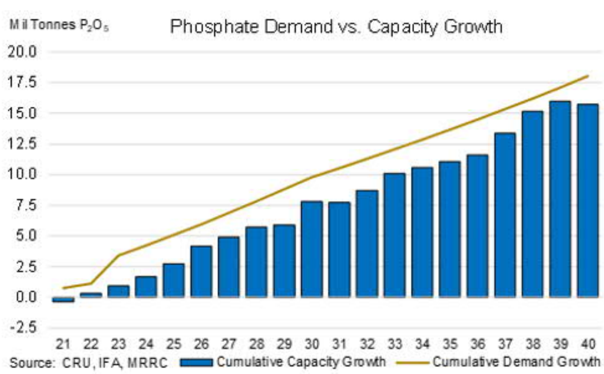


Figure 19-51: Phosphate Demand vs Capacity Growth (2021-2040)

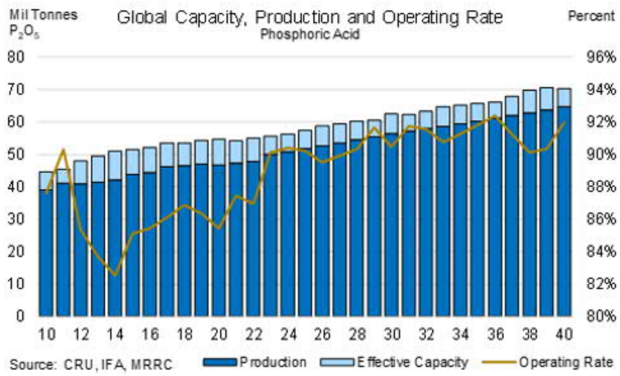


Figure 19-52: Phosphoric Acid Global Capacity, Production And Operating Rate (2010-2040)

19.5.3 Demand Forecasts

Phosphoric acid demand forecasts were built from demand forecasts by country or region for the downstream products. Table 19-19 shows phosphoric acid use for the production of these products in 2010 and 2020 as well as the change and compound annual growth rate (CAGR) during this period. Forecasts were derived by estimating CAGRs by country or region for these downstream products between 2020 and 2030 and between 2030 and 2040 based on trends and other expected developments.

Table 19-19: Phosphoric Acid Use For Production

			2020-2010			2030-2020			2040-2030	
Million Tonnes P_2O_5	2010	2020	CAGR	Change	2030	CAGR	Change	2040	CAGR	Change
Total	39.09	46.62	1.8%	7.53	56.42	1.9%	9.80	64.65	1.4%	8.22
DAP	15.01	16.16	0.7%	1.15	17.99	1.1%	1.84	19.97	1.0%	1.97
MAP	10.55	13.84	2.8%	3.29	16.68	1.9%	2.83	18.90	1.3%	2.23
NP/NPS	2.85	5.46	6.7%	2.61	7.27	2.9%	1.80	8.85	2.0%	1.59
NPK/PK/Other	1.97	1.91	-0.3%	-0.06	2.50	2.7%	0.59	2.50	0.0%	0.00
TSP	2.04	1.69	-1.9%	-0.35	2.28	3.1%	0.60	2.69	1.6%	0.40
SPA/APP	0.76	0.88	1.4%	0.11	0.96	0.9%	0.08	1.01	0.5%	0.05
Total Fertilizer	33.19	39.94	1.9%	6.75	47.68	1.8%	7.74	53.92	1.2%	6.24
Feed Phosphate	2.87	3.37	1.6%	0.50	4.05	1.9%	0.68	4.47	1.0%	0.42
PPA	3.04	3.31	0.9%	0.28	4.70	3.5%	1.38	6.26	2.9%	1.56
tMAP	*0.24*	*0.92*	14.4%	0.68	*2.29*	9.5%	1.37	*3.72*	5.0%	1.44
Other	*2.80*	*2.39*	-1.5%	-0.40	*2.41*	0.1%	0.02	*2.54*	0.5%	0.12
Total Non-Fertilizer	5.90	6.68	1.2%	0.78	8.74	2.7%	2.06	10.73	2.1%	1.98

After increasing 1.8% per year during the last decade, phosphoric acid demand is projected to grow at a comparable rate during this decade and then moderate slightly during the next decade. In particular, demand is projected to grow 1.9% per year or 9.8 mt P_2O_5 from 46.6 mt in 2020 to 56.4 mt in 2030. Demand is forecast to grow 1.4% per year or 8.2 mt to 64.6 mt in 2040 (Figure 19-53).

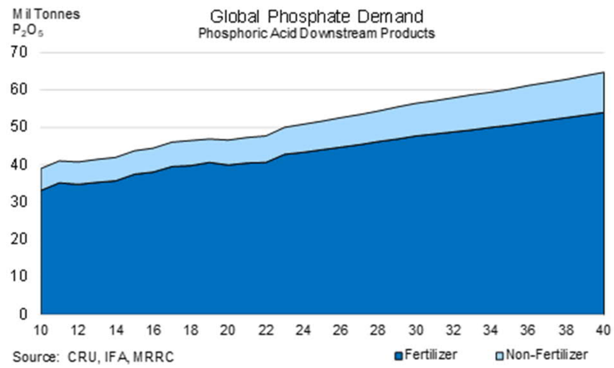


Figure 19-53: Global Phosphate Demand (Phosphoric Acid Downstream Products)

The demand forecasts used in this PEA analysis are not built from a detailed model of population and income growth, food demand, acreage and yield forecasts, nutrient recycling, and improvements in nutrient use efficiency. However, a comparison of projections in this analysis, with recent forecasts from such a sophisticated modeling exercise, concludes that the forecasts are in the same general alignment as would be produced from detailed modeling. These forecasts may turn out to be conservative in light of potential demand developments. The International Fertilizer Association (IFA) completed an extensive long term forecasting project last year. Two extracts from this presentation are included in Figure 19-54 and Figure 19-55. Nutrient demand forecasts (for crop production only) were developed based on agricultural scenarios derived from the UN-FAO long term projections to 2050, alternative estimates for the recycling of animal wastes, and different degrees of improvement in nutrient use efficiency (NUE). The results of the project were presented at the IFA Annual Conference held in Lisbon, Portugal on September 27-29, 2021.

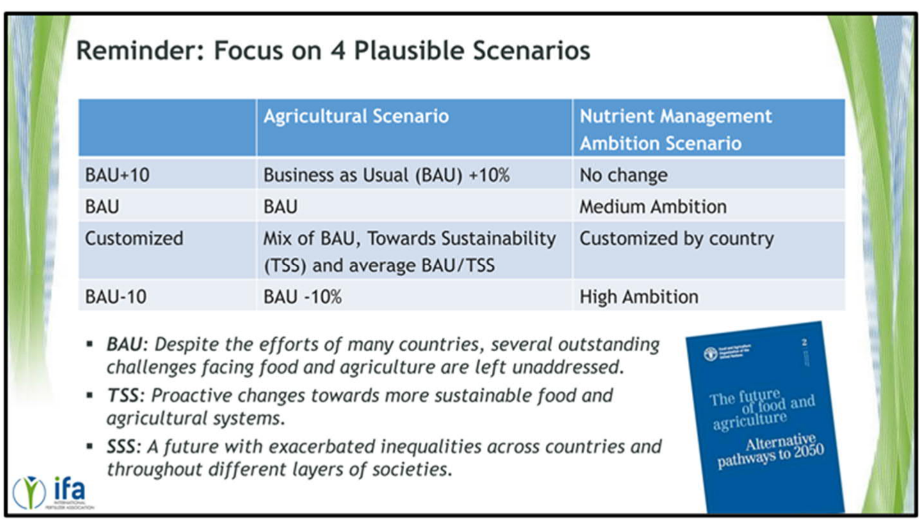


Figure 19-54: IFA Long Term Forecasting Project (Extract 1)

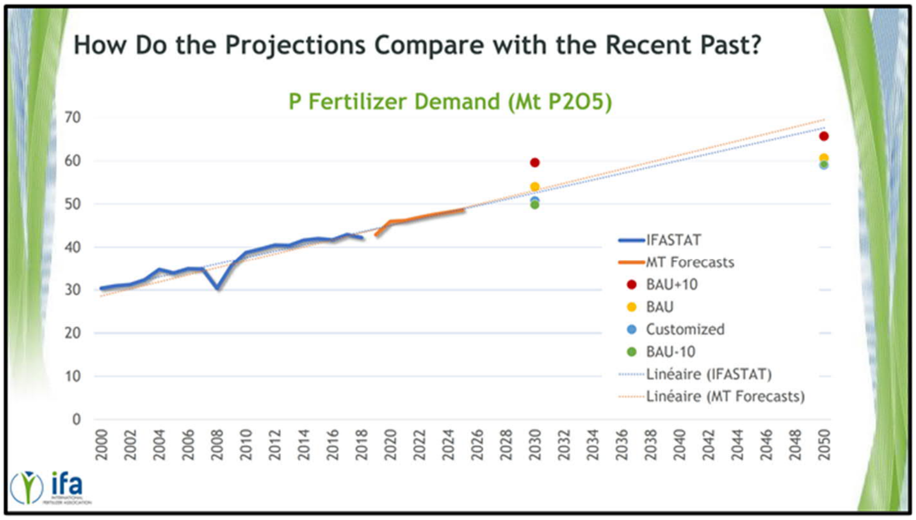


Figure 19-55: IFA Long Term Forecasting Project (Extract 2)

The IFA long term phosphate forecasts are summarized in Table 19-20. The base year used in the analysis was 2017. Phosphate use on crops was estimated to total 42 mt P_2O_5 in 2017. Demand in the low growth scenario increases at a CAGR of 1.0% to 50 mt in 2030 and to 59 mt in 2050. Demand in the medium growth scenario increases 1.1% per year to 53 mt in 2030 and to 60 mt in 2050, and demand in the high growth scenario increases 1.3% per year to 59 mt in 2030 and to 65 mt in 2050. The estimate for 2040 is based on the extrapolation from the 2030 and 2050 forecasts. A comparison of demand increases from 2017 to 2040 indicates that the forecasts used in this analysis are in the middle of the IFA scenarios.

The levels are different because the MRRC estimates exclude phosphate products (such as single superphosphate (SSP)) which are not produced with phosphoric acid as well as the amount of phosphate from rock in TSP.

Table 19-20: IFA Long Term Phosphate Forecasts

			2030-2017			2040-2017			2050-2017	
	2017	2030	CAGR	Change	2040	CAGR	Change	2050	CAGR	Change
IFA Long Term Forecast										
Low Growth	42.0	50.0	1.4%	8.0	54.5	1.1%	12.5	59.0	1.0%	17.0
Medium Growth	42.0	53.0	1.8%	11.0	56.5	1.3%	14.5	60.0	1.1%	18.0
High Growth	42.0	59.0	2.6%	17.0	62.0	1.7%	20.0	65.0	1.3%	23.0
MRRC	39.6	47.7	1.4%	8.1	53.9	1.4%	14.4	na	na	na

19.5.3.1 *The Traditional Demand Driver – The Food Story*

Phosphate demand drivers continue to look positive. Global grain and oilseed use have increased at a compound annual growth rate of 2.2% since the turn of this century. Demand has advanced at a moderate and steady pace driven by increases in population and household income. The USDA estimates that global grain and oilseed use will increase to 3.38 bt in 2021/22. Demand is projected to climb to 3.77 bt during the next five years. Steady growth in food demand, driven by population and income growth, is a stable and predictable relationship. In addition, many countries are building grain and oilseed reserves following lessons learned from the Covid-19 pandemic.

In terms of supply, the global production of grain and oilseed crops also has increased at a compound annual growth rate of 2.2% during this period. However, gains have come in significant step-ups following spikes in agricultural commodity prices in 2003/04, 2007/08, 2012/13, and now in 2021/22. It looks like at least two more step-ups are required this decade, particularly given only modest productivity/yield growth during the last several years (Figure 19-56).

Global grain and oilseed production have increased as a result of gains in both yields and harvested area. The average global yield for these crops has increased 850 kg per hectare since 2000 and accounted for about 70% of the increase in production so far this century.

Contrary to popular perception that no more land is under cultivation, harvested area worldwide has increased 150 million hectares since 2000 (Figure 19-57). To put that in perspective, the harvested area in China is 125 million hectares. Increases in harvested areas accounted for the remaining 30% or so of the increase in global production. Much of the gain was in South America, the CIS and Africa.

While “The Food Story” is not in vogue with investors today, it is still valid and these trends are expected to continue through the forecast period.

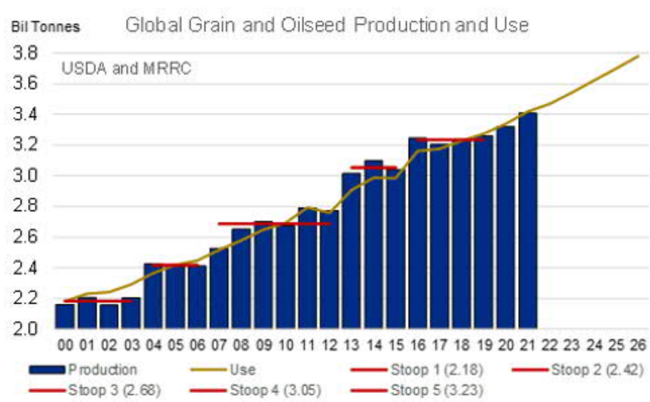


Figure 19-56: Global Grain and Oilseed Production and Use (2000-2021)

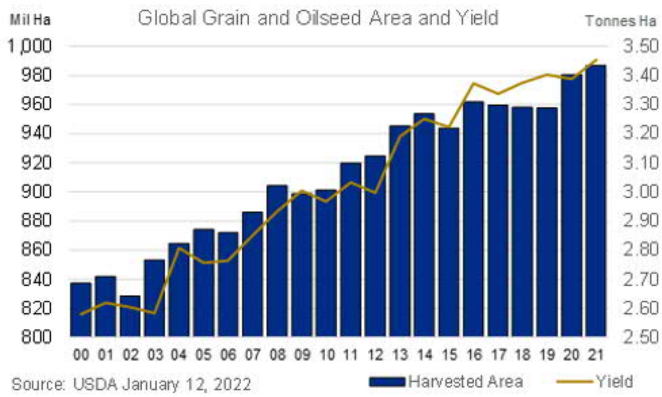


Figure 19-57: Global Grain and Oilseed Area and Yield (2000-2021)

19.5.3.2 The Demand Boosters – Renewable Diesel and Sustainable Aviation Fuel

In North America, a potential demand accelerator is the building of large renewable diesel (RD) and sustainable aviation fuel (SAF) industries. The growth in the US of renewable diesel production capacity is shown in Figure 19-58. Battery technologies are not well suited for long distance trucking or air transportation due to charging requirements and weight. As a result, trucking companies and airlines are turning to biofuels to reduce carbon footprints and meet an increasing number of state regulations and ESG demands of customers and investors.

RD is produced by hydrogenating vegetable oils or animal fats in a refinery that is similar to a petroleum refinery. RD is a perfect substitute for petroleum diesel and has none of the cold weather or other performance issues of biodiesel (a blend of petroleum diesel and non-hydrogenated biodiesel). The implementation of low carbon fuel standards in California is driving the development, but several other states have or are expected to implement similar standards.

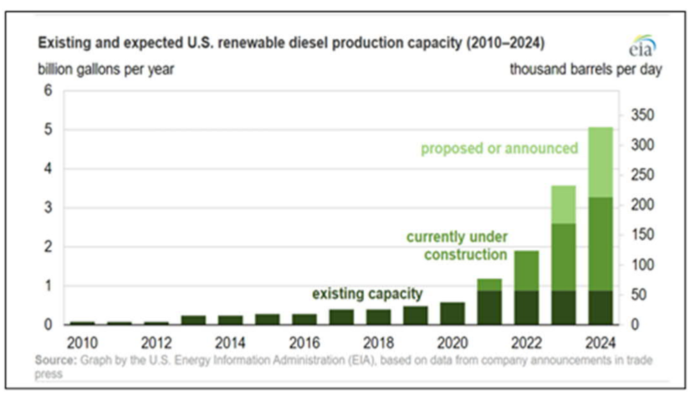


Figure 19-58: US Renewable Diesel Production Capacity (2010-2024)

There currently are more than a dozen RD refineries either operating or under construction in the US. Annual capacity is expected to reach 3.0 billion gallons by 2024, according to US Department of Energy estimates. Projects adding another 2.0 billion gallons of capacity have been announced or proposed. Renewable diesel may do for oilseeds what ethanol did for the corn about 15 years ago – and more.

Table 19-21 shows a "What If" exercise for renewable diesel. It illustrates the potential impact on US soybean oil demand and soybean acreage if RD production grows to 2.5 billion gallons by 2024/25 and to 5.0 billion gallons by 2029/30. The bottom line is that soybean acreage could grow by 15 to 35 million acres during the next eight years to meet RD demand.

Table 19-21: US Renewable Diesel Production "What If" Scenarios

U.S. Renewable Diesel Production What Ifs

The Renewable Diesel Revolution	2024/25	2029/30
Additional U.S. Renewable Diesel Capacity (Mil Gal)	2,500	5,000
Additional Feedstock Demand (Mil lbs. @ 7.7 lbs./Gal)	19,250	38,500
Percent Soybean Oil	90%	90%
Gross Additional U.S. Soybean Oil Demand (Mil lbs.)	17,325	34,650
Loss of Biodiesel/Exports/Other Soybean Oil Demand (Mil lbs.)	2,500	3,000
Additional U.S. Soybean Oil Imports (Mil lbs.)	500	750
Net Additional U.S. Soybean Oil Demand (Mil lbs.)	14,325	30,900
Additional Soybean Crush Demand (Mil Bu @ 11 lbs. Oil/Bu)	1,302	2,809
Loss of Soybean Exports (Mil Bu)	500	1,000
Net Additional U.S. Soybean Demand (Mil Bu)	802	1,809
Additional Soybean Area (Mil Acres @ 51.4 bpa)	16	35

This "What If" exercise makes generous assumptions for non-soybean-oil feedstocks (tallow, corn oil, canola oil, recycled vegetable oils), the loss of other soybean oil demand (displaced biodiesel demand, exports, other demand), increases in soybean oil imports, and decreases in soybean exports.

Large soybean processors including Cargill and ADM are expanding operations. New entrants are building renewable diesel or sustainable aviation fuel refineries (Figure 19-59). Ethanol plants are adding downstream capabilities to capture corn oil or further process ethanol into sustainable aviation fuel. Several large petroleum companies are converting refineries to produce renewable diesel or sustainable aviation fuel and are even backward integrating to crush soybeans. For example, Chevron is investing $600 million in a soybean processing partnership with Bunge to expand crush plants in Illinois and Louisiana. Conoco Phillips and Marathon Oil are investing with local partners in new crush plants in Shell Rock, IA and Spiritwood, ND, and have committed to a 100% offtake of the soybean oil for processing at converted petroleum refineries as far away as the west coast. On 28 February 2022, Chevron announced that it is buying biodiesel and renewable diesel maker Renewable Energy Group (REG) for $3.15 billion, furthering its goal of producing 100,000 b/d of renewable fuel production by 2030.

The “Build Back Better” infrastructure program in the US includes incentives for airlines to reduce carbon emissions by 20% by 2030 and to eliminate airline fossil fuel usage altogether by 2050. The major US carriers have committed to reducing their carbon footprints, and United airlines made its first flight using 100% SAF in December 2021.

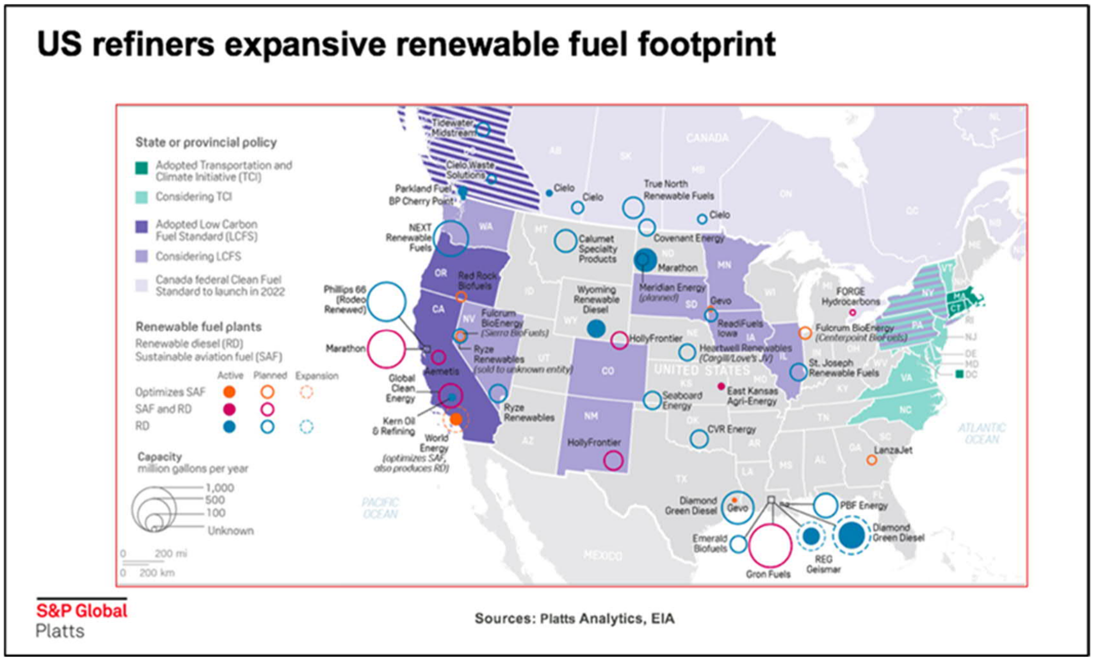


Figure 19-59: US Refiners Expansive Renewable Fuel Footprint

19.5.3.3 *The Demand Boosters – China Feed Grain Import Demand*

Some analysts, including those at Goldman Sachs, expect that China will develop a large structural feed grain deficit during this decade. This development is the result of two factors.

The first is the rapid rebuilding of the hog herd and a dramatic restructuring of the hog industry following the deadly outbreak of African Swine Fever (ASF) a few years ago. Large agribusiness firms and Chinese food companies are backward integrating into hog production using modern, and extraordinarily large scale, confinement operations and state-of-the-art feed technologies. Feed demand is growing rapidly as these complexes are built. There is a large new feed mill at the front end of these sprawling complexes which requires large quantities of mostly corn and soybeans to feed the more than 2.0 million hogs produced each year at these facilities (Figure 19-60) provides an example of one of these complexes.

The second is that domestic grain and oilseed production are hamstrung by the current structure of Chinese agriculture. China is working hard to boost feed grain production, and modern, large scale farm operations are developing, however a large share of Chinese grain production is still from small scale operations which are generally less productive. For example, current Chinese corn yields are 6.25 tonnes per hectare. This equates to 100 bushels per acre compared to average US yields of 175 bushels per acre.

As a result, demand growth, especially for feed grains, is outpacing supply growth resulting in a large structural feed grain deficit. China imported 29.5 mt of corn during the 2020/21 crop year, up from a five year average of 4.2 mt, according to USDA statistics. Corn imports are projected to decline in 2021/22 although the deficit is expected to widen over time absent a significant jump in Chinese corn production. The US is the largest corn exporter and would benefit from this development, particularly given the current setbacks in Ukraine. This could result in soybeans having to compete with corn for all the available acreage in the US.

What if China Imports 50 Million Tonnes of Corn in 2029/30?

Million Bushels Unless Noted	2020/21	2029/30	Change	All Yield Change Bu/Acre	All Area Change Mil Acres
China Corn Imports (Mil Tonnes)	29.5	50.0	20.5	na	na
China Corn Imports	1,161	1,968	807	na	na
U.S. Corn Exports (50% Scenario)	2,753	3,157	404	4.4	2.3
U.S. Corn Exports (60% Scenario)	2,753	3,237	484	5.3	2.8
U.S. Corn Exports (70% Scenario)	2,753	3,318	565	6.2	3.2

This is Muyuan Foods hog mega-farm in Neixiang county in Henan province. When fully operational, it will house 84,000 sows and produce about 2.1 million pigs per year. It is the largest hog operation in the world by a wide margin and is about 10 times the size of a typical facility in the United States. Muyuan Foods and other Chinese companies are designing higher-density automated farms, betting they can keep African Swine Fever (ASF) and other diseases out while increasing efficiency to meet the country's large appetite for pork.

Figure 19-60: Muyuan Foods Hog Mega-farm, Neixiang County, Henan Province, China

19.5.3.4 The Demand Boosters – Lithium Iron Phosphate (LFP) Batteries

Growth in purified phosphoric acid demand, for the production of technical MAP (tMAP) used in water soluble fertilizers and lithium iron phosphate (LFP) batteries for electric vehicles (EV), is expected to provide a moderate increase to non-fertilizer demand during the forecast period.

Most forecasts indicate that tMAP use for LFP batteries will increase from about 50,000 t of product in 2020 to between 750,000 mt and 1.0 mt by 2030. Further growth to around 1.5 mt by 2040 is possible depending on technological developments and government policies. This development may have the largest impact on the Chinese phosphate industry. LFP batteries, championed by Tesla, accounted for 57% of total EV battery production in China in 2021. LFP batteries cost less, and that becomes increasingly important as the Chinese government withdraws subsidies for EV during the next several years. How this battery technology evolves still is uncertain, although the use of LFP batteries, particularly in the Chinese EV industry (including Tesla), is a positive development and could raise the likelihood that the country will maintain policies to keep more phosphate in China rather than export.

19.5.3.5 Supply Forecasts

Supply increases from both higher operating rates, and massive investments in new capacity, will be required to meet the projected demand increase of 18 mt P_2O_5 during the forecast period.

For existing facilities, effective capacity was defined as lower of either nameplate capacity or 105% of the highest production achieved. Based on this criterion, effective capacity was 92% of global nameplate capacity. Estimates varied by country. For example, effective capacity was 100% of nameplate for Morocco but only 80% for India. The analysis assumes phosphoric acid production will cease at two US facilities during this period (Nutrien White Springs in 2031 and Mosaic Riverview in 2040).

Assumptions regarding new capacity additions are purely speculative and are listed in Table 19-22. This exercise does illustrate the need for massive capital investment.

Most of the new capacity will come from Morocco and Saudi Arabia. This analysis assumes Morocco will add 6.5 mt P_2O_5 of phosphoric acid capacity or more than 40% of the total increase. The completion of Line F and the Laayoune project will add 1.0 mt during the first half of this decade. However, the current plans for JPH 5-7 are apparently granulation only projects. This analysis assumes that JPH 8-10 will include phosphoric acid and that OCP will eventually add phosphoric acid capacity to the JPH 5-7. In addition, increased capacity labeled Lines G-K are included in these projections.

In Saudi Arabia, Ma'aden is expected to add Phosphate 4 and Phosphate 5 projects after it completes the Phosphate 3 project. This analysis assumes Saudi adds 4.5 mt of new phosphoric acid capacity or almost 30% of the total increase during the forecast period.

In addition, the analysis assumes the development of two world scale greenfield projects elsewhere. In particular, the supply projections assume that Mosaic will develop a world scale project around its high quality Patrocinio reserves in Brazil and that the long planned project in Algeria will eventually go forward.

The above speculation accounts for almost 90% of the projected 15.7 mt increase in phosphoric acid capacity. Other increases come from smaller projects in Kazakhstan, Russia and Egypt as well as expected increases in effective capacity in other countries such as Tunisia. The Martison project also is included in this analysis.

The effective global operating rate will need to increase from about 87% today to the 90%-92% range for most of the forecast period, even with the addition of this new capacity. Furthermore, the reliability of Chinese supplies is uncertain, if not suspect, and more mine-outs of reserves than are included in this long term analysis are possible.

Table 19-22: Potential Phosphoric Acid Capacity Additions

Potential Phosphoric Acid Capacity Additions

Nation	Firm	Facility	Project	Start-Up	2021	2022	2025	2030	2035	2040
Morocco	OCP SA	Jorf Lasfar	JPH 8	2030				500	500	500
			JPH 9	2031					500	500
			JPH 10	2032					500	500
			JPH 5+Acid	2033					500	500
			JPH 6+Acid	2034					500	500
			JPH 7+Acid	2035					500	500
			Line F	2022		250	500	500	500	500
			Line G	2036						500
			Line H	2037						500
			Line I	2038						500
			Line J	2039						500
			Line K							500
		Laayonue		2025			500	500	500	500
Total Cumulative Additions						250	1,000	1,500	4,000	6,500
Share of Total						80%	37%	19%	36%	41%
Saudi Arabia	Ma'aden/SAFCO	RAK	P-3	2026				1,500	1,500	1,500
			P-4	2032					1,500	1,500
			P-5	2037						1,500
Total Cumulative Additions								1,500	3,000	4,500
Share of Total								19%	27%	29%
Russia/Kazakhstan/Egypt				2024-26			995	1410	1410	1410
Share of Total							37%	18%	13%	9%
Brazil	Mosaic	Patrocinio		2030				1,500	1,500	1,500
Share of Total								19%	14%	10%
Algeria	SMC JV	Tebessa		2037					0	1380
Share of Total									0%	9%
ROW Cumulative Additions						63	731	1,876	1,175	429
Share of Total						20%	27%	24%	11%	3%
Global Effective Capacity					54,203	54,891	57,304	62,364	65,663	70,297
Capacity Change					-375	688	1,064	1,900	500	-255
Cumulative Capacity Change					-375	313	2,726	7,786	11,085	15,719

19.6 Price Model, Scenario Analysis and Price Forecast

19.6.1 Price Model

A statistical model was used to provide guidance in making long term price forecasts. The analysis utilized quarterly average statistics from 2002 Q1 to 2021 Q4 (80 observations). The regression model assumes that the price of DAP fob New Orleans (NOLA) barge is a function of the prices of sulfur and ammonia delivered to Tampa, the price of the front month futures corn price, and the phosphoric acid operating rate of the Moroccan industry. A variable was also included to pick up the impact of the price fly-up in 2008.

The model results were good in a statistical sense. The R2 of 89% indicates that variations in the explanatory variables noted above explained 89% of the variation in the DAP price. The standard error, however, was large and implies a wide 68% prediction interval of +/- $57per ton. All explanatory variables have correct signs and all coefficients are significantly different from zero at the .95 level of confidence (t-statistics > 2.0). For example, the model indicates that if the front month futures corn price increases 100 cents per bushel, then the price of DAP fob NOLA will increase by $32 per ton.

The regression results are listed in Table 19-23. A chart of actual vs. model forecasts as well as three tables showing price forecasts for current, upside and downside values of the explanatory variables are shown on the next page.

Table 19-23: Statistical Analysis Summary (Long Term Price Forecasts)

2002 Q1-2021 Q4
SUMMARY OUTPUT

Regression Statistics	
Multiple R	0.94
R Square	0.89
Adjusted R Square	0.88
Standard Error	57.4
Observations	80

ANOVA

	df	*SS*	*MS*	*F*	*Significance F*
Regression	5	2,014,850	402,970	122.2	2.63E-34
Residual	74	244,032	3,298		
Total	79	2,258,883			

	Coefficients	*Standard Err*	*t Stat*	*P-value*	*Lower 95%*	*Upper 95%*
Intercept	-36.870	49.036	-0.75	0.454	-134.577	60.836
2008 Fly-Up Dummy	218.566	63.965	3.42	0.001	91.112	346.020
Ammonia	0.179	0.069	2.59	0.012	0.041	0.317
Sulphur	0.931	0.192	4.85	0.000	0.548	1.313
Corn	0.321	0.070	4.56	0.000	0.180	0.461
Morocco Operating Rate	116.143	51.693	2.25	0.028	13.142	219.144

19.6.2 Scenario Analysis and Price Forecasts

Figure 19-61 shows that model forecasts closely track the actual NOLA barge although there are quarters when deviations are significant. As noted above, the model is used as a guide only for making long term forecasts.

During model development, scenarios (plausible future states of the phosphate market) are developed and then quantified in the values of the explanatory values. The sulfur and ammonia cost drivers are straightforward. Increasing concerns in regard to the long term sulfur supplies, and prices in an Electric Vehicle (EV) world or competing uses for ammonia for fuel or hydrogen transport, can be incorporated into a scenario. The same holds true for the price of corn as a proxy for the agricultural situation and phosphate demand prospects. The Moroccan operating rate also serves as a gauge for the global phosphate supply and demand balance.

Table 19-24, Table 19-25 and Table 19-26 show model forecasts for explanatory variables in January 2022, April 2022, and Downside, respectively. The 2022 Q1 sulfur price was settled at $282 per long ton cfr Tampa; the January 2022 ammonia price was settled at $1,115 per tonne cfr Tampa; the front month corn contract on the CME CBOT traded at an average price of $6.09 per bushel during January 2022; and the Moroccan industry is running at relatively high rates. Based on these values, the model results in a DAP fob NOLA barge price forecast of $731 per ton. The CRU Fertilizer Week average price for January 2022 was $698 per ton.

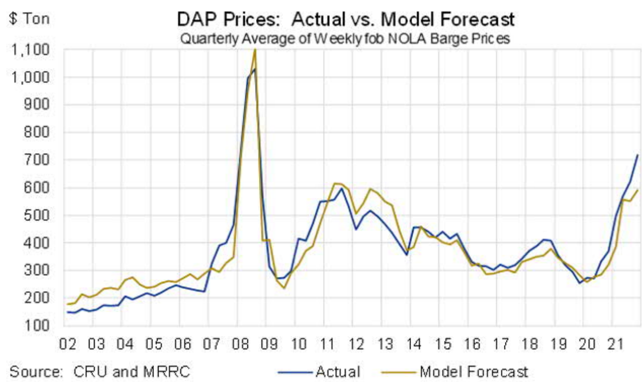


Figure 19-61: DAP Prices: Actual vs Model Forecast (2002-2022)

Table 19-24: DAP Price Forecast (January 2022)

DAP Price Forecast - January 2022	
Variable	Value
Ammonia - $ MT c&f Tampa	$1,115
Sulphur - $ LT c&f Tampa	$282
Corn - $ Bu Nearby Contract	$6.09
Morocco Operating Rate	95%
DAP $ Ton fob NOLA	$731

Table 19-25: DAP Price Forecast (April 2022)

DAP Price Forecast - April 2022	
Variable	Value
Ammonia - $ MT c&f Tampa	$1,625
Sulphur - $ LT c&f Tampa	$481
Corn - $ Bu Nearby Contract	$7.85
Morocco Operating Rate	95%
DAP $ Ton fob NOLA	$1,064

Table 19-26: DAP Price Forecast (Downside)

DAP Price Forecast - Downside	
Variable	Value
Ammonia - $ MT c&f Tampa	$800
Sulphur - $ LT c&f Tampa	$250
Corn - $ Bu Nearby Contract	$5.00
Morocco Operating Rate	90%
DAP $ Ton fob NOLA	$604

Three scenarios were developed for this analysis. Figure 19-62 and Figure 19-63 show historical prices as well as price and stripping margin forecasts. For each scenario the stripping margin is the price of DAP less the cost of sulfur and ammonia per tonne of DAP. Figure 19-64 shows the values for the explanatory variables. Prices are annual averages.

The first scenario is **Mean Reversion**. This scenario assumes that explanatory variables revert to values close to those prior to the recent fly-up and result in margin forecasts that are in line with the 20 year average.

The second scenario is **Fundamental Change**. This scenario reflects a constructive demand outlook as captured by higher corn prices. It also reflects an expected tight global phosphate supply and demand balance as indicated by high Moroccan operating rates. Sulphur and ammonia costs also are expected to remain elevated.

The third scenario is a **Green New World**. It assumes that low carbon initiatives result in strong demand for agricultural commodities for food and fuel (renewable diesel and sustainable aviation fuel in particular), require voluntary sulfur production to make up for large expected losses of involuntary output, heightens competition for ammonia for fuel and hydrogen transport, and increases the cost and delays the development of new phosphate capacity.

Each scenario is then handicapped. The weighted average of the three scenario forecasts is the Base Case. The Base Case forecasts are used for the financial evaluation in the rest of the report.

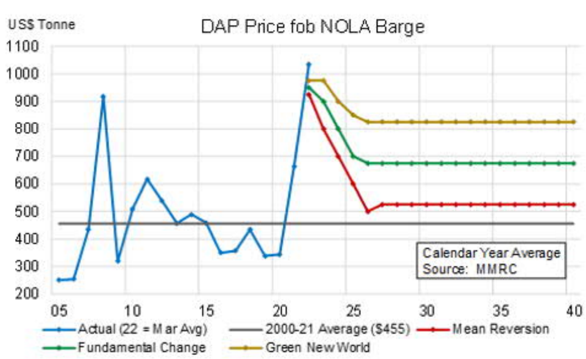


Figure 19-62: DAP Price fob NOLA Barge

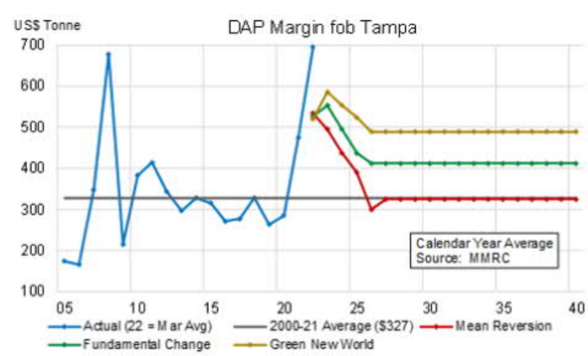


Figure 19-63: DAP Margin fob Tampa

The MAP price forecast fob NOLA was based on the historical spread to DAP. Current countervailing duties on phosphate imports from Morocco and Russia has caused this spread to widen. The Base Case assumes the duties case will be resolved by 2026 and the spread will revert to normal levels. MAP, NPS, and SPA price forecasts for the Martison Project target markets were derived from historical product and location spreads.

The charts in Figure 19-64 show the values for the driver variables for each scenario as well as the relationship between the variable and the price of DAP.

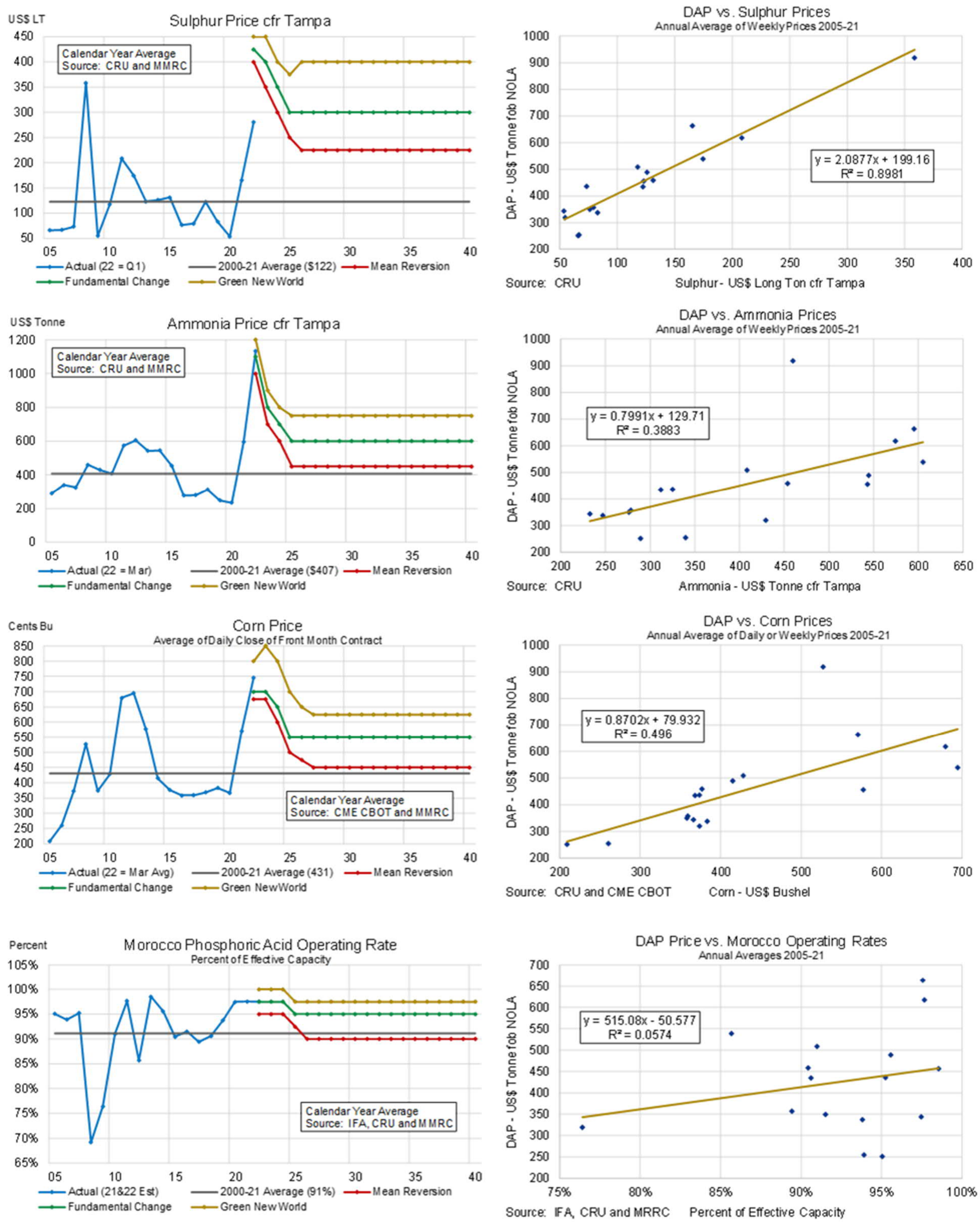


Figure 19-64: Values For The Explanatory Variables

19.7 Martison Competitive Analysis

19.7.1 Fertilizer Conversion Complex Logistics

The proposed location of the FCC is on the Ontario Northland Railway (ONR) 22 km northwest of Hearst (Figure 19-65). The FCC site is within the CN Rail interchange with the ONR at Hearst. CN Rail currently does not provide direct service to the site, and future rail service by CN Rail still needs to be determined. The rail track which will service the site has a track weight capacity of 263,000 lbs. per rail car, thus the rail loading of cars cannot exceed 92 tonnes per car.

CN Rail will provide transportation services for both inbound production inputs including sulfur and ammonia as well as outbound fertilizer products including MAP, NPS and SPA.

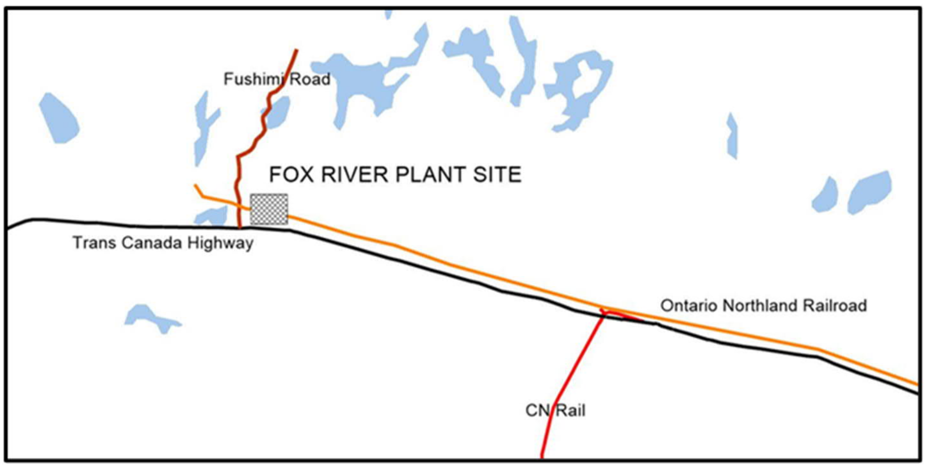


Figure 19-65: Proposed Fox River FCC Site

19.7.2 Molten Sulfur Transportation and Logistics Cost Estimates

The inbound movement of molten sulfur is expected to originate from Alberta where there are large involuntary supplies from gas, oil and oil sands refining operations. The Project will plan to source sulfur from both gas and oil refineries near Edmonton in central Alberta and from oil sands plants located farther north near Fort McMurray.

The inbound costs of delivering 425,000 t of molten sulfur from Edmonton and Fort McMurray using 2022 rail and lease car rates are summarized in Table 19-27 and Table 19-28 respectively. A blended cost based on an assessment of availability from the two Alberta locations are used in the financial analysis.

Table 19-27: Inbound Sulfur Costs From Edmonton, AB

Ex Edmonton AB Summary

- Volume: 425,000 Tonnes
- Source: Edmonton AB
- Total Inbound Logistic Cost
- Molten Shipper Tank Cars
- Service: 70-90 Car Unit Trains
- Total Cost: C$74.44 Tonne
- Total Cost: C$31.64 Million
- Service: 15 Car Block
- Total Cost: C$77.11 Tonne
- Total Cost: C$32.77 Million

Freight and Car Costs	Unit Train C$ Tonne	Block C$ Tonne
Estimated Freight Costs	67.44	69.44
Car Equipment Costs:	7.00	7.67
Freight and Car Cost	74.44	77.11

Freight and Car Costs	Unit Train Million C$	Block Million C$
Estimated Freight Costs	28.66	29.51
Car Equipment Costs:	2.98	3.26
Freight and Car Cost	31.64	32.77

Table 19-28: Inbound Sulfur Costs From Fort McMurray, AB

Ex Ft McMurray AB Summary

- Volume: 425,000 Tonnes
- Source: Ft McMurray AB
- Total Inbound Logistic Cost
- Molten Shipper Tank Cars
- Service: 70-90 Car Unit Trains
- Total Cost: C$103.56 Tonne
- Total Cost: C$44.01 Million
- Service: 15 Car Block
- Total Cost: C$106.22 Tonne
- Total Cost: C$45.14 Million

Freight and Car Costs	Unit Train C$ Tonne	Block C$ Tonne
Estimated Freight Costs	95.22	97.22
Car Equipment Costs:	8.33	9.00
Freight and Car Cost	103.56	106.22

Freight and Car Costs	Unit Train Million C$	Block Million C$
Estimated Freight Costs	40.47	41.32
Car Equipment Costs:	3.54	3.82
Freight and Car Cost	44.01	45.14

A lower sulfur requirement for the Martison operation is a competitive advantage relative to producers processing sedimentary rock. The sulfur requirement depends on rock quality. In particular, the calcium:phosphate ($CaO{:}P_2O_5$) ratio determines how much sulfuric acid is required to acidulate phosphate rock. It also determines how much by-product phosphogypsum is produced. For most sedimentary rock, 2.80-2.90 t of sulfuric acid are required per tonne of P_2O_5. All four US producers process sedimentary ore.

For igneous rock, however, only 2.40-2.50 t are needed due to a lower $CaO{:}P_2O_5$ ratio. This is a competitive advantage today, and it becomes an even greater advantage if sulfur prices increase in the future due to the expected decline in fossil fuel production. Tests to produce phosphoric acid from Martison phosphate concentrate, performed by both the International Fertilizer Development Company and Jacobs Engineering, demonstrated that sulphuric acid consumption did not exceed 2.50 tonnes per tonne of P_2O_5.

19.7.3 *Ammonia Transportation and Logistics Cost Estimates*

The inbound movement of ammonia is planned to originate from the CF Industries nitrogen complex at Courtright ON. Courtright is 950 km from the Hearst site, yet it is the nearest and best source of ammonia for the project and is serviced by CN Rail. CN Rail provided the rail freight estimate from Courtright to Hearst used in the competitive analysis. It is assumed the shipments of ammonia would occur in 15 car blocks. The car transit times were estimated from CN Rail, and leasing cost were provided from car leasing companies. The inbound costs of delivering ammonia from Courtright, ON, are summarized in Table 19-29.

Table 19-29: Inbound Ammonia Costs From Courtright, ON

Summary

- Volume: 100,000 Tonnes
- Source: Courtright ON
- Total inbound logistic cost
- Ammonia Shipper Tank Cars
- Service: 15 car blocks
- Total Cost: US$101.17 Tonne
- Total Annual Cost: US$10.12 Million

Freight and Car Costs	US$ Tonne
Estimated Freight Costs	91.27
Car Equipment Costs:	9.90
Total Freight Cost	**101.17**

Freight and Car Costs	Million US$
Estimated Freight Costs	9.13
Car Equipment Costs:	0.99
Total Freight Cost	**10.12**

19.7.4 *Cash Production Cost Estimates – Martison Project and Competitors*

Cash production costs for the Martison Project as well as for 10 other producers which supply, or could supply, the total addressable market (TAM) have been estimated. Martison rock and plant operating costs were based on engineering estimates. Phosphate rock and plant operating costs for competitors were estimated from several sources including CRU, company reports, and other public sources of information. Sulfur and ammonia costs used in the analysis were averages for 2021 rather than extraordinarily high current spot values.

The cost estimates show significant variability across companies due to a number of factors such as integrated rock or ammonia production. Based on cost estimates provided by the engineering companies, Martison cash operating costs are forecasted to rival the lowest cost offshore producers that benefit from either low cost phosphate rock (Morocco) or low cost natural gas and integrated ammonia production (Saudi Arabia and Russia). The cost advantage of the Martison operation is based on the cost and quality of its phosphate rock, lower sulfur procurement and transportation costs, and competitive ammonia costs.

The Martison fob plant costs are significantly less than estimates for its four US competitors.

Figure 19-66 highlights both the potential competitive advantages and disadvantages of the Martison project.

Lower transportation and logistics costs to target markets enhance the cost advantage for the Project. This analysis and charts on the following pages show that Martison ranks as the lowest cost supplier to the Canadian provinces and the US northern tier states.

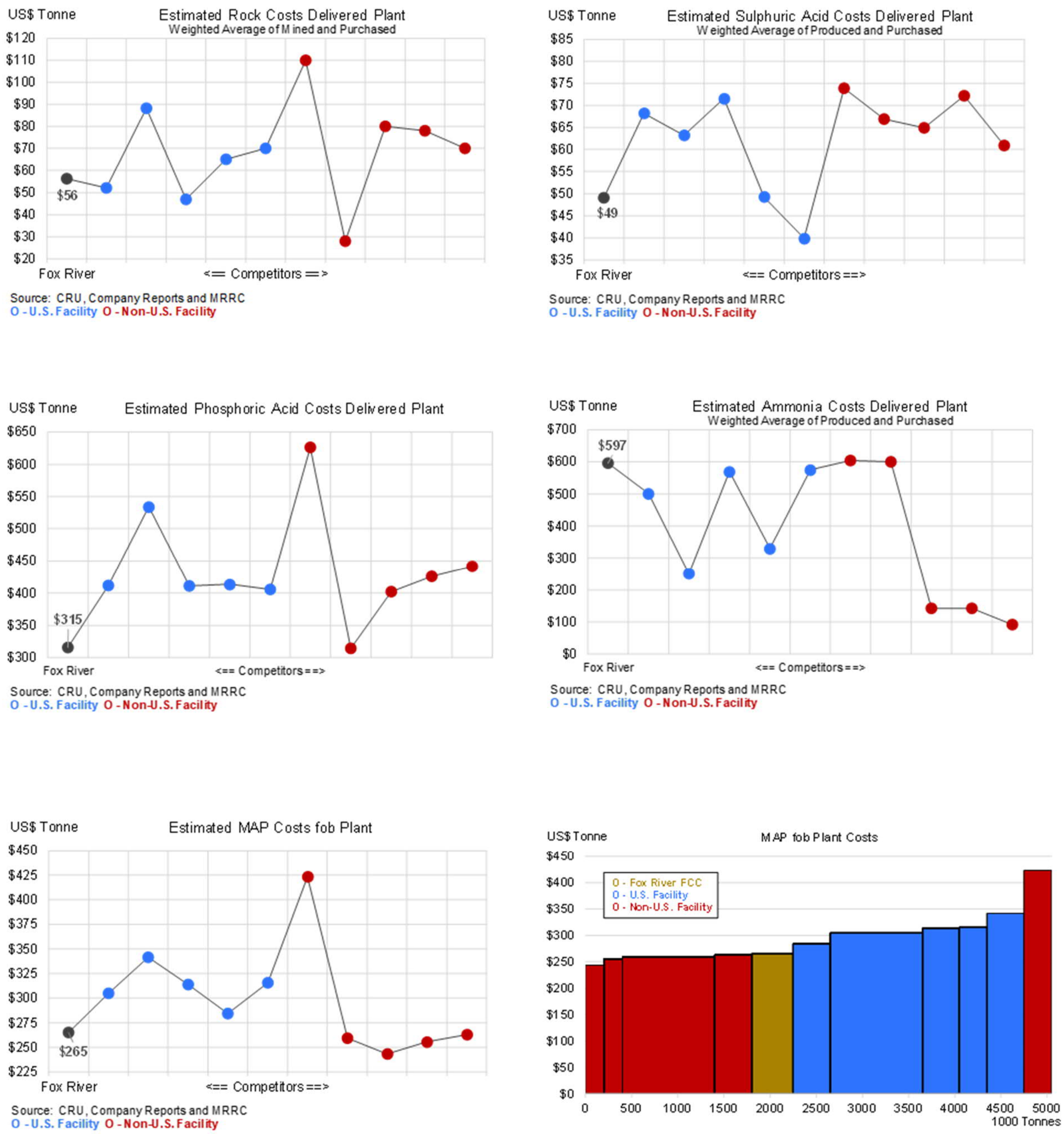


Figure 19-66: Cash Production Cost Estimates: Fox River vs Competitors

19.7.4.1 *Total Addressable Market (TAM)*

The total addressable market (TAM) for Martison project includes the five leading Canadian agricultural provinces from Quebec to Alberta as well as 10 US northern tier and leading agricultural states from Ohio to the Dakotas and Nebraska (Figure 19-67).

Demand estimates by Canadian province and US state were based on the most reliable and current information. Estimates by province came straight from Statistics Canada shipments or import statistics by province. Demand estimates by state were estimated from total US implied shipments and state shares for phosphate products from the most recent AAPFCO/TFI Commercial Fertilizer reports.

The TAM for the Martison project is expected to increase over time due to phosphate demand growth, particularly in Canada. Martison market shares for its most freight advantaged markets also are expected to increase from entry share targets in Year 1 to maximum share targets in Year 5. An assessment of the maximum share was based on the size of freight advantages and input from industry executives who had marketed phosphate products in the target markets.

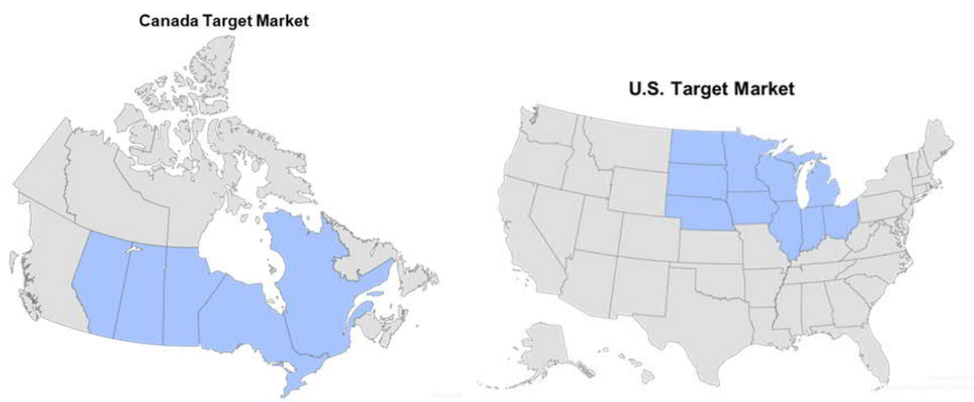


Figure 19-67: Canada and US Total Addressable Target Markets

The estimated MAP use in the five leading Canadian agricultural provinces and the 10 northern tier US states are shown in Figure 19-68 and the MAP Initial TAM and target volume are detailed in Table 19-30.

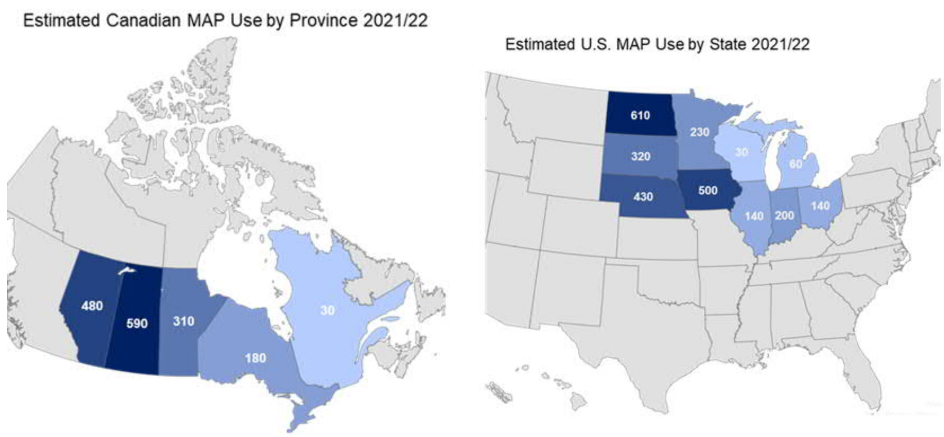


Figure 19-68: Canada and US MAP Use (2021-2022)

Table 19-30: MAP Initial Total Addressable Market and Target Volume (2021-2022)

MAP Initial Total Addressable Market and Target Volume

Tonnes	Est. MAP Demand 2021/22	Fox River Resources Corp Target Share	Target Volume	Percent of Sales
Quebec	30,000	30%	9,000	2%
Ontario	180,000	30%	54,000	11%
Manitoba	310,000	25%	77,500	16%
Saskatchewan	590,000	20%	118,000	25%
Alberta	480,000	10%	48,000	10%
Canada Total	1,590,000	19%	306,500	65%
North Dakota	610,000	10%	61,000	13%
South Dakota	320,000	2%	7,000	1%
Nebraska	430,000	2%	7,000	1%
Minnesota	230,000	15%	34,500	7%
Iowa	500,000	2%	10,000	2%
Wisconsin	30,000	10%	3,000	1%
Illinois	140,000	5%	7,000	1%
Michigan	60,000	5%	3,000	1%
Indiana	200,000	5%	10,000	2%
Ohio	140,000	15%	21,000	4%
USA Total	2,660,000	6%	163,500	35%
North America Total	4,250,000	11%	470,000	100%

Source: Stats Canada, TFI, USDC, TFI/AAPFCO, and MRRC

The estimated NPS use in the five leading Canadian agricultural provinces and the 10 northern tier US states are shown in Figure 19-69 and the NPS Initial TAM and target volume are detailed in Table 19-31.

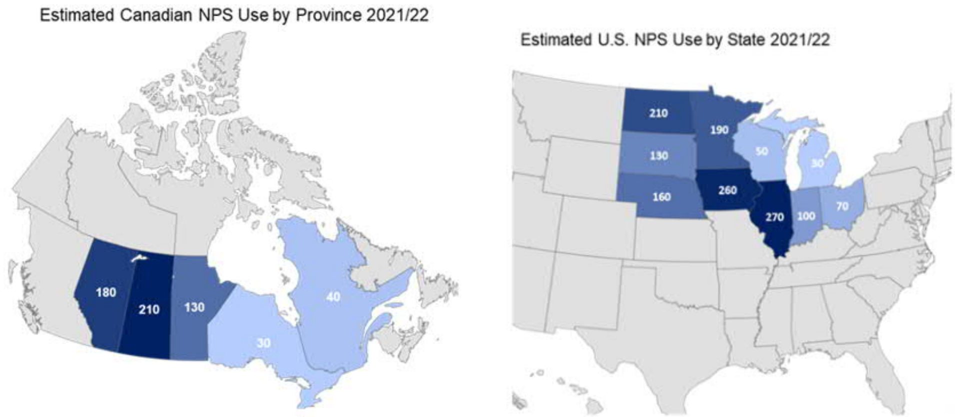


Figure 19-69: Canada and US NPS Use (2021-2022)

Table 19-31: NPS Initial Total Addressable Market and Target Volume (2021-2022)

NPS Initial Total Addressable Market and Target Volume

	Est. NPS	Fox River Resources Corp		
Tonnes	Demand 2021/22	Target Share	Target Volume	Percent of Sales
Quebec	40,000	30%	12,000	5%
Ontario	30,000	30%	9,000	4%
Manitoba	130,000	25%	32,500	13%
Saskatchewan	210,000	20%	42,000	17%
Alberta	180,000	10%	18,000	7%
Canada Total	590,000	19%	113,500	45%
North Dakota	210,000	15%	31,500	13%
South Dakota	130,000	3%	4,000	2%
Nebraska	160,000	3%	4,000	2%
Minnesota	190,000	15%	28,500	11%
Iowa	260,000	5%	13,000	5%
Wisconsin	50,000	10%	5,000	2%
Illinois	270,000	10%	27,000	11%
Michigan	30,000	15%	4,500	2%
Indiana	100,000	5%	5,000	2%
Ohio	70,000	20%	14,000	6%
USA Total	1,470,000	9%	136,500	55%
North America Total	2,060,000	12%	250,000	100%

Source: Stats Canada, TFI, USDC, TFI/AAPFCO, and MRRC

The estimated SPA use in the five leading Canadian agricultural provinces and the 10 northern tier US states are shown in Figure 19-70 and the SPA Initial TAM and target volume are detailed in Table 19-32.

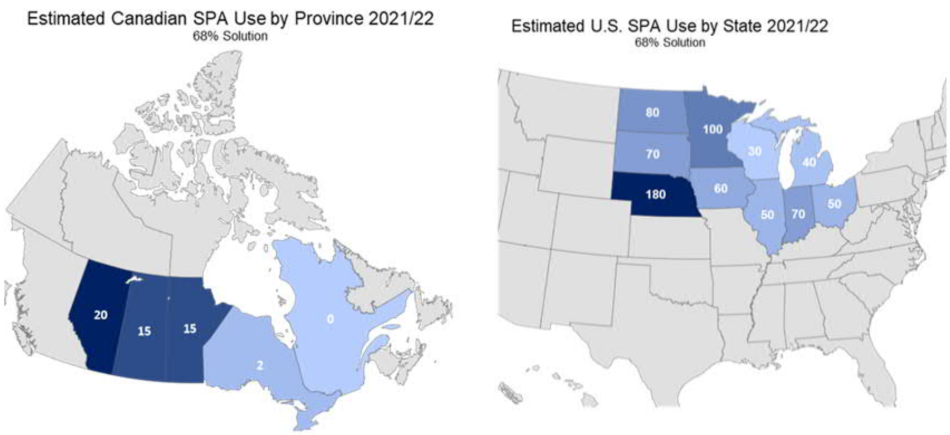


Figure 19-70: Canada and US SPA Use (2021-2022)

Table 19-32: SPA Initial Total Addressable Market and Target Volume (2021-2022)

SPA Initial Total Addressable Market and Target Volume

Tonnes 68% Solution	Est. SPA Demand 2021/22	Fox River Resources Corp Target Share	Target Volume	Percent of Sales
Quebec	0	0%	0	0%
Ontario	2,000	50%	1,000	0%
Manitoba	15,000	50%	7,500	3%
Saskatchewan	15,000	50%	7,500	3%
Alberta	20,000	25%	5,000	2%
Canada Total	52,000	40%	21,000	10%
North Dakota	80,000	30%	24,000	11%
South Dakota	70,000	20%	14,000	6%
Nebraska	180,000	18%	33,000	15%
Minnesota	100,000	35%	35,000	16%
Iowa	60,000	25%	15,000	7%
Wisconsin	30,000	35%	10,500	5%
Illinois	50,000	30%	15,000	7%
Michigan	40,000	35%	14,000	6%
Indiana	70,000	30%	21,000	10%
Ohio	50,000	35%	17,500	8%
USA Total	730,000	27%	199,000	90%
North America Total	782,000	28%	220,000	100%

Source: Stats Canada, USDC, TFI/AAPFCO, and MRRC

19.7.5 *Transportation and Logistics Outbound Cost Estimates*

Table 19-33 recaps the 2022 outbound transportation and logistics cost estimates by product for the Project. Delivery costs vary significantly by Canadian province and US state due to distance, rail car type and cost, and rail carrier service to each destination.

There are significant cost advantages to capture as much of the backyard and CN Rail freight advantaged markets as possible in Ontario, Manitoba, Quebec and Ohio. The same holds true for the second ring markets, particularly in Saskatchewan and North Dakota and other key states such as Minnesota, Illinois and Indiana.

Table 19-33: Estimated Outbound Transportation and Logistics Costs

Total Delivery Cost US$ Tonne	MAP	NPS	SPA
Manitoba	37.69	37.98	51.37
Saskatchewan	50.97	51.25	67.01
Alberta	56.37	56.95	73.89
Quebec	43.21	43.07	-
Ontario	36.24	36.10	48.70
Illinois	55.29	55.87	83.19
Indiana	54.12	54.12	80.29
Iowa	69.57	70.15	99.06
Michigan	60.57	60.57	67.98
Minnesota	55.08	55.08	71.06
Nebraska	78.09	78.67	100.96
North Dakota	46.56	46.27	78.73
Ohio	36.67	36.67	60.77
South Dakota	77.29	77.29	101.75
Wisconsin	54.79	55.66	68.73

19.7.6 *Estimated Delivery Cost Analysis – Martison vs Competitors*

Benchmarking of the target markets for both fob plant costs and delivered costs relative to the various US and Global producers is shown in Figure 19-71 by province and Figure 19-72 by state. US facilities are represented in blue and non-US facilities in red.

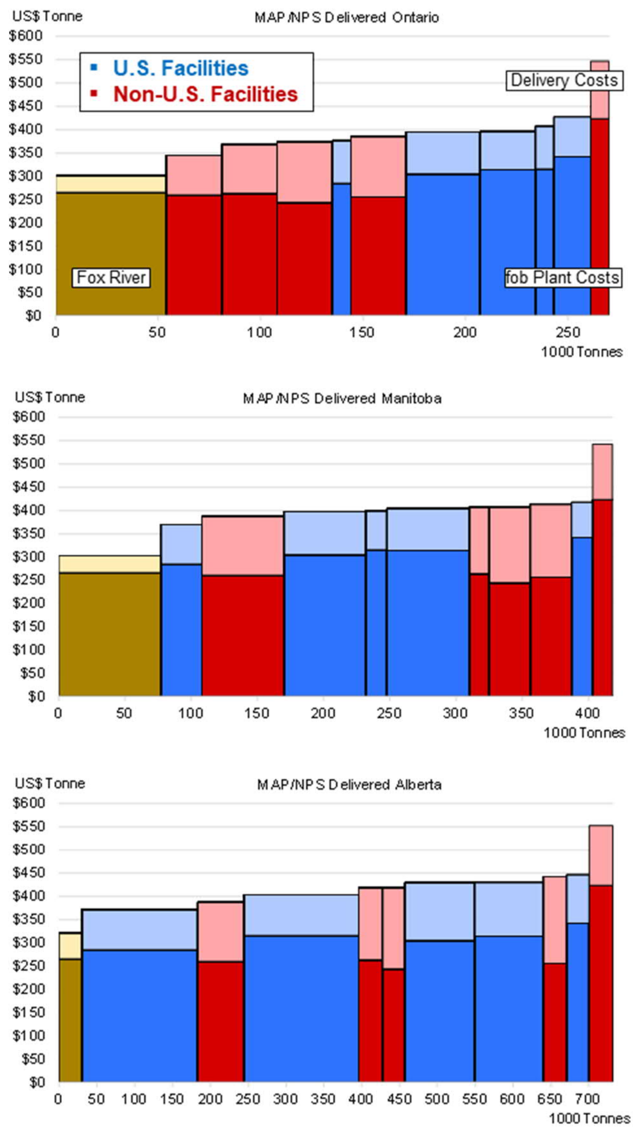
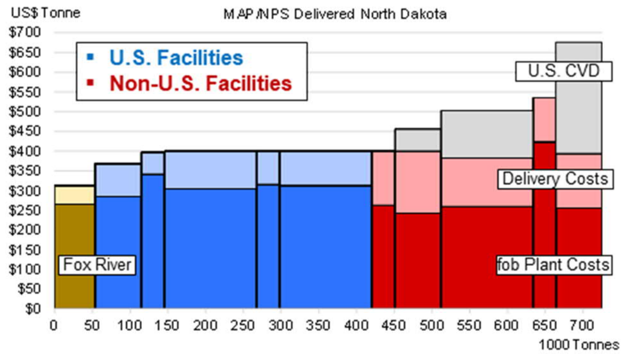
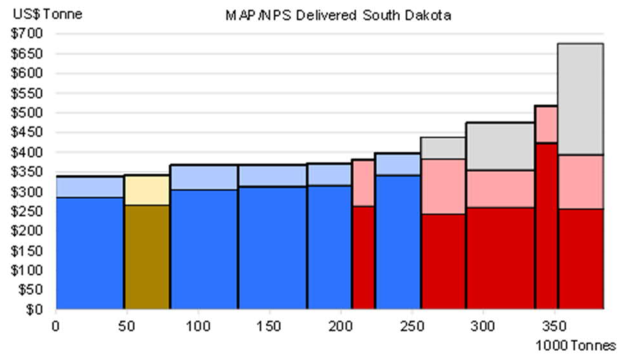
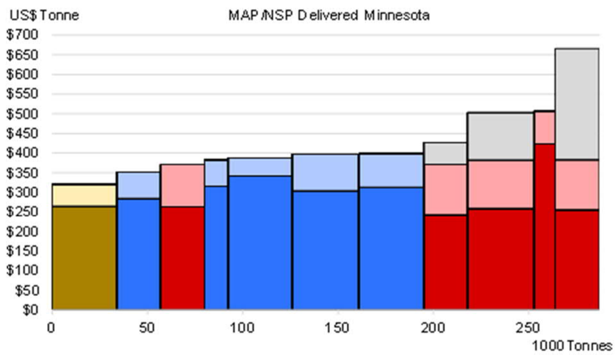
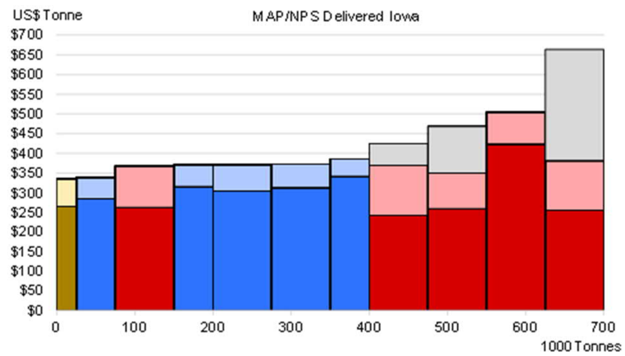


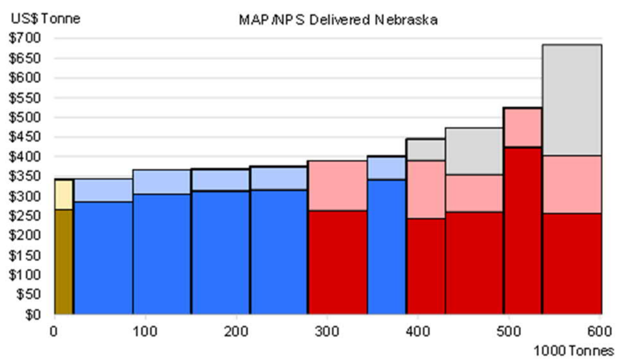
Figure 19-71: Estimated Delivered Cost Analysis by Province - Martison vs Competitors

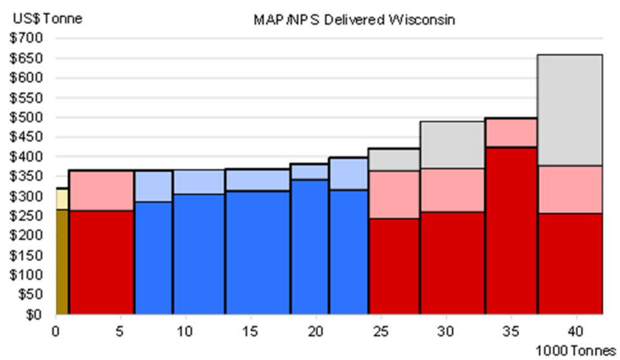

US$ Tonne
MAP/NPS Delivered North Dakota
U.S. Facilities
Non-U.S. Facilities
U.S. CVD
Delivery Costs
Fox River
fob Plant Costs
1000 Tonnes

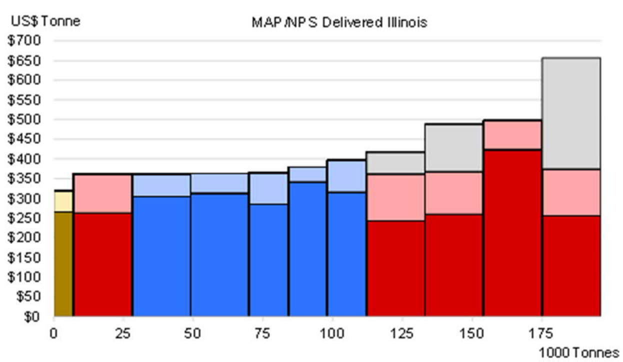

US$ Tonne
MAP/NPS Delivered South Dakota
1000 Tonnes

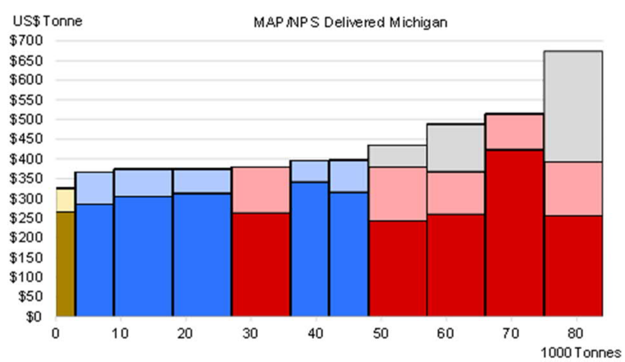

US$ Tonne
MAP/NSP Delivered Minnesota
1000 Tonnes


US$ Tonne
MAP/NPS Delivered Iowa
1000 Tonnes


US$ Tonne
MAP/NPS Delivered Nebraska
1000 Tonnes


US$ Tonne
MAP/NPS Delivered Wisconsin
1000 Tonnes


US$ Tonne
MAP/NPS Delivered Illinois
1000 Tonnes


US$ Tonne
MAP/NPS Delivered Michigan
1000 Tonnes

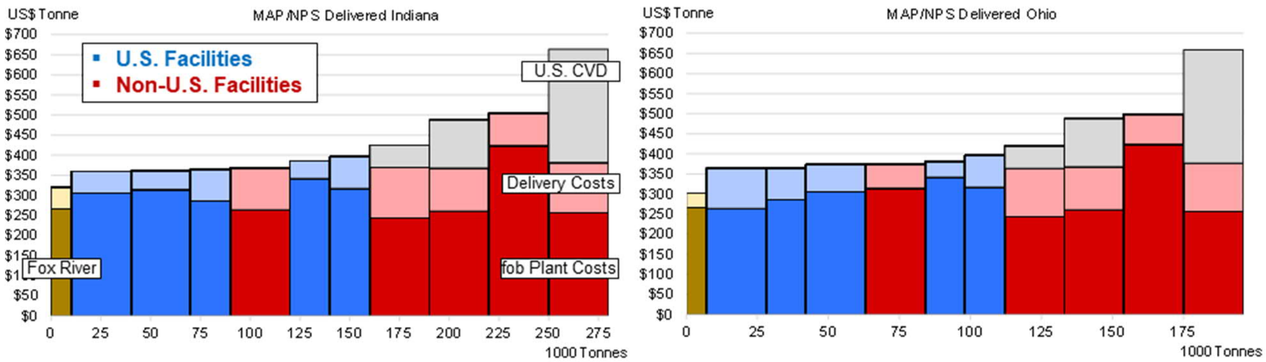


Figure 19-72: Estimated Delivered Cost Analysis by State - Martison vs Competitors

20. Environmental Baseline Studies, Permitting & Social or Community Impact

This section references the requirements from current legislation and identifies areas where further examination and details are required in the next phase of study.

20.1 Previous Work

20.1.1 Environmental Baseline Studies

Several environmental baseline studies were undertaken previously in 2008 and 2009 to support project permitting and approvals for the Project. These included the "*Baseline Biological Study, Martison Phosphate Project, Hearst Ontario*" (Golder March 2008) and "*Environmental Baseline Report, Martison Phosphate Project Mine Exploration Site*" (AMEC December 2008).

These baseline studies comprised the following subject areas:

- Cultural heritage, archaeology, Traditional Ecological Knowledge and Traditional Activities.
- Vegetation community surveys.
- Breeding bird surveys.
- Caribou habitat and movement corridors.
- Fish sampling and aquatic habitat mapping.
- Water and sediment quality sampling.

20.1.2 Cultural Heritage, Archaeology, Traditional Ecological Knowledge And Traditional Activities

Ethnohistoric documentation of CLFN use of lands in the general vicinity of the proposed mine site and associated archaeological studies (Stage 1 and 2), were carried out for the CLFN by the Mackenzie Ward Group, Professor Scott Hamilton from Lakehead University, and White Spruce Archaeology. Additional traditional land use and occupancy data were assembled by Wolverine & Associates Inc.

These studies concluded that:

- Much of the landscape in the general vicinity of the mine site are difficult to access and are generally resource poor.
- Area access by CLFN members is typically in winter through the use of snowmobile and snowshoes for the purpose of hunting and trapping.
- Fisheries values are low.
- No archaeological sites were found.

Furthermore, no culturally significant areas or archaeological sites have been identified within the vicinity of the mine site or the proposed Fushimi Road extension corridor.

20.1.3 Caribou

Baseline studies included a focus on mapping late winter caribou habitat (characterized by large lichen mats) and caribou movement corridors along the length of the proposed all-season access road. The boreal population of caribou (the forest dwelling woodland caribou) are listed as *threatened* in Ontario. In September 2009, AMEC produced a report entitled "*Caribou Winter Habitat Mapping Report, Revised Analysis*." The report concluded that the area was suitable for lichen growth although these conditions are not particularly rare in the area and, as a result, areas to the southeast and northwest may be more suitable for caribou.

20.1.4 Environmental Assessment

AMEC completed an Environmental Study Report (ESR) for the proposed all-season access road in 2008. This report is entitled "*Environmental Study Report Martison Site All-season Access Road*" (November 2008). This study was undertaken to facilitate permitting and permissions from all stakeholders towards a planned (though never subsequently undertaken) construction of the all-season access road which was originally projected to commence in late summer of 2009.

The ESR was prepared in accordance with requirements of a Category "C" Class Environmental Assessment (EA), pursuant to the Northern Development, Mines, Natural Resources and Forestry (NDMNRF) Class EA for Resource Stewardship and Facility Development Projects. The project assessed was a 40 km all season access road from the terminus of the existing Fushimi Road north to access the mine site for the purposes of advanced exploration and feasibility studies. A Statement of Completion was issued by the NDMNRF in January 2009.

20.1.5 Indigenous and Community Engagement

Constance Lake First Nation (CLFN) was engaged on the project as early as 2007 and had a significant role in determining the preferred route for the proposed access road. In addition, per the Category "C" Class EA, notices of opportunity to review the ESR were placed in local newspapers, distributed to each household on the CLFN reserve, and sent to individuals and organizations on the project mailing list. Open houses were held in Hearst and at CLFN in October 2008. Copies of the ESR were also sent to Kashechewan, Fort Albany, and Moose Cree First Nations, with a meeting held in October 2008 in Fort Albany.

20.2 Regulatory Changes and Gaps in Existing Information

The previous baseline data collected is expected to be useful to support future permitting and approvals however it will need to be updated to be in line with recent regulatory changes and to satisfy the various agencies. Specific gaps in existing information are highlighted in the following sections.

20.2.1 Environmental Assessment

The Ontario government has proposed various changes to the Environmental Assessment Act. The proposed changes include implementing a new Comprehensive EA process for major projects (formerly referred to as Individual EAs). Unlike Streamlined EAs (including Class EAs) which are completed for routine projects with predictable and manageable effects, Individual / Comprehensive EAs are prepared for large, complex projects with the potential for significant environmental effects.

Mineral development projects are not included in the proposed Comprehensive EA projects regulation. Therefore, it is not expected there will be a requirement to complete a Comprehensive EA for the Martison Phosphate Project. It should be noted however that the ESR completed for this Project in 2009 under the Class EA for Resource Stewardship and Facility Development Projects covered only the all-season access road. Supporting Project activities, such as transmission lines and on-site power generation, do trigger environmental assessments through other provincial Class EA or Streamlined EA processes. These EAs for supporting activities have not been completed for the Project and are a gap that will need to be filled. On account of each of these assessment processes having differing timelines and requirements, proponents of mining projects often opt to complete an Individual EA that captures all project and related activities for greater process certainty.

In addition, the Statement of Completion issued in 2009 for the construction of the all-season access road, the Project has passed the five year timeline specified within the NDMNRF Class EA within which work can proceed. If more than five years has elapsed, proponents may issue a Notice of Intention to Proceed for a project. The notice must describe the project and any changes that have occurred (to the project, as well as for example, to the environment, to government policies, or to technologies or engineering standards). The notice would be published in a local newspaper and sent to government agencies, and other known interested parties, for a minimum 30 day review period.

If any Indigenous person, member of the public, or other interested parties, are concerned about Treaty rights and impacts to Indigenous communities, they may make a request to the Ministry of Environment, Conservation and Parks (MECP) for an order requiring a higher level of study (for example the requirement to complete an Individual EA approval), or that other conditions be imposed such as an obligation to complete further studies.

If there have been changes to the project scope since the original Statement of Completion was issued, the project will be rescreened by NDMRNF. If the agency determines there would be an increase in potential negative effects or level of public or agency concern, a revised ESR and revised Notice of Completion will be required, with obligations for additional studies, further consultation, and public review.

20.2.2 Species at Risk permits

In 2019, jurisdiction related to species at risk in Ontario passed from NDMNRF to the MECP. In the time since the original baseline studies were completed, and the Statement of Completion was issued in 2009, significantly more caribou data has been amassed for the region, with much research led by the Ontario government in support of the *Woodland Caribou Conservation Plan*, released in 2009. The MECP normally requires significant and current data be used to support project planning and decision making. The lack of caribou data from the past 15 years for the Project area will be seen by the agency as a gap which will need to be filled prior to securing approval under the Endangered Species Act.

Since the Endangered Species Act first came into force in Ontario in 2007, Ontario Regulation 230/08 which lists specific species at risk has been modified several times, with new species added and others removed dependent upon their conservation status as assessed by the Committee on the Status of Species at Risk in Ontario (COSSARO). Several bat species (Little Brown Myotis, Northern Myotis, Eastern Small footed Myotis) were added to O. Reg 230/08 as “endangered” in 2014. In addition, the Tricoloured bat was added in 2016.

No information on bat habitat was included in the previous baseline reports and the ESR did not consider impacts to bats. This omission of bat species at risk is a gap that will need to be filled to secure approvals under the Endangered Species Act from the MECP for this project. Potential bat habitat is normally identified through ecosystem classification combined with tree snag surveys (MNRF 2015).

The 2008 baseline reports included vegetation maps for the claim boundary and along the access road. The maps identified general classes based on vegetation type and structure such as dense coniferous forest or treed wetland, however no information on forest age was included. Forest age is important for identifying bat maternity roosting habitat. The limited detail of vegetation mapping to identify potential bat habitat is considered a gap that will need to be filled to secure approvals under the Endangered Species Act from the MECP.

20.2.3 Water Crossings

Crown Land Work Permits from the NDMNRF, and approvals from Fisheries and Oceans Canada (DFO), will be required for water crossings which are impacted by the proposed all-season road and transmission line. The baseline fish, aquatics and water quality information collected in 2008 provides useful context that can be included in permitting applications, although baseline studies in 2008 sampled only a small number of lakes and rivers within the claim boundary.

The following site specific information is normally required for each crossing and is currently considered a gap that would need to be filled to support permitting and engineering detailed design:

- Flow velocity (m/s).
- Wetted and bankful width.
- Wetted and bankful depth.
- Fish habitat and fish community information.
- Foundation soil description.

20.2.4 Cultural Heritage, Archaeology, Traditional Ecological Knowledge and Traditional Activities

It is not expected that any information reviewed in these studies will have changed in regard to cultural heritage and archeology within the study area. However, in the 14 years that have passed since the assessments were completed, there have been some changes to the standards and guidelines for conducting archaeological assessments in Ontario. Pending engagement with the Ministry of Heritage, Sport, Culture and Tourism Industries (MHSCTI) it remains unclear at the time of this report whether any further work in this area will be required. Although the possibility of differing conclusions from those in 2008 is low, this status will also have to be reassessed if there is a significant divergence of the Fushimi Road extension from the route proposed at that time.

20.2.5 Indigenous Consultation and Engagement

The amount and quality of Indigenous engagement required for projects by regulatory agencies has significantly increased. Changes to both provincial and federal approvals have placed a greater emphasis on Indigenous engagement and involvement in projects. Meaningful consultation and engagement are now required both for the EA process as well as from agencies through the duty to consult for various permits and approvals. The Project will require additional consultation and engagement to support EA approvals, and to secure Provincial Crown Land Work Permits, Endangered Species Act approvals, and Fisheries and Oceans Canada (DFO) approvals.

21. Capital & Operating Costs

21.1 Capital Cost Estimate

The capital cost estimate (CAPEX) was developed to target an AACE Class 4 (preliminary) estimate accuracy range of +20% to +50% and -15% to -30%. All costs are provided in Q1 2022 US dollars, assuming a United States to Canadian currency exchange rate of 0.79365 USD/CAD. No escalation has been applied.

Included in the estimate are direct and indirect costs, contingency, and owner's costs. This estimate has included costs for all aspects of the Project and the necessary infrastructure development required to connect the facilities. The estimate which was prepared for the 2008 PFS was used as a guideline and reference document for categorization and treatment of like equipment. Less than 5% of the detail for the Project has been developed thus far, which is in line with the level of engineering definition appropriate for the AACE Class 4. New basis of estimate documents have been developed by the respective engineering teams based on the contributions to the CAPEX as follows:

- Mine site preparation – Hatch.
- Mine site infrastructure (excluding beneficiation plant) and utilities – Hatch.
- Mine access road and power transmission line – Hatch.
- Applicable mine development and mining equipment – Hatch.
- Beneficiation plant mobile equipment – Hatch.
- Tailings Management Facility (initial construction) – Hatch.
- Beneficiation plant (mine site) – JT.
- Slurry pipeline – Ausenco PSI.
- Fertilizer Conversion Complex (all areas) – JT.
- Owner's costs and other project indirects – Fox River.

This PEA includes preliminary layout drawings, process flow diagrams, equipment lists with sizing, and general specifications.

The estimate has been compiled by calculating total installed costs for each operating unit, based on line item process equipment costs, with factors applied to cover the additional costs required for related construction (foundations, support steel, piping, electrical systems) as well as installation materials and labour.

The main equipment costs were estimated using different approaches. Preliminary quotations for much of the major equipment, for example expensive items and equipment where the capacity varied significantly from available data, were obtained from vendors familiar with the equipment needed. Costs for equipment items which were not quoted directly for the PEA were obtained from historical data. Where necessary, historical figures have been adjusted for equipment size, and escalated to current day costs using published escalation rates.

The Project has been broken into multiple areas for pricing purposes. The factors applied to the equipment costs are based on historical norms and vary by area depending on the type, size, and complexity of the unit. These factors range from 1.1 (for simple utility units) to 3.8 (for the most complex processing units). With factors applied, the total installed cost for each of the equipment line items is tallied to produce the unit and overall CAPEX summaries.

In addition, preliminary quantities and costs were developed for material and installation not accounted for in the factored equipment costs. These items are typically not directly associated with a specific piece of equipment. Items that are not included in the factored equipment costs include cross country piping, conveyors, buildings, rolling stock, and roads, and have been itemized and estimated separately. Where necessary to protect against extreme cold in winter, additional enclosures or buildings have been identified and priced separately as well.

The owner's cost is estimated at 5% of direct costs. Contingency has been evaluated by a risk analysis simulation with the result rounded to USD 250 MM, or 15.5% of direct cost, plus owner's cost.

Contractor indirect allowances have been constructed from typical Hatch historical database factors and an analysis of the applicability for each indirect factor for each construction or capital purchase element.

Final assembly of all CAPEX inputs and application of other project wide allowances for the economic analysis, has been completed by JT.

The estimates assume a smooth project development without significant external factors causing delay, changes in regulation, or force majeure. The following are excluded from the CAPEX, except as can be considered part of owner's costs or operational expenses covered elsewhere:

- Sales and use taxes
- Spare parts
- Escalation

- Sunk Costs
- Consumables
- Startup costs
- Operation expenses
- Land acquisition
- Governmental fees or taxes
- Import duties
- Permits.

Capital cost summaries are provided on a unit by unit basis in the sections below.

21.1.1 Mine Site Access and Initial Site Preparation

This section includes the estimate for the initial site access and site preparation prior to the commencement of pre-production mining activities.

21.1.1.1 Access Road

Site access will be by means of an upgrade to the existing Fushimi Road followed by an extension of the road to establish an all-season route to the mine site. It has been assumed that the work will be completed by a contractor experienced at performing this form of unpaved road building in this environment. CAPEX has been estimated based on the following inputs:

- Clearing of the right of way including tree removal and grubbing.
- Backfilling of the ditch in the sections to be widened.
- Widening and resurfacing of existing Fushimi Road.
- Excavation of new ditches.
- Road expansion using aggregates.
- Establishment of several courses of aggregate layers in the new section.
- Geotextile layer on top organics on which the new road sections are built.
- Allowance for culvert installations.

This does not include the other infrastructure which will be constructed in the access road corridor (power transmission line and slurry pipeline).

21.1.1.2 Mine Site Preparation

Mine site preparation costs include the initial dewatering (draining of muskeg), clearing of trees and the removal of muskeg to the glacial till layer (where required), ahead of the development of the mining areas. This will establish the network of site roads connecting the infrastructure and prepare each area for foundation construction where buildings will be located.

Perimeter berms surrounding the key infrastructure areas will be required to redirect water ingress back into cleared areas. These will be constructed from glacial till and will be built from material being moved on the site as part of the pre-mining stripping.

Initial mine CAPEX for site access and preparation are summarized in Table 21-1 below.

Table 21-1: Mine Site Access and Initial Site Preparation CAPEX

Item	Direct CAPEX (USD MM)	Indirect Factor	Total CAPEX (USD MM)
Mine Site Access Road	20.46	1.15	23.53
Mine Site Dewatering (Muskeg Draining)	0.87	1.15	1.00
Mine Site Infrastructure and Non-Haul Roads Base Preparation	6.02	1.15	6.93
Total	**27.35**		**31.46**

21.1.2 Mining Operations

This section includes the estimate for the establishment of the open pit mining operation. This will be carried out by a contractor experienced at performing this kind of work in the environment.

21.1.2.1 Open Pit – Initial Preparation

Mining preparation will consist of the construction of dewatering wells and the requisite pumping system to direct water towards the beneficiation plant or tailings management facility, based on plant water demand. Mine preparation costing also consists of establishing the footprints for the first phase of the open pit excavation (pre-stripping of the initial box cut), and the construction of primary haul roads, waste facility, and stockpile for niobium rich material.

The pre-stripping operation's capitalized operating cost, which targets the removal and handling of glacial till during the preproduction period using contractor mining, was estimated based on a unit cost of 6.91 USD/m^3 of glacial till. This value was obtained through external benchmarking from a heavy civil and earthworks contractor experienced in this type of environment.

Mine site dewatering infrastructure requirements and costs are based on recommendations in AMEC's 2012 "Hydrogeological Critical Issues Study", which have been escalated to Q1 2022. Additional items were included and costed, including one extra well, winterization items (such as heat tracing of well heads and pipes) and pilot hole drilling for each of the wells.

Initial mine operation preparation CAPEX is summarized in the Table 21-2 below.

Table 21-2: Mine Operation Preparation CAPEX

Item	Direct CAPEX (USD MM)	Indirect Factor	Total CAPEX (USD MM)
Mine Site Water Management	11.92	1.25	14.89
Waste Facility and Pit Area Site Preparation	7.10	1.15	8.16
Haul Roads Construction	17.73	1.15	20.39
Open Pit Initial Dewatering and Pre-Stripping	39.85	1.15	45.83
Total	**76.59**		**89.27**

21.1.2.2 Owner Mobile Equipment Purchase/Leasing

Mobile equipment for use at the mine site has been divided into the following categories:

- Primary open pit production equipment.
- Mine site support and utility equipment.
- Specific equipment required in the beneficiation plant (quantities provided by JT).

Two options for mobile equipment were proposed and reviewed in the economic analysis:

1. Capital purchase option. This assumes all equipment will be purchased outright by the owner for use during the production phase of the operation. This option requires a significant up front capital commitment to establish the fleet.
2. Capital “lease” option. This option was included to test the reduction in the initial CAPEX requirement and assumes that production hauling, loading, and drilling units will be leased for the first six years of the production mine life (Years 1-6). In future years production equipment will be purchased outright by the owner (Year 7 onwards). All other mine site support and utility equipment will be purchased outright by the owner at the start of the production phase of the operation.

Unit purchasing costs per owner production and support mining equipment are based on Hatch internal benchmark data, including budgetary pricing quotes from recent projects and incorporating spare parts for initial purchase. Purchased equipment pricing is based on new units delivered to the mine site with an 8% and 4% allowance added to account for equipment assembly and transportation, respectively.

Leasing costs for the production hauling, loading, and drilling units are based on a 10% down payment of the value of the equipment, plus the cost of equipment assembly and transportation to site. Ultimately, the leasing option results in an overall higher total capital cost (initial capital plus sustaining capital), though will serve to reduce the initial upfront capital outlay.

For both options it is assumed that all vendor indirect costs have been added at source to the purchase price of the equipment. The difference in total CAPEX between the two options for mobile equipment is shown below in Table 21-3. This summarizes the initial outlay required to be able to start production ramp up (CAPEX), and then the ongoing purchase (as part of sustaining capital or SUSEX) for additional equipment and replacement of existing equipment.

Table 21-3: Mine Site Mobile Equipment CAPEX and SUSEX

Item	Initial CAPEX (USD MM)	LOM SUSEX (USD MM)	Total Capital (USD MM)
Mobile Equipment Capital Purchase Option(1)	90.24	85.27	175.51
Mobile Equipment Capital Leasing Option(1)	42.53	189.71	232.24

(1) Does not include beneficiation plant mobile equipment.

21.1.3 Mine Site Infrastructure

The estimates for infrastructure located at the mine site have been compiled from a variety of sources based on the best information available at this stage of study. With the exception of the beneficiation plant (cost estimate provided by JT), the remaining infrastructure is typical for a mine site of this type in this kind of environment. No buildings or utilities are unique and therefore costs have been assembled from the following:

- All-in approximate unit cost per square meter of building footprint for materials and construction from Hatch database benchmarks.
- Quotes from other representative projects for similar infrastructure or utilities which have been escalated where required.
- Equipment, material, and construction costs based on Hatch database for representative projects for other infrastructure categories (power transmission line for example).

Mine site infrastructure costs are summarized in Table 21-4 below.

Table 21-4: Mine Infrastructure Construction CAPEX

Item	Direct CAPEX (USD MM)	Indirect Factor	Total CAPEX (USD MM)
Power Transmission Line and Substations (Including FCC)	50.76	1.08	54.82
Site Utilities	0.83	1.25	1.04
Mine Site Buildings (Excluding beneficiation plant)	18.91	1.25	23.64
Total	**70.50**		**79.50**

The following summarizes the estimates for mine infrastructure:

- Power transmission line including a main 115kV connection to grid power and extension to the mine site, step down transformers and main substations at both FCC and mine site.
- Site utilities including potable water well (distribution and treatment), diesel fuel storage and sewage treatment.
- Mine site buildings including offices, maintenance and warehousing facilities, laboratory, core shed, explosives magazine, security, helicopter pad, fencing and communications system.
- Further refinements to the estimate through the actual building designs, and the breakdown of materials and construction labour and equipment, will be the subject of a future phase of study.

21.1.3.1 Beneficiation Plant

The beneficiation plant capital cost estimate was prepared for the following primary cost categories:

- Beneficiation plant direct and indirect cost, plus contractor's fee.
    - Factored from a priced equipment list.
- Beneficiation plant ISBL.

These items were priced from a variety of sources including vendor quotes received from local contractors and suppliers and JT database rates from recent similar projects. The table below excludes engineering, contingency, and owner's cost. Table 21-5 captures these costs.

Table 21-5: Beneficiation Plant CAPEX

Item	Direct CAPEX (USD MM)	Installation Factor	Total CAPEX (USD MM)
Beneficiation Plant Process Equipment	47.15	3.80	179.16
Beneficiation Plant Infrastructure – Building, Boiler Package, Support Buildings[1]	9.41	1.10	10.35
Total	**56.55**		**189.51**

(1) Includes site prep costs (tree clearing, muskeg removal, till fill and gravel required to construct).

21.1.4 Tailings Impoundment

The tailings impoundment capital cost was divided into two stages:

- Stage one to be completed at the time of construction operations and is considered a capital cost.
- Remaining construction for the ultimate tailings management facility is captured as a sustaining cost spread out over the first 20 years of mine life.

Table 21-6 captures these costs.

Quantities captured for the tailings management facility includes:

- Tree clearing of the total footprint.
- Removal of muskeg in the area below the dam footprint.
- Till fill and gravel required to construct the dam.
- Dam liner.
- Riprap for emergency spillways.
- Barge pumps.
- Pipelines from beneficiation plant.

These items were priced from a variety of sources including quotes received from local contractors as benchmarks and Hatch database rates from recent similar projects.

Table 21-6: Tailings Impoundment CAPEX and SUSEX

Item	Direct CAPEX (USD MM)	Indirect Factor	Total CAPEX (USD MM)
Tailings Management Facility - Stage 1 (CAPEX)	36.11	1.15	41.53
Tailings Management Facility - Stage 2 (SUSEX)	52.40	1.15	60.26
Total	**88.51**		**101.79**

21.1.5 Miscellaneous Mine Site Sustaining Costs

In addition to the sustaining capital costs which have been addressed in previous sections, the following sections provide details on other sustaining capital costs.

21.1.5.1 Waste Facility Diversion Channels

Two diversion channels will be required to divert noncontact surface runoff away from the proposed waste facility footprint to avoid any chance of contamination. These channels are assumed to be constructed in Year 15 of the mine life. These costs are considered sustaining capital costs and amount to USD4.0M.

21.1.5.2 *Progressive Site Rehabilitation*

It is assumed that progressive site rehabilitation will begin in Year 15 of the mine life until mine closure, encompassing all re-sloping and revegetation activities areas around the mine site. Rehabilitation costs have been estimated to be USD7.1M. The final closure cost has been reduced to reflect the expectation of progressive reclamation during the LOM.

21.1.5.3 *Mine Closure Costs*

Mine closure costs are applied as a sustaining capital cost in the final year of the mine life. This cost was based on the remaining site rehabilitation activities not completed during the production phase's progressive site rehabilitation efforts, as well as the cost to dismantle all mine site infrastructure and return the site to its original condition. Mine closure costs have been estimated to be USD9M.

21.1.6 *Concentrate Slurry Pipeline*

The concentrate slurry pipeline capital cost estimate was prepared for the primary cost categories in Table 21-7 below.

These items were priced from a variety of sources including vendor quotes received from contractors and suppliers and from the Ausenco PSI database of rates from recent similar projects. The pricing provided is a total installed cost by Ausenco PSI.

Table 21-7: Concentrate Slurry Pipeline CAPEX

Item	Direct CAPEX (USD MM)	Indirect Factor	Total CAPEX (USD MM)
Slurry Pipeline Pump Station(1)	26.34	1.00	26.34
Slurry Pipeline & Pressure Monitoring Stations(1)	76.40	1.00	76.40
Slurry Pipeline Terminal Station(1)	6.81	1.00	6.81
Total	**109.55**		**109.55**

(1) Includes site prep costs (tree clearing, muskeg removal, till fill and gravel required to construct).

21.1.7 *Sulfuric Acid Plant*

The sulfuric acid plant capital cost estimate was prepared for the primary cost categories in Table 21-8 below. The table excludes contingency and owner's cost.

Table 21-8: Sulfuric Acid Plant CAPEX

Item	Direct CAPEX (USD MM)	Indirect Factor	Total CAPEX (USD MM)
Chemetics Supplied Equipment	127.17	1.00	127.17
Installation of Chemetics Equipment	126.41	1.10	139.05
OSBL Equipment	2.58	3.20	8.25
Sulfuric Acid Area Infrastructure(1)	0.08	1.05	0.08
Total	**256.24**		**274.55**

(1) Includes site prep costs (tree clearing, muskeg removal, till fill and gravel required to construct).

21.1.8 Phosphoric Acid Plant

The phosphoric acid plant capital cost estimate was prepared for the primary cost categories in Table 21-9 below. The table excludes engineering, contingency, and owner's cost.

Table 21-9: Phosphoric Acid Plant CAPEX

Item	Direct CAPEX (USD MM)	Installation Factor	Total CAPEX (USD MM)
Rock Slurry Receiving	5.16	2.80	14.45
Reaction and Filtration	47.78	3.08	147.16
Gypsum Disposal Equipment (1)	1.89	2.10	3.96
Gypsum Stack & Ponds(1)	62.05	1.10	68.26
Concentration	6.39	3.50	22.38
MGA Storage	2.56	2.20	5.62
Phosphoric Acid Plant Infrastructure(1)	22.64	1.10	24.90
Total	**148.47**		**286.73**

(1) Includes site prep costs (tree clearing, muskeg removal, till fill and gravel required to construct).

21.1.9 Super Phosphoric Acid Plant

The super phosphoric acid plant capital cost estimate was prepared for the primary cost categories in Table 21-10 below. The table excludes engineering, contingency, and owner's cost.

Table 21-10: Super Phosphoric Acid Plant CAPEX

Item	Direct CAPEX (USD MM)	Installation Factor	Total CAPEX (USD MM)
54% Concentration	5.01	3.50	17.53
SPA Concentration and Reaction	16.05	3.50	56.18
SPA Storage & Shipping	5.01	2.20	11.03
SPA Plant Infrastructure(1)	3.36	1.10	3.69
Total	**29.43**		**88.43**

(1) Includes site prep costs (tree clearing, muskeg removal, till fill and gravel required to construct).

21.1.10 Granulation Plant

The granulation plant capital cost estimate was prepared for the primary cost categories in Table 21-11 below. The table excludes engineering, contingency, and owner's cost.

Table 21-11: Granulation Plant CAPEX

Item	Direct CAPEX (USD MM)	Indirect Factor	Total CAPEX (USD MM)
Raw Material Storage and Shipping	10.27	2.75	28.25
Sulfur Addition Package	1.85	2.75	5.08
Granulation Plant ISBL	23.17	3.30	76.45
Product Storage & Shipping	6.92	2.20	15.22
Building & Infrastructure(1)	30.35	1.10	33.39
Total	**72.56**		**158.39**

(1) Includes site prep costs (tree clearing, muskeg removal, till fill and gravel required to construct).

21.1.11 Infrastructure – Process Support

The infrastructure – process support capital cost estimate was prepared for the categories in Table 21-12. The table excludes engineering, contingency, and owner's cost.

Table 21-12: Infrastructure - Process Support CAPEX

Item	Direct CAPEX (USD MM)	Indirect Factor	Total CAPEX (USD MM)
Common Process Equipment for FCC	8.07	2.50	20.17
Plant Mobile Equipment	1.70	1.05	1.75
Common Infrastructure(1)	40.66	1.10	44.73
Railroad Works	25.32	1.05	26.59
Total	**75.73**		**93.25**

(1) Includes site prep costs (tree clearing, muskeg removal, till fill and gravel required to construct).

21.2 Operating Cost Estimate

Operating costs during the production phase of the project are provided in Table 21-13 and are based on individual operating unit costs, and accounting in each case for the cost of upstream intermediates and raw materials.

All costs are provided in Q1 2022 US dollars, assuming a United States dollar to Canadian dollar exchange rate of 0.79365 USD/CAD.

Table 21-13: Total and Unit Operating Cost Summary

Item	Total LOM OPEX (USD MM)	Average LOM OPEX (USD MM per year)	Unit OPEX	Unit OPEX Reported As
Mining Operations	1,055.01	40.58	$12.62	USD/t of Mill Feed
Mine Dewatering	6.88	0.27	$0.08	USD/t of Mill Feed
Mine Site Preparation	54.73	2.11	$0.65	USD/t of Mill Feed
Beneficiation Plant	540.02	20.77	$6.44	USD/t of Mill Feed
Concentrate Slurry Pipeline	40.04	1.54	$1.14	USD/t of Concentrate
Sulfuric Acid Plant(2)	3,176.15	122.16	$99.55	USD/t of H_2SO_4
Phosphoric Acid Plant	5,499.26	211.51	$423.02	USD/t of P_2O_5
Infrastructure – Process Support(3)	263.07	10.12	$21.05	USD/t of P_2O_5
Super Phosphoric Acid Plant(1)	2,178.69	83.80	$395.16	USD/t of SPA Solution
Granulation Plant (MAP)(2)	3,777.70	145.29	$319.10	USD/t MAP
Granulation Plant (NPS)(2)	1,985.80	76.38	$321.34	USD/t NPS

(1) SPA solution (68% P_2O_5).
(2) Base case prices: sulfur price of $274/t delivered, and ammonia price of $602/t delivered.
(3) Excludes: Electrical usage for FCC Heating is 12.92 kWh/t of P_2O_5 over the LOM.
(4) Each of the listed operating costs includes the relevant cost of its feedstock.
(5) Phosphoric acid unit cost $/t P_2O_5 reported on 500,000 t/yr basis.

21.2.1 Labour Cost Basis

The following Table 21-14 and Table 21-15 represent the labour costs used at both the mine site and the FCC for salaried and hourly personnel. The rates used include base salary and burdens and are comparable to other current local projects in the area.

Table 21-14: Labour Cost Summary (Salary)

Job Title	Annual Gross Salary (USD)
General Manager	$226,000
Manager	$171,000
Superintendent – Mine	$160,000
Superintendent – FCC	$145,000
Senior Engineer	$136,000
Controller	$164,000
Operations Supervisor – Mine	$127,000
Operations Supervisor – FCC	$121,000
Engineer	$121,000
Senior Geologist	$121,000
Maintenance Supervisor	$110,000
Maintenance Planner	$95,000
Senior Assayer	$130,000
Human Resources	$110,000
Environmental Engineer	$90,000
Environmental Technician	$86,000
Surveyor	$86,000
Warehouse Lead	$80,000
Geologist	$80,000
Dispatch Technician	$80,000
Operations Coordinator	$86,000
Buyer	$71,000
Assayer	$60,000
General Labourer	$80,000
Mine Technician	$75,000
Laboratory Assistant	$65,000
Clerk	$60,000
Administrative Assistant	$95,000

Table 21-15: Mine Site Labour Cost Summary (Hourly)

Job Title	Gross Hourly Wage (USD)
Truck Operators	$39.40
Shovel Operators	$54.10
Loader Operators	$49.20
Blasters & Surface Crew	$49.20
Dozer Operators	$49.20
Grader Operators	$49.20
Miscellaneous Heavy Equipment Operators	$49.20

21.2.2 Mining Operations

As detailed in Table 21-16, average mine operating costs are estimated to be USD2.42 per tonne of wet material mined, or USD13.15 per tonne of mill feed over the LOM. All owner-operated costs during the production phase of the Project includes all requisite mining activities, equipment, and labour necessary to excavate, handle and place mill feed, niobium-rich and waste materials around the mine site.

Table 21-16: Mine Operating Cost Estimate

Item	Total - LOM (USD MM)	Unit Cost (USD/t Mined - Wet)	Unit Cost (USD/t of Mill Feed)
Drilling	25.71	$0.06	$0.31
Blasting	13.33	$0.03	$0.16
Loading	93.60	$0.21	$1.12
Hauling	397.82	$0.87	$4.76
Support Equipment	211.39	$0.46	$2.53
Sub-Total (Equipment Costs)	**741.84**	**$1.63**	**$8.87**
Mine/Maintenance Hourly Labour	281.55	$0.62	$3.37
Mine/Maintenance Staff Labour	75.87	$0.17	$0.91
Sub-Total (Labour Costs)	**357.42**	**$0.79**	**$4.28**
Total Mine OPEX	**1,099.26**	**$2.42**	**$13.15**

Preliminary equipment productivities and haulage cycle times were generated and applied against the annual production quantities to estimate equipment operating hours. Consumption rates for key consumables (i.e., diesel and lubricants) and unit operating costs were applied to the equipment hours to calculate the total equipment operating costs for each period. The cost of parts and repairs (major component and minor repairs) are included in the operating costs for all owner-operated mining equipment.

Explosive quantities were calculated using an estimated powder factor derived from recent, similar projects using light blasting for consolidated and/or frozen materials. Hatch utilized internal benchmark costing data from recent projects to cost drilling, explosives, and accessories unit costs to estimate drilling and blasting consumable costs. Mine operations and mobile equipment labour and staffing requirements were estimated, and wages/salaries applied on a yearly basis to estimate total mine labour costs.

21.2.3 Miscellaneous Mine Site Operating Expenses

Additional mine site operating expenses not directly related to mining operations include the following sections. These costs are summarized in Table 21-17 below.

Table 21-17: Miscellaneous Mine Site Operating Expenses

Operating Expense	Total Yearly Average – LOM (USD MM)	Unit Cost (USD/t of Mill Feed)
Mine Dewatering & Site Preparation	2.37	$0.74
Sitewide Power Consumption(1)	8.51	$2.65
Misc. Expenses (Snow Removal, Comms)	2.72	$0.85
O&M Supplies	24.06	$7.48
Beneficiation Plant Reagents	10.14	$3.15
Sub-Total (Variable Costs)	**47.80**	**$14.87**
Sitewide Labour(1)	17.68	$5.50
Local Fees and Insurance	3.24	$1.01
General Supplies	0.88	$0.27
Sub-Total (Fixed Costs)	**21.81**	**$6.78**
Total Miscellaneous Mine Site OPEX(1)	**69.61**	**$21.65**

(1) Includes beneficiation plant and concentrate slurry pipeline.

21.2.3.1 Mine Dewatering

Mine dewatering operating costs are summarized as:

- Pipeline operating and maintenance costs.
- Additional in-pit sump pump and standby sump pumps.
- Water treatment operating and maintenance costs.

These costs were captured based on the estimate and recommendations made in AMEC's 2012 "Hydrogeological Critical Issues Study" and escalated to 2022 dollars.

21.2.3.2 *Site Preparation*

The remaining site preparation costs not captured during the project period include tree clearing of the open pit and the waste facility areas, as well as the progressive stripping of the muskeg on top of the open pit. These costs are considered an operating expense with these activities being performed on a yearly basis until all areas have been cleared for operation.

21.2.3.3 *Mine Site Infrastructure*

Operating costs for mine site infrastructure include costs that are required to operate the mine site daily. These include power consumption costs for all buildings, operating and maintenance supply costs (not already captured for the mobile mining equipment), general supply costs, and miscellaneous expenses such as snow removal and communication tower service fees.

21.2.3.3.1 Beneficiation Plant

The average beneficiation plant operating costs are estimated to be USD18.90 per tonne of concentrate produced over the LOM. All owner-operated costs during the production phase of the Project include the requisite mill feed processing activities, equipment, and labour to handle and process the mill feed, produced slurry, and waste streams generated from the beneficiation process. Table 21-18 details the beneficiation operating cost estimate.

Table 21-18: Beneficiation Plant Operating Cost Estimate

Item	Total - LOM (USD MM)	Unit Cost (USD/t of Concentrate)
Reagents	10.14	$7.47
Electric Power(1)	5.97	$4.40
O & M Supplies + Contract Maintenance	5.10	$3.76
Sub-Total (Variable Costs)	**21.21**	**$15.63**
Labour	3.23	$2.45
Local Fees & Insurance	0.95	$0.70
General Supplies	0.17	$0.12
Sub-Total (Fixed Costs)	**4.44**	**$3.27**
Total Beneficiation OPEX(1)	**25.65**	**$18.90**

(1)Assumes all power is purchased. About 83% of the power consumed at the mine site is provided by the FCC cogenerator (through offsets credits into the local grid) and consequently the net power cost is reduced to about USD0.75/t of concentrate and the total Beneficiation OPEX is reduced to USD15.25/t of concentrate.

Reagent use was based on a process material balance and usage rates from column cell flotation tests of a residuum sample from the Martison deposit. Estimated power is derived from a sized equipment list.

O & M supplies were estimated, while contract maintenance, and local fees and insurance costs were factored from the total CAPEX for the beneficiation plant. General supplies are factored from the total labour for the beneficiation plant.

Salaried and hourly labour rates are provided by Hatch and based on internal data sources. The labour rates were applied to the estimated beneficiation plant workforce to determine the total hourly labour cost. Salaries were applied to the total staff estimate to arrive at the salaried cost.

21.2.4 Concentrate Slurry Pipeline

Average concentrate slurry pipeline operating costs are estimated to be USD1.14 per tonne of concentrate over the LOM. All owner-operated costs during the production phase of the Project includes the requisite activities, equipment, and labour to handle and transfer the phosphate slurry generated from the beneficiation process. Table 21-19 details the slurry pipeline operating cost estimate.

Table 21-19: Concentrate Slurry Pipeline Operating Cost Estimate

Item	Total Yearly Average - LOM (USD MM)	Unit Cost (USD/t of Concentrate)
Electric Power(1)	1.55	$1.14
O & M Supplies + Contract Maintenance	1.03	$0.76
Sub-Total (Variable Costs)	**2.58**	**$1.90**
Labour(2)	-	-
Local Fees & Insurance	0.25	$0.18
General Supplies(2)	-	-
Sub-Total (Fixed Costs)	**0.25**	**$0.18**
Total Concentrate Slurry Pipeline OPEX(1)	**2.83**	**$2.09**

(1) Assumes all power is purchased. About 83% of the power consumed at the mine site is provided by the FCC co-generator (through offset credits into the local grid) and consequently the net power cost is reduced to about USD0.20/t of concentrate and the total concentrate slurry pipeline OPEX is reduced to USD1.14/t of concentrate.
(2) Labour and general supplies are included with the beneficiation plant OPEX.

Estimated power is derived from a sized equipment list and provided by AUSENCO.

O & M supplies were estimated, while contract maintenance, and local fees and insurance costs were factored from the total CAPEX for the concentrate slurry pipeline.

21.2.5 Sulfuric Acid Plant

Average sulfuric acid plant operating costs are estimated to be USD 99.55 per tonne of H_2SO_4 over the LOM. All owner-operated costs during the production phase of the Project include all requisite processing activities, equipment, and labour to produce the sulfuric acid, steam and power for the FCC. Table 21-20 details the sulfuric acid operating cost estimate.

Table 21-20: Sulfuric Acid Plant Operating Cost Estimate

Item	Total Yearly Average LOM (USD MM)	Unit Cost (USD/t of H_2SO_4)
Sulfur[1]	114.21	$93.07
Electric Power[2]	-	-
O & M Supplies + Contract Maintenance	4.12	$3.36
Sub-Total (Variable Costs)	**118.33**	**$96.43**
Labour	1.69	$1.38
Local Fees & Insurance	2.06	$1.68
General Supplies	0.09	$0.07
Sub-Total (Fixed Costs)	**3.83**	**$3.13**
Total Sulfuric Acid Plant OPEX	**122.16[3]**	**$99.56[3]**

(1) Base case sulfur price of $274/t delivered.
(2) Assumes all power is provided by the FCC cogeneration within the battery limit of the sulfuric acid plant. Sulfuric acid usage is 53.3 kWh/t of H_2SO_4 over the LOM. Total sulfuric acid yearly average is 65,407 MWh over the LOM.
(3) Excludes SG&A.

Sulfur consumption is calculated using a process material balance. Estimated power in derived from a sized equipment list.

O & M supplies were estimated, while contract maintenance, and local fees and insurance costs were factored from the total CAPEX for the sulfuric acid plant. General supplies are factored from the total labour for the sulfuric acid plant.

Salaried and hourly labour rates are provided by Hatch and based on internal data sources. The labour rates were applied to the estimated sulfuric acid plant workforce to determine the total hourly labour cost. Salaries were applied to the total staff estimate to arrive at the salaried cost.

21.2.6 Phosphoric Acid Plant

Average phosphoric acid plant operating costs are estimated to be USD440.05 per tonne of P_2O_5 over the LOM. All owner-operated costs during the production phase of the Project includes all requisite processing activities, equipment, and labour to produce, filter, and concentrate the phosphoric acid and the associated waste streams generated. Table 21-21 details the phosphoric acid operating cost estimate.

Table 21-21: Phosphoric Acid Plant Operating Cost Estimate

Item	Total Yearly Average LOM (USD MM)	Unit Cost (USD/t of P_2O_5)
Reagents	1.01	$2.10
Defoamer	1.44	$3.00
Gypsum Disposal	2.40	$5.00
Electric Power(1)	-	-
O & M Supplies + Contract Maintenance	6.55	$13.64
Sub-Total (Variable Costs)	**11.41(2)**	**$23.74(3)**
Labour	2.38	$4.96
Local Fees & Insurance	1.64	$3.41
General Supplies	0.12	$0.25
Sub-Total (Fixed Costs)	**4.14**	**$8.62**
Total Phosphoric Acid OPEX	**15.55(2,4)**	**$32.35(3,4)**

(1) Assumes all power is provided by the FCC cogeneration. Phosphoric acid usage is 91.1 kWh/t of P_2O_5 over the LOM. Total phosphoric acid yearly average is 43,772 MWh over the LOM.

(2) Not shown, carry forward cost for phosphate concentrate feedstock is USD76.33M and USD119.63M for sulfuric acid for total phosphoric acid OPEX of USD211.51M LOM yearly average.

(3) Not shown, carry forward costs for phosphate concentrate feedstock is USD158.81/t P_2O_5 and USD248.89/t P_2O_5 for sulfuric acid for total unit production rate of USD440.05/t P_2O_5 over LOM.

(4) Excludes S&GA.

Reagent and defoamer use were calculated using a process material balance and usage rates from similar phosphoric acid projects and test work of residuum samples from the Martison deposit. Gypsum disposal is calculated from similar phosphoric acid projects. Estimated power listed in Note 2 derived from a sized equipment list.

O & M supplies were estimated, while contract maintenance, and local fees and insurance costs were factored from the total CAPEX for the phosphoric acid plant. General supplies are factored from the total labour for the phosphoric acid plant.

Salaried and hourly labour rates are provided by Hatch and based on internal data sources. The labour rates were applied to the estimated phosphoric acid plant workforce to determine the total hourly labour cost. Salaries were applied to the total staff estimate to arrive at the salaried cost.

21.2.7 Super Phosphoric Acid Plant

Average super phosphoric acid plant operating costs are estimated to be USD395.16 per tonne of SPA solution (68% P_2O_5) over the LOM. All owner-operated costs during the production phase of the Project includes all requisite processing activities, equipment, and labour to produce, filter, and concentrate the super phosphoric acid. Table 21-22 details the super phosphoric acid operating cost estimate.

Table 21-22: Super Phosphoric Acid Plant Operating Cost Estimate

Item	Total Yearly Average – LOM (USD MM)	Unit Cost (USD/t of SPA Solution)
Reagents	0.84	3.95
Electric Power(1)	-	-
O & M Supplies + Contract Maintenance	1.33	$6.26
Sub-Total (Variable Costs)	**2.16(2)**	**$10.21(3)**
Labour	1.81	$8.55
Local Fees & Insurance	0.66	$3.13
General Supplies	0.09	$0.43
Sub-Total (Fixed Costs)	**2.57**	**$12.10**
Total Super Phosphoric Acid OPEX	**4.73(2,4)**	**$22.31(3,4)**

(1) Assumes all power is provided by the FCC cogeneration. Super phosphoric acid usage is 75.0 kWh/t of SPA solution over the LOM. Total super phosphoric acid yearly average is 10,815 MWh over the LOM.
(2) Not shown, carry forward cost for phosphoric acid feedstock is USD79.10M for total super phosphoric acid OPEX of USD83.80M LOM yearly average.
(3) Not shown, carry forward costs for phosphoric acid feedstock is USD372.86t SPA or total unit rate of USD395.16/t of SPA (solution) over LOM.
(4) Excludes S&GA.

Reagent use was calculated using a process material balance and usage rates from similar super phosphoric acid projects Reagents for the super phosphoric acid process includes filter aid, ammonium nitrate, and iron. Estimated power derived from a sized equipment list.

O & M supplies were estimated, while contract maintenance, and local fees and insurance costs were factored from the total CAPEX for the super phosphoric acid plant. General supplies are factored from the total labour for the super phosphoric acid plant.

Salaried and hourly labour rates are provided by Hatch and based on internal data sources. The labour rates were applied to the estimated super phosphoric acid plant workforce to determine the total hourly labour cost. Salaries were applied to the total staff estimate to arrive at the salaried cost.

21.2.8 Granulation Plant

Average granulation plant operating costs are estimated to be USD319.10 per tonne of MAP and USD321.34 per tonne of NPS over the LOM. All owner-operated costs during the production phase of the Project includes all requisite processing activities, equipment, and labour to produce, cool, screen, coat, store, and ship the granulation fertilizer products. Table 21-23 details the granulation operating cost estimate.

Table 21-23: Granulation Plant Operating Cost Estimate

Item	MAP Total Yearly Average – LOM (USD MM)	MAP Unit Cost (USD/t)	NPS Total Yearly Average – LOM (USD MM)	NPS Unit Cost (USD/t)
Fuel (Natural Gas)	0.87	$1.90	0.49	$2.07
Sulfur(1)	-	-	3.26	$13.70
Zinc Sulfate	-	-	5.70	$24.00
Defoamer	0.28	$0.61	0.08	$0.35
Ammonia(1)	36.46	$80.07	20.89	$87.89
Other(2)	2.34	$5.15	0.85	$3.58
Electric Power(3)	-	-	-	-
O & M Supplies + Contract Maintenance	1.04	$2.29	0.54	$2.29
Sub-Total (Variable Costs)	**40.98(4)**	**$90.01(5)**	**31.82(4)**	**$133.87(5)**
Labour	1.62	$3.56	0.85	$3.56
Local Fees & Insurance	0.78	$1.71	0.41	$1.71
General Supplies	0.08	$0.18	0.04	$0.18
Sub-Total (Fixed Costs)	**2.48**	**$5.45**	**1.30**	**$5.45**
Total Granulation OPEX	**43.46(4,6)**	**$95.45(5,6)**	**33.11(4,6)**	**$139.32(5,6)**

(1) Base case prices: sulfur price of $274/t delivered, and ammonia price of $602/t delivered.
(2) Other costs include coating oil.
(3) Assumes all power is provided by the FCC cogeneration. Granulation usage is 37.1 kWh/t of MAP over the LOM. Total granulation yearly average is 16,893 MWh for MAP. Granulation usage is 37.1 kWh/t of NPS over the LOM. Total granulation yearly average is 8,818 MWh for NPS.
(4) Not shown, carry forward cost for phosphoric acid feedstock is USD101.83M for total MAP OPEX of USD145.29M LOM yearly average. Not shown, carry forward cost for phosphoric acid feedstock is USD40.73M and USD2.53M for sulfuric acid for total NPS OPEX of USD76.38M LOM yearly average.
(5) Not shown, carry forward costs for phosphoric acid feedstock is USD223.64/t MAP for total MAP unit production cost of USD319.10/t MAP over LOM. Not shown, carry forward costs for phosphoric acid feedstock is USD171.37/t NPS and USD10.65/t NPS for sulfuric acid for total NPS unit production cost of USD321.34/t NPS over LOM.
(6) Excludes S&GA.

All components of the variable costs (natural gas, defoamer, etc.) were calculated using a process material balance and usage rates from similar granulation projects. Estimated power listed in Note 3 derived from a sized equipment list.

O & M supplies were estimated, while contract maintenance, and local fees and insurance costs were factored from the total CAPEX for the granulation plant. General supplies are factored from the total labour for the granulation plant.

Salaried and hourly labour rates are provided by Hatch and based on internal data sources. The labour rates were applied to the estimated granulation plant workforce to determine the total hourly labour cost. Salaries were applied to the total staff estimate to arrive at the salaried cost.

21.2.9 *Infrastructure – Process Support*

Average Infrastructure – Process Support operating costs are estimated to be USD21.05 per tonne of P_2O_5 over the LOM. All owner-operated costs during the production phase of the Project includes all requisite processing activities, equipment, and labour to provide process water, treated water, instrument air, sewage and wastewater treatment, mobile equipment, and other needed services for the FCC to operate. Table 21-24 details the Infrastructure – Process Support operating cost estimate.

Table 21-24: Infrastructure - Process Support Operating Cost Estimate

Item	Total Yearly Average– LOM (USD MM)	Unit Cost (USD/t of P_2O_5)
Fuel	0.13	$0.27
Reagents(1)	0.05	$0.11
Other(2)	0.21	$0.45
Electric Power(3,4)	-	-
O & M Supplies + Contract Maintenance	1.87	$3.88
Sub-Total (Variable Costs)	**2.26**	**$4.70**
Labour	6.82	$14.19
Local Fees & Insurance	0.70	$1.46
General Supplies	0.34	$0.71
Sub-Total (Fixed Costs)	**7.86**	**$16.35**
Total Infrastructure – Process Support OPEX	**10.12**	**$21.05**

(1) Water treatment expenses for RO and UF cleaning.
(2) Includes non-fuel mobile equipment operating costs.
(3) Assumes all power is provided by the FCC cogeneration. Infrastructure – Process Support usage is 16.5 kWh/t of P_2O_5 over the LOM. Infrastructure – Process Support yearly average is 7,764 MWh over the LOM.
(4) Electrical usage for FCC Heating is 12.92 kWh/t of P_2O_5 over the LOM. Infrastructure – Process Support yearly average is 6,212 MWh over the LOM.

Fuel estimate was supplied by AECOM. Estimated power listed in Note 3 derived from a sized equipment list.

O & M supplies were estimated, while contract maintenance, and local fees and insurance costs were factored from the total CAPEX for infrastructure – process support. General supplies are factored from the total labour for infrastructure – process support.

Salaried and hourly labour rates are provided by Hatch and based on internal data sources. The labour rates were applied to the estimated infrastructure – process support workforce to determine the total hourly labour cost. Salaries were applied to the total staff estimate to arrive at the salaried cost.

22. Economic Analysis

22.1 Introduction

The economic assessment for the PEA is preliminary in nature, includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The economic model provides financial calculations to determine the value of the project in terms of Net Present Value (NPV), Internal Rate of Return (IRR), payback period, and cash flow. The NPV is an evaluation of all the positive and negative cash flows over the life of the project, and accounting for the time value of money, determines the present value of the capital project. The NPV is calculated at a specific discount rate, and if the result is greater than zero, indicates that an investment adds value at that rate.

The IRR is a means of evaluating the potential return on an investment. The IRR is the rate of return required to result in a net present value of zero for all the cash flows. Higher IRRs generally indicate better investments. The payback period is a measure of the time required for the cumulative cash flows to equal the initial investment as measured from the start of production. This is considered a measure of investment risk rather than return.

The economic model was developed specifically for evaluation of this project scope on a fully funded project basis. The preliminary economic analysis uses a cash flow model at a base sulfur price of USD274/t, a base ammonia price of USD602/t and an 8% discount rate.

The financial assessment was carried out on a 100% equity basis without the inclusion of debt or other funding. Inflationary effects were not applied in the assessment. Canadian Federal tax and Ontario Provincial regulations were applied to assess the tax liabilities.

The economic model examines the capital investments, operating costs, and related factors over the life of mine, and provides results based on market analysis for the cost of key raw materials and value of the finished product. Net present value (NPV), internal rate of return (IRR) and payback period are calculated in the model and reported before and after taxes.

22.2 Returns Summary

The economic model was used to examine variations around a specific Base Case of product prices and raw material costs as defined in Section 19. Table 22-1 and Table 22-2 below indicate the sensitivity of net present value to variations pre-tax and after-tax basis, where the Base Case prices are varied collectively, reflecting the strong historical correlation between the raw material fertilizer prices.

Table 22-3 provides the sensitivity of IRR, payback, and LOP Cash Flow to the same variations around the base case.

Table 22-1: Pre-Tax NPV8% Sensitivity Analysis(1)

	30% Below Base Case	15% Below Base Case	Base Case	15% Above Base Case	30% Above Base Case
MAP Price	$560	$680	$800	$920	$1,040
SPA (68% P_2O_5) Price	$742	$901	$1,060	$1,219	$1,378
NPS Price	$567	$689	$810	$932	$1053
Sulfur Price	$192	$233	$274	$315	$356
Ammonia Price	$421	$512	$602	$692	$783
Pre-Tax $NPV_{8\%}$ (USDM)	**$457**	**$1,300**	**$2,144**	**$2,987**	**$3,830**

¹ Please see "Notes to Tables 22.1 to 22.3" below Table 22.4 for all assumptions.

Table 22-2: After-Tax NPV8% Sensitivity Analysis(1)

	30% Below Base Case	15% Below Base Case	Base Case	15% Above Base Case	30% Above Base Case
MAP Price	$560	$680	$800	$920	$1,040
SPA (68% P_2O_5) Price	$742	$901	$1,060	$1,219	$1,378
NPS Price	$567	$689	$810	$932	$1053
Sulfur Price	$192	$233	$274	$315	$356
Ammonia Price	$421	$512	$602	$692	$783
After-Tax $NPV_{8\%}$ (USDM)	**$184**	**$831**	**$1,467**	**$2,104**	**$2,737**

¹ Please see "Notes to Tables 22.1 to 22.3" below Table 22.4 for all assumptions.

Table 22-3: IRR, Payback and LOP Cash Flow Sensitivity Analysis (1)

	30% Below Base Case	15% Below Base Case	Base Case	15% Above Base Case	30% Above Base Case
MAP Price	$560	$680	$800	$920	$1,040
SPA (68% P_2O_5) Price	$742	$901	$1,060	$1,219	$1,378
NPS Price	$567	$689	$810	$932	$1053
Pre-Tax IRR	**10.9%**	**15.8%**	**20.2%**	**24.3%**	**28.2%**
After-Tax IRR	**9.3%**	**13.6%**	**17.4%**	**20.9%**	**24.2%**
After-Tax Payback (years)	**8.6**	**6.4**	**5.2**	**4.4**	**3.8**
Cumulative Cash Flow (USDM)	**$2,911**	**$4,685**	**$6,460**	**$8,235**	**$10,010**

Notes to Tables 22.1 to 22.3:

1. All results developed at an exchange rate of 0.79365 USD/CAD for the CAPEX and OPEX calculations. Product prices are in US Dollars per metric tonne.
2. The "Base Case" is a weighted average of three market forecast scenarios for the years 2022 to 2047.
3. Estimated cumulative cash flow over the life of Project, including a tax recovery in year 27. Cumulative cash flow through life of mine (year 26) is estimated at USDM6,443.

Figure 22-1 provides the sensitivity of the internal rate of return to changes in several of the key inputs to the economic analysis. Each of the key inputs is evaluated over a range of -30% to +30%. Increases in CAPEX, sulfur, and ammonia price have a negative effect on IRR, while increases in the price of sold products result in favorable increases in the expected IRR. As noted above, changes in many of these inputs are not mutually independent in the markets, however the chart indicates how strongly each individual input affects the result.

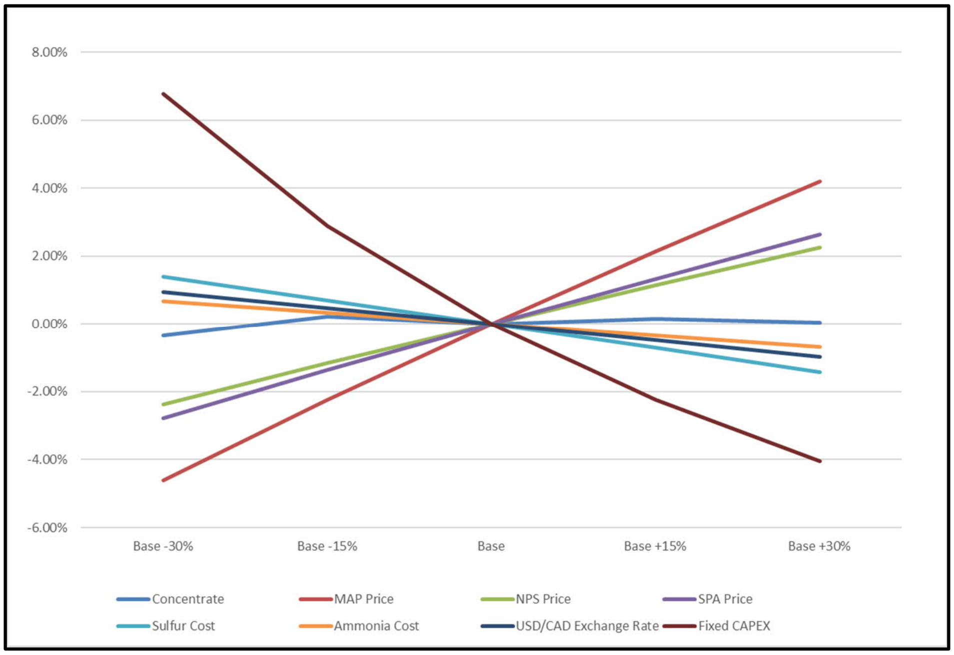


Figure 22-1: Sensitivity Analysis

22.3 PEA Base Case

22.3.1 Base Case Assumptions & Pricing

Key information for the Base Case economic analysis is provided below in Table 22-4.

Table 22-4: Key Assumptions, Parameters, and Outputs

Description	Units	Amount
Product Prices / Input Costs / FX		**Base Case[1]**
Product Prices		
Mono Ammonium Phosphate (MAP)[2]	USD/t DEL	$800
Super Phosphoric Acid 68% P_2O_5 (SPA)[3]	USD /t DEL	$1,060
Nitrogen, Phosphate, Sulfur (NPS)[4]	USD /t DEL	$810
Input Costs		
Sulfur[5]	USD/t DEL	$274
Ammonia[6]	USD/t DEL	$602
Currency Exchange Rate	USD/CAD	0.79365
Production Data		
Mine Site		
Total Tonnes Mined, Life of Mine Plan	Mt/Dry	409.48
Beneficiation Mill Feed, Life of Mine Plan	Mt/Dry	83.61
Concentrate Grade	% P_2O_5	37.28
Mine Life	Years	26
Average Mill Feed (Years 3-25)	Mt/y	3.35
Phosphate Concentrate Production (Years 3-25)	Mt/y	1.41
Average Life of Mine (LOM) Mining Cost	USD/t conc.	$31.64
Average LOM Beneficiation Cost	USD/t conc.	$15.25
Average LOM Concentrate Cost (Including Infrastructure)	USD/t conc.	$55.10
Average LOM Concentrate Cost (Including Slurry Pipeline Cost)	USD/t conc.	$56.24
Fertilizer Conversion Complex (FCC)		
Phosphoric Acid Plant Capacity	P_2O_5 t per annum	500,000
P_2O_5 Production Cash Costs	USD/t P_2O_5	$423.02
SPA Plant Capacity	P_2O_5 t per annum	150,000
SPA Production Cash Costs	USD/t SPA	$395.16
Granulation Plant Capacity	P_2O_5 t per annum	346,000
MAP Production Cash Costs	USD/t MAP	$319.10
NPS Production Cash Costs	USD/t NPS	$321.34
Sulfur Plant Capacity		
Sulfuric Acid Produced & Consumed (Years 3-25)	H_2SO_4 t per annum	1,276,000
Annual Cogeneration Production (Net)	MW	31
Average Annual Product Tonnes (Years 3-25)		
MAP	T	474,000
NPS	T	247,000
SPA	T	221,000
Average Annual Consumption (Years 3-25)		
Sulfur for H_2SO_4 production	T	421,000
Sulfur for NPS production		12,000
Ammonia for MAP production	T	63,000
Ammonia for NPS production	T	36,100
Life-of-Project (LOP) Operating Costs		
Average Annual Cash Operating Costs[7]	USD MM/y	$307.13
Average Annual OPEX + Sustaining CAPEX (SUSEX)	USD MM/y	$328.61

Description	Units	Amount
Capital Costs		
Initial CAPEX[8]	USD MM	$1,859
LOP SUSEX	USD MM	$545
Financial Analysis		
After-Tax $NPV_{8\%}$	USD MM	$1,467
After Tax IRR	%	17.4
Payback Period	Years	5.2

(1) The “Base Case” is a weighted average of three market forecast scenarios for the years 2022 to 2047.
(2) Reference prices (CAD/tonne MAP delivered Western Canada) for Base Case is $1,060.
(3) Reference prices (US/tonne P_2O_5 delivered Corn Belt) for Base Case is $1,570.
(4) Reference prices (CAD/tonne NPS delivered Western Canada) for Base Case is $1,065.
(5) Reference prices (US/long ton S CIF Tampa) for Base Case is $320.
(6) Reference prices (US/tonne NH_3 CIF Tampa) for Base Case is $630.
(7) Total operating costs includes administration, operations, maintenance costs at the Mine and FCC sites, plus SG&A costs.
(8) Includes constructed costs, contractor's fee, contingency, and owner’s costs.

22.3.2 Royalties

The economic analysis accounts for royalties under an agreement as described in Section 4.4.

22.3.3 Taxes & Depreciation

With inputs provided by Fox River’s tax advisor, the economic analysis considers the income and mining taxes applicable under the Federal and Ontario Provincial codes, applied across the life of project. Carbon tax is addressed separately, as detailed in section 21.2.3.3.

Income Taxes were calculated as if the mining and processing portions of the Project were in separate companies. A mine revenue transfer price applied based on a concentrate price of $135/tonne. The current balance of tax losses and other future deductions of CAD17m has been used to reduce future income and mining taxes in the tax calculations.

22.3.3.1 Ontario Mining Tax

The mining portion of the project would qualify as a remote mine for Ontario mining tax. Accordingly, mining tax is levied at a rate of 5% on taxable profit in excess of CAD0.5M derived from a remote mining operation in Ontario. 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 CAD10M of profit during the initial 10 year period is available to each new remote mine. In addition, a notional processing allowance deduction is allowed as a deduction in calculating taxable profit. The total Ontario mining taxes are CAD61.6M over the project life.

22.3.3.2 Federal and Provincial Income Taxes

The federal and provincial income taxes both have the same taxable income calculation. Ontario mining tax and various tax depreciation allowances are deductible for the calculation of taxable income. The first year of production has been assumed to be 2027 for the purposes of applying accelerated tax depreciation allowances. The federal income tax rate is 15% while the Ontario income tax rate is 10% applicable to the mining and processing portions of the project. For the mining company, the total federal income tax is estimated at CAD216.5M and the provincial income tax at CAD144.3M over the project life. For the processing company, the total federal income tax is estimated at CAD1,072M and the provincial income tax at CAD715M over the project life.

22.3.3.3 Carbon Tax

In January 2022, the Ontario government introduced a new carbon taxation model with an intention to encourage industrial projects to reduce carbon footprint. The basis for the legislation applies only to the consumption of fossil fuels.

Carbon tax implications are applicable to the emissions generated from the use of fossil fuels as a source of consumed energy. For the Project, this includes diesel fuel and natural gas. These are classified as Scope 1 emissions are subject to Ontario's Emissions Pricing Standards.

Consumption of electricity from the primary power grid, or from a cogeneration facility using a non-fossil fuel source, are not subject to this tax. This is classified as Scope 2 emissions, specifically grid electricity, is out of the scope for carbon pricing at the current juncture.

Cogenerated electricity in this circumstance is included in the calculation as a legitimate credit against allowable emissions and is therefore a net benefit to offsetting carbon tax impacts.

The basic calculation to determine if an operation receives carbon tax credits or a carbon tax penalty is annually as follows:

Calculated allowable emissions – Calculated expected emissions x Carbon tax rate per unit

Units of emission are expressed in tonnes of CO_2.

Allowable (annual) emissions are based on a calculation of estimated fossil fuel consumption applied against taxation formula factors and a stringency factor which declines year over year encouraging an operation to reduce emissions or be subject to an increasing difference between allowable and expected emissions.

Expected (annual) emissions are based on a calculation of estimated fossil fuel consumption from all sources. This calculation determines the tonnes CO_2 emitted directly from use of these fuel sources. It will remain unchanged year over year unless there is either a change in practice to non-fossil fuel use or, conversely, an introduction of equipment or process which increases consumption. The latter is contrary to the desired change the carbon tax is targeting to achieve and becomes an increasing tax liability as a result.

If the calculated *allowable* emissions are greater than the calculated *expected* emissions then the operation is in credit, is below the maximum annual threshold limit, and receives carbon tax credits.

If the calculated *allowable* emissions are less than the calculated *expected* emissions then the operation is in deficit, exceeds the maximum annual threshold limit, and receives a carbon tax penalty.

Additional information regarding the carbon taxation model is as follows:

- Under the current legislation the carbon tax rate per tonne of CO_2 applied to the difference between allowable and expected annual emission limits increases up to a maximum of $170/t in 2030. There is no information regarding any possible changes to this rate after this date and therefore a flat rate has been applied to future years. In addition, and due to planned regulatory reviews of this taxation model in future years, there is also no information regarding further possible reductions in the stringency factor and consequently this has also not been reduced after 2030.
- Within the model a three year grace period from the start of implementation (regardless of phase of the operation) is permitted. For the purpose of this calculation, it has been assumed that the initial production period for the Project will be 2027-2029 inclusive thus generating no tax implications in this timeframe.
- This tax does not apply to the construction phase.
- Under one integrated operation carbon tax credits generated at one site can be applied to another site. If the whole operation is in a situation of having excess credits these can be sold to other third parties to help offset deficits elsewhere.

Taxation is only applied once on each type of fossil fuel consumed. An operation will be purchasing fossil fuels through supply agreements without the carbon tax applied at the point of purchase (for example diesel fuel purchased from a pump in the town) and will be subject to the taxation through this model instead.

The calculation for this phase of study has been based on the interpretation of this recent legislation and has been completed by the Hatch Climate Change team using inputs from Hatch, JT and Fox River. Table 22-5 below summarizes the outcome from the calculations and is believed to be an accurate representation of the carbon tax impact based on the best available information at this stage of study.

Table 22-5: Project Carbon Tax Impact (Both Sites Combined)

FOX RIVER - FULL PROJECT POTENTIAL SCENARIO INCLUDING FERTILIZER COMPLEX CO-GEN OPERATION									
W/ Estimated Cogen Limit	**2027 Y (-3)**	**2028 Y (-2)**	**2029 Y (-1)**	**2030 Y (0)**	**2031 Y (1)**	**2032 Y (2)**	**2033 Y (3)**	**2034 Y (4)**	**2035 Y (5)**
Cogeneration credit limit	32,075	44,007	46,485	46,485	46,485	46,485	46,485	46,485	46,485
Diesel emissions limit	18,049	30,374	31,288	31,184	30,859	30,701	31,105	30,742	30,062
Propane emissions limit	0	0	0	0	0	0	0	0	0
Natural gas emissions limit	157,317	153,480	149,643	145,806	145,806	145,806	145,806	145,806	145,806
Total Annual Emissions Limit	207,441	227,861	227,416	223,475	223,150	222,992	223,396	223,034	222,353
Actual diesel emissions estimate	22,011	37,967	40,113	41,031	40,604	40,396	40,927	40,450	39,555
Actual propane emissions estimate	0	0	0	0	0	0	0	0	0
Actual natural gas emissions estimate	191,850	191,850	191,850	191,850	191,850	191,850	191,850	191,850	191,850
Estimated Emissions	213,862	229,818	231,963	232,882	232,454	232,246	232,778	232,301	231,406
Amount over/ under (tonnes CO2)	6,421	1,956	4,547	9,407	9,304	9,254	9,382	9,267	9,052
Carbon Tax	-	-	-	$170	$170	$170	$170	$170	$170
Credits Generated	0	0	0	($1,599,141)	($1,581,688)	($1,573,206)	($1,594,891)	($1,575,440)	($1,538,908)
	Grace period			Effective Taxation Period					

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The order-of-magnitude impact to the project has been calculated based on the following assumptions:

- For the FCC:
  - Much of the facility is designed to be electrically powered and heated using a cogeneration facility using steam provided through the burning of sulfur.
  - Fossil fuels will be consumed in the granulation process (natural gas) and for locomotives (diesel fuel).
- For the mine site:
  - All mobile equipment will use diesel fuel. The primary users are the haulage trucks and shovels. Smaller utility equipment has been included though the most significant consumer is equipment used in the open pit mining operation.
  - Most buildings will be heated in the colder months using electricity distributed from the main supply. Calculations for consumption have been based on the degree of heating required (personnel workplaces or the need to keep buildings above freezing), the approximate square metres of each building footprint and the expected annual climatic conditions for the region.
  - Explosives are another source of fossil fuel emissions under the taxation model. Nevertheless, consumption of explosives in the open pit is limited to occasional blasts in areas of higher strength material which cannot be excavated using shovels. The degree of use has been considered as having minimal impact for the purpose of carbon tax and has therefore not been included in the calculation.

The net impact of the two sites for the project is that the FCC will be in-credit and the mine site will incur a carbon tax penalty. These two sites in combination, and through the application of carbon tax credits from the processing location against the mine site penalty, results in a carbon tax penalty scenario for the project as a whole as indicated in Table 22-5.

22.3.3.4 Depreciation

Expenditures of initial and sustaining capital were assigned asset classifications for purposes of determining depreciation. The asset classes for the mining operation were Canadian development expenditures (“CDE”) and Class 41. CDE assets included capital expenditures for, mine dewatering, site preparation, and pre-stripping. Class 41 assets included capital expenditures for access road construction, mining equipment, beneficiation plant equipment and building, tailings disposal equipment and dam construction, and other site buildings. The asset classes for the FCC operations were Class 1 (costs for all buildings) and Class 43 for manufacturing and processing equipment.

22.4 Cash Flow

The individual cash flows for operating cost, revenue from sale of products, shipping costs, royalties, sales, and other expenses are evaluated over the life of the project, with *Year 2* representing the first year of capital investment in the mine and FCC, and *Year 1* representing the first year of beneficial production. The Project production rates ramp up to full capacity in the third year of operation and remain at that level through year 25. In this scenario, the production in drops off again in year 26 with mine out. For year 1 and 2, the production will be at 69% and 94.7% of capacity, while in year 26 the production will be 35.7% of full capacity.

Table 22-6 below indicates the cash flows for the project and resulting NPV at various cash discount rates.

Table 22-6: Project Cash Flow Summaries

Internal Rate Of Return on Equity Capital		
	17.4%	Post-tax
	20.2%	Pre-tax

Production Years	-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
Mine Production																												
Mill Feed (kt)	0	0	2,128	2,533	2,827	2,853	2,655	3,124	2,970	2,936	3,916	3,637	2,746	2,680	3,516	3,327	2,762	3,351	3,454	3,853	2,976	3,629	3,834	3,646	4,508	4,255	3,668	1,823
% aP_2O_5	0.00	0.00	23.03	25.78	24.70	24.50	25.81	22.81	23.75	23.81	18.87	20.10	25.27	25.56	20.58	21.65	25.21	21.50	21.05	19.24	23.72	20.25	19.25	19.97	16.92	17.76	19.99	15.33
%CaO	0.00	0.00	32.91	35.77	36.45	34.10	36.10	31.61	35.15	33.73	26.59	28.34	33.52	35.75	31.38	30.22	35.88	30.28	28.75	28.30	32.36	27.68	27.56	26.74	23.32	25.99	27.88	25.09
Revenues by Product (US $MM)																												
Mine Revenue (Transfer)	0.00	0.00	131.51	180.49	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	190.59	68.03
MGA Revenue	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
SPA Revenue	0.00	0.00	161.34	221.43	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	233.82	83.47
MAP Revenue	0.00	0.00	261.46	358.84	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	378.93	135.26
NPS Revenue	0.00	0.00	138.19	189.66	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	200.27	71.49
Electric Power from FCC to Grid			6.19	8.73	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	9.25	3.21
FCC Revenue	0.00	0.00	567.18	778.66	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	822.28	293.43
Operating Costs by Product[1] (US $MM)																												
Mine Site Operating Cost	0.00	0.00	(74.58)	(81.65)	(84.32)	(84.09)	(83.01)	(85.81)	(84.78)	(83.96)	(87.89)	(87.98)	(84.68)	(83.79)	(87.53)	(86.89)	(84.47)	(86.29)	(86.97)	(88.69)	(84.51)	(87.02)	(85.30)	(84.47)	(90.19)	(89.22)	(79.83)	(40.04)
FCC Operating Cost	0.00	0.00	(175.08)	(226.28)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(236.84)	(108.74)
Total Operating Costs	0.00	0.00	(249.67)	(307.94)	(321.16)	(320.93)	(319.86)	(322.66)	(321.62)	(320.81)	(324.73)	(324.82)	(321.52)	(320.63)	(324.37)	(323.74)	(321.31)	(323.14)	(323.81)	(325.53)	(321.35)	(323.87)	(322.15)	(321.31)	(327.03)	(326.06)	(316.67)	(148.78)
Shipping Costs by Product (US $MM)																												
MGA Shipping Cost	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
SPA Shipping Cost	0.00	0.00	(10.81)	(14.83)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(15.66)	(5.59)
MAP Shipping Cost	0.00	0.00	(15.36)	(21.08)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(22.26)	(7.95)
NPS Shipping Cost	0.00	0.00	(8.02)	(11.00)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(11.62)	(4.15)
Total Shipping Costs	0.00	0.00	(34.19)	(46.92)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(49.54)	(17.69)
Royalties (US $MM)																												
Concentrate Production	0.00	0.00	(3.01)	(0.86)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.91)	(0.33)
S&GA (US $MM)																												
Concentrate Transfer S&GA	0.00	0.00	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)	(0.86)
Fertilizer Sales S&GA	0.00	0.00	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)	(0.80)
Total S&GA	0.00	0.00	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)	(1.66)

Production Years	-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
EBITDA (US $MM)																												
Concentrate Production	0.00	0.00	53.05	97.11	104.50	104.73	105.81	103.01	104.04	104.86	100.93	100.84	104.14	105.03	101.29	101.93	104.35	102.53	101.85	100.13	104.31	101.80	103.52	104.35	98.63	99.60	108.99	26.81
Fertilizer Production	0.00	0.00	225.60	324.17	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	98.17
Combined Operations	0.00	0.00	278.65	421.28	449.00	449.23	450.30	447.50	448.54	449.35	445.43	445.33	448.63	449.53	445.79	446.42	448.85	447.02	446.35	444.62	448.81	446.29	448.01	448.85	443.13	444.10	453.49	124.98
Debt Service (US $MM)																												
Interest on Mine Debt	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Interest on FCC Debt	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Net Income BEFORE Taxes[2] (US $MM)																												
Mining Operations	0.00	0.00	53.05	97.11	104.50	104.73	105.81	103.01	104.04	104.86	100.93	100.84	104.14	105.03	101.29	101.93	104.35	102.53	101.85	100.13	104.31	101.80	103.52	104.35	98.63	99.60	108.99	26.81
FCC Operations	0.00	0.00	225.60	324.17	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	344.50	98.17
Combined Operations	0.00	0.00	278.65	421.28	449.00	449.23	450.30	447.50	448.54	449.35	445.43	445.33	448.63	449.53	445.79	446.42	448.85	447.02	446.35	444.62	448.81	446.29	448.01	448.85	443.13	444.10	453.49	124.98
Taxes Paid (US $MM)																												
Carbon Tax																												
Mine Operations	0.00	0.00	0.00	0.00	0.00	(1.66)	(1.64)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)	(1.60)
FCC Operations	0.00	0.00	0.00	0.00	0.00	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
Total Operations	0.00	0.00	0.00	0.00	0.00	(1.59)	(1.58)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)	(1.54)
Mining Tax																												
	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	(0.13)	(3.72)	(3.93)	(4.02)	(3.93)	(3.45)	(3.46)	(3.33)	(3.65)	(3.75)	(3.97)	(3.85)	(3.87)	(4.01)	(3.79)	(3.96)	(4.45)	(0.33)
Federal & Provincial Income Tax																												
Mine Operations	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	(6.58)	(18.64)	(18.56)	(20.11)	(21.09)	(20.75)	(20.36)	(20.19)	(20.25)	(20.61)	(20.52)	(21.67)	(21.34)	(21.85)	(22.16)	(21.23)	(21.76)	(24.23)	(4.69)
FCC Operations	0.00	0.00	0.00	0.00	(29.01)	(66.15)	(70.76)	(74.05)	(75.84)	(77.13)	(78.65)	(79.74)	(80.54)	(81.13)	(81.57)	(81.90)	(81.03)	(80.44)	(81.17)	(81.70)	(82.08)	(82.36)	(82.57)	(82.17)	(81.89)	(82.27)	(82.54)	(21.53)
Total Operations	0.00	0.00	0.00	0.00	(29.01)	(66.15)	(70.76)	(74.05)	(75.84)	(83.71)	(97.29)	(98.30)	(100.66)	(102.22)	(102.32)	(102.26)	(101.22)	(100.69)	(101.78)	(102.22)	(103.75)	(103.71)	(104.42)	(104.33)	(103.12)	(104.03)	(106.77)	(26.22)
Total Taxes Paid																												
Mine Operations	0.00	0.00	0.00	0.00	0.00	(1.66)	(1.64)	(1.60)	(1.60)	(8.18)	(20.38)	(23.88)	(25.65)	(26.70)	(26.28)	(25.41)	(25.25)	(25.18)	(25.86)	(25.87)	(27.24)	(26.80)	(27.32)	(27.78)	(26.62)	(27.31)	(30.28)	(6.63)
FCC Operations	0.00	0.00	0.00	0.00	(29.01)	(66.09)	(70.70)	(73.99)	(75.77)	(77.07)	(78.58)	(79.68)	(80.48)	(81.07)	(81.51)	(81.84)	(80.97)	(80.38)	(81.11)	(81.64)	(82.02)	(82.30)	(82.51)	(82.10)	(81.83)	(82.21)	(82.48)	(21.47)
Total Operations	0.00	0.00	0.00	0.00	(29.01)	(67.74)	(72.34)	(75.59)	(77.38)	(85.25)	(98.96)	(103.56)	(106.13)	(107.77)	(107.79)	(107.25)	(106.22)	(105.56)	(106.97)	(107.51)	(109.26)	(109.10)	(109.83)	(109.88)	(108.45)	(109.52)	(112.76)	(28.09)
CAPEX (US $MM)																												
Mine Site CAPEX	(278.25)	(388.07)	(106.19)	(2.64)	(5.12)	(5.12)	(9.06)	(5.12)	(10.80)	(7.45)	(10.23)	(12.65)	(5.12)	(6.72)	(5.12)	(41.94)	(13.70)	(8.05)	(5.72)	(10.83)	(9.52)	(5.85)	(13.64)	(3.08)	(3.08)	(3.08)	(3.08)	(9.60)
FCC Site CAPEX	(382.70)	(553.35)	(167.35)	(10.93)	(10.93)	(10.93)	(10.93)	(10.93)	(25.93)	(10.93)	(10.93)	(10.93)	(10.93)	(10.93)	(10.93)	(10.93)	(40.93)	(10.93)	(10.93)	(10.93)	(10.93)	(10.93)	(10.93)	(25.93)	(10.93)	(10.93)	(10.93)	0.00
Total CAPEX	(660.95)	(941.42)	(273.54)	(13.58)	(16.06)	(16.06)	(19.99)	(16.06)	(36.73)	(18.39)	(21.16)	(23.58)	(16.06)	(17.66)	(16.06)	(52.87)	(54.63)	(18.98)	(16.65)	(21.76)	(20.46)	(16.79)	(24.57)	(29.01)	(14.01)	(14.01)	(14.01)	(9.60)

Production Years	-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
Borrowed Funds[3] (US $MM)																												
Mine Site Borrowed Funds	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
FCC Site Borrowed Funds	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Principal on Mine Debt	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Principal on FCC Debt	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Equity Capital (US $MM)	(660.95)	(941.42)	(273.54)	(13.58)	(16.06)	(16.06)	(19.99)	(16.06)	(36.73)	(18.39)	(21.16)	(23.58)	(16.06)	(17.66)	(16.06)	(52.87)	(54.63)	(18.98)	(16.65)	(21.76)	(20.46)	(16.79)	(24.57)	(29.01)	(14.01)	(14.01)	(14.01)	(9.60)
Net Income AFTER Taxes (US $MM)																												
Mining Operations	0.00	0.00	53.05	97.11	104.50	103.07	104.17	101.40	102.44	96.67	80.55	76.96	78.49	78.33	75.01	76.52	79.10	77.35	75.99	74.26	77.07	75.00	76.20	76.57	72.02	72.29	78.72	20.18
FCC Operations	0.00	0.00	225.60	324.17	315.49	278.41	273.80	270.51	268.72	267.42	265.91	264.82	264.02	263.43	262.99	262.66	263.53	264.12	263.39	262.86	262.48	262.20	261.99	262.39	262.67	262.29	262.02	76.71
Total Operations	0.00	0.00	278.65	421.28	419.99	381.48	377.96	371.91	371.16	364.10	346.46	341.78	342.51	341.75	338.00	339.18	342.63	341.47	339.38	337.12	339.55	337.20	338.19	338.97	334.68	334.58	340.73	96.89
Cash Flows (US $MM)																												
Annual	(660.95)	(941.42)	5.11	407.70	403.94	365.43	357.97	355.86	334.43	345.71	325.30	318.20	326.45	324.10	321.94	286.31	288.00	322.48	322.72	315.35	319.09	320.41	313.61	309.96	320.67	320.57	326.72	87.29
Cumulative	(660.95)	(1602.37)	(1597.25)	(1189.55)	(785.61)	(420.19)	(62.21)	293.64	628.07	973.79	1299.09	1617.29	1943.74	2267.83	2589.78	2876.08	3164.08	3486.56	3809.29	4124.64	4443.73	4764.14	5077.75	5387.71	5708.38	6028.95	6355.67	6442.96

Notes

(1) Excludes Shipping Cost

(2) Before Emissions-based Carbon Tax applied

(3) Borrowed Funds Feature not accurate

NPV	
$MM	Rate
1468	8.00%
972	10.00%
528	12.50%
216	15.00%

23. Adjacent Properties

There are no currently known or identified carbonatite complexes within 50 km of the Carbonatite Complex.

The nearest known similar deposit is the Kapuskasing Carbonatite Complex (Cargill Township) located approximately 120 km to the southeast of the Martison Project which hosted an active mine owned by Agrium of Calgary which recovered phosphate (apatite) from 1999 until 2013.

24. Other Relevant Data & Information

No additional information.

25. Interpretation & Conclusions

The following identifies the key conclusions based on the outcomes of the work to complete this PEA technical report.

25.1 Geology & Mineralization

The Martison Phosphate Project has seen intermittent exploration activity since the early 1980's. Exploration work, limited to the winter months, has collectively accumulated close to 22,000 m of drill core for a total of approximately 8170 samples analysed.

Two hundred (200) drillholes for 19,840.8 m in the digital drillhole database pertain to Anomaly A and were lithologically modelled to include 499 intercepts over 18,009.92 m. Eighty seven (87) of the Anomaly A drillholes intersected residuum.

Fox River took over 100% ownership of the Project in February 2016 but have not undertaken any site investigative work since obtaining the Project rights.

There has been a closer review by DMT of the database applied in the previous MRE and Technical Report issued on behalf of Fox River in 2016. This has resulted in the re-interpretation and remodelling of the geological data, and subsequently the input parameters for the block model, which have been modified in the light of the changed technical and economic configuration of the Project. Specifically, greater emphasis has been placed on the deposit's geochemistry, and the downstream effects this may have on the modified mineral processing flowsheet.

DMT classified the resources based on consideration of drillhole spacing, zone thickness, and grade continuity. Approximately 30% of Anomaly A resources were classified as Indicated with the remainder placed in the Inferred category. No resources are currently classified as Measured. The decision to exclude Measured for now is due, in part, to the variability of the bulk density of the various sub-lithotypes (2A, 2B and 2C) within the residuum which has not been established with sufficient confidence. Some uncertainty also remains regarding the basal Residuum / Carbonatite contact and this has also influenced the decision not to classify Anomaly A resources as Measured at this time.

DMT concludes that a significant Mineral Resource exists in the Martison Anomaly A consisting of an estimated 53.8 Mt of Indicated Mineral Resources at a grade of 22.99% P_2O_5 and 0.42% Nb_2O_5, and 128.3 Mt of Inferred Mineral Resources at a grade of 17.09% P_2O_5 and 0.42% Nb_2O_5. The residuum resources are reported at a cut-off grade of 6% P_2O_5.

The lateritic material contains an estimated Indicated Mineral Resource of 6.2 Mt at 1.13% Nb_2O_5 and 7.97% P_2O_5 and an Inferred Mineral Resource of 5.3 Mt at 0.69% Nb_2O_5 and 6.40% P_2O_5 at a cut-off grade of 0.2% Nb_2O_5.

It is the opinion of DMT that opportunity exists to add additional resources to the current resources with further drilling programs, in that the Anomaly A deposit still remains open at depth in several areas, particularly in the northern part of the trough or valley feature running northwest – southeast.

Based on current drillhole intersections and supportive evidence from recently applied ground geophysics, the Anomaly A deposit also appears to have potential for further extension to the east and northeast of the proposed open pit area.

Anomaly B remains unchanged from the 2015 MRE and represents a target for further exploration currently estimated at between 35 Mt and 70 Mt of residuum containing 14% - 20% P_2O_5.

Anomaly C remains an early stage exploration target.

No resource estimate has been established for the Rare Earth Elements (“REEs”) due to the paucity of sample data, particularly from the historical drillholes.

While the Project remains ‘winter access only’ future drilling programs and other site investigative work, which would include further geophysical, hydrogeological and geotechnical work, would have to be phased over several winter seasons.

The significant hydrogeological test work undertaken in 2008 and 2012 has demonstrated that the deposit aquifer conditions to have measurable boundaries and a relatively slow recharge rate, which will allow for the design of a dewatering solution for a proposed open pit operation.

25.2 Mining Methods

A single, multi-stage pit is envisioned for mining the Anomaly A deposit, using conventional open pit mining methods. Mine equipment will consist of off-highway haulage trucks, hydraulic face shovels, wheel loaders and blasthole production drills along with typical open pit support equipment. The use of mobile mining equipment will facilitate high levels of operational flexibility, high mining rates and lower unit operating costs. Mine design and scheduling analysis indicates that the deposit can be developed in a logical progression, both in the delivery of plant feed material and equipment requirements over the LOM. Owner mine equipment requirements have been developed and costed based on an owner-operator style of operation.

The site's topography is dominated by saturated muskeg and black spruce swamp: soft ground conditions can result in rutting or sinking of large, wheeled mobile mining equipment if not adequately prepared for. Therefore, ex-pit haul roads will need to be constructed and maintained with sufficient quantities and quality of locally sourced granular fill. It is assumed for this study that sufficient fill will be sourced from nearby aggregate quarry sites, or from Anomaly A's bedrock once exposed.

In-pit haulage is also expected to be impacted by softer than typical hard rock conditions, therefore additional efforts are anticipated to be necessary for road maintenance within the pit over the life of mine. The management of subsurface geotechnical and hydrogeological conditions are expected to be a priority for the application of efficient pit mining operations, haulage ramp maintenance and pit wall slope control.

An aquifer is present mainly within a zone of weathered bedrock at the base of the residuum. Active dewatering of the mining area, utilizing dewatering wells, is required to minimize disruption to the mining operations, notably during seasonal thaw when mining at depth. Pending further geotechnical and hydrogeological work to improve site knowledge and model subsurface conditions, it has been assumed that pit wall slope stability can be improved by active depressurization via dewatering wells, as well as the placement of a toe buttress made of granular fill to support the pit wall and promote drainage.

25.3 Metallurgical Testing & Recovery Methods

The metallurgical test programs described in this report provide the basis for developing the Martison beneficiation process. The results currently available demonstrate a viable process to proceed with additional exploration and metallurgical testing as outlined in Section 26.4.

Processing risk associated with the selected recovery methods are considered minimal for the following reasons:

- Over 40 years of research, investigations, studies, and testing have been utilized to optimize and refine the beneficiation process. Metallurgical test work on samples extracted from the Martison phosphate deposit began during the early 1980's. From 2007-2010, PFS test work was conducted by Jacobs Engineering, Florida, US, who performed bench and pilot scale testing and processing of the phosphate concentrate to produce phosphatic fertilizers. In 2011, ERIEZ Magnetics laboratories, Pennsylvania, US, (ERIEZ) performed semi pilot plant scale testing of column flotation cells. The history of the Martison Phosphate project is described in Section 6.
- All processing areas incorporate industry standard beneficiation equipment that has a lengthy record of proven reliability, durability, and process efficiency in the minerals processing industry specific to phosphate beneficiation. Examples includes:
- Conventional crushing equipment (toothed roll crushers and a cone crusher).
- A homogenizing stacker/reclaimer package system to mitigate short term mill feed variability.

- Rod mills operating in closed circuit with Derrick vibrating screens.
- Desliming using hydrocyclones.
- A magnetic separation circuit consisting of low and high wet magnetic separation equipment to remove magnetic/paramagnetic materials.
- Reagent conditioning and froth flotation utilizing column cells.
- Ball mill grinding of the flotation concentrate to achieve the concentrate particle size distribution required for efficient transport using a slurry pipeline.

Fox River PEA beneficiation plant design incorporates several process modifications to the plant design utilized in the 2008 PFS study. The process modifications, which resulted from bench scale and pilot plant tests performed after the PFS, are designed to increase plant metallurgical efficiency and P2O5 recovery, and to improve operability. These process modifications includes the following:

- Addition of a third crushing stage upstream of mill feed storage to facilitate a more reasonable reduction ratio for the downstream rod mills.
- Increase the mesh of grind from 212 micron to 425 µm by utilizing two rod mills (in parallel) to replace the rod mill/ball mill combination (in series). The rod mills will operate in closed circuit with Derrick vibrating screens. This current closed circuit grinding flowsheet concept resulted from pilot plant testing conducted by Jacobs in 2009.
- Modification of the desliming circuits to remove both natural slimes and grinding slimes after grinding. The grinding slimes were problematic for flotation. The increased P2O5 losses from rejecting grinding slimes were offset by improved flotation performance and reduced reagent consumption.
- Incorporated a WHIMS magnetic product regrind mill in closed circuit to liberate P_2O_5.
- Replacement of the mechanical flotation cells with column flotation cells. The Eriez column flotation cells improved flotation performance over the mechanical cells. The flotation circuit is configured as a rougher and two cleaners.

25.4 Project Infrastructure

25.4.1 Mine Site Infrastructure

The infrastructure proposed to support the mine site has been identified at a conceptual level in order to provide the minimum services required to support the mining operation and the beneficiation plant.

25.4.1.1 Access Road

The upgrading of the existing access road and the proposed extension to the mine site has been designed as an all-season, unpaved road which will consist of a foundation of coarse material overlain by compacted engineered fill. For the extension, the route will be cleared of trees though the muskeg layer will be compressed and topped with a geotextile layer prior to the addition of fill. This right of way will be sufficient to also house the transmission line and concentrate slurry pipeline. The most optimal route for the extension will have to be confirmed and the road will require all year maintenance.

25.4.1.2 Transmission Line & Site Power

Power to the site will be supplied through an electrical transmission line which will both connect to primary power and the FCC in order to provide the surplus power generated at this location to the mine site. The design has provided for all step-down transformers at both the FCC and mine site. The line capacity is sufficient to meet the demand at the mine site. The arrangement for the extent of operation and ownership will still need to be confirmed with the local provider.

25.4.1.3 Beneficiation Plant

The beneficiation plant is located in close proximity to the open pit mining operations where mill feed will be ground and processed to produce a slurry of phosphate concentrate and rejecting phosphate-lean material to the TMF. This PEA study increases the phosphate concentrate production from 421,000 t/y (PFS) to 526,000 t/y P_2O_5 and uses the same slurry pipeline concept to deliver the concentrate to the FCC. The updated design incorporates column flotation, a third crushing stage upstream of mill feed storage and modifications to the desliming circuits. This plant is based on proven technology and will produce 1,412 ktpy of concentrate over a 26 year LOM.

25.4.1.4 Utilities

Site utilities have been designed to meet the basic requirements for the mine site. The sewage treatment plant will be an off-the-shelf, sufficiently sized to meet the expected site resourcing, with a design which will enable treated water to be sent to the tailings facility. Potable water to supply basic needs will be drawn from a drilled well and treated to a quality sufficient for domestic use with the exception of drinking water which will be supplied externally. Initial fire protection has been allowed for though will require a more thorough risk assessment to determine the extent of protection for each building in the next phase of design.

25.4.1.5 *Ancillary Buildings & Services*

The size of each building and the most effective means of construction (modular, pre-engineered or stick build) will be determined in future stages of study. The proposed site layout will still require confirmation of the most optimal layout which will be as compact as possible to minimize the extent of clearing for construction and for water retention. The current layout provides for placement of the primary maintenance facilities in proximity to the mining operations. Other buildings are in close proximity for ease of access, especially in poor weather conditions. A traffic flow analysis will also be required to ensure safe interaction between site vehicles and pedestrians. The location of the explosives magazine will also be revisited.

25.4.1.6 *Site Preparation*

The initial preparation of the site follows the same steps identified in the PFS in that this activity will be most effectively performed in the winter months to enable the water laden muskeg to be removed with minimal run-off. The approach to initial site dewatering through the establishment of drainage channels and water collection areas is a logical step prior to muskeg removal. Subsequent site preparation will focus only on the areas required to establish the mine infrastructure and operations requirements and the extent of which will need to be detailed further in a subsequent phase.

25.4.1.7 *Tailings Management Facility*

The tailings impoundment area has been identified as large footprint to the south of the beneficiation plant and contained within lined embankments built from glacial till removed during the starter open pit preparation. This containment area will be developed in phases with additional cells being constructed as required to an ultimate size for a 26 year mine life. Within the cells, the site will be clear cut although the muskeg will remain in place. The muskeg will be removed for the footprint of the lined contained embankments. The design is compliant with storage requirements for tailings in a low relief environment and based on the expectations for tailings content and settling.

25.4.1.8 *Site Water Management*

Site water management has been approached from addressing diversion of noncontact water courses as the site expands and collecting all contact (site impacted) water for collection and storage in the tailings management facility. The study has completed an analysis of the natural watershed to enable understanding of water management as well as development of the water balance based on best available information. Further detail will be required in a future stage of study.

25.4.1.9 *Concentrate Slurry Pipeline*

The slurry pipeline is a buried pipeline which follows the route of the site access road enabling ease of construction, maintenance and monitoring. The design for the pipeline facilities includes additional storage at the FCC, cathodic protection, monitoring (using Pipeline Adviser™, and SCADA systems) and telecommunication. The PEA pipeline is identical in length and pipe diameter to what was proposed in the 2008 PFS nevertheless an increase in capacity from 1.16 Mtpa to 1.41 Mtpa has been accomplished by increasing the slurry flow rate and selecting a pump capable of safely operating at higher pressures, up to 250 bar. A leak detection system, will warn the operator,

25.4.1.10 *Other*

It is envisaged that there will be no accommodation facilities on-site and all personnel will be transported to and from the Hearst area daily along the all-season road. Long daily commutes may eventually impact employee retention and this current base case will need to be examined in a future phase.

25.4.2 *Fertilizer Conversion Complex*

25.4.2.1 *Sulfuric Acid Plant*

The 4,200 t/d sulfur burning SAP will produce H_2SO_4 and steam. The majority of H_2SO_4 is used to convert the phosphate concentrate to phosphoric acid and a minor amount is used to formulate NPS. The steam is used to generate carbon free electric power and as a source of heat by other process plants. The electric power produced can operate the entire FCC and the excess provides most of the power drawn by the beneficiation plant and the slurry pipeline at the mine site.

25.4.2.2 *Phosphoric Acid Plant*

The 1,500 t/d Dihydrate Phosphoric Acid Plant will provide the P_2O_5 required by the production schedule. The dihydrate technology, coupled with wet gypsum stacking, allows for maximum P_2O_5 recovery but provides filter acid containing 28% P_2O_5. Steam from the sulfuric acid plant is used to concentrate the filter acid to the P_2O_5 strengths required to produce the fertilizer products.

25.4.2.3 *Super Phosphoric Acid Plant*

The 545 t/d SPAP converts phosphoric acid available from the PAP, using available steam from the SAP and chemical treatment, into a high strength purified liquid fertilizer product containing at least 68% P_2O_5 and a lower grade intermediate product (sludge). The sludge, which contains P_2O_5, is consumed in the granulation plant.

25.4.2.4 *Granulation Plant*

The 2,310 t/d granulation plant will produce both the MAP and NPS fertilizer products, but not simultaneously. The granular fertilizers are produced intermittently (in batches). The granulation plant uses phosphoric acid, sludge from SPAP, steam, and ammonia to produce MAP. The granulation plant uses phosphoric acid, sludge from SPAP, steam, ammonia, sulfuric acid, molten sulfur, and zinc to produce NPS. Warehouse capacity and dual load-out capability provide year round availability for both MAP and NPS products.

25.4.2.5 *Infrastructure – Process Support*

FCC process units require a significant amount of material unloading and handling, water treatment, wastewater treatment, and air systems to operate in an efficient and cost effective. The Process Support systems specified for the FCC in this PEA are designed to meet these demands.

25.5 Environmental Baseline Studies, Permitting & Social or Community Impact

The EIA and further baseline studies that were undertaken for the site and access corridor (largely in 2007-08) were to stringent government set levels. Notwithstanding any future changes in federal and/or provincial government legislation that may impact on the permissions and permitting of the Project, the fundamental environmental findings in the EIA will likely continue essentially intact or unchanged. Fauna and flora, habitats, heritage, archaeology and the socioeconomic environment will remain largely as observed in 2008. From this perspective, the study is considered to still have value and relevance to the Project.

The former owners, PhosCan, engaged and worked in close cooperation with the local First Nations at Constance Lake whose tribal lands are directly impacted by the Project. PhosCan's continued dialogue and transparency with the CLFN regarding the Martison Project has in the past had a positive impact on the permitting and other permissions required for the Project. CLFN have, in the past, expressed their support for the further development of the Project and it is anticipated that a similar approach by Fox River will maintain the good relationship with CLFN and continue to foster positive feedback for the Project.

Further exploration drilling and other associated site work and geological studies are warranted to advance the Project. Current legislation requires all site work to be permitted well in advance of the proposed start date and involves an application of the proposed site work to MNDMF and consultation with impacted First Nations communities. In this regard Constance Lake First Nations are directly impacted, with the communities of Fort Albany and Moose Creek being downstream stakeholders.

25.6 Economic Analysis

The calculated internal rate of return for the Base Case is 17.4% on an after-tax basis and 20.2% on a pre-tax basis. The payback period for the Base Case is 5.2 years. The PEA has been supported by additional test work (undertaken to better characterize the samples) and further detailing of the plant designs including superior beneficiation technology, an allowance for two granular products, combination of FCC operations, wet gypsum stacking, professional rail layout, and dual granular product loadout.

The net present value of the Project, at an 8% discounted rate, is USD 1,467 MM on an after-tax basis.

Within the limits of the estimating details, the PEA results indicate the proposed mining, beneficiation, concentrate transport, chemical processing, and fertilizer production operations of the Project are technically feasible and economically viable. Location of the facility provides freight advantages to the Canadian markets over US and overseas competitors. The product mix of MAP, NPS, and SPA provides for a wide customer base.

Overall, the results are sufficiently attractive to warrant the next phase of engineering study.

25.7 Risks & Opportunities

25.7.1 Risks

25.7.1.1 Geology & Mineral Resources

- The preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. There is a risk that the significant volume of Inferred resources (approximately 60% in the current MRE), may not be established as Indicated resources by additional drilling.
- While the Martison deposit is still at an exploration stage and an all season road does not currently exist, the site remains ‘winter access only’. Legislation and regulation for permitting of future drill programs (and for temporary winter road access) is now lengthier and more onerous. In this regard it is important that Fox River re-establish the former good relationship that the previous owners (PhosCan) had with CLFN, to help facilitate permissions being granted in a timely manner. This will allow adequate time to schedule and prepare for any future winter drilling programs.
- The climatic effects of increasingly mild winters may allow only a shorter window of opportunity for any planned site investigative work, such as drilling, to be executed. Minimum and sustained low temperatures are required to construct and maintain temporary winter access, site roads and drill pads. Milder conditions may cause a later start to road construction, or a premature earlier finish to any proposed site work because of an early spring thaw.
- Past drill programs have not adequately defined the lower deposit boundary (the residuum/carbonatite basement contact) which has a direct effect on the confidence of the geological model. There is still a risk that this important lithoboundary remains inadequately identified in future drill programs and that this will continue to lower the confidence in the geological model.
- The limited bulk density measurements of the various lithotypes remain a risk to the geological model and MRE in determining accurate tonnages for each lithology, which has a direct impact on the accuracy of mine planning and project economics.

- Geotechnical and hydrogeological investigation are still at a preliminary stage providing data from which to establish basic material engineering properties and pumping requirements for the proposed open pit. Both of these technical areas require more detailed site work to establish optimum engineering conditions for pit slope angles and foundation conditions, and to define more accurately the ground water model and optimum requirements for pit dewatering.

25.7.1.2 Mining Methods

- The mine production schedule depicted in this study includes inferred resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. In addition, the current study's estimation of total mill feed is highly sensitive to both phosphate product(s) price(s) and the geological model's CaO grade values, the latter used as a crucial parameter in mill feed categorization and partially inferred via correlation equation.
- While drilling and blasting is assumed to be required for only a portion of the deposit, it is currently uncertain how much of the deposit's consolidated material cannot be free dug. Any increase in blasting requirements will affect overall mine operation costs and equipment requirements, as well as fugitive emissions emanating from drilling and blasting operations.
- Legacy pit wall slope stability analyses (see 2007 "Technical Memorandum – Preliminary Pit Slope Design Criteria" prepared by Golder Associates) are preliminary in nature. There is the risk that further analyses can identify shallower pit slopes, thus negatively impacting Project economics with increased strip ratios. The identified aquifer in conjunction with the generally expected insitu wet ground conditions make the success of mine dewatering systems a critical aspect in maintaining designed wall slopes.
- It has been assumed that locally sourced granular material can be excavated from at least two identified quarry sites. Increased capital and operating costs could be incurred, negatively impacting Project economics should the quality and/or quantity of granular material from these sites prove insufficient; both for the construction and maintenance of the all-season access road and on-site roads.
- In addition, it has been assumed that material of sufficient quality and quantity can be produced via mining of the Anomaly A deposit's bedrock, to be used in the construction and maintenance of in-pit ramps and other mine infrastructure. To date, it has not been confirmed that Anomaly A's bedrock is of acceptable quality for road construction and maintenance, with most detailed studies to date having been completed on materials that occur above the bedrock. If the planned material is not of acceptable quality and quantity within the pit, then significant increases in support costs may be incurred, i.e., additional sourcing of suitable material from ex-pit borrow quarry(s).

- The proposed waste facility and stockpile locations require confirmation dependent on future condemnation drilling results. Potential relocation could entail increased haulage distances with resultant increases in costs and equipment requirements. This risk is deemed low based on available studies and data regarding the site's mineralization trends.
- In addition, geotechnical assessment of the proposed waste facility and stockpile locations has yet to be completed. There is the risk that increased costs may be incurred with additional pad preparation work and/or increased operational support. This risk is deemed modest in consideration of the low ultimate lift heights for the planned facilities.
- Plant feed quality parameters may require a consistent, blended feed, which in turn may require a combination of multiple run-of-mine (ROM) stockpiles ahead of the primary crusher, and/or more selective pit digging sequences.

25.7.1.3 Recovery Methods

- Some risk concerning the beneficiation plant remains. The flowsheet development has evolved from considerable test work performed on Martison samples and the selected pieces of process equipment are well proven, however, the test work to date was performed on samples with average or above average P_2O_5 grade. Consequently, the cutoff chemical quality of mill feed and the ability of below average grade mill feed to produce acceptable concentrate at an economic recovery level has not been confirmed by testing.
- Weathered igneous phosphate deposits have been mined for decades in Brazil. The Brazilian phosphate deposits have a lower P_2O_5 content than the Martison phosphate deposit and consequently the risk is not considered to be great. Nevertheless, additional testing is required to eliminate and quantify the risk associated with processing below average grade mill feed.
- Any changes to the beneficiation flowsheet design may reflect a change in CAPEX and operating expenses.

25.7.1.4 Mine Site

- An effluent/wastewater treatment facility may be required if contact water effluents are not suitable for direct discharge, both for a scenario where pit dewatering is released independently and a scenario where all contact water reports to the Tailings Management Facility (TMF).
- Potential embankment deformation, or in the event of a dam breach, presents a high risk due to the location of the TMF and its proximity to the beneficiation plant and other mine infrastructure.
- Depressurization of perched water table within the construction embankment's muskeg may be costly and challenging depending on the site's governing hydrogeological regime.

25.7.1.5 Fertilizer Conversion Complex

- The FCC facilities are proven designs with enough production and intermediate storage capacity and based on experience with similar facilities in North America and globally. As a result, these designs will be subject to typical operating risks.
- There is a risk of changes to the FCC design requirements following further development of the mine and beneficiation designs, as well as additional laboratory and pilot plant studies (which are recommended). Consequently, this could reflect a change in CAPEX and operating expenses affecting SPA and granulation plant design requirements.
- The external source for supplemental raw water supply will have to be identified and confirmed.
- Environmental permitting that may impact on process design.
- Delays to obtain operating permits and licenses which can affect the project cost and schedule.
- Licensors of the NPS technology have not been involved in the PEA and as a result some details of the process design for the granulation of the product may require modification to accommodate the specific process route chosen.

25.7.1.6 Environmental Baseline Studies, Permitting & Social or Community Impact

- Early actions to address gaps between previous legislation in place at the time of the 2008 PFS and present day will help address potential risks to permitting delays. These need to be understand and commenced as soon as possible in the next phase.
- Updated requirements for public consultation must also be an early activity to ensure that sufficient time is allowed for any potential response that the Project has not currently accounted for or anticipated.

25.7.2 Opportunities

25.7.2.1 Geology & Mineral Resources

- The Inferred resources estimated in this MRE are considered too speculative to have economic considerations applied. Further exploration drilling of the Inferred resource may provide an opportunity to improve the confidence in the geological model, by adding more mineral resources to model and provide valuable additional geochemical data to the project database for future modelling work.
- The requirement for more detailed, and statistically sound, bulk density measurements are needed to improve the accuracy of the resource tonnage estimation and for resource category confidence. While there are bulk density test data available, and which have been applied in this MRE, there still exists a low confidence in this data due to the applied methodologies, volume, distribution (lithotypes tested) and overall paucity, which requires to be addressed.

- An opportunity exists, by way of an historic hole twinning exercise, to generate missing geochemical data for the MER and other oxides. For example, the $CaO:P_2O_5$ ratio is a particularly key piece of analytical data in mill feed categorization. For the most part, the holes drilled previously in 1981, 1982 and 1983 concentrated only on basic P_2O_5 and Nb chemical analyses, without reference to the overall geochemistry and the implications to a downstream processing flowsheet. A combined drilling program will also provide much needed laboratory and bench scale testing material for further fine tuning of the mineral process flowsheet.
- Following on from the success of the ground geophysics program carried in 2008 and 2009, there is an opportunity to apply this resistivity methodology in more detail to specific areas of the deposit, such as the north and eastern parts of Anomaly A, and to 'ground – truth' the other anomalies B and C. Closer, more regularly spaced lines will be required for higher definition of the key lithoboundaries.
- By constructing an all-season access road this will provide an opportunity for site based investigative work to continue year round if necessary, eliminating the requirement to evacuate drilling equipment and de-camp due to an early spring thaw, for instance. Lengthening the opportunity to complete essential site work can, indirectly, shorten the time to which construction may commence and will provide an already constructed main artery of access to facilitate an earlier construction start date.

25.7.2.2 Mining Methods

- Future pit design work should investigate pit sequencing and phasing opportunities to increase in-pit waste disposal, as well as waste deposition to the east of the Martison pit. Increased waste deposition either in-pit and/or towards the east of the pit can reduce haulage truck requirements and associated labour costs and fuel consumption, and mitigate waste facility footprint(s), overall site disturbance and site rehabilitation costs.
- Explore ROM blending opportunities as a means of achieving consistent mill feed grades and overall material quality parameters.
- The Project's mining operation is amenable to numerous potential decarbonization options for subsequent study trade-off, namely:
    - The use of tethered/cable electric alternatives to diesel powered mining equipment, (e.g., commercially available electric rope shovels and electric powered rotary blasthole drilling units).
    - The evaluation of alternative bulk mining equipment for excavating the deposit's glacial till, (e.g., electric powered Bucket Wheel Excavators or Bucket Chain Excavators).
    - The application of bulk material handling systems for all material types, such as in-pit and/or semi-mobile crushing and conveying (notably as haulage cycle times increase during the LOM).

- The use of alternative fuels, such as renewable diesel blends, to mitigate lifecycle GHG emissions.
- The use of automated mine production equipment, in conjunction with material conveying options, can offer potential productivity improvements, reduced operating costs and fleet-wide diesel consumption.

25.7.2.3 Recovery Methods

Future test work and trade-off studies should be performed to further optimize the beneficiation process. In addition, the proposed studies may also present opportunities to lower the estimated capital cost and operating costs of the beneficiation facilities. For example, two of the five flotation reagents account for USD 5.95/t concentrate or 60% of the total reagent costs. Preliminary tests indicated that the cost impact of those two reagents can be significantly reduced.

The potential source of niobium (Nb), in both the phosphate tailings and the lateritic material should continue metallurgical tests in an effort to develop an economically feasible process to recover Nb2O5. Depending on the niobium R&D results with the phosphate tailings, incorporation of niobium recovery into a single phosphate/niobium process flowsheet may be warranted in future studies. If a flow sheet can be developed for the Nb lateritic material, then capital and operating parameters should be examined in detail.

25.7.2.4 Mine Site

- The layout and development plan of the waste facility should be further optimized in future studies to investigate the opportunity to remove the north diversion channel.
- Stockpiling of dewatered Niobium rich tailings would eliminate the need to store this tailings stream in a slurry consistency within the impoundment; hence, reducing the capacity cost of the TMF and cost to dredge Niobium tailings for reprocessing. In addition to cost reduction, this action would also reduce the operational complexity of managing two tailings streams in separate impoundments.
- Active pumping wells may be considered to reduce the phreatic level within the TMF embankment, should sourcing of drainage material be limited. This is to ensure the dam foundation is depressurized during the operational period mitigating the risk of potentially elevated phreatic levels that may cause piping or instability.
- Optimization of infrastructure and staffing requirements for the mine site has the potential to reduce initial construction costs and future operating costs.

25.7.2.5 Fertilizer Conversion Complex

- The additional studies and test work which are recommended are expected to allow further optimization of the plant processes, with a reduction in the processing risks and opportunities to reduce the capital investment and / or operating expense.
- In addition, due to inherent flexibility in the plant designs, there is a potential to offer other fertilizer products to the target markets with little or no change in the CAPEX. If there is an appropriate market opportunity, there is also a potential to modify the phosphoric acid process to hemidihydrate (HDH) in order to produce gypsum pure in sufficient quantities for sale.
- Increase of the cogeneration capacity with a goal to be able to provide all electrical power requirements for both sites and minimize GHG emissions.
- The facilities at FCC are expected to be maintained such that the life of the complex may exceed life of mine. In that scenario, arrangement for importation of phosphate rock from other sources is an additional opportunity.

25.7.2.6 Environmental Baseline Studies, Permitting & Social or Community Impact

- There is a risk of permitting delays if the environmental baseline studies, are not undertaken early enough in the next phase.
- Delays in consultation with all stakeholders may cause Project timelines and permitting process to be extended.

26. Recommendations

26.1 Introduction

This PEA technical study indicates that the economics of the Project could improve with further exploration of the opportunities described in Section 25.7. It is recommended that Fox River carry the project forward with a subsequent study phase, focusing on gathering additional geological data and improving the accuracy of the overall economic evaluation.

Table 26-1 summarizes the recommended work, with cost estimates, for the next two years in order to further enhance the level of Project maturity.

Table 26-1: Recommended Actions – Next Two Years With Costs

Activity	Cost (CAD)
Anomaly A & B Exploration and Technical Study Program – Phase I	7,965,000
Anomaly A & B Exploration and Technical Study Program – Phase IIA	6,965,000
Additional Metallurgical Testwork	395,000
FCC Site Selection Options Assessment (Including Initial Hydrogeology and Geotechnical Evaluation) and Permitting	750,000
Consultation And Community Engagement For Access Road, Mine Ste, FCC And Transmission Lines	1,000,000
Environmental Baseline Studies (Mine Site and FCC)	1,500,000
Total	**18,575,000**

The work completed in the next two years will result in the following outcomes:

- The results from additional drilling and technical studies for Anomaly A will further enhance the level of definition of the deposit as well as the understanding of the geotechnical and hydrogeological properties which exist. If results fundamentally change what is currently known then the open pit design, water management and infrastructure construction assumptions will have to be revised.
- The additional metallurgical testwork will further confirm concentrate quality and recovery.
- The assessment of options for the FCC site will enable future design work for infrastructure to advance as well as early elimination of unfavourable sites and early mitigation of localized areas of concern at the selected site. This will also enable the design of the integration with existing nearby utilities and other required infrastructure (such as the rail line).
- Completion of the environmental baseline studies for both sites will determine any gaps and further actions which have to be taken to support the lengthy permitting process.

Recommendation details by study area for the next two year period, and for future phases of work are summarized in the sections below. All dollars stated are CAD.

26.2 Geology & Mineralization

A comprehensive Project site-based program is required to follow up those open ended elements of the geological model. This is primarily the definition of the basal carbonatite contact with the phosphate bearing residuum above and also to the north-northeast where the geologically interpreted model is similarly limited by the paucity of drill hole data. Both areas (at depth and to the north) appear likely to extend the potential resources of the deposit. Further east, the ground geophysical survey conducted in 2009 would suggest that additional mineral resources may be added there.

In line with the reassessment of the deposit geochemistry and the mineralogical requirements of the downstream processing flowsheet, it is apparent that a significant number of the historical drill holes, particularly those from the 1981, 1982 and 1983 drill campaigns, lack important chemical analyses. The MER for example, will need to be addressed in any future drilling and sampling program.

This program, summarized in Table 26-2 below, includes the following activities:

- A comprehensive two phase program of site work is proposed, which would consist predominantly of drilling and sampling. As the site remains a 'winter only access', the drill programs must adopt a similar approach to that used in previous years whereby the Project site work is split into achievable and manageable work blocks which can be completed within the short winter window of typically 9-12 weeks duration.

  The drilling will provide further bulk sample volume for follow on metallurgical test work; provide much needed bulk density and moisture content information for each of the lithotypes and sub-lithotypes of the residuum.

- The use of ground geophysics to further examine the relatively unexplored Anomaly B and Anomaly C in order to further define the deposit boundaries there. The results of the limited field program, which used this technique in 2008-2009, demonstrated the ability of this method to model subsurface profiles and broad lithological boundaries.

  By adopting closer spacing between the scan lines (100-200m spacing), and modifying the system (transponder) arrays, greater detail may be added to the subsurface model. The increased detail and 'ground – truthing' of the subsurface will likely assist in planning and optimizing the design of future resource drilling programs, should the interpreted results support it.

- Further hydrogeological investigations will be required to reinforce the preliminary observations and dewatering model proposed by AMEC in 2012.

- Additional geotechnical site and laboratory testing will be required.

  - At the mine site this will be to determine, in more detail, the in–pit engineering properties of the various lithotypes with a focus on slope stability characteristics as well as foundation engineering properties for the mine based infrastructure.

  - At the FCC geotechnical work will be required for the foundation designs of all infrastructure.

  - In addition, construction materials for the all-season access road, re-engineering of mine site foundations, berms and site roads should be identified and appropriately tested for purpose.

Table 26-2: Proposed Martison Exploration and Technical Study Program and Budget Summary

Phase	Activity	Cost Estimate CAD
I	1) Combined Sonic and Diamond Drill program : 60 drillholes (appx 8,500 m) to probe deposit at depth and north (Resource generation) and twinning of historic holes	6,800,000
	2) Ground Geophysics of Anomalies B & C (High Sensitivity Resistivity). Approx. 20 profiles @ average length 1250 m (25,000m)	250,000
	3) Hydrogeological Pump Testing and Monitoring.	350,000
	4) Bulk Density & Moisture Content Testing	15,000
	5) Geotechnical site and lab based testing	300,000
	6) EIA review and revision	250,000
	Phase I Sub-Total	**7,965,000**
IIA	1) Revise/Update Geological Resource model, including Core Relogging and Variography studies.	150,000
	2) Combined Sonic and Diamond Drill program: 60 drillholes (appx 8,500 m) to probe deposit at depth, north and east (Resource generation and infill), further historic hole twinning.	6,800,000
	3) Bulk Density & Moisture Content Testing.	15,000
	Phase IIA Sub-Total	**6,965,000**
IIB	1) Revise/Update Geological Resource model.	85,000
	2) Combined Sonic and Diamond Drill program: 60 drillholes (appx 8,500 m) (Resource generation and infill).	6,800,000
	3) Bulk Density & Moisture Content Testing.	15,000
	Phase IIB Sub-Total	**6,900,000**
	Grand Total	**21,830,000**

Note: Based on approximate overall and inclusive drill program costs of CAD 800 per metre (to include temporary winter road construction, temporary camp, supervision, sampling and logging etc.).

26.3 Mining Methods

The subsequent study phase should aim to achieve the following milestones necessary towards improving the economic viability of the Project's mining operation:

- Build an updated geological block model utilizing both historical and new assay data to be collected and tested from the proposed geological drilling campaigns. Infill and step out drilling data will improve the geological confidence of the model, increasing measured and indicated mineral resource estimates and providing a basis for establishing a mineral reserve.
- Conduct a geotechnical and hydrogeological gap analysis of all available historical data and studies completed to date, ahead of the proposed geological drilling campaigns. This will establish any additional field work or testing programs that can be integrated into the geological campaigns and ensure sufficiency of data collected prior to any future geotechnical and hydrogeological modelling and analyses.
- Perform both condemnation drilling and geotechnical assessments of waste facility and stockpile locations. Review and reassess potential locations for waste deposition, as well as the geotechnical parameters for the facility designs to assess whether footprints can be reduced using higher ultimate lift height limits for the respective piles.
- Assess the Anomaly A deposit bedrock material's suitability for both haulage road and in-pit ramp construction and subsequent maintenance.
- Investigate known quarry sites to determine the available quantities and quality of locally sourced rockfill required during the Project's construction phase.
- Conduct mine design and planning work, incorporating the updated geological model as well as the latest geotechnical and hydrogeological assessments and plant processing parameters. For the selection of a preferred mining method, it is highly recommended that a focused mining methodology and equipment selection trade-off study be performed, centering on equipment sizing and the evaluation of mine decarbonization and electrification opportunities.
- Investigate mill feed blending opportunities, ensuring that mine grade control practices and ROM blending provides the best possible mill feed quality on a daily basis, i.e., maximizing P_2O_5 grade while optimizing/mitigating the presence of contaminant grades.
- Conduct detailed pit optimization and design exercises, investigating the potential to minimize pre-stripping tonnages, to optimize pit phasing and to maximize both overall Project economics and in-pit dumping opportunities.
- For basic and detailed engineering, it is recommended that mine production plans consider seasonal conditions and optimizing material extraction from pit bottoms and overburden/glacial till during the spring thaw. Although this period is challenging at most Canadian surface operations, the Project mine site appears to be particularly susceptible due to the wet and soft rock insitu conditions.

26.4 Metallurgical Testing & Recovery Methods

In the two year period post-PEA it is proposed that metallurgical testing be undertaken to determine the concentrate quality and recovery of 20 composite samples blended to specific chemical characteristics, followed by mineralogical testing of selected sample streams to identify mineral composition and locking of mineral species. Table 26-3 summarizes the work and costs. This includes:

- Five P_2O_5 composites (target % P_2O_5 = 10%, 14%, 18%, 22%, & 25%).
- Four CaO/P_2O_5 composites (target ratios = 0.8, 1.0, 1.3, & 1.6).
- Three Fe_2O_3 composites (target % Fe_2O_3 = 14%, 22%, 35%).
- Four MgO/MnO composites (target % MnO = 0.9%, 2.0%, 6.0% MnO, & 8.0% MgO).
- Four SiO_2 composites (target % SiO_2 = 2%, 6%, 10%, & 14.0%).

Table 26-3: Metallurgical Variability Testing – Phase 1

Activity	Weeks	Cost (CAD)
Prepare 20 composite samples(1)	4	60,000
Metallurgical laboratory testing(2)	20	300,000
Mineralogical laboratory testing(3)	2	20,000
Prepare test report(4)	2	15,000
Total	**28**	**395,000**

(1) Includes travel, sample preparation, and sample shipment to the selected laboratories.
(2) Includes crushing, grinding, desliming, LIMS, flotation, and WHIMS.
(3) Includes mineral identification in four waste products (slimes, LIMS magnetics, flotation Tailings, and WHIMS magnetics) and two concentrate samples.
(4) Includes documenting test procedures and results, evaluation of date, and recommend chemical cut-off criteria and procedures for subsequent laboratory testing.

Future test work should include testing to confirm that the coarser mesh of grind and incorporation of a WHIMS magnetic product regrind circuit will provide the intended increase in P_2O_5 recovery. In addition, trade off studies that may result in capital cost and operating costs savings should be considered.

Additional testing, as described below, should be designed to optimize plant metallurgical efficiency and P_2O_5 recovery, and to confirm the viability of the beneficiation process. Accordingly, the following metallurgical test programs are recommended.

- Variability testing. Two types of variability testing are recommended starting with:
  - Testing of composite samples prepared from existing core samples to evaluate the beneficiation response of specific qualities of mill feed. Testing should include a comparative analysis of sized versus unsized feed flotation performance as proposed in Table 26-3.
  - Testing of future drill core samples. This work would include (but not limited to) measurement of mill feed moisture and density as well as determining the metallurgical performance of samples from different locations in the Martison deposit.
- Optimization testing. Optimize specific beneficiation unit operations including:
  (a) confirm the reagent conditioning residence times and reagent requirements for rougher and cleaner flotation.
  (b) evaluate alternate flotation reagents to improve flotation performance and/or lower reagent cost.
  (c) evaluate sized feed flotation as an alternative to bulk flotation of -425+20 µm feed.
  (d) evaluate the effectiveness and need for sarcosine.
- Beneficiation process. The following recommendations apply to the beneficiation plant recovery process.
  - Testing to confirm that the coarser mesh of grind and incorporation of a WHIMS magnetic product regrind circuit will provide the intended increase in P_2O_5 recovery.
  - Examination of the potential benefit of post-flotation WHIMS versus pre-flotation WHIMS.
  - Examination the potential benefit of in-plant thickening of flotation tailings to lower the cost associated with pumping of low solids waste to the impoundment area and return of clarified process water to beneficiation.
  - Testing designed to further optimize plant metallurgical efficiency and P_2O_5 recovery are recommended.
  - Testing designed to further optimize specific beneficiation unit operations are proposed as well as pilot plant test programs.

- Pilot plant test programs. The pilot plant test program should be comprised of 4 major activities:
  - Semicontinuous beneficiation pilot plant testing.
  - Continuous phosphoric acid pilot plant testing of a bulk concentrate sample from the beneficiation pilot plant.
  - Batch scale testing of granular fertilizer made from pilot plant produced phosphoric acid.
  - Concentration/clarification to produce MGA and SPA for analysis and testing. Since the concentrate MER increased from 0.06 in the PFS to 0.09 for the PEA, SPA testing of the higher MER rock should be evaluated.
- Vendor test program. This program would be carried out in association with the pilot plant test program on freshly generated samples and should include:
  - Comminution tests to verify the crushing work index, universal compressive strength (UCS), ball/rod mill work index, and abrasion index.
  - Sedimentation/clarification tests of waste streams.
  - Sedimentation/flocculation testing of concentrate.
  - Testing of concentrate to define slurry rheology properties for design of the long distance slurry pipeline. The rheology tests should include yield stress, viscosity, corrosion rate as a minimum.

It is also recommended to conduct locked cycle flotation testing using Martison site water to determine if recycle of process water has an impact on flotation performance utilizing the optimized beneficiation process.

As a result of the deposit also being a potential source of niobium (Nb), continued research and development leading to an economically feasible process to recover Nb_2O_5 Is highly recommended. Depending on the niobium R&D results, incorporation of niobium recovery into a single phosphate/niobium process flowsheet may be warranted in future studies.

26.5 Project Infrastructure

26.5.1 Mine Site Infrastructure

The mine site infrastructure has been proposed at a conceptual stage and consequently the following list of recommendations should be investigated in the next phase of study:

- Examination of the sequence of site preparation steps to minimize the extent of muskeg drainage, muskeg clearing and protective berm placement.
- Improvements to the overall infrastructure layout to reduce the costs of utility distribution, determine the most optimal all season operational efficiency and complete a traffic flow study between structures.

- Analyze on-site staffing requirements, particularly non-production and non-maintenance roles. Opportunities to minimize on-site staffing by maximizing these roles at the FCC will help reduce the transportation requirements and reduce footprint of some buildings and demands on utilities.
- Determination of the most effective contract strategy for both construction and steady state support services.
- Further investigation of the most beneficial arrangement for the connection and maintenance of the electric transmission line to the HONI main grid power. Analyze the electrical demand load and power conservation at the mine site to fully realize the benefit of the supply of excess power from cogeneration at the FCC reducing the requirements for purchased power.
- A detailed water balance is required for LOM operations under different climatic conditions (wet/dry years, design rainfall and rain-plus-snowmelt events) as well as for anticipated climate change in the region.
- Staged water management plans will be required to include construction, operation and closure phases.
- Details of drainage system alignment and sizing, i.e., sediment ponds/basins, drainage ditches/culverts, pumps and pipelines, site erosion and sediment control are required in future engineering phases.
- The water quality/chemistry should be further analyzed/tested for tailings, waste rock and for existing rivers/creeks downstream of the final effluent discharge points.
- Site freshwater demand assessment and West Lake water quality test is required.
- Environmental permit conditions on discharge period, limits and criteria should be clearly communicated.
- Characterization of the tailings streams and foundation material is required to determine the tailings final density, as well as the foundation consolidation properties. These will be inputs to determine the final available capacity requirements within the TMF.
- Dewatering circuit for tailings to optimize site water flow to reduce tailings flow and reclaim water pipelines and TMF capacity requirement. Dewatering of the tailings allows for recirculation of tailings process water back into the plant for reuse. This reduces the cost to pump the larger volume of tailings (without dewatering) and reduces the size of the reclaim water pipeline.
- Identification of suitable borrow source for sand and gravel required for embankment internal drainage. This is needed to ensure that adequate material required for dam construction can be sourced within a given radius distance and avoid last minute and costly decisions.

26.5.2 Concentrate Slurry Pipeline

Opportunities for optimization have been identified which can be pursued in future phases of the project. The following items should be addressed in the next phase of the project.

- Slurry characteristics: determine parameters such as head loss and the pumping requirement, as well as thickener sizing. Hence, evaluation of slurry characteristics by testing a representative sample from the mine site is required.
- Slurry samples should be tested for corrosivity.
- A field visit by a route specialist to validate the route is recommended. A geotechnical report is recommended in future phases to determine the amount and type of rock along the route.
- Conduct surveys for route contour, hydrogeology and geotechnical, soil resistivity to define pipeline construction methodology and support a detailed construction estimate.
- Optimize pipe diameter and pumping requirements once the slurry characterization has been completed and the pipeline throughput range and the route has been finalized. Perform trade-off study for reductions for cost saving measures.
- Perform transient analysis to optimize steel requirements and provide necessary equipment for pressure containment for normal upset and emergency conditions.
- Review storage tank requirements in conjunction with likely production variability.

26.5.3 Sulfuric Acid Plant

The following additional work is recommended:

- Steam consumption and demand. The SAP can produce sufficient low pressure steam during design and average operation that an ALPHA™ System is not strictly required. A thorough review of the site steam and power distribution and utilization requirements should be included to ensure the additional CAPEX of the ALPHA™ system is justified by the value of the additional electrical power that is generated.
- Power production. The current Turbine Generator (TG) design assumes high pressure and low pressure extractions, which reduces the power production capability of the TG set and adds to the capital cost. Optimizing the extraction amounts and pressures to maximize power production should be further studied.
- Blower selection. The current design assumes the use of a dual blower configuration. There are potential cost savings associated with installation, instrumentation, and ducting with the single unit option, which can be further investigated.
- Dump condenser design basis. The dump condenser is currently designed to handle all HP steam produced by the SAP and is thus very large. This design assumption should be reviewed in the next phase once the updated steam requirements of the entire complex are established.

- Cooling water system. To reduce the size of the cooling water system it has been assumed that the dump condenser is installed in series with the acid plant coolers. This setup should be reviewed to ensure maximum return cooling water temperature is not exceeded if prolonged use of the dump condenser is foreseen.
- Cooling water blowdown. The cooling water makeup flow rate will be further refined once the cooling water quality is confirmed.

26.5.4 Phosphoric Acid Plant

The following additional work is recommended:

- Engage gypsum stack design specialists to advance the gypsum testing and wet stack design. This would include integrated cooling pond, decant pond, and return systems.
- Optimize equipment for gypsum handling.
- Consider pan filters in place of belt filters.

26.5.5 Super Phosphoric Acid Plant

The following additional work is recommended:

- Produce SPA using phosphoric acid obtained from phosphate rock having the target MER and MER^+ values.
- Determine the expected composition and quantity of solids from clarification and filtration.
- Determine composition of acid streams including metals analysis, calcium, sulfates, organic content and possibly rare earth, viscosity as a function of temperature including weak acid, strong acid, MGA and SPA.
- Perform oxidation/reduction study to confirm reactant(s) selection and dose.
- Perform corrosion study for several alloys applicable to the equipment producing SPA. Study should evaluate corrosion at process temperatures.
- Provide acid sample for plate and frame filter test. Test should evaluate the effectiveness of different types of filter aid, expected liquid composition remaining in the solids, and filtration rates as a function of temperature.
- Perform aging test on SPA as a function of MgO and polyphosphoric acid content.
- Include the MGA evaporators in the PAP scope.
- Examine whether the use of two smaller SPA evaporators would provide benefits to offset additional CAPEX.

26.5.6 Granulation Plant

The following additional work is recommended:

- Phosphoric acid specification development for granulation. Consumption of sludge from the SPA process adds another variable to the granulation design process. Initial calculations indicate that it is feasible to meet fertilizer and SPA product quality requirements, however the results suggest there is a limited tolerance for impurity fluctuations. It is therefore recommended that additional pilot plant testing be done which simulates the actual feed acid that the granulation plant will need to consume to maintain the overall site phosphoric acid balance. This should also include bench scale production of MAP and NPS granular products.
- Comparison of multiple small ammonia bullets storage versus larger ammonia bullets or sphere(s) for storage for optimal configuration.
- Analyze fabrication options (shop versus field).
- Select sulfur addition technology for NPS. Details are on hold until more information is known.
- Evaluate coating oil addition using a coating drum versus a ribbon blender.
- Investigate other sources of process water make-up to the granulation area.
- Investigate dry filler requirements (main source of filler will be SPA precipitates). This should consider the use of gypsum, unclarified 28% phosphoric acid, or reactor slurry as filler for granular products. For NPS, this may offset some sulfuric acid demand, lowering the production cost of the NPS.
- Optimize the zinc additive system. This would consider configuring the filler addition system for zinc addition.
- Optimize warehouse capacity.

26.5.7 Infrastructure – Process Support

The following additional work is recommended:

- Optimize equipment and infrastructure layout for the FCC facility.
- Consider winter control system for plant and instrument air to bypass dryers for plant air during acceptable ambient temperatures. This has the benefit of extending the desiccant service life.
- Investigate alternative wastewater treatment to reduce size of, or eliminate, several of the settling ponds.
- Investigate alternative wastewater discharge for potential reuse.

26.6 Environmental Baseline Studies, Permitting & Social or Community Impact

A summary of the information that needs to be updated to meet the current legislation for permitting requirements and approval is provided in Table 26-4. A combination of both desktop and field work may be used to fill these gaps. Where possible, it would be beneficial for this work to commence ahead of next phase of study to identify timeline and risks and completed concurrently within this future period.

The engagement of a dedicated and experienced permitting and community consultation expert to lead these activities early in the next phase is highly recommended to mitigate all risks in this area of project development.

Table 26-4: Summary of Work Required

Data to be updated	Supporting Permit / Approval
Wildlife habitat and use (caribou and bats) for the all-season access road, mine site, FCC, and transmission line	• Approval under the Endangered Species Act from the MECP • EA Notice of Intent to Proceed with all season access road • EA for other project components e.g., transmission line
Fish and fish habitat, Hydrology for the all-season access road, mine site, FCC, and transmission line.	• Authorization or letter of advice from DFO • Crown Land work permits from the NDMNRF
Consultation and engagement for the all-season access road, mine site, FCC, and transmission line	• EA notice of Intent to proceed with access road • EA for other project components e.g., transmission line • Authorization or letter of advice from DFO • Approval under the Endangered Species Act from the MECP
Cultural heritage and archaeology if required for the all-season access road, mine site, FCC, and transmission line	• Clearance letter from the MHSTCI

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