# Crawford Nickel Sulphide Project
# NI 43-101 Technical Report and Feasibility Study
## Ontario, Canada

Effective Date: October 1, 2023

Prepared for:

Canada Nickel Company, Inc.
130 King St. West
Toronto, ON, M5X 1E3, Canada

Prepared by:

Ausenco Engineering Canada ULC
15th Floor, 11 King Street West
Toronto, ON, M5H 4C7, Canada

List of Qualified Persons:

L. Paul Staples, P.Eng., Ausenco Engineering Canada ULC
Gregory Lane, FAusIMM, Ausenco Pty Ltd.
Jonathan Cooper, P.Eng., Ausenco Sustainability ULC
David Penswick, P.Eng., Gibsonian Inc.
John M. Siriunas, P.Eng., Caracle Creek International Consulting Inc.
Scott Jobin-Bevans, P.Geo., Caracle Creek International Consulting Inc.
Kenneth Arthur Bocking, P.Eng., WSP Canada Inc.
David Brown, P.Geo., WSP Canada Inc.
Steve Hales, P.Geo., WSP Canada Inc.
Michelle Fraser, P.Geo.,Stantec Consulting Ltd.
Gordon Murray, P.Eng., JL Richards & Associates Limited
Colin Hardie, P.Eng., BBA Inc.
Bruce Andrew Murphy, P.Eng., SRK Consulting (Canada) Inc.
Cameron Colin Scott, P.Eng., SRK Consulting (Canada) Inc.

# CERTIFICATE OF QUALIFIED PERSON
## L. Paul Staples, P.Eng.

I, L. Paul Staples, P.Eng., certify that I am employed as VP and Global Practice Lead with Ausenco Engineering Canada ULC ("Ausenco"), with an office address of 855 Homer Street, Vancouver, BC, Canada.

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated from Queen's University, Kingston, Ontario in 1993 with a Bachelor of Science degree in Materials and Metallurgical Engineering.
3. I am a member in good standing of the Engineers and Geoscientists of British Columbia, Licence #47367.
4. I have practiced my profession for 30 years continuously. I have been directly involved in many similar projects and studies in Canada and abroad including the nearby 38 Mt/a Dumont Nickel Project PFS and FS in Quebec, the 4 Mt/a Arctic Polymetallic Project PFS and FS in Alaska, the 80 Mt/a Grasberg Cu complex in Indonesia, the 20 Mt/a Lumwana Copper Project in Zambia and the 26 Mt/a Constancia Copper Project in Peru.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I visited the Crawford project site on August 31, 2022 for 1 day.
7. I am responsible for Sections 1.1, 1.16, 1.17.1, 1.17.2, 1.17.6.2, 1.23, 1.25, 2.1, 2.2.1, 2.4, 2.5, 2.6, 3.1, 17, 18.1, 18.5.2, 18.5.3, 18.5.4, 18.6, 18.7, 18.10.5, 18.10.6, 18.10.7, 18.12, 18.13, 21.1.1, 21.1.3, 21.1.5.2, 21.1.5.3, 21.1.5.4, 21.1.5.5, 21.1.7, 21.1.9, 21.2.5, 24, 25.1, 25.11, 25.18.5.4, 25.18.5.5, 25.19.4, 25.19.5, 26.1, 26.6, 27 of the Technical Report.
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Crawford project since 2020 and contributed to the Preliminary Economic Assessment that was filed in July 2021.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

"Signed and sealed"

L. Paul Staples, P.Eng.

# CERTIFICATE OF QUALIFIED PERSON
## Gregory Lane, FAusIMM

I, Gregory Lane, FAusIMM, certify that I am employed as a Chief Technical Officer with Ausenco Services Pty Ltd, ("Ausenco") with an office address of 189 Grey St, South Brisbane, QLD, Australia.

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated from University of Tasmania with a Master of Science.
3. I am a Fellow of Australian Institute of Mining and Metallurgy, #203005.
4. I have practiced my profession for 35 years. I have been directly involved in directing, managing and doing testwork on minerals projects, including numerous nickel sulphide projects.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Crawford project site.
7. I am responsible for Sections 1.12, 13, 25.4, 25.18.2, 25.19.2, 26.4, 27 of the Technical Report.
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Crawford project for four years and I was the QP for the Preliminary Economic Assessments in 2021 and 2022 Chapter: 13 and sub-sections: 1.10, 1.24.4, 25.5, 26.4 of those Technical Reports.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

"Signed and sealed"

Gregory Lane, FAusIMM.

# CERTIFICATE OF QUALIFIED PERSON
## Jonathan Cooper, M.Sc., P.Eng.

I, Jonathan Cooper, M.Sc., P.Eng., certify that I am employed as a Water Resources Engineer with Ausenco Sustainability ULC ("Ausenco"), with an office address of 11 King Street West, Suite 1500, Toronto, Ontario M5H 4C7.

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated from the University of Western Ontario with a Bachelor of Engineering Science in Civil Engineering in 2008, and University of Edinburgh with a Master of Environmental Management in 2010.
3. I am a I am a Professional Engineer registered and in good standing with Professional Engineers Ontario (registration #100191626), Engineers and Geoscientists British Columbia (registration #37864) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (registration # L4227).
4. I have practiced my profession continuously for over 15 years with experience in the development, design, operation, and commissioning of surface water infrastructure. Previous projects that I have worked on that have similar features to the Crawford Project are the Borden Advanced Exploration for Goldcorp, located in Ontario, Colomac Gold Project in NWT and Kwanika-Stardust for NorthWest Copper located in British Columbia.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Crawford project site.
7. I am responsible for Sections 1.17.3, 18.11, 21.1.4.2, 21.1.10.2, 21.1.10.3, 25.12.3, 25.18.5.6, 26.9, 27 of the Technical Report.
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Crawford project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

"Signed and sealed"

Jonathan Cooper, Msc., P.Eng.

# CERTIFICATE OF QUALIFIED PERSON
## David Penswick, P.Eng.

I, David Penswick, P.Eng., certify that I am an independent consultant to the mining and mining finance industries, with an office address of 163 Garden Ave, Toronto, Canada,

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated from Queen's University in Canada in 1989 with BASc in Mining Engineering, from University of the Witwatersrand in Johannesburg in 1993 with a MSc in Mining Engineering and from the University of South Africa in 1995 with an MDP in Business Management.
3. I am a member of Professional Engineers Ontario, membership number 100111644.
4. I have practiced my profession continuously for 34 years since graduating with my first degree. During this time, I have been directly involved in the operation, design and financial evaluation of numerous mining projects and operations.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I visited the Crawford project site on August 31, 2022 for 1 day.
7. I am responsible for Sections 1.14, 1.15.3, 1.19, 1.20, 1.21, 1.22, 1.24, 2.2.2, 2.3, 3.2, 3.4, 3.5, 14.11, 15, 16.3, 16.4, 16.5, 19, 21.1.1, 21.1.2, 21.1.4.1, 21.1.5.1, 21.1.6.3, 21.1.8, 21.1.10.1, 21.2.1, 21.2.2, 21.2.3, 21.2.4, 21.2.6, 22, 25.6, 25.10, 25.14, 25.15, 25.16, 25.17, 25.18.4, 25.18.7, 25.19.3, 25.19.7, 25.20, 26.5, 26.12, 27 of the Technical Report.
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Crawford project as an independent consultant since 2019 and contributed to technical reports published in 2020, 2021 and 2022 for the project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

"Signed and sealed"

David Penswick, P.Eng.

# CERTIFICATE OF QUALIFIED PERSON
## John M. Siriunas, P.Eng.

I, John M. Siriunas, P.Eng., certify that I am an associate independent consultant of Caracle Creek International Consulting Inc., with an office address of 25 3rd Side Road, Milton, Ontario, Canada, L9T 2W5.

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated from the University of Toronto (Toronto, Ontario) with a B.A.Sc. degree (Geological Engineering) in 1976 and from the University of Toronto (Toronto, Ontario) with an M.A.Sc. degree (Applied Geology and Geochemistry) in 1979.
3. I am a registered member, in good standing, of the Association of Professional Engineers of Ontario (PEO), License Number 42706010 (since June 1980), and possess a Certificate of Authorization from said Association to practice my profession.
4. I have practiced my profession continuously for over 40 years and have been involved in mineral exploration, mine site geology, mineral resource and reserve estimations, preliminary economic assessments, pre-feasibility studies, due diligence, valuation and evaluation reporting, and have authored or co-authored numerous reports on a multitude of commodities including nickel-copper-platinum group element, base metals, precious metals, lithium, iron ore and coal projects in the Americas.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I have personally visited the Crawford project site, the subject of the Technical Report, on August 11 – 12, 2022.
7. I am responsible for Sections 1.10, 1.11, 2.2.3, 11, 12, 27 of the Technical Report.
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Crawford project as an independent consultant since October 2019, having co-authored the following NI 43-101 technical reports:

   - Lane, G., Penswick, D., Jobin-Bevans, S., Siriunas, J., Staples, P., Daniel, S.E., and Van Zyl, K., 2022. Crawford Nickel Sulphide Project, NI 43-101 Technical Report, Preliminary Economic Assessment, & Updated Mineral Resource Estimate, Timmins Area, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Caracle Creek International Consulting Inc. and Ausenco Engineering Canada Inc., August 22, 2022, 412p.
   - Staples, P., Lane, G., Jobin-Bevans, S., Siriunas, J., Penswick, D., Daniel, S.E., and Van Zyl, K., 2021. Crawford Nickel Sulphide Project, NI 43-101 Technical Report & Preliminary Economic Assessment, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Ausenco Engineering Canada Inc., July 9, 2021, 383p.

- Jobin-Bevans, S., Siriunas, J., and Penswick, D., 2020. Independent Technical Report and Mineral Resource Estimates, Crawford Nickel-Cobalt Sulphide Project: Main Zone (Update) and East Zone (Initial) Deposits, Timmins-Cochrane Area, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Caracle Creek International Consulting Inc., Amended December 31, 2020, 263p.
- Jobin-Bevans, S., Siriunas, J., and Oviedo, L., 2020. Independent Technical Report and Mineral Resource Estimate, Crawford Nickel-Cobalt Sulphide Project, Timmins-Cochrane Area, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Caracle Creek International Consulting Inc., April 9, 2020, 203p.
- Jobin-Bevans, S., and Siriunas, J., 2020. Independent Technical Report on the Crawford Nickel-Cobalt Sulphide Project, Timmins-Cochrane Area, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Caracle Creek International Consulting Inc., January 27, 2020, 196p.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

“Signed and sealed”

John M. Siriunas, P.Eng.

# CERTIFICATE OF QUALIFIED PERSON
## Scott Jobin-Bevans, P.Geo.

I, Scott Jobin-Bevans, P.Geo., certify that I am the Principal Geoscientist with Caracle Creek International Consulting Inc., with an office address of 1721 Bancroft Drive, Sudbury, Ontario, Canada.

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated from the University of Manitoba (Winnipeg, Manitoba), BSc. Geosciences (Hons) in 1995 and from the University of Western Ontario (London, Ontario), PhD. (Geology) in 2004.
3. I am a registered member, in good standing, of the Professional Geoscientists of Ontario (PGO), License Number 0183 (since June 2002).
4. I have practiced my profession continuously for more than 28 years, having worked mainly in mineral exploration but also having experience in mine site geology, mineral resource and reserve estimations, preliminary economic assessments, pre-feasibility studies, due diligence, valuation and evaluation reporting. I have authored, co-authored or contributed to numerous NI 43-101 and JORC Code reports on a multitude of commodities including nickel-copper-platinum group elements, base metals, gold, silver, vanadium, and lithium projects in Canada, the United States, China, Central and South America, Europe, Africa, and Australia.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Crawford project site, the subject of the Technical Report.
7. I am responsible for Sections 1.2, 1.3, 1.5, 1.6, 1.7, 1.8, 1.9, 1.11, 1.13, 4.1, 4.2, 4.3, 4.4, 4.7, 6, 7, 8, 9, 10, 12, 14.1, 14.2, 14.3, 14.4, 14.5, 14.6, 14.7, 14.8, 14.9, 14.10, 14.12, 23, 25.2, 25.3, 25.5, 25.18.1, 25.19.1, 26.2, 26.3, 27 of the Technical Report.
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Crawford project as an independent consultant since October 2019, having co-authored the following NI 43-101 technical reports:
   - Lane, G., Penswick, D., Jobin-Bevans, S., Siriunas, J., Staples, P., Daniel, S.E., and Van Zyl, K., 2022. Crawford Nickel Sulphide Project, NI 43-101 Technical Report, Preliminary Economic Assessment, & Updated Mineral Resource Estimate, Timmins Area, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Caracle Creek International Consulting Inc. and Ausenco Engineering Canada Inc., August 22, 2022, 412p.
   - Staples, P., Lane, G., Jobin-Bevans, S., Siriunas, J., Penswick, D., Daniel, S.E., and Van Zyl, K., 2021. Crawford Nickel Sulphide Project, NI 43-101 Technical Report & Preliminary Economic Assessment, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Ausenco Engineering Canada Inc., July 9, 2021, 383p.

- Jobin-Bevans, S., Siriunas, J., and Penswick, D., 2020. Independent Technical Report and Mineral Resource Estimates, Crawford Nickel-Cobalt Sulphide Project: Main Zone (Update) and East Zone (Initial) Deposits, Timmins-Cochrane Area, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Caracle Creek International Consulting Inc., Amended December 31, 2020, 263p.
- Jobin-Bevans, S., Siriunas, J., and Oviedo, L., 2020. Independent Technical Report and Mineral Resource Estimate, Crawford Nickel-Cobalt Sulphide Project, Timmins-Cochrane Area, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Caracle Creek International Consulting Inc., April 9, 2020, 203p.
- Jobin-Bevans, S., and Siriunas, J., 2020. Independent Technical Report on the Crawford Nickel-Cobalt Sulphide Project, Timmins-Cochrane Area, Ontario, Canada. Unpublished report prepared for Canada Nickel Company Inc. by Caracle Creek International Consulting Inc., January 27, 2020, 196p.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

“Signed and sealed”

Scott Jobin-Bevans, P.Geo.

# CERTIFICATE OF QUALIFIED PERSON - KENNETH ARTHUR BOCKING

I, Kenneth Arthur Bocking, state that:

(a) I am a Fellow at:

WSP Canada Inc.

6925 Century Ave

Mississauga, ON, Canada, L5N 7K2

(b) This certificate applies to the technical report titled *Crawford Nickel Sulphide Project NI 43-101 Technical Report and Feasibility Study* with an effective date of: 1 October, 2023 (the “Technical Report”) prepared for the issuer, Canada Nickel Company.

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of the University of Saskatchewan with a Bachelor of Engineering degree in Civil Engineering in 1974 and an M.Sc. degree in geotechnical engineering in 1979. I am registered as a Professional Engineer in Saskatchewan, Ontario and Northwest Territories/Nunavut. My relevant experience after graduation for the purpose of the Technical Report includes over 49 years of consulting geotechnical engineering, which has specialized in mine waste management since 1988. During that period, I have been responsible for the design, construction and closure of numerous mine waste facilities

(d) My most recent personal inspection of each property described in the Technical Report occurred on 19 August 2022 and was for a duration of 1 day.

(e) I am responsible for Item(s) 1.17.4, 2.2.4, 18.10.1, 18.10.2, 18.10.3, 18.10.4, 25.12.1, 25.18.5.1,26.10 and part of Section 27 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) My prior involvement with the property that is the subject of the Technical Report is as follows. I was responsible for the technical review of the feasibility study level design of the proposed Tailings Management Facility.

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Mississauga, Ontario, Canada this 24 Day of November, 2023.

“Signed and Sealed”

Kenneth Arthur Bocking, P.Eng. (Ontario, Saskatchewan, Northwest Territories and Nunavut)

# CERTIFICATE OF QUALIFIED PERSON - DAVID BROWN

I, David Brown, state that:

(a) I am a Technical Fellow, Environmental Geoscientist at:
WSP Canada Inc.
25 York St, Suite 700
Toronto, Ontario, Canada, M5J 2V5

(b) This certificate applies to the technical report titled *Crawford Nickel Sulphide Project NI 43-101 Technical Report and Feasibility Study* with an effective date of: 1 October, 2023 (the "Technical Report") prepared for the issuer, Canada Nickel Company.

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of the University of Waterloo with a Bachelor of Chemistry and Environmental Studies in 1990 and a Master of Science in Earth Science in 1996. I am a registered member in good standing with the Professional Geoscientists of Ontario and the Professional Engineers and Geoscientists of Newfoundland and Labrador. My relevant experience after graduation and over the last 28 years for the purpose of the Technical Report includes designing and implementing geochemical studies to characterize mine tailings and waste rock related to proposed, operating and closed mines. I interpret those results to assess potential impacts on the surface and ground water environment and provide input to management strategies. I also develop monitoring programs to assess the effectiveness of operational and closure management plans.

(d) I did not visit the Crawford project site that is the subject of this technical report.

(e) I am responsible for Item 20.4 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Technical Report (Item 20.4) for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the part of the Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Toronto, Ontario, Canada this 24 of November, 2023.

"Signed and Sealed"

David Brown, P. Geo. (Ontario)

# CERTIFICATE OF QUALIFIED PERSON - STEVE HALES

I, Steve Hales, state that:

(a) I am a Lead Hydrogeologist at:
WSP Canada Inc.
#1 – 309 Exeter Road
London, Ontario, Canada, N6L 1C1

(b) This certificate applies to the technical report titled *Crawford Nickel Sulphide Project NI 43-101 Technical Report and Feasibility Study* with an effective date of: 1 October, 2023 (the "Technical Report") prepared for the issuer, Canada Nickel Company.

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of University of Guelph with an Honours Bachelor of Science in Earth Surface Science in 2005. I am a Professional Geoscientist (P.Geo.) of the Professional Geoscientists of Ontario (Registrant Number 2740) and the Association of Professional Engineers and Geoscientists of Alberta (Member Number 90385). My relevant experience after graduation and over 18 years includes field and office work relating to municipal, provincial and federal permitting, geotechnical, resource (oil and gas, pipeline, mining) and infrastructure sector groundwater projects, baseline hydrogeological assessments, and complex dewatering assessments. I have led a variety of projects that required skill development and training of colleagues on field instrumentation, groundwater monitoring well installation, geophysics, and hydrogeological testing of aquifers (packer tests, pumping tests, slug tests). Specific to the Crawford Nickel Sulphide Project and the Technical Report, I completed planning and execution of the hydrogeological tasks associated with site characterization for feasibility studies and the NI 43-101 report.

(d) My most recent personal inspection of each property described in the Technical Report occurred on November 1, 2021 and was for a duration of four days.

(e) I am responsible for Sections 1.15.1, 2.2.5, 16.1, 18.8.3, and 26.8 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at London, Ontario, Canada this 24 of November, 2023.

"Signed and Sealed"

Steve Hales, P. Geo. (Ontario, Alberta)

# CERTIFICATE OF QUALIFIED PERSON
## Michelle Fraser, P.Geo.

I, Michelle Fraser, P.Geo., certify that I am employed as a Senior Hydrogeologist and Water Resources Technical Discipline Leader with Stantec Consulting Ltd. (“Stantec”), with an office address of 100-300 Hagey Boulevard, Waterloo, ON, N2L 0A2.

1. This certificate applies to the technical report entitled “Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study” that has an effective date of October 1, 2023 (the “Technical Report”).
2. I graduated from the University of Waterloo, Waterloo, Ontario, Canada in 2005 with a Bachelor of Science, Honours Earth Sciences Geology Specialization and from the University of Waterloo, Waterloo, Ontario, Canada with a Masters of Science, Earth Sciences – Hydrogeology in 2007.
3. I am a Professional Geoscientist of the Association of Professional Geoscientists of Ontario (license number 1854).
4. I have practiced my profession continuously for 16 years. I have been directly involved in environmental consulting since my graduation from university. I have been involved in mining environmental studies and technical reports for 10 years, including baseline studies, environmental assessments, permitting, closure plans, and feasibility studies. This has included work on Greenstone Gold Mine, Lynn Lake Gold Project, and Marathon PGM Project.
5. I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I visited the Crawford project site on September 11, 2023 for a visit duration of 1 day.
7. I am responsible for Sections 1.4, 1.18, 2.2.6, 3.3, 4.5, 4.6, 5, 20.1, 20.2, 20.3, 20.5, 25.13, 25.18.6, 26.11, 27 of the Technical Report
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Crawford project since June 2023 supporting development of baseline reports that will be used to support the Environmental Assessment and permitting process.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

“Signed and sealed”

Michelle Fraser, P.Geo.

# CERTIFICATE OF QUALIFIED PERSON
## Gordon Murray, P.Eng.

I, Gordon Murray, P.Eng., certify that I am employed as a Senior Civil Engineer and Project Manager with J.L. Richards & Associates Limited ("JLR"), with an office address of 450 Speedvale Avenue West, Guelph ON, N1H 7Y6.

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated from the University of Toronto in 1980 with a Bachelor of Applied Science (B.A. Sc.)
3. I am a member of the Professional Engineers of Ontario, Licence # 33251604
4. I have practiced my profession for 43 years. I have been directly involved in the project management, environmental assessment, design and construction administration of roads, highways, railways and other related civil engineering projects, primarily for provincial, state and municipal government authorities throughout Canada and the United States of America.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I visited the Crawford project site on August 24, 2022, for a visit duration of six hours.
7. I am responsible for Sections 1.17.5, 2.2.7, 18.2, 18.3, 18.4, 21.1.6.1, 27 of the Technical Report
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Crawford project prior to the *Crawford Nickel Sulphide Project NI 43-101 Technical Report and Feasibility Study* report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

"Signed and sealed"

Gordon Murray, P.Eng.

# CERTIFICATE OF QUALIFIED PERSON
## Colin Hardie, P.Eng.

I, Colin Hardie, P.Eng., certify that I am employed as a Senior Consultant, Mining and Metals Business Line with BBA Inc., with an office address of 2020 Robert-Bourassa Blvd., Suite 300, Montréal, Québec, H3A 2A5, Canada.

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated from the University of Toronto, Ontario Canada, in 1996 with a BASc in Geological and Mineral Engineering. In 1999, I graduated from McGill University of Montréal, Québec Canada, with an M. Eng. in Metallurgical Engineering and in 2008 obtained a Master of Business Administration (MBA) degree from the University of Montréal (HEC), Québec Canada.
3. I am a member in good standing of the Professional Engineers of Ontario (PEO No: 90512500) since August 2000. I am also a member of the Canadian Institute of Mining, Metallurgy, and Petroleum (Member No. 140556).
4. I have practiced my profession for over 20 years by being employed for various companies in mining operations, consulting engineering and applied metallurgical research. My relevant project as a senior front- end study manager includes mine infrastructure design, electrical supply trade-offs, decarbonization studies, energy efficiency audits and mine electrification studies for numerous North American iron ore, precious metal, industrial mineral, and base metal projects.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I visited the Crawford project site on November 23, 2022 for a visit duration of *1 day.*
7. I am responsible for Sections 1.17.6.1, 2.2.8, 18.5.1, 21.1.6.2, 25.12.4, 25.18.5.3, 25.19.6, 27 of the Technical Report
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Crawford project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

"Signed and sealed"

Colin Hardie, P.Eng.

# CERTIFICATE OF QUALIFIED PERSON
## Bruce Andrew Murphy, P.Eng.

I, Bruce Andrew Murphy, P.Eng., certify that I am employed as a Principal Consultant with SRK Consulting (Canada) Inc. ("SRK"), with an office address of 2600 – 320 Granville Street, Vancouver BC, V6C 1S9 Canada.

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated from with a MSc. Eng (Mining) degree from the University Witwatersrand, in May 1996.
3. I am a professional engineer of Ontario; License # 100547515. I am also a registered engineer in British Columbia, Manitoba, Northwest Territories and Nunavut, and Newfoundland and Labrador
4. I have practiced my profession for 30 years specialising in mining rock mechanics. I worked in mining operational rock mechanics from 1989 to 2002, in both open pit and underground operations in South Africa (gold and iron ore) and in Zambia (Copper). Since joining SRK, I have carried out operational support and technical studies relating to rock mechanics for many open pit and underground projects located in Africa, Europe, Asia, and North and South America. I've gained open pit operational rock mechanics support experience including saprolites in high rainfall areas, hard structurally controlled deposits, altered and porphyry systems and cold climate environments. Additionally, involvement in open pit due diligence teams has provided me with exposure to many other sites and operating challenges.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I visited the Crawford project site between August 31, 2022 for a visit duration of one day.
7. I am responsible for Sections 1.15.2.2, 2.2.9, 16.2.2, 25.8, 25.9, 25.18.3.2, 26.7.2, 27 of the Technical Report
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Crawford project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

"Signed and sealed"

Bruce Andrew Murphy, P.Eng.

# CERTIFICATE OF QUALIFIED PERSON
## Cameron Colin Scott, P.Eng.

I, Cameron Colin Scott, P.Eng., certify that I am employed as a Principal Consultant with SRK Consulting (Canada) Inc. ("SRK"), with an office address of 2600 – 320 Granville St., Vancouver, BC.

1. This certificate applies to the technical report entitled "Crawford Nickel Sulphide Project – NI 43-101 Technical Report and Feasibility Study" that has an effective date of October 1, 2023 (the "Technical Report").
2. I graduated with a B.A.Sc. Degree in Geological Engineering from the University of British Columbia in 1974 and an M.Eng. Degree in Civil Engineering (Geotechnical Option) from the University of Alberta in 1984.
3. I am a P.Eng registered with Professional Engineers Ontario (Licence Number 100152511) and Engineers and Geoscientists British Columbia (Licence Number 11523).
4. I have practiced my profession for 49 years. I have been directly involved in the geotechnical and hydrogeological aspects of mining, including the site selection, design, permitting, operation and closure of mine waste facilities in Canada, the US, Mexico, Central and South America, Europe and various countries within the former Soviet Union.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 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 those sections of the Technical Report that I am responsible for preparing.
6. I visited the Crawford project site on August 31, 2022 for a visit duration of one day.
7. I am responsible for Sections 1.15.2.1, 2.2.10, 16.2.1, 18.8.1, 18.8.2, 18.9, 25.7, 25.12.2, 25.18.3.1, 25.18.5.2, 26.7.1, 27 of the Technical Report.
8. I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Crawford project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: November 24, 2023

"Signed and sealed"

Cameron Colin Scott, P.Eng.

## Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for Canada Nickel Company Inc. (CNC) by Ausenco Engineering Canada ULC, Ausenco Pty Ltd., and Ausenco Sustainability ULC (collectively, Ausenco); David Penswick; Caracle Creek International Consulting Inc. (Caracle Creek); WSP Canada Inc. (WSP); Stantec Consulting Ltd. (Stantec); JL Richards & Associates Limited (JLR); BBA Inc. (BBA); and SRK Consulting (Canada) Inc. (SRK); collectively the “Report Authors.” The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors’ services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by CNC subject to the terms and conditions of its contracts with each of the Report Authors. Except for the purpose legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party’s sole risk.

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# 1 SUMMARY

## 1.1 Introduction

Canada Nickel Company Inc. (CNC) commissioned Ausenco Engineering Canada ULC (Ausenco) to compile a NI 43-101 technical report and feasibility study for the Crawford Nickel Sulphide Project (Crawford Project), which is located north of Timmins, Ontario, Canada.

The responsibilities of the engineering consultants were as follows:

- Ausenco was responsible for overall compilation of the 43-101 report, metallurgical testwork management, design and costing of the following:
  - crushing and stockpiling facilities
  - processing plant and concentrate loadout
  - infrastructure buildings
  - surface water balance and water management infrastructure.
- Caracle Creek International Consulting Inc. (Caracle Creek) was responsible for the mineral resource estimates.
- David Penswick (independent consultant) was responsible for the mineral reserve, mining design and costing, marketing terms, overall compilation of the capital and operating costs, and financial modelling.
- SRK Consulting (Canada) Inc. (SRK) was responsible for site-wide geotechnical investigations, geotechnical design for soils and the open pits, and design of stockpiles and impoundment facilities.
- WSP Canada Inc. (WSP) was responsible for design of the tailings facility, hydrogeological modelling, and for the geochemical assessment.
- Stantec Consulting Ltd. (Stantec) was responsible for the environmental and permitting planning.
- JL Richards & Associates Limited (JLR) was responsible for design of the highway realignment and new rail spur.
- BBA Inc. (BBA) was responsible for design of the site power supply infrastructure.

The report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and Form 43-101 F1, and is prepared using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

The report supports disclosures by CNC in a news release dated October 12, 2023 entitled “Canada Nickel Announces Positive Bankable Feasibility Study for its Crawford Nickel Sulphide Project.”

## 1.2 Property Description

The Crawford property, located mostly in Crawford and Lucas townships with portions in Mahaffy and Carnegie townships, is about 42 km north of the City of Timmins, and can be found on 1:50 000 NTS map sheet 42A/14E and 14F, Buskegau River. The approximate centre of the property is at UTM coordinates 473380mE, 5408504mN (NAD83, UTM Zone 17 North; EPSG:2958) and elevation ranges from about 265 to 290 m above mean sea level (amsl).

## 1.3 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

### 1.3.1 Mineral Tenure and Surface Rights

The current property boundary spans approximately 9,611 ha (96.11 km$^2$), consisting of a combination of patented lands (Crown patents) and unpatented mining claims (staked claims). As of the effective date of the report, CNC holds a 100% interest in the mining lands that comprise the project, subject to certain terms and royalties.

Specifically, the property comprises 116 Crown patents (freehold patented lands) in Crawford and Lucas townships that cover approximately 7,827.42 ha, and 148 single cell mining claims (SCMC) and two multi-cell mining claims (MCMC) in Crawford Township, covering approximately 3,246.66 hectares. Some mining claims overlap areas with patented lands. In this region of Ontario, one SCMC averages approximately 21.22 hectares.

The 116 Crown patents in Crawford and Lucas townships are for mineral rights only (i.e., CNC does not control the surface rights, except for one patent) and are registered with the Land Registry Office, District of Cochrane (LRO 06). The status of patented lands can be verified online through Teranet Express. There is one patented land and three non-mining land patents within the current property boundary, that are not owned or optioned by CNC.

The *Ontario Mining Act* (2010) grants surface access to an unpatented mineral claim without owning the surface rights and given proper consultation with appropriate and relevant parties. Access to mining rights only patented lands or unpatented SCMC in which the CNC owns or has sub-surface rights only, requires that the surface rights owner be contacted in writing and that agreed-upon compensation be paid to the surface rights owner for any significant surface disturbances.

Annual holding costs for the 116 patented lands (mining tax) total approximately $37,557 and the required annual assessment work for the unpatented lands is approximately $35,200. Patented lands have an annual due date of approximately March 30 for payment of the mining land tax and related holding costs with payment due 60 days from invoicing, which is generally the end of January.

Based on the information provided by CNC and from what is available in the public domain, the authors confirm that all the unpatented and patented mining lands which comprise the Crawford Project are in good standing.

### 1.3.2 Royalties, Agreements and Encumbrances

On December 19, 2019, Noble announced that it had completed the acquisition of the 5% net smelter return royalty (NSR) applicable to ~55,000 ha of patented mineral rights on its Project 81 in the Timmins-Cochrane area of northern Ontario. As a result of doing so, those patented properties are now subject to a 2% NSR (Noble news releases dated

October 24, 2019, and November 28, 2019). The terms of this acquisition apply to the patented lands which were transferred to CNC (Noble news release dated December 3, 2019) and which comprise part of the current project.

A third party holds the rights to receive a 5% interest in the net profits (the NPI royalty) from Crawford. This royalty can be bought down to 2.5% for C$2 million.

## 1.4 Accessibility, Climate, Local Resources, Infrastructure & Physiography

The property is readily accessible year-round via the regional infrastructure network, and is bisected by Highway 655, which is a two-lane, paved, provincial highway. Access to the property is by a network of local bush roads accessed from Highway 655. Highway 655 connects to Highway 11, the Trans-Canada Highway located north of the site. The site is readily accessible from Timmins from the south, as well as the communities of Smooth Rock Falls and Cochrane to the north on Highway 11.

The Timmins Municipal airport (Timmins Victor M. Power Airport, YTS/CYTS) is serviced by several daily flights to Toronto Pearson International Airport and Billy Bishop Island Airport. There are no lakes proximal to the site that are large enough to offer float plane access. Freight rail access is available in Timmins and Cochrane.

Other local infrastructure includes a regional transmission line (500 kV), which parallels Highway 655 and brings power from the north through to the primary Hydro One Transmission Station located east of Timmins. In addition, a 115 kV transmission line is located east of the project site.

Approximately 17 km south of the project site there is a private rail spur operated by Ontario Northland Railway that connects the Kidd Mine with the federal railway network that provides access to the North American railway system.

The site has cold winters and warm summers typical of northeastern Ontario. Maximum daily average temperatures occur in July and lowest daily average temperatures occur in January. The annual temperature has a statistically significant upwards trend of 0.2°C per decade. Most precipitation occurs in the summer and fall months with July being the wettest month (91.7 mm) and February the driest month (42.3 mm). The annual rainfall has a statistically significant downwards trend of -15.6 mm per decade, indicating a shift to drier conditions (Golder, 2022).

## 1.5 History

The first aeromagnetic survey was completed in 1955, followed by a 1956 Geological Survey of Canada aeromagnetic survey, and a 1964 Geological Survey of Canada aeromagnetic survey. All three magnetic surveys showed a large, roughly circular, strongly magnetic high zone in the east-central part of the township that was interpreted to be an ultramafic rock mass (i.e., the Crawford Ultramafic Complex or “CUC”).

The 1963 discovery of the rich base metal deposit in Kidd Township (Kidd Creek Mine), about 15 km south of the CUC, led to a flurry of exploration in Crawford Township through the latter 1960s and the 1970s. The first exploration recorded in Crawford Township dates to 1964. The International Nickel Company of Canada led the way in exploring the township during the 1960s with multiple drillholes testing numerous anomalies (magnetic-electromagnetic or MAG-EM) anomalies. Anomalous base and precious metal (Cu, Zn, Pb, Ag) results were reported from intermediate to felsic volcanic rocks and long intersections (e.g., 236 m) of nickel (e.g., 0.25-0.40% Ni) in peridotite with very low sulphide content were noted (e.g., Skrecky, 1971). McIntyre Porcupine Mines Ltd. dominated exploration in the township during the 1970s with exploration waning significantly through the 1980s and thereafter.

There are at least 26 historical drillholes reported in Lucas Township which include five diamond drillholes and 21 reverse circulation (RC) drillholes (ODHD, 2023). These drillholes were completed in the 1980s by Abitibi-Price Mineral Resources (diamond and RC holes; MENDM Assessment File 42A14SE0131) and Kidd Creek Mines Ltd. (RC holes only, MENDM Assessment File 42A14SW).

Based on information that is available in the public domain, no significant work has been conducted in the project area since the 1980s.

## 1.6 Geology and Mineralization

### 1.6.1 Geology

The Crawford Project is situated in Northeastern Ontario, in the western portion of the mineral-rich Abitibi Greenstone Belt (AGB) (2.8 to 2.6 Ga), which is within the Superior Province, Canada. The AGB of the Abitibi Subprovince spans across the Ontario-Quebec provincial border and is considered the largest and best-preserved greenstone belt in the world (Jackson and Fyon, 1991; Sproule et al., 2003), covering an area approximately 700 km from the southeast to northwest and 350 km from north to south, constituting several major east-trending successions of folded volcanic and sedimentary rocks, with associated felsic to ultramafic intrusions. The supracrustal rocks of the AGB are uniquely well preserved and have mostly been overprinted only at a low metamorphic grade (Monecke et al., 2017). The economic importance of the AGB is substantial as it contains some of the main gold and base metal mining camps in Canada, as well as a long history of punctuated production from ultramafic extrusive komatiite-hosted Ni-Cu-(PGE) sulphide deposits.

More than an estimated 50% of the supracrustal rocks of the AGB, including those on the property, are under tens of metres of clay-dominated cover (referred to as the “Abitibi Clay Belt” or “Great Clay Belt” and formed from the lakebed sediments of Glacial Lake Ojibway), making mineral exploration challenging and expensive and hampering the discovery rate of new metal mines which preserves opportunities for discovery.

The AGB has been subdivided into nine lithotectonic assemblages or volcanic episodes (Ayer et al., 2002a, 2002b and 2005); however, the relationships between these assemblages are mostly ambiguous. Allochthonous greenstone belt models, with each terrane having been formed in a different tectonic environment, predict them to be a collage of unrelated fragments. Autochthonous greenstone belt models allow for the prediction of syngenetic mineral deposits hosted by specific stratigraphic intervals and formed within a structurally deformed singular terrane. Greenstone belts in the Superior Province consist mainly of volcanic units unconformably overlain by largely sedimentary “Timiskaming-style” assemblages, and field and geochronological data indicate that the AGB developed autochthonously (Thurston et al., 2008).

Proterozoic dikes of the Matachewan Dyke Swarm and the Abitibi Dyke Swarm intrude all the rock in the region. Matachewan dikes generally trend north-northwest while the younger Abitibi Dyke Swarm trends northeast.

### 1.6.2 Mineralization

The Abitibi Greenstone Belt affords a mineral exploration company with several target deposit types and commodities, including Ni-Cu-(PGE), VHMS, and orogenic gold.

Within Crawford Township, several prominent mafic-ultramafic intrusions (i.e., sills) offer the potential for sulphide-associated nickel, copper, cobalt, and platinum-group element (PGE) mineralization. Intrusion-hosted nickel sulphide mineralization is the principal target on the Crawford Project where it has been intersected by drilling, hosted in units of dunite and peridotite (largely serpentinized).

Mineralization has been drill-delineated in two principal zones of nickel sulphide mineralization, the Main Zone and the East Zone. Other areas of similar mineralization have been identified through diamond drilling by CNC, including the West Zone (Lane et al., 2022) and North Zone.

Within the Main Zone, drilling to date allows for the delineation of two higher grade (>0.30% Ni and >0.35% Ni) regions (modelled grade shells) within a larger core high-grade zone (>0.25% Ni), which in turn are within the larger enveloping low-grade zone (>0.15% Ni), all contained within the host ultramafic body of the CUC. The high-grade zone (>0.25% Ni) has a minimum modelled strike length of about 3.3 km, is between approximately 70 and 250 m wide, and contains regions of incrementally higher grade nickel (i.e., >0.30% Ni and >0.35% Ni). The high-grade zone and internal regions of higher grade nickel remain open along strike to the north-northwest and extend to a depth of at least 800 m (open at depth).

Located within the northern margin of the ultramafic to mafic body of the Main Zone, is a PGE Reef potentially 2 km long and up to 9 m wide, that is associated with a contact between a peridotite unit to the south and a pyroxenite unit to the north. Additional drillholes will be required to better define the PGE Reef.

Located about 1.2 km northeast of the Main Zone is the East Zone. The East Zone extends for about 2.6 km east-west, is open to the east but is truncated to the west by a large regional north-northwest trending fault. Like the Main Zone, mineralization in the East Zone is characterized by a relatively high-grade nickel core between approximately 70 and 200 m wide and at least 800 m deep (open at depth), within a low-grade nickel envelope and hosted by an ultramafic body comprising variably serpentinized dunite and peridotite, with lesser pyroxenite and minor gabbro.

Within the layered ultramafic unit of the East Zone, two PGE reefs have been differentiated: (1) a higher grade reef potentially 1.5 km long and up to 8 m wide at the northern boundary of the nickel-rich domain, and (2) a lower grade reef potentially 2 km long and up to 6 m wide, near the northern margin of the ultramafic body.

Although magmatic nickel sulphide (pentlandite) is present in the Crawford deposits, nickel grades have been significantly upgraded through the serpentinization (alteration) process, producing secondary minerals that are not magmatic in origin, such as heazlewoodite, awaruite, and godlevskite.

Where visible, nickel sulphide mineralization is dominated by fine-grained, disseminated pentlandite (magmatic) and heazlewoodite (hydrothermal). In general, pentlandite is the dominant nickel sulphide, except in regions of higher grade nickel (i.e., dunite core) where heazlewoodite is the dominant nickel species. This reflects a hydrothermal upgrading of nickel concentration in the CUC, also suggesting very low sulphur conditions. Overall, pyrrhotite occurs at about the same average percentage as awaruite but was identified in less of the drill holes. This suggests that although pyrrhotite occurs less often than awaruite, when it does occur, it is the dominant mineral phase of the two.

## 1.7 Deposit Types

Sulphide mineralization discovered to date in the Project area can be characterized as Komatiite-hosted Ni-Cu-Co-(PGE) deposit type, which recognizes two sub-types (Lesher and Keays, 2002): (1) Type I Kambalda-style and (2) Type II Mt. Keith-style.

Deposits of the Crawford Project, which closely resemble the Dumont nickel deposit, are most like the Type II, Mt. Keith-style of deposit, in that they are dominated by disseminated nickel sulphide mineralization (i.e., pentlandite). However, in the case of Crawford, the pervasive hydrothermal alteration (serpentinization), which led to widespread and significant upgrades in nickel concentration, is key to it being economic.

Sulphide mineralogy at Mt. Keith is dominated by disseminated (intercumulus) pentlandite, with minor millerite, violarite, and pyrrhotite, and trace amounts of pyrite, chalcopyrite, heazlewoodite, polydymite, and gersdorffite. Nickel sulphide mineralization strikes for 2 km, is about 350 m wide, and is open below 600 m depth. In 2002, the deposit had proven and probable reserves of 299 Mt grading 0.56% Ni (0.4% Ni cut-off) (Butt and Brand, 2003).

The type-example of the komatiite-hosted Ni-Co-PGE exploration model that CNC is using for the Crawford Ultramafic Complex is the Dumont nickel deposit of Dumont Nickel by Magneto Investments L.P., previously Royal Nickel Corporation (RNC), located 220 km to the east of Crawford Township, about 60 km northeast of Rouyn-Noranda, and within the Abitibi Greenstone Belt (Abitibi Region), northwestern Quebec (Ausenco, 2013).

The Dumont nickel deposit is usually categorized with its most similar counterpart, the Mt. Keith nickel deposit (Naldrett, 1989). Although the Dumont and Crawford nickel deposits share some similarities with Mt. Keith, it should be noted that nickel grades reported from reserves at Mt. Keith range from 0.48% to 0.57% Ni. In addition to its lower grade, the Crawford and Dumont deposits are differentiated from the Mt. Keith deposit by the abundance of secondary minerals heazlewoodite and awaruite. In addition, the Dumont and Crawford deposits have not been subjected to the extensive supergene weathering alteration which characterizes the Mt. Keith deposit.

Mineralization hosted by the Dumont Nickel Project and the Mt. Keith deposit is not necessarily indicative of mineralization hosted on the Crawford Project.

## 1.8 Exploration

CNC contracted Crone Geophysics and Exploration Services (G&E) to carry out a downhole electromagnetic survey on the property between June 1 and 7, 2022. One hole utilizing a transmitter loop was surveyed during this period. A 3D borehole pulse-electromagnetic (EM) system was assembled in which the axial component (Z) and cross-component (X-Y) of the induced secondary field were measured with a Crone borehole induction coil probe. The Z component detects any in-hole or off-hole anomalies and gives information on size, conductivity, and distances to the edge of conductors. The X-Y components measure two orthogonal components of the EM field in a plane orientated at right angles to the borehole. These results give directional information to the centre of the conductive body. Data is usually collected at a nominal sample interval of 10 m.

Drillhole CR22-230 was surveyed from a north-located loop offset 400 m from the collar to better couple with sub-vertical conductors at depth. Maximum coupling was achieved approximately 600 m down-hole. Only the bottom half of the hole was surveyed from 500 m to 1,100 m. The Z-component response suggests the hole is adjacent to a large conductor that decays rapidly with little remaining energy by channel 17 using a 16.66 ms time base. The non-decaying response as measured by S1 (the measured primary field reduced by the theoretical primary field) shows elevated responses peaking at 650 m, 790 m, and 960 m. The cross-components do not produce distinct crossovers at any of these peaks, however, suggesting the response is most likely due to variations in the proximity of the conductor edge relative to the drillhole at depth (i.e., pinching and swelling of a thick conductor). As the goal of the survey was to identify any massive sulphide lenses within the thick sequence of weakly conductive serpentinite, and since no obvious lenses were identified, there can be no further recommendations with this data set.

CNC is sponsoring two M.Sc. thesis studies on the Crawford Project through Laurentian University, Sudbury, Ontario. One thesis is looking to establish the in-situ paragenesis, textural relationships, and distribution of carbon minerals and carbon sequestering minerals within the CUC to provide a carbon content baseline for processing development studies. The other thesis aims to characterize the serpentinization of olivine and sulphide assemblages in the CUC, determine its relationship to nickel deportment and subsequent mineralization, and to help locate and evaluate similar deposits in the district and elsewhere.

## 1.9 Drilling and Core Sampling

Diamond drilling programs, which were initiated by Spruce Ridge in September 2018 under its option-joint venture agreement with then property owner, Noble Mineral Exploration (Noble), are ongoing. With the October 1, 2019, announcement that Noble had created a new entity, Canada Nickel Company (CNC), to focus on the Crawford Project, management and control of the drilling program shifted from Spruce Ridge to CNC.

As of April 2023, 372 drillholes totalling approximately 151,969 metres were completed by CNC and previous companies. This includes drilling metres (10,474 m) from 52 abandoned holes. Additionally, 54 holes were HQ size, completed for geotechnical and metallurgical testwork, whereas the remaining 266 drillholes used NQ size. Of these 372 drillholes, 322 were used to calculate the updated mineral resource estimates in the Main Zone and East Zone (see Section 14).

### 1.9.1 Main Zone Drilling

To date, diamond drilling has outlined a west-northwest trending ultramafic body (largely dunite-peridotite) that is at least 1.9 km in strike length, 280 m to 580 m in width, and more than 800 m deep. The focus of the ongoing 2019-2023 drilling is to extend along strike mineralization (see Section 1.9.3), to keep testing the east-northeastern and west-southwestern extents of mineralization (i.e., the contacts) as well as the deeper portions of the ultramafic body, and to complete in-fill drilling within the mineral resource envelope and its higher grade core.

### 1.9.2 East Zone Drilling

CNC began to drill-test the East Zone in late 2019 and into 2023 with relatively wide-spaced drillhole sections, followed by infill drilling in 2022-2023. To date, diamond drilling has outlined an east-west trending ultramafic body (largely dunite-peridotite) that is at least 2.6 km in strike length, 200 m to 350 m in width, and more than 800 m deep.

### 1.9.3 West Zone Drilling

In October 2020, CNC reported the discovery of previously unknown mineralization in four drillholes from the West Zone, with the first step-out hole located about 850 m northwest of the Main Zone.

Further and ongoing 2021-2023 drilling has successfully intersected mineralized dunite-peridotite, consistent with mineralization seen in the Main Zone, across a width of at least 800 m and strike length of at least 1 km.

### 1.9.4 North Zone Drilling

In December 2020, CNC drill tested previously unknown mineralization in four drillholes from the North Zone, located about 2 km northwest of the East Zone. Fifteen exploratory drillholes were completed between 2020 and 2022. All drillholes intersected the faulted continuation of the main dunite-peridotite body of the CUC and exhibited the same differentiation sequence.

## 1.10 Sample Preparation, Analyses and Security

CNC is responsible for the ongoing drilling and sampling program, including quality assurance and quality control (QA/QC). Since drilling began in 2019, the protocols followed by company personnel have changed slightly and are described below, current as of the effective date of the mineral resource estimate (August 31, 2023).

The core is marked and sampled at primarily 1.5-metre lengths and cut with diamond blade saws. Samples are bagged with QA/QC samples inserted into the sample stream (60 samples per lab batch) at a the rate of at least 1 QA/QC sample in every 20 principal samples. Samples (60 per lot) are transported in secure bags directly from the company core shack to Activation Laboratories Ltd. (Actlabs) in Timmins or by commercial truck transport (Manitoulin Transport Inc.) to SGS Canada Inc. (SGS) in Lakefield, Ontario. In general, the core recovery for the diamond drillholes on the property has been better than 95% and little core loss due to poor drilling methods or procedures has been experienced.

In the opinion of the authors, the assay data is adequate for the purpose of verifying drill core assays, estimating mineral resources, and for use in a feasibility study.

## 1.11 Data Verification

The authors have reviewed historical and current data and information regarding past and current exploration work on the property. Mr. John Siriunas (M.A.Sc., P.Eng.) visited the project on October 12, 2019 for one day, on February 3 to 4, 2020 for two days, on September 10 to 11, 2020 for two days, and on August 11 to 12, 2022 for two days. During the site visits, diamond drilling procedures were discussed and a review of the on-site logging and sampling facilities for processing the drill core was carried out.

It is the author's opinion that the procedures, policies, and protocols for drilling verification are sufficient and appropriate and that the core sampling, core handling, and core assaying methods used are consistent with good exploration and operational practices such that the data is reliable for the purpose of mineral resource estimation and for this report.

## 1.12 Metallurgical Testwork

The Crawford flowsheet will process serpentine ore to produce nickel and FeCr concentrates, while also storing $CO_2$ in the brucite component of the tailings. The concentrator comminutes the ore and separates the value minerals by a combination of flotation and magnetic separation processes. Tailings from these processes are thickened to a target 40% density and are then processed using CNC's in-process-tailings (IPT) carbonation process, which permanently fixes carbon dioxide ($CO_2$) in solid mineral form within the tailings. The carbonated tailings from the IPT Carbonation process are discharged and stored within the tailings management facility as a permanent repository for $CO_2$.

The key goals of metallurgical testing for the feasibility study were as follows:

- improve the metallurgical performance (i.e., metal recoveries and concentrate qualities) relative to the flowsheet detailed in the PEA
- develop process design criteria to support the plant design basis
- improve confidence in production forecasts, with a focus on ore that is processed during the project payback period
- quantify the carbon sequestration potential of the tailings to support $CO_2$ sequestration forecasts.

Flowsheet optimization for the flotation and magnetic recovery circuits delivered improvements in recoveries (Table 1-1) and concentrate qualities (Table 1-2) across all commodities over the life of mine when compared with the PEA.

**Table 1-1: Feasibility Study vs. PEA – Life-of-Mine Recovery Comparison**

| Description | Ni | Co | Fe | Cr |
|---|---|---|---|---|
| Feasibility Study[1] | 41.2% | 11% | 52.7% | 28.2% |
| PEA[2] | 37.3% | 8%[3] | 36.1% | 27.0% |
| **Difference** | **3.9%** | **3%** | **16.6%** | **1.2%** |

Notes: 1. Feasibility study life of mine is 41 years. 2. PEA life of mine was 25 years. 3. Cobalt was not a payable metal in the PEA.

**Table 1-2: Feasibility Study vs. PEA – Nickel and FeCr Concentrate Grade Comparison**

| Description | Nickel Concentrate – Ni Grade | FeCr Concentrate – Fe Grade |
|---|---|---|
| Feasibility Study | 34.2% | 55.0% |
| PEA | 16.0%[1] | 47.5% |
| **Difference** | **18.2%** | **7.5%** |

Notes: 1. In the PEA, a high-grade (35% nickel) and lower grade (12% nickel) concentrate were produced. The stated 16% nickel grade is a weighted average grade of the combined concentrate.

### 1.12.1 Mineralogy Studies

The mineralogy was a critical part of establishing the resource estimate, as nickel can exist in recoverable forms as minerals such as heazlewoodite, pentlandite, awaruite, and millerite, or nickel can be hosted within the matrix of silicate minerals. Silicate-hosted nickel is not recoverable by flotation, except through gangue entrainment within the final concentrate products.

Within the feasibility study mined reserve, 2,701 samples were subjected to mineralogical characterization using QEMSCAN. The main mineral at Crawford is serpentine, which has been divided into two subcategories called "iron serpentine" and "magnesium serpentine." Magnesium serpentine is typically present in more altered ores with higher recoverable mineral content. The mineralogy and microprobe data supports the production of high-grade concentrates, as Crawford is low in iron sulphide minerals such as pyrrhotite and contains high nickel tenor minerals such as heazlewoodite (71.8% Ni) and awaruite (73.4% Ni).

Brucite is a mineral that is a strong marker of the $CO_2$ sequestration potential of the ore. To support $CO_2$ sequestration forecasts, the brucite content of characterized samples was incorporated in the resource block model through kriging (Tables 1-8 and 1-9 in Section 1.13).

### 1.12.2 Comminution Testwork

The Crawford comminution circuit utilizes a semi-autogenous grinding (SAG) mill for primary grinding and a ball mill for further size reduction. The key metrics from this testwork that were used for the plant design bases were the Axb parameters from JK drop weight and SMC tests, as well as the Bond ball mill work indices.

Comminution testing was completed at SGS Lakefield on 83 samples from the key lithologies, alteration styles, and across a range of grades to evaluate variability in parameters that affect mill throughput. Most of the Axb results sit in a narrow range between 51 and 67, and most of the BWi results sit in a narrow range between 18.8 and 22.2 kWh/t. As the breakage characteristics are relatively consistent across the resource, the project risk was managed in design by using the $75^{th}$ percentile value.

### 1.12.3 Metallurgical Variability Testwork

Metallurgical variability testing was completed using a standard test procedure to understand variability in recovery and concentrate quality. Samples were selected to be representative of ore that would be processed early in project life and were selected from the key lithologies, alteration styles, and zones.

A total of 126 open circuit tests were conducted at XPS, Expert Process Solutions and COREM in parallel using a standard test procedure. Included in the test database are six synchronizing tests that were done at both laboratories on the same sample, and which demonstrated reproducibility in results between the labs. The ore sulphur grade was determined to be the key driver of nickel, cobalt, and precious metal recoveries for the project, while the magnetic susceptibility was determined to be the key driver of iron and chromium recovery. Recovery equations were developed based on the open circuit test results, which were then confirmed in locked cycle tests.

Locked cycle tests simulate continuous operation and are indicative of product grades and recoveries that would be achieved in a plant setting. Fifteen samples were subjected to locked cycle tests to evaluate the impact of recirculating streams on product grades and recoveries, and to inform the design of recovery equations. The total nickel recovery for locked cycle tests, which comprises the nickel that was recovered to both the nickel and FeCr concentrates, ranged from 30% to 63%. Nickel concentrate grades were a function of the sample's sulphur-to-nickel ratio and ranged from 18% to 46% nickel. Locked cycle testing completed using the final testwork flowsheet confirmed the ability to achieve the modelled FeCr concentrate grade of 55% iron.

Pilot plant testing was completed at SGS Lakefield in the second half of 2022 on six composites, with nickel head grades ranging from 0.20% to 0.33%. Due to mass and equipment limitations, it was decided to pilot up to the cleaner 1 flotation stage. Piloting of the full magnetic recovery circuit was completed. However, it was done in several stages. The pilot results were excellent and confirmed the ability to stabilize and control the process around target tailing grades. Recoveries were generally in line with expectations established from open circuit testing. However, chromium recoveries in the pilot plant exceeded expectations relative to the laboratory. The flotation circuit delivered higher grades than expected and the FeCr concentrate grade was in line with the modelled values.

### 1.12.4 Recovery Equations

Recovery equations estimate the recovery to the two concentrate products to support the mine design, as well as financial and production forecasts.

Four nickel recovery domains were established based on the analysis of open circuit test results, which are summarized in Table 1-3. To capture improvements made to the magnetic recovery circuit during variability testing, a nickel flowsheet evolution adjustment factor was applied to the total nickel recovery across all domains, which ranges from 1.1% to 1.3% absolute depending on the feed sulphur grade. After applying the nickel adjustment factor, a nickel splitter model is applied to estimate the deportment of nickel between the two concentrates. The open circuit test recovery definition was found to be representative of the final nickel recovery for the locked cycle tests on common samples. Based on this finding, the open circuit test recovery definition was used without applying any cleaning correction factors.

**Table 1-3: Nickel Recovery Domains and Equations**

| Metallurgical Domain Description | Magnetic Susceptibility | Zone | Ore Domain | S Grade Range | Nickel Recovery Equation |
|---|---|---|---|---|---|
| East Zone | >70 | East | All | S ≤ 0.07 | $Ni\ Rec = 23.743 * e^{14.915*(S)}$ |
| | | | | S > 0.07 | Ni Rec = 57% |
| Main Zone High-Grade Core | >70 | Main | HGC | S ≤ 0.25 | $Ni\ Rec = 28.292 * e^{3.0649*(S)}$ |
| | | | | S > 0.25 | $Ni\ Rec = 55\%$ |
| Main Zone Lower grade Domain | >70 | Main | LG | S ≤ 0.08 | $Ni\ Rec = 19.41 * e^{15.423*(S)}$ |
| | | | | S > 0.08 | Ni Rec = 53% |
| Low Magnetite Domain | <70 | Main, East | All | All | $Ni\ Rec = -5.35 * (Fe) + 19.67 * LN\left(\frac{S}{Ni}\right) + 85.27$ |

The Crawford flowsheet also concentrates cobalt, iron, chromium, and platinum group metals as byproducts. Recovery models were developed to predict the cobalt and platinum group metal recovery to the nickel concentrate, as well as the iron and chromium recoveries to the FeCr concentrate.

### 1.12.5 Concentrate Quality

Concentrate quality models were developed to estimate the grades of payable and deleterious elements within the concentrate over the life of mine.

#### 1.12.5.1 Nickel Concentrate Grade Modelling

The nickel concentrate from Crawford is the product of the flotation circuit. The concentrate contains nickel sulphide minerals, including pentlandite and heazlewoodite, and the grade is dependent on the type of ore being processed. With a life-of-mine grade of 34% nickel, 0.7% cobalt, and 11% MgO, the Crawford nickel concentrate would be the highest grade concentrate available on the market. This concentrate would be a suitable feed for the battery market or, if roasted to eliminate the sulphur, a potential feed for steel-making.

Table 1-4 summarizes the forecast nickel concentrate grades over the life of the mine. In addition to the high nickel grades, the concentrate produced is clean, with below detection limit levels of lead and arsenic, and low levels of copper and phosphorous.

**Table 1-4: Nickel Concentrate – Expected Life-of-Mine Grades**

| Ni[1] (%) | Co[1] (%) | Pt[1] (g/t) | Pd[1] (g/t) | S[2] (%) | Fe[2] (%) | Cr[4] (%) | MgO[2] (%) | $SiO_2$[3] (%) | CaO[4] (%) | Mn[4] (%) | Cu[4] (%) | P[4] (%) | As[4] (%) | Zn[4] (%) | Pb[4] (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 34 | 0.65 | 0.9 | 3.1 | 17 | 17 | 0.3 | 11 | 11 | 0.16 | 0.05 | 0.21 | 0.03 | <0.03 | 0.05 | <0.02 |

Notes: **1.** Grades are modelled using recovery and grade models developed from metallurgical variability test results. **2.** Grades are modelled as a weighted average of typical high and lower grade concentrate specifications. **3.** Estimated based on relationship between MgO and $SiO_2$ in the nickel concentrate from detailed concentrate quality analysis. **4.** Grades are estimated based on detailed concentrate quality analysis. Values presented are averages from 11 locked cycle test results.

#### 1.12.5.2 FeCr Concentrate Grade Modelling

The FeCr Concentrate is produced from the magnetic recovery circuit in the Crawford metallurgical flowsheet which is designed to recover the minerals awaruite ($Ni_{2.5}Fe$), magnetite ($Fe_3O_4$), and chrome spinel. The FeCr concentrate contains substantial quantities of nickel, iron, and chromium, which makes it a suitable feed stock for steel-making. Through analysis of locked cycle, open circuit, and pilot plant results, grade models were developed for iron, chromium, nickel, sulphur, and MgO in the concentrate.

Table 1-5 summarizes the forecast detailed specifications of the FeCr concentrate, which is based on the analysis of products from locked cycle tests as well as predictive grade models. The iron grade of the FeCr concentrate was conservatively modelled as a constant 55% iron over the life of mine. The nickel and chromium grades make the concentrate a potential direct feed for the stainless steel industry. Like the nickel concentrate, the FeCr concentrate is low in contaminants including copper, sulphur, phosphorous, arsenic, lead, and zinc. The MgO grade is high and will need to be considered when selecting downstream process routes.

**Table 1-5: FeCr Concentrate – Detailed Composition[1] (%)**

| Fe[1] | Cr[1] | Ni[1] | Co[2] | Mn[2] | Cu[2] | Zn[2] | S[1] | P[2] | MgO[1] | $Al_2O_3$[1] | $SiO_2$[1] | CaO[2] | As[2] | Na[2] | Ti[2] |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 55 | 2.6 | 0.27 | 0.03 | 0.26 | 0.05 | 0.02 | 0.04 | <0.02 | 9.3 | 1.6 | 7.4 | 0.1 | <0.005 | <0.02 | 0.05 |

Notes: **1.** Grades are modelled using recovery and grade models developed from metallurgical variability test results. **2.** Grades were calculated as averages from detailed analysis of locked cycle test products completed using the final feasibility testwork flowsheet.

### 1.12.6 IPT Carbonation Testwork

In-process tailings (IPT) carbonation is a novel $CO_2$ sequestration process developed by CNC where tailings generated by the milling process are conditioned with a concentrated source of $CO_2$ after tailings thickening and before discharge to the TMF. $CO_2$ delivered to site is sparged into the tailings slurry and reacts with the minerals to form carbonates which results in permanent sequestration of the $CO_2$. The carbonation tanks have been designed such that unreacted $CO_2$ in the head space is recompressed and recirculated to maximize $CO_2$ utilization.

A standard test procedure was developed and applied to 16 samples from Crawford with varying brucite content. It was found that the $CO_2$ sequestration potential of tailings was highly correlated with the sample brucite content. Table 1-6 states the model equation used to predict $CO_2$ sequestration from IPT carbonation.

Pilot plant testing of the IPT carbonation process was done in summer 2023, which confirmed the ability to scale up the process as well as the ability to store in excess of 1 Mt/a of $CO_2$. Pilot plant testing also validated the sequestration models that were developed.

**Table 1-6: IPT Carbonation – $CO_2$ Sequestration Capacity Model**

| Equation | Min (kg $CO_2$/t Tailings) | Max (kg $CO_2$/t Tailings) | $R^2$ |
|---|---|---|---|
| $CO_2$ Storage Capacity (kg $CO_2$/t tailings) = 19.7*LN(QEMSCAN Brucite) + 25.7 | 0 | 62 | 88% |

## 1.13 Mineral Resource Estimation & Statement

The measured, indicated, and inferred mineral resources presented herein are constrained by an RL100 pit shell generated using measured and indicated resources only, the same parameters used for the mine design pit optimization. The effective date of the mineral resource estimates is August 31, 2023. Pit-constrained, class-characterized mineral resources at a 0.10% Ni cut-off grade with respect to the higher and lower-grade nickel estimation domains within the East Zone and the Main-West Zone are presented for all elements studied in Tables 1-7 and 1-8. Note: The mineral resources stated herein are not mineral reserves, as they do not have demonstrated economic viability.

[1] MgO and $SiO_2$ grades were modelled as a function of the concentrate iron grade based on analysis of test results. $Al_2O_3$ was modelled as a function of the concentrate chromium grade.

**Table 1-7: Summary of the Pit-Constrained Updated East Zone Mineral Resource Estimate**

| Class | Tonnage (Mt) | Grade | | | | | | | | Contained Metal | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Ni (%) | Co (%) | Fe (%) | S (%) | Cr (%) | Brucite (%) | Pd (g/t) | Pt (g/t) | Ni (kt) | Co (kt) | Fe (Mt) | S (kt) | Cr (kt) | Brucite | Pd (koz) | Pt (koz) |
| Measured + Indicated | 1,034.6 | 0.23 | 0.013 | 6.31 | 0.06 | 0.60 | 1.25 | 0.012 | 0.009 | 2,363.0 | 129.6 | 65.3 | 646.5 | 6,225.9 | N/A | 406.1 | 289.9 |
| Inferred | 157.3 | 0.23 | 0.013 | 6.25 | 0.07 | 0.60 | 0.98 | 0.010 | 0.007 | 367.1 | 20.0 | 9.8 | 113.3 | 939.8 | N/A | 50.9 | 37.1 |

Source: Caracle Creek, 2023.

**Table 1-8: Summary of the Pit-Constrained Initial Main-West Zone Mineral Resource Estimate**

| Class | Tonnage (Mt) | Grade | | | | | | | | Contained Metal | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Ni (%) | Co (%) | Fe (%) | S (%) | Cr (%) | Brucite (%) | Pd (g/t) | Pt (g/t) | Ni (kt) | Co (kt) | Fe (Mt) | S (kt) | Cr (kt) | Brucite | Pd (koz) | Pt (koz) |
| Measured + Indicated | 1,527.2 | 0.24 | 0.013 | 6.91 | 0.07 | 0.58 | 2.33 | 0.016 | 0.010 | 3,672.3 | 200.6 | 105.6 | 1,041.9 | 8,840.5 | N/A | 782.6 | 492.8 |
| Inferred | 1,535.8 | 0.22 | 0.013 | 7.16 | 0.04 | 0.57 | 2.21 | 0.011 | 0.009 | 3,359.2 | 202.0 | 110.0 | 578.7 | 8,734.1 | N/A | 542.7 | 459.0 |

Source: Caracle Creek, 2023.

### 1.14 Mineral Reserve Estimation & Statement

Mineral reserves were classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). The effective date of the mineral reserves estimates is August 31, 2023. Mineral reserves include measured resources that are classified as proven reserves and indicated resources classified as probable reserves. Mineral resources are inclusive of mineral reserves.

Reserves are contained within an engineered pit design based upon a Lerchs-Grossmann (LG) pit optimization run at a revenue factor (RF) 65% of the base case prices, or $13,650/t Ni, $26,000/t Co, $58/t iron ore, $2,500/t Cr, $878/oz Pd, and $748/oz Pt.

Crawford mineral reserves are summarized in Table 1-9.

**Table 1-9: Crawford Mineral Reserves Summary**

| Description | Ore (Mt) | Grade | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | Ni (%) | Co (%) | Pd (g/t) | Pt (g/t) | Fe (%) | Cr (%) | Brucite (%) |
| Proven | 994 | 0.24 | 0.013 | 0.016 | 0.010 | 6.37 | 0.59 | 1.75 |
| Probable | 721 | 0.20 | 0.012 | 0.012 | 0.009 | 6.53 | 0.54 | 1.41 |
| Proven + Probable | 1,715 | 0.22 | 0.013 | 0.014 | 0.009 | 6.44 | 0.57 | 1.61 |
| **Description** | **-** | **Contained Metal** | | | | | | **Mt $CO_2$ Capture** |
| | | **Ni (kt)** | **Co (kt)** | **Pd (koz)** | **Pt (koz)** | **Fe (Mt)** | **Cr (kt)** | |
| Proven | - | 2,345 | 125 | 498 | 311 | 63 | 5,892 | 33 |
| Probable | - | 1,444 | 89 | 278 | 208 | 47 | 3,895 | 22 |
| Proven + Probable | - | 3,789 | 215 | 777 | 519 | 110 | 9,787 | 54 |

Notes: *Reported at a cut-off value of C$9.00 per pound and within an engineered pit design based on an LG optimized pit shell using metal prices of 13,650/t Ni, $26,000/t Co, $58/t iron ore, $2,500/t Cr, $878/oz Pd and $748/oz Pt; average metallurgical recoveries of 41% Ni, 11% Co, 53% Fe, 28% Cr, 48% Pd and 22% Pt; marginal processing and G&A costs of $6.10/t milled; a long-term exchange rate of C$1.00 equal $0.76; overall pit rock slopes of 43° to 54° depending on the sector; and a production rate of 120 kt/d. Mineral reserves include unplanned dilution of 2.0%. The proven reserves are based on measured resources while probable reserves are based on indicated resources. All figures are rounded to reflect the relative accuracy of the estimates. Some error in totals may be present due to rounding.

### 1.15 Mining Methods

#### 1.15.1 Hydrogeological Considerations

Both overburden material and bedrock are observed to contain water-bearing horizons within the site stratigraphy. The confined aquifers below the surficial clay vary widely in depth and tend to be thicker where bedrock depth is deeper, ranging in depth from a few metres to tens of metres thick. As described elsewhere in this report, the bedrock environment consists of metavolcanics, peridotite, pyroxenite, dunite, and gabbro, all of which were targeted for hydrogeological testing as part of drilling campaigns. Field hydrogeological data collection (including bedrock packer testing and overburden slug testing), data analysis, and numerical groundwater modelling were completed as part of feasibility study work to inform this technical report.

Results of the numerical groundwater model show that, while dewatering the open pit is required, dewatering can be achieved using conventional dewatering means. For the Main Zone pit, dewatering rates range from about 1,850 $m^3$/d, estimated during Years 1 to 5 (development Stage 1 of the mine life) to 9,963 $m^3$/d, estimated during Years 8 to 9 (development stage of the mine life). For the East Zone pit, dewatering rates range from about 1,690 $m^3$/d, estimated during Year 19 (development Stage 3 of the mine life) to 16,983 $m^3$/d (estimated during Years 1 to 3 (development Stage 1 of the mine life).

The groundwater model is also used to:

- assess pore water pressures under certain pit slope conditions (see Section 16.2, Geotechnical Considerations)
- consider the potential influence of nearby infrastructure according to the existing planned site layout.

### 1.15.2 Geotechnical Considerations

#### 1.15.2.1 Soil Geotechnical

Geotechnical investigations were conducted in various zones around the mine project. These investigations revealed a diversity of soil types and characteristics, such as the presence of organic soil, glaciolacustrine clay, and sand and till layers. The organic soil, primarily consisting of fibrous peat, covers much of the project area, while the glaciolacustrine clay varies in thickness and type (stiff brown and soft grey). Below this clay lies a layer of sand and till, which varies significantly in thickness and is situated atop the bedrock.

The groundwater level at the site averages around 2.5 metres below the surface. The strength of the overburden material varies, with the grey, normally consolidated clay being identified as the weakest. This layer's undrained shear strength ratio was found to be 0.30 based on CPT results. The sand and till materials beneath the clay layer exhibit similar geotechnical properties and were combined into a single unit for analysis. The esker, a significant geological feature, was also investigated.

Overall, the project area's geology is characterized by a complex interplay of various soil types and formations, each with distinct geotechnical properties. This complexity requires detailed analysis and consideration in the planning and execution of mining activities to maintain stability and safety.

Stability criteria for the overburden pit slopes were developed, considering various soil layers and groundwater levels. These analyses included both short-term (during excavation) and long-term stability assessments, incorporating the area's seismicity. Results indicated that overall pit slopes in clay should have a maximum 6H:1V inclination, while those in sand and till should have a maximum overall inclination of 5H:1V. Bench designs for excavation are based on these slope guidelines.

The project involves the construction of two stockpiles and an impoundment facility. The stockpiles and impoundment facility are located at safe distances from the open pit, mine plant, existing structures, and water ponds.

The east stockpile will be established first and is designed for a maximum capacity of 100 $Mm^3$, reaching a height of 100 m in its 20th year. The west stockpile, with a larger footprint, is designed for a maximum of 165 $Mm^3$, reaching a maximum height of 70 m.

The impoundment facility, spanning 31.5 $km^2$, includes three distinct facilities: a clay impoundment facility, a sand and till impoundment facility, and a rock impoundment facility. Each facility has specific design features, as follows:

1. The clay impoundment facility, retaining 205 $Mm^3$ of clay, is surrounded by 34-metre-high rock berms with different internal and external slopes. It will be divided into 28 cells separated by high rock ribs, which also serve as haulage roads.
2. The sand and till facility, designed for 182 $Mm^3$ of materials, will have a final height of 50 m with slopes inclined at 6H:1V.
3. The rock impoundment facility, adjacent to the sand and till facility, is expected to accommodate 1,438 $Mm^3$ of waste rock and reach a height of 110 m.

The stability criteria for these structures are based on existing guidelines and recommendations, with deformation analyses assessing the impact of the rate of rise on stability. The design accounts for pore water pressure build-up in the clay foundation, with maximum rates of rise of 5 metres per year for the rock impoundment and 2.5 metres per year for the other facilities.

#### 1.15.2.2 Rock Geotechnical

The Crawford deposit is mainly comprised of strong metavolcanic and gabbro rock masses forming the hangingwall and footwall, in general. The rock masses forming the ultramafic mineralization package, comprised of peridotite, pyroxenite and dunite, are generally weaker. Within the ultramafic mineralization package some of the dunite unit modelled to intersect the Main Zone pit is known to contain coalingite which is susceptible to a degradation process when exposed to the atmosphere. Potential rock mass failures on a bench scale are anticipated to control the pit stability and unravelling on the bench scale is likely to occur in this region.

Talc zones are modelled to intersect the east side of the East Zone pit and the east and northwest sides of the Main Zone pit within the ultramafic units. Future investigations should investigate the possible extension of these talc zones along the footwall of the mineralization package.

The findings of the structural geology assessment were consistent with structures expected to occur in a strike slip system, with steep near-vertical features splaying from one another and more moderately dipping structures accommodating movement between the near vertical structures. The structural model is based on widely spaced drillholes and no surface exposure and therefore is low confidence.

A major regional strike slip fault (i.e., the CUC fault) runs along the west side of the East Zone pit, through the proposed saddle between the pits, to the east side of the Main Zone pit. The CUC fault core is prone to slake and swell upon exposure which is anticipated to impact the bench scale and possibly the inter-ramp performance. Stiffness in the CUC fault could be critical to the performance of the upper southwest slope of the East Zone and the upper northeast slope of the Main Zone. Early emphasis on confirming CUC fault properties will be critical.

The rock geotechnical domains and slope designs are based on the lithological and alteration models. Dominant controls on the pit slope stability are expected to be kinematics in areas comprising non-ultramafic units and therefore overall slope angles were primarily controlled by bench stability, except where impacted by the CUC fault. Slope designs in areas comprised of ultramafic units were informed by structural controls and potential rock mass failures these units were largely controlled by multi-bench or overall stability. If carried too steep these could impact the overall slope.

### 1.15.3 Mine Design

The Main Zone and East Zone pits contain approximately equal tonnages of ore. The current design and sequence of the pits allows for in-pit impoundment of tailings to be maximized, thus minimizing the size of a tailings management facility (TMF) and associated environmental impacts. In-pit deposition of tailings will commence after the Main Zone is depleted in Year 17. Mining of the East Zone continues through Year 30. For the final 11 years of operation, the mill will be fed from two stockpiles of lower value ore.

Over the life of mine, 161 $Mm^3$ of various materials will be required for reclamation of disturbed areas and construction of infrastructure. Over 99% of this material will be waste from the Crawford pits. The remaining 1,825 $Mm^3$ of waste will be impounded in one large facility located north of the pits that is divided into separate zones for clay, sand and till, and rock.

Mining will be performed by a mixed fleet of equipment as follows:

- Small backhoe excavators and articulated trucks will mine 4% of total material off a clay footwall.
- 300 t face shovels and 90 t trucks will mine 6% of total material off a sand and till footwall.
- The remaining 90% of material mined from a rock footwall will be loaded by large (50 t payload) front-end loaders, 700 t face shovels, or rope shovels into 290 t trucks equipped with trolley assist and an autonomous haulage system (AHS).

## 1.16 Recovery Methods

The process plant incorporates a staged expansion to deploy capital efficiently, allowing for throughput to be increased over the life of mine as presented below:

- Phase 1 (Years 1 to 3) will have a design capacity of 60 kt/d or 21.9 Mt/a.
- Phase 2 expansion will duplicate the Phase 1 design for a total capacity of 120 kt/d or 43.8 Mt/a by Year 5 of operation.

The selected flowsheet includes two-stage crushing and stockpiling. The ore is then reclaimed into a grinding circuit consisting of a SAG mill and a ball mill circuit (SAB) operating in closed circuit with a cyclone cluster. The cyclone overflow is deslimed in a cyclone cluster whose overflow reports to tailings and underflow to coarse rougher and cleaner flotation stages. The first cleaner concentrate is reground in a dedicated regrind mill for further size reduction prior to subsequent cleaner flotation for upgrading.

The coarse flotation tailings are ground in a closed-circuit ball mill and then deslimed before fines rougher and cleaner flotation stages. The first cleaner concentrate is also reground before subsequent cleaning stages. Both flotation concentrates report to a common thickener and filter for dewatering before stockpiling. The fines flotation tailings are further processed through three stages of magnetic separation, with a dedicated regrind mill for further size reduction between the stages. The magnetic concentrate is processed through a sulphide flotation stage, the concentrate is dewatered, filtered before stockpiling.

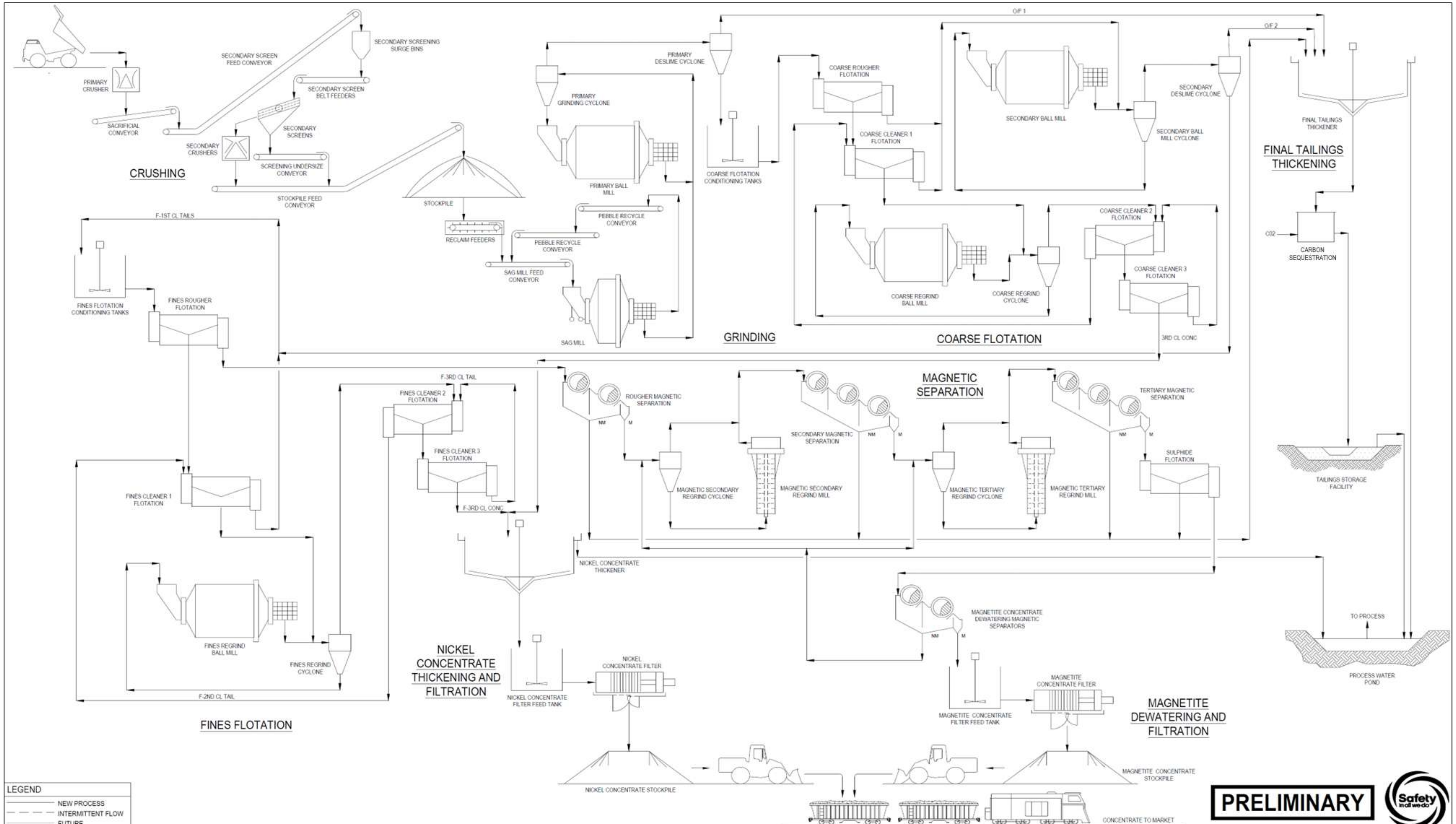
The tailings from the magnetic separation process are combined with the deslime cyclone overflows and thickened. The thickened tailings are then pumped through tanks, where carbon dioxide is introduced for capture and sequestration in the tailings facility.

A summary of the key design and performance criteria is presented in Table 1-10. The simplified flowsheet for Phase 1 is shown on Figure 1-1 on the following page.

**Table 1-10: Process Plant Key Design Criteria**

| Criteria | Units | Value |
|---|---|---|
| Operating Availability | | |
| Crushing | % | 75.0 |
| Grinding, Flotation and Magnetic Recovery | % | 91.3 |
| Concentrate Filtration | % | 84.0 |
| Plant Feed Grades, Design | | |
| Nickel | % | 0.28 |
| Chromium | % | 0.69 |
| Sulphur | % | 0.27 |
| Iron | % | 7.42 |
| Recoveries, Design | | |
| Nickel Recoveries | % | 47.4 |
| Iron Recoveries | % | 51.6 |
| Concentrate Nickel Grades, Design | % | 24.3 |
| Concentrate Grades | | |
| Nickel Concentrate, Design | % | 24.3 |
| Nickel Concentrate, Life-of-Mine Average | % | 34.2 |
| Iron Concentrate, Design | % | 51.6 |
| Iron Concentrate, Life-of-Mine Average | % | 55.0 |

**Figure 1-1: Process Flowsheet**




Source: Ausenco, 2023.

### 1.17 Project Infrastructure

Infrastructure to support the Crawford Project will consist of mining pits and storage facilities, a tailings management facility, site civil work, buildings, water management systems, existing roads and railways, new access roads, and site electrical services. The site layout is shown in Figure 1-2.

#### 1.17.1 Process Plant Infrastructure

For Phase 1 of operations, the following will be built at the crushing area near the open pits:

- primary crusher vault and building
- secondary crusher building.

The following will be built at the process plant site:

- crushed ore stockpile and cover
- process plant building
- concentrate load-out building
- plant warehouse
- plant office
- plant maintenance building.

For the Phase 2 expansion, a new processing line will be built, comprising primary crusher, secondary crusher, crushed ore stockpile cover, process plant and concentrate loadout buildings. The plant warehouse and maintenance buildings will be expanded to handle the additional activities required.

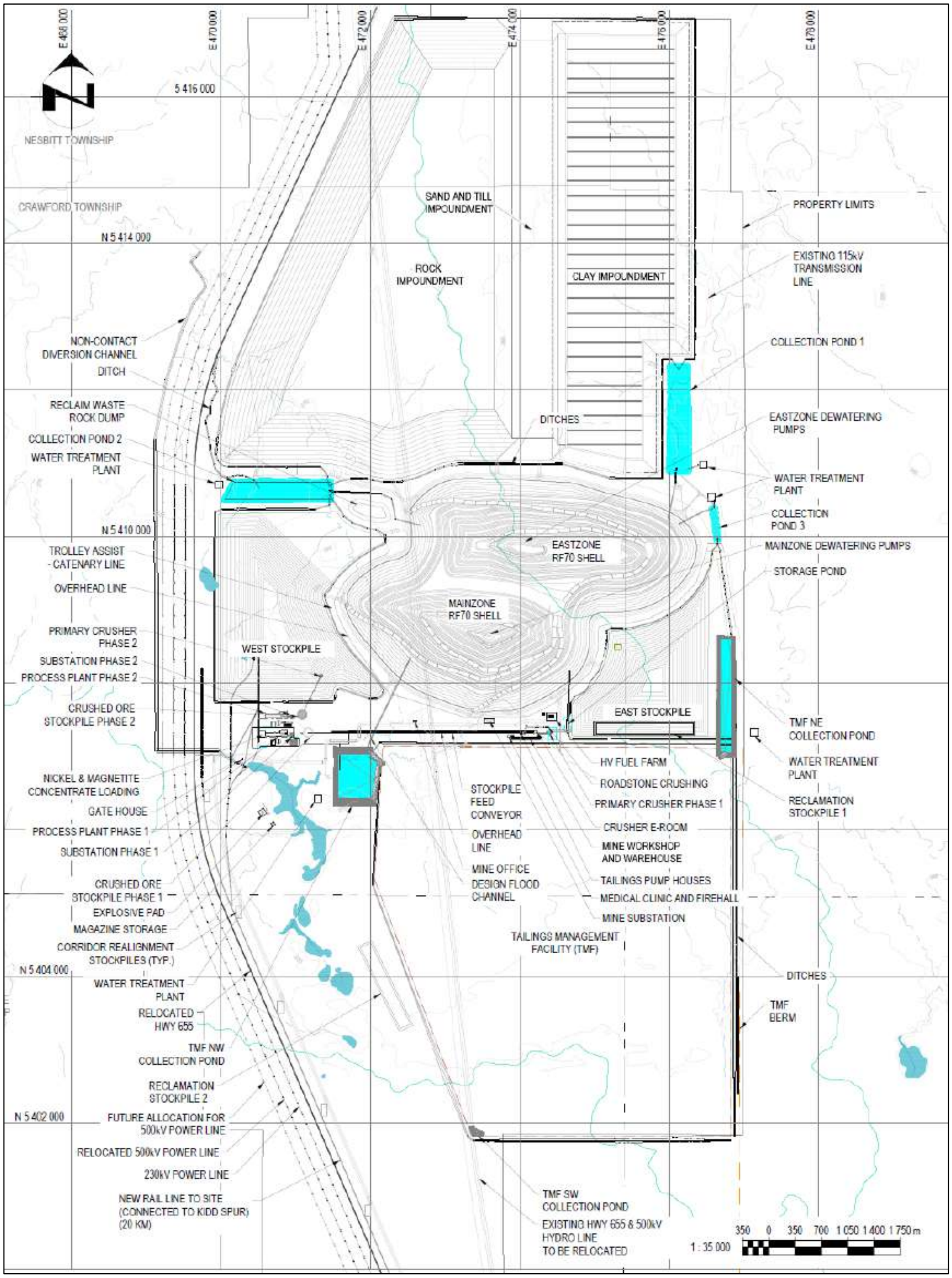
Secondary roads will be added within the plant footprint to access process plant facilities.

#### 1.17.2 Mining Building Infrastructure

The following will be built for mining support:

- a mine workshop for servicing and washing the mine fleet (built for Phase 1, and expanded for Phase 2 and afterwards)
- a mine warehouse, built adjacent to the workshop, for servicing and storing equipment parts
- a mine workshop office, built adjacent to the mine workshop, for mine workshop personnel
- a mine office, built adjacent to the pit south exit ramp, for management, technical, and administrative personnel.

**Figure 1-2: Site Layout Plan**




Source: Ausenco, 2023.

### 1.17.3 Surface Water Management

The objective of surface water management is to protect groundwater and surface water resources. Feasibility Study infrastructure and catchments for the projects were delineated based on topography data and footprints of facilities.

Contact and non-contact water are managed separately for the project. There are a number of small lakes within the site property that drain through proposed mine infrastructure. Non-contact diversions will be required to divert these flows away from mining activities. Surface water runoff that comes into contact with disturbed areas will be managed prior to being released to the surrounding environment. Runoff from disturbed areas will be collected in gravity ditches and conveyed to ponds. Collection ponds have been designed as settling ponds with a permanent water depth to aid in removing total suspended solids (TSS) prior to discharge to the receiving environment. Each pond will be equipped with a treatment plant to ensure discharge meets environmental criteria for total suspended solids.

### 1.17.4 Tailings Management Facility and Water Management

According to the mine plan, tailings will be deposited into a surface tailings management facility (TMF) until the middle of Year 18, which provides a six-month buffer after the Main Zone pit has been depleted. Thereafter, tailings will be deposited into the Main Zone pit and then the East Zone pit. It is estimated that 624 Mt of tailings will be produced before in-pit deposition begins. This corresponds to a volume of 495 $Mm^3$. The surface TMF will be located south of the mine, and will have an approximate footprint area of 2,300 ha.

Containment of tailings within the TMF will be provided by a perimeter dam that will be constructed in stages. The TMF site is relatively flat. To enhance storage capacity, the TMF will be operated as a “thickened tailings cone” with tailings deposition near the centre of the TMF. Runoff from the TMF, together with slurry water and bleed water released by the tailings, will be conveyed by gravity within the TMF and will pass through internal TMF service spillways into the adjacent external TMF water management ponds. Seepage through the TMF perimeter dam will be controlled by a low permeability core. Seepage and runoff from the downstream shell of the dam will be collected in a perimeter contact water channels and directed to the TMF water management ponds.

### 1.17.5 Highways and Rail

#### 1.17.5.1 Highway 655

Existing Provincial Highway 655 (under MTO jurisdiction) runs north to south through the western portion of the pit envelope. The existing highway will need to be removed and re-routed along the west boundary of the property. Approximately 25.7 km of new highway will need to be constructed; this is planned for completion during Year 2 of mill operations.

#### 1.17.5.2 Highway 655 Overpass

To permit access between the Phase 1 primary crusher and the process plant prior to completion of the highway realignment, a Highway 655 overpass will be constructed. The overpass will consist of a multi-plate arch structure, that will allow access for the crushed ore stockpile feed conveyor and 40 t articulated trucks underneath the highway overpass.

#### 1.17.5.3 Railway

A 25.2 km rail spur to service the process plant will be built for the project to connect with a section of the Ontario Northland Railway (ONR) line operated by Glencore for its Kidd operations.

### 1.17.6 Power Distribution

#### 1.17.6.1 Site Power Supply

Hydro One Networks Inc. (HONI) has indicated that it would be feasible to provide electrical power to the Crawford mine site for Phase 1 and Phase 2 via the Porcupine Substation located near Timmins, ON. For Phase 1, an approximately 40 km overhead 230 kV wood-pole powerline (H-frame-style structure) will be constructed to a switching station (Crawford switching station) located on the southwest corner of the Crawford mine site. For Phase 2, a second 40 km overhead 230 kV wood-pole powerline will be constructed on a new right-of way following the same general routing as Phase 1.

On February 25, 2022, a Transmission Service (Wheeling) and Project Development Agreement was signed between CNC and Transmission Infrastructure Partnerships 1 Limited (TIP1). This agreement outlined the terms and estimated costs by which the proposed transmission lines for both phases will be financed, constructed, owned, operated, and maintained by TIP1 and for which CNC will pay service and transportation fees for delivery of electricity over a 25 year period. From the Crawford switching station, there will be three 230/34.5 kV transformers that feed the mine via a 13.8 kV medium-voltage network.

#### 1.17.6.2 On-Site Power Distribution

The project will consist of two phases; Phase 1 will have an estimated combined maximum demand and operating load of 149 MW and 124 MW, respectively. Phase 2 will have an estimated combined maximum demand and operating load of 309 MW and 263 MW, respectively.

For Phase 1 operations, a new 230 kV ring bus switchyard and 230 kV / 34.5 kV, 336 / 450 MVA outdoor substation will be installed at the project site. The substation will consist of three 230 kV / 34.5 kV, 112/150 MVA, ONAN/ONAF power transformers that will be interconnected to the 230 kV switchyard ring bus via three 230 kV disconnect switches and overhead lines.

A second 230 kV / 34.5 kV, 75 / 100 MVA outdoor substation with appropriately sized electrical rooms will be installed for Phase 2.

On-site distribution across the project site will be via 34.5 kV overhead lines, and 34.5 kV and 4.16 kV cabled circuits. Larger process motors will utilize a voltage of 4.16 kV, whereas smaller loads will operate at 600 V.

## 1.18 Environmental, Permitting and Social Considerations

The project is situated within the Northern Clay Belt in northeastern Ontario, within the Timmins-Cochrane Mining Camp, a region with a strong mining history. The site is generally undeveloped except for existing provincial infrastructure (e.g., highway, transmission lines) that cross portions of the lands, and prior impacts as a result of mineral exploration and forestry operations. The following subsections summarize the social and environmental considerations for the project site.

### 1.18.1 Environmental Considerations

Most mining projects in Canada are reviewed under one or more environmental / impact assessment processes whereby design choices, environmental impacts and proposed mitigation measures are compared and reviewed to determine how best to proceed through the environmental approvals and permitting stages. The Impact Assessment Agency of Canada decided that a federal impact assessment is required for the Crawford Project and published the tailored impact statement guidelines. It is expected that completion of one or more provincial class environmental assessments will also be required based on the current project design. Upon completion of these processes, several federal and provincial authorizations, approvals, and permits will be required to construct and operate the mine, including water use authorizations, fish and fish habitat authorization, emissions and discharge approvals, and approvals for infrastructure development within the project. These processes are well understood.

Collection of environmental baseline information commenced in 2021 for the project site and surrounding areas covering aspects of atmospheric, water resources, geochemistry, terrestrial environment, aquatic environment, and human and socio-environment. Based on the information collected to date, there are no environmental aspects that appear to be limiting the project’s development. The environmental baseline studies will be used to identify environmental constraints, which may require refinements to the project components or additional mitigation measures to be implemented. It is anticipated that the results of the impact assessment and class EAs, including implementing identified mitigation measures, will limit the potential for the project causing significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the project, and cumulative effects. Compliance with terms and conditions of approvals, standards contained in federal and provincial legislation and regulations, and commitments made during the EA processes (including application of mitigation measures and monitoring and follow-up requirements), will need to be addressed throughout project planning, construction, operation, and decommissioning.

### 1.18.2 Permitting Considerations

CNC prepared and submitted the initial project description and detailed project description report to the Impact Assessment Agency of Canada to initiate the regulatory assessment process (see Section 20.1 for details). The Impact Assessment Agency of Canada decided that a federal impact assessment is required for the Crawford Project and published the Tailored Impact Statement Guidelines in response. A draft environmental impact statement is expected to be prepared by CNC in 2024. Permitting for site-specific activities related to the project’s construction and operation are anticipated to commence concurrently during the preparation of the environmental impact statement. Completion of the federal impact assessment, provincial class environmental assessments, future conditions of approval, and permitting for site-specific activities could require refinements to the project components or additional mitigation measures to be implemented.

A detailed list of anticipated permits is provided in Chapter 20. Compliance with terms and conditions of approvals, standards contained in federal and provincial legislation and regulations, and commitments made, will need to be addressed throughout project planning, construction, operation, and decommissioning. Approvals, authorizations, and permits will be required prior to initiating project construction.

Environmental baseline studies and geochemistry studies are ongoing to support a timely environmental approvals process, as well as to support the engineering design and environmental effects monitoring. Geotechnical studies are recommended to determine the potential for settling within the project site and surrounding area due to the presence of varved clay and project dewatering. Completion of Traditional Knowledge and traditional land use studies with local Indigenous groups should continue to be supported. CNC should continue to actively engage with stakeholders and local Indigenous groups on the project design elements going forward. Impact benefit agreement negotiations with local Indigenous groups should continue.

### 1.18.3 Social Considerations

As CNC advances the project towards an impact assessment, the intention of the company is to continue to fully engage with local Indigenous groups and stakeholders in a comprehensive consultation process aimed at identifying and addressing significant challenges associated with the development of the project, but also seeking to underline and optimize potential social, environmental, and economic benefits. The output of this collaborative effort is intended to be integrated in the project's impact assessment. Although the range of stakeholders is expected to evolve to reflect various levels of interest and issues over time, CNC has identified local stakeholders who have or could demonstrate an interest in the project. CNC maintains a list of stakeholders who have been contacted throughout the engagement process, though not all parties who have been contacted responded or chose to participate. CNC will continue to reach out through the environmental approvals process as appropriate so that key stakeholders are informed and engaged.

CNC is also actively engaging local Indigenous groups and have established a memorandum of understanding with three of the Indigenous groups to support future engagement and participation in the project. Additional Indigenous groups will be included in negotiations for an Impact Benefit Agreement, as appropriate. CNC will work in partnership with Indigenous Peoples to establish a mutually beneficial, cooperative, and productive relationship centred around transparent information sharing, respectful engagement, open dialogue, and meaningful partnerships.

In parallel with the consultation process, CNC is presently working on a comprehensive community contribution program that will include, without being limited to, a local procurement policy as well as a sponsorship and donation strategy adapted to CNC's guiding principles and the needs of the communities.

### 1.18.4 Closure and Reclamation Considerations

A rehabilitation and closure plan is required under the *Ontario Mining Act,* Regulation 240/00. A complete rehabilitation and closure plan has not yet been developed for the project; however, the rehabilitation and closure philosophies and concepts that will be used in the development of the project's rehabilitation and closure plan are presented below.

Three key stages of rehabilitation activities are (1) progressive rehabilitation, (2) closure or active rehabilitation, and (3) post-closure monitoring and treatment or passive closure. Progressive rehabilitation involves rehabilitation completed through mine operation, prior to closure with activities such as placement of overburden cover and revegetation of inactive areas of the TMF and rock impoundment as well as placement of tailings within the open pit(s) after the Main Zone pit is fully mined. Closure or active rehabilitation occurs in the first five years of the cessation of mining and ore processing. Active rehabilitation includes initiation of pit flooding, acceleration of pit filling by construction of channels between collection ponds and the open pit, dismantling and removing site buildings and site infrastructure, placement of remaining overburden cover on rock impoundment and lower grade value ore stockpile laydowns, and seeding. Passive closure is primarily related to water management and pit filling including construction of the pit overflow spillway and monitoring and maintenance to confirm physical and chemical characteristics of the site are deemed acceptable and stable so the project can be closed out in accordance with the *Mining Act* and Metal and Diamond Mining Effluent Regulations (MDMER).

The rehabilitation and closure plan will be drafted and finalized in consultation with provincial agencies, Indigenous groups, and public stakeholders.

## 1.19 Markets and Contracts

Pricing assumptions were developed for the various payable metals that are recovered in the nickel and magnetite concentrates produced at Crawford and are effective as of August 2023. First production from Crawford is planned for the end of 2027, which falls within the long-term period of these forecasts. For this reason, a single price, expressed in 2023 real terms, has been used for all years.

Table 1-11 summarizes the assumed pricing and payability of each metal. Commentary on these assumptions is provided below.

**Table 1-11: Metals Pricing Assumptions (2023 Basis)**

| Metal | Units | Long-Term | Payability | Source |
|---|---|---|---|---|
| Ni Concentrate | US$/t (US$/lb) | 21,000 (9.53) | 91% | Fastmarkets, company estimate |
| Ni-Fe Concentrate | US$/t (US$/lb) | 21,000 (9.53) | 91% | Fastmarkets, company estimate |
| Iron Scrap | US$/t | 325 | 50% | Fastmarkets, company estimate |
| Chromium | US$/lb | 1.75 | 65% | Fastmarkets |
| Cobalt | US$/t ($/lb) | 40,000 (18.14) | 60% | Analyst consensus[1] |
| Platinum[3] | US$/oz | 1,150 | 1 g/t deduction | Analyst consensus[2] |
| Palladium[3] | US$/oz | 1,350 | 1 g/t deduction | Analyst consensus[2] |

Notes: **1.** Aggregate of 14 analyst estimates, dated August 2023. **2.** Aggregate of 17 analyst estimates, dated August 2023. **3.** Payability based on 1 g/t combined Pt + Pd deduction. Resultant life-of-mine average payability is 76.2% and 75.2% for Pt and Pd, respectively.

The nickel concentrate, with an average nickel content of 34% nickel over the life of project, is suited for a wide range of downstream processes, including traditional smelting and refining, and the concentrate roasting/reduction approach that was successfully developed by the CNC management team for concentrate produced by the Dumont project and implemented in 2014 by Tsingshan.

This roasting/ reduction downstream process could be utilized by FeNi/NPI/matte laterite processing operations which utilize an electric furnace for primary reduction to process feed from Crawford, Dumont, and other nickel concentrates.

## 1.20 Capital Cost Estimate

A capital cost estimate was compiled with costs grouped into major scope areas by WBS. The capital cost estimate includes all costs related to the Crawford Project (e.g., local infrastructure upgrades, open pit mine development, ore processing facility, tailings management facility, high-voltage substation and power supply infrastructure, offices, maintenance shops and utilities) to support mining operations.

The estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International). Most costs have a base date of Q4 2022, except for mining, tailings management, and water management costs, which have a base date of Q2 2023.

Capital costs are summarized in Table 1-12.

**Table 1-12: Estimate Summary by WBS Level 1**

| WBS | WBS Description | Initial Capital (C$M) | Expansion Capital (C$M) | Sustaining Capital (C$M) | LOM Total Capital (C$M) | LOM Total Capital (US$M) |
|---|---|---|---|---|---|---|
| 1000 | Mining | 657 | 552 | 1,715 | 2,924 | 2,222 |
| 2000 | Process | 902 | 914 | 0 | 1,816 | 1,381 |
| 3000 | Utilities | 46 | 40 | 0 | 86 | 66 |
| 4000 | Tailings and Water Management | 129 | 111 | 136 | 375 | 285 |
| 5000 | On-Site Infrastructure | 120 | 67 | 97 | 284 | 216 |
| 6000 | Off-Site Infrastructure | 150 | 56 | 0 | 205 | 156 |
| 7000 | Indirect Costs | 244 | 174 | 0 | 418 | 317 |
| 8000 | Owner's Costs | 65 | 0 | 0 | 65 | 51 |
| 9000 | Contingency | 244 | 191 | 0 | 435 | 330 |
| | Total Capital | 2,556 | 2,105 | 1,950 | 6,611 | 5,024 |
| | Closure Costs | 0 | 0 | 175 | 175 | 133 |
| | **Total Investment** | **2,556** | **2,105** | **2,125** | **6,786** | **5,157** |

Note: Totals may not add due to rounding. Costs shown in US dollars have been converted at an exchange rate of 1 CAD = 0.76 USD.

Costs are generally grouped into three categories:

1. Initial Capital – initial project development, with a mill throughput of 60 kt/d
2. Expansion Capital – all costs from the completion of initial development through the expansion of throughput to 120 kt/d, planned for Year 4
3. Sustaining Capital – all costs subsequent to completion of the expansion.

Apart from the mining, tailings management, and water management costs noted above, there is no escalation added to the estimate from the base date of Q4 2022 forward, or allowance for exchange rate fluctuations. Additional qualifications to and exclusions from the capital cost estimate are noted in Section 21.

Working capital was also not considered in the capital cost estimate, but is included in the financial analysis (see Section 1.22).

Of the total initial capital costs, more than 85% (excluding contingency) were derived from first principles bulk material take-offs and equipment sizing calculations, with supporting quotations for mining development, major equipment, and contractor supply/installation rates.

## 1.21 Operating Cost Estimate

The operating cost estimate is subdivided into the following major areas: mining, tailings/water management, processing, and general and administrative costs (G&A). Costs have been further subdivided into the following three phases:

- Phase 1 covers the initial 42 months of operation when the nameplate capacity of the mill will be 60 kt/d.
- Phase 2 covers the following 26 ½ years of operation when nameplate capacity of the plant will be 120 kt/d and feed will predominantly come from the pits.
- Phase 3 covers the final 11 ¼ years of operation when nameplate capacity of the plant will be 120 kt/d and feed will be entirely sourced from low-grade stockpiles on surface.

Note that costs for Phases 1 and 2 include the impact of ramp-up to the steady-state production rate, while Phase 3 includes the ramp-down of production at the end of project life.

Key inputs and assumptions made in preparing the operating cost estimate include the following:

- The base date for costs inputs is Q2 2023, with pricing in Canadian dollars.
- Labour costs were estimated based on the organizational structure developed for each area. The rates of pay are based on wages and benefits at existing mining operations in the Timmins region of Ontario as detailed in a salary survey provided by Lincoln Strategic International.
- Electricity prices have been based on a 24-month average (August 2021 to July 2023) and consider the demand profiles for both peak and off-peak times. Prices also recognize that the project's demand will make it eligible to participate in the Industrial Conservation Initiative (ICI) and pay a share of the global adjustment (GA) based on actual peak demand as a percentage of total Ontario demand. The estimated life-of-mine average total price for electricity, taking account of charges for consumption, demand, and the GA, is C$75/MWh.
- Fuel prices are based on an assumed oil price (Brent) of $70/bbl. It has been assumed that during project construction, Crawford will pay carbon taxes on diesel and gasoline consumed, with prices ramping up to C$1.31 and $1.30/litre, respectively, as the carbon tax increases. With the start of mill production, Crawford's net negative status will obviate the requirement to pay these taxes and prices will fall to C$0.97/litre for diesel and C$1.02/litre for gasoline.

Operating costs are summarized in Table 1-13.

**Table 1-13: Summary of Operating Costs**

| Area | Units | Phase 1 | Phase 2 | Phase 3 | LOM Average |
|---|---|---|---|---|---|
| Mine – Tonnes Mined | C$/t mined | 2.22 | 1.82 | 0.00 | 1.96 |
| Mine – Tonnes Milled | C$/t milled | 12.92 | 8.17 | 0.81 | 6.29 |
| Tailings and Water Management | C$/t milled | 0.51 | 0.22 | 0.19 | 0.22 |
| Process | C$/t milled | 6.99 | 6.81 | 6.83 | 6.82 |
| G&A | C$/t milled | 2.58 | 1.10 | 0.47 | 0.98 |
| **Site Costs** | **C$/t milled** | **23.00** | **16.30** | **8.30** | **14.32** |

## 1.22 Economic Analysis

The economic analysis of Crawford presented in Table 1-14 reports all costs and values in United States dollars (USD, US$).

**Table 1-14: Crawford Feasibility Study Summary Metrics**

| Item | Unit | Value |
|---|---|---|
| Ore Mined | Mt | 1,715 |
| Payable Ni | Mlbs | 3,130 |
| Payable NiEq | Mlbs | 4,961 |
| Net Smelter Return (NSR) | $/t ore | 28.08 |
| Site Operating Costs | $/t ore | 10.88 |
| Net C1 Costs[1] | $/lb Ni | 0.39 |
| EBITDA | $/t ore | 16.04 |
| Peak Funding Requirement[2] | $M | 1,898 |
| Total Investment[3] | $M | 5,157 |
| Net AISC[4] | $/lb Ni | 1.54 |
| Post-Tax $NPV_{8\%}$ | $M | 2,475 |
| Post-Tax IRR | % | 17.1 |

Notes: 1. C1 costs include site operating expenditures. 2. Peak funding represents the cumulative unlevered investment prior to generation of positive cash flow. 3. Total investment includes all capital and closure expenses. 4. All-in sustaining costs include C1 costs, royalties, IBA, sustaining capital and closure expenses.

Key inputs and assumptions include the following:

- Construction to begin mid-2025 and lead to commissioning of the initial 60 kt/d plant in late 2027. A project to double production would then commence in mid-2029 and result in first incremental production 24 months later.
- The plant will process 1,715 Mt ore extracted from two pits mined sequentially. After the Main Zone is depleted in Year 17, impoundment of tailings will transition from the TMF to in-pit deposition. After the East Zone is depleted in Year 30, the plant will be fed for the remaining 11 years of project life from stockpiled lower grade ore on surface.
- Crawford will produce two concentrates: a nickel concentrate containing payable nickel, cobalt, palladium, and platinum, and an iron concentrate containing payable iron, nickel and chromium. Both concentrates would be sold free on board (FOB) Crawford.

The base case analysis includes the Clean Technology Manufacturing (CTM) Investment Tax Credit (ITC) that was outlined during the 2023 federal budget presentation. While it is anticipated that Crawford would also qualify for the Carbon Capture, Utilization and Storage (CCUS) ITC, until approval to receive the credit has been obtained, this will be included as an opportunity and is discussed under Section 25.

## 1.23 Other Relevant Information

### 1.23.1 Project Execution

Two contract strategies will be employed to deliver the detailed engineering and execution phases of the project:

1. Engineering, procurement, and construction management (EPCM) contract(s), led by an engineering consultant nominated by CNC, that generally encompasses the process plant and select on-site infrastructure
2. EPCM scope, led by CNC, that generally encompasses the development of the mining pits, bulk earthworks, and off-site infrastructure.

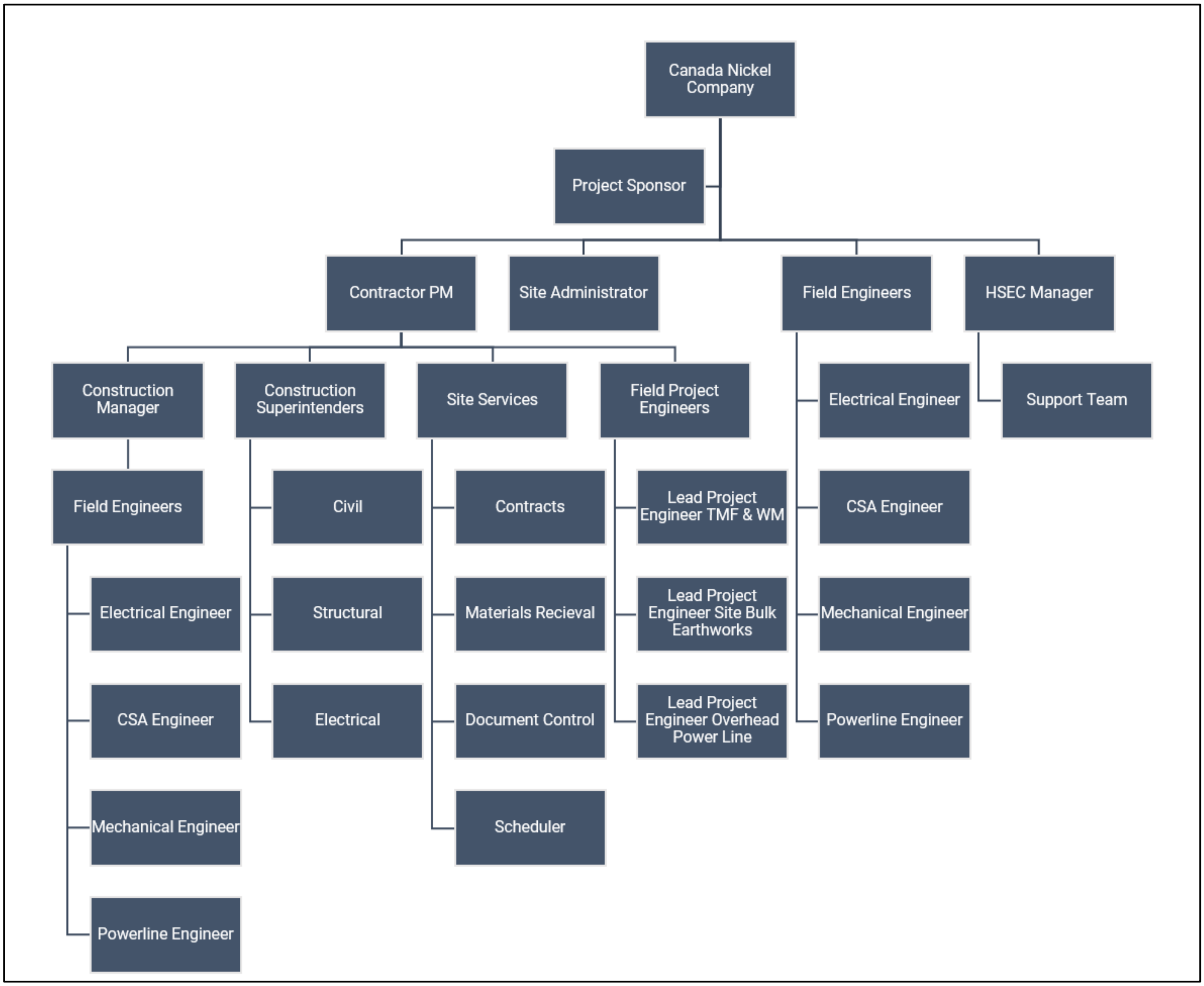
The project execution plan (PEP) will serve as the main communication tool to ensure that the integrated project teams are aware and knowledgeable of project objectives and how they will be accomplished.

### 1.23.2 Project Organization

The project team is organized based on an integrated team approach, minimizing the duplication of roles and activities between the Owner's team and major delivery partners. A project organization chart is shown in Figure 1-3.

CNC will be performing or managing a considerable portion of the project scope, including the mine design, power transmission line, rails, highway modifications, pit pre-stripping, and delivery of certain construction materials to designated work sites. Key persons will be established on both teams at site to ensure efficient coordination.

**Figure 1-3: Project Organization Chart**




Source: Ausenco, 2023.

### 1.23.3 Construction Sequencing

The key construction activites on the critical path of the project construction are summarized below. Note that the schedule has been developed assuming a mid-year construction start based on permitting timelines.

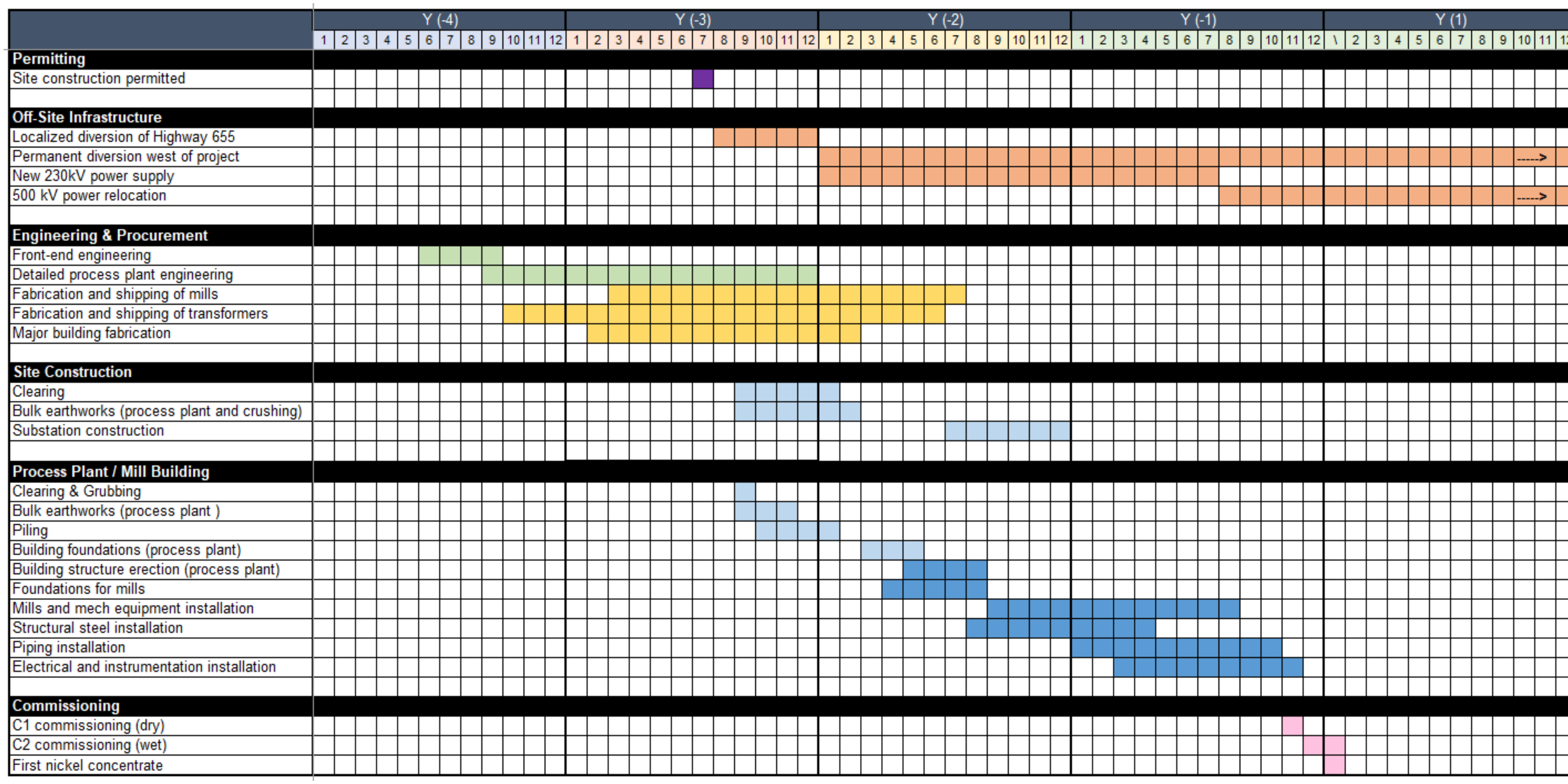
- The first contractors to mobilize to site will be the civil work contractors responsible for clearing the site.
- The infrastructure areas will then need to be levelled and graded to rough construction levels.
- Once the initial civil works are completed, the main schedule priority will be the process plant and crushing area, where piling is required to achieve suitable ground conditions for building construction.
- In parallel, contractor's execution team and subcontractor teams will mobilize to site and establish office and workshop facilities, temporary accommodations, etc.
- Mining works will continue pit development, generating and stockpiling rock and overburden materials that will be used for construction materials.
- The concrete works will begin as soon as winter conditions have eased enough to allow concrete pouring, focusing on the process plant building first, and followed by the crushing and mine infrastructure building foundations.
- Erection of the main process building steel will follow closely behind the building foundations to ensure the main building envelope is enclosed well before the second winter.
- Following this, equipment foundations will be poured, and construction will proceed through steelwork, equipment installation, piping, electrical and instrumentation.

An overall master execution schedule is included in Figure 1-4 on the following page.

### 1.23.4 Accommodations

There will be no on-site accommodations and camp for the contractors. The contractors will be responsible for camp and accommodation expenses and can stay in or near the town of Timmins.

**Figure 1-4: Preliminary High-Level Execution Schedule**




Source: Ausenco, 2023.

## 1.24 Interpretation and Conclusions

Information from legal experts and CNC support that the tenure held is valid and sufficient to support a declaration of mineral resources and mineral reserves.

The exploration programs completed to date are appropriate for the style of the deposits in the project area. Sampling methods are acceptable for mineral resource and mineral reserve estimation. The mineral reserve and mineral resource estimations for the project both conform to industry-accepted practices and are reported using the 2014 CIM Definition Standards.

The proposed mine life includes 2 ½ years of pre-stripping and 30 years of mining. Following the completion of mining, the process plant will continue to be fed from stockpiled ore for an additional 11 years.

The process plant flowsheet designs were based on testwork results and industry-standard practices and were developed for economically-optimized recovery. The initial milling throughput capacity is 60 kt/d; an expansion program will double capacity to 120 kt/d by Year 5 of mine life.

No technical or policy issues are anticipated for obtaining the required project permits and approvals.

The positive financials of the project (US$2,475 million after-tax NPV8% and 17.1% after-tax IRR) support the mineral reserve.

## 1.25 Recommendations

An analysis of the results and findings from each major area of investigation completed as part of this feasibility study suggests numerous recommendations for further investigations to mitigate risks and/or improve the base case designs. The following sections summarize the key recommendations arising from this feasibility study.

Table 1-15 presents a summary of recommended tasks, budget and detailed in the subsections that follow. Recommendations are split into two phases; Phase 2 is contingent on the project proceeding through financing approvals.

For more detailed recommendations, refer to Section 26.

**Table 1-15: Cost Estimate for Proposed Recommendations**

| Description | Cost (C$M) |
|---|---|
| **PHASE 1** | |
| Exploration Drilling – Deep Drilling | 3.40 |
| Exploration Drilling – Borehole Geophysics | 0.25 |
| Infill Drilling – Nickel Resources | 4.60 |
| Infill Drilling – PGE Reefs | 1.80 |
| Metallurgical Testwork | 5.00 |
| Mining (Bulk Sample) | 6.50 |
| Process Plant Geotechnical (Boreholes, Pile Testing and Seismic Refraction) | 1.10 |
| Soil Geotechnical | 3.25 |
| Rock Geotechnical | 0.85 |
| Hydrogeology | 0.12 |
| Surface Water Management | 0.25 |
| Tailings Management Facility (Phase 1) | 4.10 |
| Environmental | 8.00 |
| Marketing | N/A |
| **Total – Phase 1** | **39.22** |
| **PHASE 2** | |
| Tailings Management Facility | 2.10 |
| Soil Geotechnical | 0.45 |
| **Total – Phase 2** | **2.55** |

# 2 INTRODUCTION

## 2.1 Introduction and Terms of Reference

The report was prepared by Ausenco Engineering Canada ULC and Ausenco Sustainability ULC (collectively, "Ausenco") in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and Form 43-101 F1, and is prepared using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

The report supports disclosures by CNC in a news release dated October 12, 2023, titled, "Canada Nickel Announces Positive Bankable Feasibility Study For its Crawford Nickel Sulphide Project."

Inputs to the report were provided by Caracle Creek International Consulting Inc. (Caracle Creek), David Penswick, SRK Consulting (Canada) Inc. (SRK), WSP Canada Inc. (WSP), Stantec Consulting Ltd. (Stantec), JL Richards and Associates Limited (JLR) and BBA Inc. (BBA). The individuals listed on the cover of this report, by virtue of their education, experience and professional associated, are considered Qualified Persons (QPs) as defined by NI 43-101. The QPs meet the requirements of independence defined in NI 43-101.

## 2.2 Site Visits and Scope of Personal Inspections

Site visits completed by the QPs are presented in Table 2-1 and described below.

**Table 2-1: Summary of QP Site Visits**

| Qualified Person | Date of Site Visit | Duration of Visit |
|---|---|---|
| Paul Staples | August 31, 2022 | 1 day |
| Gregory Lane | No visit | - |
| Jonathan Cooper | No visit | - |
| David Penswick | August 31, 2022 | 1 day |
| John M. Siriunas | August 11 and 12, 2022 | 2 days |
| Scott Jobin-Bevans | No visit | - |
| Kenneth Arthur Bocking | August 19, 2022 | 1 day |
| David Brown | No visit | - |
| Steve Hales | November 1, 2021 | 4 days |
| Michelle Fraser | September 11, 2023 | 1 day |
| Gordon Murray | August 24, 2022 | 6 hours |
| Colin Hardie | November 23, 2022 | 1 day |
| Bruce Andrew Murphy | August 31, 2022 | 1 day |
| Cameron Colin Scott | August 31, 2022 | 1 day |

### 2.2.1 Paul Staples Site Visit

Mr. Staples, representing Ausenco, visited the site on August 31, 2022. The general topography of the project site was reviewed via ATV and truck, and locations of major infrastructure (power lines, Highway 655 and process plant) were visited. Drilled core samples were viewed in person, and a test pit excavation was reviewed in-progress to assess typical clay depths and properties for the site.

### 2.2.2 David Penswick Site Visit

Mr. Penswick completed a site visit on August 31, 2022. During this visit, he inspected potential sources of aggregate for early construction activities as well as numerous other representative areas within the overall project footprint. In addition, he walked to an area inside the proposed open pit where he witnessed and photographed a backhoe excavator completing a test pit. Furthermore, he visited the core logging facility and reviewed representative sections of drillholes intersecting the major lithologies that will be encountered during mining.

### 2.2.3 John Siriunas Site Visit

Mr. Siriunas, representing Caracle Creek, visited the project site on August 11–12, 2002 (two days). He had also previously visited the site on October 12, 2019 (one day), on February 3–4, 2020 (two days), and on September 10–11, 2020 (two days).

Visits were made to observe the general property conditions and access, and to verify the location of some of the historical and completed drillhole collars, as well as the work progress in the field. During the site visits, diamond drilling procedures were discussed and a review of the on-site logging and sampling techniques, including internal QA/QC procedures, was carried out. In 2019, the secure storage and logging facility at 3700 Highway 101 West in Timmins was visited; later visits were made to the larger facilities at 170 Jaguar Drive, Timmins, Ontario.

### 2.2.4 Kenneth Bocking Site Visit

Mr. Bocking, representing WSP, visited the site on August 19, 2022. The objective of the visit was to view the existing conditions (i.e., topography, drainage and vegetation cover) in the area of the proposed TMF. Due to swampy conditions, the site visit was carried out primarily by ATV and the visit was limited to several points on the perimeter of the TMF.

### 2.2.5 Steve Hales Site Visit

Mr. Hales, representing WSP, visited the site from November 1 to 4, 2021. The objective of the visit was to view existing groundwater and surface water conditions at the site, as well as to commence the hydrogeological (packer) testing program with field personnel. Several borehole and testing locations were observed during the visit.

### 2.2.6 Michelle Fraser Site Visit

Ms. Fraser, representing Stantec, visited the site on September 11, 2023. The objective of the visit was to view the existing conditions (i.e., topography and drainage) across the site and surrounding area. Due to drainage conditions, the site visit was carried out primarily by an amphibious all-terrain vehicle and truck. The visit was limited to several points within the open pit, the perimeter of the tailings management facility and rock stockpile, the shore of the West Buskegau River, North Driftwood River, and Mattagami River, as well as a series of lakes located along a north–south trending esker located to the west of the site.

#### 2.2.7 Gordon Murray Site Visit

Mr. Murray, P.Eng., representing JLR, visited the Crawford property on Wednesday, August 24, 2022. Weather conditions were clear and sunny. The objective of the visit was to view existing surface topography, soils and drainage conditions within the proposed Highway 655 realignment and railway corridors, using a pedestrian half-ton truck for access. Access to many sections of the proposed highway and rail corridor was not possible, however, Mr. Murray did get a good sense of the topography, soils, and general environment within the corridor. It was also possible to review the site of the existing Highway 655 at the site of a proposed future grade separation over the mine access road, to observe constraints such as existing drainage and utilities, and to walk the site of the temporary Highway 655 diversion around the grade separation construction site.

#### 2.2.8 Colin Hardie Site Visit

Colin Hardie, representing BBA, visited the project site on November 23, 2022. The objective of the visit was to review the regional infrastructure and the topography of the general routing for the proposed electrical transmission line from the Porcupine Substation located near Timmins, ON to the proposed location of the electrical substation at the Crawford project site.

#### 2.2.9 Bruce Murphy Site Visit

Mr. Murphy, representing SRK, visited the project site on August 31, 2022. During his time on site, he carried out a vehicular-based inspection of areas representative of the project site. In addition, he visited the core logging facility and reviewed representative sections of drillholes intersecting the major rock units on the project site. Specific drillholes intersecting the regional CUC Fault, other logged major fault intersections, talc alteration, as well as areas with degrading dunite containing coalingite were also reviewed.

#### 2.2.10 Cam Scott Site Visit

Mr. Scott, representing SRK, completed a site visit on August 31, 2022. During his time on site, he carried out a vehicular-based inspection of areas representative of the project site. In addition, he walked to an area inside the proposed open pit and, with the assistance of a track mounted excavator, oversaw the completion of a test pit, logged and photographed the test pit and collected a sample of the key soil stratum. With support from site geologists, representative samples from a previous test pit program were observed and compared with the single new sample.

## 2.3 Effective Dates

The report has several significant dates, as follows:

- mineral resource estimate: August 31, 2023
- mineral reserve estimate: August 31, 2023
- date of financial analysis: October 1, 2023

The effective date of the report is October 1, 2023.

## 2.4 Information Sources and References

This technical report is based on internal company reports, maps, published government reports, and public information as listed in Section 27. Additionally, it is based on information cited in Section 3.

The QPs have not performed an independent verification of land title and tenure information as summarized in Section 4 of this report, which was verified separately by CNC. The QPs did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between CNC and third parties, and as such, they express no opinion as to the ownership status of the project.

## 2.5 Previous Technical Reports

CNC has filed the following technical reports on the project:

- Lane, G., Penswick, D., Jobin-Bevans, S., Siriunas, J., Staples, P., Daniel, S., and Van Zyl, K., 2022: Crawford Nickel Sulphide Project, NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate, Timmins Area, Ontario, Canada: report prepared by Caracle Creek International Consulting Inc. and Ausenco Engineering Canada Inc. for CNC, effective date 19 May 2022.
- Staples, P., Lane, G., Jobin-Bevans, S., Siriunas, J., Penswick, D., Daniel, S., Van Zyl, K., 2021: Crawford Nickel Sulphide Project, NI 43-101 Technical Report & Preliminary Economic Assessment, Ontario, Canada: report prepared by Ausenco Engineering Canada Inc. for CNC, effective date 21 May 2021.
- Jobin-Bevans, S., Siriunas, J., Penswick, D., 2020: Independent Technical Report and Mineral Resource Estimates, Crawford Nickel-Cobalt Sulphide Project: Main Zone (Update) and East Zone (Initial) Deposits, Timmins-Cochrane Area, Ontario, Canada: report prepared by Caracle Creek International Consulting Inc. for CNC, effective date 12 December 2020.
- Independent Technical Report and Mineral Resource Estimate, Crawford Nickel-Cobalt Sulphide Project, Timmins-Cochrane Area, Ontario, Canada: report prepared by Caracle Creek International Consulting Inc. for CNC, effective date 16 March 2020.
- Independent Technical Report on the Crawford Nickel-Cobalt Sulphide Project, Timmins-Cochrane Area, Ontario, Canada: report prepared by Caracle Creek International Consulting Inc. for CNC (and Noble Mineral Exploration), effective date 10 December 2019.

## 2.6 Currency, Units, Abbreviations and Definitions

All units of measurement in this report are metric. Metals prices and economic returns are expressed in US dollars (symbol: US$; currency: USD); other costs are expressed in Canadian dollars (symbol: C$; currency: CAD) unless otherwise stated.

All material tonnes are expressed as dry tonnes (t) unless stated otherwise. A list of abbreviations and acronyms is provided in Table 2-2, and units of measurement are listed in Table 2-3.

### 2.6.1 Abbreviations and Acronyms

**Table 2-2: Abbreviations and Acronyms**

| Acronym | Definition |
|---|---|
| AP | Acidification potential |
| ARD | Acid rock drainage |
| AW | Awaruite |
| BWi | Ball mill work index |
| CAD | Canadian dollar (symbol: C$) |
| CCME | Canadian Council of Ministers of the Environment |
| CCUS | Carbon capture, utilization and storage |
| CEAA | Canadian Environmental Assessment Act |
| CEQG | Canadian Environmental Quality Guidelines |
| CFE | Concentration of frequent effects |
| CIP | Carbon-in-pulp |
| CoG | Cut-off grade |
| CSQG | Canadian Sediment Quality Guidelines |
| CTM | Clean technology manufacturing |
| CUC | Crawford Ultramafic Complex |
| CWi | Crusher work index |
| DFO | Department of Fisheries and Oceans Canada |
| DL | Detection limit |
| DTW | Down the hole |
| EA | Environmental Assessment |
| ECA | Environmental Compliance Approval |
| ECCC | Environmental and Climate Change Canada (Federal) |
| EIA | Environmental impact assessment |
| EQA | Environmental quality act |
| ESA | Endangered Species Act |
| ETP | Effluent treatment plant |
| FEQG | Federal Environmental Quality Guidelines |
| HADD | Harmful alteration, disruption or destruction |
| HARD | Half absolute relative difference |
| HZ | Heazlewoodite |
| IAA | Impact Assessment Act |
| ITC | Investment tax credit |

| Acronym | Definition |
|---|---|
| LOM | Life-of-mine |
| M&I | Measured and indicated |
| MDMER | Metal and Diamond Mining Effluent Regulations |
| MECP | Ministry of Environment, Conservation and Parks |
| MINES | Ministry of Mines |
| ML | Metal leaching |
| MNRF | Ministry of Natural Resources and Forestry |
| MOU | Memorandum of Understanding |
| MRE | Mineral resource estimate |
| MTCS | Ministry of Tourism, Culture and Sport |
| MTO | Ministry of Transportation Ontario |
| NAG | Not potentially acid generating |
| NNP | Net neutralization point |
| NP/AP | Neutralizing potential / acid potential |
| NPV | Net present value |
| NRCan | Natural Resources Canada |
| NSP | Net smelter price |
| NSR | Net smelter return |
| OEB | Ontario Energy Board |
| OEM | Original equipment manufacturer |
| OP | Open pit |
| OVB | Overburden |
| PAX | Potassium amylxanthate |
| PEA | Preliminary economic assessment |
| PFS | Prefeasibility study |
| PGA | Potential generator of acid |
| PN | Neutralization potential |
| PN | Pentlandite |
| PNN | Net neutralizing power |
| PSQG | Ontario Provincial Sediment Quality Guidelines |
| PWQO | Provincial Water Quality Objectives |
| QA/QC | Quality assurance/quality control |
| QP | Qualified person |
| ROM | Run of mine |
| RWi | Bond rod mill work index |
| SARA | Species at Risk Act |
| SG | Specific gravity |
| TC | Transport Canada |
| TMF | Tailings management facility |
| UG | Underground |
| USD | United States dollars (symbol US$) |
| W:O | Waste-to-ore ratio |

### 2.6.2 Units of Measurement

**Table 2-3: Unit of Measurement**

| Unit of Measurement | Description |
|---|---|
| % | percent |
| (‘) | minute (plane angle) |
| “ | second (plane angle) |
| < | less than |
| > | greater than |
| ° | degree |
| °C | degrees Celsius |
| µm | micron |
| A | ampere |
| a | annum (year) |
| ac | acre |
| B | Billion |
| cm | centimetre |
| cm$^2$ | square centimetre |
| cm$^3$ | cubic centimetre |
| d | day |
| d/a | days per year (annum) |
| d/wk | days per week |
| ft | feet |
| ft$^2$ | square foot |
| ft$^3$ | cubic foot |
| ft3/s | cubic feet per second |
| g | gram |
| G | gauss |
| g/cm3 | grams per cubic centimetre |
| g/L | grams per litre |
| g/t | grams per tonne |
| GPM | US gallons per minute |
| h | hour |
| h/a | hours per year |
| h/d | hours per day |
| h/w | hours per week |
| ha | hectare (10,000 m$^2$) |
| hp | horsepower |
| k | one thousand |
| kg | kilogram |
| kg/h | kilograms per hour |
| kg/m$^2$ | kilograms per square metre |
| km | kilometre |
| km/h | kilometres per hour |
| km$^2$ | square kilometre |
| kPa | kilopascal |

| Unit of Measurement | Description |
|---|---|
| kt | thousand metric tonnes |
| kV | kilovolt |
| kW | kilowatt |
| kWh | kilowatt hour |
| kWh/a | kilowatt hours per year |
| kWh/t | kilowatt hours per tonne (metric tonne) |
| L | litres |
| L/m | litres per minute |
| lb | pounds |
| lb/ton | pounds per short ton |
| m | metres |
| M | million |
| $m^2$ | square metre |
| $m^3$ | cubic metre |
| amsl | above mean sea level |
| mg | milligram |
| mg/L | milligrams per litre |
| min | minute (time) |
| mL | millilitre |
| mm | millimetre |
| Mt | million metric tonnes |
| MW | megawatt |
| MWh | megawatt hour |
| ng/g | nanogram/gram (ppb = parts per billion) |
| oz | ounce |
| ppb | parts per billion |
| ppm | parts per million |
| psi | pounds per square inch |
| rpm | revolutions per minute |
| s | second (time) |
| t | metric tonnes (1,000 kg) |
| ug/g | microgram/gram (ppm = parts per million) |
| V | volt |
| wk | week |

# 3 RELIANCE ON OTHER EXPERTS

## 3.1 Introduction

The QPs have relied upon other expert reports, which provided information regarding royalties, environmental considerations, permitting, closure, social and community impacts, taxation, and marketing for sections of this report.

## 3.2 Property Agreements, Mineral Tenure, Surface Rights and Royalties

For information related to the NSR royalty and NPI royalty, the QP, David Penswick, relied on a summary of applicable contractual terms provided by CNC (September 30, 2023).

## 3.3 Environmental, Permitting, Closure, Social and Community Impacts

The QPs have fully relied upon, information supplied by CNC and experts retained by CNC for information related to environmental (including tailings and water management) permitting, permitting, closure planning and related cost estimation, and social and community impacts as follows:

- Ausenco Engineering Canada Inc., and Caracle Creek International Consulting Inc., 2022: NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate: Prepared for Canada Nickel Company Inc., May 19, 2022, 412 pages.
- Golder Associates Ltd., part of WSP, 2022: Technical Memorandum: Borrow Source Test Pit Field Program – Crawford Project: Sent to Canada Nickel Company, dated December 5, 2022, 59 pages.
- Golder Associates Ltd., part of WSP, 2022: Application for a Category 2 Permit to Take Water, Pumping Test Program: Prepared for Permit to Take Water Director, dated April 25, 2022, 67 pages.
- Golder Associates Ltd., 2022: Crawford Open Pit Project Climate Study: Prepared for Canada Nickel Company, May 2022, 129 pages.
- Golder Associates Ltd., 2022: 2021 Hydrology Monitoring Report, Crawford Project: Prepared for Canada Nickel Company, March 2022, 41 pages.
- Wood Environment & Infrastructure Americas, a Division of Wood Canada Limited, 2022: Crawford Nickel Project 2021 Aquatic Resources Baseline: Prepared for Canada Nickel Company, September 2022, 131 pages.
- Wood Environment & Infrastructure Solutions Canada Limited, 2022: Draft Crawford Nickel Project Cultural Heritage Screening Report. Revision A: Prepared for Canada Nickel Company, May 2022, 42 pages.
- Wood Environment & Infrastructure Solutions Canada Limited, 2022: Crawford Nickel Project 2021 Terrestrial Ecology Baseline Study. Revision 0A2: Prepared for Canada Nickel Company, June 2022, 266 pages.

- WSP E&I Canada Limited, 2022: Crawford Nickel Project Stage 1 Archaeological Assessment: Prepared for Canada Nickel Limited, October 2022, 41 pages.
- WSP E&I Canada Limited, 2022: Detailed Project Description Crawford Nickel Project: Prepared for Canada Nickel Limited, December 2022, 229 pages.
- WSP E&I Canada Limited, 2022: Draft Simulated Pore Water Pressures and Groundwater Inflows – Main Zone and East Zone Pits [PowerPoint slides]: Prepared for Canada Nickel Company, October 2022, 23 pages.
- WSP E&I Canada Limited, 2023: Air Quality Monitoring Plan (Rev.2): Prepared for Canada Nickel Company Inc. March 2023, 30 pages.
- WSP E&I Canada Limited, 2023: Draft Crawford Nickel-Cobalt Sulphide Project – 2022 Terrestrial Ecology Baseline Study. Revision 0B: Prepared for Canada Nickel Company, May 2023, 130 pages.
- WSP E&I Canada Inc., 2023: Draft 2022 Hydrology Monitoring Report, Crawford Project: Prepared for Canada Nickel Company, March 2023, 111 pages.
- WSP E&I Canada Inc., 2023: Draft Interim 2022 Aquatic Resources Baseline Report, Crawford Nickel Project: Prepared for Canada Nickel Company, May 2023, 141 pages.
- WSP E&I Canada Inc., 2023: Draft Technical Memorandum: Canada Nickel Company Crawford Project – 2023 Acoustic Baseline 'Leaves Off Season' Study: Sent to Canada Nickel Company Inc., April 2023, 9 pages.
- WSP E&I Canada Inc., 2023: Draft Technical Memorandum: Geochemistry Interim Results Update – April 2023: Sent to Canada Nickel Company April 18, 2023, 235 pages.
- WSP E&I Canada Inc., 2023: Technical Memorandum: Groundwater Quality Sampling Factual Report: Sent to Canada Nickel Company, dated February 7, 2023, 8 pages.
- WSP E&I Canada Inc., 2023: Technical Memorandum: Hydraulic Conductivity Testing Summary: Sent to Canada Nickel Company, dated January 25, 2023, 51 pages.
- WSP Golder, 2022. Draft Technical Memorandum: Crawford Project: Closure Concept for Feasibility Studies. Sent to Canada Nickel Company, dated December 8, 2022, 12 pages.

This information is used in Sections 1.18, 5, 20.1, 20.2, 20.3, 20.5, 25.10, 25.12, 25.14.8 and 26.12 of the report. The information is also used in support of Sections 1.4, 4.5, and 4.6.

## 3.4 Taxation

The QP David Penswick has relied on information conveyed by experts retained by CNC related to the current taxation regime for mineral projects located in Canada and Ontario. Mr Penswick has also relied on information provided by experts retained by CNC related to proposed new investment tax credits (ITCs), including the Clean Technology Manufacturing (CTM) ITC and the Carbon Capture, Utilization and Storage (CCUS) ITC.

### 3.5 Marketing

The QP David Penswick has relied upon the following information provided by CNC with regard to payment terms and prices for the various metals that will be produced. This information is used in Section 19 of the report.

- Wood Mackenzie, Ni database Q3 2023
- CRU Nickel Monitor, August 2023
- CRU Stainless Steel Monitor Q3 2023
- Fastmarkets Ni Market Data, Q2 2023.

# 4 PROPERTY DESCRIPTION AND LOCATION

## 4.1 Introduction

The Crawford Project is situated within the Timmins-Cochrane Mining Camp in northeastern Ontario, Canada, a region with a robust mining history in gold, nickel, zinc, and lead. As a province, Ontario is favourable to mining, with regulations that reflect that history (Figure 4-1).

In general, all known economic or potentially economic mineralization that is the focus of the report and that of CNC, is located within the boundary of the project's mining lands.

## 4.2 Property Location

The Crawford Project, located mostly in Crawford and Lucas townships with portions in Mahaffy and Carnegie townships, , is about 42 km north of the City of Timmins, and can be found on 1:50 000 NTS map sheet 42A/14E and 14F, Buskegau River. The approximate centre of the property is at UTM coordinates 473380 mE, 5408504 mN (NAD83, UTM Zone 17 North; EPSG:2958). The elevation at the project site ranges from 265 to 290 m above mean sea level (amsl).

## 4.3 Mineral Tenure

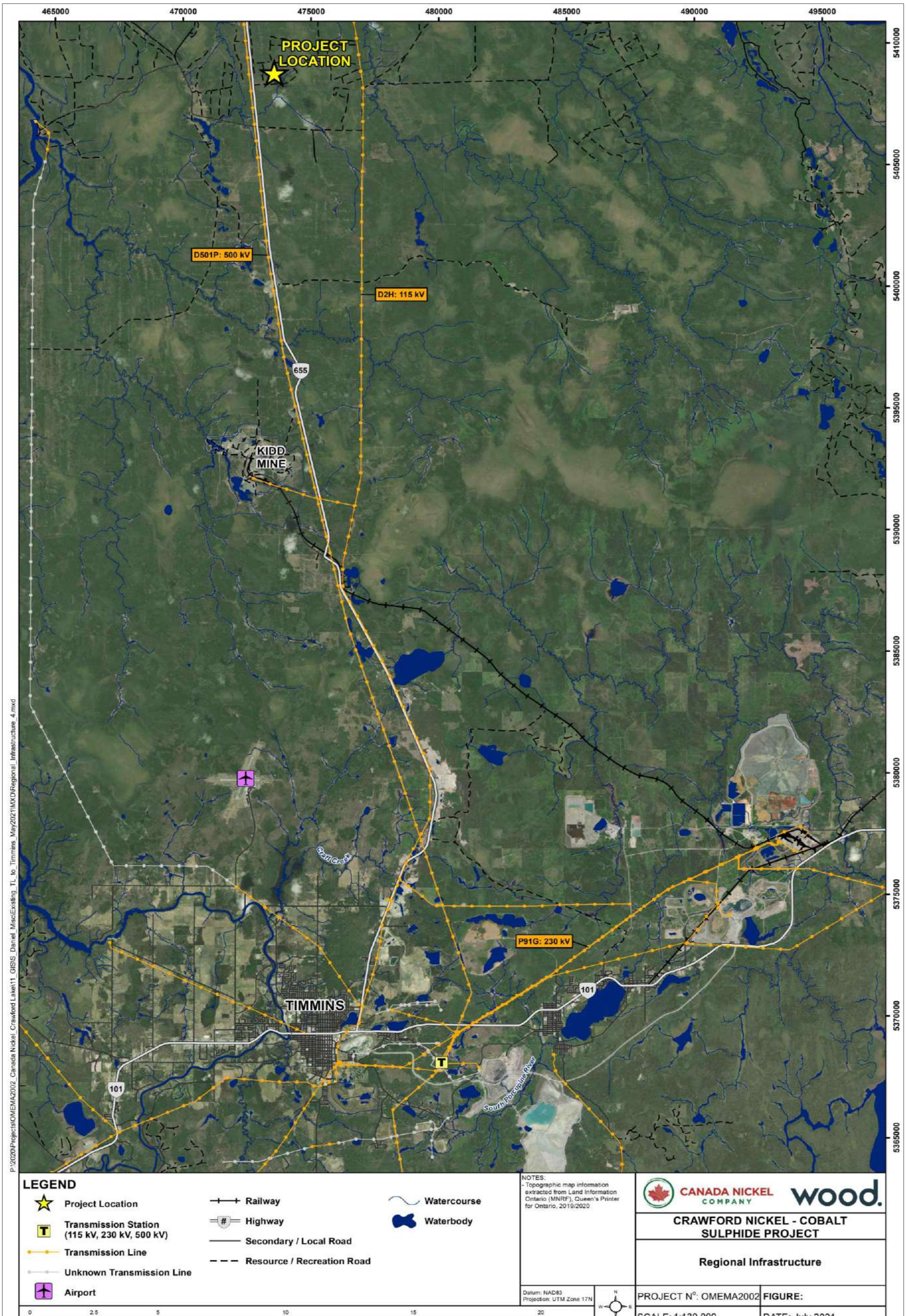
The current property boundary spans approximately 9,611 ha (96.11 $km^2$), consisting of a combination of patented lands (Crown patents) and unpatented mining claims (staked claims), as summarized in Tables 4-1 and 4-2 and shown in Figure 4-2.

As of the effective date of the report, CNC holds a 100% interest in the mining lands listed in Tables 4-1 and 4-2, subject to the terms of the Crawford Annex property purchase (CNC news release dated March 4, 2020), and a 2% NSR on the patented lands (Noble new releases dated December 3, 2019, and December 19, 2019) (see Section 4.4).

Specifically, the property comprises 116 Crown patents (freehold patented lands) in Crawford and Lucas townships that cover approximately 7,827.42 ha, and 148 single cell mining claims (SCMC) and two multi-cell mining claims (MCMC) in Crawford Township, covering approximately 3,246.66 hectares. Some mining claims overlap areas with patented lands. In this region of Ontario, one SCMC averages approximately 21.22 hectares.

The 116 Crown patents in Crawford and Lucas townships (Table 4-2) are for mineral rights only (i.e., CNC does not control the surface rights, except for one patent) and are registered with the Land Registry Office, District of Cochrane (LRO 06). The status of patented lands can be verified online through Teranet Express. There is one patented land and three non-mining land patents within the current property boundary, that are not owned or optioned by CNC (Figure 4-2).

**Figure 4-1: Regional-Scale Location of the Crawford Nickel Sulphide Project in Crawford Township**




Source: Wood Environment & Infrastructure Solutions Canada Limited, 2021.

**Figure 4-2: Land Tenure for the Crawford Property Superimposing Total Magnetic Intensity (TMI) Filtered Image**


Note: Current property boundary of the Crawford Project covering patented lands, unpatented mining claims, and Legacy Mining Claims (Tables 4-1 and 4-2). The empty areas inside of the project boundary are patented lands held by third parties. Geophysics source: MegaTEM, 2002. Source: CNC, 2023.

The *Ontario Mining Act* (2010) grants surface access to an unpatented mineral claim without owning the surface rights and given proper consultation with appropriate and relevant parties. Access to mining rights only patented lands or unpatented SCMC in which the CNC owns or has sub-surface rights only, requires that the surface rights owner be contacted in writing and that agreed-upon compensation be paid to the surface rights owner for any significant surface disturbances.

Annual holding costs for the 116 patented lands (mining tax) total approximately $37,557 and the required annual assessment work for the unpatented lands is approximately $35,200. Patented lands have an annual due date of approximately March 30 for payment of the mining land tax and related holding costs with payment due 60 days from invoicing, which is generally the end of January.

Based on the information provided by CNC and from what is available in the public domain, the authors confirm that all the unpatented and patented mining lands which comprise the Crawford Project are in good standing.

### 4.3.1 Mining Lands Tenure System

Traditional claim staking (physical staking) in Ontario came to an end on January 8, 2018; and on April 10, 2018, the Ontario Government converted all existing claims (referred to as Legacy Claims) into one or more "cell" claims or "boundary" claims as part of their new provincial grid system. The provincial grid is latitude- and longitude-based and made up of more than 5.2 million cells ranging in size from 17.7 ha in the north to 24 ha in the south. Dispositions such as leases, patents, and licenses of occupation were not affected by the new system. Mining claims are registered and administrated through the Ontario Mining Lands Administration System (MLAS), which is the online electronic system established by the Ontario Government for this purpose.

Mining claims can only be obtained by an entity (person or company) that holds a prospector's license granted by the MENDM (a "prospector"). A licensed prospector is permitted to enter onto provincial Crown and private lands that are open for exploration and stake a claim on those lands. Notice of the staked claim can then be recorded in the mining register maintained by the MENDM. Once the mining claim has been recorded, the prospector can apply for a permit to conduct exploratory and assessment work on the subject lands. To maintain the mining claim and keep it properly staked, the prospector must adhere to relevant staking regulations and conduct all prescribed work thereon. The prescribed work is currently set at $400 per annum per 16 ha claim unit. The prescribed work must be completed as no payments in lieu of work can be made in the first year. No minerals may be extracted from lands that are the subject of a mining claim—the prospector must possess either a mining lease or a freehold interest to mine the land, subject to all provisions of the *Ontario Mining Act*.

A mining claim can be transferred, charged, or mortgaged by the prospector without obtaining any consents. Notice of the change of owner of the mining claim, or charge thereof, should be recorded in the mining registry maintained by the MENDM.

**Table 4-1: Unpatented Mining Claims (SCMCs) in Crawford and Lucas Townships, Ontario**

| Legacy Claim | Tenure ID | Type | Anniversary | Holder (%) | Work Req/Year | Area (ha) | Description |
|---|---|---|---|---|---|---|---|
| 4252734 | 269339 | SCMC | 2024-10-04 | CNC (100%) | $400 | 21.22 | S 1/2 Lot 4 Con 1 |
| | 269338 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 250662 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 225503 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 332284 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 332283 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 316508 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 195379 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 130662 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| 4259542 | 333029 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 4 Con 5 |
| | 271941 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 205923 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 205922 | SCMC | 2024-10-04 | CNC (100%) | $400 | 21.22 | |
| | 167982 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 111361 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| 4263066 | 334714 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 4 Con 3 |
| | 325300 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 312574 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 305769 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 256605 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 256604 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 222029 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 171996 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 171995 | SCMC | 2024-10-05 | CNC (100%) | $400 | 21.22 | |
| 4267380 | 316442 | SCMC | 2024-09-29 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 4 Con 5 |
| | 309735 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 309734 | SCMC | 2024-09-29 | CNC (100%) | $200 | 21.22 | |
| | 309733 | SCMC | 2024-09-29 | CNC (100%) | $400 | 21.22 | |
| | 250456 | SCMC | 2024-10-04 | CNC (100%) | $400 | 21.22 | |
| | 242401 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 213242 | SCMC | 2024-09-29 | CNC (100%) | $400 | 21.22 | |
| | 193796 | SCMC | 2024-09-29 | CNC (100%) | $200 | 21.22 | |
| | 147108 | SCMC | 2024-09-29 | CNC (100%) | $200 | 21.22 | |
| | 130535 | SCMC | 2024-09-29 | CNC (100%) | $200 | 21.22 | |
| 4267381 | 327922 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 6 Con 6 |
| | 307890 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 230779 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 224133 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 224132 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 164725 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 139910 | SCMC | 2024-10-04 | CNC (100%) | $400 | 21.22 | |
| | 139909 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 113475 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| 4267382 | 337123 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 6 Con 4 |
| | 212249 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 129457 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 129456 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 109669 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| | 109668 | SCMC | 2024-10-04 | CNC (100%) | $200 | 21.22 | |
| 4267383 | 332101 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 6 Con 3 |
| | 328599 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 315319 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 308592 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 260736 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 249993 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 249992 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 212747 | SCMC | 2024-10-05 | CNC (100%) | $400 | 21.22 | |
| | 160092 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| 4267384 | 331720 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 8 Con 5 |
| | 331719 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 247901 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 247900 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 203341 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 164004 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 164003 | SCMC | 2024-10-05 | CNC (100%) | $400 | 21.22 | |
| | 158708 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |

| Legacy Claim | Tenure ID | Type | Anniversary | Holder (%) | Work Req/Year | Area (ha) | Description |
|---|---|---|---|---|---|---|---|
| 4267385 | 313693 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 8 Con 3 |
| | 275856 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 275855 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 203181 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 158482 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| 4267386 | 321076 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 9 Con 3 |
| | 254717 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 254716 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 254715 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 182533 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 169076 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 169075 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 169074 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| 4267387 | 332656 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 10 Con 3 |
| | 318442 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 271200 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 252439 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 167269 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 136045 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| 4267388 | 164776 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 12 Con 2 & S 1/2 Lot 11 Con 2 |
| | 194046 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 307924 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 260101 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 248653 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 224160 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 164775 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| 4269603 | 331930 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 10 Con 6 |
| | 314635 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| | 307898 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| | 307897 | SCMC | 2024-11-03 | CNC (100%) | $400 | 21.22 | |
| | 248624 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| | 230783 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| | 194029 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| | 159418 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| 4269604 | 321103 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 9 Con 5 |
| | 321102 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| | 291950 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 254736 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| | 105177 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| | 105176 | SCMC | 2024-11-03 | CNC (100%) | $200 | 21.22 | |
| 4269605 | 328041 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 11 Con 1 & Lot 10 Con 1 |
| | 328040 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 308512 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 260679 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 249915 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 231389 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 230810 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 165360 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 145978 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 136018 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 113580 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| 4269606 | 169058 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 12 Con 1 |
| | 152437 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 254689 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 304588 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 283834 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 217852 | SCMC | 2024-10-05 | CNC (100%) | $400 | 21.22 | |
| | 124556 | SCMC | 2024-10-05 | CNC (100%) | $200 | 21.22 | |
| | 590970 | MCMC | 2027-05-20 | CNC (100%) | $800 | 42.44 | S 1/2 Lot 4 Con 6 |
| | 585127 | SCMC | 2027-04-19 | CNC (100%) | $400 | 21.22 | N 1/2 Lot 7 Con 1 |
| | 585126 | SCMC | 2027-04-19 | CNC (100%) | $400 | 21.22 | N 1/2 Lot 7 Con 1 |
| | 585125 | SCMC | 2027-04-19 | CNC (100%) | $400 | 21.22 | S 1/2 Lot 7 Con 1 |
| | 585124 | SCMC | 2027-04-19 | CNC (100%) | $400 | 21.22 | S 1/2 Lot 7 Con 1 |
| | 585122 | SCMC | 2027-04-19 | CNC (100%) | $400 | 21.22 | S 1/2 Lot 6 Con 1 |
| | 585121 | SCMC | 2027-04-19 | CNC (100%) | $400 | 21.22 | N 1/2 Lot 6 Con 1 |
| | 585120 | SCMC | 2027-04-19 | CNC (100%) | $400 | 21.22 | N 1/2 Lot 6 Con 1 |
| | 561936 | MCMC | 2027-10-15 | CNC (100%) | $1,200 | 63.66 | N 1/2 Lot 6 Con 1 |
| | 535134 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 12 Con 3 |

| Legacy Claim | Tenure ID | Type | Anniversary | Holder (%) | Work Req/Year | Area (ha) | Description |
|---|---|---|---|---|---|---|---|
| | 535135 | SCMC | 2024-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 12 Con 3 |
| | 585128 | SCMC | 2027-04-19 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 7 Con 1 |
| | 585129 | SCMC | 2027-04-19 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 7 Con 1 |
| | 535143 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 12 Con 2 |
| | 535150 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 12 Con 2 |
| | 535151 | SCMC | 2024-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 12 Con 3 |
| | 535136 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 3 Con 2 |
| | 535137 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 2 Con 2 |
| | 535132 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 2 Con 2 |
| | 535133 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 1 Con 2 |
| | 535138 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 1 Con 2 |
| | 535140 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 3 Con 2 |
| | 535144 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | S 1/2 Lot 1 Con 2 |
| | 535146 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 2 Con 2 |
| | 535152 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 2 Con 2 |
| | 535153 | SCMC | 2026-11-17 | CNC (100%) | $200 | 21.22 | N 1/2 Lot 2 Con 2 |
| | | | | **Totals:** | **$35,200** | **3,246.66** | |

**Table 4-2: Crown Patented Lands (Mineral Rights Only) in Crawford and Lucas Townships, Ontario**

| Township | Mining Right Number | Parcel | PIN | Area (ha) | Tax |
|---|---|---|---|---|---|
| Lucas | PAT-49293 | | 65320-0144(LT) | 64.75 | $259.00 |
| Lucas | PAT-49294 | | 65320-0132(LT) | 64.75 | $259.00 |
| Lucas | PAT-49295 | | 65320-0142(LT) | 64.55 | $258.19 |
| Lucas | PAT-49301 | | 65320-0130(LT) | 64.55 | $258.19 |
| Lucas | PAT-49531 | 539SND | 65320-0176(LT) | 64.75 | $259.00 |
| Lucas | PAT-49532 | 688SND | 65320-0156(LT) | 64.75 | $259.00 |
| Lucas | PAT-49533 | 531SND | 65320-0174(LT) | 64.55 | $258.19 |
| Lucas | PAT-49534 | 511SND | 65320-0154(LT) | 64.55 | $258.19 |
| Lucas | PAT-49544 | 508SND | 65320-0198(LT) | 64.95 | $259.81 |
| Lucas | PAT-49545 | 541SND | 65320-0196(LT) | 64.95 | $259.81 |
| Lucas | PAT-49546 | 516SND | 65320-0218(LT) | 64.55 | $258.19 |
| Lucas | PAT-49547 | 513SND | 65320-0194(LT) | 64.55 | $258.19 |
| Crawford | PAT-49876 | 4479NEC | 65321-0138(LT) | 65.36 | $261.43 |
| Crawford | PAT-49877 | 4477NEC | 65321-0140(LT) | 65.36 | $261.43 |
| Crawford | PAT-49882 | 4501NEC | 65321-0150(LT) | 64.95 | $259.81 |
| Crawford | PAT-49883 | 4505NEC | 65321-0152(LT) | 64.55 | $258.19 |
| Crawford | PAT-49884 | 4519NEC | 65321-0154(LT) | 64.55 | $258.19 |
| Crawford | PAT-49885 | 818NEC | 65321-0156(LT) | 64.95 | $259.81 |
| Crawford | PAT-49886 | 4523NEC | 65321-0158(LT) | 64.95 | $259.81 |
| Crawford | PAT-49887 | 4509NEC | 65321-0160(LT) | 64.34 | $257.38 |
| Crawford | PAT-49888 | 4504NEC | 65321-0162(LT) | 64.34 | $257.38 |
| Crawford | PAT-49889 | 5336NEC | 65321-0164(LT) | 65.96 | $263.86 |
| Crawford | PAT-49890 | 4510NEC | 65321-0166(LT) | 65.96 | $263.84 |
| Crawford | PAT-49891 | 4550NEC | 65321-0168(LT) | 64.95 | $259.81 |
| Crawford | PAT-49892 | 4586NEC | 65321-0170(LT) | 64.75 | $259.00 |
| Crawford | PAT-49893 | 4518NEC | 65321-0172(LT) | 64.34 | $257.38 |
| Crawford | PAT-49894 | 4536NEC | 65321-0174(LT) | 64.55 | $258.19 |
| Crawford | PAT-49895 | 4503NEC | 65321-0176(LT) | 64.55 | $258.19 |
| Crawford | PAT-49896 | 4508NEC | 65321-0178(LT) | 63.94 | $255.76 |
| Crawford | PAT-49897 | 4555NEC | 65321-0180(LT) | 63.94 | $255.76 |
| Crawford | PAT-49898 | 4506NEC | 65321-0182(LT) | 65.96 | $263.86 |
| Crawford | PAT-49899 | 4522NEC | 65321-0184(LT) | 64.75 | $259.00 |
| Crawford | PAT-49900 | 4533NEC | 65321-0186(LT) | 64.55 | $258.19 |
| Crawford | PAT-49901 | 971NEC | 65321-0188(LT) | 64.55 | $258.19 |
| Crawford | PAT-49902 | 4094NEC | 65321-0190(LT) | 64.55 | $258.19 |
| Crawford | PAT-49903 | 4507NEC | 65321-0192(LT) | 64.95 | $259.81 |
| Crawford | PAT-49904 | 4470NEC | 65321-0194(LT) | 64.95 | $259.81 |
| Crawford | PAT-49905 | 4484NEC | 65321-0196(LT) | 64.14 | $256.57 |
| Crawford | PAT-49906 | 4558NEC | 65321-0198(LT) | 65.96 | $263.86 |
| Crawford | PAT-49907 | 4542NEC | 65321-0200(LT) | 64.75 | $259.00 |
| Crawford | PAT-49908 | 2995NEC | 65321-0202(LT) | 64.75 | $259.00 |
| Crawford | PAT-49909 | 4445NEC | 65321-0204(LT) | 64.75 | $259.00 |
| Crawford | PAT-49910 | 4116NEC | 65321-0206(LT) | 64.75 | $259.00 |
| Crawford | PAT-49911 | 972NEC | 65321-0208(LT) | 64.75 | $259.00 |
| Crawford | PAT-49912 | 4547NEC | 65321-0210(LT) | 64.95 | $259.81 |
| Crawford | PAT-49913 | 4490NEC | 65321-0212(LT) | 64.95 | $259.81 |
| Crawford | PAT-49914 | 4647NEC | 65321-0214(LT) | 63.94 | $255.76 |
| Crawford | PAT-49915 | 4645NEC | 65321-0216(LT) | 65.96 | $263.86 |
| Crawford | PAT-49916 | 4652NEC | 65321-0218(LT) | 65.96 | $263.86 |
| Crawford | PAT-49917 | 4480NEC | 65321-0220(LT) | 64.75 | $259.00 |
| Crawford | PAT-49918 | 4666NEC | 65321-0222(LT) | 63.94 | $255.76 |
| Crawford | PAT-49919 | 4668NEC | 65321-0224(LT) | 63.94 | $255.76 |
| Crawford | PAT-49920 | 4521NEC | 65321-0226(LT) | 64.55 | $258.19 |
| Crawford | PAT-49921 | 4497NEC | 65321-0228(LT) | 64.55 | $258.19 |
| Crawford | PAT-49922 | 637NEC | 65321-0230(LT) | 64.75 | $259.00 |
| Crawford | PAT-49923 | 656NEC | 65321-0232(LT) | 64.75 | $259.00 |
| Crawford | PAT-49924 | 4515NEC | 65321-0234(LT) | 64.14 | $256.57 |
| Crawford | PAT-49925 | 4659NEC | 65321-0236(LT) | 64.14 | $256.57 |
| Crawford | PAT-49926 | 5334NEC | 65321-0238(LT) | 65.96 | $263.86 |
| Crawford | PAT-49927 | 4651NEC | 65321-0240(LT) | 65.96 | $263.86 |
| Crawford | PAT-49928 | 4543NEC | 65321-0242(LT) | 60.50 | $242.00 |
| Crawford | PAT-49930 | 4679NEC | 65321-0244(LT) | 60.50 | $242.00 |
| Crawford | PAT-49931 | 4446NEC | 65321-0246(LT) | 62.93 | $251.72 |
| Crawford | PAT-49932 | 4471NEC | 65321-0248(LT) | 62.93 | $251.72 |
| Crawford | PAT-49933 | 4540NEC | 65321-0250(LT) | 64.75 | $259.00 |
| Crawford | PAT-49934 | 4541NEC | 65321-0252(LT) | 65.15 | $260.62 |
| Crawford | PAT-49935 | 4516NEC | 65321-0254(LT) | 64.55 | $258.19 |

| Township | Mining Right Number | Parcel | PIN | Area (ha) | Tax |
|---|---|---|---|---|---|
| Crawford | PAT-49938 | 4437NEC | 65321-0256(LT) | 64.55 | $258.19 |
| Crawford | PAT-49939 | 4648NEC | 65321-0258(LT) | 65.96 | $263.86 |
| Crawford | PAT-49940 | 4599NEC | 65321-0260(LT) | 64.75 | $259.00 |
| Crawford | PAT-49941 | 4657NEC | 65321-0262(LT) | 64.75 | $259.00 |
| Crawford | PAT-49942 | 3252NEC | 65321-0264(LT) | 64.75 | $259.00 |
| Crawford | PAT-49944 | 4502NEC | 65321-0266(LT) | 64.75 | $259.00 |
| Crawford | PAT-49945 | 4517NEC | 65321-0268(LT) | 64.75 | $259.00 |
| Crawford | PAT-49946 | 4524NEC | 65321-0270(LT) | 64.75 | $259.00 |
| Crawford | PAT-49948 | 4598NEC | 65321-0272(LT) | 64.75 | $259.00 |
| Crawford | PAT-49949 | 7747NEC | 65321-0274(LT) | 128.28 | $513.14 |
| Crawford | PAT-49950 | 7748NEC | 65321-0276(LT) | 131.93 | $527.71 |
| Crawford | PAT-49951 | 7742NEC | 65321-0134(LT) | 64.75 | $259.00 |
| Crawford | PAT-49952 | 7743NEC | 65321-0278(LT) | 129.50 | $518.00 |
| Crawford | PAT-49953 | 7745NEC | 65321-0283(LT) | 127.88 | $511.52 |
| Crawford | PAT-49954 | 4649NEC | 65321-0285(LT) | 64.95 | $259.81 |
| Crawford | PAT-49955 | 7741NEC | 65321-0287(LT) | 129.50 | $518.00 |
| Crawford | PAT-49956 | 4093NEC | 65321-0289(LT) | 64.75 | $259.00 |
| Crawford | PAT-49957 | 4616NEC | 65321-0291(LT) | 64.75 | $259.00 |
| Crawford | PAT-49958 | 4496NEC | 65321-0293(LT) | 64.75 | $259.00 |
| Crawford | PAT-49959 | 4537NEC | 65321-0295(LT) | 64.75 | $259.00 |
| Crawford | PAT-49960 | 663NEC | 65321-0297(LT) | 64.75 | $259.00 |
| Crawford | PAT-49961 | 4440NEC | 65321-0299(LT) | 64.75 | $259.00 |
| Crawford | PAT-49962 | 4101NEC | 65321-0301(LT) | 64.34 | $257.38 |
| Crawford | PAT-49963 | 4642NEC | 65321-0303(LT) | 65.56 | $262.24 |
| Crawford | PAT-49964 | 4646NEC | 65321-0305(LT) | 65.56 | $262.24 |
| Crawford | PAT-49965 | 4115NEC | 65321-0307(LT) | 64.75 | $259.00 |
| Crawford | PAT-49966 | 4583NEC | 65321-0309(LT) | 64.75 | $259.00 |
| Crawford | PAT-49967 | 4488NEC | 65321-0311(LT) | 64.55 | $258.19 |
| Crawford | PAT-49968 | 4580NEC | 65321-0313(LT) | 64.75 | $259.00 |
| Crawford | PAT-49969 | 4653NEC | 65321-0315(LT) | 64.75 | $259.00 |
| Crawford | PAT-49970 | 4557NEC | 65321-0317(LT) | 64.75 | $259.00 |
| Crawford | PAT-49971 | 4436NEC | 65321-0319(LT) | 64.75 | $259.00 |
| Crawford | PAT-49972 | 4100NEC | 65321-0321(LT) | 64.55 | $258.19 |
| Crawford | PAT-49973 | 4099NEC | 65321-0323(LT) | 64.55 | $258.19 |
| Crawford | PAT-49974 | 4105NEC | 65321-0325(LT) | 65.36 | $261.43 |
| Crawford | PAT-49975 | 4104NEC | 65321-0327(LT) | 65.36 | $261.43 |
| Crawford | PAT-49976 | 4658NEC | 65321-0329(LT) | 64.75 | $259.00 |
| Crawford | PAT-49977 | 4439NEC | 65321-0331(LT) | 64.75 | $259.00 |
| Crawford | PAT-49978 | 4674NEC | 65321-0333(LT) | 64.75 | $259.00 |
| Crawford | PAT-49979 | 4514NEC | 65321-0335(LT) | 64.75 | $259.00 |
| Crawford | PAT-49980 | 976NEC | 65321-0337(LT) | 64.75 | $259.00 |
| Crawford | PAT-49981 | 4511NEC | 65321-0339(LT) | 64.75 | $259.00 |
| Crawford | PAT-49982 | 4095NEC | 65321-0341(LT) | 64.75 | $259.00 |
| Crawford | PAT-49983 | 4096NEC | 65321-0343(LT) | 64.75 | $259.00 |
| Crawford | PAT-49984 | 4098NEC | 65321-0345(LT) | 64.75 | $259.00 |
| Crawford | PAT-49985 | 4097NEC | 65321-0347(LT) | 64.75 | $259.00 |
| Crawford | PAT-49986 | 4103NEC | 65321-0349(LT) | 64.55 | $258.19 |
| Crawford | PAT-49987 | 4102NEC | 65321-0351(LT) | 64.55 | $258.19 |
| Crawford | PAT-599335 | | 65321-0280(LT) | 64.34 | $257.38 |
| | | | **Total** | **7,827.42** | **$31,309.85** |

#### 4.3.2 Mining Lease

If a prospector wants to extract minerals, the prospector may apply to the MENDM for a mining lease. A mining lease, which is usually granted for a term of 21 years, grants an exclusive right to the lessee to enter upon and search for, and extract, minerals from the land, subject to the prospector obtaining other required permits and adhering to applicable regulations.

Pursuant to the provisions of the *Ontario Mining Act* (the "Act"), the holder of a mining claim is entitled to a lease if it has complied with the provisions of the Act in respect of those lands. An application for a mining lease may be submitted to the MENDM at any time after the first prescribed unit of work in respect of the mining claim is performed and approved. The application for a mining lease must specify whether it requests a lease of mining and surface rights or mining rights only and requires the payment of fees.

A mining lease can be renewed by the lessee upon submission of an application to the MENDM within 90 days before the expiry date of the lease, provided that the lessee provides documentation and satisfies the criteria set forth in the Act in respect of a lease renewal.

A mining lease cannot be transferred or mortgaged by the lessee without the prior written consent of the MENDM. The consent process generally takes between two and six weeks and requires the lessee to submit various documentations and pay a fee.

#### 4.3.3 Freehold Mining Lands

A prospector interested in removing minerals from the ground may, instead of obtaining a mining lease, make an application to the Ontario Ministry of Natural Resources (MNR) to acquire the freehold interest in the subject lands. If the application is approved, the freehold interest is conveyed to the applicant by way of the issuance of a mining patent. A mining patent can include surface and mining rights or mining rights only.

The issuance of mining patents is much less common today than in the past, and most prospectors will obtain a mining lease in order to extract minerals. If a prospector is issued a mining patent, the mining patent vests in the patentee all the provincial Crown's title to the subject lands and to all mines and minerals relating to such lands, unless something to the contrary is stated in the patent.

As the holder of a mining patent enjoys the freehold interest in the lands that are the subject of such patent, no consents are required for the patentee to transfer or mortgage those lands.

#### 4.3.4 License of Occupation

Prior to 1964, Mining Licenses of Occupation (MLO) were issued, in perpetuity, by the MENDM to permit the mining of minerals under the beds of bodies of water. MLOs were associated with portions of mining claims overlying adjacent land. As an MLO is held separate and apart from the related mining claim, it must be transferred separately from the transfer of the related mining claim. The transfer of an MLO requires the prior written consent of the MENDM. As an MLO is a license, it does not create an interest in the land.

#### 4.3.5 Land Use Permit

Prospectors may also apply for and obtain a Land Use Permit from the MNR. A Land Use Permit is considered the weakest form of mining tenure. It is issued for a period of 10 years or less and is generally used where there is no intention to erect extensive or valuable improvements on the subject lands. Land Use Permits are often obtained when the land is to be used for the purposes of an exploration camp. When a Land Use Permit is issued, the MNR retains future options for the subject lands and controls their use. Land Use Permits are personal to the holder and cannot be transferred or used as security.

### 4.4 Royalties, Agreements and Encumbrances

On December 19, 2019, Noble announced that it had completed the acquisition of the 5% net smelter return royalty (NSR) applicable to ~55,000 ha of patented mineral rights on its project 81 in the Timmins-Cochrane area of northern Ontario; as a result, those patented properties are now subject to a 2% NSR (Noble news releases dated October 24, 2019, and November 28, 2019). The terms of this acquisition apply to the patented lands transferred to CNC (Noble news releases dated December 3, 2019) and which comprise part of the current project.

A third party holds the rights to receive a 5% interest in the net profits (the NPI royalty) earned in respect of the sale of all minerals located on or under the subject lands. Net profits are calculated by deducting all costs, including capital costs, operating costs, royalties and the IBA, from the gross proceeds. The obligation to make NPI payments arises upon the commencement of commercial production. Up to 2.5% of the NPI (i.e., 50% of the total royalty) may be repurchased for C$0.8 million per 1%. For example, the royalty can be bought down by 2.5% for C$2 million.

### 4.5 Permitting Considerations

The project will require the completion of a federal Impact Assessment pursuant to the *Impact Assessment Act*, as well as provincial Class Environmental Assessments in accordance with the Ontario *Environmental Assessment Act*. An initial project description was submitted to the Impact Assessment Agency of Canada (IAAC) in August 2022 to formally start the federal process and was followed by submission of a detailed project description to IAAC on December 22, 2022. A determination was made by IAAC on January 5, 2023 that a federal impact assessment is required and the IAAC published the Tailored Impact Statement Guidelines on March 31, 2023. Completion of the federal Impact Assessment and provincial Class Environmental Assessments as well as future conditions of approval could require refinements to the project components or additional mitigation measures to be implemented.

Several acts and regulations are applicable to the project, as noted in Section 20, titled Environmental Studies, Permitting, and Social or Community Impact, and these will be addressed throughout the environmental assessment (EA) and permitting processes. Although the project is large-scale, there are currently no known indicators that these processes cannot be completed successfully.

### 4.6 Environmental Considerations

The project site is situated within the Northern Clay Belt in northeastern Ontario, within the Timmins-Cochrane Mining Camp, a region with a robust mining history. The project site is generally undeveloped except for existing provincial infrastructure (highway, transmission lines) that cross portions of the lands, and prior impacts as a result of mineral exploration and forestry operations. There are no known existing environmental liabilities. Vegetation communities consist of a mix of deciduous and coniferous forests, and low-lying wetland areas / muskeg. Drainage is mainly via the North Driftwood River and West Buskegau River.

Environmental baseline studies were initiated in 2021 and are currently ongoing covering aspects of atmospheric, water resources, geochemistry, terrestrial environment, aquatic environment, and human and socio-environment. The environmental baseline studies will be used to identify environmental constraints, which may require refinements to the project components or additional mitigation measures to be implemented.

Environmental management and monitoring plans are listed in Chapter 20 and will be developed for the project as part of the environmental approvals process. Management plans will be completed with direction from the applicable regulatory agencies in the event species at risk are encountered during project activities. The timing of construction activities will be arranged in accordance with the appropriate freshwater fisheries timing and breeding bird windows for the project area, unless otherwise approved by the applicable regulatory agency. Water takings during construction and operations will comply with applicable provincial permits to take water and guidance from the Department of Fisheries and Oceans Canada to avoid entrapment and impingement of fish.

## 4.7 Social License Considerations

Social license considerations are discussed in Chapter 20.

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

## 5.1 Accessibility

The Crawford Project is located approximately 42 km north of the City of Timmins, Ontario, in the geographic townships of Crawford, Carnegie, Kidd, Lucas, and Prosser. A small portion of the project is within the Kidd Township within the municipal boundary of the City of Timmins.

The nearest larger communities are the town of Cochrane (35 km to the northeast), the city of Timmins (42 km to the south), the town of Smooth Rock Falls (50 km to the northwest), and the town of Iroquois Falls (50 km to the east).

The property is readily accessible year-round via the regional infrastructure network, and is bisected by Highway 655, which is paved. Access to the property is by a network of local bush roads accessed from Highway 655. Highway 655 connects to Highway 11, the Trans-Canada Highway located north of the site. The Ontario Northland Railway operates a spur that services the Kidd Creek Mine located south of the site, between Timmins and the site. The Ontario Northland Railway spur connects with the provincial railway network east of Timmins which provides access to the North America railway network. The site is readily accessible from Timmins from the south, as well as the communities of Smooth Rock Falls and Cochrane, Ontario, to the north on Highway 11. Freight rail access is available in Timmins and Cochrane.

The Timmins Municipal airport (Timmins Victor M. Power Airport) is serviced by several daily flights to Toronto Pearson International Airport and Billy Bishop Island Airport. There are no lakes proximal to the site that are large enough to offer float plane access.

## 5.2 Climate

The nearest regional climate monitoring station to the project site that has an extensive data record is located at the Timmins Victor M. Power Airport (VPA) (Environment and Climate Change Canada Climate ID 6078285). This station is located approximately 28.8 km south of the site and has a data record of 57 years from 1955-2012. Timmins A station (Environment and Climate Change Canada Climate ID 6078286) is also located approximately 28.8 km from the site at the same elevation as Timmins VPA and was used to extend the climate dataset from 2012-2021. To achieve a continuous record for both precipitation and temperature, additional stations on the Project site were used to infill missing data points, with priority given to the closest sites. The combined temperature and precipitation data was then analyzed from 1990 to 2019 to determine averages and trends and are detailed in Table 5-1 (Golder, 2022).

The site has cold winters and warm summers typical of northeastern Ontario. Maximum daily average temperatures occur in July and lowest daily average temperatures occur in January. The annual temperature has a statistically significant upwards trend of 0.2°C per decade. Most precipitation occurs in the summer and fall months with July being the wettest month (91.7 mm) and February the driest month (42.3 mm). The annual rainfall has a statistically significant downwards trend of -15.6 mm per decade, indicating a shift to drier conditions (Golder, 2022).

**Table 5-1: 1990 to 2019 Climate Long-Term Averages and Trends**

| Climate Variables | Average | Minimum | Maximum | Decadal Trend | Statistical Significance |
|---|---|---|---|---|---|
| **Average Temperature (°C)** | | | | | |
| Annual | 1.6 | 0.0 | 4.1 | +0.2 | Significant at the 99th percentile |
| January | -16.8 | -25.7 | -10.0 | +0.4 | Significant at the 90th percentile |
| February | -14.8 | -23.0 | -6.8 | +0.3 | Not statistically significant |
| March | -7.9 | -14.3 | -0.2 | +0.2 | Not statistically significant |
| April | 1.1 | -3.3 | 5.8 | 0 | No trend |
| May | 9.4 | 5.8 | 13.5 | +0.1 | Not statistically significant |
| June | 14.8 | 11.9 | 18.5 | +0.2 | Not statistically significant |
| July | 17.5 | 13.7 | 20.0 | +0.2 | Significant at the 90th percentile |
| August | 15.9 | 13.0 | 18.9 | +0.2 | Significant at the 95th percentile |
| September | 11.0 | 7.8 | 14.6 | +0.4 | Significant at the 99th percentile |
| October | 4.5 | 1.0 | 9.3 | 0 | No trend |
| November | -3.5 | -9.1 | 1.8 | +0.2 | Not statistically significant |
| December | -12.5 | -21.6 | -4.5 | +0.6 | Significant at the 95th percentile |
| **Total Precipitation (mm Equivalent)** | | | | | |
| Annual | 842.1 | 521.4 | 1,129.7 | -15.6 | Significant at the 90th percentile |
| January | 51.7 | 13 | 109.6 | -2.7 | Significant at the 90th percentile |
| February | 42.3 | 7.2 | 95.7 | -2.6 | Not statistically significant |
| March | 54.8 | 9.8 | 155.6 | -3.8 | Significant at the 90th percentile |
| April | 55.7 | 6.2 | 114.8 | +2.8 | Not statistically significant |
| May | 69.8 | 21.2 | 136.8 | -2.5 | Not statistically significant |
| June | 87.0 | 25.7 | 216.1 | -1.8 | Not statistically significant |
| July | 91.7 | 33.3 | 187.2 | -3.9 | Not statistically significant |
| August | 86.1 | 9 | 202.2 | -4.7 | Not statistically significant |
| September | 88.2 | 25.9 | 168.9 | -1.4 | Not statistically significant |
| October | 81.7 | 32.2 | 182.1 | +4.6 | Significant at the 95th percentile |
| November | 73.3 | 19.6 | 150 | -1.5 | Not statistically significant |
| December | 59.9 | 20.2 | 106.9 | -2.6 | Significant at the 90th percentile |

Note: Data presented above is a combination of Environment and Climate Change Canada station Timmins VPA (ID 6078285) and Timmins A (ID 6078286) with missing precipitation or temperature data infilled by other Environment and Climate Change Canada stations, with priority given to closest stations to the project site. The European Centre for Medium-Range Weather Forecasts Re-Analysis dataset (corrected to the Timmins Victor Power A station) was used when no other observations were present. Source: (Environment and Climate Change Canada Climate ID 6078285 and 6078286) (Golder, 2022).

Current wet and dry year precipitation statistics were calculated with the combined Timmins VPA (ID 6078285) and Timmins A (ID 6078286) dataset. The wet and dry year annual total precipitation for a return period of 5 through 100 years is presented in Table 5-2. The increase of annual total precipitation with return period was found to be greater for the smaller return periods.

**Table 5-2: Wet and Dry Year Annual Total Precipitation Frequency Analysis**

| Return Period (years) | Annual Total Precipitation (mm) | |
|---|---|---|
| | Wet Year | Dry Year |
| 5 | 967.7 | 696.3 |
| 10 | 995.7 | 664.4 |
| 20 | 1,018.9 | 638.1 |
| 50 | 1,044.9 | 608.5 |
| 100 | 1,062.3 | 588.8 |

Source: Golder 2022.

Extreme precipitation events consider both rainfall and snowfall and were calculated over a range of return periods and for a duration of one to three days, as presented in Table 5-3. Extreme rainfall showed similar amounts across the durations and return periods, indicating snowfall is likely not a major component of extreme precipitation events, and these may occur more often in the summer and fall months.

**Table 5-3: Extreme Precipitation Statistics Across Durations and Return Periods (mm)**

| Duration (days) | Return Period (Years) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | 2 | 5 | 10 | 20 | 50 | 100 | 200 | 500 | 1,000 | 5,000 | 10,000 |
| 1-Day | 41.6 | 57.4 | 67.9 | 78.0 | 91.0 | 100.7 | 110.4 | 123.2 | 132.9 | 155.4 | 165.1 |
| 2-Day | 52.2 | 70.9 | 83.2 | 95.1 | 110.4 | 121.9 | 133.4 | 148.5 | 159.9 | 186.4 | 197.8 |
| 3-Day | 58.2 | 77.5 | 90.2 | 102.4 | 118.3 | 130.1 | 142.0 | 157.6 | 169.4 | 196.7 | 208.5 |

Source: Golder, 2022.

The probable maximum precipitation (PMP) was calculated using the combined climate baseline dataset and incorporates the Timmins Storm, which occurred August 31 to September 1, 1961. The probable maximum flood (PMF) corresponds to the amount of runoff generated from the PMP event and the potential 100-year snowmelt generated in the same duration. The PMP and PMF are summarized for a 1- to 3-day duration for both the spring and annual events in Table 5-4. The annual potential estimated evapotranspiration rate for the project site ranged between a minimum of 521.4 mm and a maximum of 1,129.7 mm, with an average of 842.1 mm. The highest evapotranspiration rates were observed to occur in July (150.7 mm).

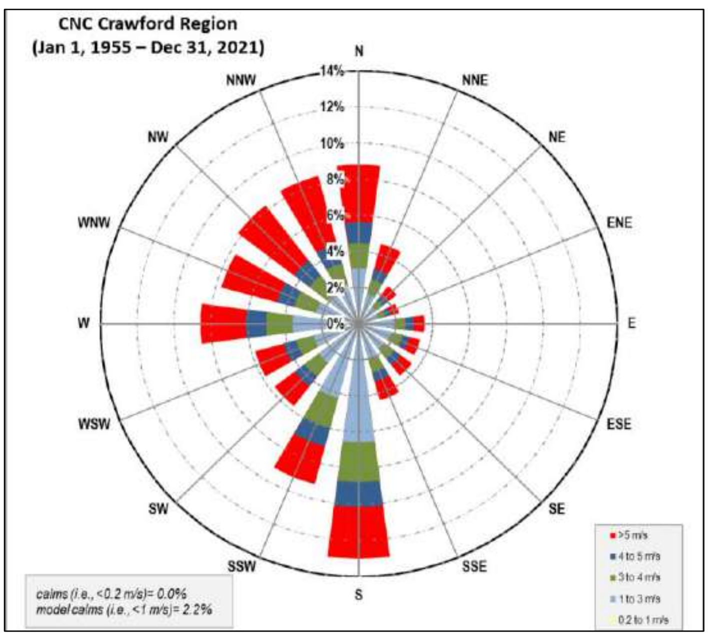
**Table 5-4: Annual and Spring PMP and PMF for Duration of 1 to 3 Days**

| PMF Event | PMP (mm) | 100-Year Snowmelt (mm) | PMF (mm) |
|---|---|---|---|
| 1-Day Spring | 195.3 | 36.9 | 232.2 |
| 1-Day Annual | 363.0 | - | 363.0 |
| 2-Day Spring | 265.7 | 70.8 | 336.5 |
| 2-Day Annual | 415.1 | - | 415.1 |
| 3-Day Spring | 328.2 | 101.3 | 429.5 |
| 3-Day Annual | 415.1* | - | 415.1 |

*Note: The 3-Day annual PMP was estimated as 414 mm, which is lower than the 2-Day annual PMP. The 2-day annual PMP is used as the 3-Day annual PMP to be conservative. Source: Golder, 2022.

The hourly wind observations for Timmins VPA and Timmins A Environment and Climate Change Canada stations were combined for a data range of 1955-2021 to develop a wind rose as presented in Figure 5-1. A southern wind was most common with relatively calm speeds between 1-3 m/s. Westerly and northerly winds had the greatest frequency of higher windspeeds in excess of 5 m/s. Maximum monthly wind gusts ranged from 85 km/h (July 2001) to 158 km/h (June 1956) from Timmins VPA station and were generally blowing from a westerly direction.

**Figure 5-1: Magnitude and Frequency of Hourly Wind Speed and Direction in the CNC Crawford Region**




Source: Golder, 2022.

Relative humidity climate normal are available for the Timmins VPA climate station for 1971-2000. The project site has an average relative humidity of 83.2% (6 AM) and 59.5% (3 PM) annually. Morning humidity ranges from 75.0% in January to 91.8% in August, while the afternoon humidity ranges from 46.3% in May to 74.6% in November.

## 5.3 Local Resources and Infrastructure

The site is readily accessible from Timmins from the south, as well as the communities of Smooth Rock Falls and Cochrane to the north on Highway 11, using Highway 655, a two-lane paved provincial highway. The property mining rights consist primarily of patented mining claims and unpatented mining claims with mining rights only. It is planned to place most of the mining facilities on patented lands, although some infrastructure may be located on provincial Crown lands.

Other local infrastructure includes a regional transmission line (500 kV), which parallels Highway 655 and brings power from the north through to the primary Hydro One transmission station east of Timmins. In addition, a 115 kV transmission line is located east of the project site. Approximately 17 km south of the project site there is a private rail spur operated by Ontario Northland Railway that connects the Kidd Mine with the federal railway network that provides access to the North American railway system.

The Cochrane District, which includes the City of Timmins and towns of Cochrane, Iroquois Falls, and Smooth Rock Falls, has a population of 77,965. The City of Timmins is the fourth largest city in northeastern Ontario in area and serves as a regional service and distribution centre. Based on the 2021 Census, the population of Timmins was 41,145, a decrease of 1.5% since 2016. Cochrane had a population of 5,390 in 2021, which indicates a 1.3% increase in population since 2016. Iroquois Falls had a population of 4,418 in 2021 which is a 3% decline from the 2016 population. Smooth Rock Falls had a population of 1,200 in 2021, which is a population decrease of 10% from 2016. The Cochrane District economy is based on natural resource extraction and is supported by industries related to the mining of metals (e.g., gold, zinc, copper, nickel, and silver), diamonds, and forestry.

The local service industry is well-prepared to service mining industry. There are more than 100 mining contractors and suppliers within Timmins and the surrounding communities listed in the Economic Development Mining Supply and Services database. Available goods and services include industrial and heavy equipment sales, rental, and servicing; analytical laboratory services; drilling; construction and engineering services; and exploration services.

## 5.4 Physiography

Topography within the project area is undulating and generally ranges between 265 and 290 m amsl, averaging about 15 m in relief. The project site is in the Northern Clay Belt, which is characterized as a low lying, undulating plain of glaciolacustrine clay. Underlying the clay is a discontinuous sand followed by glacial till (silt and/or clay) that overlies bedrock. A north-south trending esker composed of sand and gravel is located about 500 m west of the proposed tailings management facility. There is only a small proportion of outcrop exposure, mostly confined to higher ground. Overburden thickness ranges from about 10 m to 90 m thick with an average thickness of about 40 m across the project site. Drainage across the project site is poor due to the surficial clay.

The project site and local area are covered by a mix of wetland, mixed deciduous and boreal forest complexes with many streams and rivers and few small lakes. Vegetation communities that occur directly within and adjacent to the proposed project site are generally early successional mixed deciduous communities that have been shaped by timber harvest and infrastructure development; however, a few fragmented mature coniferous forest areas remain intact. As with most of northern Ontario, the site is crossed by several minor waterbodies and tributaries to larger rivers. The project site is located primarily between the North Driftwood River and the West Buskegau River both of which drain north into the Abitibi River. Jocko Creek crosses the southern portion of the project site and drains into Kidd Creek and subsequently the Mattagami River.

## 5.5 Seismicity

Northern Ontario has a very low level of seismic activity. From 1970 to 1999, on average only one or two magnitude 2.5 or greater earthquakes have been recorded in this large area. Two magnitude five earthquakes (1905, northern Michigan, and 1928, northwest of Kapuskasing) have occurred in this region (Natural Resources Canada, 2021).

The 2015 National Building Code seismic hazard calculations for the Crawford Project site are summarized in Table 5-5.

The information in this section is derived from the following sources:

- Natural Resources Canada 2021 Earthquake zones in Eastern Canada (online website)
- National Building Code of Canada 2015 NRCC no. 56190; Appendix C: Table C-3, Seismic Design Data for Selected Locations in Canada
- Structural Commentaries (User's Guide - NBC 2015: Part 4 of Division B) Commentary J: Design for Seismic Effects
- Geological Survey of Canada Open File 7893 Fifth Generation Seismic Hazard Model for Canada: Grid values of mean hazard to be used with the 2015 National Building Code of Canada.

**Table 5-5: 2015 National Building Code Seismic Hazard Calculation**

| Probability of Exceedance per Annum | 0.000404 | 0.001 | 0.0021 | 0.01 |
|---|---|---|---|---|
| **Probability of Exceedance in 50 Years** | **2%** | **5%** | **10%** | **40%** |
| Sa (0.05) | 0.177 | 0.083 | 0.041 | 0.008 |
| Sa (0.1) | 0.215 | 0.108 | 0.057 | 0.013 |
| Sa (0.2) | 0.179 | 0.093 | 0.053 | 0.015 |
| Sa (0.3) | 0.133 | 0.072 | 0.043 | 0.013 |
| Sa (0.5) | 0.092 | 0.053 | 0.033 | 0.010 |
| Sa (1.0) | 0.047 | 0.028 | 0.018 | 0.005 |
| Sa (2.0) | 0.022 | 0.013 | 0.008 | 0.002 |
| Sa (5.0) | 0.005 | 0.003 | 0.002 | 0.001 |
| Sa (10.0) | 0.002 | 0.001 | 0.001 | 0.000 |
| PGA (g) | 0.115 | 0.058 | 0.031 | 0.007 |
| PGV (m/s) | 0.072 | 0.039 | 0.023 | 0.006 |

Notes: Spectral (Sa(T), where T is the period in seconds) and peak ground acceleration (PGA) values are given in units of g (9.81 m/s2). Peak ground velocity is given in m/s. Values are for "firm ground" (NBCC2015 Site Class C, average shear wave velocity 450 m/s). NBCC2015 and CSAS6-14 values are highlighted in yellow. Three additional periods are provided - their use is discussed in the NBCC2015 Commentary. Only two significant Figures are to be used. These values have been interpolated from a 10 km-spaced grid of points. Depending on the gradient of the nearby points, values at this location calculated directly from the hazard program may vary. More than 95% of interpolated values are within 2% of the directly calculated values. Source: based on National Research Council, 2015; Geological Survey of Canada, 2015.

### 5.6 Comments on Accessibility, Climate, Location Resources, Infrastructure and Physiography

Accessibility, climate, local resources, infrastructure, and physiography are not expected to limit the project. Some of the existing infrastructure close to the site (e.g., Highway 655 and one or more transmission lines) will require relocation to fully develop the resource.

# 6 HISTORY

## 6.1 Exploration History

Prior to 1964, little was known about the geology of Crawford Township. The first aeromagnetic survey was completed in 1955 (Aeromagnetic Surveys Limited: 1 inch to ¼ mile scale), followed by a 1956 Geological Survey of Canada aeromagnetic survey (Map 301G, Crawfish Lakes: 1 inch to 1 mile), and a 1964 Geological Survey of Canada aeromagnetic survey (Map 2319G, Crawfish Lakes: 1 inch to 1 mile). All three magnetic surveys showed a large, roughly circular, strongly magnetic high zone in the east-central part of the township that was interpreted to be an ultramafic rock mass (i.e., the Crawford Ultramafic Complex or "CUC").

The 1963 discovery of the rich base metal deposit in Kidd Township (Kidd Creek Mine), about 15 km south of the CUC, led to a flurry of exploration in Crawford Township through the latter 1960s and the 1970s. The first exploration recorded in Crawford Township dates to 1964. The International Nickel Company of Canada led the way in exploring the township during the 1960s with multiple drillholes testing numerous anomalies (magnetic-electromagnetic or "MAG-EM") anomalies. Anomalous base and precious metal (Cu, Zn, Pb, Ag) results were reported from intermediate to felsic volcanic rocks and long intersections (e.g., 236 m) of nickel (e.g., 0.25-0.40% Ni) in peridotite with very low sulphide content were noted (e.g., Skrecky, 1971). McIntyre Porcupine Mines Ltd. dominated exploration in the township during the 1970s with exploration waning significantly through the 1980s and thereafter.

There are at least 26 historical drillholes reported in Lucas Township which include five diamond drillholes and 21 reverse circulation (RC) drillholes (ODHD, 2023). These drillholes were completed in the 1980s by Abitibi-Price Mineral Resources (diamond and RC holes; MENDM Assessment File 42A14SE0131) and Kidd Creek Mines Ltd. (RC holes only, MENDM Assessment File 42A14SW).

Based on what is available in the public domain, no significant work has been conducted in the project area within Crawford and Lucas townships since the 1980s.

### 6.1.1 Noble Mineral Exploration, 2012-2019

On March 2, 2012, Ring of Fire Resources Inc. announced a name change to Noble Mineral Exploration Inc. Noble continued to explore the nearby Kingsmill Ni Target, announcing March 29, 2012, that it had completed 4,922.2 m of diamond drilling which had intersected long sections (e.g., 546 m) of serpentinized peridotite.

On June 7, 2017, Noble announced the start of a 2,100 line-kilometre airborne helicopter MAG-EM survey which was completed by Balch Exploration Consulting Inc. (BECI) and covered Crawford and Carnegie Townships. The object of the survey was to identify discrete conductors that could represent copper-lead-zinc (Cu-Pb-Zn) mineralization (e.g., Kidd-Creek style) or nickel-copper sulphide, and to map weakly conductive trends that could represent gold associated with disseminated sulphide-bearing mineralization. Previous airborne work on nearby townships within project 81 identified conductive trends in bedrock that correlated with historical drilling that encountered anomalous copper, lead, zinc, and gold. The system used was the AirTEM-150, a compact and concentric helicopter time domain

EM system that can penetrate to depths of 400 m with high resolution. Measurements of the three axes of the EM secondary field are taken in a full waveform mode and the resulting profiles are used to determine the size, orientation, conductance, and depth of the anomalous source.

On May 3, 2018, Noble announced that it had commissioned Albert Mining Inc. of Brossard, Quebec to complete an artificial intelligence (AI) technology interpretation over Crawford and Carnegie townships. On October 18, 2019, Albert Mining Inc. announced a name change to Windfall Geotek Inc.

Results of the study including a final report, were delivered to Noble in June 2019 and announced July 17, 2019. The objective within Crawford Township (approximately 9,321 ha s or 93.21 km$^2$) was to use their proprietary Computer Aided Resources Detection Software (CARDS) AI technology to identify potential Cu-Zn and Ni-Co targets. By using its CARDS technology, Windfall Geotek assisted Noble in identifying targets and possible sites with the same signature as known copper-zinc and nickel occurrences. Windfall Geotek used its proprietary technology to analyze geophysical, geochemical, and geological data to discover the patterns hidden in the large amount of data that Noble has compiled over the years.

The AI study generated nine Ni targets that show +80% similarity prediction using the AGEO (aggregation of GEO-referenced model) Ni model and 12 Cu-Zn targets that show +80% similarity prediction using the AGEO Cu-Zn model. AGEO is one of two algorithms used to determine and validate the accuracy of prediction of the model. The other algorithm is clustering for classification (C-cluster) which is used to compare and validate predictions generated by the AGEO algorithm. The AI study incorporated a total of 2,632 training points that were subjected to evaluation using merged helicopter-borne time domain electromagnetic (HTEM) and magnetic surveys completed by BECI in 2017 (at 25 m resolution), together with an historical diamond drillhole database to construct the Cu-Zn and Ni “predictive models”. CARDS technology uses data-mining techniques and pattern recognition algorithms to analyze and compile the exploration data into many layers of gridded variables in order to identify target zones with high statistical similarity to known areas of mineralization.

On May 8, 2018, Noble announced that it had signed an option and joint venture agreement with Spruce Ridge Resources Ltd. to earn a 75% interest in the Crawford Township property on specific target areas having a size up 2,000 ha.

On August 27, 2018, Noble announced it had contracted CGG Multi-Physics to complete a FALCON© Airborne Gravity Gradiometer and magnetics survey over parts of Project 81 including Crawford Township. The Falcon AGG technology is a gravity gradiometer system specifically designed for airborne survey use and reportedly provides several key advantages over other standard full tensor gradiometer (FTG) systems such as lower noise, higher resolution and sensitivity, measured error and redundancy, and high production rate. Results of the survey were delivered to Noble in a final report in November 2018.

On June 11, 2019, Noble announced that it had received and released the results of mineralogical studies on drill core samples from its Crawford property. Twelve samples of drill core were selected from 1.5 m analyzed intervals to cover a range of nickel, cobalt, palladium, and sulphur contents, as well as differing degrees of serpentinization. Polished thin sections were made from the core samples and examined under reflected-light microscope and a scanning electron microscope (SEM), which provided chemical analyses of individual mineral grains to aid in their identification (see Section 6.3, Historical Mineral Processing and Metallurgical Testing).

On October 1, 2019, under the terms of a binding letter of intent, Noble announced the creation of Canada Nickel Company, Inc. (CNC) which will own a consolidated 100% interest in the Crawford property. A definitive agreement was entered into on November 14, 2019 (Noble news release dated November 29, 2019).

Noble, in conjunction with CNC, announced the results of their first phase of diamond drilling targeting the CUC on December 9, 2019. Phase 1 drilling consisted of nine diamond drillholes, totalling 5,267 m and all nine holes intersected nickel (Ni) cobalt (Co) and platinum-group element (PGE) mineralization.

### 6.1.2 Spruce Ridge Resources Inc, 2017-2019

On September 25, 2017, Spruce Ridge announced that it had signed a binding Letter of Intent (LOI) with Noble to earn a 75% interest in specific target areas having a size of up to 2,000 ha within Noble’s Crawford Township property. On May 8, 2018, Spruce Ridge announced that it had entered into an Option and Joint Venture Agreement with Noble under the terms set out in the LOI between the two companies. On September 27, 2018, Spruce Ridge announced that it has signed an additional LOI with a private group of knowledgeable mining investors to acquire up to 50% of its Option and Joint Venture agreement with Noble on its Crawford Township property.

The CUC, although geophysically recognized as early as 1964, was recently redefined by a high-resolution helicopter-borne magnetic and electromagnetic survey in 2017 (Balch, 2017) and a high-sensitivity aeromagnetic and airborne gravimetric survey in 2018 (CGG, 2018), both conducted over the entire Crawford Township, and followed up with 3D-inversion and detailed interpretation (St-Hilaire, 2019). Jobin-Bevans and Siriunas (2020) provide a review of the airborne geophysical work completed in 2017 and 2018 and the delineation of the CUC.

On November 15, 2018, Spruce Ridge (and Noble) announced they had begun a 2,000 m program of diamond drilling on the Crawford Township property. The target of the drilling program was a 3,000 m long, magnetic anomaly interpreted to be a differentiated ultramafic to mafic intrusive complex, the CUC. On March 1, 2019, Spruce Ridge announced the results of its 2018 winter drilling program which totalled 1,818 m in four drillholes; Noble announced the same on March 4, 2018.

On September 19, 2019, Spruce Ridge (and Noble on September 20) announced that it had begun a second phase of diamond drilling on the Crawford property. The phase 2 drilling program was planned to comprise approximately 4,000 metres of drilling in eight holes. Planned drillholes include infill drilling between the four drillholes put down in the winter of 2018, as well as step-out drilling to the northwest and southeast; Noble announced the same on September 20, 2019.

On October 1, 2019, Spruce Ridge announced that it had agreed to sell its interest in the Crawford property to the private Canada Nickel Company, (created by Noble). Spruce Ridge retains its interest in various base metal targets located in Crawford Township. At this time, Noble assumed care and control and management of the diamond drilling program in collaboration with management of the newly formed Canada Nickel Company.

## 6.2 Historical Drilling

In Crawford Township, between 1964 and 2018, at least 147 drillholes were completed with diamond core and reverse circulation drilling, totalling more than 14,600 metres. This drilling tested numerous geophysical anomalies, targeting base metals, gold, and nickel sulphides in volcanic and mafic-ultramafic rocks (Orix Geoscience, 2018). Reported overburden intervals are drillhole casing lengths and do not necessarily represent true thickness of overburden.

### 6.2.1 INCO Canada Ltd., 1965–1966

The earliest drilling in Crawford Township, targeting the CUC, was by INCO Canada Ltd. in 1965. A total of eight drillholes are reported, targeting magnetic anomalies "4-89", "4-313", and "4-B" which were collectively referred to as "Owl" (Table 6-1). Anomaly "4-89", "4-313", and "4-B" correspond to the "Main", "East", and "North" components of the CUC, respectively. The 1965 drilling intersected broad intervals (e.g., 467.56 m) of mafic-ultramafic rocks, largely serpentinized peridotite and/or serpentinized dunite. Overburden intervals (drillhole casing length) ranged from 34.75 to 86.87 metres.

**Table 6-1: Drillholes and Assays Summary, INCO Canada Ltd., Crawford Ultramafic Complex**

| Year | Drill Hole | Target | Anomaly | [1]OB (m) | [2]EOH (m) | From (m) | To (m) | Int (m) | Ni (%) | Comments |
|---|---|---|---|---|---|---|---|---|---|---|
| 1964 | 25050 | Main Mag | 4-89 | 34.75 | 502.31 | 39.62 | 502.31 | 462.69 | 0.25 | 34.75 m to EOH: mafic-ultramafic rocks |
| 1965 | 26636 | Main Mag | 4-89 | 43.89 | 43.89 | - | - | - | - | abandoned in overburden |
| 1965 | 26637 | Main Mag | 4-89 | 61.87 | 474.57 | - | - | - | - | 83.06 m to EOH: mafic-ultramafic rocks; no assays reported |
| 1965 | 27005 | Main Mag | 4-89 | 63.40 | 245.97 | 63.67 | 220.98 | 157.31 | 0.16 | 63.40 m to 185.93 m: mafic-ultramafic rocks |
| 1965 | 27064 | East Mag | 4-313 | 86.87 | 602.89 | 165.70 | 419.10 | 253.40 | 0.24 | 165.72 m to EOH: mafic-ultramafic rocks |
| 1966 | 27086 | Main Mag | 4-89 | 50.90 | 384.05 | 50.90 | 384.05 | 333.15 | 0.07 | 50.90 to EOH: mafic-ultramafic rocks |
| 1966 | 27095 | Main Mag | 4-89 | 37.19 | 273.41 | 37.20 | 273.40 | 236.20 | 0.34 | 37.19 to EOH: mafic-ultramafic rocks |
| 1966 | 29173 | North Mag | 4-B | 68.89 | 364.24 | - | - | - | - | 148.59 m to EOH: mafic-ultramafic rocks; no assays reported |

Notes: 1. OB = overburden. 2. EOH = End of Hole.

### 6.2.2 McIntyre Porcupine Mines Ltd., 1973

McIntyre Porcupine Mines Ltd. completed a drilling campaign in 1973 targeting a magnetic high in the north-central area of Crawford Township, near the border with Nesbitt Township to the north. The company completed four drillholes targeting a magnetic anomaly referred to as "Anomaly 3N" shown in Table 6-2. The drilling intersected broad intervals (e.g., 153.11 m) of mafic-ultramafic rocks, largely serpentinized peridotite and/or serpentinized dunite. Overburden intervals (drillhole casing length) ranged from 27.43 to 97.54 m.

**Table 6-2: Drillhole Summary with Significant Assays, McIntyre Porcupine Mines, Anomaly 3N**

| Year | Drillhole | Target | Anomaly | [1]OB (m) | [2]EOH (m) | From (m) | To (m) | Int (m) | Ni (%) | Comments |
|---|---|---|---|---|---|---|---|---|---|---|
| 1973 | 904-73-3 | Mag High | 3N | 60.96 | 163.68 | - | - | - | - | intersected felsic volcanic rocks |
| 1973 | 904-73-4 | Mag High | 3N | 60.96 | 134.42 | 120.85 | 122.38 | 1.53 | 0.35 | 120.85 m to EOH: peridotite |
| | | | | | | 129.24 | 129.69 | 0.45 | 0.43 | |
| | | | | | | 132.89 | 134.42 | 1.53 | 0.21 | |
| 1973 | 904-73-5 | Mag High | 3N | 97.54 | 208.18 | - | - | - | - | intersected felsic volcanic rocks |
| 1973 | 904-73-27 | Mag High | 3N | 27.43 | 163.37 | 35.36 | 36.88 | 1.52 | 0.17 | 27.43 m to EOH: ultramafic rocks |
| | | | | | | 57.91 | 59.44 | 1.53 | 0.30 | |

Notes: 1. OB = overburden. 2. EOH = End of Hole

### 6.2.3 Spruce Ridge Resources Ltd, 2018

#### 6.2.3.1 Drilling Program

In late 2018, Spruce Ridge completed a drilling program targeting the "Main" magnetic high that defines a portion of the CUC. Results from the four-hole, 1,818 m (NQ-size core, 47.6 mm diameter) winter drilling program were announced in March 2019 (see Table 6-3 and Spruce Ridge news release dated March 1, 2019). All four drillhole collars are located immediately east of Ontario Highway 655, about 40 km north of Timmins. The holes were drilled toward the north-northeast (azimuth 35°) at dips of -50° or -60°.

Three of the holes intersected serpentinized dunite with persistent nickel concentrations greater than 0.25% Ni over core lengths of up to 291 m. Using a lower threshold of 0.20% Ni, long intervals are present in all four holes, with a maximum core length of 558 m. Individual samples of 1.5 m core intervals reported up to 0.669% Ni and all four holes were terminated in dunite or peridotite.

Except for drillhole CR18-02, which was terminated early (216 m) and in dunite, drill core assays show increasing nickel concentrations down the holes. Drillhole CR18-01 recorded nickel grades of about 0.20% Ni in the upper peridotite, which compares favourably relative to the nickel grades in peridotite from the Dumont Sill which are generally very low to nil. Nickel grades in CR18-01 increase further down-hole and through a central intercept, and then decline toward the bottom of the hole.

Palladium concentrations show a strong correlation with increased nickel concentrations, suggesting the presence of nickel sulphides.

#### 6.2.3.2 Drill Core Characterization

Drill core samples from four drillholes completed in 2018 by Spruce Ridge, holes CR18-01, 02, 03, and 04, were used to determine average specific gravity and magnetic susceptibility of the intersected rock units and to run laboratory tests comparing recovery differences using two different analytical methods.

**Table 6-3: Summary of Drillholes Completed by Spruce Ridge Resources in Winter 2018**

| Summary of Intervals Passing 0.25% Ni cut-off | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Drillhole | Az | Dip | From (m) | To (m) | Int (m) | Ni (%) | Co (ppm) | Pt (ppb) | Pd (ppb) | Au (ppb) |
| CR18-01 | 35 | -60 | 234.00 | 525.00 | 291.00 | 0.293 | 118 | 11 | 20 | 2 |
| CR18-03 | 35 | -50 | 475.50 | 606.00 | 130.50 | 0.299 | 140 | 28 | 55 | 6 |
| CR18-04 | 35 | -50 | 205.50 | 402.00 | 196.50 | 0.332 | 135 | 10 | 27 | 2 |
| **Summary of Intervals Passing 0.20% Ni cut-off** | | | | | | | | | | |
| Drillhole | Az | Dip | From (m) | To (m) | Int (m) | Ni (%) | Co (ppm) | Pt (ppb) | Pd (ppb) | Au (ppb) |
| CR18-01 | 35 | -60 | 36.00 | 594.00 | 558.00 | 0.261 | 127 | 10 | 16 | 2 |
| CR18-02 | 35 | -50 | 24.00 | 175.50 | 151.50 | 0.224 | 126 | 5 | 5 | 1 |
| CR18-03 | 35 | -50 | 288.00 | 606.00 | 318.00 | 0.248 | 126 | 19 | 28 | 3 |
| CR18-04 | 35 | -50 | 193.50 | 402.00 | 208.50 | 0.324 | 135 | 18 | 28 | 3 |
| **Selected Intervals with Elevated PGEs** | | | | | | | | | | |
| Drillhole | Az | Dip | From (m) | To (m) | Int (m) | Ni (%) | Co (ppm) | Pt (ppb) | Pd (ppb) | Au (ppb) |
| CR18-03 | 35 | -50 | 492.00 | 493.50 | 1.50 | 0.285 | 140 | 219 | 567 | 4 |
| CR18-03 | 35 | -50 | 507.00 | 511.50 | 4.50 | 0.339 | 140 | 59 | 498 | 48 |
| CR18-04 | 35 | -50 | 165.00 | 166.50 | 1.50 | 0.182 | 120 | 69 | 570 | 6 |

Note: The lengths reported are core lengths and not true widths. Spruce Ridge has insufficient information to determine the attitude, either of the ultramafic body or of mineralized zones within it. True widths will be less than the core lengths by unknown factors.

#### 6.2.3.2.1 Specific Gravity

Drill core from CR18-01, 02, 03, and 04 had specific gravity (SG) measurements made at regular intervals using the “weight in water vs. weight in air” relative density method. Average SG for mafic volcanic rocks was 2.67 (n=60) and average SG for serpentinized ultramafic rocks was 2.66 (n=436). Specifically, with respect to the ultramafic rocks, average SG for intervals grading over 0.25% Ni was 2.61, for intervals between 0.20% and 0.25% Ni was 2.62, and for intervals less than 0.20% Ni was 2.63. Fresh, unaltered dunite and peridotite, typically have a SG in the range of 3.2 to 3.4. The process of serpentinization involves the introduction of water into the rock, resulting in a substantial volume increase. The low average SG of the CUC ultramafic rocks (2.66) implies a high degree of serpentinization.

#### 6.2.3.2.2 Magnetic Susceptibility

Magnetic susceptibility readings were collected along the drill core from the four drillholes completed in 2018. Based on more than 1,400 readings it was shown that the ultramafic rocks (average 129 units) were some 100 times higher than host mafic volcanic rocks (average 0.72 units). The serpentinized rocks are extremely magnetic relative to the host rocks and non-serpentinized ultramafic rocks, a result amplified by the serpentinization of olivine which releases iron to form magnetite.

### 6.2.4 Spruce Ridge Resources Ltd, 2019

On September 19, 2019, Spruce Ridge began a second round of drilling, planned to comprise 4,000 m (NQ size core, 47.6 mm diameter) of diamond drilling in eight holes. In conjunction with CNC, results of this phase of drilling were released by Noble (Noble news release dated December 9, 2019) and Spruce Ridge (Spruce Ridge news release dated December 10, 2019). The results, along with more recent drilling results, are discussed in Section 10.0, Drilling, and Section 11.0, Sample Preparation, Analysis and Security.

## 6.3 Historical Mineral Processing and Metallurgical Testing

### 6.3.1 CUC-SEM/BEI Mineralogical Study, 2019

In 2019, Spruce Ridge commissioned a mineralogical study of ultramafic rock material collected from drill core samples from the 2018 diamond drilling program (Noble news release dated June 11, 2019). The purpose of the study was to determine whether the nickel (and other elements) occur in the sulphide state, which could be economically extracted from the altered ultramafic host rocks of the CUC.

Twelve samples of drill core were selected from 1.5 m analyzed intervals, to cover a range of nickel, cobalt, palladium, and sulphur contents as well as differing degrees of serpentinization. Polished thin sections were made from the core samples and examined under reflected-light microscope to determine target areas for subsequent relocation and analysis using a JEOL 733 electron microprobe. Backscattered electron images (BEI) were captured and areas of interest within each grain were analyzed using an Oxford Instruments X-Act energy dispersive system (EDS) attached to the electron microprobe (Renaud, 2019).

The following minerals were identified as carrying most of the nickel and cobalt (in order of decreasing abundance): pentlandite (50%: iron-nickel sulphide), heazlewoodite (35%: sulphur poor, nickel-rich sulphide), awaruite (15%: nickel-iron alloy) and minor godlevskite (nickel-iron sulphide). Heazlewoodite is one of the most nickel-rich sulphide minerals, and is generally thought to be of hydrothermal origin, most often found in dunite and lherzolite.

### 6.3.2 Selective Leach Analysis

A selective leach analysis was performed on pulp samples of the 12 core intervals from which the mineralogy samples were taken. Table 6-4 shows a comparison between the peroxide fusion analysis and the aqua regia analysis for cobalt and nickel and establishes the potential percentages of “liberation” of these key elements (Noble news release dated June 11, 2019).

All drill core samples were initially analyzed by ICP after sample preparation using sodium peroxide fusion (FUS-ICP) for total digestion (palladium, platinum and gold were determined by fire assay). Pulps from the same 12 sample intervals selected for SEM analysis were re-analyzed using the same ICP procedure, after digestion using aqua regia (AR-ICP), which does not attack silicate minerals to any significant degree. This provided a semi-quantitative estimate of the amount of nickel and cobalt that had been liberated from their parent olivine by serpentinization. After eliminating the one sample that showed much lower liberation, the average overall nickel liberation was 62%, and the average cobalt liberation was 77% (Table 6-4).

**Table 6-4: Comparison between Peroxide Fusion and Aqua Regia Analyses for Cobalt and Nickel**

| Drillhole | From (m) | To (m) | Int (m) | Co (ppm) FUS-ICP | Co (ppm) AR-ICP | Percent Liberated | Ni (%) FUS-ICP | Ni (%) AR-ICP | Percent Liberated | S (%) FUS-ICP |
|---|---|---|---|---|---|---|---|---|---|---|
| CR18-01 | 165.0 | 166.5 | 1.5 | 240 | 193 | 80% | 0.669 | 0.431 | 64% | 0.28 |
| CR18-01 | 238.5 | 240.0 | 1.5 | 120 | 105 | 88% | 0.297 | 0.203 | 68% | 0.02 |
| CR18-01 | 243.0 | 244.5 | 1.5 | 170 | 149 | 88% | 0.487 | 0.332 | 68% | 0.15 |
| CR18-01 | 286.5 | 288.0 | 1.5 | 150 | 130 | 87% | 0.345 | 0.232 | 67% | 0.18 |
| CR18-01 | 423.0 | 424.5 | 1.5 | 120 | 85 | 71% | 0.317 | 0.203 | 64% | 0.03 |
| CR18-01 | 588.0 | 589.5 | 1.5 | 110 | 87 | 79% | 0.272 | 0.178 | 65% | 0.01 |
| CR18-03 | 508.5 | 510.0 | 1.5 | 140 | 108 | 77% | 0.332 | 0.217 | 65% | 0.01 |
| CR18-03 | 535.5 | 537.0 | 1.5 | 140 | 109 | 78% | 0.337 | 0.227 | 67% | 0.07 |
| CR18-03 | 594.0 | 595.5 | 1.5 | 150 | 110 | 73% | 0.349 | 0.205 | 59% | 0.05 |
| CR18-04 | 165.0 | 166.5 | 1.5 | 120 | 52 | 43% | 0.182 | 0.050 | 27% | <0.01 |
| CR18-04 | 216.0 | 217.5 | 1.5 | 260 | 206 | 79% | 0.647 | 0.423 | 65% | 0.60 |
| CR18-04 | 337.5 | 339.0 | 1.5 | 130 | 103 | 79% | 0.427 | 0.275 | 64% | 0.20 |
| | | | | | **Mean Co Liberation** | **77%** | | **Mean Ni Liberation** | **62%** | |

## 6.4 Historical Sample Preparation, Analysis, Security

There was no quality assurance/quality control (QA/QC) information found regarding sample preparation, analyses, and security procedures for the diamond drill core assay results prior to the 2018 and 2019 drilling programs by Spruce Ridge. No casing was left in drillholes prior to the work done by Spruce Ridge, so in the field it is not possible to confirm the location of historical, pre-2018 drillholes. The following information comes from a review of Spruce Ridge's completed 2018 program and the 2019 diamond drilling program.

### 6.4.1 Sample Collection and Transportation

Drill core (NQ-size core, 47.6 mm diameter) was placed in core boxes at the drill by the drilling contractor (NPLH Drilling of Timmins, Ontario) following industry standard procedures. Small wooden tags mark the distance drilled in metres at the end of each run. On each filled core box, the drillhole number and sequential box numbers were marked by the drill helper and checked by the site geologist. Once filled and identified, each core tray was covered and secured shut.

Core was delivered to the side of Highway 655 by the drilling contractor as the drilling progressed. Spruce Ridge personnel transported the core to the core shack from that location. Casing was left in the completed drillholes, capped, and marked with a metal flag.

### 6.4.2 Core Logging and Sampling

CNC rented a core shack in Timmins, a driving distance of approximately 50 km from the project area access point. Once the core boxes arrived at the logging facility in Timmins, they were laid out on the logging table in order and their lids removed. Core was stored sequentially, hole by hole, in racks for logging. Core logging consisted of two major parts: geotechnical logging and geological logging.

Geological core logging records the lithology, alteration, texture, colour, mineralization, structure, and sample intervals. All geotechnical and geological logging and sample data were recorded directly into a computer spreadsheet (MS Excel). As the core was logged the target rock type (dunite and/or peridotite) was marked for sampling at a nominal sample interval of 1.5 m. The entire intercept of ultramafic rocks was sampled in each drillhole. Magnetic susceptibility was measured every metre. Relative density of core samples was calculated at a variable interval of 3 to 6 m.

Samples were identified by inserting three identical pre-fabricated, sequentially-numbered, weather-resistant sample tags at the end of each sample interval. Once the core was logged, photographed, and samples marked, the core boxes were transferred to the cutting room for sampling. In general, the core recovery for the diamond drillholes on the property has been better than 95% and little core loss due to poor drilling methods or procedures has been experienced.

Sections marked for sampling were cut in half with a diamond saw; a separate cutting room was located adjacent to the logging area. Once the core was halved, it was returned to the core box. A geotechnician prepared the sample tags, selecting half of the core in each interval, placing the core in a sample bag, and sealing the bag with a cable tie. The boxes containing the remaining half core were stacked and stored on site in the secure core storage facility.

Individual samples were then placed in large polypropylene bags (rice bags), five samples to a bag, and these larger bags secured with a cable tie. CNC personnel were responsible for transporting the samples to the Activation Laboratories Ltd. (Actlabs) Timmins analytical facility, a driving distance of approximately 4.5 km from the core shack.

### 6.4.3 Analytical

A total of 952 drill core samples (CR18 drillhole series) were submitted to Actlabs (Timmins and Ancaster, Ontario) for analysis by Spruce Ridge. Actlabs, a Canadian-owned analytical and assay laboratory certified to ISO/IEC 17025 with CAN-P-1579 (Mineral Analysis), is independent of Spruce Ridge, Noble, and CNC. Analyses for precious metals (Pt, Pd, Au) were done by fire assay on 30-gram splits with ICP-OES analysis. Nickel and cobalt were determined by ICP-OES after sample preparation by sodium peroxide fusion.

Additionally, Spruce Ridge performed independent spot analysis (nickel concentration) of a duplicate pulp from approximately every fifth sample (184 samples), using a portable X-ray fluorescence (XRF) instrument. Results accorded closely to those from the Actlabs laboratory's ICP-OES peroxide fusion (ICP) analyses. With respect to the 184 samples, the percent difference between the ICP and XRF analyses ranges from -30% to +13% and the average percent difference is -5%. On average, the XRF analyzer underestimated nickel concentrations by 5%.

Concentrations of other metals such as cobalt and precious metals (i.e., gold, silver, PGE) were too low to be reliably determined by portable XRF technology.

### 6.4.4 QA/QC Data Verification

#### 6.4.4.1 Control Samples

No QA/QC samples were introduced to the sample stream by Spruce Ridge. Actlabs inserted internal certified reference material and blanks into the sample stream and carried out duplicate and replicate ("preparation split") analyses within each sample batch as part of their own internal monitoring of quality control. It is the results of Actlabs' internal quality control that Spruce Ridge relied upon to service the quality control of the project and it is those results that are reported on herein.

A total of 154 duplicate analyses (including six replicate analyses) were carried out by Actlabs in the course of their work. Of those duplicate analyses, 90 were performed by FA digestion and 82 by sodium peroxide ($Na_2O_2$) fusion digestion. A total of 83 analyses of blank material were performed by FA digestion and 91 samples of blank material were analyzed after the sodium peroxide fusion digestion. For the purposes of the report only the elements of major economic importance to the project (i.e., Ni, Co, Au, Pd, Pt) were examined in detail for an assessment of the quality of the analytical data. The elements Cu, Mg and S were also examined in a cursory manner for the assessment.

The Actlabs laboratory in Timmins, Ontario carried out the sample login/registration, sample weighing and sample preparation.

For statistical purposes within the report any analytical result reported as less than the detection limit was set to one half of that detection limit (e.g., a result reported as <0.5 was set to a numeric value of 0.25). Results reported to be greater that maximum value reportable, and where no corresponding over limit analysis was performed, were set to that maximum value (e.g., a result reported as >15.0 was set to a numeric value of 15).

#### 6.4.4.2 Blank Material

All analyses performed on blank material are considered to be acceptable, as most results were reported as below the detection limits for each element examined. The exception with respect to those elements examined in detail was S, where 5.5% of the blank samples reported at the lower limit of detection (0.01%) or above (maximum 0.06%); however, this failure rate is still considered acceptable.

#### 6.4.4.3 Certified Reference Material

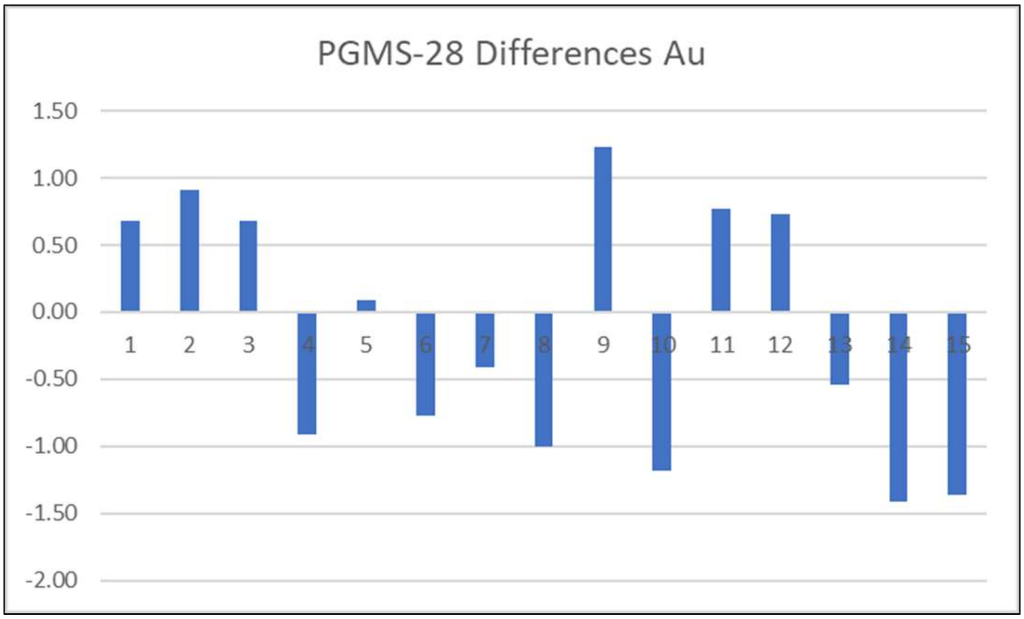
Certified reference materials (CRM) were used by Actlabs to internally monitor the accuracy of their analyses. Several different reference materials for combinations of elements were used during the analytical work being reported on herein, including: CDN-PGMS-28, DTS-2b, CCU-1e, GBW 07113, PTM-1a, CD-1, GBW 07238, OREAS 74a, OREAS 134b, MP-1b, AMIS 0129, OREAS 13b, NCS DC86314, PK2, CZN-4, W 106, OREAS 922.

For the purpose of the report, the focus on the results of the first two reference materials in the preceding list (CDN-PGMS-28 and DTS-2b) as they report certified values in ranges similar to material that was submitted to Actlabs for analysis. All the certified reference material examined in detail averaged within two standard deviations of the certified concentrations over the span of the laboratory work (Figures 6-1, 6-2 and 6-3). As all analyses of certified reference material, over time, averaged close to their certified concentration, the accuracy of the analyses was considered acceptable.

### 6.4.5 Duplicate Samples

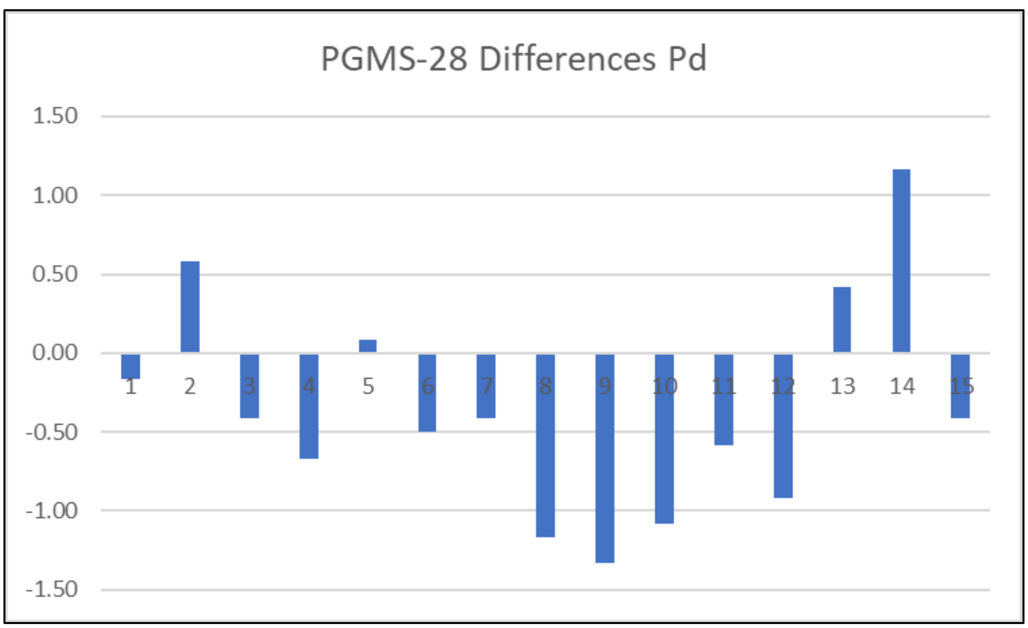
In general, the duplicate material for the precious metal analyses indicated good reproducibility of the assays (Figures 6-4, 6-5 and 6-6). Where relative differences of over 100% are observed, sample pairs generally exhibited low absolute concentrations of the precious metals and the order of magnitude difference at those levels is not considered important.

**Figure 6-1: CRM CDN-PGMS-28 – Standard Deviations of Difference for Au Analysis**



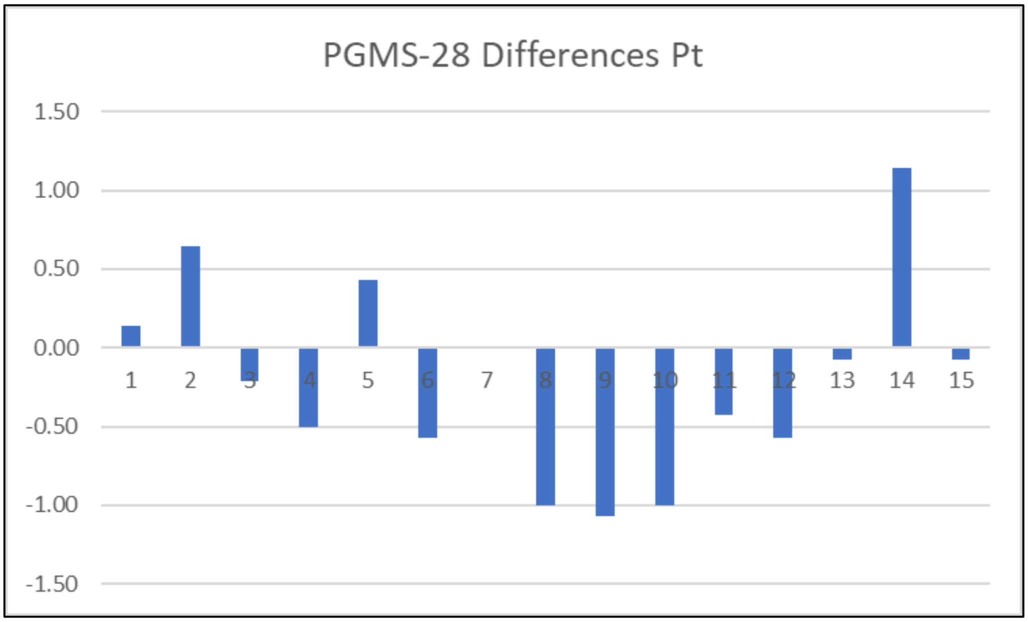

Source: Caracle Creek, 2020.

**Figure 6-2: CRM CDN-PGMS-28 – Standard Deviations of Difference for Pd Analysis**



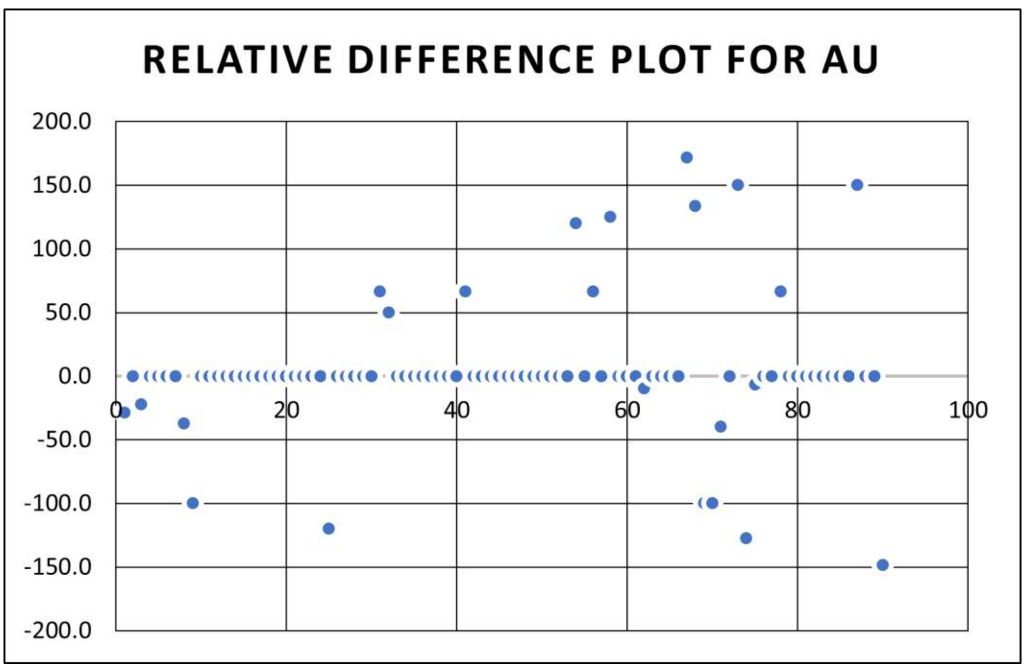

Source: Caracle Creek, 2020.

**Figure 6-3: CRM CDN-PGMS-28 – Standard Deviations of Difference for Pt Analysis**



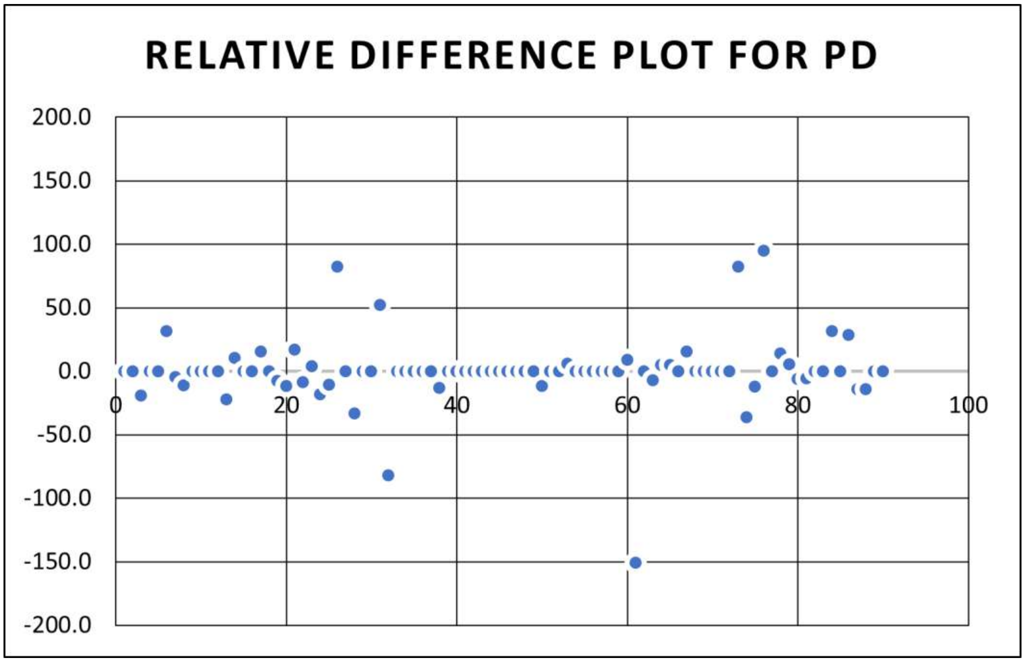

Source: Caracle Creek, 2020.

**Figure 6-4: Relative Percent Difference of Pairs of Duplicate Samples Analyzed for Au**



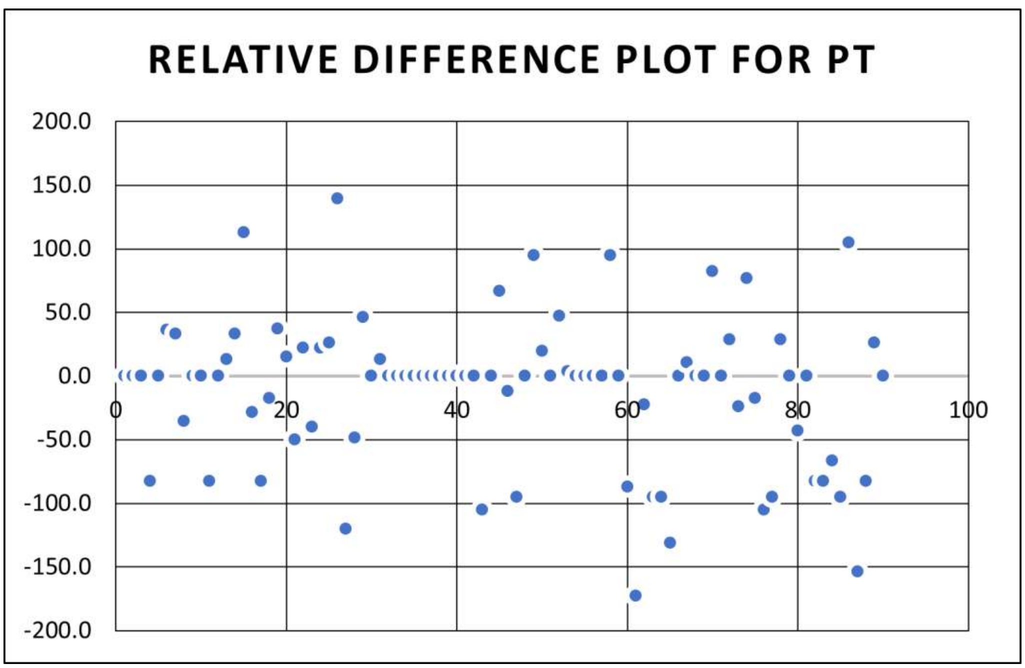

Source: Caracle Creek, 2020.

**Figure 6-5: Relative Percent Difference of Pairs of Duplicate Samples Analyzed for Pd**




Source: Caracle Creek, 2020.

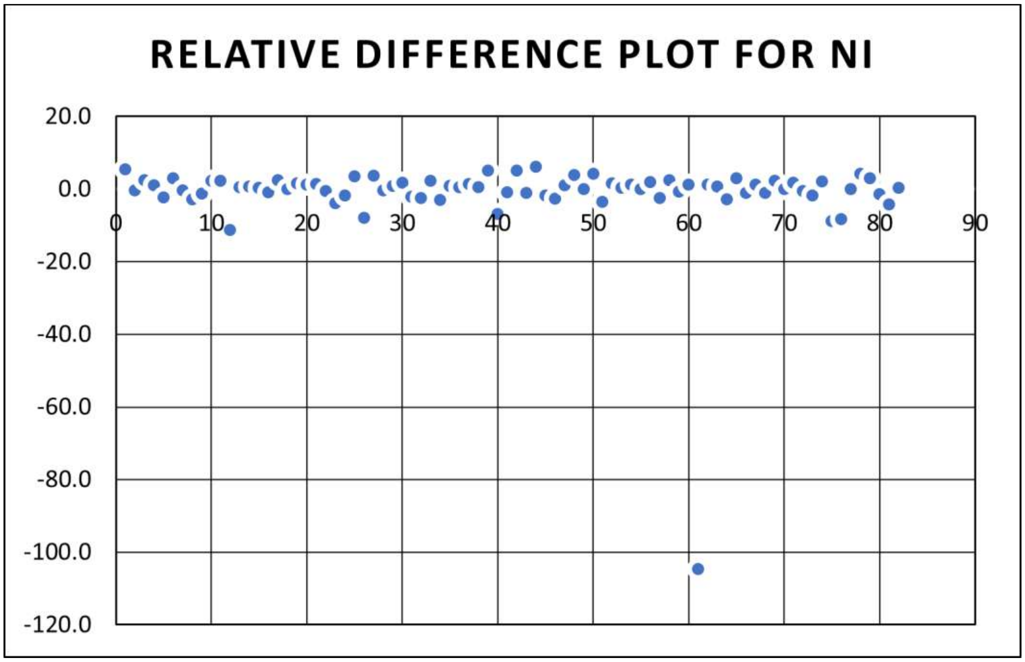
**Figure 6-6: Relative Percent Difference of Pairs of Duplicate Samples Analyzed for Pt**




Source: Caracle Creek, 2020.

The relative differences for Co and Ni were under 20% except for one sample, 701330, where the relative difference between the pair of Ni analyses was over 100% (Figure 6-7). This appears to be a case where exceptionally low nickel values were returned and as such the relative difference is not considered to be of importance.

**Figure 6-7: Relative Percent Difference of Pairs of Duplicate Samples Analyzed for Ni**




Source: Caracle Creek, 2020.

# 7 GEOLOGICAL SETTING AND MINERALIZATION

## 7.1 Regional Geology

The Crawford Project is situated in Northeastern Ontario, in the western portion of the mineral-rich Abitibi Greenstone Belt (AGB) (2.8 to 2.6 Ga), which is within the Superior Province, Canada (Figures 7-1 and 7-2). The AGB of the Abitibi Subprovince, spans across the Ontario-Quebec provincial border and is considered to be the largest and best-preserved greenstone belt in the world (Jackson and Fyon, 1991; Sproule et al., 2003), covering an area approximately 700 km from the southeast to northwest and 350 km from north to south, constituting several major east-trending successions of folded volcanic and sedimentary rocks, with associated felsic to ultramafic intrusions. The supracrustal rocks of the AGB are uniquely well preserved and have mostly been overprinted only at a low metamorphic grade (Monecke et al., 2017). The AGB is economically significant because it contains some of the most important gold and base metal mining camps in Canada, as well as a long history of punctuated production from ultramafic extrusive komatiite-hosted Ni-Cu-(PGE) sulphide deposits.

More than an estimated 50% of the supracrustal rocks of the AGB, including those on the property, are under tens of metres of clay-dominated cover (referred to as the “Abitibi Clay Belt” or “Great Clay Belt” and formed from the lakebed sediments of Glacial Lake Ojibway), making mineral exploration challenging and expensive and hampering the discovery rate of new metal mines which preserves opportunities for discovery.

The AGB has been subdivided into nine lithotectonic assemblages or volcanic episodes (Ayer et al., 2002a, 2002b and 2005); however, the relationships between these assemblages are mostly ambiguous. Allochthonous greenstone belt models, with each terrane having been formed in a different tectonic environment, predict them to be a collage of unrelated fragments. Autochthonous greenstone belt models allow for the prediction of syngenetic mineral deposits hosted by specific stratigraphic intervals and formed within a structurally deformed singular terrane. Greenstone belts in the Superior Province consist mainly of volcanic units unconformably overlain by largely sedimentary “Timiskaming-style” assemblages, and field and geochronological data indicate that the AGB developed autochthonously (Thurston et al., 2008).

Proterozoic dikes of the Matachewan Dyke Swarm and the Abitibi Dyke Swarm intrude all the rock in the region. Matachewan dikes generally trend north-northwest while the younger Abitibi Dyke Swarm trends northeast.

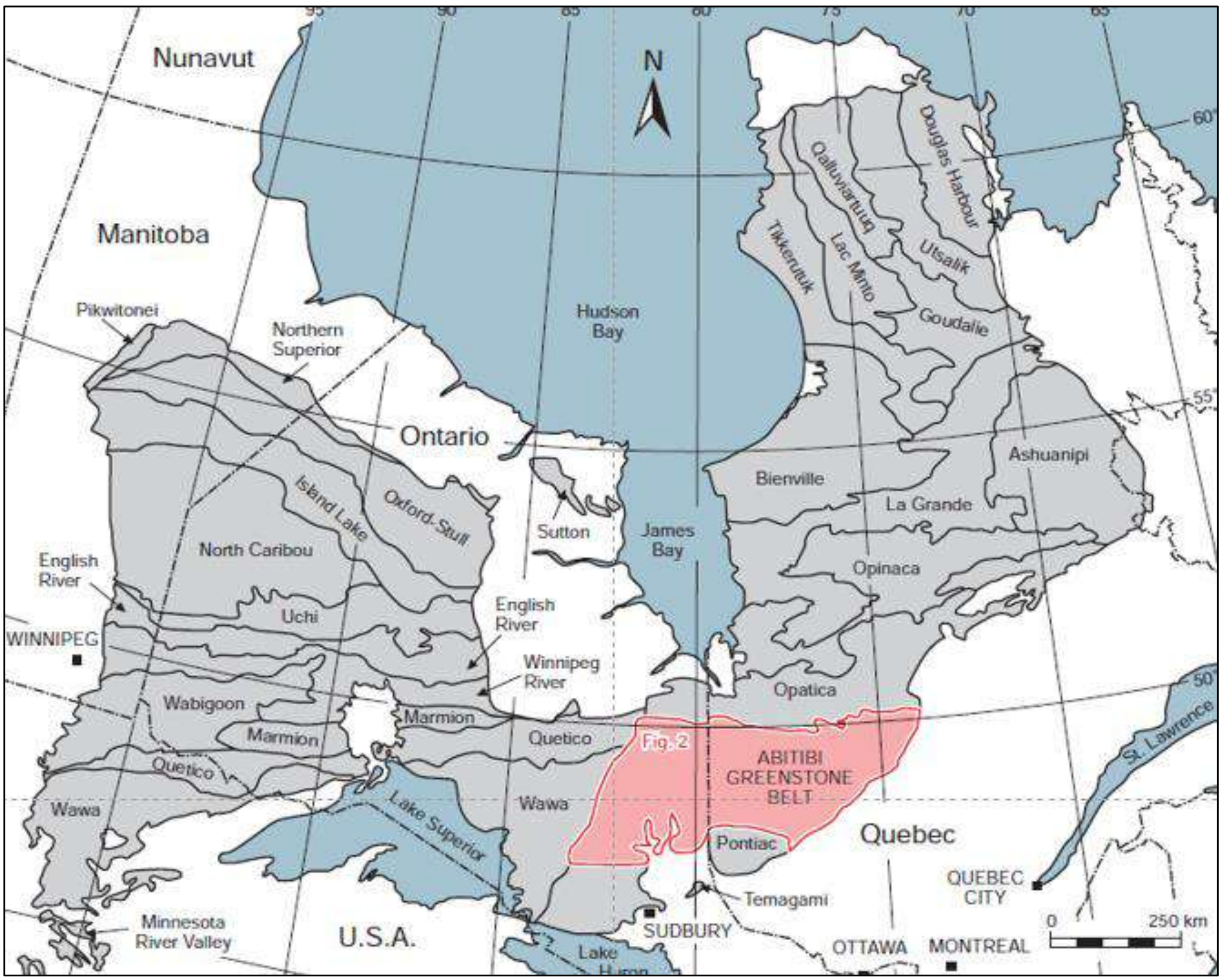
### 7.1.1 Komatiitic Rocks

Of the nine distinct lithotectonic assemblages defined in the AGB, only four of these are generally accepted to contain extrusive komatiitic rocks (ultramafic mantle-derived rock with ≥18 wt% MgO) and therefore considered prospective for komatiite-associated Ni-Cu-(PGE) sulphide deposits (Arndt et al., 2008).

These four assemblages, which differ considerably in physical volcanology and geochemistry of the komatiitic flows, have distinct and well-defined ages, as well as spatial distribution (Sproule et al., 2003; Thurston et al., 2008; Houle and Lesher, 2011):

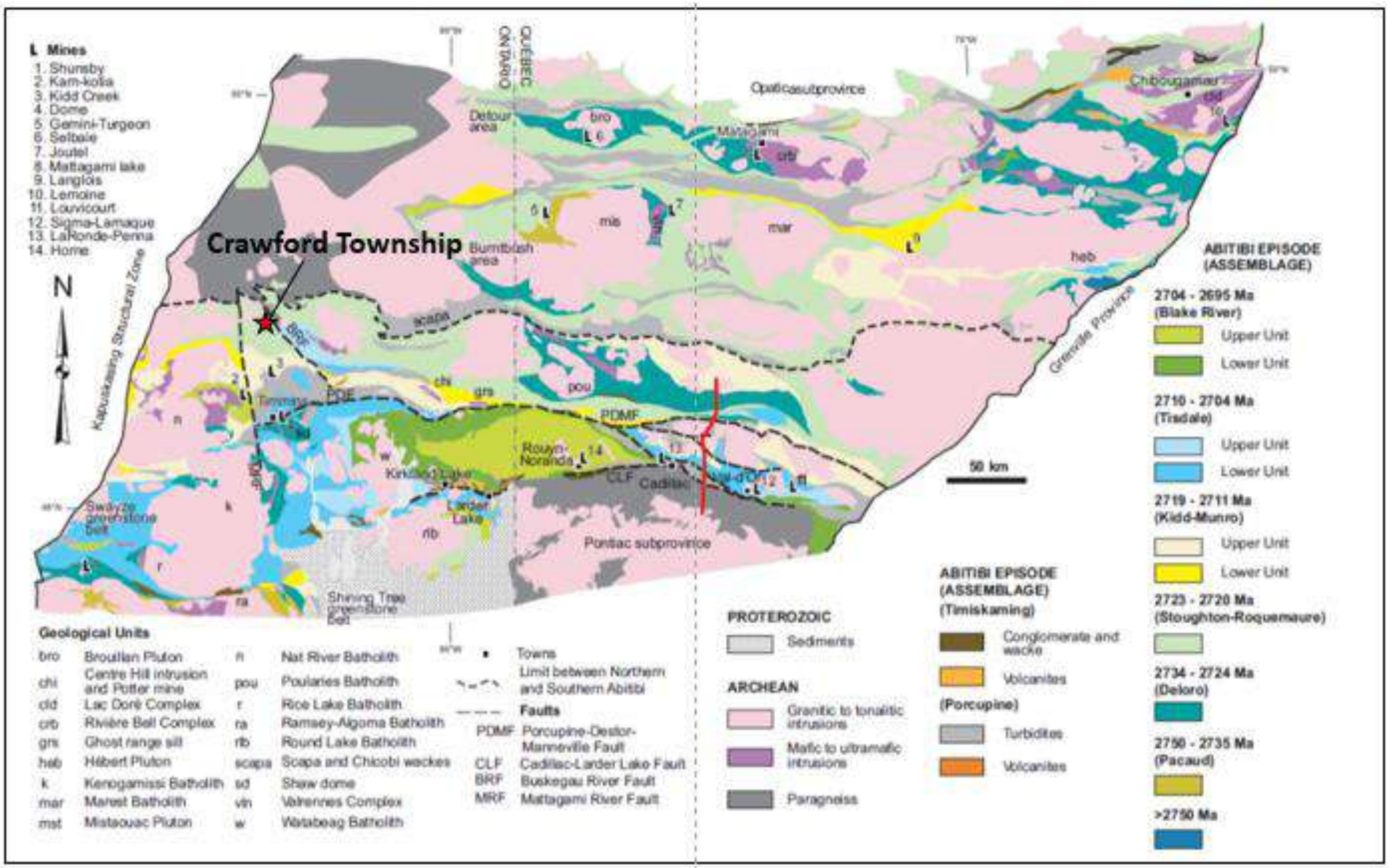
- Pacaud Assemblage (2750-2735 Ma)
- Stoughton-Roquemaure Assemblage (2723-2720 Ma)
- Kidd-Munro Assemblage (2719-2711 Ma)
- Tisdale Assemblage (2710-2704 Ma).

**Figure 7-1: Location of the Abitibi Greenstone Belt within the Archean Superior Province, Canada**




Source: Monecke et al., 2017.

**Figure 7-2: General Geology of the Abitibi Greenstone Belt and Project Location (Red Star)**




Note: The Deloro Assemblage in which the CUC is hosted is not shown at this scale. Source: Thurston et al., 2008; MERC, 2017.

The Kidd-Munro and Tisdale assemblages contain a much greater abundance of cumulate komatiites than the other assemblages. The Kidd-Munro Assemblage is east to southeast-striking and comprises komatiitic flows, magnesium to iron-rich mafic volcanic rocks, thin rhyolite units (FIII-type to calc-alkaline), clastic sedimentary rocks (argillite and greywackes, many graphitic), and chemical sedimentary rocks (limestone, dolomite) occurring as interflow horizons. These units are intruded by mafic to ultramafic bodies and minor felsic dikes (Ayer et al., 2002a and 2002b; Sproule et al., 2005; Ayer et al., 2005).

Almost all komatiite-associated Ni-Cu-(PGE) deposits in the AGB are interpreted to be localized in lava channels/channelized sheet flows (e.g., Alexo, Hart, Langmuir, Marbridge, and Texmont) or channelized sheet sills (e.g., Sothman, Dumont, Kelex-Dundeal-Dundonald South). One exception is the McWatters deposit, which occurs within a thick mesocumulate to adcumulate peridotite that is interpreted to be a synvolcanic dike (Houlé and Lesher, 2011).

### 7.1.2 Economic Geology

The Timmins Mining camp has a history of nickel production from komatiite-associated Ni-Cu-(PGE) deposits (Table 7-1 below and Figure 7-3 in Section 7.2). Several of these deposit types have been identified within the Kidd-Munro Assemblage (e.g., Alexo, Dundonald, Mickel, and Marbridge) and the Tisdale Assemblage (e.g., Hart, Langmuir, Redstone, Texmont, and Sothman).

**Table 7-1: Pre-Mining Geologic Resource Estimates Plus Mined Ore, Komatiite-hosted Ni-Cu-(PGE) Mines/Deposits, Timmins Mining Camp, Ontario**

| Name | Status | Township | Notes | Assemblage | Milled (t) | Reported (t) | Ni (%) |
|---|---|---|---|---|---|---|---|
| Alexo | Past producer | Dundonald | extrusive | Kidd-Munro | 115,000 | - | 3.18 |
| Kelex | Past producer | Clergue | intrusive (subvolcanic sill) | Kidd-Munro | 279,000 | - | 0.97 |
| Dundeal | Deposit | Dundonald | intrusive (subvolcanic sill) | Kidd-Munro | - | 400,000 | 2.00 |
| Dundonald | Deposit | Dundonald | intrusive (subvolcanic sill) | Kidd-Munro | - | 141,000 | 2.73 |
| Langmuir #1 | Deposit | Langmuir | extrusive; Shaw Dome | Tisdale | 1,834,000 | - | 0.58 |
| Langmuir #2 | Past producer | Langmuir | extrusive; Shaw Dome | Tisdale | 1,369,000 | - | 1.40 |
| McWatters | Past producer | Langmuir | intrusive; Shaw Dome | Tisdale | 1,688,000 | - | 0.75 |
| Redstone | Past producer | Eldorado | extrusive; Shaw Dome | Tisdale | 2,043,000 | - | 1.62 |
| Hart | Deposit | Eldorado | extrusive; Shaw Dome | Tisdale | 1.868,000 | - | 1.38 |
| Texmont | Past producer | Bartlett Geikie | extrusive | Tisdale | 3,369,000 | - | 0.92 |

Source: Modified after Houle et al., 2017.

The past producers and deposits listed in Table 7-1 are for illustrative purposes only, a qualified person has not completed sufficient work to verify the information, and mineralization hosted on adjacent or nearby properties is not necessarily indicative of mineralization hosted on the Crawford property.

In addition to nickel, the Timmins-Porcupine Gold Camp of Northeastern Ontario represents the largest Archean orogenic greenstone-hosted gold camp in the world in terms of total gold production (e.g., Monecke et al., 2017).

The Kidd Creek Cu-Zn deposit, north of Timmins and about 15 km south of Crawford Township, is the world's largest and highest-grade Archean volcanic-hosted massive sulphide (VHMS) deposit (aka volcanogenic massive sulphide or VMS) currently in production. Monecke et al. (2017) reported historical past production, reserves, and resources to the 2,990 m level as 170.9 Mt grading 2.25% Cu, 5.88% Zn, 0.22% Pb, and 77 g Ag/t. Discovery hole K55-1 was drilled in 1963 and encountered ore at a depth of 7 m, intersecting 190 m (entire hole) grading 1.21% Cu, 8.5% Zn, 0.8% Pb, and 138 g Ag/t. Today, the orebodies of the deposit are exploited from surface to more than 3 km depth and are open at depth, making Kidd Creek the deepest base metal mine in the world (Monecke et al., 2017).

## 7.2 Local and Project Geology

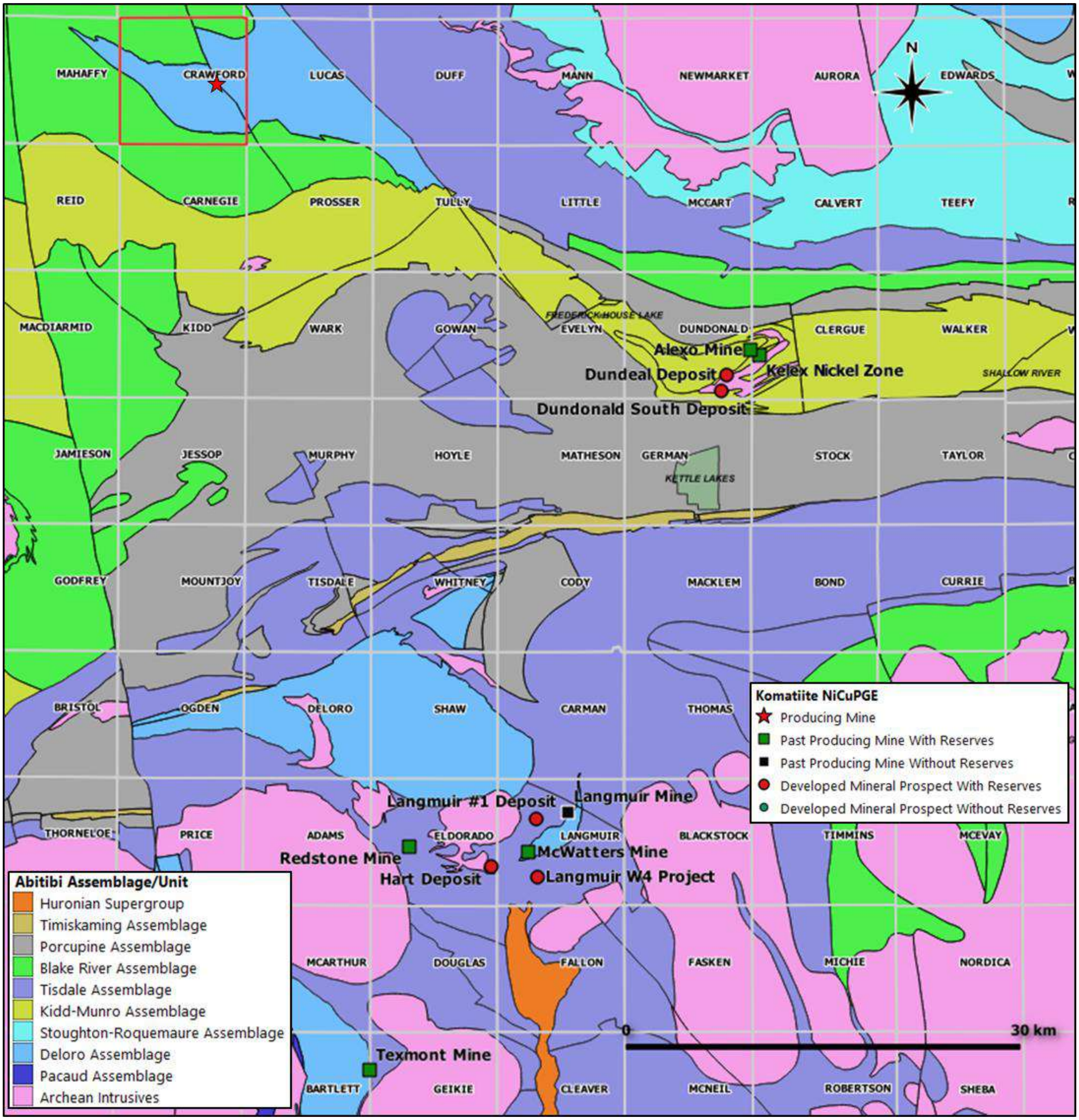
The Greenstone Architecture Project (2003-2005) and Discover Abitibi Initiative (2001-2012), led by the Ontario Geological Survey (OGS), resulted in reclassification of the lithological assemblages in the southern AGB (Ontario portion) by using detailed U/Pb geochronology, and updated geological and geophysical compilations (Ayer et al., 2005; Thurston et al., 2008). This work suggests that the rocks underlying the property are part of the Deloro Assemblage (Figures 7-3 and 7-4) (Monecke et al., 2017).

The Deloro Assemblage (2730 to 2724 Ma) consists mainly of mafic to felsic calc-alkaline volcanic rocks with local tholeiitic mafic volcanic units and an iron formation cap which is typically iron-poor, chert-magnetite (Ayer et al., 2005; Thurston et al., 2008). This assemblage (volcanic episode) is host to the CUC on the property (Crawford and Lucas townships) and other ultramafic sills in the area.

The surrounding and regional lithologies—not underlying the property—belong to the Blake River Assemblage (2704 to 2701 Ma) which consists mainly of tholeiitic mafic volcanic rocks with isolated units of tholeiitic felsic volcanic rocks and turbiditic sedimentary rocks (Ayer et al., 2005; Thurston et al., 2008). This assemblage, also referred to as the Blake River Group, is host to several mafic-ultramafic sills in the northern part of Crawford Township and in neighbouring Lucas, Mahaffy and Aubin townships. The Blake River Assemblage, the youngest volcanic-dominated package, is one of the most prospective Archean stratigraphic packages for VHMS exploration, and especially for gold-rich VHMS deposits (Ross et al., 2009).

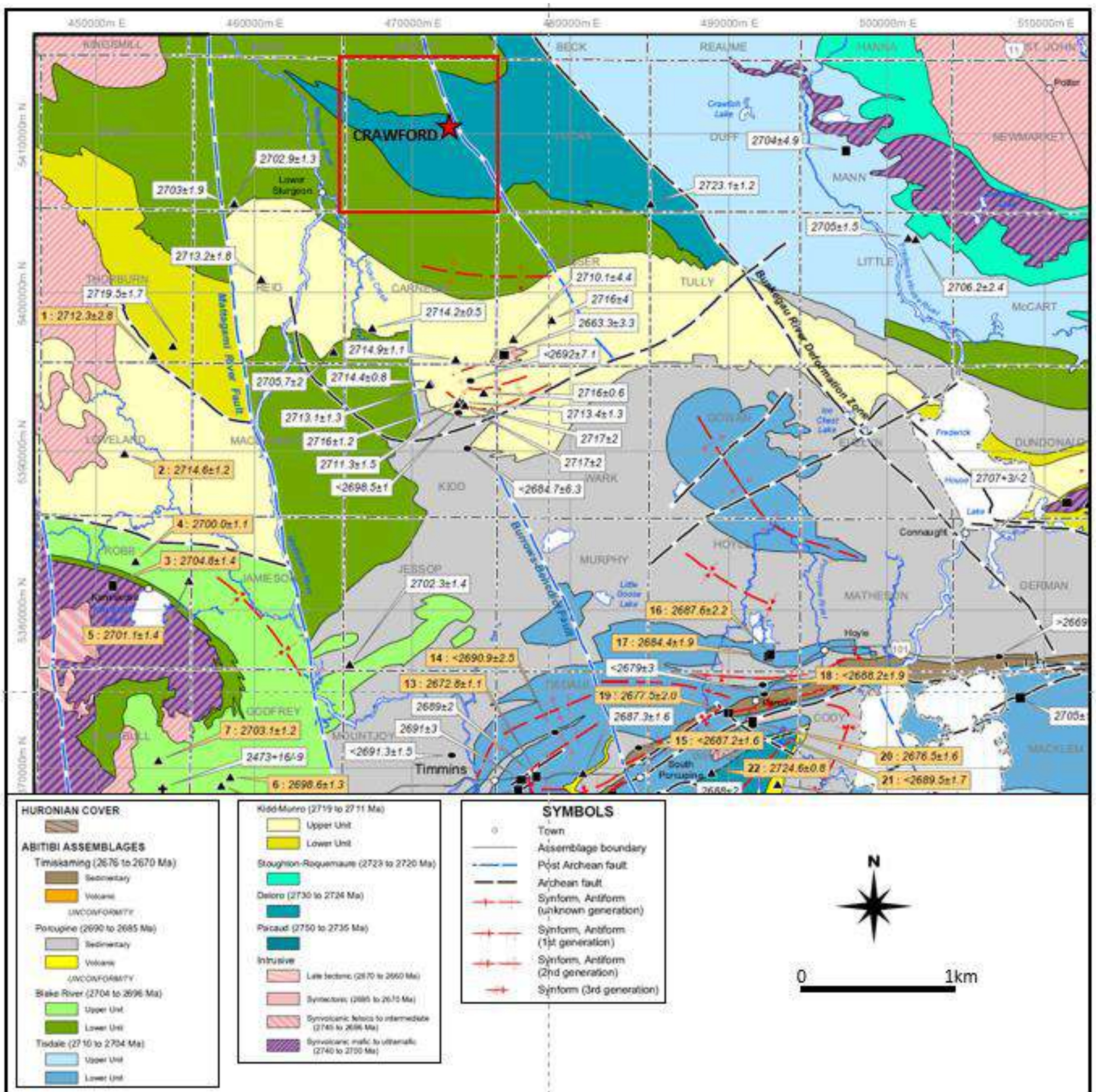
The rocks have undergone greenschist facies metamorphism with widespread carbonate, chlorite, and sericite alteration in volcanic rocks and serpentinization in ultramafic rocks (i.e., dunite, peridotite).

**Figure 7-3: Locations of Komatiite-Hosted Ni-Cu-(PGE) Deposits/Mines in the Timmins Mining Camp**




Note: Location of the project area in Crawford Township (red square) and Lucas Township, and the CUC (red star). Source: Caracle Creek, 2020. Geology of the Abitibi assemblages (volcanic episodes) is from Ayer et al., (2005) and Ontario Geological Survey MRD155.

**Figure 7-4: Regional Geology and Location of the Property and Crawford Ultramafic Complex**



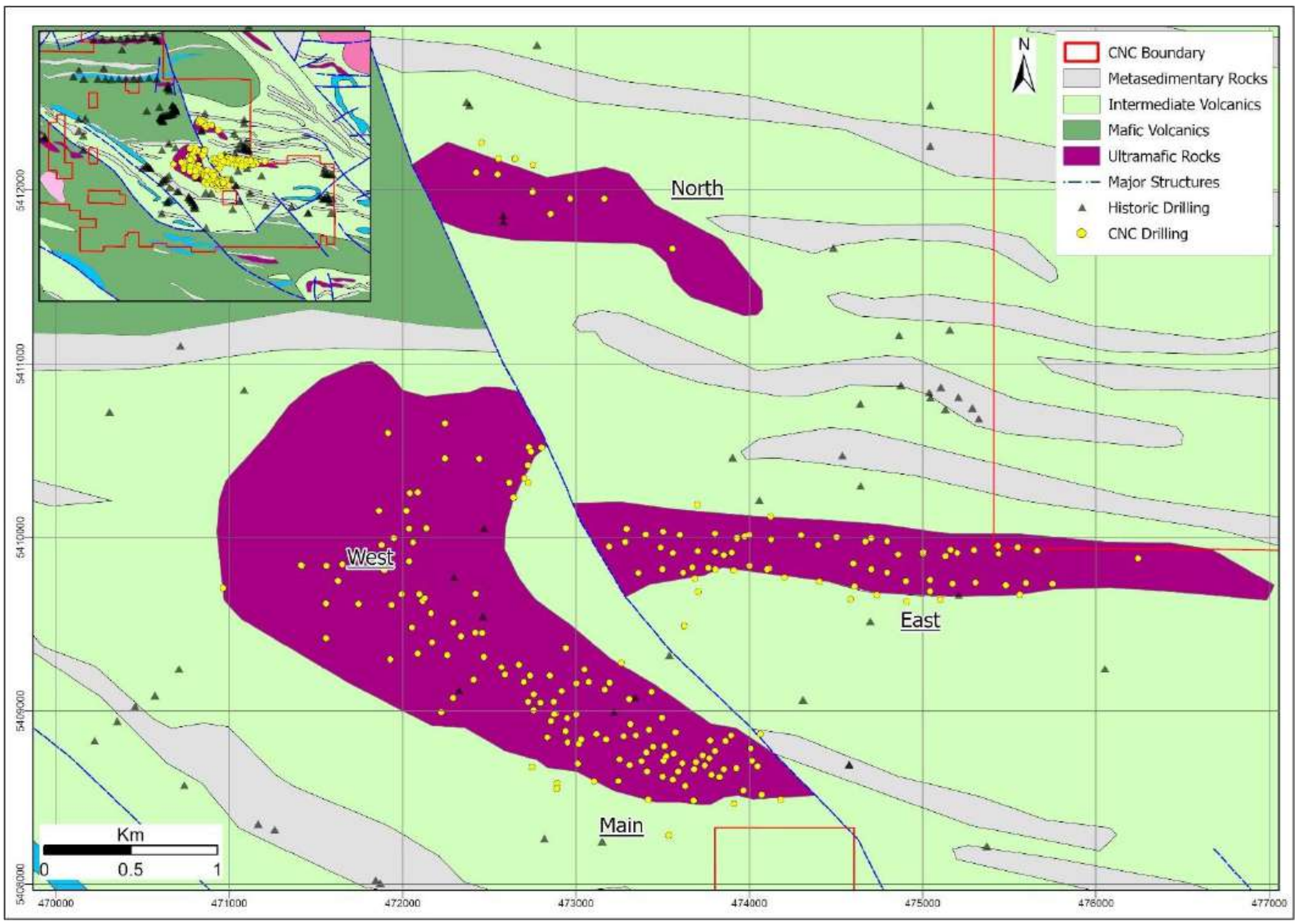

Note: Location of the property in Crawford Township (red square) and the approximate location of the CUC (red star). Also shown are the age dates from U/Pb geochronology samples taken from various Abitibi assemblages. Source: Ayer et al., 2005.

## 7.3 Crawford Ultramafic Complex

Historical work in Crawford and Lucas townships has generated several generations of geological maps with geology inferred almost entirely from diamond drill core, overburden bedrock interval sampling, and the interpretation of geophysical surveys.

The principal target, the CUC (Figure 7-5), is entirely under cover but based on geophysics and drilling is an approximately 8.0 km long by 2.0 km wide body (original estimated shape) of dunite, peridotite (and their serpentinized equivalents), and lesser pyroxenite and gabbro, as confirmed in historical diamond drillholes (Spruce Ridge Resources, 2018) and the current extensive drilling program by CNC. Historical diamond drilling in the 1960s and 1970s also reported intersections of gabbro, peridotite, pyroxenite, dunite and serpentinite (e.g., George, 1970). Descriptions from drill core logs record localized brecciation in the Main Zone at the northern contact between mafic volcanic rocks and dunite.

**Figure 7-5: General Geology of the Crawford Ultramafic Complex (Purple), Surrounding Rock Units, Major Structures, and Location of Historical and Current Drillhole Collars**




Source: CNC, 2023. Modified from OGS MRD 126.

The CUC, although geophysically recognized as early as 1964, was recently redefined by a high-resolution helicopter-borne magnetic and electromagnetic survey in 2017 (Balch, 2017) and a high-sensitivity aeromagnetic and airborne gravimetric survey in 2018 (CGG, 2018), both conducted over the entire Crawford Township, and followed up with 3D-inversion and detailed interpretation (St-Hilaire, 2019).

## 7.4 Mineralization

The Abitibi Greenstone Belt affords a mineral exploration company with several target deposit types and commodities, including Ni-Cu-(PGE), VHMS, and orogenic gold.

Within Crawford Township, several prominent mafic-ultramafic intrusions (i.e., sills) offer the potential for sulphide-associated nickel, copper, cobalt, and platinum-group element (PGE) mineralization. Intrusion-hosted nickel sulphide mineralization is the principal target on the Crawford Project where it has been intersected by drilling, hosted in units of dunite and peridotite (largely serpentinized).

Mineralization has been drill-delineated in two principal zones of nickel sulphide mineralization, the Main Zone and the East Zone (Figure 7-5). Other areas of similar mineralization have been identified through diamond drilling by CNC, including the West Zone (Lane et al., 2022) and North Zone (Figure 7-5).

Within the Main Zone, drilling to date allows for the delineation of two higher grade (>0.30% Ni and >0.35% Ni) regions (modelled grade shells) within a larger core high-grade zone (>0.25% Ni), which in turn are within the larger enveloping low-grade zone (>0.15% Ni), all contained within the host ultramafic body of the CUC (see Section 10). The high-grade zone (>0.25% Ni) has a minimum modelled strike length of about 3.3 km, is between approximately 70 and 250 m wide, and contains regions of incrementally higher grade nickel (i.e., >0.30% Ni and >0.35% Ni). The high-grade zone and internal regions of higher grade nickel remain open along strike to the north-northwest and extend to a depth of at least 800 m (open at depth).

Located within the northern margin of the ultramafic to mafic body of the Main Zone, is a PGE Reef potentially 2 km long and up to 9 m wide, that is associated with a contact between a peridotite unit to the south and a pyroxenite unit to the north (see Section 10.3.1.2). Additional drillholes will be required to better define the PGE Reef.

Located about 1.2 km northeast of the Main Zone is the East Zone (Figure 7-5). The East Zone extends for about 2.6 km east-west, is open to the east but is truncated to the west by a large regional north-northwest trending fault (Figure 7-5). Like the Main Zone, mineralization in the East Zone is characterized by a relatively high-grade nickel core between approximately 70 and 200 m wide and at least 800 m deep (open at depth), within a low-grade nickel envelope (see Section 10) and hosted by an ultramafic body comprising variably serpentinized dunite and peridotite, with lesser pyroxenite and minor gabbro.

Within the layered ultramafic unit of the East Zone, two PGE reefs have been differentiated: (1) a higher grade reef potentially 1.5 km long and up to 8 m wide at the northern boundary of the nickel-rich domain, and (2) a lower grade reef potentially 2 km long and up to 6 m wide, near the northern margin of the ultramafic body (see Section 10.3.3).

The Crawford deposits consist of large volumes of altered ultramafic rocks comprising relatively low nickel grades, derived as a result of serpentinization of the peridotitic to dunitic protolith, with serpentinization occurring when peridotite-dunite alter via metasomatism as per the following reaction:

$$3\ Mg_2SiO_4\ \text{(olivine)} + SiO_2 + H_2O \rightarrow 2\ Mg_3Si_2O_5(OH)_4\ \text{(serpentine)}\ \text{(Brownlow, 2006)}$$

During serpentinization, Ni, which also fits within the olivine structure substituting for Mg, is liberated and can form higher nickel tenor sulphides and nickel-iron alloys within the altered ultramafic rocks (Sciortino, 2014).

Although magmatic nickel sulphide (pentlandite) is present in the Crawford deposits, nickel grades have been significantly upgraded through the serpentinization (alteration) process, producing secondary minerals that are not magmatic in origin, such as heazlewoodite, awaruite, and godlevskite (see Section 6.3.2).

Where visible, nickel sulphide mineralization is dominated by fine-grained, disseminated pentlandite (magmatic) and heazlewoodite (hydrothermal). In general, pentlandite is the dominant nickel sulphide, except in regions of higher grade nickel (i.e., dunite core) where heazlewoodite is the dominant nickel species. This reflects a hydrothermal upgrading of nickel concentration in the CUC, also suggesting very low sulphur conditions. Overall, pyrrhotite occurs at about the same average percentage as awaruite but was identified in less of the drill holes. This suggests that although pyrrhotite occurs less often than awaruite, when it does occur, it is by far the dominant mineral phase.

# 8 DEPOSIT TYPES

Sulphide mineralization discovered to date in the Project area can be characterized as Komatiite-hosted Ni-Cu-Co-(PGE) deposit type, which recognizes two sub-types (Lesher and Keays, 2002):

Type I Kambalda-style: channelized flow theory; komatiite-hosted; dominated by net-textured and massive sulphides situated at or near the basal ultramafic/footwall contact with deposits commonly found in footwall embayments up to 200 m in strike length, 10s to 100s of metres in down-dip extent, and metres to tens of metres in thickness; generally on the order of millions of tonnes (generally <5 Mt) with nickel grades that are typically much greater than 1% nickel; tend to occur in clusters (e.g., Alexo-Dundonald, Ontario; Langmuir, Ontario; Redstone, Ontario; Montcalm, Ontario; Thompson, Manitoba; Raglan, Quebec).

Type II Mt. Keith-style: sheet flow theory; thick komatiitic olivine adcumulate-hosted; disseminated and bleb sulphides, hosted primarily in a central core of a thick, differentiated, dunite-peridotite dominated, ultramafic body; more common nickel sulphides such as pyrrhotite and pentlandite but also sulphur poor mineral Heazlewoodite ($Ni_3S_2$) and nickel-iron alloys such as Awaruite ($Ni_3$-Fe); generally on the order of 10s to 100s of million tonnes with nickel grades of less than 1% (e.g., Mt. Keith, Australia; Dumont Deposit, Quebec).

The Mt. Keith deposit (aka MKD5), located in the Yilgarn Craton of Western Australia, was first drill-tested and discovered in 1968 and put into production in 1993 (Butt and Brand, 2003). The MKD5 deposit is hosted by a serpentinized dunite within a larger, lenticular peridotite-dunite komatiite body, the Mt. Keith Ultramafic Complex and has a complex residual regolith profile of more than 75 m thickness (up to 120 m weathering profile).

Sulphide mineralogy at Mt. Keith is dominated by disseminated (intercumulus) pentlandite, with minor millerite, violarite, and pyrrhotite, and trace amounts of pyrite, chalcopyrite, heazlewoodite, polydymite, and gersdorffite. Nickel sulphide mineralization strikes for 2 km, is about 350 m wide, and is open below 600 m depth. In 2002, the deposit had proven and probable reserves of 299 Mt grading 0.56% Ni (0.4% Ni cut-off) (Butt and Brand, 2003).

Deposits of the Crawford Project, which closely resemble the Dumont nickel deposit, are most like the Type II, Mt. Keith-style of deposit, in that they are dominated by disseminated nickel sulphide mineralization (i.e., pentlandite). However, in the case of Crawford, the pervasive hydrothermal alteration (serpentinization), which led to widespread and significant upgrades in nickel concentration, is key to it being economic.

## 8.1 Crawford Nickel Sulphide Project Analogy – Dumont Nickel Deposit

The type-example of the komatiite-hosted Ni-Co-PGE exploration model that CNC is using for the Crawford Ultramafic Complex is the Dumont nickel deposit of Dumont Nickel by Magneto Investments L.P., previously Royal Nickel Corporation (RNC), located 220 km to the east of Crawford Township, about 60 km northeast of Rouyn-Noranda, and within the Abitibi Greenstone Belt (Abitibi Region) in northwestern Quebec (Ausenco, 2013).

The komatiitic (>18 wt% MgO), synvolcanic Dumont sill occurs within a sequence of iron-rich tholeiite lavas and volcaniclastic rocks assigned to the Amos Group and which are part of the Barraute Volcanic Complex. Although the exact age of the Dumont sill is not known, stratigraphic studies in the AGB suggest that the host rocks (Amos Group) are correlative with the Deloro assemblage (Monecke et al., 2017; Mercier-Langevin et al., 2017).

The differentiated Dumont sill, about 7 km long, up to 1 km wide, and extending to a depth of more than 500 m, dips steeply to the northeast. Its lower Ultramafic Zone (~450 m thick) comprises the Lower Peridotite Subzone, an olivine + chromite cumulate, the Dunite Subzone, an olivine ±sulphide cumulate, and the Upper Peridotite Subzone, an olivine + chromite cumulate. The overlying Mafic Zone (~250 m thick) comprises the Clinopyroxenite Subzone, a clinopyroxene cumulate, the Gabbro Subzone, a clinopyroxene + plagioclase cumulate, and the quartz gabbro which includes plagioclase + pyroxene cumulates as well as non-cumulate gabbro (Duke, 1986).

An NI 43-101 mineral resource estimate reported by RNC in July 2019 (Ausenco, 2019), quotes measured plus indicated mineral resources of 1.66 billion tonnes grading 0.27% Ni, 107 ppm Co, 9 ppb Pt and 20 ppb Pd, plus an inferred mineral resource of 0.5 billion tonnes grading 0.26% Ni, 101 ppm Co, 6 ppb Pt and 12 ppb Pd. The same study also included a mineral reserve statement with proven reserves of 163,140,000 tonnes grading 0.33% Ni, 114 ppm Co, 13 ppb Pt and 31 ppb Pd, and probable reserves of 864,908,000 tonnes grading 0.26% Ni, 106 ppm Co, 8 ppb Pt and 17 ppb Pd. The QP has been unable to verify the information; the information is not necessarily indicative of the mineralization on the property that is the subject of the technical report.

The Dumont deposit is usually categorized with its most similar counterpart, the Mt. Keith nickel deposit (Naldrett, 1989). Although the Dumont and Crawford nickel deposits share some similarities with Mt. Keith, it should be noted that nickel grades reported from reserves at Mt. Keith range from 0.48% to 0.57% Ni. In addition to the lower grade, the Crawford and Dumont deposits are differentiated from the Mt. Keith deposit by the abundance of secondary minerals heazlewoodite and awaruite. In addition, the Dumont and Crawford deposits have not been subjected to the extensive supergene weathering alteration which characterizes the Mt. Keith deposit.

Mineralization hosted by the Dumont Nickel Project and the Mt. Keith deposit is not necessarily indicative of mineralization hosted on the Crawford Project.

## 8.2 Komatiite Emplacement Models

After the discovery of the Kambalda and Mt. Keith Ni-Cu-Co-(PGE) deposits in Australia (ca. 1971), geological models were developed for these ultramafic extrusive komatiite-hosted deposits (e.g., Lesher and Keays, 2002; Butt and Brand, 2003; Barnes et al., 2004).

Komatiitic rocks are derived from high-degree partial melts of the Earth’s mantle. Due to the high degree of partial melting the komatiitic melt is enriched in elements such as nickel and magnesium. When erupted, the melts have a low viscosity and tend to flow turbulently over the substrate eroding the footwall lithologies through a combination of physical and chemical processes.

Due to the low viscosity of the komatiitic melts, the lavas tended to concentrate in topographic lows. Komatiitic eruptions have been envisaged to have a high effusion rate and large volumes of lava and/or magma. The Mt. Keith-style of deposits are no exception, interpreted to be large volume sheet flows several hundreds of metres thick by several kilometres to tens of kilometres long and are composed primarily of olivine adcumulate to mesocumulate.

Further downstream, more distal from the eruptive source, the komatiitic flows become channelized, similar to a river channel today, and begin to erode the substrate forming more defined channel feature. This channelization is the cornerstone of the Kambalda model. Denser sulphides would tend to accumulate in the bottom of the channel-like features under the influence of gravity. As the eruption continued the channel would fill with olivine mesocumulate to adcumulate because of the constantly replenished magnesium-rich komatiitic melt.

As the eruption waned, the channel would be capped by a sequence of regressive komatiitic flows composed of komatiitic pyroxenite and basalts. In order to develop Ni-Cu sulphides, the komatiitic melt must become sulphide saturated. A komatiitic melt will become sulphur saturated when an external source of sulphur is introduced to the melt by assimilation of a sulphide-rich lithology or by differentiation or contamination of a komatiitic melt until the sulphur content exceeds the saturation point. A strong relationship exists between the presence of footwall lithologies rich in sulphide and the development of Ni-Cu sulphide deposits in the overlying komatiitic flows. This association is strongest in the Kambalda-style Ni-Cu sulphide deposits. Differentiation or the assimilation of rocks rich in certain elements may result in the oversaturation of the komatiitic melt in sulphur. This is the mechanism related to the development of the Mt. Keith-style of deposits.

Komatiite-hosted Ni sulphide deposits, whether they are Archean (e.g., Kambalda, Australia) or Proterozoic (e.g., Thompson, Manitoba; Raglan, Quebec) occur in clusters of small sulphide bodies generally less than 1 Mt. At 1:250000 scale, these deposits usually occur at a pronounced thickening of ultramafic stratigraphy, and at 1:5000 scale, these deposits occur as net-textured to massive sulphide in small embayments up to 200 m in strike length, tens to hundreds of metres in down-dip length and metres to tens of metres thick. The shape can be cylindrical, podiform, or in rare instances tabular.

# 9 EXPLORATION

In addition to diamond drilling (see Section 10), CNC contracted Crone Geophysics & Exploration to carry out a downhole electromagnetic survey on the property between June 1 and 7, 2022. One hole utilizing one transmitter loop was surveyed during this period. Figure 9-1 on the following page provides borehole and loop location map of the hole surveyed.

The loop details, acquisition parameters, and survey coverage are provided in Tables 9-1 and 9-2 below. A 3D borehole pulse-EM system was assembled in which the axial component (Z) and cross component (X-Y) of the induced secondary field were measured with a Crone borehole induction coil probe. The Z component detects any in-hole or off-hole anomalies and gives information on size, conductivity, and distances to the edge of conductors.

**Table 9-1: Borehole Survey Transmitter Loop Coverage**

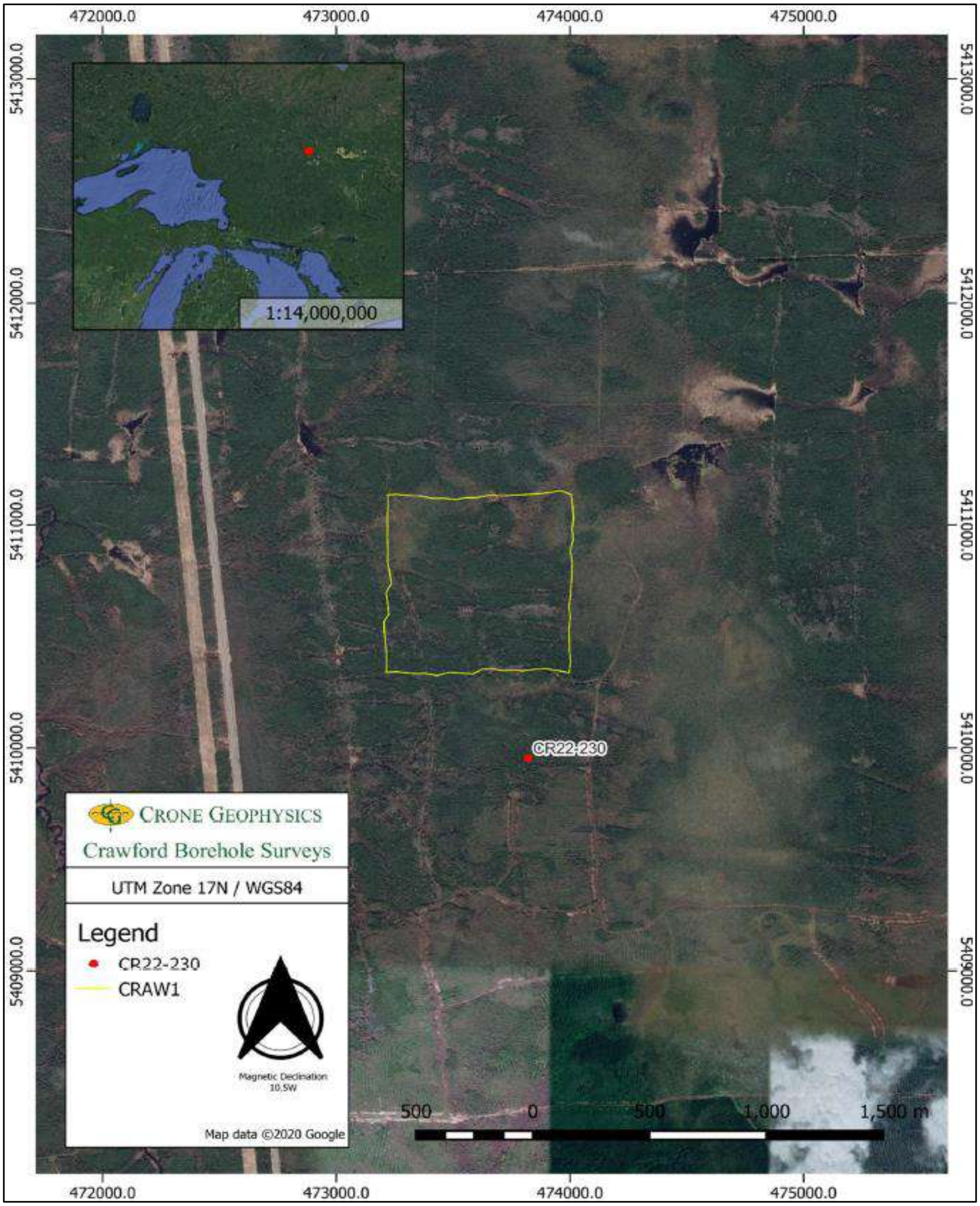
| Tx Loop | Property / Target | Size (m) | Corner Coordinates (WGS84 UTM Zone 17N) |
|---|---|---|---|
| Craw1 | Crawford | 800 x 800 | 473220 mE, 5411134 mN<br>473212 mE, 5410339 mN<br>473995 mE, 5410336 mN<br>474005 mE, 5411135 mN |

**Table 9-2: Borehole Survey Coverage**

| Hole | Zone | Time Base (ms) | Off Time Channels | Ramp (ms) | Current (A) | From (m) | To (m) | Length (m) | Comp |
|---|---|---|---|---|---|---|---|---|---|
| CR22-230 | Main Zone | 16.66 | 20 | 1.5 | 16 | 540 | 1070 | 530 | X,Y,Z |

The Z-component response suggests the hole is adjacent to a large conductor that decays rapidly with little remaining energy by channel 17 using a 16.66 ms time base. The non-decaying response as measured by S1 (the measured primary field reduced by the theoretical primary field) shows elevated responses peaking at 650 m, 790 m, and 960 metres. The cross-components do not produce distinct crossovers at any of these peaks, however, suggesting the response is most likely due to variations in the proximity of the conductor edge relative to the drillhole at depth (i.e., pinching and swelling of a thick conductor). As the goal of the survey was to identify any massive sulphide lenses within the thick sequence of weakly conductive serpentinite, and since no obvious lenses were identified, there can be no further recommendations with this data set.

**Figure 9-1: Surveyed Borehole and Geophysical Loop Location Map on the Property**




Source: CNC, 2023

## 9.1 Sponsored M.Sc. Studies

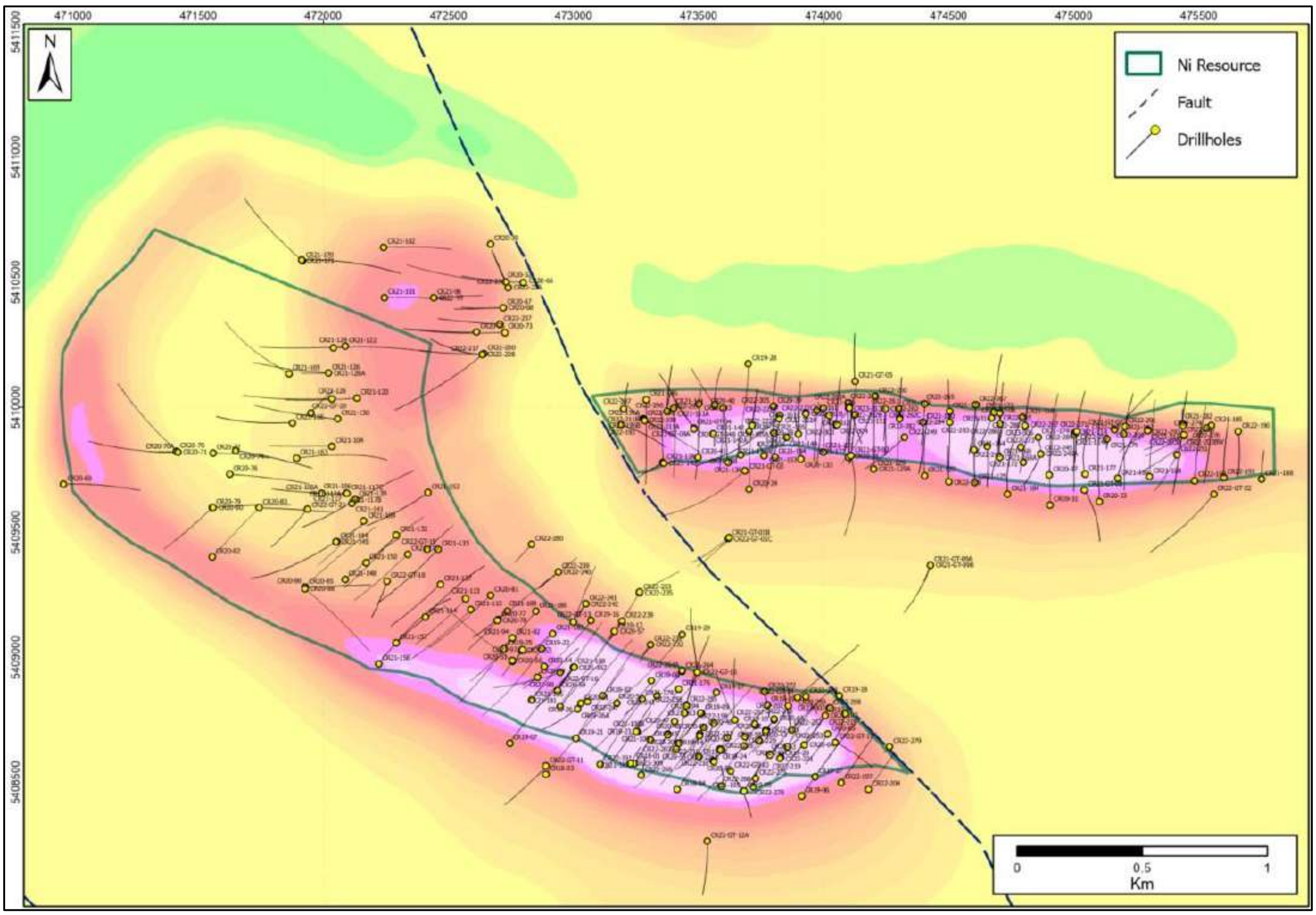
CNC is sponsoring two M.Sc. thesis studies on the Crawford Project through Laurentian University, Sudbury, Ontario. One thesis is looking to establish the in-situ paragenesis, textural relationships, and distribution of carbon minerals and carbon sequestering minerals within the CUC to provide carbon content baseline for processing development studies. The other thesis aims to characterize the serpentinization of olivine and sulphide assemblages in the CUC, determine its relationship to nickel deportment and subsequent mineralization, and to help locate and evaluate similar deposits in the district and elsewhere.

# 10 DRILLING

## 10.1 Introduction

Diamond drilling programs are ongoing, having been initiated by Spruce Ridge in September 2018, under its option-joint venture agreement with then property owner, Noble Mineral Exploration. With the October 1, 2019, announcement that Noble had created a new entity, Canada Nickel Company (CNC), to focus on the Crawford Project, management and control of the drilling program shifted from Spruce Ridge to CNC. The location of drillhole collars and traces within the Crawford project property is provided in Figure 10-1.

**Figure 10-1: Plan View of Diamond Drillhole Traces from 2018-2023 Drilling within Main Zone, East Zone, and West Zone**




Notes: Outlines showing the Main Zone and East Zone nickel mineral resource estimate envelopes superimposed on airborne total field magnetic intensity (TMI - linear colour transform from low (blue) to high (red) magnetic intensity). Source: CNC, 2023.

Results from the initial four drillholes completed by Spruce Ridge and Noble (CR18 series) are discussed in detail in Section 6.2, Historical Drilling. Following on from the initial four holes completed in late 2018 and reported in early 2019 (Noble news release dated March 4, 2019), results from CNC's first nine drillholes (CR19-05 to 13), which totalled 5,280 m, were announced by Noble on December 9, 2019.

As of April 2023, a total of 372 drillholes totalling approximately 151,969 metres (up to hole CR22-305), were completed by CNC and previous companies (Figure 10-1; Table 10-1). This includes drilling 10,474 m from 52 abandoned holes. Additionally, 54 holes were HQ size, completed for geotechnical and metallurgical testwork, whereas the remaining 266 drillholes used NQ size. A total of 322 of these 372 drillholes were used to calculate the updated mineral resource estimates in the Main Zone and East Zone (see Section 14).

## 10.2 Drillhole Collar Surveys

All the drillhole collar locations in the Main, East, and West zones, were determined through a differential GPS (DGPS) survey. Collar inclination was measured using a manual inclinometer (Table 10-1). DGPS drillhole collar surveys were carried out by Talbot Surveys Inc. of Timmins, Ontario. The APS system utilized by the drillers is a Multiwave Sensors' GPS-based compass providing true north or grid north azimuth. The unit has sub-metre accuracy with the satellite-based augmentation system (SBAS) to ±0.6 m or better and ±2.5 m accuracy when SBAS is not available. The handheld GPS unit provided location accuracy of approximately ±3 m.

In general, drillhole surveys were initiated immediately following the casing and then every 50 m afterward using a Reflex gyrocompass system. These preliminary surveys served the purpose of informing the geologists on deviation in real time. After the hole was finished, a survey was completed before removing the rods, in this case the final survey was a "continuous" survey, taking measurements approximately every 10 metres.

## 10.3 Diamond Drill Core Assay Results

As of the effective date of the mineral resource estimate (August 31, 2023), diamond drilling has been completed at the Main, East, and West zones (Figure 10-1), with 196 of the current 372 drillholes used in the updated Main-West Zone mineral resource estimate and 126 used in the initial East Zone mineral resource estimate (see Section 14.2); the last hole used in the mineral resource estimate was CR22-305.

### 10.3.1 Main Zone Drilling

The focus of the 2019-2023 drilling was to extend along strike mineralization encountered in the original historical 2018 series drillholes (Spruce Ridge) and subsequent drilling campaigns, to keep testing the east-northeastern and west-southwestern extents of mineralization (i.e., the contacts) as well as the deeper portions of the CUC, and to complete in-fill drilling within the mineral resource envelope and its higher grade core. Selective drill core assay results from the Main Zone are summarized in Table 10-2. To date, diamond drilling has outlined a west-northwest trending (~285-315Az) ultramafic body (largely dunite-peridotite) that is at least 1.9 km in strike length, 280 to 580 m in width, and more than 800 m deep (Figures 10-2 and 10-3). Mineralization remains open along strike to the northwest, and at depth. A north-northwest trending regional sinistral, strike-slip fault terminates the ultramafic body along its southeastern extent (Figure 7-5). A 3D-inversion magnetic anomaly, nearly 1 km deep, has been only partially tested at depth (Figure 10-3).

**Table 10-1: Summary of Parameters for CR18, CR19, CR20, CR21, CR22, CR23, CR21-GT, and CR22-GT Series Diamond Drillholes**

| Drillhole | UTM (mE) | UTM (mN) | Elev (m) | Length (m) | Collar Az | Collar Dip | Survey | Zone |
|---|---|---|---|---|---|---|---|---|
| CR18-01 | 473244.0 | 5408577.5 | 273.6 | 594 | 33 | -60 | DGPS | Main |
| CR18-02 | 473117.7 | 5408847.9 | 273.1 | 216 | 37 | -51 | DGPS | Main |
| CR18-03 | 472890.1 | 5408533.0 | 271.5 | 606 | 36 | -51 | DGPS | Main |
| CR18-04 | 473416.1 | 5408472.7 | 272.9 | 402 | 35 | -51 | DGPS | Main |
| CR19-05 | 473679.6 | 5408466.2 | 273.1 | 582 | 35 | -50 | DGPS | Main |
| CR19-06 | 473911.0 | 5408446.2 | 273.4 | 576 | 35 | -50 | DGPS | Main |
| CR19-07 | 472744.8 | 5408657.6 | 275.9 | 621 | 35 | -50 | DGPS | Main |
| CR19-08 | 473311.9 | 5408905.6 | 274.6 | 606 | 215 | -50 | DGPS | Main |
| CR19-09 | 473511.0 | 5408778.2 | 273.9 | 603 | 215 | -50 | DGPS | Main |
| CR19-10 | 473729.3 | 5408725.2 | 273.8 | 588 | 215 | -50 | DGPS | Main |
| CR19-11 | 474006.3 | 5408767.5 | 274.4 | 519 | 215 | -50 | DGPS | Main |
| CR19-12 | 473501.6 | 5408602.6 | 273.2 | 576 | 35 | -50 | DGPS | Main |
| CR19-13 | 473164.7 | 5409104.9 | 274.9 | 609 | 215 | -50 | DGPS | Main |
| CR19-14 | 473410.3 | 5408633.5 | 273.6 | 65 | 35 | -82 | DGPS | Main |
| CR19-14A | 473410.3 | 5408633.5 | 273.6 | 944 | 15 | -85 | DGPS | Main |
| CR19-15 | 473307.9 | 5408669.7 | 273.6 | 600 | 35 | -50 | DGPS | Main |
| CR19-16 | 473072.0 | 5409148.6 | 273.3 | 642 | 215 | -50 | DGPS | Main |
| CR19-17 | 473572.0 | 5408859.5 | 274.1 | 501 | 215 | -55 | DGPS | Main |
| CR19-18 | 474063.5 | 5408848.6 | 274.6 | 507 | 215 | -50 | DGPS | Main |
| CR19-19 | 473773.9 | 5408810.9 | 274.3 | 723 | 215 | -65 | DGPS | Main |
| CR19-20 | 473827.3 | 5408599.7 | 273.6 | 702 | 35 | -82 | DGPS | Main |
| CR19-21 | 473009.8 | 5408678.0 | 271.8 | 702 | 35 | -50 | DGPS | Main |
| CR19-22 | 472873.3 | 5409034.9 | 271.1 | 570 | 215 | -50 | DGPS | Main |
| CR19-23 | 473250.9 | 5408702.9 | 273.3 | 705 | 35 | -82 | DGPS | Main |
| CR19-24 | 473586.5 | 5408633.1 | 273.4 | 702 | 35 | -82 | DGPS | Main |
| CR19-25 | 472723.7 | 5409035.2 | 270.4 | 453 | 215 | -50 | DGPS | Main |
| CR19-26 | 473016.9 | 5408792.1 | 271.8 | 75 | 35 | -82 | DGPS | Main |
| CR19-26A | 473016.9 | 5408792.1 | 271.8 | 146 | 35 | -82 | DGPS | Main |
| CR19-27 | 473965.6 | 5408523.5 | 273.4 | 444 | 35 | -50 | DGPS | Main |
| CR19-28 | 473700.0 | 5410172.1 | 277.8 | 573 | 180 | -50 | DGPS | East |
| CR19-29 | 473435.3 | 5409092.7 | 275.0 | 750 | 215 | -50 | DGPS | Main |
| CR19-31 | 474905.4 | 5409610.1 | 276.1 | 552 | 360 | -50 | DGPS | East |
| CR20-30 | 472669.3 | 5410652.9 | 271.0 | 0 | 270 | -50 | DGPS | West |
| CR20-32 | 472728.3 | 5410500.8 | 271.6 | 633 | 270 | -50 | DGPS | West |
| CR20-33 | 475100.7 | 5409622.9 | 275.6 | 549 | 360 | -50 | DGPS | East |
| CR20-34 | 473703.4 | 5409670.8 | 277.6 | 636 | 360 | -50 | DGPS | East |
| CR20-35 | 473701.3 | 5409902.8 | 277.9 | 390 | 360 | -82 | DGPS | East |
| CR20-36 | 474102.7 | 5409798.7 | 277.1 | 483 | 360 | -50 | DGPS | East |
| CR20-37 | 474902.8 | 5409728.2 | 276.2 | 492 | 360 | -50 | DGPS | East |
| CR20-38 | 473900.0 | 5409893.2 | 278.0 | 327 | 360 | -50 | DGPS | East |
| CR20-39 | 473800.9 | 5410003.7 | 278.3 | 522 | 180 | -50 | DGPS | East |
| CR20-40 | 473597.5 | 5409994.9 | 278.0 | 378 | 180 | -50 | DGPS | East |
| CR20-41 | 473499.3 | 5409799.8 | 277.4 | 426 | 360 | -50 | DGPS | East |
| CR20-42 | 473769.8 | 5408708.5 | 274.0 | 405 | 215 | -80 | DGPS | Main |
| CR20-43 | 473743.4 | 5408666.4 | 273.8 | 402 | 215 | -80 | DGPS | Main |
| CR20-44 | 473802.3 | 5408751.9 | 274.1 | 402 | 215 | -80 | DGPS | Main |
| CR20-45 | 473613.4 | 5408678.7 | 273.6 | 408 | 35 | -80 | DGPS | Main |
| CR20-46 | 473520.7 | 5408720.6 | 273.7 | 411 | 215 | -80 | DGPS | Main |
| CR20-47 | 473404.9 | 5408742.4 | 273.9 | 402 | 215 | -80 | DGPS | Main |
| CR20-48 | 473378.5 | 5408693.3 | 273.5 | 402 | 215 | -80 | DGPS | Main |
| CR20-49 | 472855.7 | 5408922.1 | 270.9 | 402 | 215 | -80 | DGPS | Main |
| CR20-50 | 473559.2 | 5408586.4 | 273.1 | 402 | 35 | -80 | DGPS | Main |
| CR20-51 | 472755.5 | 5408985.4 | 270.4 | 405 | 215 | -80 | DGPS | Main |
| CR20-52 | 473173.1 | 5408816.6 | 273.5 | 402 | 211 | -80 | DGPS | Main |
| CR20-53 | 472794.1 | 5409028.6 | 270.9 | 402 | 212 | -80 | DGPS | Main |
| CR20-54 | 472882.9 | 5408963.3 | 270.9 | 402 | 218 | -80 | DGPS | Main |
| CR20-55 | 473410.6 | 5408633.4 | 273.4 | 240 | 35 | -82 | DGPS | Main |
| CR20-56 | 472757.0 | 5408987.4 | 270.7 | 585 | 305 | -50 | DGPS | Main |
| CR20-57 | 473165.5 | 5409105.4 | 274.8 | 174 | 216 | -50 | DGPS | Main |
| CR20-58 | 473770.4 | 5408711.1 | 273.9 | 177 | 215 | -80 | DGPS | Main |
| CR20-59 | 472937.3 | 5408865.9 | 271.4 | 513 | 35 | -50 | DGPS | Main |
| CR20-60 | 473505.2 | 5408694.5 | 273.6 | 402 | 215 | -50 | DGPS | Main |
| CR20-61 | 473030.0 | 5408816.0 | 272.0 | 504 | 35 | -50 | DGPS | Main |
| CR20-62 | 473505.6 | 5408695.0 | 273.6 | 402 | 215 | -80 | DGPS | Main |
| CR20-63 | 473783.9 | 5408614.1 | 273.4 | 402 | 35 | -80 | DGPS | Main |
| CR20-64 | 473923.9 | 5408650.8 | 273.9 | 402 | 35 | -80 | DGPS | Main |
| CR20-65 | 474015.0 | 5408693.2 | 274.2 | 402 | 215 | -65 | DGPS | Main |
| CR20-66 | 472801.1 | 5410498.8 | 272.1 | 402 | 270 | -50 | DGPS | West |
| CR20-67 | 472721.2 | 5410397.8 | 271.5 | 300 | 270 | -45 | DGPS | West |
| CR20-68 | 472720.6 | 5410397.9 | 271.6 | 402 | 270 | -60 | DGPS | West |
| CR20-69 | 470963.4 | 5409691.5 | 268.2 | 501 | 90 | -50 | DGPS | West |
| CR20-70 | 471414.3 | 5409822.4 | 267.6 | 156 | 270 | -50 | DGPS | West |
| CR20-70A | 471419.0 | 5409819.1 | 267.5 | 576 | 269 | -50 | DGPS | West |
| CR20-71 | 471419.6 | 5409820.8 | 267.6 | 594 | 314 | -53 | DGPS | West |
| CR20-72 | 471559.3 | 5409817.3 | 267.2 | 372 | 90 | -50 | DGPS | West |
| CR20-73 | 472725.0 | 5410298.8 | 271.4 | 525 | 270 | -45 | DGPS | West |
| CR20-74 | 471651.9 | 5409826.8 | 266.6 | 504 | 90 | -50 | DGPS | West |

| Drillhole | UTM (mE) | UTM (mN) | Elev (m) | Length (m) | Collar Az | Collar Dip | Survey | Zone |
|---|---|---|---|---|---|---|---|---|
| CR20-75 | 472613.8 | 5410299.2 | 271.0 | 450 | 270 | -50 | DGPS | West |
| CR20-76 | 471626.5 | 5409731.3 | 266.9 | 501 | 93 | -52 | DGPS | West |
| CR20-77 | 472699.3 | 5409150.1 | 270.5 | 405 | 223 | -66 | DGPS | Main |
| CR20-78 | 472696.6 | 5409147.4 | 270.3 | 593 | 52 | -51 | DGPS | Main |
| CR20-79 | 471558.3 | 5409599.3 | 267.9 | 395 | 225 | -52 | DGPS | West |
| CR20-80 | 471560.9 | 5409599.9 | 267.8 | 501 | 90 | -52 | DGPS | West |
| CR20-81 | 472669.5 | 5409248.0 | 270.4 | 501 | 226 | -50 | DGPS | Main |
| CR20-82 | 471557.3 | 5409400.5 | 268.6 | 422 | 45 | -50 | DGPS | West |
| CR20-83 | 471745.1 | 5409597.4 | 266.8 | 493 | 90 | -50 | DGPS | West |
| CR20-84 | 472549.8 | 5412165.5 | 276.9 | 502 | 180 | -65 | DGPS | North |
| CR20-85 | 471929.1 | 5409280.1 | 267.5 | 501 | 90 | -50 | DGPS | West |
| CR20-86 | 471927.5 | 5409282.5 | 267.4 | 553 | 45 | -50 | DGPS | West |
| CR20-87 | 472647.1 | 5412166.1 | 277.3 | 598 | 180 | -50 | DGPS | North |
| CR20-88 | 471927.7 | 5409275.4 | 267.3 | 531 | 135 | -50 | DGPS | West |
| CR20-89 | 472646.3 | 5412164.5 | 277.3 | 144 | 360 | -50 | DGPS | North |
| CR20-90 | 472454.6 | 5412256.3 | 276.8 | 540 | 180 | -50 | DGPS | North |
| CR21-100 | 473252.6 | 5408704.8 | 273.2 | 57 | 35 | -82 | DGPS | Main |
| CR21-100A | 473251.9 | 5408703.6 | 273.4 | 250 | 35 | -82 | DGPS | Main |
| CR21-101 | 472244.6 | 5410437.7 | 269.6 | 450 | 90 | -50 | DGPS | West |
| CR21-102 | 472242.8 | 5410639.0 | 270.3 | 432 | 90 | -50 | DGPS | West |
| CR21-103 | 473509.2 | 5408777.4 | 273.7 | 252 | 215 | -50 | DGPS | Main |
| CR21-104 | 472036.2 | 5409843.2 | 267.7 | 402 | 270 | -50 | DGPS | West |
| CR21-105 | 473680.8 | 5408465.9 | 273.0 | 150 | 35 | -50 | DGPS | Main |
| CR21-106 | 471994.3 | 5409655.6 | 267.9 | 67 | 270 | -50 | DGPS | West |
| CR21-106A | 471993.9 | 5409655.7 | 267.9 | 303 | 270 | -50 | DGPS | West |
| CR21-107 | 474005.5 | 5408767.1 | 274.5 | 201 | 215 | -50 | DGPS | Main |
| CR21-108 | 475303.5 | 5409721.0 | 274.7 | 501 | 360 | -50 | DGPS | East |
| CR21-109 | 472163.2 | 5409544.1 | 267.6 | 398 | 225 | -50 | DGPS | West |
| CR21-110 | 472337.7 | 5409412.6 | 268.8 | 426 | 225 | -50 | DGPS | West |
| CR21-111 | 473401.8 | 5409996.5 | 277.5 | 115 | 180 | -50 | DGPS | East |
| CR21-111A | 473401.8 | 5409997.1 | 277.6 | 444 | 180 | -50 | DGPS | East |
| CR21-112 | 475198.0 | 5409891.7 | 274.9 | 435 | 180 | -60 | DGPS | East |
| CR21-113 | 472570.3 | 5409234.4 | 269.6 | 423 | 225 | -50 | DGPS | Main |
| CR21-114 | 475001.7 | 5409895.5 | 275.8 | 227 | 180 | -60 | DGPS | East |
| CR21-114A | 475001.0 | 5409888.1 | 275.7 | 468 | 180 | -60 | DGPS | East |
| CR21-115 | 472590.5 | 5409191.3 | 269.8 | 450 | 225 | -50 | DGPS | Main |
| CR21-116 | 473998.6 | 5409996.9 | 277.7 | 510 | 180 | -50 | DGPS | East |
| CR21-117 | 472091.1 | 5409654.4 | 267.6 | 99 | 270 | -50 | DGPS | West |
| CR21-117A | 472091.7 | 5409654.3 | 267.2 | 153 | 270 | -50 | DGPS | West |
| CR21-117B | 472091.5 | 5409654.3 | 267.1 | 97 | 270 | -55 | DGPS | West |
| CR21-117C | 472096.3 | 5409656.4 | 267.6 | 33 | 270 | -55 | DGPS | West |
| CR21-118 | 474795.1 | 5409960.8 | 276.1 | 504 | 180 | -60 | DGPS | East |
| CR21-119 | 474298.8 | 5409994.7 | 276.2 | 479 | 180 | -60 | DGPS | East |
| CR21-120 | 472135.3 | 5410035.8 | 268.7 | 447 | 270 | -50 | DGPS | West |
| CR21-121 | 474704.2 | 5409977.3 | 276.2 | 507 | 180 | -60 | DGPS | East |
| CR21-122 | 472087.8 | 5410242.3 | 268.9 | 501 | 270 | -50 | DGPS | West |
| CR21-123 | 474503.2 | 5409982.6 | 276.2 | 507 | 180 | -60 | DGPS | East |
| CR21-124 | 472040.2 | 5410237.8 | 268.6 | 557 | 90 | -50 | DGPS | West |
| CR21-125 | 474605.2 | 5409698.9 | 276.3 | 376 | 360 | -60 | DGPS | East |
| CR21-126 | 472021.5 | 5410135.3 | 268.3 | 169 | 270 | -50 | DGPS | West |
| CR21-126A | 472021.1 | 5410135.4 | 268.5 | 546 | 270 | -55 | DGPS | West |
| CR21-127 | 474401.9 | 5409726.2 | 276.7 | 411 | 360 | -50 | DGPS | East |
| CR21-128 | 472036.3 | 5410034.7 | 268.1 | 549 | 270 | -50 | DGPS | West |
| CR21-129 | 474201.0 | 5409752.6 | 276.9 | 63 | 360 | -50 | DGPS | East |
| CR21-129A | 474200.9 | 5409753.0 | 277.1 | 411 | 360 | -50 | DGPS | East |
| CR21-130 | 472058.8 | 5409952.6 | 267.7 | 561 | 270 | -50 | DGPS | West |
| CR21-131 | 474000.0 | 5409818.0 | 277.3 | 402 | 5 | -65 | DGPS | East |
| CR21-132 | 472292.0 | 5409489.5 | 268.5 | 489 | 225 | -50 | DGPS | West |
| CR21-133 | 473910.4 | 5409793.3 | 277.8 | 372 | 355 | -60 | DGPS | East |
| CR21-134 | 473803.8 | 5409799.0 | 278.8 | 51 | 5 | -60 | DGPS | East |
| CR21-134A | 473803.8 | 5409799.3 | 278.6 | 351 | 5 | -60 | DGPS | East |
| CR21-135 | 472458.8 | 5409431.5 | 269.4 | 564 | 225 | -50 | DGPS | West |
| CR21-136 | 473617.3 | 5409779.7 | 277.6 | 375 | 355 | -55 | DGPS | East |
| CR21-137 | 472468.4 | 5409292.8 | 269.2 | 468 | 225 | -55 | DGPS | West |
| CR21-138 | 473359.1 | 5409778.2 | 277.1 | 420 | 360 | -60 | DGPS | East |
| CR21-139 | 472128.4 | 5409630.9 | 267.9 | 466 | 235 | -55 | DGPS | West |
| CR21-140 | 473359.8 | 5409776.6 | 277.2 | 267 | 245 | -60 | DGPS | East |
| CR21-141 | 472115.5 | 5409615.7 | 268.0 | 588 | 60 | -55 | DGPS | West |
| CR21-142 | 473557.4 | 5409893.2 | 277.5 | 69 | 178 | -86 | DGPS | East |
| CR21-142A | 473557.4 | 5409893.2 | 277.5 | 624 | 178 | -86 | DGPS | East |
| CR21-143 | 473502.2 | 5410014.1 | 277.8 | 501 | 180 | -50 | DGPS | East |
| CR21-144 | 472053.7 | 5409462.9 | 267.4 | 369 | 60 | -65 | DGPS | West |
| CR21-145 | 472050.9 | 5409461.6 | 267.2 | 675 | 235 | -55 | DGPS | West |
| CR21-146 | 473290.8 | 5410031.5 | 276.7 | 525 | 180 | -50 | DGPS | East |
| CR21-147 | 473972.0 | 5409986.5 | 277.9 | 351 | 185 | -50 | DGPS | East |
| CR21-148 | 472086.5 | 5409311.8 | 268.6 | 552 | 50 | -55 | DGPS | West |
| CR21-149 | 473853.6 | 5409880.3 | 278.0 | 474 | 178 | -87 | DGPS | East |
| CR21-150 | 472170.6 | 5409378.1 | 267.8 | 528 | 50 | -50 | DGPS | West |
| CR21-151 | 474125.0 | 5409970.0 | 277.1 | 309 | 185 | -50 | DGPS | East |

| Drillhole | UTM (mE) | UTM (mN) | Elev (m) | Length (m) | Collar Az | Collar Dip | Survey | Zone |
|---|---|---|---|---|---|---|---|---|
| CR21-152 | 472419.0 | 5409657.9 | 269.5 | 536 | 235 | -55 | DGPS | West |
| CR21-153 | 473763.5 | 5409806.7 | 278.1 | 669 | 360 | -86 | DGPS | East |
| CR21-154 | 472408.7 | 5409162.2 | 269.1 | 636 | 45 | -55 | DGPS | West |
| CR21-155 | 472834.0 | 5408828.8 | 271.0 | 579 | 35 | -62 | DGPS | Main |
| CR21-156 | 473669.7 | 5409808.9 | 277.8 | 658 | 0 | -86 | DGPS | East |
| CR21-157 | 472290.7 | 5409057.0 | 268.0 | 549 | 45 | -55 | DGPS | West |
| CR21-158 | 472222.1 | 5408975.3 | 267.8 | 699 | 45 | -55 | DGPS | West |
| CR21-159 | 473001.5 | 5408960.4 | 272.7 | 366 | 215 | -55 | DGPS | Main |
| CR21-160 | 474397.3 | 5409939.4 | 276.3 | 549 | 180 | -70 | DGPS | East |
| CR21-161 | 473928.9 | 5409975.9 | 278.0 | 231 | 175 | -79 | DGPS | East |
| CR21-161A | 473928.9 | 5409974.3 | 278.0 | 432 | 180 | -77 | DGPS | East |
| CR21-161B | 473929.0 | 5409975.4 | 277.8 | 30 | 180 | -77 | DGPS | East |
| CR21-161C | 473928.9 | 5409974.0 | 277.8 | 126 | 80 | -77 | DGPS | East |
| CR21-162 | 473001.9 | 5408960.9 | 272.7 | 396 | 215 | -75 | DGPS | Main |
| CR21-163 | 471893.3 | 5409795.7 | 267.3 | 675 | 81 | -52 | DGPS | West |
| CR21-164 | 474599.8 | 5409830.2 | 276.5 | 327 | 0 | -70 | DGPS | East |
| CR21-165 | 473801.3 | 5409897.2 | 278.6 | 555 | 180 | -86 | DGPS | East |
| CR21-165A | 473801.3 | 5409895.4 | 278.5 | 735 | 185 | -84 | DGPS | East |
| CR21-166 | 471877.8 | 5409937.8 | 266.7 | 583 | 265 | -50 | DGPS | West |
| CR21-167 | 473243.6 | 5408576.6 | 273.5 | 207 | 35 | -60 | DGPS | Main |
| CR21-168 | 474703.5 | 5409798.0 | 276.3 | 339 | 360 | -70 | DGPS | East |
| CR21-168A | 474703.7 | 5409798.3 | 276.5 | 420 | 360 | -70 | DGPS | East |
| CR21-169 | 471863.2 | 5410134.8 | 266.9 | 615 | 300 | -50 | DGPS | West |
| CR21-170 | 471916.4 | 5410585.1 | 268.1 | 531 | 315 | -50 | DGPS | West |
| CR21-171 | 471912.8 | 5410587.2 | 268.1 | 639 | 90 | -50 | DGPS | West |
| CR21-172 | 474796.3 | 5409779.1 | 276.3 | 681 | 360 | -75 | DGPS | East |
| CR21-173 | 474100.9 | 5409798.0 | 277.4 | 312 | 360 | -50 | DGPS | East |
| CR21-174 | 474856.8 | 5409880.4 | 276.1 | 630 | 168 | -85 | DGPS | East |
| CR21-175 | 473898.7 | 5409893.4 | 278.9 | 234 | 360 | -50 | DGPS | East |
| CR21-176 | 473421.1 | 5408875.0 | 274.2 | 741 | 210 | -70 | DGPS | Main |
| CR21-177 | 475043.8 | 5409733.6 | 275.8 | 567 | 360 | -70 | DGPS | East |
| CR21-178 | 473275.0 | 5408835.1 | 274.2 | 504 | 220 | -78 | DGPS | Main |
| CR21-179 | 475132.5 | 5409872.1 | 275.1 | 416 | 180 | -85 | DGPS | East |
| CR21-180 | 475174.8 | 5409714.6 | 275.6 | 501 | 10 | -65 | DGPS | East |
| CR21-181 | 472948.1 | 5408803.4 | 271.3 | 606 | 35 | -65 | DGPS | Main |
| CR21-182 | 475434.8 | 5409933.5 | 272.5 | 489 | 180 | -55 | DGPS | East |
| CR21-183 | 472916.1 | 5409095.8 | 271.3 | 596 | 220 | -75 | DGPS | Main |
| CR21-184 | 474735.7 | 5409651.3 | 276.4 | 444 | 360 | -50 | DGPS | East |
| CR21-185 | 475548.0 | 5409925.9 | 272.4 | 552 | 180 | -64 | DGPS | East |
| CR21-186 | 472848.8 | 5409184.6 | 271.0 | 621 | 215 | -72 | DGPS | Main |
| CR21-187 | 476246.1 | 5409856.8 | 272.0 | 327 | 180 | -50 | DGPS | East |
| CR21-188 | 475749.8 | 5409711.6 | 272.4 | 258 | 360 | -62 | DGPS | West |
| CR21-189 | 472735.1 | 5409186.0 | 270.4 | 663 | 220 | -76 | DGPS | Main |
| CR21-91 | 472749.5 | 5411973.5 | 277.4 | 366 | 180 | -50 | DGPS | North |
| CR21-92 | 472756.1 | 5409078.1 | 270.7 | 600 | 225 | -80 | DGPS | Main |
| CR21-93 | 472749.9 | 5411977.3 | 277.5 | 447 | 360 | -50 | DGPS | North |
| CR21-94 | 472756.1 | 5409078.0 | 270.6 | 402 | 225 | -50 | DGPS | Main |
| CR21-95 | 472854.0 | 5411846.2 | 277.7 | 444 | 360 | -50 | DGPS | North |
| CR21-96 | 472441.2 | 5410437.2 | 270.2 | 312 | 90 | -50 | DGPS | West |
| CR21-97 | 472795.0 | 5409030.1 | 270.7 | 240 | 215 | -80 | DGPS | Main |
| CR21-98 | 472935.9 | 5408866.4 | 271.3 | 162 | 35 | -50 | DGPS | Main |
| CR21-99 | 472442.0 | 5410437.3 | 270.3 | 417 | 90 | -82 | DGPS | West |
| CR21-GT-01 | 473622.8 | 5409471.0 | 272.5 | 297 | 230 | -55 | APS | East |
| CR21-GT-01A | 473622.4 | 5409472.5 | 277.2 | 115 | 230 | -65 | DGPS | East |
| CR21-GT-01B | 473619.1 | 5409473.6 | 276.8 | 0 | 230 | -60 | DGPS | East |
| CR21-GT-02 | 474109.8 | 5409804.8 | 277.4 | 626 | 170 | -60 | DGPS | East |
| CR21-GT-03 | 473686.0 | 5409746.3 | 278.2 | 618 | 230 | -55 | DGPS | East |
| CR21-GT-04 | 473479.7 | 5409917.7 | 277.9 | 113 | 325 | -65 | DGPS | East |
| CR21-GT-04A | 473480.3 | 5409916.9 | 278.0 | 396 | 325 | -65 | DGPS | East |
| CR21-GT-04B | 473482.1 | 5409914.6 | 277.7 | 600 | 325 | -65 | DGPS | East |
| CR21-GT-05 | 474125.0 | 5410104.5 | 276.7 | 450 | 30 | -65 | DGPS | East |
| CR21-GT-06 | 474671.1 | 5409957.6 | 276.2 | 573 | 10 | -65 | DGPS | East |
| CR21-GT-07 | 475160.8 | 5409909.6 | 275.0 | 525 | 45 | -65 | DGPS | East |
| CR21-GT-08 | 475043.0 | 5409670.1 | 275.9 | 525 | 150 | -65 | DGPS | East |
| CR21-GT-09 | 474428.3 | 5409368.8 | 276.7 | 40 | 200 | -65 | DGPS | East |
| CR21-GT-09A | 474428.3 | 5409369.4 | 276.7 | 170 | 200 | -65 | DGPS | East |
| CR21-GT-09B | 474427.8 | 5409368.9 | 276.6 | 525 | 200 | -65 | DGPS | East |
| CR22-190 | 475658.0 | 5409902.0 | 272.4 | 402 | 180 | -62 | DGPS | East |
| CR22-191 | 473107.9 | 5408574.0 | 273.5 | 675 | 30 | -50 | DGPS | Main |
| CR22-192 | 475482.8 | 5409707.4 | 272.7 | 491 | 360 | -55 | DGPS | East |
| CR22-193 | 473191.4 | 5409930.0 | 274.8 | 52 | 110 | -70 | DGPS | East |
| CR22-193A | 473191.0 | 5409930.2 | 274.9 | 663 | 110 | -70 | DGPS | East |
| CR22-194 | 473444.9 | 5408775.4 | 274.0 | 777 | 203 | -76 | DGPS | Main |
| CR22-195 | 475598.7 | 5409718.1 | 272.5 | 423 | 0 | -55 | DGPS | East |
| CR22-196 | 473280.8 | 5409952.3 | 277.7 | 78 | 118 | -70 | DGPS | East |
| CR22-196A | 473278.9 | 5409954.3 | 277.6 | 60 | 120 | -68 | DGPS | East |
| CR22-196B | 473281.5 | 5409951.2 | 277.7 | 762 | 120 | -68 | DGPS | East |
| CR22-197 | 474069.8 | 5408499.7 | 274.4 | 384 | 35 | -50 | DGPS | Main |
| CR22-198 | 473561.5 | 5408735.5 | 274.3 | 1044 | 205 | -75 | DGPS | Main |

| Drillhole | UTM (mE) | UTM (mN) | Elev (m) | Length (m) | Collar Az | Collar Dip | Survey | Zone |
|---|---|---|---|---|---|---|---|---|
| CR22-199 | 473279.9 | 5409948.4 | 277.2 | 276 | 310 | -60 | DGPS | East |
| CR22-200 | 472642.7 | 5410214.3 | 271.8 | 633 | 230 | -50 | DGPS | West |
| CR22-201 | 475295.2 | 5409910.9 | 275.6 | 435 | 230 | -70 | DGPS | East |
| CR22-202 | 473860.5 | 5408808.7 | 274.8 | 675 | 205 | -68 | DGPS | Main |
| CR22-203 | 475437.7 | 5409887.8 | 273.0 | 54 | 245 | -70 | DGPS | East |
| CR22-203A | 475437.7 | 5409887.7 | 273.1 | 33 | 238 | -69 | DGPS | East |
| CR22-203B | 475437.7 | 5409887.7 | 273.1 | 477 | 238 | -68 | DGPS | East |
| CR22-203BW | 475437.8 | 5409887.9 | 273.1 | 456 | 238 | -69 | DGPS | East |
| CR22-204 | 474181.0 | 5408472.8 | 275.5 | 369 | 35 | -60 | DGPS | Main |
| CR22-205 | 473769.6 | 5408709.2 | 274.4 | 354 | 215 | -80 | DGPS | Main |
| CR22-206 | 473853.5 | 5409880.7 | 278.5 | 480 | 205 | -88 | DGPS | East |
| CR22-207 | 473410.7 | 5408633.3 | 273.4 | 351 | 35 | -82 | DGPS | Main |
| CR22-208 | 473376.5 | 5408691.5 | 273.6 | 249 | 215 | -80 | DGPS | Main |
| CR22-209 | 472547.2 | 5412071.4 | 276.6 | 435 | 190 | -55 | DGPS | North |
| CR22-210 | 473562.2 | 5408584.6 | 273.1 | 369 | 35 | -80 | DGPS | Main |
| CR22-211 | 473283.4 | 5409946.7 | 276.9 | 80 | 120 | -68 | DGPS | East |
| CR22-211A | 473283.1 | 5409946.2 | 276.9 | 48 | 120 | -68 | DGPS | East |
| CR22-211B | 473283.4 | 5409946.7 | 276.9 | 402 | 117 | -68 | DGPS | East |
| CR22-212 | 473803.6 | 5408753.4 | 274.2 | 240 | 215 | -80 | DGPS | Main |
| CR22-213 | 472750.9 | 5412123.4 | 277.0 | 540 | 180 | -55 | DGPS | North |
| CR22-214 | 472546.9 | 5412070.6 | 276.8 | 378 | 5 | -65 | DGPS | North |
| CR22-215 | 474007.4 | 5408765.3 | 274.5 | 351 | 215 | -50 | DGPS | Main |
| CR22-216 | 472423.0 | 5412084.0 | 275.0 | 486 | 320 | -50 | APS | North |
| CR22-217 | 472636.3 | 5410210.4 | 271.1 | 594 | 273 | -70 | DGPS | West |
| CR22-218 | 472966.0 | 5411934.0 | 279.0 | 453 | 5 | -60 | APS | North |
| CR22-219 | 473825.8 | 5408596.1 | 273.7 | 237 | 35 | -82 | DGPS | Main |
| CR22-220 | 472423.0 | 5412084.0 | 275.0 | 450 | 0 | -55 | APS | North |
| CR22-221 | 473826.9 | 5408595.8 | 273.5 | 237 | 32 | -82 | DGPS | Main |
| CR22-222 | 473826.9 | 5409967.0 | 278.1 | 48 | 230 | -70 | DGPS | East |
| CR22-222A | 473826.9 | 5409967.0 | 278.1 | 69 | 230 | -70 | DGPS | East |
| CR22-222B | 473826.9 | 5409966.9 | 278.1 | 34 | 230 | -70 | DGPS | East |
| CR22-222C | 473826.9 | 5409966.9 | 278.1 | 456 | 230 | -70 | DGPS | East |
| CR22-223 | 473162.0 | 5411935.0 | 280.0 | 450 | 5 | -55 | APS | North |
| CR22-224 | 473783.8 | 5408610.5 | 273.3 | 252 | 40 | -80 | DGPS | Main |
| CR22-225 | 473555.3 | 5411647.8 | 279.0 | 348 | 30 | -55 | APS | North |
| CR22-226 | 473590.3 | 5408629.9 | 273.1 | 264 | 35 | -82 | DGPS | Main |
| CR22-226A | 473589.9 | 5408629.7 | 273.3 | 0 | 35 | -82 | DGPS | Main |
| CR22-227 | 473505.1 | 5408688.0 | 273.4 | 200 | 215 | -50 | DGPS | Main |
| CR22-228 | 472635.6 | 5410210.4 | 271.1 | 393 | 273 | -52 | DGPS | West |
| CR22-229 | 473681.4 | 5408648.2 | 273.3 | 204 | 218 | -69 | DGPS | Main |
| CR22-230 | 473822.4 | 5409953.4 | 278.3 | 1155 | 230 | -70 | DGPS | East |
| CR22-231 | 473309.1 | 5409051.0 | 274.7 | 171 | 210 | -70 | DGPS | Main |
| CR22-232 | 473308.6 | 5409051.3 | 274.8 | 132 | 210 | -50 | DGPS | Main |
| CR22-233 | 473265.7 | 5409257.6 | 274.2 | 291 | 60 | -50 | DGPS | Main |
| CR22-234 | 472740.9 | 5410477.5 | 271.6 | 402 | 336 | -68 | DGPS | West |
| CR22-235 | 473264.8 | 5409260.2 | 274.4 | 363 | 210 | -50 | DGPS | Main |
| CR22-236 | 472740.8 | 5410477.9 | 271.5 | 336 | 336 | -58 | DGPS | West |
| CR22-237 | 472707.6 | 5410331.1 | 271.5 | 432 | 275 | -55 | DGPS | West |
| CR22-238 | 473194.3 | 5409145.3 | 274.6 | 213 | 210 | -65 | DGPS | Main |
| CR22-239 | 472939.3 | 5409341.9 | 272.1 | 150 | 228 | -50 | DGPS | Main |
| CR22-240 | 472938.8 | 5409341.5 | 272.2 | 243 | 228 | -65 | DGPS | Main |
| CR22-241 | 473049.1 | 5409213.7 | 272.8 | 252 | 212 | -75 | DGPS | Main |
| CR22-242 | 473048.8 | 5409213.2 | 272.9 | 219 | 212 | -60 | DGPS | Main |
| CR22-243 | 473332.7 | 5408848.6 | 274.6 | 523 | 212 | -45 | DGPS | Main |
| CR22-244 | 473686.1 | 5408682.4 | 273.5 | 384 | 328 | -45 | DGPS | Main |
| CR22-245 | 474869.2 | 5409812.8 | 275.9 | 144 | 205 | -60 | DGPS | East |
| CR22-245A | 474869.2 | 5409812.8 | 275.9 | 300 | 205 | -60 | DGPS | East |
| CR22-246 | 473855.2 | 5408642.6 | 274.2 | 387 | 28 | -55 | DGPS | Main |
| CR22-247 | 473056.3 | 5408823.3 | 272.5 | 390 | 233 | -55 | DGPS | Main |
| CR22-248 | 474655.0 | 5409841.9 | 277.1 | 312 | 178 | -60 | DGPS | East |
| CR22-249 | 474324.2 | 5409878.7 | 276.7 | 471 | 228 | -64 | DGPS | East |
| CR22-250 | 474206.9 | 5410042.9 | 276.5 | 645 | 195 | -62 | DGPS | East |
| CR22-251 | 475411.0 | 5409808.8 | 273.4 | 261 | 208 | -50 | DGPS | East |
| CR22-252 | 473872.2 | 5408711.1 | 274.2 | 282 | 345 | -50 | DGPS | Main |
| CR22-253 | 473872.4 | 5408710.5 | 274.5 | 381 | 345 | -65 | DGPS | Main |
| CR22-254 | 473872.5 | 5408710.7 | 274.3 | 399 | 38 | -55 | DGPS | Main |
| CR22-255 | 473450.4 | 5408809.2 | 273.8 | 405 | 15 | -65 | DGPS | Main |
| CR22-256 | 473450.1 | 5408808.4 | 273.8 | 408 | 15 | -45 | DGPS | Main |
| CR22-257 | 473644.2 | 5408750.8 | 273.7 | 354 | 355 | -45 | DGPS | Main |
| CR22-258 | 474023.3 | 5408798.1 | 274.5 | 132 | 240 | -84 | DGPS | Main |
| CR22-259 | 474023.7 | 5408797.2 | 274.4 | 132 | 240 | -84 | DGPS | Main |
| CR22-260 | 473725.1 | 5408732.1 | 273.6 | 132 | 274 | -86 | DGPS | Main |
| CR22-261 | 474245.7 | 5409992.9 | 276.6 | 477 | 180 | -75 | DGPS | East |
| CR22-262 | 474247.4 | 5409992.7 | 276.4 | 52 | 180 | -75 | DGPS | East |
| CR22-262A | 474247.4 | 5409992.7 | 276.4 | 30 | 180 | -75 | DGPS | East |
| CR22-262B | 474247.4 | 5409992.7 | 276.4 | 33 | 180 | -75 | DGPS | East |
| CR22-262C | 474247.4 | 5409992.7 | 276.4 | 651 | 180 | -74 | DGPS | East |
| CR22-263 | 474099.6 | 5410014.7 | 277.2 | 507 | 180 | -75 | DGPS | East |
| CR22-263A | 474098.8 | 5410014.9 | 277.2 | 663 | 180 | -75 | DGPS | East |

| Drillhole | UTM (mE) | UTM (mN) | Elev (m) | Length (m) | Collar Az | Collar Dip | Survey | Zone |
|---|---|---|---|---|---|---|---|---|
| CR22-264 | 473434.0 | 5408944.9 | 274.3 | 506 | 215 | -82 | DGPS | Main |
| CR22-264A | 473435.3 | 5408946.4 | 274.0 | 615 | 215 | -82 | DGPS | Main |
| CR22-265 | 474403.4 | 5410013.5 | 276.0 | 577 | 180 | -75 | DGPS | East |
| CR22-266 | 473270.3 | 5408528.5 | 273.4 | 651 | 18 | -84 | DGPS | West |
| CR22-267 | 474606.3 | 5410009.0 | 276.2 | 650 | 180 | -75 | DGPS | East |
| CR22-268 | 473590.2 | 5408486.5 | 272.8 | 703 | 25 | -84 | DGPS | Main |
| CR22-269 | 474499.4 | 5409702.1 | 276.3 | 520 | 0 | -75 | DGPS | East |
| CR22-270 | 473717.4 | 5408484.5 | 273.2 | 417 | 10 | -84 | DGPS | Main |
| CR22-271 | 475009.1 | 5409900.5 | 275.8 | 598 | 180 | -74 | DGPS | East |
| CR22-272 | 473764.0 | 5408865.1 | 274.4 | 651 | 172 | -82 | DGPS | Main |
| CR22-273 | 474787.2 | 5409837.9 | 276.0 | 630 | 180 | -90 | DGPS | East |
| CR22-274 | 475529.0 | 5409916.9 | 272.6 | 651 | 234 | -68 | DGPS | East |
| CR22-275 | 473727.2 | 5408517.4 | 273.4 | 693 | 10 | -87 | DGPS | Main |
| CR22-276 | 475529.3 | 5409916.8 | 272.6 | 453 | 220 | -74 | DGPS | East |
| CR22-277 | 474086.0 | 5408773.6 | 274.8 | 597 | 190 | -77 | DGPS | Main |
| CR22-278 | 473929.9 | 5408843.9 | 274.7 | 651 | 142 | -82 | DGPS | Main |
| CR22-279 | 474264.1 | 5408642.4 | 276.1 | 247 | 225 | -82 | DGPS | Main |
| CR22-280 | 472833.8 | 5409451.4 | 271.2 | 180 | 220 | -55 | DGPS | Main |
| CR22-281 | 474302.5 | 5409955.6 | 276.3 | 252 | 2 | -50 | DGPS | East |
| CR22-282 | 474302.6 | 5409954.7 | 276.4 | 222 | 2 | -75 | DGPS | East |
| CR22-283 | 474504.0 | 5409938.8 | 276.1 | 222 | 2 | -75 | DGPS | East |
| CR22-284 | 474503.9 | 5409939.7 | 276.1 | 261 | 2 | -50 | DGPS | East |
| CR22-285 | 474705.0 | 5409932.1 | 276.4 | 252 | 2 | -50 | DGPS | East |
| CR22-286 | 474704.9 | 5409931.6 | 276.3 | 180 | 2 | -70 | DGPS | East |
| CR22-287 | 474805.9 | 5409922.5 | 276.2 | 252 | 4 | -50 | DGPS | East |
| CR22-288 | 474805.0 | 5409925.0 | 275.0 | 171 | 2 | -70 | DGPS | East |
| CR22-289 | 475010.5 | 5409903.5 | 275.8 | 252 | 10 | -50 | DGPS | East |
| CR22-290 | 475010.4 | 5409902.9 | 275.7 | 180 | 10 | -70 | DGPS | East |
| CR22-291 | 475204.0 | 5409922.2 | 275.0 | 210 | 10 | -50 | DGPS | East |
| CR22-292 | 475204.2 | 5409922.6 | 275.3 | 171 | 10 | -70 | DGPS | East |
| CR22-293 | 475434.4 | 5409927.6 | 272.9 | 201 | 0 | -50 | DGPS | East |
| CR22-294 | 475434.4 | 5409926.6 | 272.6 | 192 | 0 | -70 | DGPS | East |
| CR22-295 | 473373.9 | 5409984.2 | 277.4 | 282 | 2 | -50 | DGPS | East |
| CR22-296 | 473373.7 | 5409983.6 | 277.4 | 201 | 2 | -70 | DGPS | East |
| CR22-297 | 473202.1 | 5409992.6 | 276.1 | 210 | 2 | -60 | DGPS | East |
| CR22-298 | 473564.1 | 5410008.5 | 278.0 | 252 | 0 | -55 | DGPS | East |
| CR22-299 | 473564.1 | 5410007.9 | 278.1 | 201 | 0 | -75 | DGPS | East |
| CR22-300 | 473564.2 | 5410008.3 | 277.9 | 300 | 325 | -56 | DGPS | East |
| CR22-301 | 473715.8 | 5409926.1 | 278.1 | 252 | 350 | -50 | DGPS | East |
| CR22-302 | 474050.5 | 5409928.2 | 277.6 | 300 | 2 | -50 | DGPS | East |
| CR22-303 | 474050.1 | 5409927.9 | 277.6 | 264 | 320 | -65 | DGPS | East |
| CR22-304 | 474051.4 | 5409928.9 | 277.5 | 252 | 40 | -66 | DGPS | East |
| CR22-305 | 473798.0 | 5410003.2 | 278.2 | 228 | 5 | -55 | DGPS | East |
| CR22-GT-01C | 473620.3 | 5409479.2 | 278.0 | 356 | 230 | -60 | DGPS | East |
| CR22-GT-10 | 473629.4 | 5408547.2 | 273.8 | 575 | 150 | -70 | DGPS | Main |
| CR22-GT-11 | 472890.8 | 5408564.9 | 271.9 | 471 | 220 | -65 | DGPS | Main |
| CR22-GT-12 | 473534.5 | 5408264.4 | 271.8 | 0 | 180 | -65 | DGPS | Main |
| CR22-GT-12A | 473534.4 | 5408264.6 | 272.5 | 450 | 180 | -65 | DGPS | Main |
| CR22-GT-13 | 473001.2 | 5409139.9 | 273.1 | 501 | 20 | -60 | DGPS | Main |
| CR22-GT-14 | 473894.4 | 5408840.6 | 275.0 | 550 | 50 | -55 | DGPS | Main |
| CR22-GT-15 | 473496.3 | 5408941.0 | 275.2 | 600 | 350 | -65 | DGPS | Main |
| CR22-GT-16 | 472947.8 | 5408939.2 | 271.7 | 576 | 250 | -60 | DGPS | Main |
| CR22-GT-17 | 474047.5 | 5408661.5 | 274.8 | 552 | 100 | -65 | DGPS | Main |
| CR22-GT-18 | 472256.3 | 5409304.0 | 268.3 | 399 | 230 | -60 | DGPS | West |
| CR22-GT-19 | 472416.3 | 5409431.8 | 269.2 | 351 | 45 | -65 | DGPS | West |
| CR22-GT-20 | 471949.7 | 5409976.2 | 267.3 | 327 | 40 | -65 | DGPS | West |
| CR22-GT-21 | 471936.3 | 5409590.7 | 267.5 | 426 | 220 | -60 | DGPS | West |
| CR22-GT-22 | 475558.6 | 5409652.3 | 272.5 | 594 | 220 | -60 | DGPS | East |
| | | | Total: | | 151,969 | | | |

Notes: "GT" drillholes are geotechnical holes used for lithology in the modelling but do not have any assays.

**Table 10-2: Main Zone – Selected Drill Core Assays, CR19, CR20, CR21 and CR22 Series Diamond Drillholes**

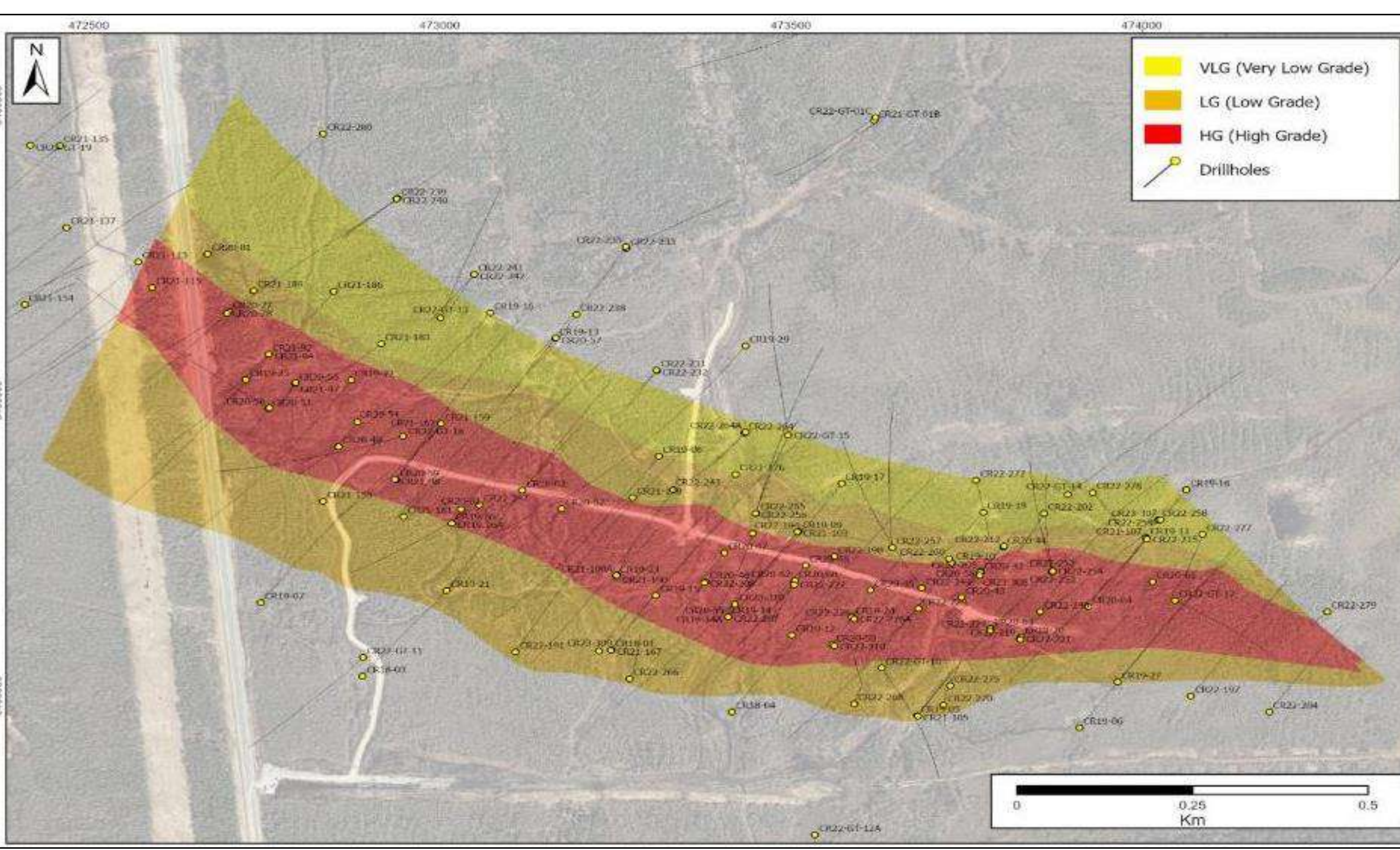
| Drillhole | From (m) | To (m) | Interval (m) | Estimated True Width (m) | Ni (%) | Co (%) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | S (%) | Fe (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| CR19-05 | 51.6 | 582.0 | 530.4 | 344.8 | 0.227 | 0.013 | 0.018 | 0.008 | 0.026 | 0.13 | - |
| *incl.* | 432.0 | 481.5 | 49.5 | 32.2 | 0.310 | 0.015 | 0.037 | 0.010 | 0.047 | 0.53 | - |
| *incl.* | 445.5 | 472.5 | 27.0 | 17.6 | 0.359 | 0.018 | 0.051 | 0.015 | 0.066 | 0.78 | - |
| CR19-06 | 207.0 | 576.0 | 369.0 | 239.9 | 0.229 | 0.011 | 0.005 | 0.004 | 0.009 | 0.04 | - |
| *incl.* | 304.5 | 453.0 | 148.5 | 96.5 | 0.275 | 0.012 | 0.001 | 0.001 | 0.002 | 0.03 | - |
| CR19-07 | 204.0 | 619.5 | 415.5 | 270.1 | 0.221 | 0.013 | 0.007 | 0.008 | 0.015 | 0.01 | - |
| *incl.* | 591.0 | 619.5 | 28.5 | 18.5 | 0.265 | 0.013 | 0.092 | 0.034 | 0.126 | 0.04 | - |
| CR19-08 | 36.0 | 592.5 | 556.5 | 361.7 | 0.251 | 0.013 | 0.020 | 0.011 | 0.031 | 0.06 | - |
| *incl.* | 70.5 | 468.0 | 397.5 | 258.4 | 0.271 | 0.013 | 0.018 | 0.011 | 0.029 | 0.06 | - |
| *incl.* | 160.5 | 363.0 | 202.5 | 131.6 | 0.314 | 0.012 | 0.032 | 0.016 | 0.048 | 0.10 | - |
| *incl.* | 183.0 | 223.5 | 40.5 | 26.3 | 0.351 | 0.013 | 0.026 | 0.013 | 0.039 | 0.18 | - |
| CR19-09 | 55.5 | 513.0 | 457.5 | 297.4 | 0.254 | 0.013 | 0.020 | 0.010 | 0.030 | 0.08 | - |
| *incl.* | 63.0 | 436.5 | 373.5 | 242.8 | 0.270 | 0.013 | 0.023 | 0.012 | 0.035 | 0.09 | - |
| *incl.* | 70.5 | 309.0 | 238.5 | 155.0 | 0.310 | 0.013 | 0.027 | 0.013 | 0.040 | 0.11 | - |
| *incl.* | 192.0 | 265.5 | 73.5 | 47.8 | 0.365 | 0.014 | 0.047 | 0.011 | 0.058 | 0.17 | - |
| CR19-10 | 55.5 | 388.5 | 333.0 | 216.5 | 0.277 | 0.013 | 0.026 | 0.010 | 0.036 | 0.34 | - |
| *incl.* | 57.0 | 271.5 | 214.5 | 139.4 | 0.320 | 0.013 | 0.030 | 0.011 | 0.041 | 0.48 | - |
| *incl.* | 208.5 | 243.0 | 34.5 | 22.4 | 0.355 | 0.015 | 0.037 | 0.013 | 0.050 | 1.18 | - |
| CR19-11 | 48.0 | 438.0 | 390.0 | 253.5 | 0.271 | 0.014 | 0.028 | 0.011 | 0.040 | 0.19 | - |
| *incl.* | 48.0 | 307.5 | 259.5 | 168.7 | 0.310 | 0.015 | 0.038 | 0.015 | 0.053 | 0.25 | - |
| *incl.* | 133.5 | 277.5 | 144.0 | 93.6 | 0.353 | 0.015 | 0.060 | 0.023 | 0.082 | 0.32 | - |
| CR19-12 | 57.0 | 571.5 | 514.5 | 334.4 | 0.210 | 0.013 | 0.017 | 0.017 | 0.034 | 0.06 | - |
| *incl.* | 57.0 | 337.5 | 280.5 | 182.3 | 0.281 | 0.012 | 0.011 | 0.003 | 0.014 | 0.07 | - |
| *incl.* | 61.5 | 157.5 | 96.0 | 62.4 | 0.310 | 0.013 | 0.017 | 0.005 | 0.022 | 0.16 | - |
| *incl.* | 72.0 | 91.5 | 19.5 | 12.7 | 0.353 | 0.014 | 0.018 | 0.004 | 0.022 | 0.23 | - |
| *incl.* | 507.0 | 520.5 | 13.5 | 8.8 | 0.060 | 0.013 | 0.300 | 0.500 | 0.800 | 0.04 | - |
| *incl.* | 508.5 | 510.0 | 1.5 | 1.0 | 0.060 | 0.014 | 0.700 | 1.600 | 2.300 | 0.02 | - |
| *incl.* | 517.5 | 520.5 | 3.0 | 2.0 | 0.050 | 0.01 | 0.700 | 0.700 | 1.400 | 0.05 | - |
| CR19-13 | 78.0 | 85.5 | 7.5 | - | 0.050 | 0.12 | 0.700 | 1.000 | 1.700 | 0.03 | - |
| and | 102.0 | 609.0 | 507.0 | 329.6 | 0.237 | 0.013 | 0.010 | 0.008 | 0.018 | 0.03 | - |
| *incl.* | 300.0 | 552.0 | 252.0 | 163.8 | 0.270 | 0.013 | 0.018 | 0.012 | 0.030 | 0.06 | - |
| *incl.* | 300.0 | 426.0 | 126.0 | 81.9 | 0.311 | 0.012 | 0.034 | 0.016 | 0.050 | 0.06 | - |
| *incl.* | 304.5 | 343.5 | 39.0 | 25.4 | 0.351 | 0.012 | 0.026 | 0.010 | 0.036 | 0.10 | - |
| CR19-14A | 43.5 | 944.2 | 900.7 | n-v* | 0.31 | 0.013 | 0.022 | 0.008 | 0.030 | 0.17 | - |
| *incl.* | 93.0 | 457.5 | 364.5 | n-v* | 0.37 | 0.014 | 0.031 | 0.011 | 0.042 | 0.26 | - |
| *incl.* | 174.0 | 225.0 | 51.0 | n-v* | 0.40 | 0.014 | 0.023 | 0.009 | 0.031 | 0.19 | - |
| and | 253.5 | 316.5 | 63.0 | n-v* | 0.40 | 0.015 | 0.030 | 0.010 | 0.040 | 0.20 | - |
| and | 357.0 | 448.5 | 91.5 | n-v* | 0.41 | 0.015 | 0.048 | 0.016 | 0.064 | 0.49 | - |
| CR19-15 | 39.0 | 447.5 | 408.5 | 265.5 | 0.25 | 0.012 | 0.008 | 0.004 | 0.013 | 0.03 | - |
| *incl.* | 60.0 | 301.5 | 241.5 | 157.0 | 0.28 | 0.012 | 0.012 | 0.004 | 0.016 | 0.05 | - |
| *incl.* | 99.0 | 184.5 | 85.5 | 55.6 | 0.30 | 0.012 | 0.019 | 0.004 | 0.023 | 0.08 | - |
| and | 519.0 | 522.0 | 3.0 | - | 0.04 | 0.008 | 0.200 | 0.100 | 0.300 | 0.04 | - |
| CR19-16 | 48.0 | 55.5 | 7.5 | - | 0.06 | 0.013 | 0.800 | 1.000 | 1.800 | 0.04 | - |
| and | 81.0 | 642.0 | 561.0 | 364.7 | 0.24 | 0.013 | 0.015 | 0.009 | 0.024 | 0.05 | - |
| *incl.* | 217.5 | 642.0 | 424.5 | 275.9 | 0.26 | 0.013 | 0.017 | 0.009 | 0.026 | 0.06 | - |
| *incl.* | 295.5 | 424.5 | 129.0 | 83.9 | 0.35 | 0.014 | 0.032 | 0.010 | 0.042 | 0.16 | - |
| *incl.* | 309.0 | 379.5 | 70.5 | 45.8 | 0.38 | 0.014 | 0.032 | 0.010 | 0.042 | 0.20 | - |
| *incl.* | 322.5 | 342.0 | 19.5 | 12.7 | 0.47 | 0.015 | 0.050 | 0.016 | 0.066 | 0.30 | - |
| CR19-17 | 36.0 | 501.0 | 465.0 | 302.3 | 0.26 | 0.013 | 0.019 | 0.009 | 0.028 | 0.08 | - |
| *incl.* | 289.5 | 501.0 | 211.5 | 137.5 | 0.32 | 0.013 | 0.034 | 0.015 | 0.049 | 0.17 | - |
| *incl.* | 400.5 | 439.5 | 39.0 | 25.4 | 0.37 | 0.015 | 0.053 | 0.022 | 0.075 | 0.23 | - |
| CR19-18 | 78.0 | 507.0 | 429.0 | 278.9 | 0.25 | 0.013 | 0.015 | 0.007 | 0.022 | 0.08 | - |
| *incl.* | 252.0 | 360.0 | 108.0 | 70.2 | 0.35 | 0.014 | 0.020 | 0.008 | 0.028 | 0.29 | - |
| *incl.* | 303.0 | 360.0 | 57.0 | 37.1 | 0.37 | 0.016 | 0.030 | 0.011 | 0.041 | 0.51 | - |
| CR19-19 | 136.5 | 723.0 | 586.5 | 381.2 | 0.26 | 0.012 | 0.013 | 0.006 | 0.019 | 0.22 | - |
| *incl.* | 393.0 | 723.0 | 330.0 | 214.5 | 0.33 | 0.012 | 0.023 | 0.007 | 0.030 | 0.39 | - |
| *incl.* | 403.5 | 442.5 | 39.0 | 25.4 | 0.41 | 0.014 | 0.030 | 0.009 | 0.040 | 0.60 | - |
| and | 570.0 | 600.0 | 30.0 | 19.5 | 0.39 | 0.012 | 0.022 | 0.007 | 0.029 | 0.26 | - |
| CR19-20 | 34.8 | 702.0 | 667.2 | n-v* | 0.26 | 0.013 | 0.017 | 0.009 | 0.026 | 0.09 | - |
| *incl.* | 249.0 | 702.0 | 453.0 | n-v* | 0.28 | 0.012 | 0.021 | 0.009 | 0.031 | 0.08 | - |
| *incl.* | 676.5 | 702.0 | 25.5 | n-v* | 0.30 | 0.012 | 0.003 | 0.004 | 0.007 | 0.09 | - |
| CR19-21 | 43.5 | 702.0 | 658.5 | 428.0 | 0.25 | 0.013 | 0.026 | 0.008 | 0.034 | 0.04 | - |
| *incl.* | 43.5 | 559.5 | 516.0 | 335.4 | 0.27 | 0.012 | 0.032 | 0.010 | 0.042 | 0.04 | - |
| *incl.* | 174.0 | 442.5 | 268.5 | 174.5 | 0.32 | 0.012 | 0.056 | 0.011 | 0.067 | 0.06 | - |
| *incl.* | 376.5 | 442.5 | 66.0 | 42.9 | 0.34 | 0.013 | 0.044 | 0.015 | 0.060 | 0.12 | - |
| CR19-22 | 55.5 | 489.0 | 433.5 | 281.8 | 0.25 | 0.013 | 0.019 | 0.010 | 0.028 | 0.05 | - |
| *incl.* | 112.5 | 301.5 | 189.0 | 122.9 | 0.31 | 0.013 | 0.031 | 0.014 | 0.045 | 0.09 | - |
| *incl.* | 139.5 | 172.5 | 33.0 | 21.5 | 0.36 | 0.014 | 0.036 | 0.016 | 0.052 | 0.20 | - |
| CR19-23 | 36.0 | 705.0 | 669.0 | n-v* | 0.30 | 0.012 | 0.019 | 0.007 | 0.026 | 0.08 | - |
| *incl.* | 357.0 | 447.0 | 90.0 | n-v* | 0.33 | 0.014 | 0.026 | 0.007 | 0.032 | 0.08 | - |
| CR19-24 | 40.5 | 702.0 | 661.5 | n-v* | 0.32 | 0.013 | 0.023 | 0.008 | 0.031 | 0.24 | - |
| *incl.* | 441.0 | 586.5 | 145.5 | n-v* | 0.38 | 0.013 | 0.029 | 0.008 | 0.037 | 0.32 | - |
| *incl.* | 511.5 | 550.5 | 39.0 | n-v* | 0.40 | 0.013 | 0.032 | 0.010 | 0.042 | 0.36 | - |
| CR19-25 | 70.0 | 387.0 | 317.0 | 196.9 | 0.22 | 0.003 | 0.014 | 0.012 | 0.026 | 0.08 | - |
| *incl.* | 70.0 | 114.0 | 44.0 | 27.3 | 0.34 | 0.006 | 0.034 | 0.012 | 0.046 | 0.26 | - |
| CR19-27 | 87.0 | 420.0 | 333.0 | 218.4 | 0.25 | 0.005 | 0.009 | 0.005 | 0.014 | 0.05 | - |

| Drillhole | From (m) | To (m) | Interval (m) | Estimated True Width (m) | Ni (%) | Co (%) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | S (%) | Fe (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| *incl.* | 166.5 | 385.5 | 219.0 | 143.7 | 0.28 | 0.004 | 0.004 | 0.003 | 0.007 | 0.04 | - |
| and | 82.5 | 91.5 | 9.0 | 5.9 | 0.51 | 0.035 | 0.320 | 0.123 | 0.443 | 0.30 | - |
| and | 82.5 | 84.0 | 1.5 | 1.0 | 0.65 | 0.094 | 0.632 | 0.263 | 0.895 | 0.53 | - |
| and | 84.0 | 85.5 | 1.5 | 1.0 | 1.09 | 0.037 | 0.699 | 0.265 | 0.964 | 0.54 | - |
| CR19-28 | 34.5 | 42.0 | 7.5 | - | 0.05 | 0.006 | 0.100 | 0.200 | 0.300 | - | - |
| and | 180.0 | 184.5 | 4.5 | - | 0.03 | 0.009 | 0.800 | 0.900 | 1.700 | - | - |
| CR19-29 | 205.5 | 210.0 | 4.5 | - | 0.05 | 0.009 | 0.300 | 0.500 | 0.800 | 0.16 | - |
| *incl.* | 208.5 | 210.0 | 1.5 | - | 0.06 | 0.011 | 0.600 | 1.000 | 1.600 | 0.21 | - |
| and | 331.5 | 445.5 | 114.0 | 74.6 | 0.21 | 0.003 | 0.004 | 0.003 | 0.007 | 0.03 | - |
| *incl.* | 382.5 | 445.5 | 63.0 | 41.2 | 0.23 | 0.003 | 0.004 | 0.003 | 0.007 | 0.03 | - |
| CR19-31 | 520.5 | 528.0 | 7.5 | - | 0.04 | 0.011 | 0.400 | 0.400 | 0.800 | - | - |
| *incl.* | 525.0 | 528.0 | 3.0 | - | 0.03 | 0.009 | 0.700 | 0.900 | 1.600 | - | - |
| CR20-42 | 43.5 | 405.0 | 361.5 | 62.8 | 0.40 | 0.017 | 0.036 | 0.012 | 0.048 | 0.52 | 6.05 |
| *incl.* | 43.5 | 349.5 | 306.0 | 53.1 | 0.42 | 0.017 | 0.041 | 0.013 | 0.054 | 0.57 | 5.71 |
| *incl.* | 304.5 | 331.5 | 27.0 | 4.7 | 0.51 | 0.019 | 0.057 | 0.019 | 0.076 | 0.74 | 5.59 |
| CR20-43 | 45.0 | 402.0 | 357.0 | *n-v | 0.33 | 0.014 | 0.027 | 0.010 | 0.037 | 0.29 | 6.11 |
| *incl.* | 45.0 | 294.0 | 249.0 | *n-v | 0.36 | 0.014 | 0.028 | 0.009 | 0.037 | 0.37 | 5.60 |
| *incl.* | 46.5 | 132.0 | 85.5 | *n-v | 0.39 | 0.014 | 0.030 | 0.010 | 0.040 | 0.41 | 5.23 |
| and | 262.5 | 294.0 | 31.5 | *n-v | 0.42 | 0.018 | 0.036 | 0.010 | 0.046 | 0.56 | 6.92 |
| CR20-44 | 36.0 | 402.0 | 366.0 | *n-v | 0.27 | 0.014 | 0.019 | 0.006 | 0.025 | 0.30 | 6.51 |
| *incl.* | 283.5 | 402.0 | 118.5 | *n-v | 0.33 | 0.016 | 0.030 | 0.009 | 0.039 | 0.58 | 6.67 |
| *incl.* | 349.5 | 402.0 | 52.5 | *n-v | 0.41 | 0.020 | 0.043 | 0.013 | 0.056 | 0.99 | 7.35 |
| CR20-45 | 39.0 | 408.0 | 369.0 | *n-v | 0.28 | 0.012 | 0.016 | 0.007 | 0.023 | 0.08 | 5.17 |
| *incl.* | 39.0 | 289.5 | 250.5 | *n-v | 0.30 | 0.012 | 0.022 | 0.008 | 0.030 | 0.11 | 4.70 |
| *incl.* | 145.5 | 219.0 | 73.5 | *n-v | 0.34 | 0.013 | 0.028 | 0.010 | 0.038 | 0.13 | 4.51 |
| CR20-46 | 48.0 | 411.0 | 363.0 | *n-v | 0.31 | 0.013 | 0.030 | 0.010 | 0.040 | 0.29 | 5.83 |
| *incl.* | 315.0 | 384.0 | 69.0 | *n-v | 0.36 | 0.017 | 0.057 | 0.019 | 0.076 | 0.97 | 8.19 |
| CR20-47 | 33.3 | 402.0 | 368.7 | *n-v | 0.30 | 0.011 | 0.020 | 0.008 | 0.028 | 0.07 | 5.91 |
| *incl.* | 40.5 | 286.5 | 246.0 | *n-v | 0.31 | 0.011 | 0.022 | 0.008 | 0.030 | 0.06 | 5.54 |
| *incl.* | 363.0 | 399.0 | 36.0 | *n-v | 0.36 | 0.014 | 0.030 | 0.009 | 0.039 | 0.14 | 6.96 |
| CR20-48 | 34.0 | 402.0 | 368.0 | *n-v | 0.31 | 0.014 | 0.026 | 0.012 | 0.038 | 0.13 | 7.22 |
| *incl.* | 34.0 | 288.0 | 254.0 | *n-v | 0.34 | 0.014 | 0.023 | 0.008 | 0.031 | 0.15 | 7.22 |
| *incl.* | 125.5 | 283.5 | 158.0 | *n-v | 0.36 | 0.014 | 0.023 | 0.008 | 0.031 | 0.16 | 7.22 |
| *incl.* | 151.5 | 223.5 | 72.0 | *n-v | 0.40 | 0.015 | 0.027 | 0.010 | 0.037 | 0.19 | 7.36 |
| CR20-49 | 36.5 | 402.0 | 365.5 | *n-v | 0.28 | 0.013 | 0.024 | 0.012 | 0.036 | 0.05 | 6.83 |
| *incl.* | 36.5 | 210.0 | 173.5 | *n-v | 0.35 | 0.012 | 0.037 | 0.011 | 0.048 | 0.10 | 6.11 |
| *incl.* | 36.5 | 91.5 | 55.0 | *n-v | 0.41 | 0.013 | 0.047 | 0.019 | 0.066 | 0.19 | 5.34 |
| CR20-50 | 36.0 | 402.0 | 366.0 | *n-v | 0.33 | 0.016 | 0.038 | 0.014 | 0.052 | 0.58 | 7.51 |
| *incl.* | 36.0 | 181.5 | 145.5 | *n-v | 0.35 | 0.017 | 0.050 | 0.020 | 0.070 | 0.67 | 7.84 |
| *incl.* | 124.5 | 162.0 | 37.5 | *n-v | 0.40 | 0.018 | 0.051 | 0.016 | 0.067 | 1.03 | 8.02 |
| CR20-51 | 49.6 | 405.0 | 355.4 | *n-v | 0.28 | 0.013 | 0.023 | 0.013 | 0.036 | 0.14 | 6.70 |
| *incl.* | 69.0 | 256.5 | 187.5 | *n-v | 0.34 | 0.013 | 0.037 | 0.013 | 0.050 | 0.23 | 6.22 |
| *incl.* | 73.5 | 174.0 | 100.5 | *n-v | 0.37 | 0.015 | 0.032 | 0.011 | 0.043 | 0.39 | 5.43 |
| *incl.* | 79.5 | 142.5 | 63.0 | *n-v | 0.39 | 0.016 | 0.038 | 0.014 | 0.052 | 0.51 | 4.92 |
| CR20-52 | 27.0 | 402.0 | 375.0 | *n-v | 0.30 | 0.012 | 0.023 | 0.017 | 0.040 | 0.07 | 5.98 |
| *incl.* | 27.0 | 157.5 | 130.5 | *n-v | 0.35 | 0.013 | 0.024 | 0.008 | 0.032 | 0.14 | 4.59 |
| *incl.* | 27.0 | 51.0 | 24.0 | *n-v | 0.40 | 0.014 | 0.033 | 0.011 | 0.044 | 0.23 | 4.43 |
| *CR20-53* | 52.0 | 402.0 | 350.0 | *n-v | 0.29 | 0.012 | 0.017 | 0.007 | 0.024 | 0.10 | 6.37 |
| *incl.* | 169.5 | 352.5 | 183.0 | *n-v | 0.35 | 0.011 | 0.021 | 0.007 | 0.028 | 0.12 | 6.66 |
| *incl.* | 169.5 | 210.0 | 40.5 | *n-v | 0.40 | 0.009 | 0.039 | 0.012 | 0.051 | 0.35 | 5.22 |
| CR20-54 | 44.4 | 402.0 | 357.6 | *n-v | 0.29 | 0.011 | 0.019 | 0.012 | 0.031 | 0.08 | 6.55 |
| *incl.* | 71.5 | 317.5 | 246.0 | *n-v | 0.32 | 0.010 | 0.025 | 0.014 | 0.039 | 0.11 | 6.21 |
| *incl.* | 74.5 | 206.5 | 132.0 | *n-v | 0.34 | 0.008 | 0.016 | 0.007 | 0.023 | 0.17 | 5.32 |
| *incl.* | 121.0 | 148.0 | 27.0 | *n-v | 0.40 | 0.010 | 0.010 | 0.005 | 0.015 | 0.23 | 4.64 |
| CR20-56 | 70.8 | 588.0 | 517.2 | - | 0.28 | 0.013 | 0.013 | 0.009 | 0.022 | 0.08 | 6.72 |
| *incl.* | 114.0 | 262.5 | 148.5 | - | 0.34 | 0.013 | 0.028 | 0.010 | 0.038 | 0.02 | 5.99 |
| *incl.* | 186.0 | 217.5 | 31.5 | - | 0.42 | 0.013 | 0.038 | 0.012 | 0.050 | 0.23 | 6.11 |
| CR20-59 | 42.6 | 390.0 | 347.4 | 223.3 | 0.26 | 0.013 | 0.015 | 0.008 | 0.023 | 0.11 | 6.08 |
| *incl.* | 42.6 | 163.5 | 120.9 | 77.7 | 0.34 | 0.015 | 0.025 | 0.008 | 0.033 | 0.28 | 5.34 |
| *incl.* | 42.6 | 99.0 | 56.4 | 36.3 | 0.38 | 0.015 | 0.032 | 0.010 | 0.042 | 0.29 | 4.73 |
| and | 460.5 | 469.5 | 9.0 | 5.9 | 0.04 | 0.010 | 0.600 | 0.900 | 1.500 | - | - |
| *incl.* | 465.0 | 468.0 | 3.0 | 2.0 | 0.05 | 0.013 | 0.600 | 1.200 | 1.800 | - | - |
| CR20-60 | 51.0 | 352.5 | 301.5 | - | 0.24 | 0.013 | 0.016 | 0.009 | 0.025 | 0.10 | 6.94 |
| *incl.* | 51.0 | 163.5 | 112.5 | - | 0.33 | 0.014 | 0.028 | 0.011 | 0.039 | 0.20 | 7.24 |
| *incl.* | 51.0 | 127.5 | 76.5 | - | 0.36 | 0.014 | 0.024 | 0.008 | 0.032 | 0.23 | 7.13 |
| *incl.* | 84.0 | 100.5 | 16.5 | - | 0.43 | 0.015 | 0.031 | 0.010 | 0.041 | 0.33 | 7.06 |
| CR20-61 | 36.8 | 276.0 | 239.2 | - | 0.30 | 0.013 | 0.022 | 0.008 | 0.030 | 0.14 | 5.39 |
| *incl.* | 36.8 | 183.0 | 146.2 | - | 0.35 | 0.014 | 0.030 | 0.010 | 0.040 | 0.20 | 5.01 |
| *incl.* | 36.8 | 156.0 | 119.2 | - | 0.38 | 0.014 | 0.034 | 0.011 | 0.045 | 0.23 | 4.84 |
| *incl.* | 67.5 | 144.0 | 76.5 | - | 0.40 | 0.015 | 0.039 | 0.012 | 0.051 | 0.26 | 4.92 |
| CR20-62 | 45.3 | 402.0 | 356.7 | *n-v | 0.29 | 0.013 | 0.018 | 0.007 | 0.025 | 0.28 | 6.83 |
| *incl.* | 49.5 | 132.0 | 82.5 | *n-v | 0.30 | 0.010 | 0.017 | 0.007 | 0.024 | 0.09 | 5.48 |
| *incl.* | 231.0 | 399.0 | 168.0 | *n-v | 0.30 | 0.015 | 0.028 | 0.011 | 0.039 | 0.49 | 7.63 |
| CR20-63 | 39.0 | 402.0 | 363.0 | *n-v | 0.27 | 0.014 | 0.027 | 0.013 | 0.040 | 0.19 | 7.01 |
| incl. | 357.0 | 402.0 | 45.0 | *n-v | 0.36 | 0.013 | 0.016 | 0.006 | 0.022 | 0.21 | 5.58 |
| CR20-64 | 32.6 | 402.0 | 369.4 | *n-v | 0.33 | 0.014 | 0.020 | 0.007 | 0.027 | 0.21 | 5.05 |
| *incl.* | 193.5 | 289.5 | 96.0 | *n-v | 0.38 | 0.014 | 0.026 | 0.010 | 0.036 | 0.20 | 4.85 |
| *incl.* | 193.5 | 22.0 | -171.5 | *n-v | 0.41 | 0.015 | 0.027 | 0.009 | 0.036 | 0.24 | 4.94 |

| Drillhole | From (m) | To (m) | Interval (m) | Estimated True Width (m) | Ni (%) | Co (%) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | S (%) | Fe (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| CR20-65 | 36.0 | 402.0 | 366.0 | - | 0.26 | 0.013 | 0.018 | 0.009 | 0.027 | 0.07 | 6.10 |
| *incl.* | 36.0 | 162.0 | 126.0 | - | 0.33 | 0.012 | 0.018 | 0.006 | 0.024 | 0.11 | 4.81 |
| *incl.* | 36.0 | 76.5 | 40.5 | - | 0.35 | 0.013 | 0.018 | 0.007 | 0.025 | 0.15 | 5.31 |
| CR21-115 | 88.5 | 450.0 | 361.5 | 232.4 | 0.21 | 0.012 | 0.003 | 0.005 | 0.008 | 0.07 | 7.23 |
| CR21-154 | 72.0 | 636.0 | 564.0 | 323.5 | 0.23 | 0.012 | 0.028 | 0.016 | 0.044 | 0.02 | 6.72 |
| *incl.* | 168.0 | 438.0 | 270.0 | 154.9 | 0.27 | 0.011 | 0.081 | 0.028 | 0.109 | 0.03 | 6.57 |
| CR21-155 | 46.0 | 579.0 | 533.0 | 250.2 | 0.28 | 0.011 | 0.016 | 0.007 | 0.023 | 0.06 | 6.60 |
| *incl.* | 198.0 | 483.0 | 285.0 | 133.8 | 0.32 | 0.011 | 0.021 | 0.007 | 0.028 | 0.07 | 6.83 |
| *incl.* | 447.0 | 481.5 | 34.5 | 16.2 | 0.37 | 0.012 | 0.034 | 0.011 | 0.045 | 0.20 | 5.83 |
| CR21-157 | 45.5 | 549.0 | 503.5 | 288.8 | 0.22 | 0.013 | 0.006 | 0.012 | 0.018 | 0.02 | 6.97 |
| *incl.* | 475.0 | 549.0 | 74.0 | 42.4 | 0.27 | 0.014 | 0.016 | 0.022 | 0.038 | 0.02 | 6.92 |
| CR21-159 | 30.2 | 366.0 | 335.8 | 192.6 | 0.27 | 0.013 | 0.015 | 0.009 | 0.024 | 0.06 | 6.27 |
| *incl.* | 81.0 | 187.5 | 106.5 | 61.1 | 0.34 | 0.013 | 0.027 | 0.011 | 0.038 | 0.14 | 5.25 |
| *incl.* | 90.0 | 135.0 | 45.0 | 25.8 | 0.39 | 0.014 | 0.037 | 0.013 | 0.050 | 0.22 | 4.33 |
| *incl.* | 94.5 | 123.0 | 28.5 | 16.3 | 0.41 | 0.014 | 0.040 | 0.014 | 0.054 | 0.25 | 4.30 |
| CR21-162 | 25.5 | 396.0 | 370.5 | 95.9 | 0.29 | 0.013 | 0.022 | 0.014 | 0.036 | 0.07 | 6.54 |
| *incl.* | 112.5 | 274.5 | 162.0 | 41.9 | 0.33 | 0.013 | 0.033 | 0.011 | 0.044 | 0.05 | 6.47 |
| CR21-166 | 49.1 | 583.0 | 533.9 | 343.2 | 0.23 | 0.015 | 0.038 | 0.027 | 0.065 | 0.04 | 7.42 |
| *incl.* | 49.1 | 103.5 | 54.4 | 35.0 | 0.27 | 0.015 | 0.011 | 0.010 | 0.021 | 0.05 | 7.21 |
| and | 124.5 | 139.5 | 15.0 | 9.6 | 0.31 | 0.017 | 0.131 | 0.074 | 0.205 | 0.18 | 7.15 |
| CR21-176 | 36.0 | 741.0 | 705.0 | 241.1 | 0.28 | 0.013 | 0.012 | 0.006 | 0.018 | 0.05 | 6.29 |
| *incl.* | 465.0 | 741.0 | 276.0 | 94.4 | 0.32 | 0.013 | 0.017 | 0.006 | 0.023 | 0.08 | 7.08 |
| CR21-178 | 35.5 | 504.0 | 468.5 | - | 0.31 | 0.01 | 0.030 | 0.011 | 0.041 | 0.11 | 5.96 |
| CR21-181 | 51.0 | 606.0 | 555.0 | - | 0.28 | 0.01 | 0.022 | 0.009 | 0.032 | 0.05 | 6.49 |
| CR21-183 | 45.0 | 596.0 | 551.0 | - | 0.26 | 0.01 | 0.023 | 0.013 | 0.036 | 0.05 | 2.07 |
| CR21-186 | 46.5 | 621.0 | 574.5 | - | 0.25 | 0.01 | 0.013 | 0.011 | 0.024 | 0.03 | 7.03 |
| CR21-189 | 73.2 | 663.0 | 589.8 | - | 0.27 | 0.01 | 0.024 | 0.012 | 0.036 | 0.03 | 6.89 |
| CR22-191 | 56.5 | 675.0 | 618.5 | - | 0.25 | 0.01 | 0.017 | 0.009 | 0.026 | 0.05 | 6.78 |
| CR22-194 | 40.5 | 775.8 | 735.3 | - | 0.28 | 0.01 | 0.025 | 0.012 | 0.037 | 0.20 | 6.28 |
| CR22-198 | 42.0 | 1044.0 | 1002.0 | - | 0.30 | 0.01 | 0.039 | 0.021 | 0.060 | 0.30 | 6.97 |
| *incl.* | 610.5 | 724.5 | 114.0 | - | 0.38 | 0.02 | 0.069 | 0.037 | 0.106 | 0.53 | 7.55 |
| CR22-202 | 46.8 | 675.0 | 628.2 | - | 0.29 | 0.02 | 0.025 | 0.010 | 0.036 | 0.30 | 6.65 |
| CR22-234 | 145.5 | 175.5 | 30.0 | - | 0.02 | 0.01 | 0.922 | 0.905 | 1.826 | 0.03 | 5.33 |
| CR22-243 | 38.5 | 523.0 | 484.5 | - | 0.25 | 0.01 | 0.016 | 0.009 | 0.025 | 0.06 | 6.66 |
| CR22-246 | 39.6 | 387.0 | 347.4 | - | 0.24 | 0.01 | 0.021 | 0.009 | 0.030 | 0.35 | 6.64 |
| CR22-277 | 62.8 | 597.0 | 534.2 | - | 0.23 | 0.01 | 0.008 | 0.008 | 0.016 | 0.04 | 6.35 |

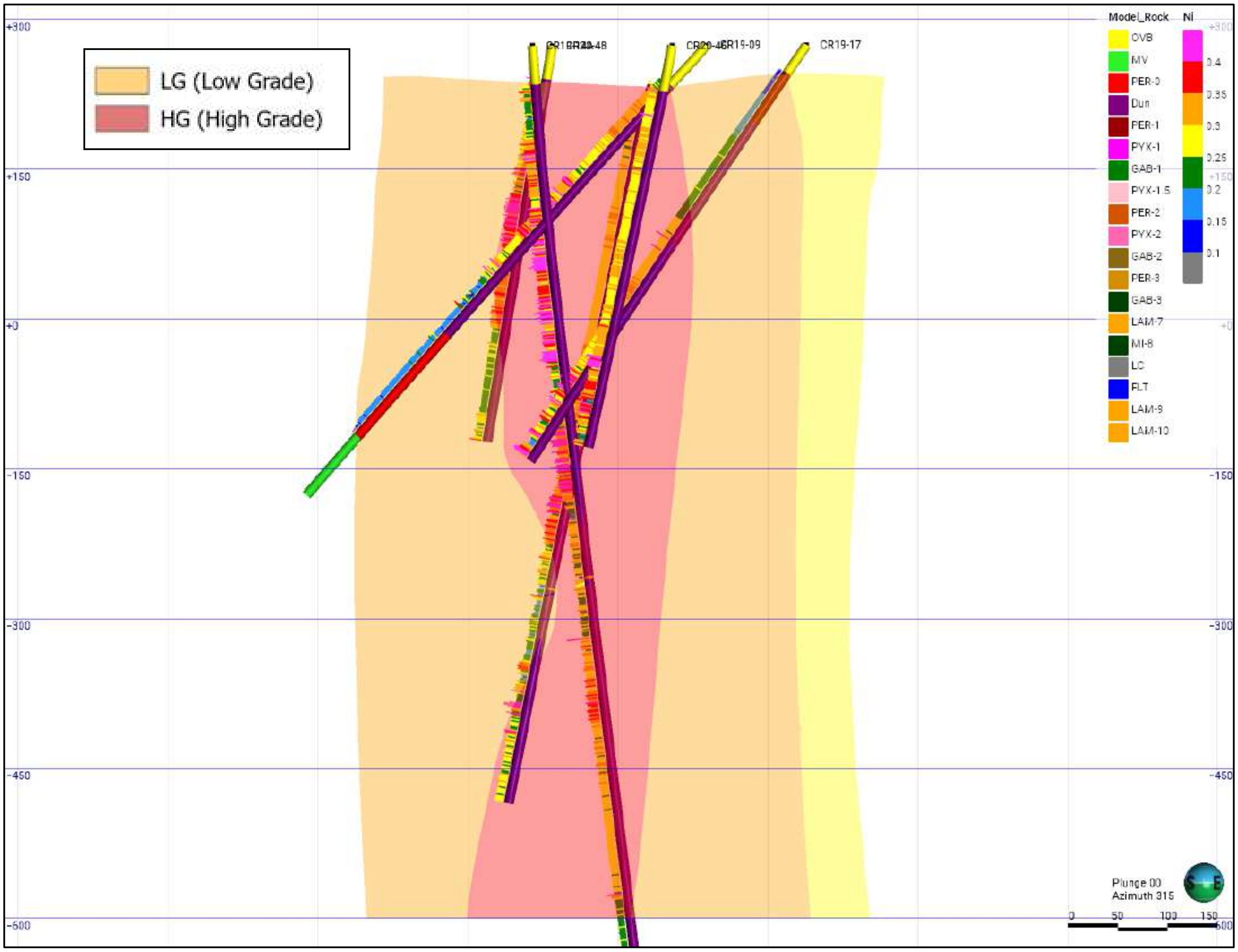
Notes: *n-v: holes drilled at steep angle of -82° or -80° and so interval length is equal to depth. Where not estimated, core intervals are not true widths. CNC has insufficient information to determine the attitude, either of the ultramafic body or of mineralized zones within it. True widths will be less than the core intervals by several factors.

**Figure 10-2: Plan View of Diamond Drillhole Traces from 2018-2022 Drilling at the Main Zone**




Note: Drilling is superimposed on the outline of the nickel grade domains within the mineral resource estimate. Source: CNC, 2023.

**Figure 10-3: Oblique Sections Looking Northwest from Main Zone with Estimation Domains for Nickel**




Note: drillhole trace histogram scale is %Ni. Source: Caracle Creek, 2023.

#### 10.3.1.1 Higher Grade Nickel Zone

Diamond drilling core assay results to date allow for the delineation of two higher grade (>0.30% Ni and >0.35% Ni) regions (modelled grade shells) within the larger core high-grade zone (>0.25% Ni), which in turn are within the larger enveloping low-grade zone (>0.15% Ni), all contained within the host ultramafic body of the CUC (Figure 10-2). The high-grade zone (>0.25% Ni) has a minimum modelled strike length of about 1.9 km, is between approximately 115 and 210 m wide, and contains regions of incrementally higher grade nickel (i.e., >0.30% Ni and >0.35% Ni). The high-grade zone remains open along strike to the west-northwest and extend to a depth of at least 650 m (Figures 10-2 and 10-3).

#### 10.3.1.2 Main Zone – PGE Reef

The Main Zone PGE reef, set within the northern margin of the ultramafic body, is associated with a contact between a peridotite unit to the south and a pyroxenite unit to the north, reflected in drillhole intercepts (Table 10-3). The reef is sufficiently well delineated in the Main Zone, though additional drillholes will be required to better define it towards the West Zone. True widths in Table 10-3 were calculated manually by measuring on-section in Leapfrog software.

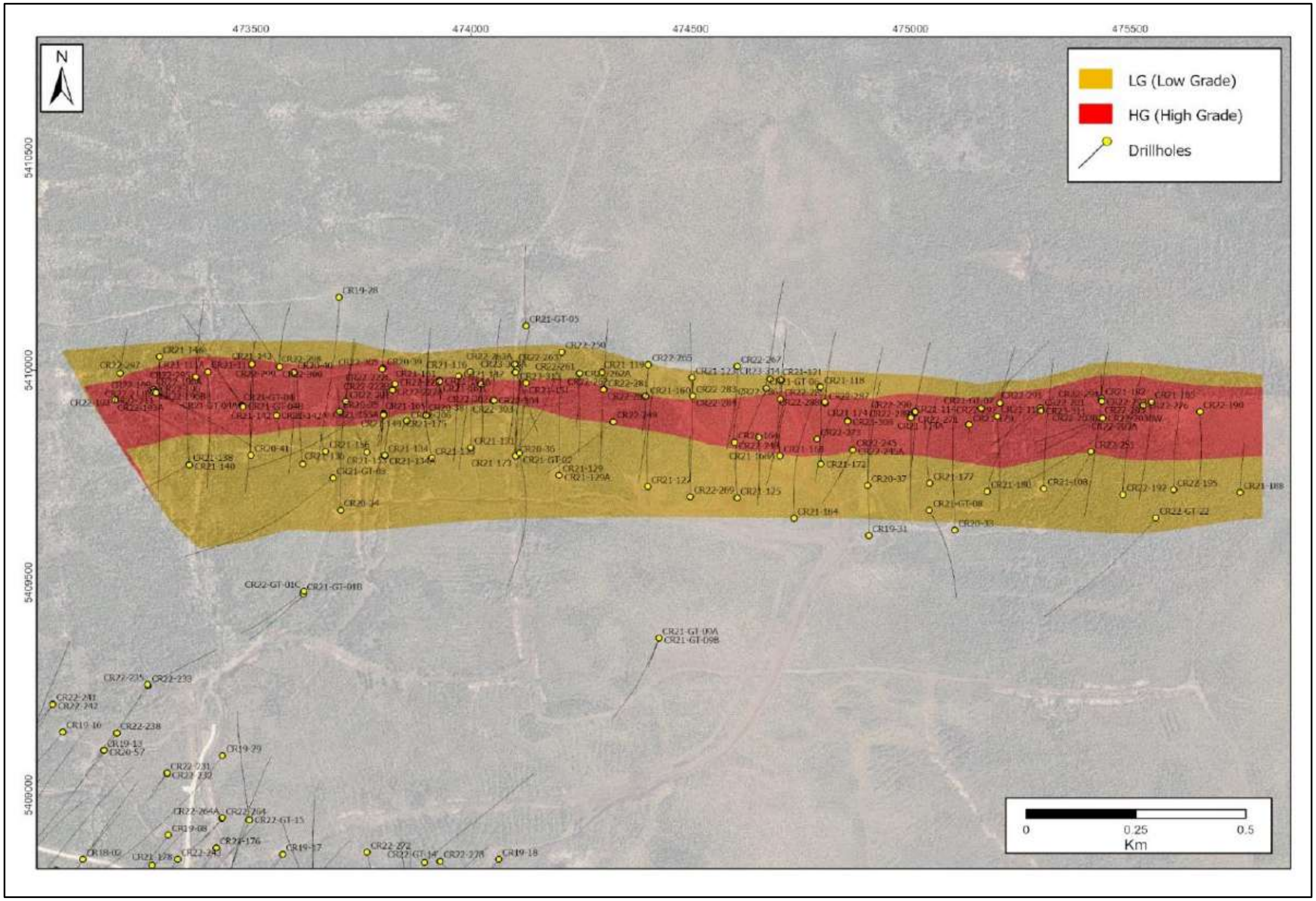
**Table 10-3: True Width Intercepts for Drillholes into the Main Zone PGE Reef**

| Drillhole | True Width (m) | Pd (ppm) | Pt (ppm) | PGE (ppm) | Ni (%) | Co (%) | Fe (%) | S (%) |
|---|---|---|---|---|---|---|---|---|
| CR19-12 | 7.70 | 0.315 | 0.493 | 0.807 | 0.06 | 0.01 | 7.38 | 0.04 |
| CR19-13 | 4.90 | 0.735 | 1.012 | 1.747 | 0.05 | 0.01 | 7.00 | 0.03 |
| CR19-15 | 0.90 | 0.298 | 0.058 | 0.356 | 0.04 | 0.01 | 4.87 | 0.01 |
| CR19-16 | 5.00 | 0.772 | 0.958 | 1.730 | 0.06 | 0.01 | 7.10 | 0.04 |
| CR19-29 | 2.80 | 0.349 | 0.484 | 0.834 | 0.05 | 0.01 | 4.81 | 0.16 |
| CR20-59 | 6.60 | 0.540 | 0.772 | 1.313 | 0.04 | 0.01 | 5.86 | 0.08 |
| CR20-61 | 0.90 | 0.127 | 0.200 | 0.327 | 0.08 | 0.02 | 7.70 | 0.03 |
| CR20-78 | 7.20 | 0.632 | 0.619 | 1.252 | 0.07 | 0.01 | 6.67 | 0.06 |
| CR22-231 | 3.60 | 0.945 | 0.858 | 1.802 | 0.06 | 0.01 | 7.29 | 0.01 |
| CR22-232 | 7.70 | 0.577 | 0.370 | 0.947 | 0.06 | 0.01 | 6.51 | 0.06 |
| CR22-235 | 6.20 | 0.703 | 0.910 | 1.613 | 0.04 | 0.01 | 6.02 | 0.11 |
| CR22-238 | 9.55 | 0.673 | 0.622 | 1.295 | 0.05 | 0.01 | 5.70 | 0.08 |
| CR22-240 | 3.65 | 0.694 | 0.814 | 1.508 | 0.04 | 0.01 | 6.17 | 1.06 |
| CR22-241 | 3.90 | 0.676 | 0.540 | 1.216 | 0.02 | 0.01 | 5.31 | 0.06 |
| *and* | 2.50 | 0.948 | 0.979 | 1.927 | 0.02 | 0.01 | 5.36 | 0.01 |
| CR22-242 | 9.00 | 0.716 | 0.587 | 1.304 | 0.04 | 0.01 | 6.43 | 0.05 |
| CR22-252 | 4.70 | 0.394 | 0.250 | 0.644 | 0.05 | 0.01 | 6.20 | 0.10 |
| CR22-253 | 1.80 | 0.319 | 0.260 | 0.579 | 0.04 | 0.01 | 4.70 | 0.05 |
| CR22-255 | 3.30 | 0.200 | 0.540 | 0.740 | 0.05 | 0.01 | 6.72 | 0.03 |
| CR22-256 | 5.50 | 0.569 | 0.700 | 1.269 | 0.04 | 0.01 | 6.52 | 0.01 |
| CR22-257 | 2.45 | 1.212 | 1.922 | 3.133 | 0.05 | 0.01 | 6.86 | 0.03 |
| CR22-272 | 5.60 | 0.693 | 0.573 | 1.266 | 0.05 | 0.01 | 6.78 | 0.08 |
| CR22-278 | 3.85 | 0.907 | 1.044 | 1.951 | 0.03 | 0.01 | 5.95 | 0.09 |
| *and* | 3.90 | 0.389 | 0.598 | 0.986 | 0.07 | 0.01 | 7.20 | 0.04 |
| *and* | 2.90 | 0.312 | 0.229 | 0.540 | 0.06 | 0.01 | 6.82 | 0.03 |
| *and* | 3.15 | 0.665 | 0.754 | 1.420 | 0.06 | 0.01 | 7.82 | 0.05 |

### 10.3.2 East Zone Drilling

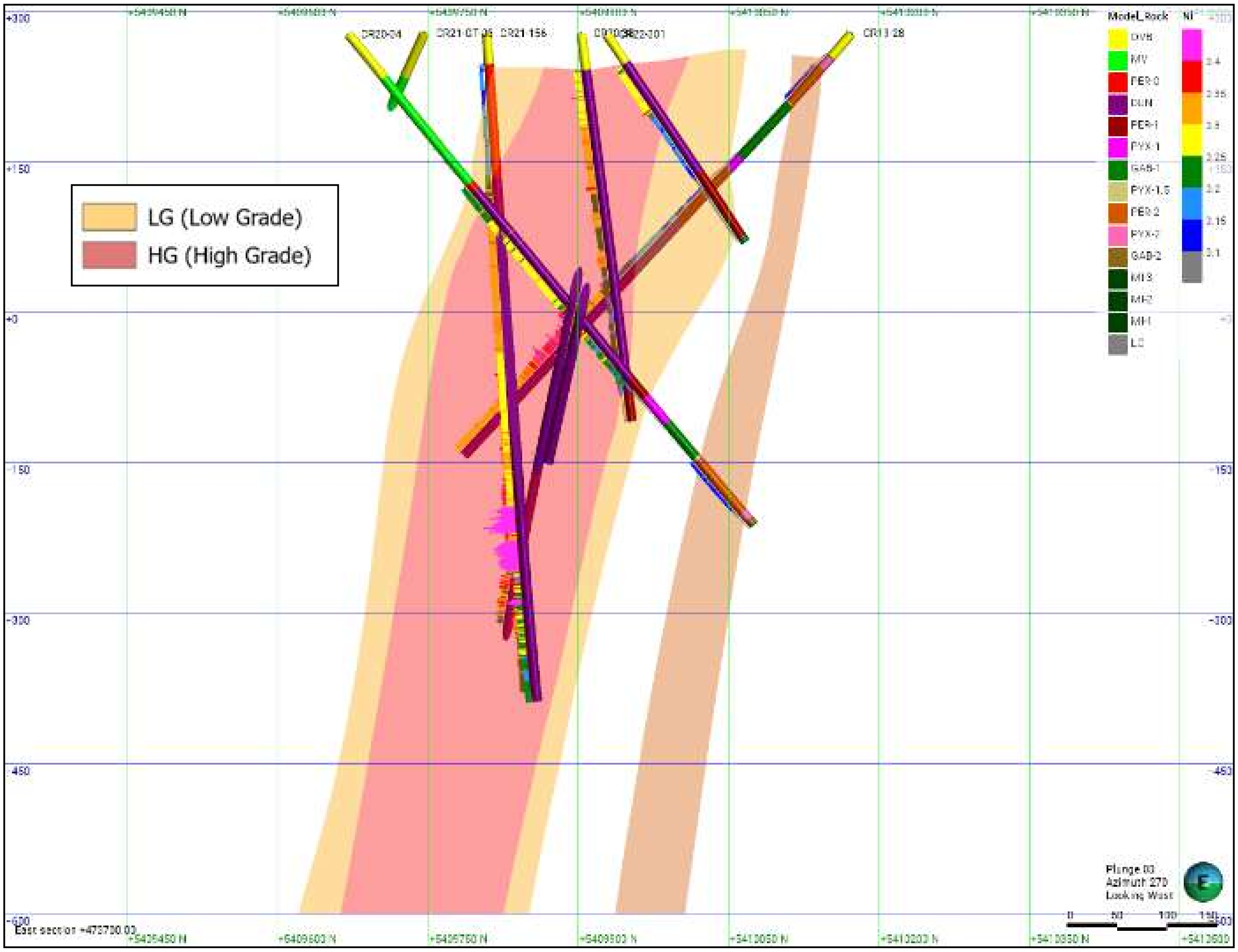
Located about 1.2 km northeast of the Main Zone (Figure 7-5), CNC began to drill-test the East Zone in late 2019 and into 2023 with relatively wide-spaced drillhole sections (Figures 10-4 and 10-5), followed by infill drilling in 2022-2023. To date, this drilling has outlined an east-west trending ultramafic body (largely dunite-peridotite) that is at least 2.6 km in strike length, 200 to 350 m in width, and more than 800 m deep. Selected drill core assays from the East Zone are summarized in Table 10-4. True widths in Table 10-4 were calculated manually by measuring on-section in Leapfrog software.

**Figure 10-4: Plan View of Diamond Drillhole Traces from 2018-2022 Drilling at the East Zone**




Source: CNC, 2023

**Figure 10-5: Section (473700N) Looking West from East Zone Showing Estimation Domains for Nickel**




Note: drillhole trace histogram scale is %Ni. Source: Caracle Creek, 2023.

**Table 10-4: East Zone: Selective Drill Core Assays, CR19, CR20, CR21 and CR22 Series of Diamond Drillholes**

| Drillhole | From (m) | To (m) | Interval (m) | Estimated True Width (m) | Ni (%) | Co (%) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | S (%) | Fe (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| CR19-28 | 252.0 | 573.0 | 321.0 | - | 0.270 | 0.013 | 0.028 | 0.017 | 0.05 | 0.07 | 6.10 |
| *incl.* | 316.5 | 573.0 | 256.5 | - | 0.300 | 0.013 | 0.032 | 0.018 | 0.05 | 0.08 | 5.77 |
| *incl.* | 406.5 | 462.0 | 55.5 | - | 0.420 | 0.014 | 0.131 | 0.071 | 0.20 | 0.78 | 6.01 |
| CR19-31 | 115.5 | 498.0 | 382.5 | - | 0.21 | 0.013 | 0.010 | 0.000 | 0.01 | 0.02 | 6.89 |
| *incl.* | 226.5 | 387.0 | 160.5 | - | 0.26 | 0.013 | 0.010 | 0.000 | 0.01 | 0.02 | 6.13 |
| *incl.* | 304.5 | 333.0 | 28.5 | - | 0.31 | 0.013 | 0.020 | 0.010 | 0.03 | 0.04 | 6.07 |
| CR19-32 | 123.0 | 135.0 | 12.0 | - | 0.02 | 0.007 | 0.900 | 0.900 | 1.80 | - | - |
| *incl.* | 123.0 | 130.5 | 7.5 | - | 0.02 | 0.007 | 1.300 | 1.300 | 2.60 | - | - |
| and | 242.0 | 245.0 | 3.0 | - | 0.03 | 0.007 | 0.900 | 1.000 | 1.90 | - | - |
| and | 277.5 | 289.5 | 12.0 | - | 0.03 | 0.008 | 0.500 | 0.500 | 1.00 | - | - |
| and | 280.5 | 286.5 | 6.0 | - | 0.02 | 0.008 | 0.900 | 0.800 | 1.70 | - | - |
| and | 390.0 | 633.0 | 243.0 | - | 0.25 | 0.013 | 0.003 | 0.003 | 0.01 | 0.02 | 6.10 |
| *incl.* | 438.0 | 633.0 | 195.0 | - | 0.27 | 0.013 | 0.003 | 0.003 | 0.01 | 0.02 | 5.80 |
| *incl.* | 576.0 | 633.0 | 57.0 | - | 0.30 | 0.013 | 0.003 | 0.003 | 0.01 | 0.01 | 5.88 |
| CR19-33 | 119.8 | 434.4 | 314.6 | - | 0.25 | 0.013 | 0.018 | 0.008 | 0.03 | 0.04 | 6.70 |
| *incl.* | 190.6 | 422.4 | 231.8 | - | 0.28 | 0.013 | 0.022 | 0.010 | 0.03 | 0.04 | 6.26 |
| *incl.* | 272.4 | 362.4 | 90.0 | - | 0.32 | 0.013 | 0.053 | 0.020 | 0.07 | 0.06 | 5.82 |
| *incl.* | 324.9 | 362.4 | 37.5 | - | 0.37 | 0.015 | 0.122 | 0.044 | 0.17 | 0.10 | 6.05 |
| *incl.* | 332.4 | 335.4 | 3.0 | - | 0.42 | 0.014 | 1.160 | 0.035 | 1.20 | 0.12 | 5.72 |
| and | 453.9 | 456.9 | 3.0 | - | 0.03 | 0.008 | 0.540 | 0.500 | 1.04 | - | - |
| and | 521.4 | 522.9 | 1.5 | - | 0.06 | 0.013 | 0.660 | 0.700 | 1.36 | - | - |
| CR20-34 | 192.0 | 445.5 | 253.5 | - | 0.26 | 0.013 | 0.032 | 0.013 | 0.05 | 0.04 | 6.28 |
| *incl.* | 274.5 | 387.0 | 112.5 | - | 0.30 | 0.012 | 0.067 | 0.026 | 0.09 | 0.04 | 5.80 |
| *incl.* | 348.0 | 381.0 | 33.0 | - | 0.37 | 0.015 | 0.216 | 0.079 | 0.30 | 0.09 | 6.06 |
| *incl.* | 349.5 | 361.5 | 12.0 | - | 0.42 | 0.015 | 0.463 | 0.166 | 0.63 | 0.08 | 6.11 |
| CR20-34 | 450.0 | 468.0 | 18.0 | - | 0.06 | 0.015 | 0.400 | 0.300 | 0.70 | - | - |
| *incl.* | 463.5 | 468.0 | 4.5 | - | 0.06 | 0.014 | 0.900 | 0.900 | 1.80 | - | - |
| CR20-36 | 33.0 | 289.5 | 256.5 | - | 0.23 | 0.013 | 0.007 | 0.006 | 0.01 | 0.05 | 6.94 |
| *incl.* | 172.5 | 247.5 | 75.0 | - | 0.30 | 0.013 | 0.011 | 0.007 | 0.02 | 0.08 | 6.09 |
| and | 432.0 | 436.5 | 4.5 | - | 0.00 | 0.000 | 0.100 | 0.200 | 0.30 | - | - |
| CR20-37 | 262.5 | 286.5 | 24.0 | - | 0.08 | 0.014 | 0.200 | 0.200 | 0.40 | - | - |
| *incl.* | 283.5 | 286.5 | 3.0 | - | 0.06 | 0.013 | 0.900 | 1.100 | 2.00 | - | - |
| CR20-38 | 51.0 | 189.0 | 138.0 | 89.7 | 0.22 | 0.012 | 0.004 | 0.004 | 0.01 | 0.04 | 6.59 |
| *incl.* | 51.0 | 117.0 | 66.0 | 42.9 | 0.26 | 0.011 | 0.003 | 0.004 | 0.01 | 0.03 | 5.78 |
| and | 189.0 | 195.0 | 6.0 | 3.9 | 0.03 | 0.008 | 0.700 | 0.700 | 1.40 | - | - |
| *incl.* | 189.0 | 193.5 | 4.5 | 2.9 | 0.03 | 0.008 | 0.800 | 0.900 | 1.70 | - | - |
| CR20-39 | 36.0 | 456.0 | 420.0 | 273.0 | 0.24 | 0.013 | 0.004 | 0.005 | 0.01 | 0.02 | 6.80 |
| *incl.* | 36.0 | 190.5 | 154.5 | 100.4 | 0.27 | 0.013 | 0.004 | 0.004 | 0.01 | 0.03 | 6.21 |
| *incl.* | 36.0 | 97.5 | 61.5 | 40.0 | 0.30 | 0.013 | 0.004 | 0.005 | 0.01 | 0.03 | 5.82 |
| CR20-40 | 48.0 | 375.0 | 327.0 | 212.6 | 0.26 | 0.012 | 0.003 | 0.003 | 0.01 | 0.03 | 5.87 |
| CR20-41 | 55.0 | 280.5 | 225.5 | 146.6 | 0.24 | 0.013 | 0.008 | 0.006 | 0.01 | 0.04 | 6.24 |
| *incl.* | 96.0 | 249.0 | 153.0 | 99.5 | 0.26 | 0.013 | 0.010 | 0.006 | 0.02 | 0.03 | 6.03 |
| *incl.* | 168.0 | 226.5 | 58.5 | 38.0 | 0.28 | 0.012 | 0.021 | 0.011 | 0.03 | 0.03 | 5.79 |
| and | 321.0 | 327.0 | 6.0 | 3.9 | 0.02 | 0.008 | 0.700 | 0.800 | 1.50 | - | - |
| *incl.* | 322.5 | 327.0 | 4.5 | 2.9 | 0.03 | 0.008 | 0.900 | 0.900 | 1.80 | - | - |
| CR21-114A | 26.0 | 494.0 | 468.0 | 197.8 | 0.21 | 0.011 | 0.008 | 0.005 | 0.01 | 0.02 | 5.75 |
| *incl.* | 26.0 | 354.0 | 328.0 | 164.0 | 0.25 | 0.012 | 0.009 | 0.005 | 0.01 | 0.02 | 5.76 |
| CR21-116 | 26.3 | 510.0 | 483.7 | 277.4 | 0.24 | 0.012 | 0.003 | 0.005 | 0.01 | 0.03 | 5.61 |
| *incl.* | 63.0 | 168.0 | 105.0 | 60.2 | 0.26 | 0.011 | 0.003 | 0.006 | 0.01 | 0.03 | 5.29 |
| CR21-121 | 12.0 | 507.0 | 495.0 | 209.2 | 0.21 | 0.012 | 0.007 | 0.006 | 0.01 | 0.01 | 5.97 |
| *incl.* | 265.5 | 394.5 | 129.0 | 54.5 | 0.26 | 0.012 | 0.005 | 0.005 | 0.01 | 0.01 | 5.32 |
| CR21-134A | 90.0 | 319.5 | 229.5 | 114.8 | 0.22 | 0.012 | 0.006 | 0.008 | 0.01 | 0.03 | 6.57 |
| *incl.* | 195.0 | 249.0 | 54.0 | 22.8 | 0.28 | 0.013 | 0.012 | 0.017 | 0.03 | 0.03 | 6.06 |
| CR21-142A | 47.5 | 624.0 | 576.5 | 10.1 | 0.32 | 0.017 | 0.017 | 0.010 | 0.03 | 0.12 | 5.72 |
| *incl.* | 441.0 | 582.6 | 141.6 | 9.9 | 0.37 | 0.022 | 0.055 | 0.011 | 0.07 | 0.38 | 6.13 |
| *incl.* | 472.5 | 490.5 | 18.0 | 3.6 | 0.43 | 0.015 | 0.040 | 0.014 | 0.05 | 0.30 | 5.57 |
| *incl.* | 530.1 | 543.6 | 13.5 | 2.3 | 0.43 | 0.019 | 0.053 | 0.017 | 0.07 | 0.72 | 6.88 |
| CR21-146 | 44.7 | 525.0 | 480.3 | 275.5 | 0.27 | 0.013 | 0.009 | 0.007 | 0.02 | 0.05 | 5.75 |
| *incl.* | 132 | 172.5 | 40.5 | 18.1 | 0.32 | 0.014 | 0.065 | 0.035 | 0.10 | 0.12 | 6.45 |
| *incl.* | 159 | 169.5 | 10.5 | 6.9 | 0.44 | 0.016 | 0.155 | 0.080 | 0.24 | 0.15 | 6.20 |
| CR21-118 | 21.0 | 504.0 | 483.0 | 241.5 | 0.23 | 0.013 | 0.015 | 0.012 | 0.03 | 0.04 | 5.89 |
| *incl.* | 276.0 | 339.0 | 63.0 | 31.5 | 0.27 | 0.013 | 0.066 | 0.030 | 0.10 | 0.03 | 5.29 |
| CR21-119 | 39.7 | 479.0 | 439.3 | 219.7 | 0.24 | 0.012 | 0.014 | 0.010 | 0.02 | 0.04 | 6.27 |
| *incl.* | 221.0 | 434.0 | 213.0 | 106.5 | 0.27 | 0.012 | 0.003 | 0.003 | 0.01 | 0.03 | 5.60 |
| CR21-123 | 22.0 | 507.0 | 485.0 | 242.5 | 0.20 | 0.012 | 0.008 | 0.007 | 0.02 | 0.02 | 6.44 |
| *incl.* | 261.5 | 430.5 | 169.0 | 84.5 | 0.26 | 0.012 | 0.006 | 0.005 | 0.01 | 0.02 | 5.74 |
| CR21-125 | 72.0 | 375.6 | 303.6 | 151.8 | 0.22 | 0.012 | 0.008 | 0.006 | 0.01 | 0.05 | 6.43 |
| *incl.* | 232.5 | 348.0 | 115.5 | 57.8 | 0.27 | 0.012 | 0.011 | 0.006 | 0.02 | 0.06 | 5.46 |
| CR21-127 | 87.0 | 300.0 | 213.0 | 136.9 | 0.21 | 0.013 | 0.007 | 0.006 | 0.01 | 0.06 | 6.54 |
| *incl.* | 177.0 | 259.5 | 82.5 | 53.0 | 0.25 | 0.012 | 0.004 | 0.005 | 0.01 | 0.06 | 5.57 |
| CR21-131 | 39.0 | 346.5 | 307.5 | 130.0 | 0.20 | 0.012 | 0.010 | 0.007 | 0.02 | 0.08 | 6.71 |
| *incl.* | 198.0 | 295.5 | 97.5 | 41.2 | 0.26 | 0.012 | 0.018 | 0.010 | 0.03 | 0.07 | 5.81 |
| CR21-133 | 99.0 | 285.0 | 186.0 | 93.0 | 0.23 | 0.012 | 0.006 | 0.006 | 0.01 | 0.06 | 6.30 |
| *incl.* | 180.0 | 271.5 | 91.5 | 45.8 | 0.26 | 0.012 | 0.007 | 0.007 | 0.01 | 0.07 | 5.68 |
| CR21-136 | 37.5 | 327.0 | 289.5 | 166.1 | 0.24 | 0.012 | 0.034 | 0.013 | 0.05 | 0.07 | 6.25 |

| Drillhole | From (m) | To (m) | Interval (m) | Estimated True Width (m) | Ni (%) | Co (%) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | S (%) | Fe (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| *incl.* | 129.0 | 285.0 | 156.0 | 89.5 | 0.28 | 0.012 | 0.056 | 0.019 | 0.08 | 0.08 | 5.76 |
| *incl.* | 235.5 | 271.5 | 36.0 | 20.6 | 0.34 | 0.013 | 0.230 | 0.065 | 0.30 | 0.11 | 5.90 |
| *incl.* | 241.5 | 255.0 | 13.5 | 7.7 | 0.40 | 0.011 | 0.560 | 0.113 | 0.67 | 0.10 | 5.64 |
| CR21-138 | 47.5 | 336.0 | 288.5 | 144.3 | 0.22 | 0.012 | 0.010 | 0.008 | 0.02 | 0.07 | 6.08 |
| *incl.* | 163.5 | 268.5 | 105.0 | 52.5 | 0.27 | 0.012 | 0.007 | 0.006 | 0.01 | 0.08 | 5.19 |
| CR21-143 | 48.6 | 499.2 | 450.6 | 289.6 | 0.24 | 0.012 | 0.003 | 0.006 | 0.01 | 0.03 | 5.94 |
| *incl.* | 48.6 | 334.0 | 285.4 | 183.5 | 0.26 | 0.012 | 0.003 | 0.005 | 0.01 | 0.03 | 5.75 |
| CR21-147 | 29.2 | 315.0 | 285.9 | 183.7 | 0.22 | 0.012 | 0.005 | 0.005 | 0.01 | 0.05 | 6.30 |
| *incl.* | 35.0 | 134.0 | 99.0 | 63.6 | 0.27 | 0.012 | 0.003 | 0.005 | 0.01 | 0.05 | 5.56 |
| CR21-149 | 31.1 | 472.6 | 441.5 | 23.1 | 0.26 | 0.013 | 0.019 | 0.012 | 0.03 | 0.12 | 6.22 |
| *incl.* | 257.5 | 353.5 | 96.0 | 5.0 | 0.31 | 0.014 | 0.053 | 0.024 | 0.08 | 0.12 | 5.93 |
| *incl.* | 313.0 | 344.5 | 31.5 | 1.6 | 0.37 | 0.017 | 0.143 | 0.049 | 0.19 | 0.19 | 6.06 |
| and | 452.5 | 472.6 | 20.1 | 1.1 | 0.34 | 0.020 | 0.055 | 0.024 | 0.08 | 0.34 | 6.41 |
| CR21-151 | 32.3 | 284.0 | 251.7 | 161.8 | 0.20 | 0.012 | 0.007 | 0.007 | 0.01 | 0.04 | 6.98 |
| *incl.* | 42.0 | 87.0 | 45.0 | 28.9 | 0.26 | 0.012 | 0.003 | 0.005 | 0.01 | 0.06 | 5.87 |
| CR21-153 | 85.0 | 669.0 | 584.0 | 40.7 | 0.30 | 0.013 | 0.020 | 0.012 | 0.03 | 0.12 | 6.04 |
| *incl.* | 397.5 | 571.5 | 174.0 | 12.1 | 0.45 | 0.015 | 0.057 | 0.026 | 0.08 | 0.29 | 5.95 |
| *incl.* | 472.5 | 537.0 | 64.5 | 4.5 | 0.71 | 0.016 | 0.115 | 0.035 | 0.15 | 0.46 | 5.53 |
| *incl.* | 492.0 | 498.0 | 6.0 | 0.4 | 1.04 | 0.018 | 0.092 | 0.033 | 0.13 | 0.82 | 5.24 |
| CR21-156 | 84.0 | 657.8 | 573.8 | 40.0 | 0.28 | 0.013 | 0.018 | 0.010 | 0.03 | 0.16 | 6.19 |
| *incl.* | 394.0 | 601.0 | 207.0 | 14.4 | 0.35 | 0.015 | 0.039 | 0.019 | 0.06 | 0.37 | 6.28 |
| *incl.* | 443.5 | 550.0 | 106.5 | 7.4 | 0.38 | 0.016 | 0.062 | 0.026 | 0.09 | 0.51 | 6.67 |
| CR21-160 | 30.7 | 549.0 | 518.3 | 177.3 | 0.24 | 0.013 | 0.007 | 0.004 | 0.01 | 0.03 | 6.10 |
| *incl.* | 304.5 | 475.5 | 171.0 | 58.5 | 0.27 | 0.012 | 0.003 | 0.003 | 0.01 | 0.02 | 5.75 |
| CR21-161A | 21.0 | 432.0 | 411.0 | 95.9 | 0.25 | 0.013 | 0.031 | 0.012 | 0.04 | 0.14 | 6.15 |
| *incl.* | 313.5 | 360.0 | 46.5 | 10.9 | 0.30 | 0.017 | 0.047 | 0.016 | 0.06 | 0.25 | 6.68 |
| AND | 393.0 | 418.5 | 25.5 | 6.0 | 0.37 | 0.017 | 0.228 | 0.060 | 0.29 | 0.29 | 6.83 |
| AND | 316.5 | 324.0 | 7.5 | 1.8 | 0.44 | 0.022 | 0.061 | 0.017 | 0.08 | 0.44 | 8.35 |
| CR21-164 | 33.0 | 210.0 | 177.0 | 60.5 | 0.20 | 0.010 | 0.006 | 0.006 | 0.01 | 0.04 | 5.80 |
| CR21-165 | 36.2 | 555.0 | 518.8 | 40.7 | 0.29 | 0.013 | 0.038 | 0.018 | 0.06 | 0.11 | 5.89 |
| *incl.* | 270.0 | 555.0 | 285.0 | 22.4 | 0.33 | 0.014 | 0.062 | 0.030 | 0.09 | 0.19 | 5.79 |
| *incl.* | 270.0 | 308.0 | 38.0 | 3.0 | 0.37 | 0.011 | 0.078 | 0.083 | 0.16 | 0.10 | 5.24 |
| and | 472.5 | 525.0 | 52.5 | 4.1 | 0.42 | 0.019 | 0.060 | 0.023 | 0.08 | 0.36 | 6.65 |
| CR21-165A | 45.0 | 735.0 | 690.0 | 72.1 | 0.30 | 0.014 | 0.042 | 0.019 | 0.06 | 0.19 | 6.09 |
| *incl.* | 270.0 | 679.5 | 409.5 | 42.8 | 0.34 | 0.015 | 0.065 | 0.028 | 0.09 | 0.29 | 6.12 |
| *incl.* | 468.0 | 579.0 | 111.0 | 11.6 | 0.40 | 0.018 | 0.046 | 0.018 | 0.06 | 0.56 | 6.85 |
| *incl.* | 469.5 | 501.0 | 31.5 | 3.3 | 0.51 | 0.017 | 0.044 | 0.019 | 0.06 | 0.39 | 6.17 |
| CR21-168A | 28.4 | 420.0 | 391.6 | 133.9 | 0.20 | 0.012 | 0.005 | 0.005 | 0.01 | 0.03 | 6.58 |
| *incl.* | 28.4 | 190.5 | 162.1 | 55.4 | 0.27 | 0.012 | 0.005 | 0.005 | 0.01 | 0.03 | 5.79 |
| CR21-174 | 27.0 | 630.0 | 603.0 | 52.6 | 0.23 | 0.012 | 0.007 | 0.007 | 0.01 | 0.04 | 6.13 |
| *incl.* | 481.5 | 630.0 | 148.5 | 12.9 | 0.27 | 0.012 | 0.005 | 0.004 | 0.01 | 0.04 | 5.84 |
| and | 27.0 | 61.5 | 34.5 | 3.0 | 0.31 | 0.014 | 0.018 | 0.007 | 0.03 | 0.07 | 5.86 |
| CR21-179 | 25.4 | 415.6 | 390.2 | | 0.29 | 0.01 | 0.028 | 0.011 | 0.039 | 0.07 | 5.74 |
| CR21-180 | 29.6 | 465.0 | 435.4 | | 0.24 | 0.01 | 0.005 | 0.004 | 0.009 | 0.04 | 6.29 |
| CR21-182 | 16.0 | 450.0 | 434 | | 0.25 | 0.01 | 0.012 | 0.011 | 0.023 | 0.04 | 6.68 |
| CR21-185 | 35.8 | 445.5 | 409.7 | | 0.27 | 0.01 | 0.003 | 0.003 | 0.006 | 0.03 | 5.63 |
| CR22-190 | 39.2 | 402.0 | 362.8 | | 0.24 | 0.01 | 0.004 | 0.004 | 0.008 | 0.03 | 5.66 |
| CR22-193A | 54.0 | 662.5 | 608.5 | | 0.29 | 0.01 | 0.023 | 0.007 | 0.031 | 0.05 | 5.60 |
| CR22-196B | 35.5 | 762.0 | 726.5 | | 0.33 | 0.01 | 0.023 | 0.011 | 0.034 | 0.12 | 5.48 |
| CR22-201 | 12.3 | 435.0 | 422.7 | | 0.32 | 0.01 | 0.029 | 0.026 | 0.056 | 0.09 | 5.50 |
| CR22-203B | 21.0 | 456.0 | 435 | | 0.30 | 0.01 | 0.027 | 0.032 | 0.058 | 0.06 | 5.29 |
| CR22-230 | 532.5 | 1155.0 | 622.5 | | 0.28 | 0.01 | 0.032 | 0.012 | 0.044 | 0.45 | 6.58 |
| CR22-262C | 477.0 | 651.0 | 174 | | 0.29 | 0.01 | 0.003 | 0.006 | 0.010 | 0.08 | 5.59 |
| CR22-271 | 26.3 | 600.0 | 573.7 | | 0.29 | 0.01 | 0.015 | 0.009 | 0.024 | 0.13 | 5.52 |
| CR22-273 | 25.8 | 630.0 | 604.2 | | 0.26 | 0.01 | 0.014 | 0.007 | 0.021 | 0.09 | 5.61 |
| CR22-274 | 35.0 | 651.0 | 616 | | 0.28 | 0.01 | 0.006 | 0.007 | 0.013 | 0.06 | 5.43 |

Note: Where not estimated, core intervals are not true widths. CNC has insufficient information to determine the attitude, either of the ultramafic body or of mineralized zones within it. True widths will be less than the core intervals by several factors.

### 10.3.3 East Zone – PGE Reefs

Within the layered ultramafic unit of the East Zone, two reefs can be differentiated: (1) directly adjacent to the main deposit, associated with a contact between the peridotite unit and its subsequent pyroxenite unit to the north, and (2) further north of the main deposit, mostly within a secondary pyroxenite unit but also associated with another peridotite unit. Several drillholes intersected one or both PGE reefs (Table 10-5). True widths in Table 10-5 were calculated manually by measuring on-section in Leapfrog software.

**Table 10-5: True Width Intercepts for Drillholes into the East Zone PGE Reef**

| Drillhole | Estimated True Width (m) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | Ni (%) | Co (%) | Fe (%) | S (%) |
|---|---|---|---|---|---|---|---|---|
| CR19-28 | 2.90 | 0.780 | 0.873 | 1.653 | 0.03 | 0.01 | 5.64 | 0.12 |
| and | 4.00 | 0.150 | 0.245 | 0.395 | 0.04 | 0.01 | 5.41 | 0.02 |
| CR19-31 | 1.70 | 0.737 | 0.849 | 1.586 | 0.03 | 0.01 | 6.51 | 0.02 |
| CR20-33 | 1.50 | 0.538 | 0.504 | 1.042 | 0.03 | 0.01 | 5.70 | 0.17 |
| and | 2.40 | 0.363 | 0.409 | 0.772 | 0.05 | 0.01 | 5.67 | 0.03 |
| CR20-34 | 4.30 | 0.865 | 0.891 | 1.755 | 0.06 | 0.01 | 8.18 | 0.06 |
| and | 3.40 | 0.171 | 0.275 | 0.445 | 0.04 | 0.01 | 5.67 | 0.04 |
| CR20-35 | 1.80 | 0.552 | 1.130 | 1.682 | 0.06 | 0.01 | 6.85 | 0.03 |
| CR20-36 | 3.10 | 0.145 | 0.226 | 0.371 | 0.04 | 0.01 | 5.77 | 0.01 |
| CR20-37 | 2.30 | 0.685 | 0.807 | 1.492 | 0.06 | 0.01 | 7.56 | 0.05 |
| and | 2.70 | 0.162 | 0.253 | 0.415 | 0.04 | 0.01 | 5.74 | 0.01 |
| CR20-38 | 3.30 | 0.836 | 0.988 | 1.824 | 0.03 | 0.01 | 6.01 | 0.17 |
| and | 3.40 | 0.153 | 0.259 | 0.412 | 0.04 | 0.01 | 5.55 | 0.02 |
| CR20-41 | 4.70 | 0.742 | 0.779 | 1.520 | 0.02 | 0.01 | 6.07 | 0.07 |
| CR21-108 | 6.45 | 0.281 | 0.385 | 0.666 | 0.03 | 0.01 | 5.48 | 0.02 |
| CR21-127 | 1.00 | 0.284 | 0.360 | 0.644 | 0.03 | 0.01 | 4.97 | 0.07 |
| CR21-129A | 5.75 | 0.621 | 0.660 | 1.281 | 0.03 | 0.01 | 5.81 | 0.07 |
| and | 1.90 | 1.240 | 1.280 | 2.520 | 0.02 | 0.01 | 5.48 | 0.00 |
| CR21-131 | 1.30 | 0.463 | 0.480 | 0.943 | 0.02 | 0.01 | 5.64 | 0.05 |
| CR21-133 | 7.85 | 0.673 | 0.693 | 1.366 | 0.02 | 0.01 | 5.81 | 0.12 |
| CR21-134A | 4.50 | 0.599 | 0.685 | 1.284 | 0.02 | 0.01 | 5.60 | 0.06 |
| CR21-138 | 2.95 | 0.384 | 0.501 | 0.886 | 0.05 | 0.01 | 6.99 | 0.03 |
| CR21-164 | 7.00 | 0.672 | 0.703 | 1.375 | 0.05 | 0.01 | 7.00 | 0.04 |
| CR21-184 | 2.00 | 0.849 | 0.989 | 1.838 | 0.04 | 0.01 | 6.51 | 0.03 |
| CR22-192 | 3.35 | 1.060 | 1.720 | 2.780 | 0.03 | 0.01 | 4.20 | 0.05 |
| and | 5.65 | 0.190 | 0.251 | 0.441 | 0.03 | 0.01 | 5.44 | 0.01 |
| CR22-195 | 3.25 | 0.303 | 0.380 | 0.683 | 0.03 | 0.01 | 4.68 | 0.02 |
| CR22-199 | 4.55 | 1.041 | 0.922 | 1.963 | 0.07 | 0.01 | 7.20 | 0.06 |
| CR22-265 | 4.50 | 0.652 | 0.832 | 1.484 | 0.03 | 0.01 | 5.72 | 0.10 |
| and | 2.15 | 0.990 | 1.288 | 2.277 | 0.03 | 0.01 | 5.95 | 0.07 |
| and | 3.45 | 1.104 | 1.277 | 2.381 | 0.03 | 0.01 | 6.14 | 0.10 |

| Drillhole | Estimated True Width (m) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | Ni (%) | Co (%) | Fe (%) | S (%) |
|---|---|---|---|---|---|---|---|---|
| CR22-281 | 3.45 | 0.624 | 0.766 | 1.390 | 0.02 | 0.01 | 5.75 | 0.02 |
| CR22-282 | 2.60 | 0.171 | 0.265 | 0.436 | 0.04 | 0.01 | 6.80 | 0.03 |
| CR22-283 | 6.80 | 0.731 | 0.927 | 1.658 | 0.03 | 0.01 | 5.05 | 0.16 |
| CR22-284 | 5.10 | 0.535 | 0.676 | 1.211 | 0.04 | 0.01 | 6.44 | 0.07 |
| and | 3.55 | 0.247 | 0.359 | 0.606 | 0.03 | 0.01 | 5.34 | 0.05 |
| CR22-285 | 2.40 | 0.339 | 0.530 | 0.869 | 0.04 | 0.01 | 5.59 | 0.01 |
| and | 4.70 | 0.161 | 0.253 | 0.414 | 0.04 | 0.01 | 5.48 | 0.01 |
| CR22-286 | 2.60 | 1.071 | 1.373 | 2.443 | 0.05 | 0.01 | 7.39 | 0.01 |
| CR22-287 | 2.40 | 0.878 | 1.022 | 1.900 | 0.07 | 0.01 | 8.03 | 0.03 |
| and | 3.60 | 0.162 | 0.261 | 0.424 | 0.03 | 0.01 | 5.50 | 0.01 |
| CR22-288 | 2.95 | 0.723 | 0.969 | 1.692 | 0.04 | 0.01 | 6.25 | 0.02 |
| CR22-289 | 2.50 | 0.897 | 1.093 | 1.990 | 0.03 | 0.01 | 5.92 | 0.03 |
| CR22-290 | 2.00 | 0.276 | 0.559 | 0.835 | 0.04 | 0.01 | 6.57 | 0.09 |
| CR22-291 | 4.05 | 0.153 | 0.269 | 0.421 | 0.04 | 0.01 | 6.04 | 0.01 |
| CR22-294 | 2.30 | 1.106 | 1.151 | 2.256 | 0.02 | 0.01 | 5.56 | 0.01 |
| CR22-295 | 3.27 | 0.535 | 0.438 | 0.973 | 0.02 | 0.01 | 4.86 | 0.02 |
| CR22-296 | 3.55 | 0.222 | 0.163 | 0.385 | 0.05 | 0.01 | 6.13 | 0.01 |
| CR22-298 | 6.70 | 0.573 | 0.657 | 1.230 | 0.02 | 0.01 | 5.39 | 0.05 |
| and | 3.25 | 0.167 | 0.280 | 0.447 | 0.04 | 0.01 | 5.32 | 0.01 |
| CR22-299 | 3.50 | 1.002 | 1.120 | 2.122 | 0.03 | 0.01 | 6.25 | 0.19 |
| CR22-300 | 4.20 | 0.397 | 0.486 | 0.882 | 0.03 | 0.01 | 5.56 | 0.06 |
| and | 3.95 | 0.183 | 0.304 | 0.487 | 0.04 | 0.01 | 5.70 | 0.01 |
| CR22-301 | 15.50 | 0.592 | 0.423 | 1.015 | 0.06 | 0.01 | 6.86 | 0.08 |
| CR22-302 | 3.45 | 0.518 | 0.635 | 1.153 | 0.06 | 0.01 | 7.81 | 0.04 |
| CR22-304 | 1.00 | 0.304 | 0.465 | 0.769 | 0.05 | 0.01 | 6.26 | 0.03 |
| CR22-305 | 4.05 | 0.768 | 0.856 | 1.624 | 0.02 | 0.01 | 5.4 | 0.13 |

Source: CNC, 2023.

### 10.3.4 West Zone Drilling

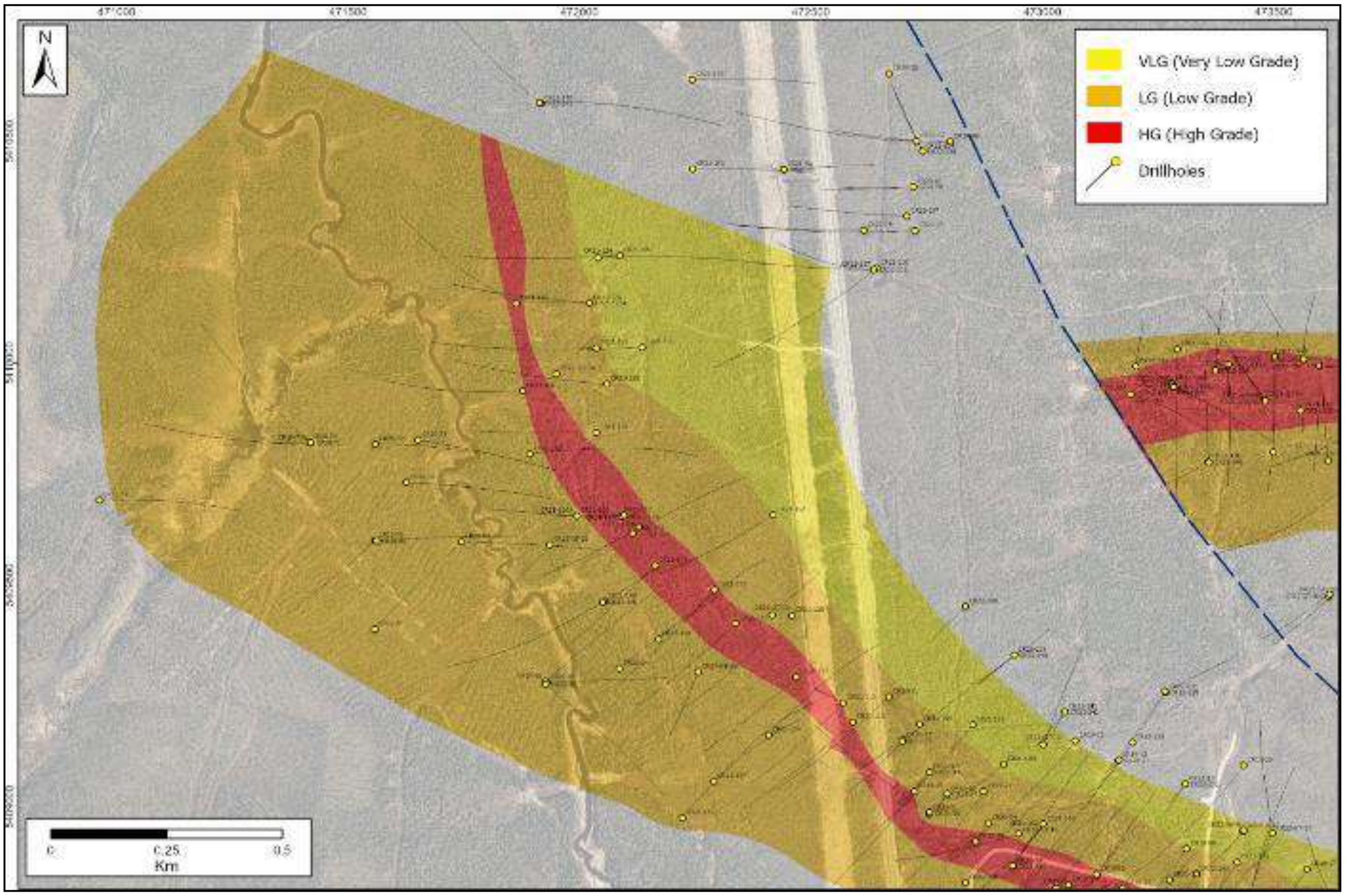
What was previously referred to as the Knuckle Zone (e.g., Ausenco, 2021) is now called the West Zone. In October 2020, CNC reported the discovery of previously unknown mineralization in four drillholes from the West Zone, with the first step-out hole located about 850 m northwest of the Main Zone. Further infill drilling during 2021-2023, and ongoing, has successfully intersected mineralized dunite-peridotite, consistent with mineralization seen in the Main Zone, across a width of at least 800 m and strike length of at least 1 km (Figure 10-6). Selective drill core assay results from the West Zone are summarized in Table 10-6.

**Table 10-6: West Zone: Selective Drill Core Assays, CR19, CR21 and CR22 Series Diamond Drillholes**

| Drillhole | From (m) | To (m) | Interval (m) | Ni (%) | Co (%) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | S (%) | Fe (%) |
|---|---|---|---|---|---|---|---|---|---|---|
| CR19-72 | 46.5 | 342.0 | 295.5 | 0.20 | 0.012 | 0.015 | 0.011 | 0.026 | 0.05 | 7.21 |
| and | 342.0 | 372.0 | 30.0 | 0.29 | 0.014 | 0.043 | 0.023 | 0.066 | 0.07 | 7.38 |
| incl. | 351.0 | 372.0 | 21.0 | 0.31 | 0.014 | 0.045 | 0.026 | 0.071 | 0.09 | 7.37 |
| CR21-110 | 57.0 | 426.0 | 369.0 | 0.23 | 0.013 | 0.007 | 0.011 | 0.018 | 0.02 | 7.39 |
| incl. | 57.0 | 160.5 | 103.5 | 0.29 | 0.012 | 0.018 | 0.024 | 0.042 | 0.03 | 7.21 |
| CR21-120 | 111.0 | 447.0 | 336.0 | 0.23 | 0.012 | 0.003 | 0.004 | 0.007 | 0.05 | 6.99 |
| incl. | 255.5 | 417.5 | 162.0 | 0.27 | 0.012 | 0.003 | 0.004 | 0.007 | 0.04 | 6.80 |
| CR21-122 | 42.3 | 501.0 | 458.7 | 0.24 | 0.012 | 0.005 | 0.004 | 0.009 | 0.02 | 6.64 |
| incl. | 231.0 | 501.0 | 270.0 | 0.28 | 0.012 | 0.006 | 0.005 | 0.011 | 0.02 | 6.15 |
| CR21-126A | 39.0 | 546.0 | 507.0 | 0.24 | 0.013 | 0.007 | 0.006 | 0.013 | 0.04 | 7.03 |
| incl. | 186.0 | 510.0 | 324.0 | 0.27 | 0.012 | 0.006 | 0.005 | 0.011 | 0.04 | 6.94 |
| CR21-128 | 40.9 | 549.0 | 508.1 | 0.24 | 0.013 | 0.017 | 0.012 | 0.029 | 0.05 | 7.27 |
| incl. | 171.5 | 312.5 | 141.0 | 0.26 | 0.012 | 0.005 | 0.004 | 0.009 | 0.05 | 6.99 |
| CR21-130 | 32.8 | 558.0 | 525.2 | 0.24 | 0.013 | 0.016 | 0.010 | 0.026 | 0.06 | 7.04 |
| incl. | 322.5 | 355.5 | 33.0 | 0.34 | 0.016 | 0.035 | 0.019 | 0.054 | 0.15 | 7.64 |
| CR21-132 | 67.0 | 489.0 | 422.0 | 0.24 | 0.014 | 0.016 | 0.015 | 0.031 | 0.02 | 7.38 |
| incl. | 67.0 | 169.5 | 102.5 | 0.29 | 0.013 | 0.052 | 0.033 | 0.085 | 0.07 | 6.96 |
| incl. | 129.0 | 165.0 | 36.0 | 0.36 | 0.013 | 0.125 | 0.077 | 0.202 | 0.12 | 6.97 |
| incl. | 135.0 | 151.5 | 16.5 | 0.43 | 0.013 | 0.104 | 0.018 | 0.122 | 0.15 | 7.16 |
| CR21-135 | 67.0 | 564.0 | 497.0 | 0.24 | 0.012 | 0.012 | 0.011 | 0.023 | 0.03 | 7.14 |
| incl. | 67.0 | 275.5 | 208.5 | 0.27 | 0.011 | 0.024 | 0.015 | 0.039 | 0.04 | 6.74 |
| CR21-137 | 61.0 | 468.0 | 407.0 | 0.21 | 0.013 | 0.003 | 0.006 | 0.009 | 0.02 | 7.18 |
| incl. | 61.0 | 109.5 | 48.5 | 0.25 | 0.013 | 0.003 | 0.005 | 0.008 | 0.02 | 7.15 |
| CR21-139 | 46.5 | 465.5 | 419.0 | 0.25 | 0.014 | 0.021 | 0.014 | 0.035 | 0.04 | 7.38 |
| incl. | 46.5 | 208.0 | 161.5 | 0.27 | 0.014 | 0.014 | 0.008 | 0.022 | 0.05 | 7.43 |
| CR21-141 | 46.5 | 586.5 | 540.0 | 0.22 | 0.012 | 0.003 | 0.005 | 0.008 | 0.04 | 6.79 |
| incl. | 46.5 | 218.5 | 172.0 | 0.25 | 0.012 | 0.004 | 0.005 | 0.009 | 0.05 | 6.30 |
| CR21-144 | 60.0 | 369.0 | 309.0 | 0.29 | 0.015 | 0.034 | 0.011 | 0.045 | 0.10 | 7.14 |
| incl. | 152.5 | 369.0 | 216.5 | 0.33 | 0.015 | 0.026 | 0.010 | 0.036 | 0.12 | 7.26 |
| incl. | 152.5 | 222.0 | 69.5 | 0.43 | 0.016 | 0.040 | 0.012 | 0.052 | 0.15 | 7.31 |
| CR21-145 | 82.0 | 667.5 | 585.5 | 0.21 | 0.013 | 0.009 | 0.010 | 0.019 | 0.06 | 7.29 |
| CR21-148 | 49.0 | 505.0 | 456.0 | 0.25 | 0.014 | 0.015 | 0.014 | 0.029 | 0.04 | 7.42 |
| incl. | 328.0 | 396.0 | 68.0 | 0.31 | 0.014 | 0.061 | 0.035 | 0.096 | 0.06 | 7.53 |
| and | 482.5 | 502.0 | 19.5 | 0.33 | 0.014 | 0.015 | 0.009 | 0.024 | 0.05 | 7.81 |
| and | 373.0 | 391.0 | 18.0 | 0.42 | 0.015 | 0.018 | 0.010 | 0.028 | 0.13 | 7.49 |
| CR21-152 | 63.9 | 535.5 | 471.6 | 0.24 | 0.012 | 0.006 | 0.005 | 0.011 | 0.05 | 7.04 |
| incl. | 262.5 | 477.0 | 214.5 | 0.28 | 0.013 | 0.009 | 0.005 | 0.014 | 0.04 | 7.31 |

Note: Core intervals are not true widths. CNC has insufficient information to determine the attitude, either of the ultramafic body or of mineralized zones within it. True widths will be less than the core intervals by several factors.

**Figure 10-6: West Zone Drillhole Traces for 2020 to 2022 Diamond Drilling**




Source: CNC, 2023

In 2019 and 2020, CNC reported six drillholes (one abandoned) from the northern West Zone, previously known as the Thumb Zone, located about 825 m west-northwest of the East Zone and about 1 km north of the west end of the Main Zone (see northern drillholes outside of the Ni resource envelope in Figure 10-6). Selective drill core assay results from the northern West Zone are summarized in Table 10-7.

**Table 10-7: Northern West Zone: Selective Drill Core Assays, CR19 Series Diamond Drillholes**

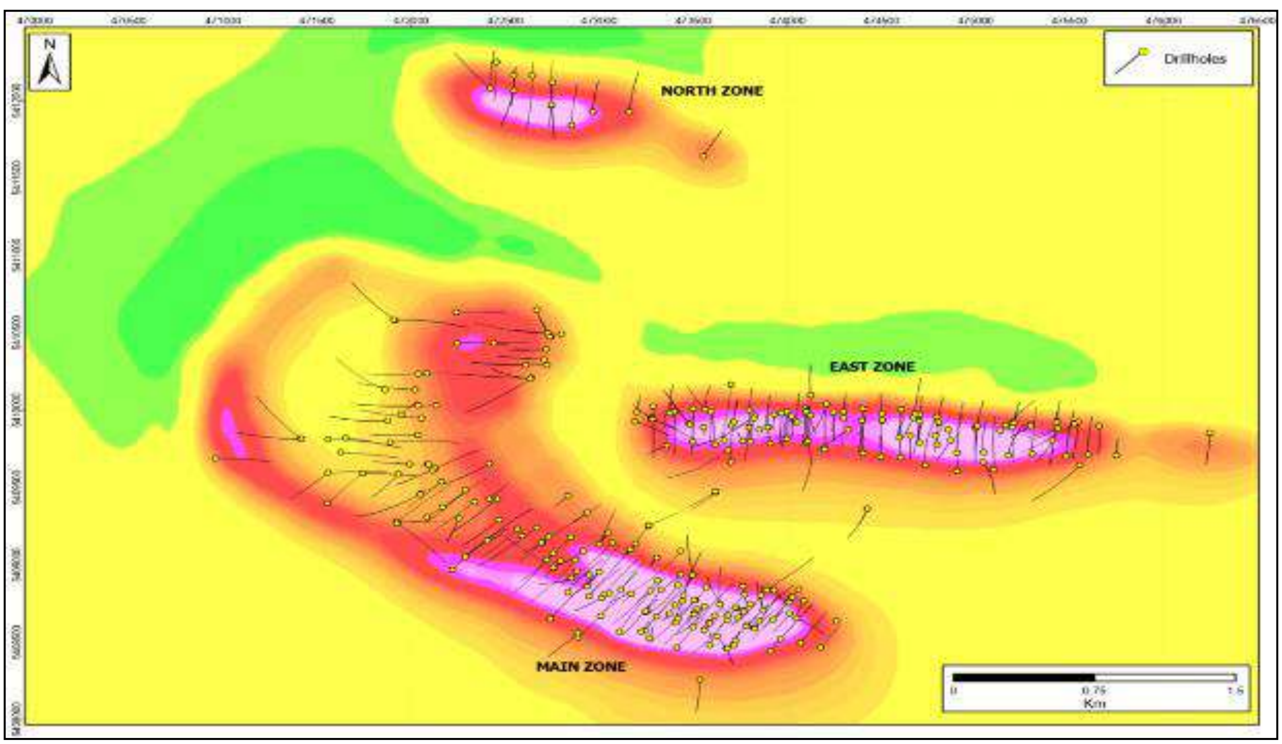
| Drillhole | From (m) | To (m) | Interval (m) | Ni (%) | Co (%) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | S (%) | Fe (%) |
|---|---|---|---|---|---|---|---|---|---|---|
| **CR19-32** | 123.0 | 135.0 | 12.0 | 0.02 | 0.007 | 0.900 | 0.900 | 1.800 | - | - |
| incl. | 123.0 | 130.5 | 7.5 | 0.02 | 0.007 | 1.300 | 1.300 | 2.600 | - | - |
| **and** | 242.0 | 245.0 | 3.0 | 0.03 | 0.007 | 0.900 | 1.000 | 1.900 | - | - |
| **and** | 277.5 | 289.5 | 12.0 | 0.03 | 0.008 | 0.500 | 0.500 | 1.000 | - | - |
| **and** | 280.5 | 286.5 | 6.0 | 0.02 | 0.008 | 0.900 | 0.800 | 1.700 | - | - |
| **and** | 390.0 | 633.0 | 243.0 | 0.25 | 0.013 | 0.003 | 0.003 | 0.006 | 0.02 | 6.10 |
| incl. | 438.0 | 633.0 | 195.0 | 0.27 | 0.013 | 0.003 | 0.003 | 0.006 | 0.02 | 5.80 |
| incl. | 576.0 | 633.0 | 57.0 | 0.30 | 0.013 | 0.003 | 0.003 | 0.006 | 0.01 | 5.88 |

Note: core intervals are not true widths. CNC has insufficient information to determine the attitude, either of the ultramafic body or of mineralized zones within it. True widths will be less than the core intervals by several factors.

### 10.3.5 North Zone Drilling

In December 2020, CNC drill tested previously unknown mineralization in four drillholes from the North Zone, located about 2.0 km northwest of the East Zone (Figure 10-7).

**Figure 10-7: North Zone Drilling and CUC Zones**




Source: CNC, 2023.

A total of 15 exploratory drillholes were completed between 2020 and 2022. All drillholes intersected the faulted continuation of the main dunite-peridotite body of the CUC and exhibited the same differentiation sequence. Selective drill core assay results from the North Zone are summarized in Table 10-8.

**Table 10-8: North Zone: Selective Drill Core Assays, CR20, CR21 and CR22 Series Diamond Drillholes**

| Drill Hole | From (m) | To (m) | Interval (m) | Ni (%) | Co (%) | Pd (g/t) | Pt (g/t) | Pd + Pt (g/t) | S (%) | Fe (%) |
|---|---|---|---|---|---|---|---|---|---|---|
| CR20-84 | 63.0 | 501.0 | 438.0 | 0.24 | 0.01 | 0.003 | 0.003 | 0.005 | 0.02 | 6.00 |
| CR20-87 | 80.0 | 453.0 | 373.0 | 0.21 | 0.01 | 0.003 | 0.004 | 0.007 | 0.02 | 6.65 |
| and | 579.0 | 588.0 | 9.0 | 0.05 | 0.01 | 0.490 | 0.513 | 1.003 | 0.05 | 7.83 |
| CR20-90 | 60.9 | 540.0 | 479.1 | 0.25 | 0.01 | 0.003 | 0.003 | 0.005 | 0.03 | 5.86 |
| CR21-91 | 171.0 | 189.0 | 18.0 | 0.03 | 0.01 | 0.395 | 0.339 | 0.735 | 0.09 | 6.02 |
| CR21-93 | 76.4 | 334.5 | 258.1 | 0.26 | 0.01 | 0.003 | 0.003 | 0.006 | 0.02 | 5.98 |
| CR21-95 | 58.5 | 444.0 | 385.5 | 0.21 | 0.01 | 0.004 | 0.004 | 0.007 | 0.02 | 6.66 |
| CR22-213 | 74.6 | 540.0 | 465.4 | 0.18 | 0.01 | 0.005 | 0.004 | 0.009 | 0.01 | 7.28 |
| CR22-214 | 63.0 | 313.5 | 250.5 | 0.24 | 0.01 | 0.007 | 0.006 | 0.013 | 0.02 | 5.54 |
| CR22-216 | 63.0 | 486.0 | 423.0 | 0.23 | 0.01 | 0.003 | 0.004 | 0.007 | 0.02 | 6.57 |
| CR22-218 | 71.7 | 431.0 | 359.3 | 0.25 | 0.01 | 0.010 | 0.013 | 0.024 | 0.02 | 6.20 |
| CR22-223 | 66.2 | 421.0 | 354.8 | 0.24 | 0.01 | 0.003 | 0.009 | 0.012 | 0.04 | 6.10 |

Note: core intervals are not true widths. CNC has insufficient information to determine the attitude, either of the ultramafic body or of mineralized zones within it. True widths will be less than the core intervals by several factors.

# 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

## 11.1 Introduction

CNC is responsible for the ongoing drilling and sampling program, including quality assurance and quality control (QA/QC). Since drilling began in 2019, the protocols followed by company personnel have changed slightly and are described below and are current as of the effective date of the mineral resource estimate (August 31, 2023).

The core is marked and sampled at primarily 1.5-metre lengths and cut with diamond blade saws. Samples are bagged with QA/QC samples inserted into the sample stream (60 samples per lab batch) at a the rate of at least 1 QA/QC sample in every 20 principal samples. Samples (60 per lot) are transported in secure bags directly from the company core shack to Activation Laboratories Ltd. (Actlabs) in Timmins or by commercial truck transport (Manitoulin Transport Inc.) to SGS Canada Inc. (SGS) in Lakefield, Ontario. In general, the core recovery for the diamond drillholes on the property has been better than 95% and little core loss due to poor drilling methods or procedures has been experienced.

In the opinion of the authors, the assay data is adequate for the purpose of verifying drill core assays, estimating mineral resources, and for use in a feasibility study.

## 11.2 Sample Collection and Transportation

Core (NQ size core, 47.6 mm diameter) is collected from the drill into core boxes and secured in closed core trays at the drill site by the drilling contractor (NPLH Drilling of Timmins, Ontario), following industry standard procedures. Small wooden tags mark the distance drilled in metres at the end of each run. On each filled core box, the drillhole number and sequential box numbers are marked by the drill helper and checked by the site geologist. Once filled and identified, each core tray is covered and secured shut.

Core is delivered by the drilling contractor to the secured laydown area inside the gravel loop road off of Highway 655; the entrance to the gravel road is gated. CNC personnel transport the core to the core shack from the laydown area. Casing is left in the completed drillholes with the casing capped, labelled, and marked with a metal flag.

## 11.3 Core Logging and Sampling Procedures

CNC originally used a rented core shack in Timmins (3700 Highway 101 West), a driving distance of approximately 50 km from the project area access point. CNC has since rented a larger facility that also provides office space at 170 Jaguar Drive in Timmins, which is marginally closer to the project area. This section describes the protocols followed at the latter facility.

Once the core boxes arrive at the logging facility in Timmins, it is stored sequentially, hole by hole, in racks ahead of the logging process. As required, core boxes they are laid out on the logging table in order and the lids are removed. The core logging process consists of two major parts: geotechnical logging and geological logging.

Core is first turned and aligned to be sure the same side of the core is being marked, cut, and sampled. Core is measured and the nominal sampling interval of 1.5 metres is marked and tagged for the entirety of the drillhole by a geotechnician. Samples are identified by inserting two identical prefabricated, sequentially numbered, weather-resistant sample tags at the end of each sample interval. Magnetic susceptibility is measured at every three-metre block, taking a minimum of two readings (averaged) and a third reading if the first two readings are significantly different. The relative density of core samples (specific gravity or "SG") is calculated from core in one out of every four core boxes that contain the target ultramafic rocks (dunite (10,402 samples) and peridotite (2,905 samples)). The geotechnician writes the SG measurement directly on the core that is measured. The logging geologist determines if additional SG measurements need to be made. In addition to the target ultramafic rocks, SG measurements were collected from pyroxenite (371 samples), gabbro (551 samples) metavolcanics (424 samples), and dikes (88 samples).

Geological core logging records the lithology, alteration, texture, colour, mineralization, structure, and sample intervals and pays particular attention to the target rock types (dunite and/or peridotite). Originally, all geotechnical logging, geological logging and sample data were recorded directly into a Microsoft Excel spreadsheet. Currently, core logging is done directly into MX Deposit, cloud-based logging software (Sequent). As the core is logged, the target rock type (dunite and/or peridotite) is marked for sampling at a nominal sample interval of 1.5 metres, with the entire intercept of ultramafic rocks sampled in each drillhole.

Once the core is logged and photographed, the core boxes are returned to the indoor storage racks prior to being transferred to the cutting room for sampling on a box-by-box basis.

Sections marked for sampling are cut in half with a diamond saw located in a separate cutting room adjacent to the logging area; three saws are available for use. The core-cutting room has been modified with a ventilation system to mitigate the possible circulation of chrysotile mineral fibres in the air. Personnel working in the room are also required to wear appropriate personal protection equipment (PPE). Once the core is cut in half it is returned to the core box. A geotechnician consistently selects the same half of the core in each interval/hole, placing the half core in a sample bag with one of the corresponding sample tags, and sealing the bag with a cable tie. Bags are also marked externally with the sample tag number. The boxes containing the remaining half core are transferred to outdoor core racks on site in the secure core storage facility.

Individual samples are placed in large, polypropylene bags (rice bags), five samples to a bag, and the larger bag is secured with a cable tie. CNC personnel are responsible for loading and transporting the samples to the Actlabs Timmins analytical facility by truck, a driving distance of approximately 3 km from the core shack location.

## 11.4 Analytical

Activation Laboratories Ltd. (Actlabs), a geochemical services company accredited to international standards, with assay laboratory ISO 17025 certification, certification to ISO 9001:2008 and CAN-P-1579 (Mineral Analysis), was used for most of the analytical requirements related to the project. The Actlabs laboratory in Timmins, Ontario carried out the sample login/registration, sample weighing, sample preparation and analyses. Actlabs certificates and report numbers are prefixed with an "A" and year designation (e.g., A19-, A20- etc.)

SGS Canada Inc. (SGS), likewise a geochemical services company accredited to the same international standards as Actlabs, was used for some of the analytical requirements as the Actlabs facility became overtaxed with service requests. Sample preparation by SGS was carried out in Lakefield, Ontario while analyses were performed at SGS' facilities in Burnaby, BC with some analyses being performed at SGS' facilities in Lima, Perú. SGS certificates and report numbers are prefixed with a "BBM" and year designation (e.g., BBM21-) for the Burnaby laboratory or "GQ" for the laboratory in Lima.

Actlabs and SGS are both independent of CNC, Noble, Spruce Ridge, and the authors.

Platinum group elements (PGE) palladium (Pd) and platinum (Pt), and precious metal gold (Au) were analyzed using a fire assay (FA) digestion of 30 g of sample material followed by an ICP-OES determination of concentration. Base metals and other elements (total of 20 elements are reported herein including Al, As, Be, Ca, Co, Cr, Cu, Fe, K, Li, Mg, Mn, Ni, Pb, S, Sb, Si, Ti, W, Zn) were determined by ICP-OES following a sodium peroxide ($Na_2O_2$) fusion digestion. The sodium peroxide fusion method is suitable for the "total" digestion of refractory minerals and samples with high sulphide content. Select samples have been analyzed for total S by combustion and infrared absorption techniques (SGS laboratory only). Detection limits for all elements at Actlabs and SGS are summarized in Tables 11-1 and 11-2. Differences between the instrumental detection limits can have a profound influence on the relative difference between analyses at low levels of elemental concentration. Samples from recent (2023) diamond drilling (not part of this report) also include total carbon analyses by infrared absorption methods; these sample results will ultimately be included in carbon sequestration studies being initiated by CNC.

For statistical purposes within the report, any analytical result that was reported to be less than the detection limit was set to one half of that detection limit (e.g., a result reported as <0.5 was set to a numeric value of 0.25). Results reported to be greater than the maximum value reportable, and where no corresponding over limit analysis was performed, were set to that maximum value (e.g., a result reported as >15.0 was set to a numeric value of 15).

**Table 11-1: Lower Limits of Detection for Elements Measured at Actlabs**

| Element | Method | LLD | Unit | Element | Method | LLD | Unit |
|---|---|---|---|---|---|---|---|
| Au | FA-ICP | 2 | ng/g (ppb) | Li | FUS-Na-$_2O_2$ | 0.01 | % |
| Pt | FA-ICP | 5 | ng/g (ppb) | Mg | FUS-Na-$_2O_2$ | 0.01 | % |
| Pd | FA-ICP | 5 | ng/g (ppb) | Mn | FUS-Na-$_2O_2$ | 0.01 | % |
| Al | FUS-Na-$_2O_2$ | 0.01 | % | Ni | FUS-Na-$_2O_2$ | 0.005 | % |
| As | FUS-Na-$_2O_2$ | 0.01 | % | Pb | FUS-Na-$_2O_2$ | 0.01 | % |
| Be | FUS-Na-$_2O_2$ | 0.001 | % | S | FUS-Na-$_2O_2$ | 0.01 | % |
| Ca | FUS-Na-$_2O_2$ | 0.01 | % | Sb | FUS-Na-$_2O_2$ | 0.01 | % |
| Co | FUS-Na-$_2O_2$ | 0.002 | % | Si | FUS-Na-$_2O_2$ | 0.01 | % |
| Cr | FUS-Na-$_2O_2$ | 0.01 | % | Ti | FUS-Na-$_2O_2$ | 0.01 | % |
| Cu | FUS-Na-$_2O_2$ | 0.005 | % | W | FUS-Na-$_2O_2$ | 0.005 | % |
| Fe | FUS-Na-$_2O_2$ | 0.05 | % | Zn | FUS-Na-$_2O_2$ | 0.01 | % |
| K | FUS-Na-$_2O_2$ | 0.1 | % | | | | |

Notes: FA-ICP=fire assay with ICP-OES finish. FUS-$Na_2O_2$=sodium peroxide fusion digestion with ICP-OES finish. %= percent by weight.

**Table 11-2: Lower Limits of Detection for Elements Measured at SGS**

| Element | Method | LLD | Unit | Element | Method | LLD | Unit |
|---|---|---|---|---|---|---|---|
| Au | FA-ICP | 5 | ng/g (ppb) | Li | FUS-Na-$_2$O$_2$ | 0.001 | % |
| Pt | FA-ICP | 10 | ng/g (ppb) | Mg | FUS-Na-$_2$O$_2$ | 0.01 | % |
| Pd | FA-ICP | 5 | ng/g (ppb) | Mn | FUS-Na-$_2$O$_2$ | 0.001 | % |
| Al | FUS-Na-$_2$O$_2$ | 0.01 | % | Ni | FUS-Na-$_2$O$_2$ | 0.001 | % |
| As | FUS-Na-$_2$O$_2$ | 0.003 | % | Pb | FUS-Na-$_2$O$_2$ | 0.002 | % |
| Be | FUS-Na-$_2$O$_2$ | 0.0005 | % | S | FUS-Na-$_2$O$_2$ | 0.01 | % |
| Ca | FUS-Na-$_2$O$_2$ | 0.1 | % | S | IR | 0.005 | % |
| Co | FUS-Na-$_2$O$_2$ | 0.001 | % | Sb | FUS-Na-$_2$O$_2$ | 0.005 | % |
| Cr | FUS-Na-$_2$O$_2$ | 0.001 | % | Si | FUS-Na-$_2$O$_2$ | 0.1 | % |
| Cu | FUS-Na-$_2$O$_2$ | 0.001 | % | Ti | FUS-Na-$_2$O$_2$ | 0.01 | % |
| Fe | FUS-Na-$_2$O$_2$ | 0.01 | % | W | FUS-Na-$_2$O$_2$ | 0.005 | % |
| K | FUS-Na-$_2$O$_2$ | 0.1 | % | Zn | FUS-Na-$_2$O$_2$ | 0.001 | % |

Notes: FA-ICP=fire assay with ICP-OES finish. FUS-$Na_2O_2$=sodium peroxide fusion digestion with ICP-OES finish. IR=infrared combustion method. %= percent by weight

## 11.5 QA/QC – Control Samples

CNC began introducing its own internal QA/QC samples into the sample stream approximately halfway through the 2019-2020 drilling program (starting with drillhole CR19-11). Prior to this point, CNC relied upon Actlabs' own use of internal monitoring of quality control to service the overall quality control of the project.

A total of 85,700 samples have been submitted for analysis by CNC since the start of the project. This includes 3,974 samples from drilling in the North Zone, 1,269 samples from diamond drilling carried out in 2023 (some results still pending) plus additional separate sampling for carbon sequestration baseline studies (74 samples).

For the purposes of this study, a total of 80,383 samples are considered. This includes 50,607 samples associated with the Main and West zones (collectively, the Main Zone) and 29,776 samples associated with the East Zone. No distinction herein has been made between the zones with respect to QA/QC, as a previous report did not indicate any appreciable difference in the QA/QC results between the zones and furthermore, the timeline for analytical results from the different zones was noted to be intermixed. Included in the sample total are 7,210 “control” samples (either a blank or CRM sample), 3,572 pulp duplicates (“replicates”) and 67 field duplicates for a total inclusion rate of 13.5%. CNC originally (as previously stated, starting with drillhole CR19-11) included QA/QC samples at the approximate rate of three samples per batch of 35 samples shipped to the laboratory (8.6%); the current rates of QA/QC sample submission are completely in-line with the rate recommended for the project (15%).

Actlabs and SGS insert internal certified reference material into the sample stream, run blank aliquots, and carry out duplicate and replicate (“preparation split”) analyses within each sample batch as part of its own internal monitoring of quality control. While CNC previously relied solely on the laboratory-provided control results to monitor the quality of the analytical results, the Company now carries out sufficient QA/QC monitoring of the laboratory results on its own account.

CNC have inserted six different samples of CRM into the sample stream: OREAS 683 (PGE ore; 330 samples), OREAS 70b (nickel sulphide ore; 1,626 samples), OREAS 70p (588 samples), OREAS 72a (nickel sulphide ore; 388 samples), OREAS 72b (nickel sulphide ore; 417 samples), and OREAS 74a (nickel sulphide ore; 40 samples).

CNC introduced 3,427 samples of blank material (variously “blank gravel”, “blank FP”, “blank sand’ or “blank silica”) into the sample stream.

CNC did quarter core-sample intervals for 65 samples to generate sampling or field duplicates. Replicate analyses of 3,365 prepared sample pulps was carried out in order to evaluate the reproducibility of the sampling procedures. Early in the program CNC did submit 320 core pulp samples to a referee lab, Bureau Veritas Commodities Canada Ltd. (see Section 11.6.4).

## 11.6 QA/QC – Data Verification

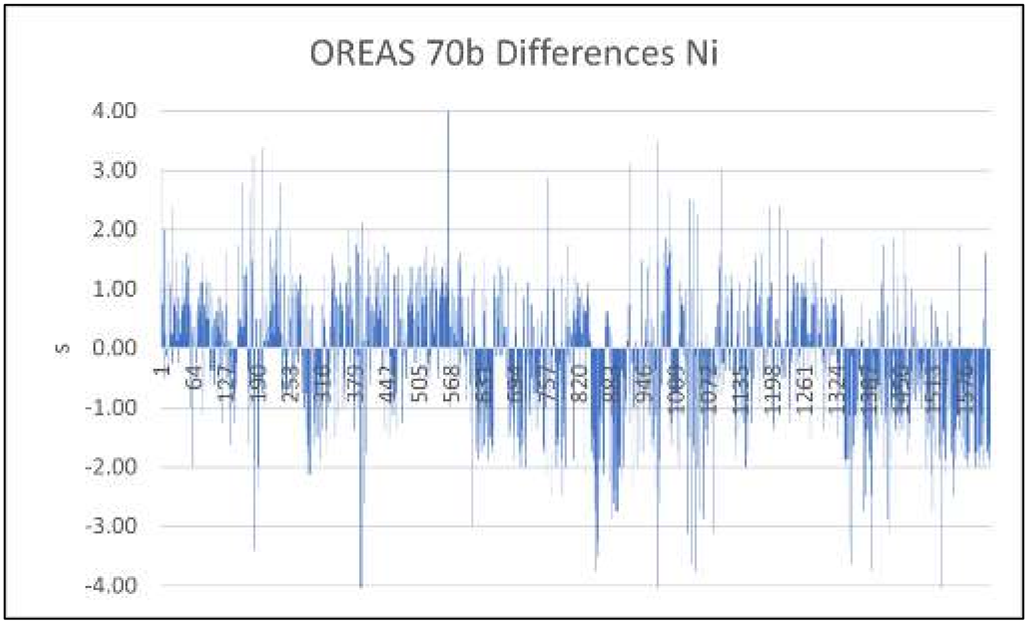
### 11.6.1 Certified Reference Material

Certified reference materials are used by CNC to monitor the accuracy of the analyses performed by Actlabs and SGS. Several different reference materials for combinations of elements were used during the analytical work being reported on herein. For the purposes of the report, we have focused on the results of the most frequently used reference materials submitted for analysis by CNC, namely OREAS 70b, OREAS 72a and OREAS 683; they reported certified values in the expected concentration ranges similar to the samples of drill core that was submitted to for analysis. It should be noted though that CRM OREAS 70P does not have certified reference values for analyses that include a sodium peroxide fusion digestion; in addition, the certified reference values for Pd and Pt are below the detection limits while that for Au is very low (13 ppb Au) for the chosen analytical method.

It is observed that in general the analyses for the certified reference material examined in detail averaged within two standard deviations of the certified concentrations over the span of the laboratory work and that, over time, averaged close to their certified concentration; this gives reason that the accuracy of the analyses be considered as acceptable.

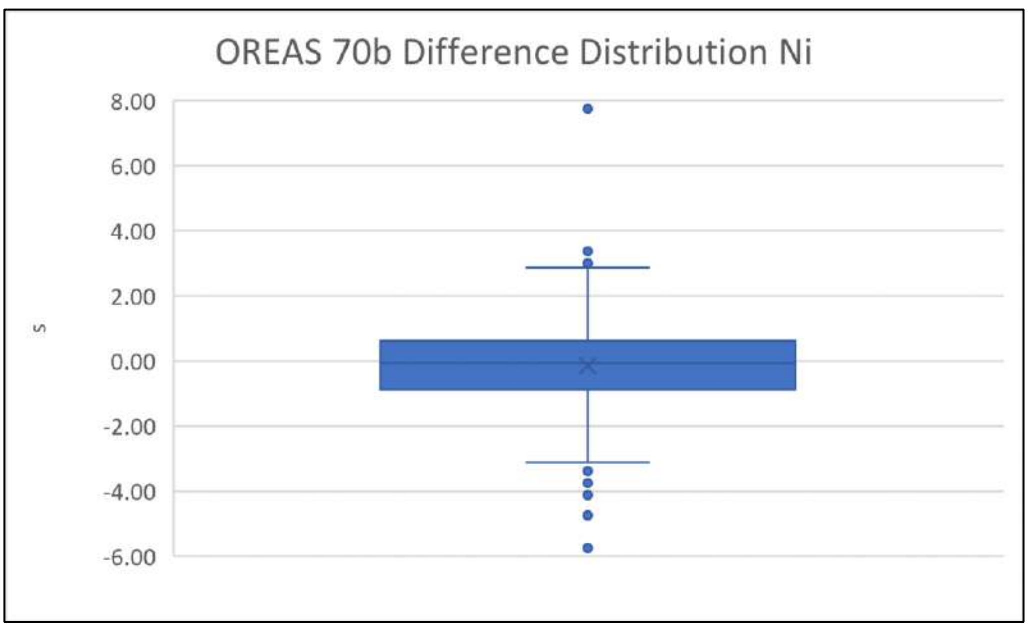
Examples of the CRM responses are shown in Figures 11-1 to 11-16.

**Figure 11-1: CRM OREAS 70b – Number of Standard Deviations Difference for Ni Analysis from the Certified Value for Various Analytical Runs**



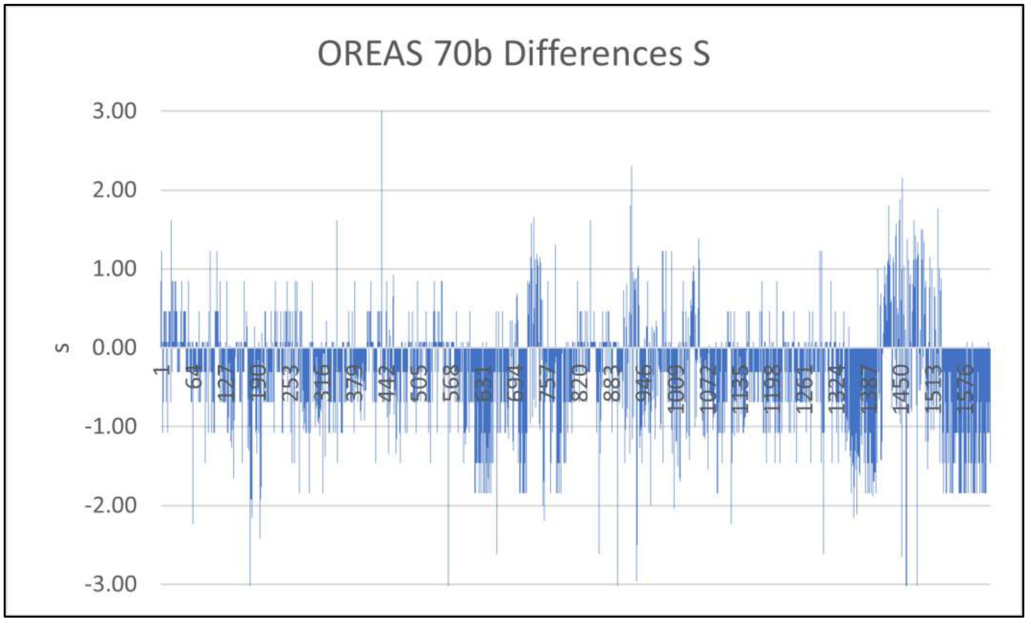

Source: Caracle Creek, 2023.

**Figure 11-2: CRM OREAS 70b – Distribution of Standard Deviations Difference for Ni Analysis from the Certified Value**



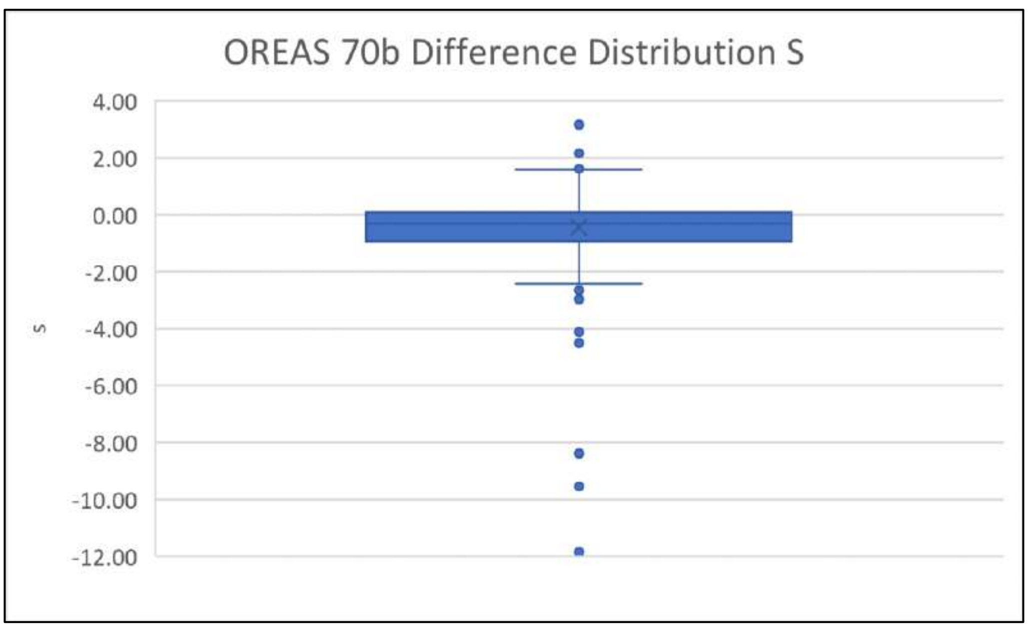

Source: Caracle Creek, 2023.

**Figure 11-3: CRM OREAS 70b – Number of Standard Deviations Difference for S Analysis from the Certified Value for Various Analytical Runs**



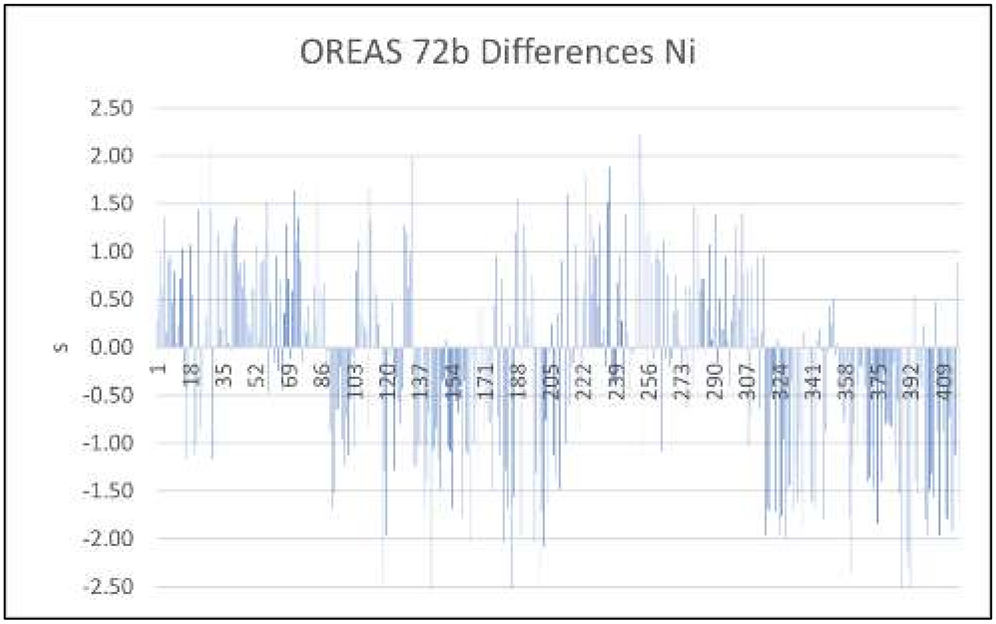

Source: Caracle Creek, 2023.

**Figure 11-4: CRM OREAS 70b – Distribution of Standard Deviations Difference for S Analysis from the Certified Value**



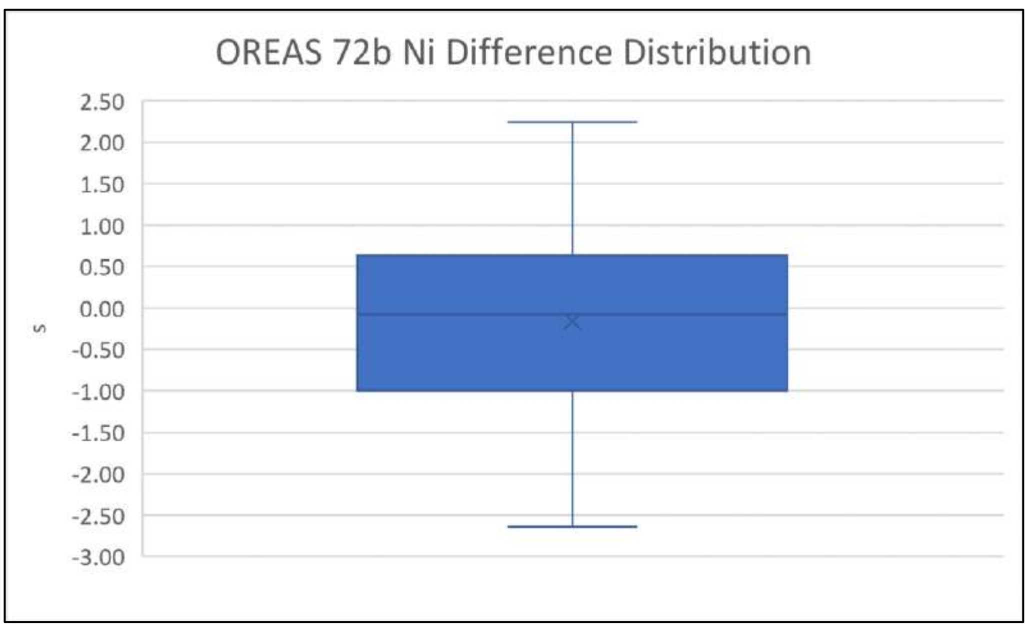

Source: Caracle Creek, 2023.

**Figure 11-5: CRM OREAS 72b – Number of Standard Deviations Difference for Ni Analysis from the Certified Value for Various Analytical Runs**



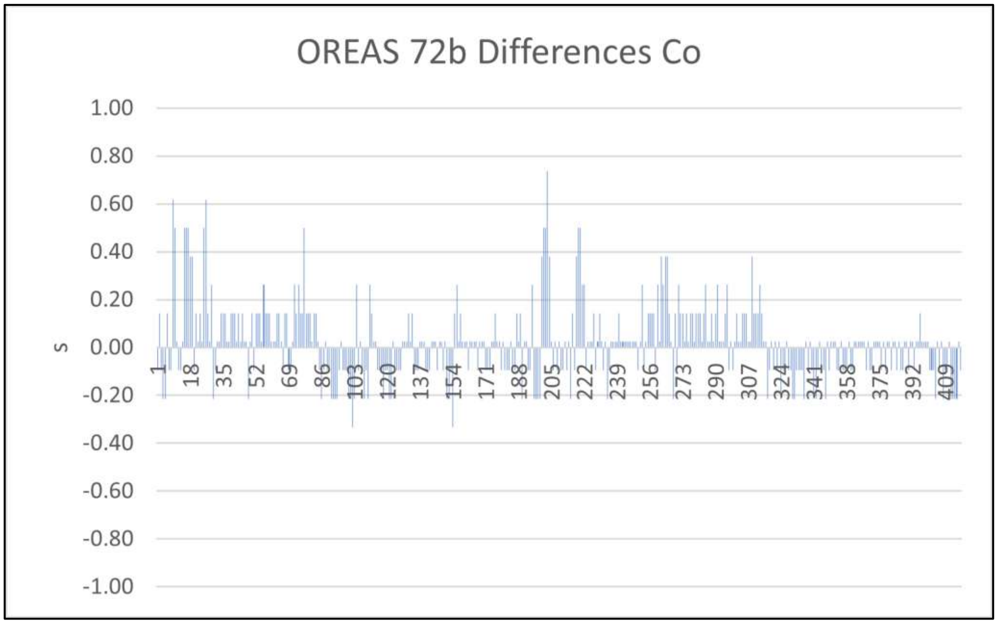

Source: Caracle Creek, 2023.

**Figure 11-6: CRM OREAS 72b – Distribution of Standard Deviations Difference for Ni Analysis from the Certified Value**



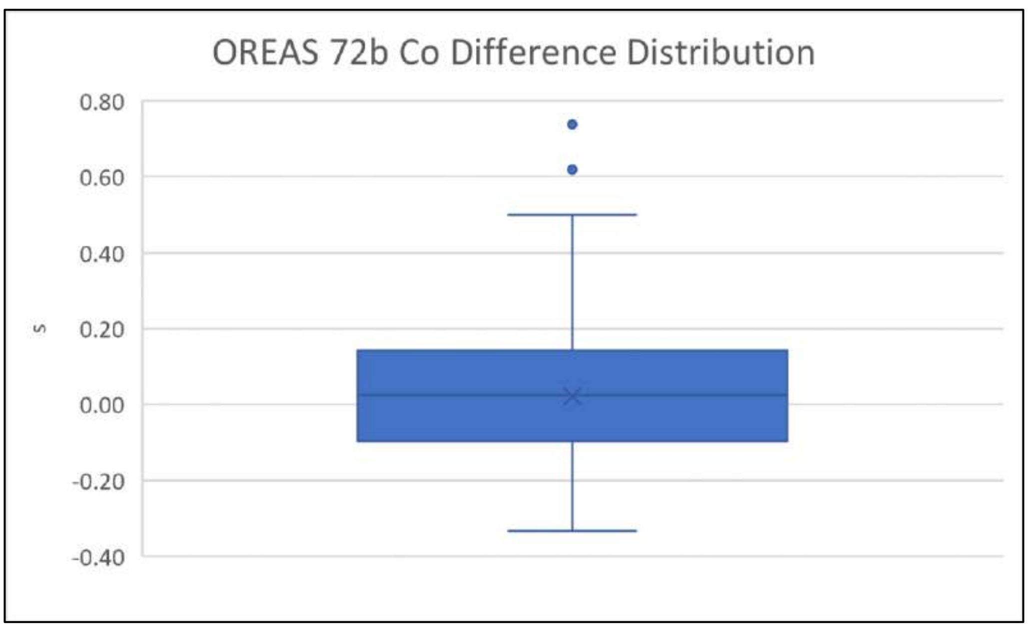

Source: Caracle Creek, 2023.

**Figure 11-7: CRM OREAS 72b – Number of Standard Deviations Difference for Co Analysis from the Certified Value for Various Analytical Runs**



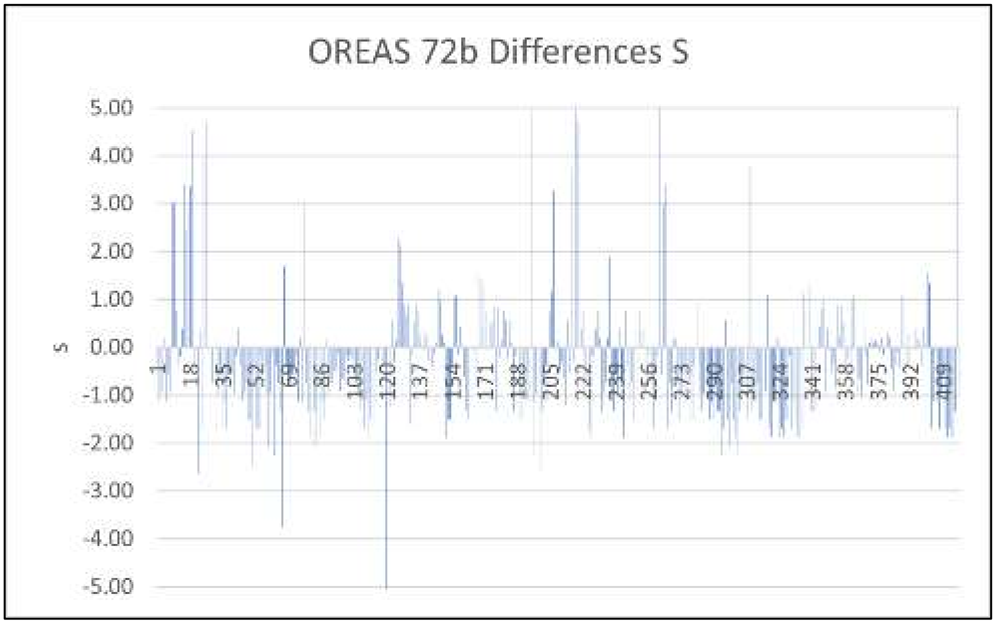

Source: Caracle Creek, 2023.

**Figure 11-8: CRM OREAS 72b – Distribution of Standard Deviations Difference for Co Analysis from the Certified Value**



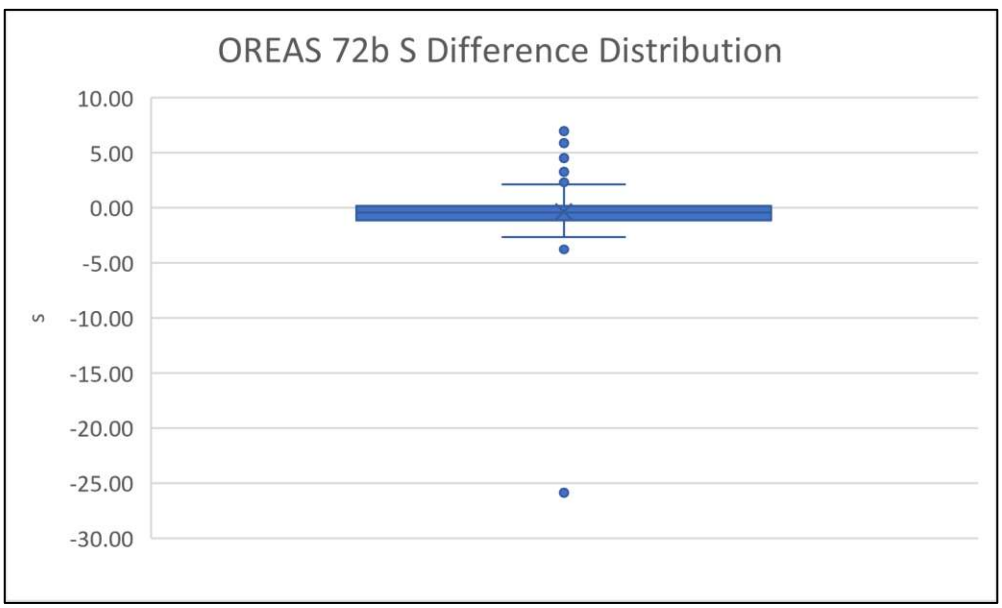

Source: Caracle Creek, 2023.

**Figure 11-9: CRM OREAS 72b – Distribution of Standard Deviations Difference for S Analysis from the Certified Value**



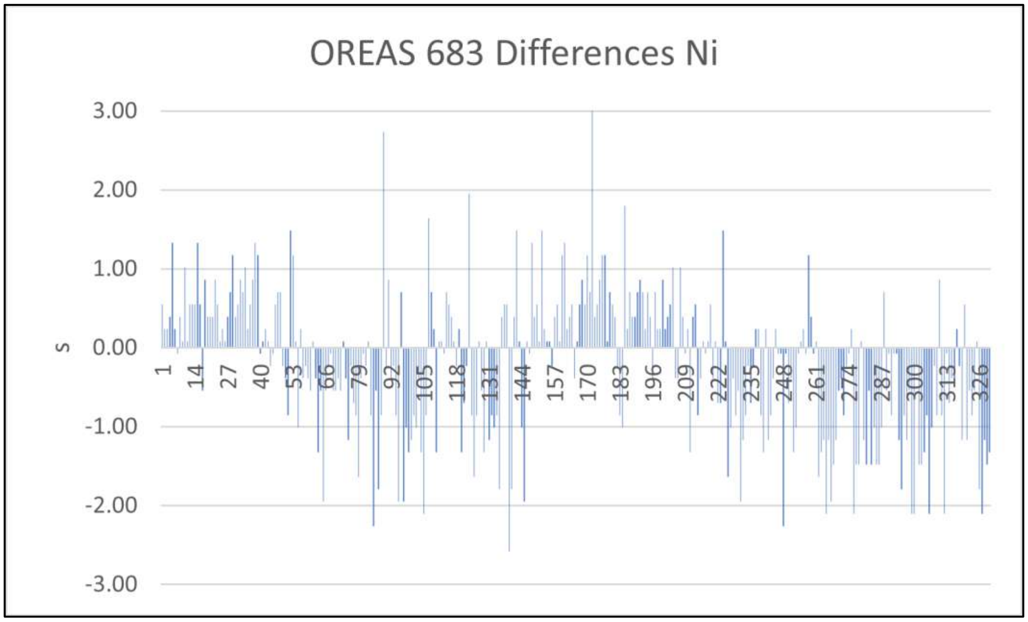

Source: Caracle Creek, 2023.

**Figure 11-10: CRM OREAS 72b – Distribution of Standard Deviations Difference for S Analysis from the Certified Value**



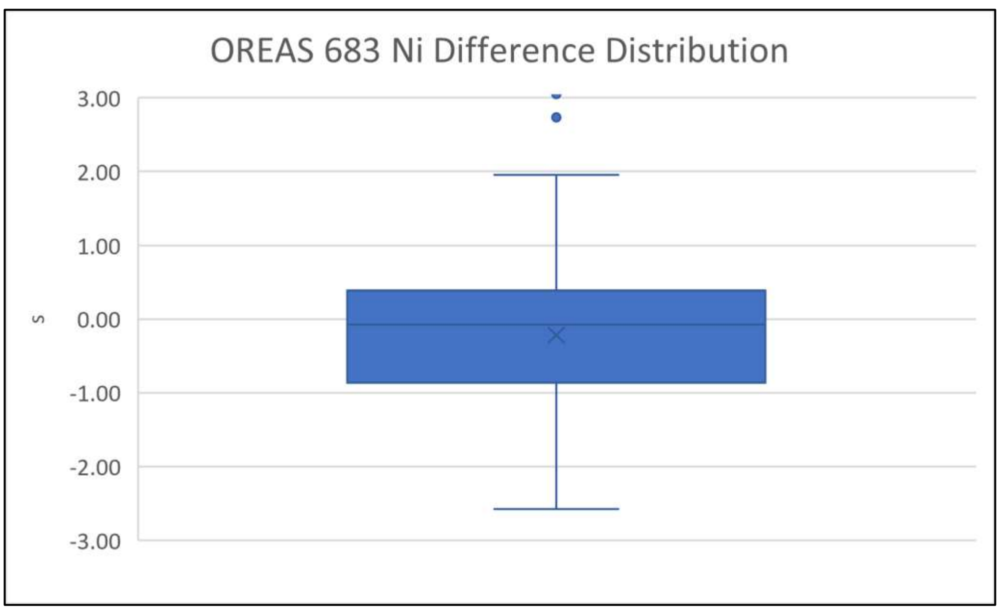

Source: Caracle Creek, 2023.

**Figure 11-11: CRM OREAS 683 – Distribution of Standard Deviations Difference for Ni Analysis from the Certified Value**



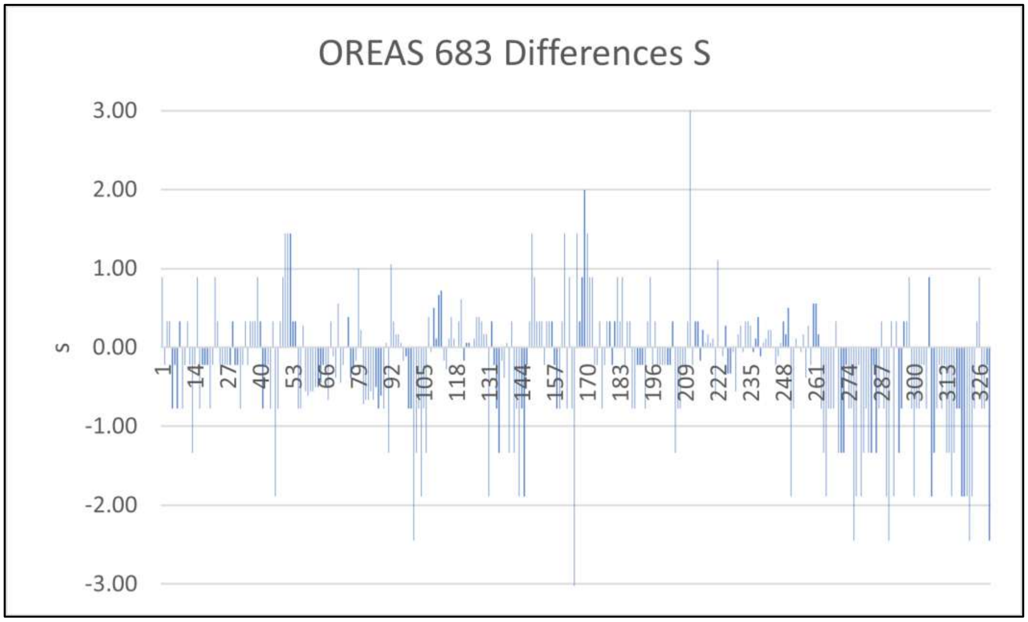

Source: Caracle Creek, 2023.

**Figure 11-12: CRM OREAS 683 – Distribution of Standard Deviations Difference for Ni Analysis from the Certified Value**



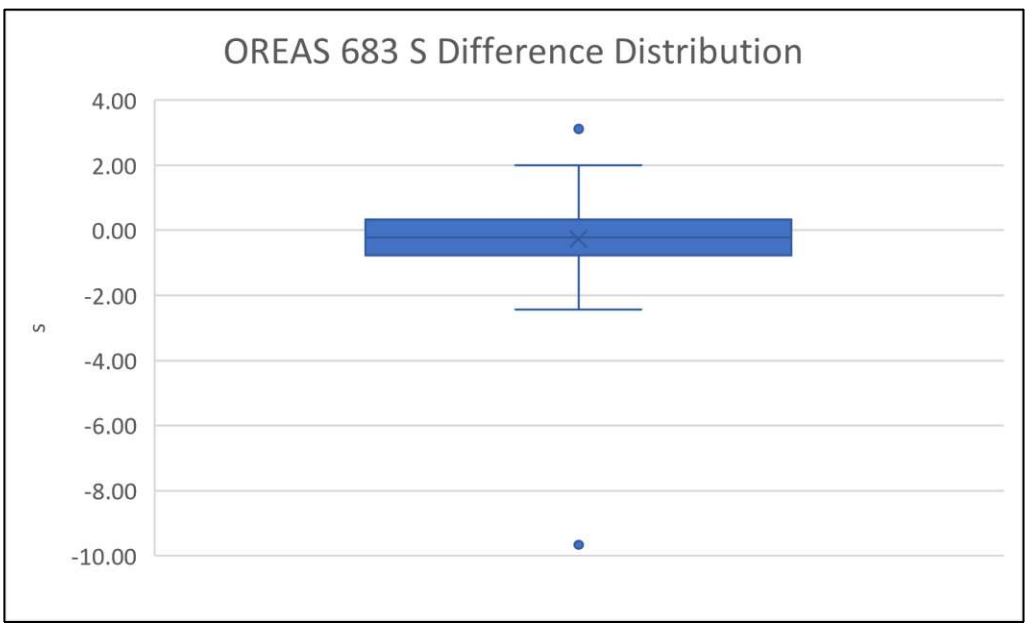

Source: Caracle Creek, 2023.

**Figure 11-13: CRM OREAS 683 – Number of Standard Deviations Difference for S Analysis from the Certified Value for Various Analytical Runs**



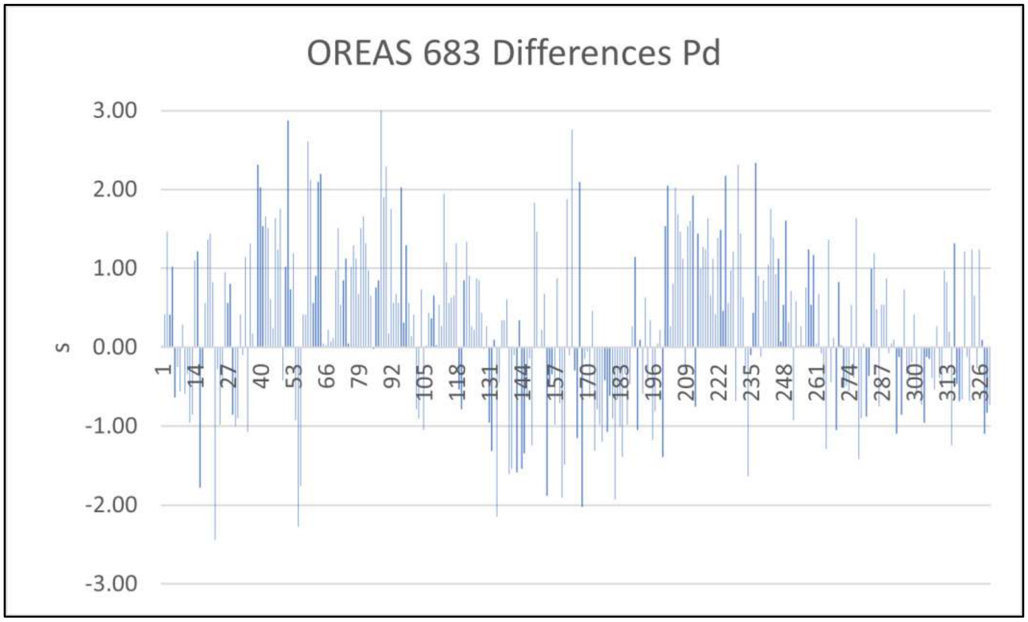

Source: Caracle Creek, 2023.

**Figure 11-14: CRM OREAS 683 – Distribution of Standard Deviations Difference for S Analysis from the Certified Value**



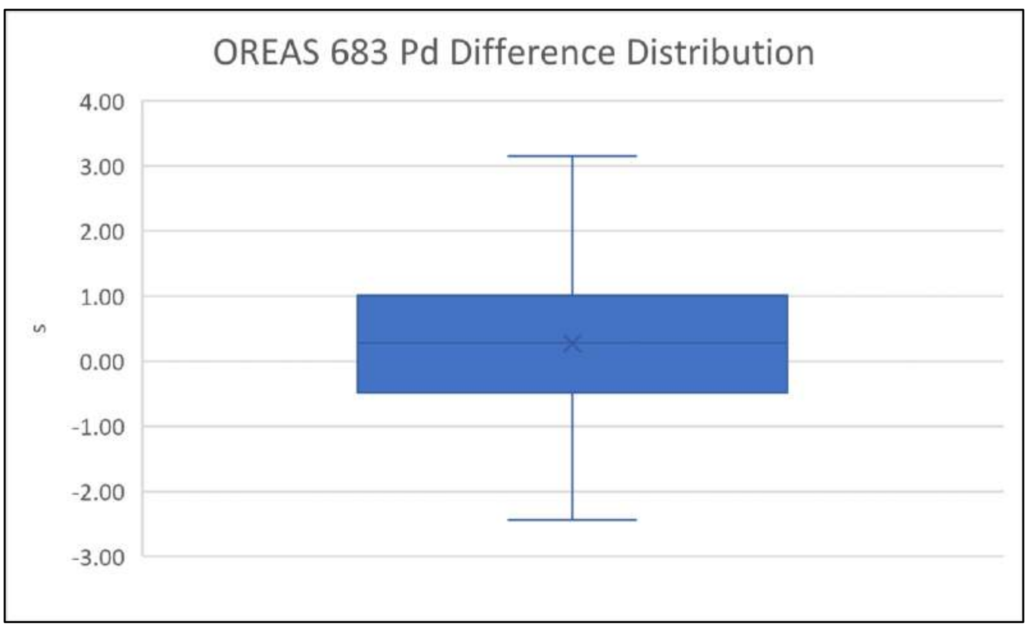

Source: Caracle Creek, 2023.

**Figure 11-15: CRM OREAS 683 – Number of Standard Deviations Difference for Pd Analysis from the Certified Value for Various Analytical Runs**




Source: Caracle Creek, 2023.

**Figure 11-16: CRM OREAS 683 – Distribution of Standard Deviations Difference for Pd Analysis from the Certified Value – East Zone**

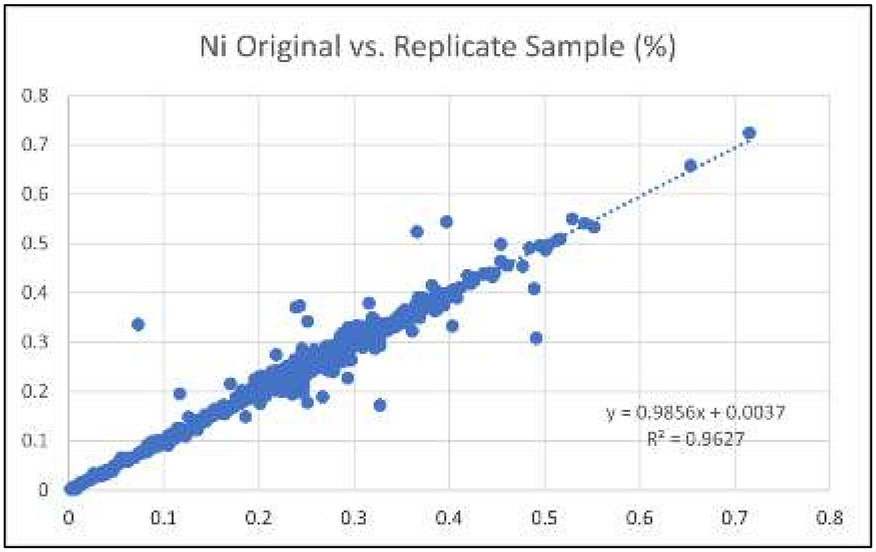



Source: Caracle Creek, 2023.

### 11.6.2 Replicate Samples – Analytical Duplicates

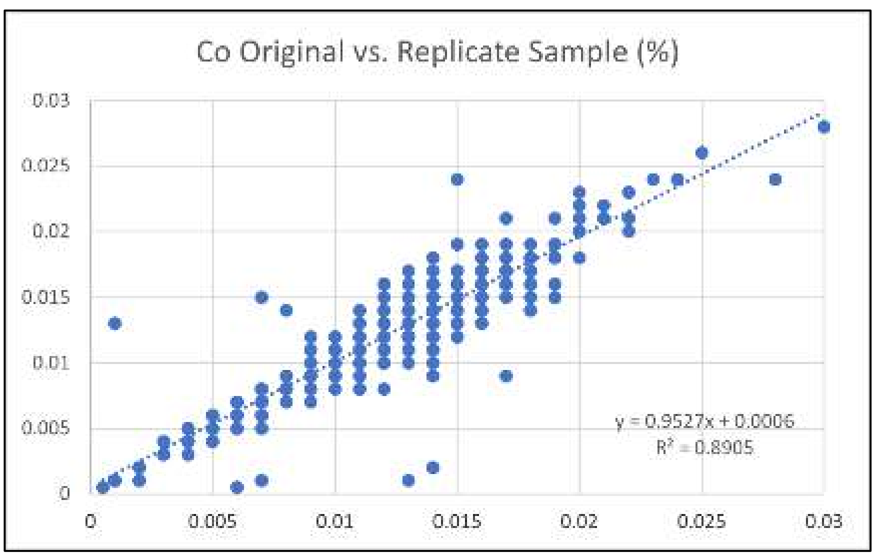
CNC, on a regular basis (about 4%), requested the labs to re-analyze prepared sample pulps. The replicate material has indicated good reproducibility of the assays as demonstrated by project examples in Figures 11-17 to 11-22.

**Figure 11-17: Plot of Absolute Concentrations of Pairs of Replicate Samples Analyzed for Ni**



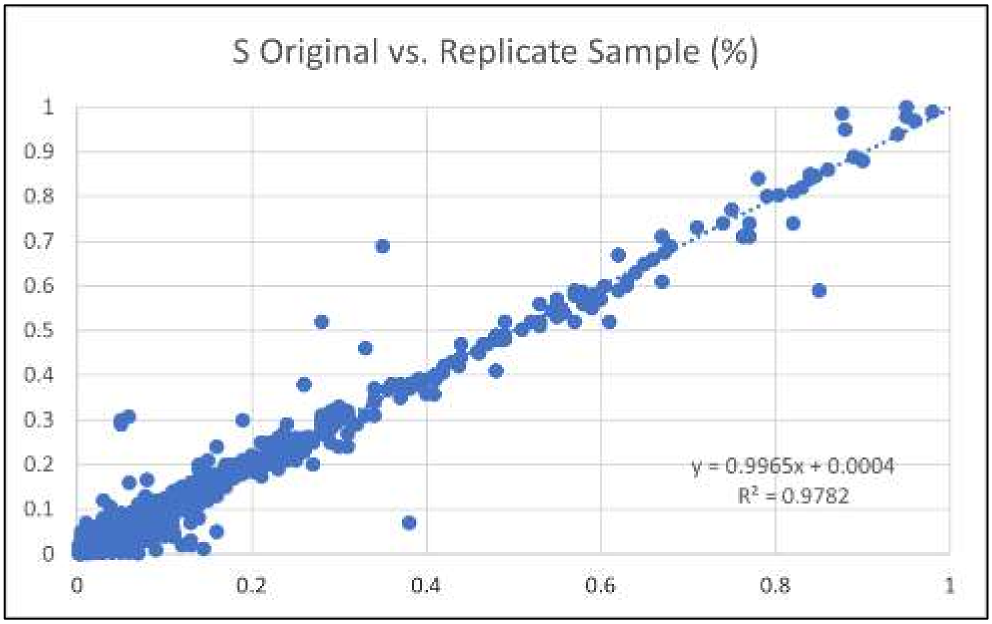

Source: Caracle Creek, 2023.

**Figure 11-18: Plot of Absolute Concentrations of Pairs of Replicate Samples Analyzed for Co**



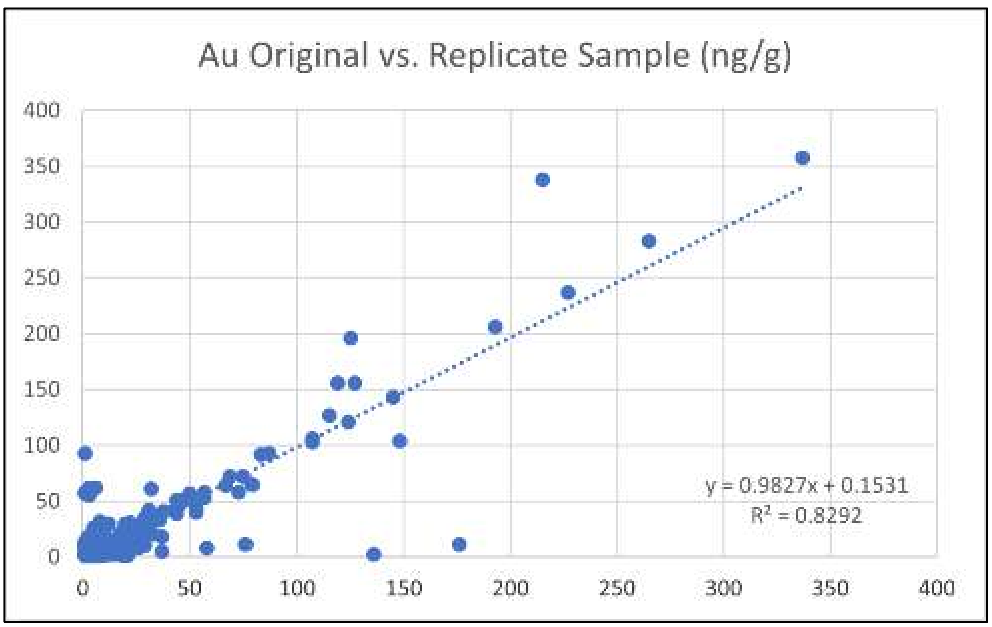

Source: Caracle Creek, 2023.

**Figure 11-19: Plot of Absolute Concentrations of Pairs of Replicate Samples Analyzed for S**



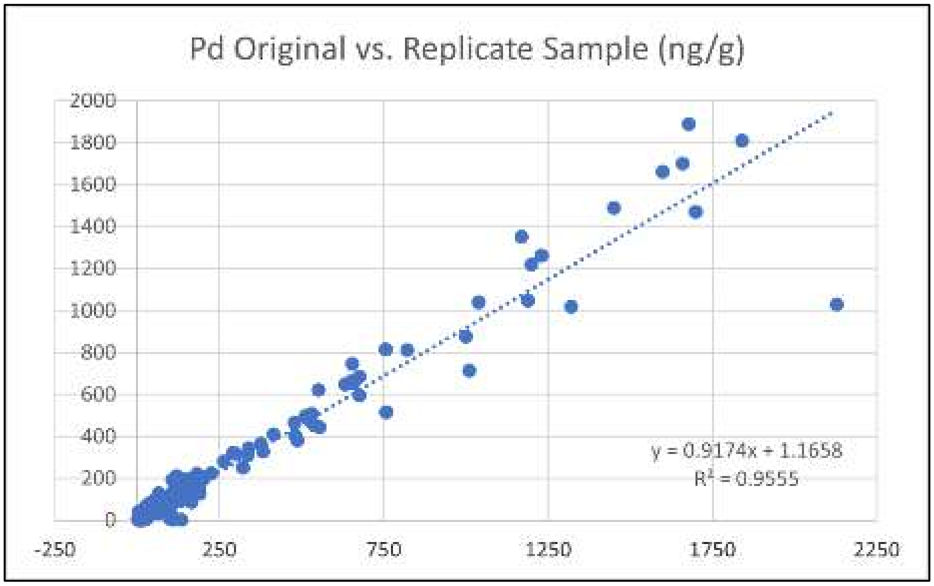

Source: Caracle Creek, 2023.

**Figure 11-20: Plot of Absolute Concentrations of Pairs of Replicate Samples Analyzed fo Au (ng/g = ppb)**



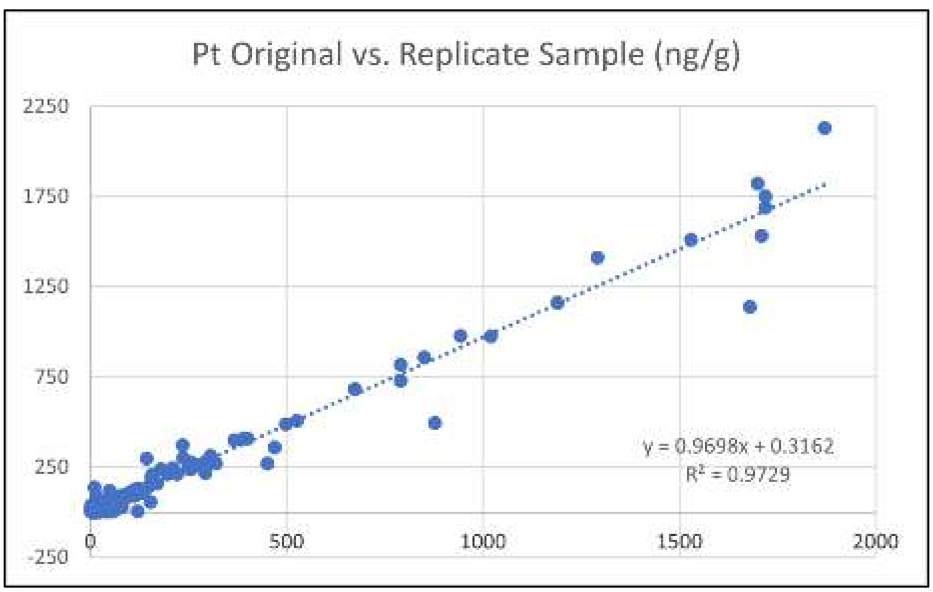

Source: Caracle Creek, 2023.

**Figure 11-21: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Pd (ng/g = ppb)**




Source: Caracle Creek, 2023.

**Figure 11-22: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Pt (ng/g = ppb)**




Source: Caracle Creek, 2023.

Where relative differences of over 100% are observed, sample pairs generally exhibit low absolute concentrations of the precious metals; the order of magnitude difference at those levels is not considered to be of importance.

The relative differences for Ni, Co and S were generally under 20% with only a few exceptions. Again, this appears to be a case where exceptionally low Ni or Co values were returned and as such the relative difference is not considered to be of importance. The results for S were like those for the precious metals (i.e., where relative differences of over 100% are observed, sample pairs generally exhibit low absolute concentrations of the precious metals and the order of magnitude difference at those levels is not considered to be of importance).

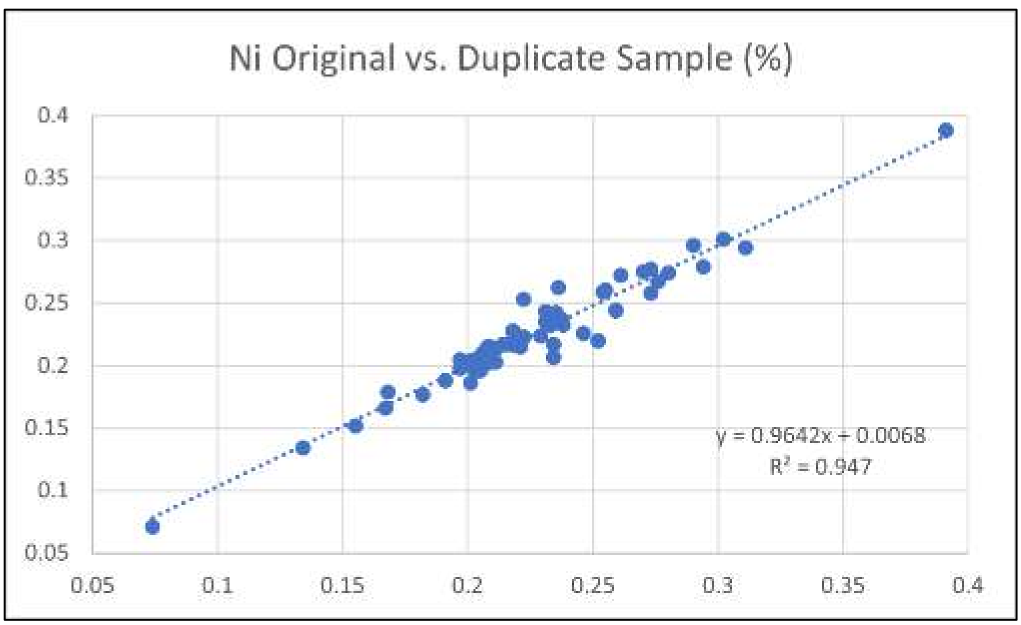
### 11.6.3 Duplicate Samples – "Field Duplicates"

CNC had a total of 65 sample intervals quarter-cut and these samples were submitted for analysis as duplicate ("field duplicate") samples.

In general, the duplicate material for the platinum group metal analyses has indicated good reproducibility of the assays though with some degree of a nuggety response. BVM typically reported higher Pd and Pt concentrations than the original analysis. Where relative differences of over 100% are observed, sample pairs generally exhibit low absolute concentrations of the precious metals; the order of magnitude difference at those levels is not considered to be of importance.

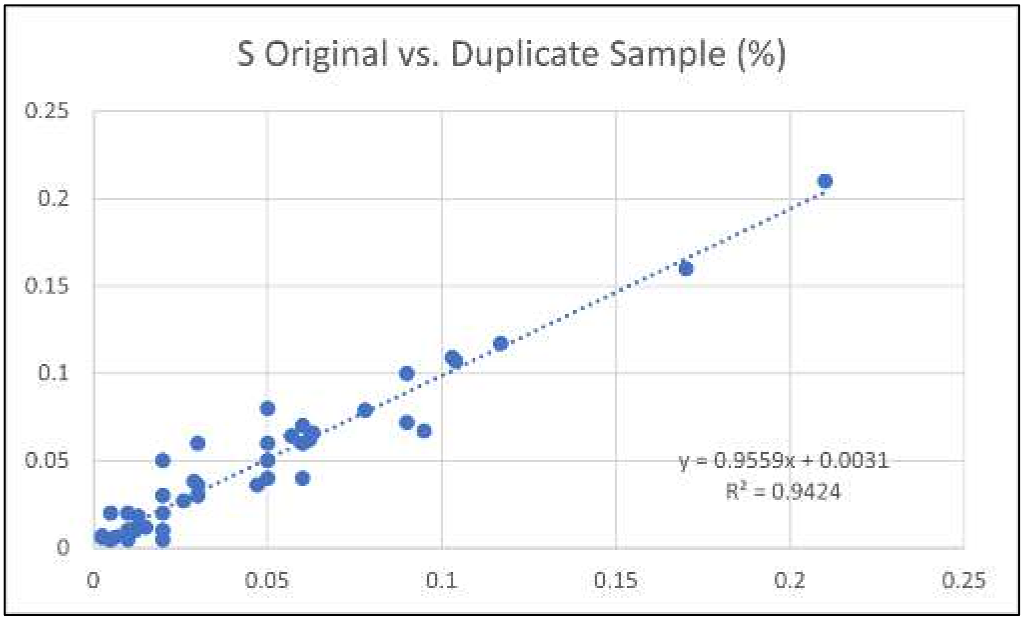
The duplicate pairs for Ni and S exhibited good correlation (Figures 11-23 and 11-24) while those for the platinum group metals were poor (example Pt, Figure 11-25); the poor correlations are attributed to the low absolute concentrations of these elements in the sample material.

**Figure 11-23: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Ni**



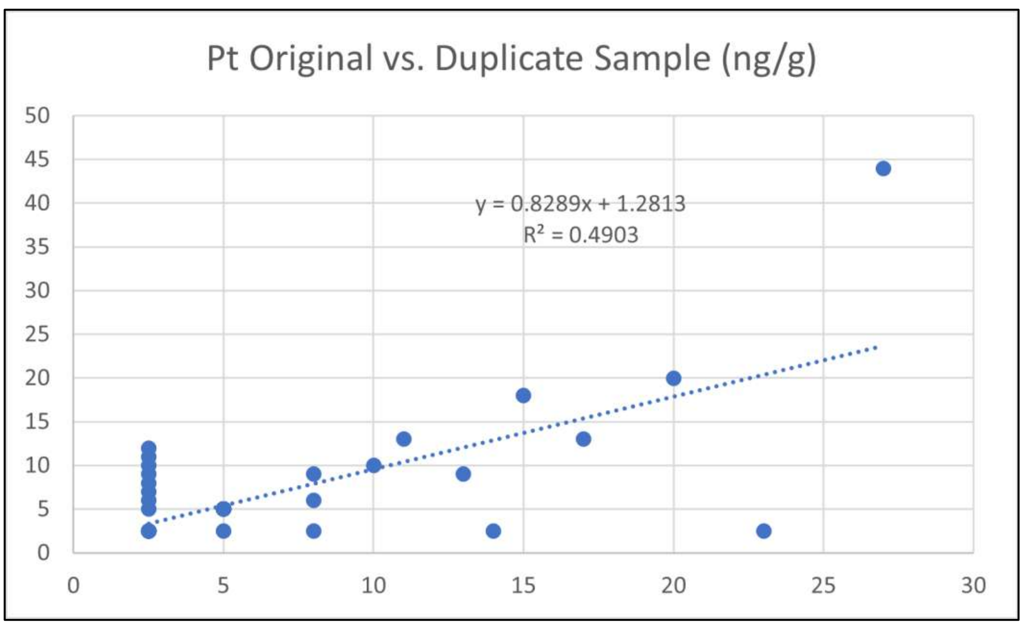

Source: Caracle Creek, 2023.

**Figure 11-24: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for S**




Source: Caracle Creek, 2023.

**Figure 11-25: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Pd (ng/g = ppb)**




Source: Caracle Creek, 2023.

### 11.6.4 Duplicate Samples – Referee Analyses

CNC had 320 sample pulps re-analyzed at an alternate laboratory or "referee laboratory", specifically Bureau Veritas Commodities Canada Ltd. (BVM), located in Vancouver, BC. BVM is an ISO 17025 accredited facility. The analytical methods used for the referee analyses were essentially identical to the original methods though the suite of elements and detection limits varied slightly.

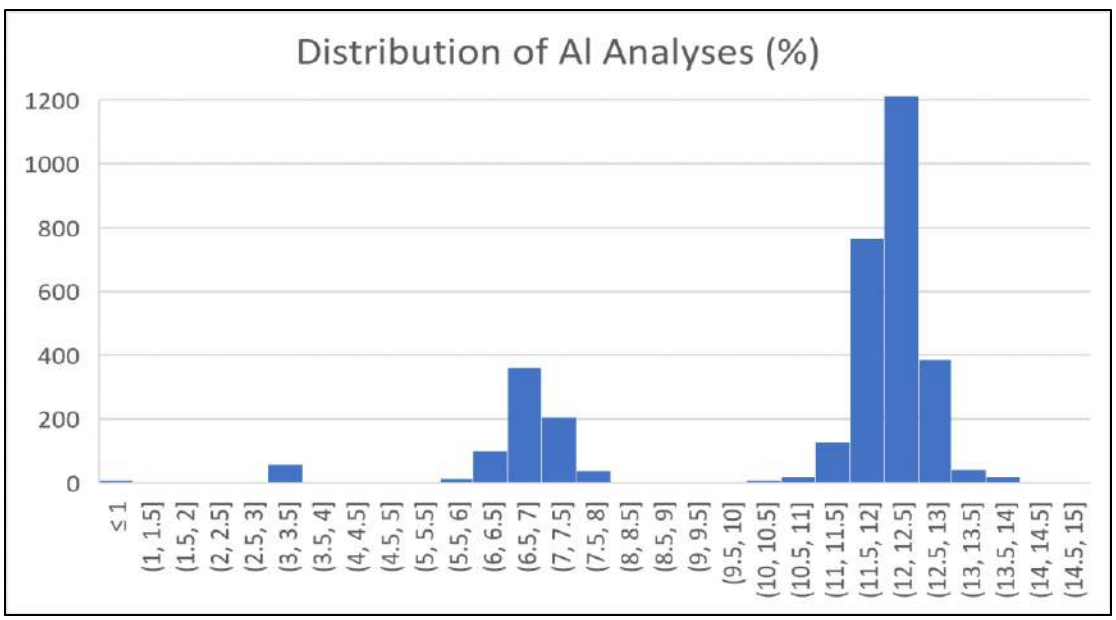
In general, the duplicate material for the platinum group metal analyses has indicated good reproducibility of the assays though with some degree of a nuggety response. BVM typically reported higher Pd and Pt concentrations than the original analysis. Where relative differences of over 100% are observed, sample pairs generally exhibit low absolute concentrations of the precious metals; the order of magnitude difference at those levels is not considered to be of importance.

The relative differences for Ni, Cr and S were all under 20% (and mostly under 10%), indicating very good reproducibility of the original analyses.

### 11.6.5 Blank Material

The blank samples introduced by CNC into their QA/QC program (variously referred to as "blank gravel," "blank FP," "blank sand' or "blank silica") appear to have a non-uniform nature and to have three different provenances of origin (Figure 11-26). Analytical results are considered to be acceptable, as the results were observed to report low or negligible variance for each element examined. The most extreme examples with respect to the Ni analyses (0.224% Ni, 0.116% Ni, 0.093% Ni) are probably due to the inherent inhomogeneity of the "blank" material used.

**Figure 11-26: Distribution of Aluminum (Al) Analyses in "Blank" Samples**




Note: Aluminum was used to crudely differentiate the rock-type populations. Source: Caracle Creek, 2023.

The failure rates of about 2% are all considered to be acceptable at the absolute concentrations being reported. There was no evidence of any systematic trend to the minor discrepancies.

In the opinion of the authors, the assay data is adequate for the purpose of verifying drill core assays, estimating mineral resources, and for a feasibility study.

### 11.7 Sample Security and Sample Storage

CNC uses a secure storage and logging facility at 170 Jaguar Drive, Timmins, Ontario that has office space for professional and technical staff. The drill core is brought to the facility from the field by CNC personnel and unloaded within the confines of the logging/office building. Once logged and sampling sections are identified, the core is split/cut by diamond saws in a room dedicated to this purpose within the facility; these sample cutting facilities have been significantly upgraded over the life of the project. Three pneumatic-feed saws are currently available for use at any given time. Individual bagged and sealed samples are stored at the facility until groups of samples are transferred to a laboratory.

Archived core is stored in covered racks, outdoors, on the grounds of the facility. Some of the core is cross-stacked in palletized piles containing up to 160 boxes prior to additional storage racks being organized. The core from the early stages of the drilling program, when a different, smaller facility was used for logging and sampling, has also been transferred to the current location.

Sample pulps and rejects that have been returned from the laboratories are also stored on site. Pulps are stored protected in intermodal shipping containers ("sea-cans") while coarse crushed reject material is currently stored outdoors.

# 12 DATA VERIFICATION

## 12.1 Internal-External Data Verification

The authors, John Siriunas and Scott Jobin-Bevans, have reviewed historical and current data and information regarding past and current exploration work on the property. More recent exploration work (i.e., 2018 to 2023) that had complete databases and documentation (e.g., assay certificates; core logging database; reporting) was thoroughly reviewed; however, older historical records (in general, pre-2018) were not as complete, so the exact methodologies used in the data collection are unknown. Nonetheless, these authors have no reason to doubt the adequacy of the historical sample preparation, security and analytical procedures and have confidence in the historical information and data that was reviewed. There were no limitations on or failures to conduct such verification.

## 12.2 Verification Performed by the QPs

Mr. John Siriunas (M.A.Sc., P.Eng.) visited the project on October 12, 2019 (one day), on February 3-4, 2020 (two days), on September 10-11, 2020 (two days), and on August 11-12, 2022 (two days). During the site visits, diamond drilling procedures were discussed and a review of the on-site logging and sampling facilities for processing the drill core were carried out. As there is no outcrop on the property, no surface grab samples of target mineralization/lithologies could be collected. After verification of existing core logs and assay results against drill core observations, Mr. Siriunas did not feel it necessary to re-sample the drill core. Random verification of several drill site locations was also carried out during Mr. Sirunas' field visits to the Crawford property. Locations and orientation of drillholes were always found to be consistent with those reported in the drillhole database files.

Dr. Scott Jobin-Bevans (Ph.D., P.Geo.) has reviewed in detail all historical and current data and information regrading past and current exploration work on the property. In addition, as part of the geological modelling and mineral resource estimations work, this author has reviewed the drillhole database inclusive of but not limited to drillhole collar locations and borehole surveys, core lithology, alteration, and mineralization, and core assays (primary and QA/QC samples). The author did not find any inconsistencies in the database and related information as provided by the Company.

## 12.3 Comments on Data Verification

It is the opinion of authors Scott Jobin-Bevans and John Siriunas that the procedures, policies, and protocols for drilling verification are sufficient and appropriate and that the core sampling, core handling and core assaying methods used are consistent with good exploration and operational practices such that the data is reliable for the purpose of mineral resource estimation and for the purpose of this report.

# 13 MINERAL PROCESSING AND METALLURGICAL TESTING

## 13.1 Introduction

The Crawford flowsheet will process serpentine ore to produce nickel and FeCr concentrates, while also storing $CO_2$ in the brucite component of the tailings. The concentrator comminutes the ore and separates the value minerals by a combination of flotation and magnetic separation processes. Tailings from these processes are thickened to a target 40% density and are then processed using CNC's in-process-tailings (IPT) carbonation process, which permanently fixes carbon dioxide ($CO_2$) in solid mineral form within the tailings. The carbonated tailings from the IPT Carbonation process are discharged and stored within the tailings management facility as a permanent bank for $CO_2$.

This section of the report provides an overview of the testwork that was completed to support the feasibility study and the interpretation of results for engineering design and financial modelling.

The Crawford project will produce two concentrates: a nickel sulphide concentrate ("nickel concentrate") and an iron and chromium concentrate ("FeCr concentrate") that also contains some nickel. The nickel concentrate is produced in the flotation circuit through recovery of nickel sulphide minerals, mostly heazlewoodite and pentlandite. This concentrate will be the highest nickel grade concentrate available on the market when in production with a life of mine forecast concentrate average grade of 34% nickel and 0.7% cobalt. The nickel concentrate is expected to be sold into the battery production chain. However, if it was roasted to eliminate the sulphur, it could also be sold as a nickel source in steel-making. The FeCr concentrate, which is produced in the magnetic recovery circuit through recovery of the minerals: magnetite, chrome-spinel and awaruite[2], contains iron, chromium and nickel and is expected to be sold to a steel melt shop as a primary feed for steel production. If the steel plant were to be co-located in Timmins, the emissions from smelting could be captured and stored at Crawford to produce low or zero carbon primary steel.

The main value-driving commodity for the Crawford project is nickel, which is hosted in recoverable forms in the minerals pentlandite, heazlewoodite and awaruite, or in unrecoverable form within the matrix of the gangue silicate minerals. Because a portion of the nickel is locked within the gangue minerals, the nickel recoveries that are achieved from Crawford ores are lower than those achieved in higher grade projects, such as those close to Sudbury. While this is often viewed as a detriment to ultramafic projects, ultramafic deposits are generally close to surface and this allows for cheaper bulk tonnage open pit mining, and the mineralogy can support the production of higher grade nickel concentrates.

One of the key goals of the metallurgical test program completed for the feasibility study was to improve the metallurgical performance (metal recoveries and concentrate qualities) relative to the flowsheet detailed in the PEA. Optimization of the flowsheet in both the flotation and magnetic recovery circuits delivered improvements in recoveries and concentrate quality across all commodities. Table 13-1 compares the nickel recovery performance for the feasibility study against the PEA for different periods of operation. It shows that improvements in nickel recovery were made across all phases of the project. Recovery gains were more substantial in Years 6 to 25. A 6.9 percentage

[2] Awaruite is a nickel iron alloy mineral with a nickel tenor of approximately 74%. Awaruite is highly magnetic.

point improvement in nickel recovery has been achieved over the first 25 years and a 3.9 percentage point improvement over the 41-year mine life.

**Table 13-1: Feasibility Study vs. PEA – Nickel Recovery Comparison**

| Description | Years 1 - 5 | Years 1 - 25 | Life of Mine |
|---|---|---|---|
| Feasibility | 47.1% | 44.2% | 41.2% (41 years) |
| PEA | 46.7% | 37.3% | 37.3% (25 years) |
| **Difference** | **0.4%** | **6.9%** | **3.9%** |

Table 13-2 compares the life of mine recoveries for nickel, cobalt, iron, and chromium for the feasibility study and the PEA. In addition to the nickel recovery gains, significant improvements in cobalt, chromium and particularly iron recovery have been achieved.

**Table 13-2: Feasibility Study vs. PEA – Life-of-Mine Recovery Comparison**

| Description | Ni | Co | Fe | Cr |
|---|---|---|---|---|
| Feasibility Study[1] | 41.2% | 11% | 52.7% | 28.2% |
| PEA[2] | 37.3% | 8%[3] | 36.1% | 27.0% |
| **Difference** | **3.9%** | **3%** | **16.6%** | **1.2%** |

Notes: 1. Feasibility study life of mine is 41 years. 2. PEA life of mine was 25 years. 3. Cobalt was not a payable metal in the PEA.

Both the nickel and FeCr concentrate grades were improved through flowsheet optimization work during the feasibility study. Table 13-3 compares the grades of the nickel and FeCr concentrates for the feasibility study and the PEA.

Optimization of the flotation cleaning circuit increased the nickel concentrate grade by 18% nickel absolute, from 16% nickel in the PEA to a forecast 34% nickel grade in the feasibility study.

The iron grade of the FeCr concentrate was conservatively modelled as a constant 55% iron over the life of mine. The nickel and chromium grades make the concentrate a potential direct feed for the stainless steel industry.

**Table 13-3: Feasibility Study vs. PEA – Nickel and FeCr Concentrate Grade Comparison**

| Description | Nickel Concentrate – Ni Grade | FeCr Concentrate – Fe Grade |
|---|---|---|
| Feasibility Study | 34.2% | 55.0% |
| PEA | 16.0%[1] | 47.5% |
| **Difference** | **18.2%** | **7.5%** |

Notes: 1. In the PEA, a high-grade (35% nickel) and lower grade (12% nickel) concentrate were produced. The stated 16% nickel grade is a weighted average grade of the combined concentrate.

The following additional goals were targeted in the feasibility study testwork program:

- Improve confidence in production forecasts, with a focus on ore that is processed during the project "payback period."[3] The results of testwork were analyzed to develop predictive models that support mine design and sequencing, recovery, concentrate grades, revenue forecasts and the development of a technoeconomic model.
    - Metallurgical variability testing, including 126 open circuit tests and 15 locked cycle tests, were completed at XPS, Expert Process Solutions in Sudbury, Ontario and COREM in Quebec City, Quebec, to support recovery and concentrate quality forecasts.
    - Comminution tests on a total of 83 samples were completed at SGS Lakefield to support throughput calculations. The key parameters used for process design were the Bond ball work index, the JKMRC Axb parameter developed through SMC and JK drop weight tests, Bond abrasion index tests and the Bond low energy impact (crusher) work index tests.
- Develop process design criteria to support the plant design basis. Results from metallurgical variability and comminution test programs, as well as additional engineering testwork including tailings thickening tests, rheology testing and deslime cyclone characterization were used to support the process design criteria.
- Quantify the carbon sequestration potential of the project with the IPT carbonation process. Using samples of fresh tailings produced from metallurgical variability, samples were tested using a standard test procedure to evaluate the $CO_2$ sequestration capacity of tailings as a function of the sample's brucite content.
- Evaluate the potential to scale up both the metallurgical and IPT Carbonation processes through pilot plant testing. A pilot test program was completed on the Crawford metallurgical process at SGS Lakefield in summer and fall 2022 using 34 tonnes of material. Tailings generated in the metallurgical pilot plant were preserved underwater until summer 2023 and were used in a pilot test of the IPT Carbonation process.

Metallurgical testwork was completed in the following sequence:

- Mineralogical assessments (Section 13.3)
- Comminution testwork to test a broader spectrum of samples (Section 13.5).
- Metallurgical optimization work to assess reagent and flowsheet options (Section 13.6). Reagent dosing was optimized before starting the metallurgical variability program. Alternative flowsheet options including the flotation cleaning circuit and the magnetic recovery circuit were optimized during the metallurgical test program.
- Evaluation of the metallurgical response of the Crawford ultramafic nickel mineralization across key lithologies, grades and alteration styles (Section 13.7). The results from variability testing including open circuit and locked cycle tests were analyzed to develop recovery and concentrate quality models to support mine planning as well as production and financial forecasts.
- Locked cycle tests on selected samples to assess final metallurgical recovery (Section 13.8).

[3] Simple payback for the Crawford project is achieved after approximately 6 years. The payback pit reflects mining that will take place during this period including ore that is processed and/or stockpiled for future use.

- Pilot plant testwork to assess flotation and magnetic performance on larger samples and generate samples for further engineering testwork (Section 13.9).
- Testwork to provide inputs for process plant design (Section 13.10). Supporting engineering testwork including rheology tests at SGS Lakefield, tailings thickening testwork at Metso and SGS Lakefield, as well as size by size characterization of the deslime cyclones was completed to guide the process plant design.
- IPT carbonation testwork to support the carbon dioxide sequestration flowsheet development (Section 13.13).

## 13.2 Previous Testwork

Prior to the current phase of study, metallurgical testwork was completed as part of the Crawford Preliminary Economic Assessment (PEA) released on May 25, 2021; this testwork is summarized below.

The metallurgical program for the Crawford PEA was carried out to understand the metallurgical response of the Crawford ultramafic nickel mineralization, provide guidance for process plant design, and provide inputs for operating cost estimation. The plant design basis in the PEA was based on metallurgical variability and comminution testing. There was no other supporting engineering testwork completed for the PEA.

### 13.2.1 Metallurgical Variability Testing

Metallurgical variability testing in the PEA was done on 16 samples including 14 open circuit tests and 7 locked cycle tests to understand the potential to recover nickel, iron and chromium from samples taken from different domains, head grades and lithologies at Crawford.

In the PEA, CNC designed the flowsheet to produce three concentrates: a high-grade nickel concentrate with a grade of 35%, a lower grade concentrate with a grade of 12% nickel and an FeCr concentrate with a grade of 47.5% iron. All the nickel and FeCr concentrates were assumed to be sold into the stainless-steel markets with payable metal components of nickel, iron, and chromium. The opportunity to recover byproducts including cobalt and precious group metals was discussed. However, these metals were not payable with steel as the target market.

The flowsheet was substantially modified between the PEA and feasibility study and the metallurgical test results from the PEA were not incorporated into the feasibility study recovery modelling database. Table 13-4 details the key changes to the flowsheet and associated impacts.

To evaluate the impact of changes made to the flotation circuit, side by side tests utilizing the PEA and the feasibility study testwork flowsheets were completed on the same sample. Table 13-5 compares the flotation circuit "rougher level"[4] grade and recovery performance for the two flowsheets and shows that the changes to the flotation circuit increased both the grade and recovery of nickel to the rougher concentrate on all three samples. The three samples tested represent heazlewoodite dominant and mixed heazlewoodite + pentlandite mineralization styles.

[4] Defined as the combined coarse first cleaner concentrate and the fine rougher concentrate.

**Table 13-4: Impact of Key Changes to the Metallurgical Flowsheet for the Feasibility Study**

| Change | Impact |
|---|---|
| Flotation circuit arrangement:<br>• Elimination of magnetic separation in the coarse flotation circuit (ahead of the secondary grinding stage).<br>• The target secondary grind $P_{80}$ size was increased from 45 µm to 100 µm.<br>• Fine rougher float time was increased by 3 minutes to 9 minutes.<br>• Optimized reagent dosing. | • The removal of the coarse magnetic separation stage from the feasibility flowsheet results in all the rougher tailings entering the secondary grinding mill, which increases the energy consumption for secondary grinding relative to the PEA flowsheet.<br>• To offset the increased energy requirements due to additional mass throughput in the secondary grinding mill, a coarser secondary $P_{80}$ grind target of 100 µm was selected. The increased nickel and cobalt recoveries in the flotation circuit justify the flowsheet changes and additional energy requirements (see Section 13.6). |
| Arrangement of the flotation circuit cleaning stages. | • The average nickel concentrate grade over the life of mine increased by 18 percentage points absolute from PEA to feasibility study (Table 13-3). |
| Removal of HG-LG flotation stage. | • Crawford is now designed to make a single nickel concentrate instead of the high (35% Ni) and low grade (12% Ni) concentrates that were targeted in the PEA. The life-of-mine grade of the single concentrate grades 34% nickel for the feasibility study. |
| Magnetic circuit arrangement including:<br>• Rougher magnetic strength increased from 1000 to 2500 G.<br>• Number of magnetic cleaning stages was increased.<br>• Cleaning magnet strengths increased to 1500 G.<br>• Grinding is done in two stages targeting $P_{80}$ 63 µm in the first stage and $P_{80}$ 25 µm in the second stage, versus a single stage grind to $P_{80}$ 35 µm in the PEA. | • Life of mine improvements in iron and chromium recoveries of 16.6 and 1.2 percentage points, respectively (Table 13-2).<br>• Increased nickel recovery in magnetic circuit by 1.1 to 1.3 percentage points (see Section 13.11.1.1.2).<br>• FeCr Concentrate grade improvement of 7.5 percentage points from 47.5% to 55% iron (Table 13-3 and Section 13.12). |

**Table 13-5: Impact of Flowsheet Changes on Flotation Circuit Grade and Recovery Performance**

| Sample ID | Head Characterization | | New Flowsheet – PEA Flowsheet Comparison | |
|---|---|---|---|---|
| | Ni (%) | S/Ni Ratio | Flotation Recovery Improvement (% Absolute) | Flotation Grade Improvement (% Absolute) |
| 103-V2 | 0.43 | 0.58 | +9.2 | +3.3 |
| 100A-V18[1] | 0.29 | 0.31 | +14.5 | +1.7 |
| CR20-55 M2[1] | 0.37 | 0.58 | +8.9 | +4.4 |
| Average | 0.36 | 0.49 | +10.9 | +3.2 |

Note: Coarse rougher float time was 3 minutes longer than the standard test procedure. The expected impact of this is approximately 1 percentage point in nickel recovery in the coarse rougher flotation stage.

### 13.2.2 Comminution Testing

Comminution testing in the PEA was done on six samples from the Crawford Main Zone. One of the samples was outside the resource and was excluded from the both PEA and the feasibility study.

As the general test methods in the PEA were the same as those used in the Crawford feasibility study, the PEA comminution test results were included in the feasibility test database. However, the Bond ball work index tests utilized a different closing screen size in the PEA (Table 13-6). Comminution testing from the PEA showed the following:

- Crawford ore is amenable to SAG/AG milling.
- Crawford ore is hard to very hard in terms of the Bond ball work index[5].
- The Bond abrasion index of Crawford ore is very low, which is typical of ultramafic rocks and leads to lower maintenance and lower grinding media consumption costs in the processing plant.

**Table 13-6: Historical Comminution Data**

| Sample ID | JK Tech Parameters | | | Work Indices | | | | Ni (%) |
|---|---|---|---|---|---|---|---|---|
| | Relative Density (t/m$^{3)}$ | Axb (DWT) | Axb (SMC) | CWi (kWh/t) | RWi (kWh/t) | BWi (kWh/t) | Ai (g) | |
| CR20-55 Comm | 2.63 | 60.7 | - | 9.5 | 14.6 | 20.7 | 0.003 | 0.35 |
| CR20-58 Comm | 2.61 | - | 46.4 | - | 15.0 | 21.1 | 0.002 | 0.38 |
| CR19-08 Comm | 2.63 | - | 58.9 | - | 13.1 | 14.4 | 0.009 | 0.30 |
| CR19-13 Comm | 2.72 | - | 44.9 | - | 15.3 | 18.2 | 0.07 | 0.28 |
| CR20-57 Comm | 2.61 | - | 49.0 | - | 13.8 | 17.9 | 0.02 | 0.11 |

## 13.3 Mineralogy Studies

Mineralogical characterization of core samples was completed at SGS Lakefield using QEMSCAN. The mineralogy program was completed to understand the distribution of key minerals across the deposit and to support the definition of geometallurgical domains. For ultramafic nickel deposits, the mineralogy is a critical part of establishing the resource estimate as nickel can exist in recoverable form as minerals such as heazlewoodite, pentlandite, awaruite and millerite, or nickel can be hosted within the matrix of silicate minerals. Silicate-hosted nickel is not recoverable by flotation, except through gangue entrainment within the final concentrate products.

[5] The Bond ball mill work index results for ultramafic ores need to be interpreted correctly due to the presence of needle-like minerals that behave differently on a mesh screen (used in the Bond test) when compared with a hydrocyclone classifier in plant operation. As a result, the laboratory Bond ball mill work indices tend to be overstated when compared to actual plant operation. This was considered in the design process by Ausenco.

The Crawford resource is contained within an ultramafic sequence, comprising primarily of dunite and peridotite rock types. Both rock types have been altered to a range of Fe-rich and Mg-rich serpentinized rocks. The Mg-rich serpentine is generally the most altered and contains high levels of magnetite, recoverable nickel, and other alteration products such as brucite.

Samples were selected from across the Crawford Main Zone and East Zone from select drillholes to cover the expected resource envelope. Caracle Creek was responsible for the resource modelling and provided support in selecting some of the samples to ensure the samples were spatially distributed across the resource. It was important to have spatial coverage of the deposit, as the mineralogy database was used to estimate the brucite content of blocks in the resource model through kriging. It was necessary to krig brucite into the block model as brucite is the key geometallurgical modelling variable for estimating the carbon sequestration potential of a sample.

Mineralogy samples were selected from select holes at a spacing of 1 sample every 7.5, 15 or 30 metres depending on the hole. The average number of samples per hole was 24. Samples representing 1.5-metre lengths of NQ drill core were shipped to SGS Lakefield for mineralogical characterization, where the samples were stage crushed to 150 µm before being mounted in a polished section and analyzed with QEMSCAN.

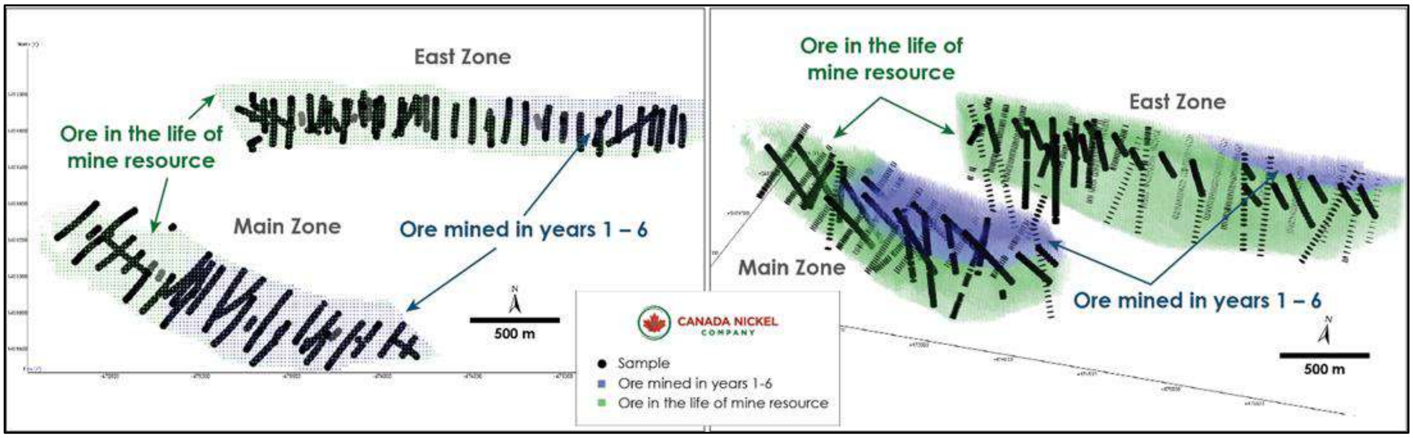
The exploration mineralogy database consists of 3,266 samples. Eighty-three percent (2,701) of these samples were determined to be within the feasibility study mined resource envelope with a head grade greater than or equal to 0.10% nickel, and were incorporated into the feasibility study mineralogy database. Table 13-7 summarizes the distribution of samples between the Main and East zones and how many drillholes were characterized.

**Table 13-7: Mineralogy Samples by Zone**

| Zone | Total No. of Samples | No. of Holes Characterized | Average No. per Hole |
|---|---|---|---|
| Main Zone | 1487 | 59 | 25 |
| East Zone | 1214 | 52 | 23 |
| **Total** | **2701** | **111** | **24** |

Figure 13-1 shows the distribution of mineralogy samples within the feasibility study resource envelope and confirms that the samples are well distributed across the deposit.

**Figure 13-1: Location of Mineralogy Samples within the Feasibility Study Mined Resource**




Source: CNC, 2023.

### 13.3.1 Mineralogy Results

Serpentinization is the Crawford alteration pathway where heat, time, pressure, and reducing fluids cause iron and nickel to leave the silicate mineral matrix of olivine and form distinct minerals such as magnetite, tochilinite and/or pyrrhotite in the case of iron and heazlewoodite, pentlandite and/or awaruite in the case of nickel. Understanding the mineralogical transformations that happen during serpentinization is important for project development, as the alteration state of the ore will impact the recoverable nickel and iron content of the ore.

At Crawford, the main mineral present in the rock is broadly classified as serpentine. To support the mineralogy program, serpentine has been divided into two subcategories called iron serpentine and magnesium serpentine. The difference between iron and magnesium serpentines relates to the composition of the mineral, where iron serpentine has more than 5% iron in the lattice structure and magnesium serpentine has less than 5% iron in the lattice structure. Iron serpentine is typically in less altered ores and sometimes in the presence of olivine, where the recoverable nickel and iron tends to be lower as they are still hosted within the matrix of silicate minerals.

Through analysis of geologically logged data, geophysics, mineralogy, and metallurgical variability results, it was found that the magnetite content of the rock was an excellent proxy for the degree of serpentinization of the rock. As olivine transforms into serpentine, iron leaves the mineral matrix and forms magnetite. Further, it was found that the magnetic susceptibility, which is a parameter that is routinely and systematically measured during core logging processes, is an excellent proxy for the magnetite content of rock and that this parameter could be used to separate completely serpentinized ore with higher recoverable mineral content, from less serpentinized, lower recovery potential ores.

Tables 13-8, 13-9, and 13-10 summarize the mineralogical data for the East Zone, the Main Zone higher magnetite domain, and the Main Zone lower magnetite domain, respectively.

**Table 13-8: Mineralogy Summary – East Zone (1214 Data Points)**

| Description | Serp (Mg) | Serp (Fe) | HZ | AW | PN | Pyrrhotite | Brucite | Talc | Magnetite | Cr Spinel |
|---|---|---|---|---|---|---|---|---|---|---|
| | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) |
| Average | 75 | 6.9 | 0.14 | 0.03 | 0.05 | 0.01 | 2.5 | 0.61 | 6.6 | 2.7 |
| Std. Deviation | 15 | 8.2 | 0.13 | 0.05 | 0.18 | 0.07 | 2.1 | 4.0 | 2.3 | 1.4 |
| Minimum | 1.4 | 0.88 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.04 | 0.11 |
| 25$^{th}$ Percentile | 76 | 3.3 | 0.05 | 0.00 | 0.00 | 0.00 | 0.40 | 0.00 | 5.3 | 1.7 |
| Median | 80 | 4.7 | 0.11 | 0.01 | 0.00 | 0.00 | 2.4 | 0.00 | 6.4 | 2.5 |
| 75$^{th}$ Percentile | 82 | 7.0 | 0.18 | 0.04 | 0.01 | 0.00 | 3.9 | 0.00 | 7.7 | 3.5 |
| Maximum | 88 | 76 | 1.18 | 0.56 | 2.4 | 1.34 | 22 | 34 | 22 | 11 |

**Table 13-9: Mineralogy Summary – Main Zone Higher Magnetite Domain (Mag Sus > 70) (641 Data Points)**

| Description | Serp (Mg) | Serp (Fe) | HZ | AW | PN | Pyrrhotite | Brucite | Talc | Magnetite | Cr Spinel |
|---|---|---|---|---|---|---|---|---|---|---|
| | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) |
| Average | 71.2 | 10.5 | 0.11 | 0.03 | 0.19 | 0.04 | 1.61 | 0.47 | 6.30 | 2.41 |
| Std. Deviation | 16.5 | 10.9 | 0.12 | 0.05 | 0.38 | 0.09 | 1.36 | 3.90 | 2.38 | 1.23 |
| Minimum | 2.07 | 1.37 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.47 | 0.06 |
| 25$^{th}$ Percentile | 70.7 | 4.17 | 0.02 | 0.00 | 0.00 | 0.00 | 0.58 | 0.00 | 4.60 | 1.49 |
| Median | 77.9 | 6.00 | 0.07 | 0.01 | 0.02 | 0.00 | 1.37 | 0.00 | 6.19 | 2.18 |
| 75$^{th}$ Percentile | 80.78 | 11.42 | 0.16 | 0.05 | 0.14 | 0.02 | 2.43 | 0.01 | 7.92 | 3.09 |
| Maximum | 87.1 | 73.73 | 0.94 | 0.28 | 2.23 | 0.86 | 13.5 | 41.5 | 15.5 | 6.49 |

**Table 13-10: Mineralogy Summary – Main Zone Low Magnetite Domain (Mag Sus < 70) (844 Data Points)**

| Description | Serp (Mg) | Serp (Fe) | HZ | AW | PN | Pyrrhotite | Brucite | Talc | Magnetite | Cr Spinel |
|---|---|---|---|---|---|---|---|---|---|---|
| | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) |
| Average | 31.5 | 36.2 | 0.04 | 0.02 | 0.08 | 0.01 | 0.98 | 0.91 | 1.64 | 2.93 |
| Std. Deviation | 20.0 | 17.2 | 0.06 | 0.04 | 0.17 | 0.07 | 1.10 | 5.27 | 1.59 | 1.20 |
| Minimum | 2.32 | 1.35 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.01 | 0.18 |
| 25$^{th}$ Percentile | 16.9 | 22.6 | 0.01 | 0.00 | 0.00 | 0.00 | 0.47 | 0.00 | 0.53 | 2.22 |
| Median | 24.9 | 37.5 | 0.02 | 0.01 | 0.01 | 0.00 | 0.73 | 0.00 | 1.11 | 2.84 |
| 75$^{th}$ Percentile | 41.4 | 49.2 | 0.05 | 0.02 | 0.06 | 0.01 | 1.08 | 0.01 | 2.25 | 3.54 |
| Maximum | 85.8 | 77.4 | 0.58 | 0.31 | 1.16 | 1.73 | 14.2 | 44.0 | 9.83 | 9.04 |

Key points are as follows:

- The East Zone is consistently altered, with most ore falling into the higher magnetite domain. Seventy-five percent of the mineralogical data points in the East Zone have a magnesium serpentine content greater than 76% with magnetite contents greater than 5.3%. Based on this, it is expected that the East Zone will have consistent recoverable nickel and iron contents throughout the resource.
- The Main Zone is more complex than the East Zone, with a higher magnetite domain and a lower magnetite domain. These domains were separated based on a magnetic susceptibility of 70, which was established from the metallurgical variability test program. Comparing Tables 13-9 and Table 13-10, the presence of pentlandite, heazlewoodite, awaruite, magnetite, and brucite are higher in the higher magnetite domain. A comparison of these tables and the difference in mineralogy shows the effectiveness of using the magnetic susceptibility to define the ore alteration state.
- Heazlewoodite and awaruite are the main nickel-bearing minerals in the East Zone with low levels of pentlandite. The East Zone is expected to yield primarily high-grade nickel concentrate with a forecast life-of-mine grade of 37.9% nickel.
- Pentlandite and heazlewoodite are the main nickel-bearing minerals in the Main Zone with lower levels of awaruite. The nickel concentrate that is produced form the Main Zone would be expected to be a lower grade than the East Zone due to the higher levels of pentlandite in the ore. The forecast life-of-mine nickel concentrate grade for the main zone is 30.8% nickel.
- Brucite is a marker for the carbon sequestration potential of a sample. The brucite contents of blocks within the resource model were estimated through kriging by Caracle Creek using the mineralogy database. Brucite is an important variable for carbon sequestration as it is the main variable used to model the carbon sequestration potential of a sample. The East Zone has higher levels of brucite than the Main Zone with an average brucite content of 2.5% based on the mineralogy data and a maximum brucite value of 22%. In the Main Zone, the brucite content is higher in the higher magnetite domain, with an average grade of 1.6% brucite. The brucite grades for the entire reserve based on kriging are 1.9% for the East Zone and 1.31% for the Main Zone.
- The presence of brucite has been found to be lithology dependent. Dunitic ores have higher levels of brucite, which decrease through transitional material towards the peridotite contact. Peridotites have very low to no brucite content. Based on this, it is expected that peridotites will have a lower carbon sequestration potential than dunites at Crawford.
- Both the Main and East Zones have low levels of iron sulphides such as pyrrhotite, which is a typical contaminant in nickel concentrates. The low levels of iron sulphides at Crawford contributes to the higher expected concentrate grades compared to typical nickel sulphide projects, such as those mined in Sudbury.
- On average the talc content across Crawford is low. However, there is a pocket of talc altered ore along the southern contact of both the Main and East Zones of the deposit. This was mapped by Caracle creek based on mineralogy data, density measurements and magnetic susceptibility.

### 13.3.2 Microprobe Results

Microprobe analysis was done on selected polished sections from the mineralogy program to measure the elemental composition of minerals within the deposit. This information is important as the microprobe analyses were used to estimate the nickel deportment within a sample.

Microprobe analysis was done on polished sections at two laboratories: XPS and Queen's University. A total of 66 samples within the resource were analyzed across 22 holes including 14 new samples that were characterized since the PEA. Seven out of 14 of the new samples were from the East Zone of Crawford. Table 13-11 summarizes the nickel tenor and composition of key minerals within the Crawford deposit from the microprobe analysis.

**Table 13-11: Microprobe Analysis – Summary of Results**

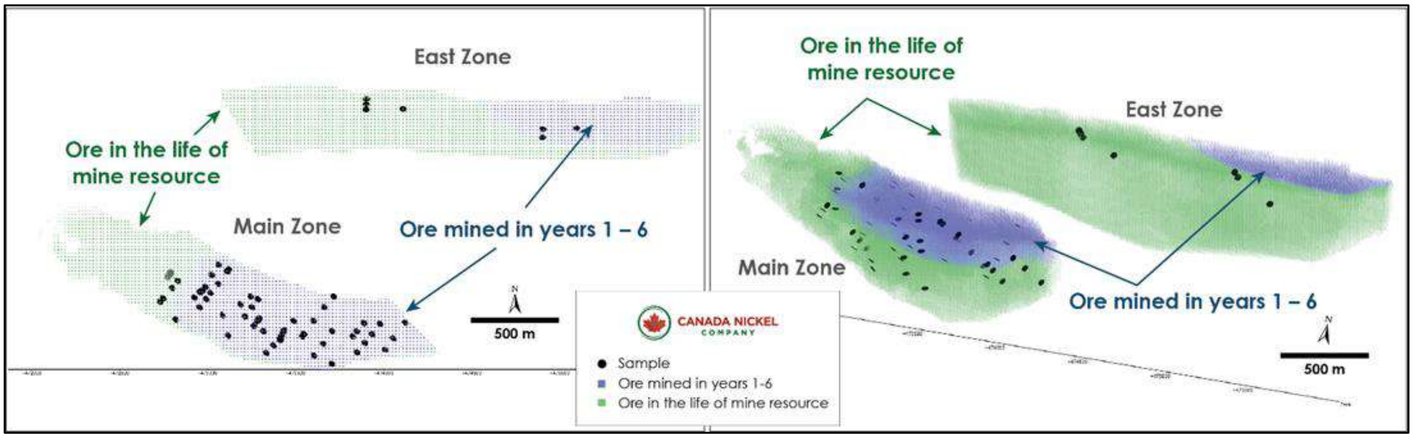
| Mineral | | Ni | | Fe | | Co | | Cr | |
|---|---|---|---|---|---|---|---|---|---|
| | No. of Pts. | Avg. (%) | St. Dev (%) | Avg. (%) | St. Dev (%) | Avg. (%) | St. Dev (%) | Avg. (%) | St. Dev (%) |
| Awaruite | 43 | 73.4 | 1.97 | 24.4 | 1.77 | 1.35 | 0.71 | - | - |
| Heazlewoodite | 279 | 71.8 | 3.82 | 1.09 | 0.88 | 0.63 | 3.39 | - | - |
| Pentlandite | 269 | 32.7 | 2.99 | 28.9 | 5.12 | 5.38 | 6.26 | - | - |
| Magnetite | 490 | 0.18 | 0.18 | 70.4 | 1.50 | 0.05 | 0.04 | 0.44 | 0.76 |
| Cr Spinel | 467 | 0.09 | 0.03 | 17.9 | 3.71 | 0.05 | 0.02 | 32.7 | 2.00 |
| Serpentine | 638 | 0.10 | 0.07 | 2.19 | 1.37 | 0.01 | 0.01 | 0.10 | 0.07 |

The updated microprobe database confirms the findings from the PEA. Key points are listed below:

- The mineral heazlewoodite is a high-grade nickel mineral with a tenor of 71.8% nickel. Heazlewoodite is primarily recovered in the flotation circuit to the nickel concentrate. The high nickel tenor of heazlewoodite allows high-grade concentrates to be produced at Crawford. Testwork has shown it is possible to produce grades in excess of 40% nickel.
- Pentlandite is a nickel sulphide mineral that also contains cobalt, iron, and sulphur within the lattice structure. The average nickel tenor of pentlandite is 32.7% nickel. The microprobe data shows high variability in the cobalt content of pentlandite, with a range of 1% to 25% cobalt in minerals identified as pentlandite. Minerals that contain 25% cobalt would be classified as a distinct mineral called "cobalt pentlandite".
- Magnetite, awaruite and chrome spinel are magnetic minerals that are recovered to the FeCr concentrate. Awaruite is a high nickel tenor mineral. It and magnetite are highly magnetic while chrome spinel is less magnetic. Nickel in the mineral awaruite as well as in the lattice of chromium and magnetite contributes to the nickel that is recovered to the FeCr concentrate.
- Serpentine is the main gangue mineral in the deposit. Microprobe analysis shows that the average nickel tenor of serpentine is 0.10% with a standard deviation of 0.07%. This nickel is hosted within the mineral matrix and is unrecoverable except through entrainment to the various products in the mineral processing flowsheet.

Figure 13-2 shows the location of samples selected for microprobe analysis at Crawford.

**Figure 13-2: Location of Samples Characterized by Microprobe within the Crawford Mined Resource**




Source: CNC, 2023.

## 13.4 Sample Selection

### 13.4.1 Comminution Samples

Comminution samples were selected from drill core from the Main and East Zones of Crawford to understand variability in ore breakage parameters across the deposit. Three types of samples were selected as part of the feasibility study:

- 3 full HQ core domain samples, which were tested for Bond low energy impact testing, Bond ball mill work index, Bond rod mill work index, Bond abrasion index, and SMC and JK drop weight indices (Axb)
- 73 half NQ core variability samples, which were tested for Bond ball mill work, Bond rod mill work, Bond abrasion and SMC Axb indices
- 1 sample of intermediate magnetic concentrate from the pilot plant peridotite composite, which was used to conduct a HIG mill signature test to support the engineering and design of the magnetic regrind mills.

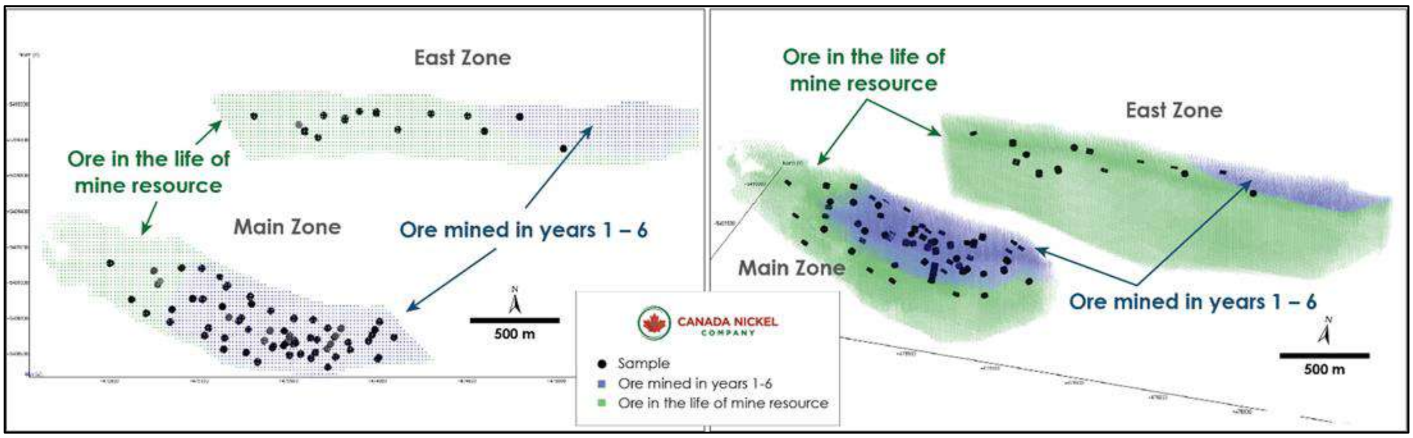
Domain samples were selected based on deposit zone and alteration style all within the dunite lithology. Two of the samples were taken from the Main Zone of Crawford and one of the samples was taken from the East Zone. Of the two samples taken from the Main Zone, one was from the western part of the Main Zone in an olivine rich zone and the other was taken from the eastern part of the Main Zone where mining is expected to start in well serpentinized, higher magnetite ore. These samples were selected to contrast different mineralization styles. The East Zone sample was from well serpentinized, higher magnetite ore in the western part of the deposit, which was outside of the payback pit. However, it still provides a good comparison to the Main Zone higher magnetite ore.

The 73 half NQ variability samples were selected based on lithology, iron grade, chromium grade and magnetic susceptibility as a proxy for alteration style and location within the deposit. The samples were selected to provide spatial coverage of the deposit zones with a bias to the Main Zone of Crawford, as this is where most of the payback period ore is expected to come from. The nickel and sulphur grades of the samples were not considered in the sample selection process as these are minor constituents in the ore which are not expected to affect comminution behaviour. However, the sample coverage does capture variability in these components.

One sample from pilot plant testing was submitted for a HIG mill signature plot test at SGS Lakefield. The sample was the magnetic cleaner 1 concentrate produced in the pilot plant from the peridotite composite. The peridotite composite was selected because it has the highest iron and magnetite content.

Figure 13-3 shows the location of the comminution samples that were tested for the feasibility study.

**Figure 13-3: Location of Comminution Samples within the Mined Crawford Resource**




Source: CNC, 2023.

### 13.4.2 Metallurgical Variability Samples

Metallurgical variability samples were selected and subjected to variability testing to understand variability in recovery and concentrate quality. Samples were selected to be representative of ore that would be processed early in project life and the sample selection was biased towards the payback period in order to de-risk early operation of the project.

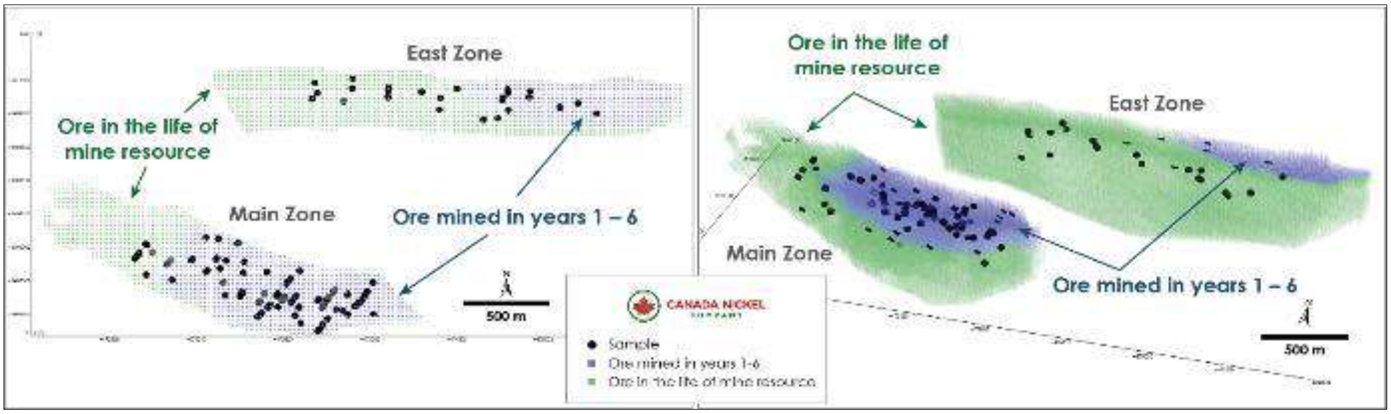
In selecting samples for metallurgical testing, the key parameters that were considered included the lithology, grade, and alteration style.

Key considerations are listed below:

- The two main lithologies at Crawford are dunite and peridotite.
- The key geochemical markers considered in the variability program were nickel and sulphur grade, with secondary components being iron and magnesium grade. The S/Ni ratio was used to target ore with different mineralization styles, as the nickel mineralization can impact process performance and resultant concentrate quality.
- Alteration styles defined as part of the metallurgical variability program included low magnetite, high magnetite and talc altered ores.

Figure 13-4 shows the locations of the samples that were selected for the metallurgical variability program within the expected Crawford reserve. Most of the samples are contained within the Main Zone of Crawford as this is where most of the payback period ore will come from. Some ore will be mined from the East Zone during the payback period to access sand and gravel in the overburden for construction purposes.

**Figure 13-4: Metallurgical Variability Sample Locations**

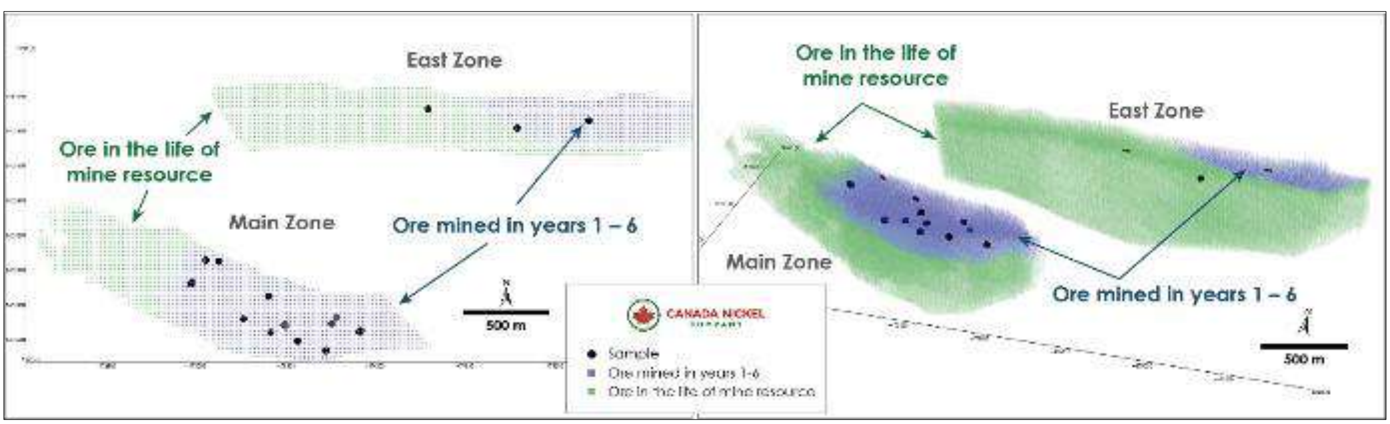



Source: CNC, 2023.

### 13.4.3 IPT Carbonation Variability Samples

Sixteen IPT carbonation variability samples were selected based on brucite content as well as location in the deposit. Thirteen samples were selected from the Main Zone of Crawford within the expected payback pit covering a brucite range of 0.01% to 5.7% and three samples were selected from the East Zone of Crawford covering a brucite range of 2.6% to 5.0% with one sample residing in the East Zone starter pit. The brucite content of samples selected for testing was measured through QEMSCAN at SGS Lakefield as part of the metallurgical program. Figure 13-5 shows the location of the IPT Carbonation variability samples within the Crawford mined reserve.

**Figure 13-5: Location of IPT Carbonation Variability Samples**




Source: CNC, 2023.

### 13.4.4 Metallurgical Pilot Plant Composites

Six pilot plant composites were built from 53 samples of large diameter drill core from 19 holes to represent the key lithologies, zones of the deposit and mineralization styles at Crawford. Drilling was focussed on areas of the deposit that were expected to be processed during the payback period and all the drillholes except for two holes drilled into the peridotite were twins of NQ core resource holes. Core was allocated to the different composites based on logged lithologies from the geologists, as well as assays from the NQ core holes that were twinned. A description of each of the composites and its relationship to the mine plan follows:

- Composite 1 was composed of Main Zone dunite with pentlandite dominant mineralization from the high-grade core domain. Composite 1 would be processed during the project payback period.
- Composite 2 was composed of Main Zone dunite with mixed pentlandite and heazlewoodite mineralization from the high-grade core domain. Composite 2 would be processed during the project payback period.
- Composite 3 was composed of Main Zone dunite from the lower grade domain. The nickel mineralization was a mix of heazlewoodite and awaruite. Composite 3 would likely be processed during the project payback period.
- Composite 4 was a mix of Main Zone and East Zone dunite and was used for plant commissioning. Material that was expected to be processed outside of the payback period was added to Composite 4. The mineralization style was mixed between heazlewoodite, pentlandite and awaruite.
- Composite 5 was composed of East Zone material. The primary lithology in this composite was dunite. However, a small amount of peridotite was in the composite to make up the required mass. Composite 5 would not be processed during the project payback period.
- Composite 6 was composed of primarily Main Zone peridotite. However, a small amount of East Zone peridotite was added to the composite to make up the required mass. Composite 6 would likely be processed during the project payback period.

Each of the pilot plant composites received the following set of tests:

- Bond ball mill work index test
- open circuit metallurgical variability test
- locked cycle test
- pilot plant test
- mineralogical characterization with QEMSCAN.

### 13.4.5 IPT Carbonation Tailings Composite

Tailings produced from the metallurgical pilot plant that were stored underwater in bins following completion of the metallurgical pilot plant were sampled to build a composite for the IPT Carbonation pilot demonstration. The tailings composite was a blend of tailings produced from each of the metallurgical pilot plant composites, which were collectively stored in bins. The tailings represent the dunite lithology as most of the drill core was taken from dunite.

However, the exact source location cannot be estimated as tailings were not segregated according to the composite during the metallurgical pilot plant.

#### 13.4.6 Engineering Testwork Samples

Samples for supporting engineering testwork were taken. An explanation of sample location is provided with the engineering testwork results overview (see Section 13.10).

### 13.5 Comminution Circuit Characterization Testwork

Comminution testwork was completed to understand the variability in competency, hardness, and abrasion of Crawford ore. The Crawford comminution circuit utilizes a SAG mill for primary grinding and a ball mill for further size reduction. The key metrics from this testwork that were used for the plant design bases were the Axb parameters from JK drop weight and SMC tests, as well as the Bond ball mill work indices. The test program was completed at SGS Lakefield and test procedures were informed from historical work done for the Crawford PEA.

#### 13.5.1 Sample Characteristics and Representativity

Comminution samples were selected from the key lithologies, alteration styles, and zones of Crawford to evaluate variability in parameters that affect mill throughput.

Eighty-three samples were tested in the comminution program. Table 13-12 summarizes the comminution sample characteristics with supporting statements:

- Iron head grades ranged from 3.9% to 8.5% which covers the expected range of mill feed grades over the life of mine. The average grade of samples tested was 6.5% versus the payback period average of 6.5%.
- Most of the samples were classified as high magnetite ores, with 75% of the samples having a magnetic susceptibility greater than 100. The well-serpentinized, higher magnetite ores are the value driver for the project and thus the bias in samples towards this material. The average magnetic susceptibility of samples tested was 124 versus the payback period average of 118.
- Eighty-one percent of the samples were from the Main Zone, as that is where most of the payback period ore will come from and 19% of samples were from the East Zone. The distribution of samples between zones of Crawford represents the expected mill feed during the payback period for which 83% is expected to come from the Main Zone and 17% from the East Zone.
- Eighty-one percent of samples tested were from the dunite lithology, 17% were from the peridotite lithology and 2% were from talc altered ores. The distribution of samples across lithologies is reasonably well distributed with the expected mill feed which is 68% dunite, 22% peridotite, and 8.4% talc during the payback period.

**Table 13-12: Comminution Sample Characteristics**

| Statistic | Ni (%) | S (%) | S/Ni Ratio | Fe (%) | Cr (%) | MgO (%) | Magnetic Susceptibility |
|---|---|---|---|---|---|---|---|
| Average | 0.25 | 0.15 | 0.50 | 6.5 | 0.61 | 39 | 124 |
| Std. Deviation | 0.07 | 0.22 | 0.64 | 1.0 | 0.13 | 2.2 | 50 |
| Minimum | 0.11 | 0.01 | 0.04 | 3.9 | 0.30 | 31 | 2 |
| 25th Percentile | 0.19 | 0.03 | 0.14 | 5.9 | 0.53 | 38 | 100 |
| Median | 0.23 | 0.06 | 0.27 | 6.7 | 0.61 | 39 | 132 |
| 75th Percentile | 0.30 | 0.17 | 0.57 | 7.3 | 0.72 | 40 | 162 |
| Maximum | 0.40 | 1.4 | 4.3 | 8.5 | 0.85 | 42 | 215 |

### 13.5.2 Grindability Testwork Results

As part of the comminution test program, the following set of tests were completed: JK drop weight test (JK DWT), SMC test, SAG power index (SPI) test, Bond low-energy impact (CWi) test, Bond rod mill work index (RWi) test, Bond ball mill work index (BWi) test and the Bond abrasion index (Ai) test. The JK DWT and CWi tests were only completed on samples from large diameter drill core.

Including data from the Crawford PEA, testing was completed on a total of 87 samples. Four of these samples were removed from the dataset as they are classified as waste material, bringing the total samples tested to 83.

Table 13-13 summarizes the grindability test results for the comminution samples which informed the feasibility study design basis.

**Table 13-13: Grindability Testwork Summary of Results**

| Statistic | JK Tech Parameters | | | | Bond Work Indices | | | | Head Grade |
|---|---|---|---|---|---|---|---|---|---|
| | Relative Density | JK DWT Axb | SMC Axb | SPI (min) | CWi (kWh/t) | RWi (kWh/t) | BWi[1] (kWh/t) | Ai (g) | Ni (%) |
| # of Data Points | 75 | 4 | 74 | 2 | 4 | 25 | 77 | 25 | 83 |
| Average | 2.63 | 50.6 | 59.0 | 27.5 | 10.2 | 15.7 | 20.3 | 0.011 | 0.25 |
| Std Deviation | 0.07 | 8.7 | 14.6 | 5.8 | 1.2 | 2.0 | 2.3 | 0.016 | 0.07 |
| Rel Std Dev. (%) | 3% | 17% | 25% | 21% | 12% | 13% | 11% | 148% | 29% |
| Minimum | 2.36 | 37.6 | 30.8 | 23.4 | 8.6 | 13.1 | 14.8 | 0.000 | 0.11 |
| 10th Percentile | 2.59 | - | 45.1 | - | - | 13.9 | 17.9 | 0.001 | 0.16 |
| 25th Percentile | 2.61 | - | 49.3 | - | - | 14.5 | 18.8 | 0.002 | 0.19 |
| Median | 2.63 | 51.9 | 57.6 | - | 10.4 | 15.3 | 20.2 | 0.004 | 0.23 |
| 75th Percentile | 2.66 | - | 65.8 | - | - | 16.3 | 21.8 | 0.010 | 0.30 |
| 90th Percentile | 2.70 | - | 69.6 | - | - | 17.2 | 22.9 | 0.031 | 0.35 |
| Maximum | 2.95 | 60.7 | 117 | 31.5 | 11.5 | 23.7 | 27.7 | 0.070 | 0.40 |

Note. 1. Bond work index testing in the PEA utilized a different closing screen size and were excluded from the dataset.

### 13.5.3 JK Drop Weight Test & SMC Test

The JK DWT and SMC test characterize ore specific parameters for sizing semi-autogenous grinding (SAG) and autogenous grinding (AG) mills. The ore specific parameter is the Axb parameter which is a measure of an ore's resistance to impact breakage. Testing confirmed that the Crawford ore is amenable to processing with SAG or AG mills.

The JK DWT was performed on four samples from Crawford and interpreted by JKTech. One of the samples tested was more competent than the others, with an Axb value of 37.6 which is defined as competent. This sample is expected to be processed in year 32 of operation. The other three samples subjected to the JK DWT had Axb values ranging from 48 to 61, with an average of 55 and are more characteristic of ore to be treated during the payback period. These samples are expected to be processed in Years 3 to 11 of operation and are defined as moderately competent.

The SMC test is an abbreviated version of the standard JK DWT that is performed on rocks of a single size fraction (-31.5 / +26.5 mm was the size range in this case). The test procedure in the feasibility study was calibrated by completing a JK DWT and a SMC test on the same samples. The results were reasonably consistent, with two out of the three results falling within 5% of each other[6].

SMC results are available for 74 samples of Crawford ore. The majority of Axb parameter results ranged from 45.1 (10$^{th}$ percentile) to 69.8 (90$^{th}$ percentile) which covers the low competence to moderate competence range. The average and median values for the Axb parameter were 59.0 and 57.6, respectively, which are defined as moderate competence.

### 13.5.4 SPI Test

The SPI is a measure of hardness for sizing SAG mills. SPI testing was completed on two samples as a substitute for the SMC test. Both samples tested with the SPI method were rich in the mineral coalingite, which is reactive to moisture and/or $CO_2$ which causes the rock to break down. Both samples from the coalingite zone, which was mapped with a wireframe, had already weathered and thus the size distribution was too fine for SMC testing. SPI values for the two samples ranged from 23.4 to 31.5 minutes with an average of 27.5 minutes, which is classified as soft. This is consistent with expectations, as ore from the coalingite zone is visually not competent and degrades quickly. There is an opportunity to increase SAG mill throughput when treating ores from the coalingite zone, which is in the western part of the Main Zone.

### 13.5.5 Bond Low-Energy Impact Test

The Bond low energy impact test determines the Bond crusher work index (CWi), which can be used to calculate power requirements for crusher sizing. For each of the four samples tested, between 12 to 18 rocks in the size range of 2 to 3 inches were selected for testing at SGS Vancouver. The CWi ranged from 8.5 to 11.5 kWh/t with an average of 10.2 kWh/t. The CWi for the olivine rich sample was 11.2 kWh/t, which fell within the range of the well serpentinized dunite samples.

[6] The sample from the PEA that was subjected to the full JK drop weight test did not receive the SMC test for side-by-side comparison.

### 13.5.6 Bond Rod Mill Work Index Test

Bond rod mill work index tests were performed at 14 mesh of grind (1,180 µm) on 25 samples. The majority of RWi values ranged from 13.9 kWh/t (10th percentile) to 17.2 kWh/t (90th percentile), which is classified as medium to hard. The RWi results aid in the calculation of SAG mill specific energy but were not required for the entire suite of samples. A strong correlation between the Bond rod and ball mill work indexes was observed with an $R^2$ of 88%.

### 13.5.7 Bond Ball Mill Work Index Test

Bond ball mill work index tests were performed with a closing screen size of 60 mesh (250 µm) targeting a $P_{80}$ grind size of 212 µm. Seventy-seven samples were subjected to the Bond ball mill work index test, with the majority of BWi values ranging from 17.9 kWh/t (10th percentile) to 23.0 kWh/t (90th percentile), with average and median values of 20.2 kWh/t and 20.3 kWh/t, respectively. The Crawford ore in this range is classified as medium to very hard in terms of the Bond ball mill work index. One of the samples in the bottom 10th percentile range was from the talc lithology, which had a BWi of 14.8 kWh/t, and was as expected, softer than the other samples tested. The majority of test product 80% passing sizes ($P_{80}$) ranged from 189 µm (10th percentile) to 202 µm (90th percentile) which are generally within 20 µm of the target 212 µm.

### 13.5.8 Bond Abrasion Test

Abrasion testing is done to understand the potential for wear of material including mill liners or pipes as well as the potential for media consumption in the ball mills. The maximum measured abrasion index was 0.041 g and the average across 25 tests was 0.008 g. These values indicate that the abrasiveness of Crawford ore is very low. This is typical of ultramafic orebodies and will result in low liner wear rates and low grinding media consumption. The low abrasiveness of ultramafic rock has a significant positive impact on mill reline time and the mill operation costs.

### 13.5.9 High-Intensity Grinding Mill Signature Plot

The magnetic recovery circuit utilizes high-intensity grinding (HIG) mills to grind the magnetic concentrate down to a final grind $P_{80}$ size of 25 µm. An intermediate sample from the pilot plant on Composite 6, which is from the peridotite lithology was taken and submitted to the SGS comminution team for a HIG mill signature plot. The HIG mill signature plot showed that at the target grind $P_{80}$ size of 25 µm, the specific energy was 15.4 kWh/t.

### 13.5.10 Breakage Domains

The comminution results were analyzed to develop relationships between key ore breakage parameters and ore characteristics. However, no relationships could be established to support the techno-economic model. It was thus decided to model the comminution parameters for the project as constants across all ore types. Further comminution testwork will be conducted in association with new drilling campaigns to attempt to develop predictive models for ore breakage parameters.

Most of the Axb results sit in a narrow range between 51 and 67, and most of the BWi results sit in a narrow range between 18.8 and 22.2 kWh/t. As the breakage characteristics are relatively consistent across the resource, the project risk was managed in design by using the 75th percentile value.

## 13.6 Metallurgical Optimization Results

### 13.6.1 Coarse Rougher Flotation Stage

The coarse rougher flotation stage treats the primary deslime cyclone underflow. The primary grind $P_{80}$ size of 180 µm as well as the six-minute rougher flotation residence time which were utilized in the PEA were maintained for the feasibility study. Reagent dosing for PAX, acid and Calgon were optimized through kinetic testing before starting the variability test program.

#### 13.6.1.1 Calgon and Acid Addition

Kinetic testing was done to evaluate the impact of Calgon and acid dosing on the coarse rougher flotation performance. The sample selected for testing was a pentlandite dominant sample with a nickel head grade of 0.34% and a S/Ni ratio of 1.0. Table 13-14 summarizes the results of kinetic testing. The conditions in test “Flot 164A” were identified as optimum in terms of the rougher concentrate grade and recovery and based on this result, a Calgon dosage of 200 g/t and acid dosage of 1000 g/t were selected for the standard test procedure.

**Table 13-14: Effect of Acid and Calgon Dosing on Coarse Rougher Flotation Performance (Ni = 0.34%, S/Ni = 1.0)**

| Test ID | Reagent Dosing | | | | Rougher Performance @ 6 min | |
|---|---|---|---|---|---|---|
| | Acid (g/t) | Calgon (g/t) | PAX to Rougher (g/t) | MIBC (g/t) | Ni Grade (%) | Ni Recovery (%) |
| Flot 164 A | 1000 | 200 | 50 + 15 | 13 | 10.4 | 53.5 |
| Flot 164 B | 1000 | 150 | 50 + 15 | 13 | 11.0 | 49.1 |
| Flot 164 C | 1000 | 125 | 50 + 15 | 13 | 9.9 | 48.2 |
| Flot 164 D | 1000 | 100 | 50 + 15 | 13 | 5.0 | 47.4 |
| Flot 164 E | 0 | 200 | 50 + 15 | 13 | 7.1 | 53.1 |
| Flot 164 F | 0 | 150 | 50 + 15 | 13 | 7.6 | 42.4 |

#### 13.6.1.2 PAX Addition

A series of 12 tests was completed on two samples that represent heazlewoodite and pentlandite dominant mineralization styles to assess the impact of PAX dosing on the rougher level grade and recovery performance. The effect of PAX addition rate to the mill and to the rougher flotation cell were investigated with the following findings:

- PAX addition to the mill at 25 g/t delivered higher concentrate grades and comparable recoveries to 50 g/t of PAX. It was decided to utilize 25 g/t of PAX in the mill for the variability test program.
- The PAX dosage to the rougher flotation cell impacted the grade and recovery of the resulting concentrate (Table 13-15). Higher PAX dosages resulted in lower concentrate grades and higher recoveries, which was expected. At the time that these kinetic evaluations were completed, the cleaning circuit had not been finalized and the flowsheet was struggling with upgrading. It was thus decided to optimize reagents for grade and a 20 g/t PAX dosage was taken forward for the variability program. There may be an opportunity to improve nickel recoveries by increasing the PAX dosage in the rougher flotation stages. However, testwork and trade-off studies are required to evaluate this.

**Table 13-15: Rougher Grade Recovery Performance as a Function of PAX Dosage**

| Sample | Test | Reagent Dosing Conditions (g/t) | | | | Rougher Flotation Results | |
|---|---|---|---|---|---|---|---|
| | | PAX to Mill | PAX to Float Cell | Calgon | Acid | Ni Grade (%) | Ni Recovery (%) |
| CBS-002 0.36% Ni 1.0 S/Ni | Flot 169A | 50 | 30 | 200 | 1000 | 10.1 | 66.5 |
| | Flot 169B | 50 | 20 | 200 | 1000 | 10.7 | 62 |
| | Flot 169C | 50 | 10 | 200 | 1000 | 12.7 | 56 |
| | Flot 170A | 25 | 30 | 200 | 1000 | 11 | 66 |
| | Flot 170B | 25 | 20 | 200 | 1000 | 15 | 64 |
| | Flot 170C | 25 | 10 | 200 | 1000 | 14 | 56 |
| 103-V1 0.30% Ni 0.37 S/Ni | Flot 171A | 50 | 30 | 200 | 1000 | 12.1 | 36.9 |
| | Flot 171B | 50 | 20 | 200 | 1000 | 12.6 | 33.8 |
| | Flot 171C | 50 | 10 | 200 | 1000 | 14.7 | 30.6 |
| | Flot 172A | 25 | 30 | 200 | 1000 | 14.0 | 36.8 |
| | Flot 172B | 25 | 20 | 200 | 1000 | 14.0 | 33.4 |
| | Flot 172C | 25 | 10 | 200 | 1000 | 15.5 | 31.7 |

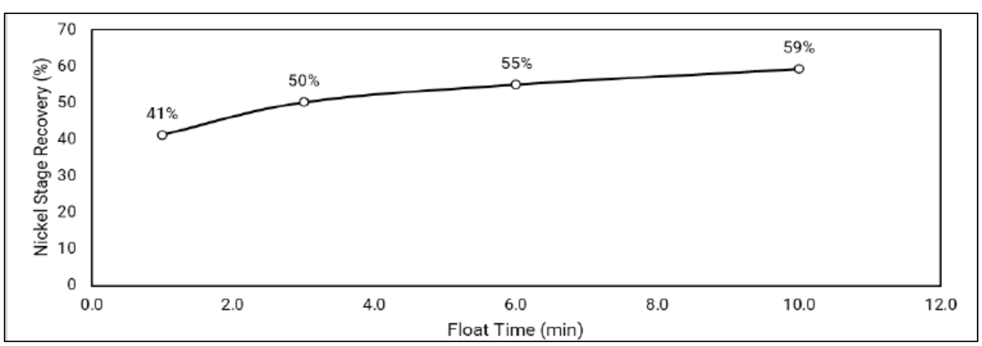
### 13.6.2 Fine Rougher Flotation Stage

Reagent and fine rougher flotation residence times were optimized through kinetic testing. Optimum conditions were established before commencing the variability test program.

#### 13.6.2.1 Flotation Residence Time

Kinetic testing in the fine rougher flotation stage was completed to establish the optimum flotation time for the standard test procedure. Kinetic testing was done on a pentlandite dominant sample with a nickel head grade of 0.34% Ni and a S/Ni ratio of 1.0. Testing was only completed on a pentlandite dominant sample as its flotation kinetics are slower than heazlewoodite. Figure 13-6 shows the kinetic stage recovery of nickel to the fine rougher concentrate, which reached a maximum of 59%. Based on this trend, it was decided to utilize a nine-minute flotation time for the fine rougher in the open circuit variability program. The additional recovery gained from six to nine minutes is minimal; however, it reduces risk of nickel sulphide mineral losses to the final tailings streams.

**Figure 13-6: Fine Rougher Kinetics – Nickel Stage Recovery (Ni = 0.34%, S/Ni = 1.0)**




Source: CNC, 2023.

#### 13.6.2.2 Calgon Dosage

Table 13-16 compares the results of two kinetic tests completed on a pentlandite dominant sample with a nickel head grade of 0.34% nickel and a S/Ni ratio of 1.0. The tests show that a Calgon dose of 100 g/t delivers superior performance compared with an addition rate of 50 g/t as it delivered both higher stage recoveries in the fine rougher float cell, and a higher nickel grade in the rougher concentrate.

**Table 13-16: Effect of Calgon Dosage on the Fine Rougher Flotation Performance**

| Test ID | Reagent Dosages | | | | Fine Rougher Flotation Performance | |
|---|---|---|---|---|---|---|
| | Calgon (g/t) | PAX (g/t) | MIBC (g/t) | A-65 (g/t) | Nickel Grade (%) | Nickel Stage Recovery (%) |
| Flot 168B | 100 | 20 | 3.4 | 0.25 | 6.0 | 38 |
| Flot 168C | 50 | 20 | 3.4 | 0.25 | 4.4 | 26 |

#### 13.6.2.3 PAX Dosage to Secondary Mill

Table 13-17 compares the results of two tests which evaluated the impact of PAX dosage to the secondary mill on the performance of the fine rougher flotation stage. Two conditions were compared including 50 g/t and 25 g/t dosages and which show that 50 g/t of PAX delivers superior performance in terms of grade and recovery. The variability test program utilized 25 g/t of PAX in the secondary mill, so there may be room to improve the fine circuit performance by increasing the PAX dosage to the secondary mill.

**Table 13-17: Effect of PAX Dosage in the Secondary Grinding Mill on the Fine Rougher Flotation Performance Units**

| Test | PAX Dosage (g/t) | | | | | Fine Rougher Flotation | |
|---|---|---|---|---|---|---|---|
| | Calgon | PAX to Mill | PAX to Float | MIBC | A-65 | Ni Grade | Ni Stage Rec |
| Flot 173A | 100 | 50 | 20 | 14.3 | 2.8 | 2.6% | 57% |
| Flot 173B | 100 | 25 | 20 | 14.3 | 2.8 | 1.7% | 53% |

### 13.6.3 Flotation Cleaning Circuit

Various flotation cleaning circuits were evaluated during flowsheet development work. Ultimately, a common layout of the cleaning circuit was established for both the coarse and fine circuits, which includes a regrind on the cleaner 1 concentrate to a $P_{80}$ of approximately 60 µm, followed by two stages of flotation cleaning. The similarity of the coarse and fine cleaning circuits results in an opportunity to merge the circuits together. However, this was not done for the variability test program in order to quantify the recoveries and resultant concentrate qualities of each of the circuits. This could be evaluated through future work.

The following decisions were made in developing the cleaner circuit including the supporting rationale:

- A target $P_{80}$ size of 55 µm in the cleaner 1 regrind was chosen to ensure the final concentrate is not penalized for fine particle size.
- The cleaner regrind location was selected as the cleaner 1 concentrate so that liberated gangue minerals have an opportunity to be rejected in the second and third flotation cleaning stages.

### 13.6.4 Magnetic Recovery Circuit

The magnetic recovery circuit was optimized to maximize the grade and recovery of Fe, Cr and Ni to the final magnetic concentrate. Refer to Table 13-4 in Section 13.2 for a summary of key changes made relative to the PEA flowsheet.

To evaluate opportunity to improve the performance of the magnetic recovery circuit, Davis tube testing was completed on three samples to evaluate the impact of the magnetic rougher strength, magnetic cleaning strength and the final grind size on process performance. Open circuit variability and pilot scale testing were completed following the flowsheet changes to confirm process performance.

#### 13.6.4.1 Davis Tube Testing

Davis tube testing was completed on three samples to evaluate the impact of the rougher magnet strength on the recovery of nickel, iron, and chromium. Three samples were tested to represent different mineralization styles as well as the two key lithologies of Crawford. Table 13-18 summarizes the characteristics of the three samples that were tested.

**Table 13-18: Sample Characteristics used for Davis Tube Testing**

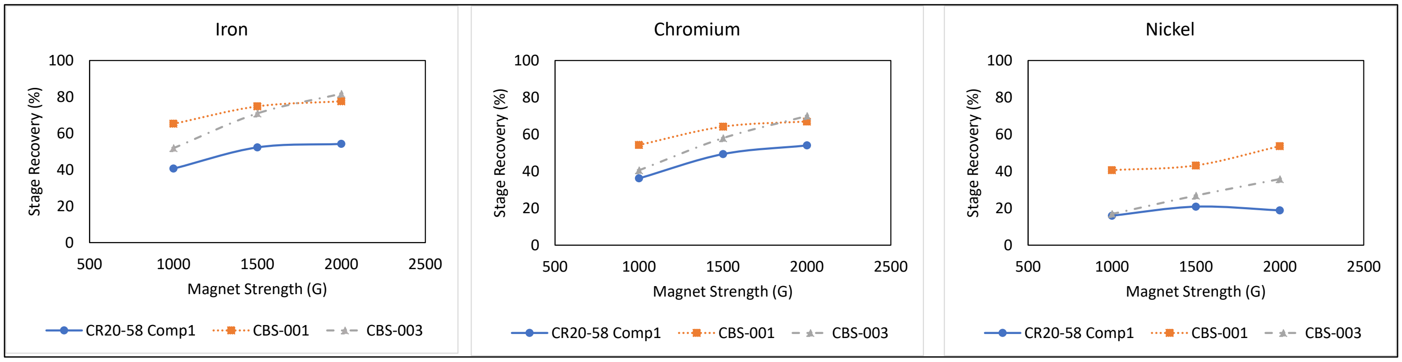
| | Sample Characteristics | | | | | |
|---|---|---|---|---|---|---|
| **Sample ID** | **Lithology** | **Ni** | **S** | **S/Ni** | **Fe** | **Cr** |
| CR20-58 Comp1 | Dunite | 0.39% | 0.68% | 1.7 | 5.6% | 0.75% |
| CBS-003 | Dunite | 0.31% | 0.20% | 0.65 | 7.9% | 0.93% |
| CBS-001 | Peridotite | 0.23% | 0.09% | 0.39 | 7.8% | 0.67% |

##### 13.6.4.1.1 Rougher Magnet Selection

Representative samples of the fine rougher tailings, which represents the feed to the magnetic recovery circuit, were generated through open circuit testing. Representative subsamples were subjected to Davis tube testing at 1000 G, 1500 G, and 2000 G to understand the impact of the magnet strength on the recovery of payable metals Ni, Fe, and Cr to the rougher magnetic concentrate.

Figure 13-7 shows the rougher stage recovery as a function of the magnet strength from Davis tube testing. For each of the payable metals, nickel, iron, and chromium, the stage recovery at the rougher level increases as a function of gauss strength. However, there are diminishing returns at higher magnet strengths. Based on these results, the standard test procedure was updated to utilize a 2000 G rougher magnet.

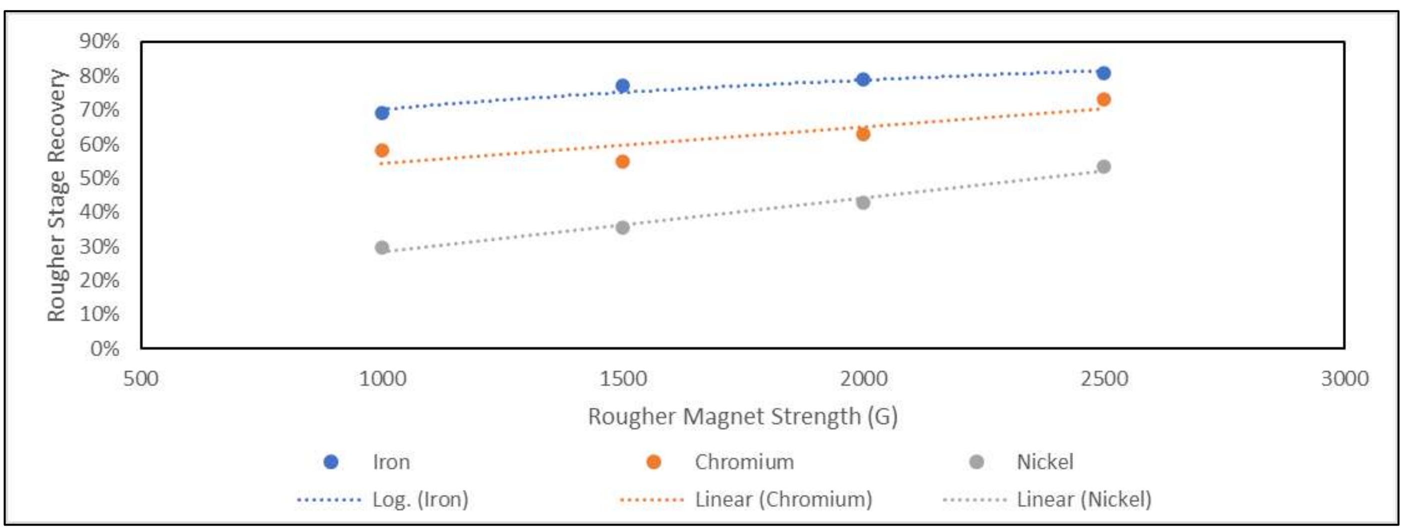
**Figure 13-7: Davis Tube Results – Magnetic Rougher Stage Recovery vs. Gauss Strength**




Source: CNC, 2023.

To support the Davis tube test results, rougher stage recoveries across the open circuit, locked cycle and pilot plant test database were analyzed to confirm the recovery gains at the rougher level. Figure 13-8 presents the stage recoveries of iron, chromium, and nickel as a function of the rougher magnet strength. The data confirms that the rougher recovery performance improves at higher magnet strengths. There was a substantial improvement in the nickel and chromium recovery in the pilot plant using a 2500 G magnet and, based on these results (see Section 13.9), a 2500 G magnet was selected for the feasibility study design basis.

**Figure 13-8: Crawford Variability Test Database – Rougher Stage Recoveries vs. Magnet Strength**




Source: CNC, 2023.

#### 13.6.4.2 Magnetic Cleaning Circuit & Final Grind Size Selection

The magnetic cleaning circuit arrangement was established based on advice from Ausenco to utilize a two-stage grind. The decision to use a two-stage grind was made to improve grinding efficiency through better gangue rejection within the magnetic recovery circuit. Additionally, Ausenco recommended the use of 1500 G magnets in the cleaning circuit to improve cleaning recovery performance relative to the 1000 G magnets utilized in the PEA.

Table 13-19 summarizes the results of Davis tube testing completed on the rougher magnetic concentrate which evaluates the impact of grind size on the cleaning stage recovery for a 1500 G magnetic field strength. Finer grind sizes result in improved concentrate grades through better gangue rejection. At a $P_{80}$ of 25 µm, the FeCr concentrate has higher iron grades and lower levels of MgO and S relative to the 35 µm grind size.

Finer grinding results in a small reduction in nickel and chromium recoveries in the magnetic cleaning circuit. These cleaning recovery losses at the finer grind size are offset by improved rougher level recoveries. Based on the results of this Davis tube work, a 25 µm grind size was selected for the feasibility study design basis. Open circuit, locked cycle, and pilot plant testing were conducted using the final flowsheet to evaluate recovery and resultant concentrate quality in the FeCr concentrate (see Section 13.12).

**Table 13-19: Cleaning Stage Recovery and Resulting Concentrate Quality vs. Grind Size (1500 G)**

| Sample ID | Grind Size $P_{80}$ | Magnetic Concentrate Grade (%) | | | | | Cleaning Stage Recovery (%) | | |
|---|---|---|---|---|---|---|---|---|---|
| | (µm) | Ni | Fe | Cr | S | MgO | Fe | Cr | Ni |
| CR20-58 Comp1 | 25 | 0.23 | 54 | 6.7 | 0.65 | 9.0 | 91 | 66 | 43 |
| | 35 | 0.27 | 52 | 7.0 | 0.71 | 11 | 92 | 73 | 53 |
| CBS-001 | 25 | 0.23 | 52 | 3.0 | 0.03 | 11 | 89 | 67 | 65 |
| | 35 | 0.21 | 49 | 3.1 | 0.03 | 13 | 91 | 76 | 68 |
| CBS-003 | 25 | 0.12 | 52 | 3.6 | 0.07 | 12 | 91 | 66 | 32 |
| | 35 | 0.14 | 48 | 3.7 | 0.08 | 15 | 93 | 76 | 42 |

### 13.7 Metallurgical Variability Testwork

Metallurgical variability testwork was completed in parallel at XPS and COREM, leveraging work completed in the PEA, optimization work discussed previously and experience from other ultramafic nickel projects. To minimize bias in results between laboratories, synchronizing tests on common samples were completed throughout the variability test program.

### 13.7.1 Sample Characteristics & Representativity

Metallurgical variability samples were selected from the key lithologies, alteration styles and zones of Crawford to evaluate variability in recovery characteristics and develop predictive models for production and financial forecasts.

126 samples were tested in the metallurgical variability program with the distribution of sample characteristics summarized in Table 13-20 with the following supporting statements:

- Nickel head grades ranged from 0.11% to 0.49%, which covers the full spectrum of expected head grades that the mill would treat.
- Sulphur grades ranged from 0.01% to 0.72% with a median value of 0.08%. The sulphur grade range that was tested covers the expected mineralization styles at Crawford. This is supported by the distribution of S/Ni ratios which ranges from 0.04 to 1.9.
- Most of the samples were classified as high magnetite ores, with 75% of the samples having a magnetic susceptibility greater than 99. The well-serpentinized, higher magnetite ores are the value driver for the project and thus the bias in samples towards this material. The average magnetic susceptibility of samples tested was 125 versus the payback period average of 118.
- Seventy-nine percent of samples tested were from the Main Zone, as that is where most of the payback period ore will come from, and 21% samples were from the East Zone. The distribution of samples between the zones of Crawford represents the expected mill feed during payback for which 83% is expected to come from the Main Zone and 17% from the East Zone.
- Fifty-five percent of samples were from the high-grade core domain and the remaining 45% of samples were from the lower grade domain.
- Eighty-three percent of samples were from the dunite lithology, 13% were from the peridotite lithology and 4% were from talc altered ores. During the payback period, the expected blend of lithologies is 68% dunite, 22% peridotite, 8% talc and 2% others. The distribution of samples across lithologies is slightly biased towards the dunite lithology.

**Table 13-20: Metallurgical Variability Sample Characteristics**

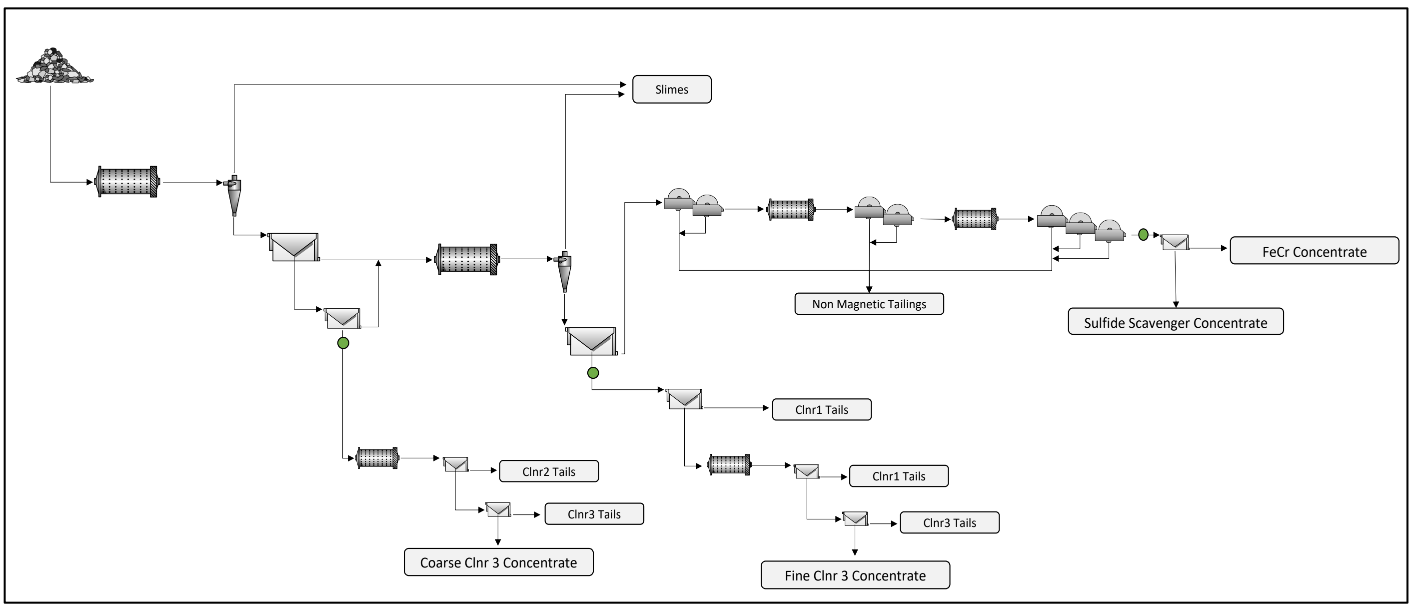
| Statistic | Ni (%) | S (%) | S/Ni Ratio | Fe (%) | Cr (%) | MgO (%) | Magnetic Susceptibility |
|---|---|---|---|---|---|---|---|
| Average | 0.28 | 0.14 | 0.45 | 6.6 | 0.63 | 40 | 125 |
| Std. Deviation | 0.08 | 0.16 | 0.38 | 1.0 | 0.16 | 2.2 | 47 |
| Minimum | 0.11 | 0.01 | 0.04 | 4.1 | 0.22 | 32 | 13 |
| 25$^{th}$ Percentile | 0.22 | 0.04 | 0.18 | 6.0 | 0.53 | 39 | 99 |
| Median | 0.27 | 0.08 | 0.32 | 6.7 | 0.65 | 40 | 124 |
| 75$^{th}$ Percentile | 0.33 | 0.21 | 0.62 | 7.5 | 0.72 | 42 | 161 |
| Maximum | 0.49 | 0.72 | 1.9 | 8.5 | 1.2 | 45 | 255 |

### 13.7.2 Standard Test Procedure

Figure 13-9 illustrates the final laboratory test flowsheet that formed the basis of the open circuit variability standard test procedure, as well as the flowsheet that was used for plant design basis. The green circles in Figure 13-9 show the streams that were used to define recovery in open circuit tests. Circuit recoveries were defined as follows:

- For open circuit tests, the flotation circuit nickel recovery was defined as that which is recovered to the coarse cleaner 1 concentrate and the fine rougher concentrate.
- In the magnetic recovery circuit, recovery was defined as that which is recovered to the final magnetic concentrate, which also represents the feed to the desulphurization float.
- Total nickel recovery is the sum of the flotation and magnetic circuit recoveries.
- Cobalt, platinum, and palladium recoveries are measured in the flotation circuit only
- Iron and chromium recoveries are measured in the magnetic circuit only.

**Figure 13-9: Open Circuit Test Flowsheet (Final Testwork Flowsheet used for Feasibility Study Design Basis)**




Source: CNC, 2023.

Samples of ½ NQ or whole HQ core from Crawford were delivered to the laboratories for metallurgical testwork. Procedures used to prepare the samples at the metallurgical test laboratories were as follows:

- Stage crush and stage screen core samples to 100% passing 10 mesh (1.7 mm). Composite and homogenize the crushed sample.
- Separate the bulk material into 5 kg test charges using a carousel splitter.
- Use three 5 kg samples to build a grind calibration curve for each sample. Confirm the grind curve with an additional 5 kg sample at the target $P_{80}$ grind size.
- Send a 200 g sample ground to 80% passing 180 µm to SGS Lakefield for mineralogical characterization to confirm nickel deportment and mineralogy.

To complete the open circuit tests, the following procedure was utilized:

- Grind two 5 kg batches of sample in wet media to the target 80% passing 180 µm.
- Treat the ground sample with a hydrocyclone to deslime the pulp with 13% to 21% of the mass reporting to the overflow.
- Perform rougher and first cleaner flotation separation on the hydrocyclone underflow.
- Grind the coarse first cleaner concentrate to 80% passing 55 µm and treat with flotation separation through two cleaning stages to produce the coarse final nickel concentrate.
- Combine the coarse rougher and cleaner tail together and regrind to 80% passing 100 µm.
- Treat the reground product with a secondary hydrocyclone to deslime the pulp with approximately 8% to 16% of the overall mass reporting to the overflow.
- Perform rougher and first cleaner flotation separation on the secondary deslime cyclone underflow.
- Grind the fine first cleaner concentrate to 80% passing 55 µm and treat with flotation separation to make the fine circuit final nickel concentrate.
- Perform two stages of magnetic separation at 2000 G on the fine rougher flotation tailings.
- Regrind the magnetic portion of the fine rougher flotation tails to a target grind size of approximately 63 µm.
- Perform two stages of magnetic separation at 1500 G on the reground product.
- Regrind the magnetic portion of the 1500 G cleaner concentrate to 25 µm.
- Perform three stages of magnetic separation at 1500 G on the reground product.
- Perform a sulphide flotation on the magnetic concentrate at a pH of 7.
- Record weights for all products.
- Send products for analysis. Assays were completed using XRF for nickel, cobalt, iron, chromium, and magnesium. Sulphur was analyzed separately with LECO.

#### 13.7.2.1 Evolution of the Standard Test Procedure

The metallurgical variability test program consisted of three main phases of testing, with each phase utilizing a different flowsheet for the standard test procedure. The changes in the flowsheet were made to the magnetic recovery circuit, while flotation circuit conditions were kept constant across all phases.

Table 13-21 summarizes the various magnetic flowsheets tested as part of the metallurgical variability program according to laboratory and phase. The key changes that were made to the magnetic recovery circuit were changes in magnet strength, final grind size, number of grinding stages and the number of magnetic cleaning stages. These changes in the magnetic recovery circuit were made to capitalize on new knowledge gained from the variability test program.

**Table 13-21: Magnetic Recovery Circuit Test Parameters by Phase of Variability Testing**

| Phase | Laboratory | Magnetic Recovery Circuit |
|---|---|---|
| 1 | XPS, COREM | 1 x 1000 G Rougher<br>Regrind to 35 µm<br>3 x 1000 G<br>Acid wash for 20 minutes at pH = 7<br>2 x 500 G Cleaners<br>Sulphide scavenger flotation |
| 2 | XPS, COREM | Phase 1 flowsheet, excluding acid wash |
| 3 | XPS | Phase 1 flowsheet, but utilizing a 1500 G rougher magnet |
| 3 | COREM | 2 x 2000 G Rougher<br>Regrind to 63 µm<br>3 x 1500 G cleaners<br>Regrind to 25 µm<br>3 x 1500 G cleaners<br>Sulphide scavenger flotation |
| Pilot | SGS | 1 x 2500 G rougher<br>Regrind to 63 µm<br>2 x 1500 G cleaners<br>Regrind to 25 µm<br>3 x 1500 G cleaners<br>Sulphide scavenger flotation |

### 13.7.2.2 Variability Program Reagent Dosing Strategy

Table 13-22 summarizes the conditions used in the open circuit metallurgical variability test program with the following supporting comments:

- Reagent dosages in the flotation circuit are presented as ranges as the dosages were adjusted according to the sample characteristics. If a range is not presented, a fixed dosage was used across the variability program.
- Sodium hexametaphosphate (Calgon) and carboxy methylcellulose (CMC) were used as slime dispersants in flotation stages. Calgon was used in the rougher flotation stages while CMC was used in the cleaning stages.
- Potassium amyl xanthate (PAX reagent named KAX51) was added as a collector for flotation.
- Sulphuric acid was added in two locations in the flowsheet including the coarse rougher flotation stage, where a constant 1000 g/t of acid was added, as well as the sulphide scavenger float in the magnetic recovery circuit, where acid was dosed to bring the pulp pH to a target of 7.
- MIBC was the main frothing agent used. However, A65, which is a stronger glycol based frother, was used in small doses as well, primarily within the cleaning circuit.

**Table 13-22: Metallurgical Variability Program Reagent Dosing Strategy (Open Circuit Tests)**

| Stage | Reagents (g/t) | | | | | | Time (min) | |
|---|---|---|---|---|---|---|---|---|
| | $H_2SO_4$ (98%) | KAX51 | Calgon | CMC | MIBC | A65 | Cond. | Froth |
| Primary Grind (180 µm) | | 25 | | | | | | |
| Deslime | | | | | | | | |
| Coarse Rougher | 1000 | 20 | 200 | | 12 - 26 | 0 - 4 | 9.5 | 6 |
| Cleaner 1 | | 2 - 5 | | 2.5 - 10 | 0 - 4 | 0 - 1 | 3 | 6 |
| Cleaner 1 Regrind (60 µm) | | | | | | | | |
| Cleaner 2 | | 1 - 3 | | 0 - 6 | 0 - 3 | 0 - 0.5 | 2 | 4 |
| Cleaner 3 | | | | | 0 - 1 | 0 - 0.5 | 0.5 | 2 |
| Secondary Grind (100 µm) | | 25 | | | | | | |
| Deslime | | | | | | | | |
| Fine Rougher | | 20 | 100 | | 14 - 24 | 0 - 2.5 | 2 | 9 |
| Cleaner 1 | | 2 | | 2.5 - 10 | 0 - 4 | 0 - 0.1 | 2 | 4 |
| Cleaner 1 Regrind (60 µm) | | | | | | | | |
| Cleaner 2 | | 1 - 4 | | 2.5 - 10 | 0 - 2 | 0 - 0.6 | 2 | 3 |
| Cleaner 3 | | | | | 0 - 1.7 | 0 - 0.6 | 2 | 2 |
| Magnetic Separation on Fine Rougher Tails | | | | | | | | |
| Regrind Magnetics to 63 µm | | | | | | | | |
| Magnetic Separation (2x 1500 G) | | | | | | | | |
| Regrind magnetics to 25 µm | | | | | | | | |
| Magnetic Separation (3x 1500 G) | | | | | | | | |
| Sulphide Scavenger Flotation | 10 - 250 | 5 | | | 0 - 2 | 0 - 2 | | 3 |

Table 13-23 summarizes the typical cell type, size, and agitation speed used for the rougher flotation stages for each laboratory. Cleaning cell sizes were adjusted based on the mass pull to concentrate.

**Table 13-23: Flotation Cell Type, Size and Agitation Rate used in Rougher Flotation Stages**

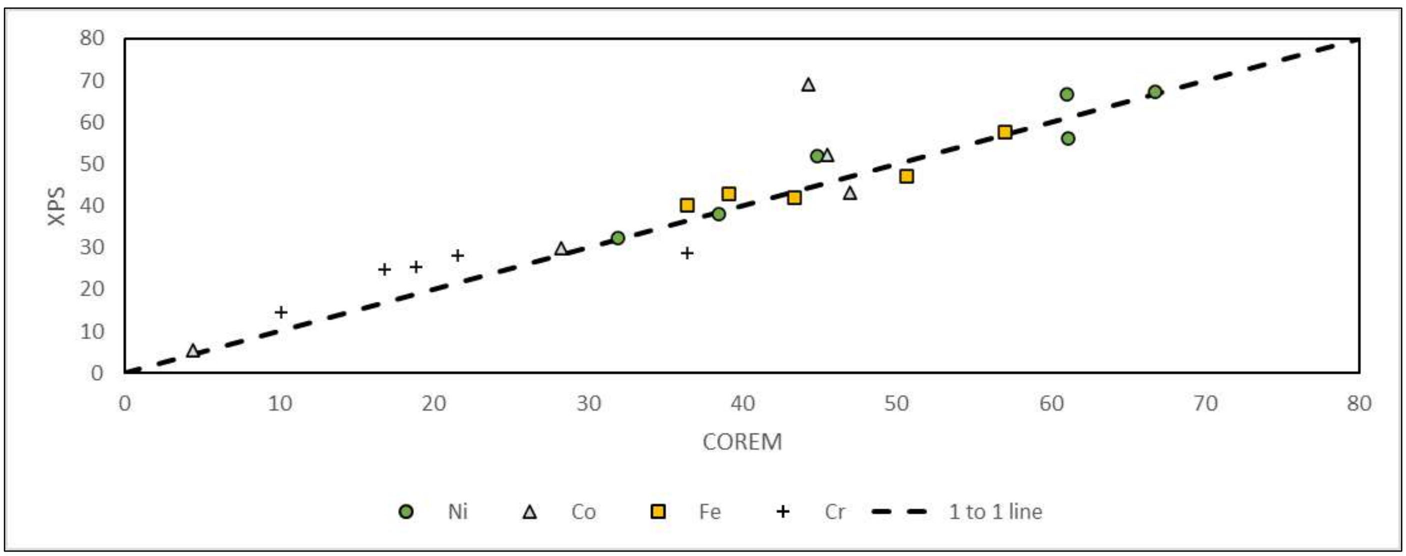
| Rougher Float Cell Parameters | XPS | COREM |
|---|---|---|
| Cell Type | Denver | Denver |
| Cell Size | 25 L | 28 L |
| Impeller Speed | 750 | 1280 |

### 13.7.3 Laboratory Synchronization

Metallurgical variability testing was conducted in parallel at COREM and XPS. To mitigate risk in biased results between laboratories, synchronizing tests on common samples were completed throughout the test program to compare recoveries and test outcomes between laboratories. Additionally, weekly meetings including the engineering team and the laboratories were held to share and discuss new results as well as new knowledge gained in the test program.

A total of six synchronizing tests were completed in the metallurgical variability test program. Figure 13-10 compares the recovery results between laboratories for nickel, cobalt, iron, and chromium with supporting observations below.

**Figure 13-10: Synchronization Test Outcomes – Recovery Comparison between Laboratories for Key Metals**




Source: CNC, 2023.

- Nickel recoveries are very comparable between laboratories with an $R^2$ of 91% and a line of best fit that aligns with the 1:1 line.
- Cobalt recoveries are comparable between laboratories except for one sample, where XPS which delivered substantially higher recoveries than COREM.
- Iron recoveries are comparable between laboratories for all tests showing synchronization.
- Chromium recoveries show a slight bias at XPS towards higher recoveries, with four out of five samples sitting above the 1:1 line. The FeCr concentrate iron grades were also lower at XPS which explains the higher recovery.

### 13.7.4 Metallurgical Variability Testing Results – Nickel

Each of the metallurgical variability samples was subjected to the standard test procedure outlined in Section 13.7.2. The results were grouped for each of the nickel recovery domains and average statistics for each domain were calculated. Table 13-24 summarizes the average open circuit test nickel recoveries and grades for each of the nickel recovery domains.

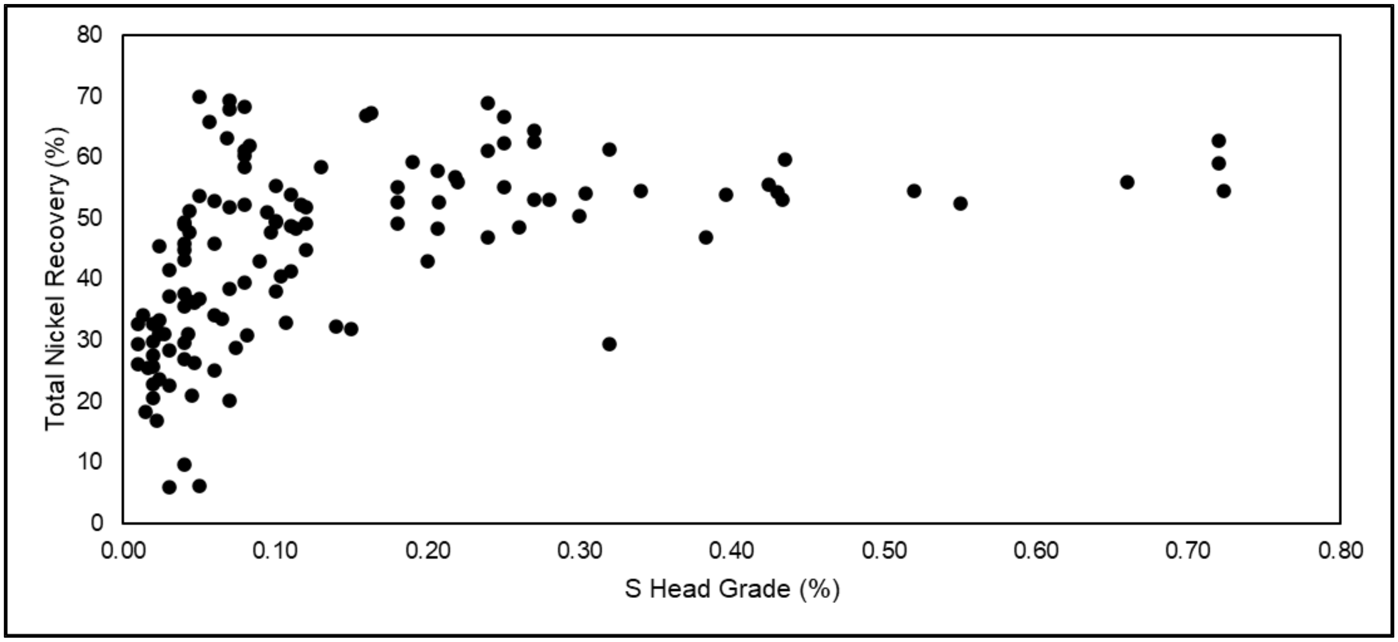
**Table 13-24: Average Nickel Grade and Recovery Performance for the Nickel Recovery Domains**

| Nickel Recovery Domain | Nickel Recovery to Coarse Cleaner 1 & Fine Rougher Concentrate (%) | Nickel Recovery to Magnetic Concentrate (%) | Nickel Concentrate Grade (Coarse Cleaner 1 & Fine Rougher Concentrate) (% Ni) | Nickel Head Grade (%) | S/Ni Ratio |
|---|---|---|---|---|---|
| Main Zone High-Grade Core | 48.1 | 2.6 | 6.7 | 0.35 | 0.75 |
| Main Zone Lower Grade Domain | 34.5 | 8.1 | 5.5 | 0.22 | 0.29 |
| East Zone | 35.6 | 10.7 | 8.1 | 0.24 | 0.25 |
| Low Magnetite | 30.8 | 2.3 | 5.9 | 0.31 | 0.42 |

For all the nickel recovery domains, sulphur is a driving variable for the nickel recovery as shown in Figure 13-11. There is variability in the nickel recoveries for samples with lower sulphur grades which is what necessitated definition of separate domains. The lowest recovery samples are from the low magnetite domain and this domain has higher levels of iron serpentine and olivine and lower levels of sulphur. Nickel recoveries plateau for high sulphur ores.

The total nickel recovery comprises the nickel that is recovered in the flotation and well as magnetic recovery circuits. A nickel splitter model was developed to estimate the deportment of nickel between the two concentrates (see Section 13.11.1.1.3). Both the nickel recovery and grade in the FeCr concentrate depend on the ore sulphur grade which is indicative of the nickel mineralization style. As the sulphur grade and S/Ni ratio of the ore decreases, the proportion of awaruite in the ore increases, and thus the nickel recovery and the final nickel grade in the FeCr concentrate generally increases. Encompassing the variability test database, nickel recoveries to the FeCr concentrate ranged from 0.6% to 21% absolute, with an average recovery of 6.1% absolute. Nickel grades in the FeCr concentrate ranged from 0.1% to 1.9% with an average grade of 0.28% nickel. The highest nickel grades achieved in the FeCr concentrate were from poorly serpentinized ores that had low levels of magnetite in the ore.

**Figure 13-11: Total Nickel Recovery vs. Sulphur Grade – Open Circuit Test Database**




Source: CNC, 2023.

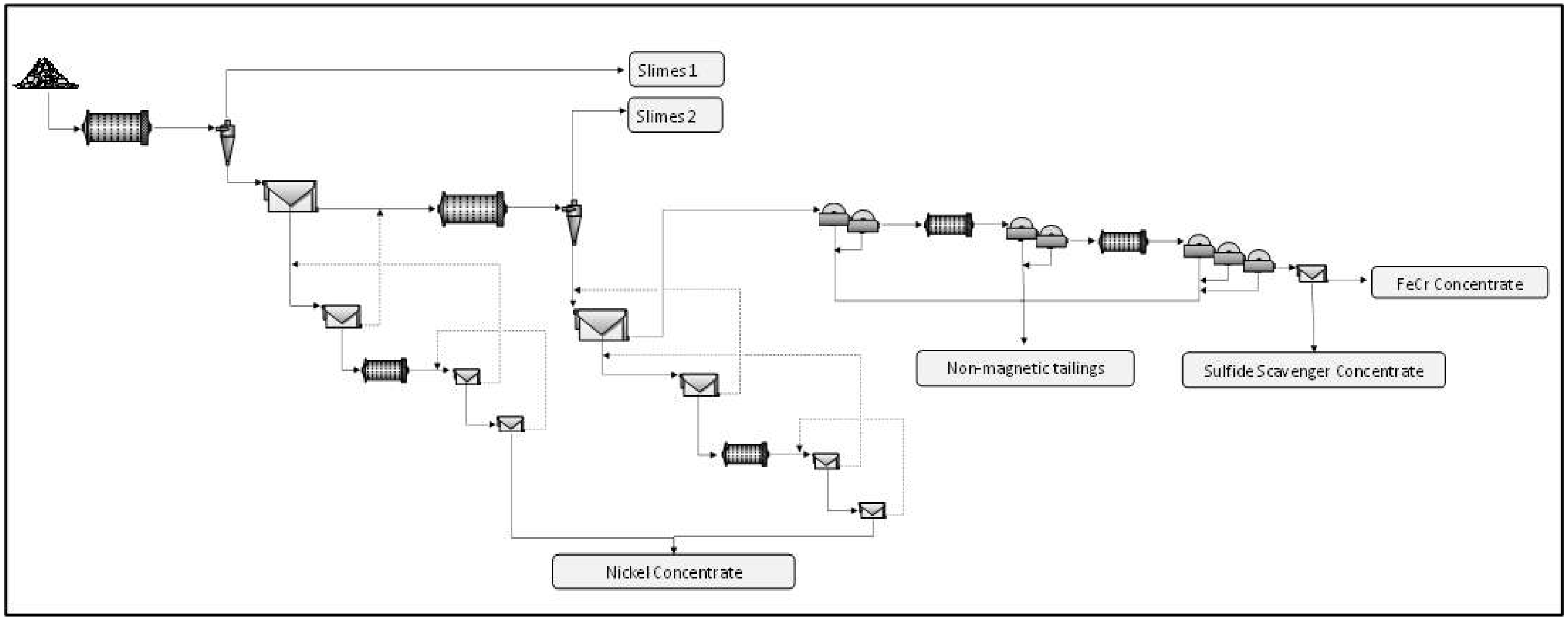
## 13.8 Locked Cycle Tests

Locked cycle tests simulate continuous operation and are indicative of product grades and recoveries that would be achieved in a plant setting. Fifteen samples were subjected to locked cycle tests to evaluate the impact of recirculating streams on product grades and recoveries, and to inform the design of recovery equations. Eight of the 15 tests were completed using the final testwork flowsheet which formed the basis for the commercial design point.

Figure 13-12 outlines the locked cycle test flowsheet that was used. The procedures outlined in Section 13.7.2 were also used in the locked cycle tests with additional steps that recirculated the cleaner tailing streams in the flotation circuits as per Figure 13-12. The nickel values in the second and third cleaner tailings eventually report to the fine flotation circuit in the locked cycle test. This is postulated to be the reason why locked cycle test nickel recovery is like the nickel recovery achieved in the open circuit tests (see Section 13.11.1.1.4).

Table 13-25 summarizes the conditions used in the locked cycle tests which were based on the conditions established in the open circuit metallurgical variability program (see Section 13.7.2.2). The values presented in Table 13-25 represent averages from the 15 locked cycle tests that were completed.

**Figure 13-12: Locked Cycle Test Flowsheet**




Source: CNC, 2023.

**Table 13-25: Locked Cycle Test Reagent Dosing Strategy**

| Stage | Reagents (g/t) | | | | | | Time (min) | |
|---|---|---|---|---|---|---|---|---|
| | $H_2SO_4$ (98%) | KAX51 | Calgon | CMC | MIBC | A65 | Cond. | Froth |
| Primary Grind (180 µm) | | 25 | | | | | | |
| Deslime | | | | | | | | |
| Coarse Rougher | 1000 | 20 | 200 | | 14.8 | | 9.5 | 6 |
| Cleaner 1 | | 5 | | 15 | 5.9 | | 3 | 6 |
| Cleaner 1 Regrind (60 µm) | | | | | | | | |
| Cleaner 2 | | 1.8 | | 2 | 0.3 | 0.4 | 2 | 4 |
| Cleaner 3 | | | | | | 0.4 | 0.5 | 2 |
| Secondary Grind (100 µm) | | 25 | | | | | | |
| Deslime | | | | | | | | |
| Fine Rougher | | 20 | 100 | | 18.5 | | 2 | 9 |
| Cleaner 1 | | 2 | | 4 | 1.8 | | 2 | 4 |
| Cleaner 1 Regrind (60 µm) | | | | | | | | |
| Cleaner 2 | | 2.5 | | | 0.4 | 0.4 | 2 | 3 |
| Cleaner 3 | | | | | 0.0 | 0.7 | 2 | 2 |
| Mag Sep on Fine Rougher Tails | | | | | | | | |
| Regrind to 63 µm | | | | | | | | |
| Mag Sep (2x 1500 G) | | | | | | | | |
| Regrind to 25 µm | | | | | | | | |
| Mag Sep (3x 1500 G) | | | | | | | | |
| Sulphide Scavenger Flotation[1] | 70 | 5 | | | | 2 | | 3 |
| Total | 1250 | 106 | 300 | 21 | 41.5 | 3.8 | 23 | 39 |

Notes: 1. Acid dosage in sulphide float is an average from 14 open circuit tests that tracked acid consumption. Acid dosages for tests completed using the final testwork flowsheet were not tracked. These are XPS results only.

Table 13-26 summarizes the locked cycle tests for nickel, Table 13-27 summarizes the locked cycle tests for byproduct metals cobalt, iron, and chromium, and Table 13-28 summarizes the locked cycle results for byproduct precious metals.

**Table 13-26: Locked Cycle Test Results Summary – Nickel**

| Sample ID | Nickel Recovery | | | Nickel Grade | | Feed Characteristics | |
|---|---|---|---|---|---|---|---|
| | Total (%) | Nickel Concentrate (%) | FeCr Concentrate (%) | Nickel Concentrate (%) | FeCr Concentrate (%) | Ni (%) | S/Ni |
| CGS-006 | **60.4** | 58.5 | 1.8 | **18** | 0.29 | 0.42 | 1.6 |
| SGS Comp 6 | **59.6** | 47.9 | 11.7 | **41** | 0.32 | 0.22 | 0.41 |
| SGS Comp 4 | **57.3** | 51.9 | 5.3 | **23** | 0.36 | 0.40 | 0.79 |
| SGS Comp 3 | **50.0** | 44.2 | 5.8 | **31** | 0.22 | 0.26 | 0.54 |
| SGS Comp 2 | **49.4** | 45.8 | 3.6 | **23** | 0.22 | 0.37 | 0.81 |
| SGS Comp 1 | **48.5** | 44.9 | 3.5 | **18** | 0.26 | 0.35 | 1.8 |
| SGS Comp 5 | **45.9** | 36.9 | 9.0 | **34** | 0.45 | 0.31 | 0.39 |
| CGS-004[1] | **63.2** | 60.4 | 2.8 | **30** | 0.09 | 0.23 | 0.70 |
| CES-006[1] | **60.7** | 53.7 | 7.0 | **46** | 0.17 | 0.19 | 0.32 |
| CDS-004A[1] | **58.4** | 55.2 | 3.2 | **25** | 0.15 | 0.28 | 0.73 |
| CKS Dunite Comp[1] | **49.1** | 33.2 | 15.9 | **27** | 0.31 | 0.18 | 0.11 |
| CKS Peridotite Comp[1] | **47.5** | 34.6 | 12.9 | **26** | 0.25 | 0.18 | 0.17 |
| CKS Low Grade Blend[1] | **37.0** | 24.3 | 12.7 | **22** | 0.20 | 0.13 | 0.15 |
| CES-005[1] | **34.7** | 32.3 | 2.4 | **37** | 0.13 | 0.27 | 0.11 |
| CGS-001[1] | **30.3** | 18.6 | 11.7 | **25** | 0.36 | 0.24 | 0.08 |

Notes: 1. Tests were completed using an outdated magnetic circuit which resulted in lower recoveries as well as lower grades in the concentrate.

**Table 13-27: Locked Cycle Test Results Summary – Byproduct Metals**

| Sample ID | Cobalt[1] | | Iron[2] | | Chromium[2] | | Feed Characteristics | |
|---|---|---|---|---|---|---|---|---|
| | Recovery (%) | Grade (%) | Recovery (%) | Grade (%) | Recovery (%) | Grade (%) | Ni | S/Ni |
| CGS-006[3] | 73 | 1.0 | 30 | 61 | 11 | 3.8 | 0.42 | 1.58 |
| SGS Comp 6[3] | 39 | 2.0 | 58 | 55 | 30 | 2.4 | 0.22 | 0.41 |
| SGS Comp 4[3] | 44 | 0.8 | 47 | 55 | 34 | 3.8 | 0.40 | 0.79 |
| SGS Comp 3[3] | 29 | 1.1 | 48 | 55 | 23 | 2.5 | 0.26 | 0.54 |
| SGS Comp 2[3] | 38 | 0.8 | 47 | 56 | 24 | 2.4 | 0.37 | 0.81 |
| SGS Comp 1[3] | 57 | 1.0 | 41 | 60 | 26 | 3.8 | 0.35 | 1.76 |
| SGS Comp 5[3] | 9 | 0.4 | 54 | 56 | 29 | 3.6 | 0.31 | 0.39 |
| CGS-004[4] | 68 | 2.1 | 52 | 51 | 32 | 2.3 | 0.23 | 0.70 |
| CES-006[4] | 2 | 0.1 | 53 | 51 | 17 | 1.2 | 0.19 | 0.32 |
| CDS-004A[4] | 50 | 1.3 | 45 | 51 | 19 | 1.5 | 0.28 | 0.73 |
| CKS Dunite Comp[4] | 2 | 0.1 | 66 | 52 | 25 | 1.9 | 0.18 | 0.11 |
| CKS Peridotite Comp[4] | 2 | 0.1 | 60 | 47 | 33 | 1.8 | 0.18 | 0.17 |
| CKS Low Grade Blend[4] | 2 | 0.1 | 60 | 51 | 27 | 1.5 | 0.13 | 0.15 |
| CES-005[4] | 1 | 0.04 | 46 | 53 | 25 | 3.6 | 0.27 | 0.11 |
| CGS-001[4] | 1 | 0.04 | 54 | 48 | 28 | 2.1 | 0.24 | 0.08 |

Notes: **1.** Cobalt grade and recoveries are measured in the nickel concentrate. **2.** Iron and chromium grades and are recovery are measured in the FeCr concentrate. **3.** Tests were completed using the final testwork flowsheet. The magnetic recovery circuit reflects the commercial design basis. **4.** Tests were completed using an outdated magnetic circuit which resulted in lower recoveries as well as lower grades in the concentrate.

**Table 13-28: Locked Cycle Test Results Summary – Precious Metals**

| Sample | Nickel Concentrate Grade (g/t) | | | Recovery (%) | | | Feed Characteristics | |
|---|---|---|---|---|---|---|---|---|
| | Pd | Pt | Au | Pd | Pt | Au | Ni | S |
| CDS-004A | 10.8 | 3.4 | 0.1 | 56 | 52 | 34 | 0.28 | 0.73 |
| CGS-004 | 4.1 | 1.8 | 0.4 | 58 | 44 | 48 | 0.23 | 0.71 |
| SGS Comp 5 | 4.1 | 1.7 | 0.3 | 57 | 31 | 4 | 0.31 | 0.39 |
| SGS Comp 4 | 3.7 | 1.7 | 0.2 | 76 | 40 | 6 | 0.40 | 0.80 |
| SGS Comp 3 | 2.6 | 1.7 | 0.4 | 61 | 23 | 5 | 0.26 | 0.54 |
| SGS Comp 2 | 3.0 | 1.2 | 0.04 | 47 | 37 | 2 | 0.37 | 0.81 |
| SGS Comp 6 | 1.6 | 2.0 | 1.2 | 28 | 20 | 13 | 0.22 | 0.41 |
| CGS-006 | 2.5 | 0.6 | 0.03 | 66 | 26 | 2 | 0.43 | 1.53 |
| SGS Comp 1 | 2.1 | 0.7 | 0.1 | 46 | 28 | 4 | 0.34 | 1.82 |
| CKS-Peridotite Comp | 1.3 | 0.6 | 0.2 | 45 | 10 | 6 | 0.18 | 0.19 |
| CES-005 | 0.8 | 0.2 | 0.04 | 32 | 12 | 4 | 0.27 | 0.11 |

Key points regarding Tables 13-26 to 13-28 are summarized below:

- Total nickel recoveries ranged from 30% to 63% for head grades between 0.13 and 0.42% nickel.
- Nickel concentrate grades were a function of the sample sulphur to nickel ratio and ranged from 18% to 46% nickel. The difference in concentrate grades is due to the different mineralization styles. Pentlandite dominant ores produced concentrates with a grade of 18% nickel and heazlewoodite dominant ores produced concentrates as high as 46% nickel. Some of the lower grade samples struggled to upgrade due to low mass pull but these samples resulted in higher recoveries than expected. It is reasonable to expect that in a continuous operation, with improved froth crowding, these lower grade samples will deliver modelled grades at target recovery.
- Cobalt recoveries ranged from 1 to 73% in locked cycle tests. Higher cobalt recoveries were achieved in samples that had higher sulphur grades, likely because cobalt is associated with pentlandite minerals. For some samples, cobalt recoveries exceeded nickel recoveries, which is likely due to a distinct cobalt mineral called cobalt pentlandite (see the microprobe results in Section 13.3.2).
- Seven of the locked cycle tests were completed with the final testwork flowsheet that formed the basis for the commercial design point. The iron grade in final concentrate for these tests ranged from 55 to 61%. The older testwork flowsheet delivered concentrate grades between 47 to 53% iron. This comparison highlights the improvements delivered by optimizations to the magnetic recovery circuit.
- Chromium recoveries using the final testwork flowsheet ranged from 11% to 34% at grades between 2.4% to 3.4% chromium. Chromium grades in the FeCr concentrate are higher in tests completed with the final testwork flowsheet relative to the outdated flowsheet due to better rejection of gangue minerals.
- Precious metal recoveries ranged from 28% to 76% at grades between 0.8 to 10.8 g/t for palladium, and 10% to 52% at grades between 0.2 to 3.4 g/t for platinum. The average combined grade of platinum and palladium for locked cycle tests was 4.7 g/t Pt + Pd. Most samples had low gold grades in the final nickel concentrate. However, sample SGS Comp 6 had a head grade of 1.2 g/t gold.

## 13.9 Pilot Plant

Pilot plant testing was completed at SGS Lakefield in the second half of 2022 to evaluate the performance of the Crawford metallurgical flowsheet with continuous feed as well as to generate concentrate for downstream flowsheet development work and marketing purposes. A total of 34 tonnes of material was tested across six composites. One composite was used for commissioning. Excluding the commissioning composite, the remaining five composites ranged in size from four to seven tonnes. The pilot plant composites were built from 53 samples of large diameter drill core from 19 holes to represent the key lithologies, zones of the deposit and mineralization styles at Crawford. Drilling was focussed on areas of the deposit that were expected to be processed during the project payback period:

- SGS Composites 1, 2 and 3 were built from material taken from the eastern half of the Main Zone where mining is expected to start and would constitute payback period ore.
- SGS Composite 4 was built from mixed material with a range of ore grades, mineralization styles, and lithologies from both the Main and East Zones of the deposit. SGS Composite 4 was used to commission the plant and the results were excluded from the feasibility study dataset. The material used to build this composite is expected to be from outside of the payback pit.
- SGS Composite 5 from the East Zone, was built from material sampled from the western part of the East Zone, which would not constitute payback period material.
- SGS Composite 6, which is the peridotite composite, was constructed primarily with Main Zone ore. However, some East Zone peridotite was added to the Main Zone composite to make up the required sample mass. Most of the material in Composite 6 would likely be processed during the project payback period.

Table 13-29 summarizes the lithologies, zones, ore domains and head grades for the five main composites. Head grades ranged from 0.20% to 0.33% nickel and 0.07% to 0.49% sulphur and encompassed material from both the Main and East Zones of Crawford. As the main lithology at Crawford is dunite, four out of the five composites targeted the dunite lithology and one of the composites was targeted in peridotite.

**Table 13-29: Pilot Plant Composite Head Grades (Excluding the Commissioning Composite)**

| Composite | Lithology | Zone | Domain | Head Grades (%) | | | |
|---|---|---|---|---|---|---|---|
| | | | | Ni | S | Fe | Cr |
| SGS Comp 1 | Dunite | Main | High-grade Core | 0.33 | 0.49 | 6.5 | 0.58 |
| SGS Comp 2 | Dunite | Main | High-grade Core | 0.30 | 0.27 | 7.0 | 0.50 |
| SGS Comp 3 | Dunite | Main | Lower Grade Domain | 0.24 | 0.12 | 6.8 | 0.62 |
| SGS Comp 5 | Dunite | East | Mixed | 0.29 | 0.08 | 6.1 | 0.65 |
| SGS Comp 6 | Peridotite | Main + East | Lower Grade Domain | 0.20 | 0.07 | 7.3 | 0.53 |

Pilot plant testing was executed in three stages at SGS. The decision to break up the flowsheet was made based on the availability of equipment for the chosen feed rate of 130 kg/hour.

The flowsheet was broken into the following sections:

- Base flowsheet including the coarse and fine flotation circuits up to and including the cleaner 1 flotation cell, the magnetic roughing stage and the first stage of magnetic cleaning. This phase of the pilot plant produced intermediate concentrates that required further upgrading. Recovery and grade targets for the base flowsheet were selected based on results from open circuit test results established in the lab. The pilot plant cleaner 1 recovery targets were set based on the open circuit test cleaner 1 recoveries. Grab assays taken for process control were used to guide the process, targeting the tail grades that were produced in the open circuit laboratory tests.

- As all stages of flotation cleaning were not included in the base flowsheet, the grade-recovery performance of locked cycle tests at the cleaner 3 concentrate cannot be compared to the base pilot for the flotation circuit which was only done up to cleaner 1.

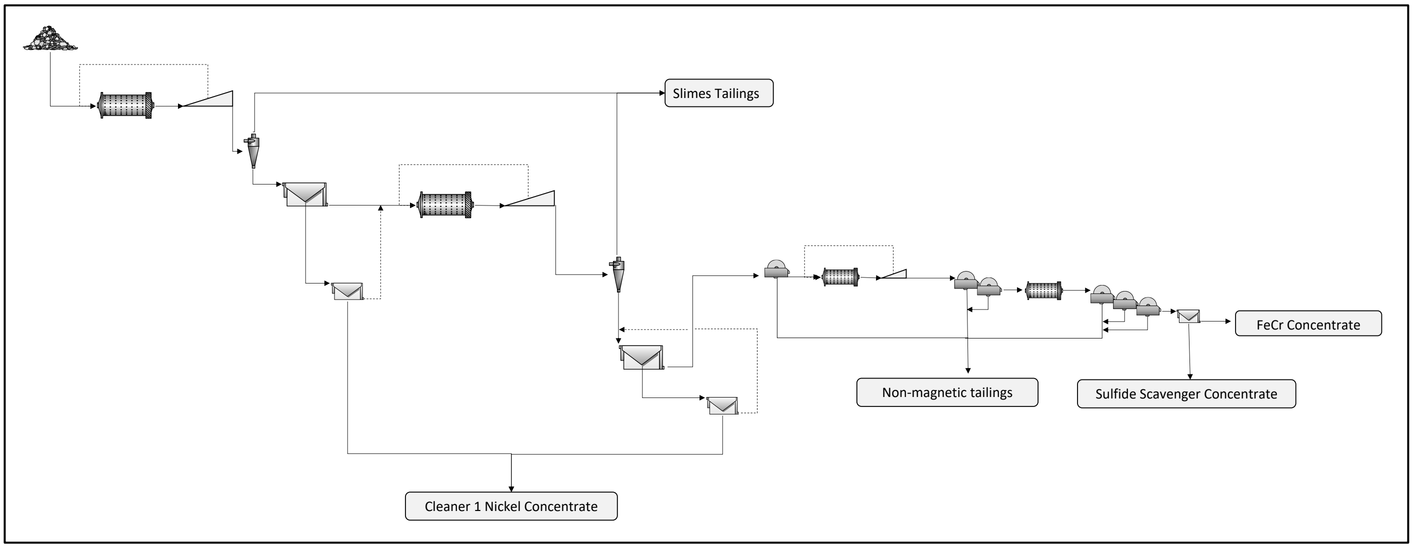
- Magnetic cleaning stages to produce the final FeCr concentrate. This second phase of pilot plant testing processed the intermediate magnetic cleaner 1 concentrate into the final FeCr concentrate through a continuous pilot run. Pilot testing on each composite was done in two campaigns to evaluate the impact of grind size on grades and recoveries to the final FeCr concentrate. The phase 2 mass balance was mapped onto the base flowsheet balance to estimate the grade-recovery performance of iron and chromium. Combining these balances was possible because the magnetic recovery circuit does not contain any recirculating tailing streams and thus, iron and chromium recoveries can be used to support recovery modelling efforts.

- Final flotation cleaning stages including the cleaner 2 and cleaner 3 flotation cells. This phase of testwork processed the intermediate cleaner 1 flotation concentrates into the final nickel concentrate through bench scale locked cycle testing. The decision to separate the flotation cleaning stages from the base flowsheet was made to improve control over the flotation cleaning stages, which if completed as part of the base pilot flowsheet, would struggle to reach stability due to the low mass pulls and a resultant large recirculating water stream within the flotation circuit. The locked cycle cleaning tests confirmed the ability to upgrade the intermediate concentrates.

The results of the locked cycle cleaning tests were not coupled with the base flowsheet results as the impact of recirculating streams in the locked cycle tests (that return to prior pilot plant stages) could not be addressed. In future pilot plants, the feed rate should be increased so that final cleaning stages can be included in the base flowsheet to assess the grade-recovery performance to the final nickel concentrate at pilot scale relative to locked cycle test results in the lab.

Figure 13-13 shows the flowsheet that was tested as part of the pilot plant. This flowsheet is similar to the feasibility study plant flowsheet.

**Figure 13-13: Crawford Flowsheet and Phases of the Pilot Plant**




Source: CNC, 2023.

### 13.9.1 Pilot Plant Results

To prepare for the pilot plant, open circuit testing was completed on each of the composites to establish flotation circuit recovery targets as well as tailing grade targets for process control. In the pilot plant, the flotation circuit recovery targets were set as the cleaner 1 recovery that was achieved in the laboratory-scale open circuit tests.

Table 13-30 summarizes the pilot plant results which includes the nickel and cobalt recoveries up to the first cleaner flotation concentrate as well as the final nickel, iron and chromium recoveries in the magnetic recovery circuit measured after the final magnetic drum separator (also the feed to the desulphurization float). The nickel recovery that is presented in Table 13-30 is the sum of recoveries to the cleaner 1 flotation concentrate and the final magnetic concentrate. The following observations are made based on Table 13-30.

- Total nickel recoveries in the pilot plant ranged from 44-65%, with the peridotite composite delivering the highest recovery. The cleaner 1 flotation concentrate grades were higher than anticipated, likely due to better froth crowding in the continuous operation.
- Cobalt recoveries to the cleaner 1 concentrate ranged from 8% to 43%. As expected, cobalt recoveries were highest in Composite 1 which is a pentlandite dominant composite. The lower cobalt recoveries achieved in Composite 3 were expected due to the heazlewoodite dominant mineralization which has low cobalt content in the nickel sulphide minerals.
- Iron recoveries ranged from 46% to 59% at grades between 54% to 56% iron. The FeCr concentrate were aligned with the modelled grade target of 55% iron except for Composite 3 which had a grade of 54% iron.

- Chromium recoveries ranged from 27% to 32% at grades between 2.1% to 3.5% The chromium recoveries achieved in the pilot plant were higher than anticipated, which is likely due to the 2500 G rougher magnet that was used. Based on the results in the pilot plant, it was decided to incorporate the 2500 G rougher magnet into the flowsheet for the commercial design point and recovery equations were designed to reflect the strong pilot plant performance.

**Table 13-30: Pilot Plant – Summary of Results for Base Flowsheet and Magnetic Recovery Circuits**

| Composite No. | Head Grade | Pilot Plant Recovery | | | | Cleaner 1 Nickel Concentrate Grade | Final FeCr Concentrate Grade | | |
|---|---|---|---|---|---|---|---|---|---|
| | Ni | Ni[7] | Co | Fe | Cr | Ni | Fe | Cr | Ni |
| SGS Comp 1 | 0.33% | 48% | 43% | 46% | 32% | 11% | 55% | 3.5% | 0.24% |
| SGS Comp 2 | 0.30% | 44% | 30% | 51% | 32% | 8% | 56% | 2.4% | 0.16% |
| SGS Comp 3 | 0.24% | 53% | 29% | 59% | 32% | 20% | 54% | 2.6% | 0.16% |
| SGS Comp 5 | 0.29% | 52% | 8% | 56% | 32% | 23% | 55% | 3.3% | 0.36% |
| SGS Comp 6 | 0.20% | 65% | 34% | 57% | 27% | 24% | 55% | 2.1% | 0.21% |

Notes: 1. Pilot plant total nickel recovery is the sum of nickel recoveries to the coarse cleaner 1 concentrate, the fine cleaner 1 concentrate and the final FeCr concentrate.

#### 13.9.1.1 Flotation Circuit Performance Evaluation

To evaluate the performance of the flotation circuit, nickel recoveries to the cleaner 1 concentrate were compared between the laboratory-scale open circuit tests and the pilot plant balances. Table 13-31 compares the outcomes with supporting observations below:

Nickel recoveries to the cleaner 1 flotation concentrate in the pilot plant exceeded open circuit test recoveries to the cleaner 1 concentrate for three out of five composites.

Composites 3, 5 and 6 delivered excellent performance with all three composites exceeding the open circuit test recovery and grade targets.

Composite 1 fell short of the target recovery by six percentage points. The campaign on Composite 1 was short and had significant process upsets due to rolling power outages on the final day of operation that affected pilot plant performance. It is expected that with more time, target recoveries could be achieved.

Composite 2 fell short of the target recovery by five percentage points and the grade of the cleaner 1 concentrate was the same as was achieved in the open circuit test. During this campaign, the pilot plant was still being commissioned which likely contributed to the poor results. CNC expects that the performance of this composite can be improved.

[7] Pilot plant total nickel recovery is the sum of nickel recoveries to the coarse cleaner 1 concentrate, the fine cleaner 1 concentrate and the final FeCr concentrate.

The pilot plant flotation grades reflect the cleaner 1 concentrate and show excellent upgrading potential with three out of five composites delivering grades greater than 20% nickel after only one cleaning stage. For all composites, with two more cleaning stages, the ability to further upgrade the concentrates was confirmed.

**Table 13-31: Pilot Plant Flotation Circuit Recovery Analysis**

| Composite | Lithology | Nickel Head Grade | Ni Concentrate Recovery (%) | | | Concentrate Grade (% Ni) | |
|---|---|---|---|---|---|---|---|
| | | | Open Circuit (Cleaner 1) | Pilot Plant (Cleaner 1) | Pilot Plant – Open Circuit | Open Circuit (Cleaner 1) | Pilot Plant (Cleaner 1) |
| SGS Comp 1 | Dunite | 0.33% | 50 | 44 | -6 | 8 | 11 |
| SGS Comp 2 | Dunite | 0.30% | 46 | 41 | -5 | 8 | 8 |
| SGS Comp 3 | Dunite | 0.24% | 45 | 49 | +4 | 12 | 20 |
| SGS Comp 5 | Dunite | 0.29% | 40 | 42 | +2 | 15 | 23 |
| SGS Comp 6 | Peridotite | 0.20% | 50 | 58 | +8 | 16 | 24 |

### 13.9.2 Magnetic Circuit Performance

Iron and chromium recoveries in the magnetic circuit were compared to the locked cycle tests as the base flowsheet and magnetic recovery circuit balances can be coupled together. Table 13-32 compares the recovery outcomes with the following comments:

- Iron recoveries in the pilot plant exceeded the locked cycle test result for all composites with an average improvement of 4 percentage points.
- Chromium recoveries exceeded the locked cycle test results for four out of five composites with an average improvement of 5 percentage points relative to the locked cycle tests. The pilot plant flowsheet utilized a 2500 G rougher magnet which delivered substantial improvements in the chromium recovery performance.

**Table 13-32: Grade Recovery Performance of Iron and Chromium Relative to Locked Cycle Tests**

| Composite | Nickel Head Grade | Iron Recovery (%) | | | Chromium Recovery (%) | | |
|---|---|---|---|---|---|---|---|
| | | Locked Cycle Test | Pilot Plant | Difference | Locked Cycle Test | Pilot Plant | Difference |
| SGS Comp 1 | 0.33% | 42 | 46 | +4 | 26 | 32 | +6 |
| SGS Comp 2 | 0.30% | 47 | 51 | +4 | 25 | 32 | +7 |
| SGS Comp 3 | 0.24% | 48 | 59 | +11 | 23 | 32 | +9 |
| SGS Comp 5 | 0.29% | 53 | 56 | +3 | 28 | 32 | +4 |
| SGS Comp 6 | 0.20% | 57 | 57 | 0 | 30 | 27 | -3 |
| | | | | **Avg = +4** | | | **Avg = +5** |

For the feasibility study design basis, a final grind $P_{80}$ size of 25 µm was selected based on the magnetic circuit optimization work completed (see Section 13.6.4). The decision to utilize a finer grind size was made to achieve higher concentrate grades in the final concentrate. As part of the pilot plant campaign, the impact of the final grind size on product qualities and recoveries to final concentrate were evaluated to support this decision.

Table 13-33 compares the grades and recoveries of the final FeCr concentrate at $P_{80}$ of 25 µm and $P_{80}$ of 35 µm with the following observations:

- Across all composites tested, the iron grade of the final FeCr concentrate improved using a $P_{80}$ of 25 µm versus a $P_{80}$ of 35 µm final grind size, which confirms the Davis tube results outlined in Section 13.6.4. The average improvement across all composites in iron grade was 3 percentage points. There was a negligible change in the iron recovery as a function of grind size, which is likely because magnetite is a highly magnetic mineral. The grade improvements were related to improved gangue rejection due to better liberation.
- Chromium recoveries to the final magnetic concentrate and resulting grades were lower at a $P_{80}$ of 25 µm versus a $P_{80}$ of 35 µm final grind size, which was expected. The average change in chromium recovery and grade at $P_{80}$ of 25 µm versus $P_{80}$ of 35 µm was -3.3% absolute and -0.1% absolute respectively.
- Nickel recoveries and grades in the final FeCr concentrate were lower at a $P_{80}$ of 25 µm versus a $P_{80}$ of 35 µm by 0.6% absolute and 0.01% absolute.

**Table 13-33: Pilot Plant Testing – Effect of Final Grind Size on the Grade and Recovery to the Final FeCr Concentrate**

| Comp | Grind Size ($P_{80}$) | Final FeCr Concentrate Grade (%) | | | Recovery to FeCr Concentrate (%) | | | |
|---|---|---|---|---|---|---|---|---|
| | | Ni | Fe | Cr | Mass % | Ni | Fe | Cr |
| SGS Comp 1 | 25 µm | 0.24 | 55 | 3.5 | 5.5 | 4.3 | 46 | 32 |
| SGS Comp 2 | 25 µm | 0.16 | 56 | 2.4 | 6.8 | 3.2 | 51 | 32 |
| SGS Comp 3 | 25 µm | 0.16 | 54 | 2.6 | 7.3 | 4.1 | 59 | 32 |
| SGS Comp 5 | 25 µm | 0.36 | 55 | 3.3 | 6.7 | 9.3 | 56 | 32 |
| SGS Comp 6 | 25 µm | 0.21 | 55 | 2.1 | 7.3 | 7.5 | 57 | 27 |
| SGS Comp 1 | 35 µm | 0.28 | 52 | 3.6 | 5.7 | 5.4 | 46 | 35 |
| SGS Comp 2 | 35 µm | 0.19 | 52 | 2.5 | 6.8 | 4.0 | 51 | 34 |
| SGS Comp 3 | 35 µm | 0.17 | 52 | 2.7 | 7.7 | 5.0 | 59 | 34 |
| SGS Comp 5 | 35 µm | 0.35 | 51 | 3.3 | 7.2 | 9.7 | 56 | 35 |
| SGS Comp 6 | 35 µm | 0.21 | 50 | 2.2 | 8.8 | 7.5 | 58 | 33 |
| **Average** | **25 µm** | **0.23** | **55** | **2.8** | **6.7** | **5.7** | **54** | **31** |
| **Average** | **35 µm** | **0.24** | **51** | **2.9** | **7.2** | **6.3** | **54** | **34** |
| **Difference** | **25 µm - 35 µm** | **-0.01** | **3** | **-0.1** | **-0.5** | **-0.6** | **0** | **-3** |

### 13.9.3 Pilot Plant Continuous Run Data

During the pilot execution, grab assays were taken to assess performance and support operational decisions including adjustments to reagents and other operating parameters. The grab assays confirmed the ability to quickly stabilize the process and control the process around target tailing and concentrate grades. Grab assays were taken at the following locations with comments:

- Coarse and fine cleaner 1 concentrates within the flotation circuit. The nickel grades of these concentrates were measured to ensure the target grades were being met or exceeded. Timed cuts of the concentrate streams were taken to estimate recoveries to the concentrates, support the grade measurements and adjust the plant operating parameters. Through inspection of the cleaner 1 concentrate visually, process performance could be diagnosed effectively. It was found that the pilot plant could exceed the laboratory results in terms of cleaner 1 concentrate grades which is likely due to the froth stability that comes with continuous operation and results in improved gangue rejection.
- Nickel grades and deslime cyclone $P_{80}$ particle sizes were monitored throughout the pilot campaign. There were no issues with cyclone performance. Mass pulls and nickel losses to the cyclone overflow streams agreed for pilot plant and bench open circuit and locked cycle tests.
- Nickel tailing grades in the non-magnetic tailing streams were measured. The low-intensity magnetic separation (LIMS) rougher non-magnetic stream contains about 50% of the tailings. Monitoring the nickel grades on this stream is essential to achieving target recoveries as any nickel sulphides that are not recovered in the flotation circuit would likely report to this stream. Three out of the four composite runs were able to achieve stable operation at or below the tailing grade target which is an excellent result. The only composite which was not able to achieve the target rougher non-magnetic tail grade for nickel was Composite 1. The Composite 1 campaign was the shortest run of the pilot plant and there were a substantial number of upsets to the process during its execution including off-target grinding and rolling power outages on the last day. Additionally, the open circuit test had higher recoveries than the locked cycle test and thus, the grade target may have been falsely low.
- Iron tailing grades in the non-magnetic tailing streams within the magnetic recovery circuit are an excellent indicator of the iron recovery and magnet performance. It was found that the best process control check on the magnets was to use a hand magnet to check for magnetite losses in the tailings streams, which could be used to adjust operation of the magnets if losses were too high.

## 13.10 Other Engineering Testwork

Engineering testwork was completed to reduce risk associated with the engineering design of the concentrator in the following areas:

- Tailings dewatering testing to support the engineering design of the thickener.
- Rheology testing to support the design of pumping systems and assess the impact of slurry rheology on design.
- Deslime cyclone characterization through cyclosizing and size by assay to support the design of the deslime cyclones.

### 13.10.1 Tailings Dewatering

Tailings thickening testwork was completed in two stages to support the sizing of the tailings thickener:

- Dynamic thickening testwork was first completed by Metso on one sample at the SGS Lakefield lab. The sample that provided the tailings was a well serpentinized dunite sample from the Main Zone high-grade core, that is expected to be processed in year 4. This sample is believed to be a good representation of typical ore that will feed the mill during the payback period.
- Follow-up Kynch settling tests on two samples of tailings from the pilot plant was then completed to validate the results from the dynamic tests. The samples were selected from the Main Zone to represent the two key lithologies, dunite and peridotite, and were generated from composites expected to be processed during the project payback period.

The results of dynamic thickening tests, conducted using Metso Outotec's bench scale 99 mm diameter thickener test, are summarized in Table 13-34, with supporting observations below:

- The flocculant used in the testwork was SNF 905 SH which was selected based on static cylinder tests.
- For all tests, the overflow water clarity was < 200 mg/L which was the target at the start of the program. This was achieved both with and without flocculant addition however flocculant addition enabled overflow clarities <100 mg/L.
- Testwork demonstrated the ability to achieve underflow densities of 40% solids both with and without flocculant. Settling rates are greater when flocculant is used.
- The yield stress of the thickened solids is proportional to the underflow density which is expected.

**Table 13-34: Dynamic Thickening Test Results**

| Run No. | Feed | | Flocculant Type | Underflow | | | Overflow |
|---|---|---|---|---|---|---|---|
| | Flux | Liquor Rise Rate | | Dose | Meas. Solids | Yield Stress | Solids Clarity |
| | (t/m$^2$/h) | (m/h) | | (g/t) | (% w/w) | (Pa) | (mg/L) |
| 1 | 0.4 | 3.09 | NA | 0 | 43.8 | 10 | 145 |
| 2 | 0.2 | 1.54 | | 0 | 50.2 | 59 | 141 |
| 3 | 0.6 | 4.63 | | 0 | 37.8 | 5 | 179 |
| 4 | 0.6 | 4.63 | SNF 905 SH | 20 | 46.6 | 147 | <100 |
| 5 | 0.6 | 4.63 | | 10 | 46.9 | 139 | <100 |
| 6 | 0.6 | 4.63 | | 30 | 42.2 | 80 | <100 |
| 7 | 0.75 | 5.79 | | 10 | 40.3 | 38 | <100 |
| 8 | 0.9 | 6.95 | | 10 | 40.2 | 38 | <100 |

Kynch settling tests were conducted to support the dynamic settling tests in Table 13-34 and to evaluate variability. Table 13-35 summarizes the results of the Kynch settling tests which confirm the ability to achieve underflow densities of 40% solids as well as overflow clarities < 200 mg/L.

**Table 13-35: Static Thickening Testwork**

| Sample ID | Dosage | Feed | Underflow | Unit Area | Total Suspended Solids |
|---|---|---|---|---|---|
| | g/t | w/w% | w/w% | $m^2/(t/day)$ | mg/L |
| Comp 2 Combined Tailings PP-10 | 0 | 19.1 | 40 | 0.54 | 79 |
| | 10 | 19.1 | 39 | 0.53 | 59 |
| | 20 | 19.1 | 42 | 0.41 | 71 |
| | 30 | 19.1 | 42 | 0.34 | 58 |
| Comp 6 Combined Tailings PP-15 | 0 | 19.9 | 38 | 0.66 | 129 |
| | 10 | 19.9 | 38 | 0.65 | 71 |
| | 20 | 19.9 | 41 | 0.48 | 106 |
| | 30 | 19.9 | 43 | 0.4 | 117 |

### 13.10.2 Rheology Testing

Rheology testing was completed on the primary deslime cyclone overflow and underflow for two metallurgical variability samples (two streams x two samples = four results). The two metallurgical variability samples that were selected for testing were from the Main Zone and were selected to represent the two key lithological units at Crawford with different brucite contents: dunite (3.7% brucite) and peridotite (0.1% brucite). Brucite is a mineral that can cause rheological issues and thus it was important to consider in the sample selection criteria.

The rheology samples were prepared at Expert Process Solutions (XPS) by grinding the feed to the target $P_{80}$ of 180 µm that was used in the standard test procedure and then passing the sample through the deslime cyclone to produce the primary cyclone overflow and underflow samples. Samples were kept in slurry form and shipped to SGS Lakefield for characterization. The results are presented in Table 13-36. As expected, the deslime overflow samples are thixotropic and any processing of this material is likely to require slurry solids from 5% to 10%.

**Table 13-36: Rheology Test Results**

| Sample | Test Code | Solids % w/w | Bingham Plastic parameters | | | | | | Observations |
|---|---|---|---|---|---|---|---|---|---|
| | | | Unsheared Sample | | | Sheared Sample | | | |
| | | | Shear Rate | Yield Stress | Plastic Viscosity | Shear Rate | Yield Stress | Plastic Viscosity | |
| | | | g | $t_{yB}$ | $h_P$ | g | $t_{yB}$ | $h_P$ | |
| | | | range, 1/s | Pa | mPa.s | range, 1/s | Pa | mPa.s | |
| | CFS-002 Underflow - CSD = ~71% solids, corresponding to ~50 Pa unsheared yield stress. | | | | | | | | |
| CFS-002 Brucite = 3.7% Dunite Deslime Underflow | T8V | 74.5 | Vane | 140 | -- | -- | -- | -- | Vane yield stress |
| | T7V | 73.3 | Vane | 101 | -- | -- | -- | -- | Vane yield stress |
| | T2 | 70.9 | 200-400 | 39 | 155 | Sheared data are invalid due to fast settling | | | Settling |
| | T3 | 68.9 | 200-400 | 30 | 52 | | | | Settling |
| | T5 | 63.4 | 200-400 | 5.7 | 33 | | | | Fast settling |
| | T6 | 60.8 | 200-400 | 2.8 | 23 | | | | Fast settling |
| | CFS-002 Overflow - CSD = ~24.5% solids, corresponding to ~30 Pa unsheared and 16 Pa sheared yield stress. | | | | | | | | |
| CFS-002 Brucite = 3.7% Dunite Deslime Overflow | T17 | 28.1 | Plug Flow | 64 | -- | 100-300 | 29 | 20 | Thixotropic |
| | T18 | 25.9 | 100-300 | 37 | 16 | 100-300 | 20 | 17 | Thixotropic |
| | T19 | 23.5 | 100-300 | 27 | 13 | 100-300 | 14 | 14 | Thixotropic |
| | T20 | 21.6 | 100-300 | 20 | 7.7 | 100-300 | 11 | 13 | Thixotropic |
| | T21 | 18.2 | 100-300 | 11 | 10 | 100-300 | 7.4 | 11 | Thixotropic |
| | T22 | 14.0 | 100-300 | 4.6 | 8.4 | 100-300 | 4.1 | 8.0 | None |
| | CFS-023 Underflow - CSD = ~67% solids, corresponding to ~ 92 Pa unsheared yield stress. | | | | | | | | |
| CFS-002 Brucite = 0.1% Peridotite Deslime Underflow | T10V | 69.0 | Vane | 273 | -- | -- | -- | -- | Vane yield stress |
| | T16V | 68.2 | Vane | 184 | -- | -- | -- | -- | Vane yield stress |
| | T9 | 67.0 | 200-400 | 51 | 202 | 200-400 | 39 | 272 | None |
| | T12 | 63.1 | 200-400 | 36 | 86 | 200-400 | 15 | 80 | Thixotropic |
| | T14 | 55.5 | 200-400 | 11 | 21 | Sheared data are invalid due to fast settling | | | Fast settling |
| | T15 | 50.3 | 200-400 | 2.8 | 15 | | | | Fast settling |
| | CFS-023 Overflow - CSD = ~17% solids, corresponding to ~12 Pa unsheared and sheared yield stress. | | | | | | | | |
| CFS-023 Brucite = 0.1% Peridotite Deslime Overflow | T23 | 21.9 | 200-400 | 27 | 73 | 200-400 | 29 | 39 | Thixotropic |
| | T24 | 20.3 | 200-400 | 24 | 49 | 200-400 | 25 | 21 | Thixotropic |
| | T25 | 16.8 | 200-400 | 17 | 19 | 200-400 | 13 | 13 | Thixotropic |
| | T26 | 13.9 | 200-400 | 9.3 | 15 | 200-400 | 8.1 | 12 | Thixotropic |
| | T27 | 9.5 | 200-400 | 3.1 | 11 | 200-400 | 4.0 | 9.8 | None |

### 13.10.3 Deslime Cyclone Size by Size Characterization

Deslime cyclones are required prior to flotation stages in order to remove deleterious elements such as MgO that negatively affect the flotation performance.

In ultramafic ores, gangue minerals such as serpentine have lower densities than the targeted nickel, iron and chromium minerals and there is a preferential deportment of minerals within the cyclones based on their density. The behaviour of minerals within the cyclones needs to be understood in order to design for the right cut point encompassing the effect of mineral density.

Deslime cyclone characterization was completed for three of the pilot plant composites, which included particle size distributions measured through cyclosizing, and size by assay (Ni, S, Fe, MgO, Co, and Cr). The finest cyclosizer cut used in the analysis was 9 µm.

Table 13-37 summarizes the mass deportment for three pilot plant balances on both the primary and secondary deslime cyclones, which indicates that the primary cyclone overflow is slightly coarser than the secondary cyclone overflow. A significant portion of the cyclone overflow is finer than 9 µm, with a range between 62 to 87% on the primary cyclone and 79 to 92% on the secondary cyclone. This data was used to support the selection of a target $P_{80}$ in the cyclone overflows of 8 µm, which is the fine end of the range for $P_{80}$ achieved in the pilot plant.

**Table 13-37: Pilot Plant Cyclone Overflow Cyclosizer Mass Deportment**

| Stream | Percent Passing 9 µm (%) | | | |
|---|---|---|---|---|
| | SGS Comp 1 | SGS Comp 2 | SGS Comp 6 | Average |
| Primary Cyclone Overflow | 87 | 62 | 75 | 75 |
| Secondary Cyclone Overflow | 87 | 79 | 92 | 84 |

Table 13-38 and Table 13-39 present the mass and nickel deportments, as well as grades for the primary and secondary deslime cyclones for the pilot plant campaigns. As expected, the primary cyclone underflow has a higher nickel grade. The total nickel losses to deslime cyclone overflow were in line with the open circuit and locked cycle test results.

**Table 13-38: Primary Cyclone – Mass Deportment, Nickel Deportment and Grades to Overflow and Underflows Streams**

| Pilot Composite | Primary Deslime Cyclone Overflow | | | Primary Deslime Cyclone Underflow | | |
|---|---|---|---|---|---|---|
| | Mass (%) | Ni Recovery (%) | Ni Grade (%) | Mass (%) | Ni Recovery (%) | Ni Grade (%) |
| SGS Comp1 | 18 | 14 | 0.27 | 82 | 86 | 0.34 |
| SGS Comp2 | 19 | 15 | 0.24 | 81 | 85 | 0.31 |
| SGS Comp3 | 19 | 12 | 0.15 | 81 | 88 | 0.26 |
| SGS Comp5 | 13 | 10 | 0.21 | 87 | 90 | 0.30 |
| SGS Comp6 | 19 | 9.0 | 0.09 | 81 | 91 | 0.22 |

**Table 13-39: Secondary Cyclone – Mass Deportment, Nickel Deportment and Grades to Overflow and Underflows Streams**

| Pilot Composite | Secondary Deslime Cyclone Overflow | | | Secondary Deslime Cyclone Underflow | | |
|---|---|---|---|---|---|---|
| | Mass (%) | Ni Recovery (%) | Ni Grade (%) | Mass (%) | Ni Recovery (%) | Ni Grade (%) |
| SGS Comp1 | 10 | 5.5 | 0.18 | 72 | 48 | 0.22 |
| SGS Comp2 | 7.0 | 3.5 | 0.15 | 73 | 52 | 0.22 |
| SGS Comp3 | 9.0 | 4.0 | 0.11 | 72 | 46 | 0.16 |
| SGS Comp5 | 12 | 5.0 | 0.12 | 75 | 47 | 0.18 |
| SGS Comp6 | 10 | 3.7 | 0.07 | 71 | 45 | 0.12 |

### 13.10.4 Concentrate Transportation Criteria

Transportable moisture limit (TML) testing was completed by SGS in Vancouver on a sample of FeCr concentrate produced in the pilot plant. Composite 1 represents ore from the Main Zone that would be mined and processed early in the project. Table 13-40 summarizes the transportable moisture specification for the sample that was tested which had a flow moisture of 12% and a transportable moisture of 11%. These values are indicative and further testing is recommended as the flow table method that was used is not approved for testing of iron ore fines derived from magnetite. The FeCr concentrate is not expected to be shipped overseas.

A TML test on nickel concentrate was not completed as the nickel concentrate is not expected to be shipped overseas.

**Table 13-40: Transportable Moisture Test Results**

| Sample ID | Flow Moisture | Transportable Moisture |
|---|---|---|
| Comp1 FeCr Concentrate | 12% | 11% |

### 13.10.5 Concentrate Chrysotile Content

Chrysotile is an asbestiform mineral present in the Crawford ore. The flowsheet was designed to reject gangue minerals and it is expected that most chrysotile would report to the tailings streams and thus, the chrysotile content in the concentrate is expected to be low.

Mineralogical characterization of the FeCr concentrate was completed at the National Resources Research Institute (NRRI) which is a research institute within the University of Minnesota Duluth. A sample of the FeCr concentrate was provided to NRRI for mineral safety screening prior to testing and through XRD analysis, no chrysotile was found. The sample submitted to NRRI was a blend of FeCr concentrates produced in the laboratory from SGS composites 3, 5 and 6. This sample blend would represent material from both the East and Main Zones of Crawford in both the dunite and peridotite lithologies. This result is a good indication that the FeCr concentrate will not contain asbestos. Further work should be completed to confirm this. The FeCr concentrate would be shipped as a wet filter cake which would minimize the risk of airborne chrysotile, if present.

No mineralogical characterizations have been done to quantify the chrysotile content of the nickel concentrate. Based on the positive results from the FeCr concentrate, it is expected that the nickel concentrate would have negligible levels of chrysotile. However, testing to confirm will be conducted. The concentrate will be shipped as a wet filter cake in closed containers which will mitigate the risk in transport.

### 13.10.6 Concentrate Self-Heating Analysis

Three samples of concentrate were submitted for self-heating analysis to assess the risk of self-ignition including the following:

- Two samples of nickel concentrate representing the two end members of mineralization styles.
    - SGS Comp 1 is a pentlandite dominant concentrate with a grade of 18% nickel and approximately 21% sulphur.
    - SGS Comp 5 is a heazlewoodite dominant concentrate with a grade of 40% nickel and 18% sulphur.
- One sample of FeCr concentrate produced from SGS Comp 1 in the pilot plant. This sample was selected as it has the highest sulphur content that would be expected in the FeCr concentrate and thus, is a worst-case scenario for self-ignition.

Table 13-41 summarizes the results of self-heating testing that was completed at SGS in Vancouver. Both nickel concentrates fell into risk region 4 which is expected as they are sulphide concentrates. The FeCr concentrate had a lower risk of self-heating which was expected as it is primarily composed of oxide minerals. The FeCr concentrate fell into risk region 3.

**Table 13-41: Concentrate Self-Heating Analysis**

| Sample ID | SHC - Stage A | SHC - Stage B | Risk Region |
|---|---|---|---|
| | J/g | J/g | |
| SGS Comp1 FeCr Concentrate | 0.9 | 3.4 | 3 - Do not expose to a heat source |
| SGS Comp1 Nickel Concentrate | 2 | 2.7 | 4 - Recommend monitoring |
| SGS Comp5 Nickel Concentrate | 4 | 6.5 | 4 - Recommend monitoring |

## 13.11 Recovery Equations

Recovery equations estimate the recovery to the two concentrate products to support the mine design, as well as financial and production forecasts. Recovery equations were designed based on open circuit variability test results and were confirmed with locked cycle tests.

The following recovery models were developed for the feasibility study:

- total nickel recovery, which is the sum of nickel recovered to the nickel and FeCr concentrates
- a nickel “splitter,” which predicts the deportment of nickel recovery between the nickel and FeCr concentrates
- cobalt recovery to the nickel concentrate
- platinum and palladium recovery to the nickel concentrate
- iron recovery to the FeCr concentrate
- chromium recovery to FeCr concentrate

In addition to the listed equations, a nickel recovery flowsheet evolution adjustment model was developed to capture the improved performance from the magnetite recovery circuit which was optimized towards the end of the variability program.

#### 13.11.1.1 Nickel

The metallurgical flowsheet is designed to recover nickel hosted in heazlewoodite, pentlandite, and awaruite. Heazlewoodite and pentlandite are nickel sulphide minerals that are recovered through flotation processes while, awaruite is a magnetic Ni-Fe alloy mineral that is recovered through magnetic separation processes.[8] Recovery models were developed to estimate the total nickel recovery as well as the percentage split of the total nickel recovery between the FeCr and nickel concentrates.

To estimate the nickel recovery, three levels of recovery models were developed and were applied in the following order:

- Model the total nickel recovery.
- Apply the nickel recovery flowsheet evolution adjustment model to the total recovery. This model was developed to capture the flowsheet improvements in the magnetic recovery circuit that were made throughout the variability program as only 16% of tests utilized the optimized flowsheet.
- Apply the nickel “splitter” model to determine the percentage split in nickel recovery between the flotation and magnetic concentrates.

##### 13.11.1.1.1 Total Nickel Recovery

Analysis of open circuit variability test results led to the development of four nickel recovery domains which capture differences in alteration state of the ore zone of the deposit as well as grade domains. The key driver of recovery across all the nickel recovery domains was the sulphur grade, which is indicative of the nickel mineralization style and quantity in the ore.

[8] Awaruite can be recovered through flotation but this was not included in the flowsheet.

Table 13-42 summarizes the definitions of each of the nickel recovery domains as well as the modelled equations with the following observations:

- For the higher magnetite domains including the Main Zone high-grade core, Main Zone lower grade domain and East Zone, recovery equations consist of a variable and a constant domain. The transition from variable to constant domain is believed to be related to mineralogical changes in the ore.
- Samples were classified broadly into two key alteration classes based on the sample magnetic susceptibility (see the Section 13.3 for further information). Samples with a magnetic susceptibility less than 70 are expected to have higher levels of iron serpentine, lower levels of magnetite and less recoverable nickel on average. This is supported by analysis of mineralogical data in Section 13.3.

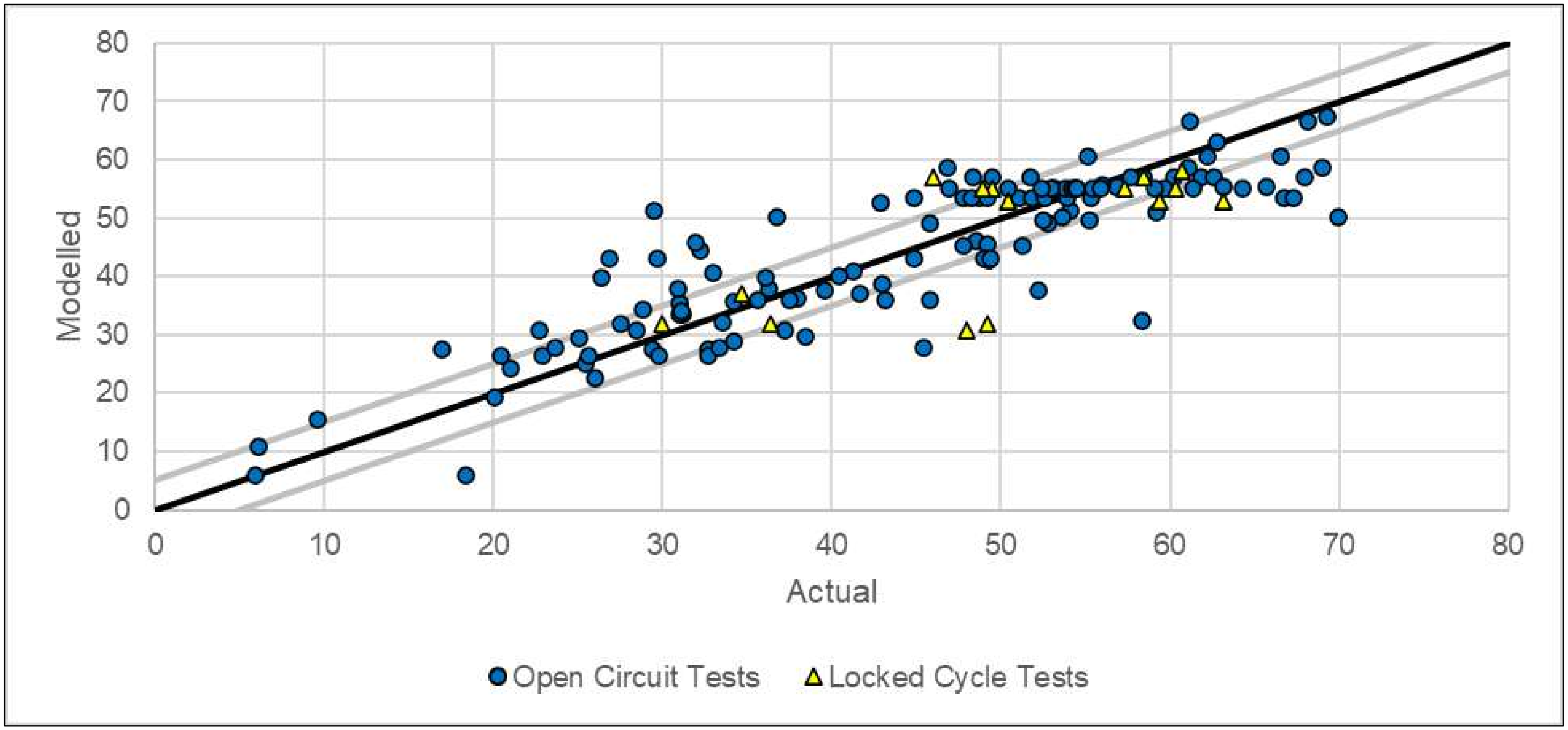
**Table 13-42: Nickel Recovery Domains and Equations**

| Metallurgical Domain Description | Magnetic Susceptibility | Zone | Ore Domain | S Grade Range | Nickel Recovery Equation |
|---|---|---|---|---|---|
| East Zone | >70 | East | All | S ≤ 0.07 | $Ni\ Rec = 23.743 * e^{14.915*(S)}$ |
| | | | | S > 0.07 | Ni Rec = 57% |
| Main Zone High-Grade Core | >70 | Main | HGC | S ≤ 0.25 | $Ni\ Rec = 28.292 * e^{3.0649*(S)}$ |
| | | | | S > 0.25 | $Ni\ Rec = 55\%$ |
| Main Zone Lower Grade Domain | >70 | Main | LG | S ≤ 0.08 | $Ni\ Rec = 19.41 * e^{15.423*(S)}$ |
| | | | | S > 0.08 | Ni Rec = 53% |
| Low Magnetite Domain | <70 | Main, East | All | All | $Ni\ Rec = -5.35 * (Fe) + 19.67 * LN\left(\frac{S}{Ni}\right) + 85.27$ |

To evaluate the performance of nickel recovery models, modelled and actual recoveries for open circuit and locked cycle tests were compared against a 1:1 line. Figure 13-14 presents the parity plot for nickel recovery models. In summary:

- Including all open circuit and locked cycle tests, the $R^2$ between modelled and actual recoveries is 74%.
- Eighty percent of the locked cycle tests and 79% of open circuit tests were within or exceeding the modelled recovery by 5 percentage points.
- The average error in recovery for locked cycle and open circuit tests was +2.5 and +0.3 percentage points in favour of the achieved result, which suggests strong model performance on average.

**Figure 13-14: Nickel Recovery Model Parity Plot**




Notes: **1.** Three samples tested in locked cycle tests achieved recoveries well above the modelled values. For two out of the three samples, the mass pull into the cleaning circuit was lower than average and the recovery above expectations – thus, there is an opportunity to move along the grade recovery curve to increase the concentrate grade. **2.** One sample tested in locked cycle test achieved recoveries significantly below the modelled value by 11 percentage points. This was a composite sample and the reason for the lower-than-expected performance is not clear. The composite is from the western half of the East Zone and is not processed during the payback period. Source: CNC, 2023.

#### 13.11.1.1.2 Nickel Recovery Flowsheet Evolution Adjustment Factor

The nickel recovery flowsheet evolution adjustment factor was developed to capture improvements made to the magnetic recovery circuit throughout the test program. This model was applied to the total nickel recovery for each of the nickel recovery domains outlined in Table 13-42.

Table 13-43 details the flowsheet evolution adjustment equation that was developed.

**Table 13-43: Nickel Recovery Flowsheet Evolution Adjustment Model**

| Adjustment | Minimum Cap | Maximum Cap |
|---|---|---|
| Adjustment = (-0.064)*Ln(S) + 1.03 | 1.1 | 1.3 |

#### 13.11.1.1.3 Nickel Splitter Model

To estimate the nickel recovery to each of the nickel and FeCr concentrates, a nickel "splitter" model was developed as a function of the sulphur grade. The sulphur grade is indicative of the nickel mineralization style and, thus, it makes sense that it would reflect the distribution of recovered nickel between the concentrates. Table 13-44 summarizes the nickel "splitter" model that was developed.

**Table 13-44: Nickel "Splitter" Model to Estimate Nickel Deportment between Concentrates**

| Domain | Nickel Splitter Relationship |
|---|---|
| S < 0.12 | $\%\ of\ Nickel\ Recovery\ to\ Flotation\ Circuit = 3.0585 * S + 0.5671$ |
| S ≥ 0.12 | $\%\ of\ Nickel\ Recovery\ to\ Flotation\ Circuit = 94\%$ |

#### 13.11.1.1.4 Locked Cycle vs. Open Circuit Test Recovery Comparison

After developing the recovery models with the open circuit tests, locked cycle tests were completed to confirm the recovery at final concentrate grade Table 13-45 compares the total nickel recoveries between open circuit and locked cycle tests and Figure 13-15 presents the results as a parity plot. Logically, the open circuit tests should give higher Ni recoveries than the locked cycle tests as the open circuit tests did not include the final stages of cleaning to reach final grade. However, the average difference between the open and locked cycle tests' Ni recovery was within assay error.

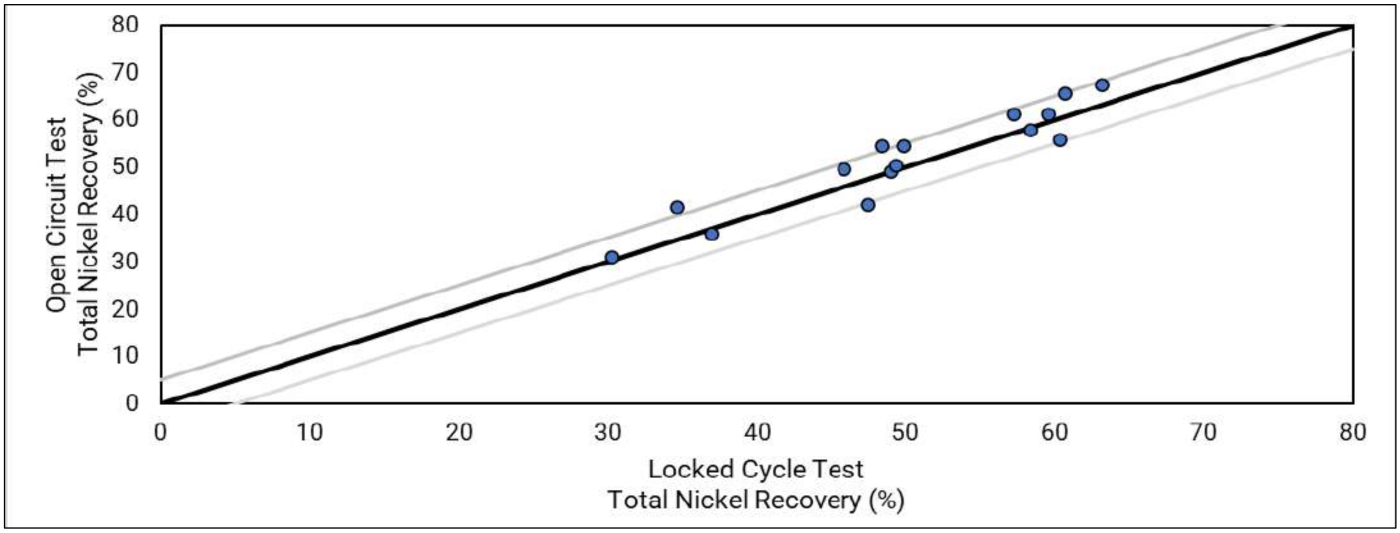
The open circuit test definition of nickel recovery and locked cycle tests final nickel recovery are in good agreement with 12 out of 15 locked cycle tests achieving nickel recoveries within the assay reconciliation error and 13 out of 15 locked cycle tests achieving nickel recoveries within 5 percentage points of the open circuit test recovery result. The nickel recovery models that were developed using the open circuit test recovery definition are representative of the final nickel recovery for the locked cycle tests on common samples. Based on these conclusions, the open circuit test recovery definition was used without applying any cleaning correction factors.

**Table 13-45: Comparison of Locked Cycle (LCT) and Open Circuit (OCT) Test Nickel Recovery Results**

| Sample ID | Sample Characteristics | | | | Nickel Recovery | | | LCT Performance | |
|---|---|---|---|---|---|---|---|---|---|
| | Deposit | Domain | Ni (%) | S/Ni Ratio | LCT Total (%) | OCT Total (%) | LCT- OCT (%) | Error due 0.01 Diff in Ni Grade[1] | Acceptable Error |
| CKS Peridotite Comp | Main | LG | 0.18 | 0.17 | 48 | 42 | 6 | 6 | Yes |
| CGS-006 | Main | HGC | 0.42 | 1.57 | 60 | 56 | 4 | 2 | Yes |
| CDS-004A | East | HGC | 0.28 | 0.75 | 58 | 58 | 0 | 4 | Yes |
| CKS Low-Grade Blend | Main + East | LG | 0.13 | 0.15 | 37 | 37 | 0 | 8 | Yes |
| CKS Dunite Comp | East | LG | 0.18 | 0.11 | 49 | 49 | 0 | 6 | Yes |
| SGS Comp 2 | Main | HGC | 0.37 | 0.81 | 49 | 50 | -1 | 3 | Yes |
| SGS Comp 6 | Main | LG | 0.22 | 0.41 | 60 | 61 | -1 | 5 | Yes |
| CGS-001 | East | HGC | 0.24 | 0.08 | 30 | 31 | -1 | 4 | Yes |
| SGS Comp 5 | East | All | 0.31 | 0.39 | 46 | 49 | -3 | 3 | Yes |
| CGS-004 | Main | LG | 0.23 | 0.70 | 63 | 67 | -4 | 4 | Yes |
| SGS Comp 3 | Main | LG | 0.26 | 0.54 | 50 | 54 | -4 | 4 | Yes |
| CES-006 | East | LG | 0.19 | 0.32 | 61 | 66 | -5 | 5 | Yes |
| SGS Comp 4 | Main | HGC | 0.40 | 0.80 | 57 | 61 | -4 | 3 | No |
| SGS Comp 1 | Main | HGC | 0.35 | 1.74 | 48 | 54 | -6 | 3 | No |
| CES-005 | East | HGC | 0.27 | 0.11 | 35 | 42 | -7 | 4 | No |

Notes: **1**. Due to the low-grade nature of the deposit, a deviation in reconciled head grade can have a significant impact on estimated recovery. For example, a 0.01 percentage point difference in reconciled head grade for a sample with a head grade of 0.30 percentage points results in a recovery difference of 3.3 percentage points. To compare the locked cycle and open circuit tests, the error in recovery related to a 0.01 percentage point difference in nickel head grade was calculated and which was rounded to the nearest whole number. The difference in open circuit and locked cycle test recovery was then calculated to determine if it was within the error bar for each sample.

**Figure 13-15: Total Nickel Recovery Comparison between Locked Cycle and Open Circuit Tests**




Source: CNC, 2023.

#### 13.11.1.2 Byproduct Recoveries

In addition to nickel, Crawford will produce iron (20% of metal value), chromium (15% of metal value), cobalt (1.2% of metal value), and PGE (1.0% of metal value) byproducts which excludes the contribution of carbon sequestration to project revenue. Recovery equations were developed to model each of these byproduct streams.

##### 13.11.1.2.1 Iron

Iron recovery models were designed to estimate the amount of iron that is recovered to the final FeCr concentrate. Iron recovery modelling was done in three steps:

1. Rougher level iron recoveries were measured from open circuit tests and were then adjusted to reflect the expected performance at 2500 G, which is the commercial design point. The adjustment factors used to normalize iron recoveries to a magnet strength of 2500 G were calculated based on a relationship between the average rougher stage recovery of iron versus the magnet strength, which showed a positive, logarithmic relationship with an $R^2$ of 93%.

2. Iron cleaning stage recoveries were modelled using data from tests completed using the final testwork flowsheet which utilized the same cleaning circuit arrangement as was used in the commercial design. Results achieved with outdated test flowsheets were excluded from the dataset.

3. Final iron recovery to the FeCr concentrate is calculated as the product of rougher level recovery and the cleaner stage recovery. A 6 percentage point iron recovery adjustment factor was applied to the final recovery in the high sulphur rougher domain to improve model accuracy.

Table 13-46 summarizes the rougher level recovery equations. The key driver of the rougher iron recovery was found to be the ore magnetic susceptibility, which is a proxy for the magnetic content of the ore and which is measured systematically during core logging processes. A high sulphur subdomain was created to reflect material that has some of the iron tied up in iron sulphide minerals such as pentlandite and tochilinite (note: tochilinite is a non-magnetic iron-sulphide hydroxide mineral that does not float and reports to the non-magnetic tailing streams).

**Table 13-46: Rougher Level Iron Recovery Domains and Recovery Equations**

| Recovery Domain | Magnetic Susceptibility | S Grade (%) | Equation | $R^2$ | Minimum Cap | Maximum Cap |
|---|---|---|---|---|---|---|
| Magnetic Domain 1 | <70 | All | $Ro\ Fe\ Rec = 24.978 \cdot \ln(Mag\ Sus) - 55.474$ | 73% | 9 | 79 |
| | 70 -145 | <0.2% | | | | |
| High Sulphur Domain | 70 -145 | ≥0.2% | $Ro\ Fe\ Rec = -14.16 \cdot \ln\left(\frac{S}{Fe}\right) + 9.8332$ | 57% | 37 | 63 |
| High Magnetic Susceptibility Domain | ≥145 | All | $Ro\ Fe\ Rec = 67$ | - | - | - |

Two cleaning recovery domains were identified, which account for the ore alteration state and the mineralization style and which are summarized in Table 13-47. Cleaning recoveries are multiplied by the rougher recoveries to generate the iron recovery to the final FeCr concentrate.

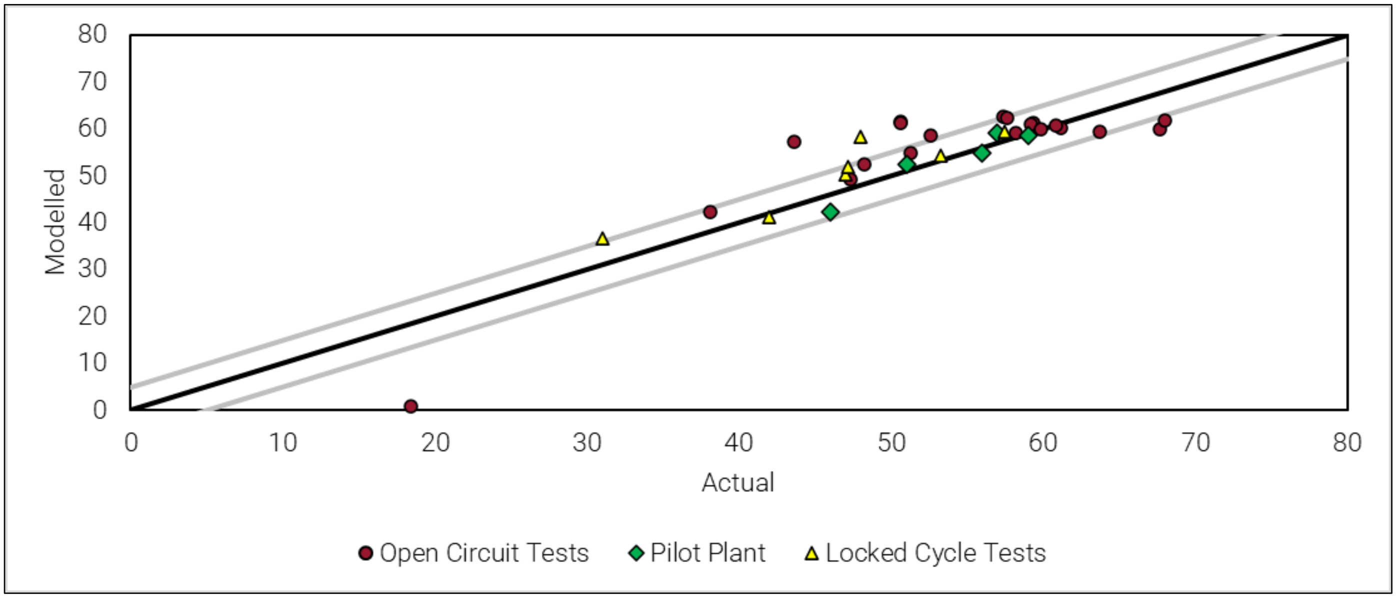
**Table 13-47: Magnetic Circuit Iron Cleaning Recovery Domains and Equations**

| Recovery Domain | Magnetic Susceptibility | Equation | Minimum Cap | Maximum Cap |
|---|---|---|---|---|
| Low Magnetic Susceptibility | <70 | $Fe\ Clnr\ Rec = (0.4323 \cdot \ln(Mag\ Sus) - 1.0) * 100\%$ | 16% | 87% |
| Higher Magnetic Susceptibility | >70 | $Fe\ Clnr\ Rec = \left(-0.044 \cdot \ln\left(\frac{S}{Fe}\right) + 0.69\right) * 100\%$ | 77% | 94% |

Figure 13-16 presents the parity plot for iron recovery models and shows the modelled versus actual final recoveries that were achieved in open circuit, locked cycle, and pilot plant tests. The dataset is condensed to the samples that were tested using the feasibility study flowsheet, as the magnetic circuit arrangement evolved throughout the test program. The following observations are made:

- Iron recoveries achieved in the pilot plant are in close agreement with the modelled result with an average error of -1 percentage points and an $R^2$ of 92% between modelled and actual values.
- Locked cycle and open circuit tests are also accurately modelled with an $R^2$ fit of 83% and 80%, respectively. Five out of seven or 86% of locked cycle tests are within or exceeding the modelled recovery by 5 percentage points. Fifteen out of twenty or 75% of open circuit tests are within or exceeding the modelled recovery by 5 percentage points.

**Figure 13-16: Iron Recovery Model Parity Plot (Showing Only the Results with the Final Flowsheet)**




Source: CNC, 2023.

### 13.11.1.2.2 Chromium

Chromium recovery models estimate the amount of chromium that is recovered to the FeCr concentrate. Chromium recovery modelling was done in three steps:

1. Rougher level recoveries were modelled using the dataset completed with a 1000 G rougher.
2. Rougher level recoveries were adjusted using a factor of 1.26 to account for the expected improvement when increasing the rougher magnet strength to 2500 G. This factor was developed based on a comparison of results between the laboratory and pilot plant data.
3. A cleaning stage recovery model was applied to the adjusted rougher recovery. The cleaning stage recovery was established based on results from laboratory and pilot tests and was a constant 50% across all domains.

Table 13-48 summarizes the chromium recovery equations that were used in the feasibility study.

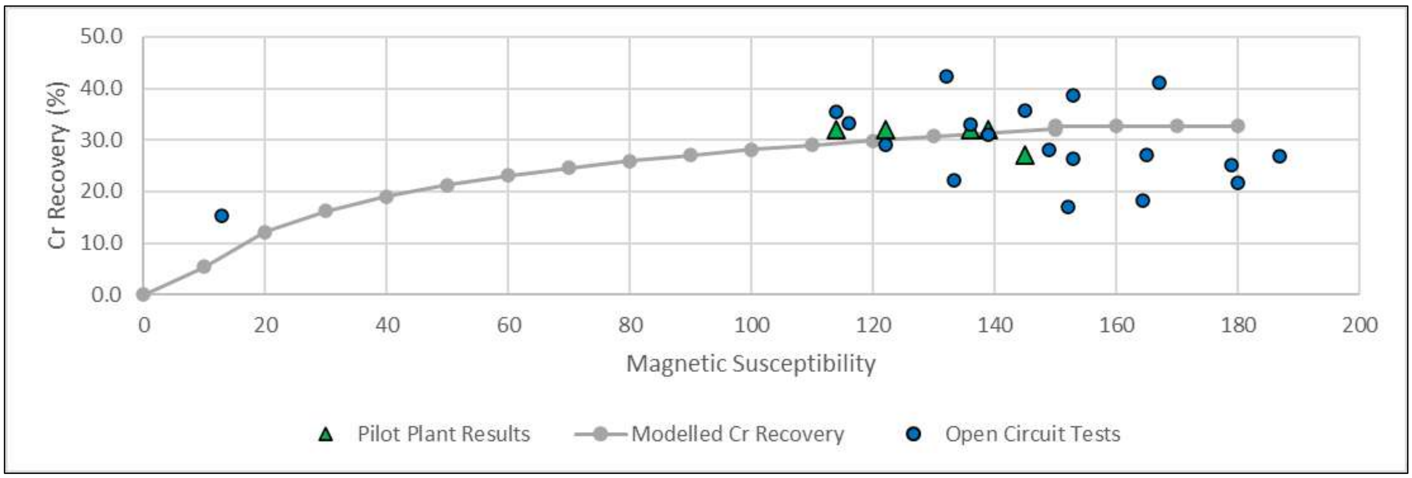
**Table 13-48: Chromium Recovery Equations, Recovery to the FeCr Concentrate (Combined Rougher, Cleaner)**

| Magnetic Susceptibility | Equation | Minimum Cap | Maximum Cap | $R^2$ |
|---|---|---|---|---|
| ≤150 | $Cr\ Rec = (14.684 \cdot Ln(Mag\ Sus) - 23.22) * 1.26 * 0.5$ | 9.45 | 33 | 55% |
| > 150 | $Cr\ Rec = 52 * 1.26 * 0.5 = 32.76$ | - | - | - |

Figure 13-17 illustrates the modelled chromium recovery curve and shows the individual pilot plant and open circuit test results. As the flowsheet evolved over the course of the variability test program, only the results that were achieved using the flowsheet taken forwards for engineering, design and costing were evaluated for model performance. The following observations support Figure 13-17:

- Chromium recoveries achieved on the pilot plant composites are being accurately modelled with four out of the five composites exceeding the modelled recovery. The grades of chromium in the final FeCr concentrate that were produced in the pilot plant are aligned with the forecast grades over the life of mine. The pilot plant results were generated from five composites that represent the key lithological and grade domains of the deposit, four of which were composed of ore expected to be processed early in the project life.
- Open circuit tests for chromium show some variability in the modelled versus actual chromium recoveries.

**Figure 13-17: Pilot Plant and Laboratory Test Chromium Recoveries Relative to Modelled Values**




Source: CNC, 2023.

#### 13.11.1.2.3 Cobalt

Cobalt is a byproduct metal in the nickel concentrate which represents 1.2% of the metal value from Crawford. Cobalt recovery models were developed to predict recovery to the nickel concentrate. The key drivers of cobalt recovery were determined to be the ore sulphur grade and the Co/S ratio. Ores with higher sulphur grades that contain pentlandite style mineralization exhibit higher cobalt recoveries as the cobalt is hosted in the mineral lattice structure of pentlandite or in a distinct mineral which is called cobalt pentlandite. For heazlewoodite dominant ores, cobalt recoveries were confirmed to be lower as there are low levels of cobalt in the heazlewoodite lattice structure.

Table 13-49 summarizes the cobalt recovery equations and the characteristics of the recovery domains that were established.

**Table 13-49: Cobalt Recovery Equation Summary by Domain**

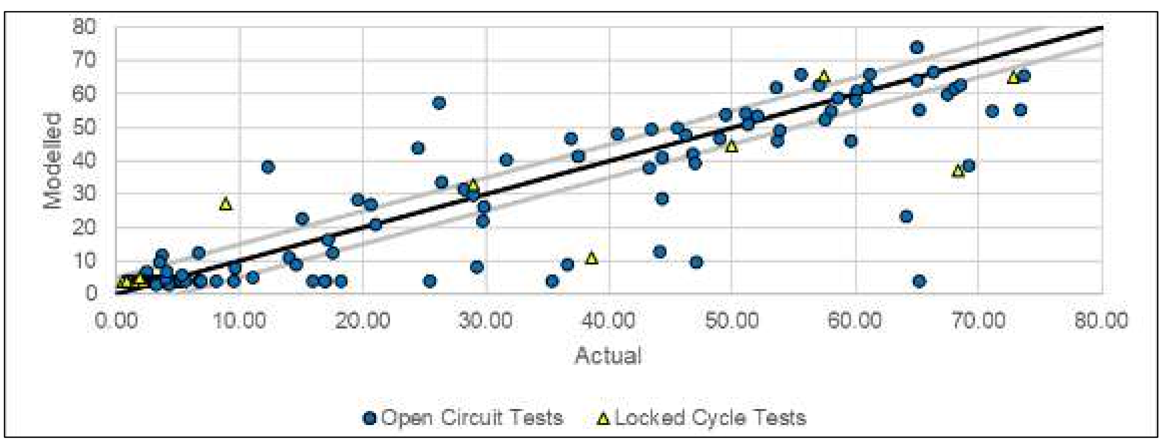
| Recovery Domain | Excess S[1] | S/Ni | Deposit Domain | No. of Tests | Equation | Minimum Cap | Maximum Cap | $R^2$ |
|---|---|---|---|---|---|---|---|---|
| 1 | > 0 | All | All | 54 | $Co\ Rec = -422.74\left(\frac{Co}{S}\right) + 76.707$ | 2% | 74% | 56% |
| 2 | ≤ 0 | < 0.25 | All | 47 | $Co\ Rec = 4$ | - | - | - |
| 3 | ≤ 0 | ≥ 0.25 | Lower Grade Domain[2] | 11 | $Co\ Rec = 1655e^{-24.77*\left(\frac{Co}{S}\right)}$ | 3% | 44% | 82% |
| 4 | ≤ 0 | ≥ 0.25 | High-Grade Core[3] | 10 | $Co\ Rec = 206.84\left(\frac{Co}{S}\right) - 17.065$ | 4% | 17% | 54% |

Notes: **1.** Excess sulphur is a calculated variable that reflects the amount of additional sulphur that is available after allocating all available sulphur to heazlewoodite. It is calculated as: Excess Sulphur=(S-0.37·Ni). **2.** The lower grade domain was defined by the resource modellers as the outer edges of the Crawford deposit which have grades less than 0.25% nickel. The lower grade domain consists of dunite and peridotite lithologies. **3.** The higher grade core domain was defined by the resource modellers as the inner core of the Crawford deposit which has a grade of greater than or equal to 0.25% nickel. The higher grade core domain is primarily of the dunite lithology.

Figure 13-18 shows the parity plot for cobalt which compares the achieved and modelled cobalt recoveries with the following observations:

- Including all open circuit and locked cycle tests, the $R^2$ between modelled and actual recoveries is 74%.
- The average error between modelled and actual recoveries for locked cycle and open circuit tests was +1.9 and +2.6 percentage points in favour of the achieved recoveries.
- 85% of all tests are within or exceeding the modelled recovery by 5 percentage points.

**Figure 13-18: Cobalt Recovery Model Parity Plot**




Source: CNC, 2023.

#### 13.11.1.2.4 Platinum Group Metals

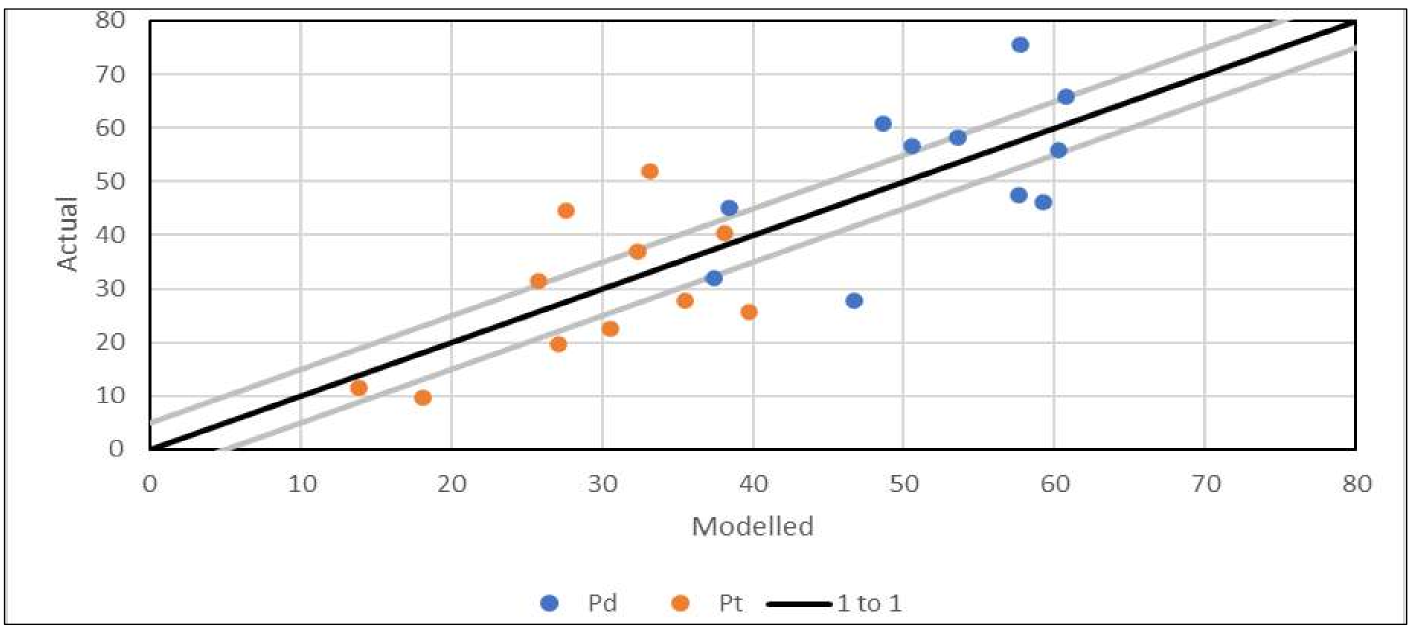
Platinum and palladium recovery models were designed to predict the recovery to the nickel concentrate based on the results of locked cycle tests. The key drivers of recovery for both platinum and palladium were the precious metal head grades as well as the sulphur grade. Samples with higher sulphur grades are thought to provide a more stable froth which helps with precious metal recovery. Table 13-50 summarizes the platinum and palladium recovery equations that were applied across the deposit.

**Table 13-50: Platinum and Palladium Recovery Equations**

| Component | Recovery Equation | $R^2$ | Minimum Cap | Maximum Cap | Avg Abs. Error |
|---|---|---|---|---|---|
| Palladium | Pd Rec = 0.0601*Ln(Pd ppm) + 0.0343*LN(S head) + 0.803 | 35% | 28% | 76% | 9.5 |
| Platinum | Pt Rec = 3.57*(Pt ppm) + 0.0527*Ln(S) + 0.305 | 36% | 10% | 52% | 8.7 |

Figure 13-19 presents the parity plot for platinum and palladium recovery models.

**Figure 13-19: Palladium and Platinum Recovery Model Parity Plot**




Source: CNC, 2023.

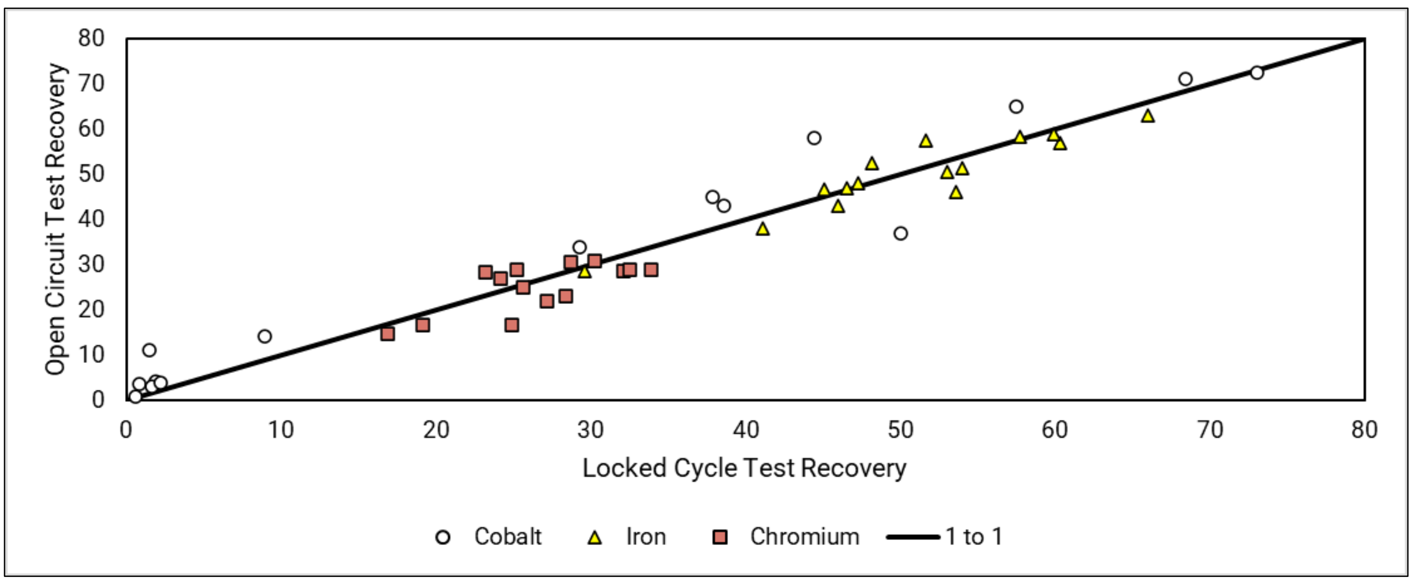
Figure 13-19 shows the following:

- The $R^2$ fit between modelled and actual recoveries for platinum and palladium are 36% and 35%, respectively. There is some variability in the recovery estimates for these commodities. However, the model accuracy is suitable considering the relative value contributed to the project from precious metals (1.6% of payable metal value). As well considering that the head grades are near to the detection limit, additional error is introduced to the recovery estimates based on the assay method.
- For 7 out of 11 or 64% of the locked cycle tests, the achieved palladium recovery is within or exceeding the modelled value by at least 5 percentage points.
- For 6 out of 11 or 55% of the locked cycle tests, the achieved platinum recovery is within or exceeding the modelled recovery by at least 5 percentage points.

### 13.11.2 Byproduct Metals – Locked Cycle vs. Open Circuit Test Recovery Comparison

Locked cycle test recoveries to final concentrate were compared to the recoveries achieved in open circuit for the byproduct metals iron, chromium, and cobalt. Figure 20 compares the recovery results against a 1:1 line. The $R^2$ values between datasets for iron, chromium and cobalt were 87%, 52% and 91%, respectively. There is close agreement between the two tests results which supports the decision to use open circuit tests for recovery modelling.

**Figure 13-20: Open Circuit vs. Locked Cycle Test Recovery Comparison – Byproduct Metals**




Source: CNC, 2023.

## 13.12 Concentrate Quality Modelling

Concentrate quality models were developed to estimate the grades of payable and deleterious elements within the concentrate over the life of mine.

### 13.12.1 Nickel Concentrate Grade Modelling

The nickel concentrate to be produced from Crawford is the product of the flotation circuit. The concentrate contains nickel sulphide minerals including pentlandite and heazlewoodite and the grade is dependent on the type of ore being processed. With a life of mine grade of 34% nickel and 0.7% cobalt, the Crawford nickel concentrate would be the highest grade concentrate available on the market. This concentrate would be a suitable feed for the battery market or, if roasted to eliminate the sulphur, it would be a potential feed for steel-making.

Table 13-51 summarizes the forecast nickel concentrate grades over the life of the mine. In addition to the high nickel grades, the concentrate produced is clean with below detection limit levels of lead and arsenic, and low levels of copper and phosphorous.

The MgO grade of the nickel concentrate is expected to range from 7% to 12% and has a life of mine average of 11%. MgO is a deleterious element for smelters as it has a high melting point and can cause viscosity issues in slag. The Crawford nickel concentrate MgO levels are above typical flash furnace smelter levels which are usually set at 5% for concentrates. However, the nickel grade of the concentrate is also significantly higher than conventional concentrates processed through smelters. Custom smelters that use electric furnaces should consider the Ni/MgO ratio of concentrates, as this can help trade-off opportunities with other MgO containing feeds that may have lower nickel and MgO grades.

**Table 13-51: Nickel Concentrate – Expected Life-of-Mine Grades**

| Ni[1] (%) | Co[1] (%) | Pt[1] (g/t) | Pd[1] (g/t) | S[2] (%) | Fe[2] (%) | Cr[4] (%) | MgO[2] (%) | $SiO_2$[3] (%) | CaO[4] (%) | Mn[4] (%) | Cu[4] (%) | P[4] (%) | As[4] (%) | Zn[4] (%) | Pb[4] (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 34 | 0.65 | 0.9 | 3.1 | 17 | 17 | 0.3 | 11 | 11 | 0.16 | 0.05 | 0.21 | 0.03 | <0.03 | 0.05 | <0.02 |

Notes: **1.** Grades are modelled using recovery and grade models developed from metallurgical variability test results. **2.** Grades are modelled as a weighted average of typical high and lower grade concentrate specifications. **3.** Estimated based on relationship between MgO and SiO2 in the nickel concentrate from detailed concentrate quality analysis. **4.** Grades are estimated based on detailed concentrate quality analysis. Values presented are averages from 11 locked cycle test results.

Products from locked cycle tests were submitted for detailed chemical characterization with ICP. Table 13-52 summarizes the detailed analysis of locked cycle products. There is variability in the MgO grade in products from locked cycle tests, which ranges from 6.7% to 20.9% MgO.

The upper limit of expected MgO grades is 12% MgO in the high-grade concentrate. The high MgO grades for some samples in Table 13-52 is related to the nickel and sulphur head grades and associated low mass recoveries in the flotation circuit which is limitation for laboratory scale testing. It is expected that with continuous operation and better froth stability, MgO rejection and concentrate quality will improve. Discussion is provided for each of the samples that did not achieve the target concentrate grade:

- CKS Low Grade Blend achieved a nickel concentrate grade of 21.5% versus a modelled 40% grade. This sample head grade is close to detection limit and is expected to be processed towards the end of the 41-year mine life, which minimizes risk to product quality during the first 10 years of operation.
- CKS Dunite and CKS Peridotite composite samples yielded concentrate grades of 27% and 26% nickel, which is below the modelled grade of 40% nickel. These samples significantly outperformed the modelled recoveries and thus, the concentrate grade can be improved by moving along the grade-recovery curve while still maintaining production forecasts.
- SGS Comp5 produced a nickel concentrate with a grade of 33.7% nickel versus a modelled grade of 40%. This composite was built from material from outside of the payback period and thus does not pose a risk to the project payback period.

**Table 13-52: Locked Cycle Test Nickel Concentrate Quality – Detailed Assays**

| LCT Sample | Ni | Co | Pt | Pd | Au | Cu | S | Fe | Cr | MgO | P | As | Zn |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | (%) | (%) | (g/t) | (g/t) | (g/t) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) |
| CES-006 | 45 | 0.11 | 0.87 | 1.1 | 0.15 | - | 17.2 | 7.34 | 0.1 | 12.7 | - | - | - |
| SGS Comp6 | 40.7 | 2.0 | 2.0 | 1.6 | 1.2 | 0.09 | 19.4 | 9.8 | 0.14 | 10.1 | 0.02 | 0.03 | 0.03 |
| CES-005 | 37.3 | 0.0 | 0.2 | 0.8 | 0.0 | 0.01 | 13.1 | 15.1 | 0.32 | 13.1 | 0.03 | <0.005 | 0.02 |
| SGS Comp5 | 33.7 | 0.4 | 1.7 | 4.1 | 0.3 | 0.16 | 13.5 | 20.3 | 0.43 | 9.5 | 0.04 | <0.005 | 0.03 |
| SGS Comp3 | 31.6 | 1.1 | 2.6 | 1.7 | 0.4 | 0.11 | 16.0 | 17.4 | 0.19 | 6.7 | 0.01 | <0.005 | 0.04 |
| CGS-004 | 30.0 | 2.2 | 1.7 | 4.1 | 0.4 | 0.15 | 24.9 | 25.0 | 0.10 | 7.8 | 0.04 | <0.005 | 0.03 |
| CKS Dunite | 27.1 | 0.1 | 0.8 | 1.8 | 0.2 | 0.08 | 10.2 | 11.1 | 0.25 | 19.0 | 0.02 | <0.005 | 0.02 |
| CKS Peridotite | 26.0 | 0.1 | 0.6 | 1.3 | 0.2 | 0.12 | 9.4 | 7.8 | 0.38 | 19.5 | 0.03 | <0.005 | 0.02 |
| CDS-004A | 25.2 | 1.3 | 2.6 | 8.1 | 0.1 | 1.16 | 18.0 | 29.8 | 0.29 | 9.5 | 0.02 | <0.005 | 0.12 |
| CGS-001 | 25.2 | 0.04 | - | - | - | - | 8.9 | 9.4 | 0.37 | 20.9 | - | - | - |
| SGS Comp2 | 23.5 | 0.8 | 3.0 | 1.2 | 0.0 | 0.18 | 15.4 | 27.6 | 0.25 | 12.3 | 0.02 | <0.006 | 0.09 |
| SGS Comp4 | 23.5 | 0.8 | 3.7 | 1.7 | 0.2 | 0.20 | 17.4 | 28.2 | 0.32 | 10.7 | 0.03 | <0.005 | 0.09 |
| CKS Low Grade Blend | 21.5 | 0.13 | - | - | - | - | 8.4 | 8.3 | 0.27 | 20.1 | - | - | - |
| SGS Comp1 | 20.0 | 1.1 | 0.7 | 2.1 | 0.1 | 0.06 | 21.1 | 35.5 | 0.26 | 6.7 | 0.01 | <0.005 | 0.04 |
| CGS-006 | 18.3 | 0.97 | - | - | - | <0.2 | 20.9 | 37.9 | 0.15 | 7.0 | - | - | - |
| **Average** | **28.6** | **0.74** | **1.7** | **2.5** | **0.28** | **0.21** | **15.6** | **19.4** | **0.25** | **12.4** | **0.03** | **<0.03** | **0.05** |

#### 13.12.1.1 Nickel Concentrate – Nickel Grade Model

Locked cycle test results were used to model the grade of the nickel sulphide flotation concentrate. Table 13-53 presents the combined coarse and fine flotation concentrate nickel grades as a function of the sample sulphur to nickel ratio. The sulphur to nickel ratio is an excellent proxy for mineralization style and thus for concentrate quality. The nickel flotation grade model was broken into three domains based on the S/Ni ratio including:

1. High-Grade Domain – Heazlewoodite Dominant – S/Ni < 0.37
   - For samples with S/Ni less than 0.37, the mineralization style is mixed between heazlewoodite and awaruite, both of which minerals have a nickel tenor greater than 70%. It is expected that this material will deliver a 40% Ni concentrate grade. Within this domain, some of the samples tested in locked cycle did not meet the modelled specification. This grouping of samples likely did not achieve the target grade because they are samples with low sulphur grades between 0.02% to 0.03% sulphur which have very low mass pulls. In the laboratory it is hard to replicate the froth crowding that would be expected in the commercial operation which would improve gangue rejection and concentrate grade. The pilot plant results support this, where upgrading improvements at the cleaner 1 flotation stage were observed. Three of the four samples that were below the target concentrate grade exceeded the modelled recovery by 4 to 17 percentage points. By moving along the grade recovery curve, the concentrate quality can be improved, which would also result in lower MgO grades.
2. Mixed Domain – Heazlewoodite + Pentlandite – S/Ni between 0.37 – 0.93
   - For this range of S/Ni grades, the mineralization style of the ore is expected to be mixed between pentlandite and heazlewoodite and as the S/Ni ratio increases, the proportion of pentlandite in the ore also increases. As pentlandite has a nickel tenor of approximately 33% and heazlewoodite a nickel tenor of approximately 74%, it is expected that the concentrate grade would decrease as the mineralization style becomes more dominant in pentlandite. Thus, the trend of decreasing grades with increasing S/Ni ratio makes sense.
3. Lower Grade Domain – Pentlandite Dominant – S/Ni > 0.93
   - The modelled nickel grade for this domain, which is a pentlandite dominant domain, is 18% Ni. This value was selected based on the two locked cycle test results for this domain, both of which achieved an 18% nickel grade. Note that although this concentrate is called a lower grade concentrate, its grade is still well above the typical current industry nickel concentrate grades.

Table 13-53 summarizes the nickel concentrate nickel grade domains and models.

**Table 13-53: Nickel Concentrate – Nickel Grade Model Equations**

| S/Ni Domains | Modelled Nickel Grade | Minimum Cap | Maximum Cap | $R^2$ |
|---|---|---|---|---|
| < 0.37 | 40% | | | |
| 0.37 - 0.93 | Ni Grade = ((-0.39431)*(S/Ni) + 0.54789)*100% | 18% | 40% | 86% |
| > 0.93 | 18% | | | |

#### 13.12.1.2 Nickel Concentrate – Byproduct Grade Models for Cobalt, Platinum, and Palladium

The nickel recovery and concentrate grade models were used to define the mass pull to the flotation concentrate. Byproduct grades for Co, Pt and Pd in the nickel concentrate were then calculated using modelled recoveries and relative to the mass pull to the nickel concentrate. The following equations show how this was done.

$$Nickel\ Concentrate\ Production\ (tonnes) = \frac{Mass\ of\ Nickel\ Recovered\ to\ Nickel\ Concentrate}{Modelled\ Nickel\ Concentrate\ Grade\ (\%Ni)}$$

$$ByProduct\ Grade\ in\ Nickel\ Concentrate\ (Co, Pt, Pd) = \frac{Mass\ Recovered\ Metal}{Nickel\ Concentrate\ (tonnes)} 100\%$$

### 13.12.2 FeCr Concentrate

The FeCr Concentrate is produced from the magnetic recovery circuit in the Crawford metallurgical flowsheet which is designed to recover the minerals awaruite ($Ni_{2.5}Fe$), magnetite ($Fe_3O_4$) and chrome spinel. The FeCr concentrate contains substantial quantities of nickel, iron, and chromium, which make it a suitable feed stock for steel-making. Through analysis of locked cycle, open circuit and pilot plant results, grade models were developed for iron, chromium, nickel, sulphur and MgO in the concentrate.

Table 13-54 summarizes the forecast detailed specifications of the FeCr concentrate, which is based on the analysis of products from locked cycle tests as well as predictive grade models. Like the nickel concentrate, the FeCr concentrate is low in contaminants including copper, sulphur, phosphorous, arsenic, lead, and zinc. Although this concentrate has a lower iron grade than high-grade iron concentrates, there is substantially higher value contained in this concentrate due to the polymetallic nature of it. With substantial quantities of Ni, Fe and Cr in the FeCr concentrate, this is a potential feedstock for stainless steel-making.

The FeCr concentrate would likely be treated with an electric furnace smelting process to produce precursor products for steel production. Although the MgO grade is above typical levels, this is not expected to affect the ability to process the ore, as RKEF operations in Indonesia have demonstrated the ability to treat lateritic ores with similar or higher MgO grades.

**Table 13-54: FeCr Concentrate – Detailed Composition**[9]

| Fe[1] (%) | Cr[1] (%) | Ni[1] (%) | Co[2] (%) | Mn[2] (%) | Cu[2] (%) | Zn[2] (%) | S[1] (%) | P[2] (%) | MgO[1] (%) | $Al_2O_3$[1] (%) | $SiO_2$[1] (%) | CaO[2] (%) | As[2] (%) | Na[2] (%) | Ti[2] (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 55 | 2.6 | 0.27 | 0.03 | 0.26 | 0.05 | 0.02 | 0.04 | <0.02 | 9.3 | 1.6 | 7.4 | 0.1 | <0.005 | <0.02 | 0.05 |

Notes: **1.** Grades are modelled using recovery and grade models developed from metallurgical variability test results. **2.** Grades were calculated as averages from detailed analysis of locked cycle test products completed using the final feasibility testwork flowsheet.

[9] MgO and $SiO_2$ grades were modelled as a function of the concentrate iron grade based on analysis of test results. $Al_2O_3$ was modelled as a function of the concentrate chromium grade.

Products from locked cycle tests (completed using the final testwork flowsheet that is representative of the commercial design basis) were submitted for detailed characterization with ICP. Table 13-55 summarizes the detailed analysis of locked cycle products which are expected to be representative of the concentrate produced from the commercial operation.

**Table 13-55: Detailed Analysis of FeCr Concentrate at a Grind Size $P_{80}$ of 25 µm**

| Sample ID | Fe | Cr | Ni | Co | MgO | Al | Si | Ca | S | P | Cu | Pb | Zn | Mn | Mo |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) | (%) |
| CGS-006 | 61 | 3.7 | 0.29 | 0.03 | 4.9 | 0.74 | 1.4 | <0.07 | 0.29 | <0.04 | <0.2 | <0.02 | <0.02 | 0.29 | - |
| SGS Comp 1 | 60 | 3.8 | 0.25 | 0.02 | 7.1 | 0.42 | 2.06 | 0.08 | 0.44 | 0.01 | 0.05 | <0.02 | 0.01 | 0.27 | <0.01 |
| SGS Comp 6 | 55 | 2.4 | 0.32 | 0.06 | 10.1 | 0.48 | 4.27 | 0.14 | 0.03 | 0.01 | 0.01 | <0.02 | 0.02 | 0.32 | <0.01 |
| SGS Comp 5 | 56 | 3.7 | 0.45 | 0.06 | 9.8 | 0.40 | 3.33 | 0.08 | 0.04 | 0.01 | 0.02 | <0.02 | 0.01 | 0.21 | <0.01 |
| SGS Comp 4 | 55 | 3.8 | 0.14 | 0.02 | 8.0 | 0.69 | 3.24 | 0.03 | 0.12 | 0.01 | 0.07 | <0.02 | 0.01 | 0.22 | <0.01 |
| SGS Comp 2 | 57 | 2.4 | 0.19 | 0.02 | 7.9 | 0.37 | 3.09 | 0.02 | 0.14 | 0.01 | 0.01 | <0.02 | 0.01 | 0.28 | <0.01 |
| SGS Comp 3 | 55 | 2.5 | 0.19 | 0.02 | 8.3 | 0.52 | 4.50 | 0.06 | 0.08 | 0.02 | 0.11 | <0.02 | 0.05 | 0.25 | <0.01 |
| **Average** | **57** | **3.2** | **0.26** | **0.03** | **8.01** | **0.52** | **3.13** | **0.07** | **0.16** | **0.01** | **0.05** | **<0.02** | **0.02** | **0.26** | **<0.01** |

#### 13.12.2.1 FeCr Concentrate – Iron Grade

The FeCr concentrate grade was conservatively modelled as a constant 55% iron over the life of the project based on the results of open circuit, locked cycle, and pilot plant results with supporting comments.

- All the locked cycle test results in Table 13-55 achieved or exceeded the stated 55% Fe grade, with the average concentrate grade achieved in locked cycle tests being 57% and covering a range of 55-61% Fe. These results support the selection of 55% Fe for the assumed FeCr concentrate grade and point to some upside potential in the grade.
- The results of pilot plant testing also support the selection of a 55% iron grade, as four out of the five composites exceeded the target 55% grade and one composite achieved a 54% grade target.
- Sixteen open circuit tests with on target grinding that were tested using the final testwork flowsheet were analyzed to evaluate the iron grade model. Twelve out of 16 samples exceeded the modelled 55% iron grade target, 2 out of 16 samples were close to the 55% grade target and 2 out of 16 samples fell far short of the grade target. The two samples that had upgrading challenges are expected to be processed well after the project payback period which presents minimal risk to financiers.

#### 13.12.2.2 FeCr Concentrate - Nickel and Chromium Grade Models

The iron recovery and concentrate grade models were used to define the mass pull to the FeCr concentrate. Nickel and chromium grades within the FeCr concentrate were then calculated using modelled recoveries and relative to the mass pull to the FeCr concentrate. The following equations show how this was done.

$$FeCr\ Concentrate\ Production\ (tonnes) = \frac{Mass\ of\ Iron\ Recovered\ to\ FeCr\ Concentrate}{Modelled\ FeCr\ Concentrate\ Grade\ (\%Fe)}$$

$$Grade\ in\ FeCr\ Concentrate\ (Ni, Cr) = \frac{Mass\ Recovered\ Metal}{FeCr\ Concentrate\ (tonnes)} 100\%$$

#### 13.12.2.3 FeCr Concentrate – Sulphur Grade Model

Sulphur is a deleterious element in steel production processes as it is recovered with the Ni, Fe, and Cr metals during smelting processes. Sulphur can be removed from alloy phases through manipulation of slag chemistry in primary smelting furnaces or through desulphurization reactions an AOD reactor. Understanding the sulphur content of the FeCr concentrate is critical as it is a deleterious component in steel. Models for the sulphur content in the FeCr concentrate were developed based on the results of open circuit tests completed using the final testwork flowsheet. Table 13-56 summarizes the equations used for modelling. This relationship shows that the FeCr concentrate sulphur content is dependent on the feed sulphur grade:

- For samples with a S grade below 0.1% S, the FeCr concentrate has low sulphur content ranging between 0.02% to 0.04% S, with an average of 0.03% S. Most of the ore treated during the payback period and over the life of mine has a sulphur grade < 0.10% and thus, most of the FeCr concentrate is expected to have low sulphur content.
- For samples with a sulphur grade greater than or equal to 0.10%, the FeCr concentrate sulphur grade shows a linear relationship with the feed sulphur grade with an $R^2$ of 97%. This linear relationship was used to model the sulphur content of the FeCr concentrate for material with a sulphur grade ≥0.10%.

**Table 13-56: FeCr concentrate – Sulphur Grade Model**

| S Domain | Equation | Minimum Cap | Maximum Cap | $R^2$ |
|---|---|---|---|---|
| S < 0.10% | Mag Concentrate S Grade = 0.03% | - | - | - |
| S ≥ 0.1% | Mag Concentrate S Grade = 0.531*(S Head Grade)-0.023 | 0.03 | 0.37 | 97% |

## 13.13 In-Process Carbonation Testwork

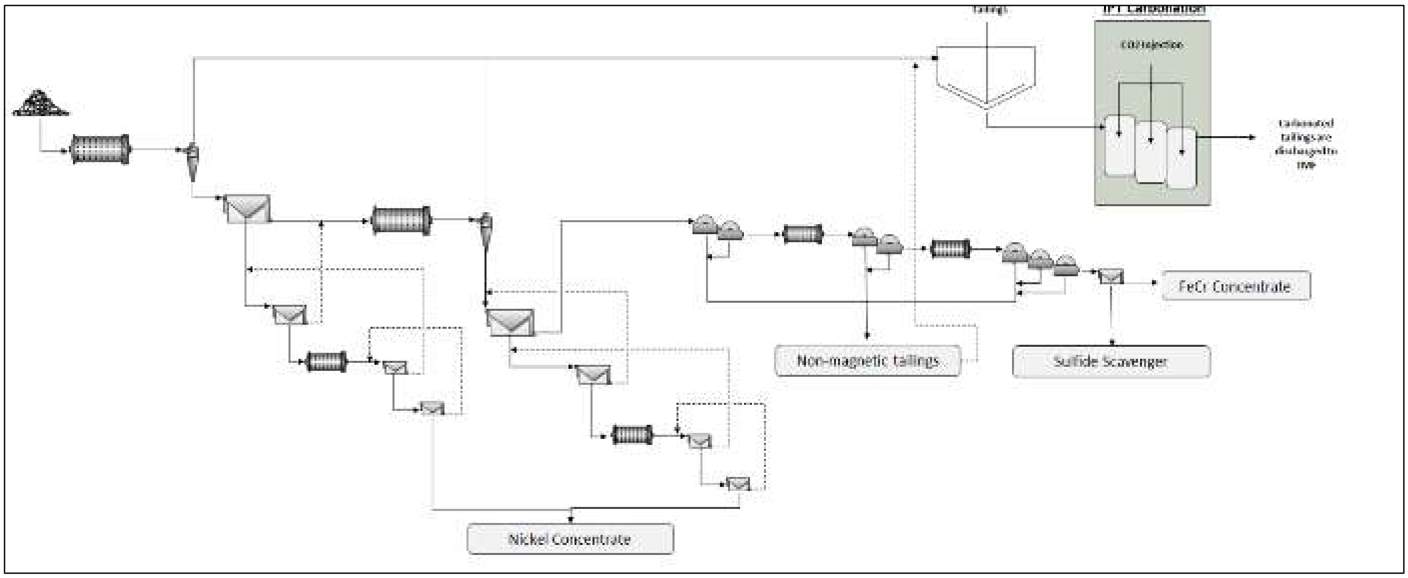
In-process tailings (IPT) carbonation is a novel $CO_2$ sequestration process developed by CNC where tailings generated by the milling process are conditioned with a concentrated source of $CO_2$ after tailings thickening and before discharge to the TMF. $CO_2$ delivered to site is sparged into the tailings slurry and reacts with the minerals to form carbonates which results in permanent sequestration of the $CO_2$. The carbonation tanks have been designed such that unreacted $CO_2$ in the head space is recompressed and recirculated to maximize $CO_2$ utilization.

Brucite is the key mineral that defines the $CO_2$ sequestration potential of Crawford's tailings. The relevant mineral carbonation reaction for IPT carbonation is shown below involving brucite.

$$Mg(OH)_2 + CO_2 + 2H_2O \rightarrow MgCO_3 \cdot 3H_2O$$

Figure 13-21 shows where the IPT Carbonation process was integrated into the Crawford metallurgical flowsheet.

**Figure 13-21: IPT Carbonation Location within Overall Crawford Metallurgical Flowsheet**




Source: CNC, 2023.

The following testwork was completed to support the incorporation of the IPT Carbonation process in the feasibility study:

- Variability testing was performed on different samples of ore from key lithologies and of varying brucite content. A predictive model for the $CO_2$ sequestration capacity of tailings was developed from these test results which was used in technoeconomic modelling to support estimates of $CO_2$ sequestration and associated revenue forecasts.
- Semi-continuous testing on one sample was completed to generate process control knowledge and inform operating conditions ahead of piloting.
- Pilot testing was completed to show the ability to scale up the process and to confirm the sequestration models.
- Supporting engineering testwork was conducted to verify the brucite content of tailings, evaluate the impact of carbonation reactions on rheological properties of tailings, and quantify $CO_2$ degassing from carbonated tailings.

Variability and semi-continuous testing were completed in the laboratory at Kingston Process Metallurgy. The pilot campaign was conducted at SGS Lakefield in summer 2023.

### 13.13.1 Sample Characteristics & Representativity

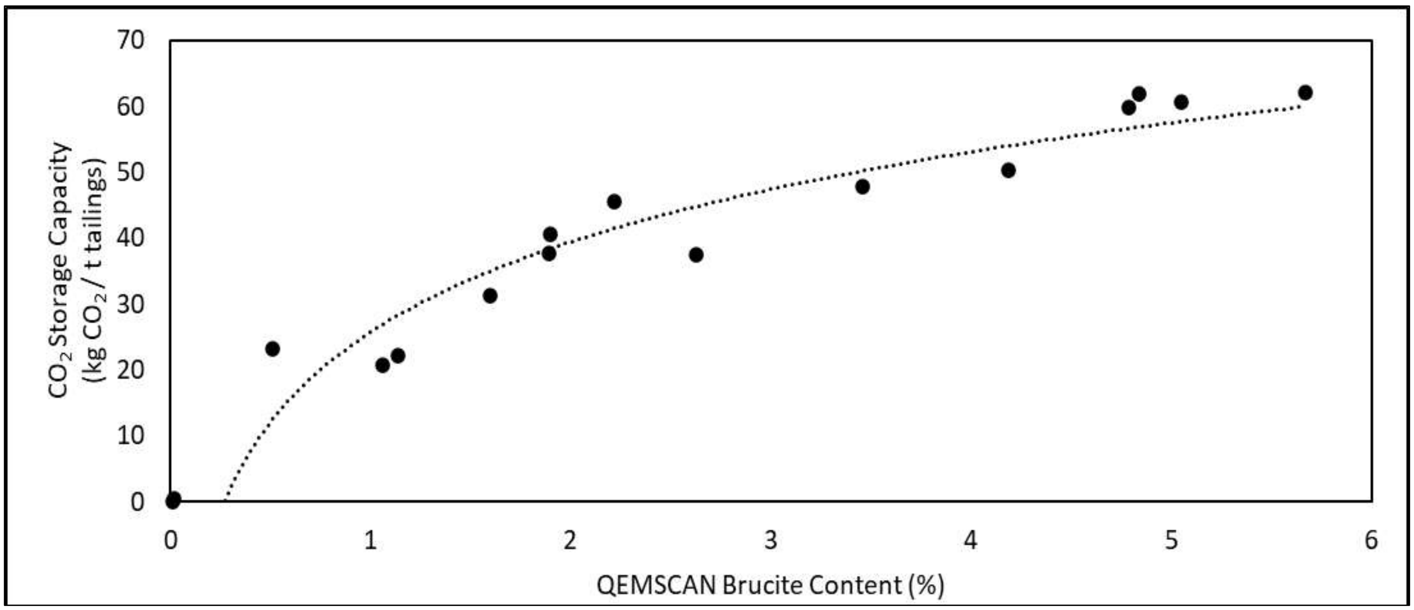
Variability samples were selected to represent the dunite and peridotite lithologies, which are the two main lithological units at Crawford. Samples were selected from both the Main and East zones of Crawford to represent ore that is expected to be processed during the payback period. Twelve samples from the dunite lithology were tested with

QEMSCAN brucite grades ranging from 0.5% to 5.7% with an average of 3.2%. Four samples from the peridotite lithology were tested with QEMSCAN brucite grades ranging from 0.01% to 1.6% and an average of 0.7%. The average QEMSCAN brucite grade of all the samples was 2.5%. The sample distribution between Main and East zones represents the relative proportions of ore coming from each zone during the project payback period.

### 13.13.2 IPT Carbonation Variability Test Results

Samples of representative tailings were subjected to a standard test procedure to quantify the $CO_2$ sequestration potential of the tailings. Using test results, models were developed to forecast $CO_2$ sequestration over the life of the project. The $CO_2$ sequestration capacity of the tailings was calculated based on the change in carbon assay before and after IPT Carbonation. The tailings storage capacity showed a strong correlation with the QEMSCAN measured brucite as shown in Figure 13-22 and supported by Table 13-57 which states the equation used to model $CO_2$ sequestration.

**Figure 13-22: IPT Carbonation Variability Testing Results and Model**




Source: CNC, 2023.

**Table 13-57: IPT Carbonation – $CO_2$ Sequestration Capacity Model**

| Equation | Min (kg $CO_2$/t tailings) | Max (kg $CO_2$/t tailings) | $R^2$ |
|---|---|---|---|
| $CO_2$ Sequestration Capacity (kg $CO_2$/t tailings) = 19.7*LN(QEMSCAN Brucite) + 25.7 | 0 | 62 | 88% |

### 13.13.3 Semi-Continuous Laboratory Testing

Semi-continuous laboratory testing was conducted to study the impact of fresh feed on the IPT Carbonation process, to gain knowledge ahead of piloting and inform the operation and design of the pilot plant. The key operating parameters studied were residence time and $CO_2$ flow rate with the goal of maximizing $CO_2$ sequestration and $CO_2$ utilization[10].

Results generated by this testwork demonstrated the relationship between operating conditions (residence time, $CO_2$ flow rate) and key process metrics ($CO_2$ sequestration and utilization). Increasing the residence time and $CO_2$ flow rate results in higher $CO_2$ sequestration rates, but lower $CO_2$ utilization. Decreasing residence time and $CO_2$ flow rate has the inverse effect. The $CO_2$ utilization for continuous tests ranged from 49% to 98% with an average of 74%. Continuous showed that a 3- to 4-hour residence time would be sufficient to achieve the modelled sequestration rate, which was taken forwards for piloting.

### 13.13.4 Pilot Plant Testing

The IPT Carbonation pilot plant program was run in the summer of 2023 at SGS Lakefield to demonstrate the ability to scale up the process. An estimated 7 tonnes of tailings from the previous metallurgical pilot plant[11] were processed over 12 tests.

Fresh feed, pulped to approximately 40% solids, was continuously pumped in and out of carbonation vessels. Eriez CavTube spargers were used to sparge pure $CO_2$ into the bottom of the vessels. The % solids density was measured using a Marcy scale during operation. Grab samples of product discharge were taken hourly. Process surveys were completed every 5 hours, which consisted of four sample cuts taken over the course of an hour to build a composite. Grab and survey composites were submitted for total carbon analysis.

Table 13-59 summarizes the results of a pilot plant test which confirmed the ability to reproduce the modelled $CO_2$ sequestration.

Survey samples averaged 32 kg of $CO_2$ stored per tonne tailings throughout the pilot test. Sixty-five percent of injected $CO_2$ was sequestered in a single pass and 94% of the predicted sequestration capacity was used. Grab samples indicated even better process performance. Feed, grab, and survey assays supporting this result are shown in Figure 23, plotted against time of day.

The baseline tailings carbon content which represents the feed to the IPT Carbonation process was measured during surveys. For all four surveys, the feed carbon assay was stable between 0.18 to 0.19% carbon The IPT carbonation process increased the tailings carbon content by close to 8X to an average of 1.12% for the grab assays and 1.1% for the surveys. The grab and survey results are in close agreement.

Pilot plant testing confirmed the ability to reproduce the modelled $CO_2$ sequestration estimates for the sample tested. The confirmation of the modelled estimate in the pilot test supports the forecast $CO_2$ sequestration rates of up to 1.5 Mt/a. The pilot also made improvements to the process design, reducing the residence time from 6.5 to 3 hours,

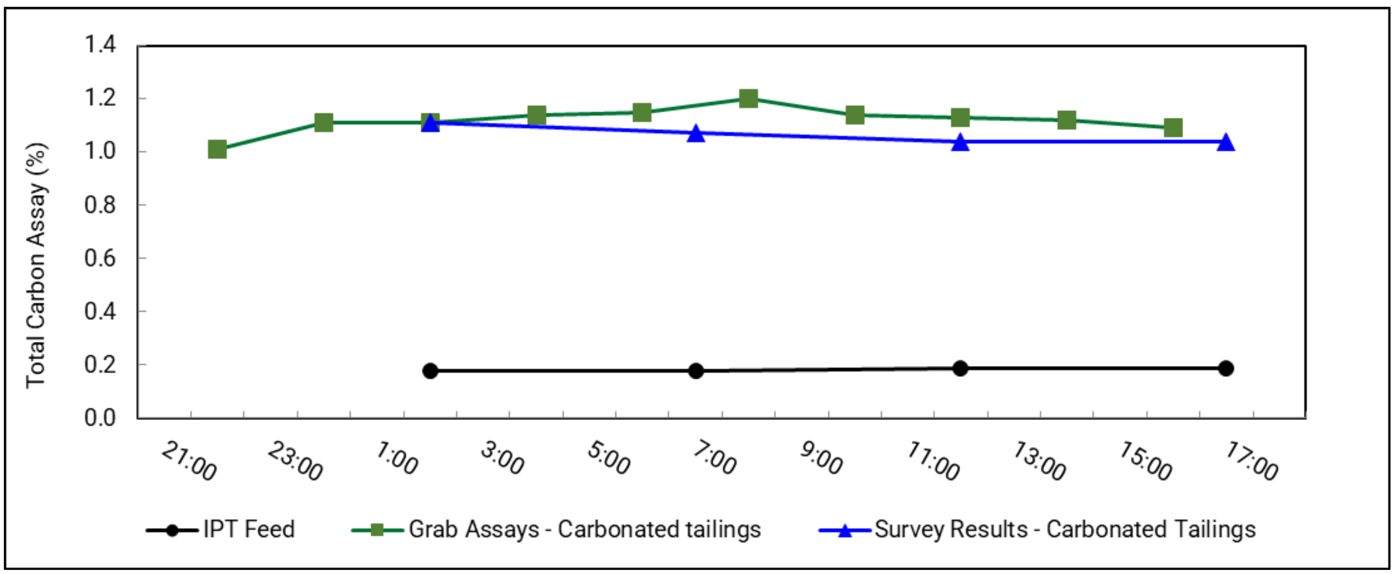
[10] $CO_2$ utilization is the fraction of $CO_2$ reacted in a single pass through the process.
[11] The tailings produced in the metallurgical pilot plant from summer 2022 were preserved under water to limit weathering of the tailings.

and increasing $CO_2$ utilization from 50% to 65%, although these improvements were not reflected in the project design basis. It is expected that IPT carbonation operational and capital costs can be reduced once these improvements are incorporated into the engineering design.

**Table 13-58: IPT Carbonation Pilot Plant Results**

| Description | Feed Carbon Content (%) | Average Product Carbon Content (%) | kg $CO_2$ stored/t Tailings | Model Performance (Actual / Modelled) (%) | $CO_2$ Utilization (%) |
|---|---|---|---|---|---|
| Grab Sample Results | 0.18 - 0.19 | 1.12 | 35 | 101% | 70% |
| Survey Results | 0.18 - 0.19 | 1.07 | 32 | 94% | 65% |

**Figure 13-23: IPT Carbonation Pilot Plant Continuous Run Data – Total Carbon Content of Feed and Products**




Source: CNC, 2023.

### 13.13.5 Supporting Engineering Testwork

#### 13.13.5.1 Rheology

A rheological study was performed on Crawford tailings to assess the impact of IPT carbonation on the pumpability of the tailings. Three types of samples were tested as follows:

- baseline uncarbonated tailings
- carbonated tailings from the pilot plant after a 0-, 3-, 6-, and 24-hour (short) time delay after IPT carbonation treatment

- carbonated tailings generated from semi continuous testing at Kingston Process Metallurgy with a weeks-long time delay after IPT carbonation treatment.

Rheological testing shows that IPT Carbonation affects tailings properties through an increase in yield stress and viscosity. The impact on rheology is a function of time and is thought to be related to the slow precipitation of carbonates. A significant change in the properties of the tailings was observed for tailings that had rested for several weeks. Tailings that had rested for up to 24 hours post carbonation showed only a very minor change in rheology. To mitigate this risk, operational procedures during periods of extended downtime should be developed to minimize the amount of time that tailings sit idle.

#### 13.13.5.2 Brucite Quantification by Thermogravimetric Analysis

The $CO_2$ sequestration model that was developed using empirical laboratory results showed that more $CO_2$ was being stored than was stoichiometrically possible based on the QEMSCAN measured brucite. The risk to $CO_2$ sequestration forecasts is negligible since a consistent approach has been taken to characterize brucite across the resource, however additional testing was completed to assess the brucite content of the rock and understand this discrepancy.

Two alternative methods for quantifying brucite were used to evaluate the brucite content of three samples, which were then compared to the QEMSCAN measured brucite:

- Quantitative X-Ray Diffraction (qXRD) is a common analytical technique used to estimate mineralogy based on the crystallinity of minerals.
- Quantitative thermogravimetric analysis (qTGA) has been used in academia to quantify brucite content based on its characteristic thermal decomposition over a specific temperature range.

The results of both qTGA and qXRD testing, which are summarized in Table 13-60, suggests that QEMSCAN is underpredicting the brucite content of the rock. On all three samples tested, the alternative quantification methods predicted higher brucite than was reported by QEMSCAN and than what was stoichiometrically reacted in variability testing. This explains why the $CO_2$ sequestration capacity of samples exceeds the QEMSCAN brucite capacity. In addition, it is possible that some of the serpentine is also being carbonated as well.

**Table 13-59: Brucite Quantification – Comparison of Methods**

| Sample ID | QEMSCAN Brucite (%) | Stoichiometric Brucite Reacted in Variability Testing (%) | qXRD Brucite (%) | qTGA Brucite (%) |
|---|---|---|---|---|
| CGS-009 | 1.4 | 2.7 | 4.8 | 5.8 |
| CGS-007 | 2.1 | 6.0 | 7.5 | 6.8 |
| CGS-006 | 4.8 | 8.2 | 8.9 | 8.8 |

#### 13.13.5.3 Degassing Assessment after IPT Carbonation

IPT carbonation conditions tailings slurry with $CO_2$ which results in some of the $CO_2$ dissolving into the water. Most of the dissolved $CO_2$ precipitates as solid carbonate minerals to store the $CO_2$; however, some of the $CO_2$ degasses after leaving the IPT carbonation vessels. $CO_2$ degassing does not impact the sequestration forecasts but rather the maximum $CO_2$ sequestration efficiency that can be achieved. Testing on one sample showed that approximately 0.6 kg $CO_2$ per tonne of tailings degasses after the carbonation process which limits the $CO_2$ sequestration efficiency to 98%. Notably, the $CO_2$ degassing is from unreacted dissolved $CO_2$ in the water, rather than from carbonate minerals.

# 14 MINERAL RESOURCE ESTIMATES

## 14.1 Introduction

Caracle Creek was retained by CNC to prepare two updated mineral resource estimates, supported by a technical report, for the Crawford Project. Drillhole information that could be confidently confirmed up to March 6, 2023, was utilized in the preparation of the updated mineral resource estimates. The mineral resource estimates (MREs) have an effective date of August 31, 2023.

The updated mineral resource estimates for the Main-West Zone and the East Zone, disclosed herein, were prepared by Miguel Vera (B.Sc., Geology) and this work was reviewed and signed off on by qualified person Dr. Scott Jobin-Bevans (P.Geo.). The mineral resources herein are not mineral reserves as they do not have demonstrated economic viability. The results disclosed in this report are nickel, cobalt, platinum, palladium, chromium, sulphur, and iron mineral resources estimated to be contained within a large, relatively homogenous body of ultramafic rock, the CUC. The mineral resource estimates include indicated, inferred, and measured mineral resources, interpreted on the assumption that the mineralization has reasonable prospects for eventual economic extraction using open pit mining methods.

The QP, Dr. Jobin-Bevans, is not aware of any legal, political, environmental, or other risks that could materially affect the potential development of the mineral resources.

## 14.2 Resource Database

The drillhole and project database provided by CNC was initially filtered by zones of interest (Main, West, and East), then validated and refined (e.g., ignored duplicate data, conflicting twin holes or statistical outliers that are clear mistakes, among other correction measures) for geomodelling and resource estimation purposes.

### 14.2.1 Main-West Zone

Within an area of approximately 3.3 km along strike, 280 to 1000 m in width, and 800 m deep, the working database of the Main-West Zone contains the following:

- Collar: 196 holes amounting to 89,234.93 m, with an approximate mean depth of 450 m and a maximum of 1,048 m. Included are 12 geotechnical holes (with geologs only) and 21 metallurgical holes (twin holes with complete logs but no samples).
- Survey: 193 measured holes, with the remaining 3 holes estimated from their planned direction.
- Lithology: 193 holes with 22 unique rock codes, grouped into 10 codes for modelling purposes (see Section 14.4).
- Assays: 158 holes with 43,933 core samples of 1.5 m average length; 34 elements reported.

- Magnetic Susceptibility: 182 holes with 53,401 handheld magnetic susceptibility measurements on drill core, taken approximately every 3 m during the initial campaigns, and later every 1 metre.
- Specific Gravity: 180 holes with 9,904 density measurements from drill core, taken every 7 m on average.
- Mineralogy (QEMSCAN): 66 holes with 1,667 core samples of 1.5 m average length, mostly taken every 15 m; 34 minerals reported, including brucite.

Secondary data sources include alteration, mineralization, and structural drillhole logs, historical geophysical surveys (e.g., magnetic susceptibility, EM, and gravity), geological maps and various work reports.

### 14.2.2 East Zone

Within an area of approximately 2.6 km along strike, 200 to 350 m in width, and 800 m deep, the working database of the East Zone contains the following:

- Collar: 126 holes amounting to 53,978.8 m, with an approximate mean depth of 410 m and a maximum of 1,155 m. Included are 13 geotechnical holes (with geologs only) and 6 metallurgical holes (twin holes with complete logs but no samples).
- Survey: 123 measured holes, with the remaining 3 holes estimated from their planned direction.
- Lithology: 123 holes with 20 unique rock codes, grouped into 10 codes for modelling purposes (see Section 14.4).
- Assays: 103 holes with 25,135 core samples of 1.5 m average length; 34 elements reported.
- Magnetic Susceptibility: 124 holes with 44,496 handheld magnetic susceptibility measurements on drill core, taken approximately every 3 m during the initial campaigns, and later every 1 metre.
- Specific Gravity: 114 holes with 5,045 density measurements from drill core, taken every 8 m on average.
- Mineralogy (QEMSCAN): 52 holes with 1,253 core samples of 1.5 m average length, mostly taken every 15 m; 34 minerals reported, including brucite.

Secondary data sources include alteration, mineralization, and structural drillhole logs, historical geophysical surveys (magnetic susceptibility, EM, and gravity), geological maps and various work reports.

## 14.3 Methodology

The main stages of the resource estimation process in each zone are very generally described below:

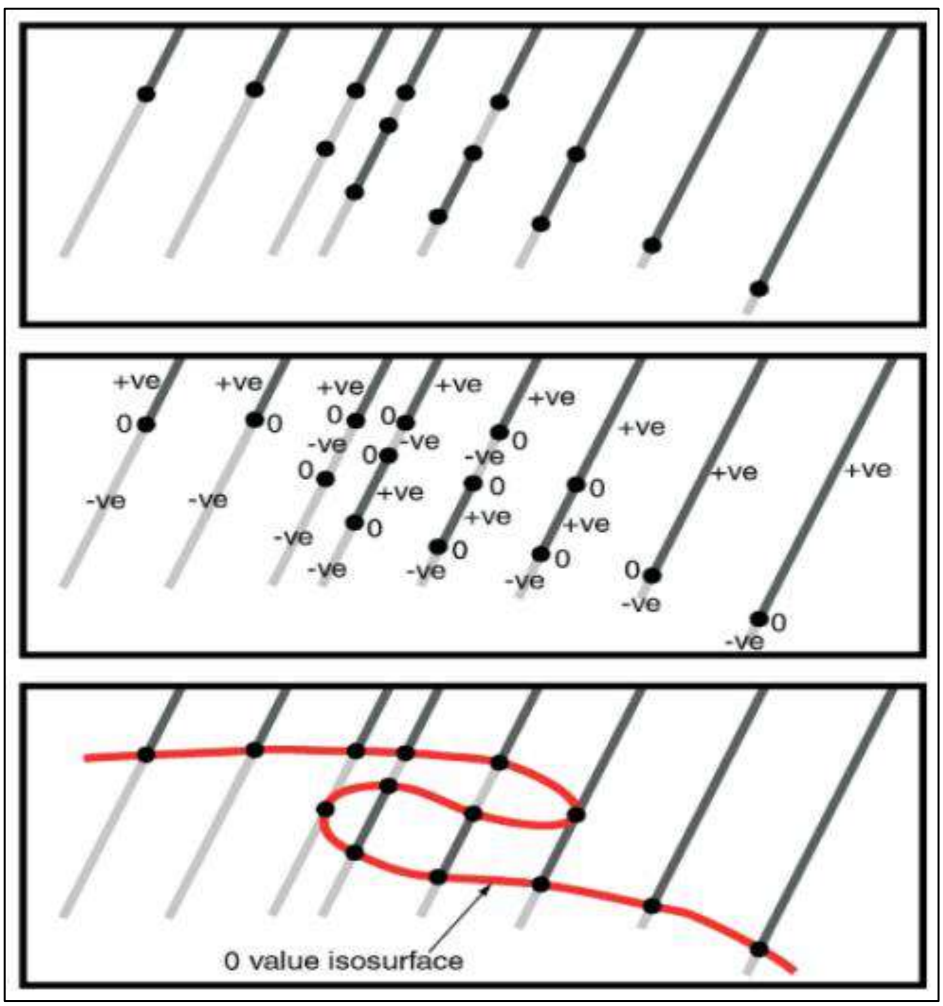
- compilation of historical and CNC drillhole databases; generation of the working database for subsequent stages
- 3D modelling of geological (rock types, alterations) and mineralized domains based on revised lithological codes, densities, magnetic susceptibility, and assay grades
- exploratory data analysis (EDA), capping and compositing of assay grades within the modelled domains; estimation strategy definition
- variogram modelling and cross-validation

- block modelling, grade interpolations (kriging, IDW, NN) and validations (visual, statistical, swath plots)
- resource classification and class smoothing.

These steps involve the use of mining software packages such as Leapfrog Geo 2022.1.1 (3D modelling) and Isatis.neo 2022.04 (geostatistics).

Leapfrog Geo operates through implicit modelling techniques (Cowan et al., 2003). Implicit modelling uses interval and/or point data along with structural trends and other user-defined parameters to interpolate geological surfaces and volumes (Figure 14-1), which can then be improved through manual editing. In order to work with categorical data, the software converts it into distance points relative to a zero value that usually corresponds to a lithological contact. Volumes can then be extracted through Boolean operations against a primary model box or previous volumes.

**Figure 14-1: Implicit Modelling Technique**



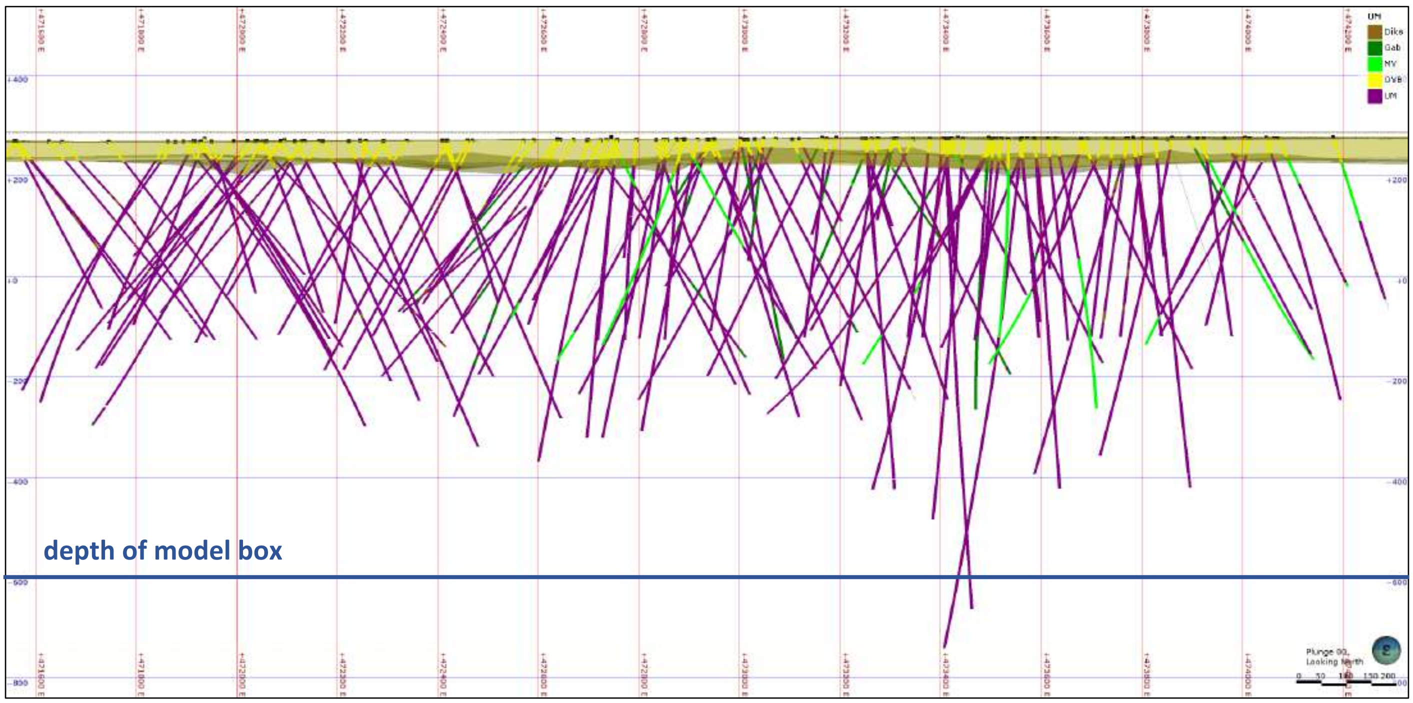

Note: Two sets of intervals (upper panel) converted into positive ("+ve" or inside) and negative ("-ve" or outside) distance points (middle panel) and the resulting interpolation through zero distance ("0" or contact) value points (lower panel). Source: Modified from Cowan et al. (2003).

## 14.4 Geological Interpretation and Modelling

### 14.4.1 Overburden and Topography

The entire project area is covered by a mix of barren clay and gravels (considered overburden) that average 38 m depth (Figure 14-2). This volume was extracted from the model using the topography and the "bedrock" surface, which represents the base of overburden. The topographic surface was prepared by CNC after a LiDAR survey (presenting a good match with collar heights), while the bedrock surface was prepared by SRK by interpolating through the base of the "OVB" drillhole intervals plus additional test holes outside of the project area.

**Figure 14-2: Main-West Zone Longitudinal View (Looking North) Showing Drillholes and the OVB Volume (Transparent Yellow)**




Note: The ultramafic unit, coloured purple in the drillhole traces, is the target lithology that forms the bulk of the resource model area. Green intercepts are mafic volcanic and metavolcanic rocks. Source: Caracle Creek, 2023.

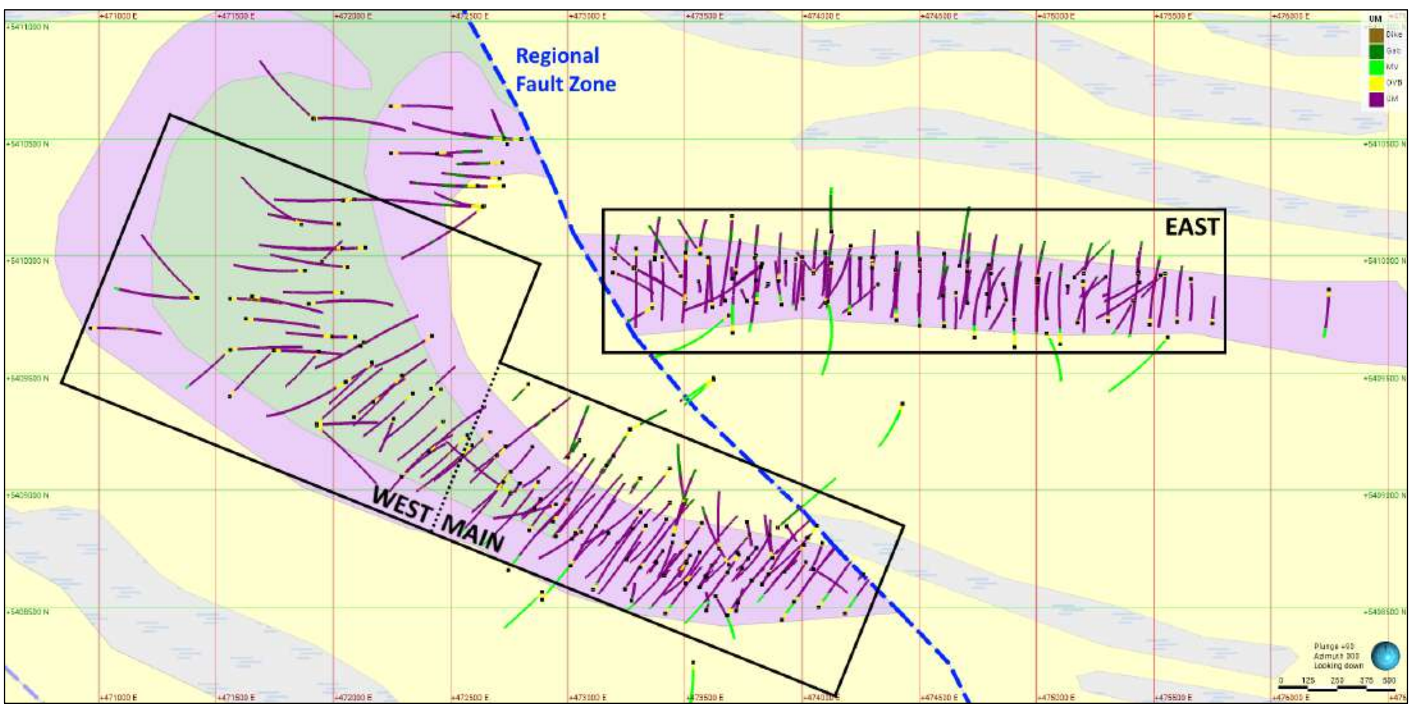
### 14.4.2 Lithology

Located at either side of a regional fault zone, approximately 1 km apart and with complementary geometries, the so-called Main-West and East zones are interpreted as once being part of the same body, displaced by this northwest trending structure (Figure 14-3). As such, lithologies identified in both zones share common features:

- a combination of consistently ordered, subvertical ultramafic (dunite, peridotite and pyroxenite) rock horizons which usually transition into each other, as the main feature
- gabbroic rock horizons, seemingly conformable in between ultramafic packages, marking the northern contact of the ultramafic core body and its separation with other, less mineralized ultramafic/gabbro horizons further north
- metavolcanic rocks of mafic/intermediate and lesser felsic composition, marking the southern contact of the ultramafic core body.

These lithologies encompass most of the deposit, the remaining ones corresponding to diabase/mafic intrusion and lamprophyre dikes, found mostly in the Main-West Zone, while in the East Zone they are few and far between.

**Figure 14-3: Plan View of the Main-West and East Zones**




Note: Background geology from Ontario Geological Survey (MRD126), shown against the ultramafic unit, coloured purple in the drillhole traces.
Source: Caracle Creek, 2023.

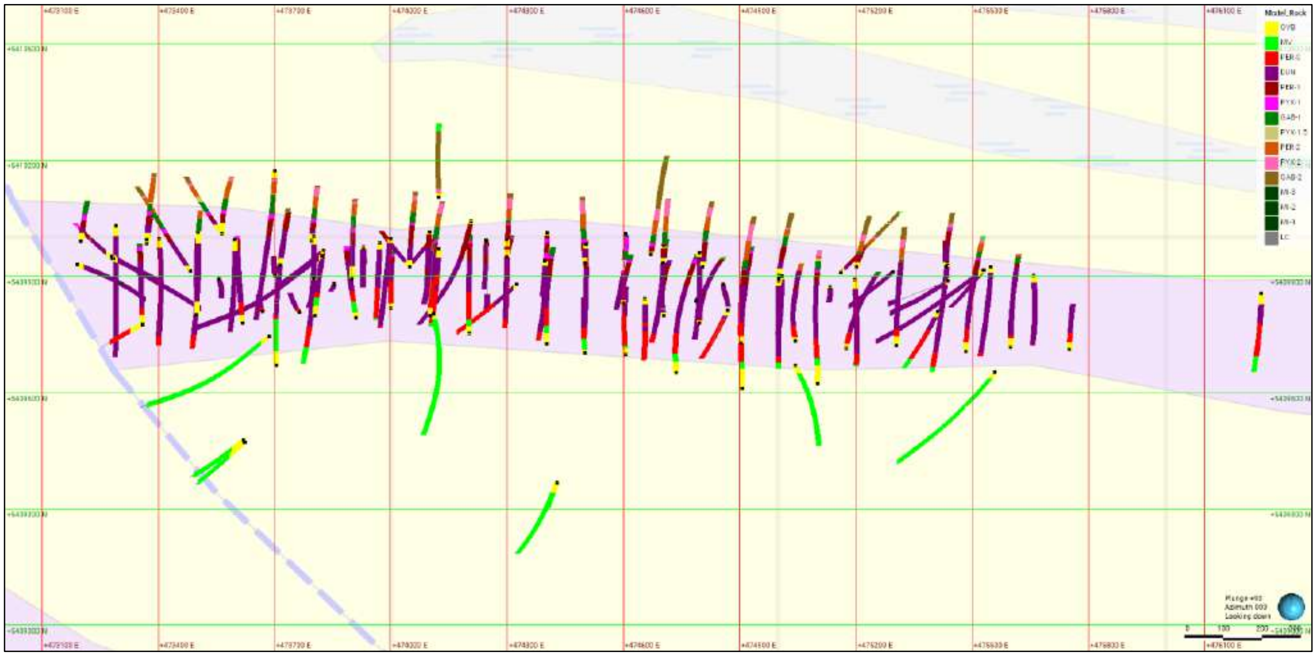
Using available core logging information, as well as aluminum/magnesium ratios for rock type validation, lithologies from the Main-West and East zones were initially generalized and grouped into broader categories to facilitate correlation between major units (Table 14-1, Figures 14-4 and 14-5). Subsequently, individual horizons from each category were singled out and codified, given the regularity and ordered appearance exhibited by the different ultramafic units and gabbro/metavolcanics in drilling logs across both zones, akin to stratigraphic layering though in this case with distinct subvertical fashion.

**Table 14-1: Lithologies with Respective Original and Model Rock Codes**

| LITHOLOGY | MODEL CODE | |
|---|---|---|
| Overburden | OVB | |
| Lamprophyre | LAM | |
| Diabase | MI | |
| Mafic Intrusive | MI / GAB | |
| Gabbro | GAB | UM |
| Pyroxenite | PYX | UM |
| Talcose Ultramafics | PYX / PER / DUN | UM |
| Peridotite | PER / DUN | UM |
| Dunite | DUN | UM |
| Bleached Dunite | DUN | UM |
| Carbonatized Dunite | DUN | UM |
| Laminated Dunite | DUN | UM |
| Mafic Metavolcanics | MV | |
| Intermediate Metavolcanics | MV | |
| Felsic Metavolcanics | MV | |
| Metasediments | MV | |
| Major Fault | FT | |
| Lost Core | LC | |
| Anorthosite | Not modelled, few intervals merged within other units | |
| Intermediate Intrusive | Not modelled, few intervals merged within other units | |
| Rodingite Vein | Not modelled, few intervals merged within other units | |
| Quartz Vein | Not modelled, few intervals merged within other units | |
| Quartz-Feldspar Porphyry | Not modelled, few intervals merged within other units | |

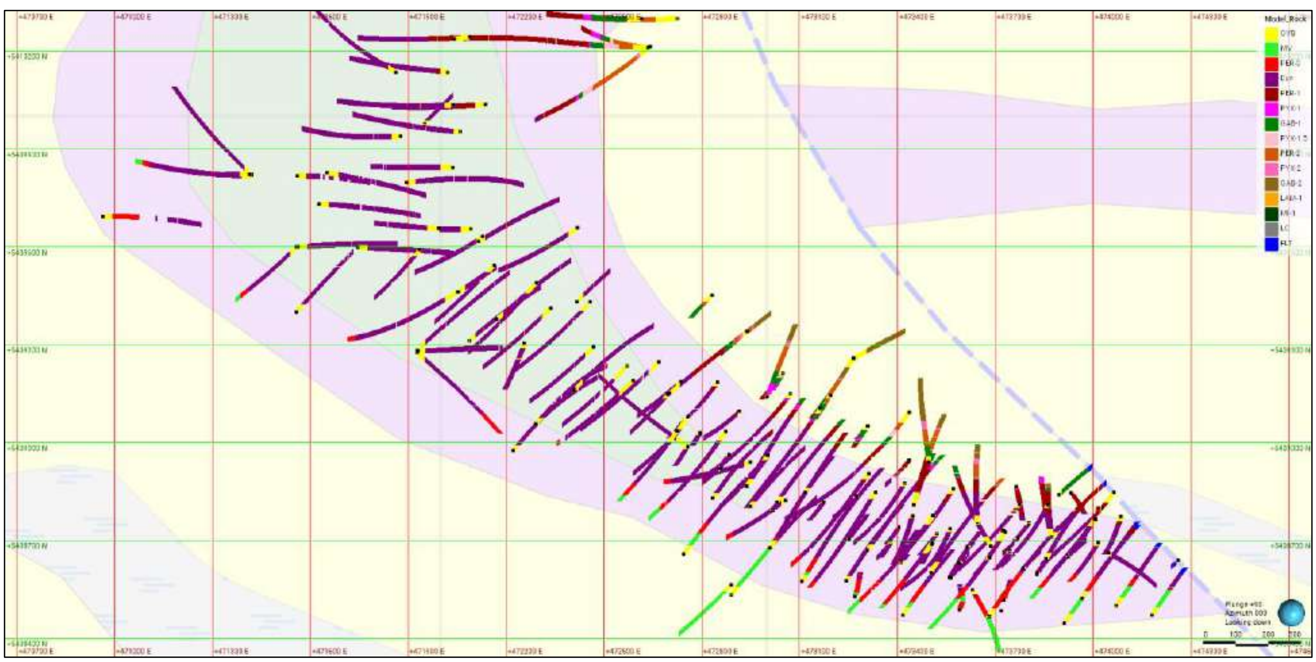
Source: Caracle Creek, 2023.

**Figure 14-4: East Zone Plan View of Drillhole Intercepts Showing Grouped Lithologies**



Note: Background geology from Ontario Geological Survey (MRD126). Source: Caracle Creek, 2023.

**Figure 14-5: Main-West Zone Plan View of Drillhole Intercepts Showing Grouped Lithologies**



Note: Background geology from Ontario Geological Survey (MRD126). Source: Caracle Creek, 2023.

Horizons were numbered relative to their location, from south to north:

- metavolcanics (MV), the southernmost unit, though also found encompassing the subsequent ultramafic/mafic rock sequence (UM unit) as it bends or wedges out.
- pyroxenite (PYX-0), very rarely present (mostly talc-altered), not enough to be considered for modelling.
- peridotite (PER-0).
- dunite (DUN).
- peridotite (PER-1).
- pyroxenite (PYX-1).
- gabbro (GAB-1).
- pyroxenite (PYX-1.5), irregularly present, though enough to be considered for modelling.
- peridotite (PER-2).
- pyroxenite (PYX-2).
- gabbro (GAB-2), the last correlatable unit in the sequence.

Diabase and lamprophyre dikes, common and easily identifiable in the Main-West Zone compared to the East Zone, were singled out and codified through interval correlations. Lastly, the fault zone was codified based on titanium/aluminum ratios and rock behaviour in the proximity of the known fault trace.

Complementary datasets (i.e., assays, density, hole mag-sus) facilitated verification when lithological boundaries were unclear. For example, nickel and most other assay grades drop noticeably outside of the core dunite-peridotite unit, density differences between ultramafic and gabbroic rocks are apparent, while magnetic susceptibility allows for discrimination of ultramafic rocks from the southern metavolcanic rocks. Regional geophysics datasets provided further information to identify the general eastern and western extents of the UM unit, as well as confirmation of its overall shape and dimensions.

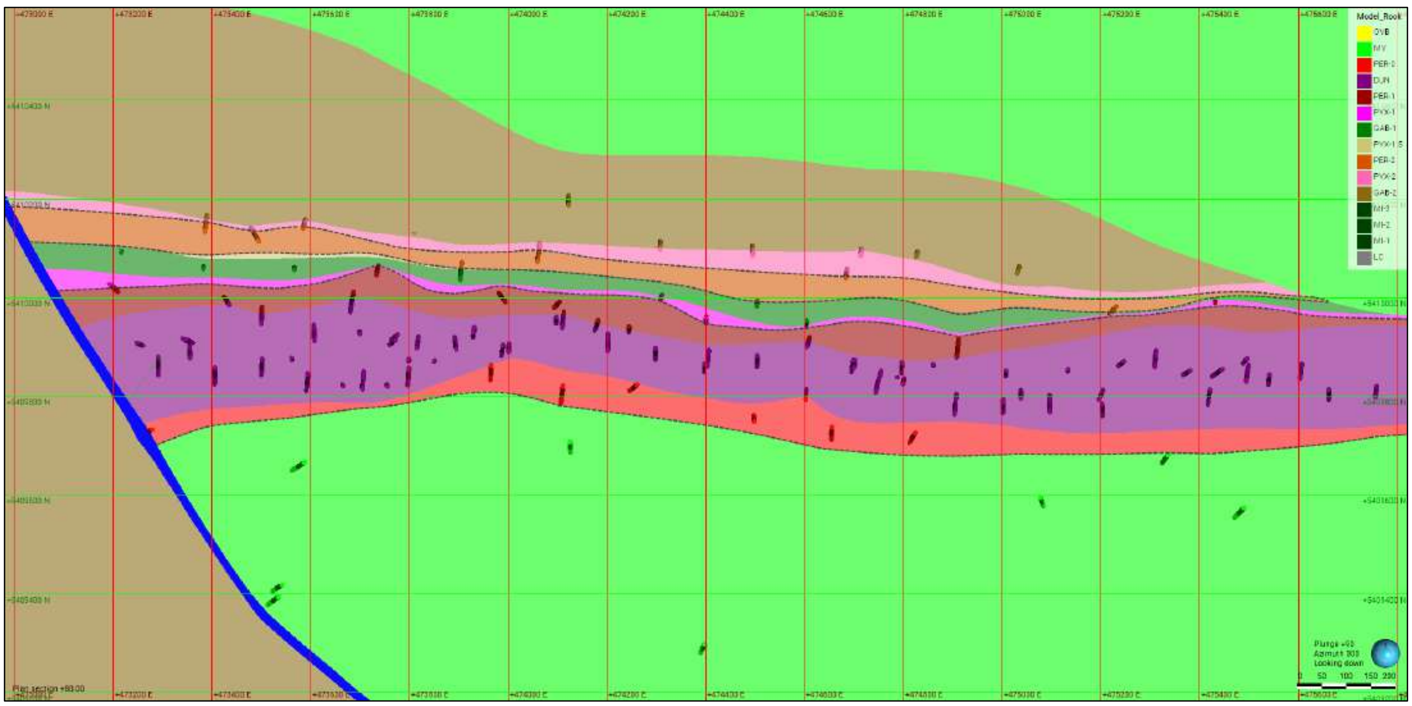
The modelling area, covering both zones, is 7.5 km long (from 469500 mE to 477000 mE) by 5 km wide (from 5407000 mN to 5412000 mN), with a maximum depth set at -600 m elevation, approximately 835 m below overburden. Despite this bottom limit, both zones can be considered open at depth due to two drillholes in the Main-West Zone and one in the East Zone extending past this limit, with intercepts containing mostly moderate nickel grades (0.20-0.25% Ni) with a few higher grades (>0.25% Ni).

Each zone was modelled separately, divided by an approximately 15-metre-wide fault zone volume based after the Ontario Geological Survey map trace and the fault intercepts codified in the database (Table 14-1). Note that, despite the naming conventions, the Main and West zones comprise a single orebody (Figure 14-3), and as such are treated as one zone and modelled together.

The newly codified database also served as the base for the lithology models (Figures 14-6 through 14-9), with second pass verifications using complementary datasets. Given the relatively simple nature of the geological sequence, cross-section interpretation was deemed unnecessary before modelling. Contacts between horizons were interpolated individually and sequentially, adding polylines to maintain or modify their shape where necessary. This process helped improve the predictability of the models and, to some extent, compensates for the lack of information in some areas, especially at great depths.

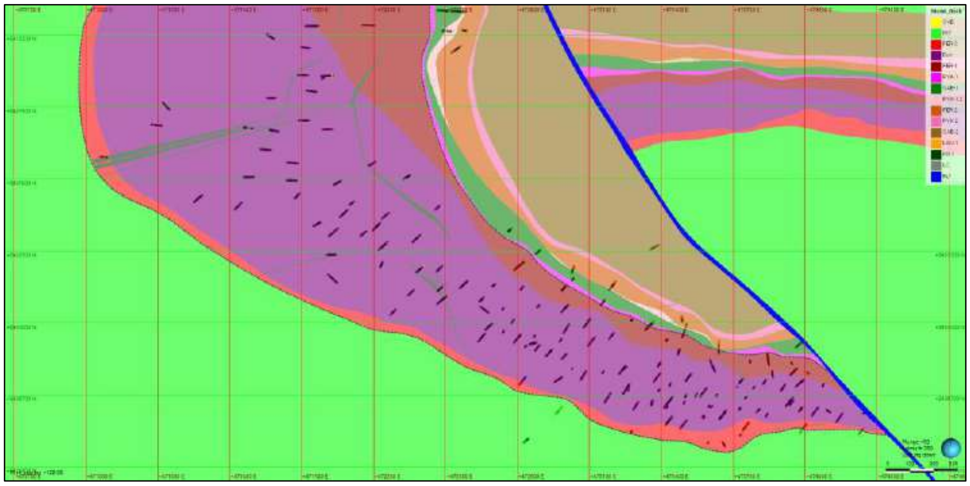
Dikes were modelled individually, then merged and cut against the previously modelled lithologies, with diabase structures following a subvertical trend and lamprophyre structures with a currently interpreted sub-horizontal orientation.

**Figure 14-6: East Zone Plan Section (80 m) Showing the Drillhole Traces and the Final Lithology Wireframes**



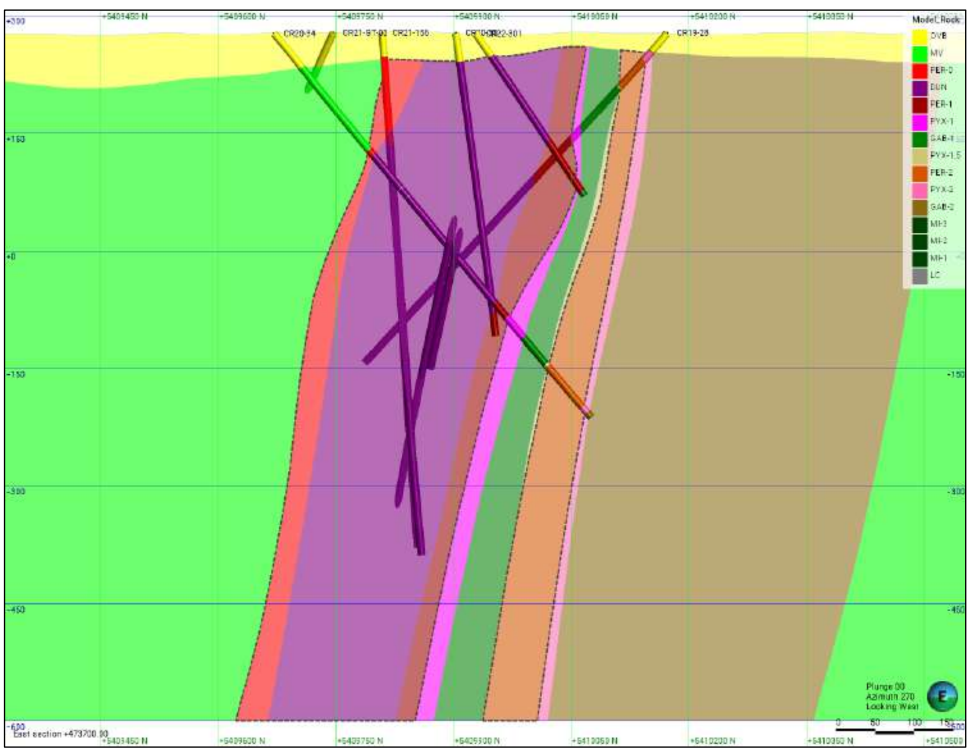
Note: Mineralization envelopes are marked within dashed lines. Source: Caracle Creek, 2023.

**Figure 14-7: Main-West Zone Plan Section at 120 m Showing Drillhole Traces and Final Lithology Wireframes**



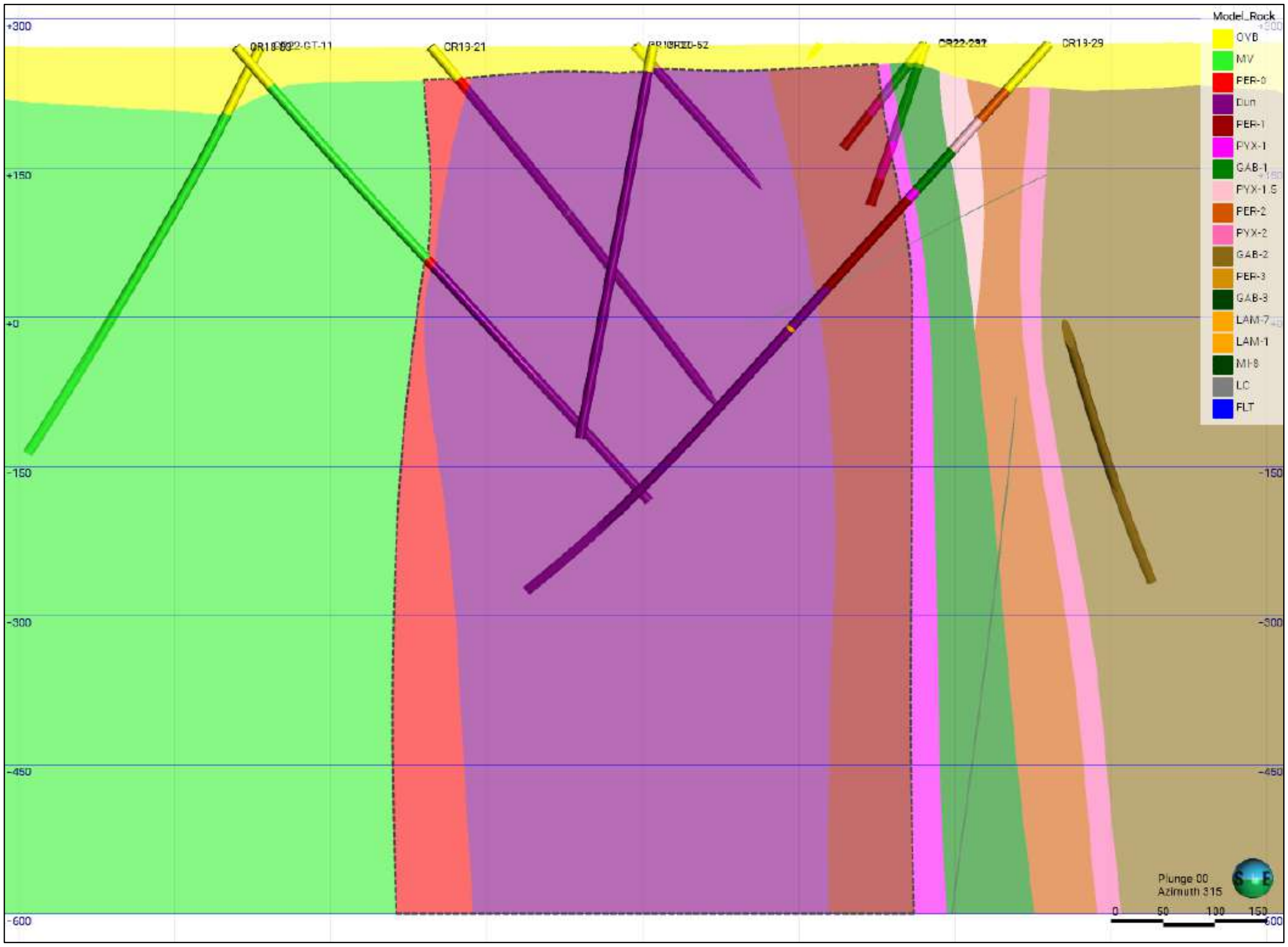
Note: Mineralization envelopes are marked within dashed lines. Source: Caracle Creek, 2023.

**Figure 14-8: East Zone Section (473700 mN) Looking West Showing Drillhole Traces and the Final Lithology Wireframes**




Note: Mineralization envelopes are marked within dashed lines. Source: Caracle Creek, 2023.

**Figure 14-9: Main-West Zone Oblique Section Looking West Showing the Drillhole Traces and the Final Lithology Wireframes**




Note: Mineralization envelopes are marked within dashed lines. Source: Caracle Creek, 2023.

The geological models for the Main-West and East zones developed by Caracle Creek constitute the basis for the interpretation of mineralization and the corresponding mineral estimation domains.

### 14.4.3 Alteration

Ultramafic rocks of the CNC are largely serpentinized which influences rock density (representative of serpentinization degree) and magnetic susceptibility (representative of magnetite content mostly derived from serpentinization) are distributed, which are necessary variables for economic analysis. The study of alteration in the project, besides the variables involved, comes from alteration and lithology logs, as well as QEMSCAN analysis for validation purposes.

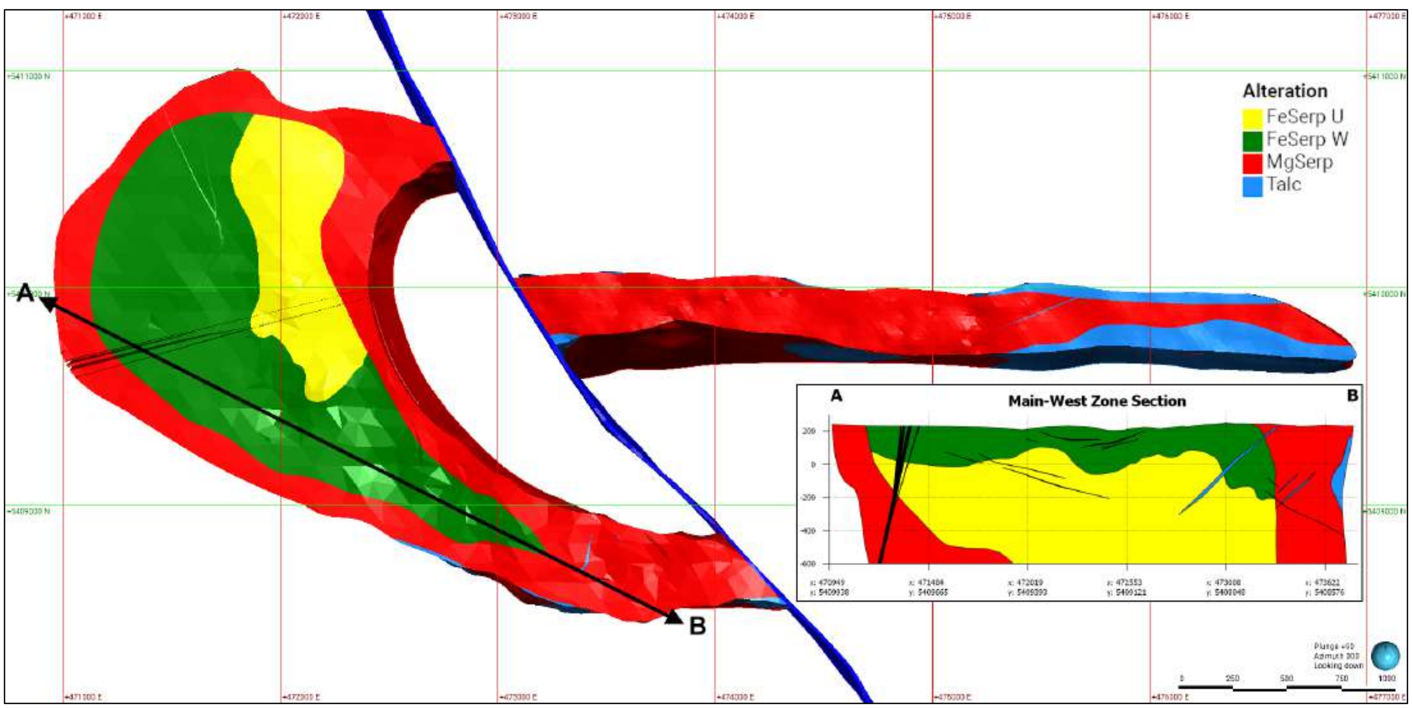
The main alteration in the Main-West and East zones is serpentinization, given the ultramafic rock type prevalence, with talc-carbonation as a secondary occurrence. Other alteration types (silicification, sericitization, albitization, among others) are only found in minor quantities and affecting very limited areas so as to become relevant for study. Thus, alteration modelling was restricted to the main area of influence of the two relevant alteration types, represented by the mineralized ultramafic horizons PER-0, DUN and PER-1 (Figure 14-10).

Serpentinization presents two recognizable phases: an early to intermediate phase ("FeSerp," <60-75% serpentinization) forming Fe-rich serpentine/brucite with very low to no magnetite; and an advanced to complete phase ("MgSerp," 75-100% serpentinization), forming Mg-rich (Fe-poor) serpentine/brucite with secondary magnetite. The first phase is metastable, meaning it is susceptible to oxidation (i.e., weathering), which is an ongoing process in the project. As such, weathering was included in the model as an additional alteration type.

The datasets used and the criteria applied for each alteration domain are briefly described:

- MgSerp/FeSerp: Serpentinization phase discrimination was based on a ~75 magnetic susceptibility (magnetite content) cutoff, with low mag-sus values representing low magnetite contents and thus a FeSerp phase, and vice versa for the MgSerp phase. Both phases are present in the Main-West Zone, while the East Zone only shows MgSerp.
- Weathered/Unweathered FeSerp: Weathering was identified in the Main-West Zone's FeSerp domain due to unusually low-density values close to the overburden, increasing gradually with depth. Thus, this subdivision was based on a 2.65 g/cm$^3$ cutoff.
- Talc-carbonate: Generated from the "talcose ultramafic" lithology logs, complemented with density values.

**Figure 14-10: Plan View of the Alteration Model for Main-West and East Zones**




Note: Inset shows a vertical section cutting through the Main-West Zone (see A-B trace in map). Source: Caracle Creek, 2023.

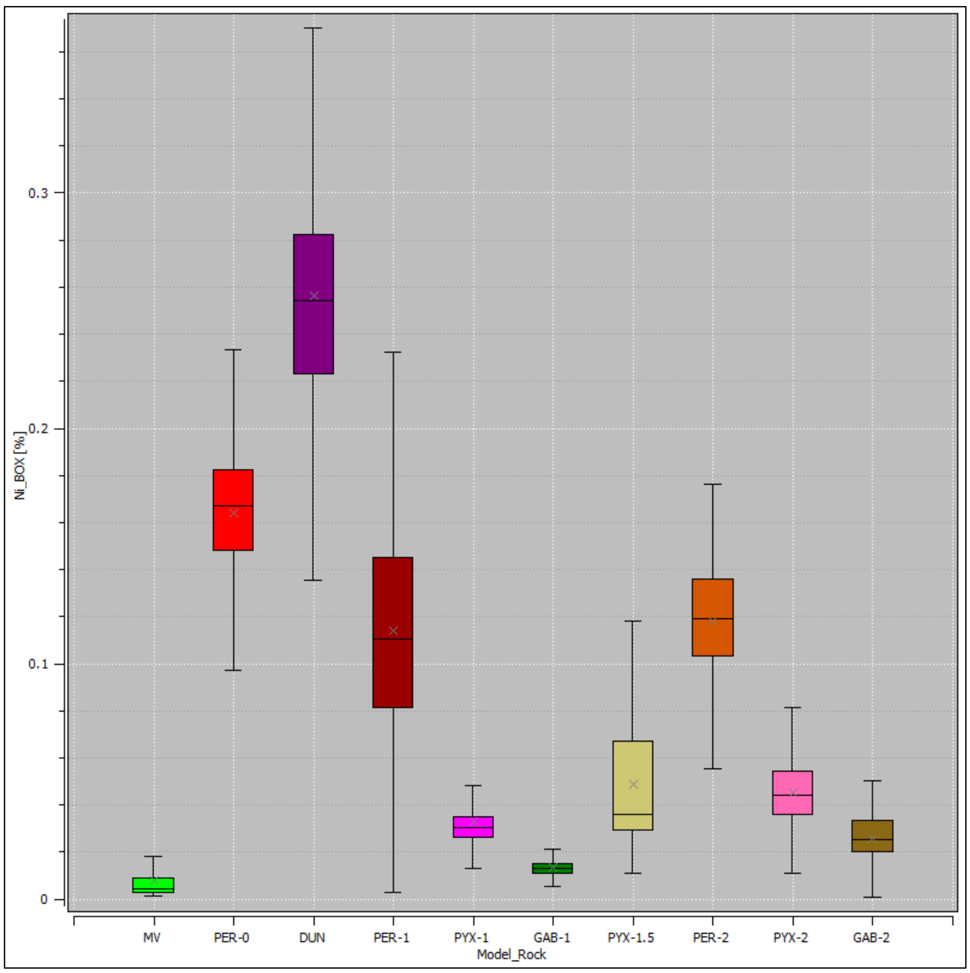
The alteration models for the Main-West and East zones developed by Caracle Creek constitute the basis for density and magnetic susceptibility estimation.

## 14.5 Data Analysis and Estimation Domains

### 14.5.1 East Zone: Exploratory Data Analysis (EDA)

The East Zone assay database comprises 25,135 results from drill core samples. Statistical analysis of nickel and other mineral grades filtered by lithology (Figure 14-11) reveals 4 horizons of interest, three of which (PER-0, DUN, PER-1) constitute the main mineralization target (EST-1) and thus contain most of the drilling and sampling (22,439 samples), while the last horizon (PER-2) is a secondary target (EST-2) isolated north of the main target and poorly tested (731 samples). This section will be focused on the EST-1 domain analysis.

**Figure 14-11: East Zone Boxplot Showing the Distribution of Nickel Grades According to Lithological Units**

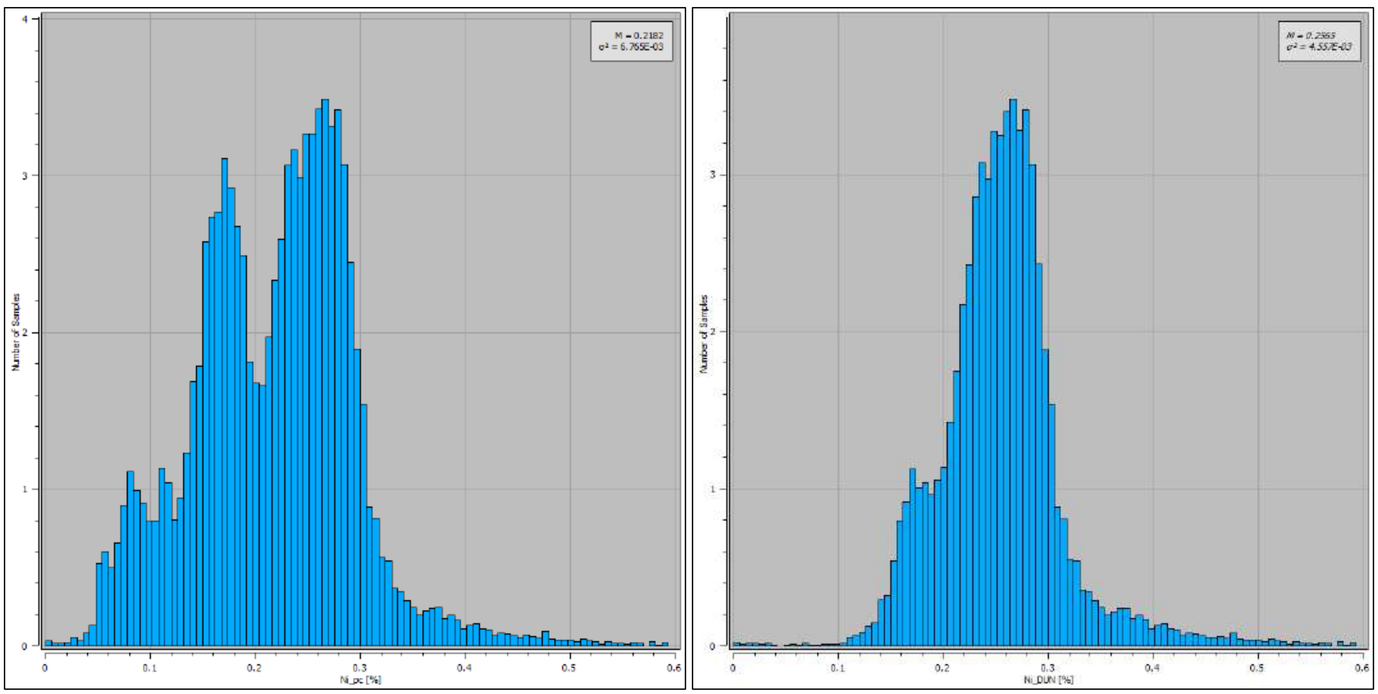



Source: Caracle Creek, 2023.

A histogram of nickel grades within the EST-1 domain shows a bimodal distribution, with a lower grade population of 0.15-0.19% Ni and a higher grade population of 0.22-0.29% Ni (Figure 14-12, left). Given that mineralization visibly follows the lithological trend, these two populations also manifest as horizons: A central one comprising moderate to high grades, and two contiguous northern and southern horizons with lower grade populations, occasionally dropping to very low concentrations, especially towards the north.

Though at first sight it would seem like there is a direct relationship between these three mineral horizons and the three lithologies that make up the EST-1 domain (PER-0, DUN, PER-1), a histogram of nickel grades within the DUN unit still shows a slight bimodality (Figure 14-12, right).

**Figure 14-12: East Zone Histograms Showing the Bimodal Distribution of Nickel Grades within the EST-1 Domain (Left) and its Subordinate DUN Unit (Right)**

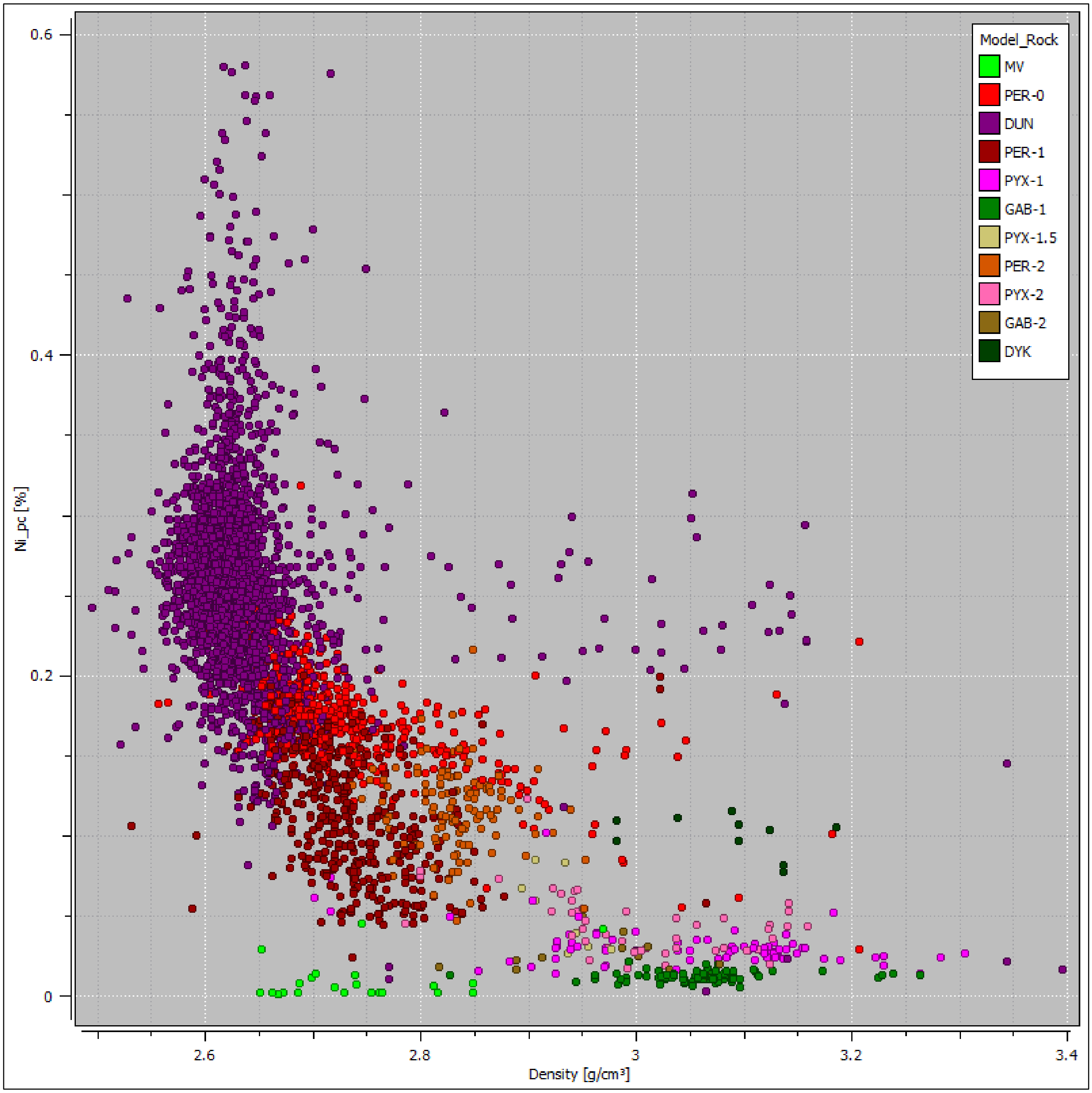



Source: Caracle Creek, 2023.

It is important to note that the lower grades (<0.2% Ni) preserving the bimodality within the DUN unit are always found adjacent to the southern PER-0 and northern PER-1 units. This is evidence that the contact between dunite and peridotite is not hard but transitional, an observation further supported by core descriptions found in lithology logs.

To completely remove this effect and effectively separate both populations, a combination of aluminum and magnesium cutoffs (~23.5% Mg and ~0.7% Al) were first established, elements which previously aided with lithological discrimination, allowing for a sufficiently reliable separation of actual dunite with high grades and “transitional” dunite, which shares characteristics with the adjacent peridotite though with a mix of high and low grades. The latter thus calls for an additional filter based on densities (representative of serpentinization degree) due to their reasonable correlation with nickel grades (Figure 14-13), with a chosen window of >2.65 g/cm$^3$ (<80% serpentinization degree) to achieve a proper selection of low grades (<0.2% Ni) within transitional dunite.

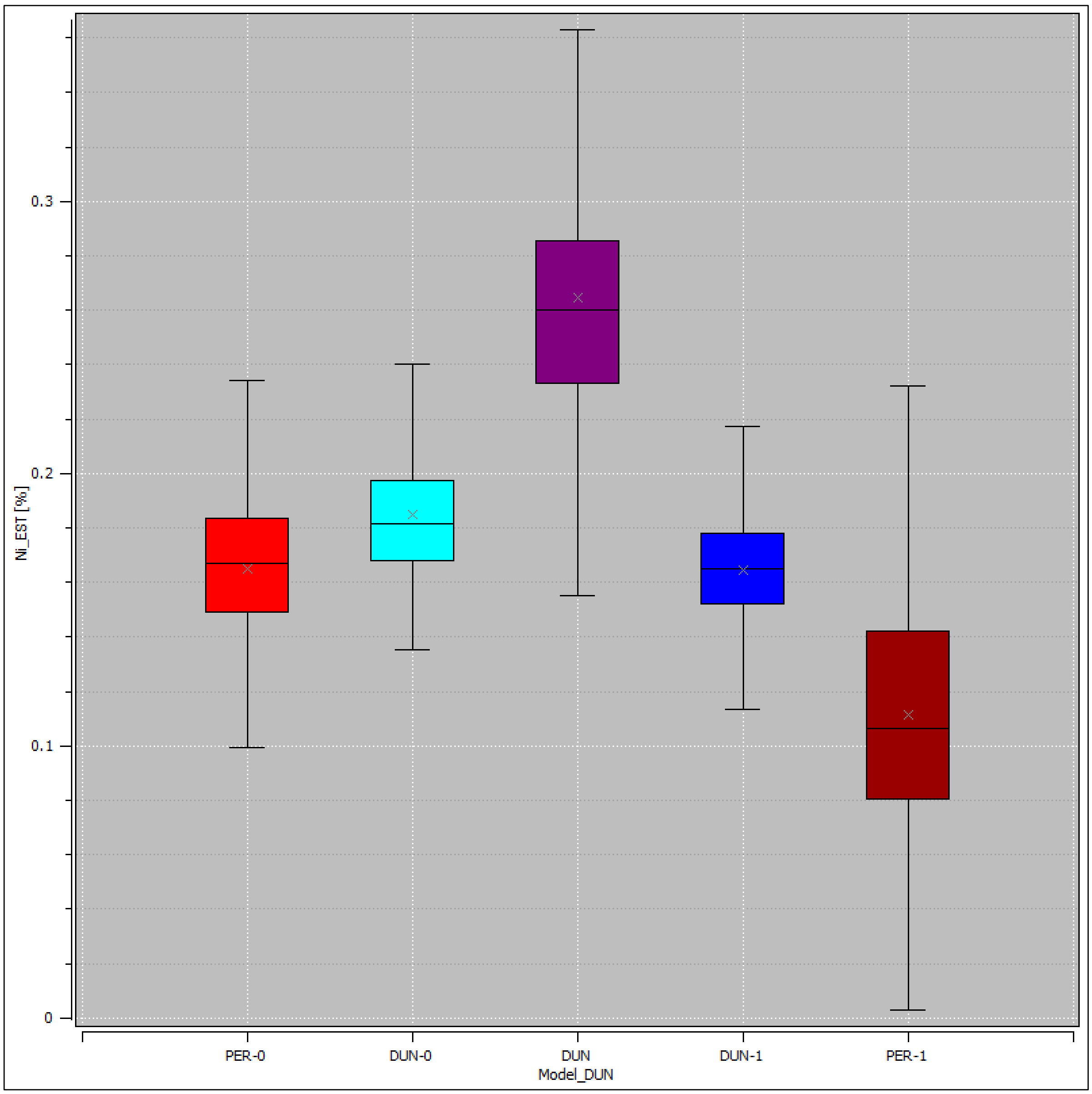
**Figure 14-13: East Zone Cross-Plot Showing the Correlation of Density Values to Nickel Grades According to Lithological Units**




Source: Caracle Creek, 2023.

The result, after application of the previous rules, is a successful discrimination of two dunite types, each corresponding to a specific nickel grade population from the observed bimodality: one “pure,” highly serpentinized and higher grade central dunite, and a secondary “transitional,” moderately serpentinized and lower grade dunite, located (when present) between the northern and southern dunite/peridotite contacts. Statistical analysis of these subdivisions confirms that the transitional dunite units (DUN-0, DUN-1) are more like their peridotite counterparts (PER-0, PER-1) than to the central dunite unit (Figure 14-14).

**Figure 14-14: East Zone Boxplot Showing the Distribution of Nickel Grades According to the Dunite Unit Subdivision and Adjacent Peridotite Units**



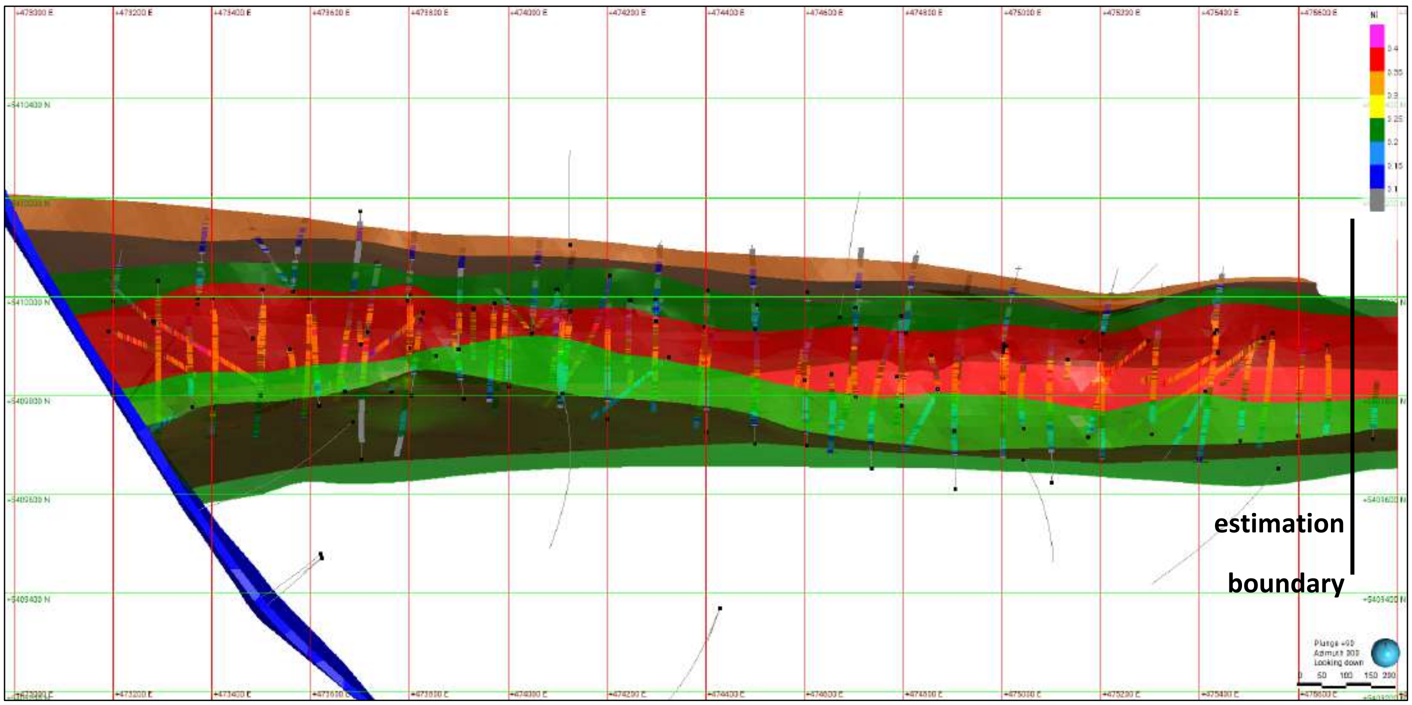

Source: Caracle Creek, 2023.

### 14.5.2 East Zone: Estimation Domains (Grade Shells)

The two nickel grade populations identified in the EDA for the EST-1 domain are thus proven to correspond to a central higher grade dunite unit and a combination of lower grade transitional dunite and peridotite units, both to the north and south. The subdomains representing these units were generated by the same interpolation process used for the lithological model, given that the mineralization trend follows the lithology, resulting in a higher grade (HG) domain and two adjacent lower grade (SLG, NLG) domains (Figure 14-15).

The estimation domains were limited to the lithologies of interest (PER-0, DUN, PER-1, PER-2) up to 50 m beyond the second to last easternmost drillhole (estimation boundary, Figure 14-15), given that the last is an exploratory hole located almost 500 m away from the previous hole.

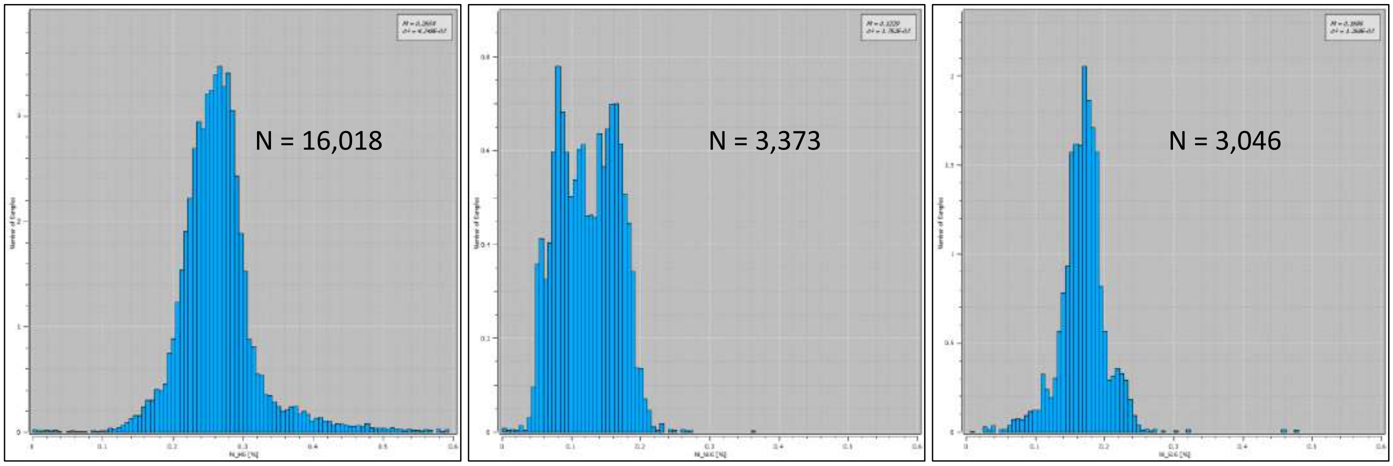
**Figure 14-15: East Zone Plan View with Nickel Grade Drillhole Traces and Estimation Domains**




Note: Shows HG (red), SLG (light green), NLG (dark green) and EST-2 (brown) estimation domains. Source: Caracle Creek, 2023.

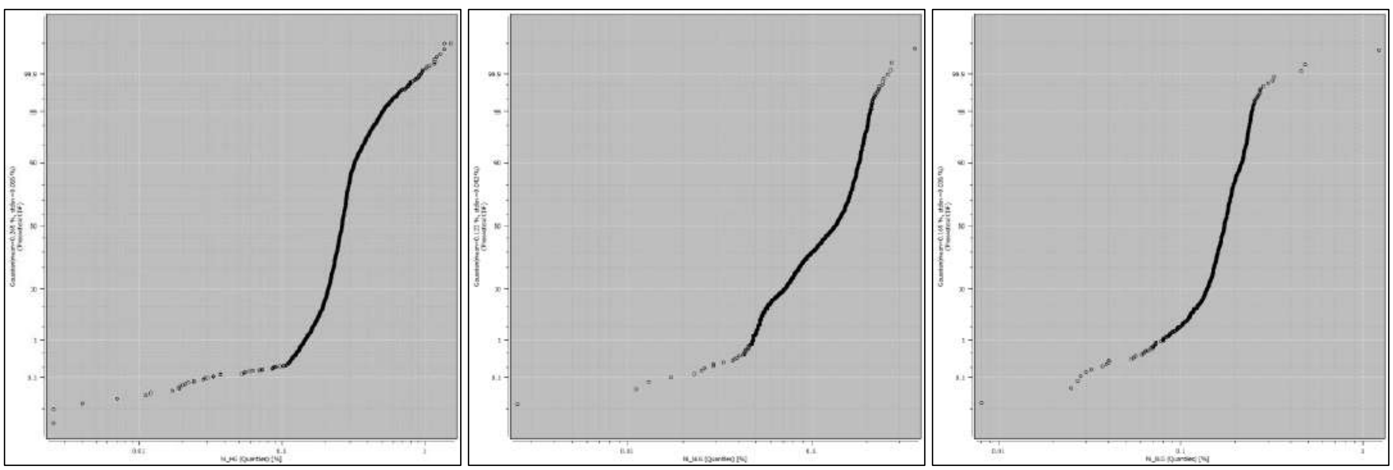
Statistical analysis within the HG/LG domains confirmed the appropriate separation of the two nickel grade populations, evidenced by adequate distributions and statistical parameters for each estimation domain, though with lower sample numbers in the case of the lower grade domains (Figures 14-16 and 14-17). Further analyses for the other estimation variables (see Section 14.7) suggested no need for additional domaining.

**Figure 14-16: East Zone Nickel Grade Histograms with Sample Amounts for the HG, NLG and SLG Estimation Domains**




Source: Caracle Creek, 2023.

**Figure 14-17: East Zone Nickel Grade Probability Plots for the HG, NLG and SLG Estimation Domains**



Source: Caracle Creek, 2023.

### 14.5.3 East Zone: Compositing and Capping

Considering that 99% of drillhole samples are 1.5 m in length, the 15.0 m height of the blocks (see Section 14.6, Block Modelling) and the sample database size, composites of 7.5 m were deemed the most appropriate. These were generated within the general EST-1 domain (meaning that HG/LG boundaries were not considered) and EST-2 domain for the seven studied elements (Ni, Co, Fe, Cr, S, Pd, Pt).

The density database consists of 4,196 data points for the serpentine-altered portion of the EST-1 domain and 105 data points for the EST-2 domain. Similarly, the brucite (mineralogy) database consists of 1,195 samples for the serpentine-altered portion of the EST-1 domain, and none for the EST-2 domain. Given that both density and brucite data are essentially punctual, as opposed to continuous intervals, they cannot be composited.

Capping was applied before compositing and only for “true” outliers (values out of context such as a single high-grade among low-grade values). Capping values were mainly based on histogram and probability plot distributions, which yielded top caps of 0.6% for nickel, 0.03% for cobalt, 1.2% for chromium, 1 ppm for palladium and none for sulphur, iron, platinum, brucite or density. Anomalous zones that transition into very high values were not capped at this stage, instead their influence was limited to a set distance during estimation if deemed necessary (Table 14-2).

**Table 14-2: East Zone Capping Values and Summary Statistics of Declustered Samples and Composites by Element and their Selected Estimation Domain**

| Domain | Element | 1.5 m Drillhole Samples | | | | | 7.5 m Composites | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Count | Mean | Std Dev | CV | Med | Count | Mean | Std Dev | CV | Med |
| **HG** | Ni % | 16,018 | **0.26** | 0.07 | 0.25 | 0.26 | 3,204 | **0.27** | 0.06 | 0.21 | 0.26 |
| | Fe % | 16,018 | **5.80** | 0.85 | 0.15 | 5.71 | 3,204 | **5.80** | 0.68 | 0.12 | 5.71 |
| | Cr % | 16,018 | **0.65** | 0.11 | 0.17 | 0.66 | 3,204 | **0.65** | 0.10 | 0.15 | 0.67 |
| | Density | 3,058 | **2.64** | 0.08 | 0.03 | 2.62 | | | | | |
| | Brucite | 897 | **3.09** | 1.50 | 0.49 | 3.02 | | | | | |
| **NLG** | Ni % | 3,373 | **0.12** | 0.04 | 0.35 | 0.12 | 682 | **0.12** | 0.04 | 0.34 | 0.12 |
| | Fe % | 3,373 | **7.32** | 0.75 | 0.10 | 7.33 | 682 | **7.31** | 0.54 | 0.07 | 7.37 |
| | Cr % | 3,373 | **0.52** | 0.19 | 0.37 | 0.50 | 682 | **0.52** | 0.15 | 0.29 | 0.49 |
| | Density | 633 | **2.72** | 0.05 | 0.02 | 2.71 | | | | | |
| | Brucite | 120 | **0.21** | 0.48 | 2.28 | 0.01 | | | | | |
| **SLG** | Ni % | 3,046 | **0.17** | 0.04 | 0.21 | 0.17 | 611 | **0.17** | 0.03 | 0.17 | 0.17 |
| | Fe % | 3,046 | **7.36** | 0.74 | 0.10 | 7.42 | 611 | **7.36** | 0.59 | 0.08 | 7.45 |
| | Cr % | 3,046 | **0.50** | 0.10 | 0.21 | 0.50 | 611 | **0.50** | 0.09 | 0.18 | 0.51 |
| | Density | 505 | **2.73** | 0.09 | 0.03 | 2.71 | | | | | |
| | Brucite | 178 | **0.04** | 0.13 | 3.14 | 0 | | | | | |
| **EST-1** | Co % | 22,437 | **0.013** | 0.002 | 0.165 | 0.012 | 4,497 | **0.013** | 0.002 | 0.123 | 0.012 |
| | S % | 22,431 | **0.06** | 0.11 | 1.69 | 0.03 | 4,497 | **0.06** | 0.10 | 1.62 | 0.03 |
| | Pd ppm | 22,382 | **0.013** | 0.071 | 5.408 | 0.003 | 4,490 | **0.010** | 0.050 | 3.700 | 0.000 |
| | Pt ppm | 22,382 | **0.010** | 0.042 | 4.150 | 0.005 | 4,490 | **0.010** | 0.030 | 3.334 | 0.000 |
| **EST-2** | Ni % | 731 | **0.12** | 0.03 | 0.21 | 0.12 | 146 | **0.12** | 0.02 | 0.18 | 0.12 |
| | Co % | 731 | **0.013** | 0.002 | 0.154 | 0.013 | 146 | **0.013** | 0.002 | 0.126 | 0.013 |
| | Fe % | 731 | **9.70** | 1.42 | 0.15 | 9.90 | 146 | **9.69** | 1.26 | 0.13 | 9.97 |
| | Cr % | 731 | **0.35** | 0.10 | 0.29 | 0.34 | 146 | **0.35** | 0.08 | 0.23 | 0.35 |
| | S % | 731 | **0.05** | 0.05 | 0.97 | 0.04 | 146 | **0.05** | 0.03 | 0.67 | 0.04 |
| | Pd ppm | 729 | **0.011** | 0.009 | 0.745 | 0.011 | 146 | **0.010** | 0.010 | 0.473 | 0.010 |
| | Pt ppm | 729 | **0.006** | 0.004 | 0.721 | 0.005 | 146 | **0.010** | 0.000 | 0.486 | 0.000 |
| | Density | 105 | **2.83** | 0.04 | 0.014 | 2.83 | | | | | |

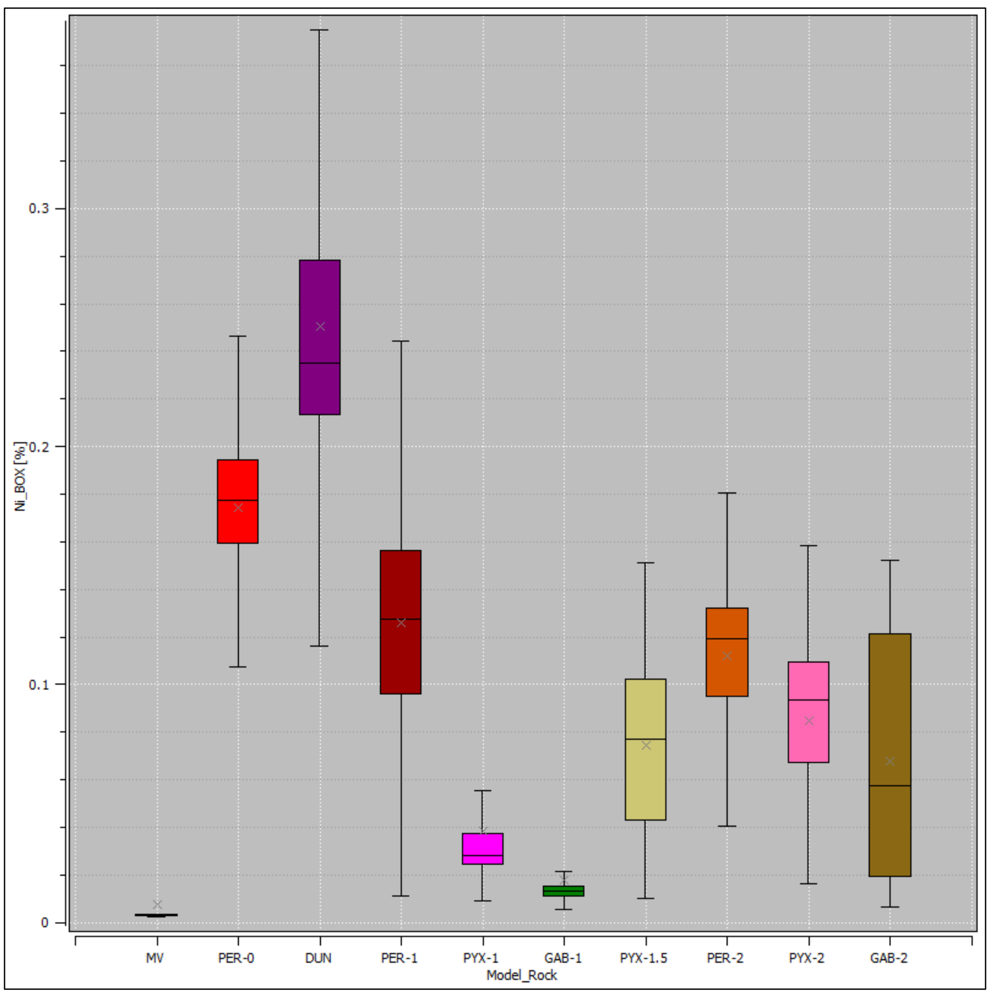
Note: Composites for PGE elements (Pd, Pt) have a lower count due to the exclusion of very high grades located at the northern contact with pyroxenite, corresponding to a high-grade structure or “PGE reef” that will be evaluated in a future estimate. Density and brucite count values only consider samples within the serpentine-altered portion of each domain. Source: Caracle Creek, 2023.

The resulting capped composites show more than adequate distributions and statistical parameters for most elements to carry out ordinary kriging (OK) estimation within the EST-1 domain and its subdomains, with S, Pd and Pt presenting some complexities due to their high CV values. Composites within the EST-2 domain are not enough to carry out OK estimation, opting instead for inverse distance weighting (IDW) estimation.

### 14.5.4 Main-West Zone: Exploratory Data Analysis (EDA)

The Main-West Zone assay database comprises 43,933 results from drill core samples. Statistical analysis of nickel and other mineral grades filtered by lithology (Figure 14-18) reveals 3 horizons of interest (PER-0, DUN, PER-1) which constitute the main mineralization target (EST) and thus contain most of the drilling and sampling (38,015 samples). There seems to be some potential in horizon PER-2 as a secondary target, like in the East Zone, though in this case it is not as well sampled considering its extension (699 samples).

**Figure 14-18: Main-West Zone Boxplot Showing the Distribution of Nickel Grades According to Lithological Units**



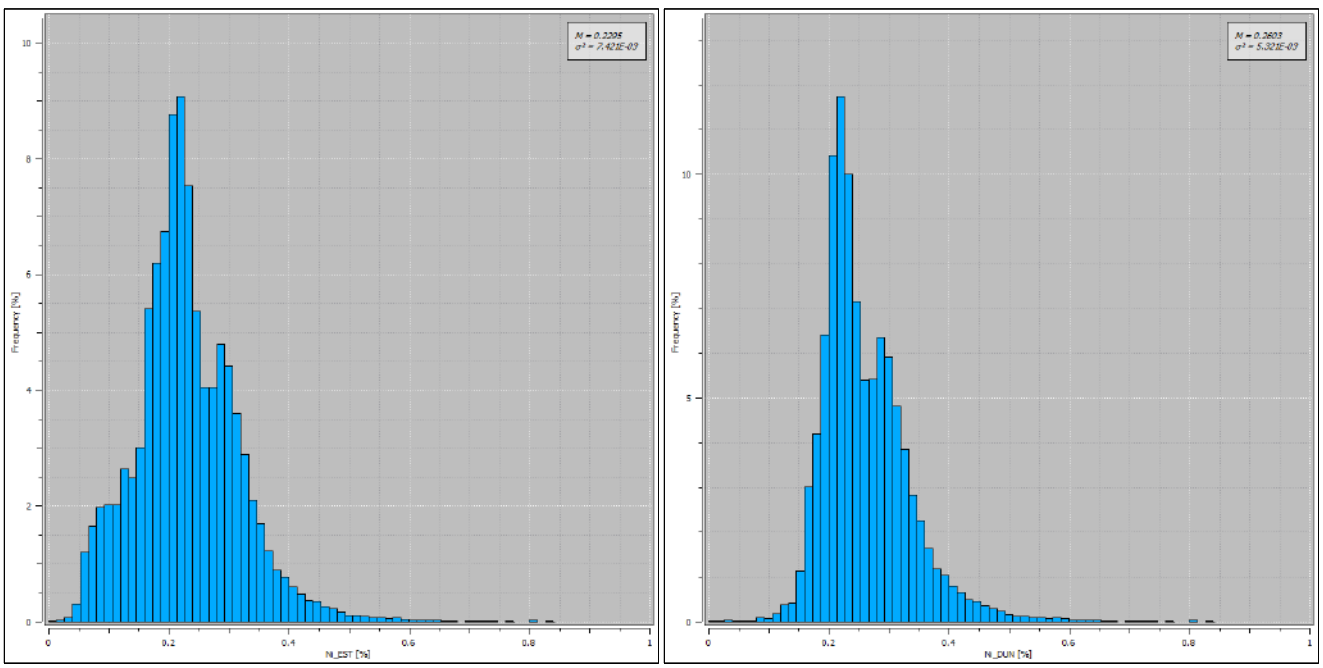

Source: Caracle Creek, 2023.

It should be noted that, even though mineralization trends and grades in the Main Zone (25,329 samples) and West Zone (12,686 samples) are continuous and justify treating them as a single zone, nickel and sulphur do show higher grades and a more distinct grade progression from centre to sides in the Main Zone compared to the West Zone. Because of this, nickel and sulphur estimation strategies were based primarily on the EDA of Main Zone samples.

A histogram of nickel grades within the Main Zone's EST domain shows a bimodal distribution, with a lower grade population of 0.19-0.25% Ni and a higher grade population of 0.27-0.31% Ni (Figure 14-19, left). Given that mineralization visibly follows the lithological trend, these two populations also manifest as horizons: A central one comprising moderate to high grades, and two contiguous northern and southern horizons with lower grade populations. In addition, it is possible to discern a third, very low-grade population (<0.15% Ni), manifested as an additional horizon running through the northern limit of the EST domain.

As also shown in the East Zone, there is no direct relationship between these mineral horizons and the lithologies that make up the EST domain (PER-0, DUN, PER-1). A histogram of nickel grades within the DUN unit still shows a clear bimodality (Figure 14-19, right panel).

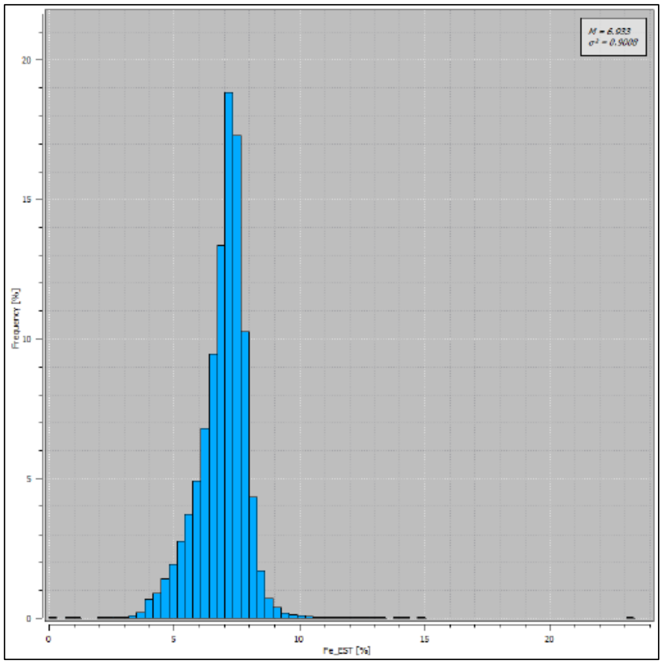
**Figure 14-19: Main Zone Histograms Showing the Bimodal Distribution of Nickel Grades within the EST Domain (Left) and its Subordinate DUN Unit (Right)**

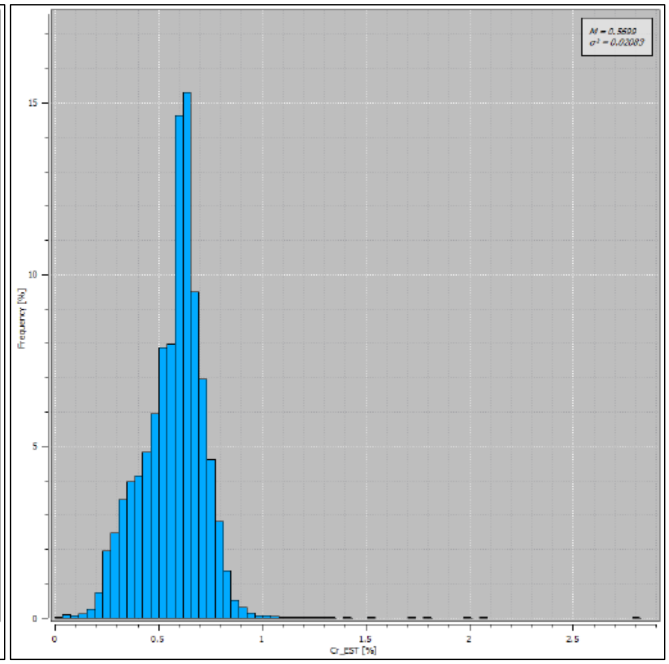



Source: Caracle Creek, 2023.

Other elements that present bimodal distributions within the EST domain are iron and chromium, though in an inverse and less prominent fashion than nickel, presented as lower grade populations of 4.0-6.0% Fe and 0.2-0.4% Cr within a central horizon, grading to two northern and southern horizons with higher grade populations of 6.0-9.0% Fe and 0.5-1.0% Cr (Figure 14-20). In the case of chromium, like with nickel before, it is possible to discern a northernmost lower grade (<0.5% Cr) population running parallel to the previous ones.

**Figure 14-20: Main-West Zone Histograms Showing the Bimodal Distribution of Iron Grades (Left) and Chromium Grades (Right) within the EST Domain**




Source: Caracle Creek, 2023.

Even though the Main Zone shares very similar characteristics to the East Zone, the transition and change in alteration from MgSerp to FeSerp towards the West Zone (see Section 14.4.3 and Figure 14-10) generates a major modification in the density arrangement and its correlation with nickel grades, preventing discrimination of high-grade and low-grade transitional dunite sections. This negates the method applied in the East Zone to separate dunite types and generate estimation domains.

Moreover, grade distributions and trends for elements such as nickel, iron and chromium have noticeable differences among them (as opposed to their more consistent behaviour in the East Zone), each case requiring specific domaining conditions, which makes the use of grade cutoffs the most practical way of separating their bimodal populations and generating estimation domains. This is also justified by the fact that grade populations manifest as reasonably correlatable horizons following the general lithology trend, allowing to individualize them as one would do with mineral veins.

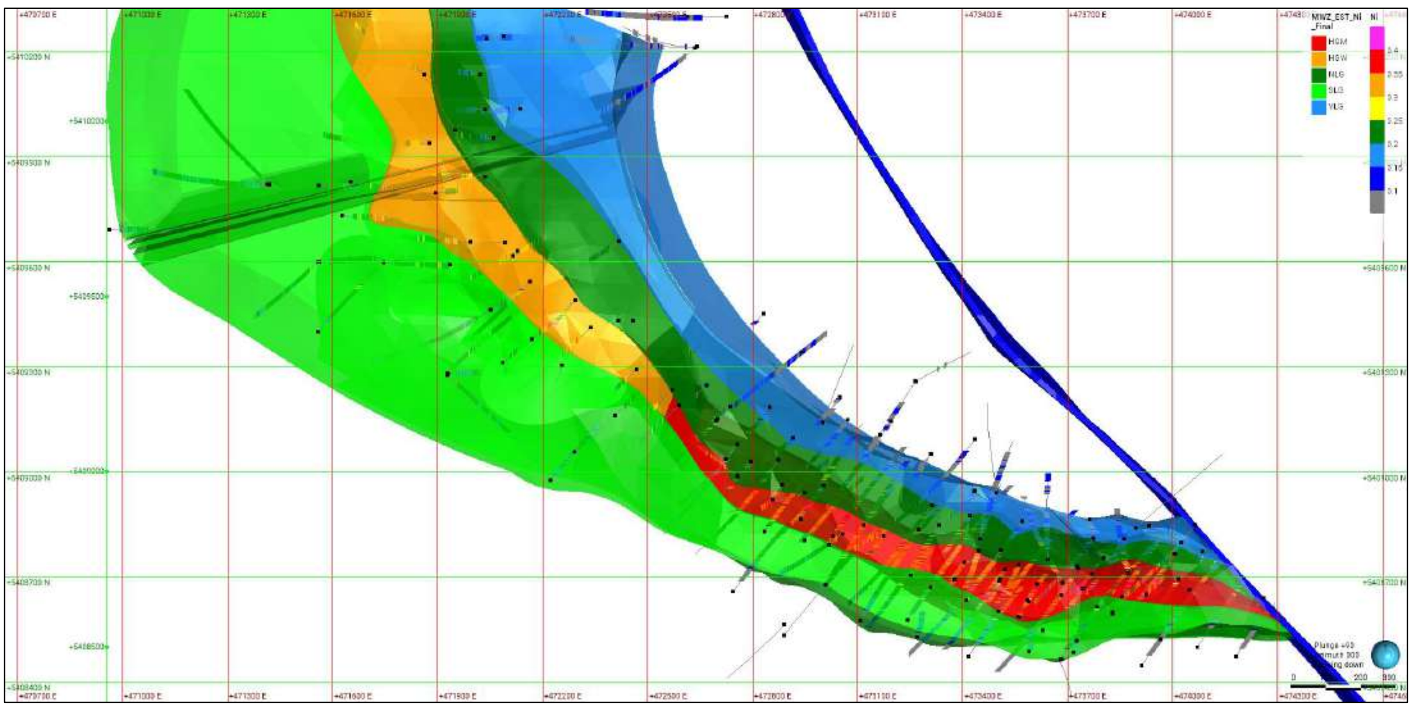
### 14.5.5 Main-West Zone: Estimation Domains (Grade Shells)

Having settled for grade cutoffs to separate bimodal populations means that estimation domains must be based on grade shells. These subdomains were generated within the estimation boundary by the same interpolation process used for the lithological model, resulting in a higher grade (HG) domain with two adjacent lower grade (SLG, NLG) domains and a northernmost very-low-grade (VLG) domain for nickel (Figure 14-21); a lower grade (LFE) domain with two adjacent higher grade (SHFE, NHFE) domains for iron (Figure 14-22), and a lower grade (LCR) domain with two adjacent higher grade (SHCR, NHCR) domains and a northernmost low-grade (VLCR) domain for chromium (Figure 14-23).

An additional subdivision for the HG nickel domain was defined to account for the observed nickel and sulphur high-grade variability differences between Main and West zones (HGM, HGW). This is the only instance where both zones were treated separately.

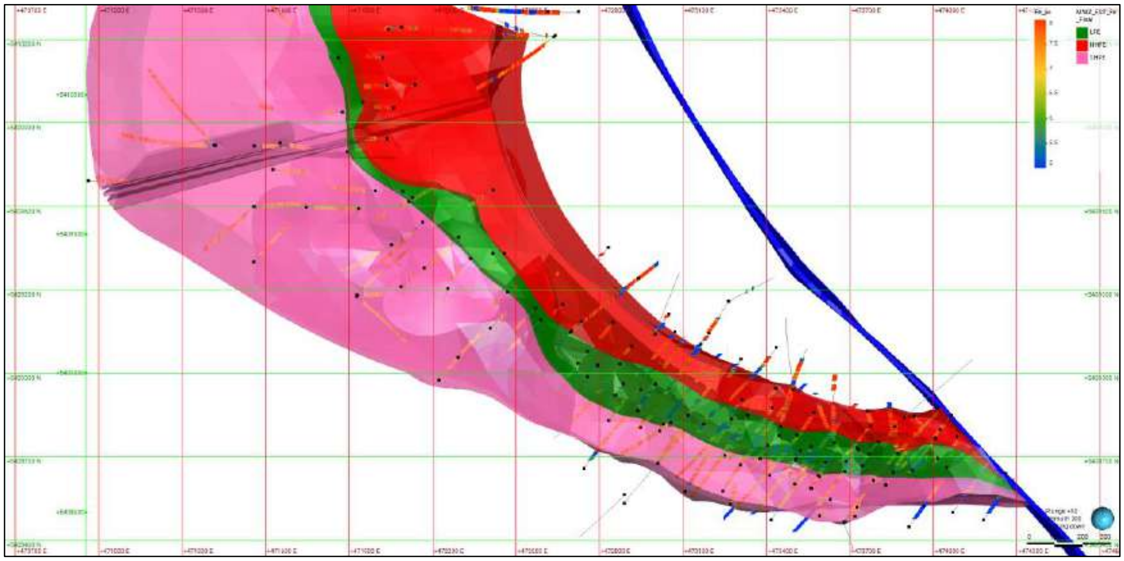
The estimation domains were limited to the lithologies of interest (PER-0, DUN, PER-1) up to 100 m beyond the second to last northernmost drillhole, given that the last is an exploratory hole located over 350 m away from the previous hole.

**Figure 14-21: Main-West Zone Plan View with Nickel Grade, Drillhole Traces, and Estimation Domains**



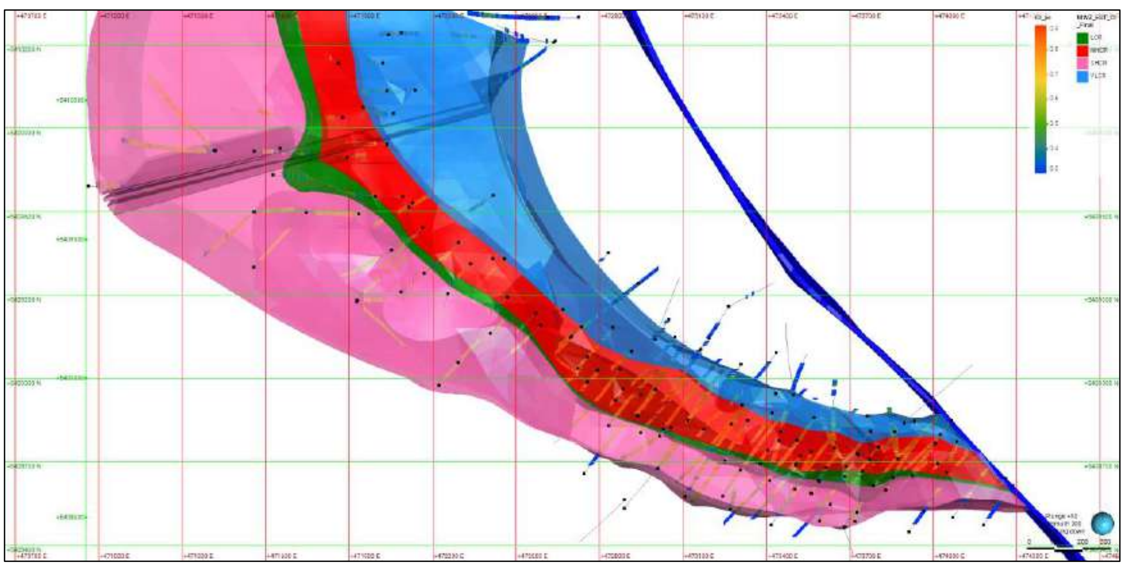
Note: Shows HGM (red), HGW (orange), SLG (light green), NLG (dark green) and VLG (light blue) estimation domains. Source: Caracle Creek, 2023.

**Figure 14-22: Main-West Zone Plan View with Iron Grade, Drillhole Traces, and Estimation Domains**



Note: Shows LFE (dark green), SHFE (pink) and NHFE (red) estimation domains. Source: Caracle Creek, 2023.

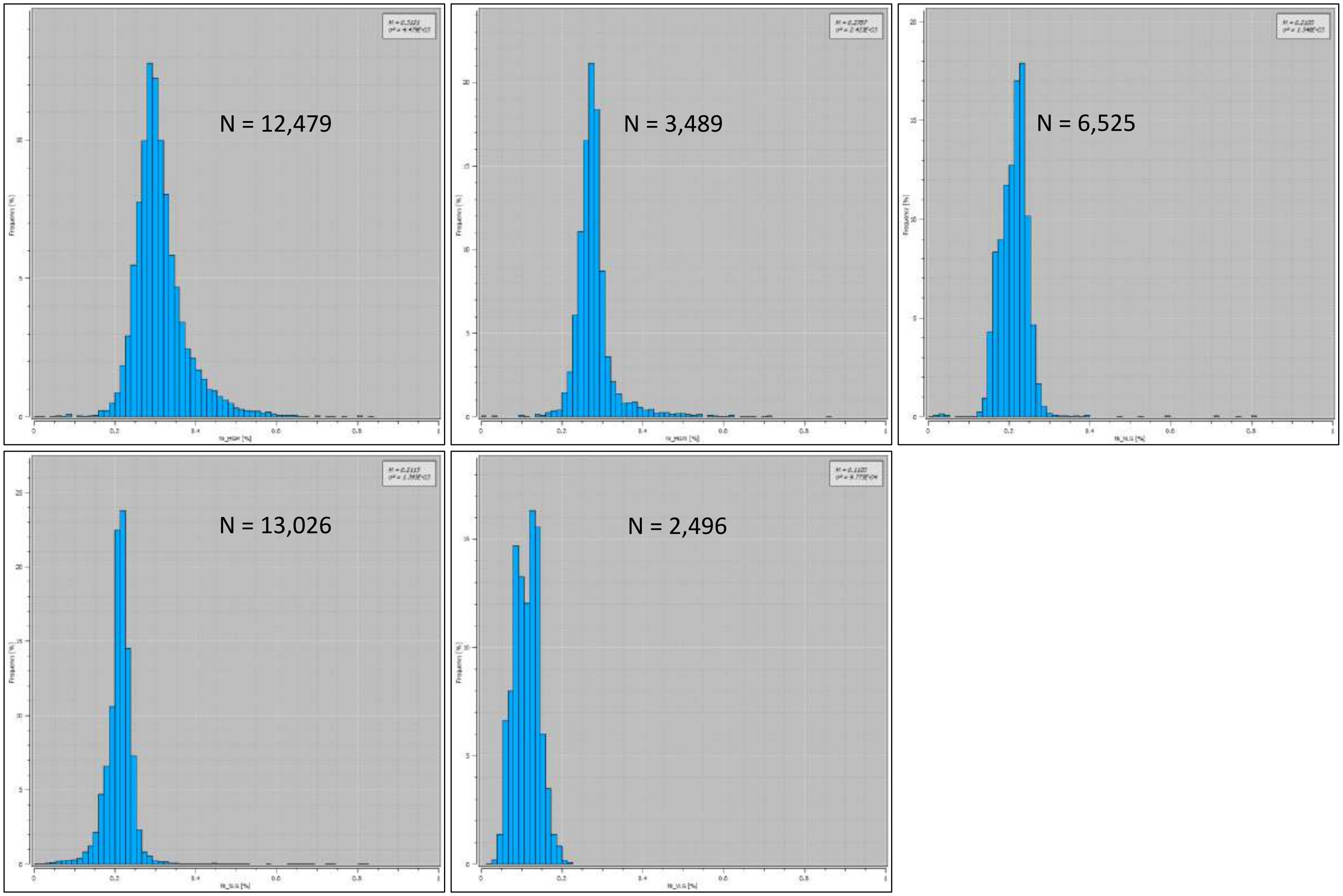
**Figure 14-23: Main-West Zone Plan View with Chromium Grade, Drillhole Traces, and Estimation Domains**



Note: Shows LCR (dark green), SHCR (pink), NHCR (red) and VLCR (light blue) estimation domains. Source: Caracle Creek, 2023.

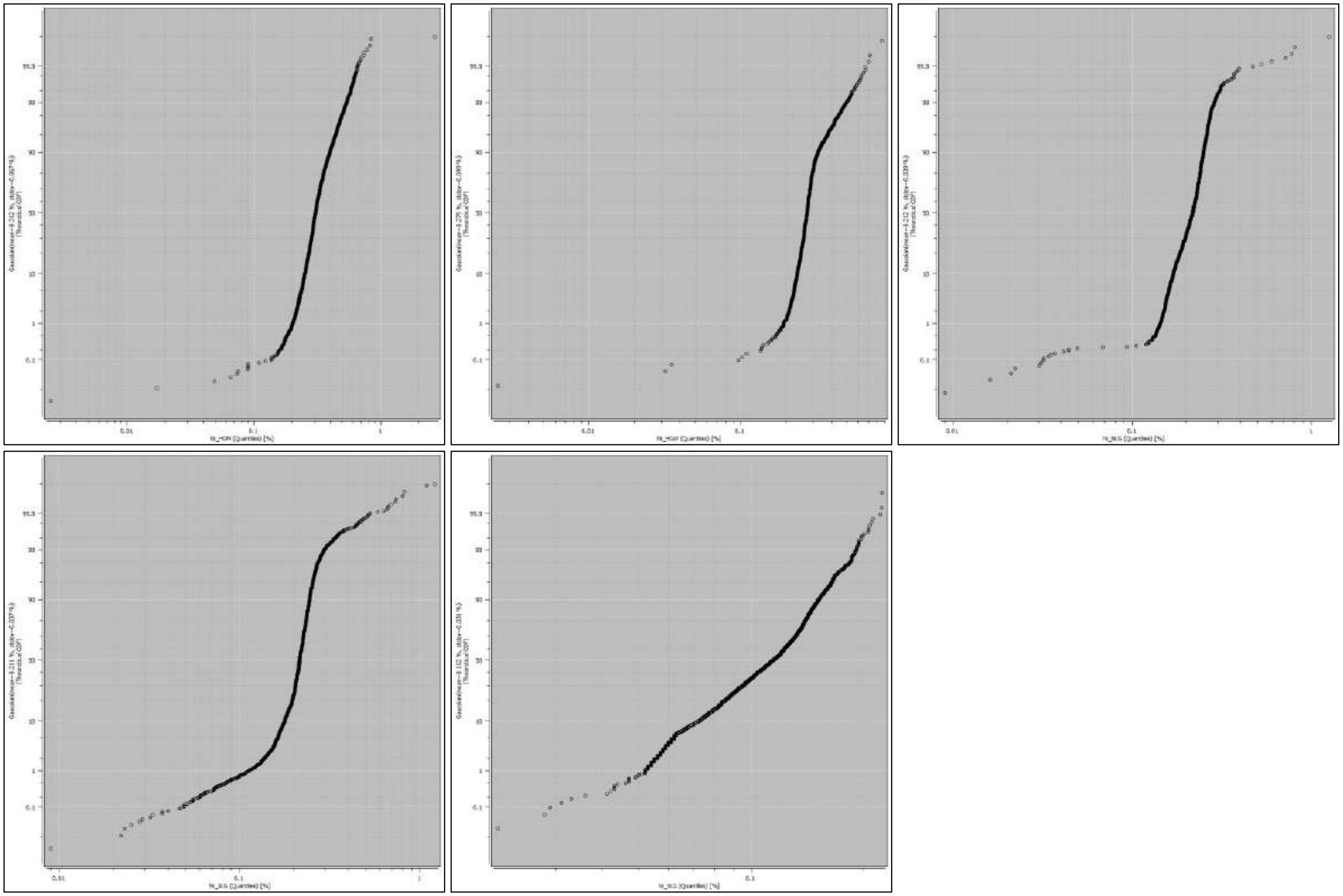
Statistical analysis within the set of estimation domains for each element confirmed the appropriate separation of grade populations, evidenced by adequate distributions and statistical parameters (Figures 14-24 and 14-25 on the following pages). Further analyses for the other estimation variables (see Section 14.7.3) suggest no need for additional domaining.

**Figure 14-24: Main-West Zone Nickel Grade Histograms with Sample Amounts for the HGM, HGW, NLG, SLG and VLG Estimation Domains**




Source: Caracle Creek, 2023.

**Figure 14-25: Main-West Zone Nickel Grade Probability Plots for the HGM, HGW, NLG, SLG and VLG Estimation Domains**



Source: Caracle Creek, 2023.

### 14.5.6 Main-West Zone: Compositing and Capping

Considering that 99% of drillhole samples are 1.5 m in length, the 15.0 m height of the blocks (see Section 14.6, Block Modelling) and the sample database size, composites of 7.5 m were deemed the most appropriate. These were generated within the general EST domain (meaning that estimation domain boundaries were not considered) for the seven studied elements (Ni, Co, Fe, Cr, S, Pd, Pt). The density database consists of 7,906 data points and the brucite (mineralogy) database consists of 1,528 samples for the serpentine-altered portion of the EST domain. Given that both density and brucite data are essentially punctual, as opposed to continuous intervals, they cannot be composited.

Capping was applied before compositing and only for "true" outliers (values out of context such as a single high grade among low-grade values). Capping values were mainly based on histogram and probability plot distributions, which yielded top caps of 0.75% for nickel, 0.03% for cobalt, 12.0% for iron, 1.2% for chromium, 0.5% for sulphur and none for palladium, platinum, brucite, or density. The latter, however, did receive a bottom cap of 2.3 g/cm$^3$. Anomalous zones that transition into very high values were not capped at this stage; instead, their influence was limited to a set distance during estimation if deemed necessary. The resulting capped composites show more than adequate distributions and statistical parameters for most elements to carry out OK estimation within the EST domain and its subdomains, with S, Pd and Pt presenting slight complexities due to their somewhat high CV values (Table 14-3).

**Table 14-3: Main-West Zone Capping Values and Summary Statistics of Declustered Samples and Composites by Element and their Selected Estimation Domain**

| Domain | Element | 1.5 m Drillhole Samples | | | | | | 7.5 m Composites (Except Density) | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Count | Mean | Std. Dev. | CV | Med | | Count | Mean | Std. Dev. | CV | Med |
| HGM | Ni % | 12,479 | **0.31** | 0.07 | 0.21 | 0.30 | | 2,496 | **0.31** | 0.05 | 0.17 | 0.31 |
| | S % | 12,479 | **0.18** | 0.22 | 1.23 | 0.10 | | 2,496 | **0.18** | 0.22 | 1.20 | 0.10 |
| HGW | Ni % | 3,489 | **0.28** | 0.05 | 0.19 | 0.27 | | 698 | **0.28** | 0.04 | 0.14 | 0.27 |
| | S % | 3,489 | **0.06** | 0.05 | 0.92 | 0.04 | | 698 | **0.06** | 0.04 | 0.78 | 0.05 |
| SLG | Ni % | 13,026 | **0.21** | 0.04 | 0.17 | 0.22 | | 2,581 | **0.21** | 0.03 | 0.13 | 0.22 |
| | S % | 13,026 | **0.04** | 0.04 | 1.25 | 0.02 | | 2,581 | **0.04** | 0.04 | 1.11 | 0.02 |
| NLG | Ni % | 6,525 | **0.21** | 0.04 | 0.17 | 0.22 | | 1,306 | **0.21** | 0.03 | 0.14 | 0.22 |
| VLG | Ni % | 2,496 | **0.11** | 0.03 | 0.28 | 0.11 | | 500 | **0.11** | 0.03 | 0.27 | 0.11 |
| NLG+VLG | S % | 9,021 | **0.05** | 0.04 | 0.99 | 0.03 | | 1,806 | **0.05** | 0.04 | 0.93 | 0.03 |
| LFE | Fe % | 10,387 | **5.66** | 0.86 | 0.15 | 5.74 | | 2,076 | **5.66** | 0.76 | 0.13 | 5.78 |
| SHFE | | 21,950 | **7.28** | 0.61 | 0.08 | 7.28 | | 4,349 | **7.28** | 0.45 | 0.06 | 7.28 |
| NHFE | | 5,678 | **7.09** | 0.82 | 0.12 | 7.09 | | 1,121 | **7.10** | 0.62 | 0.09 | 7.11 |
| LCR | Cr % | 5,792 | **0.39** | 0.12 | 0.31 | 0.37 | | 1,129 | **0.38** | 0.10 | 0.25 | 0.37 |
| SHCR | | 14,793 | **0.62** | 0.10 | 0.16 | 0.62 | | 2,948 | **0.62** | 0.09 | 0.14 | 0.62 |
| NHCR | | 13,590 | **0.66** | 0.11 | 0.16 | 0.67 | | 2,734 | **0.66** | 0.09 | 0.14 | 0.67 |
| VLCR | | 3,840 | **0.42** | 0.10 | 0.24 | 0.41 | | 770 | **0.42** | 0.07 | 0.17 | 0.42 |
| EST | Co % | 38,015 | **0.013** | 0.002 | 0.145 | 0.013 | | 7,581 | **0.013** | 0.002 | 0.120 | 0.013 |
| | Pd ppm | 38,015 | **0.019** | 0.079 | 4.146 | 0.003 | | 7,565 | **0.016** | 0.039 | 2.487 | 0.005 |
| | Pt ppm | 38,015 | **0.015** | 0.074 | 5.107 | 0.005 | | 7,565 | **0.011** | 0.018 | 1.718 | 0.006 |
| | Density | 7,906 | **2.68** | 0.12 | 0.05 | 2.67 | | | | | | |
| | Brucite | 1,528 | **1.25** | 1.26 | 1.01 | 0.88 | | | | | | |

Note: Composites for PGE elements (Pd, Pt) have a lower count due to the exclusion of very high grades located at the northern contact with pyroxenite, corresponding to a high-grade structure or "PGE reef" that will be evaluated in a future estimate. Density and brucite count values only consider samples within the serpentine-altered portion of each domain. Source: Caracle Creek, 2023.

### 14.6 Block Modelling

The selection of a block model size was based on mining recovery and dilution tests, as well as economic analyses, arriving to a 20 m x 20 m x 15 m dimension as the more optimal choice for both the Main-West and East zones, with a possibility of reducing it to 10 m x 10 m x 7.5 m in the future. To avoid centroid mismatches between both models in overlapping regions, they were derived from a 40 m x 40 m x 15 m parent block model covering the complete modelling area (see Section 14.4.2), and as result overlapping blocks share the same ID number.

The area covered by these block models had to be sufficiently large to host a theoretical open pit, therefore opting for an extension of 1 km beyond the estimation domain limits of each zone in all horizontal directions (Table 14-4). Vertical constraints came from the topographic surface at the top, and from the modelling depth at the bottom (-600 m elevation). Sub-blocks were not defined, instead a column of fill percentage was generated for each geological model that was flagged into the block model, for tonnage calculations.

**Table 14-4: Block Model Properties**

| East Zone Block Model | X | Y | Z | Main-West Zone Block Model | X | Y | Z |
|---|---|---|---|---|---|---|---|
| Minimum Centroid Coordinates | 472,010 | 5,408,530 | -592.5 | Minimum Centroid Coordinates | 470,010 | 5,407,450 | -592.5 |
| Box Extents | 5,000 | 2,680 | 885 | Box Extents | 5,360 | 4,260 | 885 |
| Block Size | 20 | 20 | 15 | Block Size | 20 | 20 | 15 |
| Number of Blocks | 250 | 134 | 59 | Number of Blocks | 268 | 213 | 59 |
| Rotation | - | - | - | Rotation | - | - | - |

Source: Caracle Creek, 2023.

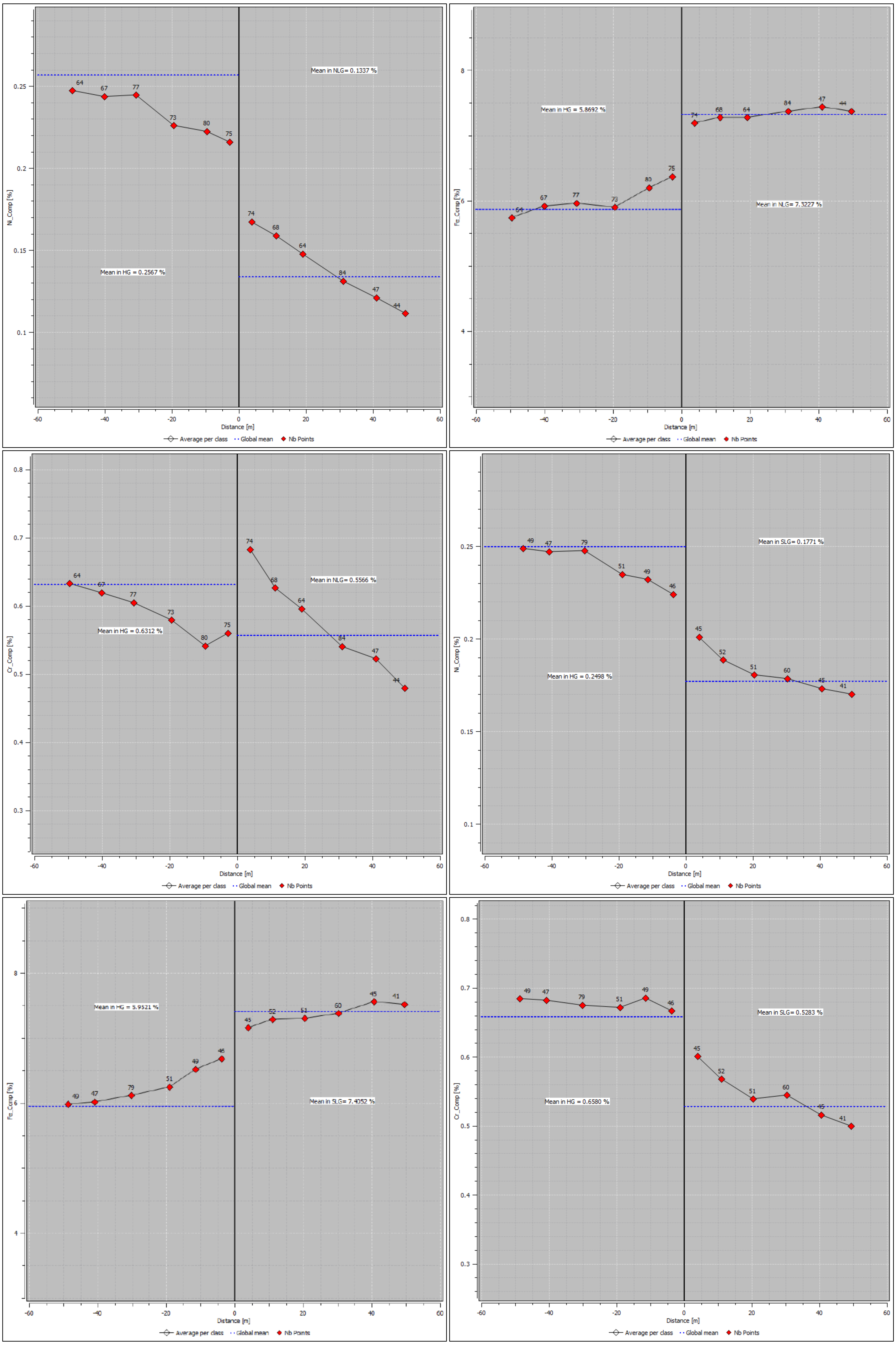
### 14.7 Estimation Strategy

#### 14.7.1 East Zone: Estimation Methodology

Due to noticeable grade differences between the HG, NLG and SLG domains (Figure 14-26), composites for nickel, iron and chromium were filtered for OK estimation within each one. Density and brucite values were also considered for OK estimation within the predominant, MgSerp-altered portion of these subdomains, owing to their good correlation with nickel, but limited to IDW estimation within the scarcely sampled talc-carbonate domain. Other elements such as cobalt, sulphur, palladium and platinum did not show sufficiently relevant grade variations between subdomains and were considered for OK estimation within the general EST-1 domain.

Contacts between estimation domains are transitional (semi-soft boundaries), making it desirable to include composites of adjacent domains up to a certain distance to better represent this transition. The selected distance was 10 m for all variables within each subdomain. The only exception was brucite, for which hard contacts were deemed more appropriate due to sharper transitions, meaning no composites were shared between domains.

**Figure 14-26: East Zone Contact Analysis Plots of Nickel (Left Column), Iron (Middle Column) and Chromium (Right Column) for the HG/NLG (Top) and HG/SLG (Bottom) Domain Transitions**




Source: Caracle Creek, 2023.

As previously stated, the EST-2 domain contains only a few composites, which is not ideal for an OK estimation approach, instead opting for IDW estimation.

### 14.7.2 East Zone: Estimation Parameters

Neighbourhood search radii were based on set ranges for all elements, mostly based on deposit geometry rather than variography, while interpolation parameters were replicated (Table 14-5). Blocks were discretized to a 2 x 4 x 4 ratio. A three-pass strategy was implemented, each with successively larger neighbourhood search radii and more relaxed parameters. A fourth pass with an “infinite” range was considered for any remaining block that did not meet previous criteria.

**Table 14-5: East Zone General Neighbourhood Search and Estimation Parameters for all Estimation Domains**

| Parameter | Neighbourhood | | | |
|---|---|---|---|---|
| Pass | 1st | 2nd | 3rd | 4th |
| Sector Search | Single | Single | Single | Single |
| Minimum Sectors | NO | | | |
| Maximum Points per Sector | 20 | 20 | 20 | 20 |
| Minimum Total Points | 8 | 4 | 2 | 1 |
| Maximum Points per Drillhole | 4 | 4 | 4 | 4 |
| Minimum Points per Drillhole | - | - | - | - |
| Minimum Drillholes | 2 | 1 | 1 | 1 |
| Search Radius Directions | 90-92° Az / 80° Dip / 280° Pitch | | | |
| Search Radius Axis 1 | 200 | 2x | 4x | ∞ |
| Search Radius Axis 2 | 300 | 2x | 3x | ∞ |
| Search Radius Axis 3 | 50 | 2x | 4x | ∞ |

Source: Caracle Creek, 2023.

### 14.7.3 Main-West Zone: Estimation Methodology

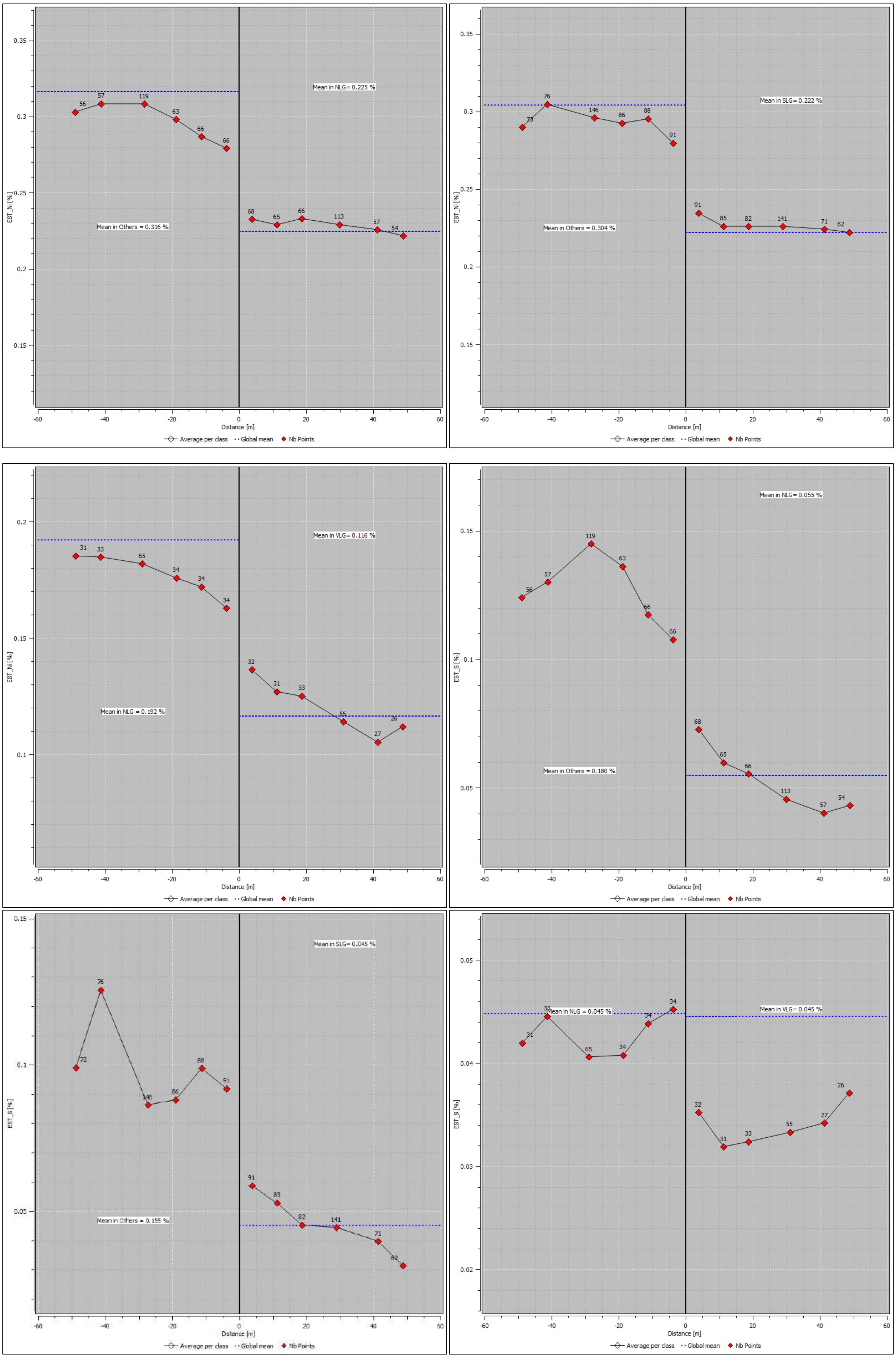
The use of grade shells to separate populations of nickel, iron and chromium guaranteed that estimation domains would display noticeable grade differences between them (Figures 14-27, 14-28 and 14-29), justifying composite filtering for separate OK estimation within each one. Additionally, a moderate correlation between sulphur and nickel grades with an acceptable visual match of their grade populations provided sufficient support for separate sulphur OK estimation within the nickel subdomains, except for the VLG domain, which showed no transition or grade difference to the NLG domain (Figure 14-27), meaning that they could be treated as a single domain.

Other elements such as cobalt, palladium and platinum did not show relevant grade variations between subdomains and were considered for OK estimation within the general EST domain. Palladium also showed a slight bimodal distribution that could be resolved by estimating within the nickel subdomains, but this was disregarded to maintain consistency with platinum, which did not display this bimodality and is otherwise highly correlated to palladium. Density values were considered for OK estimation within their corresponding alteration domains MgSerp and

Weathered/Unweathered FeSerp (see Section 14.4.3), but not the scarcely sampled talc-carbonate domain, where they were limited to IDW estimation. Lastly, brucite values were considered for OK estimation within the serpentine-altered portions of the PER-0, PER-1, and DUN lithological domains (resource domains do not separate mineralogical populations appropriately in this instance), with the talc-carbonate domain limited to IDW estimation.

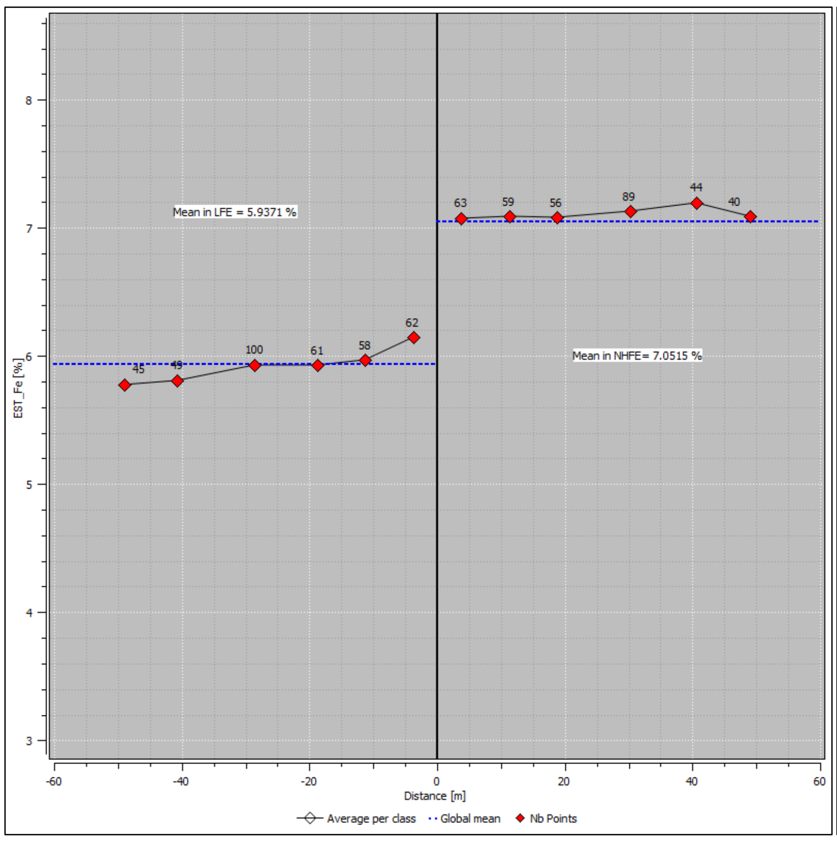
Contacts between estimation domains are transitional (semi-soft boundaries), making it desirable to include composites of adjacent domains up to a certain distance to better represent this transition. The selected distance was 10 m for all variables within each subdomain, except for chromium within the LCR domain and density within the FeSerp Weathered domain, which only included adjacent composites up to 5 m away due to their narrowness. For brucite, hard contacts were deemed more appropriate due to sharper transitions, meaning no composites were shared between domains.

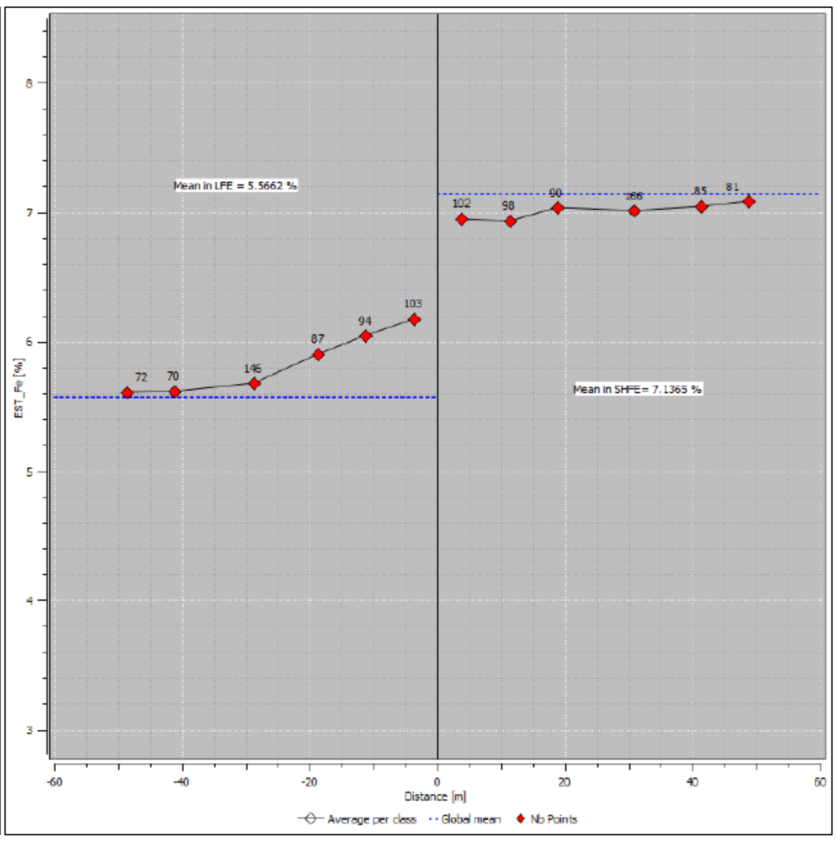
**Figure 14-27: Main-West Zone Contact Analysis Plots of Nickel (Top Row), and Sulphur (Bottom Row) for the HG/NLG (Left Column), HG/SLG (Middle Column) and NLG/VLG (Right Column) Domain Transitions**




Source: Caracle Creek, 2023.

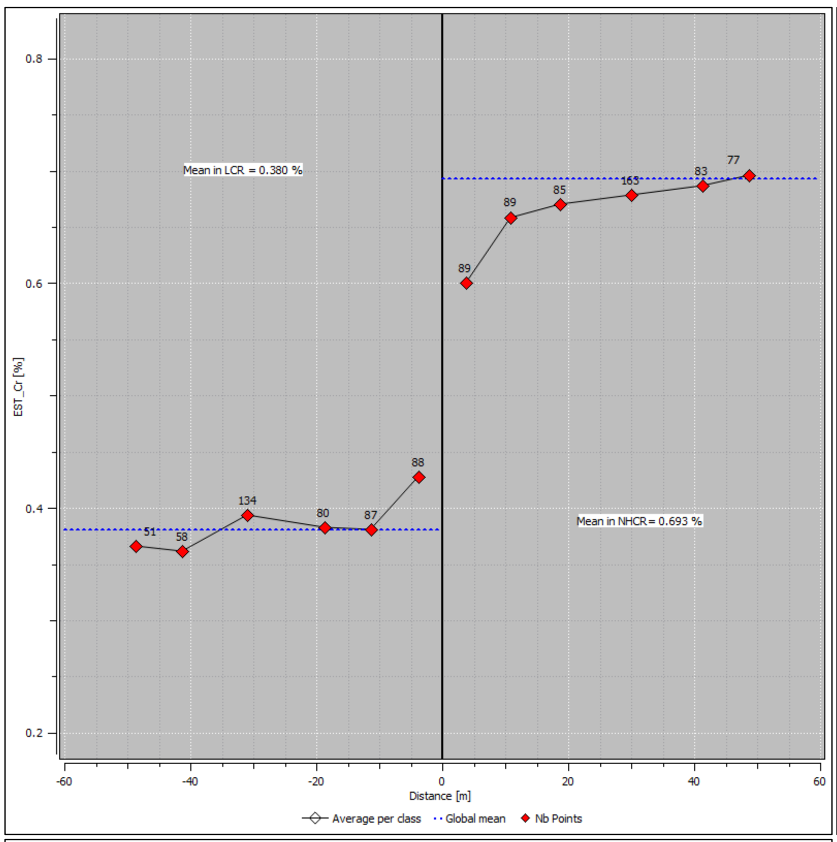
**Figure 14-28: Main-West Zone Contact Analysis Plots of Iron for the LFE/NHFE (Left) and LFE/SHFE (Right) Domain Transitions**

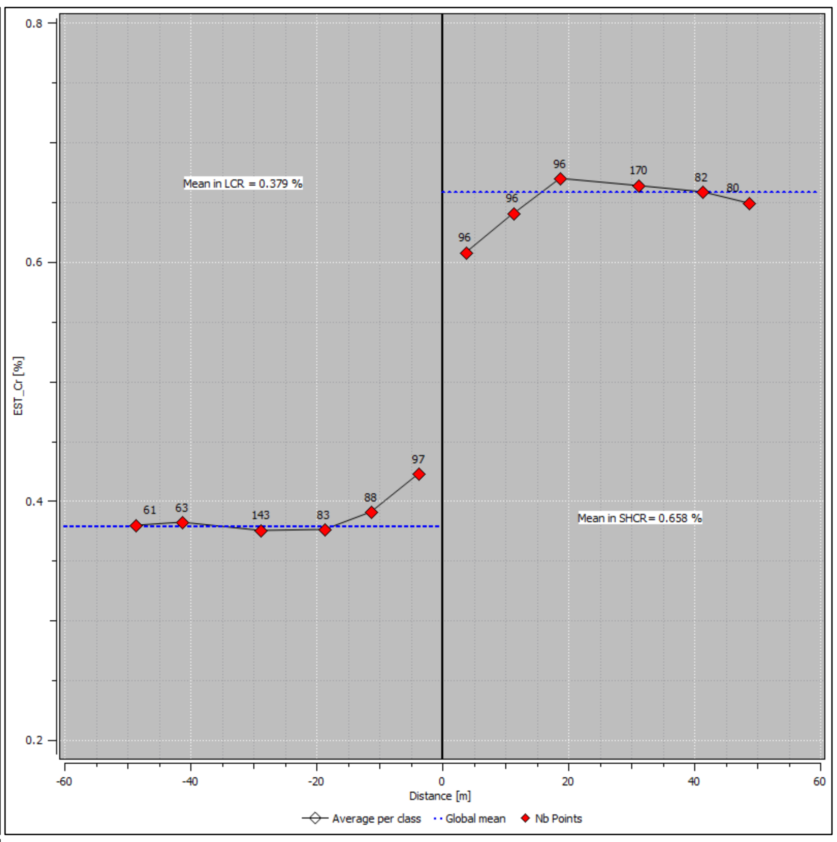


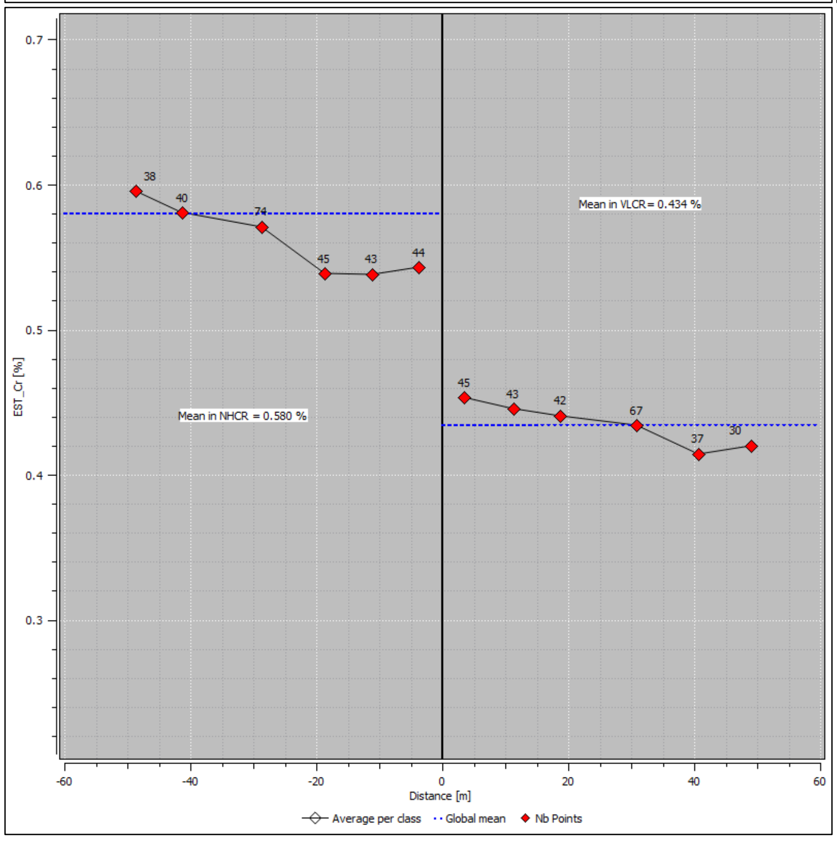




Source: Caracle Creek, 2023.

**Figure 14-29: Main-West Zone Contact Analysis Plots of Chromium for the LCR/NHCR (Left), LCR/SHCR (Middle) and NHCR/VLCR (Right) Domain Transitions**




Source: Caracle Creek, 2023.

### 14.7.4 Main-West Zone: Estimation Parameters

Neighbourhood search radii were based on variogram ranges for each element and domain, while interpolation parameters were replicated (Table 14-6). Given the curved nature of this deposit, with a mineralization trend that starts in an approximate east-west direction in the Main Zone and progressively changes to an almost north-south direction in the West Zone, it was deemed necessary to assign local anisotropies for estimation within each block. These set of directions were derived from the general trends of the contact surfaces for the EST domain and its subdomains.

**Table 14-6: Main-West Zone Neighbourhood Search and Estimation Parameters for all Estimation Domains**

| Parameter | Neighbourhood | | | |
|---|---|---|---|---|
| Pass | 1st | 2nd | 3rd | 4th |
| Sector Search | Single | Single | Single | Single |
| Minimum Sectors | NO | | | |
| Maximum Points per Sector | 20 | 20 | 20 | 20 |
| Minimum Total Points | 8 | 4 | 2 | 1 |
| Maximum Points per Drillhole | 4 | 4 | 4 | 4 |
| Minimum Points per Drillhole | - | - | - | - |
| Minimum Drillholes | 2 | 1 | 1 | 1 |
| Search Radius Directions | Local Anisotropies | | | |
| Search Radius Axis 1 (range) | 150-300 | 2x | 4x | ∞ |
| Search Radius Axis 2 (range) | 100-250 | 2x | 4x | ∞ |
| Search Radius Axis 3 (range) | 50-150 | 2x | 4x | ∞ |

Source: Caracle Creek, 2023.

Blocks were discretized to a 2 x 4 x 4 ratio. A three-pass strategy was implemented, each with successively larger neighbourhood search radii and more relaxed parameters. A fourth pass with an “infinite” range was considered for any remaining block that did not meet previous criteria.

## 14.8 Variography

### 14.8.1 East Zone

Variography was carried out for the seven studied elements, density, and brucite within the EST-1 domain or its subdomains, depending on the spatial distribution and variability of their grades. The EST-2 domain did not require variography. A comprehensive list of variables and their estimation domains is presented next:

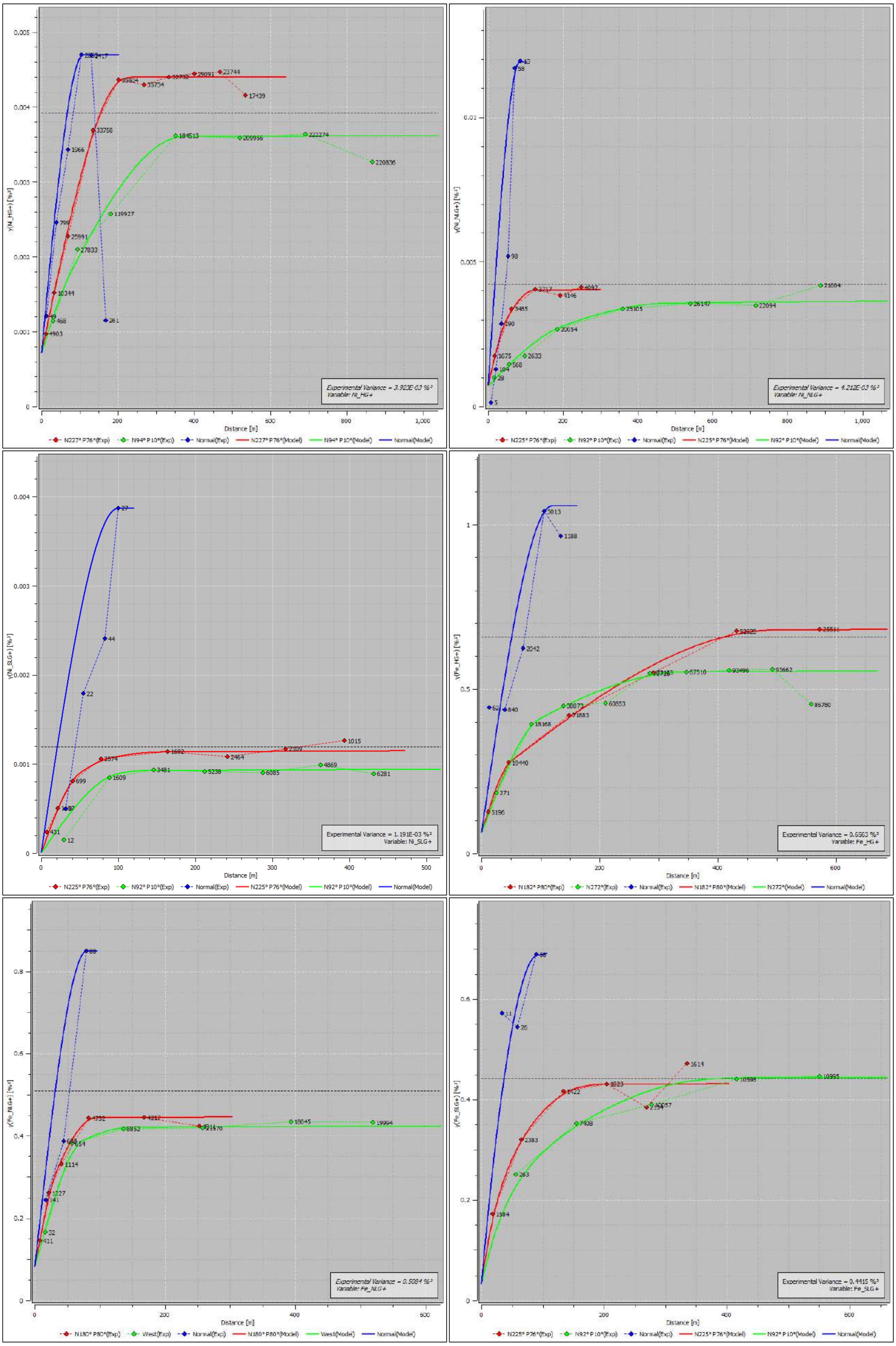
- Nickel: HG, NLG and SLG domains (Figure 14-30). Additional nickel variogram within the general EST-1 domain for resource classification (Figure 14-44).
- Iron: HG, NLG and SLG domains (Figure 14-30).

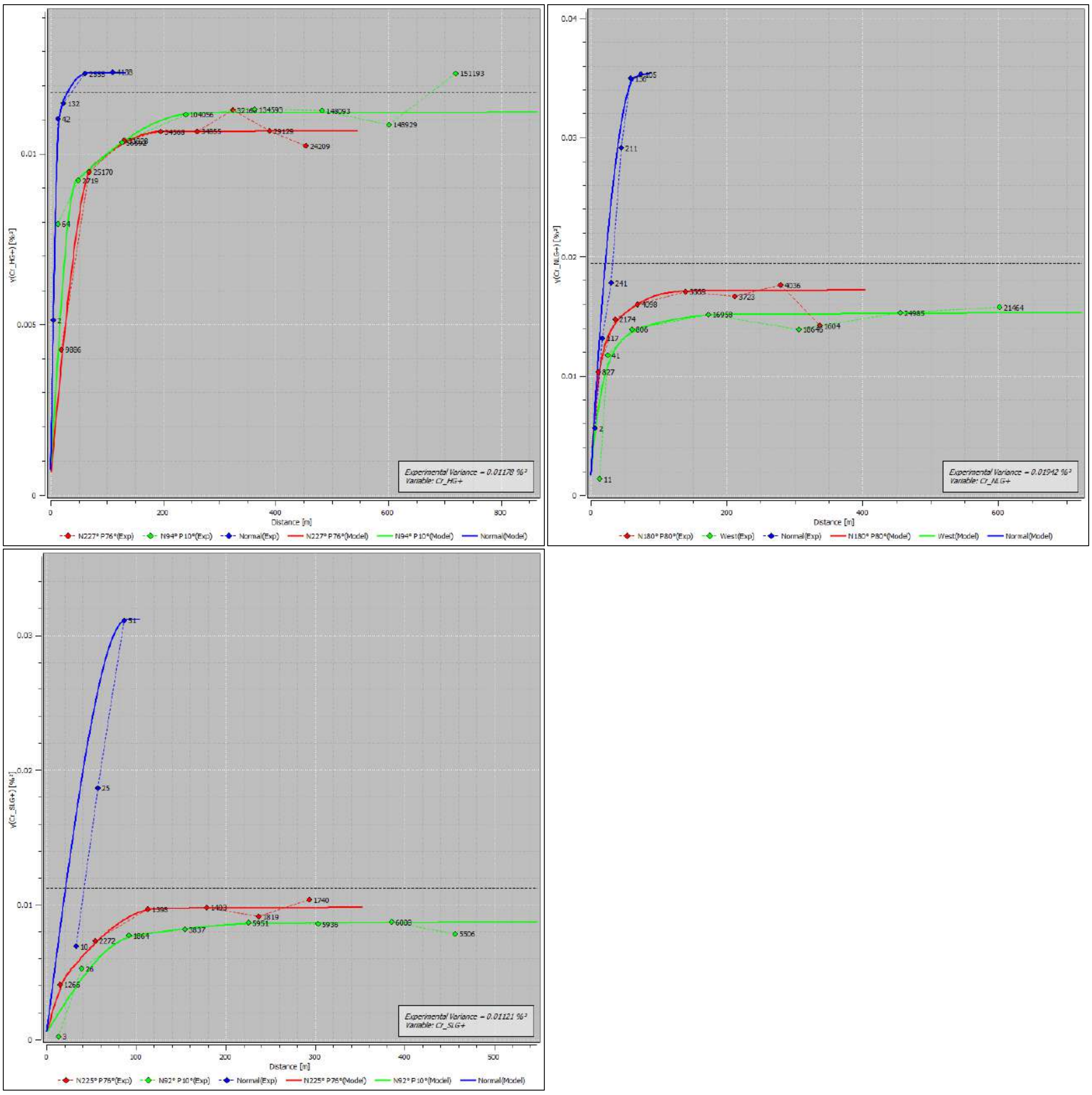
- Chromium: HG, NLG and SLG domains (Figure 14-30).
- Cobalt: EST-1 domain (Figure 14-31).
- Sulphur: EST-1 domain (Figure 14-31).
- Palladium: EST-1 domain.
- Platinum: EST-1 domain.
- Density: MgSerp-altered portion of the HG, NLG and SLG domains. The talc-carbonate domain did not require variography.
- Brucite: MgSerp-altered portion of the HG, NLG and SLG domains. The talc-carbonate domain did not require variography.

Variogram directions were based on the geometry of the deposit, mineralization trends and drilling directions, with variogram maps for each element serving as further reference. The result was a preferential direction of 90-92 azimuth and 80°S dip.

Multidirectional variograms were modelled using declustering weights and considering zonal anisotropies due to the significant variability differences between directions. Some notable grade outliers were excluded in a few instances to reduce noise. Down-the-hole variograms were also modelled in each case for an initial approach to the nugget value. Cross-validation was carried out for evaluation of variogram robustness and recalibration of estimation parameters to improve validation results.

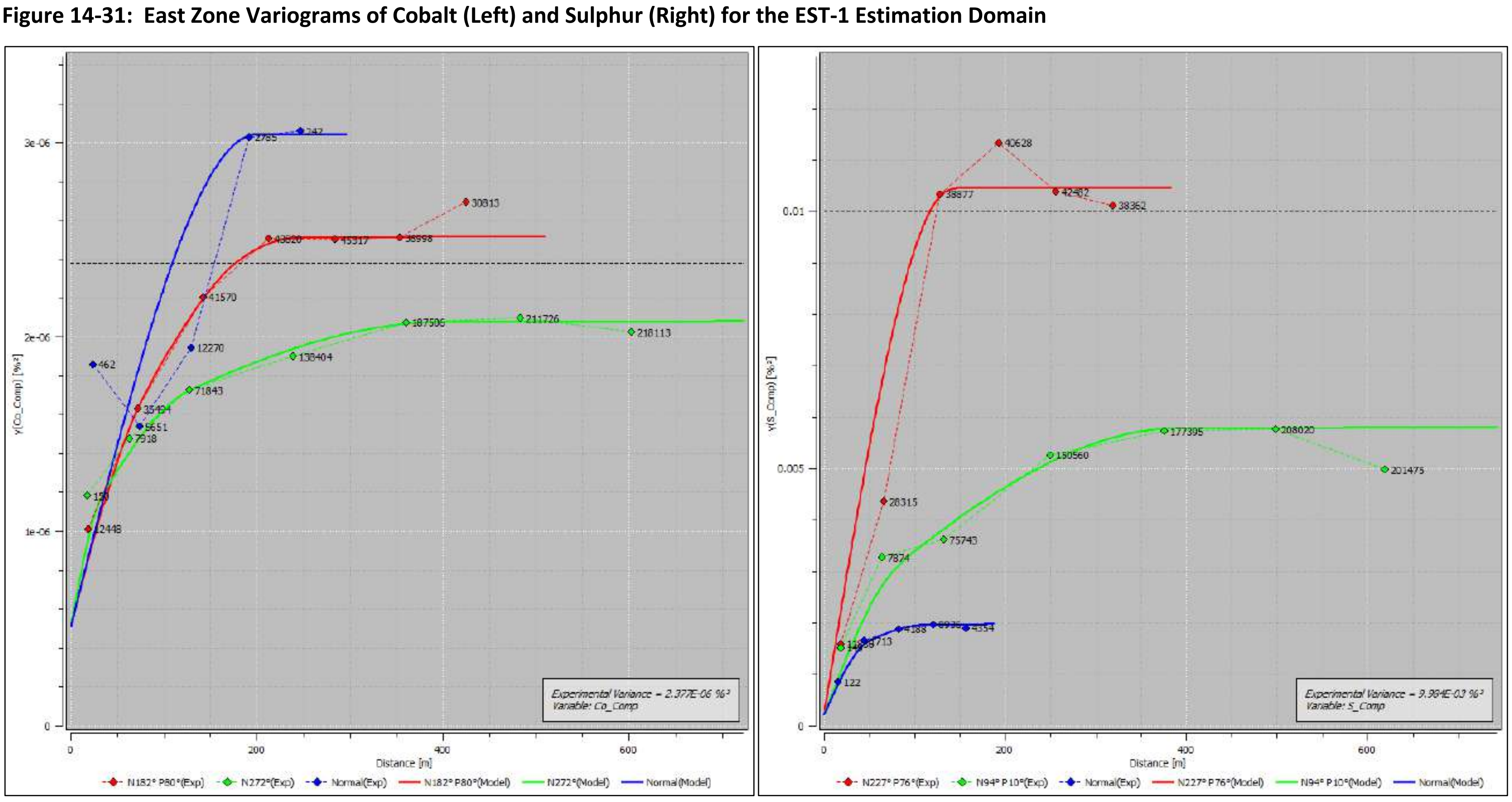
**Figure 14-30: East Zone Variograms of Nickel (Top Row), Iron (Middle Row) and Chromium (Bottom Row) for the HG (Left Column), NLG (Middle Column) and SLG (Right Column) Estimation Domains**






Source: Caracle Creek, 2023.

**Figure 14-31: East Zone Variograms of Cobalt (Left) and Sulphur (Right) for the EST-1 Estimation Domain**




Source: Caracle Creek, 2023.

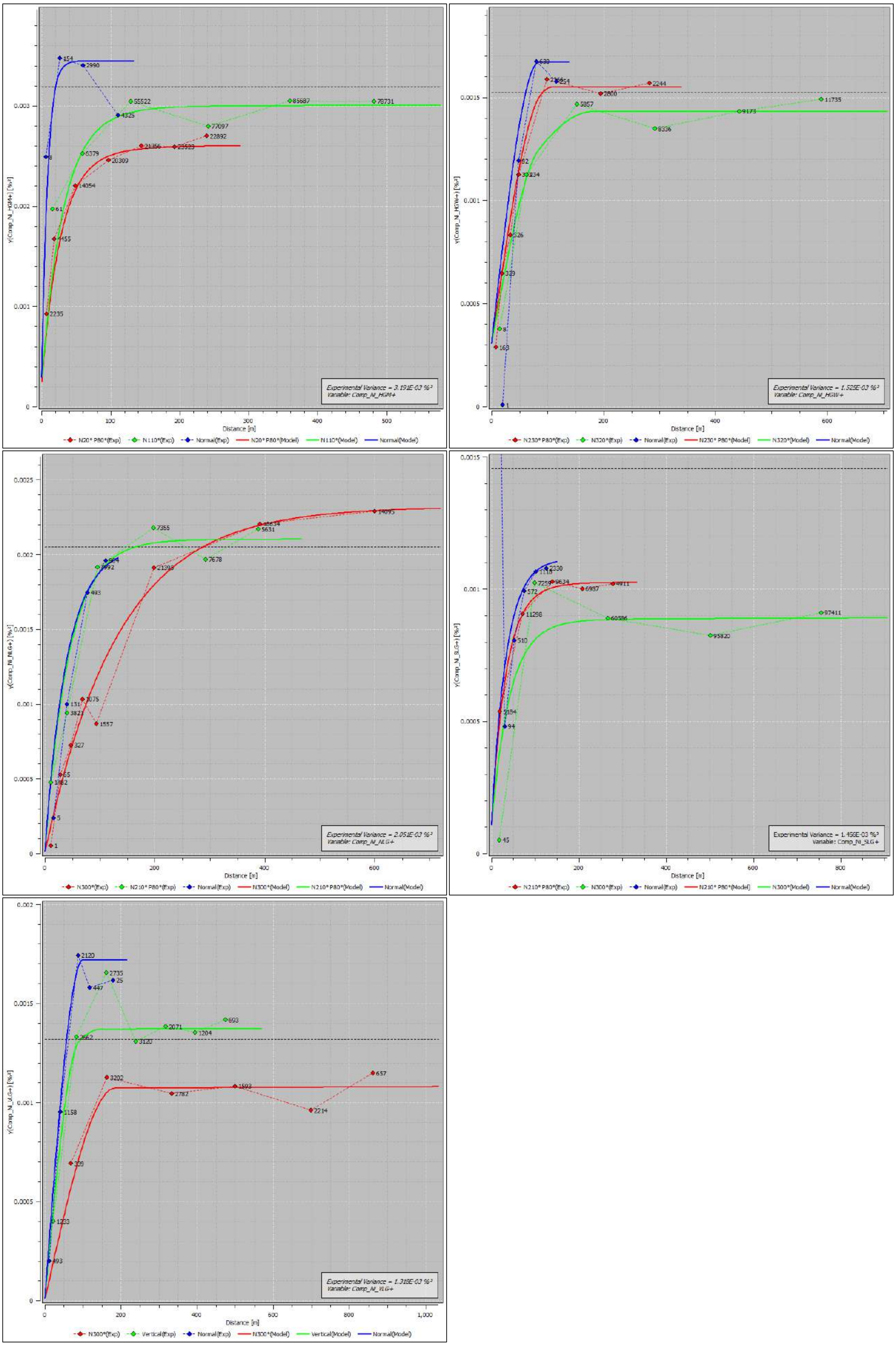
### 14.8.2 Main-West Zone

Variography was carried out for the seven studied elements and density within the EST domain or its subdomains, depending on the spatial distribution and variability of their grades. A comprehensive list of variables and their estimation domains is presented next:

- Nickel: HG, NLG, SLG and VLG domains (Figure 14-32). Additional nickel variogram within the general EST domain for resource classification (Figure 14-46).
- Cobalt: EST domain.
- Sulphur: HG, NLG, SLG+VLG (combined) domains (Figure 14-33).
- Iron: LFE, NHFE and SHFE domains (Figure 14-34).
- Chromium: LCR, NHCR, SHCR and VLCR domains (Figure 14-35).
- Palladium: EST domain.
- Platinum: EST domain.
- Density: MgSerp, FeSerp Unweathered and FeSerp Weathered alteration domains (see Section 14.4.3). The talc-carbonate domain did not require variography.
- Brucite: MgSerp-altered portion of the PER-0 and PER-1 lithological domains, MgSerp- and FeSerp-altered portions of the DUN lithological domain. The talc-carbonate domain did not require variography.

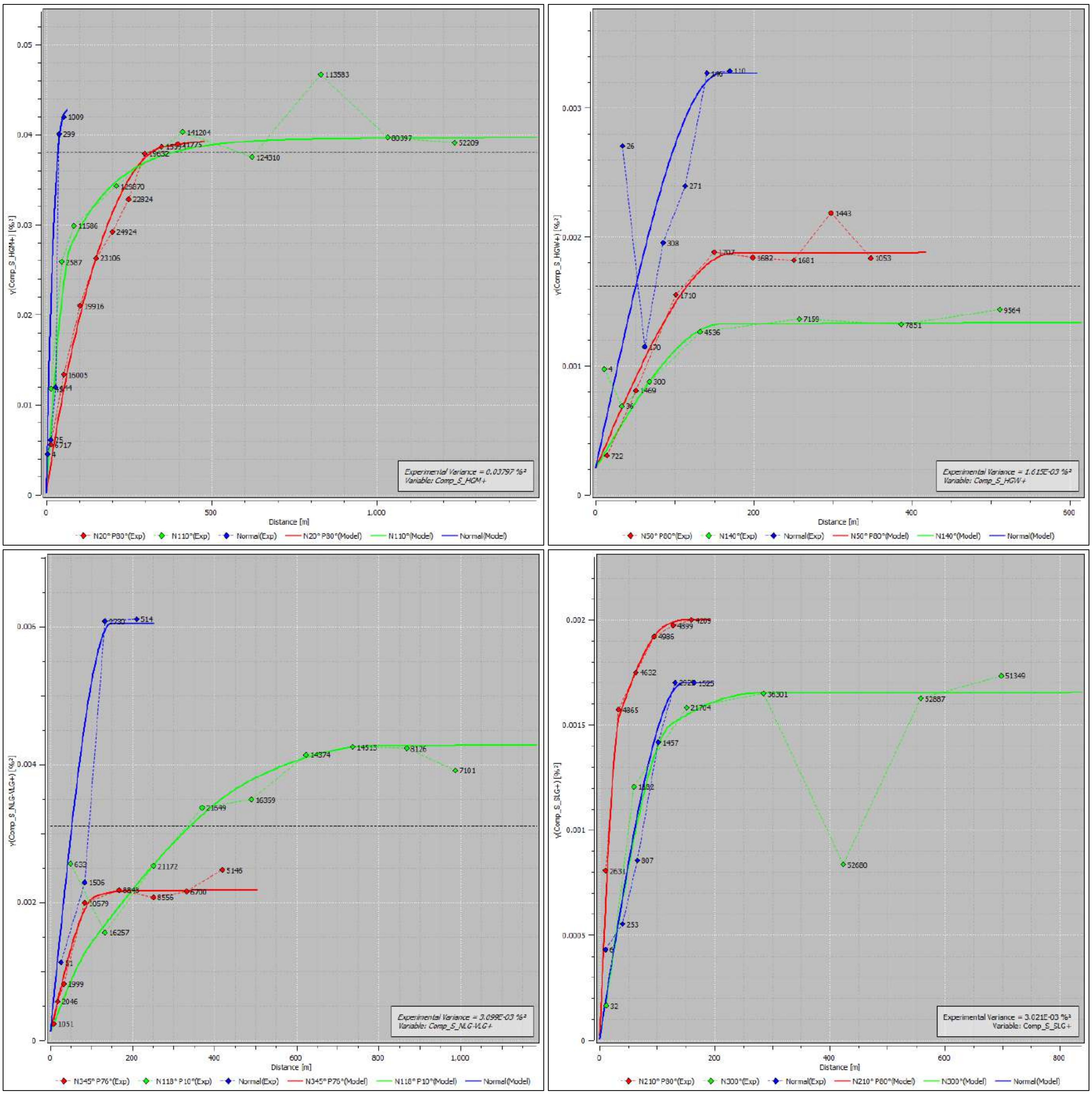
Variogram directions were based on the geometry of the deposit, mineralization trends and drilling directions, with variogram maps for each element serving as further reference. The result for most domains were directions ranging between 105-125 azimuth and 90-75°N dip, except for the sub-horizontal density trend within FeSerp domains.

**Figure 14-32: Main-West Zone Variograms of Nickel for the HGM, HGW, NLG, SLG and VLG Estimation Domains**



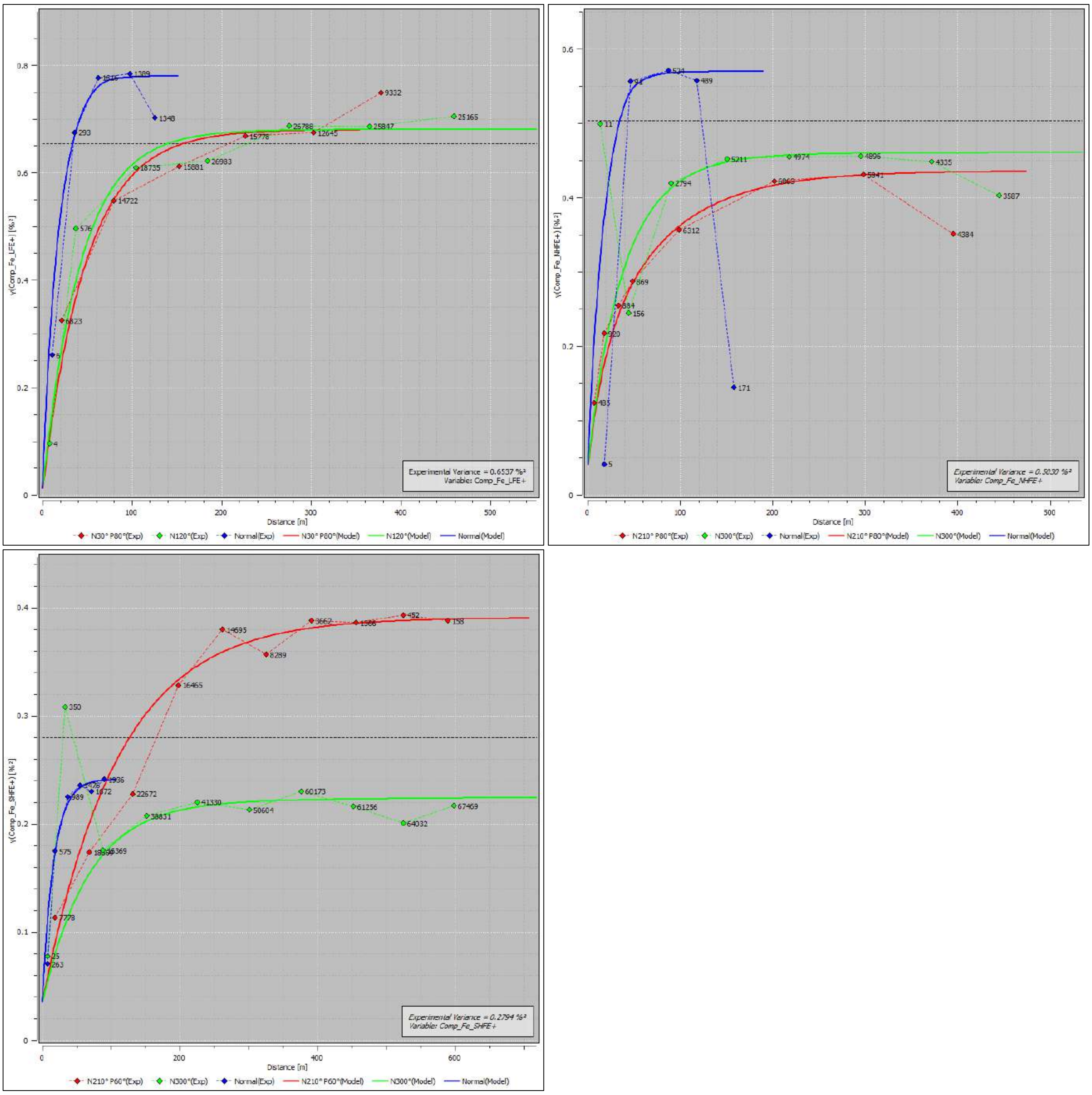

Source: Caracle Creek, 2023.

**Figure 14-33: Main-West Zone Variograms of Sulphur for the HGM, HGW, NLG+VLG and SLG Estimation Domains**



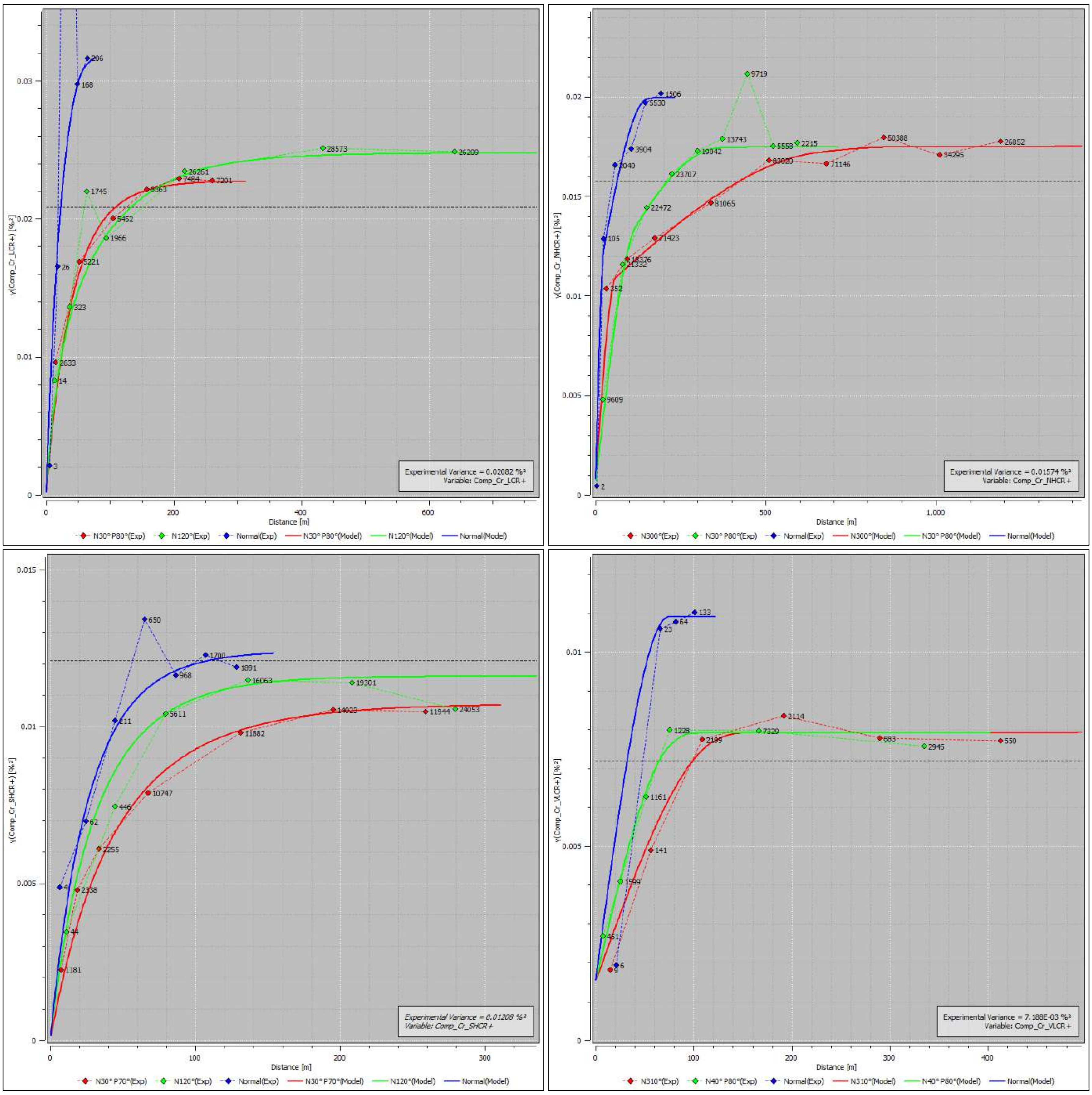

Source: Caracle Creek, 2023.

**Figure 14-34: Main-West Zone Variograms of Iron for the LFE (Left), NHFE (Middle) and SHFE (Right) Estimation Domains**




Source: Caracle Creek, 2023.

**Figure 14-35: Main-West Zone Variograms of Chromium for the LCR (Top Left), NHCR (Top Right), SHCR (Bottom Left) and VLCR (Bottom Right) Estimation Domains**




Source: Caracle Creek, 2023.

Multidirectional variograms were modelled using declustering weights and considering zonal anisotropies due to the significant variability differences between directions. Some notable grade outliers were excluded in a few instances to reduce noise. Down-the-hole variograms were also modelled in each case for an initial approach to the nugget value. Cross-validation was carried out for evaluation of variogram robustness and recalibration of estimation parameters to improve validation results.

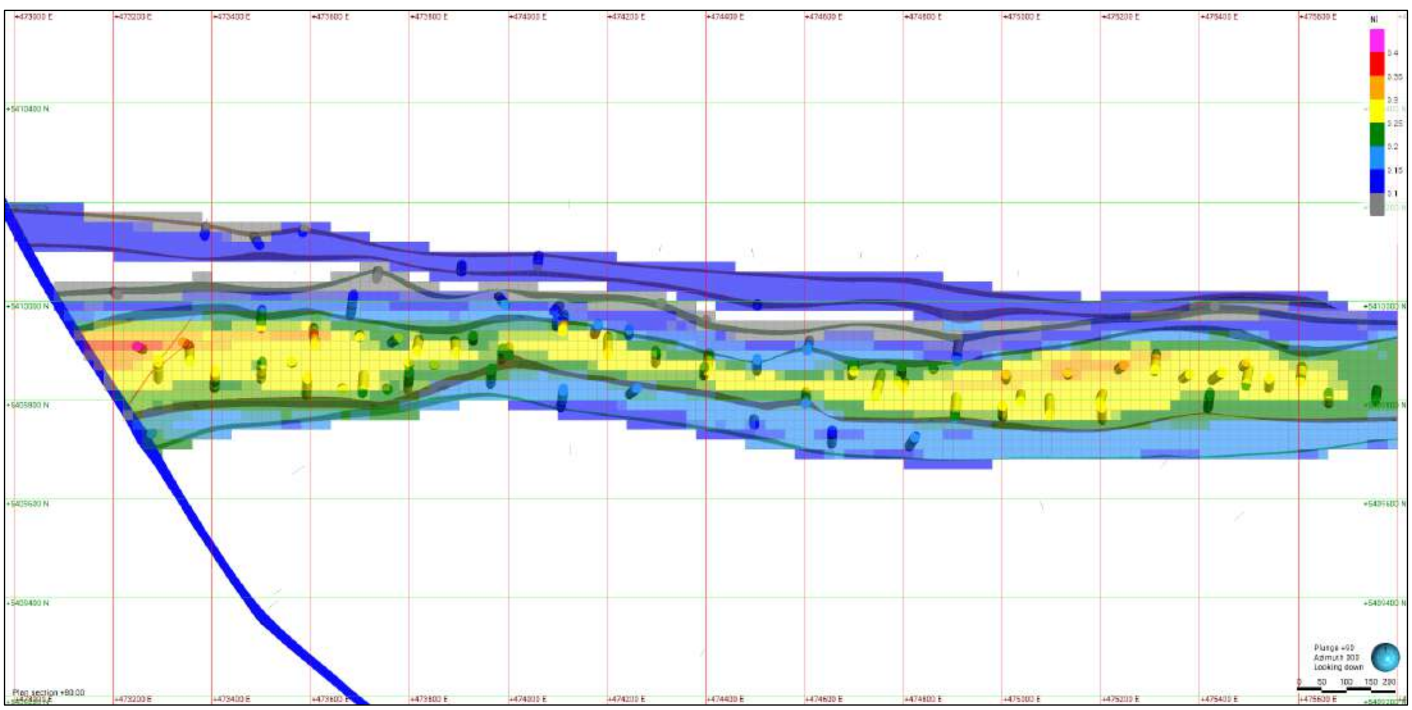
## 14.9 Block Model Validation

Estimation results were validated by three methods: (1) visual; (2) statistical; and (3) moving window mean plots (or swath plots). Validations are shown mainly for the corresponding main elements and, when possible, for other elements.

### 14.9.1 Visual Validation

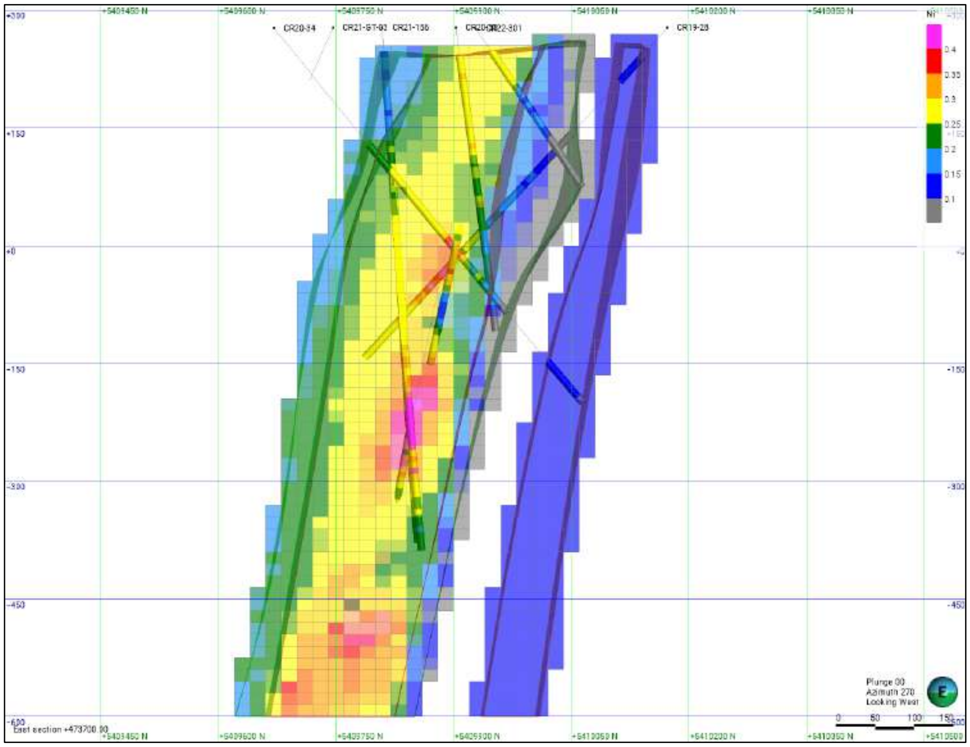
Plan views and predefined sections (Figures 14-36 through 14-39) based on drillhole direction and location were used for visual comparison of block models and composites. These show generally good consistency between estimates and composites.

**Figure 14-36: East Zone Plan Section (80 m) Comparing Block Model Grades and Nickel Composites for all Estimation Domains**



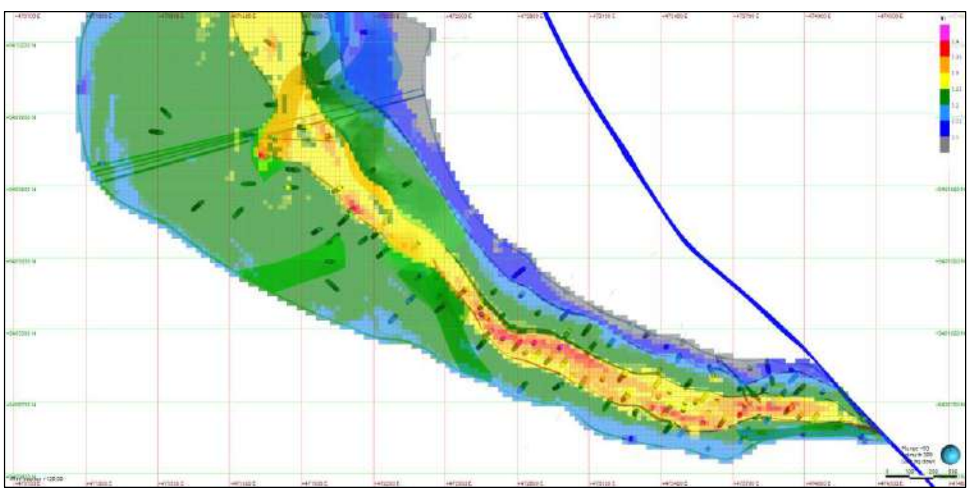
Source: Caracle Creek, 2023.

**Figure 14-37: East Zone Section (473700 mN) Looking West Comparing Block Model Grades and Nickel Composites for all Estimation Domains**



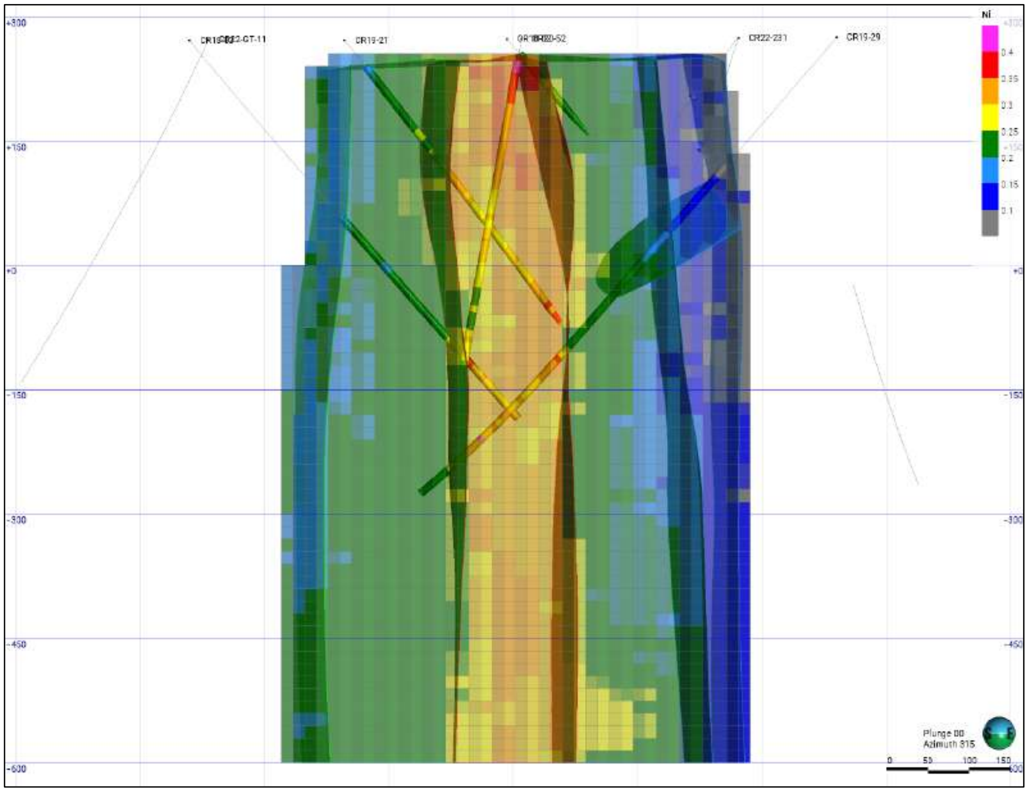

Source: Caracle Creek, 2023.

**Figure 14-38: Main-West Zone Plan Section (120 m) Comparing Block Model Grades and Nickel Composites for all Estimation Domains**



Source: Caracle Creek, 2023.

**Figure 14-39: Main-West Zone Oblique Section Looking West Comparing Block Model Grades and Nickel Composites for all Estimation Domains**




Source: Caracle Creek, 2023.

### 14.9.2 Statistical Validation

Global bias measures the percentage difference between declustered composites and estimation results (OK, IDW and NN), which preferably should not exceed 5% (Tables 14-7 and 14-8). Statistical parameters for all studied variables are also presented for comparison. It should be noted that even though values are rounded, calculations are based on non-rounded values, and that very low grades tend to produce large percentage differences.

All variables within the East Zone show generally good consistency, with Pd/Pt estimations as minor exceptions (bold red values in Table 14-7), both showing apparent underestimation. These are mostly cases of low mean values which, as previously stated, tend to produce large percentage differences for small variations, especially given the high coefficient of variance of the variables.

Similarly, most variables within the Main-West Zone show generally good consistency, with low-grade sulphur and palladium estimations as notable exceptions (bold red values in Table 14-8), all showing apparent underestimation. These results are not unexpected, given the high coefficient of variance of these variables. Both cases involve low mean values which, as previously stated, tend to produce large percentage differences for small variations, but it is also an indication, especially for palladium, that alternative estimation strategies should be investigated.

**Table 14-7: East Zone Global Statistical Comparisons Between Estimates and Declustered Composites in Resource Domains**

| Element | Domain | Data | Mean | Bias | Std Dev | CV |
|---|---|---|---|---|---|---|
| Ni % | HG | Composites | 0.25 | - | 0.004 | 0.25 |
| | | OK | 0.25 | **-0.19%** | 0.002 | 0.16 |
| | | IDW | 0.26 | **0.77%** | 0.002 | 0.17 |
| | | NN | 0.25 | **-0.53%** | 0.004 | 0.24 |
| | NLG | Composites | 0.14 | - | 0.004 | 0.47 |
| | | OK | 0.14 | **1.53%** | 0.002 | 0.33 |
| | | IDW | 0.14 | **3.24%** | 0.002 | 0.32 |
| | | NN | 0.15 | **4.90%** | 0.006 | 0.53 |
| | SLG | Composites | 0.18 | - | 0.001 | 0.20 |
| | | OK | 0.18 | **0.63%** | 0.001 | 0.14 |
| | | IDW | 0.18 | **1.28%** | 0.001 | 0.12 |
| | | NN | 0.18 | **2.29%** | 0.001 | 0.21 |
| Co % | EST-1 | Composites | 0.013 | - | 0.000002 | 0.12 |
| | | OK | 0.013 | **0.52%** | 0.000001 | 0.07 |
| | | IDW | 0.013 | **0.55%** | 0.000001 | 0.08 |
| | | NN | 0.013 | **0.35%** | 0.000003 | 0.13 |
| Fe % | HG | Composites | 5.97 | - | 0.66 | 0.14 |
| | | OK | 5.99 | **0.31%** | 0.32 | 0.09 |
| | | IDW | 5.92 | **-0.77%** | 0.30 | 0.09 |
| | | NN | 5.96 | **-0.14%** | 0.68 | 0.14 |
| | NLG | Composites | 7.09 | - | 0.51 | 0.10 |
| | | OK | 7.04 | **-0.64%** | 0.17 | 0.06 |
| | | IDW | 7.05 | **-0.54%** | 0.16 | 0.06 |
| | | NN | 6.89 | **-2.76%** | 0.70 | 0.12 |
| | SLG | Composites | 7.26 | - | 0.44 | 0.09 |
| | | OK | 7.17 | **-1.15%** | 0.17 | 0.06 |
| | | IDW | 7.17 | **-1.22%** | 0.20 | 0.06 |
| | | NN | 7.17 | **-1.13%** | 0.50 | 0.10 |
| Cr % | HG | Composites | 0.64 | - | 0.012 | 0.17 |
| | | OK | 0.64 | **-1.28%** | 0.003 | 0.08 |
| | | IDW | 0.64 | **-0.76%** | 0.004 | 0.09 |
| | | NN | 0.64 | **-1.24%** | 0.011 | 0.17 |
| | NLG | Composites | 0.52 | - | 0.019 | 0.27 |
| | | OK | 0.51 | **-3.08%** | 0.005 | 0.14 |
| | | IDW | 0.52 | **-1.34%** | 0.006 | 0.15 |
| | | NN | 0.51 | **-2.52%** | 0.017 | 0.25 |
| | SLG | Composites | 0.52 | - | 0.011 | 0.20 |
| | | OK | 0.53 | **1.90%** | 0.006 | 0.14 |
| | | IDW | 0.54 | **3.00%** | 0.005 | 0.14 |
| | | NN | 0.53 | **1.53%** | 0.011 | 0.20 |
| S % | EST | Composites | 0.06 | - | 0.010 | 1.62 |
| | | OK | 0.06 | **2.12%** | 0.004 | 1.05 |
| | | IDW | 0.06 | **-4.79%** | 0.004 | 1.03 |
| | | NN | 0.06 | **2.27%** | 0.010 | 1.57 |
| Pd ppm | EST | Composites | 0.013 | - | 0.0023 | 3.700 |
| | | OK | 0.012 | **-5.30%** | 0.0003 | 1.505 |
| | | IDW | 0.012 | **-11.00%** | 0.0003 | 1.463 |
| | | NN | 0.013 | **-1.31%** | 0.0017 | 3.227 |
| Pt ppm | EST | Composites | 0.010 | - | 0.0011 | 3.334 |
| | | OK | 0.010 | **-6.84%** | 0.0002 | 1.389 |
| | | IDW | 0.009 | **-11.53%** | 0.0001 | 1.256 |
| | | NN | 0.011 | **8.67%** | 0.0010 | 2.892 |
| Density (MgSerp) | HG | Composites | 2.64 | - | 0.006 | 0.03 |
| | | OK | 2.64 | **-0.24%** | 0.001 | 0.01 |
| | | IDW | 2.63 | **-0.33%** | 0.001 | 0.01 |
| | | NN | 2.64 | **-0.20%** | 0.003 | 0.02 |
| | NLG | Composites | 2.70 | - | 0.004 | 0.02 |
| | | OK | 2.70 | **-0.25%** | 0.002 | 0.02 |
| | | IDW | 2.70 | **-0.24%** | 0.001 | 0.01 |
| | | NN | 2.70 | **-0.25%** | 0.005 | 0.03 |
| | SLG | Composites | 2.72 | - | 0.007 | 0.03 |
| | | OK | 2.72 | **0.01%** | 0.002 | 0.02 |
| | | IDW | 2.72 | **-0.01%** | 0.003 | 0.02 |
| | | NN | 2.71 | **-0.24%** | 0.007 | 0.03 |
| Brucite (MgSerp) | HG | 2.82 | - | 3.19 | 0.63 | 2.82 |
| | | 2.64 | **-6.21%** | 1.43 | 0.45 | 2.64 |
| | | 2.91 | **3.17%** | 1.23 | 0.38 | 2.91 |
| | | 2.72 | **-3.48%** | 3.36 | 0.67 | 2.72 |
| | NLG | 0.78 | - | 1.60 | 1.61 | 0.78 |
| | | 0.58 | **-25.86%** | 0.32 | 0.98 | 0.58 |
| | | 0.61 | **-22.16%** | 0.32 | 0.92 | 0.61 |
| | | 0.6 | **-23.87%** | 1.32 | 1.92 | 0.6 |
| | SLG | 0.2 | - | 0.28 | 2.57 | 0.2 |
| | | 0.2 | **-2.89%** | 0.06 | 1.27 | 0.2 |
| | | 0.17 | **-16.53%** | 0.04 | 1.12 | 0.17 |
| | | 0.18 | **-13.91%** | 0.21 | 2.62 | 0.18 |

Source: Caracle Creek, 2023.

**Table 14-8: Main-West Zone Global Statistical Comparisons between Estimates and Declustered Composites in Resource Domains**

| Element | Domain | Data | Mean | Bias | Std Dev | CV |
|---|---|---|---|---|---|---|
| Ni % | HGM | Composites | 0.30 | - | 0.0032 | 0.19 |
| | | OK | 0.29 | **-1.41%** | 0.0009 | 0.10 |
| | | IDW | 0.29 | **-1.48%** | 0.0011 | 0.11 |
| | | NN | 0.30 | **-0.93%** | 0.0032 | 0.19 |
| | HGW | Composites | 0.28 | - | 0.0015 | 0.14 |
| | | OK | 0.27 | **-1.29%** | 0.0003 | 0.06 |
| | | IDW | 0.27 | **-1.72%** | 0.0003 | 0.06 |
| | | NN | 0.26 | **-4.11%** | 0.0013 | 0.14 |
| | NLG | Composites | 0.21 | - | 0.0021 | 0.21 |
| | | OK | 0.22 | **3.75%** | 0.0006 | 0.11 |
| | | IDW | 0.22 | **3.70%** | 0.0006 | 0.11 |
| | | NN | 0.21 | **0.87%** | 0.0017 | 0.19 |
| | SLG | Composites | 0.22 | - | 0.0015 | 0.18 |
| | | OK | 0.22 | **0.03%** | 0.0005 | 0.10 |
| | | IDW | 0.22 | **1.80%** | 0.0005 | 0.10 |
| | | NN | 0.22 | **2.66%** | 0.0014 | 0.17 |
| | VLG | Composites | 0.12 | - | 0.0013 | 0.31 |
| | | OK | 0.12 | **4.60%** | 0.0006 | 0.20 |
| | | IDW | 0.12 | **1.48%** | 0.0008 | 0.23 |
| | | NN | 0.12 | **0.28%** | 0.0021 | 0.38 |
| Co % | EST | Composites | 0.013 | - | 0.000002 | 0.12 |
| | | OK | 0.013 | **1.03%** | 0.000001 | 0.08 |
| | | IDW | 0.013 | **1.11%** | 0.000001 | 0.08 |
| | | NN | 0.013 | **0.75%** | 0.000003 | 0.12 |
| Fe % | LFE | Composites | 5.89 | - | 0.65 | 0.14 |
| | | OK | 6.1 | **3.65%** | 0.24 | 0.08 |
| | | IDW | 6.05 | **2.73%** | 0.24 | 0.08 |
| | | NN | 6.23 | **5.86%** | 0.62 | 0.13 |
| | NHFE | Composites | 6.99 | - | 0.50 | 0.10 |
| | | OK | 6.98 | **-0.19%** | 0.16 | 0.06 |
| | | IDW | 6.99 | **0.01%** | 0.20 | 0.06 |
| | | NN | 7.14 | **2.05%** | 0.62 | 0.11 |
| | SHFE | Composites | 7.25 | - | 0.28 | 0.07 |
| | | OK | 7.41 | **2.26%** | 0.08 | 0.04 |
| | | IDW | 7.35 | **1.45%** | 0.09 | 0.04 |
| | | NN | 7.34 | **1.31%** | 0.26 | 0.07 |
| Cr % | LCR | Composites | 0.46 | - | 0.023 | 0.33 |
| | | OK | 0.47 | **1.28%** | 0.005 | 0.14 |
| | | IDW | 0.46 | **0.65%** | 0.005 | 0.15 |
| | | NN | 0.50 | 7.55% | 0.022 | 0.30 |
| | NHCR | Composites | 0.60 | - | 0.019 | 0.23 |
| | | OK | 0.59 | **-1.21%** | 0.006 | 0.14 |
| | | IDW | 0.59 | **-2.16%** | 0.007 | 0.14 |
| | | NN | 0.59 | **-1.92%** | 0.017 | 0.22 |
| | SHCR | Composites | 0.59 | - | 0.012 | 0.19 |
| | | OK | 0.59 | **0.54%** | 0.004 | 0.10 |
| | | IDW | 0.59 | **0.32%** | 0.004 | 0.11 |
| | | NN | 0.58 | **-2.77%** | 0.011 | 0.19 |
| | VLCR | Composites | 0.44 | - | 0.007 | 0.19 |
| | | OK | 0.44 | **1.06%** | 0.002 | 0.11 |
| | | IDW | 0.44 | **-0.10%** | 0.003 | 0.12 |
| | | NN | 0.44 | **0.17%** | 0.007 | 0.19 |
| S % | HGM | Composites | 0.13 | - | 0.0323 | 1.36 |
| | | OK | 0.13 | **-4.47%** | 0.0183 | 1.07 |
| | | IDW | 0.13 | **-2.56%** | 0.0192 | 1.07 |
| | | NN | 0.13 | **1.28%** | 0.0360 | 1.42 |
| | HGW | Composites | 0.05 | - | 0.0016 | 0.83 |
| | | OK | 0.04 | -10.44% | 0.0005 | 0.53 |
| | | IDW | 0.05 | **-3.01%** | 0.0005 | 0.46 |
| | | NN | 0.04 | -15.45% | 0.0012 | 0.86 |
| | NLG+VLG | Composites | 0.05 | - | 0.0031 | 1.20 |
| | | OK | 0.04 | -18.07% | 0.0017 | 1.09 |
| | | IDW | 0.04 | -22.59% | 0.0013 | 1.02 |
| | | NN | 0.04 | **-4.73%** | 0.0045 | 1.52 |
| | SLG | Composites | 0.04 | - | 0.0030 | 1.35 |
| | | OK | 0.03 | -24.94% | 0.0010 | 1.02 |
| | | IDW | 0.03 | -17.00% | 0.0011 | 0.99 |
| | | NN | 0.03 | -16.44% | 0.0017 | 1.22 |
| Pd ppm | EST | Composites | 0.016 | - | 0.00153 | 2.487 |
| | | OK | 0.012 | -25.65% | 0.00018 | 1.132 |
| | | IDW | 0.012 | -21.75% | 0.00018 | 1.087 |

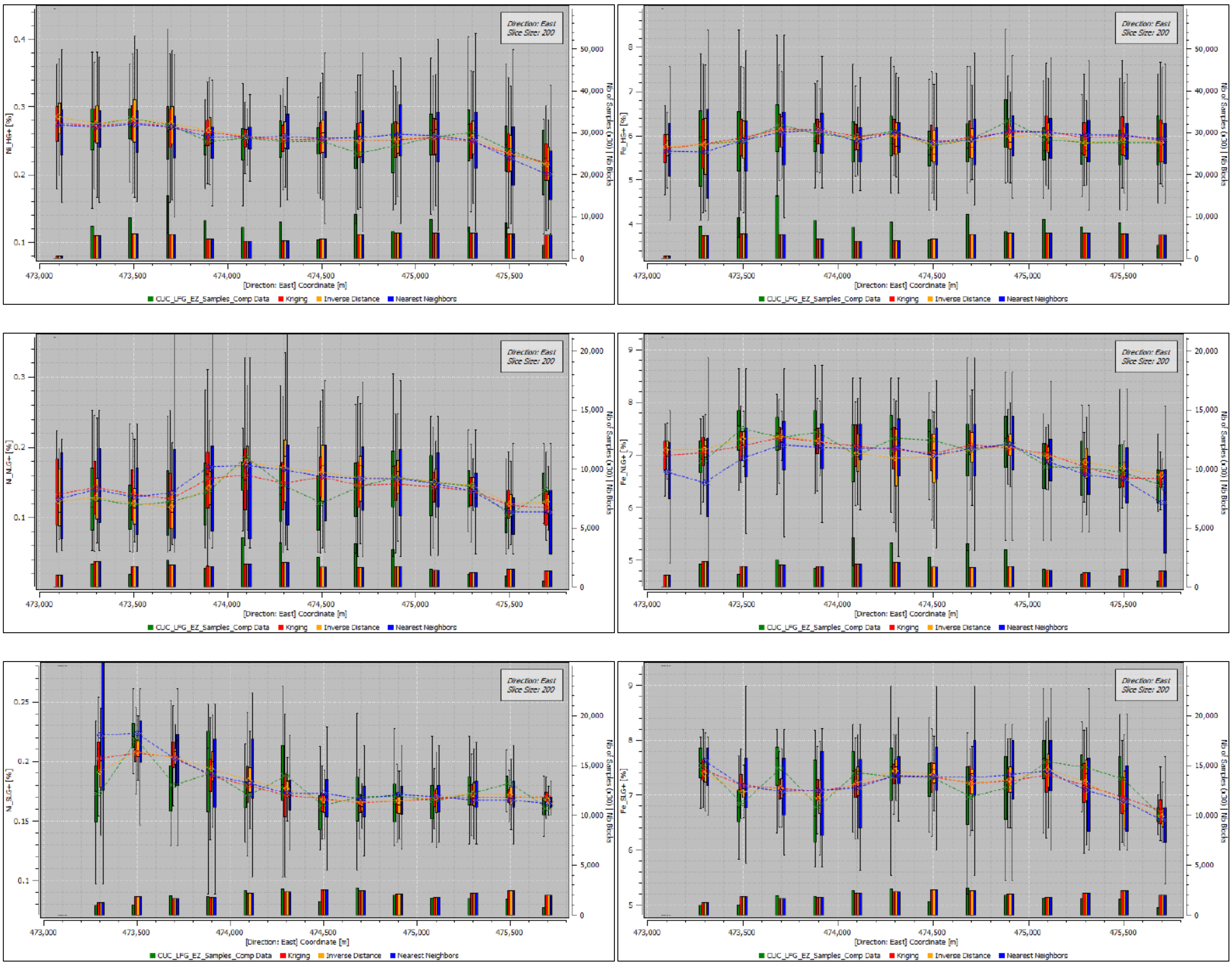
| Element | Domain | Data | Mean | Bias | Std Dev | CV |
|---|---|---|---|---|---|---|
| | | NN | 0.014 | **-9.20%** | 0.00079 | 1.968 |
| Pt ppm | EST | Composites | 0.011 | - | 0.00032 | 1.718 |
| | | OK | 0.009 | **-11.04%** | 0.00004 | 0.712 |
| | | IDW | 0.010 | **-7.22%** | 0.00004 | 0.688 |
| | | NN | 0.011 | **3.97** | 0.00023 | 1.402 |
| Density | MgSerp | Composites | 2.67 | - | 0.006 | 0.03 |
| | | OK | 2.67 | **0.17%** | 0.002 | 0.02 |
| | | IDW | 2.68 | **0.42%** | 0.003 | 0.02 |
| | | NN | 2.69 | **0.64%** | 0.006 | 0.03 |
| | FeSerp Unweathered | Composites | 2.77 | - | 0.016 | 0.05 |
| | | OK | 2.77 | **0.14%** | 0.007 | 0.03 |
| | | IDW | 2.77 | **0.16%** | 0.006 | 0.03 |
| | | NN | 2.75 | **-0.59%** | 0.017 | 0.05 |
| | FeSerp Weathered | Composites | 2.57 | - | 0.008 | 0.04 |
| | | OK | 2.57 | **0.13%** | 0.002 | 0.02 |
| | | IDW | 2.57 | **0.09%** | 0.002 | 0.02 |
| | | NN | 2.56 | **-0.05%** | 0.010 | 0.04 |
| Brucite | DUN (MgSerp) | Composites | 1.81 | - | 1.31 | 0.72 |
| | | OK | 1.84 | **1.16%** | 1.05 | 0.57 |
| | | IDW | 1.89 | **3.99%** | 1.06 | 0.56 |
| | | NN | 1.94 | **6.81%** | 1.42 | 0.73 |
| | DUN (FeSerp) | Composites | 0.87 | - | 0.71 | 0.81 |
| | | OK | 0.83 | **-4.84** | 0.34 | 0.40 |
| | | IDW | 0.80 | **-8.16%** | 0.33 | 0.41 |
| | | NN | 0.88 | **0.85** | 0.78 | 0.89 |
| | PER-0 (MgSerp) | Composites | 0.35 | - | 0.67 | 1.90 |
| | | OK | 0.26 | **-26.59%** | 0.35 | 1.33 |
| | | IDW | 0.29 | **-17.46%** | 0.35 | 1.19 |
| | | NN | 0.27 | **-23.03%** | 0.64 | 2.34 |

Source: Caracle Creek, 2023.

### 14.9.3 Moving Window Validation

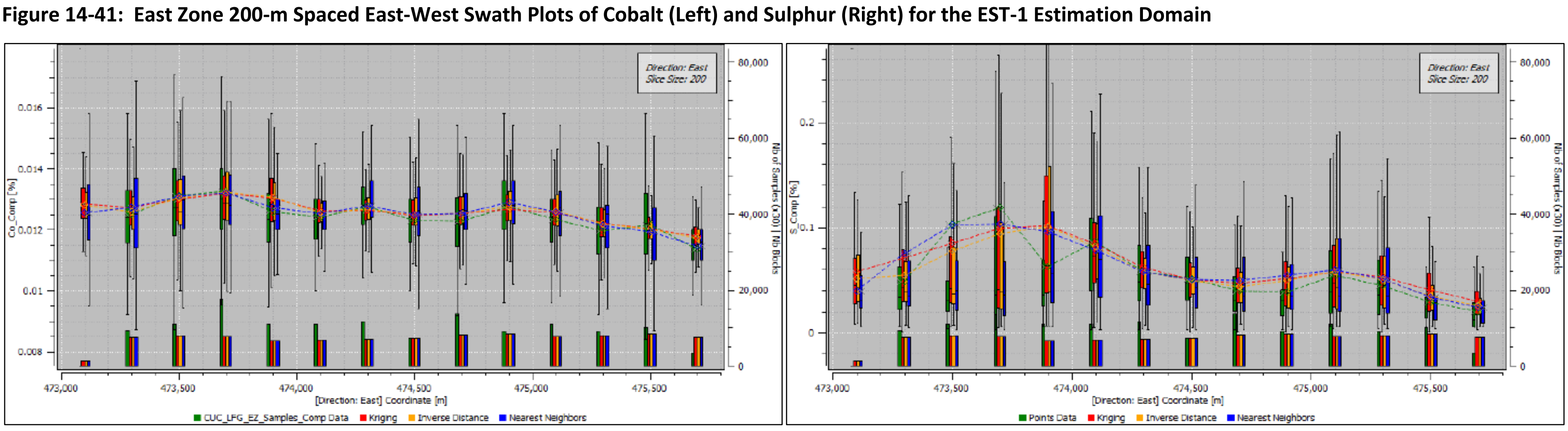
Swath plots allow for localized statistical comparisons by averaging grades in sequential slices (or windows) through the estimated resource. Slice directions were aligned with the blocks (east-west, north-south and vertical), and the slice width was selected depending on sample distribution in each direction. The presented plots (Figures 14-40 through 14-43) show composite (green), OK (red), IDW (orange) and NN (blue) means and box plot for each step, as well as value counts. All variables show generally good consistency between estimates and composites, especially in windows with high composite counts. Deviations between composites/estimates and NN results manifest in some cases given that the latter are currently obtained from a single neighbourhood search and direction.

**Figure 14-40: East Zone 200-m Spaced East-West Swath Plots of Nickel (Left Column) and Iron (Right Column) for the HG (Top Row), NLG (Middle Row) and SLG (Bottom Row) Estimation Domains**



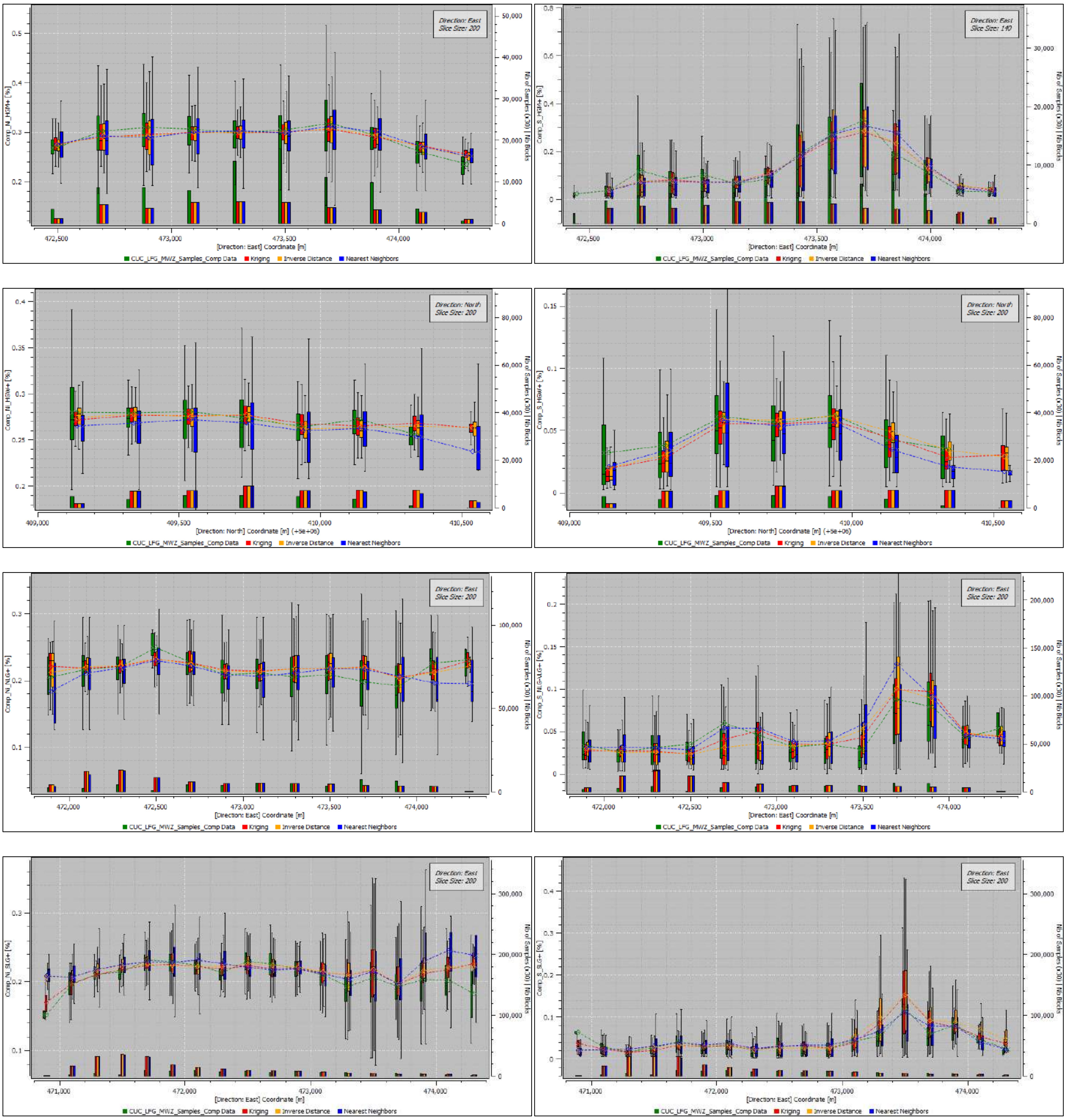

Source: Caracle Creek, 2023.

**Figure 14-41: East Zone 200-m Spaced East-West Swath Plots of Cobalt (Left) and Sulphur (Right) for the EST-1 Estimation Domain**



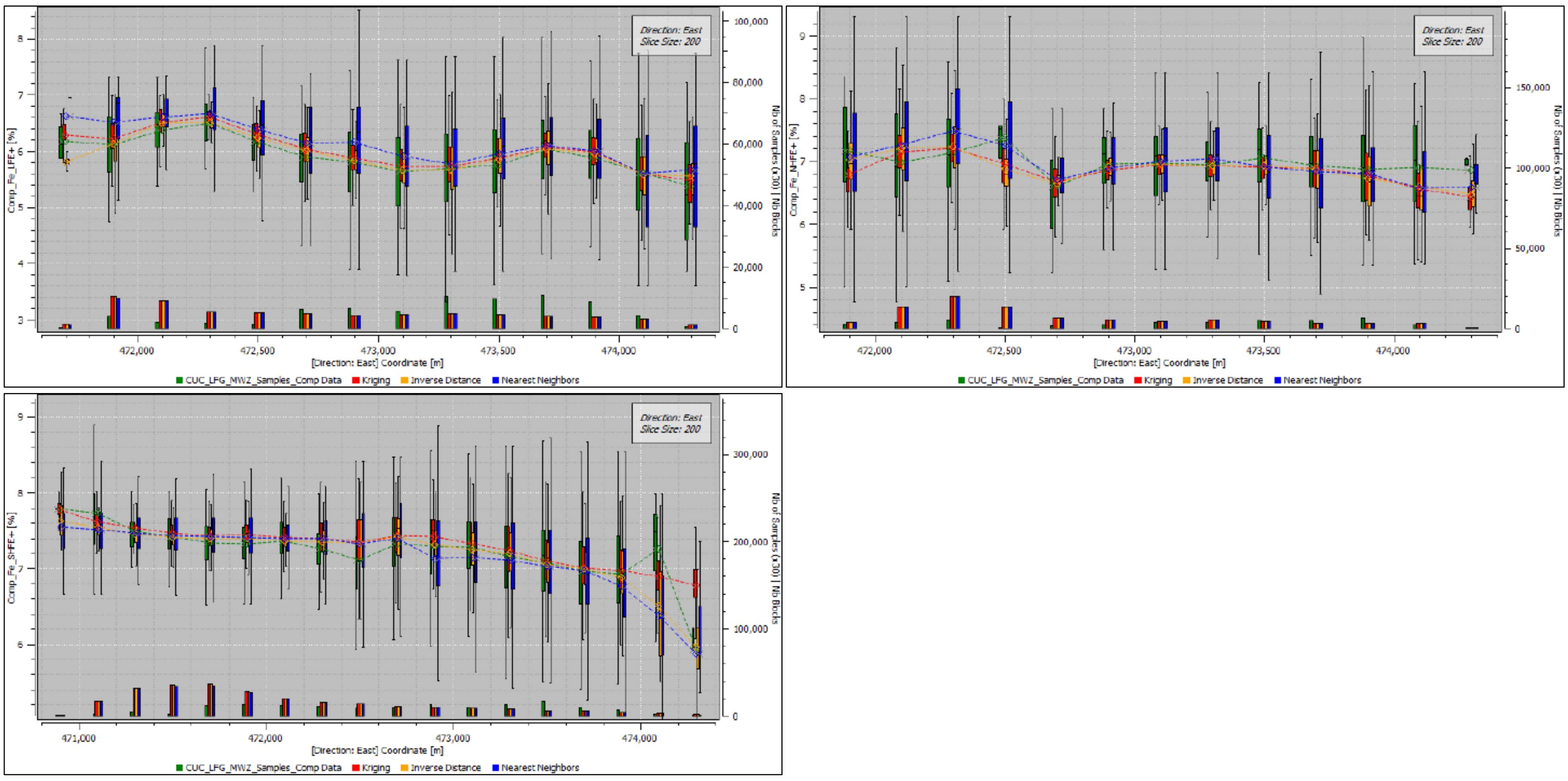

Source: Caracle Creek, 2023.

**Figure 14-42: Main-West Zone 200-m Spaced East-West Swath Plots of Nickel (Left Column), and Sulphur (Right Column) for the HGM (Top Row), HGW (second Row) NLG (third Row) and SLG (Bottom Row) Estimation Domains**




Note: The sulphur variogram for the NLG domain also includes the VLG domain, as they were combined for this element. The nickel variogram for the VLG estimation domain was not included for practical purposes. Source: Caracle Creek, 2023.

**Figure 14-43: Main-West Zone 200-m Spaced East-West Swath Plots of Iron for the LFE (Top), NHFE (Bottom Left) and SHFE (Bottom Right) Estimation Domains**




Source: Caracle Creek, 2023.

## 14.10 Mineral Resource Classification and Estimate

The mineral resources for the project were classified in accordance with the most current CIM Definition Standards (CIM, 2019). The “CIM Definition Standards for Mineral Resources and Reserves” prepared by the CIM Standing Committee on Resource Definitions and adopted by the CIM council on November 29, 2019, provides standards for the classification of Mineral Resources and Mineral Reserves estimates as follows:

- Inferred Mineral Resource – An inferred mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated based on limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that most inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.
- Indicated Mineral Resource – An indicated mineral resource is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with sufficient confidence to allow the application of modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An indicated mineral resource has a lower level of confidence than that applying to a measured mineral resource and may only be converted to a probable mineral reserve.
- Measured Mineral Resource – A measured mineral resource is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of modifying factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A measured mineral resource has a higher level of confidence than that applying to either an indicated mineral resource or an inferred mineral resource. It may be converted to a proven mineral reserve or to a probable mineral reserve.

### 14.10.1 East Zone: Mineral Resource Classification

Resource classification in the East Zone consisted in evaluating block proximity to drillholes using range values from a nickel variogram at different percentages of the sill, with measured blocks representing the range at 50% of the sill, indicated blocks from the range at 75% of the sill and inferred blocks from the range at 90% of the sill. Any blocks that did not meet previous criteria were classified as Potential. The reference variogram was generated using nickel composites from the most recent update (Lane et al., 2022) within the general EST-1 domain (Figure 14-44), to maintain consistency with the previous classification criteria. The neighbourhood search was carried out according to a stricter set of parameters than the ones used for resource estimation (Table 14-9), like the use of octant search or the requirement of three drillholes to achieve measured status and two for indicated.

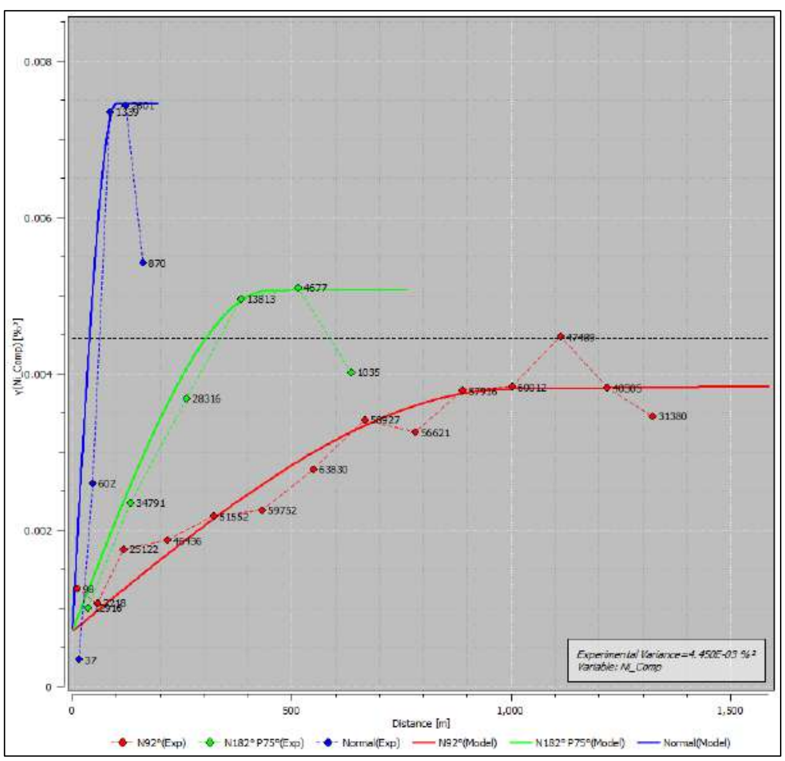
Smoothing was carried out by digitizing a rough outline of the block distribution for the initial classification as seen in cross-section every 50 m, in order to model shells representing the final volumes of the four classes. These volumes were then flagged into the block model, generating the final classification column (Figure 14-45).

**Table 14-9: East Zone Neighbourhood Search and Estimation Parameters for Resource Classification**

| Parameter | Neighbourhood | | | |
|---|---|---|---|---|
| Pass | 1st | 2nd | 3rd | 4th |
| Sector Search | Octants | | | |
| Minimum Sectors | 6 | 6 | 6 | 6 |
| Maximum Points per Sector | 4 | 4 | 4 | 4 |
| Minimum Total Points | 12 | 8 | 4 | 1 |
| Maximum Points per Drillhole | 4 | 4 | 4 | 4 |
| Minimum Points per Drillhole | - | - | - | - |
| Minimum Drillholes | 3 | 2 | 1 | 1 |
| Search Radius Directions | 92° Az / 75° Dip / 0° Pitch | | | |
| Search Radius Axis 1 | 275 | 485 | 675 | ∞ |
| Search Radius Axis 2 | 135 | 230 | 320 | ∞ |
| Search Radius Axis 3 | 35 | 55 | 70 | ∞ |

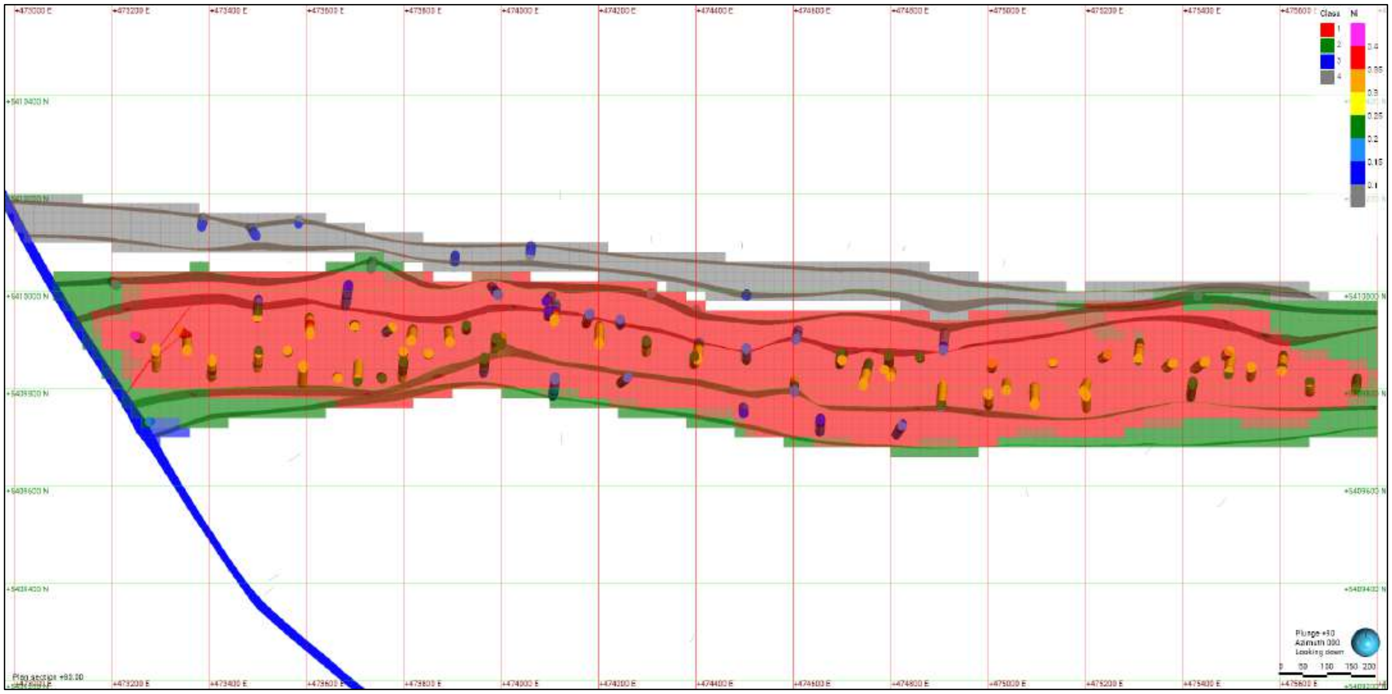
Source: Caracle Creek, 2023.

**Figure 14-44: East Zone EST-1 Domain Nickel Variogram for Resource Classification**




Source: Caracle Creek, 2022.

**Figure 14-45: East Zone Plan Section (80 m) of Resource Classification**



Note: Shows measured (red), indicated (green), inferred (blue) and potential (grey) blocks for all estimation domains.
Source: Caracle Creek, 2023.

### 14.10.2 Main-West Zone: Mineral Resource Classification

Like the East Zone, resource classification in the Main-West Zone consisted in evaluating block proximity to drillholes using range values from a nickel variogram at different percentages of the sill, with measured blocks representing the range at 50% of the sill, indicated blocks from the range at 70% of the sill and inferred blocks from the range at 90% of the sill. Any blocks that did not meet previous criteria were classified as Potential. The reference variogram was generated using nickel composites from the most recent update (Lane et al., 2022) within the general EST domain (Figure 14-46), to maintain consistency with the previous classification criteria. The neighbourhood search was carried out according to a stricter set of parameters than the ones used for resource estimation (Table 14-10), like the use of octant search or the requirement of three complete drillholes to achieve measured status and three "incomplete" drillholes for Indicated.

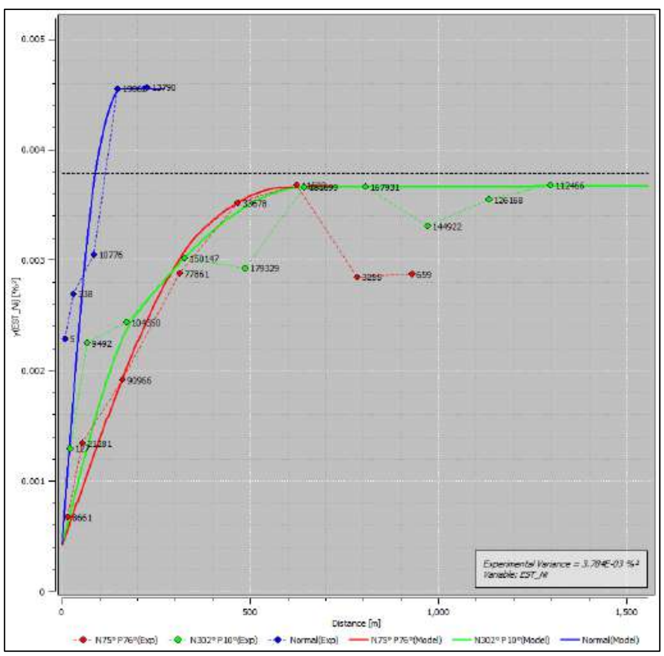
**Table 14-10: Main-West Zone Neighbourhood Search and Estimation Parameters for Resource Classification**

| Parameter | Neighbourhood | | | |
|---|---|---|---|---|
| Pass | 1st | 2nd | 3rd | 4th |
| Sector Search | Octants | | | |
| Minimum Sectors | 6 | 6 | 6 | 6 |
| Maximum Points per Sector | 4 | 4 | 4 | 4 |
| Minimum Total Points | 12 | 10 | 8 | 1 |
| Maximum Points per Drillhole | 4 | 4 | 4 | 4 |
| Minimum Points per Drillhole | - | - | - | - |
| Minimum Drillholes | 3 | 2.5 | 2 | 1 |
| Search Radius Directions | 120° Az / 80° Dip / 280° Pitch | | | |
| Search Radius Axis 1 | 145 | 240 | 380 | ∞ |
| Search Radius Axis 2 | 115 | 205 | 410 | ∞ |
| Search Radius Axis 3 | 40 | 65 | 100 | ∞ |

Source: Caracle Creek, 2023.

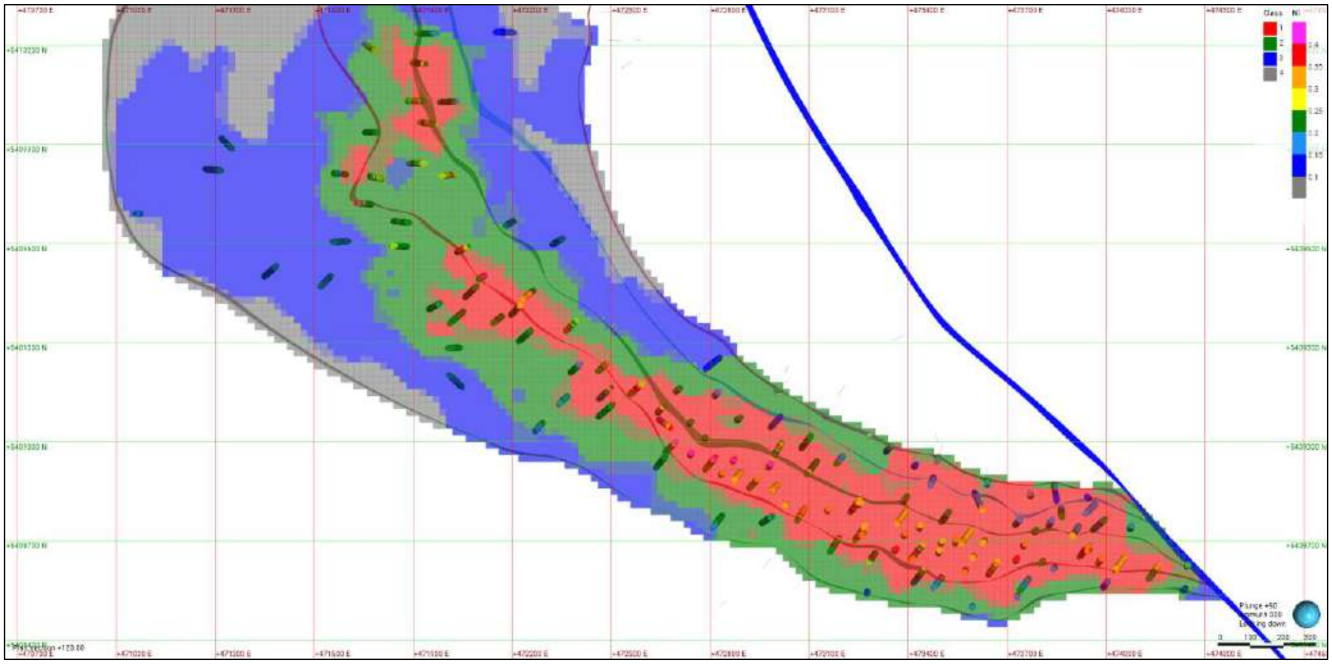
Smoothing was carried out by digitizing a rough outline of the block distribution for the initial classification as seen in cross-section every 50 m, in order to model shells representing the final volumes of the four classes. These volumes were then flagged into the block model, generating the final classification column (Figure 14-47).

**Figure 14-46: Main-West Zone EST-1 Domain Nickel Variogram for Resource Classification**




Source: Caracle Creek, 2022.

**Figure 14-47: Main-West Zone Plan Cross-Section of Resource Classification, Looking Down at 0 Elevation**



Note: Shows measured (red), indicated (green), inferred (blue) and potential (grey) blocks for all estimation domains.
Source: Caracle Creek, 2023.

## 14.11 Pit Optimization and Cut-off Grade

According to CIM (2019), for a mineral deposit to be considered a mineral resource it must be shown that there are "reasonable prospects for eventual economic extraction". As Crawford will be mined using open pit methods, the 'reasonable prospects' are considered satisfied by limiting mineral resources to those constrained within a conceptual pit shell and grading above a cut-off grade.

The pit shell was generated under the supervision of David Penswick, who is the Qualified Person for the mine design and economic analysis components of the study, using the Lerchs-Grossmann (LG) algorithm, As discussed in Section 15.4, the LG algorithm is the industry standard tool used to define the limits of an open pit.

Specific inputs to the LG algorithm are discussed at length in Sections 15.3, 15.4, and 16.2. Key inputs include the following (note: except where stated otherwise, prices are in US dollars):

- nickel price of $21,000 and payability of 91% (Ni generates 63% of total metal revenue)
- iron price and payability equivalent to an iron ore price of $89/t (Fe generates 20% of total metal revenue)
- chromium price of $1.75/lb and payability of 65% (Cr generates 15% of total metal revenue)
- total marginal costs (process, G&A and water management) of C$9.21/tonne milled.

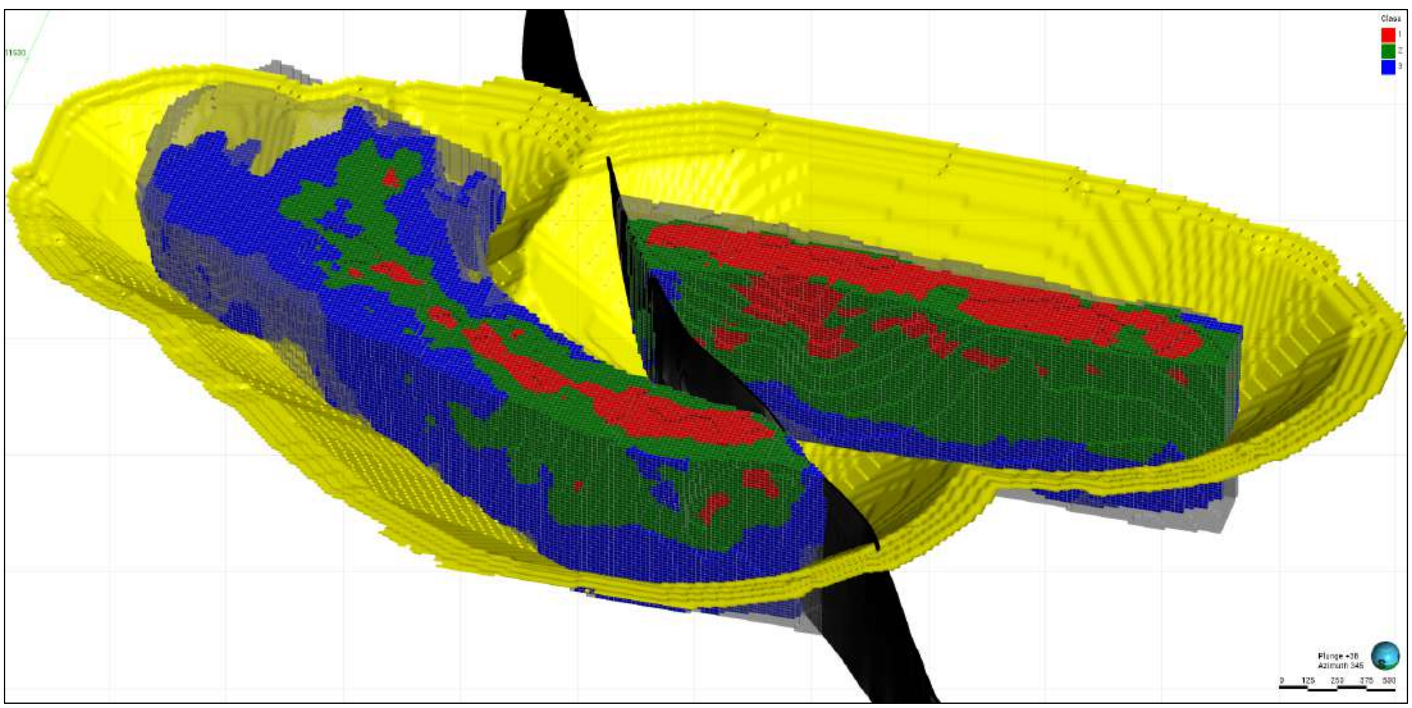
It is important to note that the results from this pit optimization exercise are used solely for testing the “reasonable prospects for eventual economic extraction” by open pit mining methods and do not represent an economic study. Figure 14-48 shows a 3D view of the generated pit around the East Zone and Main-West Zone mineral resource estimates.

The cut-off grade was calculated using the following parameters:

- Marginal costs reported in Section 15.7 that aggregate to C$9.21/t milled (US$7.00/t at the exchange rate of $0.76).
- Average Ni recovery of 41%. Note that the parameters controlling Ni recovery do not include nickel grade, so it is not unreasonable to assume average recovery for lower grade material. This assumption is supported by testwork (see Section 13.7, where testwork includes that on samples grading as low as 0.11% nickel).
- Nickel price and payability, as reported above, of US$21,000/t and 91%, respectively.

These parameters yield a cut-off grade of 2.0 lbs contained nickel per tonne of ore, or 0.09% Ni. This has been rounded up to 0.10% Ni.

**Figure 14-48: Main-West and East Zone 3D View Looking North-Northwest**



Note: The theoretical pit surface (yellow) is shown against measured (red), indicated (green), inferred (blue) and potential (transparent grey) blocks for all estimation domains. The regional fault zone is represented by the black volume. Source: Caracle Creek, 2023.

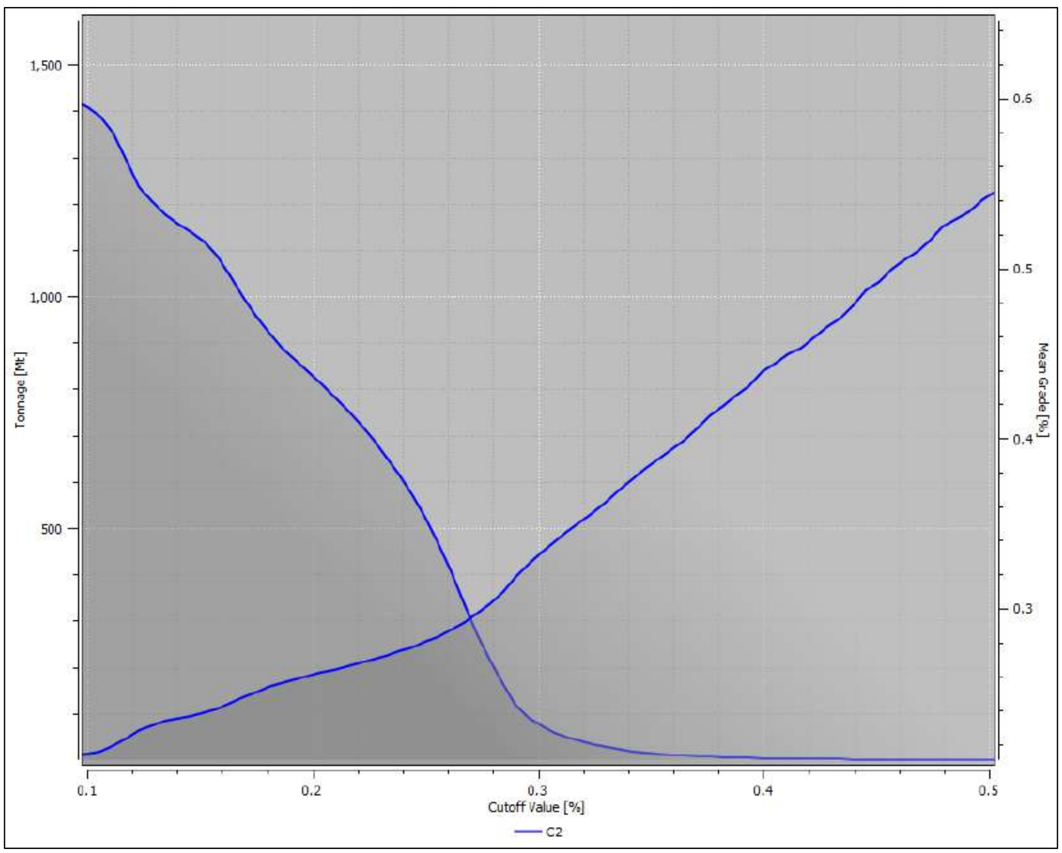
### 14.11.1 Grade Sensitivity Analysis: East Zone

Based on the combined block model from Section 14.10.1 (Mineral Resource Classification) and constrained by the RF100 pit, a grade-tonnage curve was calculated for the nickel domains (Figure 14-49), marking a nickel cut-off grade of 0.269% Ni, included as a data point in the grade sensitivity analysis. The reader is cautioned that the numbers presented in Figure 14-49 should not be misconstrued as a mineral resource statement (see Section 14.12 for details).

### 14.11.2 Grade Sensitivity Analysis: Main-West Zone

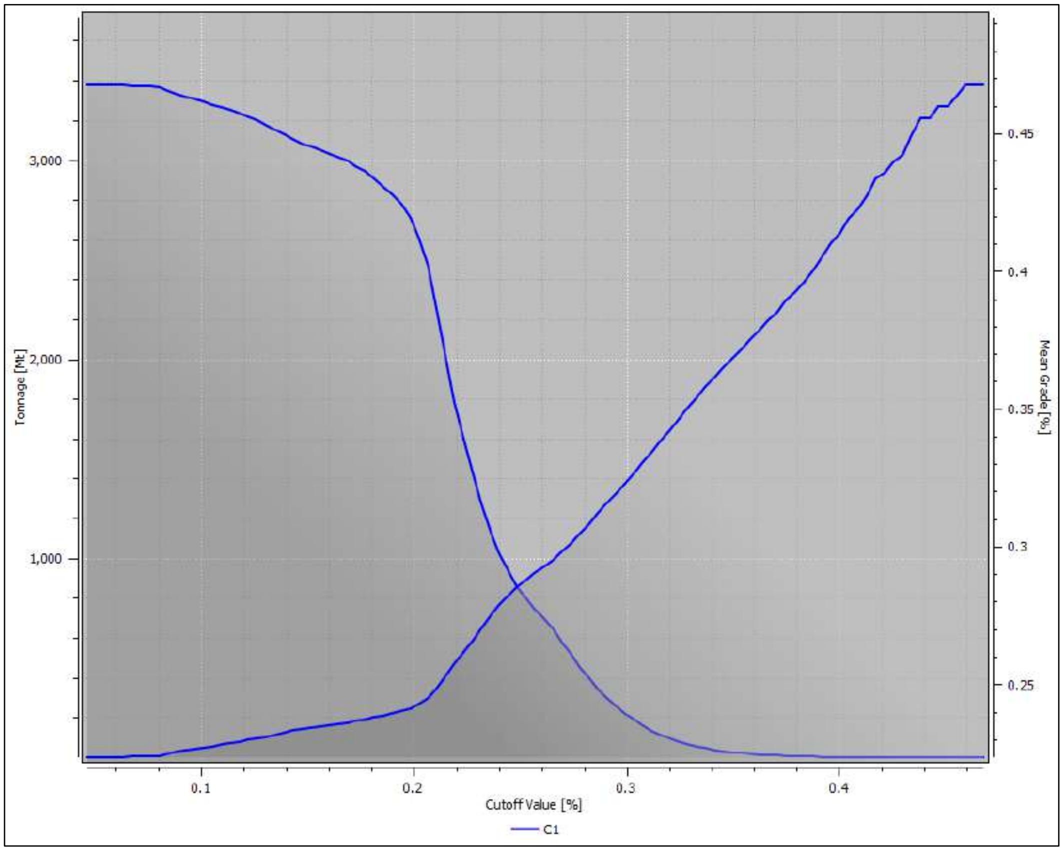
Based on the combined block model from Section 14.10.2, Main-West Zone: Mineral Resource Classification, and constrained by the RF100 pit, a grade-tonnage curve was calculated for the nickel domains (Figure 14-50), marking a nickel cut-off grade of 0.249% Ni, included as a data point in the grade sensitivity analysis. The reader is cautioned that the numbers presented in Figure 14-50 should not be misconstrued with a mineral resource statement (see Section 14.12).

**Figure 14-49: East Zone Grade-Tonnage Curve for Nickel Grades**




Source: Caracle Creek, 2023.

**Figure 14-50: Main-West Zone Grade-Tonnage Curve for Nickel Grades**




Source: Caracle Creek, 2023.

## 14.12 Mineral Resource Statement

The measured, indicated, and inferred mineral resources presented herein are constrained within the pit shell developed for the Main-West and East zones from the pit optimization analysis discussed above. The effective date of the mineral resource estimates is August 31, 2023.

### 14.12.1 East Zone: Mineral Resource Estimate

Pit-constrained, class-characterized mineral resources at a 0.10% Ni cut-off grade with respect to the higher and lower grade nickel estimation domains within the East Zone are presented for all elements studied in Table 14-11.

### 14.12.2 Main-West Zone: Mineral Resource Estimate

Pit-constrained, class-characterized mineral resources at a 0.10% Ni cut-off grade with respect to the higher and lower grade nickel estimation domains within the Main-West Zone are presented for all elements studied in Table 14-12.

**Table 14-11: Summary of the Pit-Constrained Updated East Zone Mineral Resource Estimate**

| Domain | Class | Tonnage (Mt) | Ni (%) | Ni Content (kt) | Co (%) | Co Content (kt) | Fe (%) | Fe Content (Mt) | S (%) | S Content (kt) | Cr (%) | Cr Content (kt) | Brucite (%) | Pd (g/t) | Pd Content (koz) | Pt (g/t) | Pt Content (koz) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Higher Grade | Measured | 394.2 | 0.26 | 1,021.6 | 0.012 | 49.2 | 5.92 | 23.3 | 0.07 | 271.3 | 0.65 | 2,545.6 | 3.10 | 0.015 | 184.9 | 0.009 | 119.5 |
| | Indicated | 299.6 | 0.26 | 773.6 | 0.013 | 37.8 | 5.85 | 17.5 | 0.08 | 239.3 | 0.63 | 1,886.7 | 3.19 | 0.011 | 102.6 | 0.007 | 68.9 |
| | Measured + Indicated | 693.7 | 0.26 | 1,795.2 | 0.013 | 87.1 | 5.89 | 40.9 | 0.07 | 510.6 | 0.64 | 4,432.3 | 3.14 | 0.013 | 287.5 | 0.008 | 188.3 |
| | Inferred | 111.8 | 0.26 | 289.1 | 0.013 | 14.2 | 5.90 | 6.6 | 0.08 | 86.5 | 0.62 | 695.4 | 2.89 | 0.010 | 36.7 | 0.007 | 25.5 |
| Lower Grade | Measured | 169.2 | 0.16 | 278.6 | 0.013 | 21.3 | 7.25 | 12.3 | 0.04 | 66.9 | 0.54 | 907.8 | 0.40 | 0.011 | 57.5 | 0.009 | 49.1 |
| | Indicated | 171.7 | 0.17 | 289.2 | 0.012 | 21.2 | 7.11 | 12.2 | 0.04 | 69.0 | 0.52 | 885.9 | 0.93 | 0.011 | 61.1 | 0.009 | 52.4 |
| | Measured + Indicated | 340.9 | 0.17 | 567.8 | 0.012 | 42.5 | 7.18 | 24.5 | 0.04 | 135.9 | 0.53 | 1,793.7 | 0.67 | 0.011 | 118.6 | 0.009 | 101.6 |
| | Inferred | 45.5 | 0.17 | 78.0 | 0.013 | 5.8 | 7.11 | 3.2 | 0.06 | 26.9 | 0.54 | 244.4 | 0.55 | 0.010 | 14.2 | 0.008 | 11.6 |
| **Total Grade** | **Measured + Indicated** | **1,034.6** | **0.23** | **2,363.0** | **0.013** | **129.6** | **6.31** | **65.3** | **0.06** | **646.5** | **0.60** | **6,225.9** | **1.25** | **0.012** | **406.1** | **0.009** | **289.9** |
| | **Inferred** | **157.3** | **0.23** | **367.1** | **0.013** | **20.0** | **6.25** | **9.8** | **0.07** | **113.3** | **0.60** | **939.8** | **0.98** | **0.010** | **50.9** | **0.007** | **37.1** |

Source: Caracle Creek, 2023.

**Table 14-12: Summary of the Pit-Constrained Initial Main-West Zone Mineral Resource Estimate**

| Domain | Class | Tonnage (Mt) | Ni (%) | Ni Content (kt) | Co (%) | Co Content (kt) | Fe (%) | Fe Content (Mt) | S (%) | S Content (kt) | Cr (%) | Cr Content (kt) | Brucite (%) | Pd (g/t) | Pd Content (koz) | Pt (g/t) | Pt Content (koz) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Higher Grade | Measured | 253.3 | 0.30 | 770.4 | 0.013 | 33.1 | 6.40 | 16.2 | 0.15 | 381.9 | 0.59 | 1,502.8 | 1.73 | 0.027 | 219.0 | 0.012 | 95.5 |
| | Indicated | 295.9 | 0.28 | 829.7 | 0.013 | 39.0 | 6.93 | 20.5 | 0.08 | 245.3 | 0.57 | 1,694.2 | 1.36 | 0.023 | 218.0 | 0.012 | 111.8 |
| | Measured + Indicated | 549.2 | 0.29 | 1,600.1 | 0.013 | 72.1 | 6.68 | 36.7 | 0.11 | 627.2 | 0.58 | 3,197.0 | 1.53 | 0.025 | 437.0 | 0.012 | 207.3 |
| | Inferred | 211.9 | 0.28 | 587.1 | 0.013 | 28.2 | 6.91 | 14.6 | 0.06 | 122.6 | 0.56 | 1,190.2 | 1.21 | 0.018 | 123.2 | 0.011 | 72.9 |
| Lower Grade | Measured | 280.4 | 0.22 | 607.3 | 0.013 | 36.8 | 6.89 | 19.3 | 0.04 | 121.0 | 0.59 | 1,646.0 | 1.15 | 0.011 | 96.3 | 0.009 | 78.8 |
| | Indicated | 697.5 | 0.21 | 1,464.9 | 0.013 | 91.7 | 7.10 | 49.6 | 0.04 | 293.6 | 0.57 | 3,997.5 | 1.07 | 0.011 | 249.3 | 0.009 | 206.7 |
| | Measured + Indicated | 977.9 | 0.21 | 2,072.2 | 0.013 | 128.5 | 7.04 | 68.9 | 0.04 | 414.7 | 0.58 | 5,643.5 | 1.10 | 0.011 | 345.6 | 0.009 | 285.4 |
| | Inferred | 1,324.0 | 0.21 | 2,772.1 | 0.013 | 173.8 | 7.20 | 95.4 | 0.03 | 456.1 | 0.57 | 7,543.8 | 0.94 | 0.010 | 419.5 | 0.009 | 386.2 |
| **Total Grade** | **Measured + Indicated** | **1,527.2** | **0.24** | **3,672.3** | **0.013** | **200.6** | **6.91** | **105.6** | **0.07** | **1,041.9** | **0.58** | **8,840.5** | **2.33** | **0.016** | **782.6** | **0.010** | **492.8** |
| | **Inferred** | **1,535.8** | **0.22** | **3,359.2** | **0.013** | **202.0** | **7.16** | **110.0** | **0.04** | **578.7** | **0.57** | **8,734.1** | **2.21** | **0.011** | **542.7** | **0.009** | **459.0** |

Source: Caracle Creek, 2023.

# 15 MINERAL RESERVE ESTIMATES

## 15.1 Summary

The Crawford mineral reserves are summarized in Tables 15-1 and 15-2.

Mineral reserves were estimated by Dave Penswick, P.Eng. These are based on the mineral resource block model described in the previous section. Mineral reserves are contained within an engineered pit design that has been based on a Lerchs-Grossmann (LG) pit optimization run at a revenue factor (RF) 65% of the base case prices (reported in US dollars); or $13,650/t Ni, $26,000/t Co, $58/t iron ore, $2,500/t Cr, $878/oz Pd and $748/oz Pt. Mineral reserves include unplanned dilution of 2.0%.

**Table 15-1: Mineral Reserve Estimate by Grade (Effective Date: August 31, 2023)**

| Description | Ore | Grade | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | (Mt) | Ni % | Co % | Pd g/t | Pt g/t | Fe % | Cr % | Brucite % |
| **HG Main Zone** | | | | | | | | |
| Proven | 208 | 0.31 | 0.013 | 0.027 | 0.011 | 6.23 | 0.60 | 1.78 |
| Probable | 64 | 0.29 | 0.013 | 0.023 | 0.012 | 6.47 | 0.54 | 1.98 |
| **LG Main Zone** | | | | | | | | |
| Proven | 213 | 0.21 | 0.013 | 0.011 | 0.009 | 6.69 | 0.58 | 1.15 |
| Probable | 368 | 0.18 | 0.013 | 0.011 | 0.009 | 6.82 | 0.53 | 1.03 |
| **HG East Zone** | | | | | | | | |
| Proven | 375 | 0.26 | 0.012 | 0.014 | 0.009 | 5.92 | 0.64 | 2.84 |
| Probable | 148 | 0.25 | 0.012 | 0.009 | 0.007 | 5.83 | 0.63 | 2.87 |
| **LG East Zone** | | | | | | | | |
| Proven | 198 | 0.15 | 0.012 | 0.011 | 0.011 | 7.00 | 0.50 | 0.32 |
| Probable | 141 | 0.15 | 0.011 | 0.012 | 0.010 | 6.54 | 0.47 | 0.60 |
| **Total Crawford** | | | | | | | | |
| Proven | 994 | 0.24 | 0.013 | 0.016 | 0.010 | 6.37 | 0.59 | 1.75 |
| Probable | 721 | 0.20 | 0.012 | 0.012 | 0.009 | 6.53 | 0.54 | 1.41 |
| **Proven + Probable** | **1,715** | **0.22** | **0.013** | **0.014** | **0.009** | **6.44** | **0.57** | **1.61** |

Notes: **1.** $6.29/lb Ni, $11.97/lb Co, $58/t iron ore, $1.16/lb Cr, $891/oz Pd and $759/oz Pt; average metallurgical recoveries of 41% Ni, 11% Co, 53% Fe, 28% Cr, 48% Pd and 22% Pt; marginal processing and G&A costs of $6.10/t milled; a long-term exchange rate of C$1.00 equal $0.76; overall pit rock slopes of 43° to 54° depending on the sector; and a production rate of 120 kt/d. **2.** Mineral reserves include unplanned dilution of 2.0%. **3.** The proven reserves are based on measured resources while probable reserves are based on indicated resources. **4.** All figures are rounded to reflect the relative accuracy of the estimates. Some error in totals may be present due to rounding.

**Table 15-2: Mineral Reserve Estimate by Contained Metal (Effective Date: August 31, 2023)**

| Description | Contained Metal | | | | | | Mt $CO_2$ |
|---|---|---|---|---|---|---|---|
| | Ni (kt) | Co (kt) | Pd (koz) | Pt (koz) | Fe (Mt) | Cr (kt) | Capture |
| **HG Main Zone** | | | | | | | |
| Proven | 641 | 27 | 180 | 74 | 13 | 1,249 | 8 |
| Probable | 185 | 8 | 47 | 24 | 4 | 348 | 3 |
| **LG Main Zone** | | | | | | | |
| Proven | 445 | 27 | 75 | 58 | 14 | 1,226 | 6 |
| Probable | 678 | 47 | 133 | 106 | 25 | 1,961 | 10 |
| **HG East Zone** | | | | | | | |
| Proven | 965 | 47 | 170 | 112 | 22 | 2,418 | 18 |
| Probable | 369 | 18 | 44 | 32 | 9 | 926 | 7 |
| **LG East Zone** | | | | | | | |
| Proven | 295 | 24 | 73 | 67 | 14 | 998 | 1 |
| Probable | 212 | 16 | 53 | 46 | 9 | 659 | 2 |
| **Total Crawford** | | | | | | | |
| Proven | 2,345 | 125 | 498 | 311 | 63 | 5,892 | 33 |
| Probable | 1,444 | 89 | 278 | 208 | 47 | 3,895 | 22 |
| **Proven + Probable** | **3,789** | **215** | **777** | **519** | **110** | **9,787** | **54** |

Notes: **1.** \$6.29/lb Ni, \$11.97/lb Co, \$58/t iron ore, \$1.16/lb Cr, \$891/oz Pd and \$759/oz Pt; average metallurgical recoveries of 41% Ni, 11% Co, 53% Fe, 28% Cr, 48% Pd and 22% Pt; marginal processing and G&A costs of \$6.10/t milled; a long-term exchange rate of C\$1.00 equal \$0.76; overall pit rock slopes of 43° to 54° depending on the sector; and a production rate of 120 kt/d. **2.** Mineral reserves include unplanned dilution of 2.0%. **3.** The proven reserves are based on measured resources while probable reserves are based on indicated resources. **4.** All figures are rounded to reflect the relative accuracy of the estimates. Some error in totals may be present due to rounding.

Proven reserves are based on measured resources while probable reserves are based on indicated resources. All Figures are rounded to reflect the relative accuracy of estimates. Mineral resources reported in Section 14 are inclusive of mineral reserves.

The base case assumes that Crawford will produce the following two concentrates:

- a nickel (Ni) concentrate that contains payable levels of Co and 2E PGE (Pd and Pt)
- an iron (Fe) concentrate that contains payable levels of Fe, Ni, and chromium (Cr).

The base case also assumes that the carbon-capture potential of Brucite contained in Crawford ore will be commercially utilized with $CO_2$ provided by third parties and injected into the process stream where capture will be affected using CNC's proprietary IPT carbonation process.

## 15.2 Reserve Estimation Process Overview

Reserves were estimated using the following process:

- The feasibility study mine design uses the resource block model described in the previous section. This model includes the estimated content of the economic metals nickel, cobalt, iron, chromium, palladium, and platinum as well as the mineral brucite that is responsible for carbon sequestration. The resource block model also includes the estimated content of sulphur and the magnetic susceptibility (MagSus), that are key determinants for the metallurgical recovery of each economic metal.
- The metallurgical recovery of each metal to each of the two concentrates that will be produced was calculated on a block-by-block basis. The net smelter return (NSR) for each block was then calculated from the estimated recovered content of each metal along with CNC's forecast of commercial terms. including long-term metal prices, exchange rate, and payability. As net revenues associated with brucite and carbon sequestration were excluded from individual block values, the calculation can be considered conservative.
- The LG algorithm was employed to define the optimal final pit shell to be used as the basis for a subsequent engineered design. There are no meaningful points of inflection in either revenue generation or costs along the entire continuum to RF100. Consequently, a primary consideration in selecting revenue factor (RF) 65% (RF65) as the basis for the engineered design was the associated waste products (including tailings) approximated available capacity within CNC's current land holdings to the east of the realigned Highway 655 corridor.
- An engineered pit design was produced for the RF65 shell. This design used inter-ramp angles as recommended by the geotechnical consultants and ramps of variable width depending on inclusion of trolley-assist infrastructure and density of traffic.
- Unplanned dilution was applied to the mineral reserve estimate to reflect the possibility of waste being mixed with ore when mining at the mineralized contact.
- A theoretical calculation of the cut-off grade was supported by an iterative investigation, which confirmed the highest project $NPV_{8\%}$ was achieved with the selected NSR cut-off of $9/t.
- Measured resources were classified as proven reserves while indicated resources were classified as probable reserves.
- Inferred resources were treated as waste and not considered in the generation of pit shells. The engineered pit design contains 84 Mt inferred resources with an average NSR 26% lower than that of the mineral reserves.

## 15.3 Net Smelter Return Model

Each block of mineralization within the resource block model has a unique estimate of grade, metallurgical recovery, and concentrate grade. Table 15-3 summarizes the parameters used to calculate the net smelter return (NSR) per tonne. The following should be noted:

- Payability for palladium and platinum is based on a one-unit 2E PGE deduction (i.e., one unit of combined palladium + platinum). Note that for purposes of the deduction, palladium and platinum are treated equally despite differences in recovery and price.

- Iron is priced per tonne of concentrate grading 55% Fe, where all other metals are priced per payable content within the respective concentrates.
- While the current assumption is that cobalt contained within the iron concentrate will not be paid, this represents a potential upside.

**Table 15-3: Net Smelter Return Assumptions**

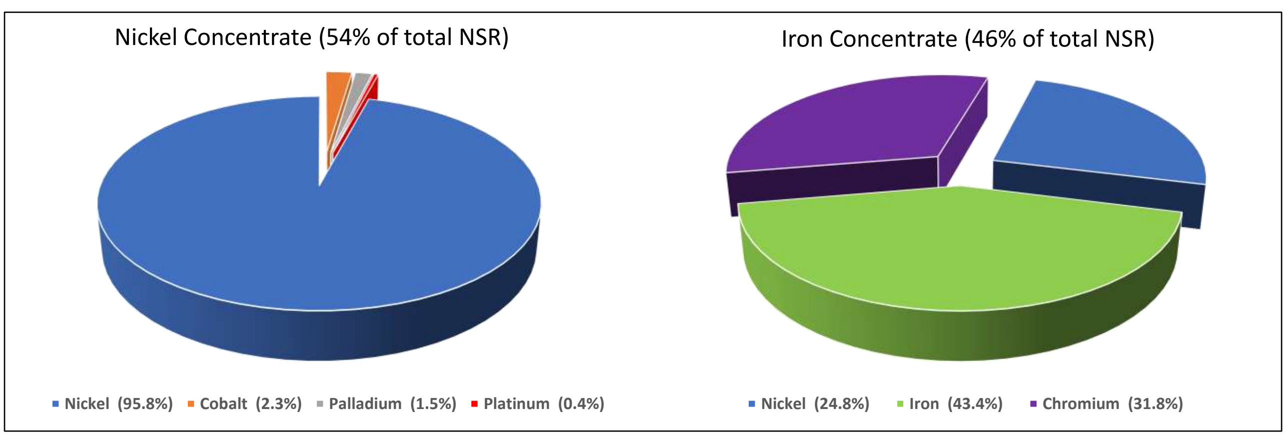
| Concentrate | Metal | Average Grade | Payability % | Price (USD) |
|---|---|---|---|---|
| Nickel Concentrate | Nickel | 34.2% | 91 | $21,000/t |
| | Cobalt | 0.7% | 60 | $10,000/t |
| | Palladium[1,2] | 3.1 g/t | 75.2 | $1,350/oz |
| | Platinum[1,2] | 0.9 g/t | 76.1 | $1,150/oz |
| Iron Concentrate | Iron[3] | 55% | N/A | $89/t |
| | Chromium | 2.6% | 60 | $3,858/t |
| | Nickel | 0.3% | 91 | $21,000/t |
| | Cobalt | 0.1% | 0.0 | n/a |

Notes: **1.** Life-of-mine average payability, based on a one-unit deduction and 4.1 g/t 2E grade. **2.** Quarterly (3 monthly) payability ranges from 42% to 91%. **3.** Scrap iron price of $325/t and payability of 50% equates to price for 55% Fe concentrate of $89/t.

The assumptions given in Table 15-3 and the feasibility study production schedule result in the nickel concentrate generating 54% of total NSR over the life of project. Figure 15-1 illustrates the contribution to overall NSR for each of the concentrates' constituent metals.

There is greater confidence in the global estimate of brucite content and associated carbon sequestration than for the local variation in block values. Additionally, a portion of the carbon sequestration will be utilized by Crawford (to achieve net-zero status and thus avoid paying carbon taxes on fuel) and not be sold to third parties. For these reasons, revenue from carbon sequestration has been excluded from the calculation of block values and these values can therefore be considered conservative.

**Figure 15-1: Contribution to NSR**




Source: CNC, 2023.

## 15.4 Lerch-Grossmann Pit Shells

The Lerch-Grossmann (LG) algorithm is the industry standard tool used to define the limits of an open pit. The design process was initiated by calculating the net value of each block in the model by subtracting estimated costs for mining, processing, and G&A from the NSR of each block. Waste blocks with no NSR value have a negative net value. As the native currency for all costs at Crawford will be the Canadian dollar, that currency has been used in the LG optimization with an assumed long term exchange rate of C$1.00 = US$0.76. Table 15-4 summarizes assumptions used in generating the net value for cells.

Overall slope angles were assigned to the various sectors based on recommendations provided in Section 16. The LG algorithm then selected a ‘cone’ of ore and associated waste stripping that maximizes NPV. By varying the revenue factor (percentage of the long-term metal prices), it was possible to generate higher value nested cones that can be used to identify the optimal development sequence. Figure 15-2 illustrates the progression in total ore and value per tonne of ore of both NSR and EBITDA from RF50 to RF100.

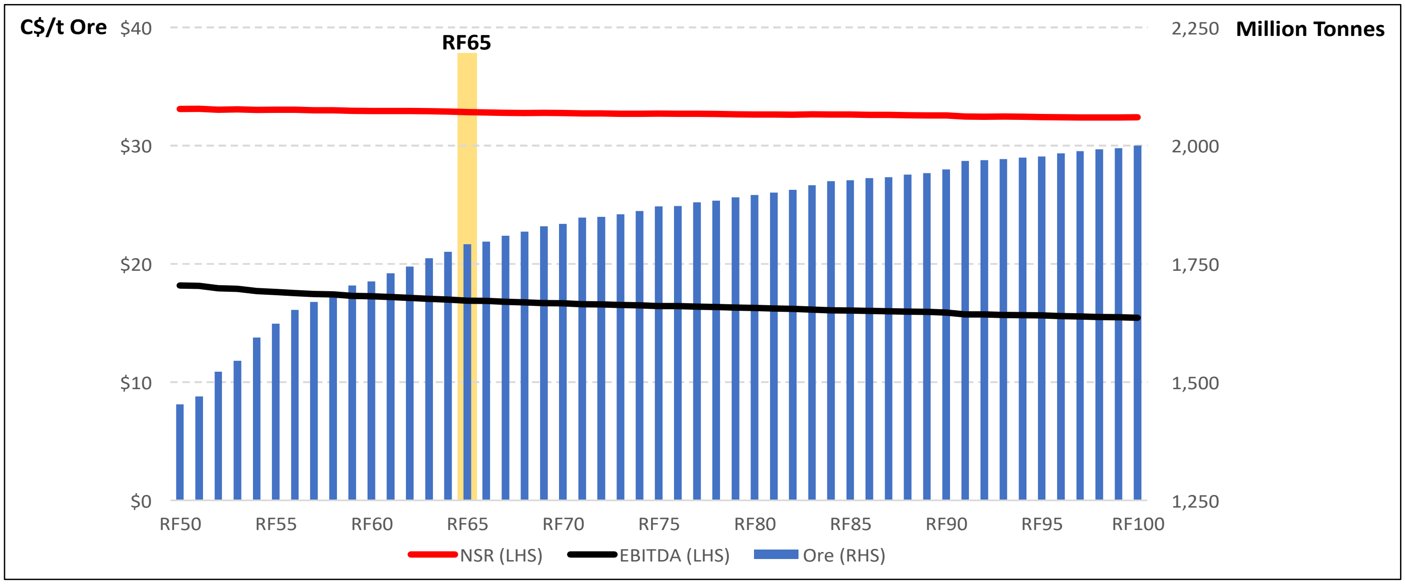
There are no meaningful points of inflection along the continuum. Testing of various alternatives demonstrated that overall project value would be maximized with the largest scope that could be accommodated within the current land position to the east of the realigned corridor for Highway 655. As the waste content of RF65 approximated the total capacity for impoundments including the tailings management facility (TMF) this shell was selected as the basis for the engineered design. Figure 15-3 presents a plan view of RF65 that was selected as the basis for the engineered design.

**Table 15-4: Lerch-Grossmann Cost Assumptions**

| Area | Element | Units | Value C$ | Comment |
|---|---|---|---|---|
| Base Mining Costs | 45 t truck | C$/t mined | 2.76 | 4% of FS total mined |
| | 90 t truck | C$/t mined | 2.13 | 6% of FS total mined |
| | 290 t truck | C$/t mined | 1.35 | 90% of FS total mined |
| Mining Cost Adjustment Factor (MCAF) | 45 t truck | C$/t mined per 15 m bench | 0.24 | |
| | 90 t truck | C$/t mined per 15 m bench | 0.07 | |
| | 290 t truck | C$/t mined per 15 m bench | 0.03 | |
| Rehandle Costs[1] | Higher Value Ore [2,3] | C$/t ore | | $0.08 |
| | Lower Value Ore [2,4] | C$/t ore | | $0.47 |
| Process Costs | Mill | C$/t milled | 6.82 | Applies to all ore |
| | G & A | C$/t milled | 0.98 | Applies to all ore |
| | Water management | C$/t milled | 0.38 | Applies to all ore |
| | Royalties | C$/t milled | 3% NSR | Applies to all ore |

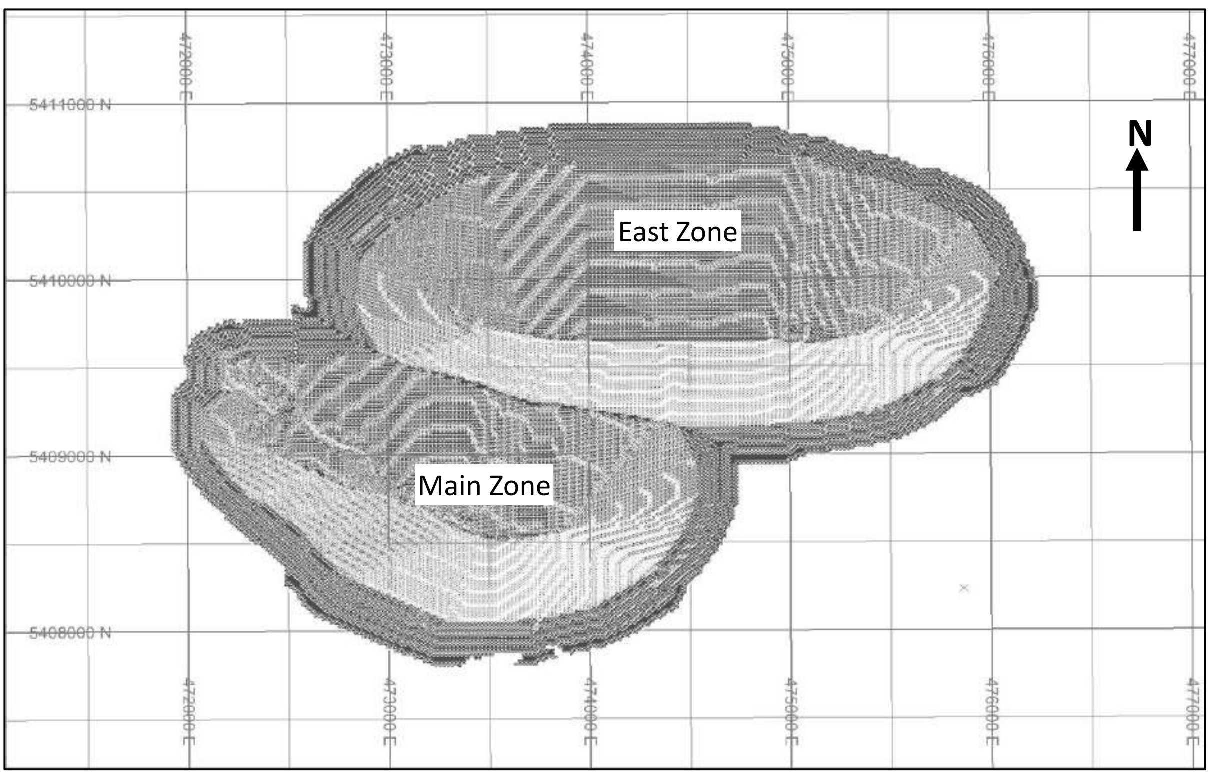
Notes: **1.** Rehandle applied probabilistically by grade bin, based on % of ore expected to be rehandled. **2.** Average one-way haulage distance of 3.1 km and 1.3 km for lower-value and higher-value stockpiles, respectively. **3.** Percentage of higher value ore rehandled ranges from 0% to 53% depending on bin, averaging 19%. **4.** 100% of lower-value ore is rehandled.

**Figure 15-2: Progression of Lerch-Grossmann Nested Shells**




Source: CNC, 2023.

**Figure 15-3: RF65 Basis for Engineered Pit Design**




Source: CNC, 2023.

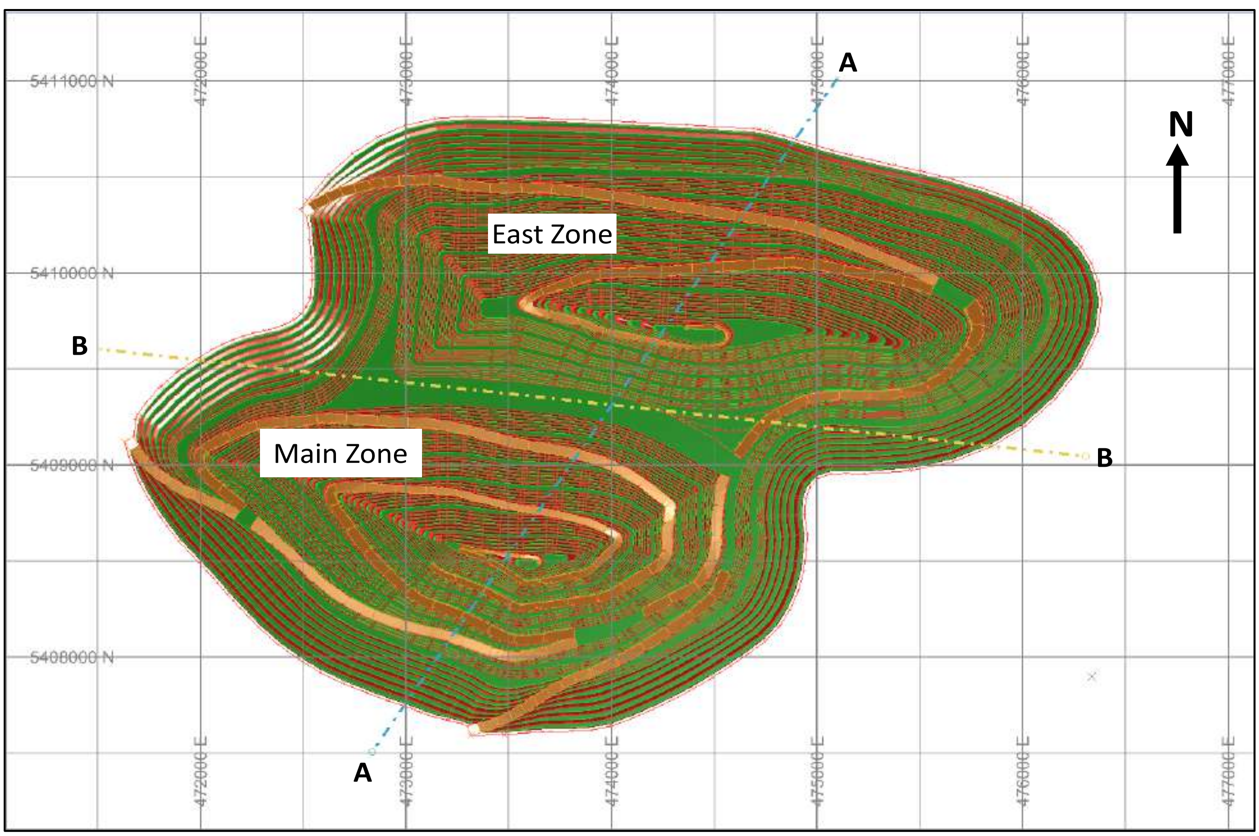
## 15.5 Engineered Pit Design

Pit shells generated using the LG algorithm represent a theoretical design and, while the final walls honour the imposed overall slope constraints, it cannot be considered a practical design as no provision is made for ramps. The engineered design includes ramps of the following widths:

- 35 m for two-way traffic with 290 t trucks where trolley assist will not be utilized (the ramp width approximates 3.5 times the running width of trucks and allows for two lanes of traffic with a ditch and berm)
- 50 m for two-way traffic with 290 t trucks where trolley assist will be utilized (this provides both 5 m for the trolley infrastructure as well as space for a third lane of traffic, in order that any truck stopped under the trolley line can be passed)
- 25 m for areas where low density traffic and/or 90 t trucks will be employed
- 15 m for 'good-bye' cuts at the extreme bottom of ore stages.

Figure 15-4 illustrates the final engineered pit.

**Figure 15-4: Engineered Final Pit Design**



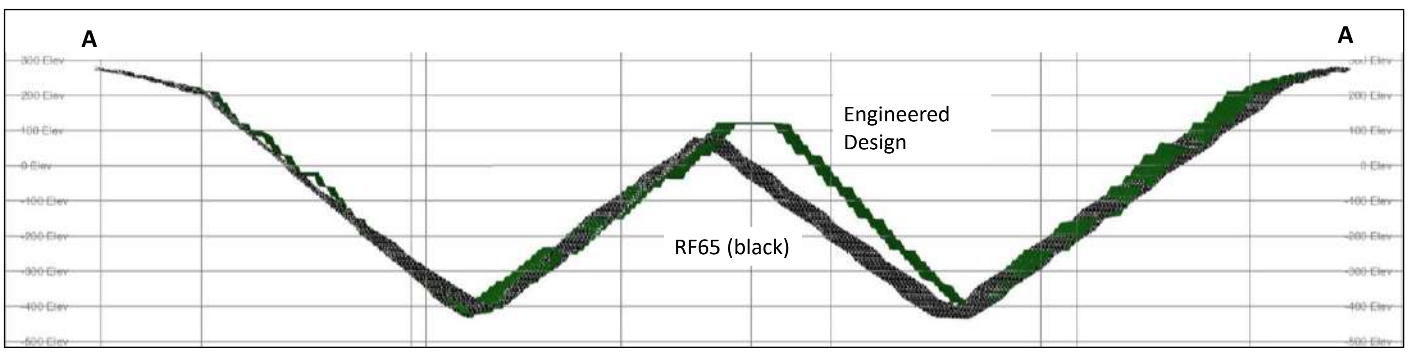

Source: CNC, 2023

Attention is directed towards the 'saddle' separating the Main Zone and East Zone. This has been extended to RL120 (approximately 150 m below the nominal surface elevation) as part of the overall tailings impoundment strategy, which can be summarized as follows:

- Tailings will initially be impounded within the TMF on surface. For the current design criteria, which include a maximum nominal height of the retaining dyke of 17 m and a beach slope of deposited tails of 2.5%, the maximum capacity of this facility is 495 $Mm^3$.
- By the time the TMF is at capacity, mining must be complete within the Main Zone in order that the resulting void may be used to impound tailings.
- The capacity of the Main Zone void below the 'saddle' elevation is 484 $Mm^3$, which is sufficient for all tailings that will be produced while the East Zone remains in production.
- Upon completion of pit operations, tailings produced from processing of low-grade stockpiles will be impounded in the mined out East Zone void. In total, 61% of total tailings will be impounded within the mined-out pits.

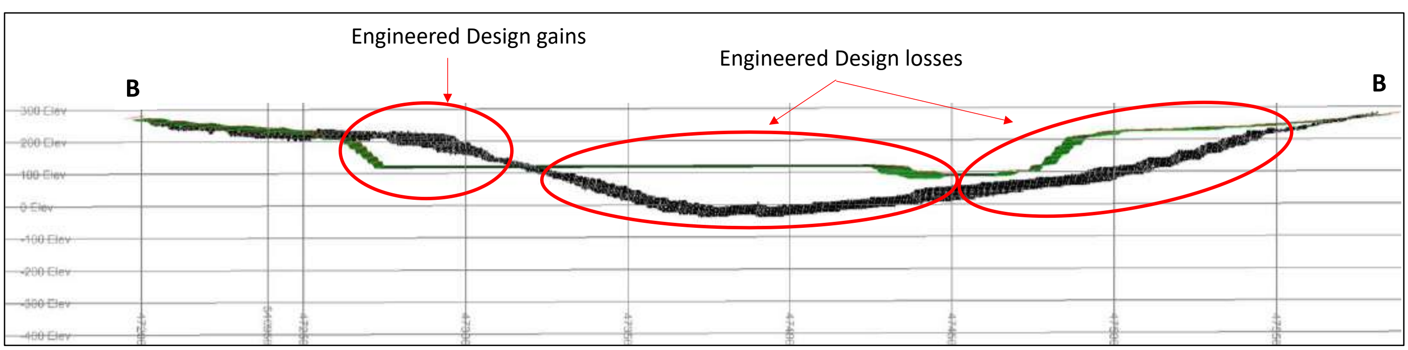
Figures 15-5 and 15-6 provide a cross section (Section line A-A) and longitudinal section (Section line B-B) of the saddle designs for RF65 and the engineered design while Table 15-5 compares the content of the engineered design to RF65.

**Figure 15-5: Cross-Section of RF65 vs. Engineered Design**




Source: CNC, 2023.

**Figure 15-6: Longitudinal Section of RF65 vs. Engineered Design**




Source: CNC, 2023.

**Table 15-5: Comparison of Engineered Design and LG RF65 Shell**

| Description | Undiluted Ore (Mt) | Grade (% NiEq) | Contained NiEq (kt) | Waste (Mt) | Strip Ratio |
|---|---|---|---|---|---|
| RF65-LG | 1,756 | 0.38 | 6,685 | 3,991 | 2.27 |
| Engineered | 1,672 | 0.38 | 6,334 | 4,035 | 2.41 |
| Variance | -4.8% | -0.4% | -5.2% | 1.1% | 6.2% |

The engineered design achieves a mining recovery equal to 94.8% of equivalent nickel (NiEq) contained within the RF65 shell. The 84 Mt of undiluted ore losses are almost entirely associated with the 'saddle'. If it is possible to increase the capacity of the TMF—by raising the dykes, increasing the beach slope, or by other means—it would be possible to reduce losses associated with the 'saddle' and increase mining recovery closer to unity. This opportunity is discussed further in Section 25.

It can also be seen that the engineered design achieves a relatively low planned dilution of 0.4%.

The conversion of cobalt, iron, chromium, palladium, and platinum to NiEq considers both the prices and payability as demonstrated in Table 15-6. For example, one pound of contained cobalt is 1.3 pounds of contained NiEq while one tonne of iron is 18.7 pounds of NiEq.

**Table 15-6: NiEq Conversion**

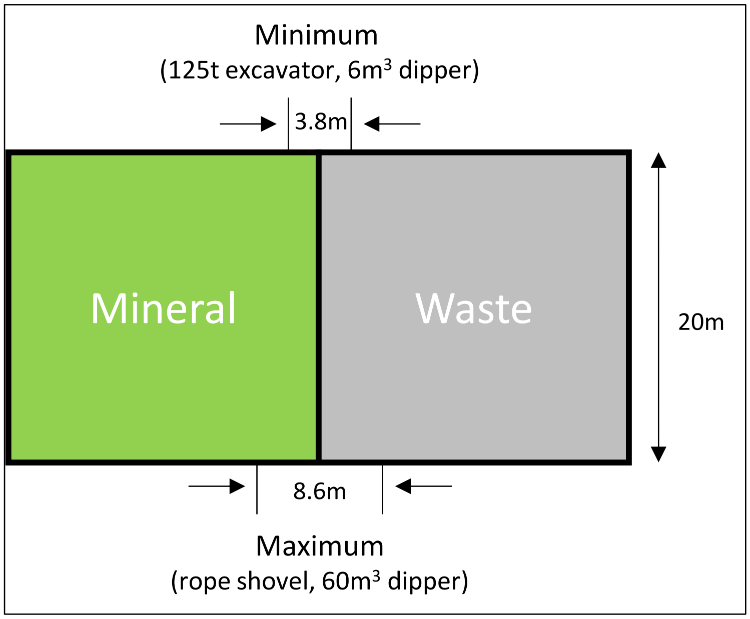
| Metal | Unit | Price ($) | Payability (%) | NiEq Conversion |
|---|---|---|---|---|
| **Nickel** | **pound** | **9.53** | **91** | **1** |
| Cobalt | pound | 18.14 | 60 | 1.3 |
| Iron | tonne | 325 | 50 | 18.7 |
| Chrome | pound | 1.75 | 65 | 0.1 |
| Palladium | ounce | 1,350 | 75.2 | 117.1 |
| Platinum | ounce | 1,150 | 76.1 | 101.0 |

## 15.6 Dilution and Mining Losses

Planned dilution and mining losses have been discussed in the previous section. Unplanned dilution will potentially occur at the contact between ore and waste, which is limited as mineralization at Crawford is massive, disseminated, and continuous. On a typical level in either the Main Zone or East Zone, the mineralized thickness exceeds 100 m and with no interstitial waste.

An allowance has been made for inclusion of barren waste (i.e., zero grade) equating to one half dipper width or 1.9 m to 4.3 m of mineralization at the contact (Figure 15-7).

**Figure 15-7: Unplanned Dilution**




Source: CNC, 2023.

As blasts would be designed to honour the ore-waste contact and technology will be employed to assist the definition of this contact, the allowance for unplanned dilution is considered conservative.

Table 15-7 summarizes the impact of unplanned dilution on reserves.

**Table 15-7: Diluted Reserves**

| Classification | Grades | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | Ore (Mt) | Ni % | Co % | Pd g/t | Pt g/t | Fe % | Cr % | Brucite % |
| **Proven Reserves** | 994 | 0.24 | 0.013 | 0.016 | 0.010 | 6.37 | 0.59 | 1.75 |
| Measured Resources | 983 | 0.24 | 0.013 | 0.016 | 0.010 | 6.43 | 0.60 | 1.77 |
| Dilution | 11 | 0.00 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | 0.00 |
| **Probable Reserves** | 721 | 0.20 | 0.012 | 0.012 | 0.009 | 6.53 | 0.54 | 1.41 |
| Indicated Resources | 698 | 0.21 | 0.013 | 0.012 | 0.009 | 6.74 | 0.56 | 1.45 |
| Dilution | 23 | 0.00 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | 0.00 |
| **Total** | **1,715** | **0.22** | **0.013** | **0.014** | **0.009** | **6.44** | **0.57** | **1.61** |

## 15.7 Cut-Off Value

As Crawford is a polymetallic deposit, with three different metals each contributing more than 10% of total revenue, the cut-off for mine planning will not be expressed as a grade of one or more metals but rather the NSR value of material.

The methodology for determining the NSR value has been described in Section 15.3 and includes the following parameters:

- grades of the various recoverable economic metals
- metallurgical recovery
- commercial terms, including the metal price and percentage payable.

As is normal for open pit designs, the calculation of cut-off values excludes mining costs and includes only the following marginal costs:

- any incremental haulage costs from the pit rim that would be incurred for re-handling low-grade ore (as material at the marginal cut-off grade would initially be stockpiled)
- milling costs
- costs associated with water management and tailings management
- general and administrative costs.

The cut-off calculation also accounts for royalties; including the 2% NSR royalty and 2.5% net profits interest (NPI) royalty, with the latter being simplified as an equivalent 1% of NSR.

Table 15-8 illustrates that the theoretical cut-off grade equates to C$9.21/t.

**Table 15-8: Cut-Off Calculation**

| Area | Units | Value (C$) |
|---|---|---|
| Milling | C$/t milled | 6.82 |
| General and Administration (G&A) | C$/t milled | 0.98 |
| Water Management | C$/t milled | 0.38 |
| Low-grade Rehandle | C$/t ore rehandled | 0.75 |
| **Subtotal** | | **8.94** |
| Royalties | %NSR | 3% |
| **Theoretical Required NSR** | **C$/t milled** | **9.21** |

Note: Approximation includes 2% NSR and 2.5% NPI royalty.

The following related to the theoretical calculation should be noted:

- As the native costs for all inputs to the calculation are in Canadian dollars, the same currency has been used for expressing the cut-off value
- Costs associated with the TMF and water management are front-loaded, so marginal material treated during the later years of project life would have a lower associated cost.
- Some of the lowest grade material would be treated as run-of-mine material and therefore not accrue the rehandle costs.
- There is a limited tonnage of lower value mineralization, specifically less than 4 Mt in the increment between C$9.00 and the theoretical C$9.21.

As a result of these factors, the cut-off has been rounded to C$9/t. The validity of this value was then tested by re-running the life-of-mine production schedule using cut-off values of C$8/t and C$10/t and comparing a basket of key economic metrics against those generated by the C$9/t cut-off.

The following results confirm the C$9/t cut-off, in aggregate, is the optimal choice:

- C$8/t NPV8% 0.15% lower / IRR 0.06% lower / simple payback 0.05% shorter
- C$10/t NPV8% 0.20% lower / IRR 0.08% lower / simple payback 0.16% longer.

## 15.8 Classification of Mineral Reserves

Diluted measured resources contained within the engineered design and above the cut-off value have been classified as proven mineral reserves. Diluted indicated resources contained within the engineered design and above the cut-off value have been classified as probable mineral reserves.

## 15.9 Factors That May Impact the Statement of Mineral Reserves

As discussed in Section 15.7, reserves are the measured and indicated resources with an NSR value that exceeds marginal costs of approximately $9/t. Accordingly, factors that may impact the statement of mineral reserves are those that either increase the marginal cost or decrease the NSR and include the following:

- The prices of various metals produced, with a 10% reduction in the basket of metal prices reducing the tonnage of reserves by 19 Mt or 1.1%.
- The Canadian dollar exchange rate, as metal prices are set in United States dollars while costs are denominated by Canadian dollars. A 10% strengthening of the Canadian dollars, from $0.76 to $0.84, would reduce the tonnage of reserves by 17 Mt or 1.0%.
- Prices for the various goods and services consumed at site, with a 10% increase in total site costs also reducing the tonnage of reserves by 17 Mt or 1.0%.

# 16 MINING METHODS

## 16.1 Hydrogeological Considerations

A 3D numerical groundwater model has been constructed with the widely used FEFLOW code, incorporating overburden modelling results completed using LeapFrog and bedrock data from diamond drilling and hydrogeological (bedrock packer and overburden slug) testing campaigns. The bedrock packer testing results available for the bedrock in the vicinity of the planned open pits demonstrate a trend of decreasing hydraulic conductivity with increased depth below top of rock surface. Except for areas influenced by the regional "CUC" fault, no unique relationship has been identified between the bedrock packer testing hydraulic conductivity results and the location of mapped structures.

The numerical groundwater modelling was reported to CNC for the purpose of estimating dewatering volumes during the proposed pit progression schedule. Results of the model show that pit dewatering can be achieved using conventional dewatering means. For the Main Zone pit, dewatering rates range from about 1,850 $m^3$/d, estimated during Years 1 to 5 (development Stage 1 of the mine life) to 9,963 $m^3$/d, estimated during Years 8 to 9 (development stage of the mine life). For the East Zone pit, dewatering rates range from about 1,690 $m^3$/d, estimated during Year 19 (development Stage 3 of the mine life) to 16,983 $m^3$/d (estimated during Years 1 to 3 (development Stage 1 of the mine life).

Sensitivity analyses were completed to assess the effects of anisotropy in the overburden and to provide conservative estimates for inflow to the Main Zone pit.

The groundwater model is also used to:

- assess pore water pressures under certain pit slope conditions, which are discussed further in Section 16.2, Geotechnical Considerations
- consider the potential influence of nearby infrastructure according to the existing planned site layout, which is discussed in Section 18.8.3, Hydrogeology.

Slope angles for the pit and impoundments assume a similar groundwater regime as experienced at existing operations or more advanced projects within the Abitibi Region of Ontario and Quebec.

## 16.2 Geotechnical Considerations

### 16.2.1 Soil Geotechnical

#### 16.2.1.1 Surficial Geology

The project lies within the Abitibi Uplands of the James Physiographic Region of the Canadian Shield (Bostock, 1970). The upland surface is characterized by low ridges and knobs of Archean rock, predominately covered by flat-lying glacial sediments. The thick cover of glaciolacustrine sediments gives rise to the region's informal name, the "Clay Belt." The overburden stratigraphy is complex and consists of seven map units. Three units, older till, Barlow-Ojibway formation, and Cochrane till, are extensive sheetlike deposits. The remaining units, consisting of glaciofluvial, aeolian, and organic deposits, have less extension and irregular distribution. Figure 16-1 depicts the layout of the main mine infrastructure on the regional surficial geology. The open pit, according to the surface geology shown in the Figure, involves glaciolacustrine fine-grained deposits (Barlow-Ojibway formation), upper till (Cochrane formation) and organic deposits (peat and black muck).

The Barlow-Ojibway formation is probably the most extensive unit. This formation includes varved stratification of clay, silty clay, and silt. The irregular thickness of the varved sediments range from thick to very thin.

Most glaciofluvial deposits are present west of Highway 655. These deposits include sediments which grade from coarse quartz sand to coarse gravel containing boulders. Included with these deposits is an esker, the approximate location of which is shown on Figure 16-2. On the east side of Highway 655, the fine sands within this unit interfinger with glaciolacustrine clay to form thick to thin varves within the Barlow-Ojibway formation.

Organic deposits consist typically of partially saturated peat with roots and decomposed wood (known as fibrous peat).

#### 16.2.1.2 Geotechnical Characteristic of the Overburden Soils

The overburden soils at the mine site were characterized at a feasibility study level based on the field investigation conducted during the Q3 2021 campaign. In Q1 2023, an additional drilling program was carried out to further characterize areas that were under-represented in the initial Q3 2021 program.

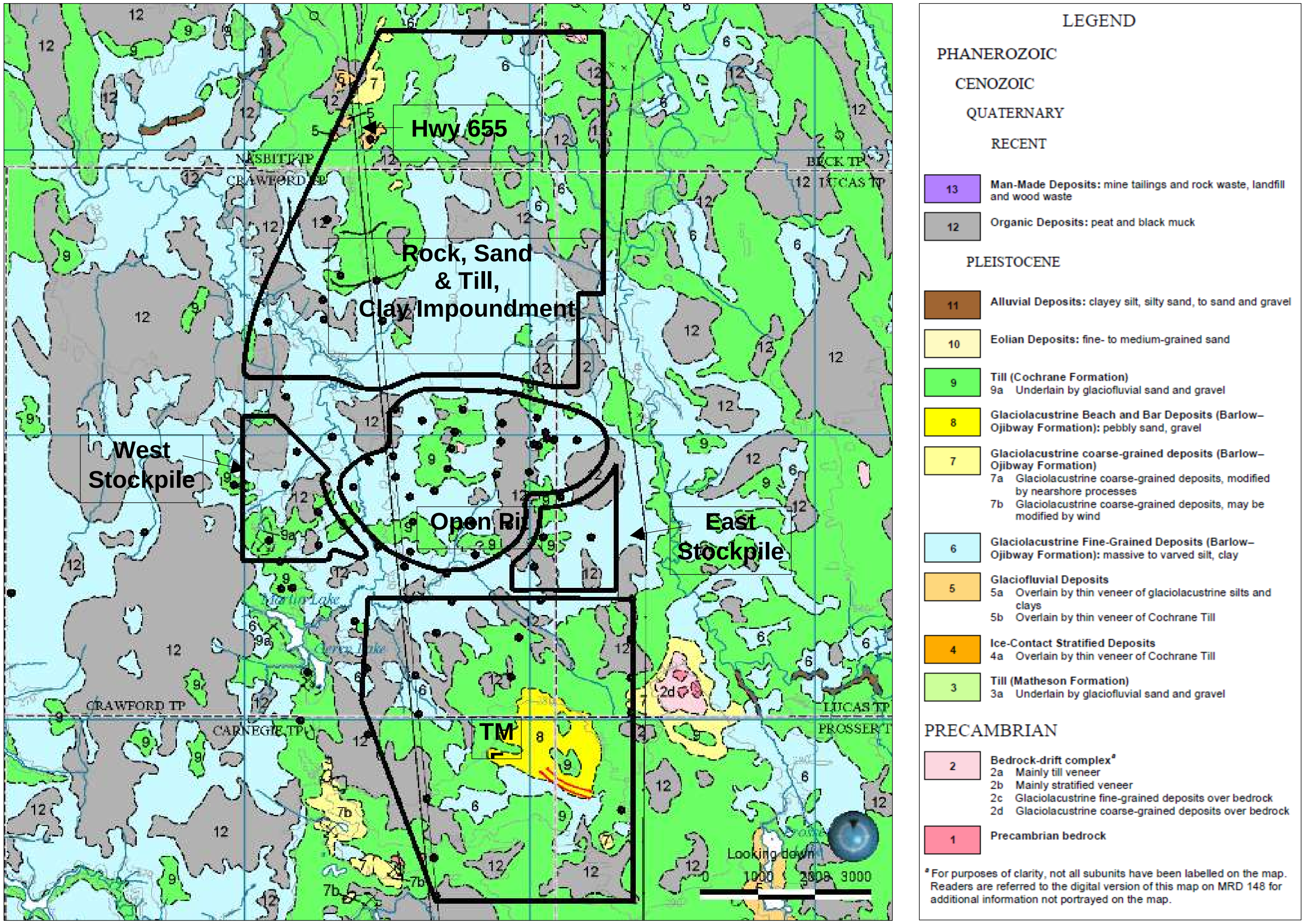
The geotechnical database for the soils in the mine site area, derived from the investigations conducted during the Q3 2021 campaign and the infill drilling program of Q1 2023, comprises 79 sonic drillholes (3,164 m), most of which extended to bedrock, and 55 cone penetration tests (CPTs) (1,100 m) performed to refusal (typically on dense granular soils).

The distribution of the drillholes and CPTs in the vicinity of the mine facilities was as follows:

- 29 drillholes and 16 CPTs in the open pit area (9.9 $km^2$)
- 21 drillholes and 14 CPTs in the tailings management facility area (22.9 $km^2$)
- 8 drillholes and 9 CPTs in the WRD1 complex area (17.1 $km^2$)
- 11 sonic drillholes and 12 CPTs in the west low-grade stockpile area (4.6 $km^2$)
- 4 sonic drillholes and 3 CPTs in the east low-grade stockpile area (2.4 $km^2$)
- 2 drillholes in the plant area
- 4 drillholes in the planned location for the bulk sample.

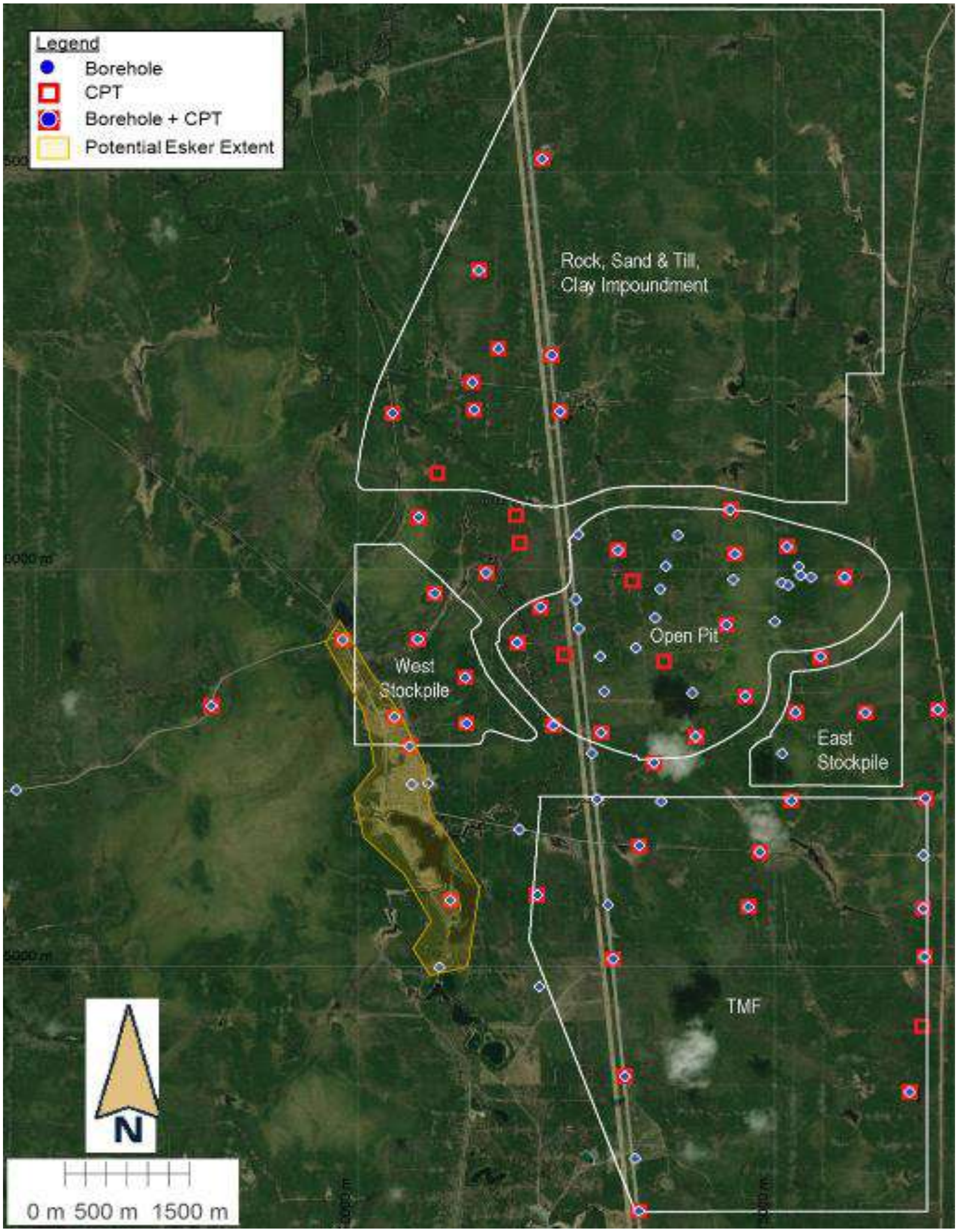
Figure 16-2 displays the project area with the location of drillholes and CPT from the Q3 2021 and Q1 2023 geotechnical campaigns, overlaid on the proposed site layout.

**Figure 16-1: Regional Surficial Geology and Main Infrastructure of the Crawford Project**




Note: Black dots represent the location of boreholes from the Q3 2021 site investigation. Source: Ontario Geological Survey 2005. Three-dimensional modelling of overburden in the Timmins – Kirkland Lake region: Discover Abitibi Initiative; Ontario Geological Survey.

**Figure 16-2: Location of Drillholes and CPT Probes from Q3 2021 and Q1 2023 Geotechnical Campaigns, Overlaid on the Proposed Site Layout**

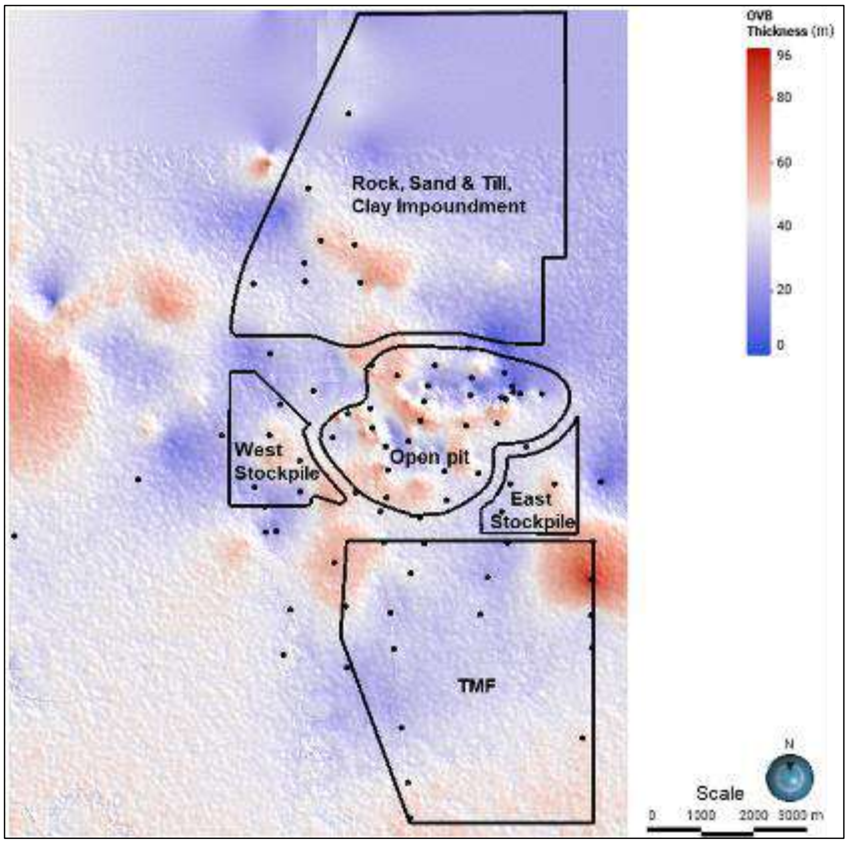



Source: SRK, 2023.

#### 16.2.1.3 General Stratigraphy and Geotechnical Conditions

Based on the data from the Q3 2021 field investigation and laboratory test results, the overburden (soil) thickness in the mine site area is shown on Figure 16-3, as interpreted by Leapfrog software. The thickness of the overburden approaches its maximum, close to 84 m, in the central portion of the proposed Main Zone pit. Conversely, the overburden is generally thinnest in the East Zone pit.

**Figure 16-3: Interpreted Overburden Thickness based on Data from Q3 2021**




Source: SRK, 2023.

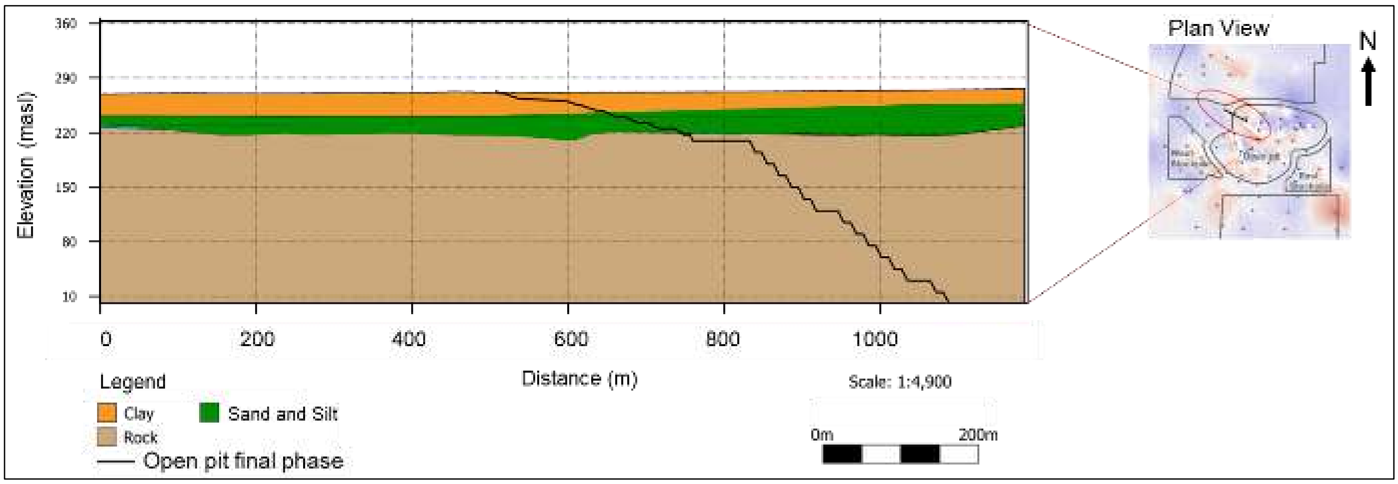
Based on the field program, the following soil types, listed in descending stratigraphic order, were identified and are illustrated in Figure 16-4 for a specific cross-section:

- Organic soil, which consists mainly of fibrous peat. This layer blankets much of the project area and extends to a depth of up to 4.0 m with an average thickness of 0.9 m.
- Glaciolacustrine clay, which is generally found immediately below the organic soil and typically ranges in thickness from 5 to 30 m. Two types of clay were encountered: stiff brown clay and soft grey clay. The brown clay overlies the grey clay. Varves were not clearly identified during the FS site investigation program.

- Sand and till, which is generally found below the clay, on top of the bedrock, and typically ranges in thickness from 0.5 to 30 m and can achieve depths around 70 m.
- Regarding the esker that extends in a northwest-southeast direction, boreholes reached a maximum depth of 40 m. Figure 16-2 above provides a very rough approximation of the esker's location and extent. The northern and southern boundaries of the esker were determined based on available data from CPTs and boreholes from the site investigations conducted in Q3 2021 and Q1 2023. The eastern and western boundaries were inferred from topographical features and the presence of water bodies.

Not all soil types are present in all areas of the mine site.

**Figure 16-4: Cross-Section Highlighting the Overburden Interpretation with the Final Pit Walls Superimposed**



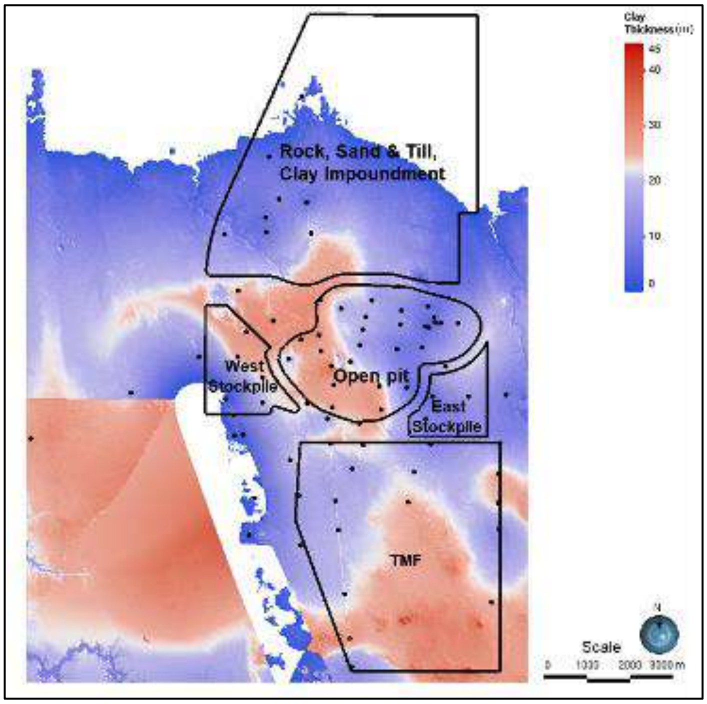

Source: SRK, 2023.

On average, the groundwater surface at the mine site is approximately 2.5 m below the ground surface. The brown clay encountered below the organic soil is over-consolidated and has a thickness of about 10 m. The grey clay underlying the brown clay is lightly over-consolidated to normally consolidated and extends to a depth of up to 30 m below the ground surface. The total thickness of the clay deposits at the mine site is illustrated in Figure 16-5. This Figure is based on data from the Q3 2021 field investigation and Leapfrog software. There are two white zones on Figure 16-5. The large white zone at the top is an area where, due to data limitations, the Leapfrog model was unable to declare the presence and/or depth of the clay. Similarly, the smaller white zone in the lower part of the Figure shows an area within which the Leapfrog model generated a different shape to the esker deposit and was unable to declare the presence and/or depth of the clay.

In terms of strength, the saturated, grey, normally consolidated clay is the weakest unit within the overburden materials. This clay is not sensitive, and the 30$^{th}$ percentile of the peak undrained shear strength ratio obtained from all CPT results is $(s_u/\sigma'_{v0})_{Peak}$ equal to 0.30. Table 16-1 summarizes the average geotechnical properties of the grey clay based on laboratory testing.

The sand and till materials found beneath the clay exhibit similar geotechnical properties according to both field and laboratory tests. These two types of soils were grouped into a single geotechnical unit for the PF geotechnical analysis. Table 16-2 summarizes the average geotechnical properties of the sand and till unit based on laboratory tests.

**Figure 16-5: Interpreted Clay Thickness based on Data from Q3 2021**




Source: SRK, 2023.

**Table 16-1: Average Properties of the Saturated Grey Clay**

| USCS Classification | CF (%) | $w$ (%) | $w_L$ (%) | $w_P$ (%) | $G_s$ (-) | $k_v$ (m/s) | $C_c$ (-) | $\varphi'$ (°) |
|---|---|---|---|---|---|---|---|---|
| CL | 40 ($\sigma = 18.5$) | 28 ($\sigma = 9.6$) | 32.5 ($\sigma = 8.7$) | 18 ($\sigma = 3.4$) | 2.75 | 2E-10 | 0.18 ($\sigma = 0.09$) | 27 ($\sigma = 2.2$) |

Notes: CF – clay fraction; $w$ – water content; $w_L$ – liquid limit; $w_p$ – plastic limit; Gs – specific gravity; $k$ – hydraulic conductivity; $C_c$ – compression index; $\varphi'$: friction angle. Standard deviation in parenthesis. * Value represents the 30th percentile of the friction angles determined using laboratory and field tests.

**Table 16-2: Properties of the Sand and Till**

| USCS Classification | CF (%) | $w$ (%) | $w_L$ (%) | $w_P$ (%) | $G_s$ (-) | $k_v$ (m/s) | $\varphi'$ (°) |
|---|---|---|---|---|---|---|---|
| SM | 27 ($\sigma = 16.4$) | 14 ($\sigma = 5.8$) | 25.5 ($\sigma = 9.3$) | 16.8 ($\sigma = 5.4$) | 2.69 ($\sigma = 0.03$) | 2.5E-5 | 32.1* ($\sigma = 4.8$) |

Notes: CF – clay fraction; $w$ – water content; $w_L$ – liquid limit; $w_p$ – plastic limit; Gs – specific gravity; $k$ – hydraulic conductivity; $C_c$ – compression index; $\varphi'$: friction angle. Standard deviation in parenthesis. * Value represents the 30th percentile of the friction angles determined using laboratory and field tests.

#### 16.2.1.4 Trafficability

The overburden soils will not be trafficable for normal mining equipment unless a waste rock layer at least 1.5 m thick is placed over them. The thickness of the waste rock layer may be reduced depending on the actual conditions (stiffness and strength) of the subgrade and the equipment type and size. The mechanical characteristics of the subgrade can be improved through compaction to reduce the required thickness of waste rock. Other materials that are placed on the subgrade and under the waste rock require specified compaction. Water infiltration into the subgrade can impact its stiffness and strength, potentially undermining the trafficability of structures constructed with waste rock. This circumstance calls for a strict, well-designed, and appropriate water management system to maintain the operational integrity of these structures.

#### 16.2.1.5 Mining

The characteristics of the clay material at the Crawford project will influence the selection of mining equipment. This soft, sticky material requires a careful strategic choice of equipment to optimize performance and efficiency. Emphasizing wear resistance, traction, and the ability to manage sticky material will be critical components in the operational planning phase.

Excavating normally consolidated, soft, saturated clays, and sand and till layers, with a combined thickness nearing 60 m and an average of approximately 35 m, presents technical challenges. The inherent weakness of the saturated clay soils could potentially affect the structural integrity of the overburden slopes within the pit. Therefore, recommended slope angles should be maintained strictly to uphold stability and operational safety. In addition, monitoring systems for detecting pit overburden slope displacements are essential to identify and facilitate the timely execution of preventative measures to preserve pit stability.

The high degrees of saturation within the clay, and sand and till layers add an additional layer of complexity to the stability of the pit overburden slopes. A comprehensive dewatering strategy is required to mitigate the risk of slope instability within the pit overburden slopes. Moreover, if the sand and till layers potentially connect to other layers carrying significant amounts of water, like eskers, factors such as seepage forces or internal erosion could destabilize the pit's overburden slope. This potential issue underscores the importance of a carefully devised dewatering strategy and continual monitoring throughout the operations.

The challenging ground conditions can also reduce excavation rates and overall productivity. However, the development of effective operational strategies can mitigate these impacts. These strategies might include methods to enhance the rate of excavation or innovative ways to manage equipment efficiently.

Environmental impacts cannot be overlooked. Managing large volumes of water and fine-grained materials during excavation can potentially lead to significant environmental repercussions if not handled correctly. Therefore, integrating environmentally friendly practices throughout all stages of the operation - from planning to execution and throughout the lifespan of the pit overburden slopes - is a key factor in the overburden excavation.

In the face of these challenges, it is evident that a high degree of technical expertise, paired with extensive planning, is required to excavate the open pit overburden using practices that are safe, efficient, and environmentally responsible.

#### 16.2.1.6 Slope Design

Stability criteria for the overburden pit slope stability analysis were developed. Stability analyses were conducted on simplified cross-sections through the overburden, considering varying thicknesses of the clay and sand and till layers with the groundwater at the ground and slope surface, for both short-term (during excavation) and long-term (pit overburden slopes) conditions. Pseudo-static stability assessments of the pit overburden slopes were conducted considering the project area's seismicity reported in the National Building Code of Canada. Simplified displacement analyses were performed with the peak ground acceleration relevant to the project area.

Based on these analyses, the maximum overall pit slopes in the clay materials should be 6H:1V, while the maximum overall pit slopes in the sand and till layer should be 5H:1V. Selection of bench slopes, widths, and heights for the excavation of the overburden materials must align with these overall slopes. Specifically, for excavation in the clay, a height of 5 metres between benches with a slope not exceeding 2H:1V, and bench widths of 20 m should be used. Similarly, in the sand and till layer, a height of 5 m between benches with a maximum slope inclination of 3H:1V, and bench widths of 10 m should be applied.

Assessment of detailed deformations of the pit's overburden slopes under seismic loads, specific to the site's seismicity, should be considered in future project phases.

### 16.2.2 Rock Geotechnical

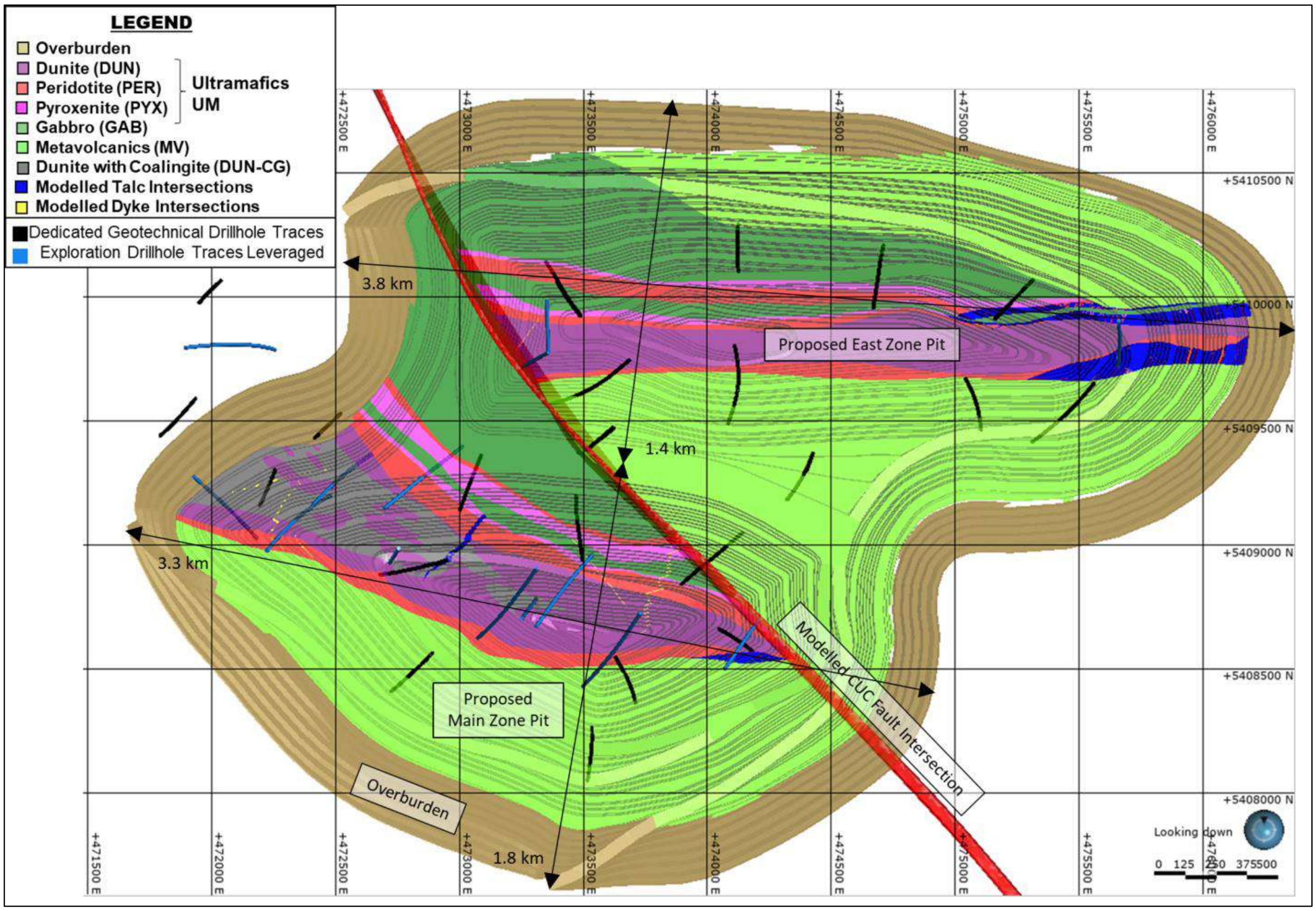
The Crawford project comprises two large proposed open pits, the East Zone pit is approximately 3.8 km long and 1.4 km wide, the Main Zone pit is approximately 3.3 km long and 1.8 km wide in rock. These pits are designed to mine a mineralized ultramafic package, namely the Dunite (DUN) unit, with Gabbro and Metavolcanics units comprising the hangingwall and footwall units. This lithological assemblage is offset by a regional strike slip fault, the CUC Fault, which runs along the west side of the East Zone pit, through the proposed saddle between the pits, to the east side of the Main Zone pit, as seen in Figure 16-6.

During exploration a preliminary model for the CUC Fault was created to understand the impact on the mineralization. Dedicated geotechnical drilling was not undertaken to support the PEA study and exploration drillholes were confined to the extents of the mineralized package as well as the immediate hangingwall and footwall. To support the geotechnical characterization of the rock that will comprise the proposed pit slopes, detailed geotechnical information on small- and large-scale structural orientations, rock mass quality and strength were required across the project site. In support of this, the following work was undertaken:

- Dedicated geotechnical logging and orientation information was obtained from 22 geotechnical specific holes: 10 drillholes in the East Zone pit, 8 drillholes in the Main Zone pit and 4 drillholes northwest of the proposed Main Zone pit completed during the feasibility study field investigation, undertaken during the winter 2021/2022 field season. These drillholes are displayed with black traces in Figure 16-6 as well as additional exploration drillholes that were leveraged in the evaluation of the slope areas.
- Laboratory testing of representative lithology/alteration samples of intact rock and joint strengths were taken from the dedicated geotechnical drillholes.

- Additional orientation data was obtained from acoustic televiewer (ATV) surveys completed on the accessible dedicated geotechnical drillholes (18) as well as 14 selected geological drillholes, displayed with light blue traces in Figure 16-6.
- A review of core from geological drillholes, targeting areas between dedicated geotechnical drillholes, was carried out.

**Figure 16-6: Plan View of the Proposed East Zone and Main Zone Pits Displaying the Location of the Dedicated Geotechnical Drillholes and the Exploration Drillholes where ATV Surveys were Undertaken**




Source: SRK, 2023.

The dedicated geotechnical drillholes were geotechnically logged using oriented core techniques, collecting detailed parameters that included joint conditions, rock mass and major structure description, minor structure orientation and others. Televiewer surveys and corresponding structural picking were attempted to be carried out on all dedicated geotechnical drillholes. However, due to ground instability and issues with access, four of the dedicated geotechnical drillholes were not able to be surveyed. Selected exploration drillholes were also surveyed, as summarized in Table 16-3.

**Table 16-3: Summary of Dedicated Geotechnical Drillholes and Downhole Geophysical Surveys**

| Description | Drillholes with Detailed Geotechnical Logging | Drillholes with Successful Downhole Geophysical Surveys |
|---|---|---|
| East Zone Pit | 10 drillholes (~5,296 m) | 8 drillholes (1) |
| Main Zone Pit | 8 drillholes (~4,275 m) | 7 drillholes (1) |
| Northwest of Main Zone Pit | 4 drillholes (~1,503 m) | 3 drillholes (2) |
| Exploration Drillholes | N/A | 14 drillholes<br>(3 - East Zone pit, 9 - Main Zone pit, and 2 - NW of Main Zone pit) |
| **Total** | **22 (~11,074 m)** | **32 drillholes** |

Notes: **1**. Drillholes were not able to be surveyed with downhole geophysics because of boreholes stability. **2.** Drillhole was not able to be surveyed with downhole geophysics because of site access.

In the slope areas, the lithological model developed by Caracle Creek uses the lithological information logged by CNC personal from the dedicated geotechnical drillholes.

The modelled sulphide mineralized orebody dips steeply towards the South at approximately 75-90° in the East Zone pit and nearly subvertical towards the south in the Main Zone pit. It should be noted that the DUN in the Main Zone pit comprises modelled sectors with coalingite (DUN-CG), modelled by CNC, as presented in Table 16-4, and shown in Figure 16-6, which is susceptible to degradation processes when exposed to the atmosphere. In addition, Talc zones modelled by Caracle Creek are intersecting the eastern side of the East Zone and Main Zone planned pits as well as the northwest side of the Main Zone pit. Finally, occasional narrow (< 10 m) lamprophyre dykes were modelled by Caracle Creek in the lithological model that cut through the entire geological assemblage in the Main Zone pit only.

**Table 16-4: Geological Units in the East Zone and Main Zone Pits**

| Proposed Pit & Location | | Modelled Lithological Units | Modelled Coalingite | Modelled Alteration Units (1) |
|---|---|---|---|---|
| East Zone Pit | Footwall | Gabbro (GAB) and Metavolcanics (MV) | - | - |
| | Mineralization / Ultramafic (UM) | Peridotite (PER); Pyroxenite (PYX); and Dunite (DUN) | - | MgSerp and Talc |
| | Hangingwall | Metavolcanics (MV) | - | - |
| Main Zone Pit | Footwall | Gabbro (GAB) | - | - |
| | Mineralization / Ultramafic (UM) | Peridotite (PER); Pyroxenite (PYX); and Dunite (DUN) | Yes | Unweathered FeSerp, Weathered FeSerp, MgSerp and Talc |
| | Hangingwall | Metavolcanics (MV) | - | - |

Notes: 1. MgSerp: Magnetite-rich serpentinite/brucite, FeSerp: Iron-rich serpentinite/brucite.

#### 16.2.2.1 Structural Model

Following the completion of the geotechnical drilling program, a detailed structural geology study was undertaken as a part of the feasibility study. The deposit-scale structural geometries were modelled using regional to sub-regional scale gravity and magnetic data, drillhole data from CNC's geological database, dedicated geotechnical drillholes, and acoustic televiewer surveys.

The structural modelling identified three dominant fault patterns across the Crawford area, with the mineralized zones themselves being separated by a major strike slip fault structure called the CUC Fault. The CUC Fault strikes approximately northwest-southeast, and is vertical or near vertical, with an estimated damage zone true width of up to 26 m. An example of the rock retrieved from a drillhole that pierced the modelled CUC Fault is presented in Figure 16-7.

**Figure 16-7: Example Image of Drill Core from the Modelled CUC Fault Core and Damage Zone**


Source: SRK, 2023.

The fault patterns that occur in the rest of the fault system, in the proposed open pit areas, were analyzed by plotting oriented datasets with the highest confidence on stereonets, with a focus on structures that have larger apertures.

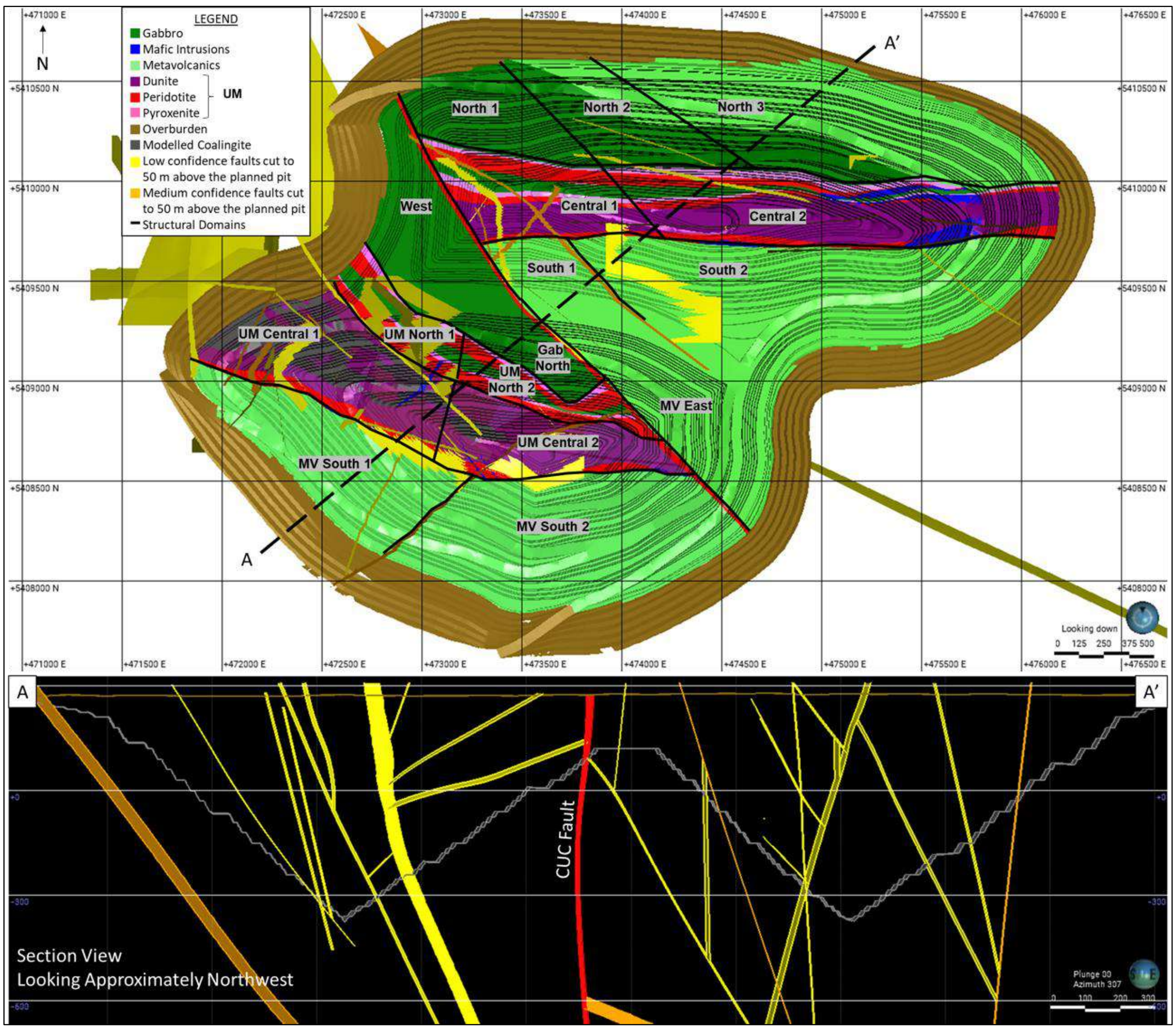
The identified patterns include the following:

- steep broadly northwest-southeast striking structures (broadly CUC Fault parallel, and in places splaying from the CUC Fault)
- moderate to shallowly dipping faults dipping either northeast or southwest

- steep faults striking towards the north or northeast in the Main Zone pit and east in the East Zone pit; this difference in pattern is a key difference between the two zones.

These patterns, displayed on the proposed pits in Figure 16-8, are consistent with those expected to occur in a strike slip system, with steep near vertical features splaying from one another, and more moderately dipping structures accommodating movement between them. It should be noted that the structural model is currently based on widely spaced drillhole data (>300 m in slope areas) and no surface exposure.

**Figure 16-8: Plan View of 3D Fault Model, Structural Domains, and Proposed Pit, showing Section Location and NW Facing Section, Perpendicular to CUC Fault, Displaying Overall Pattern of Strike Slip Fault System**




Source: SRK, 2023.

The structural model is based on widely spaced drillholes and no surface exposure and therefore is low confidence, especially in the upper areas of the pit.

#### 16.2.2.2 Geotechnical Model

The geotechnical model is based on the lithological and alteration models developed by Caracle Creek, SRK interpreted fault network, and rock mass specific parameters. Using the litho-structural model as a framework, an analysis of geotechnical data was undertaken for the rock masses identified in the project area. The main assessed parameters included the following:

- rock quality designation (RQD)
- fracture frequency per metre (ff/m)
- empirical field estimates of intact rock strength (IRS, ISRM-based system)
- field testing of rock strength (point load index test, PLT) and laboratory testing of strength (uniaxial compressive strength, triaxial)
- joint shear strength (direct shear testing of natural joints)
- foliation orientation (in applicable units)
- rock mass ratings ($RMR_{89}$).

Upon review, the modelled lithological units were determined to adequately represent the large-scale geotechnical units across the project site. These units then needed to be further refined to capture weak areas resulting from the following:

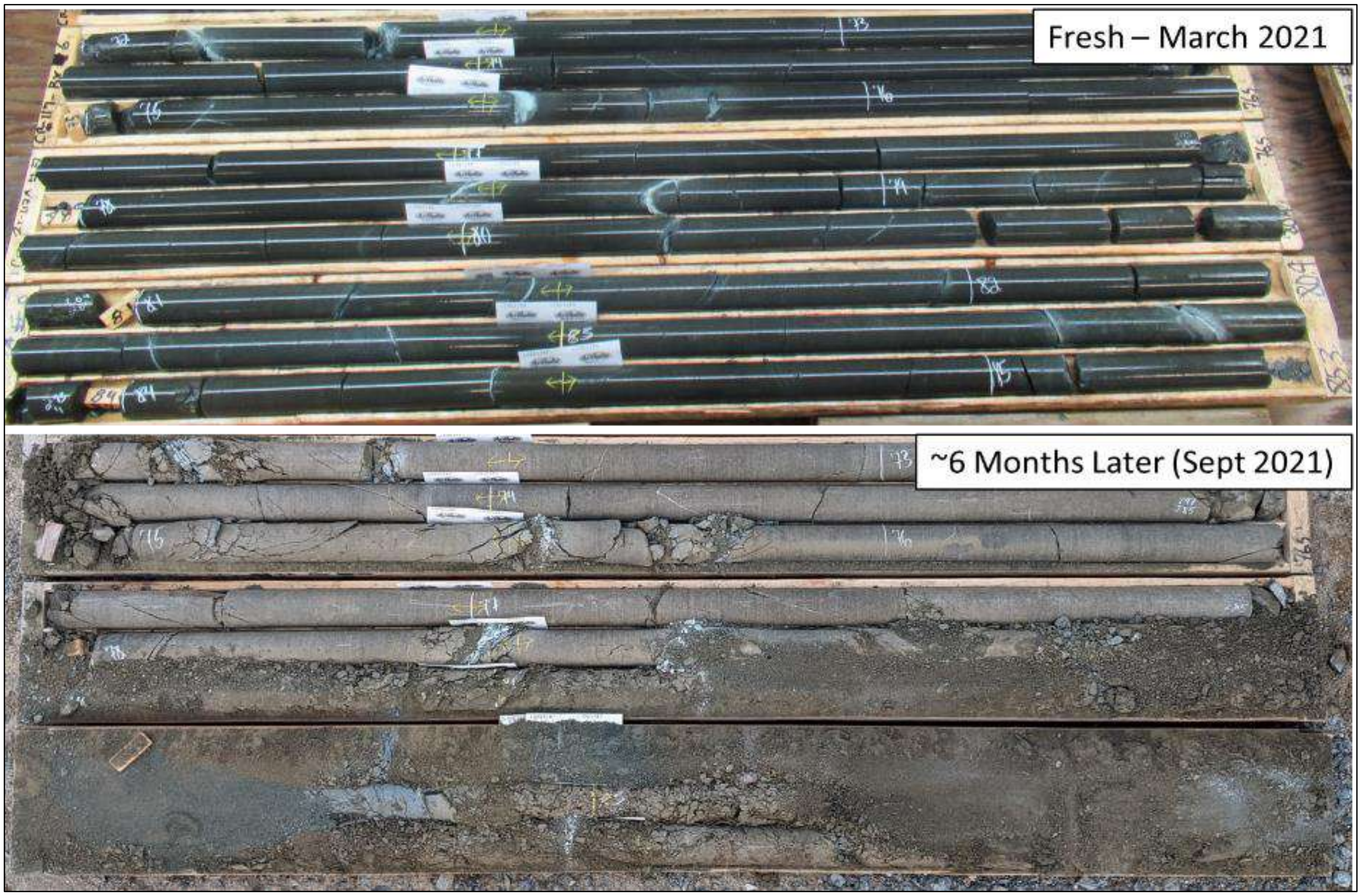
- presence of DUN-CG within the modelled DUN unit
- damage zones of major structures, specifically the modelled CUC Fault
- presence of talc within the ultramafic units.

In the northwest corner of the planned East Zone pit the foliation increases in complexity and may change orientations more rapidly than in other areas of the planned pits, as a result of 'dragging' along the regional CUC Fault.

In areas where the dunite contains coalingite (DUN-CG), a rapid degradation of the rock mass quality and joint conditions can occur following the rock being exposed to the atmosphere, as presented in Figure 16-9, depending on the intensity of alteration. Where the dunite is modelled to contain coalingite, potential unravelling and rock mass failures on a bench scale are anticipated to control the bench and inter-ramp stability.

The regional modelled CUC Fault intersection is known to impact rock mass quality in-situ within the damage/influence zone of the fault. Site observations suggest that the CUC Fault core is prone to slake and swell upon exposure, especially towards the MV unit, adversely impacting bench and possibly inter-ramp performance along the structure. A design mitigation for these areas is that the exposed CUC Fault core should be shotcreted at the bench scale to limit the exposure of this material to the atmosphere and the unravelling of the affected bench areas. The stiffness of the CUC Fault can also impact the performance of the overlying slope and needs to be considered during future evaluations.

**Figure 16-9: Example of Rock Mass Degradation from Presence of Coalingite**

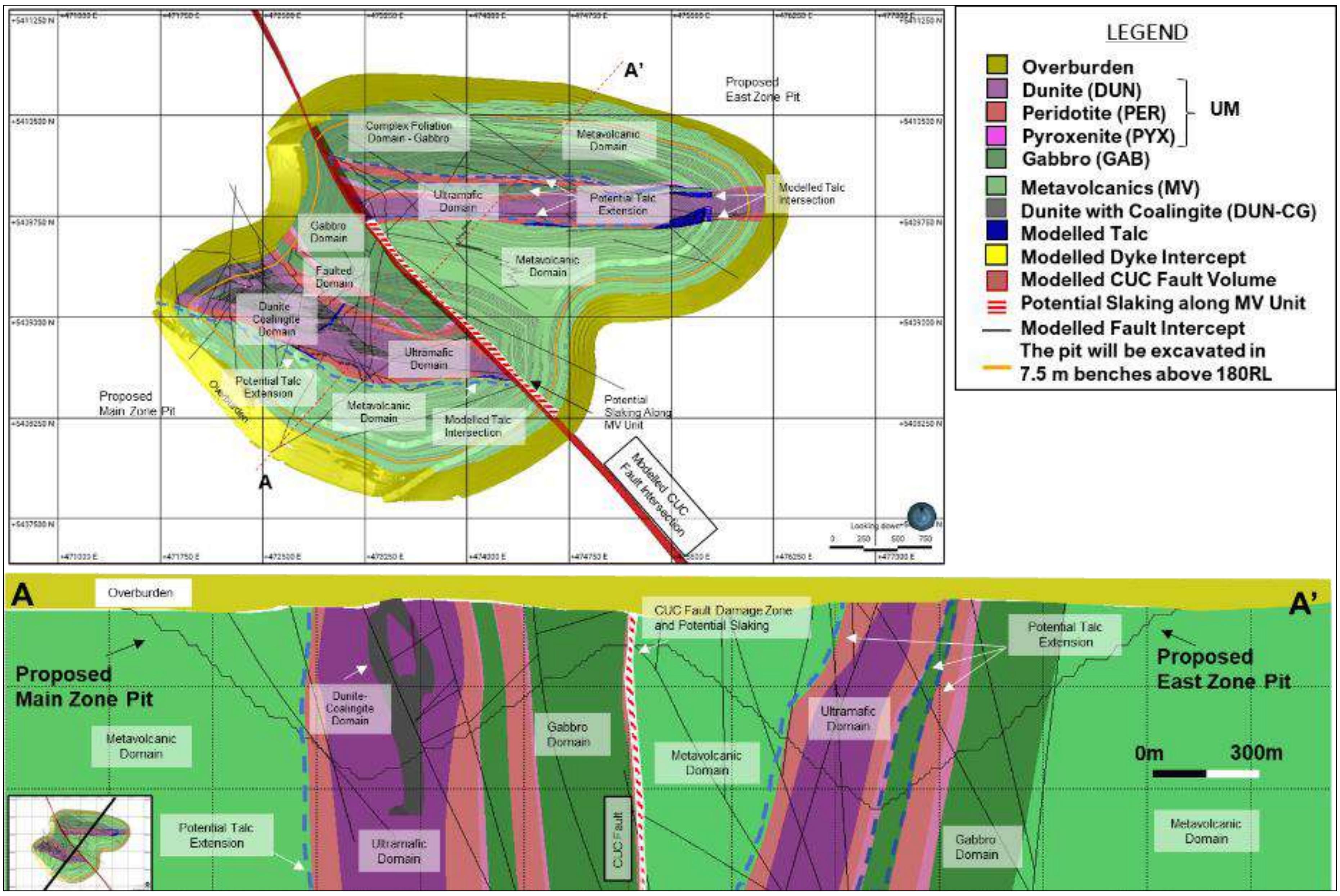



Note: The length of core that fits into these core boxes is approximately 1.5 m. Source: SRK, 2023.

Caracle Creek has modelled talc solids which currently intersect the east side of the East Zone pit, and the east and west sides of the Main Zone pit, as seen in Figure 16-10. The extension of these Talc zones along the footwall of the deposit is not currently confirmed and requires further definition during the early stages of mining. The intersection of these modelled Talc zones is anticipated to decrease the rock mass and joint strength resulting in local unravelling on a bench-scale and can impact overlying slope performance based on their thickness and stiffness. To manage this risk local single benching may be required. These talc zones may also act as a tension crack on the multi-bench and inter-ramp scale. Representative ranges of geotechnical parameters for each modelled lithological unit are provided in Table 16-5.

As a result of the scale of the project and the variability observed in these parameters local parameters were selected for the assessments, rather than using average values obtained from the entire deposit.

**Figure 16-10: Representative Plan and Section through the Proposed Crawford Pits showing the Geotechnical Domains**




Source: SRK, 2023

**Table 16-5: Representative Ranges of Geotechnical Characteristics of the Modelled Lithological Units at the Crawford Project Site**

| Pit | Modelled Lithological Units | Rock Quality Designation [RQD] (%) | Uniaxial Compressive Strength (MPa) | Fracture Frequency (ff/m) | Joint Condition Rating [$JCR_{89}$](1) | Rock Mass Rating [$RMR_{89}$](1) | Comments |
|---|---|---|---|---|---|---|---|
| East Zone | Metavolcanics (MV) | 93 (18) | 137 (50) | 3.8 (6.5) | 18 (3) | 75 (11) | The metavolcanics that makeup the hangingwall of the CUC Fault have been observed to experience a degradation process in the influence zone of the CUC Fault. This process and the areas it impacts are currently not well understood. |
| | Dunite (DUN) | 95 (15) | 91 (30) | 2.8 (3.9) | 17 (4) | 73 (10) | Where talc is modelled to intersect these units the rock mass strength and joint conditions are expected to consistently perform below median values. |
| | Peridotite (PER) | 82 (23) | 176 (90) | 7.1 (7) | 17 (4) | 67 (11) | |
| | Pyroxenite (PYX) | 95 (13) | 168 (55) | 3.4 (5.8) | 18 (3) | 76 (9) | |
| | Gabbro (GAB) | 98 (7) | 174 (56) | 1.8 (2) | 19 (3) | 79 (6) | |
| Main Zone | Metavolcanics (MV) | 95 (13) | 134 (37) | 2.5 (4.4) | 18 (3) | 78 (9) | The metavolcanics that makeup the hangingwall of the CUC Fault have been observed to experience a degradation process. This process and the areas it impacts are currently not well understood. |
| | Dunite (DUN) | 94 (13) | 156 (50) | 3 (5.5) | 17 (4) | 74 (12) | Where talc is modelled to intersect these units the rock mass strength and joint conditions are expected to consistently perform below median values.<br><br>Where coalingite is present, the rock mass quality (rock strength and joint conditions) is known to rapidly deteriorate once the rock has been exposed to the atmosphere. |
| | Peridotite (PER) | 94 (10) | 265 (86) | 2.7 (3) | 17 (4) | 75 (9) | Where talc is modelled to intersect these units the rock mass strength and joint conditions are expected to consistently perform below median values. |
| | Pyroxenite (PYX) | 84 (30) | 131 (60) | 6.8 (11.4) | 15 (6) | 68 (17) | |
| | Gabbro (GAB) | 92 (15) | 219 (71) | 3.4 (5.3) | 18 (3) | 75 (10) | |

Notes: 1. Each cell indicates the median followed by the standard deviation in brackets. 2. Bieniawski, 1989.

#### 16.2.2.3 Geotechnical Domains

The combined litho-structural, alteration model and the geotechnical model including known weaker areas were used to inform the geotechnical domains for the deposit. A representative plan and cross-section of the geotechnical domains for the proposed Crawford pits is provided above in Figure 16-10.

#### 16.2.2.4 Rock Slope Design

Analysis of the site geology, structural geology findings, geotechnical evaluation, and resource targets indicates that the Crawford deposit is mainly comprised of relatively strong rock masses in most of the slope areas of the planned pits comprising the Metavolcanics and the Gabbro units. The Ultramafic units (peridotite, pyroxenite and dunite) are, in general, weaker rock masses. Furthermore, within the Dunite unit on the western side of the proposed Main Zone pit, there are unfavourable rock mass conditions where the Dunite unit contains coalingite, which contributes to degrading the rock mass properties over time. Modelled talc intersections on the east side of the East Zone and Main Zone pits are also areas of weakness within the rock mass forming the proposed pit walls. The current modelled fault representing the regional CUC Fault intersects the west side of the East Zone pit and the east side of the Main Zone pit and is associated with slacking in the metavolcanic unit however the extent of this potential rockmass adverse alteration/damage and the consistency of this damage across/along the fault is not understood. The CUC Fault stiffness influences the stability of the northeast corner of the Main Zone Pit and will require further evaluation.

The dominant controls on the pit slope stability are expected to be kinematic in areas comprising non-ultramafic units. The slope design geometry in these units were primarily designed to control potential planar and wedge failures generated by unfavourable steeply dipping structures and foliation. In areas comprising ultramafic rock units, slope designs were informed by structural controls and potential rock mass failures. A summary of the anticipated dominant kinematic failure modes and estimations of the probability of occurrence for each zone is detailed in Table 16-6.

**Table 16-6: Summary of Potential Kinematic Failure Modes by Proposed Pit**

| Proposed Pit | Pit Wall | Domain Rock Type(1) | Bench Scale | | Inter-Ramp and Global Scale | |
|---|---|---|---|---|---|---|
| | | | Dominant Failure Mode and Likelihood of Occurrence | | Dominant Failure Mode and Likelihood of Occurrence | |
| East Zone Pit | South | MV | Planar | Moderate | Planar | Low |
| | North | MV and GAB | Planar | Low | Planar | Low |
| | West | GAB | Planar | Moderate | Planar | Low |
| | East and Central | DUN, PER, PYX | Wedges and planar | High | Planar | Low |
| Main Zone Pit | South | MV | Wedges | Moderate | Wedges | Low |
| | East | MV | Wedges | Moderate | Wedges | Low |
| | North | DUN, PER, PYX | Wedges and planar | Moderate | Planar | Low |
| | North | GAB | Wedges and planar | High | Planar | Low |
| | North-West | DUN-CG | Wedges | High | Planar and Wedges | Low |

Notes: 1. MV – metavolcanics, GAB – gabbro, DUN – dunite, PER - peridotite, PYX – pyroxenite, DUN-CG – dunite with coalingite

Assessment of the proposed pit slopes indicates that in sectors comprising non-ultramafic units (East Zone pit: hangingwall, footwall and west wall, Main Zone pit: hangingwall and east wall), the inter-ramp and overall slope angles designs were primarily controlled by bench stability, slope stability results showed that the risk of multi-bench and overall instability, including deep-seated failures, is low given the level of geotechnical information available at this time. As the kinematics are based on widely spaced drillhole data the orientation of the discontinuities and associated kinematic assessments will need to continue to be refined following rock exposure.

Considering the geotechnical domains, presented in Figure 16-10, and the likely slope directions, slope design sectors were generated, as seen in Figure 16-11, and slope design parameters developed for each design sector. The recommended maximum stack height for the structurally controlled areas (Metavolcanics and Gabbro units) is 120 m, whereas 90 m is recommended for areas within the coalingite containing dunite (DUN-CG). All stacks are recommended to be separated by a geotechnical safety berm of 25 m. A 20 m step-off is required at the boundary between the overburden and rock slopes to account for the potential construction of a buttress and the potential need for overburden slope armouring. These design specifications are presented in Table 16-5.

The impact of individual or combinations of modelled major structures on the proposed slope design were reviewed and indicated that structures could impact the following:

- upper central portion of the south slope of the East Zone pit with a potential wedge intersection
- upper central portion of the north slope of the Main Zone pit in a zone of structural complexity
- lower portion of the west slope of the Main Zone pit with potential wedge and planar intersections.

These identified intersections are between faults which are currently defined as low confidence and therefore the location and orientation of these faults will need to be further assessed with additional definition drilling during the early stages of overburden mining to understand if and how a slope design change should be implemented to promote slope stability. However, the structural model is currently based on widely spaced drillhole data and no surface exposure and therefore potential risks to the slope and ramps therefore major structures will need to continue to be reviewed following rock exposure and refinement of the structural model.

Limit equilibrium and finite element numerical modelling was undertaken on representative sections of the proposed final slope walls. The rock mass properties including strengths and orientations of discontinuities, influence of modelled major structures as well as hydrogeological considerations (see Section 16.1) were included in these assessments. The phreatic surface tends to be close to the slope face with the benches being free drained because of blasting induced fracturing and damage. In regions of the pit controlled by kinematics, these models allowed for the slope design guidelines to be confirmed.

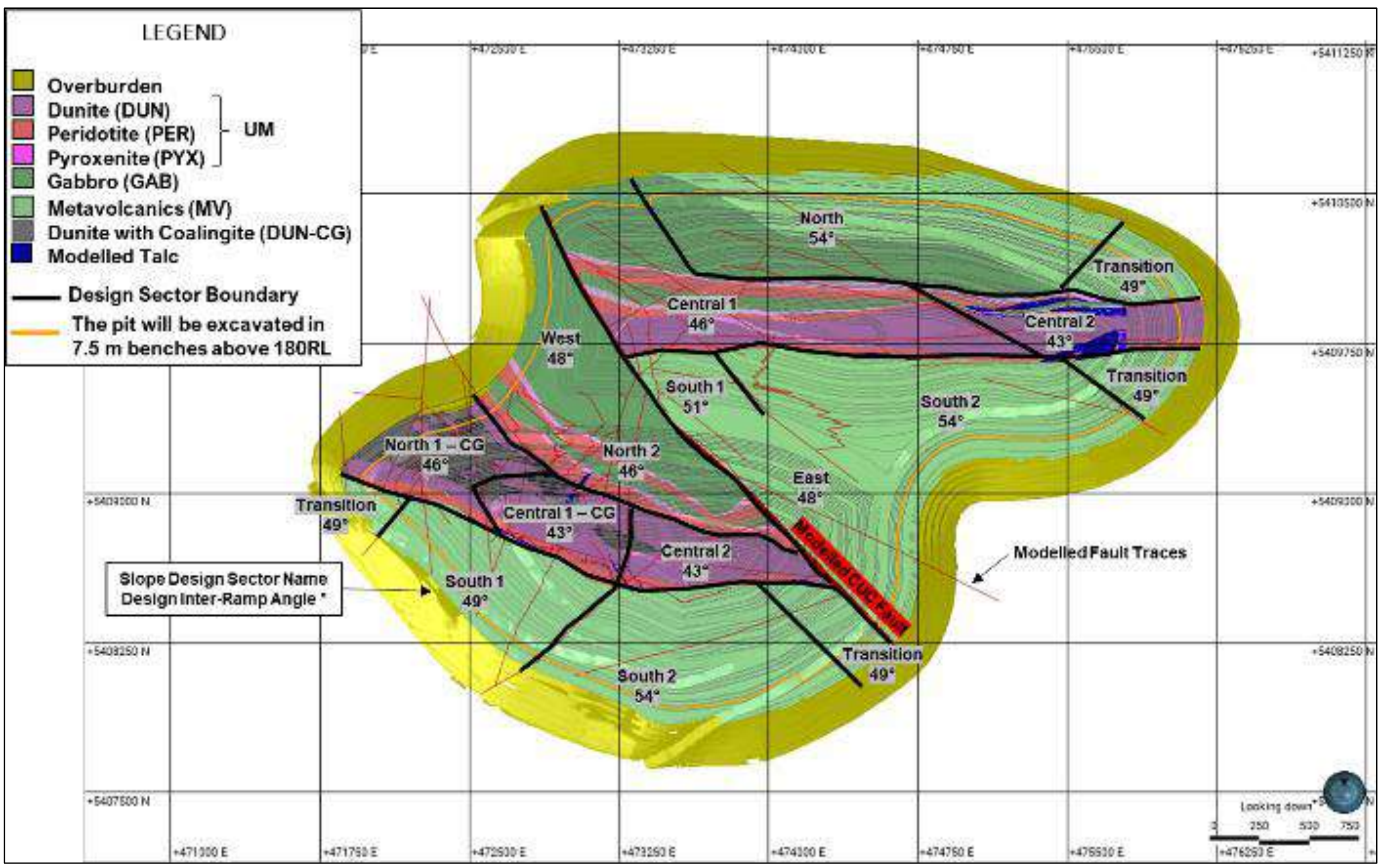
In slope design sectors controlled by rock mass failure, these models were used to inform slope designs. The models also informed the validation of the designed minimum saddle width, of 150 m. These models indicated that dewatering of the saddle between the East Zone and the Main Zone pits may be required to meet the slope design acceptance criteria. A refinement of this assessment is required in the following stages of the project to validate considerations such as the mining sequence and effects of deconfinement on the saddle stability. The geomechanical properties of the rock mass as well as the hydrogeological properties of the rock mass and the major structures will need to be further confirmed in this area prior to construction.

The pit ramp infrastructure can be impacted by potential geotechnical instability and require additional evaluation during mining in the following areas:

- in the East Zone pit where a ramp switch back is planned in an area of talc in the east wall
- sections of the ramp in the northwestern side of the Main Zone pit that are planned in the dunite modelled to contain coalingite
- where the ramp is currently planned in the structurally complex section of the central northwestern Main Zone pit
- ramp sections intersected by the hangingwall side of the CUC Fault.

The provided slope design guidelines, presented in Figure 16-11 and Table 16-7, assume that the use of pre-split and controlled blasting is strongly implemented on the intermediate and final walls, to assist in limiting the damage to the rock mass and related impacts on bench and inter-ramp stability.

**Figure 16-11: Plan View of the East Zone and Main Zone Pit Slope Design Sectors and Proposed Inter-Ramp Angles**




Source: SRK, 2023.

**Table 16-7: Crawford Pit Design Guidelines by Slope Design Sector**

| Design Domain / Sector (Face Dip° – Direction°) | | Bench Height (m) | Bench Width (m) | Bench Face Angle (°) | Inter-Ramp Angle (°) | Stack Height Restriction (m) |
|---|---|---|---|---|---|---|
| East Zone | North (170° – 220°) | 30 | 10.5 | 70 | 54 | 120 |
| | Transition Zone (220° – 310°) | 15 | 7.5 | 70 | 49 | 90 |
| | Central 1 (000° – 360°) | 15 | 7.5 | 65 | 46 | 90 |
| | Central 2 (000° – 360°) | 15 | 7.5 | 60 | 43 | 90 |
| | West (015° – 100°) | 15 | 8.0 | 70 | 48 | 90 |
| | South 1 (360° – 015°) | 30 | 10.5 | 65 | 51 | 120 |
| | South 2 (310° – 360°) | 30 | 10.5 | 70 | 54 | 90 |
| Main Zone | North 1 – CG (105° – 190°) | 15 | 7.5 | 65 | 46 | 90 |
| | North 2 (150° – 270°) | 15 | 7.5 | 65 | 46 | 90 |
| | East (180° – 270°) | 15 | 8.0 | 70 | 48 | 90 |
| | Transition Zones | 15 | 7.5 | 70 | 49 | 90 |
| | Central 1 – CG (180° – 055°) | 15 | 7.5 | 60 | 43 | 90 |
| | Central 2 (000° – 360°) | 15 | 7.5 | 60 | 43 | 90 |
| | South 1 (040° – 050°) | 30 | 12.5 | 65 | 49 | 120 |
| | South 2 (310° – 050°) | 30 | 10.5 | 70 | 54 | 120 |

## 16.3 Open Pit Mine Plan

### 16.3.1 Overview

The Crawford plan includes mining of two discreet, though overlapping, pits (Figure 16-12). Both pits contain an approximately equal tonnage of ore, with the Main Zone having a lower stripping ratio and the East Zone higher average ore values. The current design and sequence of the pits allows for in-pit impoundment of tailings to be maximized, thus minimizing the size of a tailings management facility (TMF) and associated environmental impacts. This sequence allows in-pit deposition of tailings to commence after the Main Zone is depleted in Year 17. Mining of the East Zone continues through Year 30. Thereafter, the mill is fed for the final 11.25 years of operation from two stockpiles of lower value ore.

Over the life of mine, 161 $Mm^3$ of various materials will be required for reclamation of disturbed areas and construction of infrastructure. Over 99.6% of this material will be waste from the Crawford pits. There are numerous potential sources for the remaining 0.7 $Mm^3$ of construction aggregate that will be required during the first year of project construction. Approximately 90% of the reclamation and construction materials will be run of mine. The remaining 10% will be temporarily stockpiled at sites close to where they will ultimately be required.

**Figure 16-12: Crawford Open Pit and Associated Infrastructure**


Source: CNC, 2023.

The remaining 1,825 $Mm^3$ of waste will be impounded in one large facility located north of the pits. This will be divided into the following three zones:

- Furthest to the east will be the clay impoundment. To facilitate delivery and containment of clay, ribs constructed from waste rock (included in the volume required for construction) will be spaced at a nominal 200 m.
- Adjacent to the clay impoundment will be a zone for sand and till.
- The western portion of the impoundment will store waste rock.

A total of 1,715 Mt ore will be mined. As discussed in Section 15, this total includes mining losses of 5.2% (i.e., mining recovery is 94.8%), planned dilution of 0.4% and unplanned dilution of 2.0%. Of this total, 1,078 Mt will be delivered directly to the crushers as run-of-mine feed.

Lower value ore will be impounded in two stockpiles, the east stockpile and west stockpile, which are near the two primary crushers. Each stockpile will have zones of higher and lower values, with the higher value zones reclaimed first. Over the life of mine, 37% of total ore will be stockpiled.

The pits will employ two bench heights. To a depth of 90 m below surface, which is the maximum depth of overburden, the bench height will be 7.5 m. Below this horizon, all material will be rock and benches will be 15 m.

Mining will be performed by a mixed fleet of equipment as follows:

- For the 4% of total material that will be mined from a footwall located in clay, small backhoe excavators and 40 t articulated trucks will be used.
- For the 6% of total material that will be mined from a footwall located in sand and till, a medium-size (300 t class) electric face shovel and 90 t trucks will be used.
- The remaining 90% of material will be mined from a footwall located in rock. This will be loaded using large (50 t payload) front-end loaders (FEL), 700 t class electric face shovels or rope shovels and hauled using 290 t class haul trucks equipped with trolley-assist and autonomous haulage systems (AHS).

To support operations, the pits will have extensive systems for dewatering and supply of electricity.

Steps performed in developing the design and schedule for Crawford included the following:

- A techno-economic model was developed to evaluate design and schedule alternatives. In general, the optimal alternative was defined as delivering maximum post-tax net present value (NPV8%), though internal rate of return (IRR) and cashflow index (CFI; the ratio of NPV: maximum at-risk investment) were also considered.
- The LG optimization determined the sequence and limits for pit development. Nested shells generated by the algorithm were aggregated to define the stages of pit development. Stages were tested iteratively to determine the optimal mining sequence and ultimate limits for each, therefore forming the basis for phase designs.
- An engineered pit design was developed that incorporated the specified slope design and ramps of sufficient width for the planned equipment and density of traffic.
- A 'generic' engineered schedule was developed for the optimal sequence and optimized using the techno-economic model.

A key element of the overall Crawford mining strategy is the decoupling of production rates for the mine and mill. With the mine releasing ore an average of 39% faster than will be digested by the mill, higher value ore will be processed immediately while lower value will be temporarily impounded in one of two stockpiles.

Other mining-related infrastructure includes the waste impoundments, TMF, and a network of surface roads.

Each of these topics that has been outlined will be discussed in more detail below.

### 16.3.2 Techno-Economic Model

The key design tool was a techno-economic model. This allowed unique production schedules to be evaluated, with bottom-up estimates of costs and revenues generated for each unique schedule. Examples of the elements considered in these cost estimates include:

- unique haulage profiles broken into horizontal, uphill, and downhill segments – within the pit, enroute to an exit ramp, and ex-pit to the ultimate destination
- different truck speeds and rates of diesel consumption on horizontal, uphill, and downhill segments
- the impact of fleet age on maintenance costs.

Costs and revenues flowed through to a set of post-tax financial statements from which metrics such as NPV, IRR, CFI and Simple Payback could be calculated. A macro would step through the various degrees of freedom, typically generating >> 10,000 unique schedules for which key production, cost and valuation metrics were recorded. These were then ranked, allowing the optimal to be identified.

### 16.3.3 LG Optimization

The Crawford LG optimization is described in some detail in Section 15.4 and is thus summarized below. Inputs to the LG optimization included the following:

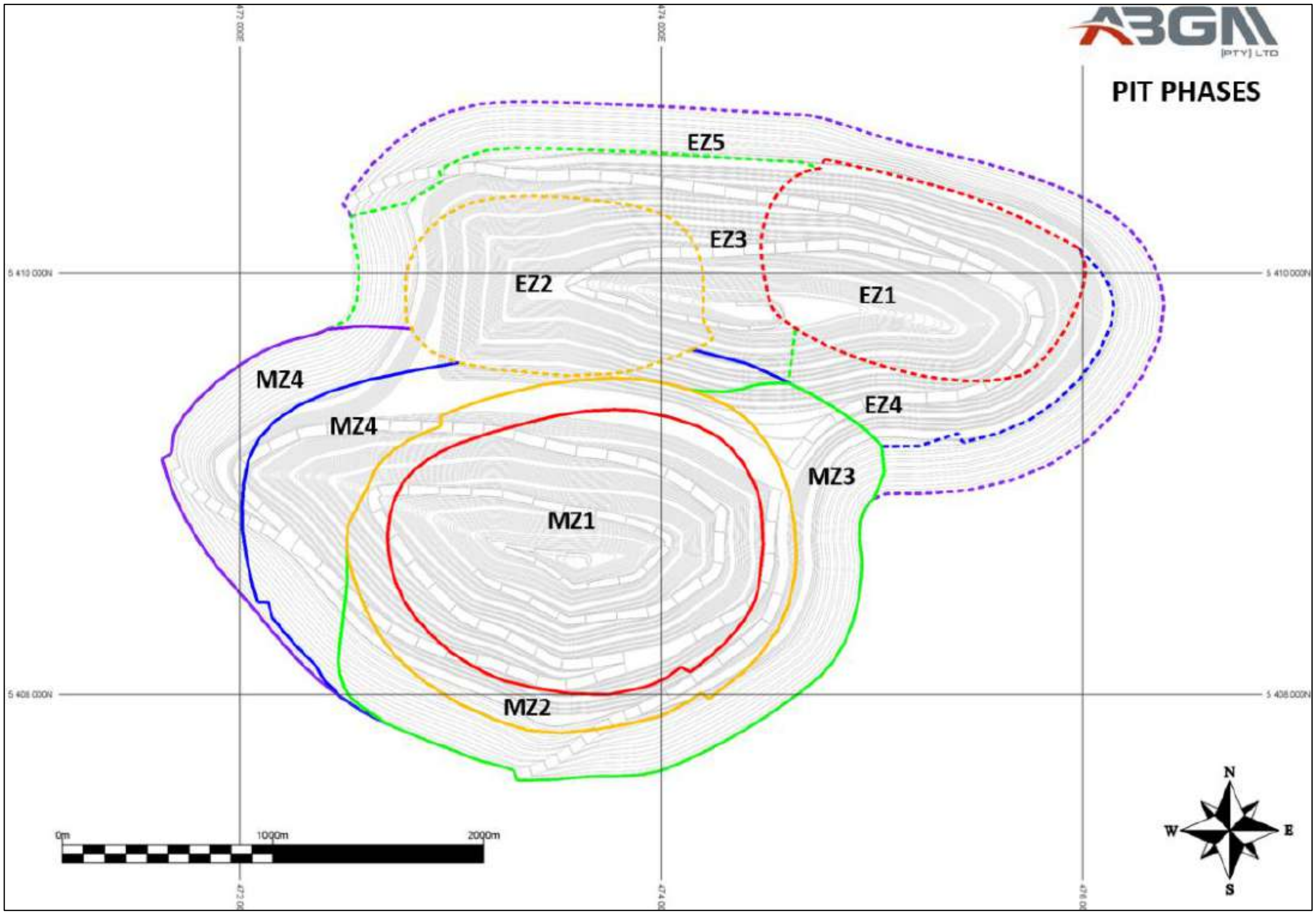
- slope angles for various sectors of the pits and materials that will be mined (see Section 16.2)
- NSR based on a block-by-block estimate of mill recovery for each metal and CNC's forecast payment terms for the two concentrates that will be produced (see Section 15.3, Table 15-2).
- cost assumptions as provided in Section 15.4, Table 15-3.

The LG algorithm then selected a 'cone' of ore and associated waste stripping that maximizes NPV. By varying the revenue factor (RF, which is the percentage of the long-term metal prices), it was possible to generate higher value nested cones that can be used to identify the optimal development sequence. As discussed in Section 15, there are no meaningful points of inflection on the continuum from RF50 – RF100. Evaluation of various alternatives demonstrated that overall value would be maximized with the largest scope that could be accommodated within the current land position to the east of the realigned corridor for Highway 655. It should be noted that it would be feasible to expand the project west of the realigned corridor; at this time the incremental NPV associated with such an expansion would be negligible when the project life already exceeds 40 years. As the waste content of the RF65 shell approximated the total capacity for impoundments including the TMF, this shell was selected as the basis for the engineered design.

### 16.3.4 Phase Design and Sequence

The nested shells contained within the RF65 shell were used to guide ten stages of mine development. A key element of the Crawford design is in-pit storage of tailings, with maximized in-pit storage achieved by mining the two pits sequentially and the early depletion of one pit. As a result, individual stages were confined to one or the other pit. These can be summarized as follows (Figure 16-13).

**Figure 16-13: Crawford Open Pit Stages**




Source: CNC, 2023.

### 16.3.4.1 Main Zone Stages

- MZ1 is centrally located within the Main Zone and the highest value (both NSR and EBITDA). It has the lowest overall strip ratio, allowing for a high mine speed factor (MSF, the ratio of ore mined to milled). However, it contains limited waste rock that could be used for construction. It also has the largest tonnage of clay, which requires waste rock for armouring the mining surface and for constructing the clay impoundment.
- MZ2 is a concentric expansion of MZ1 that contains average ore values and stripping ratio.
- MZ3 is a waste pushback to access the deeper ore contained in MZ4 and MZ5.

MZ4 and MZ5 have lower strip ratios due to the stripping performed in MZ3. This will allow an increase to the MSF, to partially offset the lower average value of ore in both stages.

#### 16.3.4.2 East Zone Stages

- EZ1 is a small stage that primarily targets waste rock (for construction) underlying the zone of shallowest clay. As a result, the stripping ratio is relatively high. Ore that is released is of average value.
- EZ2 is another small zone that targets a shallow, high value zone that is not connected to EZ1.
- EZ3 is a pushback into lower value material that connects EZ1 and EZ2.
- EZ4 and EZ5 are large, higher value zones located at depth.

The overall mining sequence prioritizes the Main Zone for early depletion as it is the smaller pit and thus can be available as an impoundment for tailings approximately three years sooner than the East Zone. MZ1 is also the highest value individual stage and thus logically a source of early feed to the mill. The TMF capacity is sufficient to allow a modest tonnage of East Zone to be mined concurrently with the Main Zone, which will be utilized by:

- The first stage mined will be EZ1, primarily to provide waste materials that will be used for project construction.
- EZ2 will be mined concurrently with MZ3 waste pushback, to maintain a supply of higher value run-of-mine ore.

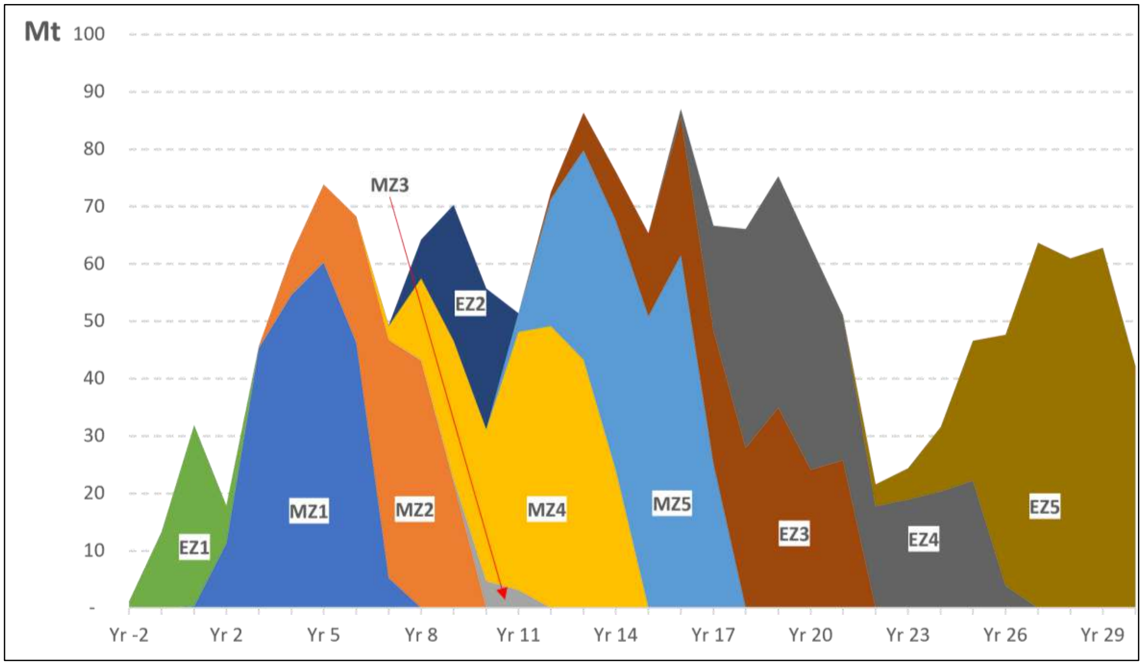
Figure 16-14 and Figure 16-15 illustrate the sequence of ore mining and overall mining, respectively, by stage.

The following constraints placed on the current design and schedule relate to the capacity of the TMF:

- As discussed in Section 15.5, the ‘saddle’ elevation has been raised to RL120 to ensure sufficient capacity within the Main Zone void for tailings generated while the East Zone remains active. As a result, 78 Mt of measured and indicated resources contained within the RF65 shell are sterilized. While these resources have a high strip ratio of 4.3: 1, the average NSR is 41% higher than that of the overall mine.
- The average NSR and EBITDA of East Zone ore are 13% and 12% higher, respectively, than that of the Main Zone. Some of this margin is offset by the impact of the high value MZ1 providing early feed (i.e., the impact on discounted cash flows). Additionally, the lower strip ratio of the Main Zone allows for a higher MSF and the associated increase in value of run-of-mine feed (albeit with a higher percentage of lower value ore stockpiled). Nonetheless, incremental value could be realized by reversing the priorities of the Main and East Zone, particularly if MZ1 could be maintained as the initial source of mill feed.

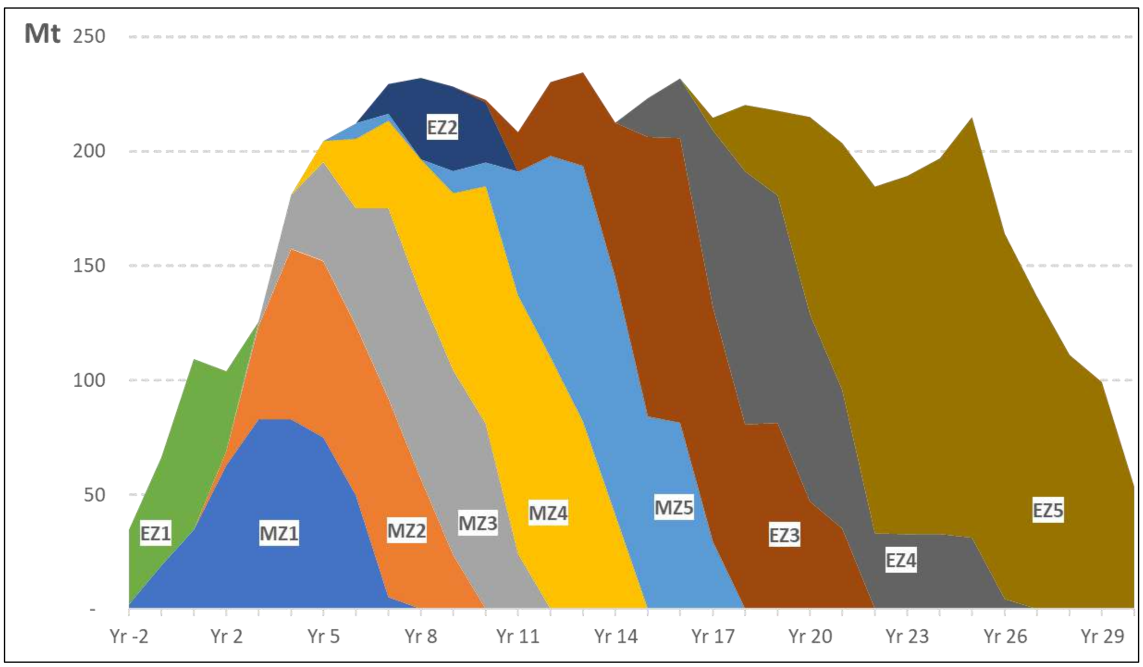
In the event the TMF capacity could be increased, by increasing the height of embankments or by achieving a steeper beach slope than has been assumed, it would be feasible to reduce the saddle elevation and/or reverse the priority of pits. These scenarios and the associated incremental value are discussed further in Section 25.16.

**Figure 16-14: Crawford Ore Mining by Stage**




Source: CNC, 2023.

**Figure 16-15: Crawford Total Mining by Stage**




Source: CNC, 2023.

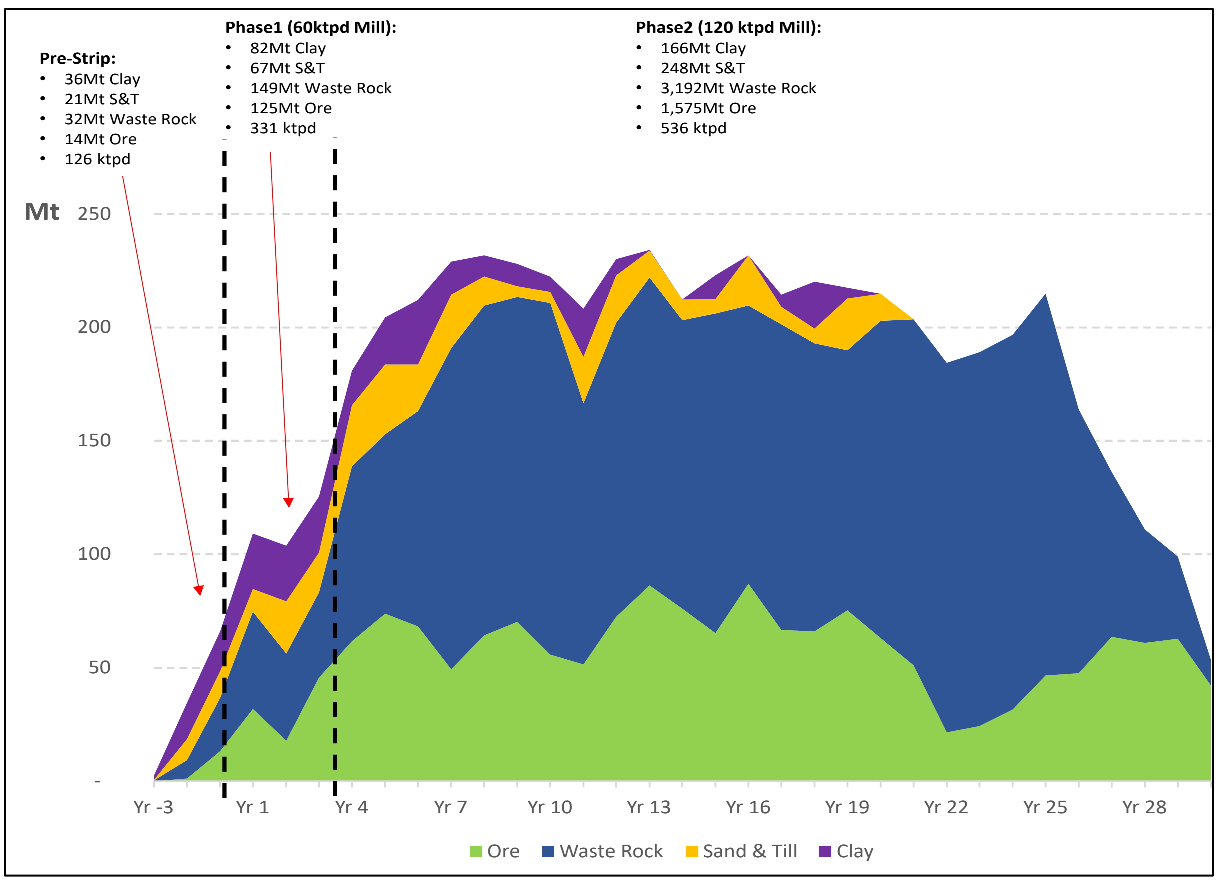
### 16.3.5 Schedule

The phase designs were used as the basis for the overall mine schedule. This was constructed in the following iterative manner:

- A 'generic' schedule was constructed for each phase using mine planning software (Datamine).
- This schedule was used to assign individual cells in the block model values for both the period they would be mined (the schedule increment being a three-month quarter) and the sequence within the quarter.
- Alternative schedules were then tested using the techno-economic model. Key degrees of freedom tested included the start date for a phase and mining rate for each of the three fleets of equipment (i.e., articulated trucks in clay, 90 t trucks in sand and till, 290 t trucks in rock).
- The schedule determined to be optimal was then re-imported to the planning software to confirm its feasibility.

The resultant schedule is summarized in Figure 16-16 and Table 16-8. Note that due to the very flat topography, there are no material downhill loaded hauls.

**Figure 16-16: Crawford Summary Mine Production Schedule**



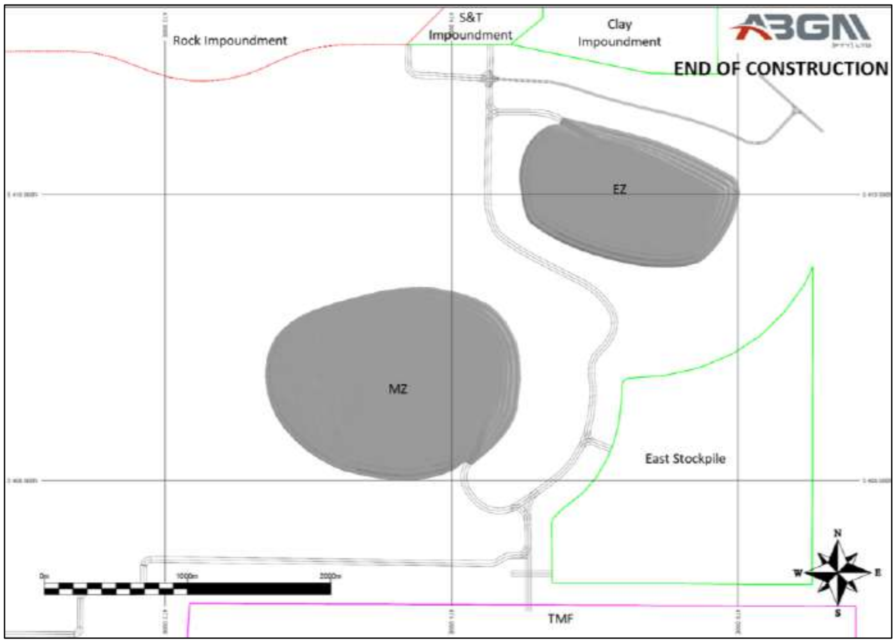

Source: CNC, 2023.

**Table 16-8: Crawford Life of Mine Physicals**

| Total Mining | Units | Total | Yr -3 | Yr -2 | Yr -1 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Yr 10 | Yr11-15 | Yr16-20 | Yr21-25 | Yr26-30 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Clay | Mt | 283 | 2 | 16 | 18 | 25 | 25 | 25 | 15 | 21 | 28 | 15 | 9 | 10 | 7 | 39 | 31 | - | - |
| Sand & Till | Mt | 336 | 0 | 9 | 12 | 10 | 23 | 18 | 27 | 31 | 21 | 24 | 13 | 5 | 5 | 69 | 71 | - | - |
| Waste Rock | Mt | 3,372 | 0 | 8 | 23 | 43 | 38 | 37 | 77 | 79 | 95 | 142 | 145 | 143 | 155 | 648 | 638 | 813 | 286 |
| Ore | Mt | 1,715 | - | 1 | 13 | 32 | 18 | 46 | 62 | 74 | 68 | 49 | 64 | 70 | 56 | 352 | 358 | 175 | 277 |
| **Articulated Truck** | | | | | | | | | | | | | | | | | | | |
| Total Hauled | Mt | 231 | 2 | 12 | 15 | 22 | 20 | 20 | 13 | 19 | 23 | 13 | 6 | 10 | 6 | 27 | 25 | - | - |
| Horizontal Haul | 1-way, metres | 5,317 | 2,861 | 3,508 | 5,793 | 6,010 | 6,641 | 6,363 | 5,618 | 6,163 | 6,464 | 3,904 | 3,060 | 5,864 | 5,084 | 4,103 | 4,204 | - | - |
| Uphill Loaded | vertical metres | 38 | 6 | 8 | 15 | 24 | 33 | 36 | 34 | 47 | 57 | 32 | 38 | 56 | 62 | 45 | 51 | - | - |
| **90 t Truck** | | | | | | | | | | | | | | | | | | | |
| Total Hauled | Mt | 331 | 1 | 12 | 12 | 12 | 22 | 22 | 25 | 31 | 21 | 20 | 13 | 4 | 5 | 67 | 64 | - | - |
| Horizontal Haul | 1-way, metres | 5,511 | 1,496 | 3,632 | 4,830 | 7,014 | 7,730 | 7,270 | 6,850 | 5,720 | 6,361 | 6,589 | 4,077 | 4,574 | 6,099 | 4,726 | 4,289 | - | - |
| Uphill Loaded | vertical metres | 60 | 13 | 16 | 25 | 36 | 47 | 43 | 52 | 58 | 69 | 77 | 49 | 65 | 69 | 69 | 77 | - | - |
| **290 t Truck** | | | | | | | | | | | | | | | | | | | |
| Total Hauled | Mt | 5,145 | 0 | 10 | 39 | 75 | 62 | 84 | 144 | 155 | 168 | 196 | 212 | 214 | 211 | 1,013 | 1,009 | 988 | 563 |
| Horizontal Haul | 1-way, metres | 5,491 | 1,369 | 3,813 | 4,060 | 5,084 | 4,632 | 4,903 | 4,852 | 6,093 | 7,285 | 7,000 | 6,330 | 6,200 | 6,736 | 5,820 | 5,274 | 5,222 | 4,022 |
| Uphill Loaded | vertical metres | 262 | 11 | 24 | 42 | 89 | 117 | 79 | 115 | 154 | 178 | 177 | 183 | 191 | 231 | 275 | 243 | 290 | 500 |
| **Ore Mining** | **Units** | **Total** | **Yr -3** | **Yr -2** | **Yr -1** | **Yr 1** | **Yr 2** | **Yr 3** | **Yr 4** | **Yr 5** | **Yr 6** | **Yr 7** | **Yr 8** | **Yr 9** | **Yr 10** | **Yr11-15** | **Yr16-20** | **Yr21-25** | **Yr26-30** |
| Ore Mined | Mt | 1,715 | - | 1 | 13 | 32 | 18 | 46 | 62 | 74 | 68 | 49 | 64 | 70 | 56 | 352 | 358 | 175 | 277 |
| Grade Ni | % Ni | 0.22 | - | 0.17 | 0.22 | 0.22 | 0.19 | 0.23 | 0.22 | 0.23 | 0.24 | 0.22 | 0.22 | 0.23 | 0.23 | 0.22 | 0.21 | 0.21 | 0.23 |
| Grade Co | % Co | 0.01 | - | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 |
| Grade Pd | g/t Pd | 0.01 | - | 0.01 | 0.01 | 0.01 | 0.01 | 0.02 | 0.02 | 0.02 | 0.02 | 0.01 | 0.02 | 0.02 | 0.01 | 0.01 | 0.01 | 0.01 | 0.02 |
| Grade Pt | g/t Pt | 0.01 | - | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 |
| Grade Fe | % Fe | 6.44 | - | 6.20 | 6.16 | 6.12 | 6.36 | 6.51 | 6.47 | 6.43 | 6.38 | 6.44 | 6.45 | 6.56 | 6.39 | 6.78 | 6.51 | 6.22 | 6.06 |
| Grade Cr | % Cr | 0.57 | - | 0.56 | 0.62 | 0.60 | 0.55 | 0.57 | 0.56 | 0.57 | 0.57 | 0.55 | 0.57 | 0.58 | 0.58 | 0.56 | 0.56 | 0.58 | 0.59 |
| Grade Brucite | % Brucite | 1.61 | - | 1.40 | 1.39 | 2.07 | 1.51 | 0.72 | 1.19 | 1.61 | 1.64 | 1.40 | 1.20 | 1.12 | 1.45 | 1.11 | 1.61 | 2.41 | 2.22 |

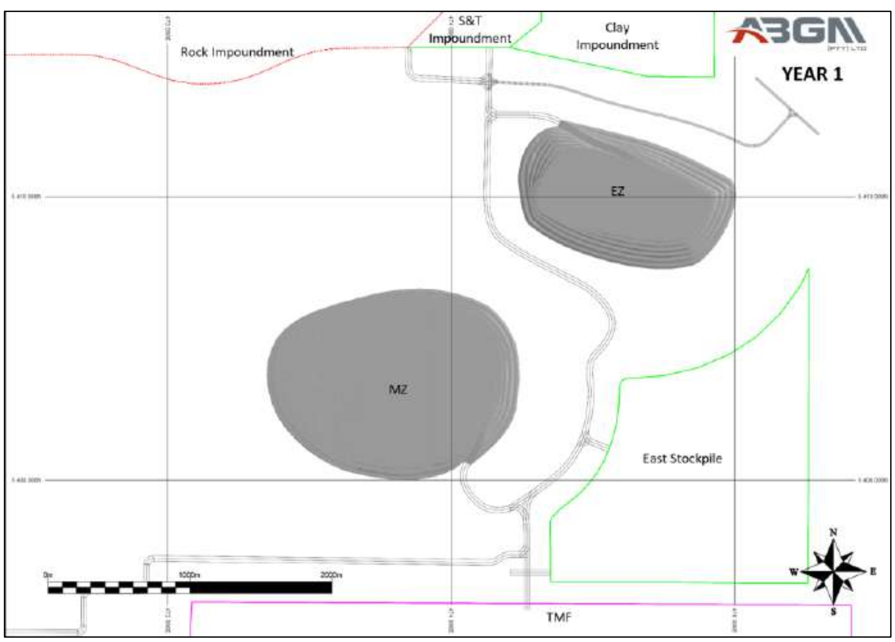
Figures 16-17 through 16-27 illustrate the progression of the pit mining faces.

**Figure 16-17: Crawford Mine Development, End of Pre-Strip**



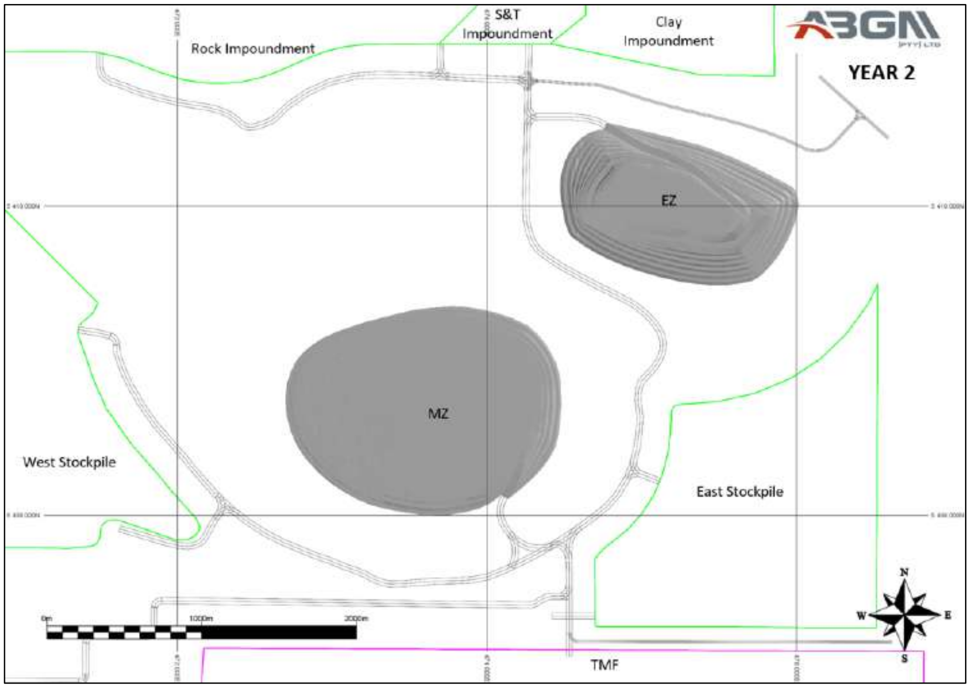

Source: CNC, 2023.

**Figure 16-18: Crawford Mine Development, End of Year 1**




Source: CNC, 2023.

**Figure 16-19: Crawford Mine Development, End of Year 2**

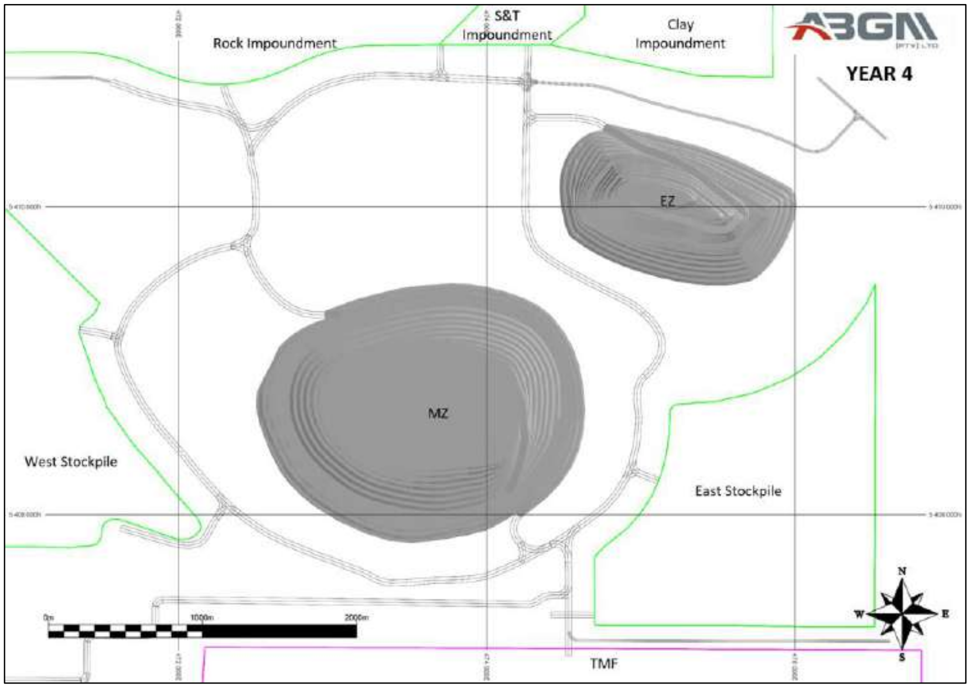



Source: CNC, 2023.

**Figure 16-20: Crawford Mine Development, End of Year 3**

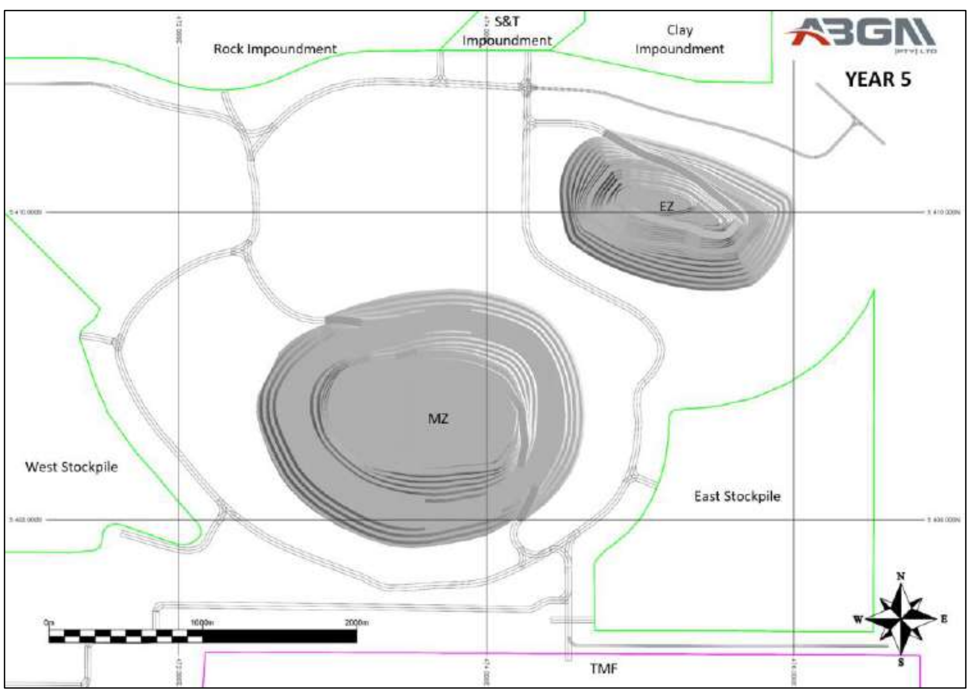

Source: CNC, 2023.

**Figure 16-21: Crawford Mine Development, End of Year 4**



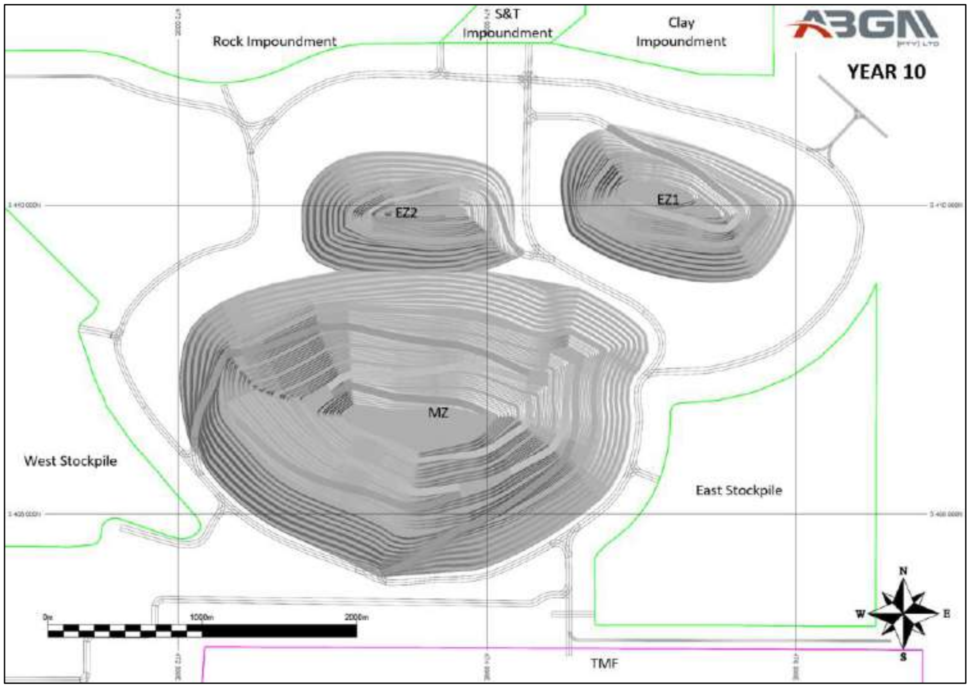

Source: CNC, 2023.

**Figure 16-22: Crawford Mine Development, End of Year 5**



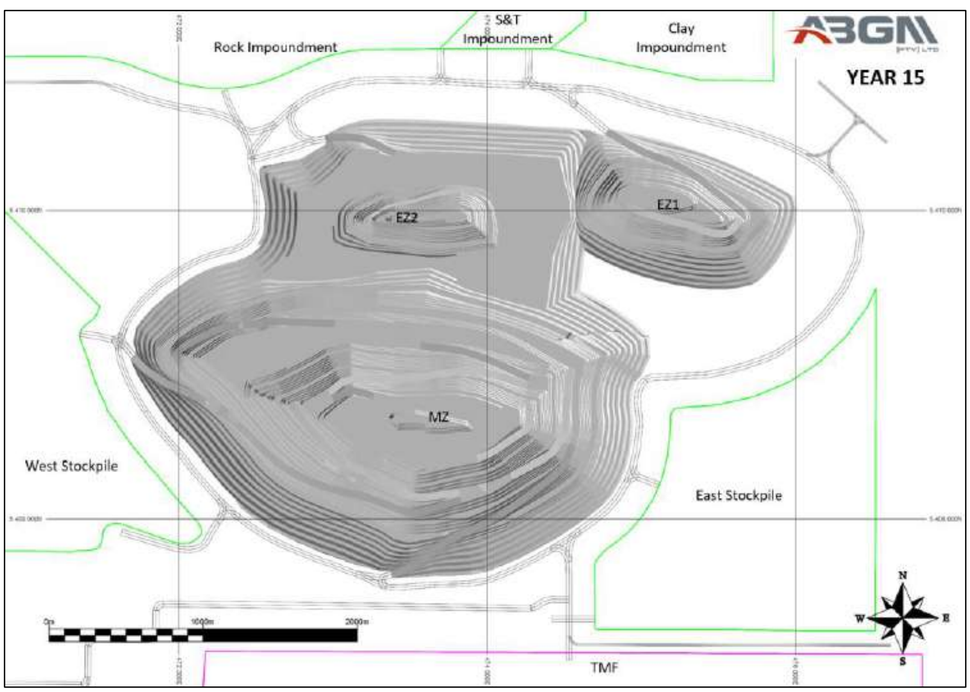

Source: CNC, 2023.

**Figure 16-23: Crawford Mine Development, End of Year 10**



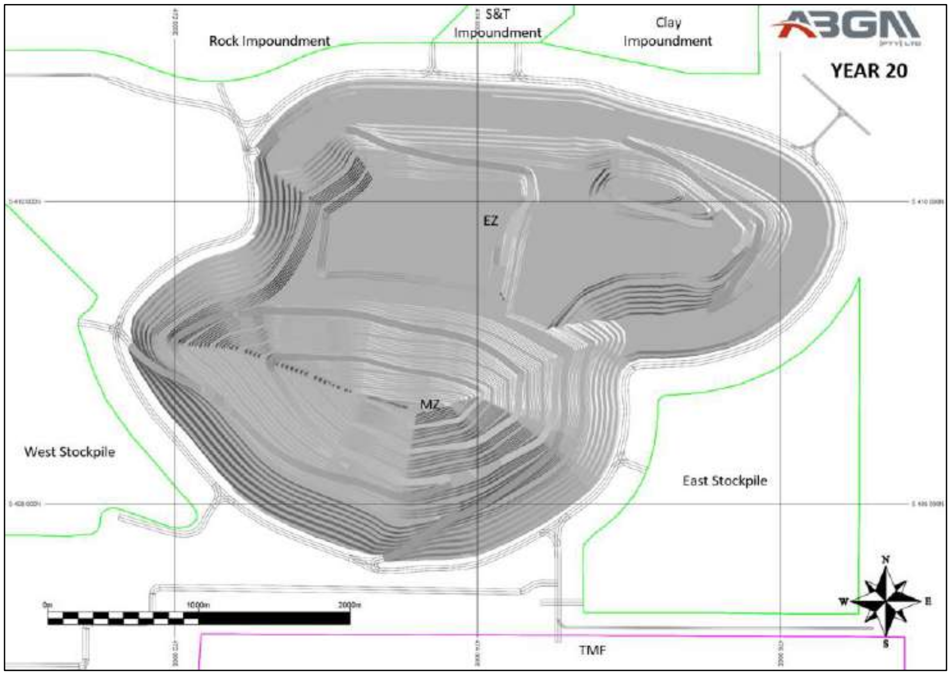

Source: CNC, 2023.

**Figure 16-24: Crawford Mine Development, End of Year 15**



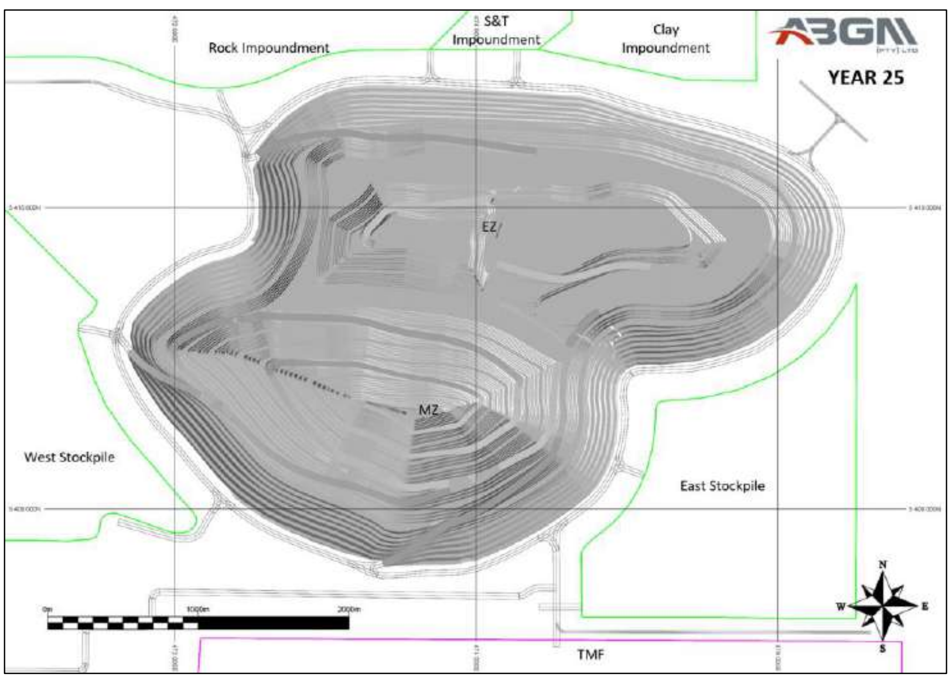

Source: CNC, 2023.

**Figure 16-25: Crawford Mine Development, End of Year 20**



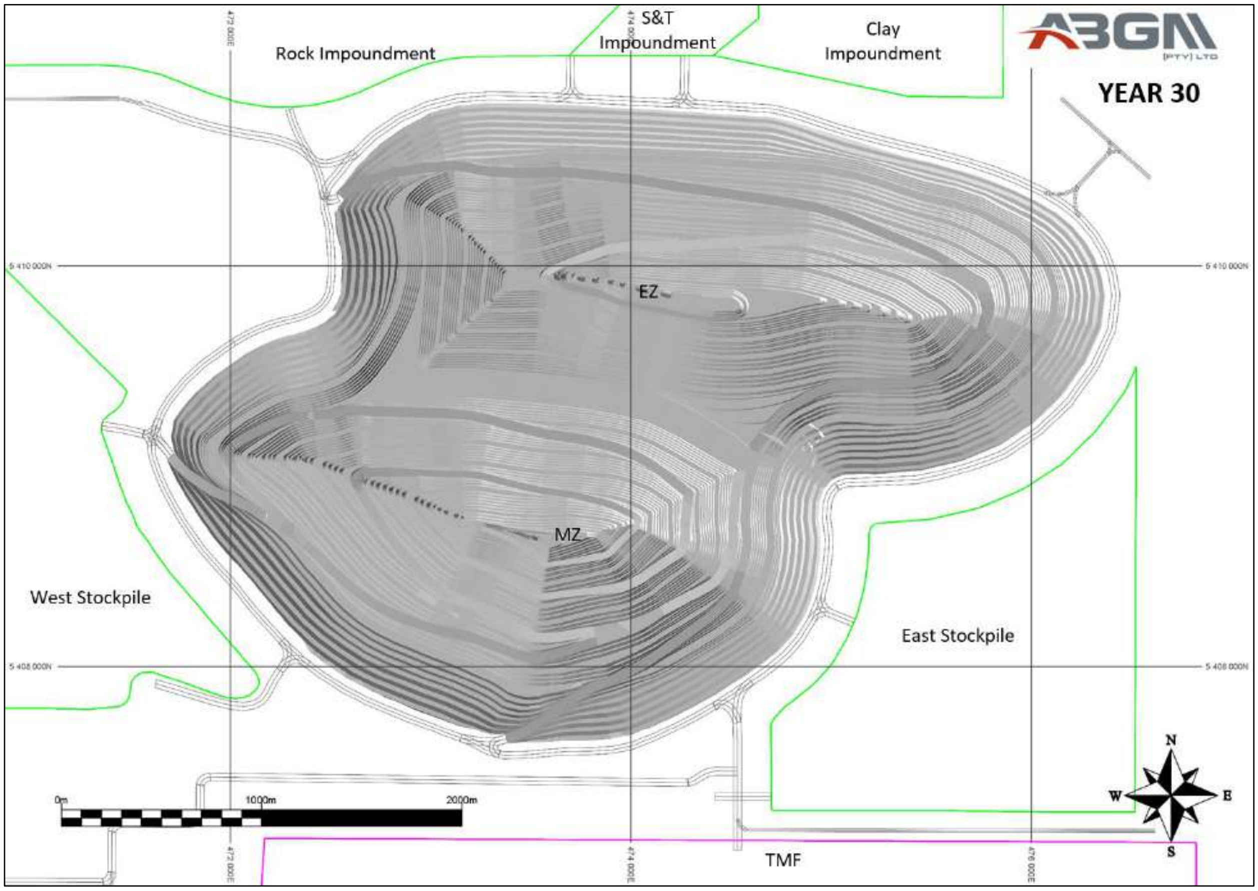

Source: CNC, 2023.

**Figure 16-26: Crawford Mine Development, End of Year 25**




Source: CNC, 2023.

**Figure 16-27: Crawford Mine Development, End of Year 30 (End of Open Pit Life)**




Source: CNC, 2023.

### 16.3.6 Ore Stockpiles

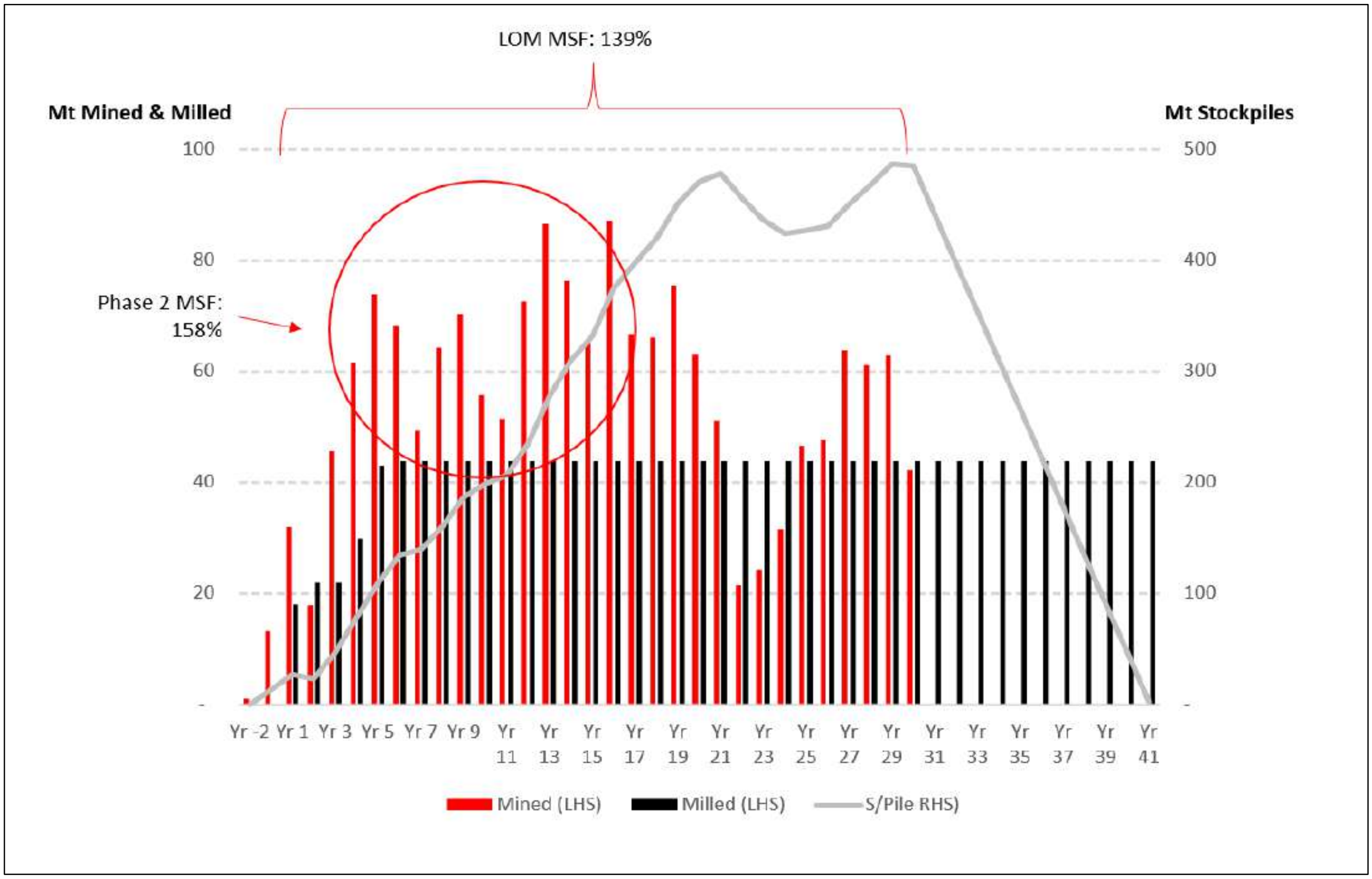
A key element of the mining strategy is the decoupling of mine and mill production rates, with the mine releasing ore at a faster rate than can be digested by the mill. This has three significant benefits, as follows:

1. Higher value ore is directly fed to the mill while lower value ore is temporarily stockpiled. Over the active life of the open pits, the average mine speed factor (MSF) is 139% (i.e., the mining rate of ore from the pit is 39% higher than the milling rate. During the period the Main Zone is active, the MSF is higher at 158% (Figure 16-28)

2. The mined-out pits provide an impoundment for tailings. Crawford's mine plan further enhances this benefit by mining the pits sequentially, with the Main Zone available for impoundment while the East Zone is still active. Over the life of mine, 61% of total tailings production will be impounded in-pit.

3. There are buffer stocks to ensure the mill is continually fed during any short-term disturbances in the mine resulting from mechanical, geological, climatic, or other factors.

**Figure 16-28: Ore Production and Stockpiles**




Source: CNC, 2023.

The primary downside of stockpiling is the additional cost of rehandling ore. At Crawford, this cost will be minimized by the scale of equipment used (rope shovels and 290 t AHS trucks). Additionally, the lower unit costs resulting from higher mining rates while the pits are active will largely offset subsequent rehandle costs and, on an undiscounted basis, total costs will be approximately neutral.

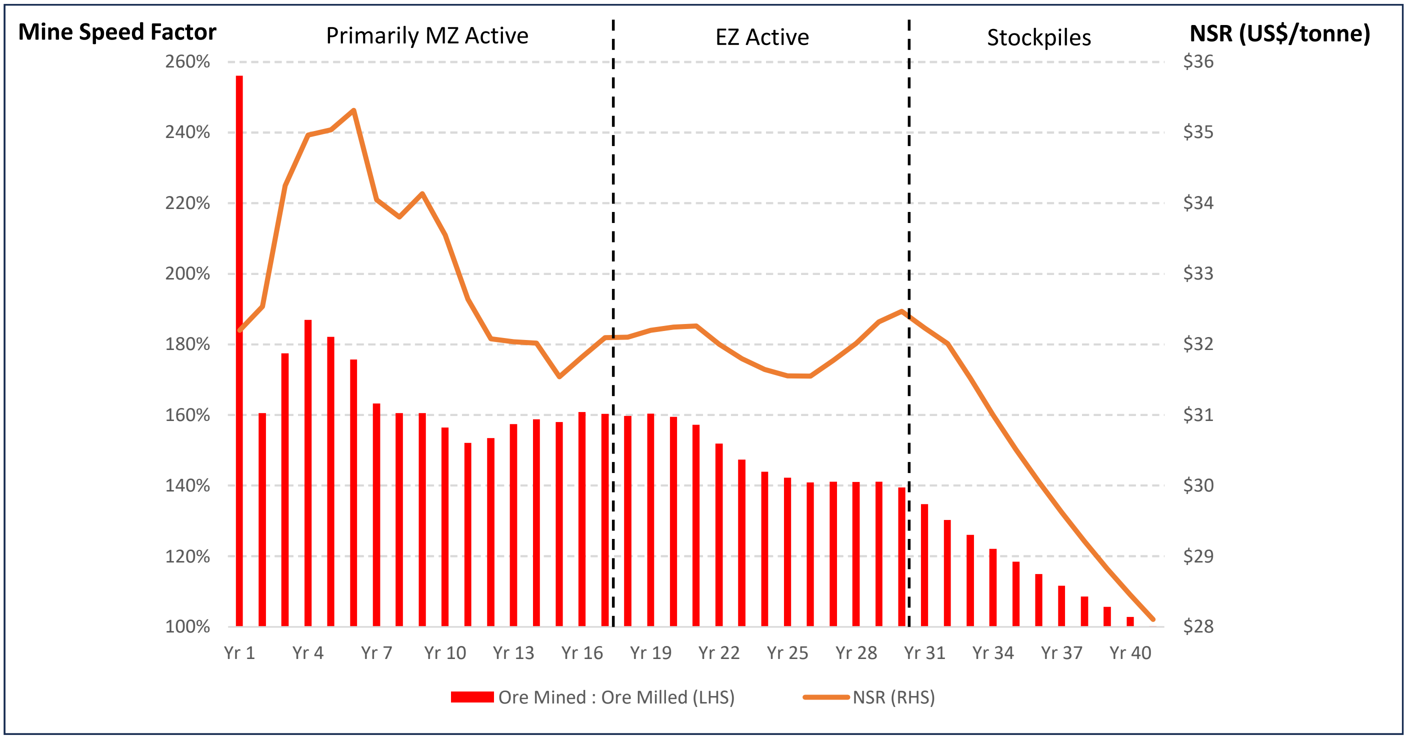
Both stockpiles will employ overall slope angles of 6H:1V. This will be achieved through a combination of the angle of repose (nominally 37°) and a perimeter access road on each lift.

The east stockpile will briefly reach its maximum design height of 100 m in Year 20, and will average 50 m elevation during the 43 years it is active (including pre-stripping).

The west stockpile has also been designed to 100 m; the current plan results in it reaching a maximum elevation of 70 m and averaging 30 m over its 37-year active life.

Figure 16-29 illustrates the impact of MSF on the NSR value of ore milled. During the initial year of operation, the primary source of ore is EZ1 and the average value milled is C$32/t. Thereafter, as production transitions to the higher value and lower stripping ratio MZ1, the MSF can be ramped up to a cumulative value that exceeds 180%, resulting in the cumulative NSR exceeding C$35/t. Thereafter, lower value and higher stripping ratio stages result in the cumulative NSR dropping below C$32 in the last years of Main Zone operation. Thereafter, the NSR improves, first as a result of lower stripping ratio / higher MSF EZ3-4, then subsequently as a result of the higher NSR (but lower MSF) EZ5.

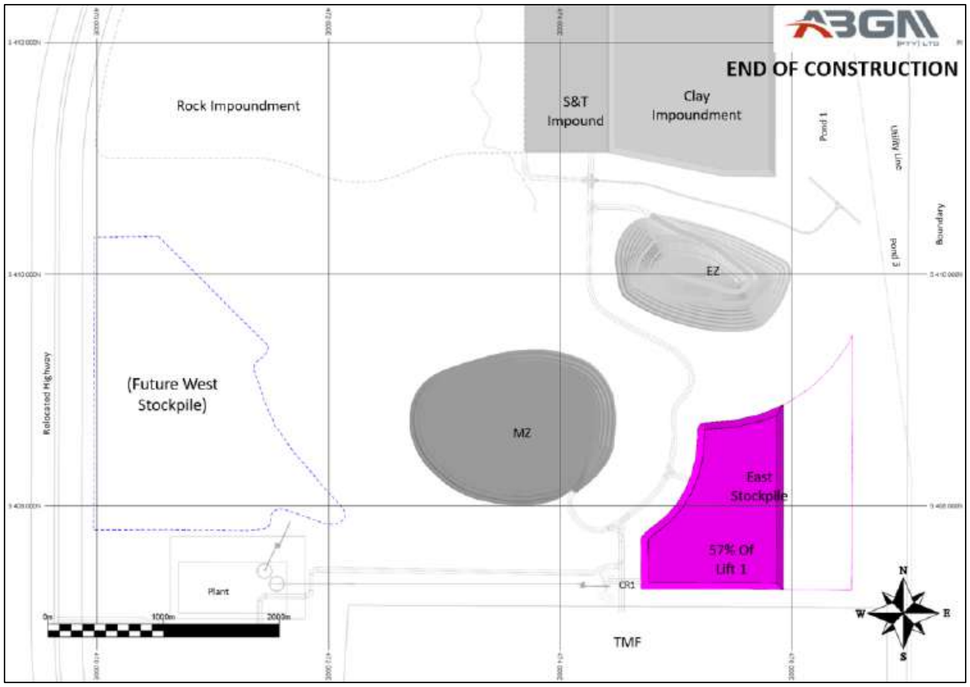
**Figure 16-29: Cumulative NSR vs. MSF**




Source: CNC, 2023.

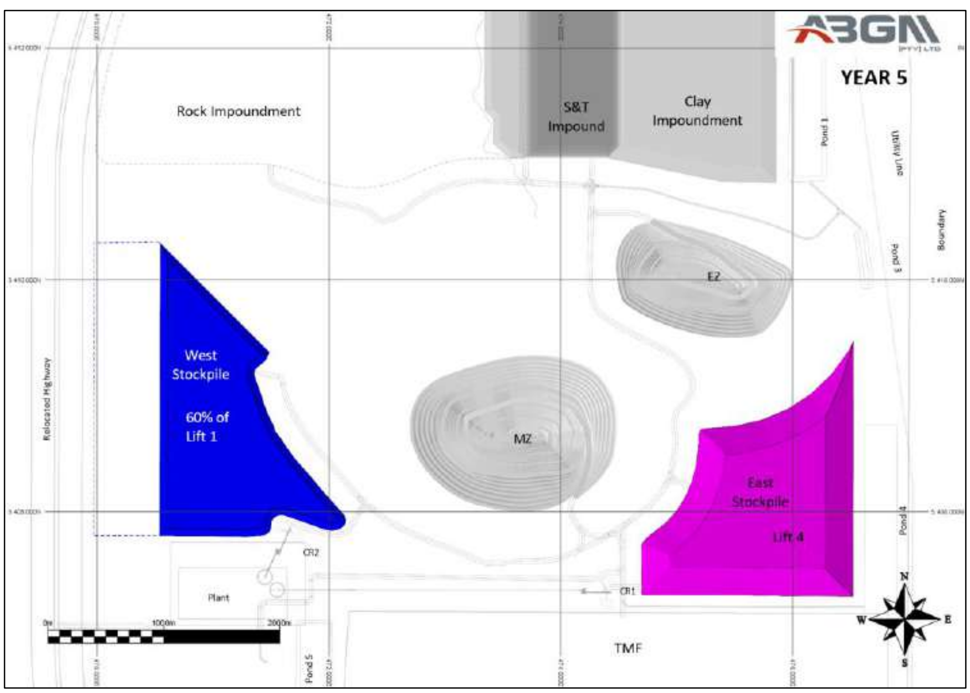
There will be two distinct stockpiles. Development of the east stockpile, located near crusher 1 (Cr1) will commence during the construction period. Development of the west stockpile will begin during commissioning of the mill expansion, which includes Cr2. The surface footprint available for the west stockpile is larger than that available for the east stockpile and that stockpile will consequently receive approximately 64% of total ore to be stockpiled while the two facilities are active. The west stockpile will also be drawn down at a faster rate while the pits are active to ensure both stockpiles have approximately equal inventory when the pits are depleted. In this manner, rehandle haulage costs will be minimized (the additional distance to haul from the east stockpile to Cr2 is 3.2 km while hauling from the west stockpile to Cr1 adds 3.6 km). Figures 16-30 to 16-34 illustrate the progression in stockpile development over the life of mine.

**Figure 16-30: Stockpile Development, End of Construction**



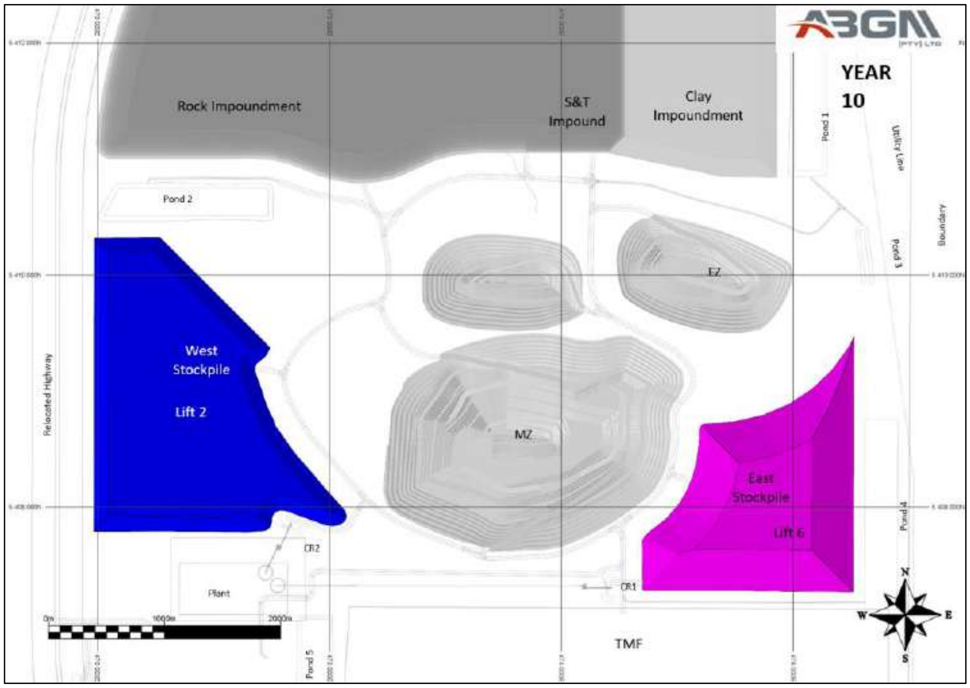

Source: CNC, 2023.

**Figure 16-31: Stockpile Development, End of Year 5**



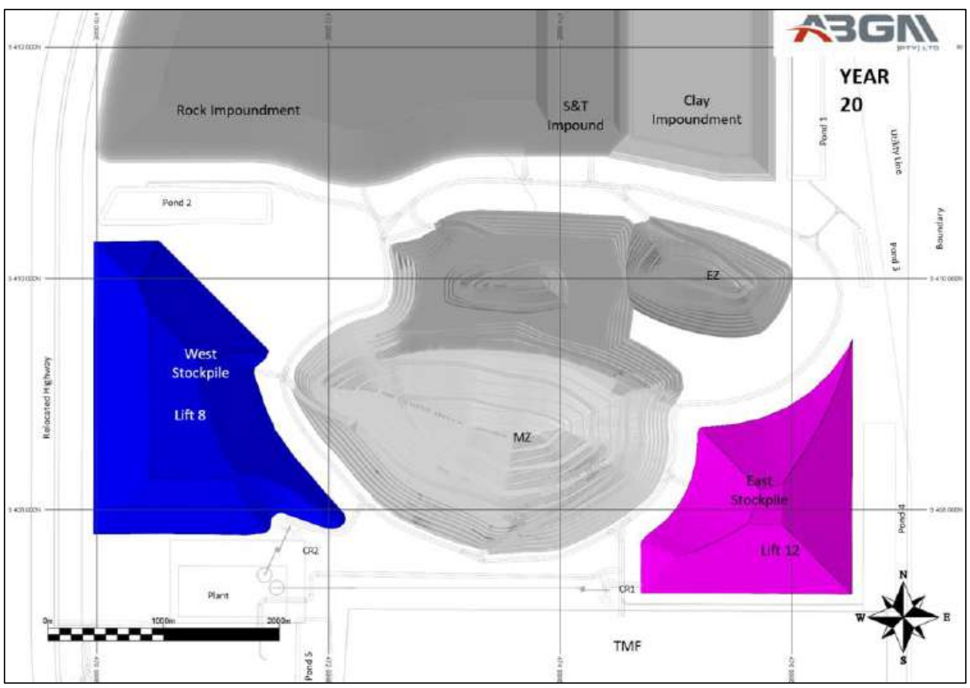

Source: CNC, 2023.

**Figure 16-32: Stockpile Development, End of Year 10**



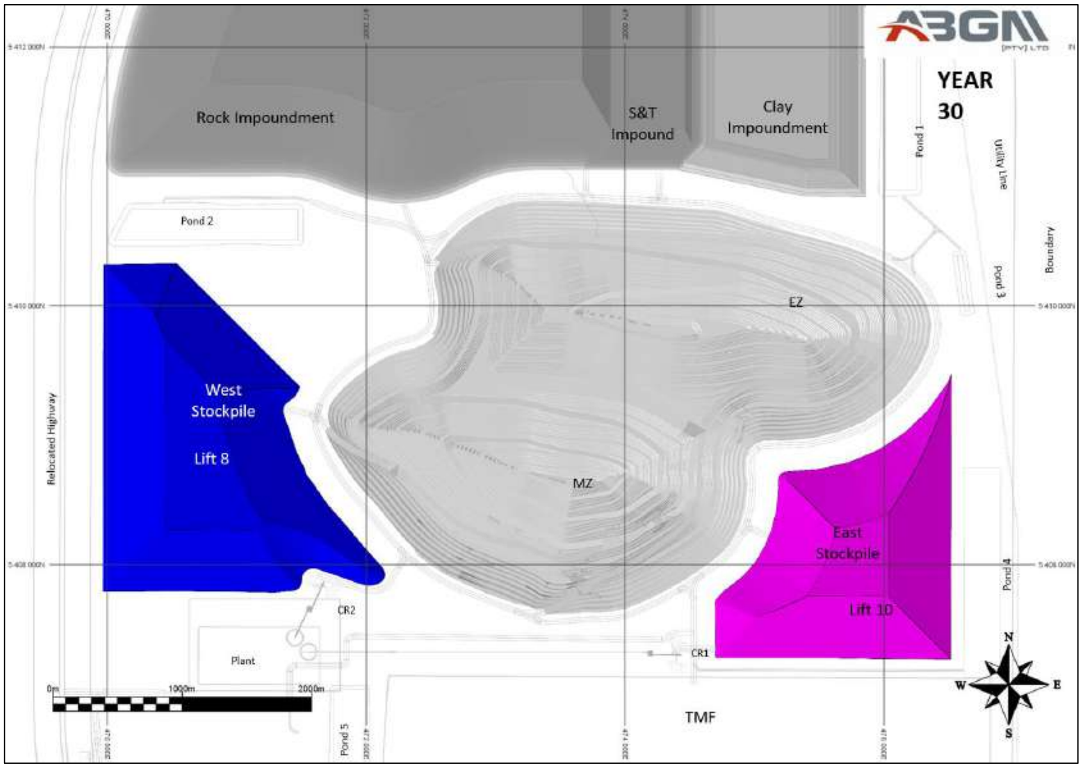

Source: CNC, 2023.

**Figure 16-33: Stockpile Development, End of Year 20**




Source: CNC, 2023.

**Figure 16-34: Stockpile Development, End of Year 30 (End of Open Pit Life)**




Source: CNC, 2023.

### 16.3.7 Waste Impoundments

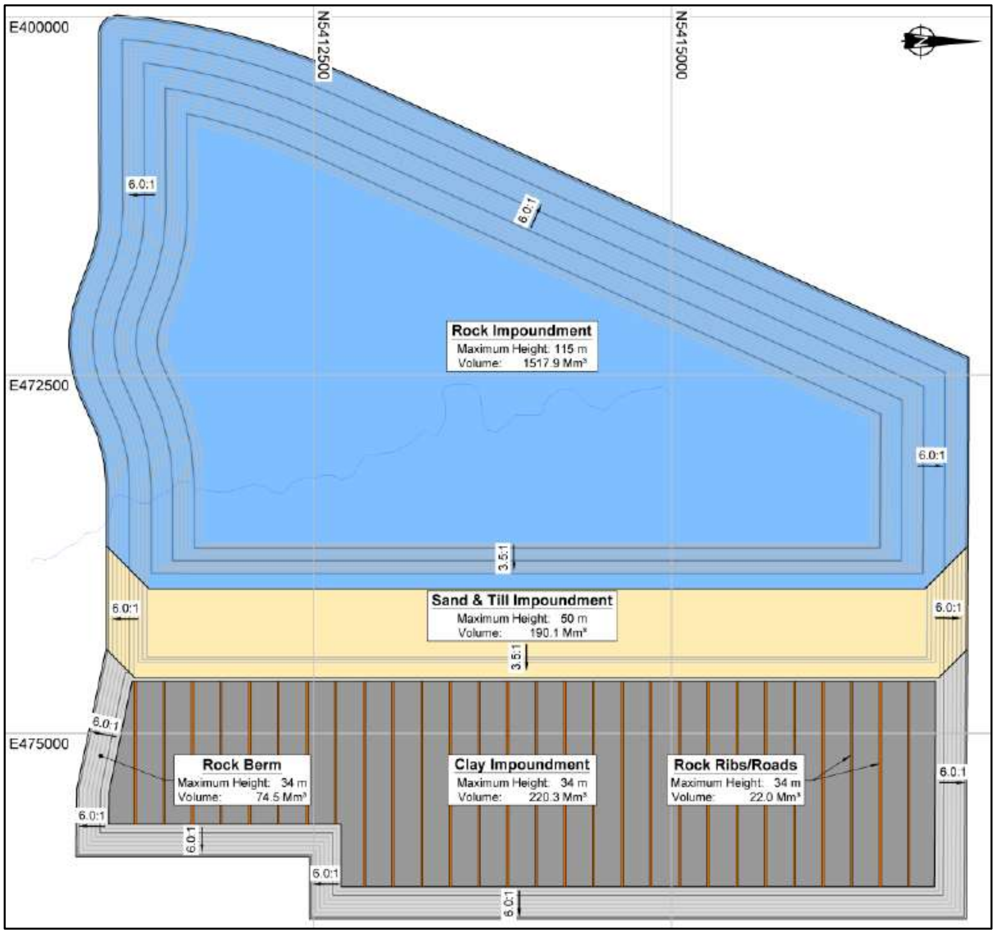
In addition to 1,715 Mt ore, the pit excavation includes the following quantities of waste:

- 14 Mt peat
- 270 Mt clay
- 336 Mt sand and till
- 3,372 Mt waste rock.

Materially, all the peat will be consumed reclaiming disturbed areas, with 5,700 ha disturbance covered to a nominal depth of 15 cm. Approximately 12% of the peat will be transported directly to the site being reclaimed. The remainder will be temporarily stockpiled at numerous sites, close to the location of ultimate use.

The remaining waste materials not used for reclamation or construction will be impounded within a single, large facility located north of the open pits. This facility will be subdivided into three areas, depending upon the material being impounded (Figure 16-35). Note that the capacity and ultimate elevations shown for each section represent design quantities. As the actual amounts that will be impounded are less, the maximum elevations will be marginally lower.

**Figure 16-35: Crawford Waste Impoundment**




Source: SRK, 2023.

Clay has been sub-classified as follows, based on its composition within the 7.5 m bench:

- Clay 1 will be loaded from a footwall located in clay and thus comprises the entire bench and will be loaded with a small backhoe excavator into an articulated truck. Clay 1 represents 81% of the total clay to be excavated.
- Clay 2 will be loaded from a footwall located in sand and till, thus allowing for use of a larger face shovel and 90 t trucks, and comprises greater than or equal to 30% of the total material within the bench. The remainder will be sand and till. Clay 2 represents 18% of total clay. Sand and till mixed with Clay 2 has been categorized as “S&T1.”
- Clay 3 represents the remaining 1% of total clay and will also be loaded from a footwall located in sand and till but will comprise less than 30% of the total bench height. Sand and till mixed with Clay 3 has been categorized as “S&T2.”

Approximately 3 Mt Clay 1 will be used in construction of the TMF. The remainder and all Clay 2 / S&T1 will be stored in a clay impoundment located on the eastern side of the waste facility. To facilitate storage of clay within the impoundment, it will be surrounded by a perimeter constructed of waste rock. Downstream slopes for this perimeter will be the same as for the stockpiles at 6H:1V. The crest will be 15 m wide, which is sufficient for the articulated trucks that will deliver clay into the impoundment.

Delivery of clay will also be facilitated by ribs constructed from waste rock using the centreline method and spaced at 200 m. With this construction method, the clay impoundment is a major consumer of waste rock; to its ultimate design height of 34 m, the 220 $Mm^3$ storage capacity requires 96 $Mm^3$ waste rock for construction. An opportunity to reduce the quantity of waste rock required through the stabilization of clay with lime and its impact on the mine plan will be discussed in Section 25.16.

Sand and till has been sub-classified in a similar manner as clay:

- S&T1 is associated with Clay 2, comprising less than or equal to 70% of the total bench height. S&T1 represents 11% of the total sand and till.
- S&T2 is associated with Clay 3, comprising more than 70% of the total bench height and represents 8% of the total.
- S&T3 comprises the full 7.5 m bench and represents 64% of the total.
- S&T4 overlies rock and will be cleaned off, either by dozer or backhoe, prior to drilling of blastholes and represents the remaining 17% of the total.

Approximately 9% of total sand and till will be used for revegetation (lining rock faces prior to addition of organic peat) or in construction of the TMF. As previously discussed, S&T1 will be impounded in the clay cells. The remaining sand and till (and Clay 3) will be impounded in a zone sandwiched between those for clay and waste rock. The long-term stability of the zone will benefit from its being buttressed by rock walls to the east (i.e., the rock perimeter wall for the clay cells) and west. As with the clay impoundment, it will be tipped in 2.5 m lifts. This zone has been designed to an ultimate height of 50 m.

Waste rock will be stored in the western portion of the facility. The 1,438 $Mm^3$ impounded will reach a maximum height of 110 m, compared to the design height of 115 m. The downstream slopes will be as per the stockpiles, at 6H:1V.

### 16.3.8 Tailings Management Facility

This section discusses mining-related aspects of the TMF. For more details on the TMF in general, refer to Section 18.10.

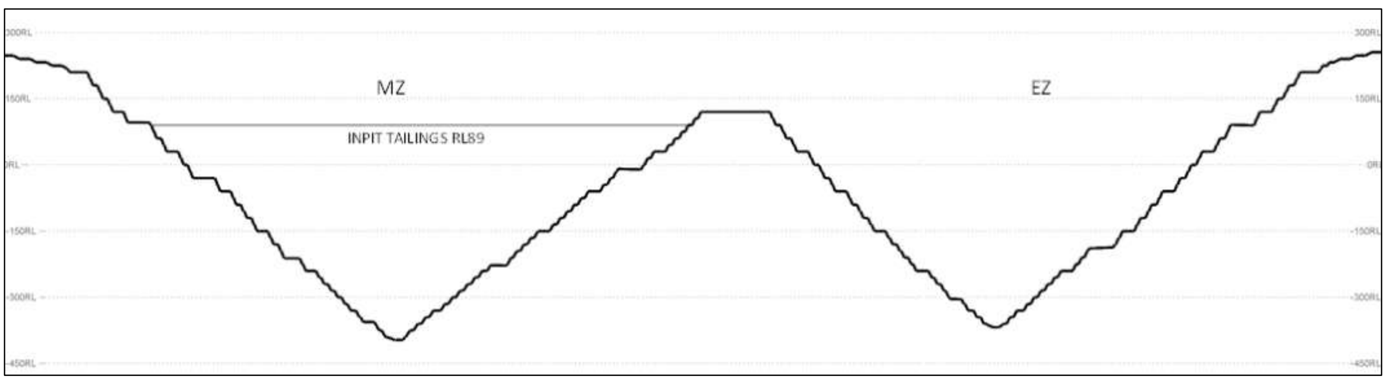
Crawford tailings will capture and store carbon dioxide. In addition to the positive environmental impacts, the chemical reaction effectively cements the tailings into a substance with significantly improved trafficability. Based on the witnessed transformation of tailings at the Dumont Project (which are of similar mineralogy as Crawford) after being exposed to atmospheric concentrations of $CO_2$ (i.e., significantly lower concentration than will be achieved at Crawford with IPT), the ability to locate infrastructure for the delivery of tailings on recently delivered tailings has been assumed. Tailings will therefore be delivered along the centreline of the TMF, with the beach sloping down to the perimeter embankments. The TMF measures approximately 5 km (N-S) x 4.5 km (E-W) and is currently expected to have a beach slope of 2.5%. Consequently, while the facility reaches a maximum elevation of 61 m, the embankments are relatively low, averaging approximately 17 m elevation around the entire perimeter. The deposition method has the added advantage of allowing the initial 104 $Mm^3$ tailings to be contained by topography, with the initial lift of the embankment not required to be complete before Year 5 of mill operations.

The TMF will consist of a clay core enclosed within a filter zone of sand, and a transition zone of crushed rock before the downstream and upstream shells that will be constructed from waste rock. Clay will be delivered during a six-month period (May to October). The timing of clay release from the pits will allow 80% of required clay to be delivered directly, with the remainder temporarily stockpiled. All other materials will be delivered throughout the year.

The current TMF design has a capacity of 495 $Mm^3$. This will be sufficient for a life exceeding 17 years, resulting in annual raises to the embankment of approximately one metre. This life also results in the depleted Main Zone pit being available for receipt of tailings six months before required.

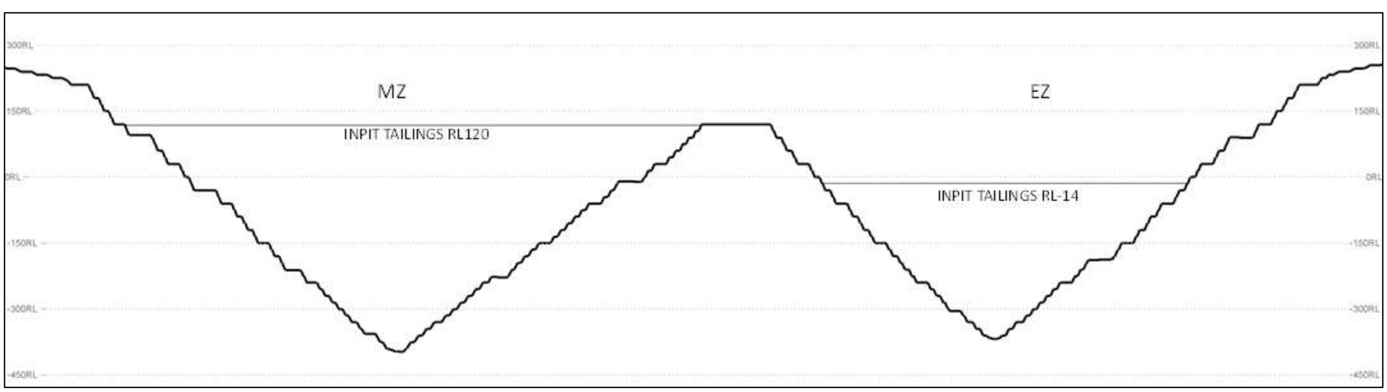
After the capacity limit of the TMF has been reached, deposition will transition to the depleted Main Zone pit. This will provide capacity for a further 484 $Mm^3$, representing 15 years mill production. By this stage, mining in the East Zone would have ceased 27 months earlier and deposition would transition to the East Zone void. Figure 16-36 and Figure 16-37 provide a cross section of in-pit tailings deposition in Year 30 (at the end of activities in the East Zone) and Year 42 (at the end of project life).

**Figure 16-36: In-Pit Tailings Deposition – Year 30**




Source: CNC, 2023.

**Figure 16-37: In-Pit Tailings Deposition – Year 42**




Source: CNC, 2023.

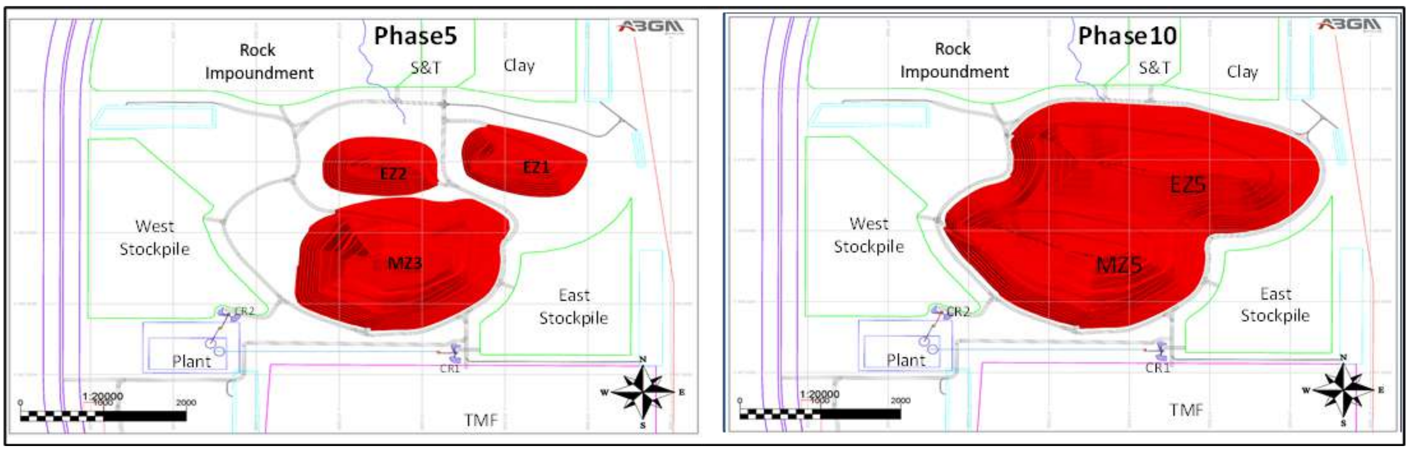
### 16.3.9 Haul Roads

Over the life of project, 44.6 km of surface haul roads will be constructed. Figure 16-38 illustrates stages in the evolution of the road network. The maximum extent of interior roads is realized by Phase 5, with access provided to the starter East Zone pits (EZ1 and EZ2) as well as the interim Main Zone stage, MZ3. By the end of life, these have been replaced by single road covering the entire perimeter of the pits.

To minimize dust and maximize tire life, roads will be constructed using only gabbro and basalt rock types. Additionally, allowance has been made to cover all roads with crushed aggregate from the roadstone. A total of 36.2 Mt of roadstone will be placed over the life of mine.

Surface haul roads for the 290 t class trucks will be 35 m wide. Where trolley assist is used on in-pit ramps, the width will be 50 m, which allows for trolley infrastructure and an extra lane to pass any vehicle (including service vehicles) that may be stopped under the trolley line. Other roads, will measure 15 m wide.

**Figure 16-38: Evolution of Surface Haul Roads**




Source: CNC, 2023.

## 16.4 Mining Process Description

### 16.4.1 Overview

Open pit mining operations at Crawford will be performed by a mixed fleet of mining equipment as shown in Figure 16-39 and described below:

- Areas where the footwall is in clay will be mined with 120 t class backhoe excavators loading 40 t articulated trucks.
- Areas where the footwall is in sand and till will be mined with 300 t electric face shovels loading 90 t trucks. This will include clay contained in mixed clay / sand and till benches.
- A bench height of 7.5 m will be employed to RL180 (approximately 90 m below the mean surface elevation), which is below the lowest horizon where overburden will be encountered. The 1,037 Mt of rock contained within these

benches (63% of all 7.5 m bench material) will be mined predominantly with 700 t face shovels. A lesser tonnage of rock will be loaded by 50 t payload front-end loaders and 100 t payload rope shovels. All three loading units will load 290 t trucks equipped with AHS and trolley assist.

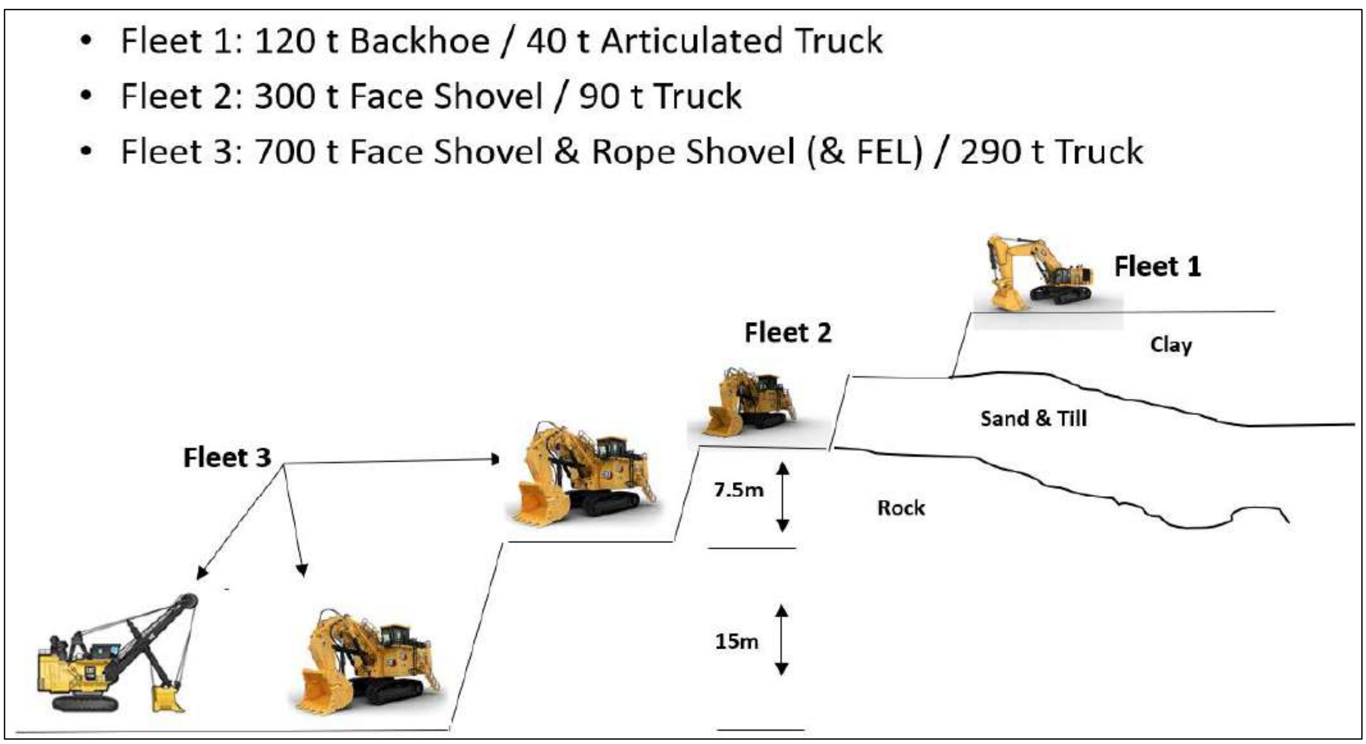
- The 4,047 Mt rock that will be mined below RL180 will be predominantly loaded by rope shovels, supported by face shovels and front-end loaders. Over the life of mine, 2% of total rock will be loaded by front-end loaders, 30% by face shovels and the remaining 68% by rope shovels.
- No drilling and blasting of overburden will be required. For pioneer operations on the initial bench of rock mining, small diesel powered and conventionally operated drills will be used to drill 127 mm blast holes. Below this initial bench, larger electrically powered units equipped with an autonomous drilling system (ADS) will be used for drilling 229 mm blast holes on 7.5 m benches and 271 mm blast holes on 15 m benches. Final walls will be pre-split. Pre-split holes will be drilled using the same machine as for pioneering.

Production equipment will be supported by various units of support equipment, including tracked dozers, wheel dozers, front-end loaders, graders, water tankers and utility excavators.

A mining contractor will be used to expedite the start-up, with particular focus on sourcing aggregate from off-site and establishing the initial benches in clay. Thereafter, all mining fleet will be Owner-operated.

The duty cycle for production units was estimated by first principles, based on the production plan.

**Figure 16-39: Crawford Mining Fleets**




Source: CNC, 2023.

The following infrastructure will be provided to support mining activities:

- workshop and associated warehouse
- storage tanks for fuel an associated fuelling infrastructure
- explosives manufacture facility and magazine (as is the norm in Canada, this will be operated by the explosives supplier)
- roadstone crusher to produce aggregate for maintaining roads, along with sized material required for other infrastructure, such as the TMF
- input dewatering system
- electrical reticulation system.

As Ontario's electricity is generated primarily using non-carbon sources (hydro and nuclear), the diesel-powered equipment and blasting associated with the open pit will be the primary source of $CO_2$ emissions. With current haul truck technology, the open pit will be responsible for 83% of the estimated 7.5 Mt $CO_2$ scope 1 and 2 emissions by Crawford over its life of mine. Crawford's emissions will represent only 14% of the capture and storage potential of Crawford's tailings using the IPT process. As a result, Crawford will achieve net-negative status and not be required to pay carbon taxes on fuel purchases. The 46.9 Mt of capture and storage capacity that are above and beyond its own requirements will be sold to third parties, with the associated revenues based on current carbon prices.

The open pit labour complement will average 418 full-time equivalents (FTEs) over the life of mine, reaching a peak of 1,135 during the early years when maximum stripping of clay is being performed. During the final 11 years of operation, when the mill will be fed entirely from reclaimed stockpiles, the complement will average 89 FTE.

### 16.4.2 Mining Fleet

#### 16.4.2.1 Summary

The required number of production equipment was calculated based on the following assumptions:

- The mine will operate 24 hours per day, 365 days per year.
- The mechanical availability and utilization of equipment will vary based on the unit of equipment and the degree of technology employed, with autonomous equipment achieving higher levels than manually operated ones.

Table 16-9 summarizes the main units that will be employed in the open pit.

**Table 16-9: Crawford Mining Fleet**

| Process | Unit | Application | Size | Examples |
|---|---|---|---|---|
| Drilling | Diesel Percussion Drill | Pioneer Operations<br>Pre-Splitting | 127 mm hole<br>114 mm hole | Epiroc T45<br>Sandvik DX800 |
| | Electric Production (Rotary or Percussion) Drill | Rock | 229 mm hole / 7.5 m bench<br>271 mm hole / 15 m bench | Epiroc PV271<br>Sandvik DR412i |
| Loading | 120 t Diesel Excavator | Backhoe Clay | 6 $m^3$ bucket (8 t) | Cat 6015 / Komatsu PC1250 |
| | 300 t Electric Excavator | Face Shovel, Overburden | 17 $m^3$ bucket (30 t) | Cat 6030 / Komatsu PC 3000 |
| | 700t Electric Excavator | Face Shovel, Rock | 34 $m^3$ bucket (60 t) | Cat 6060 / Komatsu PC 7000 |
| | Electric Rope Shovel | Rock | 60 $m^3$ bucket (100 t) | Cat 7495 / P&H 4100 |
| Hauling | Articulated Truck | Matched to Backhoe | 40 t payload | Cat 745 / Komatsu HM400 |
| | 90 t Truck | Matched to 290 t Face Shovel | 90 t payload | Cat 777 / Komatsu HD785 |
| | 290 t Truck | Rock | 290 t payload | Cat 794 / Komatsu 930E |
| Support Equipment | Large Front-End Loader | Cleanup and rehandle to 290 t trucks | 50 t payload | Komatsu WE1850 |
| | Medium Front-End Loader | Cleanup and rehandle to 120 t trucks | 20 t payload | Cat 992 / Komatsu WA900 |
| | Small Front-End Loader | Cleanup and rehandle to articulated trucks | 10 t payload | Cat 988 / Komatsu WA600 |
| | Large Dozer | Support Rock Operations | 800 HP | Cat D11 / Komatsu D475 |
| | Medium Dozer | Support Mixed Overburn Operations | 600 HP | Cat D10 / Komatsu D375 |
| | Small Dozer | Support Clay Operations | 300 HP | Cat D8 / Komatsu D155 |
| | Wheel Dozer | Road Clean Up | 600 HP | Cat 834 |
| | Large Grader | Maintain Permanent Roads | 18 ft blade | Cat 18M / Komatsu GD955 |
| | Small Grader | Maintain Roads in Overburden | 14 ft blade | Cat 14M / Komatsu GD655 |
| | Water Tanker | Suppress Dust | 150 t payload | Cat 777 / Komatsu HD 785 |
| | Utility Excavator | Construction (digging trenches)<br>Wall Scaling<br>Mobile Rock Breaker | | Cat 390<br>Komatsu PC900 |

The average fleet size for the various units is given in Table 16-10.

**Table 16-10: Crawford Mining Fleet Numbers**

| Unit | Pre-Strip | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr6-10 | Yr11-15 | Yr16-20 | Yr21-25 | Yr26-30 | Yr31 + |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Percussion Drill | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 2 | 1 | 1 | 1 | 0 |
| Production Drill | 2 | 4 | 3 | 5 | 6 | 4 | 6 | 6 | 6 | 6 | 3 | 0 |
| Backhoe | 6 | 9 | 9 | 8 | 6 | 7 | 5 | 3 | 3 | 0 | 0 | 0 |
| 290 t Excavator | 2 | 2 | 2 | 2 | 2 | 3 | 2 | 2 | 2 | 0 | 0 | 0 |
| 700 t Excavator | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 0 |
| Rope Shovel | 0 | 0 | 0 | 0 | 2 | 2 | 3 | 3 | 3 | 3 | 3 | 1 |
| Articulated Truck | 44 | 89 | 94 | 84 | 50 | 68 | 38 | 16 | 15 | 1 | 0 | 1 |
| 120 t Truck | 10 | 15 | 28 | 25 | 30 | 33 | 18 | 15 | 12 | 1 | 1 | 1 |
| 290 t Truck | 5 | 17 | 15 | 16 | 30 | 38 | 55 | 56 | 52 | 54 | 38 | 5 |
| Small Dozer | 6 | 9 | 8 | 8 | 7 | 9 | 7 | 5 | 4 | 1 | 0 | 1 |
| Medium Dozer | 3 | 4 | 3 | 4 | 4 | 4 | 4 | 3 | 2 | 0 | 0 | 1 |
| Large Dozer | 4 | 5 | 6 | 4 | 9 | 9 | 10 | 9 | 8 | 7 | 5 | 0 |
| Wheel Dozer | 2 | 2 | 2 | 1 | 1 | 1 | 2 | 2 | 1 | 1 | 1 | 0 |
| Small Grader | 2 | 3 | 3 | 3 | 3 | 3 | 2 | 2 | 1 | 1 | 0 | 1 |
| Large Grader | 1 | 2 | 3 | 2 | 3 | 4 | 4 | 4 | 4 | 3 | 2 | 1 |
| Water Tanker | 1 | 2 | 2 | 2 | 2 | 2 | 3 | 2 | 2 | 1 | 1 | 1 |
| Small Front-End Loader | 1 | 1 | 1 | 1 | 1 | 1 | 2 | 1 | 1 | 1 | 0 | 1 |
| Medium Front-End Loader | 2 | 2 | 2 | 1 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Large Front-End Loader | 1 | 2 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 | 0 |
| Utility Excavator | 5 | 6 | 5 | 3 | 3 | 3 | 5 | 6 | 4 | 3 | 3 | 0 |

#### 16.4.2.2 Production Drilling and Blasting

Geotechnical parameters for the various Crawford rock types were determined from testwork performed by SRK and provided to an explosives OEM (Dyno-Nobel), who performed simulations to predict the required powder factor. These simulations indicated that an acceptable particle size distribution for all rock types could be achieved with the following powder factors:

- Pioneer Benches; average = 0.25 kg/t, range = 0.23 (peridotite) – 0.35 (gabbro)
- 7.5 m Benches; average = 0.29 kg/t, range = 0.25 (pyroxenite) – 0.35 (gabbro)
- 15 m Benches; average = 0.25 kg/t, range = 0.22 (pyroxenite) – 0.32 (gabbro)

The resulting drilling patterns will range from approximately 3.3 to 3.9 m (square) for pioneer benches, increasing to 4.9 to 5.8 m (square) for 7.5 m benches and 7.7 to 8.9 m (square) on 15 m benches. In all cases, the smaller dimension is for higher S.G. gabbro while the increased dimension is for lower S.G. peridotite and dunite. Over the life of mine, an average of 137 tonnes broken is yielded per metre drilled.

There are no outcrops at Crawford that would allow for testing by drilling OEMs and the assumed penetration rates and bit life are based on the performance from peer operations and projects. The assumed instantaneous rates are as follows:

- Pioneer drilling in all rock lithologies: 95 m/h
- 229 mm and 271 mm drilling in 'soft rocks' (dunite, peridotite and talc): 60 m/h
- 229 mm drilling in 'hard rocks' (pyroxenite, gabbro and basalt): 36 m/h
- 271 mm drilling in 'hard rocks' (pyroxenite, gabbro and basalt): 33 m/h.

The 'hard' rocks comprise 60% of the total rock that will be mined and the life-of-mine weighted average penetration rate is 41.7 m/h. The calculated working time for drills also included the following. (In all cases, these values reflect the planned use of ADS – typical values for manually operated drills are given in parenthesis):

- An allowance for re-drilling 1% of holes (manually operated drills: 3%).
- Delays moving between holes: 2 min/hole (manually operated drills: 5 min/hole).
- Delays moving between patterns: 6% of total percussion and move between holes time (manually operated drills: 15%).

After including these delays, overall ADS blast hole drill productivity is estimated at 35.2 m/h (with manually operated units, this value would fall to 28.1 m/h).

The productivity calculations take account of the planned implementation of a high-precision GPS (HPGPS) guidance, monitoring, and rock recognition system for the fleet of large drills. A key benefit of the system is elimination of the requirement for surveyors to stake the X-Y collar positions of holes and conduct subsequent checks of drilled holes. Other benefits of the system include the following:

- A more correct determination of collar elevation, improving footwall control and minimizing over-drilling.
- Recognition of rock types encountered down the hole will allow more accurate mapping of lithologies, which will improve planning – including the design of blast patterns to reduce dilution and ore loss at contacts.
- The technology ensures more consistent drill performance, particularly when lithologies change down the hole. In turn, this ensures closer adherence to the drill instructions, regardless of the operator experience and level of skills.

The HPGPS guidance, monitoring and rock recognition system is also an essential building block for operation of autonomous drill systems (ADSs), which is discussed in Section 16.4.4.

Rock would be blasted using emulsion produced at an on-site facility operated by the explosives OEM. Typically, blasting would be performed 2 to 3 times weekly, to minimize blast associated delays. Multiple patterns would be blasted on a single blast day.

#### 16.4.2.3 Pre-Splitting

The mine design assumes that all final walls will be pre-split. The design of pre-split blasts was based on simulations performed by Dyno-Nobel, which indicated that with 12.5 kg of explosive placed in a 15 m x 114 mm diameter hole, hole spacings ranging from, a minimum of 1.1 m (pyroxenite) to a maximum of 1.7 m (gabbro). A weighted average of 1.48 m was then estimated, based on the volumes of the different rock types.

The total pre-splitting requirement was based on a combined measured 917 km of total final wall perimeter over the 46 rock benches that would be mined in each of the Main and East zones. The resulting 544,660 presplit holes would require 8,876 km drilling. Pre-splitting would start in advance of the initial final walls becoming exposed in the Main Zone, continue for a period of 27 years and be performed by a fleet of two drills.

#### 16.4.2.4 Loading and Hauling

Multiple fleets of load and haul equipment will be employed to ensure the various material types at Crawford will be mined most productively. Smaller fleet will be used for overburden, with areas where the footwall will be in clay stripped using 120 t backhoe excavators and 40 t articulated trucks and areas where the footwall is in sand and till mined using 300 t face shovels and 90 t trucks. Rock will be mined with a combination of large front-end loaders, 700 t face shovels and rope shovels. The backhoe excavators and front-end loaders will use diesel while the larger face shovels and rope shovels will be electric.

The backhoes, 300 t face shovels and large front-end loaders will all be operated by an on-board operator. To take full advantage of AHS haul trucks, which will not be delayed for operator delays, the 700 t face shovels and rope shovels will be operated tele-remotely. Additionally, an additional operator will be provided for each fleet per shift, to facilitate operator breaks (e.g., lunch). For example, at peak production there will be two 700 t face shovels and three rope shovels operational. These will be operated by a team of three face shovel operators and four shovel operators on each shift. Criteria used to calculate the productivity of various loading units are given in Table 16-11.

**Table 16-11: Crawford Loading Unit Design Criteria**

| Parameter | Units | 120 t Backhoe | 300 t Face Shovel | 700 t Face Shovel | Rope Shovel |
|---|---|---|---|---|---|
| Average Bucket Factor | tonnes | 11.3 | 27.4 | 65.0 | 100.0 |
| Average Truck Payload | tonnes | 39 | 89 | 290 | 290 |
| Total Passes per Load[1] | number | 3.75 | 3.55 | 4.75 | 3.25 |
| Cycle Time per Bucket | seconds | 40 | 30 | 30 | 30 |
| Spot Time | seconds | 30 | 30 | 30 | 30 |
| Total time to Load Truck | seconds | 180 | 137 | 173 | 128 |
| Availability x Utilization | % | 71% | 71% | 79% | 82% |
| Utilized Hours per Year | hours | 6,180 | 6,180 | 6,925 | 7,169 |
| Non-Productive Time per Year[2] | hours | 1,030 | 1,030 | 1,154 | 1,195 |
| **Maximum Productivity per Unit** | **Mt/a** | **3.9** | **11.7** | **33.6** | **47.0** |

Notes: **1.** Includes allowance of 0.25 passes/cycle to reflect inability to achieve target bucket factor with every load. **2.** Delays include waiting for trucks, shovel moves and blasts.

One constraint in mine planning was the capacity of the available fleet (as discussed in Section 16.3.5, a key degree of freedom was the start-up date and ultimate number for each of the various loading units). For rock mining, the total tonnage was preferentially allocated to rope shovels, followed by the 700t face shovel and the large front-end loader last. Consequently, the life-of-mine average productivity of shovels of 45.6 Mt/a is 97% of the theoretical maximum, while the 700 t face shovel (29.8 Mt/a) is 89% of the theoretical and the front-end loader is lower still at 85%. Mining rates in overburden are driven by the requirement to expose underlying rock (and not the capacity of loading units) and the actual productivity of the 300 t face shovels and backhoes is 78% and 91%, respectively, of the theoretical maximum.

Table 16-12 compares the economic life of the various units to the required usage. Note that usage for units loading the 290 t trucks (large front-end loader, 700 t face shovel and rope shovel) includes hours required for stockpile rehandle. For the 120 t backhoe, a minimum of eight units would be required. However, the peak fleet required to achieve the mine plan is 9 units, which is the number of units purchased. The minimum number of required 700 t face shovels and rope shovels exceeds the maximum number of operating fleets and two replacements units of each machine will be purchased.

**Table 16-12: Crawford Loading Unit Purchases**

| Parameter | Units | 120 t Backhoe | 300 t Face Shovel | Large Front-End Loader | 700 t Face Shovel | Rope Shovel |
|---|---|---|---|---|---|---|
| Economic Life | engine hours | 60,000 | 108,000 | 90,000 | 108,000 | 120,000 |
| Total Usage | engine hours | 449,041 | 167,621 | 42,302 | 326,044 | 579,599 |
| Minimum Required Units | number | 8 | 2 | 1 | 4 | 5 |
| Maximum Fleet Required[1] | number | 9 | 3 | 2 | 2 | 3 |
| Initial Purchases | number | 9 | 3 | 2 | 2 | 3 |
| Replacement Units | number | 0 | 0 | 0 | 2 | 2 |

Note: 1. Number of units required to achieve planned production schedule.

The bearing pressure of the 120 t backhoes is approximately 130 kPa and the experience at peer operations suggests that supporting with clay surface with cleared lumber will provide a sufficiently competent surface for these machines to operate. However, the bearing pressure of the articulated trucks that will be loaded is significantly higher, exceeding 300 kPa. The clay surface will therefore need 'armouring' with a nominal 1.5 m of crushed waste rock. The armour need only be applied to the roads travelled by the trucks, which represent approximately 65% of the total surface area. As the clay would be mined in 7.5 m benches, the required volume of armour is 13% of the volume of clay to be excavated. The life-of-mine requirement is 66 Mt of crushed rock. The experience at peer operations shows that approximately 75% of armour can be re-used from one bench to the next, reducing Crawford requirement to 19.5 Mt fresh rock, that would be supplied from mine waste.

Both the articulated trucks and 90 t trucks will be conventionally diesel-powered and operated by an on-board operator. However, the 290 t trucks will utilize both trolley assist, to reduce diesel consumption and improve speed on uphill hauls, and AHS, to reduce limitations associated with on board operators (including operator error and the need to take breaks). Both technologies are discussed in more detail in Section 16.4.4.

Design criteria for the various fleets of trucks are given in Table 16.13.

**Table 16-13: Crawford Hauling Design Criteria**

| Parameter | Units | Articulated | 90 t | 290 t |
|---|---|---|---|---|
| Payload | tonnes | 39 | 89 | 290 |
| Loading Time[1] | min / load | 3 | 2.3 | 2.4 |
| Dumping Time | min / load | 2 | 1 | 1 |
| Queuing Time | min / load | 2 | 2 | 2 |
| Speed – In-Pit Flat (Empty & Full) | km/h | 18.8 | 25.0 | 30.0 |
| Speed – Ex-Pit Flat (Empty & Full) | km/h | 26.3 | 35.0 | 50.0 |
| Speed – Uphill Loaded Conventional | km/h | 10.0 | 11.7 | 11.0 |
| Speed – Uphill Loaded Trolley | km/h | n/a | n/a | 22.0 |
| Speed – Downhill Empty | km/h | 22.5 | 40.0 | 59.0 |
| **Average Cycle Time[2]** | **min / load** | **36.1** | **29.6** | **32.1** |
| **Average Fuel Burn[2]** | **L/h** | **28** | **90** | **161** |
| **Average Productivity per Unit[2]** | **Mt/a** | **0.4** | **1.0** | **3.8** |

Notes: 1. Loading time varies as function of loading unit, weighted average of all units for 290 t trucks. 2. Averages over life of mine, given selected mine plan.

Table 16-14 compares the economic life of the various trucks to the required usage. There is potential scope for optimization of the mine plan, as the maximum fleet of 90 t trucks required to execute the mine plan (44 units) exceeds the minimum number required based on equipment life by 16 units. Note that the economic life of machines was not taken as a hard limit. Based on data provided by the various OEMs, maintenance costs were modelled as a logarithmic function and adjusted quarterly based on the average life of the operating fleet. Where the remaining duty cycle for a particular unit did not justify its replacement, the machine was run past its economic life with an associated increase in operating costs.

**Table 16-14: Crawford Truck Purchases**

| Parameter | Units | Articulated | 90 t | 290 t |
|---|---|---|---|---|
| Economic Life | engine hours | 30,000 | 80,000 | 100,000 |
| Total Usage | engine hours | 4,791,699 | 2,215,767 | 10,022,445 |
| Minimum Required Units | number | 160 | 28 | 101 |
| Maximum Fleet Required[1] | number | 99 | 44 | 72 |
| Initial Purchases | number | 99 | 44 | 72 |
| Replacement Units | number | 61 | 0 | 29 |

Note: 1. Number of units required to achieve planned production schedule.

The efficiency of the load and haul operation will be maximized through purchase of several technology systems, including:

- The fleet management system (FMS), which assigns, tracks, and monitors the fleet of mobile equipment. Assignments are continually updated, taking account of the current status of all equipment, and thus ensuring the plan is achieved with the minimal resources and operating expenditure.

- The tire monitoring systems, which involve the deployment of sensors within the tires on haul trucks and front-end loaders. These report the real-time pressure and heat data that can be used to generate assignments that maximize both safety and tire life. For example, a haul truck with tires approaching the thermal limit that would lead to a heat separation could be re-routed on profiles where lower speeds are realized, allowing the tires to cool.

- HPGPS guidance and monitoring, which is like the system that will be deployed on drills. This will allow loading units to load more closely to planned X-Y-Z boundaries, reducing dilution and ore loss while improving footwall conditions. The system will also be employed on support equipment such as dozers and graders, to ensure footwall and ramp conditions are maintained to the highest standard. For example, the FMS may identify a section of road where haul trucks are reducing speed. A dozer and/or grader would then be automatically dispatched to the exact location, where cutting and/or filling was required to return the road to grade. Potential damage to the truck suspension and/or tires is obviated while trucks can travel at faster speeds.

- Loading unit tooth detection, which uses cameras to monitor the status of dipper teeth and send an alarm when a tooth is missing. This allows identification of the haul truck containing the broken tooth before the tooth is delivered to a crusher and causes a blockage.

- Payload monitoring, which provides loading unit operators with the dipper-by-dipper mass delivered to a haul truck to ensure over- and under-loading is minimized. In turn, this maximizes utilization of haul trucks while minimizing excessive wear on the haul truck frame, suspension, and tires.

#### 16.4.2.5 Support Equipment

Open pit roads and working places will be maintained by a fleet of support equipment, including the following:

- Track dozers for ripping footwalls, cutting ramps, dozing rock and other construction activities. The fleet requirement has been estimated by the empirical relationship of 0.5 dozers per active loading unit with an additional unit for each active dumping location. Three classes of dozer have been provided, with 40 t class machines (e.g., Komatsu D155 or Cat D8) provided for the clay operations, 75 t class machines (Komatsu D375 or Cat D10) provided for sand and till operations, and 100t class machines (Komatsu D475 or Cat D11) for rock operations. As discussed above, these machines will be equipped with HPGPS to improve control of footwalls.
- Wheel dozers, for light clean-up, particularly that associated with truck spillage at trolley line switch backs, and towing portable substations. A single 50 t class unit (e.g., Cat 834) will be provided for each operating pit.
- Graders, for maintaining road surfaces. The fleet requirement has been estimated by the empirical relationship of 1 grader for every 20 active trucks. Two sizes of graders will be provided, with 14 ft units (e.g., Komatsu GD655 or Cat 14M) matched with the articulated and 120 t trucks, while 18 ft units (Komatsu GD955 or Cat 18M) matched

with the 290 t trucks. As discussed above, these machines will be equipped with HPGPS to improve control of roads.

- Water tankers for suppressing dust. Tankers would be modified 150 t class haul trucks (e.g., Komatsu HD1500 or Cat 785). The fleet requirement was calculated using the average rate of evaporation (peaking at 340 mm in the summer (three-month period) and the number of active roads in a given period. These machines will be equipped with AHS to improve productivity.
- Front-end loaders for construction and clean-up activities, including the loading of roadstone into 90 t trucks. Three sizes of front-end loader have been considered. Small units, with a 10 t payload (e.g., Komatsu WA600 or Cat 988) would support clay operations, medium units with a 20 t payload (Komatsu WA900 or Cat 992) would support sand and till operations, while large units with a 50 t payload (Komatsu WE1850) would support rock operations. Note the large front-end loader is the same machine that would also be used to load 2% of total rock.
- Utility excavators will be provided for activities such as construction, scaling highwalls and breaking oversize.

### 16.4.3 Infrastructure

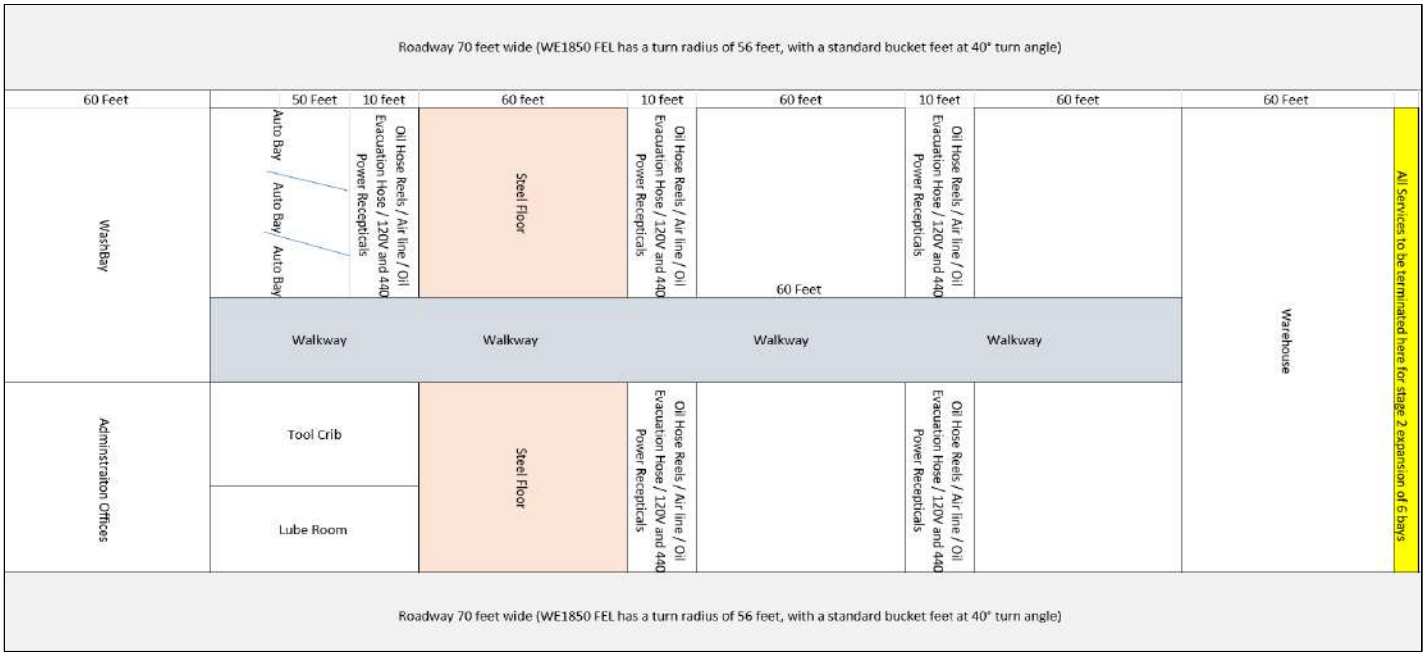
#### 16.4.3.1 Workshop

A workshop and associated warehouse will be provided to maintain the fleet of mining equipment. While a mixed fleet with three sizes of truck will be employed, a single size of bay will be provided. The following empirical relationships were used to determine the total number of bays required:

- 1 bay for every five 290 t trucks
- 1 bay for every seven-and-a-half 90 t trucks (i.e., for 50% of maintenance activities, it will be possible to fit two 90 t trucks in a single bay)
- 1 bay for every ten articulated trucks (it will be possible to fit two articulated trucks in a single bay for 100% of activities).

Additional bays will be provided for washing equipment and light vehicle service. The layout of a portion of the workshop is given in Figure 16-40.

**Figure 16-40: Mine Workshop Design**




Source: SMS, 2022.

#### 16.4.3.2 Fuel Farm / Fuelling Station

Over the life of mine, 2,222 thousand $m^3$ (or 2.2 billion litres) of diesel will be consumed at Crawford, as illustrated in Figure 16-41.

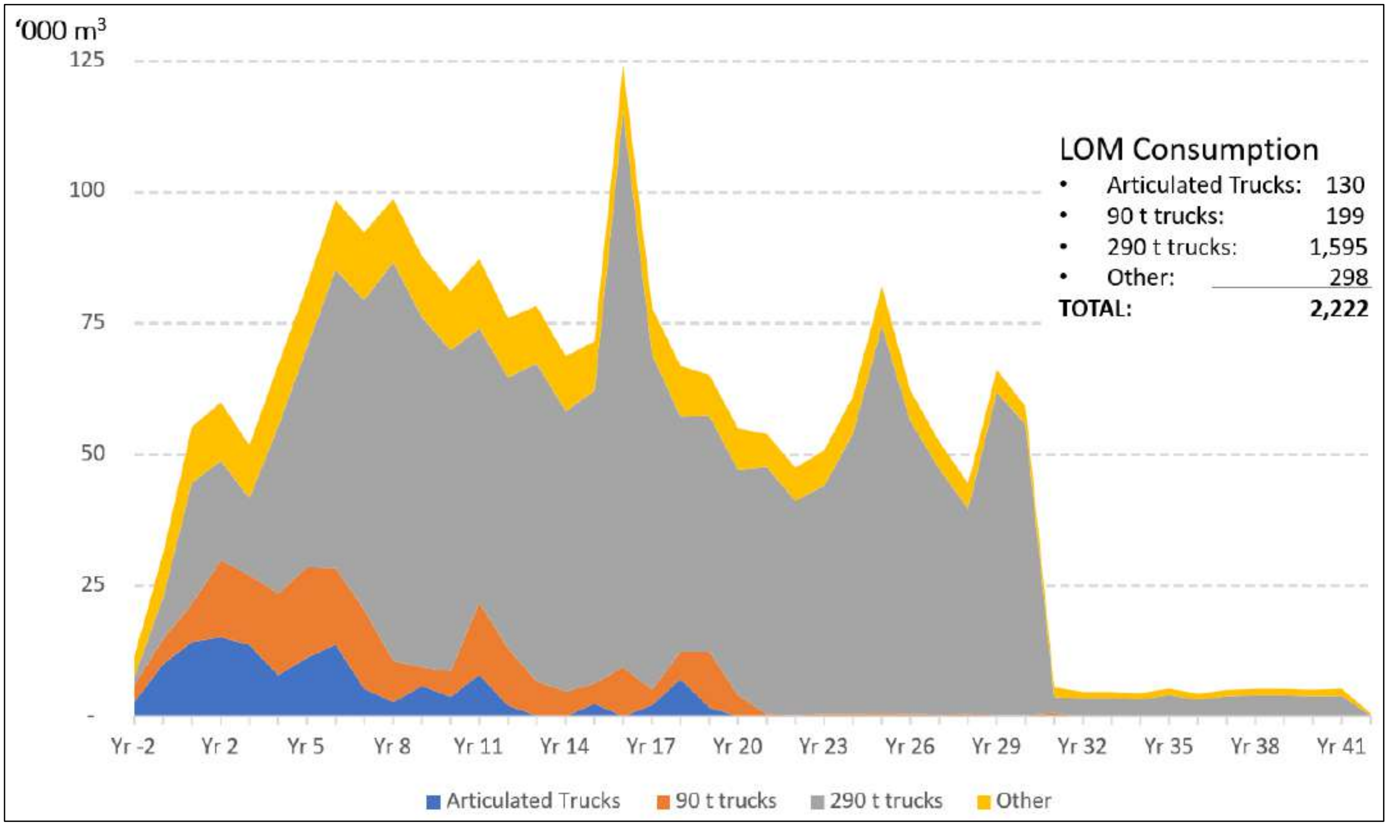
Consumption of gasoline and diesel exhaust fluid (DEF) will be approximately one and a half orders of magnitude lower, at 47 million litres (ML) and 76 ML, respectively.

The layout and all concepts for the fuel farm and fuelling station have been based on recommendations provided by Canada Clean Fuels Limited (CCF). Fuel would be stored in 80 $m^3$ capacity tanks, with the size of tank approximating the maximum size of commercially available tanks that can be transported to site (otherwise, tanks would need to be assembled and welded at site, significantly adding to costs). Fuel would be delivered by road from facilities in the Greater Toronto Area (GTA) to ensure regular delivery. In turn, this will allow site storage to be minimized at three days' consumption. The initial scope for the fuel farm would comprise four tanks for diesel and a single tank for each of gasoline and DEF. There would be no need to increase the capacity of gasoline and DEF storage. Diesel tankage would increase in response to the mine plan, ultimately reaching 13 tanks in Year 16.

To minimize delays associated with refuelling, vehicles would be fuelled at the fuel farm, with each tank interconnected (to ensure consistent levels across the facility) and equipped with fuelling nozzles. In this way, multiple machines can be refuelled concurrently.

The capital and operating cost estimates assume that CCF would be responsible for costs associated with the installation, maintenance and decommissioning of the facility and would recoup their investment via a fixed cost per litre consumed. The only capital cost borne by CNC will be for the required concrete pad and supply of electricity.

**Figure 16-41: Crawford Diesel Consumption**




Source: CNC, 2023.

#### 16.4.3.3 Explosives Manufacture and Storage

Rock will be blasted using emulsion explosives. There are two prospective OEMs that could supply explosives to the project, Dyno-Nobel or Orica.

For the initial 15 months of project construction (during pre-stripping), while an explosives facility is being constructed on site, finished bulk explosives products could be trucked from either company's existing facilities, both of which are located within 420 km of the Crawford mine site. The total explosives requirement during this period will be relatively low, averaging 16 tonnes daily, which could be transported in four deliveries by 20 t trucks every five days. On-site storage facilities totalling 70 t capacity would be erected at site, providing in excess of four days' supply. A single bulk re-pump truck to deliver product to the blast holes would be sufficient for the required duty cycle, but a second would be provided for back-up. Two magazines will be erected for the separate storage of boosters and detonators.

Following commissioning of the on-site facility, emulsion will be manufactured at site. The main ingredient is ammonium nitrate solution (ANSOL), which is non-explosive and can be delivered by larger trucks, with a capacity of 38 t. At the peak production rate, explosives demand would be 151 t/d or approximately four truckloads daily.

The technology under consideration for Crawford would result in explosives only being sensitized (i.e., prior to sensitizing, the product is inert) when injected with a gassing agent in the bulk delivery truck used to charge the holes. As a result, materials stored on site will not be considered explosive. This would allow the explosives storage facilities to be located no further than 270 m from other buildings, reducing the footprint of surface operations and cycle times for the bulk delivery trucks.

The explosives manufacture facility will use intellectual property owned by the explosives supplier. In line with North American practices, the facility would be owned and operated by the explosives supplier. Based on budgetary quotations provided by Orica and Dyno-Nobel, the financial model assumes that the capital cost associated with all equipment and facilities, including those associated with decommissioning, will be borne by the explosives OEM and recovered by way of a service charge applied for the initial 5 years once the project begins generating cashflow. As with the fuel farm, the only capital cost borne by CNC would be for the required concrete pad and supply of electricity.

#### 16.4.3.4 Roadstone Crusher

To ensure the truck fleet achieves high productivity (including an average life of 7,200 hours for tires on the 290 t haul trucks), roads will be continually re-surfaced with crushed waste rock. This re-surfacing will be particularly necessary on ramps equipped with trolley assist, where undulations in the road surface would lead to arcing as the pantographs lose contact with the line. The quantum of crushed rock required for road dressing (life of mine requirement = 36 Mt) will make it economically and operationally advantageous to install a permanent plant. The required size of crushed product is a $P_{100}$ of 50 mm, and a $P_{80}$ of 38 mm. Fines < 10 mm are not desired as they lead to dust.

The existence of a plant to produce roadstone will, in turn, allow the mine to produce aggregates for other requirements that include the following:

- armour of clay surfaces (19.4 Mt of product with $P_{80}$ of 150 mm)
- transition zone of TMF core (9.6 Mt with $P_{100}$ of 38 mm, $P_{80}$ of 25 mm and fines < 10 mm removed)
- road construction (2.3 Mt with a similar size range as the TMF core)
- stemming of blast holes (1.6 Mt with a size range down to 8 mm).

It should be noted that the conventional approach is to not dress roads and for construction aggregates to be provided by a contractor, at a unit cost that is typically greater than 5 times the cost of an Owner-operated plant.

The Crawford plant will be fed waste rock by a medium front-end loader (20 t payload). The circuit will consist of the following:

- a primary 32-inch x 54-inch jaw crusher operating in open circuit
- a 400 horsepower (HP) secondary cone crusher operating in a closed circuit with a screening plant.

The production rate will vary as a function of the mine plan, with a peak throughput of 3.6 Mt/a. To assist the plant achieving this level of output, feed will be blasted separately using only material from 7.5 m benches and an increased powder factor of 0.03 kg/t. Product will be loaded from multiple stockpiles using a 20 t payload front-end loader onto 90 t trucks.

#### 16.4.3.5 Pit Dewatering

All inflows to the open pit will be captured in a sump at the lowest operating level, with the sump being a box cut down to the next bench and measuring approximately 100,000 $m^3$. Water will be pumped in 150 m (vertical) stages, with each stage below the 'saddle' elevation (150 m below surface) equipped with two 676 kW pumps, each capable of pumping 1,000 $m^3$/h. Pumping stages above the sump will be provided with one-hour surge capacity or approximately 2,000 $m^3$ volume.

At the 'saddle' elevation, which for periods will handle discharge from the Main and East Zones concurrently, the interim stage will be equipped with three pumps to handle a maximum of 3,000 $m^3$/h.

A key element of the overall water management plan is the use of the pits as surge capacity for periods where precipitation events exceed the 1:10-year events for which the surface ponds were designed. Surge inflows to the pits will be managed as follows:

- Years -3 to 2: The bottom bench of MZ1 will be allowed to flood, with adequate ore exposure in EZ1 and/or surface stockpiles to ensure uninterrupted mill operation.
- Years 3 to 10: Surge will be directed into the dormant EZ1.
- Years 3 to 16: The bottom bench of East Zone will be allowed to flood, with adequate ore exposure in the Main Zone and/or surface stockpiles to ensure uninterrupted mill operation.
- Years 17 to 30: Surge will be directed into the depleted Main Zone.
- Year 31+: Surge will be directed into the depleted East Zone.

#### 16.4.3.6 Electrical Reticulation

Electrical demand in the pit will peak at 70 MW (operating load) in Year 13 and average 30 MW over the life of mine. The main customer will be the trolley assist system, consuming 62% of the total kilowatt-hours. The in-pit dewatering system will consume a further 9%, while workshops and the blasting plant require 1%. The remaining 28% will be consumed by mobile electric equipment, including blast hole drills, face shovels, and rope shovels.

Extending power to the various units of electric equipment will require a network of overhead lines that progressively extends, with a total of 68.7 km installed over the life of mine. This total includes 19.0 km of lines that will have been previously removed (as the pit extends into previously installed roads – see Figure 16-38 above) and reinstalled.

Mobile electrical equipment (shovels and drills) and pumps will be supplied from mobile substations that are mounted on skids or wheels and can be towed by a wheel dozer.

### 16.4.4 Technology

#### 16.4.4.1 Overview

CNC retained the technology consultant PeckTech to assist with the design and implementation of autonomous drilling systems (ADS) and autonomous haulage systems (AHS). CNC also retained the global technology company ABB to assist with the design and implementation of trolley-assisted truck haulage.

Collectively, these technologies will achieve the following:

- A reduction in the unit mining operating cost of 26%, with attendant impact on the economic limits of open pit mining
- A reduction in the open pit labour component of 33%. The jobs being eliminated are lower skilled equipment operator positions that peer operations are having difficulty filling. These positions will be partially replaced by higher skilled positions associated with the implementation and maintenance of technology.
- A reduction in site-wide $CO_2$ emissions of 39%.

Following is a brief discussion of these technologies. Statistics regarding global adoption of autonomous systems have been provided by PeckTech.

#### 16.4.4.2 Autonomous Drilling System (ADS)

ADS blasthole drills can operate safely and efficiently without the requirement for an onboard operator. Basic functional characteristics of the technology include the following:

- wireless connectivity to a fleet management system to provide continuous real-time location and status updates
- automated guidance and navigation, allowing the drill to move to and position itself on a hole to be drilled; the hole location is transmitted to the drill, obviating the requirement for a surveyor to lay out a pattern of stakes
- automated drilling of the hole, with the applied pulldown force and bit rotation speed optimized in response to the encountered lithologies
- site awareness, including proximity and obstacle detection in order that the drill can operate safely
- self diagnostic capabilities related to the state of machine health.

Currently, there are approximately 100 ADS equipped machines operating globally, with 80% supplied by Epiroc.

At Crawford, ADS machines would be supervised remotely in an office control room, or locally (i.e., in the pit) via a tablet. The nominal span of control will be one supervisor for every three operating ADS units.

#### 16.4.4.3 Autonomous Haulage Systems (AHS)

AHS-equipped haul trucks can operate safely and efficiently without the requirement for an onboard operator. Basic functional characteristics of the technology are as follows:

- wireless connectivity to a fleet management system to provide continuous real-time location and status updates
- automated navigation and route optimization (through real time communications with the fleet management system)
- optimized acceleration, braking, and steering controls for all cycle elements
- site awareness, including proximity and obstacle detection, with a limited ability to apply collision avoidance (currently, AHS fleets typically operate in zones from which other mobile equipment are excluded and will shut down if a non-autonomous machine is detected)
- ability to work in a coordinated manner with loading units (which may be manually or tele-remote operated), using automated spotting
- self-diagnostic capabilities related to the state of machine health.

As of 2021, there were over 1,000 AHS-equipped machines operating globally with Komatsu and Caterpillar being the joint leaders. While 62% of operating units are within the Pilbara region of Australia, the Canadian Oil Sands are considered a fast follower. Other Canadian AHS sites include Highland Valley Copper and the Cote Gold Mine near Timmins.

At Crawford, AHS machines would be supervised by a team of engineers, technicians, coders, ‘runners’ (who monitor the status of equipment in the pit), and dispatchers. As some positions require a fixed number of personnel, irrespective of the number of operational units, and other positions require additional personnel if the total fleet exceeds a certain number, the overall span of control varies. For Crawford’s mine plan, the life of mine average is seven trucks per person, per shift.

#### 16.4.4.4 Trolley Assist

##### 16.4.4.4.1 Background

A typical diesel-electric haul truck utilizes a diesel engine to drive a traction alternator, which produces the electricity used to drive the wheel motors. The on-board control cabinet conditions the voltage and amperage of the power in order that the motors provide the desired speed and torque, in an analogous manner that the transmission does in a mechanical drive truck. The speed of the vehicle on grade is limited by the horsepower output of the diesel engine.

With trolley assist, two pantographs mounted on the truck enable collection of electric power from overhead lines. The lines are supported by rigid poles, and electricity is fed to the line by a direct current (DC) sub station. Additional control devices are added to the truck, so that power from the overhead lines can be properly applied to the wheel motors. Figure 16-42 provides views of a trolley-equipped truck and infrastructure at the Copper Mountain mine in British Columbia.

When on trolley, a truck's diesel engine and alternator are not used for propulsion. The engine's speed automatically drops to an idle, with all the power for propulsion coming from the overhead lines. The speed of a diesel-powered truck is limited by its engine horsepower, but the speed of a trolley truck is limited by the capabilities of its traction motors.

Savings realized from trolley assist can be categorized as follows:

- Energy Cost Savings – This occurs as power is supplied to wheel motors from an overhead line (and thus from the electrical grid) rather than being generated using the on-board diesel engine. The value of savings is a function of the kilometres travelled on trolley and the relative prices for fuel and electricity. With the energy prices that have been forecast for Crawford, the energy savings on trolley is estimated at C$31/km travelled.
- Productivity Savings – Savings result from the increased speed of haul trucks traveling uphill on trolley. For the class of truck planned at Crawford, a doubling of speed on trolley is possible. This would lead to an overall reduction in average cycle time over the life of mine of 14%. This allows the mine plan to be achieved with fewer trucks, with the additional benefit of reducing congestion associated with 'bunching' of units.
- Reduced Maintenance Costs – The maintenance interval for diesel engines can best be modelled as a function of fuel consumption. With the lower consumption rate for a truck traveling on trolley, the interval between overhauls and replacements can be extended.

In addition to the cost benefits listed above, trolley assist also has significantly environmental benefits, resulting from the reduction in particulate matter and greenhouse gases associated with generating energy from hydrocarbons. In the event trolley assist were not used at Crawford, diesel consumption by the fleet of 290 t trucks would approximately double leading to a 53% increase in $CO_2$ emissions.

**Figure 16-42: Trolley Assist at Copper Mountain**


Source: SMS, 2023.

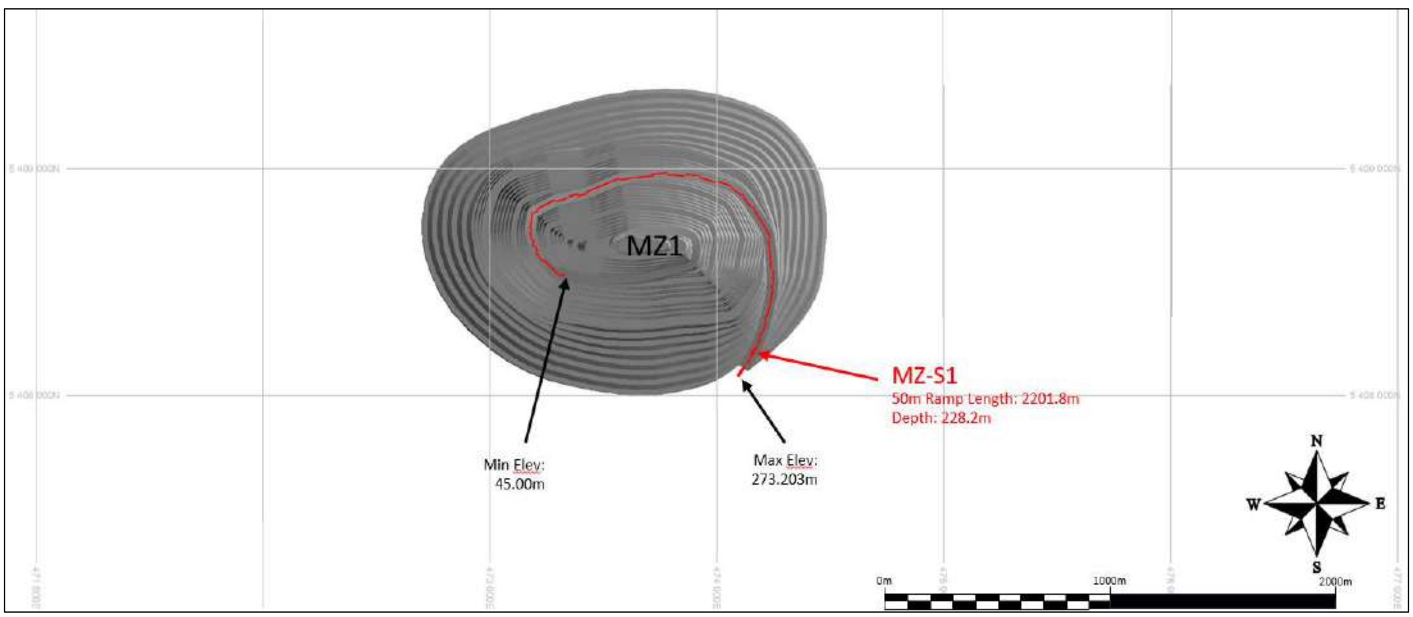
Savings associated with trolley-assist are partially offset by costs associated with operating the system that include the following:

- fixed infrastructure, including the trolley line, pole, and substation
- truck infrastructure, including the pantograph and associated on-board control devices
- ongoing maintenance of fixed and truck-based infrastructure
- wider ramps to accommodate trolley-assist infrastructure (primarily the substations)—the width of the equipped ramps would be increased by 5 m (note: the width of Crawford trolley-equipped ramps was increased a further 10 m to 50 m in total to allow stationary traffic under the trolley line, including disabled haul trucks and trolley maintenance equipment).

#### 16.4.4.4.2 Crawford Trolley Assist Design

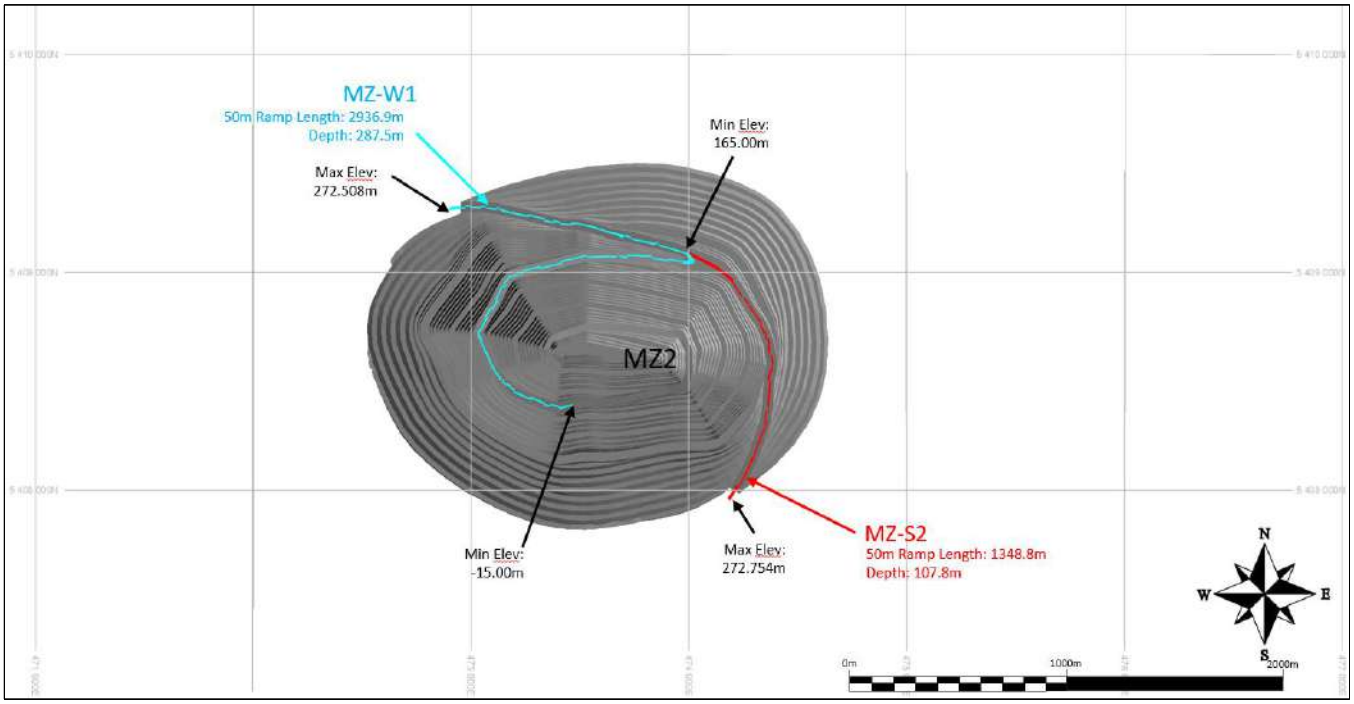
Eight of the ten stages would include trolley-assist infrastructure, with just the small East Zone starter phases EZ1 and EZ2 not having sufficient travel on the ramps to justify the technology. Figures 16-43 through 16-50 illustrate the extent of trolley assist that will be installed in each of the remaining eight stages. Trolley assist will also be provided to each of the stockpiles and to the waste rock impoundment.

**Figure 16-43: Trolley Assist Installation – Phase 2, MZ1**



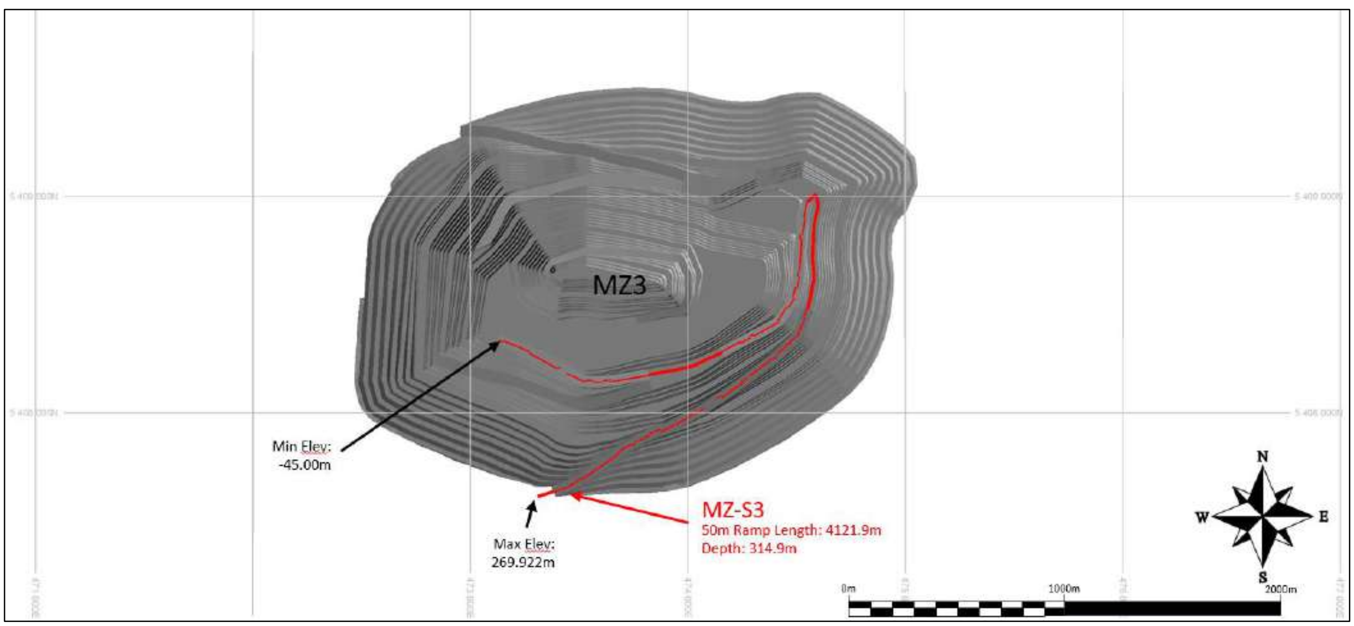

Source: CNC, 2023.

**Figure 16-44: Trolley Assist Installation – Phase 3, MZ2**



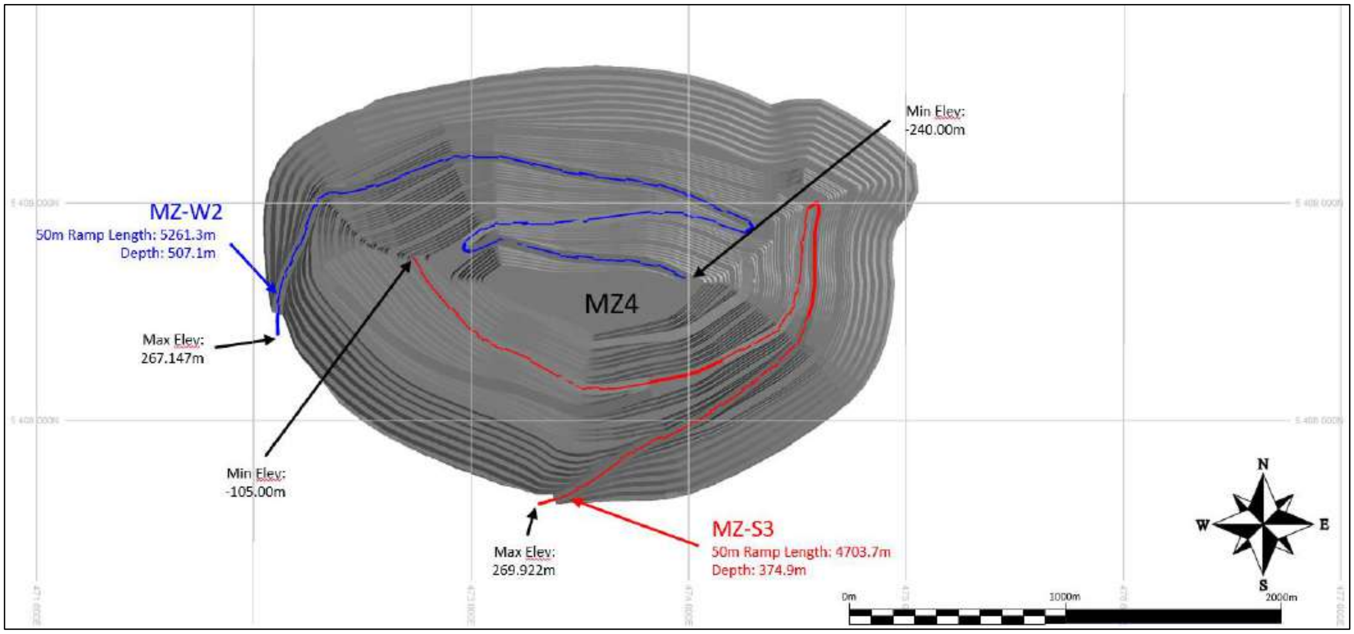

Source: CNC, 2023.

**Figure 16-45: Trolley Assist Installation – Phase 4, MZ3**



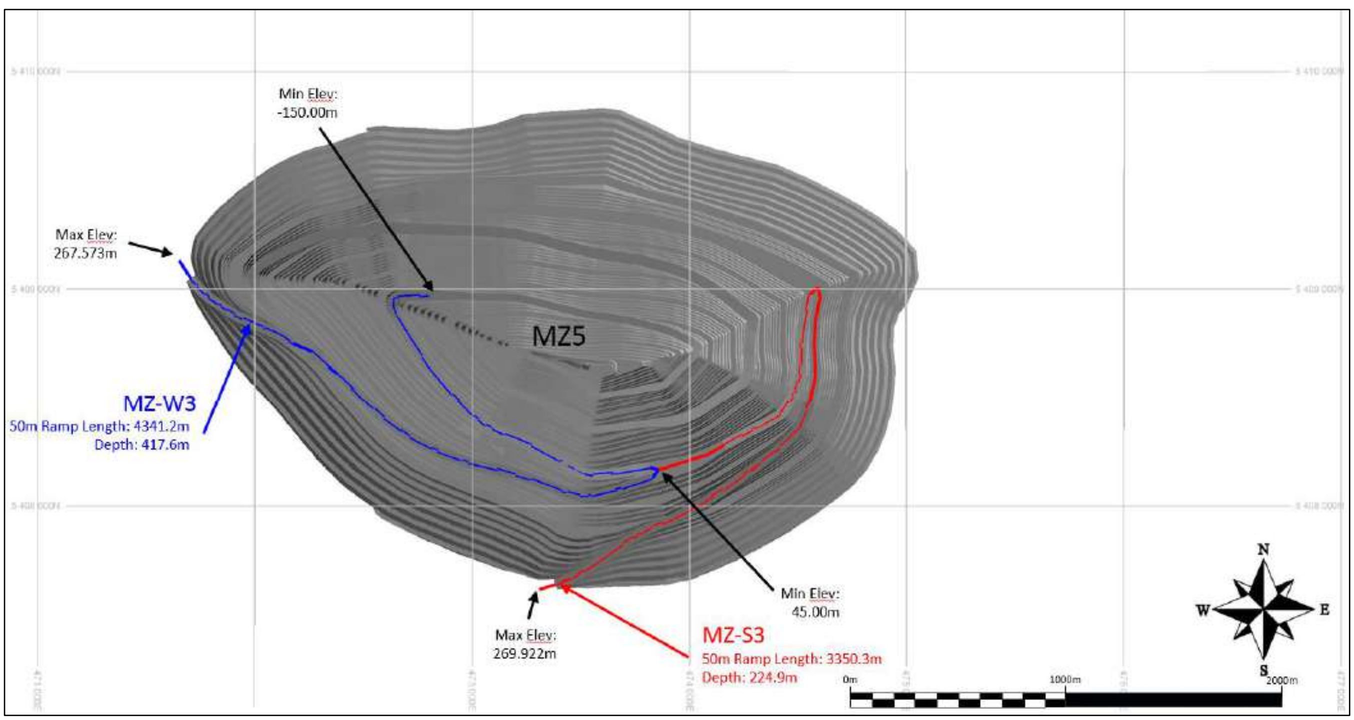

Source: CNC, 2023.

**Figure 16-46: Trolley Assist Installation – Phase 6, MZ4**



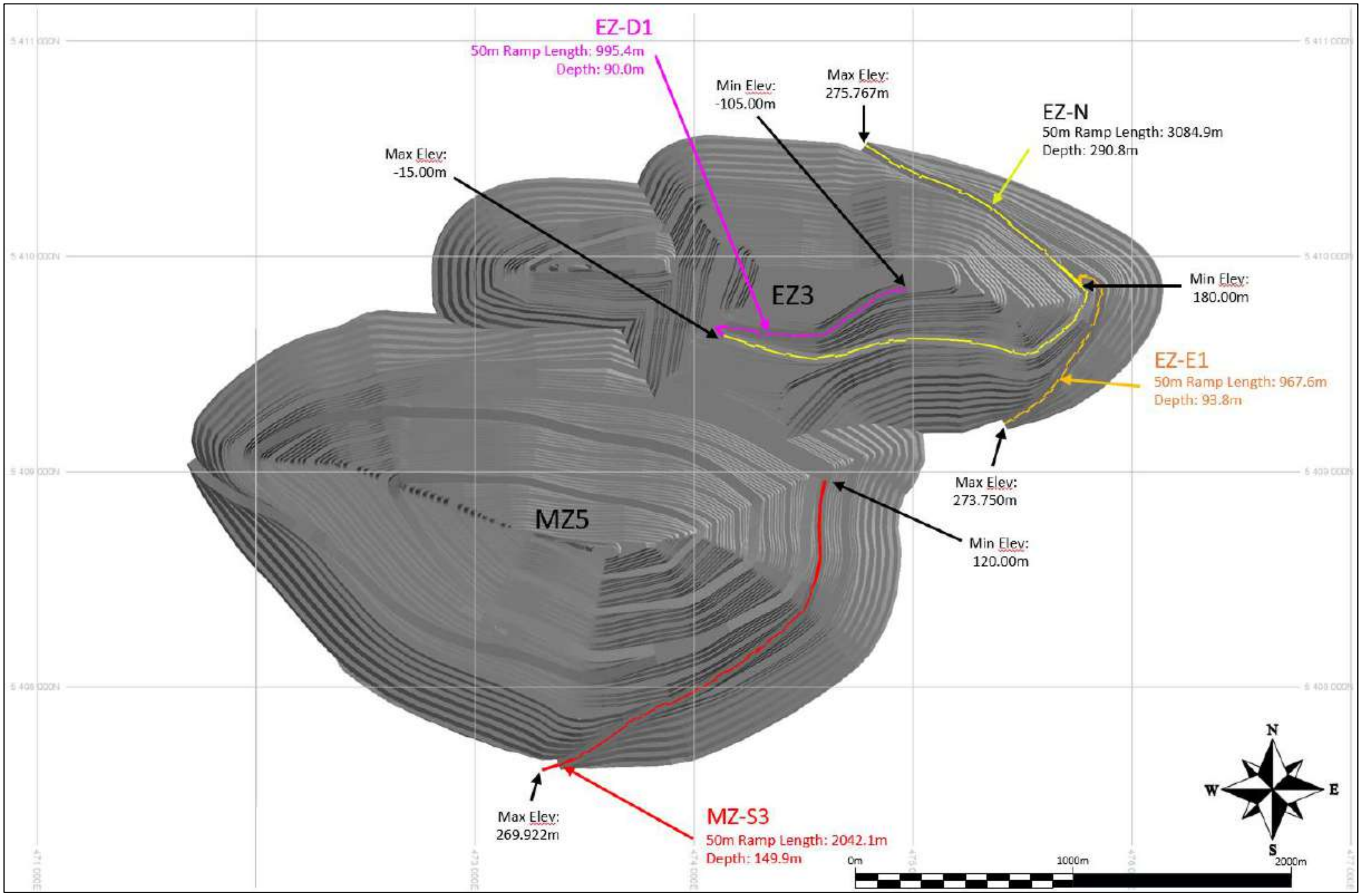

Source: CNC, 2023.

**Figure 16-47: Trolley Assist Installation – Phase 7, MZ5**



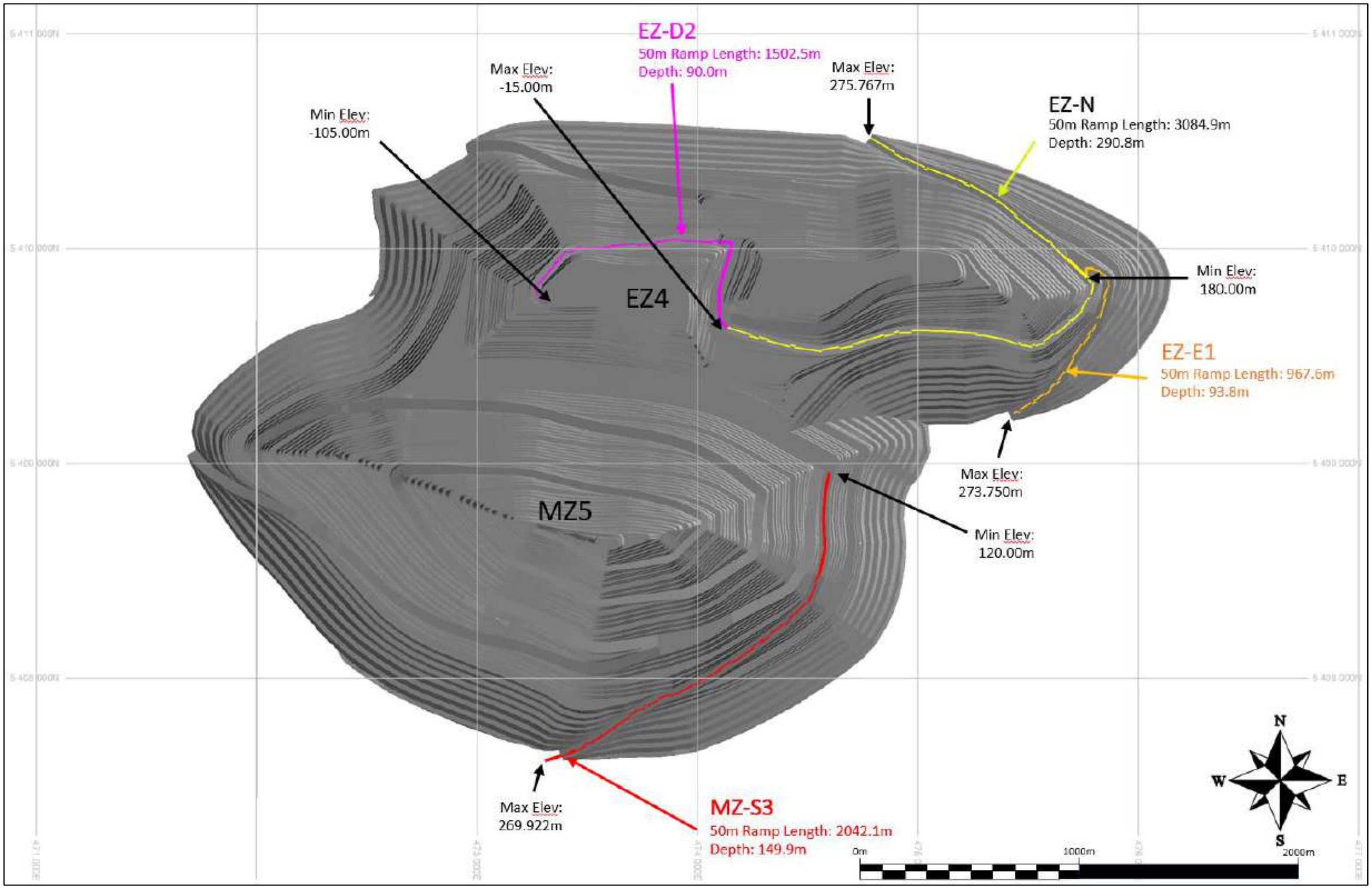

Source: CNC, 2023.

**Figure 16-48: Trolley Assist Installation – Phase 8, EZ3**



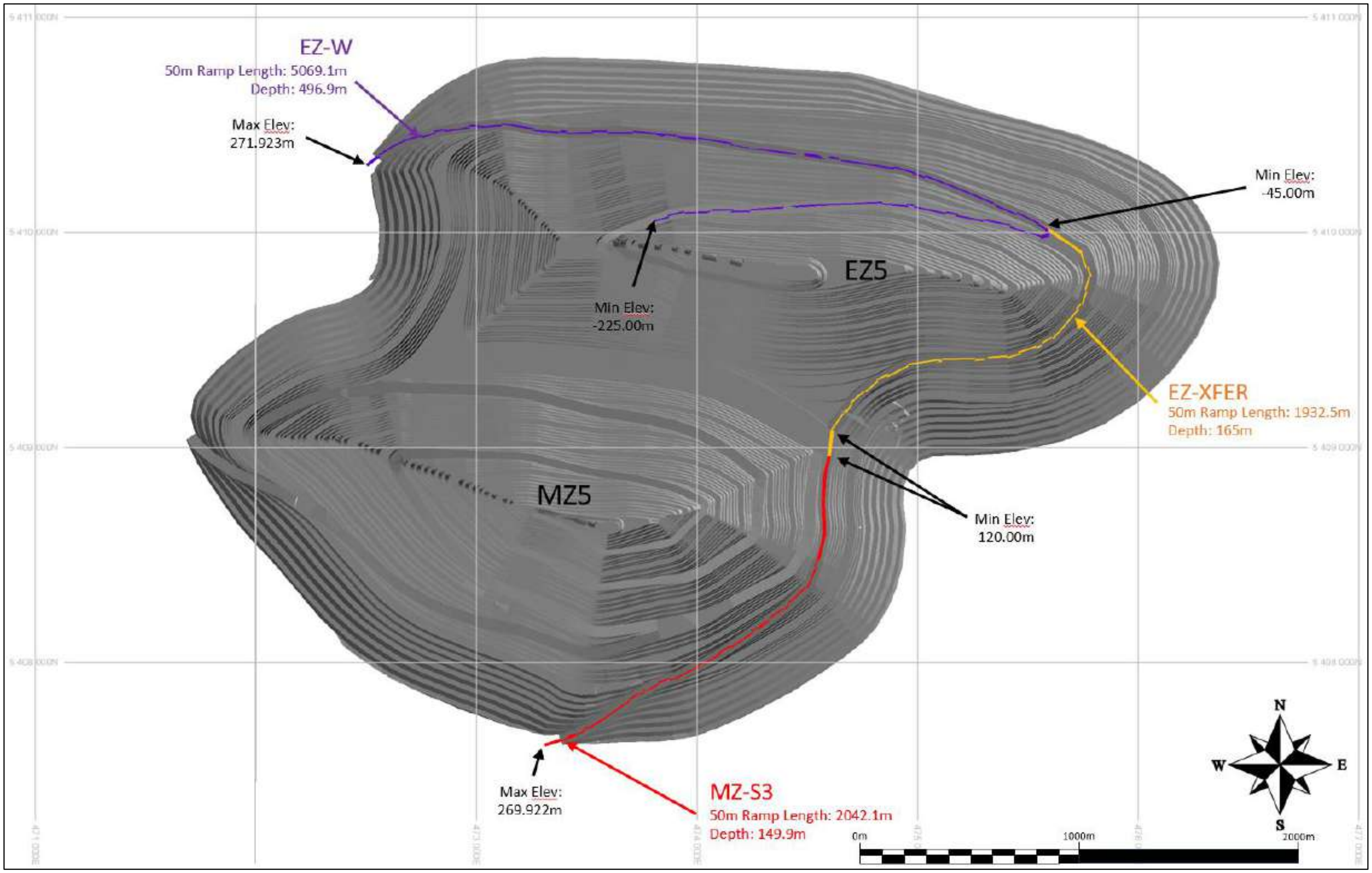

Source: CNC, 2023.

**Figure 16-49: Trolley Assist Installation – Phase 9, EZ4**




Source: CNC, 2023.

**Figure 16-50: Trolley Assist Installation – Phase 10, EZ5**




Source: CNC, 2023.

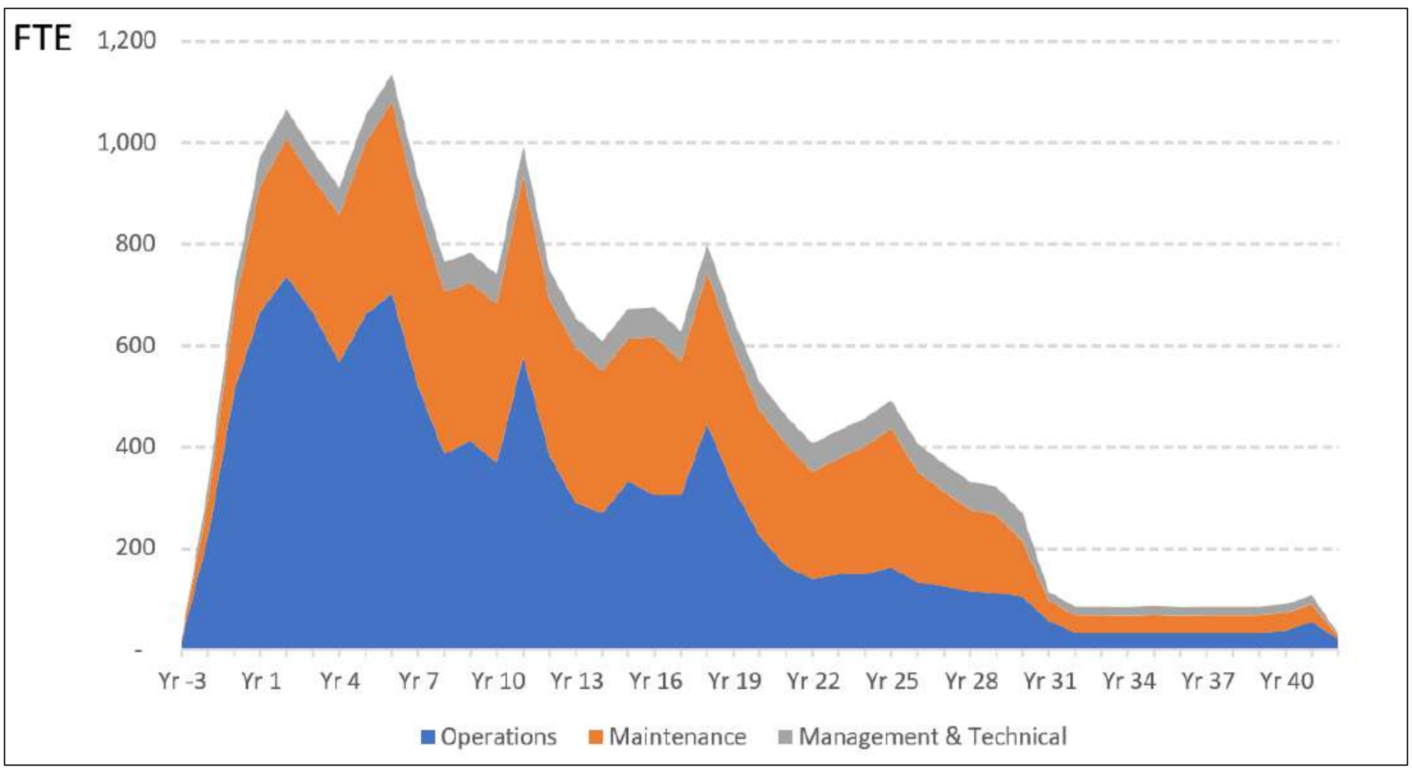
A key assumption in the design, based on operating experience at Palabora and Sishen, is that steady-state utilization of each trolley equipped ramp (measured in percentage of potential tonnes x equipped kilometres) would be 90%. It was also assumed each new in-pit segment would take 18 months to reach this utilization, with a key constraint being the time required to open a bench sufficiently that fly rock from blasting would not damage the system. For the dump, where no blasting would take place, the time required to reach steady-state was assumed to be 12 months. Over the life of mine, 73% of total uphill tonnes x kilometres travelled by the 290 t trucks would be on trolley-assist. The smaller 90 t and articulated trucks will not be equipped for trolley assist.

### 16.4.5 Labour

The Crawford open pit will operate continuously, with two 12-hour shifts daily and 365 days per year. This will be achieved by four crews each working an average of 42 hours per week and labour costs allow for two hours of planned overtime weekly (in addition to unplanned overtime). Staff personnel will work on a conventional five-day week schedule.

The life-of-mine labour complement illustrated in Figure 16-51 was calculated from first principles based on the number of units of equipment required to achieve the planned production schedule. Included in the numbers are the personnel associated with tailings and water management, the bulk of whom will be using the mining fleet to construct the TMF embankment or water management ponds.

**Figure 16-51: Crawford Open Pit Labour**




Source: CNC, 2023.

During the period of pre-stripping, the complement will average 418 full-time equivalent (FTE) personnel. This increases to 994 FTE during the initial phase of mill operations at 60 kt/d. The maximum complement of 1,135 FTE is reached in Year 6. Thereafter, the complement decreases in line with the decrease in relative quantities of overburden mined. For the entire 26.5-year period of Phase 2 operations, when the pit is active and mill producing at 120 kt/d, the average complement is 632 FTE. During Phase 3, when the open pits are depleted and mill fed from reclaimed stockpiles, the complement averages 89 FTE.

The ratio of maintenance-to-operations personnel at Crawford of 0.74 is higher than typical and reflects the following:

- The use of technologies such as ADS, AHS, and trolley assist, that both reduce the number of personnel required to operate equipment and/or increase the productivity of these units.
- The use of such advanced technologies increases the number of maintenance personnel required to maintain a single machine.

## 16.5 Comment on Mining Method

Crawford will employ conventional open pit mining techniques to mine 1,715 Mt ore and 3,992 Mt waste over a 33 ½ year life, including 2 ½ years of pre-stripping.

Over the life of mine, production rates will average 512 kt/d and reach a maximum of 641 kt/d. These rates are considered appropriate, give the scale of reserves.

Crawford will employ three different fleets of mining equipment that are tailored to specific issues of the different lithologies that will be mined. The two different fleets that will be employed for mining overburden (including clay as well as sand and till), will be conventionally operated by on-board operators. Assumed rates of productivities for these units are in line with what is achieved at numerous peer operations. The fleet of electric face shovels, rope shovels and 290 t payload trucks that will be employed to mine rock will employ several technologies that are already commercially available and considered proven. As these technologies are not yet widely adopted in Canada, assumed rates of productivity for this fleet are higher than has been achieved in peer Canadian operations, though in line with what has been achieved by similar fleets globally.

The Crawford pit will operate continuously, 24 hours per day and seven days per week. This is appropriate for open pit mining.

# 17 RECOVERY METHODS

## 17.1 Overview

The process plant is designed to process an ultramafic komatiite-hosted Ni-Cu-Co-(PGE) deposit type ore with an iron formation cap, recovering a nickel flotation concentrate and a magnetite concentrate. Cobalt, platinum, and palladium are recovered in the nickel concentrates as well.

To deploy capital efficiently, the design incorporates a staged expansion approach. The throughput is expanded over the life of mine as presented below:

- Phase 1 will have a design capacity of 60 kt/d or 21.9 Mt/a.
- The Phase 2 expansion will duplicate the Phase 1 design for a total capacity of 120 kt/d or 43.8 Mt/a by Year 5 of operation.

The overall circuit design is the based on the interpretation of the testwork conducted by SGS-Lakefield, COREM and XPS in 2022 and 2023, as well as optimizations based on Ausenco's experience, and industry best practices.

The process plant can be divided into five major areas: comminution and desliming, coarse/fine nickel flotation, magnetic separation, concentrate thickening/filtration and tailings processing.

The ore (from stockpile or run-of-mine) is delivered to the crushing area for primary/secondary crushing and conveyed to the crushed ore stockpile. The ore is then reclaimed into a grinding circuit consisting of a semi-autogenous grinding mill and a ball mill circuit (SAB) operating in closed circuit with a cyclone cluster. The cyclone overflow is deslimed in a deslime cyclone cluster. The deslime cyclone overflow reports to the tailings thickener and the deslimed underflow reports to the coarse nickel flotation circuit.

In the coarse nickel flotation circuit, the deslimed slurry feeds the rougher. Three stages of cleaning are used with a cleaner regrind between the first and second flotation cleaning stage. Product from the coarse cleaners reports to the nickel concentrate thickener. Tails from the coarse rougher and cleaner circuit are reground again at the fines regrind ball mill for additional size reduction and upgrades in the fines cleaner cells. Products from the fines cleaners also report to the nickel concentrate thickener and filter for dewatering before stockpiling. Tails from the fines rougher and cleaners are directed to the magnetic separation circuit.

The magnetic separation circuit has three stages of magnetic separations and a dedicated regrind mill for further size reduction between each stage. The magnetic concentrate is processed through a sulphide flotation stage, and the concentrate is dewatered and filtered before stockpiling.

The tailings from the magnetic separation process are combined with the deslime cyclone overflows and thickened at the tailings thickener. The tailings thickener underflow is pumped through tanks where carbon dioxide is introduced to convert brucite into nesquehonite in the slurry for carbon absorption, reducing carbon emission to the environment. The slurry is then pumped either to the tailings management facility (TMF) or to the pits for in-pit deposition.

A summary of the expected performance, for both Phase 1 and 2, is as follows:

- primary crushing availability of 75%
- grinding and flotation availability of 91.3%
- concentrate filtration availability of 84.0%.

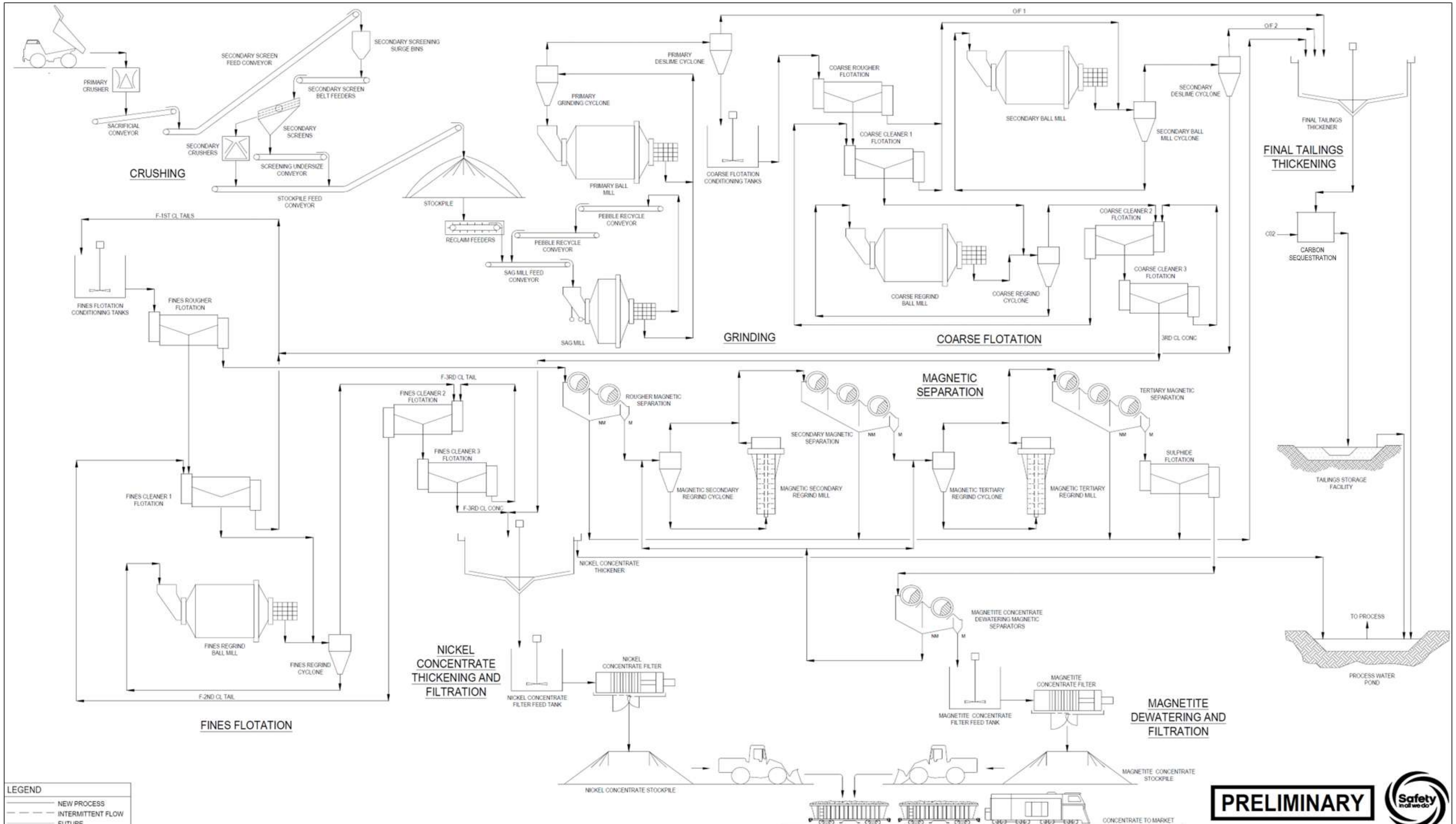
## 17.2 Process Flowsheet

The overall process plant flowsheet for Phase 1 (Figure 17-1) shows all the major unit equipment and project boundaries. Figure 17-2 to 17-6 shows the 3D model isometric view of the overall processing plant and the key process areas.

The process plant will consist of the following unit operations:

- primary and secondary crushing
- crushed ore stockpile and reclaim
- open-circuit SAG mill grinding
- closed-circuit primary ball mill grinding and desliming
- coarse rougher and cleaner flotation with regrind
- closed-circuit secondary ball mill grinding and desliming
- fines rougher and cleaner flotation with regrind
- magnetic separation with regrind
- sulphide flotation
- flotation concentrate thickening and filtration
- magnetic concentrate dewatering and filtration
- tailings thickening
- carbon sequestration
- reagent mixing and distribution
- services and utilities.

**Figure 17-1: Process Flowsheet**




Source: Ausenco, 2023.

## 17.3 Design Criteria

The process design criteria were established based on the metallurgical data and requirements to deliver a robust design for optimum recovery. General design criteria are presented in Table 17-1, and comminution design criteria are shown in Table 17-2. Flotation, magnetic separation, carbon sequestration, and tailings criteria are listed in Table 17-3.

**Table 17-1: Process Plant General Design Criteria**

| Process Design Criteria | Units | Value |
|---|---|---|
| Plant Design Capacity – Phase 1 | kt/d | 60 |
| | Mt/a | 21.9 |
| Plant Design Capacity – Phase 1 & 2 (Total) | kt/d | 120 |
| | Mt/a | 43.8 |
| **Operating Availability** | | |
| Crushing | % | 75.0 |
| Grinding, Flotation and Magnetic Recovery | % | 91.3 |
| Concentrate Filtration | % | 84.0 |
| **Plant Feed Grades, Design** | | |
| Nickel | % | 0.28 |
| Cobalt | % | 0.014 |
| Iron | % | 7.42 |
| Chromium | % | 0.69 |
| Sulphur | % | 0.27 |
| **Flotation Concentrate Recoveries, Design** | | |
| Nickel | % | 47.4 |
| Cobalt | % | 21.8 |
| **Magnetic Concentrate Recoveries, Design** | | |
| Nickel | % | 5.0 |
| Iron | % | 51.1 |
| Chromium | % | 32.2 |
| **Flotation Concentrate Grades, Design** | | |
| Nickel | % | 24.3 |
| Cobalt | % | 0.55 |
| **Magnetic Concentrate Grades, Design** | | |
| Nickel | % | 0.19 |
| Iron | % | 51.6 |
| Chromium | % | 3.05 |
| **Ore Characteristics, Design** | | |
| Specific Gravity | | 2.65 |
| Moisture Content | % | 3.0 |
| Bulk Density for Volume calculations | $t/m^3$ | 1.58 |
| Bulk Density for Weight Calculations | $t/m^3$ | 1.72 |

Note: Phase 2 expansion assumes the addition of 60 kt/d capacity for a total design capacity of 120 kt/d.

**Table 17-2: Comminution Design Criteria**

| Process Design Criteria | Units | Value |
|---|---|---|
| **Comminution Characteristics, Design** | | |
| Bond Crushing Work Index | kWh/t | 14.7 |
| Bond Rod Mill Work Index | kWh/t | 16.8 |
| Bond Ball Mill Work Index | kWh/t | 21.0 |
| Drop weight test – Axb Parameter | Axb | 50 |
| Bond Abrasion Index | g | 0.020 |
| Ball Mill Circulating Load | % | 350 |
| **Particle Size Distributions, Design** | | |
| Primary Crusher Feed, $F_{80}$ | mm | 500 |
| Grinding Circuit Feed, $F_{80}$ | mm | 40 |
| Primary Ball Mill Product, $P_{80}$ | µm | 230 |
| Secondary Ball Mill Product, $P_{80}$ | µm | 100 |
| Deslime Cyclone Overflow, $P_{80}$ | µm | 8 |
| Flotation Cleaner Regrind Product, $P_{80}$ | µm | 55 |
| Magnetic Secondary Regrind Product, $P_{80}$ | µm | 60 |
| Magnetic Tertiary Regrind Product, $P_{80}$ | µm | 25 |

**Table 17-3: Flotation, Magnetic Separation, Carbon Sequestration and Tailings**

| Process Design Criteria | Units | Value |
|---|---|---|
| **Coarse Flotation Mass Recovery, Design** | | |
| Rougher | % Plant Feed | 2.89 |
| Cleaner 1 | % Plant Feed | 1.04 |
| Cleaner 2 | % Plant Feed | 0.39 |
| Cleaner 3 | % Plant Feed | 0.27 |
| **Fines Flotation Mass Recovery, Design** | | |
| Rougher | % Plant Feed | 1.29 |
| Cleaner 1 | % Plant Feed | 1.01 |
| Cleaner 2 | % Plant Feed | 0.39 |
| Cleaner 3 | % Plant Feed | 0.28 |
| **Magnetic Separation Mass Recovery, Design** | | |
| Rougher | % Circuit Feed | 29.9 |
| Secondary | % Stage Feed | 44.5 |
| Tertiary | % Stage Feed | 74.6 |
| **Concentrate Dewatering, Thickening and Filtration, Design** | | |
| Unit Area Thickening Rate | $t/m^2/h$ | 0.25 |
| Thickener Underflow Density | %w/w | 50 |
| Nickel Concentrate Filter Cake Moisture | %w/w | 10 |
| Magnetic Concentrate Filter Cake Moisture | %w/w | 7 |
| **Tailings Thickening and Carbon Sequestration, Design** | | |
| Unit Area Thickening Rate | $t/m^2/h$ | 0.90 |
| Thickener Underflow Density | %w/w | 40 |
| $CO_2$ Feed Rate | Mt/a | 0.5 |
| $CO_2$ to Dry Tailings Ratio | kg/t | 32 |
| $CO_2$ to Capture Efficiency, per Stage | % | 50 |
| Residence Time, per Stage | h | 6.5 |

## 17.4 Plant Description

### 17.4.1 Phase 1 (60 kt/d)

The process plant is designed for a mill feed rate of 60 kt/d in Phase 1. The individual circuits and equipment are sized to accommodate the grades and mass yields to be encountered in Phase 1.

#### 17.4.1.1 Primary Crushing

Run-of-mine or stockpiled ore is transported from the mine and dumped into a 580-tonne live capacity hopper above the primary gyratory crusher. The crusher is enclosed to minimize dust and reduce noise. An overhead crane is provided for servicing. A partially buried crusher design has been selected to minimize run-of-mine pad elevation and reduce mine haulage costs. The nominal capacity of the crushing plant is approximately 3,333 t/h.

The gyratory crusher operates at a closed-sized setting of 120 mm to reduce the ore from an 80% passing feed size ($F_{80}$) of 500 mm to a product size of 80% passing ($P_{80}$) 128 mm. A mobile rock breaker is employed to break rocks that are larger than the maximum top size of 1,200 mm. Crusher product is conveyed from the crusher discharge vault by a variable-speed primary crusher discharge apron feeder and discharged onto a sacrificial conveyor. The sacrificial conveyor, equipped with a tramp magnet for tramp metal removal, feeds the secondary screen feed conveyor.

The major equipment in the primary crushing area includes the following:

- 750 kW, 5960 gyratory crusher
- tracked mobile rock breaker
- 112 kW, 8 m long apron feeder
- 64,000 $m^3$/h pulse jet dust collector.

#### 17.4.1.2 Secondary Screening and Crushing

The secondary screen feed conveyor directs the crushed ore into a 667-tonne capacity surge bin, feeding two identical open circuit secondary screening/crushing streams. A bypass chute allows the crushed ore to bypass the surge bin and feed directly to the stockpile feed conveyor.

In each stream, the ore is drawn from the surge bin via a belt feeder onto a double-deck secondary screen with 90 mm top deck and 38 mm bottom deck apertures. The screen oversize is fed into a cone crusher, and the undersize deposits onto a conveyor that transfers the ore onto the stockpile feed conveyor. The cone crushers have a nominal feed rate of approximately 1,200 t/h and operate at a closed size setting of 38 mm to reduce the ore from an 80% passing feed size ($F_{80}$) of 141 mm to a product size of 80% passing ($P_{80}$) 44 mm. The crusher product is deposited directly onto the stockpile feed conveyor, where it is combined with the secondary screen undersize for transport to the crushed ore stockpile. The combined material has a product size of 80% passing ($P_{80}$) 40 mm.

The major equipment in the secondary screening and crushing area includes the following:

- two 1.8 m long belt feeders
- two 3.0 x 7.3 double-deck banana vibrating screens
- two 970 kW cone crushers
- 10,000 $m^3$/h pulse jet dust collector
- Crushed ore stockpile feed conveyor.

#### 17.4.1.3 Crushed Ore Stockpile

The crushed ore stockpile has a 32,860-tonne live capacity, which is equivalent to 12 hours of mill feed at the nominal mill feed rate. The total capacity of the stockpile is approximately 155,770 tonnes. Earth-moving equipment can be used to mechanically reclaim the dead volume and expand the full stockpile capacity by moving material into additional stockpile areas. This increases the feed available to the downstream operations and de-couples the grinding circuit from the crushing circuits and mining operations.

The stockpile will be a covered facility to mitigate freezing of the stockpiled material and fugitive dust emissions. Access to earth-moving equipment is provided. The stockpile is equipped with a dust-collecting system. Ore will be reclaimed from the stockpile using three variable-speed apron feeders, each delivering up to 50% of the nominal mill feed rate.

The major equipment in the crushed ore stockpile area includes the following:

- three 1.5 m x 7 m apron feeders
- two 25,000 $m^3$/h pulse jet dust collectors.

#### 17.4.1.4 Primary Grinding and Classification

The primary grinding circuit consists of a SAG mill followed by a ball mill arranged in a closed circuit with an 838 mm diameter cyclone cluster. This is followed by de-sliming of the ball mill cyclone overflow. Small diameter (102 mm) cyclone clusters are selected to remove fine fibrous particles that can interfere with flotation kinetics. The nominal feed throughput of the circuit is approximately 2,738 t/h. The circuit is designed to reduce the crushed ore particle size from an 80% passing feed size ($F_{80}$) of 40 mm to a product size of 80% passing ($P_{80}$) of 230 µm. The deslime cyclone overflow has a target product size of 80% passing ($P_{80}$) approximately 8 µm.

Ore reclaimed from the stockpile is fed into the SAG mill via a conveyor. Discharge from the SAG mill goes through a 45 mm aperture trommel screen. Oversized pebbles from the trommel screen are recycled via recycle conveyors back onto the mill feed conveyor. A cross-belt self-cleaning electromagnet removes tramp steel from the recycle stream. Undersize from the SAG trommel screen flows by gravity into the primary ball mill cyclone feed pumpbox, where the slurry is pumped by a single-duty pump to the primary ball mill cyclone cluster.

The primary ball mill cyclone cluster underflow gravity flows to the primary ball mill. Discharge from the primary ball mill gravity flows through a 25 mm aperture trommel screen back into the primary ball mill cyclone feed hopper for reclassification and further grinding. The trommel oversize is discharged into a scats bunker. The primary ball mill circuit is designed for a recirculating load of 350%.

The primary ball mill cyclone overflow is split into two streams, each passing through a 0.7 mm aperture trash screen to a cyclone feed pumpbox. Each pumpbox has two pumps that each feed a dedicated cyclone cluster for a total of four deslime cyclone clusters. The overflow from all clusters is combined and sent to the final tailings thickener. The underflow is sent to coarse flotation.

Steel balls are added to the mills as required to maintain mill charge volume and power draws. Process water is added to maintain the target slurry density of 65% solids (w/w).

The major equipment in the primary grinding and classification area includes the following:

- 18 MW SAG mill (11.6 m diameter x 6.71 m EGL)
- 18 MW ball mill (8.1 m diameter x 13.4 m EGL)
- 24 place 838 mm cyclone cluster (19 operating, 3 standby and 2 blanks)
- two 5 x 7 m vibrating, single-deck deslime trash screens
- four 10 pod clusters with 10 x 101 mm cyclones per pod (366 operating and 34 standby).

#### 17.4.1.5 Coarse Flotation and Regrind

The primary deslime cyclone underflow is fed by gravity to two banks of two agitated coarse rougher conditioning tanks per bank, where flotation reagents are added. The flotation reagents are a combination of collector, frother, and depressant. The conditioning tanks allow for the optimal homogenization of the slurry and reagents. Additional dosing points for the flotation reagents are located along the coarse rougher flotation banks.

The conditioned slurry is gravitated to two banks of conventional forced air flotation cells for rougher flotation. The coarse rougher concentrate is pumped to a single bank conventional cell first cleaning stage operating in an open-circuit configuration. The tailings from the rougher and first cleaning stages are combined and sent to secondary milling for further liberation and recovery in the fines flotation circuit.

The first cleaner concentrate is pumped to the coarse regrind circuit. The coarse regrind circuit consists of a ball mill in a closed circuit with hydrocyclones designed to produce a product size of 80% passing ($P_{80}$) 55 µm. Steel balls are added to the mill as required to maintain mill charge volume and power draws. Process water is added to maintain the target slurry density of 65% solids (w/w). A 25 mm aperture trommel is used to scalp scats.

The regrind cyclone overflow is sent to a conventional cell second coarse cleaner flotation stage and dosed with collector and depressant. The tailings from this stage are recycled to the first coarse cleaner cells, and the concentrate is pumped to the third cleaner cells. The third cleaner concentrate is sent to the nickel concentrate thickener, and the tailings are recycled back to the second cleaner bank.

The major equipment in the coarse floatation and regrind area includes the following:

- two banks of 5 x 311 $m^3$ rougher forced air flotation cells
- five 36.5 $m^3$ first cleaner forced air flotation cells
- two 36.5 $m^3$ second cleaner forced air flotation cells
- two 36.5 $m^3$ third cleaner forced air flotation cells
- 355 kW regrind ball mill (2.7 m diameter x 4.0 m EGL).

#### 17.4.1.6 Secondary Grinding and Classification

The secondary grinding circuit consists of a ball mill arranged in a closed circuit with a cyclone cluster followed by de-slime. The circuit is designed to reduce the coarse flotation tailings to a product size of 80% passing ($P_{80}$) 100 µm and remove fibrous particles before fines flotation.

The coarse flotation tailings are pumped into the secondary mill cyclone feed pumpbox, where they are transferred to the secondary mill cyclone cluster. The cyclone cluster underflow gravity flows to the secondary ball mill. Discharge from the ball mill gravity flows through a 25 mm aperture trommel screen, then into the cyclone feed hopper for reclassification and further grinding. The trommel oversize is discharged into a scats bunker. The secondary ball mill circuit is designed for a recirculating load of 350%. Steel balls are added to the mill as required to maintain mill charge volume and power draws. Process water is added to maintain the target slurry density of 65% solids (w/w).

The secondary ball mill cyclone overflow is split into two deslime streams. Each stream has a pumpbox with two pumps that feed to a dedicated cyclone cluster, for a total of four deslime cyclone clusters. The overflow from all clusters, with a target product size of 80% passing ($P_{80}$) approximately 8 µm, is combined and sent to the final tailings thickener. The underflow is sent to fines flotation.

The major equipment in the secondary grinding and classification area includes the following:

- 18 MW ball mill (8.1 m diameter x 13.4 m EGL)
- 20 place 838 mm cyclone cluster (15 operating, 3 standby and 2 blanks)
- four 7-pod clusters, with 10 x 101 mm cyclones per pod (366 operating and 34 standby).

#### 17.4.1.7 Fines Flotation and Regrind

The secondary deslime cyclone underflow is fed to two agitated fines rougher conditioning tanks where collector, frother, and depressant are added. The conditioned slurry is then gravitated to two banks of flotation cells for rougher flotation. The fines rougher concentrate is pumped to a single bank of flotation cells at the first cleaning stage. The tailings are pumped to the magnetic separation circuits for magnetic concentrate recovery.

The first cleaner concentrate is pumped to the fines regrind circuit. The fines regrind circuit consists of a ball mill in a closed circuit with hydrocyclones designed to produce a product size of 80% passing ($P_{80}$) 55 µm. Steel balls are added to the mill as required to maintain mill charge volume and power draws. Process water is added to maintain the target slurry density of 65% solids (w/w). A 25 mm aperture trommel is used to scalp scats.

The regrind cyclone overflow is sent to a second fines cleaner flotation stage and dosed with frother. The tailings from this stage are recycled to the first coarse cleaner cells, and the concentrate is pumped to a conventional cell in the third cleaner stage. The third cleaner concentrate is sent to the nickel concentrate thickener, and the tailings are recycled back to the second cleaner bank.

The major equipment in the fines flotation and regrind area includes the following:

- two banks of 8 x 311 $m^3$ rougher forced air flotation cells
- six 58.1 $m^3$ first cleaner forced air flotation cells
- two 58.1 $m^3$ second cleaner forced air flotation cells
- one 58.1 $m^3$ third cleaner forced air flotation cells
- 315 kW regrind ball mill (2.5 m diameter x 3.2 m EGL).

#### 17.4.1.8 Magnetic Separation and Regrind

A three-stage low-intensity magnetic separation (LIMS) circuit is used to recover nickel contained in magnetic alloys in the fines rougher flotation tailings. Fines rougher tailings are pumped and distributed to two trains of rougher magnetic separators, each consisting of seven double-drum magnetic separators (3.05 m long x 1.2 m diameter) per train. The rougher magnetic separation concentrate gravity flows to a common pumpbox, then pumped to the magnetic secondary regrind circuit.

An open-circuit regrind stage is used to grind the rougher magnetic concentrate stream to a product size of 80% passing ($P_{80}$) of 60 µm. The concentrate is pumped to a cyclone where the underflow feeds the vertical stirred magnetic secondary regrind mill. The mill product is combined with the cyclone overflow and pumped to the secondary magnetic separation stage, which consists of six triple-drum magnetic separators (3.05 m long x 1.2 m diameter) fed via a feed distributor. The secondary magnetic separation concentrate gravity flows to a common pumpbox to be pumped to the magnetic tertiary regrind circuit.

The open-circuit tertiary regrind vertical stirred mill reduces the secondary magnetic separation concentrate to a product size of 80% passing ($P_{80}$) of 25 µm. The magnetic tertiary regrind product is then pumped to a train of four triple-drum tertiary magnetic separators (3.05 m long x 1.2 m diameter), fed via a feed distributor. The tertiary magnetic separation concentrate is pumped to a bank of conventional flotation cells where the collector and frother are dosed for sulphide flotation. The sulphide flotation concentrate is then pumped to the magnetic concentrate dewatering stage.

The number of magnetic separators in use per stage can be varied based on variations in throughput and magnetic recoveries. Magnetic separation tailings from all stages, including sulphide flotation, are combined in the magnetic tailings pumpbox and pumped to the final tailings thickener.

The major equipment in the magnetic separation area includes the following:

- two trains of seven 3.05 m x 1.2 m double-roll, counter-current, 2,000 G, wet, low-intensity, rougher magnetic separators
- six 3.05 m x 1.2 m triple-roll, counter-current, 1,500 G, wet, low-intensity, secondary magnetic separators
- four 3.05 m x 1.2 m triple-roll, counter-current, 1,500 G, wet, low-intensity, tertiary magnetic separators
- two 4.5 MW regrind vertical stirred mills
- four 58.1 $m^3$ forced-air sulphide flotation cells.

#### 17.4.1.9 Concentrate Dewatering

The coarse and fines nickel concentrate from the third cleaner flotation is combined and passed over a 1.2 mm aperture trash screen into the 10 m diameter above-ground nickel concentrate thickener, where it is thickened to approximately 50 % w/w solids. The thickened underflow is pumped to a filter press for further dewatering to 90% w/w solids before stockpiling.

The magnetic concentrate is dewatered using 4,000 G magnetic separators to 65% w/w solids. The dewatered concentrate is split into two filter press streams for further dewatering to 93% w/w solids before stockpiling.

The filtered concentrates are handled by a front-end loader for further stockpiling and loadout to market.

The major equipment in the concentrate dewatering areas includes the following:

- 1.8 m x 1 m vibrating, single-deck trash screen
- 10 m diameter nickel concentrate thickener
- a horizontal, fast-membrane nickel concentrate filter press
- two 3.05 m x 1.2 m double-roll, counter-dewatering magnetic separators
- two horizontal, fast-membrane, magnetic concentrate filter press.

#### 17.4.1.10 Carbon Sequestration and Tailings Disposal

The deslime cyclone overflows and magnetic separation tailings are combined and thickened in a 63 m diameter high-rate thickener to an underflow density of 40% w/w solids. The thickened slurry is pumped into three carbon-capture streams. In each stream, the slurry is split into three agitated, enclosed tanks where carbon dioxide is passed through the slurry for capture and sequestration. The carbon dioxide is supplied as a gas at 97% purity and recirculated with draft fans until absorbed. The tailings from all carbon sequestration three streams are combined with a thickener bypass stream and pumped to the tailings management facility for long-term storage. Process water released from the consolidation of tailings in the facility is decanted and recycled into the process.

The major equipment in the carbon sequestration and tailings disposal area are as follows:

- 63 m diameter, high-rate thickener
- nine 15.5 m diameter, agitated, carbon-capture tanks.

### 17.4.2 Phase 2 Expansion (120 kt/d)

The ramp-up to Phase 2 capacity is starts in Year 4 of operations, with a total design capacity of 120 kt/d available in Year 5. The current design assumes the duplication of the Phase 1 (60 kt/d) to achieve the full design capacity.

The Phase 2 process plant will consist of the following unit operations:

- primary and secondary crushing
- crushed ore stockpile and reclaim
- open-circuit SAG mill grinding
- closed-circuit primary ball mill grinding and desliming
- coarse rougher and cleaner flotation with regrind
- closed-circuit secondary ball mill grinding and desliming
- fines rougher and cleaner flotation with regrind
- magnetic separation with regrind
- sulphide flotation
- flotation concentrate thickening and filtration
- magnetic concentrate dewatering and filtration
- tailings thickening
- carbon sequestration
- reagent mixing and distribution
- services and utilities are shared with Phase 1

The key process design criteria for the mill during Phase 2 are listed in Table 17-4. Any repeated plant availability and comminution characteristics identical to Phase 1 have been omitted.

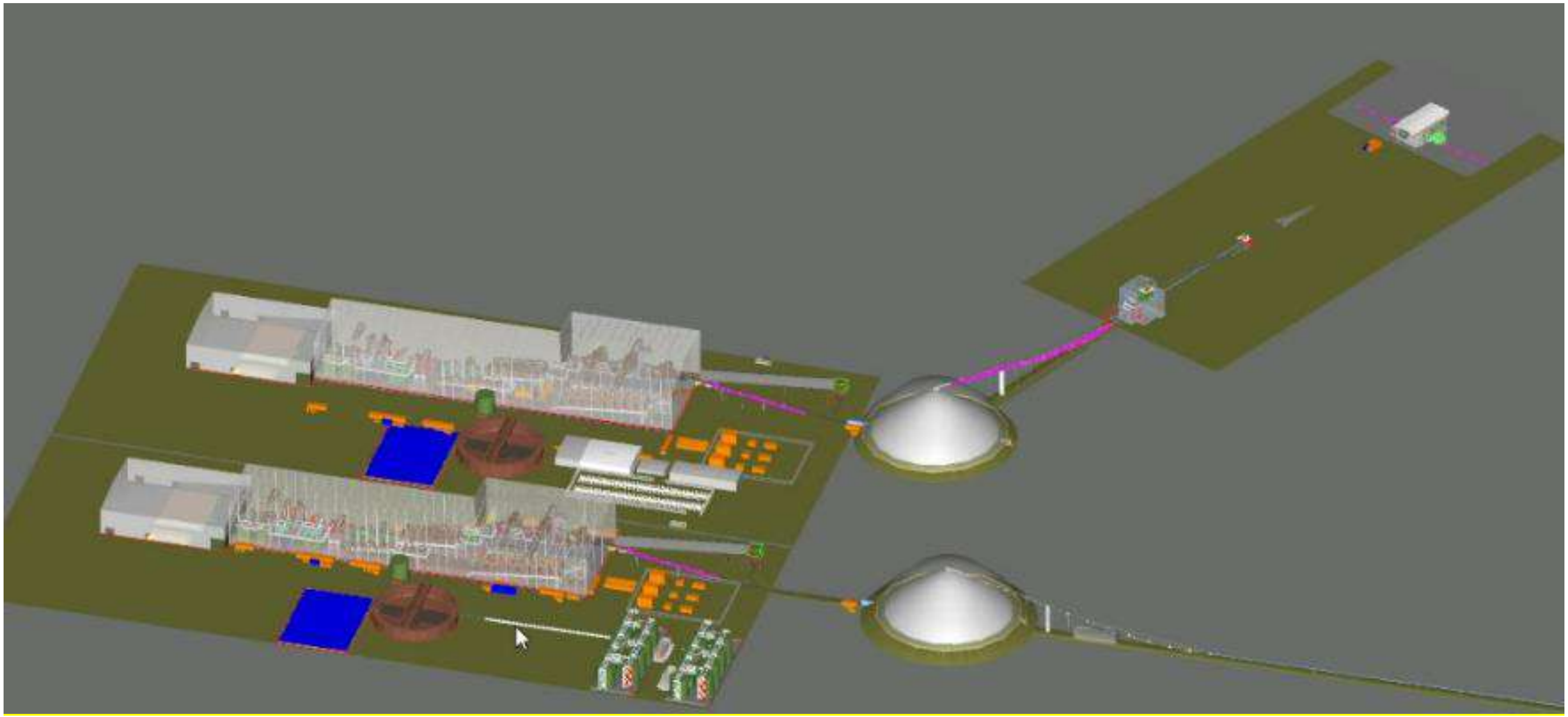
**Table 17-4: Process Design Criteria – Phase 2**

| Process Design Criteria | Units | Value |
|---|---|---|
| **Plant Feed Grades, Design** | | |
| Nickel | % | 0.28 |
| Cobalt | % | 0.014 |
| Iron | % | 8.33 |
| Chromium | % | 0.65 |
| Sulphur | % | 0.32 |
| **Flotation Concentrate Recoveries, Design** | | |
| Nickel | % | 48.3 |
| Cobalt | % | 7.4 |
| **Magnetic Concentrate Recoveries, Design** | | |
| Nickel | % | 5.0 |
| Iron | % | 42.6 |
| Chromium | % | 32.1 |
| **Flotation Concentrate Grades, Design** | | |
| Nickel | % | 24.3 |
| Cobalt | % | 0.19 |
| **Magnetic Concentrate Grades, Design** | | |
| Nickel | % | 0.19 |
| Iron | % | 51.6 |
| Chromium | % | 3.05 |

### 17.4.3 Process Plant 3D Model

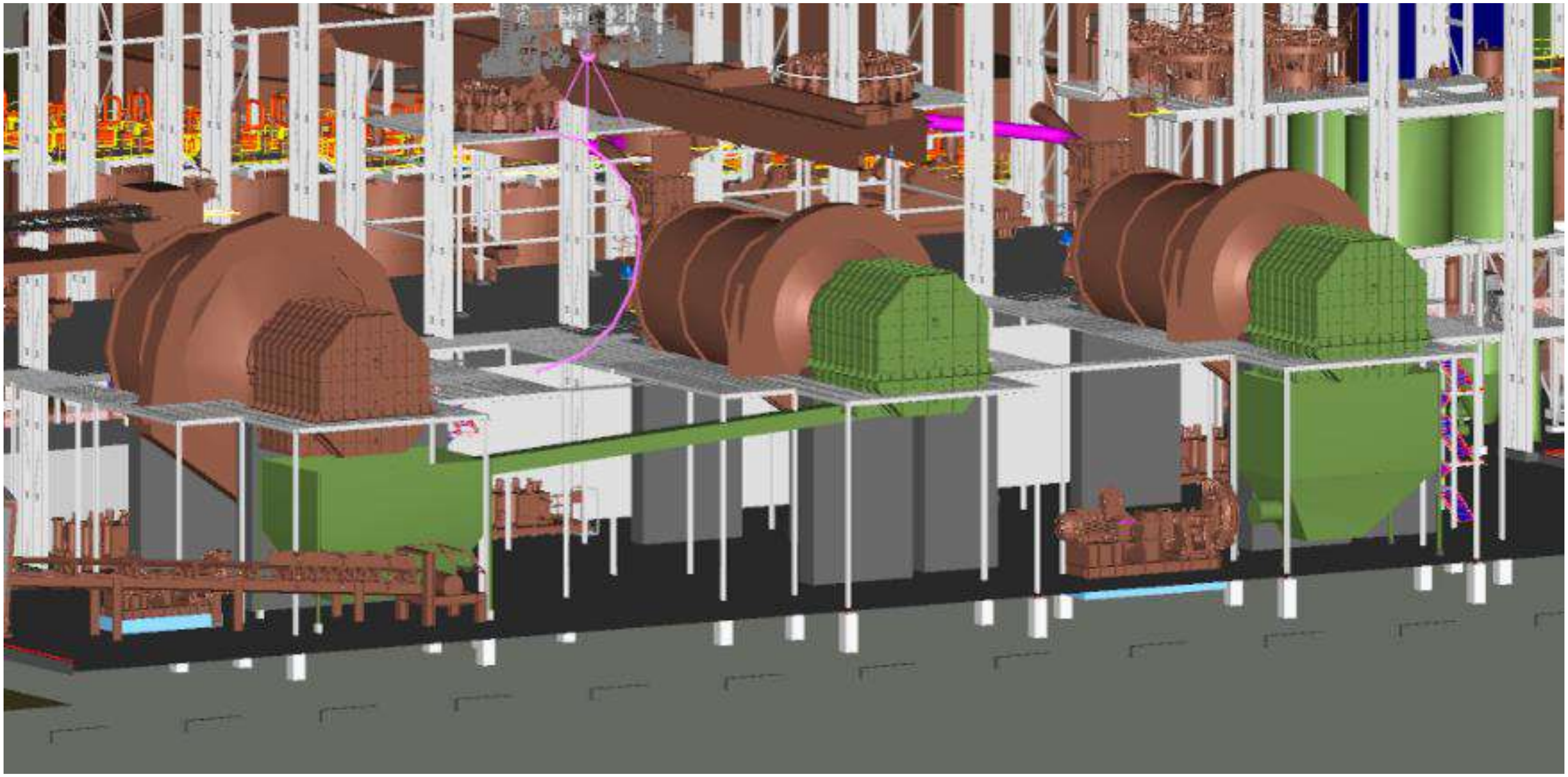
Ausenco prepared a 3D model of the Phase 1 and Phase 2 crushing areas and process plants. Views of the major areas are shown in Figures 17-2 to 17-6.

**Figure 17-2: Overall Process Plant and Crushing Layout (3D Model View)**



Source: Ausenco, 2023.

**Figure 17-3: Grinding Area (3D Model Isometric View)**



Source: Ausenco, 2023.

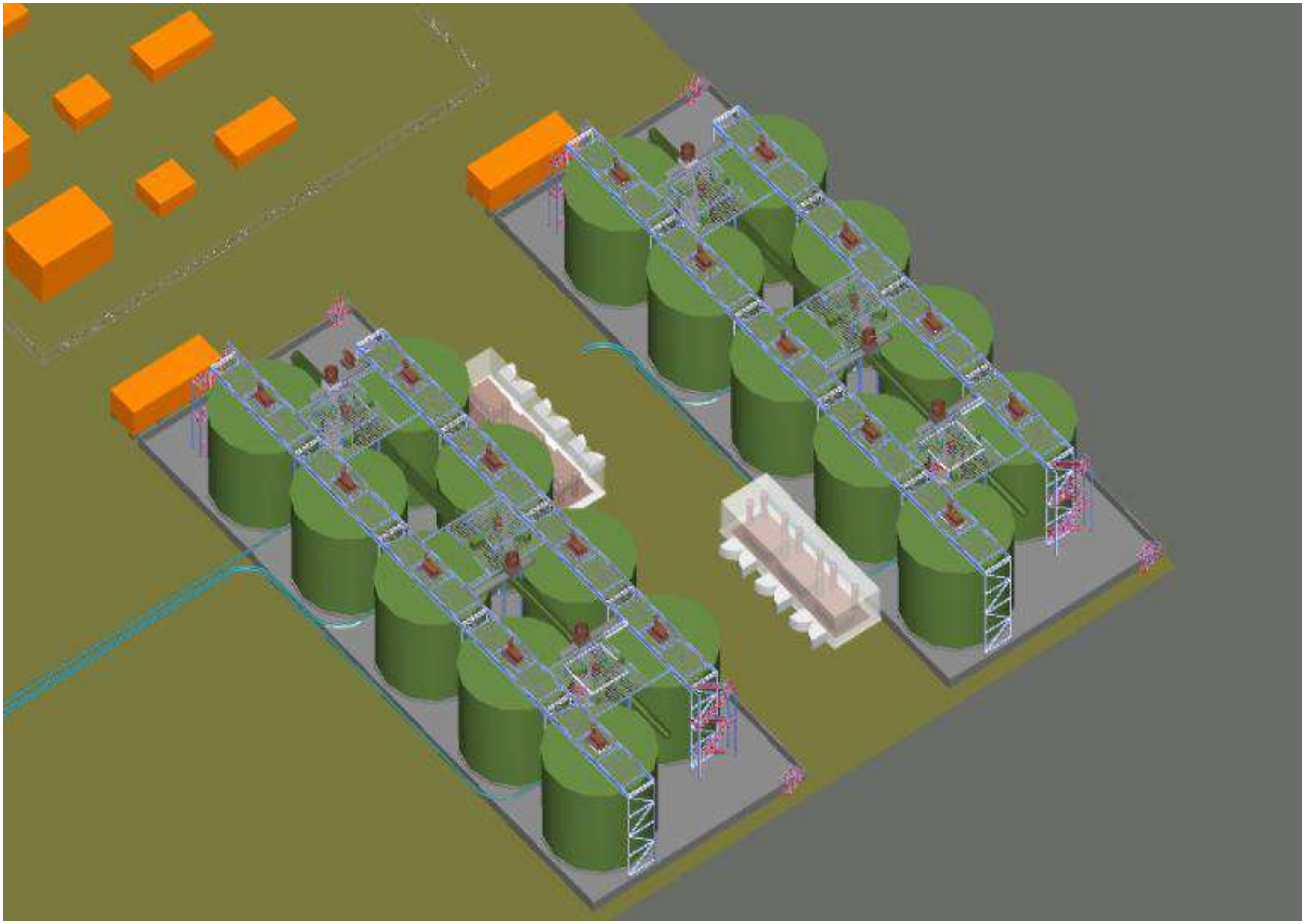
**Figure 17-4: Coarse and Fine Flotation (3D Model Isometric View)**

Source: Ausenco, 2023.

**Figure 17-5: Magnetic Separation Area (3D Model Isometric View)**

Source: Ausenco, 2023.

**Figure 17-6: Carbon Sequestration Area (3D Model Isometric View)**



Source: Ausenco, 2023.

## 17.5 Consumption Rates

### 17.5.1 Reagents

Various reagents will be added to the process to modify the mineral particle surfaces and enhance the effectiveness of a unit process. The reagents will be prepared, metered, and distributed as required. The estimated reagent consumption per tonne of mill feed is shown Table 17-5.

**Table 17-5: Phase 1 (60 kt/d) Annual Reagent Consumption**

| Reagent | Reagent Consumption (g/t feed) |
|---|---|
| Potassium Amyl Xanthate (PAX) | 184 |
| Methyl Isobutyl Carbinol (MIBC) | 36 |
| Cytec 65 | 4 |
| Calgon | 300 |
| Carboxy Methyl Cellulose (CMC) | 21 |
| Sulphuric Acid ($H_2SO_4$) | 1,000 |
| Flocculant | 19 |

#### 17.5.1.1 Collector

Potassium amyl xanthate (PAX) will be supplied in 750 kg bulk bags as a dry reagent. PAX will be diluted to a solution concentration of 20% w/w in an agitated mixing tank and transferred to the day tank for distribution to the flotation circuit using the metering pump on the ring main.

#### 17.5.1.2 Frothers

Methyl isobutyl carbinol (MIBC) will be supplied as a liquid by bulk tankers and offloaded into a storage tank with a seven-day storage capacity at design flow rates. The frother is distributed to the flotation circuit dosing points by dedicated metering pumps.

Cytec 65 is supplied in intermediate bulk containers (IBCs) and offloaded into a storage tank with a four-day storage capacity at design flow rates. The frother is distributed to the flotation circuit dosing points by dedicated metering pumps.

#### 17.5.1.3 Depressants

Calgon (sodium hexametaphosphate) is supplied in 1,000 kg bulk bags as a dry reagent. Calgon will be diluted to a solution concentration of 5% w/w in an agitated mixing tank and transferred to the day tank for distribution through the cleaner flotation process as required using dedicated metering pumps.

Carboxy methyl cellulose (CMC) is supplied in 600 kg bulk bags as a dry reagent. CMC will be diluted to a solution concentration of 0.5% w/w in an agitated mixing tank and transferred to the day tank to distribute the process using the metering pump on the ring main.

#### 17.5.1.4 pH Modifier

Sulphuric acid ($H_2SO_4$) is supplied by bulk tankers and offloaded into a storage tank into a storage tank with a 13-hour storage capacity at design flow rates. The sulphuric acid is distributed to the flotation circuit dosing points as required using dedicated metering pumps.

#### 17.5.1.5 Flocculant

Flocculant will be supplied in 750 kg bulk bags as a dry reagent. It will then be diluted to a solution concentration of 0.25% w/w in an agitated mixing tank and transferred to the flocculant storage tank. Dedicated dosing pumps transfer the flocculant through inline mixers, where the solution is further diluted to 0.025% w/w before dosing in the thickeners.

### 17.5.2 Consumables

A summary of the expected key process consumable usage, based on testwork and industry experience, during Phase 1 (60 kt/d) is shown in Table 17-6. This usage is expected to double in Phase 2 (120 kt/d).

**Table 17-6: Phase 1 Annual Consumable Usage**

| Consumable | Units | Phase 1 Consumption |
|---|---|---|
| Primary Crusher Mantle Standard | set/a | 2 |
| Primary Crusher Concaves | set/a | 1 |
| Cone Crusher Bowl and Mantle | set/a | 3 |
| SAG Mill Liner | set/a | 1 |
| Ball Mill Liner | set/a | 1.5 |
| Regrind Mill Liners | set/a | 0.75 |
| Magnetic Regrind Mill Liners | set/a | 2 |
| Screen Media | set/a | 3 |
| Ball Mill Media | t/a | 4,027 |
| SAG Mill Media | t/a | 1,539 |
| Steel Regrind Media | t/a | 95 |
| Regrind Media (10 mm) | t/a | 303 |
| Regrind Media (5 mm) | t/a | 311 |

## 17.6 Services and Utilities

The services and utilities discussed in this section reflect the Phase 1 (60 kt/d) design unless otherwise specified. Phase 2 usage is assumed to be double Phase 1's usage; this may change based on operational knowledge and optimizations during Phase 1.

### 17.6.1 Water Services

Process water is recovered from the thickener overflows and magnetic concentrate dewatering, as well as supernatant from the consolidated tailings facility. These streams flow into the process water pond for collection before distribution and use across the process plant. Fresh water is used to make up the deficit due to water entrained in the concentrate and tailings. The expected process water usage is approximately 423,054 $m^3$/d.

Raw water is used for gland water and the preparation of reagents. The 2,598 $m^3$ raw water tank contains both a firewater reserve of 1,050 $m^3$ and raw water for 6 hours of consumption at design rates. A dedicated pump skid with an electrical pump, diesel pump, and jockey pump supply water to the firewater piping network. Raw water pumps distribute raw water to users in the process plant as required. The expected raw water usage, excluding makeup water, is approximately 11,044 $m^3$/d.

Part of the raw water is directed to the 53 $m^3$ freshwater tank, which is then treated in a reverse-osmosis system to produce potable water. The water is stored in a 53 $m^3$ potable water tank before distribution to potable water users.

### 17.6.2 Plant Air Services

Low-pressure process air is supplied to the flotation cells by four flotation blowers. Pressure reducers are used to step down the pressure to meet the flotation cells' pressure requirements.

Three screw air compressors provide low- and high-pressure compressed air for plant and instrument air requirements. Two duty and one standby compressor will operate in lead-lag mode.

Plant air is stored in the plant air receivers to account for variations in demand before being distributed throughout the process plant. An additional compressor is dedicated to the crushing area requirements due to the relative distant location of the primary crusher from the mill. Instrument (high-pressure) air is dried and supplied at 900 kPa-g. Low-pressure air is supplied at 690 kPa-g.

Four other air compressors are dedicated to the supply pressing and cake-drying air to the concentrate filters. They have a dedicated air dryer and receivers.

### 17.6.3 Power Supply

Power to the process plant will be supplied from the power grid. Table 17-6 shows the projected power usage at nominal rates, excluding site water management.

**Table 17-7: Power Usage Summary**

| Parameter | Units | Phase 1 (60 kt/d) | Phase 2 (120 kt/d) |
|---|---|---|---|
| Maximum Power Demand | MW | 115 | 221 |
| Nominal Operating Power | MW | 100 | 193 |
| Annual Consumption | GWh | 800 | 1,565 |
| Consumed Power Specific Energy | kWh/t | 36.5 | 35.7 |

## 17.7 Assay/Metallurgical Laboratory and Quality Control

Several samplers are provided throughout the process plant to generate samples for on-stream analysis and composite shift samples from key process streams. The composite samples will be assayed in the assay laboratory (located on site, adjacent to the process building), where standard assays will be performed. The resultant data (e.g., metal content, moisture, and particle size distribution) will be used for product quality control and routine process optimization.

# 18 PROJECT INFRASTRUCTURE

## 18.1 Introduction

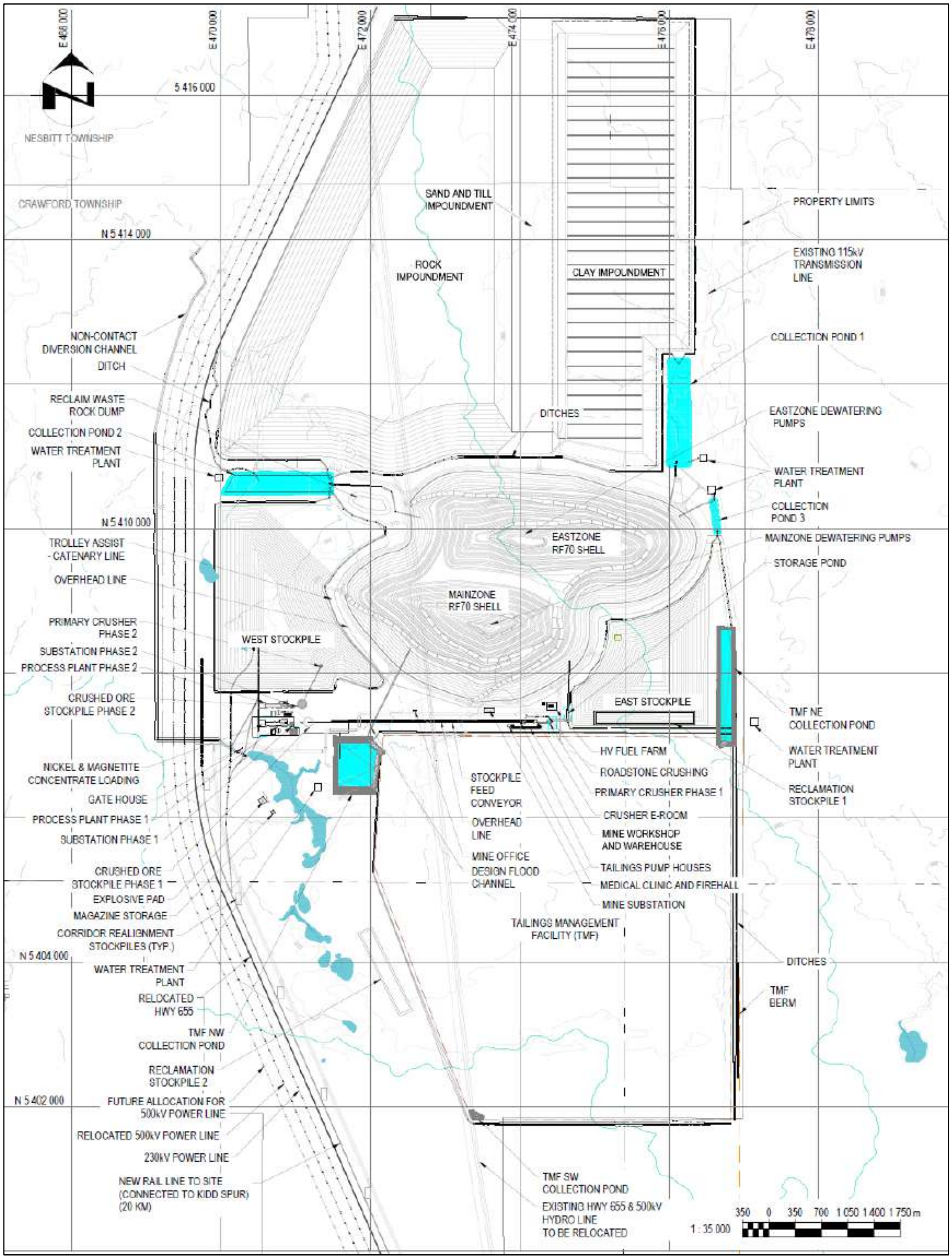
The project site layout (shown in Figure 18-1) has been developed during the feasibility study to take into consideration the following key constraints:

- ore and rock stockpiles located immediately adjacent to open pits
- run-of-mine pad, primary crushing and fuel farm located next to south exit from open pits
- infrastructure locations will avoid water bodies and fish habitats as much as possible
- process plant located near the relocated highway, powerline and rail corridor for personnel and power access and ease of transport of products.

Figure 18-1 shows the major project facilities, which include:

- open pit mines
- tailings management facility (TMF)
- impoundment facility (comprising three separate zones for the impoundment of clay, sand and till, and rock)
- Phase 1 and Phase 2 crushing facilities and crushed ore stockpiles
- Phase 1 and Phase 2 processing facilities
- site support buildings, including the mine workshop and offices
- water management structures
- existing and new powerlines
- existing and relocated Highway 655
- new rail line.

**Figure 18-1: Site Layout Plan**




Source: Ausenco, 2023.

## 18.2 Roadways

### 18.2.1 Site Access

The project site is accessed from Ontario Highway 655, which links Ontario Highway 101 and the Trans-Canada Highway 11. An existing secondary local road leads directly to the property, west from Highway 655. New roads will be developed as necessary to provide access to different areas of the site.

A portion of Highway 655 will be relocated around the project site. The new highway section, approximately 25.7 km in length, will be constructed to the specifications of the Ontario Ministry of Transportation.

### 18.2.2 Highway 655 Overpass

The realignment of Highway 655 will be completed during Year 2 of mill operations. To permit access between the primary crusher and the plant site prior to this time, an overpass will be constructed. This will entail the following:

- construction of a temporary bypass (two lanes wide and approximately 2,700 m long) to route highway traffic around the construction zone
- installation of a multi-plate arch structure that will allow road access between the primary crusher and the process plant, underneath Highway 655.

The arch has been sized (14.8 m wide x 6.8 m high) to permit one-way passage of 40 t articulated trucks alongside the coarse ore conveyor.

## 18.3 Highways

Highway 655 is a secondary highway in the Cochrane District of Northern Ontario. It links Highway 101 in Timmins and Highway 11 near Cochrane. The alignment of the existing Highway 655 runs north to south, through the western portion of the Crawford Mine property, dividing the proposed Crawford open pit footprint into two parts. A 25.7 km section of Highway 655 will be realigned to the western boundary of the Crawford Mine site and the existing Highway 655 section being by-passed will be decommissioned.

The existing two-lane highway operates with a design speed of 110 km/h and is posted with a regulatory speed limit of 90 km/h.

## 18.4 Rail Infrastructure

The Crawford mine site will require a new rail spur line to transport materials to and from the mine site. The closest freight rail spur line is located approximately 20 km south of the Crawford property. This existing rail spur line is owned by Ontario Northland Railway (ONR), (Government of Ontario), and utillized by Glencore Canada's Kidd Operations, who currently transports mineralized rock from its mine site to its metallurgical site.

A new 25.2 km rail spur line is proposed to branch off of ONR's existing spur line near its intersection with the existing Highway 655 corridor and run parallel with the realigned section of highway (on the east side) to the new Crawford mine site.

The new rail line construction will include spur lines at the plant site to accommodate loading and unloading of materials.

Design standards for the proposed ONR spur line will match the design standards for the existing spur line. Due to the proximity of the proposed rail embankment to the realigned Highway 655, the profile of the highway and rail will need to be coordinated, to avoid high/steep slopes between the highway and railway.

## 18.5 Power Supply

### 18.5.1 Site Power Supply

Hydro One Networks Inc. (HONI) will provide electrical power to the Crawford Mine site. Since the Crawford mine's load is expected to exceed 150 MW over the life of mine, the electrical transmission system necessary to feed the mine will be built in two phases, which are as follows:

- Phase 1 will require an approximately 40 km, single-circuit, wooden-pole 230 kV transmission line (H-frame style structure) that would be constructed between Hydro One's Porcupine Substation located near Timmins, ON, and a newly constructed 230/34.5 kV switching station located on the southwest corner of the Crawford mine site. This Phase 1 powerline will be sufficient to meet the initial 150 MW of load.

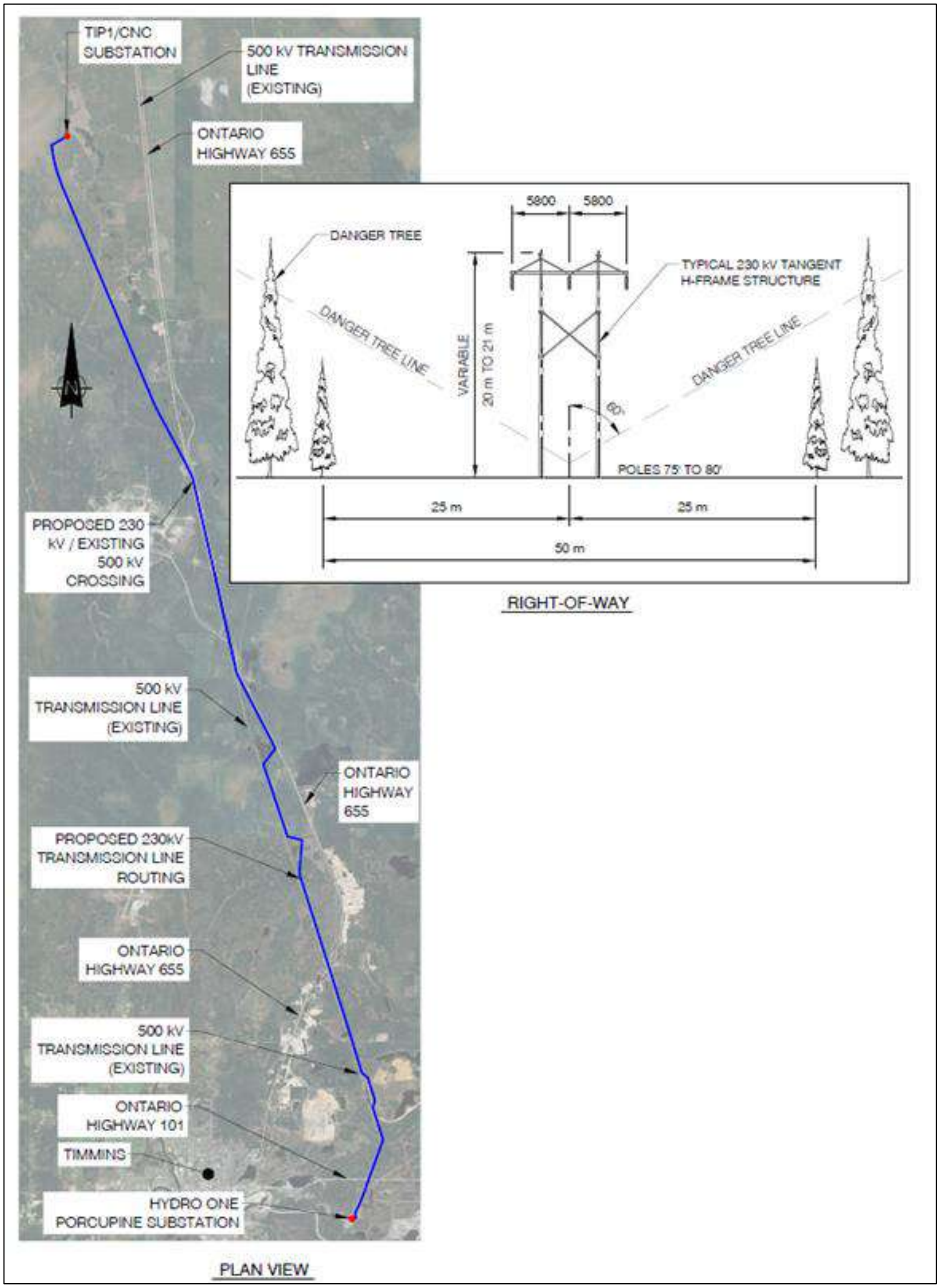
- Phase 2 will be constructed once the load increases beyond 150 MW. This phase will require a second approximately 40 km, single-circuit, wooden-pole 230 kV transmission line that would be constructed between the same Hydro One Porcupine Substation to the switching station at the mine site constructed for Phase 1. It is assumed that the Phase 2 transmission line will be constructed on a new right-of-way and not within the same right-of-way as used in Phase 1.

The proposed wire cable is 795 MCM aluminum conductor steel-reinforced (ACSR) conductor because of its economic, mechanical (strength to weight), and electrical properties (resistance to electromagnetic interference) along with its long service record in similar applications.

Figure 18-3 shows the proposed transmission line routing and a preliminary arrangement drawing for the H-frame style wood-pole structures.

The proposed transmission lines for both phases will be financed, constructed, owned, operated, and maintained by Transmission Infrastructure Partnerships 1 (TIP1). TIP1 will recover their investment from CNC over a 25-year period, with the payments recorded as a G&A operating cost expense.

**Figure 18-2: Electrical Powerline Routing and Wood Pole H-Frame Structure General Arrangement**




Source: BBA, 2023.

### 18.5.2 Electrical System Demand

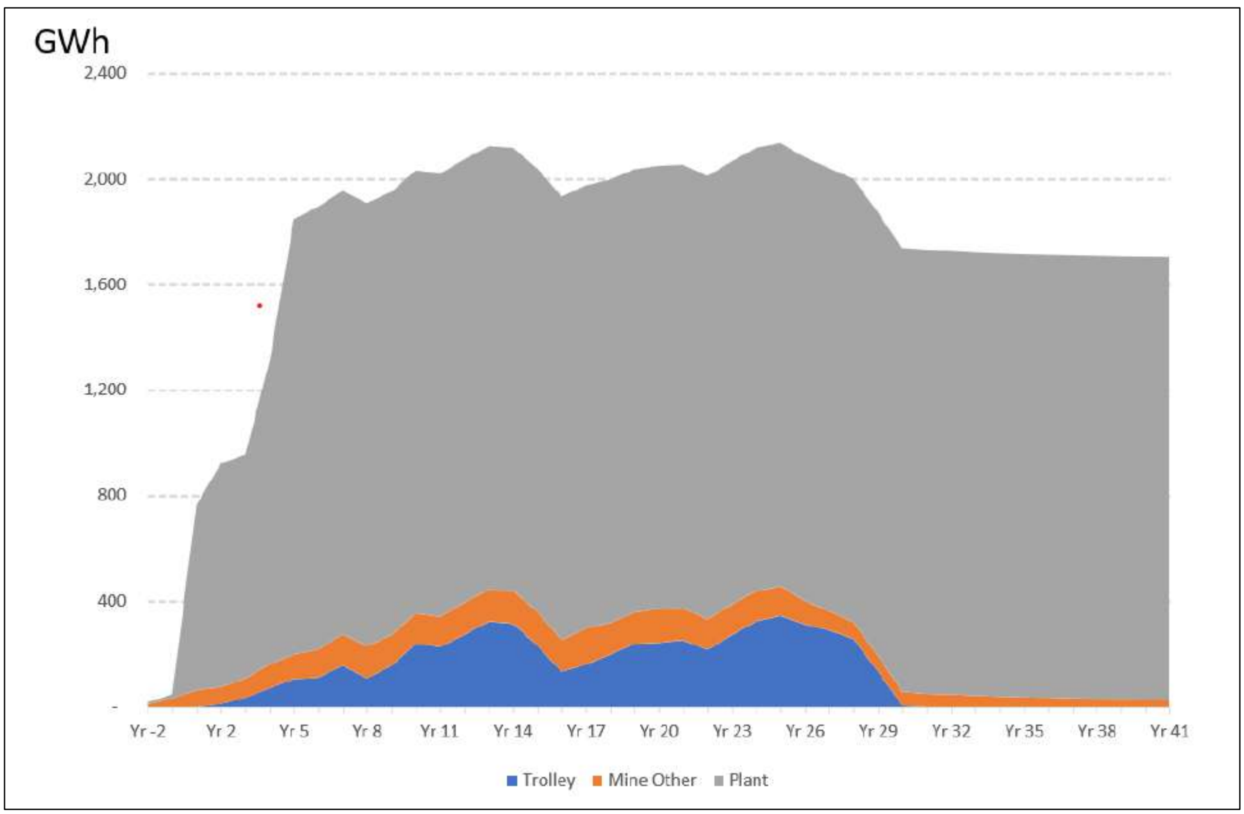
Table 18-1 shows the maximum demand and operating loads for both project phases.

**Table 18-1: Maximum Demand and Operating Loads**

| Description | Maximum Demand MW | Operating Load MW |
|---|---|---|
| Process Plant Load | | |
| Phase 1 | 115 | 100 |
| Phase 2 | 221 | 193 |
| Mine Site Load | | |
| Phase 1 | 34 | 24 |
| Phase 2 | 88 | 70 |
| **Phase 1 Loads** | **149** | **124** |
| **Phase 2 Loads** | **309** | **263** |

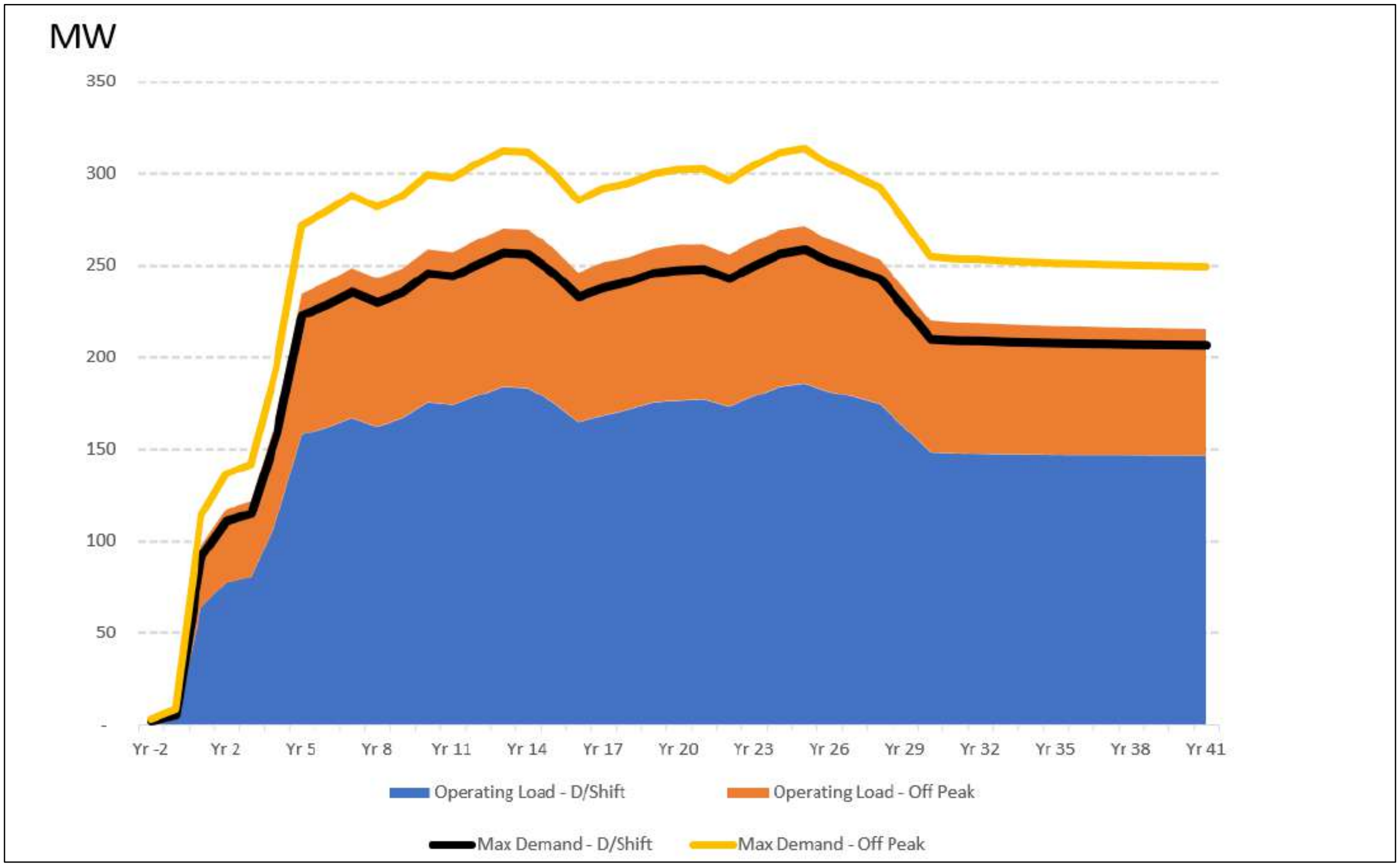
The cost of power in Ontario is based on consumption (total megawatt-hours) and time of usage demand (megawatts). Figure 18-3 and Figure 18-4 illustrate total consumption and time of usage demand over the project life.

**Figure 18-3: Crawford Electrical Consumption by Area**




Source: CNC, 2023.

**Figure 18-4: Crawford Electrical Demand by Time of Use**




Source: CNC, 2023.

### 18.5.3 On-Site Power Distribution

#### 18.5.3.1 Main Substation

The Crawford, 230 kV / 34.5 kV, 336 / 450 MVA, air-insulated outdoor substation is sized to accommodate the Phase 1 process plant and mine site loads (Phases 1 and 2). The substation will consist of three 112/150 MVA, ONAN/ONAF-rated power transformers in radial distribution with an N-1 transformer contingency and will be located to the southeast corner of the process plant. It will be supplied via three 230 kV, 150 MVA overhead lines that originate at the 230 kV ring bus switchyard.

The 230 kV / 34.5 kV, 336 / 450 MVA air-insulated outdoor substation will consist of a dead-end structure, bus bars, motorized disconnect switches, circuit breakers, and a three-transformer arrangement with one transformer in hot standby service. The transformers will not be designed to operate in parallel but will be sized and interconnected so that if one transformer is out of service, the remaining two transformers will be capable of sustaining the entire process plant and mine site operating load.

During normal Phase 1 substation operation (i.e., three transformers in service), the operating load on any of the power transformers is not forecast to exceed 53% of the transformer's base rating, whereas, under abnormal operation (i.e., one failed transformer, two out of three transformers are still in service), the operating load is projected at 78% of the transformer's forced cooled rating.

For Phase 2, a dedicated 230 kV / 34.5 kV, 75 / 100 MVA air-insulated outdoor substation is planned to be constructed to facilitate the new process plant load requirements. Under normal operation (i.e., three transformers in service), the operating load on any of the power transformers is not forecast to exceed 56% of the transformer's base rating, whereas under abnormal operation (i.e., one failed transformer, so two out of three transformers in service), the operating load is projected at 84% of the transformer's forced cooled rating.

#### 18.5.3.2 Site Power Reticulation

The 230 kV incoming supply by Hydro One will be stepped down at the Crawford, 230 kV / 34.5 kV, 336 / 450 MVA, air-insulated outdoor substation and distributed to the process plant and mine site consumers via the plant's primary distribution electrical room and 34.5 kV circuits.

Power distribution to the process plant major consumers (mills and distribution electrical rooms) will be achieved via 34.5 kV feeders and cabled circuits, whereas distribution to the mine site area, water management ponds, and crushing area will be by 34.5 kV overhead lines.

The 34.5 kV overhead distribution lines will be constructed using ACSR cable and will be supported on wooden poles.

#### 18.5.3.3 Standby / Emergency Power Supply

Standby power will be provided to the emergency loads identified in the mechanical equipment and electrical load list, via distributed standby diesel generators in weatherproof enclosures. These generators will be appropriately sized and specified, located adjacent to the designated electrical room, and connected to the respective emergency motor control centre via an automatic transfer switch.

#### 18.5.3.4 Plant Power Distribution

The primary plant distribution voltage will be 34.5 kV and will be used to deliver bulk power to large electrical consumers and remote electrical rooms around the process plant, mine site and off-site locations.

The SAG mill, primary ball mill, and secondary ball mill are the largest electrical consumers at the process plant. All three mills will be operated using dedicated transformers and variable frequency drive assemblies, supplied via 34.5 kV cabled circuits which originate from the plant's primary distribution electrical room. All other electrically operated loads at the process plant, crushing area, and stockpile area will be supplied by one of 12 strategically located distributed electrical rooms.

The electrical rooms for the process plant and stockpile area will be supplied via 34.5 kV cabled circuits. This distribution voltage will be stepped down to 4.16 kV and 600 V using resistance-grounded, secondary substation type, oil-filled distribution transformers that will be located adjacent to the respective electrical room. Larger electrical-driven process loads (e.g., cyclone feed pumps, conveyors, coarse and fines flotation cells, deslime pumps, and regrind mills) will operate at 4.16 kV, whereas the smaller three-phase loads will operate at 600 V.

Power to the mine workshop and crushing area will be supplied by 34.5 kV overhead lines. Resistance and solidly grounded, substation-type and pad-mounted transformers will be used to step down the distribution voltage for utilization. An electrical room will house the electrical distribution equipment at the crushing area.

All electrical rooms will be adequately rated for the environment and will be outfitted with heating and ventilation, lighting and small power transformers, distribution boards, and uninterrupted power supply (UPS) systems. To reduce installation time, the electrical rooms will be prefabricated modular buildings, installed on structural framework above ground for bottom cable entry. Electrical rooms will be located as close as practical to the electrical loads, thereby minimizing voltage drop concerns and cable cost.

### 18.5.4 Mine Area Electrical Distribution

The overall maximum demand of the mine area electrical load (i.e., Phases 1 and 2 mine fleet, trolley assist, dewatering and water management systems) is an estimated 67 MW at 0.98 power factor lag.

The maximum demand of the Phase 1 mine site loads (i.e., fleet, trolley assist, dewatering and water management systems) is an estimated 37 MW at 0.98 power factor lag, with an additional 30 MW at 0.98 power factor lag in Phase 2.

Infrastructure (e.g., transformers and switchgear) required to meet the mine site load demand is included in Phase 1 development. Bulk power will be delivered from the process plant's primary distribution electrical room to the mine site via two 34.5 kV overhead lines with an estimated combined capacity of 100 MW to support mine operations.

The 34.5KV overhead lines shall deliver power to the mine area substation which will further distribute power to trolley assist, mine fleet, dewatering pumps, and water management system loads.

## 18.6 Process Plant Area Buildings

The process plant area buildings are listed in Table 18-2. Additional details are provided in the following subsections.

### 18.6.1 Primary Crushing

There are two identical primary crushing buildings. One building will be built for Phase 1 and the other for Phase 2.

The primary crushing building is a buried concrete vault complete with a stick-built crane-supporting superstructure. This building will house the primary crushing plant, including the run-of-mine bin, rock breaker, primary crusher, apron feeder, dust collector, access platforms, and support structures.

The building and its interior components are supported on a reinforced concrete raft foundation on bedrock 30 m below finished grade. The raft foundation also serves as the flooring within the building.

There is a 150 m long conveyor tunnel bringing ore from the primary crusher to the surface.

The building will be equipped with adequate heating, ventilation, air-conditioning (HVAC), lighting, dust collection, and a 90-tonne overhead crane.

**Table 18-2: Process Plant Area Buildings**

| Building Name | Building Construction Type | Phase | Geometry | | |
|---|---|---|---|---|---|
| | | | L (m) | W (m) | H (m) |
| Primary Crushing | Buried concrete vault with stick-build superstructure | 1 | 23 | 13 | 30 |
| Secondary Crushing | Stick-Built | 1 | 34 | 34 | 39 |
| Crushed Ore Stockpile Cover | Geodesic Dome | 1 | 108 m diameter | | 44 |
| Process Plant Building | Pre-Engineered | 1 | 288 | 88 | 35 |
| Concentrate Loadout Building | Pre-Engineered | 1 | 79 | 100 | 12 |
| Plant Warehouse | Pre-Engineered | 1 | 40 | 34 | 8 |
| Plant Maintenance Building | Pre-Engineered | 1 | 36 | 18 | 9 |
| Plant Office | Modular | 1 | 22 | 18 | 3 |
| Medical Clinic & Firehall | Modular | 1 | 20 | 40 | 3 |
| Gatehouse | Modular | 1 | 12 | 4 | 3 |
| Assay Lab | Modular | 1 | 15 | 9 | 4 |
| Tailings Pumphouse | Modular | 1 | 13 | 7 | 3 |
| Primary Crushing | Buried concrete vault with stick-build superstructure | 2 | 23 | 13 | 30 |
| Secondary Crushing | Stick-Built | 2 | 34 | 34 | 39 |
| Crushed Ore Stockpile Cover | Geodesic Dome | 2 | 108 m diameter | | 44 |
| Process Plant Building | Pre-Engineered | 2 | 288 | 88 | 35 |
| Concentrate Loadout Building | Pre-Engineered | 2 | 79 | 100 | 12 |
| Plant Warehouse Expansion | Pre-Engineered | 2 | 40 | 34 | 8 |
| Plant Maintenance Building Expansion | Pre-Engineered | 2 | 36 | 18 | 9 |
| Tailings Pumphouse | Modular | 2 | 13 | 7 | 3 |

### 18.6.2 Secondary Crushing

There are two identical secondary crusher buildings. One building will be built for Phase 1 and the other for Phase 2.

The secondary crushing building is a stick-built building. This building will house a feed bin, pair of secondary screens, and pair of secondary crushers.

The building and its interior components are supported on a reinforced concrete pile cap and steel piles down to bedrock. The pile cap also serves as the flooring within the building.

The building will be equipped with adequate HVAC, lighting, dust collection, and a 40-tonne overhead crane.

For Phase 1, there is a 3 km conveyor connecting the secondary crushers to the crushed ore stockpile. Reinforced precast concrete foundations are used for the overland portion of the conveyor and reinforced concrete foundations on steel piles down to bedrock are used for the elevated portion of the conveyor.

#### 18.6.3 Crushed Ore Stockpile Cover

There are two identical stockpile covers. One building will be built for Phase 1 and the other building will be built for Phase 2. Each stockpile cover is a geodesic dome. This building will house the crushed ore stockpile.

The building comes complete with the following:

- Cladding: single skin (non-insulated) metal cladding
- HVAC: passive exhaust on the roof, unheated
- Lighting: high-bay LED lighting and emergency lighting
- Overhead doors: two 5.5 m (wide) x 4.5 m (high) doors.

The building is supported on reinforced concrete strip footing pile caps and steel piles down to bedrock. Beneath the stockpile is a reinforced concrete reclaim tunnel, which is also supported by steel piles down to bedrock.

The ground supporting the crushed ore stockpile requires improvement to increase its bearing capacity. Ground improvement will be done using the wick drain technique.

#### 18.6.4 Process Plant Building

There are two identical process plant buildings. One building will be built for Phase 1 and the other for Phase 2.

The process plant building is a pre-engineered, rigid-frame, metal building. This building will house the grinding and chemical processing facilities, including the semi-autogenous grinding (SAG) mill, primary and secondary ball mills, regrind mills, thickeners, coarse flotation cells, fine flotation cells, reagents, and access and operating platforms.

The building comes complete with the following:

- Cladding: insulated metal panel (IMP) roof and wall cladding
- HVAC: exhaust fan, electric unit heater (for shutdown only) as needed to maintain minimum indoor temperature of 5°C
- Lighting: high-bay LED lighting and emergency lighting

- Overhead doors: two 7 m (wide) x 8 m (high) doors and ten 5 m (wide) x 8 m (high) doors
- Overhead cranes: one 140-tonne with a 10-tonne auxiliary hook and five 30-tonne cranes.

The building and its interior components are supported on a reinforced concrete pile cap and steel piles down to bedrock. The pile cap also serves as the flooring within the building.

There is a 63 m diameter thickener, near the process plant building, supported on pile caps and steel piles down to bedrock. The pile cap will constitute the floor of the thickener. Thickener walls are steel plate walls.

#### 18.6.5 Concentrate Loadout Buildings

There are two identical concentrate loadout buildings containing the nickel concentrate filter press, nickel concentrate stockpile, magnetite concentrate filter presses, and magnetite concentrate stockpile. One building will be built for Phase 1 and the other building will be built for Phase 2.

The concentrate loadout building is a pre-engineered, rigid-frame, metal building.

The building comes complete with the following:

- Cladding: insulated metal panel (IMP) roof and wall cladding
- HVAC: exhaust fan, electric unit heater (for shutdown only) as needed to maintain minimum indoor temperature of 5°C
- Overhead doors: four 7 m (wide) x 8 m (high) doors.

The building and its interior components are supported on a reinforced concrete pile cap and steel piles down to bedrock. The pile cap also serves as the flooring within the building.

#### 18.6.6 Plant Warehouse

A plant warehouse building will be built for Phase 1 and the building will be expanded in Phase 2.

The plant warehouse is a pre-engineered, rigid-frame, metal building.

The building comes complete with the following:

- Cladding: insulated metal panel (IMP) roof and wall cladding
- HVAC: exhaust fan, electric unit heater (for shutdown only) as needed to maintain minimum indoor temperature of 5°C
- Overhead doors: four 4 m (wide) x 4 m (high) doors after Phase 1 and eight 4 m (wide) x 4 m (high) doors after Phase 2

The building and its interior components are supported on a reinforced concrete raft foundation on in-situ soil. The raft foundation also serves as the flooring within the building.

### 18.6.7 Plant Maintenance Building

A pre-engineered, rigid-frame, metal, plant maintenance building will be built for Phase 1 and expanded in Phase 2. The building comes complete with the following:

- Cladding: insulated metal panel (IMP) roof and wall cladding
- HVAC: exhaust fan, electric unit heater (for shutdown only) as needed to maintain minimum indoor temperature of 5 °C
- Overhead doors: eight 4 m (wide) x 4 m (high) doors after Phase 1 and sixteen 4 m (wide) x 4 m (high) doors after Phase 2

The building and its interior components are supported on a reinforced concrete raft foundation on in-situ soil. The raft foundation also serves as the flooring within the building.

### 18.6.8 Ancillary Buildings

A plant office, medical facility and firehall, gatehouse, assay laboratory, and tailings pumphouse will be built in Phase 1. An additional tailings pumphouse will be built in Phase 2. All of these will be modular buildings with the necessary offices, cubicles, meeting rooms, lunchrooms, lobbies, washrooms, HVAC, lighting, windows, and doors.

The assay laboratory will come complete with all laboratory equipment.

These modular buildings are all supported by precast concrete footings on in-situ soil, except for the tailings pumphouses, each of which are supported on a reinforced concrete pile cap and steel piles down to bedrock, in large part to provide adequate support to the pumps within the facilities.

## 18.7 Mine Infrastructure Buildings

The mine infrastructure buildings are listed in Table 18-3. Additional details are provided in the following subsections.

**Table 18-3: Mine Area Buildings**

| Building Name | Building Construction Type | Phase | Geometry | | |
|---|---|---|---|---|---|
| | | | L (m) | W (m) | H (m) |
| Mine Workshop | Pre-Engineered | 1 | 256 | 25 | 15 |
| Mine Workshop Expansion | Pre-Engineered | 2 | 96 | 25 | 15 |
| Mine Warehouse | Fabric | 1 | 36 | 18 | 6 |
| Mine Workshop Office | Modular | 1 | 25 | 25 | 3 |
| Mine Office | Modular | 1 | 44 | 18 | 3 |

### 18.7.1 Mine Workshop

A mine workshop for servicing and washing the mine fleet will be constructed during Phase 1. The workshop will be expanded with additional service bays during Phase 2. A further expansion to the ultimate configuration of 26 bays will take place after Phase 2 and has been included as sustaining capital.

The mine workshop building is a pre-engineered, rigid-frame, metal building that comes complete with the following:

- Cladding: insulated metal panel (IMP) roof and wall cladding
- HVAC: exhaust fan, electric unit heater (for shutdown only) as needed to maintain minimum indoor temperature of 5°C
- Overhead doors: sixteen 7 m (wide) x 8 m (high) doors after Phase 1 and twenty-two 7 m (wide) x 8 m (high) doors after Phase 2
- Overhead cranes: two 36-tonne cranes for Phase 1 and one 36-tonne crane for the Phase 2 expansion.

The mine workshop building and its interior components are supported on a reinforced concrete pile cap and steel piles down to bedrock. The pile cap also serves as the flooring within the building.

### 18.7.2 Mine Warehouse

A mine warehouse will be built during Phase 1. The mine warehouse is a fabric building that comes with the following:

- structural steel framing and insulated fabric membrane
- four 4 m (wide) x 4 m (high) overhead doors.

The mine warehouse is supported by shipping container foundations on in-situ soil. No concrete is required.

### 18.7.3 Offices

A workshop office will be built during Phase 1. It will be a modular building with four offices, 16 cubicles, meeting room, lunchroom, lobby, two washrooms, HVAC, lighting, windows, and personnel doors. The workshop office will be supported by precast concrete foundations on in-situ soil.

A mine office will be built during Phase 1. The mine office is a modular building with with five offices, 20 cubicles, meeting room, lunchroom, and lobby, two washrooms, HVAC, lighting, windows, and personnel doors. The mine office will be supported by precast concrete foundations on in-situ soil.

## 18.8 Site-Wide Geotechnical Considerations

### 18.8.1 Overview

Due to the evolution of the mine design and issues linked to land tenure, an iterative approach to sighting infrastructure for the project was utilized. This is specifically true with respect to the location and function of the stockpiles and impoundment facility. The infrastructure locations presented reflect the feasibility-level designs for the project.

### 18.8.2 Geotechnical

The site-wide geotechnical considerations, including field program investigations and resulting geotechnical model findings, are detailed in Section 16.2.

Stability criteria for the geotechnical analysis were based on existing guidelines and recommendations. The final evaluation of geotechnical stability involved numerical deformation analyses to assess how the rate of rise affects the stability of the facilities. During construction, pore water pressure will build up in the saturated clay foundation, significantly impacting stability. The design accounted for a maximum rate of rise of 5 metres per year for the rock impoundment and 2.5 metres per year for the other facilities (see Section 18.9). The rate of rise was determined from estimated rock production, foundation clay thickness and its hydraulic conductivity. The analyses assumed a 30 m thick clay layer with a vertical hydraulic conductivity of 2.0 E-10 m/s, determined from limited laboratory permeability tests on Shelby-tube clay samples, and assumed the groundwater to be at ground level. The assumed 30 m thickness is considered conservative, and it represents a typical package for the foundation clay layer based on the available data and geological understanding.

### 18.8.3 Hydrogeology

The numerical groundwater model generally simulates relatively low infiltration from the nearby infrastructure when compared to the groundwater inflows simulated for the open pits. This is due to the low hydraulic conductivity of the (upper) clay as well as its thickness and continuity. The numerical model does predict higher infiltration for the western low-grade stockpile, since a relatively small part of its footprint is estimated to be located above more permeable material, associated with an esker trending north-south slightly to the west of the proposed pits and most of the site infrastructure, according to the LeapFrog overburden model.

Infiltration from infrastructure to the open pits will be handled using the same operational dewatering system as proposed for open pit dewatering elsewhere in this report.

Based on the current dataset and overburden models, the application of 350 millimetres per year (mm/y) for the inferred area of the esker is considered conservative yet applicable. There are multiple test pits exhibiting sand at the surface near the axis of the esker. These test pits do appear to be concentrated in the southern portion of the esker relative to the site (i.e., south of Mel and Davis lakes); however, there are mixed results in the portion of esker near the northern edge of the TMF. Recharge to the pits can occur through surficial sand, if there is indeed a direct hydraulic connection, which is currently inferred by modelling.

## 18.9 Stockpiles and Impoundment Facility

The locations designated for the construction of the stockpiles and impoundment facility are overlain by a layer of organic soil (fibrous peat), averaging around 1.0 m in thickness. Stability analyses indicate this layer does not need to be removed prior to initiating the construction process. The prevailing ground conditions are characterized by fine-grained, plastic soils that are saturated, have a sticky consistency and vary from very stiff to soft at a depth pf approximately 7 m.

The stockpiles and impoundment facility are located at the following minimum distances from specific structures (Figure 18-5):

- 250 m away from the open pit, mine plant, and other existing structures (e.g., roads, powerlines, etc.) at a minimum.
- minimum distance of 100 m from the project's water ponds.

**Figure 18-5: Ore Stockpiles and Impoundment Facility General Arrangement**


Source: SRK, 2023.

### 18.9.1 Lower Value Stockpiles

A key element of the mining strategy is the decoupling of mine and mill production rates, with the mine releasing ore at a faster rate than can be digested by the mill. This allows higher-value ore to be fed to the mill as run of mine while lower-value ore is temporarily stockpiled in one of two stockpiles that are located near the primary crushers.

The east stockpile will be established first, with the initial ore stockpiled during the construction period. The east stockpile has been designed to impound a maximum of 100 $Mm^3$, during the planned mine life the maximum impounded volume will be reached resulting in the design height of 100 m in Year 20.

First material will be delivered to the west stockpile in preparation for commissioning of the expansion, which includes the western Phase 2 primary crusher. This stockpile has a larger footprint than the east stockpile and has been designed to impound a maximum of 165 $Mm^3$, but the maximum impounded at any period in the planned mine is 140 $Mm^3$. This facility has also been designed to a height of 100 m but will only reach a maximum of 70 m.

Further details regarding the stockpiles are given in Section 16.3.6.

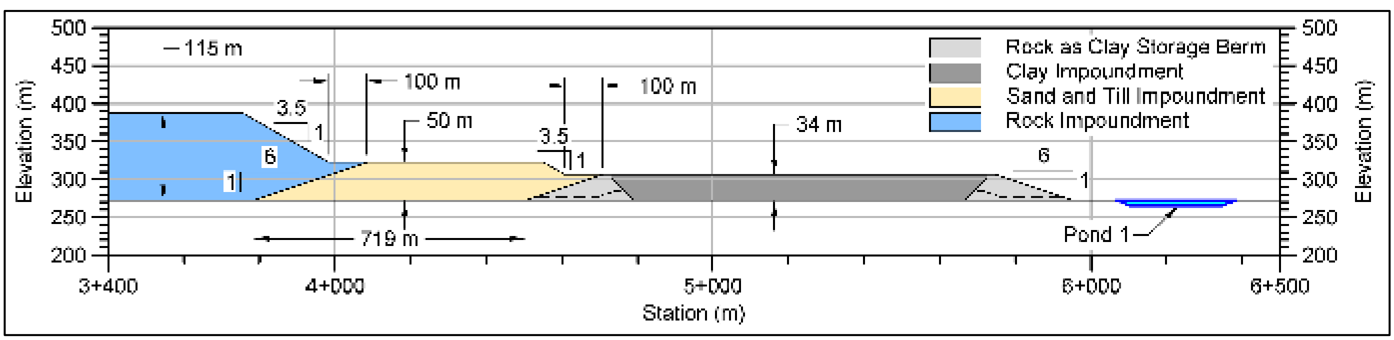
### 18.9.2 Impoundment Facility

The impoundment facility is located north of the open pit (Figure 18-5) and has an area of 31.5 $km^2$. The facility includes the following:

- clay impoundment facility to impound 205 $Mm^3$ of clay
- sand and till impoundment facility to store 182 $Mm^3$ of sand and till
- rock impoundment facility to store 1,438 $Mm^3$ of waste rock.

Figure 18-6 shows a cross-section in the west-to-east direction of the impoundment facility.

**Figure 18-6: Cross-Section in the West-to-East Direction of the Impoundment Facility**



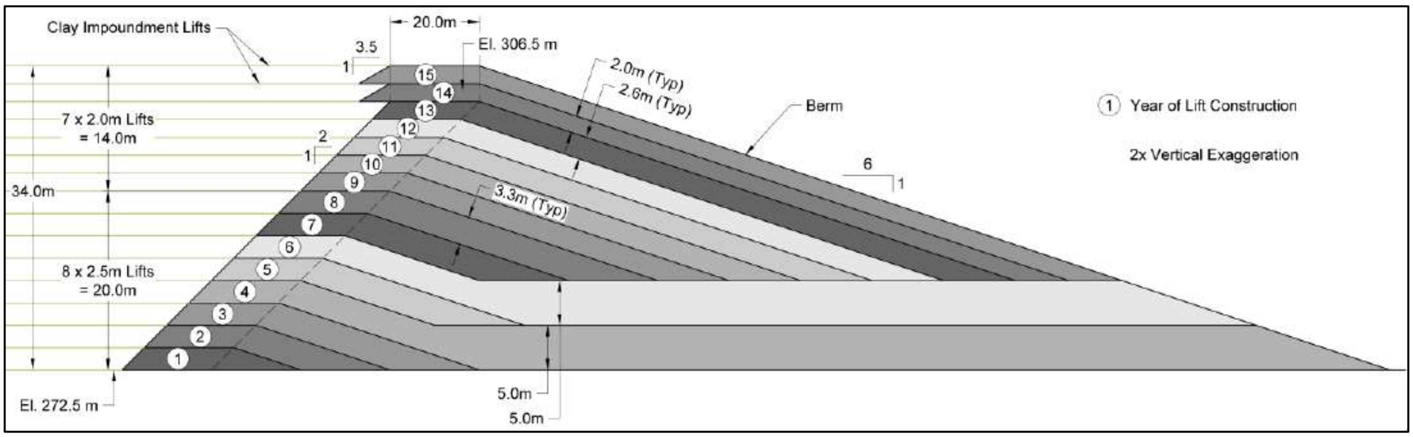

Source: SRK, 2023.

### 18.9.2.1 Clay Impoundment Facility

The clay storage facility will impound 205 $Mm^3$ of clay retained by 34 m high perimetral waste rock clay storage berms. The internal slope of the perimetral berm (i.e., the berm slope facing the impounded clay in the facility) will be inclined 2.0 H:1 V. The external berm slope will have an inclination of 6H:1V (Figure 18-7). The total volume of waste rock required to build the berms is 74.5 $Mm^3$.

The berms will be raised in a combination of downstream construction, horizontal lifts, and centreline construction (Figure 18-7).

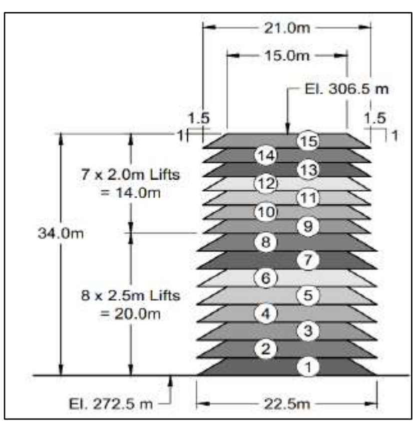
**Figure 18-7: Details of the Clay Storage Berm**




Note: The numbers in circles refer to the construction sequence. Source: SRK, 2023.

The clay impoundment facility will be divided into 28 cells, each separated by 34 m high waste rock ribs. These structures are designed to contain clay as it is gradually deposited into the cells, rising incrementally with the increasing levels of clay. In addition to their containment role, they will also serve a dual purpose as roads for clay haulage. The construction of the ribs requires 22 $Mm^3$ of waste rock. Figure 18-8 provides a detailed view of the ribs.

**Figure 18-8: Detail of the Ribs**




Source: SRK, 2023.

#### 18.9.2.2 Sand & Till Impoundment Facility

The sand and till facility will store 182 $Mm^3$ of sand and till (Figure 18-5). The facility will have a final height of 50 m with slopes inclined at 6H:1V. The facility will be raised in horizontal lifts at a rate not exceeding the specified 2.5 m/y to allow for sufficient dissipation of excess pore water pressure, contributing to the stability of the facility during construction. The specified rate of rise is based on the assumed thickness of the foundation clay layer and its hydraulic conductivity determined from laboratory tests.

#### 18.9.2.3 Rock Impoundment Facility

The rock impoundment facility will be constructed adjacent and to the west of the sand and till facility. It is expected that the facility will accommodate 1,438 $Mm^3$ and reach a height of 110 m (Figure 18-5).

## 18.10 Tailings Management Facility

### 18.10.1 Introduction

According to the mine plan, tailings will be deposited into a surface tailings management facility (TMF), until the middle of Year 18, then the tailings will be deposited into the Main Zone pit. In Year 33, tailings deposition will switch from the Main Zone pit to the East Zone pit. It is estimated that 624 Mt of tailings will be produced before in-pit deposition begins. This corresponds to a TMF volume of 495 $Mm^3$. As shown on Figure 18-1, the TMF will be located south of the open pit and will have an approximate footprint area of 2,300 ha. Containment of tailings within the TMF will be provided by a perimeter dam that will be constructed in stages.

The TMF site is relatively flat with an average slope of 0.4% from south to north (an approximate 20 m decrease in elevation from south to north). To achieve storage capacity requirements and to accommodate the site geotechnical conditions, the TMF will be operated as a “thickened tailings cone” with tailings deposition near the centre of the TMF. The thickened tailings technology has been adopted because it is expected to produce a steeper beach slope (relative to conventional slurry deposition) which contributes to increasing the TMF storage capacity and limiting the perimeter dam height (required for dam stability). This approach is similar to the one adopted for many years at the nearby Kidd Metsite. A tailings beach slope of 2.5% has been assumed based on precedence. The target tailings discharge density is 39% solids.

Runoff from the TMF, together with slurry water and bleed water released from the tailings, will be conveyed by gravity within the TMF, and will pass through internal TMF service spillways into the adjacent external TMF water management ponds (WMPs). Seepage and runoff from the downstream shell of the TMF perimeter dam will be collected in perimeter contact water channels and will be directed to the TMF WMPs.

A preliminary geotechnical investigation was carried out at the TMF site to characterize the subsurface conditions to support the feasibility level design. Results of the investigation indicate that the site is underlain by thick overburden deposits between 23 and 67 m deep. The overburden deposits (from bottom to top) consist of glacial till, glacio-fluvial sands, and glacio-lacustrine clay, with surficial organic deposits. The stability of the TMF perimeter dam is governed by the low shear strength of the glacio-lacustrine clay deposits, which range in thickness from approximately 10 to 30 m.

### 18.10.2 TMF Design

The perimeter dam has been designed to meet the requirements of the Canadian Dam Association (CDA) Dam Safety Guidelines. In accordance with the latest CDA dam classification methodology (CDA, 2013), the proposed TMF dam has been classified in a “very high” failure consequence category during the construction, operation, transition, and closure active care phases and as “significant” for the closure passive care phase. In accordance with the CDA (2013) guidelines, an inflow design flood (IDF) of 2/3 between the 1:1000-year return flood and the probable maximum flood (PMF) was considered for the construction, operation, transition, and closure active care phases. An IDF of 1/3 between the 1:1000-year flood and the PMF was considered for the closure passive care phase.

To reduce potential for storm inflow to overtop the perimeter dams, the TMF internal service spillways and TMF WMP spillways will be designed to pass the IDF flow. Other water management facilities within the TMF, such as the downstream seepage collection channels, will be designed to safely pass the runoff from the following events:

- construction, operation, transition, and closure active care phases: 1 in 100 years
- closure – passive care phase: 1 in 100 years.

In accordance with CDA (2013), the peak ground acceleration (PGA) considered in the dam pseudo-static stability analysis was selected corresponding to the dam classification and annual exceedance probability (AEP) earthquake as follows:

- construction, operation, transition, and closure active care phases: AEP of ½ between 1:2475-year return and 1:10000-year return
- closure passive care phase: AEP of 1:2475-year return.

The PGA corresponding to the AEP was sourced from the online 2020 National Building Code of Canada Seismic Hazard Tool and corresponding to Site Class D (NRCC, 2020).

Dam stability criteria are based on the most recent CDA Dam Safety Guidelines (2013). The applicable stability factors of safety are as follows:

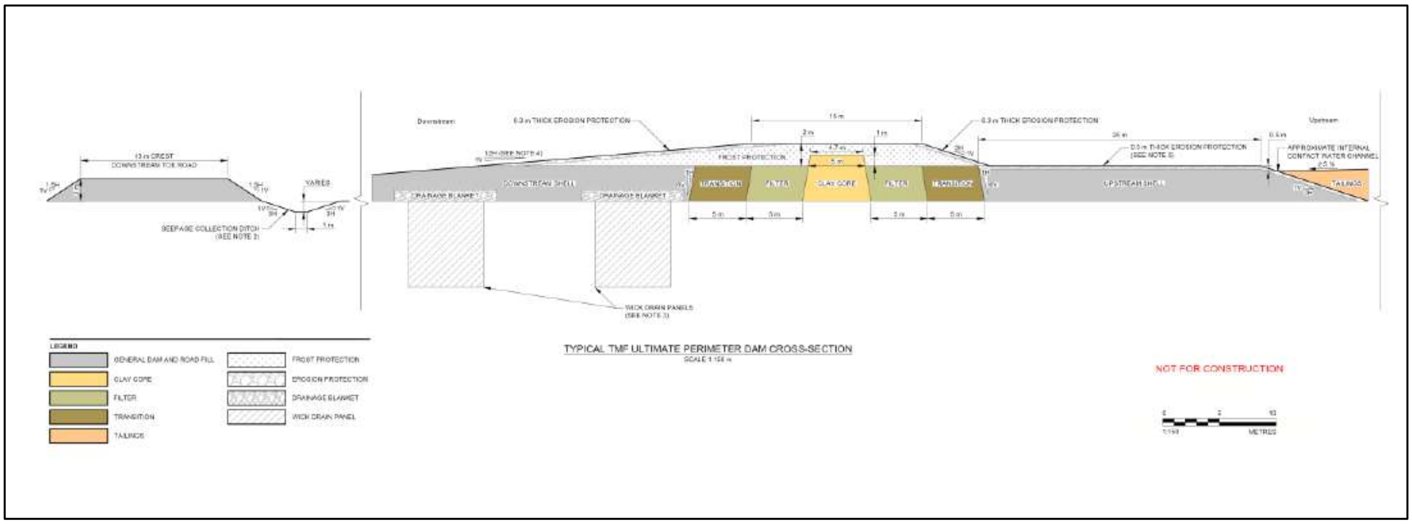
- static, short-term, end of construction = 1.5
- pseudo-static = 1.0
- post-earthquake = 1.1.

The perimeter dam typical cross-section consists of a clay core, filter, and transition zones, downstream and upstream shells constructed from waste rock and pit run sand, respectively. The clay core will be covered by a layer of frost protection material (sand) to insulate the core and reduce potential for degradation from freeze-thaw action. Rip-rap erosion protection will be provided over the exposed sand slopes of the frost protection and upstream shell. Wick drains will be installed in two zones below the downstream shell to accelerate the rate of the foundation glacio-lacustrine clay consolidation and strength gain. Drainage blankets located over the wick drain panels will be constructed from crushed waste rock. The dam will have a downstream slope of 12H:1V and an upstream slope 3H:1V. The dam will be progressively raised by the centreline method. The ultimate dam height varies from 9 m to 23 m.

The filter sand, frost protection, clay core, and upstream shell sand will be obtained from the open pit stripping activities. It is expected that obtaining filter sand of suitable gradation will require selective excavation and/or processing (e.g., screening). Selective excavation of clay core material will also be required to obtain material at a workable moisture content. Waste rock for the downstream shell, rip rap, drainage blankets, and transition granular will be sourced from the open pit. Crushing and/or screening of the waste rock to obtain transition and drainage blanket material of suitable gradation will be required.

Figure 18-9 shows the typical TMF perimeter dam cross-section.

**Figure 18-9: Typical TMF Cross-Section**



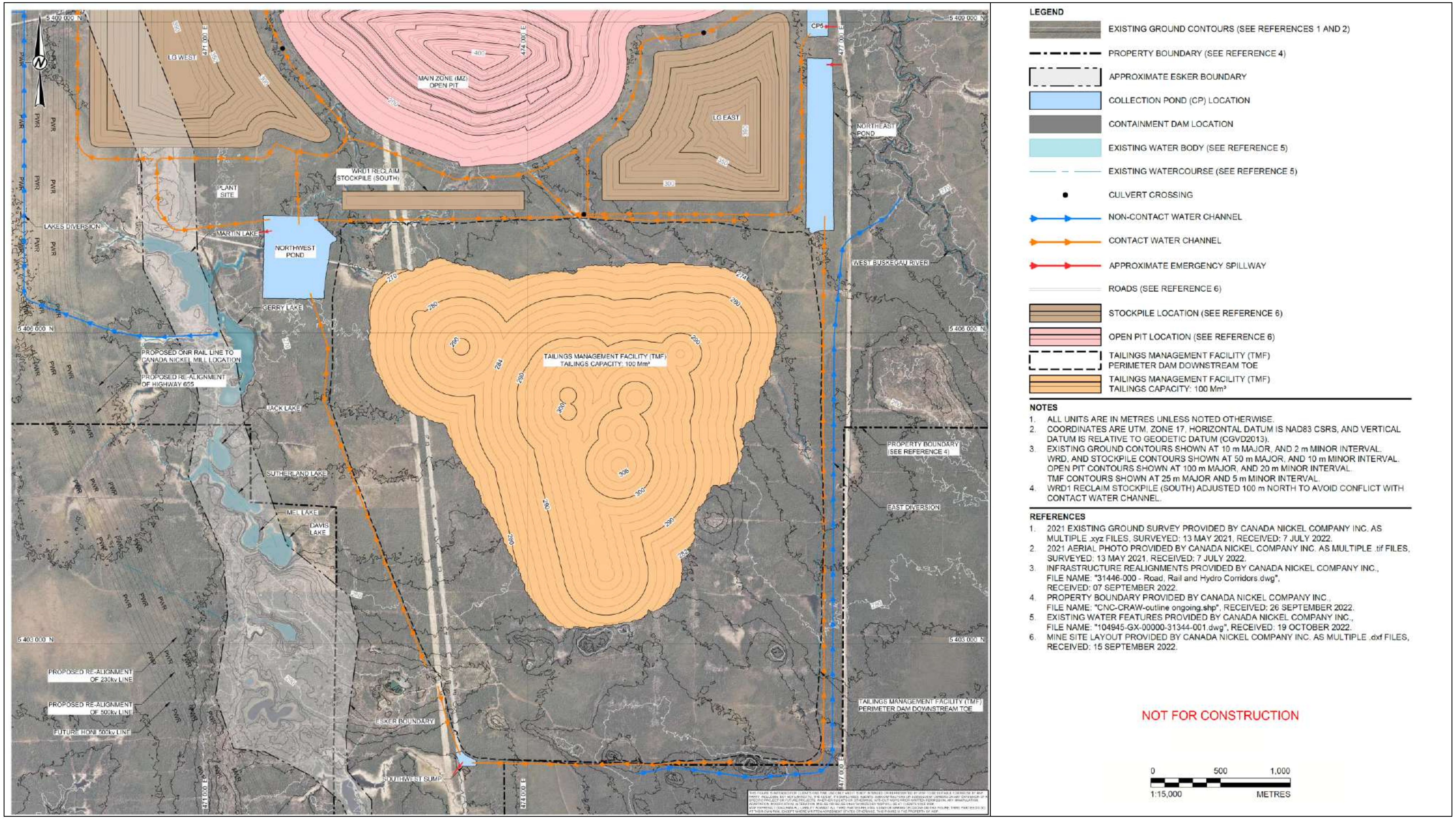

Source: WSP, 2023.

## 18.10.3 Tailings Deposition

Tailings will be thickened to approximately 39% solids in the mill and then pumped to the TMF where they will be discharged from multiple locations near the centre of the facility. The discharge locations and their elevations will vary throughout the life of the TMF to suit the tailings deposition plan.

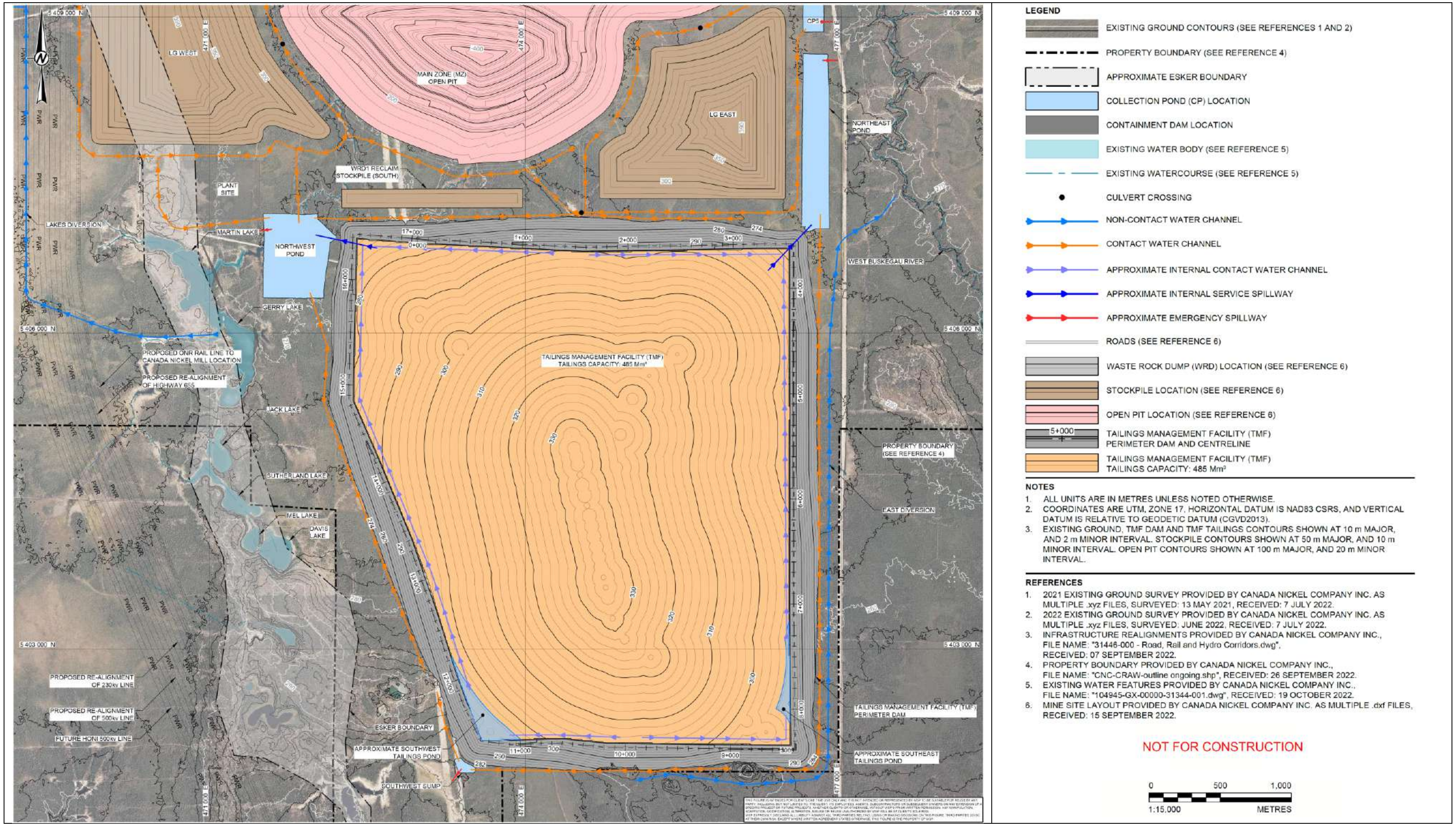
Currently, the TMF tailings deposition plan is divided into four stages: start-up, starter dam, 350 $Mm^3$ of containment, and 495 $Mm^3$ of containment. The start-up phase will accommodate 100 $Mm^3$ of tailings deposition within the central area of the TMF and will not require perimeter containment dams (though perimeter channels for contact water management will be in place). The starter dam phase refers to the configuration when the TMF dam has approximately 1.0 m of tailings impounded against the upstream slope. The proposed start-up and final deposition stages are shown in Figure 18-10 and Figure 18-11, respectively.

**Figure 18-10: Deposition Plan – 100 Mm³ of Tailings**




Source: WSP, 2023.

**Figure 18-11: Deposition Plan – End of TMF Operations**




Source: WSP, 2023.

### 18.10.4 Mine Closure Requirements – TMF

Once the tailings deposition is switched into the open pits, the TMF will be progressively closed. Geochemical testing carried out indicates that the tailings will not be acid generating or metal leaching.

The tailings surface will be revegetated to control erosion. The revegetation will involve the placement of a 0.15 m thick layer of growth media consisting of stripped overburden and/or organics. The growth media will be seeded with herbaceous plants. Erosion protected ditches and channels will be incorporated on the closed TMF surface to collect clean runoff water from the facility and to direct it to one or more external sedimentation ponds. After mining ceases, the TMF runoff will be directed into the open pits to accelerate pit flooding.

TMF water management seepage and runoff collection channels and ponds will be removed when water quality monitoring indicates that water can be discharged directly to the receiving environment and after the pit has been fully flooded.

### 18.10.5 Water Supply

The raw water makeup for the plant will be sourced from the TMF collection ponds and pit dewatering. The nominal raw water makeup flowrate will be approximately 504 m3/hr or 140 L/s. Pontoon-mounted pumps will pump the makeup water via HDPE pipeline to the raw water storage tank within the process plant.

### 18.10.6 Tailings Delivery System

Tailings from the process plant (after passing through the carbon sequestration system) will be pumped to the TMF through an HDPE pipeline. The tailings pipeline will run east from the process plant in a trench along an access road, to a booster pump station located near the secondary crusher building. The booster pumps will push the tailings south, into the TMF.

The HDPE distribution piping within the TMF will have spigots to distribute the tailings slurry, creating a deposition pattern to optimize space within the pond. Additional piping branches and spigots will be added throughout the life of mine as the TMF deposition is formed.

### 18.10.7 Return Water Delivery System

Return water from the TMF will run off into two ponds, the TMF northwest collection pond and the TMF northeast collection pond.

## 18.11 Site Water Management

### 18.11.1 Hydrologic Considerations

The monthly distribution of precipitation for the site was developed based on Environment Canada data for the Timmins A and Timmins Victor Power A stations, as follows:

- capture and attenuate contact runoff from disturbed areas including impoundment facility, stockpiles, and TMF in collection ponds.
- average yearly precipitation: 831 mm
- typical wet year precipitation: 1,057 mm.

Annual average lake evaporation was estimated based on the mean annual lake evaporation map from the Hydrology Atlas of Canada. The total annual lake evaporation for the project is 470 mm.

### 18.11.2 Site Water Management System

Water management requires consideration of the amounts of water (flows) between facilities and associated catchments areas. Surface water runoff that comes into contact with disturbed areas will need to be managed prior to being released to the surrounding environment. The general site-wide water management strategy is shown in Figure 18-12.

The overall water management strategy for the site aims to accomplish the following:

- capture and attenuate contact runoff from disturbed areas including the impoundment facility, stockpiles, and TMF in collection ponds,
- treat runoff to desired discharge criteria utilizing modular treatment plant near each pond,
- utilize pit dewatering and TMF collection ponds to provide freshwater make-up water for mill processing purposes.

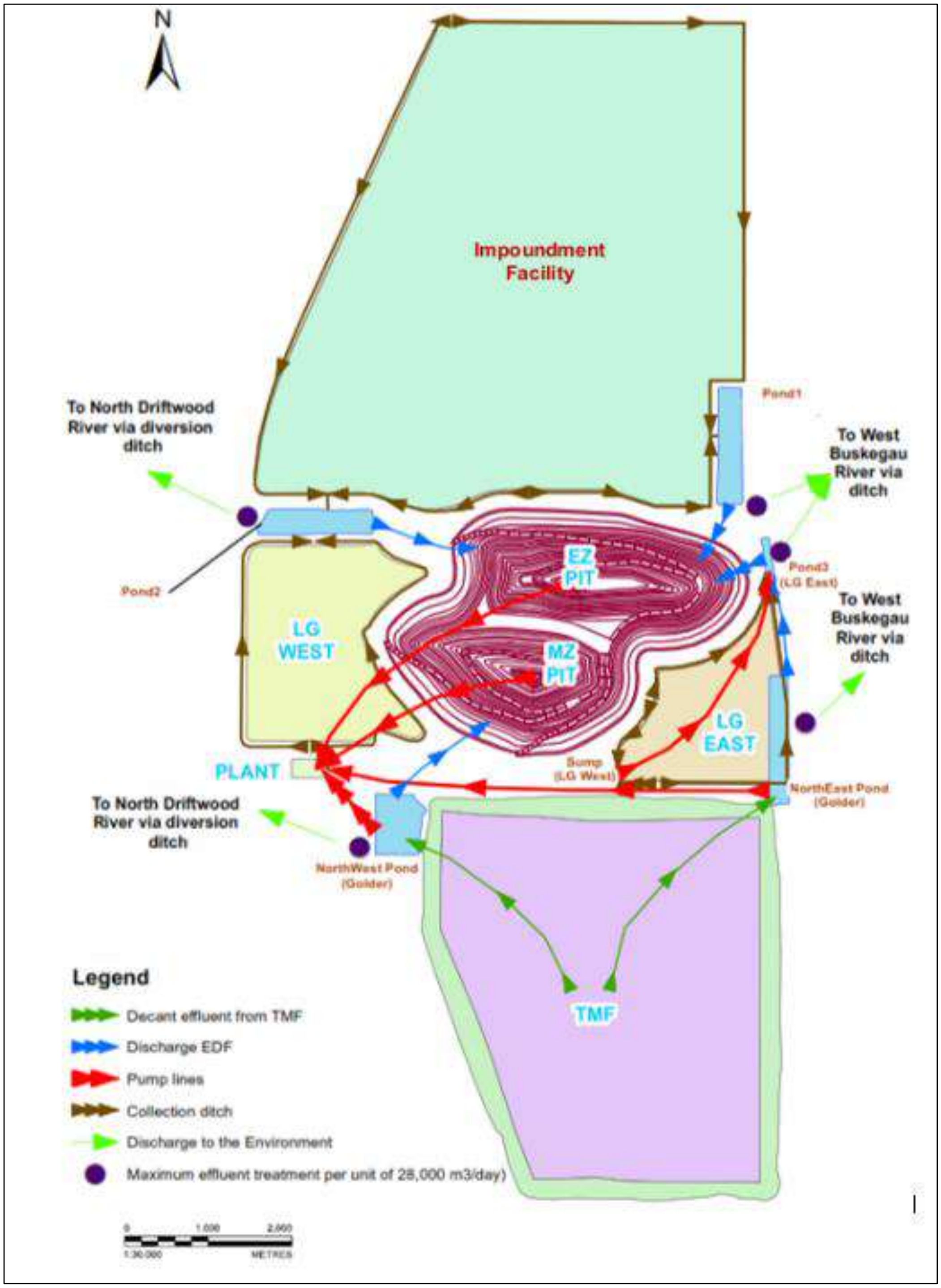
#### 18.11.2.1 Collection Ponds

Runoff from disturbed areas will be collected in gravity ditches and conveyed to ponds. The ponds have been designed as settling ponds with a permanent water depth to aid in removing total suspended solids (TSS) prior to discharge to the receiving environment. The required volume of the collection ponds was based on the storage required to attenuate the design inflows (1:10-year, 24-hour storm) and runoff from a mean average monthly precipitation, assuming release to the environment was limited to 28,000 $m^3$/d (i.e., the capacity of the modular treatment plant).

The design configuration of the collection ponds is shown in Table 18-4 and summarized below:

- Pond 1 will be constructed in conjunction with the east segment of the rock impoundment facility to manage runoff from that facility. It is envisioned that the pond will be constructed in stages and expanded to match the required runoff management as the rock impoundment facility footprint expands.
- Pond 2 will be constructed to manage runoff from the western segment of rock impoundment facility, once the highway is re-aligned. This pond will also serve to manage water from the western stockpile.
- Pond 3 will manage contact water from the eastern stockpile. Due to topographic constraints, capturing runoff from the entire stockpile into a single pond utilizing gravity is not feasible. An excavated sump at the southwest corner of the eastern stockpile will be required to capture runoff from a portion of the facility. A pumping system will convey accumulated runoff to pond 3 for treatment.

**Figure 18-12: Site-Wide Water Management Strategy Schematic**




Source: Ausenco, 2023.

**Table 18-4: Collection Pond Configurations**

| Pond | From Where Runoff is Received | Approx. Length (m) | Approx. Width (m) | Side-Slopes (H:V) | Depth (m) | Approx. Volume ($m^3$) | Forebay (Y or N) | Permanent Water Depth (m) |
|---|---|---|---|---|---|---|---|---|
| Pond 1 | WRD (East) | 1500 | 300 | 5:1 | 6 | 3.05 M | Yes | 2 |
| Pond 2 | WRD(West)<br>Western Stockpile | 1500 | 300 | 5:1 | 6 | 3.24 M | Yes | 2 |
| Pond 3 | Eastern Stockpile | 500 | 100 | 5:1 | 6 | 240 k | Yes | 2 |
| Pond 3 Sump | Eastern Stockpile | 200 | 80 | 5:1 | 5 | 80 k | No | 0 |

To facilitate cleanout and to promote settling, the ponds have been sized with a forebay. Forebays will be approximately 20% of the total permanent pool volume and will be separated from the rest of the ponds via a riprap lined berm and an overflow weir. All inflows to the pond will enter at the forebay and travel the length of the pond before being discharged to the environment.

At this stage, additional TSS treatment has been accounted for via modular treatment plants located at each pond. Barge pumps will be positioned near the far side of the pond, as far as possible from the inlet, and will draw water from the pond to the adjacent modular treatment plants before being released to the environment via an excavated gravity channel. If discharge criteria are achieved within the sediment ponds and additional treatment is not required, each pond will have a reverse sloped gravity outlet that can discharge directly to the excavated channel. Outlets will be fitted with a valve to control or stop the release if there is an upset condition or discharged needs to be routed to the treatment modular.

#### 18.11.2.2 Environmental Design Flood Management

The environmental design flood (EDF) will be managed by conveying excess flows from the sediment ponds to the pits for temporary storage via an overflow spillway and excavated channel. The adopted EDF was the 1:100-year, 24-hour event.

#### 18.11.2.3 Contact Water Collection Ditches

The impoundment facility and western and eastern stockpiles will have perimeter ditching to capture and convey runoff to sediment ponds. At this stage, based on the topographical information, a shallow slope ditch is feasible; however, given the length of the channels and the flat terrain, the northern segment of the impoundment collection system will require a combination of berming, and excavated channel. For the eastern stockpile, a portion of the runoff will be captured in an excavated sump and be pumped to Pond 3. Ditches have been sized to capture and convey the EDF.

#### 18.11.2.4 Non-Contact Diversion Channel and Berms

There are a number of small lakes within the site property that drain through proposed mine infrastructure. Non-contact diversions will be required to divert these flows away from mining activities. An excavated channel is proposed as a new outlet for Martin Lake to divert flow along the re-aligned highway and tie back into North Driftwood River further downstream. A diversion berm will also be required to cut off the existing outlet of the lake which flows through the proposed TMF northwest pond and the pits. Similarly, a diversion will be required on Gerry Lake to redirect flow.

### 18.11.3 Site-Wide Water Balance

A site-wide water balance (Goldsim) was developed based on the conceptual water strategy to quantify the amount of runoff generated from disturbed areas and inform water management infrastructure design and pumping rates. The climatic model considered deterministic scenarios using precipitation for average, wet, and dry years.

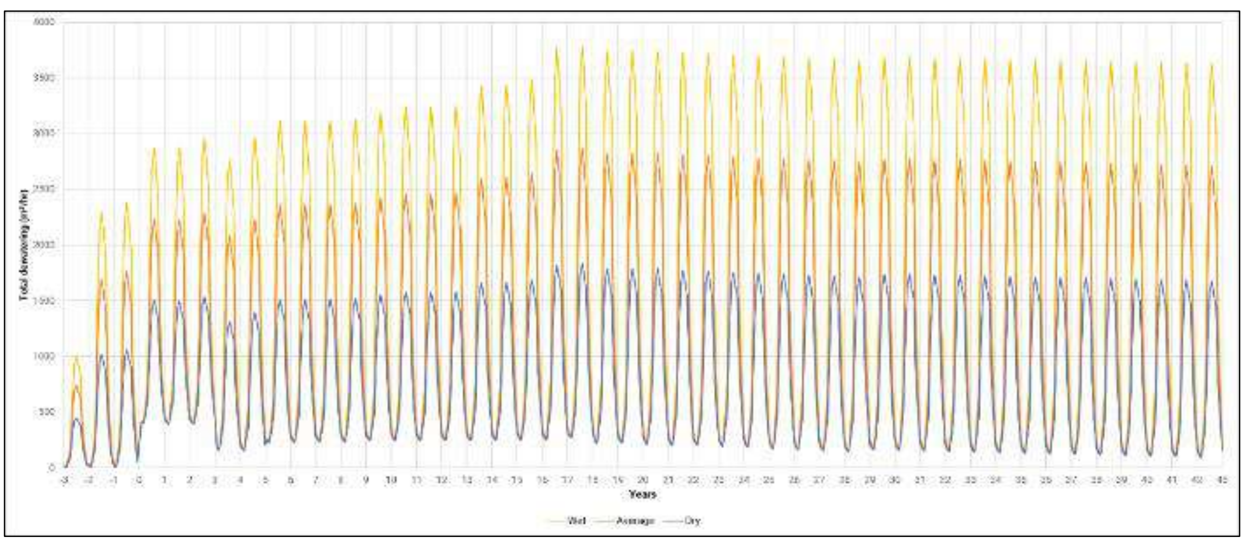
#### 18.11.3.1 Pit Dewatering

Total pit dewatering was estimated considering runoff and seepage as inflows and dust suppression as a demand. Figure 18-13 shows monthly variations of the total pit dewatering, and Table 18-5 below shows results for average conditions.

**Table 18-5: Total Pit Dewatering**

| Facility | Total Pit Dewatering ($m^3/h$) | | |
|---|---|---|---|
| | Average Year | Wet Year | Dry Year |
| Average | 1,205 | 1,553 | 813 |
| Monthly Maximum | 2,874 | 3,790 | 1,838 |

**Figure 18-13: Total Pit Dewatering**



Source: Ausenco, 2023.

### 18.11.3.2 Collection Ponds

Runoff for the different mine facilities were estimated using runoff coefficient according to mine facilities characteristics. Table 18-6 shows the estimated treatment rates for each pond under the different climatic conditions assessed and Table 18-7 shows the inflow to each pond for the 1:10-year, 24-hour design storms as well as the anticipated storage requirements during winter operations and the EDF event.

**Table 18-6: Water Treated Flow Rates: Collection Ponds**

| Description | Water Treated Flow Rate ($m^3$/d) | | |
|---|---|---|---|
| | Pond 1 | Pond 2 | Pond 3 |
| Average Precipitation | 17,821 | 13,915 | 2,817 |
| Average Wet Month[1] | 25,854 | 23,406 | 5,070 |
| Average Dry Month[1] | 9,788 | 4,424 | 563 |
| Monthly maximum | 28,000 | 28,000 | 7,358 |

Note: 1. Average is based on the climate normal for wet/dry years. Source: Ausenco, 2023.

**Table 18-7: Inflows and EDF Storage Requirements – Collection Ponds**

| Description | Pond 1 | Pond 2 | Pond 3 |
|---|---|---|---|
| Winter Operations (Average)[1] | 0.48 $Mm^3$ | 0.33 $Mm^3$ | 0.05 $Mm^3$ |
| 1:10-Year, 24-Hour Storm[2] | 6.2 $m^3$/s | 10.9 $m^3$/s | 5.0 $m^3$/s |
| EDF Pit Storage | 1.20 $Mm^3$ | 1.22 $Mm^3$ | 0.07 $Mm^3$ |

Notes: 1. Would be stored in the ponds until the spring. 2. Peak flows, which would be stored and attenuated at each pond.

### 18.11.3.3 TMF Ponds and Water Balance

An internal water balance for the process plant was developed separately as part of the project. For the site-wide water balance, process plant inflows and outflows have been considered constant over time. These are listed in Table 18-8.

**Table 18-8: Process Plant Inflows and Outflows**

| Description | Unit | Phase 1 | Phase 2 |
|---|---|---|---|
| Tailings Production | kt/d | 60 | 120 |
| Water in Tailings to TMF | $m^3$/h | 3,812 | 7,623 |
| TMF Decant Water | $m^3$/h | 2,752 | 5,503 |

TMF collection ponds were designed by WSP and are reported separately within the document. Both ponds will supply decanted water to the process plant and excess inflows will be treated at modular unit plants. It was assumed that each pond will receive 50% of flows from the TMF (i.e., decant water and runoff). Excess pit dewatering flows (after dust suppressant and make-up water requirements) will be conveyed to NW pond for treatment. In months when the anticipated inflows exceed the storage and treatment capacity of the ponds, flows will be directing via a gravity channel to the pits for temporary storage. Under average precipitation conditions, it is estimated that the pits will temporarily store water for three months. The design configuration of the TMF collection ponds are shown in Table 18-9.

**Table 18-9: TMF Collection Pond Configurations**

| Pond | From Where Runoff is Received | Approx. Length (m) | Approx. Width (m) | Side-Slopes (H:V) | Depth (m) | Approx. Volume ($m^3$) | Permanent Water Depth (m) |
|---|---|---|---|---|---|---|---|
| Northwest Pond | TMF | 700 | 600 | 6:1 | 6.8 | 2.8 M | No |
| Northeast Pond | TMF | 1400 | 200 | 6:1 | 6.8 | 1.9 M | No |

Table 18-10 shows the estimated treatment rates for each TMF pond under the different climatic conditions assessed and Table 18-11 shows the inflow to each pond for the 1:10-year, 24-hour design storms as well as the anticipated storage requirements during winter operations and the EDF event.

**Table 18-10: Water Treated Flow Rates – TMF Ponds**

| Facility | Water Treated Flow Rate ($m^3$/day) | |
|---|---|---|
| | NW Pond | NE Pond |
| Average | 18,737 | 17,088 |
| Average Wet Month[1] | 27,662 | 25,509 |
| Average Dry Month[1] | 9,812 | 8,666 |
| Monthly maximum | 28,000 | 28,000 |

Note: 1. Average is based on the climate normal for wet/dry years. Source: Ausenco, 2023.

**Table 18-11: Inflows and EDF Storage Requirements – TMF Ponds**

| Facility | NW Pond | NE Pond |
|---|---|---|
| Winter Operations (Average)[1] | 0.43 $Mm^3$ | 0.40 $Mm^3$ |
| 1:10-Year, 24-Hour Storm[2] | 30.1 $m^3/s$ | 30.1 $m^3/s$ |
| EDF Pit Storage | 0.93 $Mm^3$ | 0.93 $Mm^3$ |

Notes: 1. Would be stored in the ponds until the spring. 2. Peak flows, which would be stored and attenuated at each pond.

### 18.11.4 Water Treatment Plant

Based on the geochemical characterization, certain primary constituents of concern exceed Provincial Water Quality Objectives (PWQO), need further treatment before being discharged to the receiving environment. The potential need for treatment is compounded by the fact that the overburden is predominately clay, which will be stored as part of the impoundment facility. There is limited information on the anticipated sediment loading or sediment characteristics (i.e., particle size, settling time). It has been assumed that TSS removal within a traditional settling pond may not be sufficient to meet discharge criteria and additional water treatment may be required. As previously noted in Section 18.11.2, the site-wide water management strategy accounts for additional TSS removal by modular treatment plants. At this stage, it has been assumed that electric treatment plants will be utilized with a nameplate treatment capacity of 28,000 $m^3$/d.

## 18.12 Water Supply & Distribution

### 18.12.1 Potable Water Supply

The potable water for the plant will be supplied by groundwater sourced from water wells located west of the process plant. The nominal potable water flowrate will be approximately 3.33 $m^3$/hr or 80,000 L/d. Pontoon mounted pumps will transfer the water via HDPE pipeline to the freshwater tank within the process plant. The fresh water will be treated by a reverse osmosis system, to meet provincial drinking water standards, and then report to the potable water storage tank. Dedicated potable water pumps will distribute the water to safety showers and other points in the process plant facilities. The distribution piping will be heat traced and insulated wherever it is not inside a heated building.

### 18.12.2 Fire Water Services

Surface infrastructure buildings will have a fire suppression system in accordance with the structure’s function. These fire detection and protection systems will generally consist of smoke detectors, manual pull stations, alarm strobes/sounders, portable fire extinguishers, and hose stations. Automated fire sprinkler systems will be provided for critical areas such as the primary crushing, secondary crushing, and grinding area, in accordance with insurer requirements.

The process plant area will have a fire reticulation system consisting of fire water storage, a fire water pump station (including a backup diesel-powered pump), and a fire water main around the perimeter of the area. Electrical and control rooms will be equipped with dry-type fire extinguishers. For the reagents area within the process plant building, appropriate fire suppression systems will be included according to their material safety datasheets.

## 18.13 Construction Camps

As the project is adjacent to the greater Timmins municipality (43 km north of the City of Timmins), construction contractors and suppliers were directed during the budgetary quotation process to include costs for housing their own workforce off the project site (as this is typically preferred over residing in a project controlled camp facility). Historically in the area independent accommodation providers have followed the trends of demand and provided accommodations for temporary workers when required for large projects. During the execution phase, this principle will continue to be put in place with potential project participants.

# 19 MARKET STUDIES AND CONTRACTS

## 19.1 Introduction

The following information is summarized from information provided by CNC and listed in Section 3 and 27.

## 19.2 Price Assumptions

The price and payability assumptions for various payable metals that will be produced by Crawford are based on aggregated forecasts by analysts and other market participants as of August 2023. The first full year of production from Crawford is planned for 2028, which falls within the long-term period of these forecasts. For this reason, a single price, expressed in 2023 real terms, has been used for all years.

Table 19-1 summarizes the assumed pricing and payability of each metal. Commentary on these assumptions is provided below.

**Table 19-1: Pricing Assumptions (2023 Long-Term Real Basis)**

| Metal | Units | Long-Term | Payability | Source |
|---|---|---|---|---|
| Ni Concentrate | US$/t ($/lb) | 21,000 (9.53) | 91% | Fastmarkets, company estimate |
| Ni-Fe Concentrate | US$/t ($/lb) | 21,000 (9.53) | 91% | Fastmarkets, company estimate |
| Iron Scrap | US$/t | 325 | 50% | Fastmarkets, company estimate |
| Chromium | US$/lb | 1.75 | 65% | Fastmarkets |
| Cobalt | US$/t ($/lb) | 40,000 (18.14) | 60% | Analyst consensus[1] |
| Platinum[3] | US$/oz | 1,150 | 1 g/t deduction | Analyst consensus[2] |
| Palladium[3] | US$/oz | 1,350 | 1 g/t deduction | Analyst consensus[2] |

Notes: 1. Aggregate of 14 analyst estimates, dated August 2023. 2. Aggregate of 17 analyst estimates, dated August 2023. 3. Payability based on 1 g/t combined Pt + Pd deduction. Resultant life-of-mine average payability is 76.2% and 75.2% for Pt and Pd, respectively.

The long-term nickel price assumption of $21,000 per tonne is lower than the forecast provided by Fastmarkets of ($23,000) per tonne. While CNC believes in the longer-term potential of the nickel market, particularly for low carbon supply in North America, it believes that this is a prudent forecast to be used in this feasibility study.

Feeds for use in the stainless and alloy steel industry are priced on their iron and chromium content. The benchmark for the iron used in stainless steel feed is #1 heavy melting scrap price. CNC is utilizing $325 per tonne, which it believes is a prudent forecast to be utilized in this feasibility study. This is lower than the Fastmarkets forecast of $450 per tonne, which is at the upper end of the recent range ($289 to $510 per tonne). Current prices are due to increasing demand for low carbon sources of raw material like iron scrap as the entire steel industry shifts away from high carbon blast furnace production towards lower carbon processes utilizing electric arc furnaces.

The chromium benchmark in North America is the US ferrochrome price. CNC is utilizing $1.75 per pound, which is line with the study by Fastmarkets. This is the middle of its recent range since the beginning of 2021 of $0.90 to $3.90.

The metal price assumption for cobalt of $40,000 per tonne is lower than the forecast long-term consensus average price of 14 analysts published by Scotiabank Capital Markets in August 2023 of $54,700 per tonne. CNC believes that analysts are underestimating the impact on the cobalt market from the ramp-up of high-pressure acid leach (HPAL) production in Indonesia over the coming decade combined with the push from batterymakers to maximize the use of nickel and minimize the use of cobalt in their lithium-nickel-manganese-cobalt oxide (NMC) battery chemistries.

Assumed prices per ounce for platinum and palladium of $1,150 and $1,350, respectively, are consistent with the forecast long-term consensus average prices of 17 analysts published by Scotiabank Capital Markets in August 2023 of $1,145 per ounce for platinum and $1,390 per ounce for palladium.

Sensitivities for these pricing assumptions are provided in Section 22.

## 19.3 Market Outlook

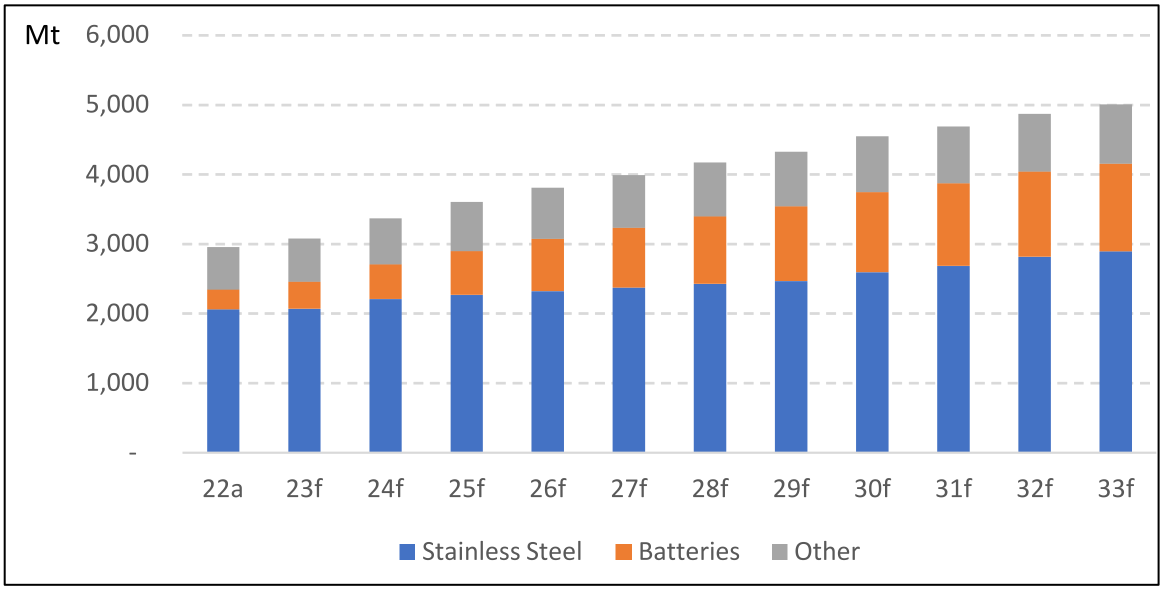
### 19.3.1 Nickel Market Outlook

According to the long-term outlook provided by Fastmarkets in September 2023, primary nickel demand will rise at a CAGR of 4.9% between 2022 and 2033, driven by growth in the stainless steel and battery sectors. Demand for primary nickel in batteries will grow at a CAGR of 14.2% between 2022 and 2033, consolidating its position as a strong second largest consumer of primary nickel after the stainless-steel industry.

According to Fastmarkets, demand from the stainless-steel sector will grow at a CAGR of 3.2% over the forecast period. Given the sector's size, this represents an additional 838 kt or nickel required by 2033. Other sectors are forecast to grow at a CAGR of 3%, representing a further increase of 237 kt of nickel by 2033 (Figure 19-1).

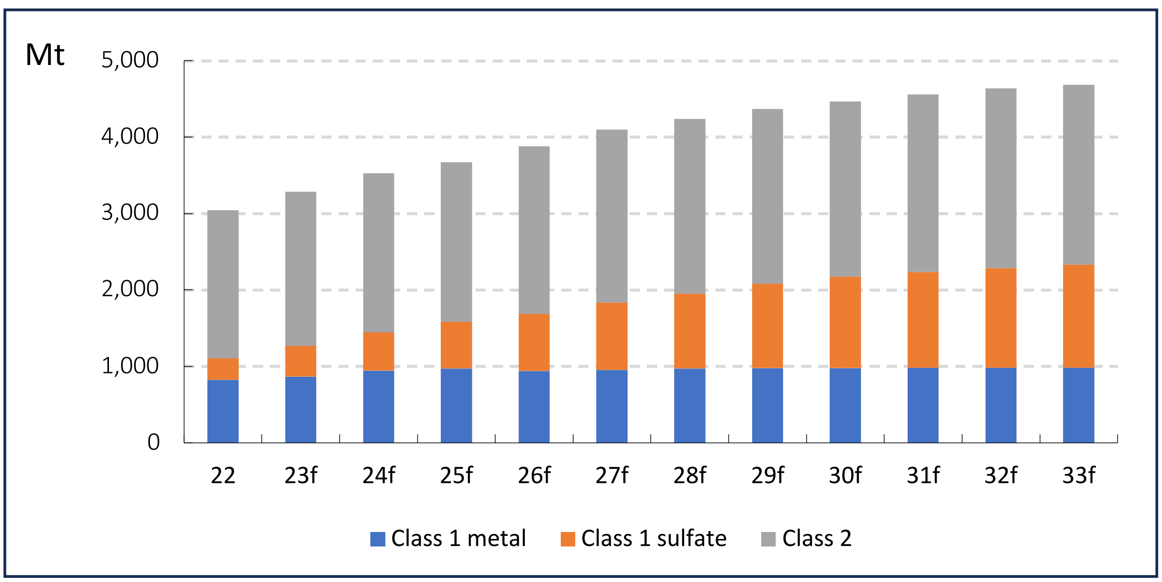
According to Fastmarkets, the refined nickel supply will grow at a CAGR of 4% between 2022 and 2033 (Figure 19-2). The Chinese refined supply will grow at a CAGR of 6.1% to 1.6 Mt over that same period. Despite a lower growth rate of 5.3%, Indonesia will remain the top producer with a forecast supply of 2.1 Mt. The importance of Indonesia as a primary supply of metal is underscored by the rise in Chinese refined production being highly reliant on Indonesian intermediate supply, whether matte or MHP.

**Figure 19-1: Nickel Demand**




Source: Fastmarkets, 2023.

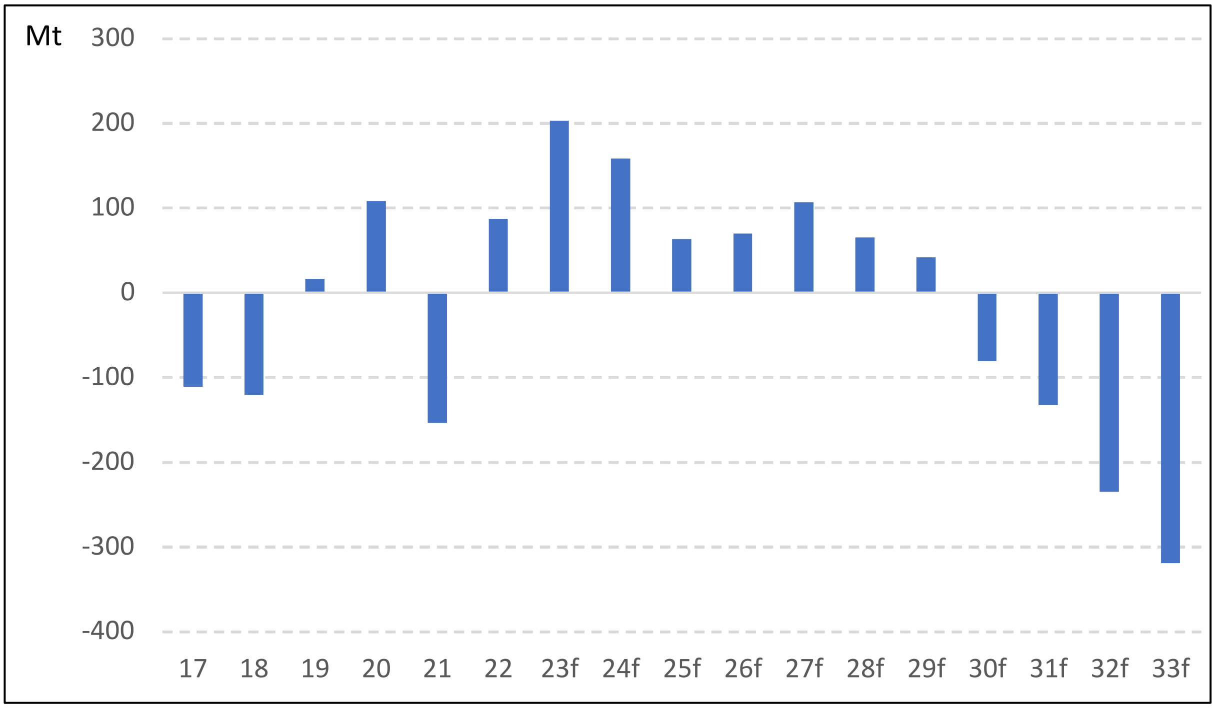
**Figure 19-2: Nickel Supply**




Source: Fastmarkets, 2023.

According to Fastmarkets, supply will keep pace with demand in the short to medium term, leading to an oversupplied market. In the longer term, a lack of new projects will result in deficits beginning in 2030 and widening to in excess of 300 kt annually by 2033 (Figure 19-3)

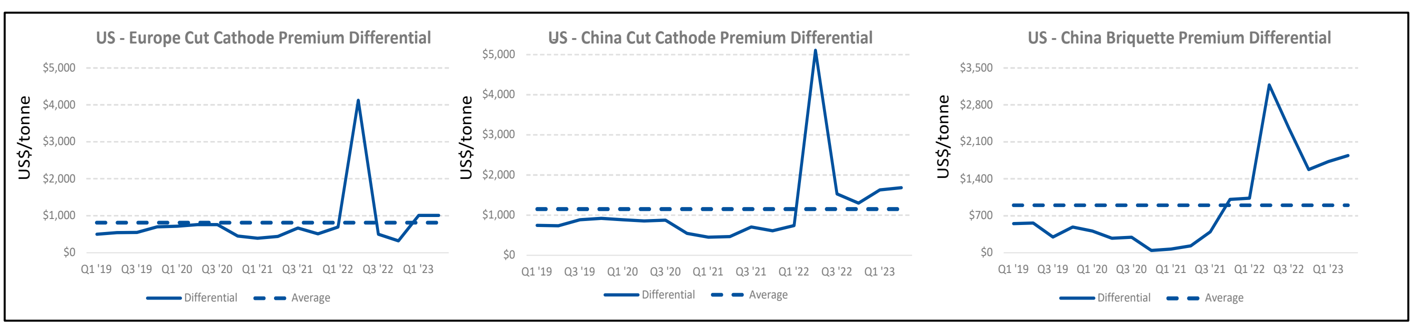
**Figure 19-3: Nickel Market Balance**




Source: Fastmarkets, 2023.

CNC notes that the United States is fully dependent on imports of finished nickel products. As a result, CRU reports that in 2022, premia for various nickel products in the US market range up to $2,000 per tonne higher than for the same products in Asian or European markets.

**Figure 19-4: Nickel Product Premia Differentials**




Source CRU Nickel Monitor, August 2023.

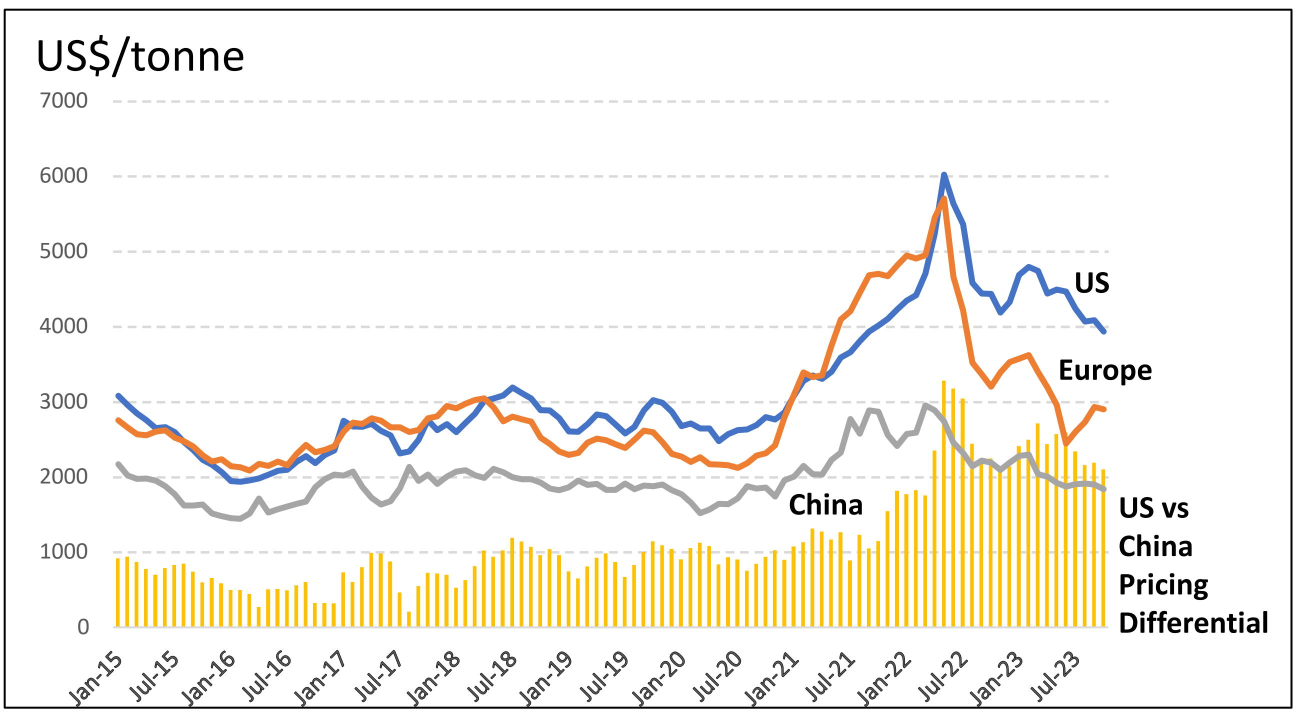
### 19.3.2 Stainless Steel Market Outlook

Steel & Metals Market Research GmbH (SMR), a global leader in providing market intelligence for the global speciality and stainless-steel industry, estimates that North American stainless-steel consumption in 2023 will be 2.7 Mt, with 2.1 Mt supplied by domestic producers and the remaining 0.6 Mt imported. These imports occur despite significant import restrictions (section 232 and various dumping duties). SMR expects a further 0.8 Mt of growth in North American stainless consumption by 2030, driven by a combination of stainless steel continuing to expand market share from other materials and re-shoring of demand from Asia.

SMR believes that the combination of significant reliance on imports and future supply growth provides an opportunity for a new stainless steel meltshop on the North American continent. This is particularly so given pressures faced by existing European meltshops due to uncertainty caused by regional high and volatile energy prices. While there are many factors which determine the optimal location for a meltshop, SMR believes that a location in the Timmins, Ontario, region offers significant benefits due to lower cost, low carbon grid electricity, substantial existing infrastructure, the potential to store any $CO_2$ generated by the meltshop in the Crawford tailings, and significant government funding support. Furthermore, proximity to raw material supply mimics the approach of some Chinese stainless-steel mills.

Because North America is a net importer of stainless steel and has applied various import restrictions, stainless steel prices continue to trade at significant premiums to other markets (Figure 19-5). This creates opportunities for new meltshop capacity in North America.

**Figure 19-5: CRU Stainless Steel Prices**




Source: CRU, Stainless Steel Flat Products Monitor, August 2023.

#### 19.3.3 Scrap Iron Market Outlook

According to the long-term outlook by Fastmarkets (September 2023), iron scrap prices are forecast to trend upwards over the next ten years in the key North American markets of Cleveland, Pittsburgh, and Montreal. This trend will be primarily by the increased demand for domestic scrap attributed to significant electric arc furnace (EAF) capacity expansions within North America as well as globally. The long-term real (2023 basis) prices are expected to average $545-$605 per tonne

According to Fastmarkets, notable projects in the United States by industry leaders such as Nucor, SDI, and US Steel, along with Canada's Algoma and Arcelor Mittal Dofasco's shift towards EAF production, are key contributors to this trend. This will support domestic scrap consumption in both Canada and the US. Indeed, scrap consumption is forecast to rise by 23% over the next ten years, according to Fastmarkets projections. Rising demand in major export markets for North American obsolete scrap will also support scrap prices. In the US, scrap exports are an important outlet, creating a floor for prices as overseas buyers take excess supplies from the market which would otherwise depress prices.

According to Fastmarkets, North American crude steel production is expected to increase at a rate of just over 1% per year (CAGR) between 2022 and 2032, representing an increase of more than 12 Mt over the period. These gains represent the additional EAF production planned for the period. We also note that a portion of the new steel-making capacity will displace old existing capacity which is mainly blast furnace / basic oxygen furnace (BF/BOF) method. A substantial portion of the EAF capacity expansions will be focused on producing HR coil, requiring low residual scrap and metallics such as prime-grade scrap.

According to Fastmarkets, prime scrap generation, however, is not expected to increase at the same rate as regional scrap consumption in the next decade. This shortfall is reflected in car and commercial vehicle production forecasts—a key source of prime scrap generation. Oxford Economics expect output in the US and Canada to rise by 15% and 9% over the next ten years, respectively. Moreover, the automotive sector's efforts to reduce prime scrap wastage as well as light-weighting vehicles to reduce steel usage will result in reduced flows of prime scrap per unit produced. The tight prime scrap supply will place additional pressure on obsolete scrap to meet the rising demand.

#### 19.3.4 Chromium Market Outlook

Ferrochrome prices are expected to decline from current levels in 2023 to 2024 towards a long-term real (2023 basis) price of $1.75 and remain stable well above average prices in the 2010 to 2021 period. Much of this is based on the cost outlook. Prices of high-carbon ferrochrome are understood to have historically moved very closely in line with costs of production given the relatively oversupplied nature of the market. The costs of production for high-carbon ferrochrome are largely accounted for by chrome ore, reductants, and energy.

Beyond 2023, prices will be rangebound in a narrow band as we expect little influence on prices to materialize and shock to the upside nor downside. After 2023, coke prices are expected to moderate on overall reduced demand from the steel industry’s efforts to decarbonize. Reduced supply in the global market will provide a level of support to prices, however, over the longer term. Rising electricity costs in South Africa are a key threat to the industry there. Electricity prices have risen by an average of 8% to 9% per year since 2009. They rose by 16% in 2021 and by another 10% in 2022.

## 19.4 Concentrate Marketing

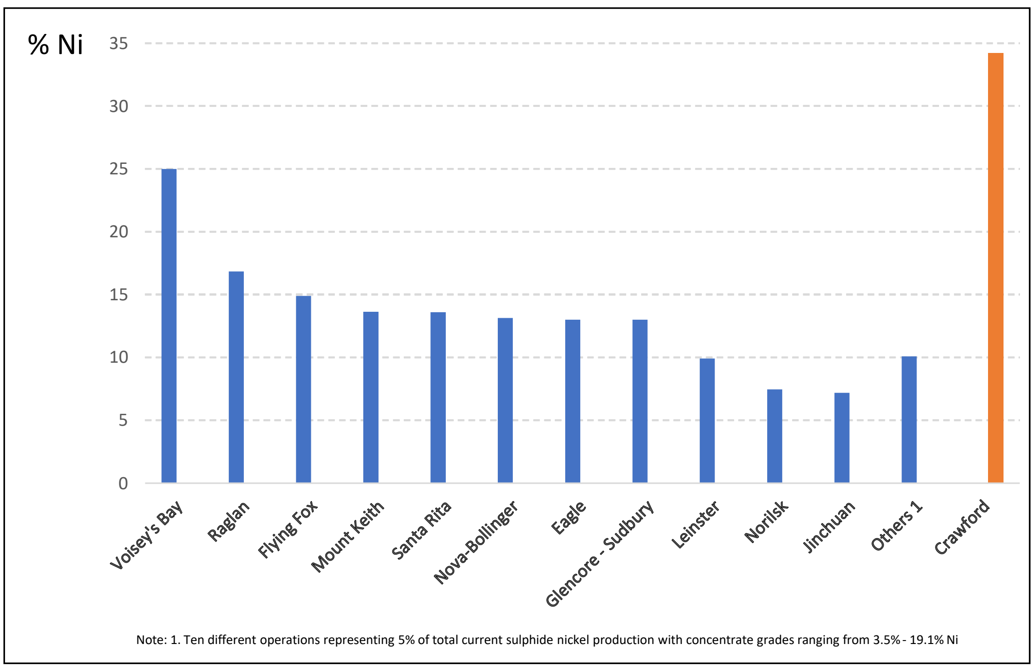
### 19.4.1 Nickel Concentrate

The Crawford nickel concentrate, with an average nickel content of 34% nickel over the life of project, is suited for a wide range of downstream processes including traditional smelting and refining, and the concentrate roasting/reduction approach successfully developed by the CNC management team for concentrate produced by the Dumont project and implemented in 2014 by Tsingshan.

This roasting/ reduction downstream process could be utilized by FeNi/NPI/matte laterite processing operations which utilize an electric furnace for primary reduction to process feed from Crawford, Dumont, and other nickel concentrates.

Figure 19-6 was drawn from Wood Mackenzie's database of current operations and represents more than 80% of current sulphide nickel production. The production weighted average concentrate grade of these operations is 10.5% Ni. Crawford's average grade of 34% will be more than three times higher and will results in lower costs per pound of nickel contained in concentrate, particularly in the smelting process where costs are largely driven per tonne of feed.

**Figure 19-6: Concentrate Grade of Select Current Operations**




Source: Wood Mackenzie, 2023.

The only deleterious element of note in the concentrate is MgO which averages 11% and ranges between 7% and 12%. For several processing options, there is little to no impact of this MgO content. The specific impact of MgO content on each processing option is discussed below.

Nickel payability for nickel concentrate across the range of processing options ranges from 70% to 91%. In general, the nickel concentrate processing market can be broken down into three broad segments: Asian smelters, North American/European smelters, and the roasting-reduction process.

Asian smelters, with the most significant facilities based in Western Australia and Gansu, China, are primarily based on flash furnace technology and are located significantly inland, further than 1000 km from major nickel-consuming regions. The primary feed sources for each of those smelters also contain higher amounts of MgO, limiting the abilities of these flash furnaces (which are less able to tolerate higher MgO feeds) to acquire third-party feeds which may have higher MgO content. These Asian processing facilities also face the market challenge of lower finished product premiums than are available in North America and Europe (Figure 19-4). The combination of the logistics challenges, the lower market premiums, and limits on MgO result in payabilities from these smelters for third-party concentrates in the 70% to 80% range. Given the availability of other higher value concentrate processing alternatives, these smelters have not been included in the analysis.

North American and European smelters (two of which are located in Canada and one in Finland) utilize both electric and flash furnace technology. Electric furnaces are generally able to tolerate much higher levels of MgO than a flash furnace. The flash furnace in Sudbury operated by Vale has a primary low MgO feed and is understood to be able to accommodate some quantity of higher MgO third-party feeds.

Specific processing terms for a nickel sulphide concentrate processed by both Canadian smelters was provided in a technical report for Lundin's Eagle Mine (Clarke et al., 2022). This nickel concentrate, grading 12% nickel, receives an 81% payability at a $21,000 LME nickel price. There was also an MgO penalty of $2.50 per tonne of concentrate for each percentage point of MgO content in excess of 6%. This payability is in line with the 75% to 85% payability expected from these facilities for typical nickel grade concentrates (10% to 15% nickel). Payability is higher is higher than in Asia because of a combination of higher finished product premiums (Figure 19-4) and lower logistics costs due to proximity to major nickel consumers.

Utilizing this benchmark and adjusting for the nickel grade of the Crawford concentrate (34% versus 12% benchmark concentrate) and incorporating the MgO penalty (average of 11% incurring $12.50 penalty per tonne of concentrate), it is possible to calculate a nickel payability percentage in the mid 80s / low 90s for the Crawford concentrate.

The roasting-reduction process developed by CNC management for the Dumont Project (with a similar nickel concentrate) and implemented by Tsingshan, allows the utilization of a nickel concentrate in any electric furnace operation once the concentrate has been roasted (see Section 19.4.1.4 for more information).

Given the high-grade nature of the Crawford concentrate (average of 34%), the potential to utilize a roasting/reduction process, and the finished product premiums available in the US market equal to an additional 2.5% to 5% of payability, this feasibility study assumes a payability of 91% and a cobalt payability of 60%. These levels of payability are in line with the proposed terms from potential downstream JV partners.

There are various nickel processing options globally. Brief profiles of the most likely processing options are provided in the subsections below.

#### 19.4.1.1 Glencore

The Glencore smelter located in the Sudbury region currently treats concentrates from Glencore's mining operations in Sudbury and Quebec, as well as from third parties. The smelter uses electric furnace technology, which is more suitable for treating concentrates containing elevated levels of MgO than facilities utilizing flash smelting technology. The facility is not filled by Glencore's existing sulphide mines and is understood to treat a wide variety of third-party feeds with an overall capacity of approximately 450 kt of feed. As such, it should be possible to treat Crawford concentrate at the Glencore smelter without exceeding MgO limits. Matte produced by the smelter is shipped to the Nikkelverk refinery in Norway. Overall cobalt recovery through the smelter and refinery is approximately 70%.

#### 19.4.1.2 Vale

Vale's main smelter is in the Sudbury region. The smelter uses flash smelting technology, which is less suitable for treating concentrates containing elevated levels of MgO. Vale's own Sudbury basin mines do not currently fill the facility capacity and typically produce concentrates with low MgO, and the large capacity of the facility (an estimated 1 Mt/a feed) coupled with the high nickel grade of Crawford concentrate would result in concentrates from Crawford representing a small portion of the total feed tonnage. As a result, it should be possible to treat Crawford concentrate at Vale's main smelter without exceeding MgO limits.

#### 19.4.1.3 Boliden

Boliden currently operates the Harjavalta smelter in Finland utilizing the DON smelting process. Harjavalta is part of a polymetallic complex that treats separate copper and nickel concentrates. The Harjavalta smelter was recently expanded to process 370 kt/a of nickel feeds and only a fraction of that capacity is utilized by Boliden's Kevitsa mine. The smelter also treats a wide range of third-party feeds. The smelter process can accommodate some quantity of MgO-bearing concentrates. The complex achieves high recoveries for cobalt.

#### 19.4.1.4 Roasting Reduction (Tsinghsan) Process

Company management was involved in the initial demonstration the potential of roasted nickel concentrate as a more valuable alternative to traditional smelting and refining in 2011 and worked with Tsingshan in 2012 to validate the concept. In 2014, Tsingshan began constructing the first plant to directly utilize nickel sulphide concentrate as part of the stainless steel-making process and has since built an additional plant utilizing the roasted nickel concentrate approach. The production from nickel pig iron operations like these Tsingshan operations have subsequently been modified to produce matte, allowing nickel and cobalt to be fed directly to the battery supply chain.

In the Dumont Feasibility Study published in 2019 (Staples et al., 2019), the study reported that work with a large Japanese trading house indicates that roasters in Asia can process feed at an approximate cost of $30 per tonne. When combined with the average integrated NPI/stainless conversion cost of approximately $80 to $90 per tonne, (according to Wood Mackenzie), the implied conversion cost is approximately $110 to $120 per tonne of concentrate (equivalent to approximately $350 per tonne of contained nickel for a 34% concentrate or approximately 2% of the assumed long-term LME price of $21,000 per tonne). The Dumont feasibility study utilized a nickel payability of 91.5%. Based on recent Wood Mackenzie data, the average NPI/stainless conversion cost has increased to a range of $85 to $110, which would translate to an implied conversion cost of approximately $115 to $140 or $340 to $410, which is still approximately 2% of the assumed long-term LME nickel price of $21,000 per tonne.

### 19.4.2 Magnetite Concentrate

The magnetite concentrate produced by CNC will have average grades of 55% Fe, 2.6% Cr, and 0.26% Ni. The chromium contained in this concentrate would make Crawford the sole miner of chromium in North America (a listed critical mineral in Canada and the United States). The average Phase 2 production rate of 75 kt/a equates to 15% of total North American primary consumption and would thus make Crawford a material source of chromium for North America.

As iron, nickel, and chromium comprise the key elements in 300 series stainless steel and a range of alloy steels, all three elements in this concentrate are valuable to be utilized as feed for these products and could be utilized by a wide range of production facilities across North America as well as the additional meltshop capacity that SMR indicated would be required. Iron and chromium feeds for stainless steel in North America are priced using #1 heavy melting iron scrap price and US ferrochromium price.

At the assumed payability and iron prices in the feasibility study of 50% of the iron content and assumed long-term iron scrap price, the iron ore-equivalent price of this concentrate would be $89 per tonne, which is above the long-term consensus price of $83 per tonne (62% iron ore based on average of 16 analysts). This additional value reflects the additional byproduct value from the contained nickel and chrome (additional incremental value of nearly $155 per tonne of magnetite concentrate at a long-term price of $21,000 per tonne nickel and $1.75 per pound chromium), the very low phosphorous levels relative to standard iron ore concentrates, and the potential to utilize the Crawford carbon sequestration potential to create a zero-carbon transformation of this feed. The assumed chromium payability of 65% and nickel payability of 91% in the magnetite concentrate reflects the recoverable by-product values in this feed and is in-line with proposed terms from potential downstream JV partners.

## 19.5 Carbon Sequestration Fees

The company commissioned a study by a leading strategy house that confirms that the project could reasonably expect in excess of C$25 per tonne of $CO_2$ in capture and storage fees from its IPT carbonation process based on publicly known storage fees and communicated carbon price and policy status. The study also confirmed the potential requirement for more than 20 Mt of annual storage capacity of $CO_2$, given the communicated carbon price and Carbon Capture, Utilization, and Storage (CCUS) ITC status, from a population of approximately 150 potential emitters, with approximately 50 emitters concentrated in four distinct clusters in Sudbury, Sault Ste. Marie, Toronto, and Sarnia.

## 19.6 Contracts

CNC has not entered into any contracts at this time.

## 19.7 QP Comment

The QP has reviewed the information listed in Section 3 and 27 and notes that it supports the assumptions in the report.

# 20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This section provides an overview of the regulatory framework and environmental studies and consultation efforts related to obtaining federal and provincial environmental approval for the project.

## 20.1 Regulatory Framework

Both federal and provincial regulatory requirements apply to the project. These include a federal impact assessment, a provincial environmental assessment (EA), and corresponding jurisdictional environmental approvals.

### 20.1.1 Impact / Environmental Assessment

The project is subject to both federal and provincial requirements as detailed in the following sections. A cooperation agreement is in place between the Province of Ontario and Government of Canada to allow a coordinated federal and provincial process to reduce duplication of effort in both document preparation and consultation efforts for the federal impact assessment and provincial Class EA requirements.

#### 20.1.1.1 Federal Impact Assessment Requirements

The following conditions of the Physical Activities Regulations (SOR/2019-285) pursuant to the *Impact Assessment Act* (IAA) apply to the project based on the preliminary project design:

*Section 18 – The construction, operation, decommissioning and abandonment of one of the following:*

*(c) a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with an ore production capacity of 5,000 tonnes per day (t/d) or more*

*(d) a new metal mill, other than a uranium mill, with an ore input capacity of 5,000 t/d or more.*

Based on the current project design, the maximum rate of ore extraction will be up to 237,000 t/d during Year 13 of operations and an average rate of 150,000 t/d over the life of mine. The ore processing plant and associated service facilities will process run-of-mine ore delivered to primary crushers to produce nickel concentrate, iron concentrate, and tailings at a rate of approximately 60,000 t/d at the start of mine life, ramping up to a maximum of 120,000 t/d.

The maximum rate of ore production and processing at the project is expected to be substantially more than 5,000 t/d therefore meets conditions (c) and (d) listed above from the Physical Activities Regulations.

CNC submitted an initial project description to IAAC in August 2022 to initiate the regulatory assessment process. Based on public and regulatory consultation on the initial project description, a detailed project description (WSP, 2022b) was submitted to IAAC on December 22, 2022. In accordance with Section 16(2) of the *Impact Assessment Act*, IAAC decided on January 5, 2023, that a federal impact assessment is required for the Crawford Project. IAAC published the Tailored Impact Statement Guidelines in March 2023.

An Impact Statement will be prepared by CNC. A public review period and regulatory technical review of the Environmental Impact Statement will follow prior to the issuance of a Decision Statement about the project by IAAC. Permitting for site-specific activities related to the project's construction and operation are anticipated to commence concurrently during the preparation of the Environmental Impact Statement.

#### 20.1.1.2 Provincial Environmental Assessment Requirements

In Ontario, mining development projects are carried out by private sector proponents and are not subject to provincial Individual EA requirements, although certain ancillary activities associated with a mining development may be subject to one or more prescribed Class EA processes. Provincial Class EA processes that apply to the ancillary activities include the following:

- Class EA for resource stewardship and facility development projects (Category B or C) is required for the disposition of Crown resources. It is assumed the Ministry of Natural Resources and Forestry (MNRF) will delegate the authority to complete the Class EA for resource stewardship and facility development projects to CNC.
- Class EA for minor transmission facilities is required for the 500 kV transmission line relocation. This Class EA is the responsibility of Hydro One Networks Inc.
- Class EA for minor transmission facilities is required for 230 kV transmission line. This Class EA is the responsibility of Transmission Infrastructure Partnerships Limited.
- Class EA for minor transmission facilities is required for the transformers. This Class EA is the responsibility of CNC.
- Class EA for provincial transportation facilities (Group A or B) is required for the realignment of Highway 655 and potentially for the proposed overpass. It is assumed Ministry of Transportation Ontario (MTO) will delegate the authority to complete the Class EA to CNC.
- Class EA for the 120 m reserve around water bodies is required if infrastructure is placed within the 120 m reserve. There is potential for infrastructure such as pipelines and roads to be placed within the reserve. It is assumed the ministry of Mines (MINES) will delegate the authority to complete the MINES Class EA to CNC.

### 20.1.2 Environmental Approvals

#### 20.1.2.1 Federal Environmental Approvals

In addition to the federal impact assessment, federal environmental approvals related to the *Fisheries Act*, *Species at Risk Act*, *Canadian Navigable Waters Act, Explosives Act,* and *Migratory Bird Convention Act* are anticipated to be required for the project. Environment and Climate Change Canada (ECCC), Fisheries and Oceans Canada (DFO), Transport Canada, NAV Canada and Natural Resources Canada (NRCan) are the agencies primarily involved with approvals under these statutes.

Table 20-1 summarizes the federal environmental approvals that could potentially be required for the construction, operation, and closure of the project (including engineering approvals related to explosives manufacturing and/or storage). Additional details regarding each federal act are provided in the following sections. Other approvals may arise through consultation with federal agencies or with changes to the design of the project.

These federal departments have a broad range of responsibilities. The *Fisheries Act* gives the DFO responsibility for the management of fisheries, habitat, and aquaculture, including aquatics species at risk. DFO has the role to manage the harmful alteration, disruption, or destruction of fish habitat. ECCC is also responsible for the protection of fish and more broadly protection of the environment, and leads approvals related to the deposition of deleterious substances in fish bearing waters (a regulatory amendment to list a waterbody to Schedule 2 of the Metal and Diamond Mining Effluent Regulations [MDMER]). TC reviews the impacts on waterway navigability to ensure there is no interference with passage. NRCan is responsible for authorizing the manufacturing and storage of explosives through licenses and issues permits for the transportation of explosives.

It is anticipated that the overprinting of waters frequented by fish by the TMF and/or stockpiles may be necessary and could require a listing under Schedule 2 of the federal MDMER, pursuant to the *Fisheries Act*. In addition, a compensatory fisheries program, referred to as an Offsetting Plan for the Paragraph 35(2)(b) Authorization, and a Fish Habitat Compensation Plan for the Schedule 2 MDMER listing is anticipated to be required for the development of the project.

#### 20.1.2.1.1 Fisheries Act

The federal *Fisheries Act* includes provisions for the protection of fish and fish habitats, notably the prohibition against harmful alteration, disruption, or destruction (HADD) of fish habitat. The Act also prohibits activities that cause the "death of fish" (other than permitted fishing activities), considers the cumulative effects of development activities, and provides additional protection for highly productive, sensitive, rare, or unique fish and/or fish habitats. If death of fish or the HADD of fish habitat will likely result from a project, proponents are required to apply for an authorization from the Minister of Fisheries, Oceans and the Canadian Coast Guard as per Paragraph 34.4(2)(b) or 35(2)(b) of the *Fisheries Act* Regulations. The application must include an offsetting plan to counterbalance the HADD, along with a financial guarantee as an assurance mechanism if the offsetting plan is not completed. The *Fisheries Act* authorization will include terms and conditions the proponent must follow to avoid, mitigate, offset, and monitor impacts to fish and fish habitat resulting from a project.

#### 20.1.2.1.2 Fisheries Act – Metal & Diamond Mining Effluent Regulations

The MDMER, pursuant to the federal *Fisheries Act*, establishes effluent quality standards and requirements for environmental effects monitoring.

The MDMER requires a fish study for selenium (under specified monitoring results), as well as additional substances to be monitored (i.e., chloride, chromium, cobalt, sulphate, thallium, uranium, phosphorus, and manganese) as part of effluent characterization and water quality monitoring studies. Sub-lethal toxicity testing focuses on the most sensitive test species, and biological monitoring studies focus on aquatic communities facing situations of higher risk for environmental effects. Exemptions may be allowed from some biological monitoring requirements for mines with effluent presenting lower risks of affecting fish and fish habitat.

**Table 20-1: Anticipated Federal Environmental Approvals and Applicable Acts**

| Act or Approval | Responsible Ministry | Description |
|---|---|---|
| *Fisheries Act*, Metal and Diamond Mining Effluent Regulation Schedule 2 Listing | ECCC | Storage of potentially deleterious mineral waste covering minor tributaries that are frequented by fish.<br>An alternative assessment for mineral waste disposal in the prescribed format could be required along with an approved fish habitat compensation plan |
| Species At Risk Act | ECCC | Impacts to listed threatened and endangered species and their critical habitat. |
| Migratory Bird Convention Act | ECCC | Impacts to migratory birds, their eggs, and their nests within Canada and the United States. |
| *Fisheries Act*, authorization for harmful alteration, disruption or destruction of fish habitat or death of fish by means other than fishing | DFO | Direct impacts to fish habitat including overprinting of waterbodies and construction of structures in waterbodies / watercourses.<br>Indirect impacts to fish habitat, including flow reductions.<br>An approved fisheries offset plan will be required. |
| Canadian Navigable Waters Act | NAV Canada | Activities that will affect navigation in navigable waters. |
| Explosives Act | NRCan | Work with industrial explosives or restricted components. |
| *Canada Navigable Waters Act,* approval under the Navigation Protection Program | Transport Canada | Alteration of navigable waters and crossing of navigable waters with infrastructure.<br>Diversion of unscheduled watercourse to provide for safe mining. |
| *Aeronautics Act,* Land Use Clearance | NAV Canada | Construction of tall structures, use of cranes, transmission line towers.<br>Notification to the Land Use office of blasting to limit potential interference and safety issues with airports and air navigation infrastructure. |
| *Aeronautics Act,* aeronautical obstruction clearance Canadian Aviation Regulations (SOR / 96-433) | Transport Canada | Marking and lighting for structures that could interfere with aeronautical navigation. |

#### 20.1.2.1.3 Canadian Navigable Waters Act

The *Canadian Navigable Waters Act* applies to anyone planning activities that will affect navigation in navigable waters. The *Canadian Navigable Waters Act* regulates major works and obstructions on navigable waters, even those not listed on the schedule of navigation, and creates a new category for “major works.” Major works are those likely to interfere substantially with navigation, and always require approval from Transport Canada. Transport Canada administers the *Canadian Navigable Waters Act* through the Navigation Protection Program.

#### 20.1.2.1.4 Species at Risk Act (SARA)

Both federal and provincial governments regulate species at risk and their protection through specific legislation. The federal *Species at Risk Act* (SARA) is intended to protect species at risk in Canada and their "critical habitat" (as defined by SARA). Under SARA, proponents are required to demonstrate that no harm will occur to listed species, their residences or critical habitat, or identify adverse effects on specific listed wildlife species and their critical habitat, followed by the identification of mitigation measures to avoid or reduce effects. Activities must comply with SARA, with prohibitions against (1) the killing, harming, or harassing of endangered or threatened species at risk (SAR) (Sections 32 and 36); and (2) the destruction of critical habitat of and endangered or threatened SAR (Sections 58, 60 and 61).

#### 20.1.2.1.5 Explosives Act

The federal *Explosives Act* regulates the manufacture, testing, acquisition, possession, sale, storage, transportation, importation, and exportation of explosives to ensure they are handled safely. Under the *Explosives Act*, licenses or permits may be required to work with explosives or restricted components such as fertilizers, cleaning products, stump removers, and paint thinners. An explosives licence is required to buy, sell, and store industrial explosives.

#### 20.1.2.1.6 Migratory Bird Convention Act, 1994

The purpose of the *Migratory Birds Convention Act* is to protect and conserve migratory birds, their eggs, and their nests within Canada and the United States. Within the *Migratory Birds Convention Act*, the disposal of any substances harmful to migratory birds in any waters or areas frequented by migratory birds is prohibited. Similarly, damaging, destroying, removing, or disturbing migratory bird habitat is prohibited.

#### 20.1.2.1.7 Aeronautics Act

Transport Canada requires that structures, which are obstacles to air navigation, are marked and/or lighted so they can be easily identified during the day and night. Under the *Aeronautics Act*, the completion of an aeronautical assessment is required for structures that may be obstacles to air navigation.

### 20.1.2.2 Provincial Environmental Approvals

In addition to the provincial Class EAs, it is anticipated that the following four primary provincial ministries will be involved in environmental approvals for the project:

- Ministry of Mines (MINES) has a responsibility to ensure the orderly development of mineral resources in Ontario, including responsibilities for the disposition of Crown lands for mining, and has primary responsibility for mine reclamation planning activities.
- Ministry of Environment, Conservation and Parks (MECP) grants permits and approvals that address aspects that are related to water quality and quantity, air quality, noise, and species at risk.

- Ministry of Natural Resources and Forestry (MNRF) have the role to ensure the protection and wise use of Crown resources including merchantable timber; and Crown lands not otherwise disposed such as through the *Mining Act* administered by MINES.
- Ministry of Transportation (MTO) manages the provincial highway system amongst other aspects, including Highway 655 which bisects the project site.

Additional agencies that may be involved in permitting and approvals include the following:

- Ontario Energy Board (OEB)
- Ministry of Citizenship and Multiculturism (MCM).

Provincial environmental approvals that are anticipated to construct and operate the project are shown in the preliminary list in Table 20-2 with details regarding each provincial act provided in the following sections.

##### 20.1.2.2.1 Mining Act

The Ontario *Mining Act* and associated regulations provide a framework for authorizing mineral exploration and development activities in the province.

The *Mining Act* requires proponents to submit a closure plan to MINES before undertaking activities in the development or operational stages of the project. Proponents are also required to provide the MINES with financial assurance, which amounts to the estimated costs of the rehabilitation measures described in the closure plan and allows the province access to this money to carry out the rehabilitation work outlined in a closure plan if the proponent is unable or unwilling to undertake the work itself.

##### 20.1.2.2.2 Ontario Water Resources Act

The Ontario *Water Resources Act* is regulated by the MECP and focuses on both groundwater and surface water throughout the province. The *Water Resources Act* regulates the taking of water, sewage disposal, and “sewage works” and prohibits the discharge of polluting materials that may impair water quality.

Ontario Regulation 387/04: Water Taking and Transfer Regulation (O.Reg. 387/04) requires a permit for water takings of more than a total of 50,000 L/d (with some exceptions). Section 34 of the OWRA requires the proponent to obtain a Permit To Take Water and Section 9 of O.Reg. 387/04 requires all permit holders to collect, record, and report data on daily volumes of water withdrawals.

Section 53 of the OWRA requires that an Environmental Compliance Approval (ECA) be obtained for industrial sewage systems that release or discharge, store, or transport contaminants to groundwater or surface water.

##### 20.1.2.2.3 Environmental Protection Act

The *Environmental Protection Act* is regulated by the MECP. This Act regulates the discharge of contaminants and pollutants into the environment if the discharge has the potential to cause an adverse effect.

**Table 20-2: Anticipated Provincial Environmental Approvals**

| Act or Approval | Responsible Ministry | Description |
|---|---|---|
| Closure Plan, *Mining Act* | MINES | Progressive reclamation and final closure of the site, including financial assurance<br>Construction of dams above the high-water mark of watercourses, if any |
| Environmental Compliance Approval (ECA) – Waste Disposal Site / Waste Management, *Environmental Protection Act* | MECP | For establishment of a waste disposal / transfer site, if required, and for the transportation of waste materials offsite and onto provincial highways |
| ECA – Noise and Air, Environmental Protection Act | MECP | Atmospheric and noise emissions from the project, such as from haul trucks and structures including the ore processing plant, refuelling / oil / lubrication areas, administration complex, on-site laboratory, and other buildings |
| ECA – Domestic Sewage, Ontario Water Resources Act | MECP | Establishment and operation of a domestic sewage treatment plant, treatment of domestic waste produced at the project site including back wash and sludge produced in the sewage treatment plant |
| ECA(s) - Industrial Sewage Works, *Ontario Water Resources Act* | MECP | Mine / mill water treatment system(s) discharging into the environment, process water, contact water, and tailings management<br>Management and treatment of grey water, domestic sewage, etc. |
| Overall Benefits Permit, *Endangered Species Act* | MECP | Impacts to species or their habitat identified as endangered or threatened.<br>Project is within the Kesagami range of woodland caribou, and as a minimum a screening is required |
| Permit(s) to Take Water, Ontario Water Resources Act | MECP | Dewatering activities in support of construction and longer term mine dewatering (in excess to 50,000 L/d) |
| Clearance Letter, *Heritage Act* | MCM | Confirmation that appropriate archaeological studies and mitigation, if required, have been completed for the project |
| Aggregate Resource License, *Aggregate Resource Act* | MNRF | If the proposed field investigations are successful in finding an appropriate resource, CNC may pursue an aggregate resource permit to provide a source of aggregate to support mine construction and operation |
| Forest Resource License, Crown Forest Sustainability Act | MNRF | For cutting of merchantable timber for site development or as part of construction of the transmission line |
| Land Use Permit(s) / Sale of Crown Land / License of Occupation (lake bottom), *Public Lands Act* | MNRF | Temporary land tenure for facilities off the mining lease (transmission line, shoreline structures), if required<br>Consultation with MNRF planned regarding shoreline tenure at north end of main pit, given that the shoreline will be mined as part of the main pit<br>Consultation with MNRF planned regarding tenure for lake bottom where main pit is located |
| Work Permit(s), Lakes and Rivers Improvement Act, Public Lands Act | MNRF | Construction of facilities on Crown land, including below the high-water mark of waterbodies / watercourses<br>Construction of a dam in any lake or river (in circumstances as set out in the regulations) requires written approval of the Minister for the location of the dam, and its plans and specifications |
| Permit to Collect Fish for Scientific Purposes, *Fish and Wildlife Conservation Act* | MNRF | Potential fish transfer during construction from watercourses and waterbodies to be overprinted or removed<br>Fisheries investigations during construction, operation, and closure<br>Authority to remove beavers and/or beaver dams |
| Entrance Permit and Encroachment Permit, Public Transportation and Highway Improvement Act | MTO | May or may not be applicable depending on the final project design / construction management approach |
| Leave to Construct, Ontario Energy Board Act | OEB | Approval to construct a transmission line |

#### 20.1.2.2.4 Endangered Species Act

Both federal and provincial governments regulate species at risk and their protection through specific legislation. The Ontario *Endangered Species Act* (ESA) provides special protection for native plant and animal species considered to be endangered or threatened in Ontario.

#### 20.1.2.2.5 Public Lands Act

The Ontario *Public Lands Act* regulates the sale or lease of public lands for any purpose other than agricultural purposes. Approval under the *Public Lands Act* is required to obtain tenure for long-term facilities on Crown land, such as for a transmission line, or shoreline structures (dock, pumphouse and pipeline).

#### 20.1.2.2.6 Lakes and Rivers Improvement Act

The Ontario *Lakes and Rivers Improvement Act* regulates management, protection, preservation, and use of the waters of the lakes and rivers of Ontario and the land under them. Furthermore, the *Lakes and Rivers Improvement Act* governs the design, construction, operation, maintenance, and safety of dams in Ontario, as well as other works, including water crossings, channelizations, enclosures, cables, and pipelines.

#### 20.1.2.2.7 Crown Forest Sustainability Act

The purpose of the Ontario *Crown Forest Sustainability Act* is to manage the Crown forests to meet the social, economic, and environmental needs of present and future generations. As part of planning for sustainability in our forests, a forest management plan is prepared for each and every forest management unit in Ontario.

"Sustainability" with respect to the Act means "long-term Crown forest health". If CNC's mine leases have trees reserved to the Crown, the approval for removal of the trees would be completed as per the *Crown Forest Sustainability Act*.

#### 20.1.2.2.8 Fish and Wildlife Conservation Act

The *Ontario Fish and Wildlife Conservation Act* is a provincially regulated Act for the hunting, trapping, and fishing within the province. The aim of the *Fish and Wildlife Conservation Act* is to preserve at-risk wildlife and habitat. Within the Act, it is prohibited to damage, destroy, remove, or disturb the nest or eggs of an indigenous bird.

#### 20.1.2.2.9 Aggregate Resources Act

Most of Ontario's pits and quarries are regulated under the *Aggregate Resources Act*. Some areas of private land are not covered by the Act. In these areas, the local municipality may regulate pit and quarry operations. On private land, a Class A license is required if more than 20,000 tonnes of aggregate are removed annually. A Class B license is required if 20,000 tonnes or less of aggregate is removed annually. On Crown land, an aggregate permit is required to operate a pit or quarry or to extract Crown-owned aggregate or topsoil.

### 20.1.3 Additional Permits and Authorizations

A small portion of the project footprint is located within Kidd Township and within the municipal boundary of the City of Timmins. Project components within this area include portions of the Highway 655 realignment and the rail spur line. Notably, part of the linear infrastructure corridor, the 230 kV transmission line proposed by TIP-1 will also pass through this area. CNC will engage with the City of Timmins regarding municipal approvals related to constructing infrastructure on municipal lands. Additional authorizations may be required during construction and operation such as building permits, electrical safety authority inspections/authorizations, an explosives permit, etc.

## 20.2 Environmental Studies

The project site is in a remote part of northeastern Ontario, with existing provincial infrastructure including highway and transmission lines that overlap part of the site, and prior impacts from exploration or forestry operations (Figure 4-1). The primary disturbance on site to date is related to exploration activities and engineering investigations. CNC has been conducting ongoing environmental baseline investigations associated with the project since early 2021.

The following sections summarizes the baseline studies completed to date and the physical and biological environmental settings of the project site and surrounding area.

### 20.2.1 Baseline Studies

CNC began environmental baseline studies at the project site in 2021, and has committed to ongoing studies in acknowledgment that programs will be required to meet the anticipated baseline environmental data needs of the following:

- Federal Impact Assessment Act
- Provincial Environmental Assessment Act
- future environmental approval applications, such as requirements under various federal and provincial legislation including but not limited to the federal *Fisheries Act* and the provincial *Environmental Protection Act*.

Studies carried out in 2021, 2022 and into 2023, are listed in Table 20-3. The results of these studies provide the basis for the information in Sections 20.2 to 20.5 of this report.

Results of the environmental baseline studies to date have been as expected, with project site conditions generally typical of the area. As the project design is advanced, follow-up studies will be carried out as required to address information gaps.

**Table 20-3: Existing and Ongoing Environmental Baseline Studies**

| Technical Discipline | Reports |
|---|---|
| Atmospheric and Noise | Crawford Open Pit Project Climate Study. Prepared for Canada Nickel Company, May 2022. Golder Associates Ltd.<br>Draft CNC Crawford Project Nighttime Light Baseline Report. September 2023. WSP E&I Canada Inc.<br>CNC Crawford Mine Baseline Noise Field Program, 2023 Baseline Noise Monitoring & Measurements – Leaves-On and Leaves-Off, November 2023. WSP E&I Canada Inc.<br>Interim Baseline Air Quality Monitoring Report. Prepared for Canada Nickel Company, September 27, 2023. WSP E&I Canada Inc. |
| Aquatic Ecology | Crawford Nickel Project 2021 Aquatic Resources Baseline. Prepared for Canada Nickel Company, September 2022. Wood Environment & Infrastructure Americas, a Division of Wood Canada Limited.<br>Draft Interim 2022 Aquatic Resources Baseline Report, Crawford Nickel Project. Prepared for Canada Nickel Company, May 2023. WSP E&I Canada Inc.<br>Crawford Nickle Project 2021-2023 Fish and Fish Habitat Baseline. Prepared for Canada Nickel Company, October 2023. WSP E&I Canada Limited. |
| Hydrology | 2021 Hydrology Monitoring Report, Crawford Project. Prepared for Canada Nickel Company, March 2022. Golder Associates Ltd.<br>2022 Hydrology Monitoring Report, Crawford Project. Prepared for Canada Nickel Company, May 2023. WSP Canada Inc. |
| Hydrogeology | Technical Memorandum: Groundwater Quality Sampling Factual Report, dated February 7, 2023. Sent to Canada Nickel Company. WSP Canada Inc.<br>Technical Memorandum: Hydraulic Conductivity Testing Summary, dated January 25, 2023. Sent to Canada Nickel Company. WSP Canada Inc.<br>Technical Memorandum: Borrow Source Test Pit Field Program – Crawford Project, dated December 5, 2022. Sent to Canada Nickel Company. WSP Canada Inc.<br>PowerPoint PDF: Simulated Pore Water Pressures and Groundwater Inflows – Main Zone and East Zone Pits, dated October 3, 2022 and as updated November 25, 2022. Prepared for Canada Nickel Company. WSP Canada Inc.<br>Application for a Category 2 Permit to Take Water, Pumping Test Program, dated April 25, 2022. Prepared for Canada Nickel Company. WSP Golder Associates Ltd. |
| Terrestrial Ecology | Crawford Nickel Project 2021 Terrestrial Ecology Baseline Study. Revision 0A2. Prepared for Canada Nickel Company, June 2022. Wood Environment & Infrastructure Americas, a Division of Wood Canada Limited.<br>Draft Crawford Nickel-Cobalt Sulfide Project – 2022 Terrestrial Ecology Baseline Study. Revision 0B. Prepared for Canada Nickel Company, May 2023. WSP E&I Canada Ltd.<br>Draft Crawford Nickel Project Species at Risk Summary Information Technical Memorandum. Prepared for Canada Nickel Company, June 2023. WSP E&I Canada Limited. |
| Geochemistry | Technical Memorandum: Geochemistry Interim Results Update – April 2023. Sent to Canada Nickel Company June 22, 2023. WSP Canada Inc. |

| Technical Discipline | Reports |
|---|---|
| | Draft Technical Memorandum: Interim Geochemistry Source Terms. Prepared for Canada Nickel Company, September 15, 2023. WSP Canada Inc. |
| Human Environment | Draft Crawford Nickel Project Cultural Heritage Screening Report. Revision A. Prepared for Canada Nickel Company, May 2022. Wood Environment & Infrastructure Solutions Canada Limited. (Wood 2022d)<br>Draft Crawford Nickel Project: Socio-Economic Baseline Report. Prepared for Canada Nickel Company, September 7, 2023. Stantec Consulting Ltd.<br>Flying Post First Nation Socio-Economic Study for Canada Nickel Company's Crawford Nickel Sulphide Project. August 2023. Firelight Research Inc. with Flying Post First Nation.<br>Matachewan First Nation Socio-Economic Study for Canada Nickel Company's Crawford Nickel Sulphide Project. August 2023. Firelight Research Inc. with Matachewan First Nation.<br>Mattagami First Nation Socio-Economic Study for Canada Nickel Company's Crawford Nickel Sulphide Project. August 2023. Prepared by Firelight Research Inc. with Mattagami First Nation.<br>Socio-Economic Study for the Crawford Nickel Project: Summary of Findings and Recommendations Report. Prepared for Taykwa Tagamou Nation, August 18, 2023. Shared Value Solutions. |
| Archaeology and Heritage Resources | Crawford Nickel Project Stage 1 Archaeological Assessment. Prepared for Canada Nickel Limited, October 2022. WSP E&I Canada Ltd.<br>Crawford Nickel Project Stage 1 Archaeological Assessment. Prepared for Canada Nickel Company, August 2023. WSP E&I Canada Ltd. |
| Indigenous Traditional Knowledge/ Traditional Land Use | Flying Post First Nation Knowledge and Use Study for Canada Nickel Company's Crawford Nickel Project. July 11, 2023. Jonathan Taggart and Firelight Research Inc. with Flying Post First Nation.<br>Matachewan First Nation Knowledge and Use Study for Canada Nickel Company's Crawford Nickel Project. July 19, 2023. Jonathan Taggart PhD and Firelight Research Inc. with Matachewan First Nation.<br>Mattagami First Nation Knowledge and Use Study for Canada Nickel Company's Crawford Nickel Project. August 16, 2023. Jonathan Taggart and Firelight Research Inc. with Mattagami First Nation.<br>Taykwa Tagamou Nation Crawford Nickel Project Traditional Knowledge and Land Use Study. Prepared for Taykwa Tagamou Nation, February 24, 2023. Shared Value Solutions. |

### 20.2.2 Environmental Setting

The project site is situated within the Northern Clay Belt in northeastern Ontario, within the Timmins-Cochrane Mining Camp, a region with a robust mining history (e.g., gold, nickel, zinc, lead). The site is generally in a natural condition other than existing provincial infrastructure (e.g., highway, transmission lines) that cross portions of the lands, and prior impacts as a result of mineral exploration and forestry operations. Vegetation communities consist of a mix of deciduous and coniferous forests, and low-lying wetland areas and muskeg. The closest communities are the Town of Cochrane (35 km to the northeast), the City of Timmins (42 km to the south), the town of Smooth Rock Falls (50 km to the north), and the Town of Iroquois Falls (50 km to the east).

The following subsections summarize the physical and biological environmental settings of the project site and surrounding area and are based on literature reviews and baseline studies conducted between 2021 and 2022.

#### 20.2.2.1 Terrestrial Ecology

The project is located within the Ecoregion 3E, the Lake Abitibi Ecoregion, within the Canadian Shield, and is subject to the Boreal Ecological Land Classification for classifying the vegetation communities (Wood, 2022c). Specifically, the project site is situated within the northeastern part of the ecoregion, which is associated with the Northern Clay Belt. The Northern Clay Belt is a low lying, undulating plain of glaciolacustrine clay that limits drainage. Provincial significant wetlands (such as Kraft Creek/Murphy Creek Wetland and the Little Goose Creek Wetland) are located within 20 km of the project site. A conservation reserve (Mahaffy Township Ground Moraine) is located within 5 km of the project site. Provincial parks and First Nation reserves are located within 100 km of the project footprint.

Three hundred and forty-one species of vascular plants, bryophytes, and lichens were identified during field investigations and 28 distinct plant communities (upland and wetland) were recorded (WSP, 2023a). Groundwater-fed fens, bogs fed solely by rainwater, swamps, and marshes were all present throughout the project site. Coniferous forests dominate the project site, with typical tree and shrub species identified on site including black spruce, tamarack, speckled alder, bog birch, balsam willow, trembling aspen, Labrador tea, eastern white cedar, white birch, balsam fir, balsam poplar, red-osier dogwood, and paper birch.

Nine mammal species were identified through aerial surveys. Moose and red fox were directly observed during the surveys. Tracks of American beaver, North American river otter, grey wolf, Canada lynx, American marten, and snowshoe hare were also observed throughout the project site.

During targeted bird surveys in 2021 and 2022, 89 species were visually observed. An additional 24 species were recorded during other non-targeted investigations (WSP, 2023a). Acoustic monitoring stations using Wildlife Acoustics' song meter micros were used to identify 101 species of birds within the project site and surrounding area, 20 of which were not detected visually. The most abundant species observed include the white-throated sparrow, Tennessee warbler, and ruby-crowned kinglet. Three species were confirmed breeding (Canada goose, common loon, and green-winged teal), 49 species were noted as probable breeding, and 35 species were recorded as possible breeding (WSP, 2023a).

Bat maternity roosting habitats were surveyed across the project site. Cavity trees identified were 58.1% white birch, 18.6% trembling aspen, and others in order of popularity were balsam poplar, black spruce, northern white cedar, tamarack, willow sp., balsam fir, yellow birch, and eastern cottonwood. In 2021, 14,967 bat passes were recorded and 32,500 bat passes were recorded in 2022 (WSP, 2023a). The most frequent species observed were the silver-haired bat, the hoary bat, and big brown bat. There was an absence of suitable bat hibernacula across the project site.

Six species of frogs and amphibians were identified during the amphibian call survey in 2021 and 2022, specifically the mink frog, spring peeper, American toad, green frog, boreal chorus frog, and wood frog (WSP, 2023a). Although targeted surveys for salamanders were not completed, a blue-spotted salamander was observed and documented on site. Suitable turtle basking habitat was also found throughout the project site, although no turtles were recorded during the turtle basking surveys.

#### 20.2.2.2 Species at Risk

Habitats in the project site and surrounding area also support designated species at risk (SAR). The provincial *Endangered Species Act* (ESA) and the federal *Species at Risk Act* (SARA) provide their own designations for each of the species. Not all SAR are offered protection under provincial and/or federal law and regulation.

The flora and vegetation surveys identified one plant SAR in the study area. Black ash is ranked as “endangered” (ESA) due to its rapid decline as a result of the invasive emerald ash borer beetle. As a listed SAR, black ash would be afforded individual and habitat protection; however, a minister’s order was approved, suspending protection for two years (Ontario Regulation 23/22), beginning January 26, 2022 and ending on January 25, 2024. There were two recorded observations of black ash within the study area.

Woodland caribou are listed as a “threatened” (ESA) species and were not observed either visually or through tracks or sign. The project is located along the southern boundary of the Kesagami caribou range for woodland caribou. The northern portion of this caribou range is characterized by the James Bay Lowlands with extensive wetland and boreal forest complexes with many rivers and few small lakes throughout; caribou occurrence is highly coupled with these intact wetland and mature black spruce forests. In contrast, the southern portion of the Kesagami caribou range, where the project occurs, is highly impacted by previous and current human activity, most notably timber harvest and settlement, with fragmented mature coniferous forest areas remaining and consequently caribou occurrence is minimal. This southern part of the range is targeted by the MECP for restoration of habitat for woodland caribou as habitat function has become degraded.

Blanding’s Turtle was reported to be within the study area by the community and is identified as Threatened under ESA and Endangered under SARA. Locations in which Blanding’s Turtle were reported to be observed by the communities were studied using eDNA sampling in 2023 with results to follow. Protected areas for Blanding’s Turtle under the *Endangered Species Act* are required, if observed.

Six SAR were observed during the bird surveys. The chimney swift is recognized as “threatened” by SARA and ESA. The bald eagle, Canada warbler, common nighthawk, olive-sided flycatcher, and rusty blackbird are all documented as “special concern” by ESA. SARA also lists the Canada warbler as “threatened,” and the common nighthawk, olive-sided flycatcher, and rusty blackbird as “special concern.” The bald eagle has no status under SARA. Long-tailed duck were also observed on site but were documented outside of the breeding period and not considered to be a species of concern.

Little Brown Myotis was documented to be on site through bat maternity habitat acoustic surveys and is listed as Endangered through SARA and the ESA. The presence of Northern Myotis (Endangered under SARA and ESA) could not be confirmed on site, but the presence of this species should not be ruled out. The Tricoloured bat (Endangered under SARA and ESA) was also not observed on site, and is unlikely to occur as the project is located north of the documented range of the species. Additional field surveys are recommended to confirm the presence of SAR bat species.

Other species potentially present based on desktop review and/or comments received, but not observed during field investigations are listed below. CNC is mindful that observations may occur, however based on the extent of the field studies conducted so far, the likelihood of observing these species during additional field campaigns is limited. As such, apart from the lake sturgeon, no additional field campaigns are planned regarding these species. The species include the following:

- bank swallow (“threatened” under SARA and ESA)
- yellow rail (“special concern” under SARA and ESA)
- peregrine falcon (“special concern” under ESA)
- monarch butterfly (“special concern” under SARA and ESA)
- yellow-banded bumble bee (“special concern” under SARA and ESA).

Discussions have been initiated with the MECP regarding information sharing for species of conservation concern, and determination of next steps to be undertaken in support of mitigation measures which may be required for development of the project. Further studies to identify and/or confirm SAR presence on the site is underway for 2023.

#### 20.2.2.3 Aquatic Ecology

The project is located within the Hudson Bay watershed, within the headwaters of the West Buskegau River and North Driftwood River. The Mattagami River borders the project area to the west, with a small part of the TMF (1.3 $km^2$) located within the Jocko Creek (tributary of Mattagami River) catchment area. The Lower Sturgeon Falls hydroelectric generating station is located on the Mattagami River, approximately 9 km west of the proposed TMF footprint and open pit. Commissioned in 1923, the generating station operates as a concrete ogee and pier dam with a reservoir and is a barrier to fish passage. The dam has a contributing upstream catchment area of 7,899 $km^2$ and is owned and operated by Ontario Power Generation.

Aquatic baseline studies were undertaken in 2021 and 2022 on the project site and surrounding area, focusing on the following watercourses and associated tributaries:

- North Driftwood River (tributary of Abitibi River)
- West Buskegau (tributary of Abitibi River)
- Mattagami River.

The studies included fish habitat and community assessments, fish collection for fish tissue analyses, and benthic invertebrate and sediment analyses. Habitat assessments and a multi-season physico-chemical surface water quality analysis were conducted on riverine (lotic) and ponded or lake (lentic) stations.

The fish habitat within the river systems in the area of the project is typical of northeastern Ontario, composed of channels with dense shrubby riparian vegetation, wetland segments with ponds, as well as abundant evidence of beaver activity. Sparse shrubs within the riparian habitat provide some overhead cover nearshore, while submerged woody debris provide in-stream cover. The channels are mostly characterized by a large width-to-depth ratio with a slow flow. The substrate is primarily composed of fine-grained sediment with high organic content attributed to the wetland habitats and beaver inputs. Beaver dams provide some seasonal fragmentation of these watercourses; however, they do not pose year-round barriers to fish passage as demonstrated by fish presence throughout the sampled areas of the project.

Fish community studies were completed at 11 locations on the North Driftwood River, including Gerry Lake and Martin Lake, and four locations on the West Buskegau River in the summer and fall of 2021. The field program in 2022 expanded to an additional six locations on the North Driftwood River (including four named lakes), three locations on the West Buskegau River, and one location on Jocko Creek, a tributary of the Mattagami River. Preliminary observations from the initial baseline studies and fish community studies completed by MNRF documented the presence of 17 fish species within the investigation areas, including northern pike, brook stickleback, fathead minnow, Iowa darter, and yellow perch. The local fish communities are mostly represented by small bodied, forage fish species such as dace, shiners and minnows that prefer a cool water thermal regime. Some fish species also represent a range of cool-to-warm fish habitat and some cold-water species such as burbot are also present within the inland tributaries. Large-bodied fish species, including northern pike and white sucker are found mostly in their juvenile life stages, whereas adults of these species can be found within the larger waterbodies such as Gerry Lake and Martin Lake, as well as the Mattagami River to the west of the project. Lake sturgeon of the Southern Hudson Bay – James Bay population are listed as "special concern" (SARA and ESA) and are known to occur within the Mattagami River. The baseline studies to date have not detected lake sturgeon within the study areas. In addition, fish community study results from surveys provided by the MNRF do not include the presence of Lake Sturgeon within the reach of the Mattagami River downstream of the Lower Sturgeon Falls hydroelectric generating station dam.

Small-bodied sentinel fish species (northern pearl dace and northern redbelly dace) and large-bodied sentinel fish species (northern pike) were used for the 2021 tissue analysis within two lakes on the North Driftwood River. Additional small-bodied sentinel species (finescale dace, brook stickleback, lake chub, and fathead minnows) were used in the 2022 tissue analysis within an additional eight locations on the North Driftwood River, two locations on the West Buskegau River, and one location on Jocko Creek. Fish tissue samples were analyzed for total metals, including mercury, methylmercury, and selenium. Northern pike showed an increasing trend in total mercury observed with increasing length, weight, and age. Some northern pike were found to have mercury concentrations over the recommended consumption guidelines for Health Canada and the Ontario consumption guidelines for the general population during the 2021 sampling event, with concentrations below guidelines during the 2022 sampling program. The average total methylmercury concentrations for northern pike, brooke stickleback, lake chubb, and northern pearl dace were also above the Canadian Council of Ministers of the Environment (CCME) guidelines during both the 2021 and 2022 sampling events for some sampling locations (Wood, 2022a; WSP, 2023c). No exceedances of either the provincial or federal consumption guidelines were noted for selenium.

Sediment and benthic samples were taken at eight locations on the North Driftwood River and four locations on the West Buskegau River in the fall of 2021. During 2022, sampling was conducted at an additional four locations on the North Driftwood River and two locations on the West Buskegau River. In 2021, exceedances of the Ontario Provincial Sediment Quality Guidelines (PSQG; MOE 2008) and/or Canadian Sediment Quality Guidelines (CSQG) for the

Protection of Aquatic Life (CCME 2001) for either the lowest effect level or severe effect level were observed for concentrations of total organic carbon, cadmium, chromium, copper, iron, manganese, and nickel on some lakes and streams of North Driftwood River (Wood, 2022a). Concentrations for total kjeldahl nitrogen, total arsenic, and zinc were also found to have some exceedances of the PSQG and/or CSQG on stations on the North Driftwood River during the 2022 sampling program (WSP, 2023c). Some exceedances of the PSQG and/or CSQG in the tributaries of West Buskegau River of total organic carbon, cadmium, chromium, iron, lead, manganese, and nickel were noted during 2021 and 2022. Many benthic invertebrate communities were observed, with the taxa of greatest abundance including *chironomidae*, *caenidae*, *naididae*, and *planorbidae*. Phytoplankton and zooplankton were also found to have seasonal variability for biomass and density across the project area, with higher biomass and density occurring in September/October.

Water quality sampling was completed at 19 stations across the project site during the summer and fall of 2021, including two lake sampling locations. An additional 11 stations were added to the program in 2022, while two stations from the 2021 program were discontinued. Between the 2021 and 2022 sampling programs, three sampling stations were located on the Mattagami River or Jocko Creek, 18 sampling stations on the North Driftwood River or tributaries thereof, and 9 sampling stations on the West Buskegau River or tributaries thereof. Site water quality was compared for characterization purposes to the federal (Federal Environmental Quality Guidelines [FEQG]; CCME Canadian Environmental Quality Guidelines [CEQG] for the Protection of Aquatic Life [CCME 2022]) and Provincial Water Quality Objectives (PWQO [MOEE 1999]) to assess existing surface water quality.

Water quality on the North Driftwood River system was below the PWQO, FWQG, and CWQG for most nutrient, total metal, and dissolved metal parameters, except for some parameters listed below:

- North Driftwood 2021 riverine sampling locations exceeded the CWQG for total mercury, and some exceedances in the CWQG for total aluminum, cobalt, iron, manganese, mercury, and dissolved strontium. Riverine sampling locations also exceeded the PWQO for dissolved oxygen, total aluminum, and iron during both 2021 and 2022 sampling events.
- West Buskegau riverine sampling locations exceeded the CWQG for total mercury in 2021, and the CWQG for total aluminum, copper, and iron during 2021 and 2022 sampling events. Some riverine sampling locations exceeded in the PWQO for total iron during both 2021 and 2022 sampling events.
- Mattagami River sampling locations exceeded the CWQG for total mercury during the 2021 sampling event. Total aluminum and total iron concentrations exceeded the CWQG guideline during the 2022 sampling event.

Additional studies were completed by CNC during the 2023 field season to gather additional baseline data and confirm the presence of Lake Sturgeon within the project area. The results of these studies are pending.

#### 20.2.2.4 Social, Cultural & Economic Environment

Crawford Township has been an area of interest for mineral potential since 1955, with many mining companies and government bodies investigating the area up until the 1980s. The project site was more actively explored starting in 2017, with airborne surveys and diamond drilling undertaken by Noble Exploration in the area in 2017 and 2018. Noble Exploration announced the creation of CNC, with a 100% interest in the Crawford property, and with interests of Spruce Ridge Resources sold to CNC. Since that time, work undertaken on the project site has consisted of exploration and resource definition drilling, and geotechnical drilling in support of ongoing engineering studies.

The project site is located within the Abitibi River Forest. The Abitibi River Forest encompasses approximately 35,000 km$^2$, extending westward from the Ontario / Québec border for 190 km to the southern limit, south of Timmins, to the northern most extent of the province's managed forest land (ARFMI, 2022). The Abitibi River Forest is currently managed by Abitibi River Forest Management Inc., which is made up of forest resource management partners responsible for forest management planning and operations.

There are no federal parks near the project site. The closest provincial parks are Greenwater Provincial Park (a non-operating, natural environment park with no facilities) approximately 49 km to the north and Kettle Lakes Park (day use and overnight camping facilities), located approximately 80 km away from the project site. There are several provincial Conservation Reserves in the region, including the Mahaffy Township Ground Moraine Conservation Reserve located approximately 10 km to the northwest of the project site, and the Northern Claybelt Forest Complex Conservation Reserve, approximately 50 km to the west.

The project site is in the geographic townships of Crawford, Carnegie, Kidd, Lucas, and Prosser within the Cochrane District in northeastern Ontario, with a total population of 77,965. A small portion of the project extent within Kidd Township also lies within the municipal boundary of the City of Timmins. The primary industries include mining, healthcare and social assistance, education, construction, and retail trade (Statistics Canada 2017). There are several mining operations within the district, including Newmont Porcupine and Glencore's Kidd Creek mines, as well as exploration programs, such as the West Cache Gold Project (Galleon Gold) and Fenn-Gib Project (Mayfair Gold). Eco-tourism is a popular recreational activity in the district, with activities including snowmobiling, ATV touring, camping, and water sports. A snowmobile trail crosses the project footprint in the area of mine rock stockpile no. 1. Available information indicates that there are no designated ATV trails, canoe, or trail routes which overlap with the project site (Adventure North Ontario, 2022). According to the Abitibi River Forest 2022-2032 Forest Management Plan, there are no recorded trapper cabins, access points, beaches, boat caches, clubhouses, designated camp sites, fishing access points, commercial campgrounds, main base lodges, outpost camps, shooting ranges, recreation camps, or youth camps in the project site footprint. Although not identified in the Abitibi River Forest 2022-2032 Forest Management Plan, camps and hunting blinds have been observed within the project site footprint.

There are no First Nation Reserve lands proximal to the site, although the project site is within or close to the traditional or operating region of several Indigenous Peoples. In addition, IAAC issued the Indigenous Engagement and Partnership Plan, the purpose of which is to provide the opportunity for meaningful engagement with Indigenous communities and to carry out collaborative assessment of the impacts of the project on the Aboriginal or treaty rights of the Indigenous community. Indigenous groups that have expressed interest in the project are as follows:

- Apitipi Anicinapek Nation, located approximately 104 km southeast of the project site
- Flying Post First Nation, located approximately 59 km southwest of the project site
- Matachewan First Nation, located approximately 100 km southeast of the project site
- Mattagami First Nation, located approximately 115 km south of the project site along the Mattagami River
- Métis Nation of Ontario – Region 3.
- Taykwa Tagamou Nation, located approximately 45 km northeast from the project site in the Cochrane District along the Abitibi River

Through engagement activities and primary research, CNC will continue to engage and work with communities and Indigenous Nations to gather information on economic activities and better understand the potential impacts of project related activities in these areas. Socio-economic studies have recently been provided by Taykwa Tagamou, Mattagami, Matachewan, and Flying Post First Nations. The economic context of Indigenous Nations and Peoples will be further assessed through the impact assessment process.

The Porcupine Health Unit serves primarily Cochrane District, including Timmins, Cochrane, Iroquois Falls, and Smooth Rock Falls. According to the Porcupine Health Unit, people in the District of Cochrane and surrounding area fare better than the provincial average on some measures of well-being, such as sense of community belonging, self-reported physical activity during leisure time, and compliance rates for vaccination of school-aged children. In comparison to provincial averages, however, the residents within the Porcupine Health Unit service area experience higher rates of obesity, alcohol use, and smoking, a lower percentage of food secure households, and higher rates of teenage pregnancy. Furthermore, residents within the Porcupine Health Unit service area have a lower life expectancy with 4.4 years less than the provincial average for men and 4.1 years less for women. Residents also have heightened risks for potentially avoidable mortality issues e.g., deaths due to smoking, excessive drinking, or injuries, at 1.6 times the Ontario average.

There is a gap in available data on First Nations on Reserve and Indigenous People living off reserve within the Porcupine Health Unit area. CNC will continue to engage and work with Indigenous Peoples to gather information on the health of Indigenous Peoples, including social determinants of health and community well-being, and how the Indigenous Peoples define these aspects.

There are no known structures, sites or objects that are of historical, archaeological, paleontological, or architectural significance to Indigenous Peoples present within the development area that could be affected by the Crawford Project. Initial preliminary desktop studies have identified areas of higher archaeological potential, mostly on the banks of watercourses. Background research for the project identified one potential cultural heritage landscape and two properties in the project area with buildings or structures more than 40 years old; however, none of these are predicted to be directly or indirectly impacted by the project (Wood, 2022b). A Stage 2 archaeological field program will be executed, if needed, to confirm the presence or absence of archaeological features and will be informed by Traditional Knowledge and Traditional Land Use studies which are currently ongoing.

#### 20.2.2.5 Hydrology

Baseline hydrology studies for the project site and surrounding area have been ongoing since 2021. The following summarizes key baseline hydrology observations and findings for data collected in 2021 and 2022.

The project area is located within the Hudson Bay watershed. The project site is mainly located in the headwaters of the West Buskegau River and North Driftwood River subwatersheds, with a small portion of the site extending into the Jocko Creek watershed. The West Buskegau River is a tributary of the Buskegau River, which discharges to the Frederick House River and then the Abitibi River. The North Driftwood River discharges directly into the Abitibi River, which flows northward before discharging into Hudson Bay via Moose River. The Mattagami River bounds the project to the west flowing northward and discharging directly into Moose River. The Lower Sturgeon Falls hydroelectric generating station (dam) is located on the Mattagami River approximately 6.5 km east of Martin Lake, downstream of the confluence with Kidd Creek, and controls the flow rate of the river downstream of the dam.

The proposed TMF location is at the southern end of the project footprint, with a small portion (1.3 $km^2$) within the Jocko Creek watershed, a tributary of Kidd Creek, which discharges to the Mattagami River. The North Driftwood watershed area, downstream of the border of the project area, is 105.5 $km^2$, while the West Buskegau River, downstream of the border of the project area, has a watershed area of 199.0 $km^2$. The Mattagami River has a watershed area of 8,363 $km^2$ downstream of the proposed effluent discharge location.

Runoff characteristics and stream flows are directly impacted by climate conditions within the project area. The project area has a mean annual temperature of 1.6°C, ranging from 17.5°C in July and -16.8°C in January and a mean annual precipitation of 842.1 mm, based on the Environment Canada and Climate Change VPA and Timmins A climate stations (Golder, 2022a). Total annual precipitation for the 100-year wet year is 1,062.33, while the 100-year dry year is 588.8 mm (Golder, 2022a).

The low-lying topography of the project area is consistent with the slow flowing and low-gradient channels observed in the baseline flow monitoring stations (Golder, 2022c; WSP, 2023b). The watercourses were observed to have high width-to-depth ratios. Brush and grasses dominated the banks and part of the soft watercourse beds with shallow root zones and active beaver dams observed along the North Driftwood River. Small trees and brush were observed along the banks of the West Buskegau River and tributaries thereof, while a beaver dam was observed on the West Buskegau River near the downstream end of the project area. Small trees along the banks and light aquatic vegetation were observed on Jocko Creek, with no beaver activity noted at the time of the surveys.

Local hydrological conditions have been assessed using the automated water level data collected within the project area. The location of the hydrometric monitoring stations was updated following the site visits in 2021 to provide a more robust dataset. Stage and discharge data were collected through hydrometric monitoring stations at five locations on the North Driftwood River, including two hydrometric monitoring stations located on Martin Lake to collect stage data within the lake and stage and discharge data at the lake outlet. Three hydrometric monitoring stations are located on the West Buskegau River (or tributaries thereof) and two stations located on Jocko Creek. Two site visits were completed in 2021 and four site visits were carried out in 2022, with manual measurements completed when safe to do so. Water level elevations, discharge, and precipitation records were acquired for the baseline study period from the Lower Sturgeon Falls hydroelectric generating station. Periods of high flow and elevated water levels were observed for the North Driftwood River, West Buskegau River, and Jocko Creek between mid-March and June, peaking at the beginning of May after the onset of spring freshet. High flows and elevated water levels were also observed in 2021 and 2022 between September and October, corresponding with an increase of precipitation during the fall. Unit runoff was estimated for stations with developed rating curves for the 2022 program and was estimated to be 6.14 L/s/$km^2$ for the North Driftwood River and range between 7.79 L/s/$km^2$ to 10.2 L/s/$km^2$ for the West Buskegau River. Jocko Creek has an estimated unit runoff of 7.74 L/s/$km^2$ (WSP, 2023b).

Additional hydrology data was collected in 2023 to supplement the baseline dataset and support the assimilative capacity of the Mattagami River, West Buskegau River, and North Driftwood River for potential effluent discharge from the project.

##### 20.2.2.6 Hydrogeology

The surficial geology of the regional area is dominated by organics overlying deposits of the Barlow-Ojibway Formation (consisting of massive to varved silts and clays) and till of the Cochrane formation and up to five other distinct till units (consisting of clayey silt till, including minor glaciolacustrine sediments, and sand and gravel) overlying bedrock (Smith, 1992). In some areas, discontinuous nearshore sand and gravel deposits are located between the surficial varved silts and clays and glacial till and/or bedrock. An esker complex, consisting of coarse sands and gravel, has also been mapped in the southwest corner of the property boundary (OGS, 2005). The discontinuous confined sand unit may be connected to the esker complex. Borehole drilling programs completed in 2021 and early 2022 by SRK, and in 2022 by WSP Golder, indicate that the overburden thickness across the project site was found to range between 9 m to over 86 m, with an average depth of 40 m (Golder, 2022a).

Bedrock is part of the Deloro Assemblage (Monecke et al., 2017). The Deloro Assemblage hosts the Crawford Ultramafic Complex (CUC) and consists mainly of mafic to felsic calc-alkaline volcanic rocks with local tholeiitic mafic volcanic units and an iron formation cap which is typically iron-poor, chert-magnetite (Ayer et al., 2005; Thurston et al., 2008). Based on geophysics and drilling, the CUC is approximately 8.0 km long by 2.0 km wide and has a northwest to southeast placement through the proposed location of the open pit. The open pit has been designed to mine the main zone and east zone of the CUC. A north-northwest trending regional sinistral strike-slip fault has displaced the main zone and the east zone (by about 1 km) which were once part of the same body.

Local groundwater flow is anticipated to follow local topography and watershed divides and may also be influenced by bedrock topography below the overburden deposits. Shallow groundwater flow in the eastern portion of the project site is interpreted to flow east towards the West Buskegau River while the western portion of the project site is interpreted to report to the North Driftwood River, in line with surface water flow. A local recharge zone exists along the western edge of the project area, along the sand and gravel deposits of the esker complex, which may act as a recharge zone for the deeper confined aquifers in the area.

Hydraulic testing was completed using slug and/or bail tests on 23 overburden monitoring wells installed during 2021 and 2022 within or near the footprint of the open pit, mill, impoundment facilities, and TMF. Existing water levels prior to the slug tests showed groundwater levels ranging between -0.64 m below ground surface to 6.89 m below ground surface within the project footprint. The hydraulic conductivities ranged from $2x10^{-4}$ m/s to $2x10^{-10}$ m/s, with a mean of $2x10^{-5}$ m/s. The screened overburden material included sand (mean hydraulic conductivity of $5x10^{-5}$ m/s), silty sand (mean hydraulic conductivity of $6x10^{-6}$ m/s), sandy silts (mean hydraulic conductivity of $1x10^{-6}$ m/s), finer-grained tills (mean hydraulic conductivity of $9x10^{-8}$ m/s), and well-sorted clays (hydraulic conductivity of $2x10^{-10}$ m/s) (WSP, 2023f). Hydraulic conductivity testing was completed on 69 bedrock intervals (via packer testing) within the vicinity of the open pit which demonstrated a slight trend of decreasing hydraulic conductivity with increased depth below top of rock surface. The geometric mean for weathered rock was calculated as $1x10^{-6}$ m/s, unweathered rock (<200 m below rock surface) as $3x10^{-9}$ m/s, and unweathered rock (>200 m below rock surface) as $6x10^{-10}$ m/s (WSP, 2022c). The bedrock hydraulic conductivity near the regional fault has a geometric mean of $1x10^{-7}$ m/s (WSP, 2022c).

Groundwater quality sampling of monitoring wells commenced in fall 2022 and data loggers were installed in select monitoring wells to capture seasonal groundwater quality and level data which will continue throughout 2023.

### 20.2.3 Environmental Sensitivities

Based on the information available to date, and understanding of the proposed development, it is anticipated that the environmental sensitivities identified will not be limiting to project development. There are species at risk that may be present in the project site and surrounding area, based on experience with other mining developments in northern

Ontario and confirmation by baseline studies. As the project design and environmental studies progress, this assumption will be confirmed and management plans will be completed with direction from the applicable regulatory agencies if species at risk are encountered during project activities.

The timing of construction activities will be arranged in accordance with the appropriate freshwater fisheries timing and breeding bird windows for the project area, unless otherwise approved by the applicable regulatory agency. Water takings during construction and operations will comply with applicable guidance from DFO to avoid entrapment and impingement of fish.

Completion of the federal impact assessment and provincial environmental assessments as well as future conditions of approval and engagement with Indigenous groups and stakeholders could require refinements to the project components or additional mitigation measures to be implemented to avoid and/or mitigate environmental sensitive features.

### 20.2.4 Future Environmental Management and Monitoring Plans

As the project is currently in the design stage, environmental monitoring plans for the construction and/or operation of the project have not been developed, although environmental baseline studies are ongoing. Environmental monitoring plans are expected to be developed for the site as part of the environmental approvals process. Monitoring activities will be developed for each stage of the project (construction, operations, closure, and post-closure), and are expected to include, but are not limited to, the following:

- Environmental Protection Plan
- Chemical and Hazardous Materials Storage and Handling Plan
- Waste Management Plan
- Contingency Plan
- Explosives and Blasting Management Plan
- Fish Habitat Offset Plan
- Water Management Plan
- Tailings Management Facility Operations, Maintenance and Surveillance Manual
- Community Cooperation Agreements
- Soil and Rock Management Plan
- Public (Stakeholder) Safety Plan
- Effluent Monitoring Plan
- Tailings/Effluent Release Emergency Response Plan
- Emergency Response and Spill Contingency Plans
- Follow-up and Monitoring Plan(s)
- Rehabilitation and Closure Plan

## 20.3 Social and Community Considerations

Consultation with Indigenous groups and stakeholders (i.e., community members, agencies, interested parties) is an integral part of the project. CNC was created at the end of 2019 and listed on the TSX Venture Exchange in early 2020 to advance the development of the project. As a new player in the socio-economic landscape of the Timmins area, CNC proactively engaged with relevant Indigenous groups and communities, and with a number of local stakeholders to introduce CNC and its intention to pursue the development of processes to allow the production of net zero carbon nickel, cobalt, and iron products from the Crawford deposit.

CNC set the basis of its stakeholder engagement strategy by hiring a vice-president of sustainability at the end of 2020 and a community relations and communications coordinator in June 2021. Initial discussions with project stakeholders began in June 2021.

CNC has established a core set of guidelines on which engagement activities have been, and will continue to be, based. These guidelines include:

- early, ongoing, and proactive engagement that is tailored to the community's interests and expectations
- engaging stakeholders by proximity to the project and relevance to potential project impacts, and providing interested stakeholders with a chance to obtain information and share feedback
- sharing project information that transparently addresses issues, concerns, and opportunities, and helps develop solutions suited to all involved parties
- taking project decisions per engineering and regulatory requirements, in addition to Indigenous and stakeholder feedback
- obtaining a plurality of perspectives from the community by reaching out to groups not often involved in mining projects.

The COVID-19 pandemic situation significantly reduced CNC's capacity to engage in person with Indigenous Peoples and the project's stakeholders. To reduce the risk of transmission of COVID-19, CNC's internal policy focused on reducing non-essential human contacts. As a result, CNC's staff optimized the use of technological tools during that period to maintain efficient communication channels with key groups and individuals, such as using videoconferences wherever possible.

As CNC advances the project, the intention of the company is to pursue engagement with local Indigenous groups and stakeholders in a comprehensive consultation process aimed at identifying and addressing significant challenges associated with the development of the Crawford Project, but also seeking to underline and optimize potential social, environmental, and economic benefits. The output of this collaborative effort is intended to be integrated in the project's impact statement. Although the range of stakeholders is expected to evolve to reflect various levels of interest and issues over time, CNC has identified local stakeholders who have or could demonstrate an interest in the project as outlined in Table 20-4.

**Table 20-4: List of Indigenous Nations, Stakeholders, and Governmental Organizations with Whom Consultation Has Commenced and/or Will Commence**

| | | |
|---|---|---|
| **Indigenous Groups** | | |
| • Apitipi Anicinapek Nation<br>• Mattagami First Nation | • Flying Post First Nation<br>• Taykwa Tagamou Nation | • Matachewan First Nation<br>• Metis Nation of Ontario – Region 3 |
| **General Public** | | |
| **Community Groups / Organizations** | | |
| • Aboriginal People Alliance of Northern Ontario<br>• Access Better Living<br>• Anti Hunger Coalition of Timmins & District<br>• Apatisiwin Employment and Training Program<br>• Canadian Mental Health Association Cochrane – Timiskaming Branch<br>• Cochrane District Social Planning Council<br>• Cochrane District Social Services Administration Board<br>• Cochrane Local Citizens Committee | • Ellevive<br>• Ininew Friendship Centre Jubilee Centre<br>• Keepers of the Circle<br>• Living Space Timmins<br>• Nature and Outdoor Tourism Ontario<br>• NORCAT<br>• North Eastern Ontario Family & Children Services<br>• Northglen Community<br>• Ojibway and Cree Cultural Centre | • South Cochrane Addiction Services<br>• The Venture Centre<br>• Timmins and Area Women in Crisis<br>• Timmins and District Multicultural Centre Timmins Community Development Committee<br>• Timmins Economic Development Housing CommitteeTimmins Native Friendship Centre<br>• Timmins Shared Safety & Well Being Committee |
| **Environmental Groups / Organizations** | | |
| • Friends of the Porcupine River Watershed<br>• Northwatch | | |
| **Economic Groups / Organizations** | | |
| • Abitibi Institute<br>• Cochrane Board of Trade<br>• Cochrane Economic Steering Board | • Iroquois Falls Economic Development Corporation<br>• Porcupine Prospectors and Developers Association<br>• Smooth Rock Falls Economic Development Corporation<br>• Timmins Chamber of Commerce<br>• Timmins Downtown Association (BIA) | • Timmins Economic Development Corporation<br>• Timmins Fur Council |
| **Educational Groups / Organizations** | | |
| • College Boreal<br>• Far Northeast Training Board / Commission de Formation du Nord-Est | • Mushkegowuk Environmental Research Centre | • Northern College |
| **Recreational Groups / Organizations** | | |
| • Arctic Riders of Smooth Rock Falls<br>• Big Water Campground<br>• Canadian Parks and Wilderness Society – Wildlands League<br>• Hardwood Lake Hunt Club | • Iroquois Falls Cross Country Ski Club<br>• Jackpine Snowmobile Club<br>• Ontario Federation of Anglers and Hungers (Northeastern)<br>• Polar Bear Riders (Cochrane) Snowmobile Club | • Porcupine Ski Runners<br>• Timmins ATV Club<br>• Timmins Snowmobile Club |
| **Government** | | |
| **Local** | | |
| • Black River-Matheson Township<br>• Town of Cochrane | • Town of Iroquois Falls<br>• City of Timmins | • Town of Smooth Rock Falls |
| **Provincial** | | |
| • Ministry of Mines<br>• Ministry of Environment, Conservation and Parks (MECP)<br>• Ministry of Natural Resources and Forestry (MNRF)<br>• Ministry of Economic Development, Job Creation and Trade | • Ministry of Transportation<br>• Ministry of Tourism, Culture, and Sport<br>• Mattagami Region Source Protection Committee<br>• Northern Claybelt Complex Conservation Reserve<br>• Porcupine Health Unit<br>• Hydro ONE | • Ontario Power Generation<br>• Ontario Northland Transportation Commission<br>• Mattagami Region Conservation Authority |
| **Federal** | | |
| • Innovation, Science and Economic Development Canada<br>• Environment and Climate Change Canada | • Impact Assessment Agency of Canada<br>• Fisheries and Oceans Canada (DFO) | • Natural Resources Canada<br>• Transport Canada |

As part of the consultation process, CNC, under advisement from a Socio-Economic Committee, comprised of social, economic, and municipal representaives, a comprehensive community contribution program that includes a local procurement policy as well as a sponsorship and donation strategy adapted to CNC's guiding principles and the needs of the communities.

CNC maintains a list of stakeholders who have been contacted throughout CNC's engagement process, though not all parties who have been contacted responded or chose to participate. CNC will continue to reach out through the environmental approvals process as appropriate to ensure key stakeholders are informed and engaged.

The following sections highlight regulatory, stakeholder, and Indigenous engagement for the project.

### 20.3.1 Regulatory Engagement

CNC met with representatives from individual provincial and federal departments and agencies through general consultation engagement and throughout the preparation of the detailed project description to support the federal impact assessment. CNC will continue to meet with provincial and federal ministries as needed through the federal impact assessment and provincial Class EA process and permitting. The regulatory authorities that have an interest in the project are identified in Table 20-4.

### 20.3.2 Stakeholder Engagement

CNC began engagement activities with stakeholders in June 2021 to share preliminary information regarding the project. A wide range of stakeholders were identified based on proximity to the project and the potentially affected areas, by specific perceived interests, and by past or current interest in similar projects or major project development. Key community and stakeholder engagement activities have included (WSP, 2022b) the following:

- Information-sharing by email regarding proposed activities, meetings, and project updates
- community newsletters (published quarterly)
- project website with a community specific page (www.canadanickel.com/sustainability), which includes general project information, project documents (including publicly available meeting reports and summarized factsheets, as they become available), and an inquiry submission form
- an email address dedicated to community relations (administered daily by the community relations and communications coordinator)
- individual and group meetings (help primarily virtually during the COVID-19 Pandemic) with stakeholders
- meeting reports produced by the consulting firm Transfert Environnement et Société following scheduled meetings, distributed to participants for validation, and shared on project website
- anonymous feedback surveys to collect stakeholder feedback on various subjects (the summary results of the feedback surveys were shared during early meetings and used in the development of the project's Preliminary Engagement Plan)

- factsheet summarizing the federal impact assessment process and how CNC will integrate it into the project's engagement process, made available at the Timmins office and on the project website
- factsheet summarizing the project's preliminary economic assessment, made available at the CNC Timmins office
- summary document for the initial project description, made available on the project website and distributed to public meeting registrants and to interested communities
- formation of a socioeconomic committee, consisting of stakeholders (chosen by demonstrated interest or expertise) and focused on identifying and discussing potential social, economic, and health impacts related to the project to jointly define and implement potential solutions, with meetings held quarterly
- formation of an environmental committee, consisted of stakeholders (chosen by demonstrated expertise) and focused on review of the Crawford Project's potential environmental impacts and planned mitigation measures, with meetings held quarterly
- formation of a workforce planning committee that brings together regional employment and training experts to discuss CNC's labour requirements and plan for community and company actions needed to address gaps and challenges in meeting this demand
- letters posted to known cabins, hunting blinds, and other evidence of activity on all the properties covered by CNC exploration projects inviting the user(s) to contact CNC for information on exploration activities and safe coordination of property use.

To date, CNC has completed numerous stakeholder meetings:

- Meetings were held in June/July 2021 to share preliminary information regarding CNC, the project, company values and objectives, and to identify how CNC should proceed with future engagement activities.
- Project baseline meetings were held in September/October 2021 to share the preliminary baseline results of the environmental studies, and to present the Preliminary Engagement Plan, its proposed tools, and estimated timeline.
- Initial project description meetings were held in May/June 2022, including two virtual public meetings, to present the initial project description and obtain feedback.

Feedback surveys were distributed to participants following the meetings to gauge stakeholder preferences for engagement activities as well as primary areas of interest/concern related to the project.

#### 20.3.2.1 Main Issues from Stakeholders

Open discussions, feedback surveys, and the presentations given during summer 2021, fall 2021, and for the initial project description, in spring 2022, are the primary sources of feedback collection to date, in addition to some comments received via the community email address and during CNC's attendance at community and industry events. Comments and concerns voiced by project stakeholders have been, and will be, taken into consideration during project design and implementation. Comments submitted to IAAC throughout the impact assessment process will be fully considered and addressed in the environmental impact statement.

Key issues raised to date by project stakeholders include the following:

- water management practices and discharge into the surrounding environment
- project’s potential impact on land use, mainly on hunting and fishing
- surface and ground water quality and flow
- project footprint and potential impacts on wildlife
- workforce requirements and early planning
- project’s potential impacts to socio-economic conditions, including housing availability and healthcare.

#### 20.3.2.2 Plan for Future Engagement

Those communication methods previously mentioned will continue to be implemented, alongside additional activities expected to include the following:

- openness to initiating discussion with newly interested groups and individuals
- interviews, focus groups, and discussions, as appropriate, to facilitate socio-economic primary research
- potentially hosting a public open house.

CNC has and will continue to pursue engagement with diverse population groups to support an understanding of unique perspectives and socio-economic conditions. To date, this has included efforts to contact organizations focused on representation of some of these populations for engagement opportunities, and approaching such organizations at community events. This engagement will support the gender-based analysis plus (GBA+) framework that will be completed as part of the impact assessment to illustrate the unique experiences (including potential impacts and benefits from the project) of diverse population groups.

### 20.3.3 Indigenous Engagement

Below are the Indigenous Peoples that have specific interest in the project, as listed in IAAC’s Indigenous Engagement and Partnership Plan, and with whom CNC has engaged:

- Apitipi Anicinapek Nation
- Flying Post First Nation
- Matachewan First Nation
- Mattagami First Nation
- Taykwa Tagamou Nation
- Métis Nation of Ontario

CNC maintains regular consultations with the Indigenous communities listed above. These consultations have been instrumental throughout the planning stages, and have involved the negotiation and implementation of the following agreements and structures:

- Early exploration agreements designed to mitigate the impact of exploration activities on traditional land and provide compensation accordingly.
- Impact assessment process agreements aimed at fostering comprehensive participation of Indigenous communities in the federal impact assessment process. These agreements outline effective communication channels and platforms for meaningful engagement while facilitating substantial capacity building within the communities, with a focus extending beyond the Crawford Project and CNC.
- Community committees that are representative of the community's diverse demographics, incorporating representatives of land users, elders, youth, and other stakeholders. These committees are created to provide support for impact assessment and engagement activities.
- Operational project agreements including impact and benefit agreements as well as mutual support agreements, to address and manage potential project impacts and opportunities.
- Business partnerships focused on key project components and infrastructure, aiming to ensure long-term tangible benefits and the expansion of economic opportunities for the involved parties.

At the time of the writing, CNC has the following agreements in place with the Indigenous groups:

- Exploration Agreements:
  - Apitipi Ancinapek Nation
  - Matachewan First Nation
  - Mattagami First Nation
  - Taykwa Tagamou Nation
- Impact Assessment Agreements:
  - Matachewan First Nation
  - Mattagami First Nation
  - Taykwa Tagamou Nation
- Relationship Building Agreement established with the MNO (Métis Nation of Ontario) Region 3
- Transmission Business Development Agreement secured with Transmission Infrastructure Partnership 1 (TIP1) a Taykwa Tagamou Nation joint venture for the construction, operation, and maintenance of the powerline required to connect Crawford to the grid.

CNC conducted information-sharing and engagement activities with Indigenous communities, which vary from community to community, depending on the stage of the relationship and specific community interests. In addition to those activities mentioned for stakeholder engagement which also apply to Indigenous engagement, CNC community specific activities include the following:

- participation in community events, including open houses and community meetings
- exploration agreements, business MOUs, IBAs, and other agreements as relevant under development, signed or upcoming, as appropriate
- formation of committees, hiring of community liaisons, and initiation of regularly scheduled meetings, as appropriate, requested, and/or included in agreements
- participation in baseline studies, including site visits, field work accompaniment, and review of baseline work plans and schedules, as appropriate, requested, and/or included in agreements
- provision of draft impact documents for review, such as sharing of the draft initial project description prior to formal submission
- community meetings led by CNC, hosted in the community when appropriate, to present the detailed project description and other updates, with opportunities for comprehensive question and answer periods
- provision of funding, support, and opportunities for participation relating to the impact assessment and baseline study programs, including Traditional Knowledge and land use, to support capacity building, information sharing, and meaningful collaboration
- sharing of maps and other material to support identification by community members, specifically elders, of culturally significant or potential archaeological sites
- sharing of job opportunities and contracts; future training opportunities and programs, job postings, and business opportunities will also be shared, with an emphasis in CNC's procurement and hiring programs placed upon Indigenous Peoples, Indigenous-owned businesses, and joint ventures
- regular reporting of environmental performance
- sponsorship and contributions to community activities and organizations, including support provided to date for sporting events/teams, powwows, and youth programs.

CNC's engagement with Matachewan First Nation, Mattagami First Nation, and Flying Post First Nation is regularly supported by the Wabun Tribal Council.

#### 20.3.3.1 Main Issues by Indigenous Peoples

Comments and concerns voiced by Indigenous Peoples will be taken into consideration during project design and implementation. Comments submitted to IAAC throughout the impact assessment process will be considered and addressed in the Impact Statement, if an impact assessment is required.

The main topics discussed to date are described below:

- training and employment opportunities, in particular opportunities for women and youth
- capacity building as it relates to participation in business opportunities
- involvement in environmental and impact assessment studies
- environmental topics, relating to transparent reporting, potential impacts to water quality and aquatic species, and potential impacts to wildlife from site activities
- project impacts on practices, activities, and ways of life, including traplines, fishing and hunting
- discretionary sharing of Traditional Knowledge.

#### 20.3.3.2 Plan for Future Engagement

CNC intends to continue engagement activities with interested Indigenous Peoples, with an emphasis on open, respectful dialogue, clear communication channels and meaningful participation. A specific plan for future engagement in connection with the impact assessment process will be designed and reviewed with Indigenous Peoples and IAAC at an appropriate time.

Main topics and objectives of future engagement activities, to occur alongside those activities already outlined above, are as follows:

- involvement of Indigenous Peoples in the environmental baseline studies process according to each community or group’s interests, expectations, and capacity for participation, with particular emphasis on the archaeology program
- validating the interpretation and use of Traditional Knowledge, land use, and socio-economic studies as provided by the Indigenous Nations in impact assessment documentation (accounted for or to be accounted for in the relevant Agreement and plans for engagement)
- identification of Indigenous land use activities through ongoing community engagement and Traditional Knowledge and land use studies, and discussion with primary community contacts and impact assessment committees, where appropriate
- information sharing by email regarding proposed activities, meetings, and project updates.

CNC will work with Indigenous Peoples to establish a mutually beneficial, cooperative, and productive relationship centred around transparent information sharing, respectful engagement, open dialogue, and meaningful partnerships.

## 20.4 Preliminary Geochemical Assessment

The geochemical characteristics of waste rock (234 samples)/ore (63 samples), tailings (4 samples) and overburden (30 samples) were assessed to evaluate the potential for acid rock drainage (ARD) and metal leaching. This characterization includes Acid Base Accounting (ABA), elemental analysis, Shake Flask Extraction (SFE), Net Acid Generation, analysis of tailings supernatant liquid, and Humidity Cell Tests (HCT). This characterization program was designed to be consistent with Canadian industry standard guidance for geochemical characterization of mine waste as described by Mine Environment Neutral Drainage (MEND, 2009).

Testing related to ARD and metal leaching potential are described as follows:

- ARD Potential – ABA quantifies the potential of a material to generate acid and to neutralize acid generated through the measurement of acid generating and neutralizing minerals. The resulting ratio of acid potential to neutralization potential is used to determine the net acid generation potential based on MEND (2009) guidelines. Net acid generation testing provides an indication of the net buffering capacity of a material by oxidizing available sulphide minerals in a sample and allowing reaction with buffering mineral phases.
- Metal Leaching Potential – SFE tests provide an indication of the soluble metals that can be readily leached from the test materials when mixed with deionized water under laboratory conditions. HCT assess material reactivity under long-term laboratory conditions. Tailings supernatant liquid was collected during pilot scale test to assess the potential process water quality. Although short- and long-term leachate and tailings supernatant parameter concentrations were compared to Ontario and Canadian water quality guidelines the test results do not directly measure the expected effluent chemistry of the material under ambient conditions. Rather, these comparisons are conducted to qualitatively identify parameters that may be present at concentrations that require further evaluation in the context of the overall site water quality.

### 20.4.1 Tailings Geochemical Assessment

#### 20.4.1.1 ARD Potential

Dunite and peridotite tailings samples were found to have low acid generation potential, having sulphur content of 0.1% or less, and a substantial surplus neutralizing potential. The non-potentially acid generating (non-PAG) classification based on ABA data is supported by long-term humidity cells results, which indicated all samples in a preliminary group tested to date produce neutral to alkaline pH leachate.

#### 20.4.1.2 Metal Leaching Potential

Geochemical characterization of tailings from two ore lithologies – dunite and peridotite – is on-going and interpretations are considered preliminary. The results for the analysis of the supernatant liquid from the dunite tailings samples indicated that cobalt, copper, nickel, and zinc supernatant concentrations were greater than the screening criteria in one or two samples, whereas chromium (VI) concentrations were greater than the screening criteria in all three samples. HCT results indicated that after the flushing of the initial process water leachate concentrations were below the screening criteria. Metal parameter concentrations in the supernatant water from the peridotite tailings were less than the screening criteria; however, HCT leachate indicate that chromium (VI) concentrations were greater than the screening criteria. Except for chromium (VI) in the peridotite HCT, which requires further testing to verify long-term leaching potential, preliminary tailings HCT results indicate that tailings have low long term metal leaching potential.

### 20.4.2 Ore, Waste Rock, and Overburden Geochemical Assessment

#### 20.4.2.1 ARD Potential

Samples were found to have low acid generation potential for waste rock and ore, with most materials having sulphur content, less than 0.1%, and substantial surplus neutralizing potential. Buffering capacity is provided primarily by silicate minerals, with negligible carbonate buffering capacity present in some lithologies. The overall non-PAG classification based on ABA data is supported by long-term humidity cells results, which indicated all samples in a preliminary group tested to date produce neutral to alkaline pH leachate and have excess neutralization potential.

Although all waste rock lithologies are expected to be non-PAG in aggregate, samples from several groups were found to have a discrepancy in classification between bulk neutralization (i.e., from silicate and carbonate minerals) potential and neutralization potential solely from carbonate minerals due to carbon concentrations at or below the method detection limit. Silicate buffering capacity is generally considered to be less conservative than carbonate buffering capacity, due to the potential for relatively slower buffering reactions in silicate minerals (MEND 2009). To provide greater certainty regarding the ARD potential of waste rock lithologies with negligible carbonate buffering capacity, net acid generation was conducted on a subset of 20 samples which reported carbonate neutralization potential below or near the method detection limit. Net acid generation testing was conducted to evaluate the available buffering capacity of samples and the ability for the silicate minerals present to neutralize any potential acidity released during the test. Samples with negligible carbonate buffering capacity were selected and included materials with a range of bulk neutralization potential values, and samples from all lithologies. All samples included in the subset for net acid generation testing demonstrated that negligible acidity was released by the test procedure, with neutral to alkaline leachate pH generated in all samples. This result indicates that despite the lack of carbonate buffering capacity in some project samples, these materials are still expected to be non-PAG. As a result, where project samples report negligible carbonate buffering capacity, bulk neutralization potential was used to classify the waste rock and ore as non-PAG.

Overburden samples are considered to have negligible potential for acid generation, based on excess carbonate buffering capacity in all but one sample.

#### 20.4.2.2 Metal Leaching Potential

Leach data indicate that for waste rock and ore samples the concentrations of several parameters (e.g., aluminum, arsenic, chromium (VI), copper, vanadium) are above screening criteria in some SFE tests. HCT leachate concentrations, however, were below the screening criteria for all parameters for all waste rock and ore samples after the initial flush. Additional HCTs will be completed on representative waste rock samples to confirm the low metal leaching potential.

Testing to assess potential metal leaching from overburden (e.g., silt, sand, clay, and till) removed during mine development is ongoing.

## 20.5 Reclamation Approach

Rehabilitation is defined as measures taken to restore a property as close to its former use or condition as practicable, or to an alternate use or condition that is deemed appropriate and acceptable by the Ontario Ministry of Mines. For mining projects, a Rehabilitation and Closure Plan is a requirement under the Ontario Mining Act Regulation 240/00.

A complete Rehabilitation and Closure Plan has not yet been developed for the Crawford Project; however, the following sections outline the rehabilitation and closure philosophies and concepts that will be used in the development of the project's Rehabilitation and Closure Plan. The Rehabilitation and Closure Plan will be drafted and finalized in consultation with provincial agencies, Indigenous communities, and public stakeholders.

As the planning and design stages of the project continue, consideration for the future closure issues and requirements will continue to be incorporated into project design. In efforts to be proactive with rehabilitation activities, the following steps will be implemented:

- disturbances of terrain, soil, and vegetation will be limited to the areas necessary to complete the required work as defined by the project
- organic soils, mineral soils, glacial till, and excavated rock will be stockpiled separately, where practicable, and protected for future use
- stabilization of disturbances will be completed to reduce erosion and promote natural revegetation
- natural revegetation will be encouraged throughout the project area
- Volume calculations during construction to ensure that sufficient soil is available for rehabilitation, while avoiding excavating and stockpiling soils in greater quantities than required. This will limit project footprint growth and closure cost impacts.
- Previous ground and surface water studies and geochemistry analysis have demonstrated there is reduced potential for metal leaching or PAG within the ore body, however sampling will continue through the rehabilitation and closure planning stage to validate this conclusion. Potential metal concentrations above screening criteria in initial flush of water through tailings and/or waste rock may be related to aluminum, arsenic, cobalt, copper, chromium (VI), vanadium, and zinc.

### 20.5.1 Phases of Reclamation

There are three key stages of rehabilitation activities that occur over the life span of a mine, which include:

- progressive rehabilitation
- closure or active rehabilitation
- post-closure monitoring and treatment or passive closure.

Progressive rehabilitation involves rehabilitation completed throughout the mine operation, prior to closure, wherever practical to do so. This includes activities that contribute to the overall rehabilitation effort and would otherwise be carried out as part of the closure rehabilitation at the end of mining life.

Closure rehabilitation involves activities that are completed after mining operation ceases, to restore and/or reclaim the project to as close to its pre-mining condition. Such activities include demolition and removal of site infrastructure, re-vegetation of disturbed areas, and other activities to achieve the requirements and goals as detailed in the project's rehabilitation and closure plan.

Once closure rehabilitation activities have been completed, a period of post-closure monitoring is required to show that the rehabilitation has been successful. The following sections describe the rehabilitation and closure concepts for each phase of reclamation.

#### 20.5.1.1 Progressive Rehabilitation

As the mine advances from development to operations and throughout the operational phase of the project, opportunities for progressive rehabilitation are possible. These include, but are not limited to, the following:

- placement of TMF overburden cover and revegetation with rehabilitation and closure of the TMF after Year 18 of operation when tailings will be placed within the open pit(s)
- progressive cover of individual lifts within the rock impoundment facility once the individual lifts have been tipped to their limits
- construction of a safety berm around the open pits
- implementation of mobile water treatment units to treat site seepage and runoff from impoundment facilities, stockpiles, plant site, and the TMF
- milling of lower value ore stockpiles prior to end of operations
- monitoring rehabilitation grading and/or scarifying disturbed areas, covering these with overburden and organic materials, where required, and seeding to promote natural re-vegetation
- placement of tailings within the open pit(s) after the Main Zone pit is fully mined
- initiation of pit flooding in the Main Zone, after it is fully mined, coinciding with placement of tailings.

#### 20.5.1.2 Active Closure Year 1 to 5

Closure rehabilitation activities will be carried out once it is no longer economical to mine, or once resources have been exhausted. Many of the closure activities will take place within the first five years of the cessation of mining and ore processing, including decommissioning the process plant and other site infrastructure. In general, the closure activities that will be completed for the site include, though are not limited to the following, and will be conducted in accordance with regulations at the time of closure:

- initiation of pit flooding in the East Zone
- construction of channels between collection ponds and the open pit to accelerate pit flooding
- dismantling and removing site buildings and surface infrastructure for re-use, disposal, or recycling at approved facilities
- removal of petroleum products, chemicals, reagents, waste, and explosives to an approved facility
- placement of remaining overburden cover on the rock impoundment facility
- placement of overburden on lower value ore stockpile laydowns and seeding

- monitoring rehabilitation grading and/or scarifying disturbed areas, covering these with overburden and organic materials, where required, and seeding to promote natural re-vegetation.

#### 20.5.1.3 Passive Closure Year 6 to Pit Flooded

Following the removal of major site infrastructure and rehabilitation of mine features the site will transition into passive closure monitoring and maintenance to confirm reclamation efforts are established and functioning as intended. During this final phase of rehabilitation and closure, emphases will be placed on water quality monitoring, the ongoing filling of the open pit with water to create a pit lake, performance monitoring of the completed closure work, and maintenance, as required. Through preliminary modelling, it is predicted that filling the open pit with water may take 60 to 80 years. In general, the closure activities that will be completed in the passive closure phase of mine life include, though are not limited to, the following:

- Rehabilitation and/or removal of remaining infrastructure, primarily related to water management and pit filling, once water quality meets criteria for discharge to the environment (i.e., ditches, ponds, water treatment, etc.)
- Pit overflow spillway will be constructed, and pit flooding will be completed
- Monitoring and maintenance will continue until the physically and chemical characteristics of the site are deemed acceptable and stable and can be closed out in accordance with applicable regulations.

### 20.5.2 Rehabilitation and Closure Strategy for Mine Infrastructure

The following provides the rehabilitation and closure strategy specific to select mine infrastructure.

#### 20.5.2.1 Site Infrastructure

Currently, it is assumed that the buildings and associated infrastructure will be removed during active closure, however some buildings will need to remain in place to support passive closure monitoring. As operations progress, buildings and infrastructure not required to support operation, will be progressively rehabilitated. Once into active closure, rehabilitation efforts will include:

- Site buildings, concrete structures, crushers, and above and below ground pipelines will be removed and either disposed of in an approved landfill facility or sold as reuse or scrap.
- Machinery and equipment will be returned to the rental vendor or sold.
- In passive closure, powerlines and transformers will be removed, including supporting infrastructure. During passive closure, buildings that remain in place will be powered by generators.
- Laydowns, parking lots, haul roads, and access roads will be scarified, graded, and revegetated during the transition into passive closure as locations are no longer required. As roads are reclaimed, culverts will be removed and disposed of off site.

#### 20.5.2.2 Petroleum Products, Chemicals, Waste and Explosives

The types of solid waste anticipated to accumulate on site over the life of the project include domestic waste, special management waste, and demolition waste.

Domestic wastes will include food waste, clothing, scrap metal, glass, plastic, and fibrous material (e.g., wood and paper). These materials will be transported off site for management, in accordance with regulations. CNC will evaluate the feasibility of segregating waste streams (e.g., domestic waste, recyclable materials) and available facilities to reduce the amount of material directed to a landfill.

Special management waste will include vehicle maintenance wastes (waste petroleum products, waste glycol and packaging), petroleum contaminated soil (in case of a spill), waste explosives, and biomedical waste. Special management wastes produced in the construction, operation, decommissioning, and closure phases of the project will be stored indoors and/or in sealed containers in an area with secondary containment until they can be appropriately transported to a licensed facility off site.

Demolition waste will include concrete, steel, wallboard, and other materials resulting from demolition activities. Salvageable machinery, equipment, and other materials will be dismantled and taken off site for sale or re-use if economically feasible.

Domestic sewage waste will be limited as there will not be a permanent accommodations complex on site. Waste will be generated from office and administrative buildings, as well as the mine dry. During the construction and operation phases of the project, domestic sewage will be treated by an appropriately sized method, such as a sewage treatment plant. Effluent discharged from the facility will be treated to meet regulatory requirements and either directed to a pond on site or discharged into the environment. Following closure, the sewage treatment facility will be demobilized from site.

Unused explosives will be returned to the manufacturer and the storage building will be demolished as part of the removal of site infrastructure.

#### 20.5.2.3 Tailings Management Facility (TMF)

The tailings that are produced from the milling process will be deposited in the TMF for the first 18 years of the project operation phase, using a thickened tailings process with no internal ponding. Once the Main Zone open pit has been mined out (anticipated by Year 17 of operation), tailings from processing ore from the East Zone will be stored within the Main Zone open pit. The tailings from the processing of lower-value ore temporarily stockpiled during mining operation will be stored in the East Zone open pit. Storage of these mine wastes within the open pit will help to reduce the overall project footprint.

The TMF impounding dams are being designed for closure in accordance with the guidance provided by the Canadian Dam Association (CDA) such that the geometry of the dams will not require modification during the mine closure phase to provide long-term stability of the facility. When the tailings deposition is moved to the open pits, in Year 18 of operation, the process of closure and rehabilitation of the TMF will commence.

After closure, covered tailings beaches are not expected to produce acidic runoff and/or have high or moderate leaching; therefore, the surface will be covered with 0.15 m of overburden prior to revegetation during operations (progressive reclamation). During progressive and active closure, a mobile water treatment plant will treat seepage prior to discharging to the North Driftwood River and/or West Beskagau River. Elevated total suspended solids will begin to stabilize once the tailings facility is reclaimed and vegetated, allowing for water treatment to halt likely in the early stages of passive closure. Once monitoring confirms water quality is stable and meets criteria for discharge, seepage collection ditches and collection ponds will be graded and vegetated to simulate the environment envisioned in the closure plan.

The regulatory landscape regarding tailings management has been changing because of significant dam failures in recent years, and it is anticipated that regulation and guidance will continue to change with respect to tailings management, closure of tailings facilities, and needed alignment with climate change. CNC is committed to working with provincial and federal regulators and following CDA guidelines so that the TMF is designed, constructed, operated, and ultimately rehabilitated, in a safe and responsible manner that will protect the environment in the long term.

#### 20.5.2.4 Open Pits

Upon closure, equipment and dewatering infrastructure will be removed, and the open pit(s) will be allowed to fill with surface water runoff, precipitation, and groundwater seepage. Runoff from rehabilitated areas from the various collection ponds will be directed to the open pit. Filling of the pits commences in operations and is preliminary forecasted to require from 60 to 80 years.

The overburden thickness in the open pit ranges between 20 to 80 m, and averages 38 m. Approximately 779 $Mm^3$ of tailings will be placed in the open pits during operation which will remain about 30 m below the top of bedrock. The tailings placed in the open pits and the pit wall rock are predicted to be non-PAG. Preliminary metal leaching rates are predicted to be result in runoff and seepage quality similar to the Provincial Water Quality Objectives (PWQO). Based on existing geochemistry data no further water treatment is anticipated in passive closure.

Once confirmed that the water quality is stable and meets criteria for discharge, a passive spillway will be constructed. A detailed assessment of the as-constructed pit geometry and spill elevation in relation to the surrounding terrain will be required to determine where the water will ultimately flow from the pits post closure. A channel(s) is expected to reconnect the open pits drainage to the natural, adjacent waterbodies. Monitoring of water quality within the open pits during filling will be completed to assess the potential discharge water quality and to determine if water treatment could be required until water quality meets the appropriate criteria.

Portions of the open pit's perimeter berm will be graded to make a shallow shoreline around the open pits to allow egress for people and animals once the site is closed out according to the *Mining Act*.

#### 20.5.2.5 Stockpiles

During construction, rock and overburden materials will be managed and sorted into high-grade ore that is sent to process and lower value ore, rock, and overburden stockpiles and impoundment facilities. Each will be managed throughout operations and into closure.

Baseline geochemical and water sampling surveys indicate that materials generated are not potentially acid generating (PAG). However, the project is located within a clay belt, where total suspended solids may exceed permitting thresholds and therefore require water management and treatment of runoff from the stockpiles. Runoff from stockpiles and impoundment facilities will be collected by ditches around the perimeter of the stockpiles and impoundment facilities which will report to a series of sediment ponds, followed by treatment through a mobile water treatment plant and either discharged to the open pit or discharged to the environment.

###### 20.5.2.5.1 Rock Impoundment Facilities

The rock impoundment facility will be established to the north of the open pit. The pile will be sloped and benched in accordance with the future closure plan design as they are developed, creating overall safe slopes for final closure of six horizontal to one vertical (6H:1V). The rock impoundment facility will also be progressively rehabilitated when areas will no longer be disturbed by material placement at which time overburden placement and vegetation can be applied. To limit erosion of the cover material, runoff channels will be incorporated into the reclamation design of the rock impoundment facility. Ditching and sedimentation ponds constructed to manage the runoff from the facility will be left in place into early stages of active closure until open pit flooding commences. At that time, water may be pumped into the pit to expedite pit flooding. Geochemistry analysis indicated that the waste rock is non-PAG and that runoff captured in seepage collection systems may meet or be proximal to the PWQO, however additional testing is ongoing. Once vegetation has established, and monitoring confirms water quality is stable and meets criteria for discharge to the environment, the seepage collection ditches and ponds will be graded and vegetated.

###### 20.5.2.5.2 Ore Stockpiles

Two stockpiles of lower value ore will be developed in early operations, as ore will be produced at a faster rate than the mill throughput will allow. Lower value ore will pass through the mill at various times throughout operations. Near the end of operations, remaining lower value ore will be processed in the mill. The former lower value ore stockpile pads will be covered with overburden, graded, and revegetated.

###### 20.5.2.5.3 Reclaim (Overburden) Impoundment Facilities

Approximately 12% of organic material and 57% of sand that will be used as cover will be delivered directly to the site being reclaimed. The remainder of organic material and overburden will be removed from various project development areas and stockpiled for progressive and final rehabilitation activities. Overburden material will be separated into a clay impoundment facility and a sand and till impoundment facility. Some overburden (suitable glacial till) may be used as a low-permeability fill material for dams, ditching, and as a base for stockpile pads to assist in drainage control. As the project design process moves forward, the volume of soils required for all rehabilitation activities will be assessed, to confirm that sufficient soils are available for rehabilitation, while avoiding excavating and stockpiling soils in greater quantities than required, avoiding an increased project footprint and soils excavation, management, and closure impacts.

As required through progressive reclamation and into closure, material will be utilized from these locations to reclaim laydowns, roads, and other disturbed areas requiring vegetation. Once the reclaimed have been covered, the final stage of passive closure will be final grading, if necessary and revegetation.

### 20.5.3 Passive-Closure and Long-Term Monitoring

The passive closure and long-term monitoring plans are not yet developed. These programs will be developed during formal closure plan development and based on the experience gained through testing and monitoring plans during construction and operation. It is anticipated that the closure monitoring plans will align with the operational monitoring program to provide continuity of data and be traceable to a historical baseline. It is also anticipated that, as the post-closure monitoring program moves forward, the monitoring requirements will decrease until, they will no longer be required. A full schedule of monitoring requirements will be outlined in the Crawford Project Closure Plan, that will be prepared and submitted to the Ministry of Mines.

Although the monitoring programs are not yet developed, it is assumed they will include physical stability monitoring of the open pit, waste rock slopes, TMF dams, and water conveyance structures on an annual basis during active and passive closure phases.

Surface water and groundwater monitoring programs initiated during operations will continue during active closure. After Year 3, when the project transitions into passive closure, the scope and frequency of monitoring will be reduced from operational monitoring. The monitoring period in passive closure will be based on achieving a flooded open pit and stable water quality that meets criteria for discharge to the environment.

Reclaimed and vegetated areas will require annual monitoring post rehabilitation until the location is successfully established, then the frequency will be reduced as targets for coverage and quality are reached.

Ongoing closure monitoring and maintenance activities will be carried out throughout passive closure until the closure requirements and objectives have been satisfied. The monitoring period will be based on the satisfaction of regulators that physical and chemical characteristics of the site are acceptable and stable. When the project is deemed physically and chemically stable and the closure objectives have been met, the project will transition to 'closed-out' status as defined under the Mining Act and a Recognized Closed Mine under the Metal and Diamond Mining Effluent Regulations (MDMER).

### 20.5.4 Financial Assurance

As defined in the *Mining Act*, a lessee shall provide financial assurance as part of a rehabilitation and closure plan prior to site development. The financial assurance amount is based on the cost estimate for the closure activities as presented in a rehabilitation and closure plan. The rehabilitation and closure plan is yet to be developed for the Crawford Project. Refer to Section 21 of this report for further closure cost details.

# 21 CAPITAL AND OPERATING COSTS

The majority of project costs (capital and operating) have been sourced in Canadian dollars, and therefore all costs in this section are presented in Canadian dollars (CAD or C$), unless otherwise noted.

Note that costs presented in the press release were in United States dollars (USD or US$), and were converted at a rate of 1 CAD = 0.76 USD.

## 21.1 Capital Cost Estimate

### 21.1.1 Overview

Table 21-1 summarizes the project capital cost estimate, with costs grouped into major scope areas. The capital cost estimate includes all costs related to the Crawford Project (e.g., local infrastructure upgrades, open pit mine development, ore processing facility, tailings management facility, high-voltage substation and power supply infrastructure, offices, maintenance shops and utilities) to support mining operations, except costs for connection to the provincial 230 kV power grid, which will be borne by TIP1 and recovered from the project as an operating cost.

**Table 21-1: Estimate Summary by WBS Level 1**

| WBS | WBS Description | Initial Capital (C$M) | Expansion Capital (C$M) | Sustaining Capital (C$M) | LOM Total Capital (C$M) | LOM Total Capital (US$M) |
|---|---|---|---|---|---|---|
| 1000 | Mining | 657 | 552 | 1,715 | 2,924 | 2,222 |
| 2000 | Process | 902 | 914 | 0 | 1,816 | 1,381 |
| 3000 | Utilities | 46 | 40 | 0 | 86 | 66 |
| 4000 | Tailings and Water Management | 129 | 111 | 136 | 375 | 285 |
| 5000 | On-Site Infrastructure | 120 | 67 | 97 | 284 | 216 |
| 6000 | Off-Site Infrastructure | 150 | 56 | 0 | 205 | 156 |
| 7000 | Indirect Costs | 244 | 174 | 0 | 418 | 317 |
| 8000 | Owner's Costs | 65 | 0 | 0 | 65 | 51 |
| 9000 | Contingency | 244 | 191 | 0 | 435 | 330 |
| | Total Capital | 2,556 | 2,105 | 1,950 | 6,611 | 5,024 |
| | Closure Costs | 0 | 0 | 175 | 175 | 133 |
| | **Total Investment** | **2,556** | **2,105** | **2,125** | **6,786** | **5,157** |

Note: Totals may not add due to rounding.

The estimate conforms to Class 3 guidelines for a feasibility study estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International). Most costs have a base date of Q4 2022, except for mining, tailings management, and water management costs, which have a base date of Q2 2023.

The estimate is based on an EPCM execution approach. The following qualifications should be noted:

- No allowance has been made for exchange rate fluctuations.
- Except for the mining, tailings management, and water management costs noted above, there is no escalation added to the estimate from the base date of Q4 2022 forward.
- A growth allowance was included.
- Data for the estimates have been obtained from numerous sources, including the following:
  - mine schedules
  - geotechnical investigations
  - feasibility-level engineering designs
  - budgetary equipment quotes from Canadian and international suppliers
  - budgetary unit costs from numerous local contractors for civil, concrete, steel, electrical, and mechanical works
  - data from similar recently completed studies and projects.

The following items were not considered in this cost estimate:

- financing charges
- residual value of temporary equipment and facilities
- environmental approvals
- further project studies
- pipeline for carbon dioxide delivery to site
- force majeure events
- future scope changes
- special incentives (e.g., schedule, safety, or others)
- strikes or other work stoppages
- management reserve above the project contingency included
- foreign exchange exposure
- land acquisition.

Working capital was also not considered in the capital cost estimate, but is included in the financial analysis (see Section 22).

As outlined in Table 21-1, the life-of-mine capital cost of the project will be approximately C$6,611 million. Costs are also shown in United States dollars with a conversion of 1 CAD = 0.76 USD.

### 21.1.2 Mining Work Breakdown Structure (WBS 1000)

Table 21-2 summarizes elements of the mining capital estimate for initial and sustaining capital expenditures. Note that installation of the trolley-assist system would begin in Year 2 of mill production, so it is excluded from the initial estimate. Sources for the estimates presented in Table 21-2 are described in the following subsections.

**Table 21-2: Summary of Mining Capital Costs**

| WBS | WBS Description | Initial Capital (C$M) | Expansion Capital (C$M) | Sustaining (C$M) | Total (C$M) |
|---|---|---|---|---|---|
| 1100 | Site Preparation | 3 | 3 | 3 | 8 |
| 1200 | Stripping | 311 | 0 | 0 | 311 |
| 1300 | Fleet | 220 | 427 | 1,376 | 2,022 |
| 1400 | Buildings and Mechanical Infrastructure | 84 | 13 | 38 | 135 |
| 1500 | Electrical Infrastructure | 40 | 43 | 34 | 116 |
| 1600 | Trolley Assist | 0 | 68 | 265 | 333 |
| **1000** | **Mining Subtotal** | **657** | **552** | **1,715** | **2,924** |

#### 21.1.2.1 Site Preparation

The estimate is based on quotations submitted by contractors and the clearing of 3,190 ha, including all land overlying the pits, plant site, water management excavations and roads. For impoundments (i.e., TMF, waste rock facility, and low-grade stockpiles), only the perimeter would be cleared.

Land would be cleared as required by the mining plan, with work extending into Year 18 of mill operation. Trees that would be cleared are of no commercial value; these would be used as corduroy support for mining excavators.

Except over the pits and water management excavations, organic material would not be removed, as this typically provides a better working surface than the underlying clay. Where organic material would be removed, the associated cost is included under stripping or operating costs.

#### 21.1.2.2 Stripping

The estimate for stripping includes work that would be performed by a contractor and the Owner.

Costs for the contractor are based on unit rates provided by contractors for specific activities the contractor would perform (i.e., early mining and hauling of waste rock from a nearby quarry or other source for use as construction aggregates; mining of clay, sand, till, and rock at site; and crushing of rock to produce sized material for construction).

Costs for the Owner have been estimated using a detailed estimate of volumes and other physical parameters for the work and the techno-economic model discussed in Section 16.3.2.

#### 21.1.2.3 Fleet

Items include the mining production fleet and associated technology, such as autonomous haulage systems, for which quotations were submitted by original equipment manufacturers (OEMs). Also included in this area is ancillary equipment such as cranes, flatbeds, and service trucks. Costs for these items are a mix of recent quotations provided by OEMs and older quotations that have been escalated to present terms.

#### 21.1.2.4 Buildings and Mechanical Infrastructure

Items include mine offices, the truckshop, and the road stone crusher, for which designs have been produced and costed. The truckshop would be expanded incrementally: 14 bays in the initial estimate, 6 bays in the expansion, and 6 final bays (for a total 26) included under sustaining capital costs.

Costs for construction of the explosives plant and fuel storage and dispensing infrastructure would be borne by the suppliers and charged back to the project as either a fixed monthly rate (by the explosives supplier) or a cost-per-litre rate (for the fuel supplier). Only the costs for establishing concrete pads and power supply are included in the capital estimate.

The pit would be dewatered in 150 m lifts, with most sumps equipped with two pumps.

#### 21.1.2.5 Electrical Infrastructure

Given the variable nature of loads from the mine, power would be supplied by a separate electrical room that is constructed in two stages. The initial stage would provide power to electrical equipment and dewatering equipment from the outset. In the second stage, the facility would be upgraded to handle the increased loads associated with the trolley-assist system after it is commissioned in Year 2 of mill operations.

Mobile substations and physical infrastructure (poles and cables) for supplying power to the various equipment, including pumps, is included under this area.

#### 21.1.2.6 Trolley-Assist System

Designs were produced and costed for 13 discrete trolley lines that would be active in the pits plus three additional lines for surface impoundments. A total of 36,455 m of trolley line would be installed, as follows:

- 17,902 m of new lines
- 18,553 m of previously installed lines to be removed and relocated.

### 21.1.3 Process Plant and Process Utilities (WBS 2000 and 3000)

The capital cost estimate for these areas includes the following scope:

- plant site earthworks, piling, concrete and structural steel
- buildings, including process buildings
- mechanical equipment (e.g., mills, drives, crushers, conveyors, flotation cells, thickeners)
- electrical equipment (e.g., electrical rooms and transformers)
- electrical and instrumentation bulks
- in-plant pipework
- pipe corridors for distribution pipelines
- water supply, air systems, control systems, and lighting.

The detailed cost breakdown associated with these areas is presented in Table 21-3. The cost breakdown per discipline is presented in Table 21-4.

**Table 21-3: WBS 2000 / 3000 – Summary of Process Plant & Utilities Capital Costs**

| WBS | WBS Description | Initial Capital (C$M) | Expansion Capital (C$M) | LOM Total Capital (C$M) |
|---|---|---|---|---|
| 2100 | Crushing | 162 | 160 | 322 |
| 2200 | Grinding Circuit | 341 | 341 | 682 |
| 2300 | Desliming | 23 | 23 | 46 |
| 2400 | Coarse Nickel Flotation | 57 | 58 | 115 |
| 2500 | Fines Nickel Flotation | 68 | 69 | 137 |
| 2600 | Magnetic Separation | 70 | 70 | 140 |
| 2700 | Concentrate Thickening, Filtration, and Stockpile | 74 | 73 | 147 |
| 2800 | Tailings Thickening, Discharge, and Return Water | 94 | 107 | 201 |
| 2900 | Reagents | 14 | 14 | 28 |
| **2000** | **Process Subtotal** | **902** | **914** | **1,816** |
| 3100 | Air Systems | 12 | 13 | 25 |
| 3200 | Water Systems | 30 | 23 | 53 |
| 3300 | Process Control System | 3 | 3 | 6 |
| 3400 | Plant Area Lighting | 1 | 1 | 2 |
| 3000 | Utilities Subtotal | 46 | 40 | 86 |
| | **Process and Utilities Subtotal (2000 and 3000 WBS)** | **948** | **955** | **1,903** |

Notes. 1. Numbers may not add due to rounding. 2. There are no sustaining capital costs included in these WBS numbers.

**Table 21-4: WBS 2000 / 3000 – Summary of Process Plant & Utilities Capital Costs by Discipline**

| Discipline Code | Discipline | Initial Capital (C$M) | Expansion Capital (C$M) | LOM Total Capital (C$M) |
|---|---|---|---|---|
| A | Architectural | 114 | 113 | 227 |
| B | Earthworks | 35 | 37 | 72 |
| C | Concrete | 139 | 138 | 277 |
| E | Electrical | 51 | 51 | 102 |
| F | Platework and Mechanical Bulks | 43 | 43 | 86 |
| I | Instrumentation | 40 | 40 | 80 |
| M | Mechanical Equipment | 334 | 334 | 668 |
| N | Plant and Miscellaneous Equipment | 9 | 9 | 18 |
| P | Pipework | 90 | 97 | 187 |
| Q | Electrical Bulks | 41 | 41 | 82 |
| S | Structural Steelwork | 52 | 52 | 104 |
| | **Total (2000 and 3000 WBS)** | **948** | **955** | **1,903** |

Note: There are no sustaining capital costs included in these WBS numbers.

#### 21.1.3.1 Estimate Sources

The capital cost estimate for the process plant included a provision for all mechanical and electrical equipment, buildings, and quantities or factors for major bulks materials such as earthworks (piling), concrete, steel, piping, instrumentation, and electrical.

All major processing equipment were sized based on the process design criteria. Once the mechanical equipment list was outlined, mechanical scopes of work were compiled and sent to suppliers for budgetary pricing. As shown in Tables 21-5 to Table 21-8, 88% of the mechanical equipment costs and 93% of the electrical equipment costs were sourced from budgetary quotations, while the remaining pricing for minor equipment was derived from recent reference projects and studies.

Installation rates and contractors' indirects were also sourced from local contractors for major construction packages such as earthworks, concrete installation, structural/mechanical installation, and field-fabricated tanks.

In support of the major mechanical and electrical equipment packages, the process plant and infrastructure engineering design were completed to a feasibility study level of definition, allowing for the bulk material quantities (i.e., earthworks, concrete, structural steel, platework, piping, electrical and instrumentation bulks) to be derived for major commodities.

**Table 21-5: Mechanical Equipment Supply Price Basis for Process Plant (WBS 2000) and Utilities (WBS 3000)**

| Source | Initial Phase Supply (C$M) | Initial Phase Supply Percentage of Total (%) | Expansion Supply (C$M) | Expansion Supply Percentage of Total (%) | LOM Total Capital (C$M) | Percentage of LOM Total Capital (%) |
|---|---|---|---|---|---|---|
| Budget Quote | 214 | 88% | 214 | 88% | 428 | 88% |
| Historical | 29 | 12% | 29 | 12% | 58 | 12% |
| Factored | 0 | 0% | 0 | 0% | 0 | 0% |
| Allowance | 0 | 0% | 0 | 0% | 0 | 0% |
| **Total** | **242** | **100%** | **242** | **100%** | **484** | **100%** |

Note: Excludes cost of freight. There is no cost associated with sustaining capital for the process plant (WBS 2000) and utilities (WBS 3000).

**Table 21-6: Mechanical Equipment Supply Price Basis**

| Package No. | Equipment |
|---|---|
| P001 | Primary and Secondary Crushers |
| P002 | Conveyors and Feeders |
| P003 | SAG and Ball Mills |
| P004 | Flotation Cells |
| P005 | Regrind Ball Mills |
| P006 | Final Tailings Thickeners |
| P007 | Cyclones |
| P008 | Concentrate Filter Press |
| P009 | Magnetic Separators |
| P010 | Water and Slurry Pumps |
| P012 | Large Agitators |
| P013 | Concentrate Thickeners |
| P014 | Screens |
| P015 | Apron Feeders |
| P016 | Cooling Towers |
| P019 | Carbon Sequestration Compressors |
| P020 | Carbon Sequestration Agitators |

**Table 21-7: Electrical Equipment Supply Price Basis for Process Plant (WBS 2000) and Utilities (WBS 3000)**

| Source | Initial Phase Supply (C$M) | Initial Phase Supply Percentage of Total (%) | Expansion Supply (C$M) | Expansion Supply Percentage of Total (%) | LOM Total Capital (C$M) | Percentage of LOM Total Capital (%) |
|---|---|---|---|---|---|---|
| Budget Quote | 39 | 93% | 39 | 93% | 78 | 93% |
| Historical | 3 | 7% | 3 | 7% | 6 | 7% |
| Factored | 0 | 0% | 0 | 0% | 0 | 0% |
| Allowance | 0 | 0% | 0 | 0% | 0 | 0% |
| **Total** | **42** | **100%** | **42** | **100%** | **84** | **100%** |

Note: Excludes cost of freight. There is no cost associated with sustaining capital for the process plant (WBS 2000) and utilities (WBS 3000).

**Table 21-8: Electrical Equipment Supply Price Basis**

| Package No. | Equipment |
|---|---|
| P201 | Outdoor Substation |
| P202 | Distribution Transformers |
| P203 | Power Factor Correction Capacitors |
| P206 | Standby Diesel Generators |
| P207 | Process Control System |
| P208 | Cable Bus |
| P209 | Integrated Electrical Rooms |
| P210 | Instrumentation |

After deriving the bulk material quantities for the process plant and utilities areas, major construction contracts were formed and tendered to experienced Canadian contractors for budgetary pricing bids, as per Table 21-9.

**Table 21-9: Construction Contract Packages**

| Package No. | Contract |
|---|---|
| P212 | Steel Supply & Fabrication |
| P501 | Earthworks |
| P502 | Structural Mechanical Platework Piping |
| P503 | Concrete Contract |
| P504 | Pre-Engineered Buildings |
| P507 | Electrical & Instrumentation Bulks Supply & Installation |
| P508 | Field Fabricated tanks |
| P509 | Other – Platework |
| P510 | Medium-Voltage Overhead Powerlines |
| P515 | Piling Contract |
| P518 | Geodesic Dome |
| P520 | Off-Plot Piping |
| P530 | Fire Safety System |
| P540 | Sewage Treatment Plant |
| P550 | In-Plant Piping |

### 21.1.4 Tailings and Water Management (WBS 4000)

A summary of the capital costs for the tailings and water management WBS area (4000) is shown in Table 21-10.

**Table 21-10: WBS 4000 – Summary of Tailings and Water Management Capital Costs**

| WBS | WBS Description | Initial Capital (C$M) | Expansion Capital (C$M) | Sustaining Capital (C$M) | LOM Total (C$M) |
|---|---|---|---|---|---|
| 4100 | Tailings Management Facility | 7 | 64 | 126 | 197 |
| 4200 | Water Management, General | 3 | - | - | 3 |
| 4211 | Water Management Excavations | 22 | 37 | 10 | 68 |
| 4212 | Water Management Backfill and Earthworks | 14 | 1 | - | 14 |
| 4220 | Water Treatment Plant | 28 | 5 | - | 34 |
| 4240 | Pumping Equipment | 32 | 3 | - | 35 |
| 4250 | Pipelines | 10 | 1 | - | 11 |
| 4200 | Water Management | 108 | 46 | 10 | 164 |
| 4400 | Capitalized Operating Costs | 13 | - | - | 13 |
| | **Subtotal Tailings and Water Management** | **129** | **111** | **136** | **375** |

Note: Numbers in table may not sum due to rounding.

#### 21.1.4.1 Tailings Management Facility (TMF), Water Management Excavation, and Water Treatment

Tailings and water management infrastructure will be excavated by the same fleet as the open pit while the TMF dyke will be constructed using waste rock, sand, and clay mined from the pit. It has been assumed that management and quality control of excavations and construction would be performed by open pit management and technical personnel, and that the engagement of an outside consultant will not be required.

Some tailings and water management earth movement costs have been classified as operating costs. Refer to Section 21.2.4 for details.

#### 21.1.4.2 Site-Wide Water Management (Excluding Earthworks and Water Treatment)

Costs for water treatment equipment and structures (WBS 4200, 4212, 4240 and 4250) include the following:

- ponds backfilling, culverts, erosion protection, culverts, diversion channels, hydroseeding, grading
- pipelines
- pumping equipment
- pumping equipment, including storage pond pumps and pumping floating systems
- pipelines conveying runoff water from collection ponds to water treatment plants and from the TMF to the process plant.

Budget pricing was sourced from the market for mechanical equipment supply and installation, electrical equipment, and bulk earthworks. Installation rates and contractor indirects were quoted by local contractors in Q4 2022.

### 21.1.5 On-Site Infrastructure (WBS 5000)

Costs for on-site infrastructure include the following:

- earthworks for process plant and non-process buildings (note: this includes bulk excavation, backfilling, piling, culverts, trenching, ditching, and topsoil stripping)
- ancillary buildings
- electrical equipment, electrical rooms, distribution transformers, 230 kV overhead powerline, and electrical bulks
- plant mobile equipment.

On-site infrastructure costs are summarized in Table 21-11.

**Table 21-11: WBS 5000 – Summary of On-Site Infrastructure Capital Costs**

| WBS | WBS Description | Initial Capital (C$M) | Expansion Capital (C$M) | Sustaining Capital (C$M) | LOM TOTAL (C$M) |
|---|---|---|---|---|---|
| 5000 | Site Maintenance | 0 | 0 | 91 | 91 |
| 5100 | Process Plant Earthworks | 25 | 17 | 0 | 42 |
| 5300 | Communications | 1 | 1 | | 2 |
| 5400 | Plant Mobile Equipment | 21 | 9 | 6 | 36 |
| 5500 | Ancillary Buildings | 12 | 2 | 0 | 14 |
| 5600 | On Site Power Distribution | 61 | 38 | 0 | 0 |
| **5000** | **Subtotal On-Site Infrastructure** | **120** | **67** | **97** | **284** |

#### 21.1.5.1 General Site Infrastructure Maintenance (WBS 5000)

Sustaining capital costs of $3.2 million per year following completion of the Phase 2 expansion have been included in the overall estimate to allow for the general maintenance of site infrastructure and facilities outside of the process plant. These costs are intended to cover the maintenance and repair of site roads, fencing, and other infrastructure. Costs are maintained, but at a lower rate of C$1.0 million per year, after the pits are depleted and the mill is fed from stockpiles.

#### 21.1.5.2 Process Area Earthworks (WBS 5100)

This area includes the costs to construct the process plant and infrastructure pads, as well as plant access roads and in-plant roads (including culverts, gates, and fencing).

All process area earthworks quantities were estimated from quantity take-offs from the model, layout drawings, and/or historical data.

Budgetary rates for earthworks construction and indirects were obtained from local contractors. The rates were then reviewed and compared, and the selected contractor's rates were used in the estimate.

#### 21.1.5.3 Mobile Equipment (WBS 5400)

This area includes the mobile equipment required for process plant and concentrate transport, as follows:

- process plant light vehicles
- locomotives
- concentrate train wagons
- magnetite railcar side dumper
- rock breaker, excavator
- front-end loaders
- railcar unloading excavator.

#### 21.1.5.4 Ancillary Buildings (WBS 5500)

Ancillary buildings included in the initial capital cost estimate include the following:

- medical clinic and fire hall
- assay laboratory (processing samples only)
- gatehouse.

Pricing for the assay laboratory was obtained from a budgetary quotation, whereas the pricing for the medical clinic, fire hall, and gatehouse was derived from recent reference projects and studies.

#### 21.1.5.5 On-Site Power Distribution (WBS 5600)

The estimate allows for the supply and installation of electrical equipment and electrical bulks for the process plant buildings, mining buildings, and site-wide power distribution.

An electrical equipment list was developed based on the mechanical equipment list, load list, single line diagrams, and general arrangement drawings. The estimate also allows for the design, supply, and construction of the above-ground power network within the process plant. This includes overhead lines, transformers, and surge arrestors.

Electrical equipment, electrical bulks, and overhead powerline pricing is a mixture of budget quotes and historical data.

Major equipment prices were solicited from vendors and the data returned was technically and commercially evaluated. The supply costs from the recommended vendor's pricing have been included in the estimate. Minor equipment has been costed using either engineering estimates or Ausenco's in-house data.

Installation of the electrical equipment and the supply and installation of electrical bulks (e.g., cables, terminations, light fittings, and cable ladder) was priced using bulk material take-offs and the electrical equipment list, which were then issued to contractors for pricing as an electrical and instrumentation package.

The returned rates were compared and evaluated, and the rates from the selected contractor have been carried forward in the estimate.

### 21.1.6 Off-Site Infrastructure (WBS 6000)

Costs for off-site infrastructure (WBS 6000) are summarized in Table 21-12.

**Table 21-12: WBS 6000 – Summary of Off-Site Infrastructure Capital Costs**

| WBS | WBS Description | Initial Capital (C$M) | Expansion Capital (C$M) | Sustaining Capital (C$M) | LOM Total (C$M) |
|---|---|---|---|---|---|
| 6400 | 500 kV Transmission Line | 83 | 0 | 0 | 83 |
| 6500 | Highway 655 | 13 | 56 | 0 | 69 |
| 6600 | Rail | 54 | 0 | 0 | 54 |
| **6000** | **Off-Site Infrastructure** | **150** | **56** | **0** | **205** |

Note: Numbers in table may not sum due to rounding.

#### 21.1.6.1 Highway and Rail Realignment (WBS 6500 and 6600)

The current corridor for Highway 655, which also includes a 500 kV powerline, bisects the property and will be realigned to the west. To aid delivery of bulk consumables to, and shipping of concentrate from, the project site, a 25.2 km railway spur will be added to the realigned corridor, which measures approximately 25.7 km.

Work will be completed in two phases, as described below.

1. During the initial project phase, the corridor will be realigned and equipped with the new railway spur. Highway 655 will temporarily remain in its current position with an overpass installed to provide access between the pit and process plant.
2. During the expansion project in Phase 2, Highway 655 will be relocated to the realigned corridor.

The capital cost estimate for the realignment and associated infrastructure includes the following:

- clearing of the new corridor, including management of soils where required
- installation of a multi-plate arch structure to allow road access between the primary crusher and the process plant underneath Highway 655
- construction of the new highway embankment (note: bulk material quantity estimates were developed from a design plan and profile developed by J.L. Richards—materials would be sourced from open pit waste and delivered to stockpiles located near the realigned corridor)

- construction of a new two-lane highway along the entire length of realignment, including ditching and culvert watercourse crossings, as well as tie-ins on both ends of the existing highway and realigned sections
- a new 25.2 km railway spur line, including a tie-in to the existing railway that terminates to the south (note: estimate includes upgrading the existing road infrastructure that will intersect the new rail corridor).

Designs have been prepared in accordance with guidelines produced by regulatory bodies, including the Ministry of Transportation Ontario (MTO) and Transportation Association of Canada (TAC) for the highway and CN's Engineering Specifications for Industrial Track Standards and the American Railway and Maintenance-of-Way for the railway.

Excavation quantities and thicknesses were estimated based on local norms, as pavement and rail designs have not been completed due to a lack of geotechnical information and analysis at this time. As such, accuracy to AACE standards for the infrastructure associated with the realignment cannot be provided at this time. Once geotechnical recommendations are available and the design is updated to reflect this information, material quantities and costs will be refined to achieve AACE Class 3 accuracy.

Unit prices and lump sum costs for the current estimate have been obtained via bids received from heavy civil contractors capable of performing highway and railway construction work. These rates are quantity-based and include all labour, equipment, and materials required to complete the work.

#### 21.1.6.2 Off-Site Power Distribution – 230 kV (WBS 6400)

No capital costs will be incurred by CNC related to the electrical transmission line, as the proposed transmission lines for both phases will be financed, constructed, owned, operated, and maintained by Transmission Infrastructure Partnerships 1 (TIP1). TIP1 will recover their investment by fees charged to CNC on a declining balance basis over 25 years that have been included under G&A operating expenditures.

#### 21.1.6.3 Off-Site Power Distribution – 500 kV (WBS 6400)

An existing 500 kV Hydro One powerline that passes through the open pit footprint must be relocated to the west of the project site (i.e., immediately west of the future relocated Highway 655). A total of 29 km of powerline as well as 68 support structures require realignment.

Costs for this relocation have been included in the initial capital estimate based on a design and estimate provided by Hydro One.

### 21.1.7 Indirect Costs (WBS 7000)

Indirect costs include the costs necessary for project completion but which are not related to direction construction costs. The following items are included as indirect costs for the process plant:

- temporary construction facilities (e.g., site offices, induction centre, first aid facilities, administration, portable toilets, temporary fencing, temporary roads, and parking)
- temporary utilities (e.g., power supply, temporary grounding and generators, construction lighting)

- construction support (e.g., site clean-up and waste disposal, material handling, maintenance of buildings and roads, testing and training, service labour, site transport, site surveys, and security)
- construction equipment, tools and supplies purchased by the owner or EPCM contractor (e.g., heavy equipment and cranes, large tools, consumables, scaffolding, and purchased utilities)
- freight associated with EPCM contractor temporary construction and services, agents, staging, and marshalling (note: freight costs for equipment and materials are included under direct costs)
- site office (local services and expenses, communications, and office furniture)
- engineering, procurement, and construction management (EPCM) costs for the process plant, including home office engineering, site and home office expenses, commissioning services and subcontractors bonding (note: EPCM costs for mining and tailings facilities are included under direct costs)
- fuel for process plant construction equipment, light vehicles, and temporary services (note: costs do not include fuel for mining development)
- spares (e.g., operating, commissioning, and strategic spares)
- process and mining first fills
- vendor representatives
- commissioning.

The indirect cost estimate was developed using a blend of first principles methods and recent historical costs. Process plant EPCM costs were estimated using the first principles method. Temporary construction facilities, temporary services, and construction equipment were developed using semi-detailed item list. The temporary construction camp bed count was developed using a built-up staffing histogram and historical contractor rental prices. Process first fill estimates were based on supplier's price submissions. Spares were based on percentages of direct equipment supply costs.

The indirect cost estimate is presented in Table 21-13.

**Table 21-13: WBS 7000 – Summary of Indirect Costs**

| WBS | Description | Initial Capital (C$M) | Expansion Capital (C$M) | LOM Total Capital (C$M) |
|---|---|---|---|---|
| 7100 | Temporary Facilities | 6 | 7 | 13 |
| 7200 | Temporary Services | 29 | 22 | 51 |
| 7300 | Accommodations | 4 | 4 | 8 |
| 7400 | Construction Equipment | 4 | 2 | 6 |
| 7500 | Freight and Logistics | 3 | 3 | 6 |
| 7600 | EPCM Costs | 152 | 120 | 272 |
| 7700 | Commissioning Support & Mod Squad | 8 | 0 | 8 |
| 7800 | First Fills – Process / Spares | 29 | 16 | 45 |
| 7900 | First Fills – Mining | 8 | 0 | 8 |
| **7000** | **Indirects Subtotal** | **244** | **174** | **418** |

Note: There are no sustaining capital costs for indirects.

### 21.1.8 Owner's Costs (WBS 8000)

The Owner's costs shown in Table 21-14 have been estimated by David Penswick using information provided by CNC. There will be no Owner's costs associated with the expansion, as these are covered under operating costs.

**Table 21-14: WBS 8000 – Summary of Owner's Costs**

| WBS No. | Description | Initial Capital Cost (C$M) |
|---|---|---|
| 8110 | Payments to TIP1* | 14 |
| 8120 | Capitalized G&A | 18 |
| 8200 | Owner's Team Logistics | 4 |
| 8310 | Recruitment | 6 |
| 8320 | Training | 2 |
| 8330 | Operational Readiness | 10 |
| 8400 | Construction Insurance | 11 |
| | **Owner's Costs Subtotal** | **65** |

Note: *Involves recovery of costs associated with connection to electrical grid.

### 21.1.9 Estimate Growth, Estimate Contingency, and Accuracy

#### 21.1.9.1 Growth Allowance

Each line item of the direct cost estimate was developed initially at a base cost only. A growth allowance has been added to line items in the process plant and site infrastructure areas to reflect the level of definition of design and pricing strategy.

The following statements apply to estimate growth:

- Estimate growth is intended to account for items that cannot be quantified based on current engineering status but empirically known to appear.
- The accuracy of quantity take-offs and engineering lists is based on the level of engineering and design undertaken at feasibility study level
- It represents pricing growth for the likely increase in cost due to development and refinement of specifications as well as re-pricing after initial budget quotations and after finalization of commercial terms and conditions to be used on the project.

Growth has been calculated by commodity and by evaluating the status of the engineering scope definition and maturity and the ratio of the various pricing sources for equipment and materials used to compile the estimate. The capital cost growth allowance for is presented in Table 21-15.

**Table 21-15: Growth Allowances**

| Discipline Code | Discipline | Growth Applied (%) | Initial Capital (C$M) | Expansion Capital (C$M) |
|---|---|---|---|---|
| A | Architectural | 6 | 8.6 | 2.1 |
| B | Earthworks | 6 | 4.6 | 2.3 |
| C | Concrete | 6 | 9.1 | 3.0 |
| E | Electrical | 6 | 5.5 | 0.7 |
| F | Platework | 7 | 2.4 | 2.1 |
| I | Instrumentation | 10 | 1.8 | 2.3 |
| M | Mechanical Equipment | 6 | 22.4 | 3.0 |
| N | Plant and Miscellaneous Equipment | 17 | 1.4 | 0.7 |
| O | Mobile Equipment | 17 | 2.8 | 2.1 |
| P | Pipework | 9 | 2.8 | 2.3 |
| Q | Electrical Bulks | 6 | 3.2 | 3.0 |
| S | Structural Steel | 6 | 3.0 | 0.7 |
| V | Third-Party Packages | 0 | 0 | 2.1 |
| | **Total** | | **67.5** | **57.3** |

#### 21.1.9.2 Contingency

Estimate contingency has been included to address anticipated variances between the specific items contained in the estimate and the final actual project cost.

Contingency is an allowance that is included, over and above the base cost, to ensure the success of the project by providing for the various uncertainties. The level of contingency will vary depending on the nature of the contract and the client's requirements. Due to uncertainties at the time the capital estimate is developed (either in terms of the level of engineering definition, the basis of the estimate, or the schedule development), it is essential the estimate includes a provision to cover risk from these uncertainties.

The amount of risk is assessed with due consideration of the preliminary level of design work, method of deriving pricing, and the preliminary nature of the plan for project implementation.

The estimate contingency does not allow for the following:

- abnormal weather conditions
- changes to market conditions affecting the cost of labour or materials
- changes of scope within the general production and operating parameters
- effects of industrial disputes.

A deterministic contingency analysis was completed to develop the contingency values. The estimate was summarized and presented for major disciplines such as concrete, steel, contractor labour, and mechanical equipment. Through experience, judgement, discussion, and reviews, the major cost components were evaluated in terms of confidence of pricing and quantity basis to provide input ranges for potential underrun/overrun. The inputs were applied as percentages to the base estimate.

A summary of contingency for the initial capital and expansion capital estimates can be found in Table 21-16 and Table 21-17, respectively.

**Table 21-16: Initial Capital – Contingency**

| Discipline Code | Discipline | Initial Capital Pre-Contingency (C$M) | Contingency (%) | Initial Capital Contingency (C$M) | Initial Capital Total (C$M) |
|---|---|---|---|---|---|
| A | Architectural | 153 | 11 | 17 | 170 |
| B | Earthworks | 81 | 20 | 16 | 97 |
| C | Concrete | 161 | 12 | 19 | 180 |
| E | Electrical | 97 | 11 | 11 | 108 |
| F | Platework and Mechanical Bulks | 43 | 12 | 5 | 48 |
| I | Instrumentation | 42 | 14 | 6 | 48 |
| M | Mechanical | 393 | 9 | 36 | 429 |
| N | Plant and Miscellaneous Equipment | 10 | 10 | 1 | 11 |
| O | Mobile Equipment | 21 | 15 | 3 | 24 |
| P | Pipework | 105 | 15 | 16 | 121 |
| Q | Electrical Bulks | 56 | 12 | 7 | 63 |
| S | Structural Steelwork | 52 | 12 | 6 | 58 |
| U | Field Indirect | 45 | 15 | 7 | 52 |
| V | Third-Party Packages / Other | 836 | 9 | 73 | 909 |
| W | EPCM | 152 | 12 | 18 | 170 |
| Y | Owner's Costs | 65 | 5 | 3 | 68 |
| | **Total** | **2,312** | **11** | **244** | **2,556** |

**Table 21-17: Expansion Capital – Contingency**

| Discipline Code | Discipline | Expansion Capital Pre-Contingency (C$M) | Contingency (%) | Expansion Capital Contingency (C$M) | Expansion Capital Total (C$M) |
|---|---|---|---|---|---|
| A | Architectural | 122 | 11 | 14 | 136 |
| B | Earthworks | 51 | 20 | 10 | 61 |
| C | Concrete | 149 | 12 | 18 | 167 |
| E | Electrical | 87 | 11 | 9 | 96 |
| F | Platework and Mechanical Bulks | 43 | 12 | 5 | 48 |
| I | Instrumentation | 40 | 14 | 6 | 46 |
| M | Mechanical | 339 | 9 | 31 | 370 |
| N | Plant and Miscellaneous Equipment | 9 | 10 | 1 | 10 |
| O | Mobile Equipment | 9 | 15 | 1 | 10 |
| P | Pipework | 97 | 15 | 15 | 112 |
| Q | Electrical Bulks | 41 | 12 | 5 | 46 |
| S | Structural Steelwork | 52 | 12 | 6 | 58 |
| U | Field Indirect | 36 | 15 | 5 | 41 |
| V | Third-Party Packages / Other | 719 | 7 | 50 | 769 |
| W | EPCM | 120 | 12 | 14 | 134 |
| Y | Owner's Costs | 0 | 0 | 0 | 0 |
| | **Total** | **1,914** | **10** | **191** | **2,105** |

### 21.1.10 Closure Costs

#### 21.1.10.1 Closure Cost Summary

The closure cost estimate makes provision for the following:

- decommissioning of the process plant and infrastructure
- reclamation and revegetation of disturbed areas, including waste dumps and the TMF, as well as the footprint of the low-grade stockpiles and decommissioned plant
- ongoing monitoring to ensure (1) run-off from the decommissioned site is on track to meet post-mining land use objectives; (2) pits, tailings, and waste rock piles are geotechnically stable; and (3) effects to receiving environment and aquatic communities are being managed
- costs associated with the placement of a closure bond.

The disturbed area that would be reclaimed, including the side slopes of waste dumps, totals 5,800 ha. Reclamation costs assume that these disturbed areas would be covered by waste material stripped from the pits. In general, rock is first covered with 0.35 m of sand before 0.15 m of organic material. Non-rock surfaces, such as those that will be encountered at the TMF and footprints of the decommissioned plant site and low-grade stockpiles, require only 0.15 m of organic material. A total of 9.2 $Mm^3$ sand and 8.7 $Mm^3$ organic material will be required for cover. When possible, cover material would be transported directly without stockpiling, with 57% of sand and 12% of organic tipped as run-of-mine material. The cover would be hydroseeded to facilitate new growth.

Closure costs would be expended as the various plants and infrastructure are closed. For example, impoundment of tailings in the TMF is completed in Year 17, at which time impoundment transitions to the mined out Main Zone pit and closure of the TMF commences. However, a bond to cover closure expenses must be placed prior to this time. The current approach is for the bond to be updated in five-year increments. The bond amount must satisfy requirements to decommission and reclaim all activities that will be performed during that five-year window, along with the cumulative amount of previous five-year windows less any actual closure expenditures.

As summarized in Table 21-18, closure expenditures will total C$142 million; however, with the reclamation work performed in a progressive manner before ultimate closure of the plant site in Year 42, the peak bond amount is C$91 million. A surety would be put in place to cover the bond with the assumed premium of 1.13% being the midpoint of a range of 0.75% to 1.50% suggested by a financial institution.

**Table 21-18: Closure Expenses**

| Activity | Total (C$M) | Estimating Scope |
|---|---|---|
| Decommissioning – Phase 1 Process infrastructure | 24 | Ausenco |
| Decommissioning – Phase 2 Process infrastructure | 21 | Ausenco |
| Reclamation | 84 | Dave Penswick |
| Monitoring | 13 | Ausenco |
| Subtotal | 142 | |
| Premium Expenses for Bond Surety | 33 | Dave Penswick |
| **Total** | **175** | |

#### 21.1.10.2 Process Plant and Building Closure Costs

Closure costs for the process plant structures, equipment, and site buildings were compiled based on the preliminary closure plan (see Section 20). These are summarized in Table 21.18 above.

The closure cost estimate allows for demolition and removal of the following areas for the Phase 1 and Phase 2 processing facilities:

- crushing infrastructure, including conveyors, and the crushed ore stockpile, including foundations
- process plant equipment and buildings, including foundations
- ancillary buildings, including truckshop, offices, and warehouses
- site water management pumps and pipelines
- fuel storage farm and explosives storage facilities
- powerlines
- switchyard.

Additional costs have been included for the following:

- contaminated site assessment (no costs allocated for contaminated site remediation)
- contractor mobilization, demobilization, and indirects
- project management
- 15% contingency.

#### 21.1.10.3 Post-Closure Monitoring Costs

Post-closure monitoring costs were compiled based on the preliminary requirements outlined in the project description. These are summarized in Table 21.18 above.

The monitoring costs include the following:

- site maintenance, including maintaining erosion control, inspection, and maintenance of surface water ditches, and infill planting
- surface water and groundwater monitoring
- aquatic effects monitoring
- TMF safety inspections, reviews, and replacement of instrumentation as required
- open pit and rock stockpiles geotechnical inspections and reviews
- vegetation reclamation monitoring
- 15% contingency.

Post-closure monitoring costs assume a 100-year post-closure period and apply a 4% discount rate beginning in the first year of closure.

## 21.2 Operating Cost Estimate

### 21.2.1 Summary

Operating costs for the open pit were based on the production schedule, performance parameters for mining equipment as recommended by OEMs, the current cost of key consumables from supplier quotations, regional benchmark costs for other commodities, and labour rates for the Timmins region, as determined from a salary survey. Rates from the salary survey were also used as the basis for labour costs in the process and G&A areas.

Operating costs for tailings and water management were estimated in the same manners as the open pit.

Operating costs for the concentrator were based on rates of consumption for reagents and other consumables determined from metallurgical testwork and a labour structure that is appropriate for the current flowsheet.

General and administrative (G&A) costs were based on the level of support required for the operation, including an organizational chart provided by the Owner.

Average costs for the following three phases of operation are summarized below and in Table 21-19:

- Phase 1 covers the initial 42 months of operation when the nameplate capacity of the mill will be 60 kt/d.
- Phase 2 covers the following 26 ½ years of operation when nameplate capacity of the plant will be 120 kt/d and feed will predominantly come from the pits.
- Phase 3 covers the final 11 ¼ years of operation when nameplate capacity of the plant will be 120 kt/d and feed will be entirely sourced from low-grade stockpiles on surface.

Note that costs for Phases 1 and 2 include the impact of ramp-up to the steady-state production rates while Phase 3 includes the impact of a ramp-down at the end of project life.

**Table 21-19: Summary of Operating Costs**

| Area | Units | Phase 1 | Phase 2 | Phase 3 | LOM Average |
|---|---|---|---|---|---|
| Mine – Tonnes Mined | C$/t mined | 2.22 | 1.82 | 0.00 | 1.96 |
| Mine – Tonnes Milled | C$/t milled | 12.92 | 8.17 | 0.81 | 6.29 |
| Tailings and Water Management | C$/t milled | 0.51 | 0.22 | 0.19 | 0.22 |
| Process | C$/t milled | 6.99 | 6.81 | 6.83 | 6.82 |
| G&A | C$/t milled | 2.58 | 1.10 | 0.47 | 0.98 |
| **Site Costs** | **C$/t milled** | **23.00** | **16.30** | **8.30** | **14.32** |

### 21.2.2 Key Assumptions in Estimate

The bulk of prices for goods and services were obtained in Q2 2023. Prices in Canadian dollars obtained prior to the cost basis date and that are specific to the mining industry (i.e., related to equipment or for reagents) have been escalated to Q2 2023 terms using the reported Canadian producer price index (PPI) for the sector (primary non-ferrous metal products). Other prices in Canadian dollars have been escalated using the consumer price index (CPI). Prices obtained in United States dollars prior to the cost basis date are all specific to the mining industry and have been escalated to these terms using the reported US PPI for the sector (WPU 010).

Labour costs were estimated based on the organizational structure developed for each area. The rates of pay are based on wages and benefits at existing mining operations in the Timmins region of Ontario as detailed in a salary survey provided by Lincoln Strategic International.

Electricity prices have been based on a 24-month average (August 2021 to July 2023) and consider the demand profiles for both peak times of Monday to Friday, 8h00 to 18h00, as well as off-peak times. Prices also recognize that the project's demand will make it eligible to participate in the Industrial Conservation Initiative (ICI) and pay a share of the global adjustment (GA) based on actual peak demand as a percentage of total Ontario demand. The life-of-mine share was calculated on the forecast year of peak demand (2040, where a daytime peak of 208 MW is expected) and Ontario's current maximum demand of approximately 22 GW and is thus conservative. The forecast life-of-mine average total price for electricity, taking account of charges for consumption, demand, and the GA, is C$75/MWh.

Fuel prices are based on an assumed oil price (Brent) of $70/bbl. It has been assumed that during project construction, Crawford will pay carbon taxes on diesel and gasoline consumed, with prices ramping up to C$1.31 and $1.30/litre, respectively, as the carbon tax increases. With the start of mill production, Crawford's net negative status will obviate the requirement to pay these taxes and prices will fall to C$0.97/litre for diesel and C$1.02/litre for gasoline. It has been assumed that all diesel equipment will require diesel exhaust fluid (DEF), and that the average consumption rate will be 3.5% that of the fuel.

### 21.2.3 Mine Operating Capital Costs

A summary of mining costs by function and area is provided in Table 21-20 and Table 21-21, respectively. It should be noted that the forecast mining costs for Crawford may be considered low relative to some existing large-scale Eastern Canadian open-pit hard rock mines, but can be explained by the following factors:

- Crawford is located close to Timmins (approximately 42 km from downtown), so a camp and associated labour premium will not be required. It is also sufficiently remote that no constraints are expected to be placed on the operation in terms of mitigating noise, vibration, or dust.
- The ultramafic rocks (i.e., dunite and peridotite) that host Crawford ore have include very low abrasion indices, which will result in a lower consumption of ground engaging tools (GETs).
- The mechanical properties of the ore (which will be crushed) and scale of equipment that will be used for mining waste rock will make it possible to blast all Crawford rock with a relatively low powder factor, leading to lower costs for drilling and blasting.
- The geometry of mineralization will allow for a highly productive bulk mining method. Over 60% of the total material will be mined on 15 m benches using rope shovels that will load 290 t payload class trucks in three-passes.
- The 290 t trucks hauling 90% of the total material will use a trolley-assist system, which will reduce energy and maintenance costs while increasing productivity on uphill hauls, as well as an autonomous haulage system (AHS), which will increase utilization and speed while simultaneously reducing maintenance costs.

Notably, the costs summarized in Tables 21-20 and 21-21 assume steady-state levels of efficiency will not be achieved from the outset, but will be achieved following a 36-month learning curve for manually operated equipment and after 18 months for the autonomous haulage system.

**Table 21-20: Mining Operating Costs by Function**

| Activity | Units | Total | Capitalized [1] | Expensed | % of Total Expensed |
|---|---|---|---|---|---|
| Total Mined | Mt | 5,707 | 103 | 5,604 | 98.2% |
| Contractor | C$M | 89 | 74 | 15 | 0.1% |
| Owner by Process: | | | | | |
| Drilling and Blasting | C$M | 1,582 | 17 | 1,565 | 15.4% |
| Loading | C$M | 987 | 28 | 959 | 9.4% |
| Hauling | C$M | 5,363 | 115 | 5,248 | 51.7% |
| Stockpile Rehandle [2] | C$M | 284 | 1 | 283 | 2.8% |
| Revegetation | C$ M | 84 | 84 | 0 | 0.0% |
| Services [3] | C$M | 774 | 30 | 743 | 7.3% |
| Maintenance Labour | C$M | 1,157 | 30 | 1,127 | 11.1% |
| Management, Technical, and Administration | C$M | 863 | 15 | 848 | 2.1% |
| **Total** | **C$M** | **11,182** | **394** | **10,787** | **100.0%** |
| $/t mined | C$/t | 1.96 | 3.82 | 1.92 | |

Notes: 1. Includes capitalized stripping and life-of-mine costs for revegetation. 2. Includes stockpiles of lower value material and construction materials. 3. Includes road maintenance, pit dewatering, and other infrastructure maintenance.

**Table 21-21: Mining Operating Costs by Area**

| Area | Units | Total | Capitalized [1] | Expensed | % of Total Expensed |
|---|---|---|---|---|---|
| Total Mined | Mt | 5,707 | 103 | 5,604 | 98.2% |
| Contractor | $M | 89 | 74 | 15 | 0.1% |
| Owner by Area: | | | | | |
| Labour Cost | C$M | 2,441 | 119 | 2,322 | 21.5% |
| Consumables | C$M | 2,331 | 79 | 2,252 | 20.9% |
| Maintenance | C$M | 3,240 | 54 | 3,186 | 29.5% |
| Fuel | C$M | 2,168 | 62 | 2,107 | 19.5% |
| Power | C$M | 700 | 3 | 697 | 6.5% |
| Other | C$M | 212 | 4 | 208 | 1.9% |
| **Total** | **C$ M** | **11,182** | **394** | **10,787** | **100.0%** |
| Unit Rate | $/t Rock | 1.96 | 3.82 | 1.92 | |

Notes: 1. Totals may not sum due to rounding. 2. Includes capitalized pre-stripping and LOM costs for revegetation.

Contractor mining represents less than 1% of operating costs, as the plan is to transition to Owner mining as quickly as possible. The contractor would be used to assist with surge requirements and/or specialist activities such as stripping clay. Contractor costs are based on rates provided by a local contractor.

As Crawford staff will be able to live in Timmins and not commute to a camp, it was considered unnecessary to provide premium compensation. Labour rates have been based on the midpoint (50th percentile) of the salary survey except for specialist functions, such as maintenance personnel associated with the autonomous systems, where the 75th percentile has been selected. The key consumables include:

- Explosives and accessories – Powder factors for different lithologies have been calculated using simulations performed by an explosives OEM and the desired product size. These range from 0.22 kg/t to 0.32 kg/t and average 0.245 kg/t. Prices have been provided by OEMs.
- Tires – The expected life of tires is based on guidance from a tire OEM and takes account of the planned use of AHS (which will reduce operator error, which is a leading cause of premature failure) along with inclusion of a roadstone crusher in the project scope (continually resurfaced roads will extend tire life). The 53.80R63 tires that will be used on the 290 t haul trucks are expected to achieve an average life of 7,200 hours.
- Ground engaging tools – The expected life and costs have been provided by equipment OEMs.

The cost of maintenance spares for most units is based on typical rates of cost per operating hour. One item that is tracked discretely is the 290 t haul truck engines, for which a life of 4.164 ML has been assumed. This life is based on the use of AHS; for conventionally operated machines, life falls to 3.785 ML (1 million gallons).

Fuel consumption for the various classes of haul truck has been estimated based on rates of consumption provided by OEMs that differentiate between status (empty / full) and profile type (flat / uphill / downhill). With the use of trolley-assist, the 290 t class haul trucks are responsible for over 70% of total fuel. Without trolley, consumption by these units would almost double.

The impact of power factor has been included in the calculated cost of electrical power.

### 21.2.4 Tailings Operating Costs

Tailings and water management costs allocated to operating expenses are listed in Table 21-22. Note that the capital costs of TMF and water management earthworks include a proportional distribution of capitalized operating costs associated with the maintenance of equipment used to perform the earthworks.

**Table 21-22: Tailings and Water Management Operating Costs by Function**

| Function | Units | Total | Initial Capital | Expansion Capital | Sustaining Capital | Operating Expense |
|---|---|---|---|---|---|---|
| TMF Earthworks | C$M | 222 | 8 | 64 | 126 | 23 |
| Water Management Earthworks | C$M | 76 | 25 | 37 | 10 | 5 |
| Water Treatment | C$M | 339 | 9 | 0 | 0 | 329 |
| TMF Operation | C$M | 19 | 0 | 0 | 0 | 19 |
| **Total** | **C$M** | **655** | **42** | **101** | **136** | **377** |
| Unit Rate | C$/t ore | 0.38 | 0.02 | 0.06 | 0.08 | 0.22 |

### 21.2.5 Process Operating Costs

Processing costs are estimated based on a milling rate of 60 kt/d for Phase 1 and 120 kt/d for Phases 2 and 3. The operating costs are separated into the categories of labour, power, consumables, maintenance materials, mobile equipment, and laboratory and assays. A breakdown of the process operating costs is presented in Table 21-23 and a description of how the costs were derived is provided in the following subsections.

**Table 21-23: Average Annual Process Operating Cost**

| Area | Phase 1 (C$/t) | Phase 2 (C$/t) | Phase 3 (C$/t) |
|---|---|---|---|
| Labour | 0.54 | 0.43 | 0.43 |
| Power | 2.86 | 2.84 | 2.84 |
| Consumables | 2.97 | 2.95 | 2.96 |
| Maintenance Materials | 0.48 | 0.45 | 0.46 |
| Mobile Equipment | 0.02 | 0.02 | 0.02 |
| Laboratory and Assays | 0.11 | 0.11 | 0.11 |
| **Total** | **6.99** | **6.81** | **6.83** |

#### 21.2.5.1 Labour

Staffing was estimated by benchmarking against similar North American base metal projects. The labour costs incorporate requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay laboratory, and contractor allowance.

The total operational labour averages 96 employees for Phase 1 and 168 for Phases 1 and 2. Salaries were provided by CNC, who performed a salary survey for each expected role. CNC also confirmed the specific benefits and bonuses to be allocated.

Rates were estimated as overall, including all burden costs. An organizational staffing plan outlining the labour requirement for the process plant is shown in Table 21-24.

**Table 21-24: Process Operation and Maintenance (O&M) Staffing Plan**

| Area | Phase 1 | | | Phases 2 and 3 | | |
|---|---|---|---|---|---|---|
| **Labour / Contractor Summary** | **#/Shift** | **# Shifts** | **Quantity** | **#/Shift** | **# Shifts** | **Quantity** |
| Metallurgy | | | | | | |
| Mill Manager | 1 | 1 | 1 | 1 | 1 | 1 |
| Secretary/Assistant | 1 | 1 | 1 | 1 | 1 | 1 |
| Chief Metallurgist | 1 | 1 | 1 | 1 | 1 | 1 |
| Production Metallurgist | 2 | 1 | 2 | 2 | 1 | 2 |
| Metallurgical Laboratory Supervisor | 1 | 1 | 1 | 1 | 1 | 1 |
| Metallurgical Technicians | 2 | 1 | 2 | 3 | 1 | 3 |
| Mill Trainer | 2 | 1 | 2 | 2 | 1 | 2 |
| Production (Operations) | | | | | | |
| Shift Supervisor | 1 | 4 | 4 | 1 | 4 | 4 |
| Control Room Operator | 1 | 4 | 4 | 2 | 4 | 8 |
| Crushing Operator | 1 | 4 | 4 | 2 | 4 | 8 |
| Grinding Operator | 2 | 4 | 8 | 4 | 4 | 16 |
| Flotation/Deslime/Magnetic Separation Operator | 2 | 4 | 8 | 4 | 4 | 16 |
| Thickening/Filtration Operator | 2 | 4 | 8 | 4 | 4 | 16 |
| Reagents Operator | 2 | 4 | 8 | 4 | 4 | 16 |
| Concentrate Loadout Operator | 2 | 4 | 8 | 4 | 4 | 16 |
| Training/Annual Leave/Day Services | 4 | 1 | 4 | 8 | 1 | 8 |
| Track Mobile Operator | 1 | 1 | 1 | 1 | 1 | 1 |
| Mill Maintenance | | | | | | |
| Maintenance Superintendent | 1 | 1 | 1 | 1 | 1 | 1 |
| Maintenance General Supervisor | 1 | 1 | 1 | 2 | 1 | 2 |
| Chief Electrician | 1 | 1 | 1 | 1 | 1 | 1 |
| Maintenance Planner | 2 | 1 | 2 | 3 | 1 | 3 |
| Electrical Engineer | 1 | 1 | 1 | 1 | 1 | 1 |
| Mechanical Engineer | 1 | 1 | 1 | 1 | 1 | 1 |
| Electricians | 3 | 2 | 6 | 5 | 2 | 10 |
| Millwrights | 4 | 2 | 8 | 8 | 2 | 16 |
| Welders | 4 | 1 | 4 | 6 | 1 | 6 |
| Instrumentation Technicians | 3 | 1 | 3 | 6 | 1 | 6 |
| Lineworker/High-Voltage Switching | 1 | 1 | 1 | 1 | 1 | 1 |
| **Total** | **50** | **54** | **96** | **80** | **54** | **168** |

#### 21.2.5.2 Power

The processing power draw was based on the nominal power utilization of each motor on the electrical load list for the process plant and associated services. Power will be supplied by the Hydro One grid to service the facilities at the site. Power consumption rates of C$0.066/kWh and annual power demand cost of C$64,440/MW were provided by CNC.

#### 21.2.5.3 Consumables

Individual reagent consumption rates were estimated based on the metallurgical testwork results, industry practice, and peer-reviewed literature. All reagent costs were obtained from reagent vendors in Q2 2023. A detailed description of the reagents required for the process is provided in Chapter 17.

Other consumables (e.g., liners for the primary crusher, SAG mill, ball mill, and ball media for the mills) were estimated using metallurgical testwork results (abrasion) and modelling simulations to forecast nominal power consumption.

A breakdown of consumables by area is listed in Table 21-25, and consumables by type are listed in Table 21-26.

**Table 21-25: Consumables Cost by Area**

| Area | Phase 1 (C$/t) | Phase 2 (C$/t) | Phase 3 (C$/t) |
|---|---|---|---|
| Crushing | 0.06 | 0.06 | 0.06 |
| Grinding | 0.97 | 0.96 | 0.97 |
| Rougher Flotation and Regrind | 1.19 | 1.18 | 1.19 |
| Cleaner Flotation | 0.42 | 0.42 | 0.42 |
| Magnetic Separation | 0.19 | 0.19 | 0.19 |
| Sulphide Flotation | 0.03 | 0.03 | 0.03 |
| Thickening and Filtration | 0.11 | 0.11 | 0.11 |
| **Total** | **2.97** | **2.95** | **2.96** |

**Table 21-26: Consumables Cost by Type**

| Area | Phase 1 (C$/t) | Phase 2 (C$/t) | Phase 3 (C$/t) |
|---|---|---|---|
| Mantle, Concaves, etc. | 0.06 | 0.06 | 0.06 |
| Grinding Media | 0.56 | 0.57 | 0.58 |
| Mill Liner | 0.34 | 0.34 | 0.35 |
| Reagents | 2.01 | 2.05 | 2.07 |
| **Total** | **2.97** | **2.95** | **2.96** |

#### 21.2.5.4 Maintenance

Annual maintenance consumable costs were calculated based on a total installed mechanical capital cost by area using a weighted average of 2% to 4.5%. The total maintenance consumables operating cost is C$0.44/t milled or approximately 7% of the total process operating cost.

#### 21.2.5.5 Laboratory and Assays

Operating costs associated with laboratory and assay activities were estimated according to the anticipated number of assays per day and per year. Assay costs include daily sampling/assaying and additional counts for sampling campaigns and re-assays. The laboratory and assays comprise approximately 0.6% of the total process operating cost, and the forecasted annual requirement for internal assays will be around 18,400 for Phase 1 and 36,800 for Phases 1 and 2 for the processing plant.

Assay costs for mine grade control or exploration samples are not included in these costs; mining sampling is accounted for separately under mining operating costs.

#### 21.2.5.6 Mobile Equipment

Vehicle operating costs are based on a scheduled number of light vehicles and mobile equipment. Included in the costs are fuel, maintenance, spares and tires, annual registration, and insurance fees.

### 21.2.6 General and Administrative (G&A) Operating Costs

The estimated cost for G&A expenses is based upon the level of service required for the size of Crawford's operation and accounts for existing local services. These costs are summarized in Table 21-27.

Key elements include the following:

- compensation for a G&A complement that reaches a maximum of 68 full-time equivalents (FTEs) during Phases 1 and 2 of operation
- transportation of employees to the site (note: costs assume the entire complement would be bussed from Timmins)
- costs associated with training employees and providing personal protective equipment (PPE), as well as information technology (IT) and communications equipment
- costs related to an annual audit and insurance
- site-wide contracts for security, cleaning, and garbage disposal
- recovery of the capital cost associated with connecting to the electrical grid by TIP1, over a 25-year period.

**Table 21-27: General and Administrative (G&A) Costs**

| Description | Units | Phase 1 | Phase 2 | Phase 3 | LOM Average |
|---|---|---|---|---|---|
| Average Total Complement | FTE[1] | 1,165 | 867 | 309 | 740 |
| G&A Complement | FTE[1] | 68 | 68 | 52 | 64 |
| Labour Directs[2] | C$M/a | 8 | 8 | 6 | 7 |
| Employee Transport | C$M/a | 7 | 5 | 3 | 5 |
| Other Labour Indirects[3] | C$M/a | 11 | 6 | 2 | 6 |
| Regulatory[4] | C$M/a | 2 | 2 | 2 | 2 |
| Security, Cleaning, and Garbage Disposal | C$M/a | 2 | 2 | 2 | 2 |
| TIP1 Charge | C$M/a | 19 | 19 | 0 | 14 |
| Other | C$M/a | 6 | 6 | 6 | 6 |
| **Total** | **C$M/a** | **54** | **48** | **20** | **41** |
| | C$/t milled[5] | $2.46 | $1.09 | $0.46 | $0.98 |

Notes: **1.** Full-time equivalent**. 2.** Compensation for G&A personnel. **3.** Includes IT, communications equipment, personal protective equipment, training, and recruitment. **4.** Includes audit and insurance**. 5.** Unit rate at steady-state production rates of 60 kt/d (Phase 1) and 120 kt/d (Phases 2 and 3).

# 22 ECONOMIC ANALYSIS

## 22.1 Forward Looking Information

This economic analysis of the feasibility study represents forward-looking information that is subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements include, but are not limited to, the following:

- timing and conditions of permits required to initiate project construction, sustain operations, and perform mine closure
- assumptions regarding geotechnical and hydrogeological factors
- time required to develop the project
- estimation and realization of the mineral resource estimates within the Feasibility Study mine plans
- assumptions regarding mine dilution and losses along with the associated grade and quantity of ore delivered to the mill
- metallurgical recovery rates
- forecast production rates and amounts of payable metal produced
- operating costs associated with the mine, mill, tailings and water management, and G&A
- initial, expansion, and sustaining capital costs
- costs associated with closure, including decommissioning, reclamation, and monitoring
- expected future prices of Ni, Co, Pd, Pt, Fe and Cr, along with the realized price for carbon sequestration.

All the above will impact the timing and amount of future cash flows

## 22.2 Summary

The economic analysis of Crawford (Table 22-1) focuses on the following base case scope. All amounts are in United States dollars (USD, US$) except where otherwise noted:

- Construction to begin mid-2025 and lead to commissioning of the initial 60 kt/d plant in Q4 2027. A project to double production would then commence in mid-2029 and result in first incremental production 24 months later.
- The plant will process 1,715 Mt ore extracted from two pits mined sequentially. After the Main Zone is depleted in Year 17, impoundment of tailings will transition from the TMF to in-pit deposition. After the East Zone is depleted in Year 30, the plant will be fed for the remaining 11 years of project life from stockpiled lower value ore on surface.

- Crawford will produce two concentrates: a nickel concentrate containing payable cobalt, palladium, and platinum, and an iron concentrate containing payable nickel, chromium, and cobalt. Both concentrates would be sold free on board (FOB) Crawford.

The base case analysis includes the Clean Technology Manufacturing (CTM) Investment Tax Credit (ITC) that was outlined during the 2023 federal budget presentation. While it is anticipated that Crawford would also qualify for the Carbon Capture, Utilization and Storage (CCUS) ITC, until approval to receive the credit has been obtained, this will be included as an opportunity and is discussed under Section 25.

**Table 22-1: Crawford Feasibility Study Summary Metrics**

| Item | Unit | Value |
|---|---|---|
| Ore mined | Mt | 1,715 |
| Payable Ni | Mlbs | 3,130 |
| Payable NiEq | Mlbs | 4,961 |
| Net Smelter Return (NSR) | $/t ore | 28.08 |
| Site Operating Costs | $/t ore | 10.88 |
| Net C1 Costs[1] | $/lb Ni | 0.39 |
| EBITDA | $/t ore | 16.04 |
| Peak Funding Requirement[2] | $M | 1,898 |
| Total Investment[3] | $M | 5,157 |
| Net AISC[4] | $/lb Ni | 1.54 |
| Post-Tax $NPV_{8\%}$ | $M | 2,475 |
| Post-Tax IRR | % | 17.1 |

Notes: 1. C1 costs include site operating expenditures. 2. Peak funding represents the cumulative unlevered investment prior to generation of positive cash flow. 3. Total investment includes all capital and closure expenses. 4. All-in sustaining costs include C1 costs, royalties, IBA, sustaining capital and closure expenses.

## 22.3 Assumptions

Key price and payability assumptions for Crawford products include the following:

- A long-term nickel price of $21,000/t, based on forecasts provided by the price reporting agency, Fastmarkets. Based on proposed terms from prospective downstream processors and publicly disclosed terms for other producers, a payability of 91% for nickel contained in both the nickel and iron concentrates is assumed.

- A long-term cobalt price of $40,000/t, based on consensus forecasts from analysts. Based on proposed terms from prospective downstream processors and publicly disclosed terms for other producers, a payability of 60% for cobalt contained in nickel concentrate is assumed.

- Long-term prices of $1,350/oz and $1,150/oz for palladium and platinum, respectively, based on consensus forecasts from analysts. Payability for both metals is based on an assumed 1 g/t combined palladium and platinum deduction and averages of 75% for palladium and 76% for platinum.
- An equivalent long-term price for iron ore concentrate of $89/t, based on forecast scrap prices of $325/t provided by Fastmarkets and a payability of 50%.
- A long-term chromium price of $3,858/t, based on forecasts from Fastmarkets. Based on proposed terms from prospective downstream processors, a payability of 65% for chromium contained in iron concentrate has been assumed.
- Based on a study produced by a leading strategy house, a long-term price of C$25/t for third-party $CO_2$ captured and stored.
- Based on consensus forecasts from analysts, a flat, long-term Canadian dollar exchange rate of $0.76

Assumptions used in the pricing of energy include:

- A long-term oil price (Brent) of $70/bbl, exchange rate of C$1.00 = $0.76, regression of recent (March 2021 to September 2023) rack rates and current excise taxes to arrive at pre-carbon tax, long term prices for 87 octane gasoline and ULS diesel of C$1.02/L and C$0.97/L, respectively. Carbon taxes have been assumed to increase as currently legislated, from the current rate of C$0.17/L to C$0.45/L by 2030. Crawford would pay carbon taxes while under construction. Once in production, an average of 14% of the life-of-mine capacity to capture and store $CO_2$ would be allocated to site emissions to ensure Crawford does not pay carbon taxes.
- The electricity price is based on a 24-month (August 2021 to July 2023) average rate for consumption, the current price for demand, and a share of the Global Adjustment (GA) based on Crawford's forecast life-of-mine peak daytime demand and the province's current peak demand. The forecast life-of-mine average total price for electricity, taking account of charges for consumption, demand, and the GA, is C$75/MWh.
- Working capital has been calculated based on the following:
    - Contractual terms for the sale of concentrate would make provision for payment within 7 days of shipping.
    - Accounts payable would be settled within 30 days.
    - The stores holding of consumables would be 1 month, with the exceptions of tires (4 months), fuel (nominally 5 days' consumption at site), and electricity (no holding).
- The royalty and other payments include the following:
    - existing 2% NSR royalty.
    - assumed Impacts Benefits Agreement (IBA) in favour of the local Indigenous communities. Note that an IBA has yet to be negotiated for Crawford and the terms used are based on those negotiated for other projects.
    - existing 5% net profits interest (NPI) royalty (note: it has been assumed that the option to buy this royalty down to 2.5% for C$2 million would be exercised).

The analysis was carried out on both a pre-tax and post-tax basis. The post-tax evaluation incorporates the following features of the Canadian income and Ontario income and mining tax codes:

- The Ontario Mining Tax (OMT) is applied at a rate of 10% on resource profits, which exclude revenues associated with carbon sequestration while adding back expenditures on royalties and the assumed IBA. The OMT also allows an annual allowance up to 5.2% of the cumulative expenditure on processing assets. Income taxes allow a deduction of the OMT from taxable income.
- Income tax rates of 15% (federally) and 10% (provincially) are included, with the provincial rate including a 1.5% deduction from the general income tax rate of 11.5% that is applied to certain manufacturing industries, including mining.
- A capital cost allowance (CCA) of 25% is included, with the previous accelerated rate of 100% for pre-production expenditures having been phased out in 2020.
- Deductions for Canadian development expenses (CDE) of 30% are included. The accelerated rate of 100% for future pre-production expenditures (Canadian exploration expenses, CEE) were phased out in 2017, though expenditures incurred in prior years could still be depreciated at the accelerated rate.
- Net operating losses (NOLs) can be pooled for up to 20 years and deducted in their entirety in a single year.

The base case assumes Crawford would be eligible for the CTM ITC that was outlined during the 2023 federal budget presentation. This tax credit would apply to capital invested in the plant and equipment necessary for production of the defined critical minerals necessary for Clean Technology Manufacturing (which include nickel, cobalt, chromium, and platinum group elements). Disclosure to date indicates the ITC will take the form of a refund of 30% of the eligible expenditure through 2031. The refund percentage will ramp down during 2032 to 2034 before being phased out in 2035.

## 22.4 Base Case Results

The project life project can be subdivided into the following periods:

- Construction for a period of 30 months.
- Phase 1 at a nominal concentrator throughput of 60 kt/d/ for a period of 42 months (3.5 years).
- Phase 2 at a nominal concentrator throughput of 120 kt/d and the bulk of ore being provided from the operating pits for a period of 318 months (26.5 years).
- Phase 3 is the final 135 months of operation after the pits have been depleted. The mill will continue to operate at a nominal concentrator throughput of 120 kt/d and be fed entirely from low-grade stockpiles.

Table 22-2 provides a summary of metrics by phase.

**Table 22-2: Crawford Feasibility Study Summary Metrics by Phase**

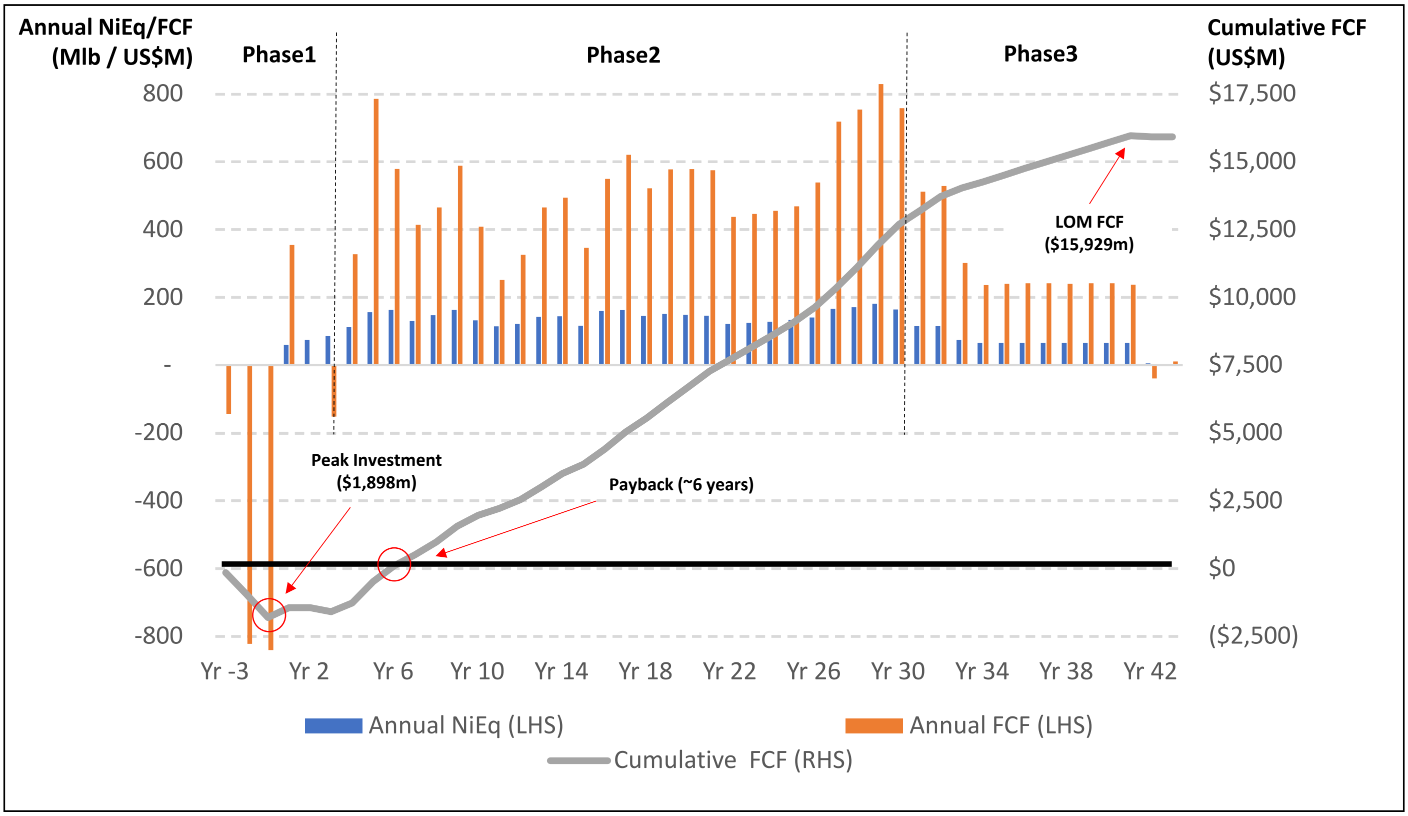
| Item | Units | Construction | Phase 1 | Phase 2 | Phase 3 | Life of Mine |
|---|---|---|---|---|---|---|
| Ore Mined | Mt | 14 | 125 | 1,575 | 0 | 1,715 |
| Ore Milled | Mt | 0 | 73 | 1,157 | 485 | 1,715 |
| Strip Ratio | Waste:Ore Mined | 6.2 | 2.4 | 2.3 | n/a | 2.3 |
| **Grade** | | | | | | |
| Nickel Head Grade | % | | 0.26 | 0.24 | 0.17 | 0.22 |
| Cobalt Head Grade | % | | 0.013 | 0.013 | 0.012 | 0.013 |
| Palladium & Platinum Head Grade | g/t | | 0.030 | 0.024 | 0.021 | 0.024 |
| Iron Head Grade | % | | 6.20 | 6.43 | 6.49 | 6.44 |
| Chromium Head Grade | % | | 0.63 | 0.60 | 0.49 | 0.57 |
| **Recovery** | | | | | | |
| Nickel Recovery | % | | 48% | 46% | 25% | 41% |
| Cobalt Recovery | % | | 19% | 14% | 4% | 11% |
| Palladium & Platinum Recovery | % | | 40% | 39% | 33% | 38% |
| Iron Recovery | % | | 54% | 56% | 46% | 53% |
| Chromium Recovery | % | | 28% | 29% | 26% | 28% |
| **Annual Production** | | | | | | |
| Recovered Nickel | kt/a | | 26 | 48 | 18 | 38 |
| Recovered Cobalt | kt/a | | 0.5 | 0.8 | 0.2 | 0.6 |
| Recovered Palladium & Platinum | koz/a | | 8 | 13 | 10 | 12 |
| Recovered Iron | Mt/a | | 0.7 | 1.6 | 1.3 | 1.4 |
| Recovered Chromium | kt/a | | 37 | 76 | 54 | 67 |
| Carbon Sequestration | Mt/a | | 0.6 | 1.5 | 1.1 | 1.3 |
| **Revenue & Costs** | | | | | | |
| NSR | US$/t milled | | 34.96 | 32.31 | 16.96 | 28.08 |
| Mining Operating Costs | US$/t milled | | 9.82 | 6.21 | 0.62 | 4.78 |
| Milling Operating Costs | US$/t milled | | 5.31 | 5.18 | 5.19 | 5.19 |
| G&A Operating Costs | US$/t milled | | 2.35 | 1.00 | 0.50 | 0.92 |
| Total On-Site Costs | US$/t milled | | 17.48 | 12.38 | 6.31 | 10.88 |
| Gross C1 Cash Cost | US$/lb NiEq | | 4.82 | 3.72 | 3.64 | 5.96 |
| Net C1 Cash Cost | US$/lb Ni | | 2.67 | 0.68 | (2.39) | 0.39 |
| Net AISC | US$/lb Ni | | 2.98 | 1.87 | (1.19) | 1.54 |
| Total Investment | US$M | 1,946 | 1,602 | 1,450 | 159 | 5,157 |
| **Taxation** | | | | | | |
| Federal and Provincial Income Taxes | US$M | 0 | 0 | 4,222 | 1,062 | 5,284 |
| Provincial Mining Tax | US$M | 0 | 0 | 1,653 | 341 | 1,995 |
| CTM ITC | US$M | (69) | (459) | (340) | 0 | (868) |
| **Cash Flow** | | | | | | |
| Annual EBITDA | US$M | 0 | 349 | 811 | 426 | 667 |
| Annual Free Cash Flow | US$M | (723) | 17 | 545 | 292 | 431 |

Notes: 1. Byproducts converted to nickel at the ratio of prices. 2. Net of byproduct credits. 3.Total investment includes capital costs and closure.

Figure 22-1 provides a life-of-project graph of cash flow and NiEq production. The following information is highlighted:

- The peak funding requirement of $1,898 million is reached in the initial quarter of commercial production, as the mill is ramping up to nameplate output. Thereafter, and despite further capital expenditure of $1,600 million in the expansion to 120 kt/d, the combination of cash flow and the CTM ITC results no additional funding is required.
- Simple payback, which excludes costs associated with finance, is achieved approximately 6 years after the start of mill production.
- After the expansion to 120 kt/d, for the remaining 26 ½ years of pit life, production, and cash flow average 145 Mlbs NiEq and $545 million, respectively.
- During the final 11 years when ore is sourced from stockpiles, output reduces approximately 47% from Phase 2 levels. However, the low-cost base result in healthy free cash generation of almost $300 million annually. Life-of-mine free cash totals $15,929 million.

**Figure 22-1: Crawford Production and Cash Flow**




Source: CNC, 2023.

Table 22-3 provides details of annual production and cash flow.

**Table 22-3: Crawford Detailed Production and Cash Flow**

| Production | Unit | Total | Construct | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Yr 10 | Yr 11 | Yr 12 | Yr 13 | Yr 14 | Yr 15 | Yr 16 | Yr 17 | Yr 18 | Yr 19 | Yr 20 | Yr 21-30 | Yr 30-41 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Total Mined | Mt | 5,707 | 103 | 109 | 104 | 125 | 181 | 204 | 212 | 229 | 232 | 228 | 222 | 208 | 230 | 234 | 212 | 223 | 232 | 214 | 220 | 217 | 215 | 1,552 | - |
| Ore Mined | Mt | 1,715 | 14 | 32 | 18 | 46 | 62 | 74 | 68 | 49 | 64 | 70 | 56 | 51 | 72 | 86 | 76 | 65 | 87 | 67 | 66 | 75 | 63 | 452 | - |
| Ore Milled | Mt | 1,715 | - | 18 | 22 | 22 | 30 | 43 | 44 | 44 | 44 | 44 | 44 | 44 | 44 | 44 | 44 | 44 | 44 | 44 | 44 | 44 | 44 | 438 | 485 |
| | | | | | | | | | | | | | | | | | | | | | | | | | |
| Nickel Grade | % | 0.22 | - | 0.26 | 0.24 | 0.27 | 0.27 | 0.27 | 0.27 | 0.23 | 0.25 | 0.26 | 0.24 | 0.22 | 0.22 | 0.23 | 0.24 | 0.21 | 0.24 | 0.25 | 0.23 | 0.24 | 0.24 | 0.24 | 0.17 |
| Cobalt Grade | % | 0.013 | - | 0.012 | 0.012 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.013 | 0.012 | 0.012 | 0.013 | 0.013 | 0.012 |
| Palladium Grade | g/t | 0.014 | - | 0.015 | 0.013 | 0.021 | 0.022 | 0.020 | 0.023 | 0.015 | 0.019 | 0.019 | 0.013 | 0.013 | 0.012 | 0.015 | 0.018 | 0.010 | 0.017 | 0.019 | 0.009 | 0.013 | 0.014 | 0.014 | 0.011 |
| Platinum Grade | g/t | 0.009 | - | 0.017 | 0.011 | 0.010 | 0.010 | 0.010 | 0.010 | 0.009 | 0.010 | 0.010 | 0.008 | 0.009 | 0.010 | 0.010 | 0.010 | 0.008 | 0.009 | 0.010 | 0.007 | 0.009 | 0.009 | 0.009 | 0.009 |
| Iron Grade | % | 6.44 | - | 5.84 | 6.23 | 6.35 | 6.45 | 6.41 | 6.40 | 6.49 | 6.46 | 6.56 | 6.41 | 6.67 | 6.73 | 6.76 | 6.87 | 6.87 | 6.60 | 6.55 | 6.32 | 6.20 | 6.29 | 6.25 | 6.49 |
| Chromium Grade | % | 0.57 | - | 0.67 | 0.63 | 0.60 | 0.60 | 0.60 | 0.60 | 0.59 | 0.61 | 0.61 | 0.61 | 0.59 | 0.59 | 0.58 | 0.59 | 0.58 | 0.55 | 0.57 | 0.62 | 0.63 | 0.62 | 0.61 | 0.49 |
| Brucite Grade | % | 1.61 | | 2.51 | 1.54 | 0.91 | 1.59 | 1.97 | 1.86 | 1.49 | 1.41 | 1.44 | 1.66 | 1.31 | 1.30 | 1.37 | 1.32 | 1.24 | 1.91 | 1.88 | 2.08 | 2.39 | 2.13 | 2.17 | 1.02 |
| | | | | | | | | | | | | | | | | | | | | | | | | | |
| Nickel Production | Mlbs | 3,439 | - | 46 | 54 | 65 | 85 | 117 | 123 | 89 | 107 | 124 | 94 | 73 | 80 | 102 | 104 | 73 | 120 | 121 | 104 | 112 | 108 | 1,082 | 456 |
| Cobalt Production | Mlbs | 54 | - | 0 | 1 | 2 | 2 | 3 | 4 | 2 | 3 | 3 | 1 | 1 | 1 | 2 | 3 | 1 | 3 | 3 | 1 | 1 | 1 | 11 | 6 |
| Palladium Production | koz | 375 | - | 4 | 4 | 8 | 11 | 14 | 17 | 10 | 14 | 14 | 8 | 8 | 8 | 10 | 12 | 7 | 12 | 14 | 6 | 9 | 10 | 95 | 80 |
| Platinum Production | koz | 113 | - | 3 | 2 | 2 | 2 | 3 | 3 | 2 | 3 | 3 | 2 | 2 | 3 | 3 | 3 | 2 | 3 | 4 | 2 | 3 | 3 | 32 | 28 |
| Iron Production | kt | 58 | - | 1 | 1 | 1 | 1 | 1 | 2 | 2 | 2 | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 16 | 14 |
| Chromium Production | Mlbs | 6,091 | - | 68 | 91 | 85 | 115 | 163 | 168 | 166 | 168 | 167 | 164 | 159 | 162 | 161 | 154 | 162 | 161 | 165 | 179 | 179 | 177 | 1,733 | 1,346 |
| | | | | | | | | | | | | | | | | | | | | | | | | | |
| Payable Nickel | Mlbs | 3,130 | - | 42 | 49 | 59 | 77 | 107 | 112 | 81 | 97 | 113 | 85 | 66 | 73 | 93 | 95 | 67 | 109 | 110 | 94 | 102 | 98 | 984 | 415 |
| Payable Cobalt | Mlbs | 32 | - | 0 | 0 | 1 | 1 | 2 | 2 | 1 | 2 | 2 | 1 | 1 | 1 | 1 | 2 | 0 | 2 | 2 | 0 | 1 | 1 | 7 | 3 |
| Payable Palladium | koz | 282 | - | 3 | 3 | 5 | 8 | 10 | 12 | 7 | 10 | 9 | 6 | 7 | 6 | 7 | 9 | 5 | 8 | 10 | 4 | 6 | 7 | 68 | 70 |
| Payable Platinum | koz | 86 | - | 2 | 1 | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 3 | 1 | 2 | 2 | 23 | 25 |
| Payable Iron | Mt | 29 | - | 0 | 0 | 0 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 8 | 7 |
| Payable Chromium | Mlbs | 3959 | - | 44 | 59 | 55 | 75 | 106 | 110 | 108 | 109 | 108 | 107 | 103 | 105 | 104 | 100 | 105 | 105 | 107 | 116 | 117 | 115 | 1,126 | 875 |
| Third-Party Carbon Stored | Mt | 47 | | 0.6 | 0.5 | 0.3 | 0.7 | 1.3 | 1.2 | 1.1 | 1.0 | 1.1 | 1.2 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.2 | 1.3 | 1.4 | 1.5 | 1.5 | 14.5 | 11.5 |
| **Revenue** | | | | | | | | | | | | | | | | | | | | | | | | | |
| Nickel Concentrate | US$M | 25,493 | - | 343 | 398 | 535 | 690 | 940 | 1,009 | 667 | 851 | 997 | 673 | 511 | 563 | 777 | 820 | 501 | 951 | 967 | 735 | 814 | 792 | 7,938 | 3,025 |
| Iron Concentrate | US$M | 21,764 | - | 228 | 312 | 278 | 384 | 549 | 552 | 578 | 557 | 567 | 592 | 579 | 599 | 588 | 553 | 611 | 582 | 584 | 646 | 636 | 628 | 6,183 | 4,982 |
| Carbon Revenue | US$M | 891 | - | 11 | 9 | 6 | 14 | 25 | 24 | 20 | 19 | 20 | 23 | 19 | 19 | 20 | 20 | 18 | 22 | 24 | 27 | 29 | 28 | 276 | 219 |
| **Net Smelter Return** | **US$M** | **48,148** | **-** | **582** | **719** | **819** | **1,087** | **1,513** | **1,584** | **1,265** | **1,426** | **1,584** | **1,288** | **1,108** | **1,181** | **1,385** | **1,393** | **1,130** | **1,555** | **1,574** | **1,408** | **1,479** | **1,447** | **14,397** | **8,225** |

| Production | Unit | Total | Construct | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Yr 10 | Yr 11 | Yr 12 | Yr 13 | Yr 14 | Yr 15 | Yr 16 | Yr 17 | Yr 18 | Yr 19 | Yr 20 | Yr 21-30 | Yr 30-41 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Operating Costs** | | | | | | | | | | | | | | | | | | | | | | | | | |
| Mining | US$M | 8,199 | - | 187 | 204 | 207 | 238 | 295 | 336 | 330 | 331 | 314 | 316 | 345 | 322 | 329 | 307 | 315 | 380 | 294 | 285 | 275 | 252 | 2,036 | 301 |
| Tails & Water Management | US$M | 289 | - | 8 | 9 | 8 | 9 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 7 | 7 | 7 | 7 | 6 | 6 | 63 | 70 |
| Milling | US$M | 8,891 | - | 99 | 115 | 115 | 158 | 223 | 227 | 227 | 227 | 227 | 227 | 227 | 227 | 227 | 227 | 227 | 227 | 227 | 227 | 227 | 227 | 2,265 | 2,517 |
| G&A | US$M | 1,282 | - | 42 | 39 | 37 | 49 | 52 | 52 | 48 | 46 | 45 | 44 | 46 | 43 | 41 | 40 | 41 | 42 | 41 | 42 | 40 | 36 | 244 | 172 |
| **Site Costs** | **US$M** | **18,661** | **-** | **336** | **367** | **367** | **454** | **579** | **623** | **613** | **612** | **594** | **595** | **625** | **599** | **605** | **581** | **590** | **655** | **569** | **560** | **548** | **521** | **4,608** | **3,060** |
| | | | | | | | | | | | | | | | | | | | | | | | | | |
| Royalties, etc. | US$M | 1,990 | - | 12 | 15 | 17 | 22 | 31 | 32 | 26 | 56 | 70 | 51 | 38 | 45 | 59 | 61 | 44 | 69 | 73 | 63 | 69 | 68 | 692 | 378 |
| **Net C1 cash costs** | **US$/lb** | **0.39** | **-** | **3.90** | **2.52** | **2.02** | **1.53** | **0.99** | **1.13** | **1.69** | **1.34** | **0.95** | **1.68** | **2.54** | **1.80** | **1.37** | **1.16** | **1.70** | **1.49** | **0.63** | **0.84** | **0.67** | **0.39** | **(0.14)** | **(2.38)** |
| | | | | | | | | | | | | | | | | | | | | | | | | | |
| **Investment** | | | | | | | | | | | | | | | | | | | | | | | | | |
| Initial Capital | US$M | 1,943 | 1,943 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Expansion Capital | US$M | 1,600 | 0 | 182 | 436 | 616 | 367 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Sustaining Capital | US$M | 1,481 | 0 | 0 | 0 | 0 | 96 | 129 | 159 | 101 | 102 | 81 | 79 | 90 | 64 | 45 | 36 | 21 | 16 | 27 | 12 | 12 | 14 | 300 | 96 |
| W/C and Closure | US$M | 134 | (67) | 63 | (41) | 3 | 50 | (3) | (2) | (1) | 9 | 3 | (10) | 0 | 8 | 7 | 3 | (8) | 15 | 0 | 19 | 8 | 2 | 25 | 50 |
| **Total Investment** | **US$M** | **5,157** | **1,876** | **245** | **395** | **619** | **513** | **126** | **157** | **100** | **111** | **85** | **69** | **91** | **72** | **52** | **39** | **12** | **31** | **27** | **31** | **21** | **16** | **325** | **145** |
| | | | | | | | | | | | | | | | | | | | | | | | | | |
| **Cash Flow** | | | | | | | | | | | | | | | | | | | | | | | | | |
| EBITDA | US$M | 27,497 | | 234 | 337 | 434 | 611 | 904 | 930 | 626 | 758 | 919 | 641 | 445 | 537 | 721 | 751 | 496 | 831 | 932 | 784 | 863 | 859 | 9,097 | 4,786 |
| Investment | US$M | (5,157) | (1,876) | (245) | (395) | (619) | (513) | (126) | (157) | (100) | (111) | (85) | (69) | (91) | (72) | (52) | (39) | (12) | (31) | (27) | (31) | (21) | (16) | (325) | (145) |
| Income Tax | US$M | (5,284) | - | - | - | - | - | (86) | (147) | (87) | (125) | (169) | (114) | (75) | (100) | (145) | (155) | (100) | (178) | (202) | (166) | (189) | (189) | (1,995) | (1,062) |
| Mining Tax | US$M | (1,995) | - | - | - | - | - | (0) | (52) | (35) | (57) | (77) | (48) | (28) | (38) | (58) | (62) | (36) | (72) | (83) | (65) | (75) | (75) | (790) | (341) |
| ITC | US$M | 868 | 69 | 365 | 60 | 34 | 229 | 95 | 7 | 9 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Free Cash Flow** | **US$M** | **15,929** | **(1,807)** | **355** | **2** | **(151)** | **328** | **786** | **579** | **414** | **466** | **588** | **409** | **251** | **326** | **466** | **495** | **347** | **550** | **621** | **522** | **578** | **579** | **5,987** | **3,238** |

## 22.5 Sensitivity Analysis

Figure 22-2 to Figure 22-4 illustrate the sensitivity of project NPV, IRR, and net C1 cash costs to macro-economic factors that are outside of management's ability to control, including the following:

- prices for the key metals that will be produced
- price for energy that will be consumed
- rate of exchange for the Canadian dollar.

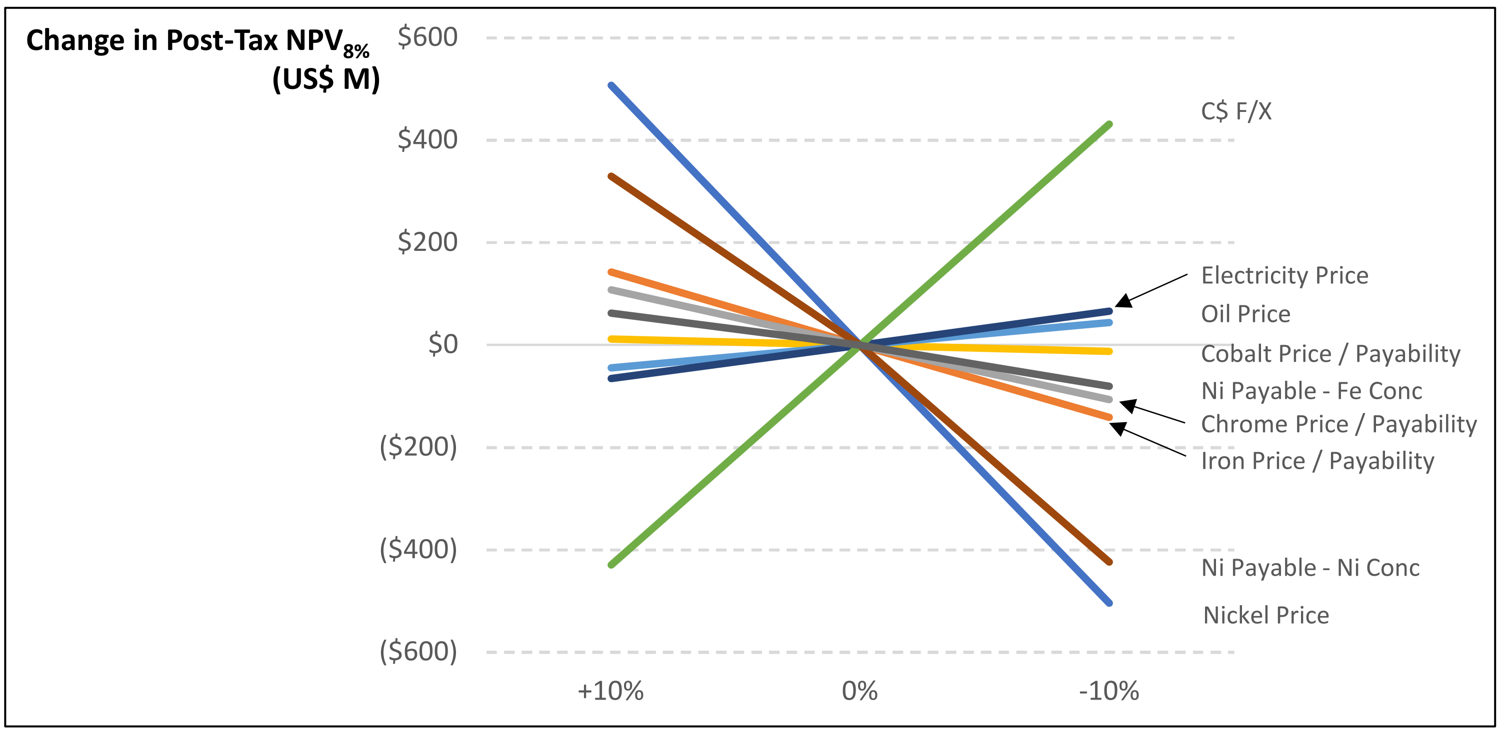
Figure 22-2 shows the NPV is most sensitive to the nickel price, with a 10% change in prices having an impact of ±$506 million, or 20% of the base case NPV and therefore 2.0x greater than the variation in price. The NPV is also highly sensitive to the Canadian dollar exchange rate and the payability of nickel in the Ni concentrate, at 1.7x and 1.4x, respectively.

The NPV is 50% more sensitive to variation in the iron price or payability, at 0.6x, than chromium (0.4x). The NPV is equally sensitive to changes in the price of electricity and payability of nickel in the Fe concentrate, at 0.3x. This is 50% greater than the sensitivity to oil price, at 0.2x. The NPV is insensitive to variation in cobalt prices or payability (0.05x). The sensitivity to PGE prices or payability is lower still and therefore has not been shown

Figure 22-3 shows that the sensitivity of the IRR to variation in macro-economic factors exhibits the same overall trend as for the NPV, with a key difference being the reversal of ranking for nickel prices and exchange rate. A 10% variation in exchange rate leads to a +2.4% / -2.1% change in IRR, the average relative variation in IRR being 1.3x the 10% variation in assumption. This can be compared to the ± 1.8% change in IRR resulting from a 10% change in nickel price (1.0x the base case value). The ranking of the remaining macro-economic elements is unchanged (from the rankings for NPV), though the spread between the impact of variance in electricity and oil price reduces from 50% (seen with NPV) to 20%.

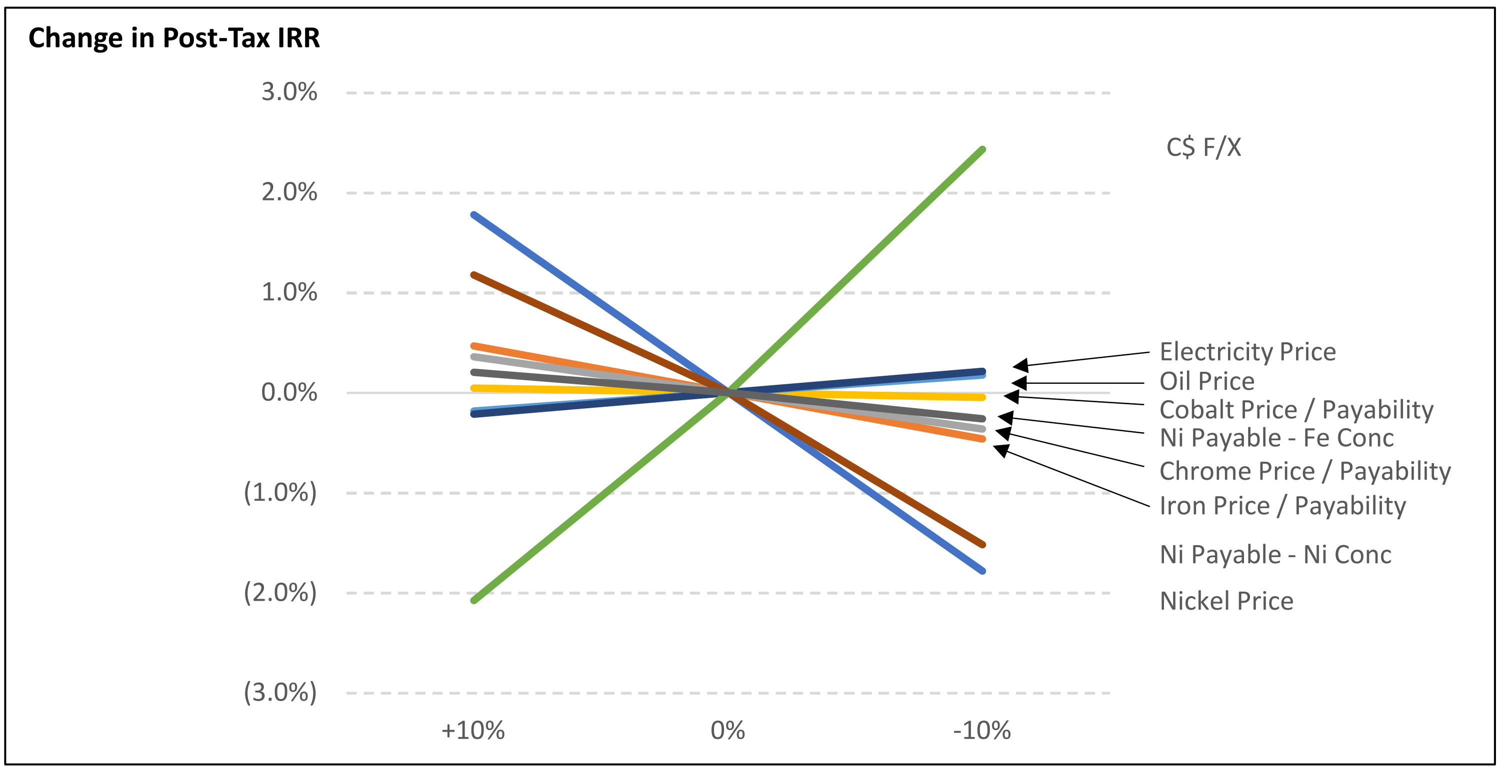
Figure 22-4 excludes nickel prices but includes nickel payability and other metal prices as they influence the quantum of byproduct credit. The overall ranking remains unchanged from the NPV and IRR graphs, though the change in net C1 costs resulting from a 10% change in electricity prices is $0.14/lb, more than 2x the impact from change in oil prices.

**Figure 22-2: Crawford Macro-Economic Sensitivities – NPV**



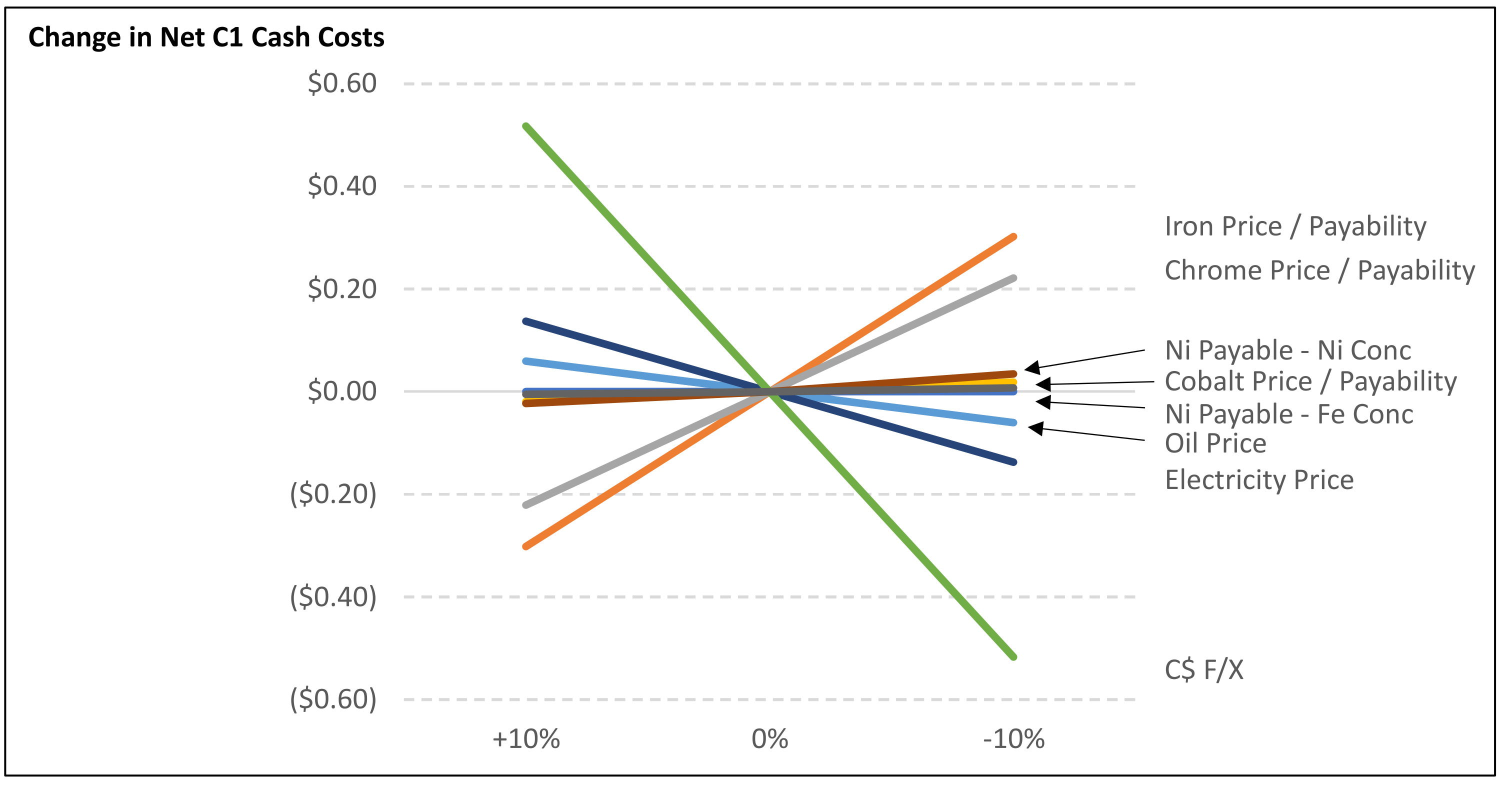

Source: CNC, 2023.

**Figure 22-3: Crawford Macro-Economic Sensitivities – IRR**




Source: CNC, 2023.

**Figure 22-4: Crawford Macro-Economic Sensitivities – Net C1 Cash Costs**

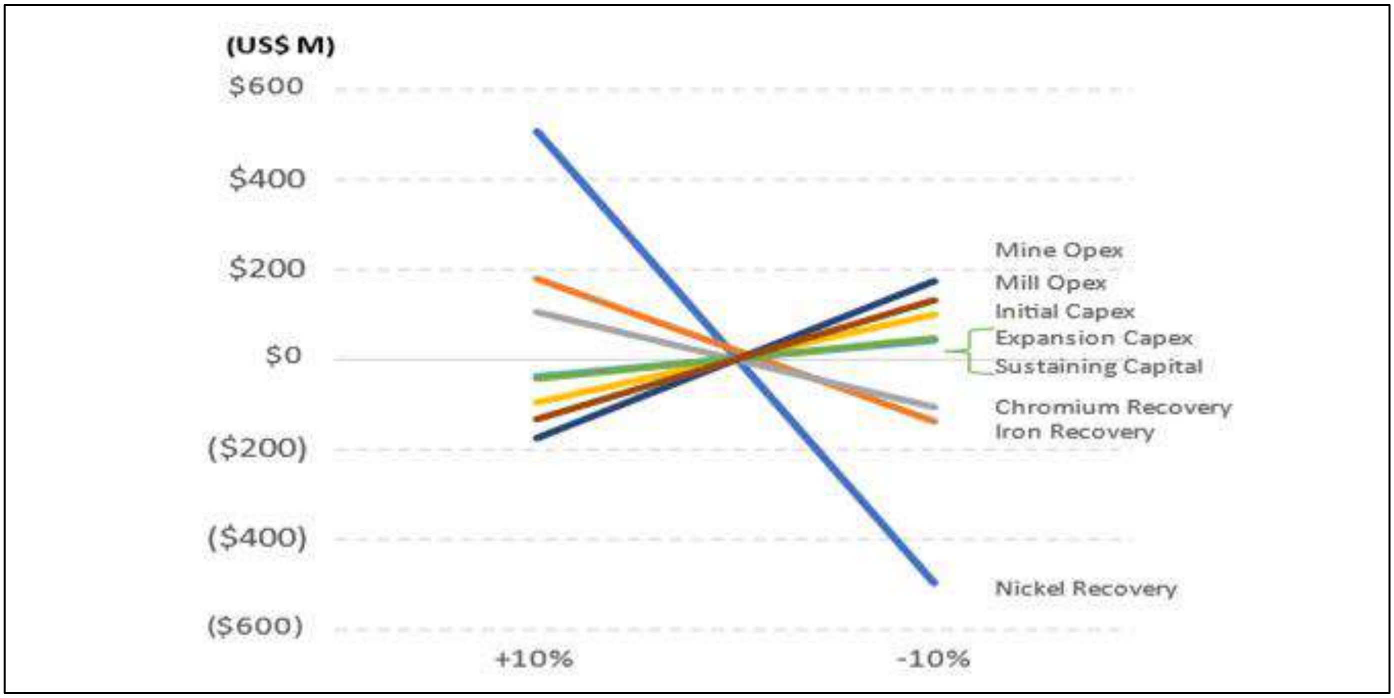



Source: CNC, 2023.

Figure 22-5 to Figure 22-7 perform a similar analysis of the sensitivity of project NPV, IRR, and net C1 cash costs to operating factors that are largely within management's scope of control, including the capital costs, operating costs, and metallurgical recovery.

Figure 22-5 shows the NPV is as sensitive to variation in recovery as to price, with curves for nickel, iron, and chromium recovery being identical to those in Figure 22-2. Due to the quantum of total expenditure, the NPV is more sensitive to variation in both mine and mill operating costs than initial capital costs. Despite total mill operating costs being 9% higher than those for the mine, the mine operating costs have a greater impact on NPV as they are front loaded.

**Figure 22-5: Crawford Operational Sensitivities – NPV**

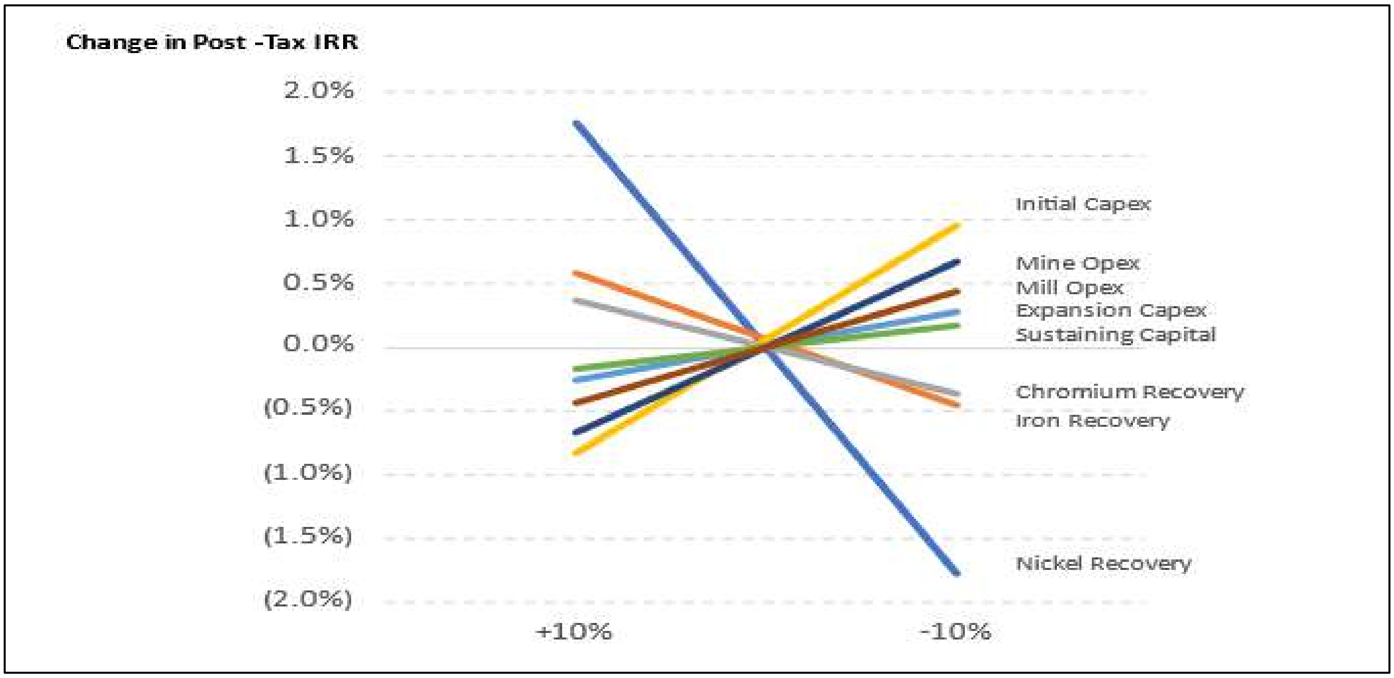



Source: CNC, 2023.

In Figure 22-6, the impact of expenditure timing can clearly be seen with the sensitivity to initial capital costs overtaking both mine and mill operating costs, while the sensitivity to expansion capital costs is 60% higher than to sustaining capital costs.

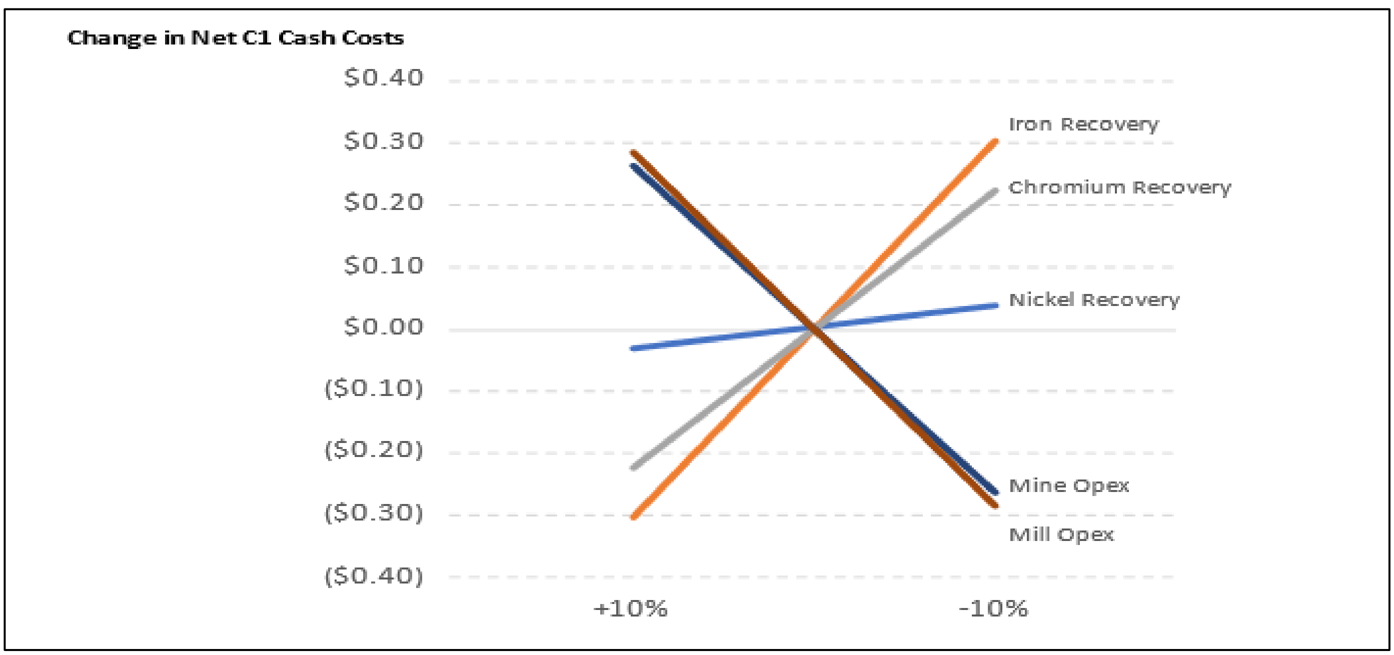
The capital cost items have been removed from Figure 22-7. The impact of nickel recovery (which impacts the pounds of nickel used as the cash cost divisor) is muted compared to iron and chromium recovery, which affect the quantum of byproduct credit. The higher life-of-mine expenditure at the mill translates to an 8% higher impact on cash costs relative to the mine.

**Figure 22-6: Crawford Operational Sensitivities – IRR**




Source: CNC, 2023.

**Figure 22-7: Crawford Operational Sensitivities – Net C1 Cash Costs**




Source: CNC, 2023.

# 23 ADJACENT PROPERTIES

There are no adjacent properties that are actively being explored that would materially affect the Authors' understanding of the project or the interpretations and conclusions presented in this report.

# 24 OTHER RELEVANT DATA AND INFORMATION

## 24.1 Project Execution and Organization

The Project Execution Plan (PEP) is a governing document that establishes the means to execute, monitor, and control the execution phase of the Crawford Project. The plan will serve as the main communication tool to ensure that the integrated project teams are aware and knowledgeable of project objectives and how they will be accomplished.

The following subsections summarize the contents of the preliminary PEP.

### 24.1.1 Summary

The PEP provides the following information at a minimum:

- an overview of the project
- the scope of work and services for the project
- the project execution schedule with key activities and target dates
- an organization chart for the execution phase
- project execution sub-plans.

The PEP is supported by various sub-plans that include, but are not limited to, the following areas:

- health, safety, environment, and community management
- project management
- contracting strategy
- construction execution
- project quality
- logistics.

### 24.1.2 Objectives

CNC aims to bring the project into operation while satisfying the following objectives:

- achieve zero harm to personnel involved with construction, operation, and maintenance of the facilities, and zero unintended environmental impact or incidents
- preserve or improve the project value through effective control of project costs and completion of construction and commissioning on or ahead of schedule
- satisfy quality and performance targets
- comply with company policies and legislative requirements, negotiated benefits agreements
- maintain positive community relations.

### 24.1.3 Execution Strategy

Two contract strategies will be employed to deliver the detailed engineering and execution phases of the project:

1. Engineering, procurement, and construction management (EPCM) contract(s), led by an engineering consultant nominated by CNC, that generally encompasses the process plant and select on-site infrastructure
2. EPCM scope, led by CNC, that generally encompasses the development of the mining pits, bulk earthworks, and off-site infrastructure.

These are described in more detail in the following subsections.

#### 24.1.3.1 EPCM Scope Led by Engineering Consultant

The delivery strategy for the process plant and infrastructure EPCM scope can be summarized as follows:

- Engineering and design for construction will be completed by the engineering consultant. Detailed design will start in Year -4 (prior to start of production), with the procurement of long-lead equipment.
- Procurement of equipment and services, expediting and contract management will be performed by the engineering consultant.
- The engineering consultant will perform commercial management of contractors during construction, and provide technical supervision and support on-site as required.

#### 24.1.3.2 EPCM Scope Led by CNC

The following remaining scope areas will be managed by CNC on an EPCM basis:

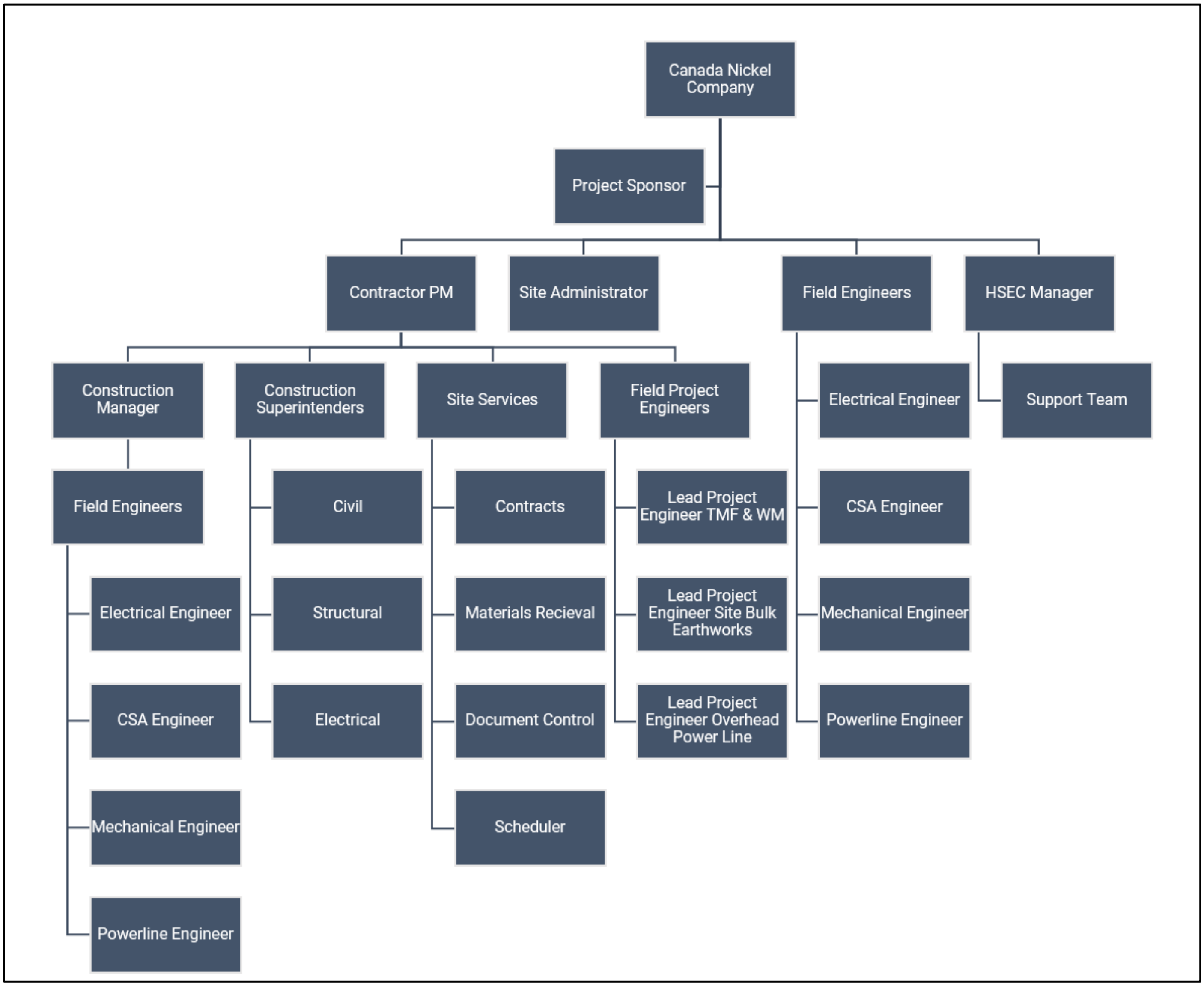
- site bulk earthworks
- tailings and water management earthworks (by mining fleet)

- open pit development by mining fleet
- off-site infrastructure, including 500 kV transmission line by Hydro One, Highway 655 modifications and rail.

### 24.1.4 Project Organization

The project team is organized based on an integrated team approach, minimizing the duplication of roles and activities between the Owner's team and major delivery partners. A project organization chart is shown in Figure 24-1.

**Figure 24-1: Project Organization Chart**




Source: Ausenco, 2023

CNC will be performing or managing a considerable portion of the project scope, including the mine design, power transmission line, rails, highway modifications, pit pre-stripping, and delivery of certain construction materials to designated work sites. Key persons will be established on both teams at site to ensure efficient coordination.

### 24.1.5 Construction Execution Strategy

This section outlines the high-level execution sequencing constraints that were evaluated in order to determine the execution schedule baseline in the feasibility study. An overall master execution schedule is included in Figure 24-2.

#### 24.1.5.1 Construction Sequencing

After completion of the feasibility study, submission of required permits will be progressed, as this is the critical path activity for the next phase of the project.

The key construction activites on the critical path of the project construction are summarized below. Note that the schedule has been developed assuming a mid-year construction start based on permitting timelines..

The construction phase of the project will consist of the following activities:

- The first contractors to mobilize to site will be the civil work contractors responsible for clearing the site.
- The infrastructure areas will then need to be levelled and graded to rough construction levels.
- Once the initial civil works are completed, the main schedule priority will be the process plant and crushing area, where piling is required to achieve suitable ground conditions for building construction.
- In parallel, contractor's execution team and subcontractor teams will mobilize to site and establish office and workshop facilities, temporary accommodations, etc.
- Mining works will continue pit development, generating and stockpiling waste rock material that will be used for construction materials.
- The concrete works will begin as soon as winter conditions have eased enough to allow concrete pouring, focusing on the process plant building first, and followed by the crushing and mine infrastructure building foundations.
- Erection of the main process building steel will follow closely behind the building foundations, to ensure that the main building envelope is enclosed well before the second winter.
- Following this, equipment foundations will be poured, and construction will proceed through steelwork, equipment installation, piping, electrical and instrumentation.

The timing of the construction phase is dictated by CNC receiving permit approvals.

#### 24.1.5.2 Winter Construction

The construction duration for this project will involve ongoing work throughout the winter seasons over multiple years. To mitigate downtime and loss of productivity, considerations were included in the execution schedule. The concrete works and piling for the process plant are largely scheduled to be carried out during the spring, summer, and fall months. The construction sequence for the process plant has been developed to avoid outdoor concrete pouring during the worst of the winter months (December to February). Piling activities can occur during this time, as frozen ground conditions ameliorate access for heavy equipment. Priority will also be given to erect the contractors' temporary warehouse and fabrication buildings for additional all-weather storage during the winter months.

## 24.2 Construction Facilities

Temporary construction infrastructure will be dedicated to the EPCM contractor. The construction of common facilities such as security, emergency response and medical facilities, communication, and camp and accommodations will be prioritized at the beginning of the construction phase.

### 24.2.1 Site Accommodations

There will be no on-site accommodations and camp for the contractors. The contractors will be responsible for camp and accommodation expenses and can stay in or near the town of Timmins.

### 24.2.2 Maintenance Facilities

The maintenance building for the process plant and the warehouse will be prioritized to be available early in the construction phase. The temporary maintenance and cleanup services for temporary facilities will be managed by CNC.

### 24.2.3 Emergency Response and Medical Facilities

New emergency response facilities will be built early in the construction phase, housing both ambulance and fire trucks. The construction of the medical facility and fire hall will occur at the outset of the construction phase, while temporary medical facilities on the plant site will be used until permanent facilities become operational.

### 24.2.4 Water Supply

The construction of freshwater wells for the site will be completed at the beginning of the construction phase.

### 24.2.5 Fuel Supply and Storage

The fuel storage and dispensing facility will be established in two phases. The initial stage will entail development of a temporary facility to be utilized by all equipment and located close to the centroid of early mining activities in the East Zone. This temporary facility will include a single 80 $m^3$ capacity tank for each of dyed diesel (diesel for off-road equipment and exempt from provincial excise taxes), clear diesel (subject to the provincial excise tax of 13%) and gasoline. When a concrete pad and electricity supply have been established at the permanent mine facility, located south of the ultimate pits and close to the mine workshops, the temporary facility will be decommissioned. Its storage tanks will be relocated to a permanent facility close to the concentrator, where it will also be utilized by G&A equipment. The mine facility will be progressively expanded, and include four tanks for dyed diesel by the time commissioning of the mill commences. Ultimately, at peak mine production, there will be 13 tanks for dyed diesel and a single tank for each of clear diesel and gasoline. Installation and commissioning of the tanks, as well as all associated costs, will be borne by the fuel supplier and recaptured during operations as a fixed charge per litre of fuel consumed.

## 24.3 Shared Site Services

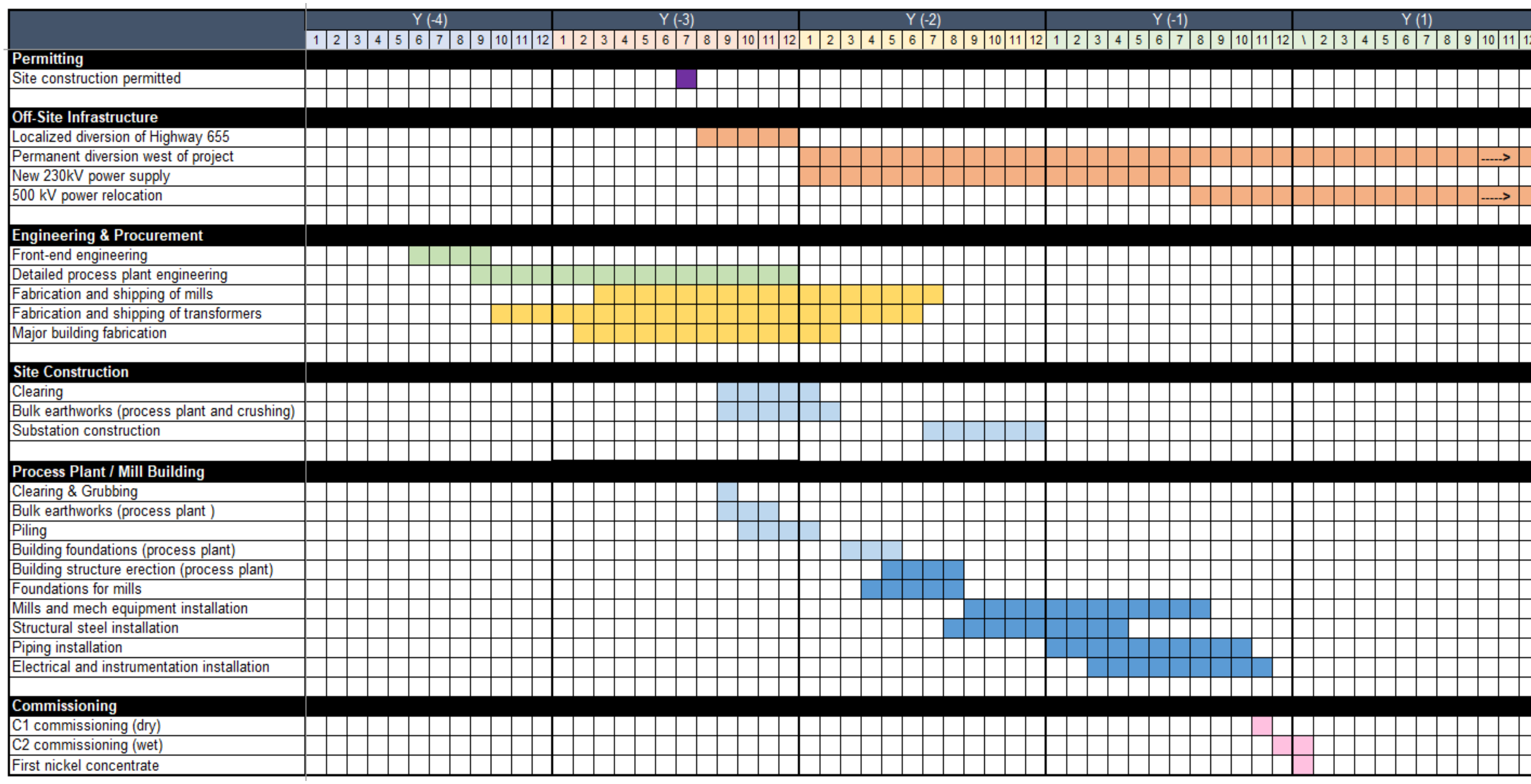
Several services were identified during the feasibility study that are common across the work fronts during construction. It may be advantageous to offer these common services to the contractors from a cost perspective, as well as to favour site service contracts to local businesses. These services include the following:

- diesel fuel supply
- temporary power supply
- road maintenance and snowclearing
- garbage removal
- upfront purchase or lease of mobile equipment that will be required by operations and can be free-issued to the contractors during construction.

## 24.4 Execution Schedule

The preliminary project execution schedule is shown in Figure 24-2.

**Figure 24-2: Preliminary High-Level Execution Schedule**




Source: Ausenco, 2023.

# 25 INTERPRETATION AND CONCLUSIONS

## 25.1 Introduction

The QPs have provided the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this report.

## 25.2 Mineral Tenure, Surface Rights, Royalties and Agreements

Information from legal experts supports that the mining tenure held is valid and is sufficient to support declaration of mineral resources and mineral reserves.

## 25.3 Geology

The Crawford Project hosts a new economic deposit type, referred to as Crawford-style komatiite-hosted nickel sulphide. Crawford-style deposit types consist of large volumes of altered ultramafic rocks comprising relatively low nickel grades.

The understanding of the Crawford deposit settings, lithologies, mineralization, and the geological, structural, and alteration controls on mineralization is sufficient to support estimation of mineral resources and mineral reserves.

## 25.4 Metallurgical Testwork

The key goals of metallurgical testing for the feasibility study were as follows:

- improve the metallurgical performance (i.e., metal recoveries and concentrate qualities) relative to the flowsheet detailed in the PEA
- develop process design criteria to support the plant design basis
- improve confidence in production forecasts, with a focus on ore that is processed during the project payback period (note: the results of testwork—126 open circuit tests, 15 locked cycle tests, and 83 comminution tests—were analyzed to develop predictive models that support mine design and sequencing, recovery, concentrate grades, throughput, revenue forecasts, and the development of a technoeconomic model)
- quantify the carbon sequestration potential of the tailings to support $CO_2$ storage forecasts.

Flowsheet optimization for the flotation and magnetic recovery circuits delivered improvements in recoveries and concentrate qualities across all commodities over the life of mine when compared with the PEA.

## 25.5 Mineral Resource Estimate

The mineral resource estimate has been completed using CIM standards and best practice guidelines (CIM 2014, 2019). The data used for the resource estimate and methods employed are considered reasonable for this level of study for the Crawford Project.

## 25.6 Mineral Reserve Estimate

The mineral reserve estimation for the project conforms to industry-accepted practices and is reported using the 2014 CIM Definition Standards. Mineral reserves include measured resources that are classified as proven reserves and indicated resources classified as probable reserves.

Factors that may affect the estimate include changes to long-term metal price assumptions; changes to recovery assumptions based on further metallurgical testwork; changes to marketing terms due to future negotiations; ability of the mining operation to meet the annual production rate, operating cost assumptions, process plant and mining recoveries; the ability to meet and maintain permitting and environmental license conditions; and the ability to maintain the social license to operate.

## 25.7 Soil Geotechnical

The open pit, covering an area of 10 km², has slopes in the overburden materials that have been designed using routine procedures with seismic considerations. Clay slopes are designed with an overall maximum angle of 6H:1V, and sand and till slopes at 5H:1V. For clay excavation, the recommended bench configurations include interbench heights of 5 m with a maximum slope of 2:1 and 20 m bench widths. In the sand and till layer, benches are designed with 5 m interbench heights, a 3:1 slope, and 10 m widths.

## 25.8 Structural Geology

The findings of the structural geology assessment were consistent with structures expected to occur in a strike-slip system, with steep near-vertical features splaying from one another and more moderately dipping structures accommodating movement between the near vertical structures.

Based on the current structural geological understanding the likelihood of having individual or combinations of major structures impacting the slope design beyond what could be managed operationally is currently low. However, the structural model is based on widely spaced drillholes, with no surface exposure and therefore is of low confidence. This is particularly true in the upper areas of the pit slopes. Additional data acquisition will be required to manage the risk in some early slope areas, as detailed Section 26, Recommendations.

### 25.9 Rock Geotechnical

The Crawford deposit is mainly comprised of strong metavolcanic and gabbro rock masses generally forming the hangingwall and footwall, respectively. The ultramafic mineralization package (peridotite, pyroxenite and dunite) are generally weaker.

The geotechnical domains and slope designs are largely based on the lithological and alteration models. Dominant controls on the pit slope stability on the bench and inter-ramp scale are expected to be kinematics in areas comprising non-ultramafic units, except where the slopes are impacted by the larger fault structures (CUC fault). Slope designs in areas comprised of ultramafic units were informed by structural/kinematic controls and potential rock mass failure, that in some cases influence overall slope stability.

The saddle between the East Zone and Main Zone pits should be depressurized to avoid increased pore pressures, especially once tailings are being placed in the Main Zone pit. When additional geotechnical and hydrogeological data is available in this region, the need of a depressurization program in this sector, including deep drain holes and piezometers, should be further evaluated.

Based on the observed slaking/degradation within its core and its thickness, the CUC Fault will pose a stability risk when exposed in the slopes. Where thicker, the weak CUC Fault is expected to impact slope stability on the inter-ramp scale.

The Dunite containing Coalingite is susceptible to a degradation process when exposed to the atmosphere. Potential rockmass failures on a bench scale are anticipated to control the immediate slope stability as unravelling on the bench-scale is likely to occur. A 15 m single bench geometry has been adopted in these areas.

Talc zones are modelled to intersect the east side of the East Zone pit and the east and northwest sides of the Main Zone pit within the ultramafic units. Narrow lamprophyre dykes are modelled to cut through the geological assemblage in the Main Zone `pit. Extent of these units as well as potential` impacts on stability are currently not confirmed.

The maximum stack height for some structurally controlled areas (metavolcanics and gabbro units) is 120 m; other slope domains are limited to stack heights of 90 m. In all mining areas stacks are to be separated by a 25 m safety berm. The minimum width of the saddle between the proposed East Zone and Main Zone pits was recommended to be 150 m. A 20 m step-off is required at the overburden/rock boundary to account for the construction of buttress and/or slope armoring, if required.

### 25.10 Mining

Crawford will employ conventional open pit mining techniques to mine 1,715 Mt ore and 3,992 Mt waste over a 33 ½ year life, including 2 ½ years of pre-stripping. Over the life of mine, production rates will average 512 kt/d and reach a maximum of 641 kt/d.

Crawford will employ three different fleets of mining equipment that are tailored to specific issues of the different lithologies that will be mined. The two different fleets that will be employed for mining overburden (including clay as well as sand and till), will be conventionally operated by on-board operators. The fleet of electric face shovels, rope shovels and 290 t payload trucks that will be employed to mine rock will employ several technologies that are already commercially available and considered proven. As these technologies are not yet widely adopted in Canada, assumed rates of productivity for this fleet are higher than has been achieved in peer Canadian operations, though in line with what has been achieved by similar fleets globally.

The Crawford pit will operate continuously, 24 hours per day and seven days per week.

## 25.11 Recovery Methods

The process plant will process ore at a nominal rate of 60 kt/d (21. 9 Mt/a) for Years 1 to 3, followed by a ramp up to 120 kt/d (43.8 Mt/a) by Year 5 of operation. The recovery methods align with conventional nickel sulphide practices in the industry. Ore comminution, flotation, and magnetic recovery of payable metals and handling of tailings are achieved through typical processes that are commonly used in the industry for similar projects. Previous studies, industry experience coupled with testwork results, were used to develop the resulting flowsheet suitable for optimal nickel recovery.

The recovery methods utilize a stage-wise expansion approach to appropriately manage desired production rates throughout the life of mine without incurring excessive capital costs early in the project. The expansions utilize twinned or parallel equipment wherever possible, as well as de-risked brownfield expansion activities and simplified engineering.

## 25.12 Infrastructure

The infrastructure for the project consists of open pit mines, a phased processing plant design, TMF, mine services, water management infrastructure and site roads. In addition, off-site infrastructure will include the relocation of the Highway 655, as well as a high-voltage powerline and rail spur from Timmins.

### 25.12.1 Tailings Management Facility

According to the mine plan, tailings will be deposited into a surface TMF until the middle of Year 18, after which time the tailings will be deposited into the Main Zone pit and then into the East Zone pit. It is estimated that 495 $Mm^3$ of tailings will be produced before in-pit deposition begins, and the surface TMF has been designed to hold 495 $Mm^3$ of tailings. The surface TMF will be located south of the mine and will have an approximate footprint area of 2,300 ha.

### 25.12.2 Stockpiles and Mining Impoundments

The stockpiles and mining impoundment facilities are designed considering the mine's operational life. The east and west stockpiles are designed to impound maximum volumes of 100 $Mm^3$ and 165 $Mm^3$, achieving design heights of 100 m and 70 m, respectively. The mining impoundment facility, spanning 31.5 $km^2$, comprises areas for clay (205 $Mm^3$), sand and till (182 $Mm^3$), and waste rock (1,438 $Mm^3$). The clay impoundment is divided into 28 cells separated by 34 m high waste rock ribs. The sand and till facility will achieve a final height of 50 m, and the adjacent rock facility is anticipated to reach 110 m. The construction strategy for the facilities includes specified rates of rise to ensure safety during construction.

### 25.12.3 Site Water Management Ponds

Collection ponds have been sized to attenuate the design inflows (1:10-year, 24-hour storm) and runoff from a mean average monthly precipitation, assuming release to the environment was limited to 28,000 $m^3/d$ (i.e., the capacity of the modular treatment plant).

At this stage, additional TSS treatment has been accounted for via modular treatment plants located at each pond. If discharge criteria are achieved within the collection ponds and additional treatment is not required, each pond will have a reverse-sloped gravity outlet that can discharge directly to the excavated channel.

The EDF will be managed by conveying excess flows from the collection ponds to the pits for temporary storage via an overflow spillway and excavated channel. The adopted EDF was the 1:100-year, 24-hour event.

### 25.12.4 Power Supply and Distribution

To meet the power requirements of both project phases, Hydro One Networks Inc. has indicated that it would be feasible to supply electrical power to the Crawford mine site from the Porcupine Substation located near Timmins, ON.

For Phase 1, an approximately 40 km overhead 230 kV wood-poled powerline will be constructed to a new 230 kV ring bus switchyard and 230 kV / 34.5 kV, 336 / 450 MVA outdoor substations located on the southwest corner of the Crawford mine site.

For Phase 2, a second 40 km overhead 230 kV wood-poled powerline will be constructed on a new right-of-way¸ following the same general routing as Phase 1. The proposed transmission lines will be financed, constructed, owned, operated, and maintained by Transmission Infrastructure Partnerships 1 (TIP1). CNC will pay service and transportation fees to TIP1 for delivery of electricity over a 25-year period.

On-site distribution across the project site will be via 34.5 kV overhead lines, 34.5 kV and 4.16 kV cabled circuits. Larger process loads will have a utilization voltage of 4.16 kV, whereas smaller loads will operate at 600 V.

## 25.13 Environmental, Permitting and Social Considerations

Environmental considerations identified to date are not expected to pose risks or uncertainties that affect the reliability or confidence in the exploration information, mineral resource, or mineral reserve estimates, or projected economic outcomes for the project. Sufficient schedule should be allowed for the federal impact assessment and provincial class environmental assessment processes as well as any subsequent federal and provincial permitting processes.

CNC is actively engaging with stakeholders and local Indigenous groups and has established a path forward with local Indigenous peoples to support future engagement and participation in the project to set the basis of negotiation for long-term collaboration agreements, such as impact benefit agreements, relationship-building agreements, and/or mutual support agreements. Long-term collaboration agreement negotiations are ongoing, and CNC will continue to work with Indigenous groups to understand potential effects of the project on traditional land uses and activities.

Environmental baseline information collection on the atmosphere, water resources, geochemistry, terrestrial, aquatic, and human and socio-environment commenced in 2021 for the site and surrounding areas. Based on the information collected to date, there are no environmental aspects that appear to be limiting project development. The environmental baseline studies will be used to identify environmental constraints, which may require refinements to the project components or additional mitigation measures to be implemented.

It is anticipated that the results of the impact assessment and class EAs, including implementing identified mitigation measures, will limit the potential for the project to cause significant adverse environmental effects, including effects from accidents and malfunctions, effects to the environment, and cumulative effects. Compliance with terms and conditions of approvals, standards contained in federal and provincial legislation and regulations, and commitments made during the EA processes (including application of mitigation measures and monitoring and follow-up requirements), will need to be addressed throughout project planning, construction, operation, and decommissioning.

## 25.14 Markets and Contracts

Pricing assumptions have been derived for all payable metal products, effective as of August 2023, based on third-party market surveys. First production from Crawford is planned for the end of 2027, which falls within the long-term period of these forecasts. For this reason, a single price, expressed in 2023 real terms, has been used for all years.

The nickel concentrate will be high grade, averaging 34% nickel over the life of mine, and relatively low levels of impurities. It will therefore be suited to either a traditional smelting and refining process, or for the concentrate roasting/reduction approach successfully developed by CNC management personnel and implemented in 2014 at Tsingshan. It is expected that the grade, range of processing options open to Crawford nickel concentrate and regional premiums will allow Crawford to command relatively high levels of payability.

The iron concentrate will be a potential direct feed for the stainless steel industry. Payability for this concentrate has consequently been based on typical terms for the various feeds to this industry.

## 25.15 Capital Cost Estimates

A capital cost estimate was compiled with costs grouped into major scope areas by WBS. The capital cost estimate includes all costs related to the Crawford Project (e.g., local infrastructure upgrades, open pit mine development, ore processing facility, tailings management facility, high-voltage substation and power supply infrastructure, offices, maintenance shops and utilities) to support mining operations.

The estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International). Most costs have a base date of Q4 2022, except for mining, tailings management, and water management costs, which have a base date of Q2 2023.

The estimate was prepared in Canadian dollars (CAD or C$). Life-of-mine project capital costs total C$6,611 million, which can be broken down as follows:

- Initial capital costs (required to construct all the surface facilities, and develop open pits to commence a 60 kt/d operation) are estimated to be C$2,556 million.
- Expansion capital costs (to expand mill throughput to 120 kt/d) are estimated to be $2,105 million.
- Sustaining capital costs (for mining and tailings development) are estimated to be $1,950 million.
- Additionally, closure costs (costs required to close, reclaim, and complete ongoing monitoring of the mine once operations conclude) are estimated to be $175 million.

## 25.16 Operating Cost Estimate

Operating cost estimates are based on a combination of first-principal calculations, experience, reference projects and factors. Operating costs include provision for mining, processing, tailings, and G&A. Operating costs have a base date of Q2 2023, and are expressed in Canadian dollars.

Operating costs have been calculated in three phases, as follows:

- Phase 1 covers the initial 42 months of operation when the nameplate capacity of the mill will be 60 kt/d. Operating costs for this phase are an average of C$23.00 per tonne milled.
- Phase 2 covers the following 26 ½ years of operation when nameplate capacity of the plant will be 120 kt/d and feed will predominantly come from the pits. Operating costs for this phase are an average of C$16.30 per tonne milled.
- Phase 3 covers the final 11 ¼ years of operation when nameplate capacity of the plant will be 120 kt/d and feed will be entirely sourced from low-grade stockpiles on surface. Operating costs for this phase are an average of C$8.30 per tonne milled.

Overall operating costs over the life of mine are an average of C$14.32 per tonne milled.

## 25.17 Economic Analysis

A techno-economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the project. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

The economic analysis was performed assuming an 8% discount rate. The post-tax net present value discounted at 8% ($NPV_{8\%}$) is US$2,475M and the post-tax IRR is 17.1%.

Macro-economic parameters that project returns are most sensitive to are the nickel price, CAD:USD exchange rate, and payability of nickel in Ni concentrate. For all three items, the relative variation in returns (either NPV or IRR) exceeds the relative variation in the macro-economic parameter. Project returns are less sensitive to variation in the price or payability of iron, and chrome, the payability of nickel in the iron concentrate, and energy prices (oil and electricity). For all these, the relative variation in returns is 0.2x to 0.5x variation in the macro-economic parameter. Returns are relatively insensitive to variation in the price or payability of cobalt and PGEs.

The operating parameter that project returns are most sensitive to is nickel recovery, with the relative variation in returns being 1.9x the relative variation in recovery. The sensitivity to recovery of other metals is similar to the sensitivity of prices for those metals. Returns are more sensitive to operating costs than capital costs. Despite life-of-mine expenditures at the concentrator being higher than for the mine, the front loading of mining costs results in returns being more sensitive to mine operating costs than the concentrator.

## 25.18 Risks

### 25.18.1 Geology

It is the opinion of the authors that at this stage of the project, there are no reasonably foreseen contributions from risks and uncertainties identified in the report that could affect the project's continuance at its current stage of exploration.

### 25.18.2 Metallurgy

The key metallurgical risks at Crawford relate to the ability to meet production and product quality forecasts. These risks are as follows:

- Recovery, throughput and concentrate quality forecasts were modelled based on the results of metallurgical variability testwork, which was completed on a limited number of samples that do not provide complete coverage of the Crawford reserve. Although the samples represent the expected mill feed variability, there is a risk that the models do not capture the behavior of zones of the deposit that have not yet been tested. This risk was considered in the sampling approach which targeted ore types that are expected to be processed during the payback period to minimize the risk to financiers. Areas of the deposit that would benefit from additional variability testwork include the East Zone starter pit, the low magnetite domain, as well as at depth in both the Main and East Zones.

- The potential for ore properties to differ from the engineering design parameters which could result in reduced plant throughput. Future engineering testwork is recommended to support the sizing of the regrind mills in the Crawford flowsheet, as well as to establish material handling properties for the design of stockpile reclaim systems, transfer chutes and bins.

### 25.18.3 Geotechnical

#### 25.18.3.1 Soil Geotechnical

After completing the fieldwork and assessments, several potential risk areas, related to the overburden slope design and mining, have been identified. These areas will require further investigation and refinement in future studies prior to the commencement of mining:

- The selection of the mining equipment was based on the stratigraphic database. However, the variable characteristics of the clay material with depth may reduce the efficiency of the selected mining equipment.
- Due to the expected higher hydraulic conductivity in the sand and till layer, as detailed in the hydro-geotechnical model produced by WSP and discussed in Section 16.1, seepage forces and/or potential internal erosion (either backward erosion or contact erosion) in the sand & silt layer could impact the stability of the overburden in the intermediate and final pit slopes, particularly under high hydraulic gradients.
- Should there be a significant hydraulic connection between the sand and till in the open pit and the esker, it could significantly challenge both the mining of the overburden and the maintenance of overburden slopes.
- Trafficability on the overburden units, even in the winter, will be a challenge and require careful road design for trafficability during mining of the overburden materials.
- A seismic event with a magnitude and intensity surpassing those considered in the project's design, resulting in the opening of cracks on the slope and causing concentrated flows, which in turn lead to significant erosion of the exposed clay slopes at the open pit.
- Management of large volumes of water and fine-grained materials during excavation is insufficient to prevent serious and localized slope failures within the overburden. These failures can have potentially negative impacts on both safety and production, and may also affect the water quality on local streams.

#### 25.18.3.2 Rock Geotechnical

Following the evaluation of the structural and geotechnical data, having considered the hydrogeological inputs and undertaken the slope design, the following are considered as potential risks that would need to be resolved pre-production or during the early stages of the rock slope development:

- The density of structural and rockmass data in the planned pit slopes is considered low based on the proposed large pit excavations, especially in the upper areas of the slopes. This poses the risk of not identifying major structures that are discontinuous, variations in minor structures, including the orientation and strength of the foliation, and/or rock mass variability that could impact the slope performance on the multi-bench and inter-ramp scales.
- The possibility of unidentified low angle faults, dipping 30° to 60° into the pit, which might cause slope stability issues. Especially possible in an identified region of high structural complexity in the northwest section of the Main Zone. The reliability of the structural model is impacted by the lack of outcrop, widely spaced drilling in the proposed open pit slopes and thick overburden package.
- There are three elevated risk areas, associated with current modelled major structures intersecting the final (Phase 10) planned pit. The upper centre of the south wall of the East Zone and lower centre south wall and the northwest wall of the Main Zone. The slope design has not been altered to account for these until confidence (currently low) in the orientation and location of these structures can be improved.
- The extent and influence of the coalingite (potential degradation) within the dunite (DUN-CG) is not well understood. The current DUN-CG model, produced by CNC, is believed to be conservative.

- The character and influence of the CUC Fault and location of other large structures are not well understood. The stiffness of the CUC Fault can also impact the performance of the overlying slope (southeast wall of the Main Zone) and needs to be considered during future evaluations. Furthermore, there is an indication that the CUC Fault damage zone causes slaking in the Metavolcanics unit; however, the extent of this potential rockmass damage and the consistency of this damage across/along the fault is not fully understood.
- Implementing a single ramp strategy for 345 m of the East Zone pit and 360 m of the Main Zone pit, as currently designed, results in access to the bottom of the pit being at risk should the single access ramp become compromised.
- The foliation in the northwest corner of the planned East Zone pit is currently interpolated with limited data but is modelled to drag slightly along the CUC Fault resulting in a region of complex foliation. This region requires additional assessment to confirm the foliation orientations and potential impact on slope designs.
- Caracle Creek has modelled talc units that intersect both planned pits. The extent of these modelled talc units could be limited by lack of data. Additional data acquisition and assessments will be required to confirm the extent of these talc units, along the footwall of the mineralized zones and the slope design updated accordingly.

### 25.18.4 Mining

The key mining risks at Crawford relate to the package of overburden overlying the deposits that average 38 m in depth. These risks are as follows:

- Challenges faced in the excavation and storage of various overburden materials will lead to productivities that are lower than expected with associated higher costs. Lower productivity in overburden could have a further negative impact of delaying the release of underlying ore. The feasibility study plan recognizes these risks and has taken a conservative approach selecting equipment (e.g., the overburden fleet selected for Crawford is smaller than what some peer operations and projects have attempted to use under similar conditions). Crawford's plan also mitigates schedule risks by the decoupling mining rates from the mill, resulting in 15 Mt of stockpiled ore at the time the plant is commissioned, including 11 Mt of high-grade ore.
- The potential for hydraulic connectivity to the nearby granular geological anomaly (esker), which could result in significantly higher water inflows to the pit than have been considered in the current design. This would have implications for schedule and cost. Some mitigation is provided by incorporating overburden stripping well in advance of mining the underlying ore. While testing to date has not revealed hydraulic connectivity, this possibility should be explored in future geotechnical investigation programs.

The Crawford mine plan makes extensive use of technology, in large part to mitigate risks associated with attracting and retaining the workforce required to operate a large fleet of conventional haul trucks, as well as the productivity and efficiency of manually operated equipment. All the technologies planned for use at Crawford are considered proven and are in commercial use elsewhere. However, it will be important to attract, train, and retain a higher skilled workforce to operate and maintain these technologies.

The water management and ramp designs have the following calculated risks:

- Surface water management ponds have been designed to a 1:10-year event. During periods when precipitation exceeds these limits, the pits would be allowed to flood the lowest bench.
- The lower portion of both Main and East Zone pits are equipped with a single ramp and, in the event of a significant failure, access could be lost.

As the Crawford design includes considerable measures of redundancy (both pits are available to provide ore, if necessary, for the initial 17 years of the 30-year total pit life after the mill has been commissioned) and contingent sources of ore feed (large surface stockpiles). Consequently, while flooding of a bench and/or loss of access at depth would be undesirable, neither would materially affect the supply of ore to the mill.

### 25.18.5 Infrastructure

#### 25.18.5.1 Tailings Management

The TMF tailings storage capacity assumes that the slope on the surface of the tailings "cone" will be 2.5% or higher, which is a reasonable assumption based on precedence. However, if the actual slope is less than 2.5%, it may become necessary to augment the capacity of the TMF.

#### 25.18.5.2 Stockpiles and Impoundments

After conducting fieldwork and evaluations, potential risks related to the low-grade ore stockpiles and rock impoundment facilities have been identified. These risks are expected to require further study and adjustments in future research before construction starts:

- The peat in some areas may be closer to amorphous peat than the fibrous peat that has been observed during the field investigations completed to date. This could potentially lead to slope stability concerns at the external slopes of the low-grade stockpiles and the rock impoundment facility, specifically unacceptable safety factors and deformation along the peat unit.
- Insufficient characterization of the foundation clay layer, especially in the rock impoundment facility area. Greater clay thickness than considered in the design could lead to lower safety factors for the facilities and potentially cause bearing capacity type failures (e.g., foundation failures).
- Uncertainty in the foundation clay's mechanical properties used for numerical modelling and stability analysis could impact the accuracy of the results and predictions for the facilities' stability.
- Uncertainty in the hydraulic conductivity of the foundation clay, specifically if the actual value is lower than what has been determined in the laboratory and used in the design, could compromise the stability of the structures.
- A rate of rise of structures exceeding that which has been considered in the design, based on the determined hydraulic conductivity and assumed foundation clay thickness from available site investigations, could lead to foundation failure in the facilities within the rock impoundment facility and low-grade stockpiles.

- Differences in the modelled volumes and locations of overburden units and the soils/rock boundary, due to the ongoing design of the project, could necessitate additional storage in the rock impoundment facility. This risk is mitigated by the current impoundment design having more than 100 $Mm^3$ storage capacity in excess of the current requirements.
- Long-term total and differential settlements on the order of meters due to slow dissipation of excess pore pressure accumulated in the foundation clay during construction and the foundation clay's low hydraulic conductivity. These settlements could compromise the stability and performance of various facilities during the closure phase.
- Seismic events could lead to significant build-up of pore water pressure in the foundation clay, inducing large deformations in the facilities and potentially compromising their performance or stability.

#### 25.18.5.3 Power Supply

The following potential risk to the project electrical power supply has been identified:

- Ongoing environmental baseline studies along the Phase 1 or Phase 2 routings may discover areas that need to be protected, which require the routing to be modified. This could result in higher capital costs and construction schedule delays.

#### 25.18.5.4 Processing

Risks for mineral processing include the following:

- The scale-up factors for flotation residence time and froth carrying capacity are typical for ultramafic nickel ores; however, they may not represent the ultimate plant performance, as they are based on historical benchmarks.
- The regrind technologies employed in the process plant are commonly available and employed in similar duties; however, fine grinding of ultramafic nickel ores can be challenging due to the presence of fibrous materials, which may impact recovery.
- The final iron concentrate is dewatered by magnetic separation. Potential poor performance of this stage could impair the filterability of the iron concentrate.

#### 25.18.5.5 Process and Infrastructure Buildings

Risks associated with construction of the surface infrastructure buildings (process plant, crushing, and mining buildings) include:

- Piling drivability conditions are unfavourable (e.g. due to rocky soil), requiring heavier piling rigs and/or a longer piling installation schedule.
- Bedrock is deeper than expected, requiring unanticipated field splicing of piles in order to reach bedrock, increasing piling cost and schedule.
- Soil lateral resistance is lower than expected, requiring larger diameter piles at higher cost.

- Clearing of the process plant area is currently planned for mid-summer, based on the anticipated date for receipt of permits. Clearing in summer may take longer than expected due to swampy or boggy conditions, potentially delaying the construction schedule.

The piling risks will be quantified better after completion of the borehole and piling test program in the process plant and crushing areas (see Section 26).

#### 25.18.5.6 Water Management

Risks associated with the water management assessment are as follows:

- Kinetic testing on the proposed tailings supernatant liquid was conducted on three samples of dunite and one sample of peridotite. The tailings have been represented by three lithologies and tested for their composition. Depending on the ultimate production process, the assessment may not be representative of the ultimate tailings and further assessment should be undertaken to verify the chemical parameters of the tailings.
- Limited data is available on the sediment loading and sediment characteristics expected from the site. The treatment efficiency of the settling ponds is uncertain at this time and as such, additional modular treatment has been included in the design. Bench-scale treatability testing should be carried out to confirm the targets in the receiving environment and to verify if additional constituents of concern need to be treated prior to discharge.

### 25.18.6 Environmental, Permitting and Social

After conducting fieldwork and baseline studies for the Crawford Project, the potential risks related to environmental, permitting, and social considerations are identified as follows:

- Following the Supreme Court of Canda ruling that the *Impact Assessment Act* is largely unconstitutional, there is uncertainty as to the process and enforcement of the Act, particularly as it relates to the preperation, consultation with regulators, and approvals for the Environmental Impact Assessment for the Crawford Project.
- Regulators determine if consultation/engagement efforts have been sufficient.
- There will be conditions of the Decision Statement for the Environmental Impact Statement and permit approvals that are required to be completed prior to construction commencing, the extent of which are unkown.
- With ongoing constraints in the public sector, CNC is monitoring the risk of agencies not meeting reasonable or the mandatory timeframe for permitting approvals.
- The extent of information requests by the governments, Indigenous Groups, and stakeholders is unknown.

The above risks have the possibility of extending the permitting and approval process, which may delay the start date for project construction and/or start of operations. To limit and/or minimize approval delays, CNC has engaged and will continue to engage with regulators, Indigenous groups, and stakeholders through the baseline studies, environmental assessment preparation, and permitting process.

### 25.18.7 Marketing and Financial

Note that all prices and valuations in the following paragraphs are given in US dollars (US$ or USD).

Crawford is a large-scale, low-grade operation, and overall project success is reliant on achieving the planned cost structure and associated profit margins. The cost structure and margins, in turn, are somewhat dependent on the macro-economic environment. It is conceivable that when Crawford begins operating, actual prices for goods and services consumed by the mine and/or prices for products that will be produced will differ adversely from those that have been assumed for this study. The chief mitigants to the financial risks are as follows:

- The project's high margins, with the life-of-mine EBITDA margin of 57%, indicate that the operation would remain profitable despite a simultaneous decrease in revenue and increase in operating costs of 40%.
- Over the long 41-year life of the mine, it is to be expected that macro-economic factors would revert to the mean.

It is also conceivable that realization terms will vary adversely from what has been assumed in this study. The impact of variation in payability has been illustrated in Section 22.5. Most significant would be a reduction in the payability of nickel in the Ni concentrate, with each 1% reduction (in absolute terms) from the currently assumed 91% reducing NPV by $47 million. A 1% reduction in the payability of iron (equivalent to a $2/t reduction in the iron ore equivalent price, from $89/t to $87/t) would have a $28 million impact on NPV. A 1% reduction in chromium payability (in absolute terms) would reduce NPV by $16 million. As reported in Section 22.5, returns are relatively insensitive to variation in the payability of cobalt or PGEs, with a 1% reduction in payability of either having less than a $2 million impact on NPV.

It is possible that the structure of offtake contracts will differ from what has been assumed for this study, which is that the cost of downstream treatment and refining is captured through a payability less than the recovery of the downstream facilities. It has been further assumed that the spread (between payability and recovery) would aim to capture an all-in treatment charge (TC) in line with that charged to other concentrate producers. The example given in Section 19 for Lundin's Eagle Mine (81% payability for a 12% Ni concentrate and $21,000/t Ni price) equates to an effective all-in TC of approximately $400/t for a smelter with recovery of 97%. The 91% payability that has been assumed for Crawford in this study results in a downstream processor achieving a modestly higher all-in TC. The cost structure for conventional downstream processors is such that, broadly, approximately 50% of costs are incurred in smelting and the remainder in refining. Crawford's much higher grade concentrate would result in a smilar increase to the refining component of costs and therefore significantly higher overall costs. Nonetheless, CNC believes that, for a North American based conventional downstream processor, the higher costs associated with refining Crawford concentrate would be offset by regional premia for end products (Figure 19-4) and/or the much lower cost roast reduction/reduction process could be employed.

In the event Crawford were charged an all-in TC that aimed to capture the entirety of costs incurred by its higher grade concentrate, payability for Ni and Co in the nickel concentrate could be reduced as low as 85.6% and 23%, respectively. Parameters underpinning these payabilities include:

- life-of-mine average concentrate grades of 34% Ni, 0.7% Co and 11% MgO
- metal prices of $21,000/t Ni and $40,000/t Co
- full cost TCs of $180/t and RCs of $0.83/lb N and $4.00/lb Co
- MgO penalty of $2.50/t for grades higher than 6%.

The lower paybilities would reduce NPV and IRR by $325 million and 1.2%, respectively.

### 25.19 Opportunities

#### 25.19.1 Geology

##### 25.19.1.1 Additional Targets

Excluding opportunities that are universal to all mining projects, such as improvements in grade and tonnage, higher metal prices, improved exchange rates, etc., there are several opportunities, mostly technical, that could enhance the project. Despite being covered by 10 m (or more) of overburden, with 196 holes targeting the project's Main Zone and 126 holes targeting the East Zone, there is an overall (at medium- to large-scale) good understanding of the geology, alteration, mineralization, geochemistry, and geometry of the ultramafic bodies. Because of this, the Crawford Ultramafic Complex (CUC) offers good potential for developing further low-grade, large-tonnage nickel (Co, Pt, Pd, Fe) resources and should continue to be investigated.

One of such instances is the continuity of the Main and East Zones beyond their current boundaries. To begin with, both seem to be open at depth, based on a few deep drillholes. Additionally, opposite to their faulted side, the Main Zone is open and has been proved to extend into what's now known as the West Zone; while the East Zone has yet to prove an apparent easternmost extension.

Other potential secondary targets are the Northern West Zone (see Section 10.3.4) as well as the North Zone (see Section 10.3.5).

##### 25.19.1.2 PGE Reefs

Drilling to date has identified three discrete zones (<10 m wide) of peridotite-pyroxenite-hosted higher grade PGE mineralization, referred to elsewhere in this report as the PGE reefs. One of these reefs is located in the immediate footwall of the Main Zone, with the other two in the immediate footwall of the East Zone. The current density of drilling is sufficient only to classify a portion of these reefs as inferred resources. Provided infill drilling were to confirm the behaviour witnessed thus far, it is possible the reefs could aggregate to 20 to 25 Mt grading 0.7 to 0.9 g/t combined Pt and Pd, for 450 to 700 koz combined Pd + Pt.

Preliminary geological and geostatistical analyses have determined that, in order to achieve indicated-level resources, it is necessary that the average sample distance approximates 75m. This estimate suggests that 20 to 30 infill holes totaling between 7,000 and 8,000 m would be required to bring this material to the indicated category.

The style and geometry of reef mineralization would require more selective mining and smaller equipment than will be employed in the main nickel orebody. The required equipment will already be available on site, as it will be used for mining overburden. Mining the PGE reefs will not require additional stripping as their proximity to the nickel orebody mean they have already been included in the mine plan, and are currently considered waste.

Preliminary metallurgical testing of reef-grade material suggests that a recovery of 50% to 70% for both platinum and palladium would be achievable, resulting in incremental production of 225 to 490 koz combined Pd + Pt. The grade and recovery of the reefs is such the resultant value would lead this material to be fed as run-of-mine material to the mill and not stockpiles. Consequently, incremental metal production would be realized over the 30 years that the two pits were active.

#### 25.19.2 Metallurgy

Additional opportunities for improvements in the project metallurgy have been identified:

- There is potential for additional nickel recovery by flotation of the deslime cyclone overflow slimes stream. This stream currently reports to tailings without further processing.
- The carbon sequestration system was designed for 6.5 hours of residence time in the sequestration tanks. Preliminary results from test work indicate there is an opportunity to shorten the residence time, which would reduce the required tank volume and capital costs.
- The carbon sequestration system was designed assuming 50% utilization. Laboratory and pilot testing indicates that higher $CO_2$ utilizations could be achievable, which would result in capital and operating cost savings associated with gas recompression and recirculation.

These opportunities will be further evaluated as part of the planned bench scale and pilot plant test work in 2024.

#### 25.19.3 Mining

Key opportunities include the following:

- The use of lime to stabilize clay in order that it can be used instead of waste rock for certain construction activities, thus reducing the tonnage of required pre-stripping.
- In the event capacity of the TMF can be increased beyond the current design of 495 Mm3, it would be possible to reduce the elevation of the 'saddle' separating the Main and East Zones, and thus release some of the high value material sterilized with the current design.
- Increasing the TMF capacity would also permit a more optimal mining sequence, with the East Zone being prioritized over the Main Zone.

These opportunities are discussed in more detail below.

##### 25.19.3.1 Stabilized Clay

Clay is commonly referred to as a fine grained material, but it often includes non-plastic, medium to fine silt. Clay has different chemical properties than sand or silt. Clay particles also have a different geometry, being planar where silt particles are more rounded, angular, or subangular. The chemical properties of clays allows a mixture of lime and water to form cementitious bonds, which in turn enhances the strength of the clay to the extent that it may be used as an engineered construction material. In practice, stabilization to a level allowing the material to be used as an engineered material typically occurs when a minimum of 25% of the material is comprised of clay-sized particles.

Early analysis has identified that Crawford clay-containing soils of interest contain approximately 40% clay-sized particles. As a result, it was not surprising that laboratory scale test work demonstrated the ability of lime to strengthen Crawford clays. However, the quantum of strengthening was unexpected, with some samples achieving a compressive strength of 220 kPa, which significantly exceeds requirements for many construction activities.

The key unknowns at this stage are the ability to mix the clay with lime at an industrial scale despite the observed stickiness of the dominant clay, and the ability to achieve laboratory results on an industrial scale given issues associated with cold weather and potential access limitations.

If it were possible to stabilize clay on an industrial scale, applications where it would be considered for use during the pre-stripping period are as follows:

- within the perimeter embankment and ribs of the clay cells
- within the highway realignment embankment
- temporary site roads
- armouring of the clay surface for mining.

The total volume of waste rock currently required for construction is approximately 14 $Mm^3$, with the largest single requirement being the embankment and ribs for the clay cells. If 30% of the rock planned for each of the above applications were replaced with stabilized clay (i.e., the bulk of construction material for each application would remain rock), the total volume of stabilized clay would be 3.8 $Mm^3$. This would reduce the pre-strip mine plan by a similar volume of waste rock (or 8 Mt), which would also impact the volume of overlying overburden requiring stripping. In turn, the reduction in overburden stripping and removal of 3.8 $Mm^3$ stabilized clay from the clay cells would lead to a virtuous circle of further reductions. It has been conservatively estimated that the total tonnage of pre-stripping under this scenario could be reduced by approximately 20% to 80 Mt. The estimated saving associated with this reduction in pre-stripping, net of the lime addition required for stabilization, is on the order of C$100 million.

While this material and associated costs would not be permanently removed from the mine plan, their mining and associated costs could be deferred to such time as the operation is generating positive cash flows and would therefore not require financing.

#### 25.19.3.2 Reduced Saddle Pillar

As discussed in Section 15.5, the 'saddle' has been raised to the RL120 horizon, 150 m below surface, in order to ensure in-pit storage for tailings is available when the TMF capacity of 495 $Mm^3$ is reached. This horizon is 45 m higher than the elevation in the LG RF65 pit shell used as the basis for the engineered pit design. As a result, 78 Mt of high value measured and indicated resources (the average NSR is 41% higher than that of the overall mine) are sterilized.

Two key assumptions used in calculating the storage capacity of the TMF are as follows:

- An average clay thickness around the entire perimeter of the TMF of 30 m. While this value corresponds to the deepest clay encountered, it was considered a prudent assumption given the relative lack of data. The depth of clay, in turn, is a key factor limiting the ultimate height of embankment to a nominal 17 m. In the event the average depth of clay is less than assumed, it may prove possible to raise the embankments. Given the area of TMF within the embankment is approximately 2,000 ha, each metre of raise would increase the capacity of the TMF by approximately 20 $Mm^3$, which translates to 30 Mt of ore.

- The beach slope of tailings was assumed to be 2.5%, based on that reportedly achieved by neighbouring Kidd Creek. Notwithstanding the geographic proximity, there are significant differences between the two tailings products, in part due to Crawford's IPT process and associated sequestration of $CO_2$. Preliminary work has shown that beach slope angles achievable at Crawford may be markedly higher. In turn, this would allow the centreline to be raised higher for the current embankment height.

The increase in TMF capacity necessary to release the entire tonnage of sterilized ore (i.e., by dropping the 'saddle' 45 m) has been estimated at 100 $Mm^3$, or 20%. In the event this were achieved, the release of sterilized ore would extend life by two years or 5%. As it has a higher stripping ratio, the total tonnage mined would increase by 7% to 6,120 Mt. As the sterilized ore is higher value than the overall mine average, total undiscounted cashflow could rise as much as 6%. However, the incremental material would be released later in the mine life (likely in Years 30 to 33 of the now 33 years of open pit mining) and the impact on NPV would be less, at $50 to $80 million, or 2% to 3%.

#### 25.19.3.3 Mine Sequence

Despite the Ni and Fe grades in the Main Zone being approximately 5% higher than those at the East Zone, the average NSR is more than 10% lower due to forecast lower recoveries. This lower value is partially offset by the lower strip ratio at the Main Zone, which allows ore to be mined at a higher mining speed factor (MSF, being the ratio of ore mining rate to mill processing rate). The initial Main Zone phase, MZ1, is also the highest value phase (measured by EBITDA) as a function of both a high NSR and low strip ratio. The low strip ratio has the added benefit of allowing MZ1 to be mined at a high MSF—the results of which were illustrated in Figure 16.30 in Section 16.

In the event it was possible to expand capacity of the TMF and therefore mine more material before a depleted pit was available as a tailings impoundment, it would be beneficial to prioritize the higher value East Zone ore ahead of the Main Zone. A conceptual model was developed that allowed the sequence of phases to be altered. This model relied on the base case timing of infrastructure development by phase (notably the trolley-assist system), and scenarios that accelerated the East Zone were penalized as the southern exit trolley line was assumed to be established only with the later Main Zone stages. The lower resultant trolley utilization means that scenarios prioritizing the East Zone have higher capital and operating costs that collectively result in a 1% reduction in undiscounted free cash flow relative to the base case. It is likely that an engineered design and schedule tailored to an East Zone start-up would eliminate this cost premium.

This model showed the value of maintaining MZ1 in its current sequence. As a result of the higher operating and capital costs associated with accelerating the East Zone that are discussed above, a plan that defers MZ1 until completion of the entire East Zone would lead to an NPV improvement comparable to that of releasing sterilized ore in the 'saddle' or $50 to $80 million.

Much greater value would be realized with a mine plan that maintained the sequence for MZ1 then mined the entire East Zone to completion before transitioning back to Main Zone. This scenario has an NPV improvement of $225 to

$275 million. In both cases, returns would be further improved by an engineered design and schedule tailored to the East Zone start-up, that eliminated the current cost premium associated with the lower trolley utilization.

The current mine plan results in the depleted Main Zone pit shell being available to receive tailings approximately six months before the limits of the TMF are reached. The six months buffer equates to approximately 100 Mt of mining. This compares to an additional 685 Mt mining before the depleted East Zone would become available for tailings, in the scenario where MZ1 maintains its sequence. The net 585 Mt represents 2.5 to 3 years of activity, or 80 $Mm^3$ to 100 $Mm^3$ in tailings production.

### 25.19.4 Processing

Opportunities for mineral processing include the following:

- A decant pond could be considered for nickel thickener overflow, filtrates from the nickel and magnetic concentrate filters. Collecting these streams in the decant pond before overflowing to the process water pond posts additional metal recovery and ensures process water quality.
- The IPT carbonation process is completed in a series of agitated, enclosed tanks where carbon dioxide is passed through the slurry, facilitating capture and sequestration. There is an opportunity to optimize equipment selection for this process, exploring alternative solutions that offer increased energy efficiency and reduced capital costs.

The current mass balance predicts a total loss of 24% nickel between the primary and secondary deslime cyclone overflows. A deslime flotation circuit could be considered to recover the metal loss from these two streams.

### 25.19.5 Process and Infrastructure Buildings

Opportunities associated with construction of the surface infrastructure buildings (process plant, crushing, and mining buildings) include the following:

- Bedrock depth is shallower than anticipated within the building footprints, requiring less pile depth and piling installation time.

### 25.19.6 Power Supply

The following potential opportunity for the project electrical power supply has been identified:

- Ongoing discussions with Hydro One may identify the possibility of the proposed TIP1 electrical line becoming part of the Hydro One Network as a regulated network asset under certain conditions. This may result in lower transmission services fees and lower overall power operating costs for the project.

### 25.19.7 Financial Opportunities

As discussed in Section 22, the base case analysis assumes Crawford would be eligible for the Clean Technology Manufacturing (CTM) Investment Tax Credit (ITC), that was outlined during the 2023 federal budget presentation. However, the Carbon Capture, Utilization and Storage (CCUS) ITC, for which draft legislation was recently published, has not been included because Ontario has not been included with Saskatchewan, Alberta, and British Columbia as an eligible jurisdiction. It is believed this is solely due to the lack of a fossil fuel industry in Ontario and the associated historical lack of carbon sequestration opportunities.

The long life (exceeding 40 years) and scale (steady-state capture and storage in excess of 1% of Ontario's annual output) are aligned with the federal government's overall carbon strategy which should allow Crawford to qualify for the ITC.

Adoption of the draft legislation would result in a sub-set of the capital expenditure qualifying for both the CTM ITC and CCUS ITC; however, it is not possible to claim both. The refund under CCUS is higher than with CTM, as outlined below:

- CCUS – Carbon capture investments are eligible for a 50% refund until 2030, and then a 25% refund until the ITC is discontinued in 2041.
- CCUS – Carbon storage investments are eligible for a 37.5% refund until 2030, and then an 18.75% refund until the ITC is discontinued in 2041.
- CTM investments are eligible for a 30% refund through 2031, after which they ramp down to 5% by 2034 and are discontinued in 2035.

Eligible CCUS are also classified as 'development' (associated with the capital project) and 'refurbishment' (associated with sustaining capital). Aggregate refunds under 'refurbishment' are capped at 10% of the total for development. At this stage, it has not been specified how expansion capital expenditures would be treated. However, it is assumed that the claimant would have discretion over the specific ITC being claimed for specific expenditures, which would minimize the impact of an adverse decision regarding expansion expenditures at Crawford (i.e., expansion expenditures could be claimed under CTM if classified as refurbishment under CCUS).

With the combination of a higher refund and longer life that the CCUS ITC provides select investments, the overall project NPV would increase by $125 to $175 million, while the required peak investment would be reduced by $200 to $250 million.

## 25.20 Conclusions

Based on the assumptions and parameters presented in the report, the project has a mine plan that is technically feasible and economically viable. The positive financials of the project (US$2,475 million after-tax $NPV_{8\%}$ and 17.1% after-tax IRR) support the mineral reserve.

# 26 RECOMMENDATIONS

## 26.1 Overall

Analysis of the results and findings from each major area of investigation completed as part of this feasibility study suggests numerous recommendations for further investigations to mitigate risks and/or improve the base case designs. The following sections summarize the key recommendations arising from this feasibility study. Table 26-1 presents a summary of recommended tasks, budget and detailed in the subsections that follow. Recommendations are split into two phases; Phase 2 is contingent on the results from the tailings and geotechnical work in Phase 1.

**Table 26-1: Cost Estimate for Proposed Recommendations**

| Description | Cost (C$M) |
|---|---|
| **PHASE 1** | |
| Exploration Drilling – Deep Drilling | 3.40 |
| Exploration Drilling – Borehole Geophysics | 0.25 |
| Infill Drilling – Nickel Resources | 4.60 |
| Infill Drilling – PGE Reefs | 1.80 |
| Metallurgical Testwork | 5.00 |
| Mining (Bulk Sample) | 6.50 |
| Process Plant Geotechnical (Boreholes, Pile Testing and Seismic Refraction) | 1.10 |
| Soil Geotechnical | 3.25 |
| Rock Geotechnical | 0.85 |
| Hydrogeology | 0.12 |
| Surface Water Management | 0.25 |
| Tailings Management Facility (Phase 1) | 4.10 |
| Environmental | 8.00 |
| Marketing | N/A |
| **Total – Phase 1** | **39.22** |
| **PHASE 2** | |
| Tailings Management Facility | 2.10 |
| Soil Geotechnical | 0.45 |
| **Total – Phase 2** | **2.55** |

## 26.2 Exploration Drilling

Future drilling should also test the deposits for deeper higher grade nickel. Consideration should be given to borehole electromagnetic surveys (BHEM) in selected areas to target regions of higher percentage sulphide (semi-massive sulphides) along with in-hole induced polarization (IP) surveys to target regions of lower percentage sulphides (disseminated sulphides).

Approximately 15,000 m of deep drilling is proposed. At an estimated all-in drilling cost of $225 per metre, the program is estimated to cost $3.4 million.

Twenty-five holes are recommended for BHEM and IP surveys, at an estimated cost of $10,000 per hole, for a total cost of $250,000.

## 26.3 Mineral Resource Infill Drilling

### 26.3.1 Current Nickel Resources

Based on the work completed for the mineral resource estimate, it is recommended that programs aimed at infill drilling be undertaken to upgrade current resources to the measured category and continue to improve the geological models.

Approximately 20,000 m of infill drilling is proposed. At an estimated all-in drilling cost of $225 per metre, plus an allowance for updates to the mineral resource estimate and models, the program is estimated to cost $4.6 million.

### 26.3.2 PGE Reefs

To realize the opportunity presented by the PGE Reefs, it is recommended these targets be infill drilled to potentially upgrade current inferred resources and exploration potential to the achieve the indicated category. Approximately 7,000 m of infill drilling would be required, at an all-in cost of $1.8 million.

## 26.4 Metallurgical Testwork

The next phase of testwork will be completed to support detailed engineering of Crawford. The following testwork is recommended:

- additional metallurgical and comminution variability testing on samples from the East zone starter pit and low magnetite ore domains
- flume testing of carbonated tailings to confirm the design criteria for the tailings beach slope in the tailings management facility (CNC considers the current design basis to be conservative and that there is an opportunity to reduce the TMF footprint)
- further testwork on the IPT carbonation process to support the engineering of the $CO_2$ gas recirculation systems, improve the quantification of $CO_2$ degassing from the tailings, and evaluate the impact of lower $CO_2$ gas strengths on the carbonation process performance

- grindability testing (Bond ball work index and HIG mill signature plots) on streams that feed the regrind mills
- dynamic settling and filtration testwork on the final concentrates
- material handling testwork.

To generate samples for engineering testwork, a pilot plant will be required. The scale of the pilot plant should be increased to include the full flotation cleaning circuit and demonstrate the grade-recovery performance of the entire metallurgical flowsheet. As part of the pilot plant, the potential to recover nickel from the slimes stream should be evaluated. Slimes flotation is very challenging to test in the laboratory. However, CNC believes that there could be between 1 to 5 percentage points of additional nickel recovery from this stream. Finally, the IPT carbonation process should be integrated into the flowsheet to develop operational knowledge on the integrated plant.

The expected cost to complete this test program is approximately $5 million, assuming that ore can be collected from the bulk sample program that is planned for winter 2024.

## 26.5 Mining

The feasibility study plan uses assumptions relating to the mining of overburden, including both clay and sand and till, based on the experiences at peer operations. Trial mining should be undertaken at Crawford as part of a bulk sample program. Notwithstanding the importance of budget and schedule, the primary focus of trial mining should be data collection and analysis. The cost of this bulk sampling program, including costs associated with testing the stabilization of clay and drill performance, is estimated at C$6.5 million.

The feasibility study plan assumes that clay will have no use as a construction material other than as the impermeable core of the TMF. As a result, in excess of 100 Mt of pre-stripping is required to liberate sufficient waste rock to meet all the construction requirements. Clay that is excavated during bulk sampling should be tested with lime addition to determine if it can be suitably stabilized for use in certain construction activities, notably the construction of embankments for the highway realignment and for clay storage.

The feasibility study plan also uses assumptions regarding drilling performance, including penetration rate and bit life, that are also based on the experiences at peer operations. An OEM should be invited to perform test drilling of Crawford’s lithologies as part of the bulk sample program. The focus of these tests should the ore lithologies (dunite and peridotite) as these are not as prevalent in the Abitibi and therefore less well understood than the waste lithologies (gabbro and basalt).

The feasibility study plan assumes it will not be possible to store more than 495 $Mm^3$ tailings in the TMF. Ensuring that there is an in-pit deposition site available when this capacity is reached necessitates the sterilization of 78 Mt ore as well as a potentially sub-optimal mining sequence. Geotechnical data that aims to better quantify the achievable height of TMF embankments and beach slope of deposited tailings should be gathered (and therefore the overall capacity of the TMF). The cost of this geotechnical work is estimated at C$0.1 million. As the optimal mining sequence would still entail early mining of the MZ1, it is not essential that this work and a definitive conclusion be drawn prior to the start of mining.

The feasibility study plan includes 84 Mt of inferred resources above the marginal cut-off grade that is currently considered as waste. Shorter-term mine plans should allow sufficient time for this material to be potentially upgraded, either through blast hole sampling or infill resource drilling, so upgraded material may be included as mill feed.

## 26.6 Process Plant Geotechnical

The following activities are recommended to be completed prior to detailed engineering:

- Additional geotechnical borehole drilling at the Phase 1 process plant and primary crusher locations is recommended to provide further definition of bedrock depth and rock quality. Thirty-seven boreholes are recommended at an average cost of $15,000 per borehole. Including an allowance for laboratory testwork, the estimated program cost is $700,000.
- A pile testing program should be carried out to verify pile capacity, effective length of the pile, and constructability at the process plant and crushing area locations. Twelve piles are recommended to be driven of various sizes and in various locations. The total program cost is estimated at $300,000.
- Seismic refraction (geophysics) testwork is recommended to establish a 3D profile of the underlying bedrock in the area of the process plant and crushing areas. The total cost of this program is estimated at $100,000.

## 26.7 Soil & Rock Geotechnical

### 26.7.1 Soil Geotechnical

Based on the identified risks related to the design of the overburden pit slopes and mining impoundments, the recommendations described below are made.

#### 26.7.1.1 Phase 1 Recommendations

- Geomorphological Study – A detailed geomorphological study should be carried out to investigate and analyze the landforms, soil, and sub-soil characteristics, as well as the natural processes that have shaped the terrain in the project area. The cost of this study is estimated at $200,000.
- Soil Characterization – Based on the outcomes of the geomorphological study, further detailed geotechnical site investigations, including boreholes, field tests, and advanced laboratory testing on high-quality soil samples, should be conducted. The proposed geotechnical site investigation includes 60 boreholes, 30 cone penetration tests, and 15 in-situ soil permeability tests, among others. The estimated cost for the geotechnical site investigation, including laboratory tests, is $2 million.
- Hydraulic Conductivity Reassessment – Additional laboratory and field permeability tests should be conducted to better characterize the hydraulic conductivities of the overburden materials. The estimated costs for the hydraulic conductivity reassesment are included in the above estimated cost for the geotechnical site investigation.
- Large-Scale Consolidation Field Test – A large-scale, in-situ test should be completed at the site by constructing an embankment, approximately 5 m high x 10 m long and with 3H:1V slopes, over the normally consolidated clay. Instruments should be installed to measure settlement and pore pressure in the foundation prior to embankment

construction. Prior to the test, the underlying clay should be sampled and tested. The estimated cost for the field test is $800,000. This cost includes construction of the embankment (using the bulk sample contractor), laboratory tests, four vibrating wire piezometers, two extensometers and installation, and data collection/processing over three months.

- Large-Scale Open Pit Model – During the bulk sample campaign in 2024, the excavation should be used as a scale model of the open pit. The excavation should be instrumented to monitor slope deformations and groundwater drawdown in the overburden due to water pumping. The cost of this scale model is estimated to be $400,000. This cost includes 12 vibrating wire piezometers and their accessories, surface displacement measurements, installation of instruments, supervisory personnel during excavation, data collection, processing and interpretation. It does not include the costs of earthworks associated with the open pit excavation (assumed to be part of the bulk sample program).

#### 26.7.1.2 Phase 2 Recommendations

- Trafficability – Trafficability on the overburden units should be evaluated in more detail, including during winter conditions. A detailed road design will be required to promote safe and efficient mining operations. The estimated cost for the evaluation of trafficability and a detailed road design is $150,000.
- Seismic Load Consideration – During detailed design and stability analyses, site-specific seismic loads should be included. The estimated costs for considering seismic loads is $200,000. This cost includes a site-specific seismic hazard study and deformation analysis of the overburden pit slopes and mining impoundments under seismic loads.
- Comprehensive Water and Fine-Grained Materials Management Plan – This plan should be developed to mitigate potential safety and environmental impacts during excavation. The cost of this study is estimated at $100,000.

#### 26.7.1.3 Operational Recommendations

- Continuous Design Refinement – As new data becomes available from ongoing investigations, design optimizations, and operations, the model of overburden units and soils/rock boundary should be updated. These ongoing adjustments should continue during operations.
- Clear Construction Specifications – The construction specifications will clearly outline the allowable rate of rise, considering the site-specific hydraulic conductivity and clay thickness.

As these recommendations are intended to be carried out during mining operations and are not part of Phase 1 or Phase 2, no costs have been estimated.

### 26.7.2 Rock Geotechnical

Following the design assessment and with consideration to the identified risk areas above, the recommendations listed below are made for Phase 1:

- Prior to construction of the pits, more structural and rock mass information should be collected, and the current models should be tested to confirm the presence and orientation of structures in the interim and final planned pit slopes as well as rock mass assumptions. It is recommended that eight geotechncial drillholes be completed on the CUC fault and upper areas of the pit slopes. The approximate cost to undertake this proposed program will be $600,000.

- During the bulk sample program, limited rock slopes should be used to calibrate rock mass parameters. The approximate cost to undertake this program will be $40,000.
- Following additional rock geotechnical drilling and bulk sample data aquisiotion, additional validation of the stability of the final proposed pit depths (~660 to 700 m) is recommended. This will consist of an analysis of multi-bench and overall stability as well as potential deep-seated slope deformation. The slope designs should be reviewed based on this information and the information obtained from the above drilling program should be used to update the slope design in affected areas. The approximate cost to undertake this work is $150,000.
- A core review, with an approximate cost of $60,000, should be undertaken with the following objectives:
  - to investigate the extent, influence, and time dependency of the dunite with coalingite (DUN-CG) unit degradation and its impact on bench and ramp stability
  - to investigate the extent and influence of the slaking in the damage zone of the CUC fault on the metavolcanics side and the possibility of slaking in damage zones of other modelled major structures
  - to investigate the extent and influence of talc zones along the footwall of the mineralized zone in both the Main Zone and East Zone pits.

## 26.8 Hydrogeology

The numerical groundwater model currently predicts higher infiltration above the esker which is slightly west of the open pit footprints. Recharge to the pits can occur through surficial sand if there is a direct hydraulic connection as inferred by the model. To confirm this, a multi-day pumping test is recommended to provide clarity on sand exposure at surface and to calibrate the recharge rates in the groundwater model. The estimated cost for this program is approximately $120,000.

## 26.9 Surface Water Management

Recommendations associated with surface water management are as follows:

- Considering the mining footprint makes up a significant portion of the overall watercourse catchment, potential impacts to the aquatic receiving environment due to reductions in flows during the winter period should be further assessed to ensure no adverse impacts.
- A trial pad or other investigations should be carried out to better understand the anticipated sediment loading and sediment characteristics. This information can be used in the next stages of design to refine the collection pond sizing and further assess the design criteria of the modular treatment plants.
- The assimilative capacity of the proposed receiving watercourses appears to be adequate during open flow season (i.e., April to November) to accept predicted effluent discharge. In conjunction with assessing the sediment loading,

the assimilative capacity of the watercourses should be reviewed for the EDF (1:100-year storm) to confirm if pond treatment alone is adequate to meet assimilative background conditions.

- Further refinement of the modular treatment capacity and efficiency should be undertaken. Bench scale treatability testing should be carried out to confirm the targets in the receiving environment. The characterization of the different inflow (i.e., WRD/LG stockpiles vs. TMF) should be reviewed as the two inflows water may have a different chemistry.

- Progressive reclamation should be reviewed further to assess if treatment assumptions and overall life of mine treatment requirements can be reduced.

- Test pits should be dug in the footprint of the future water management ponds and ditches to confirm bedrock and soil profiles prior to construction.

The estimated cost for the above program is approximately $250,000.

## 26.10 Tailings Management Facility

To support the detailed design of the TMF, a Phase 1 program of additional work is recommended, as summarized in Table 26-2. To support the planning for closure of the TMF, a Phase 2 program of additional work is recommended, as summarized in Table 26-3.

**Table 26-2: TMF – Phase 1 Recommended Work Program**

| Program Component | Estimated Total Cost ($M) |
|---|---|
| Flume testing of tailings to help confirm the achievable tailings slope and the solids content required from the tailings thickener. | 0.1 |
| Complete a more extensive geotechnical investigation of the TMF construction areas prior to tendering. | 2.0 |
| Complete a test embankment program to determine the efficacy of wick drains at various spacings. | 1.0 |
| Conduct wind tunnel testing on tailings to estimate the extent of dusting that will occur during operations. Include observations of self-cementing of the tailings surface and analysis of the chemistry and mineralogy of the dust particles. | 0.5 |
| Carry out a detailed study and inventory of potential sources of materials for the construction of the TMF perimeter dykes and prepare a mass flow plan over the life of the facility. | 0.5 |
| **Total** | **4.1** |

**Table 26-3: TMF – Phase 2 Recommended Work Program**

| Program Component | Estimated Total Cost ($M) |
|---|---|
| Carry out test plots for revegetation of the surface of the TMF to optimize the addition of organics and fertilizers and the seed mix. | 2.0 |
| Carry out trafficability studies to determine the type of equipment to be used to construct the revegetated cover over the TMF. | 0.1 |
| **Total** | **2.1** |

## 26.11 Environmental, Permitting & Community Relations

As indicated in Section 20.1, CNC prepared and submitted the Crawford Initial Project Description and Detailed Project Description Report to the Impact Assessment Agency of Canada to initiate the regulatory assessment process. The Impact Assessment Agency of Canada decided that a federal impact assessment is required for the project and published the Tailored Impact Statement Guidelines in response. A draft environmental impact statement is expected to be prepared by CNC in 2024. Permitting for site-specific activities related to the project's construction and operation are anticipated to commence concurrently during the preparation of the environmental impact statement. Completion of the federal impact assessment, provincial class environmental assessments, future conditions of approval, and permitting for site-specific activities could require refinements to the project components or additional mitigation measures to be implemented.

A detailed list of anticipated permitting is provided in Chapter 20. Compliance with terms and conditions of approvals, standards contained in federal and provincial legislation and regulations, and commitments made, will need to be addressed throughout project planning, construction, operation, and decommissioning. Approvals, authorizations, and permits will be required prior to initiating project construction.

Environmental baseline studies and geochemistry studies are recommended to be progressed in order to support a timely environmental approvals process, as well as to support the engineering design and environmental effects monitoring. Completion of Traditional Knowledge, land use, and socio-economic studies with local Indigenous groups should continue to be supported. CNC should continue to actively engage with stakeholders and local Indigenous groups on the project design elements going forward. Impact benefit agreement negotiations with local Indigenous groups should continue.

The estimated cost to support environmental and permitting approvals and community relations is approximately $8 million.

## 26.12 Marketing Terms

As Crawford progresses through permitting and financing stages in 2024, it is recommended that CNC continue to engage downstream customers of the concentrates that will be produced. The objective of this engagement will be to better define marketing terms and would likely entail modeling of Crawford's specifications and/or testing of samples. As this work would be performed by the CNC management team and potential customers, no additional costs would be incurred.

# 27 REFERENCES


Abitibi River Forest Management Inc. (ARFMI), 2022. Abitibi River Forest Management Plan 2022-2032. Publish Start Date March 17, 2022. Retrieved from https://nrip.mnr.gov.on.ca/s/published-submission?language=en_US&recordId=a0z3g000000slvfAAE

Adventure North Ontario, 2022. Retrieved from https://adventurenord.ca/en/.

Arndt, N.T., Lesher, C.M., and Barnes, S.J., 2008. Komatiite: Cambridge University Press, Cambridge, UK, 467p.

Ausenco, 2013. Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada. Prepared for Royal Nickel Corporation by Ausenco Solutions Canada Inc. in conjunction with SRK Consulting (Canada) Inc., 432p.

Ausenco, 2019. Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada. Prepared for Royal Nickel Corporation by Ausenco Solutions Canada Inc., in conjunction with SRK Consulting (Canada) Inc., Golder Associates Ltd., Wood, and WSP, 425p.

Ayer, J.A., Thurston, P.C., Bateman, R., Dubé, B., Gibson, H.L., Hamilton, M.A., Hathway, B., Hocker, S.M., Houlé, M.G., Hudak, G., Ispolatov, V.O., Lafrance, B., Lesher, C.M., MacDonald, P.J., Péloquin, A.S., Piercey, S.J., Reed, L.E. and Thompson, P.H., 2005. Overview of results from the Greenstone Architecture Project: Discover Abitibi Initiative; Ontario Geological Survey, Open File Report 6154, 175p.

Ayer, J., Amelin, Y., Corfu, F., Kamo, S., Ketchum, J., Kwok, K., and Trowell, N., 2002a. Evolution of the southern Abitibi greenstone belt based on U-Pb geochronology: Autochthonous volcanic construction followed by plutonism, regional deformation and sedimentation. Precambrian Research. v115. pp.63-95.

Ayer, J.A., Ketchum, J.W.F. and Trowell, N.F., 2002b. New geochronological and neodymium isotopic results from the Abitibi greenstone belt, with emphasis on the timing and the tectonic implications of Neoarchean sedimentation and volcanism; In Summary of Field Work and Other Activities 2002, Ontario Geological Survey, Open File Report 6100, pp. 5-1 to 5-16.

Canadian Council of Ministers of the Environment (CCME), 2001. Canadian Sediment Quality Guidelines for the Protection of Aquatic Life. Available at https://ccme.ca/en/summary-table.

Canadian Council of Ministers of the Environment (CCME), 2022. Canadian Water Quality Guidelines for the Protection of Aquatic Life. Available at http://ceqg-rcqe.CWQG.ca.

Cowan, Jun & Beatson, Rick & Ross, H.J. & Fright, W.R. & McLennan, T.J. & Evans, T.R. & Carr, J.C. & Lane, R.G. & Bright, D.V. & Gillman, A.J. & Oshust, P.A. & Titley, M., 2003. Practical implicit geological modelling. Fifth International Mining Geology Conference, pp.17-19.

Duke, J.M., 1986. Petrology and economic geology of the Dumont Sill: an Archean intrusion of komatiitic affinity in northwestern Quebec. Geological Survey of Canada, Economic Geology Report, v35, 56p.

Environment Canada, 1961. Storm rainfall in Canada Analysis Code No. Ont. 9(2)-61.

Firelight Research Inc. with Flying Post First Nation, 2023. Flying Post First Nation Socio-Economic Study for Canada Nickel Company's Crawford Nickel Sulphide Project. August 2023.

Firelight Research Inc. with Matachewan First Nation, 2023. Matachewan First Nation Socio-Economic Study for Canada Nickel Company's Crawford Nickel Sulphide Project. August 2023.

Firelight Research Inc. with Mattagami First Nation, 2023. Mattagami First Nation Socio-Economic Study for Canada Nickel Company's Crawford Nickel Sulphide Project. August 2023

Golder Associates Ltd (Golder), 2022b. Crawford Open Pit Project Climate Study. Prepared for Canada Nickel Company, May 2022.

Golder Associates Ltd (Golder), 2022c. 2021 Hydrology Monitoring Report, Crawford Project. Prepared for Canada Nickel Company, March 2022.

Golder Associates Ltd. (Golder), 2022. Crawford Open Pit Project – Climate Study. Prepared for Canada Nickel Company, May 2022.

Golder Associates Ltd., part of WSP (Golder). 2022a. Technical Memorandum: Borrow Source Test Pit Field Program – Crawford Project, dated December 5, 2022. Sent to Canada Nickel Company.

Houle, M.G., and Lesher, C.M., 2011. Komatiite-associated Ni-Cu-(PGE) mineralization in the Abitibi greensone belt. Reviews in Economic Geology. v17. pp.89-121.

Jobin-Bevans, S., and Siriunas, J., 2020. Independent Technical Report on the Crawford Nickel Sulphide Project, Timmins-Cochrane Area, Ontario, Canada. Prepared for Canada Nickel Company Inc. and Noble Mineral Exploration Inc., January 27, 2020 (Effective December 10, 2019), 124p.

MEND, 2009. Price, W.A. Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials. MEND Report 1.20.1, December.

Mercier-Langevin, P., Goutier, J., and Dubé, B. (ed.), 2017. Precious- and base-metal deposits of the southern Abitibi greenstone belt, Superior Province, Ontario and Quebec: 14th Biennial Society for Geology Applied to Mineral deposits meeting field trip guidebook; Geological Survey of Canada, Open File 8317, 86p. https://doi.org/10.4095/306250.

Ministry of Environment (MOE), 2008. Guidelines for Identifying, Assessing and Managing Contaminated Sediments in Ontario: An Integrated Approach. May 2008.

Ministry of Environment and Energy (MOEE), 1999. Water Management Guidelines Provincial Water Quality Objectives of the Ministry of Environment and Energy. July 1994, reprinted February 1999. PIBS 3303E.

Ministry of Natural Resources (MNR), 2006. PMP for Ontario. Draft completed by the IBI Group for the Ministry of Natural Resources.

Monecke, T., Mercier-Langevin, P., Dube, B., and Frieman, B.M., 2017. Geology of the Abitibi Greenstone Belt (Chapter 1); In Archean Base and Precious Metal deposits, Southern Abitibi Greenstone Belt, Canada, Reviews in Economic Geology, v19, pp.7-49.

Naldrett, A. J., 1989. Magmatic Sulphide Deposits: Oxford, Oxford University Press, 196p.

Natural Resources Canada, 1978. Hydrological Atlas of Canada Plate 17. Mean Annual Lake Evaporation, Plate 25. Water Balance, Derived Precipitation and Evapotranspiration.

Natural Resources Canada (NRCan), 2021. Canada Announces Critical Minerals List. https://www.canada.ca/en/natural-resources-canada/news/2021/03/canada-announces-criticalminerals-list.html

ODHD, 2023. Ontario drillhole database, current to 29-09-2023, Ontario Ministry of Mines, electronic data set.

Ontario Geological Survey (OGS), 2005. Surficial Geology of the North Timmins Region; Ontario Geological Survey, Map P. 3654, scale 1: 100 000

Orix Geoscience, 2018. Project 81 - Crawford Township Targeting Compilation, February 10, 2018, Progress Update Presentation, 13p.

Renaud, 2019. Ni-Sulphide EDS Report – Crawford Nickel Project. Prepared for Spruce Ridge Resources Ltd., 451p.

Shared Value Solutions, 2023a. Taykwa Tagamou Nation Crawford Nickel Project Traditional Knowledge and Land Use Study. Prepared for Taykwa Tagamou Nation, February 24, 2023.

Shared Value Solutions, 2023b. Socio-Economic Study for the Crawford Nickel Project: Summary of Findings and Recommendations Report. Prepared for Taykwa Tagamou Nation, August 18, 2023.

Smith, S.L., 1992. Quaternary Stratigraphic Drilling Transect, Timmins to the Moose River Basin, Ontario. Geological Survey of Canada Bulletin 415

Stantec Consulting Ltd. (Stantec), 2023. Draft Crawford Nickel Project: Socio-Economic Baseline Report. Prepared for Canada Nickel Company, September 7, 2023.

Sproule, R.A., Lesher, C.M., Houlé, M., Keays, R., Ayer, J., and Thurston, P., 2005. Chalcophile Element Geochemistry and Metallogenesis of Komatiitic Rocks in the Abitibi Greenstone Belt, Canada. Economic Geology. v100. pp.1169-1190.

Sproule, R.A., Lesher, C.M., Ayer, J., Thurston, P., and Herzberg, C.T., 2003. Erratum to “Spatial and temporal variations in the geochemistry of komatiites and komatiitic basalts in the Abitibi greenstone belt.” Precambrian Research v126, pp.169–172.

Statistics Canada, 2017. Census Profile of Cochrane, District (DIS) [Census division], Ontario and Ontario [Province]. 2016 Census. Statistics Canada Catalogue no. 98-316-X2016001. Ottawa, Ontario, Canada.

Taggart, J., and Firelight Research Inc. with Flying Post First Nation, 2023a. Flying Post First Nation Knowledge and Use Study for Canada Nickel Company's Crawford Nickel Project. July 11, 2023.

Taggart, J. and Firelight Research Inc. with Matachewan First Nation, 2023b. Matachewan First Nation Knowledge and Use Study for Canada Nickel Company's Crawford Nickel Project. July 19, 2023.

Taggart, J. and Firelight Research Inc. with Mattagami First Nation, 2023c. Mattagami First Nation Knowledge and Use Study for Canada Nickel Company's Crawford Nickel Project. August 16, 2023.

Thurston, P.C., Ayer, J.A., Goutier, J., and Hamilton, M.A., 2008. Depositional Gaps in Abitibi Greenstone Belt Stratigraphy: A Key to Exploration for Syngenetic Mineralization. Economic Geology v103, pp. 1097-1134.

Wood Environment & Infrastructure Americas, a Division of Wood Canada Limited (Wood), 2022a. Crawford Nickel Project 2021 Aquatic Resources Baseline. Prepared for Canada Nickel Company, September 2022.

Wood Environment & Infrastructure Solutions Canada Limited (Wood), 2022b. Draft Crawford Nickel Project Cultural Heritage Screening Report. Revision A. Prepared for Canada Nickel Company, May 2022.

Wood Environment & Infrastructure Solutions Canada Limited (Wood), 2022c. Crawford Nickel Project 2021 Terrestrial Ecology Baseline Study. Revision 0A2. Prepared for Canada Nickel Company, June 2022.

WSP E&I Canada Inc. (WSP), 2023b, 2022. Hydrology Monitoring Report, Crawford Project. Prepared for Canada Nickel Company, May 2023.

WSP E&I Canada Inc. (WSP), 2023c. Draft Interim 2022 Aquatic Resources Baseline Report, Crawford Nickel Project. Prepared for Canada Nickel Company, May 2023.

WSP E&I Canada Inc. (WSP), 2023d. Technical Memorandum: Geochemistry Interim Results Update – April 2023. Sent to Canada Nickel Company June 22, 2023.

WSP E&I Canada Inc. (WSP), 2023e. Technical Memorandum: Groundwater Quality Sampling Factual Report, dated February 7, 2023. Sent to Canada Nickel Company.

WSP E&I Canada Inc. (WSP), 2023f. Technical Memorandum: Hydraulic Conductivity Testing Summary, dated January 25, 2023. Sent to Canada Nickel Company.

WSP E&I Canada Inc. (WSP), 2023g. CNC Crawford Mine Baseline Noise Field Program, 2023 Baseline Noise Monitoring & Measurements – Leaves-On and Leaves-Off. Prepared for Canada Nickel Company, November 2023.

WSP E&I Canada Inc. (WSP), 2023h. Crawford Nickle Project 2021-2023 Fish and Fish Habitat Baseline. Prepared for Canada Nickel Company, October 2023. October 2023.

WSP E&I Canada Inc. (WSP), 2023i. Draft Technical Memorandum: Interim Geochemistry Source Terms. Prepared for Canada Nickel Company, September 15, 2023.

WSP E&I Canada Inc. (WSP), 2023j. Draft Crawford Nickel Project Species at Risk Summary Information Technical Memorandum. Prepared for Canada Nickel Company, June 2023.

WSP E&I Canada Inc. (WSP), 2023k. Crawford Nickel Project Stage 1 Archaeological Assessment. Prepared for Canada Nickel Company, August 2023. WSP E&I Canada Ltd.

WSP E&I Canada Limited. (WSP), 2022a. Crawford Nickel Project Stage 1 Archaeological Assessment. Prepared for Canada Nickel Limited, October 2022.

WSP E&I Canada Limited. (WSP), 2022b. Detailed Project Description Crawford Nickel Project. Prepared for Canada Nickel Limited, December 2022.

WSP E&I Canada Limited. (WSP), 2022c. Draft Simulated Pore Water Pressures and Groundwater Inflows – Main Zone and East Zone Pits [PowerPoint slides]. Prepared for Canada Nickel Company, October 2022.

WSP E&I Canada Limited. (WSP), 2023a. Draft Crawford Nickel Sulphide Project - 2022 Terrestrial Ecology Baseline Study. Revision 0B. Prepared for Canada Nickel Company, May 2023.