Crawford Nickel Sulphide Project

NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate

Timmins Area, Ontario, Canada

Prepared for: Canada Nickel Company Inc.

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Effective Date: May 19, 2022

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John Siriunas, P.Eng., Caracle Creek International Consulting Inc.
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CERTIFICATE OF QUALIFIED PERSON

L. Paul Staples

I, L. Paul Staples, P.Eng., certify that I am employed as VP and Global Practice Lead with Ausenco Engineering Canada ("Ausenco"), with an office address of 855 Homer Street, Vancouver, BC. This certificate applies to the technical report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate", with an effective date of May 19, 2022 (the "Technical Report").

I graduated from Queen's University, Kingston, Ontario in 1993 with a Bachelor of Science degree in Materials and Metallurgical Engineering. I am a member in good standing of the Engineers and Geoscientists of British Columbia, Licence #47467. I have practiced my profession for 28 years continuously. I have experience in process operation and design from over 20 similar projects and studies in Canada and abroad including the nearby 38 Mt/y Dumont Nickel Project PFS and FS in Quebec, the 4 Mt/y Arctic Polymetallic Project PFS and FS in Alaska, the 80 Mt/a Grasberg Cu complex in Indonesia, the 20 Mt/y Lumwana Copper Project in Zambia and the 26 Mt/y Constancia Copper Project in Peru.

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.

I have not visited the Crawford property. I am responsible for Sections: 1.2, 1.3, 1.14, 1.15, 1.17, 1.18, 1.19, 1.22, 1.24.1, 1.24.5; 2.3.1; 3.1, 3.3; 5.5; 17; 18.1, 18.2, 18.3, 18.4, 18.5, 18.6, 18.8.7, 18.8.8, 18.10, 18.11, 18.14, 18.15, 18.16, 18.17; 19; 21 (except 21.3.1, 21.3.3, 21.4.6, 21.4.9, 21.5.3); 24; 25.10, 25.11, 25.13, 25.14, 25.16 ; 26.1, 26.5 and 27 of the Technical Report.

I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101. I was previously involved in co-authoring the report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report & Preliminary Economic Assessment", with an effective date of May 21, 2021.

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: August 22, 2022

"Signed and sealed"

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L. Paul Staples, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Greg Lane

I, Greg Lane, FAusIMM, certify that I am employed as the Chief Technical Officer with Ausenco Services Pty Ltd ("Ausenco"), in Australia. This certificate applies to the technical report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate", with an effective date of May 19, 2022 (the "Technical Report").

I graduated from the University of Tasmania with a M.Sc. I am a Fellow of AusIMM (#203005). I have practiced my profession for 30 years.

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.

I have not visited the Crawford Property. I am responsible for sections 1.10, 1.24.4; 13, 25.5, and 26.4 of the Technical Report.

I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101. I was previously involved in co-authoring the report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report & Preliminary Economic Assessment", with an effective date of May 21, 2021.

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Dated: August 22, 2022

"Signed"

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Greg Lane, FAusIMM #203005

CERTIFICATE OF QUALIFIED PERSON

Scott Jobin-Bevans

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, P3B 1R9. This certificate applies to the technical report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate", with an effective date of May 19, 2022 (the "Technical Report").

I graduated from the University of Manitoba (Winnipeg, Manitoba) with a B.Sc. Geosciences (Hons) in 1995 and from the University of Western Ontario (London, Ontario) with a Ph.D. (Geology) in 2004. I am a member, in good standing, of the Association of Professional Geoscientists of Ontario, License Number 0183.

I have practiced my profession continuously for more than 25 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 NI-43-101 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.

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.

I have not visited the Crawford Property. I am responsible for sections 1.1, 1.4 to 1.9, 1.11, 1.12, 1.23, 1.24.2; 2; 4.1 to 4.4, 4.7; 6.1 to 6.3; 7; 8; 9; 10; 12; 14 (except 14.11); 15; 23; 25.1 to 25.4, 25.6 to 25.8, 25.17; 26.1, 26.2; 27 of the Technical Report.

I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101. I have been involved with the Crawford Nickel Sulphide Project as an independent consultant and Qualified Person since October 2019 with the most recent technical report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report & Preliminary Economic Assessment", with an effective date of May 21, 2021. I was also involved in co-authoring the report titled, "Independent Technical Report and Mineral Resource Estimates Crawford Nickel-Cobalt Sulphide Project: Main Zone (Update) and East Zone (Initial) Deposits", with a report effective date of December 12, 2020, a mineral resource estimate effective date of December 11, 2020, an original report date of December 4, 2020, and an amended report date of December 31, 2020.

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: August 22, 2022

"Signed"

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Scott Jobin-Bevans, P.Geo. #0183

CERTIFICATE OF QUALIFIED PERSON

John M. Siriunas

I, John M. Siriunas, P.Eng., certify that I am an associate independent consultant of Caracle Creek International Consulting Inc. (Caracle) and have an address at 25 3rd Side Road, Milton, Ontario, Canada, L9T 2W5. This certificate applies to the technical report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate", with an effective date of May 19, 2022 (the "Technical Report").

I graduated from the University of Toronto (Toronto, Ontario) with a B.A.Sc. (Geological Engineering) in 1976 and from the University of Toronto (Toronto, Ontario) with an M.A.Sc. (Applied Geology and Geochemistry) in 1979. I have been a member, in good standing, of the Association of Professional Engineers of Ontario since June 1980 (Licence Number 42706010) and possess a Certificate of Authorization to practice my profession.

I have practiced my profession continuously for 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.

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.

I visited the Crawford property on October 12, 2019, on February 3-4, 2020, September 10-11, 2020, and August 11-12, 2022. I am responsible for sections 2; 6.4; 11; and 12 of the Technical Report.

I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101. I have been involved with the Crawford Nickel Sulphide Project as an independent consultant and Qualified Person since October 2019 with the most recent technical report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report & Preliminary Economic Assessment", with an effective date of May 21, 2021. I was also involved in co-authoring the report titled, "Independent Technical Report and Mineral Resource Estimates Crawford Nickel-Cobalt Sulphide Project: Main Zone (Update) and East Zone (Initial) Deposits", with a report effective date of December 12, 2020, a mineral resource estimate effective date of December 11, 2020, an original report date of December 4, 2020, and an amended report date of December 31, 2020.

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: August 22, 2022

"Signed and sealed"

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John Siriunas, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
David Penswick

I, David Penswick, P. Eng., certify that I am self-employed as a consultant to the mining and mining finance industries, with an office address of 163 Garden Ave, Toronto, Canada. This certificate applies to the technical report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate", with an effective date of May 19, 2022 (the "Technical Report").

I graduated from Queen's University in Canada in 1989 with B.A.Sc. in Mining Engineering; from University of the Witwatersrand in Johannesburg in 1993 with an M.Sc. in Mining Engineering; and from the University of South Africa in 1995 with an MDP in Business Management. I am a member of Professional Engineers Ontario, membership number 100111644. I have practiced my profession continuously for 32 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.

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.

I have visited the Crawford site on several occasions, most recently on May 18, 2022. I am responsible for Sections 1.11, 1.13, 1.16.3, 1.20, 1.21, 1.24.3; 14.11; 16; 18.7, 18.9, 18.12, 18.13; 21.3.1, 21.3.3, 21.4.6, 21.4.9, 21.5.3; 22; 25.9, 25.15; and 26.3 of the Technical Report.

I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101. I have been involved with design and evaluation of the Crawford Project since December 2019 and I was previously involved in co-authoring the report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report & Preliminary Economic Assessment", with an effective date of May 21, 2021.

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: August 22, 2022

"Signed and sealed"

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David Penswick, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Sheila Ellen Daniel, P.Geo.

I, Sheila Ellen Daniel, P.Geo., certify that I am employed as a Principal Geoscientist, Discipline Lead Mining Environmental Management Approvals with Wood Environment & Infrastructure Americas ("Wood"), with an office address of 2020 Winston Park Drive, Suite 600, Oakville, Ontario, Canada, L6H 6X7. This certificate applies to the technical report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate", with an effective date of May 19, 2022 (the "Technical Report").

I graduated with an M.Sc. from McMaster University in 1990 and with a B.Sc. (Honours) from the University of Western Ontario in 1988. I am a registered Professional Geoscientist of Ontario (#0151). I have practiced my profession for 30 years, during which time I have been directly involved in mining environmental consulting, including support for engineering reports, and environmental assessments and approvals for a large number of Ontario mining projects.

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.

I have not visited the Crawford Property. I am responsible for sections 1.16.1, 1.16.2, 1.24.8; 3.2; 4.5, 4.6; 5.1, 5.2, 5.3, 5.4, 5.6; 20; 25.12; and 26.8 of the Technical Report.

I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101. I was previously involved in co-authoring the report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report & Preliminary Economic Assessment", with an effective date of May 21, 2021.

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: August 22, 2022

"Signed and sealed"

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Sheila Daniel, P.Geo., #0151

CERTIFICATE OF QUALIFIED PERSON

Karel Van Zyl, M.Eng., P.Eng.

I, Karel Van Zyl, M.Eng., P.Eng., certify that I am employed as a Senior Geotechnical Engineer with Wood Environment & Infrastructure Americas ("Wood"), with an office address of 2020 Winsor Park Drive, Suite #600, Oakville, ON, L6H 6X7. This certificate applies to the technical report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report, Preliminary Economic Assessment & Updated Mineral Resource Estimate", with an effective date of May 19, 2022 (the "Technical Report").

I graduated from University of Stellenbosch, South Africa, 1991; University of Calgary, Canada, 1999 with a B.Eng.; M.Eng. I am a registered Professional Engineer of Professional Engineers Ontario (#100510746). I have practiced my profession for 15 years. I have been directly involved in the areas of tailings dams, dam safety reviews, dam safety inspections, dam risk assessment, soil/rock investigation, fill placement, monitoring of staged construction, and slope stability.

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.

I have not visited the Crawford Property. I am responsible for sections 1.24.6, 1.24.7; 18.8.1 to 18.8.6; 26.6, and 26.7 of the Technical Report.

I am independent of Canada Nickel Company Inc. as independence is defined in Section 1.5 of NI 43-101. I was previously involved in co-authoring the report titled, "Crawford Nickel Sulphide Project – NI 43-101 Technical Report & Preliminary Economic Assessment", with an effective date of May 21, 2021.

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: August 22, 2022

"Signed and sealed"

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Karel Van Zyl, P.Eng.

Important Notice

This Technical Report was prepared as National Instrument 43-101 Technical Report for Canada Nickel Company Inc. by Ausenco Engineering Canada Inc., Caracle Creek international Consulting Inc., Wood Environment & Infrastructure Americas, and David Penswick – Independent Consultant, 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 Canada Nickel Company Inc. subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of this Report by any third party is at that party's sole risk.

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1 SUMMARY

This section (except 1.1, 1.4 to 1.9, 1.10, 1.11, 1.23, 1.24.2) has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021, and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

1.1 Introduction

This report (the "Report") was prepared by Caracle Creek International Consulting Inc (Caracle Creek) with support from Ausenco Engineering Canada Inc. (Ausenco), and Wood Environment & Infrastructure Solutions Canada Limited (Wood) (collectively the "Authors") for Canada Nickel Company Inc. (CNC) to provide the updated mineral resource estimate as announced by CNC on July 6, 2022 and to re-iterate the results of the Crawford Nickel Sulphide Project Preliminary Economic Assessment (PEA) dated July 9, 2021 and with an effective date of May 21, 2021.

The Report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

The responsibilities of the engineering and geological consultants are as follows:

- Caracle Creek International Consulting Inc. was commissioned by CNC to manage and coordinate the work related to the report and to complete the mineral resource estimates, inclusive of a review of the drilling campaigns to date, sample preparation, and analyses for use in the mineral resource estimate.
- Ausenco was commissioned to lead the development of the PEA-level design and cost estimating of the process plant and surface infrastructure.
- Wood Environment & Infrastructure Solutions Canada Limited (previously Wood Canada Limited) was commissioned to support environmental planning, assessment, licensing, permitting, and conceptual design of the tailings management facility (TMF) and complete the layout and sizing of water management structure.
- David Penswick, an independent mining consultant, was commissioned to design the open pit mine plan, mine production schedule, mine capital and operating costs and economic evaluation.

1.2 Property Description

The Crawford Nickel Sulphide Project is situated within the Timmins-Cochrane Mining Camp in Northeastern, Ontario, Canada, a region with a strong mining history (gold, nickel, zinc, lead, etc.), and a Canadian province favourable to mining with regulations that reflect that history.

In general, all known mineralization, economic or potentially economic, that is the focus of this Report is located within the boundary of the mining lands that comprise the Crawford Nickel Sulphide Project.

1.3 Project Setting

The project is located approximately 42 km north of Timmins, Ontario (Figure 1-1). The property is highly accessible to the regional infrastructure network, and is bisected by Highway 655, a paved highway offering year-round accessibility. Access to the property itself is by a network of local bush roads accessed from Highway 655. Highway 655 connects to Highway 11, the Trans-Canada Highway, north of the site. The site is readily accessible from both Timmins and South Porcupine from the south, 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) 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.

1.4 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

The Project comprises approximately 11,433.22 ha (114.33 km^2), consisting of a combination of patented lands (Crown patents) and unpatented mining claims (staked claims)

Specifically, the property comprises 140 Crown patents (freehold patented lands) in Crawford and Lucas townships that cover approximately 9389.32 ha and 173 single cell mining claims (SCMC), and 2 multi-cell mining claims (MCMC) in Crawford Township covering approximately 2043.9 ha. In this region of Ontario, one SCMC averages approximately 21.22 hectares.

The 140 Crown patents in Crawford and Lucas townships are mineral rights only (CNC does not control the surface rights), 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 (mineral and surface rights) within the boundary of the Project area that is not currently owned or optioned by CNC.

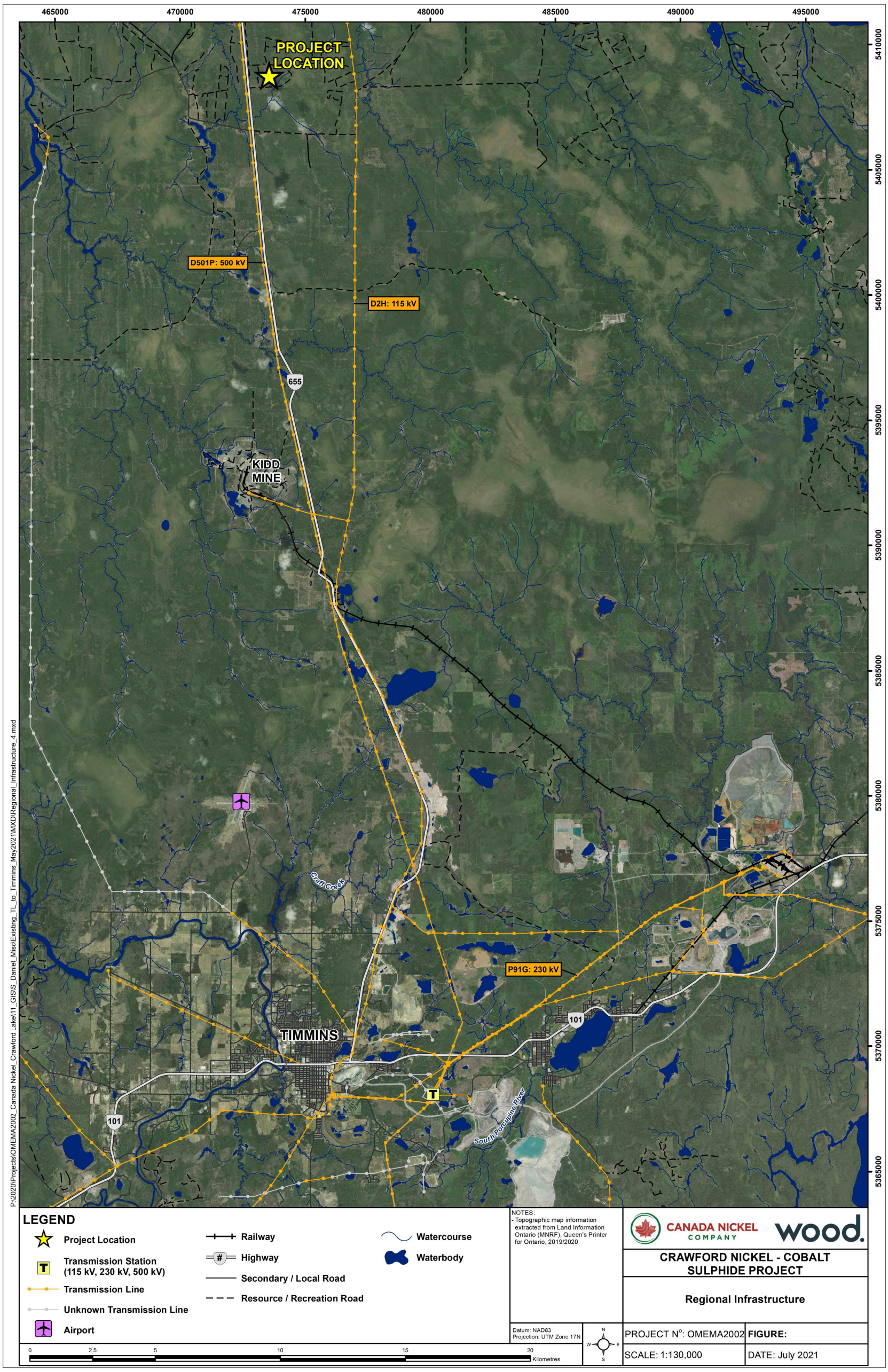
The *Ontario Mining Act* (2010) grants surface access to an unpatented mineral claim without owning the surface rights and given proper consultation with appropriate stakeholders. Access to mining rights on patented lands or unpatented SCMC in which the CNC owns or has sub-surface rights only, requires 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.

On December 19, 2019, Noble announced that it had completed the acquisition of the 5% net smelter return royalty (NSR) applicable to ~55,000 hectares 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 (see 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 (see Noble news release dated December 3, 2019) and which comprise part of the current Project.

As part of the Crawford property acquisition from Noble, the Company acquired the right to purchase up to 15,000 acres of surface rights associated with the patented mineral rights which comprise part of the Project.

Annual holding costs for the 140 patented lands (mining tax) total approximately $37,557 and the required annual assessment work for the unpatented lands is approximately $40,200.

Figure 1.1: Project Location Map



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

1.5 Geology and Mineralization

The Project is situated in the Timmins-Cochrane Mining Camp of Northeastern Ontario, in the western portion of the mineral-rich Abitibi Greenstone Belt (2.8 to 2.6 Ga), which is within the Superior Province, Canada. The Abitibi Greenstone Belt 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). The Timmins-Cochrane Mining camp has a history of nickel production from komatiite-associated nickel-copper-platinum group element (Ni-Cu-(PGE)) deposits.

Recent work (2003-2012) suggests that the rocks underlying the property are part of the Deloro Assemblage (Monecke et al., 2017). The Deloro Assemblage (2730 to 2724 Ma) 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).

The surrounding Lower Blake River Assemblage (2704 to 2701 Ma), not underlying the property, consists predominantly 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) and is host to several mafic-ultramafic sills in the northern part of Crawford Township and in neighbouring Mahaffy and Aubin townships.

The principal target, the CUC 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 recent historical diamond drill holes (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.

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).

Sulphide mineralization discovered to date on the Crawford property can be characterized as komatiite-hosted Ni-Cu-Co-platinum-group-elements (PGE) deposit type, which recognizes two sub-types (Lesher and Keays, 2002). Sulphide nickel-copper-cobalt-PGE mineralization in the CUC is interpreted as most similar to Mt. Keith-style. Mt. Keith-style (Type II) is based on sheet flow theory (Lesher and Keays, 2002) and is characterized by 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, are present along with the sulphur-poor mineral heazlewoodite (Ni_3S) and nickel-iron alloys such as awaruite (Ni_3-Fe). These deposit types are 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 deposit type example of the komatiite-hosted Ni-Co-PGE exploration model that CNC is using for the CUC is the Dumont nickel deposit of Dumont Nickel by Magneto Investments L.P., previously Royal Nickel Corporation (RNC). The Dumont deposit is located 220 km to the east of Crawford Township. The Dumont sill has undergone pervasive serpentinization and local talc-carbonate alteration due to metamorphism to mid-upper greenschist facies (*e.g.*, Eckstrand, 1975; Sciortino et al., 2015). The observed mineralogy of the Dumont is a result of the serpentinization of a dunite protolith (>90% olivine), which locally hosted a primary, disseminated (intercumulus) magmatic sulphide assemblage and contained "trapped" nickel within the unaltered olivine. The pervasive serpentinization process, whereby olivine reacts with water to produce serpentine, magnetite and brucite, creates a strongly reducing environment where the nickel released from the

decomposition of olivine is partitioned into low-sulphur nickel sulphides (*i.e.*, heazlewoodite) and newly formed awaruite. The final mineral assemblage and texture of the disseminated nickel mineralization in the Dumont deposit and the variability has been controlled primarily by the variable degree of serpentinization that the host dunite has undergone.

1.6 History

Prior to 1964, little was known about the geology of Crawford Township. The 1963 discovery of the rich base metal deposit in Kidd Township (Kidd Creek Mine), about 15 km south of the property, led to a flurry of exploration in Crawford Township through the latter 1960s and the 1970s. Historical aeromagnetic surveys (1950s and 1960s) show a large, roughly circular, strongly magnetic high zone in the east-central part of the township that is interpreted to be an ultramafic rock mass (the CUC). The International Nickel Company of Canada Limited led the way in exploring the township during the 1960s with multiple drill holes testing numerous geophysical MAG-EM anomalies.

There are at least 26 historical drill holes reported in Lucas Township which comprise five diamond drill holes and 21 reverse circulation (RC) drill holes (ODHD, 2020). These drill holes 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).

In late 2017, Spruce Ridge Resources entered into an agreement with Noble Mineral Exploration to explore certain lands in Crawford Township, including the CUC. Spruce Ridge began a diamond drilling program in late 2018 and continued to drill in 2019. In 2017, Noble Mineral Exploration completed a 1,031.3 line-km airborne helicopter MAG-EM survey and in 2018, a fixed-wing 936.1 line-km FALCON© airborne gravity gradiometer and magnetics survey, both covering Crawford Township and the property.

1.7 Exploration

Other than diamond drilling, Canada Nickel has not completed any other exploration work on the Project.

1.8 Drilling and Core Sampling

The current drilling program is 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 Nickel Sulphide Project, management and control of the drilling program shifted from Spruce Ridge to CNC.

As of the effective date of the Mineral Resource Estimate (May 19, 2022), a total of 76 drill holes totalling approximately 32,293 m (up to hole CR20-73) have been completed by Canada Nickel and Spruce Ridge. This includes drilling metres (635 m) from six abandoned holes (CR19-14, CR19-26, CR19-26A, CR20-30, CR20-40, CR20-70). Three of the 76 drill holes, CR20-55, CR20-57, and CR20-58, were HQ size, completed for metallurgical testwork, whereas the remaining 73 drill holes used NQ size. Diamond drilling has been completed at the Main, East, and West zones. Forty-nine of the current 76 drill holes were used in the updated Main Zone Mineral Resource Estimate and 11 were used in the initial East Zone Mineral Resource Estimate.

1.8.1 Main Zone

The focus of the 2019-2022 drilling at the Main Zone was to extend along strike mineralization encountered in the original historical 2018 series drill holes (Spruce Ridge), to test the east-northeastern and west-southwestern extents of

mineralization (*i.e.*, the contacts), to test deeper portions of the CUC, and to complete in-fill drilling within the Main Zone mineral resource envelope and its higher-grade core and the East Zone mineral resource envelope.

As of the effective date of the Crawford Mineral Resource Estimate (May 19, 2022), a total of 291 drill holes totalling approximately 123,617.68 metres have been completed by Canada Nickel and previous companies. This includes drilling metres (7,564.7 m) from 36 abandoned holes. Additionally, 55 of the 76 drill holes were HQ size, completed for geotechnical and metallurgical testwork, whereas the remaining 236 drill holes used NQ size. Diamond drilling has been completed at the Main, East, and West zones, with 177 of the current 291 drill holes used in the updated Main Zone Mineral Resource Estimate and 81 used in the initial East Zone Mineral Resource Estimate.

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 metres deep. 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. A 3D-inversion magnetic anomaly, nearly 1 km deep, has been only partially tested at depth.

1.8.2 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 (modeled 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. 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 and internal regions of higher-grade nickel remain open along strike to the west-northwest and extend to a depth of at least 650 metres.

1.8.3 Main Zone – PGE Reef

The Main Zone PGE reef, located within the northern margin of the ultramafic to mafic body, is associated with a contact between an ultramafic (pyroxenite) unit to the south and a gabbroic unit to the north, reflected in seven drill hole intercepts. Additional drill holes will be required to better define the PGE reef and as such the PGE reef was restricted to the central region of the modelling area.

1.8.4 East Zone

In addition to Main Zone drilling, CNC began to drill-test the East Zone, located about 1.2 km northeast of the Main Zone in late 2019 and into 2020 with relatively wide-spaced drill hole sections (11 drill holes in total). Drilling and interpretation have defined an ultramafic body that is about 2.6 km long oriented east-west, 200 to 350 m wide, and at least 800 m deep. The East Zone is open to the east and west and is open at depth.

1.8.5 East Zone – PGE Reefs

Within the layered ultramafic unit of the East Zone, two domains can be differentiated: (1) a high-nickel, PGE-poor domain to the south, comprising mainly dunite and peridotite, and (2) a low (to barren) nickel domain, comprising peridotite and pyroxenite, with major PGE occurrences interpreted as horizons or “reefs” proximal too the northern margin of the ultramafic body. Nine of the 11 drill holes in the East Zone intersected one or both of the two PGE reefs, with five holes intersecting both the south reef (PGE-1) and north reef (PGE-2).

1.9 Data Verification

The Authors 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 2021), having complete databases and documentation such as assay certificates, was thoroughly reviewed. However, older historical records (in general, pre-2018) are not as complete and so the Authors do not know the exact methodologies used in the data collection. Nonetheless, the Authors have no reason to doubt the adequacy of the historical sample preparation, security and analytical procedures and have complete confidence in all historical information and data that was reviewed.

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 (2 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.

1.10 Metallurgical Testwork

Metallurgical testwork was completed at two labs: XPS in Sudbury, Ontario, and COREM in Quebec City, Quebec between August 2020 and March 2021 to formulate a standardized flowsheet. The standardized flowsheet was then used to evaluate the metallurgical response of metallurgical variability samples from across the ore body, including 14 open circuit tests and 7 locked cycle tests to achieve the following:

- understand the metallurgical response of the Crawford ultramafic nickel mineralization
- provide guidance for process plant design
- provide inputs for operating cost estimation

Test work completed subsequent to March 2021 for the feasibility study was not used in the resource statement update and is not reported herein due to the incomplete nature of the work.

1.10.1 Comminution Testwork

Comminution circuit testwork was completed at SGS Lakefield on six samples, three from whole HQ core and three from half NQ core. One sample was determined to be waste grade and was not reported.

The Axb parameter from the JK drop weight test was 60.7, which is characterized as moderate competency. The variability samples tested using the SMC test were more competent and were classified as moderate to medium competency. The results were similar to those from other low grade ultramafic nickel deposits.

The Bond ball mill work index tests indicate that the ore is hard to very hard. However, these results are impacted by the acicular nature of the ultramafic mineralization.

1.10.2 Mineralogy

For ultramafic nickel deposits, the mineralogy is a critical part of establishing the resource estimate. Nickel can exist in recoverable form as minerals such as heazlewoodite, pentlandite, awaruite and millerite, or it 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. Serpentine is the main gangue mineral in the deposit. Microprobe analysis shows that the average nickel tenor of serpentine is 0.08% with a standard deviation of 0.05%.

For the study, 999 distinct samples across 26 drill holes were submitted for mineralogical characterization. The goal of the Phase 1 mineralogy program was to cover off the Main Zone of Crawford with respect to the east-west and north-south axes.

Magnesium serpentine is the main silicate mineral in Crawford. There appears to be some areas with higher proportions of iron serpentine in the Main Zone. In these areas, there may be a greater proportion of nickel hosted within the silicate matrix.

1.10.3 Flowsheet Development Testwork

Fourteen metallurgical variability samples were selected from HQ and half NQ cores with the samples from each of the mine domains (high-grade core, north low-grade and south low-grade) and a range of compositions.

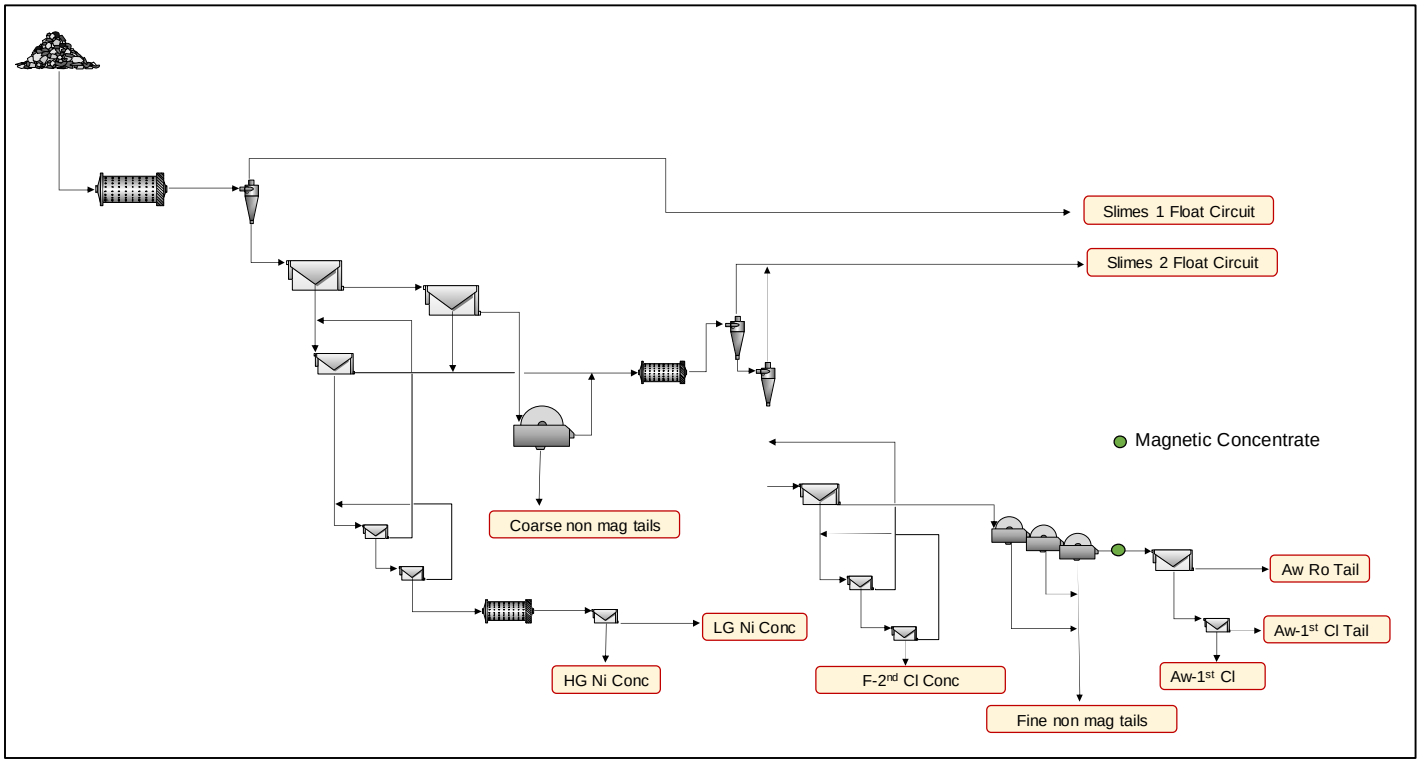
Testwork was completed in parallel at both XPS and COREM, leveraging work completed on other ultramafic nickel deposits such as Mt. Keith and Dumont. Initial flowsheet development started with kinetic testwork in the coarse circuit rougher-scavenger flotation cells and was done on a heazlewoodite dominant sample and a pentlandite dominant sample.

Metallurgical variability tests were conducted on 14 samples using batch and locked cycle tests. The samples represented the variability in head grades and mineralogy.

The average total nickel recovery for samples taken from the high-grade core was 51%. The average grade of flotation and magnetic concentrates for samples taken from the high-grade core was 6.2% Ni and 0.27% Ni, respectively.

Seven locked cycle tests (see Figure 1.2) were completed between August 2020 and May 2021 of which three were done with the final metallurgical variability flowsheet at a primary grind size of 80% passing 180 µm and final reagents. All samples tested in locked cycle were from the high-grade core domain.

Figure 1.2: Locked Cycle Test Flowsheet



Source: Ausenco, 2021.

1.10.4 Recovery Equations

The recovery equations were designed using both open circuit and locked cycle test results. Nickel recovery equations were developed for three domains: high-grade core, north low-grade, and south low-grade. These domains represent the degree of serpentinization of the material and the sulphur content, as both have an impact on nickel recovery. Table 1.1 summarizes the nickel recovery equations according to the mine and metallurgical domains, and Table 1.2 summarizes the split in Ni recovery between the high- and low-grade concentrates.

Table 1.1: Ni Recovery Equations for the Metallurgical Domain S/Ni < 0.3

Metallurgical Domain	Ore Domain	Recovery Equation
S/Ni < 0.3	High-Grade Core	Ni Recovery (%) = 90 · (S/Ni) + 20
	North Low-Grade	Ni Recovery (%) = 77 · (S/Ni) + 18
	South Low-Grade	Ni Recovery (%) = 77 · (S/Ni) + 15
0.3 ≥ S/Ni < 0.5	High-Grade Core	Ni Recovery (%) = 48
	North Low-Grade	Ni Recovery (%) = 45
	South Low-Grade	Ni Recovery (%) = 42
S/Ni ≥ 0.5	All Domains	Ni Recovery (%) = 52

Table 1.2: Modelled Equations for the Distribution of Nickel Between High- and Low-Grade Concentrates

Metallurgical Domain	% of Nickel Recovered to High-Grade Concentrate	% of Nickel Recovered to Low-Grade Concentrate
S/Ni < 0.1	30	70
0.1 ≥ S/Ni < 0.5	40	60
0.5 ≥ S/Ni < 1	50	50
S/Ni ≥ 1	0	100

Crawford has the potential to produce iron, chromium, and cobalt by-products. Recovery equations were developed to model each of these by-product streams. The metallurgical variability tests and locked cycle tests were used to validate the recovery equations.

For iron, a single recovery equation was applied across the mine domains and no metallurgical domains were created. Iron recovery was capped at 70% of the magnetite within a given mining block. The equation used to model iron recovery was:

$$Fe\ Recovery\ (\%) = 45.214 \cdot LN\cdot(\%\ Fe\ in\ Feed) - 38.401$$

The chromium recovery across the deposit was assumed to be 27%, which was the average chromium recovery from the locked cycle tests. Table 1.3 summarizes the Ni domain; cobalt recovery was capped at 40.

Table 1.3: Cobalt Recovery Equations

Metallurgical Domain	Domain	Recovery Equation
S/Ni < 1	All Domains	Cobalt Recovery (%) = 89 · (S/Ni) - 28[1]
S/Ni ≥ 1	All Domains	Ni Recovery (%) = 40

Platinum group metals were assumed to be not payable in the economic model for this preliminary economic assessment.

1.10.5 Concentrate Quality

The concentrate composition from locked cycle tests from the Eastern part of the Main Zone, where mining is expected to start, and capture two metallurgical domains: an intermediate S/Ni domain and the high S/Ni domain (Table 1.4).

Table 1.4: Quality of the Ni Concentrates

Sample ID	Conc. Type	% Ni Rec to Conc.	S/Ni	Ni %	Co %	Fe %	Cr %	S %	Mg %
103-V13-Mar21	High grade	75	0.46	40	0.3	11	0.2	16	6.6
103-V13-Mar21	Low grade	25	0.46	15	0.2	18	0.3	6.5	13
107-V12-Mar21	Low grade	100	1.1	15	0.8	32	0.6	16	6.7
COREM PN-Master	Low grade	100	1.6	15	0.8	42	0.3	19	3.8

The magnetic concentrates across these three locked cycle tests had an average grade of 50% Fe and 3.8% chromium. There has been very little optimization work done on the magnetic circuit (Table 1.5).

Table 1.5: Quality of the Magnetic Concentrate

Sample ID	S/Ni	Ni %	Co %	Fe %	Cr %	S %	Mg %
103-V13-Mar21	0.46	0.1	BDL[1]	49	1.6	0.1	7.0
107-V12-Mar21	1.1	0.2	BDL[1]	53	5.4	0.4	3.9
COREM PN-Master	1.6	0.3	BDL[1]	48	4.3	0.8	7.3

Note: BDL = below detection limit.

1.11 Mineral Resource Estimation & Statement

1.11.1 Mineral Resource Estimation

Caracle Creek was retained by CNC to prepare two updated mineral resource estimates (MREs) in accordance with NI 43-101, supported by one technical report for the Crawford Nickel Sulphide Project, which incorporates all current diamond drilling for which the drill hole data could be confidently confirmed. Drill hole information up to 19 May 2022 was utilized in the preparation of the mineral resource estimates. The updated MRE for the Main Zone and the MRE for the East Zone, disclosed herein, were prepared by Miguel Vera (B.Sc., Geology) and this work was reviewed and signed off by qualified person Dr. Scott Jobin-Bevans (P.Geo.).

The deposit type being considered for nickel mineralization discovered to date in the Crawford Ultramafic Complex, komatiite-hosted Ni-Cu-Co-(PGE), is comparable to the Dumont nickel deposit, located in Quebec, Canada. The host Archean Dumont Sill is about 7 km long, up to 1 km in width, and like the Crawford ultramafic deposit is located within the Abitibi Greenstone Belt.

The drill hole and project database provided by CNC was filtered by zones of interest, validated, and refined for geomodelling and resource estimation purposes. The working database for the Main Zone, with complementary numbers in parenthesis for the aggregate database of Main Zone plus the new "West Zone" (given that the two were treated as a single entity), contains the following:

- Collar: 111 (177) holes amounting to 49,661.68 m (80,934.63 m), with an approximate mean depth of 450 m. Included are 8 (12) geotechnical holes (with geologs only) and 19 metallurgical holes (twin holes with complete logs but no samples).
- Survey: 110 (174) measured holes, with the remaining hole estimated from its planned direction.
- Lithology: 109 (172) holes with 23 unique rock codes, grouped into 14 codes for modelling purposes.
- Assays: 65 (116) holes with 23,270 (34,276) core samples of 1.5 m average length; 34 elements reported.
- Magnetic Susceptibility: 101 (161) holes with 24,324 (44,328) handheld magnetic susceptibility measurements on drill core, taken every 3 m during the initial campaigns, and later every 1 m.
- Specific Gravity: 102 (162) holes with 6,176 (8,977) density measurements from drill core, taken every 7 m on average.

The working database for the East Zone contains the following:

- Collar: 81 holes amounting to 36,452.34 m, with an approximate mean depth of 450 m. Included are 13 geotechnical holes (with geologs only) and 3 metallurgical holes (twin holes with complete logs but no samples).
- Survey: 79 measured holes, with the remaining 2 holes estimated from their planned direction.

- Lithology: 77 holes with 19 unique rock codes, grouped into 13 codes for modelling purposes.
- Assays: 47 holes with 12,101 core samples of 1.5 m average length; 34 elements reported.
- Magnetic Susceptibility: 67 holes with 24,148 handheld magnetic susceptibility measurements on drill core, taken every 3 m during the initial campaigns, and later every 1 metre.
- Specific Gravity: 68 holes with 3,166 density measurements from drill core, taken every 8 m on average.

Secondary data sources include alteration, mineralization, and structural drill hole logs, QEMSCAN data, historical geophysical surveys (magnetic susceptibility, EM and gravity), geological maps and various work reports.

The nickel resource area in the Main Zone measures approximately 1.9 km along strike, 280 to 580 m in width, and 800 m deep, while the nickel resource in the East Zone is approximately 2.6 km along strike, 200 to 350 m in width, and 800 m deep. Estimates are based on a compilation of a few historical and numerous recent diamond drill holes, along with mineralized zones prepared by Caracle.

The main steps in the resource estimation methodology were as follows:

- CNC drill hole database compilation and preparation of the working database to be used in the resource estimate.
- Modelling of 3D geological units and mineralized domains based on revised lithological codes, densities, magnetic susceptibility, and assay grades.
- Generation of drill hole intercepts for each mineralized zone in all relevant data tables.
- Exploratory data analysis (EDA), capping and compositing of assay grades.
- Variogram modelling and cross-validation.
- Grade interpolations (kriging, IDW, NN) and validations (visual, statistical, swath plots).
- Resource classification.

The mineral resource estimates detailed in the report were prepared using Leapfrog Geo 2021.2 (3D modelling) and Isatis.neo 2022.04 (geostatistics) software packages. The estimation used 3D block modelling, applying the ordinary kriging (OK) and inverse distance weighting (IDW) interpolation methods, depending on the domains and variables. Leapfrog Geo uses implicit modelling techniques for 3D modelling (Cowan et al., 2003).

The modelling area for the entire project is 7.5 km long (from 469500E to 477000E) by 5 km wide (from 5407000N to 5412000N), with a modelling depth 800 m below overburden (at -600 m elevation). The main regional fault dividing the project was modelled to arrive at the corresponding Main Zone and East Zone general domains. Note that, for modelling purposes, the Main Zone cannot be separated from its western extension (West Zone), which was also completely modelled (and estimated) along with it, even if not being considered in this MRE. Both zones can be considered open at depth, with at least two drill holes in the Main Zone extending past the modelling depth, with intercepts containing mostly medium to low nickel grades with a few >0.25% Ni intercepts. The East Zone is currently being tested below this limit too.

Resource classification in the Main Zone consisted in evaluating block proximity to drill holes 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. The reference variogram was generated using all nickel composites and the block evaluation was carried out according to a stricter set of parameters than the ones used for resource estimation, like the use of octant search or the requirement of three complete drill holes to achieve Measured status and three “incomplete” drill holes for Indicated.

Similarly, resource classification in the East Zone used 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. The reference variogram was generated using all nickel composites within and the block evaluation was carried out according to a stricter set of parameters than the ones used for resource estimation, like the use of octant search or the requirement of three drill holes to achieve Measured status and two for Indicated.

1.11.2 Mineral Resource Statement

The measured ("Mea"), indicated ("Ind") and inferred ("Inf") mineral resource estimates presented herein are constrained within pit shells developed for the Main and East zones from pit optimization analysis. The effective date of the mineral resource estimate is May 19, 2022.

Pit constrained, class-characterized mineral resources with respect to the higher and lower-grade nickel estimation domains within the Main Zone are presented for all elements studied in Table 1.6.

Pit constrained, class-characterized mineral resources with respect to the higher and lower-grade nickel estimation domains within the East Zone are presented for all elements studied in Table 1.7.

Table 1.6: Summary of the Pit-Constrained Updated Main Zone Mineral Resource Estimate

Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
Higher-Grade	Measured	159.0	0.31	493.5	0.013	20.6
	Indicated	135.6	0.30	402.3	0.013	17.6
	Mea+Ind	294.5	0.30	895.8	0.013	38.2
	Inferred	128.7	0.28	366.9	0.013	17.0
Lower-Grade	Measured	92.1	0.23	213.2	0.013	12.0
	Indicated	337.5	0.22	732.2	0.013	43.9
	Mea+Ind	429.6	0.22	945.3	0.013	56.0
	Inferred	396.5	0.21	839.3	0.013	51.3
Domain	Class	Tonnes (Mt)	Fe (%)	Fe Content (Mt)	S (%)	S Content (kt)
Higher-Grade	Measured	159.0	6.31	10.0	0.17	265.8
	Indicated	135.6	6.55	8.9	0.13	182.9
	Mea+Ind	294.5	6.42	18.9	0.15	448.7
	Inferred	128.7	6.78	8.7	0.10	134.0
Lower-Grade	Measured	92.1	6.78	6.2	0.05	42.7
	Indicated	337.5	6.87	23.2	0.04	142.1
	Mea+Ind	429.6	6.85	29.4	0.04	184.8
	Inferred	396.5	7.01	27.8	0.05	190.7
Domain	Class	Tonnes (Mt)	Cr (%)	Cr Content (kt)		
Higher-Grade	Measured	159.0	0.59	942.9		
	Indicated	135.6	0.57	771.7		
	Mea+Ind	294.5	0.58	1,714.6		
	Inferred	128.7	0.55	708.5		
Lower-Grade	Measured	92.1	0.61	564.2		
	Indicated	337.5	0.59	1,990.9		
	Mea+Ind	429.6	0.59	2,555.0		
	Inferred	396.5	0.58	2,290.7		
Domain	Class	Tonnes (Mt)	Pd (g/t)	Pd Content (koz)	Pt (g/t)	Pt Content (koz)
Higher-Grade	Measured	159.0	0.027	136.0	0.011	56.8
	Indicated	135.6	0.024	103.1	0.012	50.6
	Mea+Ind	294.5	0.025	239.1	0.011	107.4
	Inferred	128.7	0.017	70.1	0.011	46.7
Lower-Grade	Measured	92.1	0.012	35.5	0.009	26.7
	Indicated	337.5	0.010	113.0	0.008	83.6
	Mea+Ind	429.6	0.011	148.5	0.008	110.3
	Inferred	396.5	0.012	149.4	0.009	109.7
SUMMARY						
Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
Total Grade	Mea+Ind	724.1	0.25	1,841.2	0.013	94.1
	Inferred	525.3	0.23	1,206.2	0.013	68.3
Domain	Class	Tonnes (Mt)	Fe (%)	Fe Content (Mt)	S (%)	S Content (kt)
Total Grade	Mea+Ind	724.1	6.68	48.4	0.09	633.5
	Inferred	525.3	6.95	36.5	0.06	324.8
Domain	Class	Tonnes (Mt)	Cr (%)	Cr Content (kt)		
Total Grade	Mea+Ind	724.1	0.59	4,269.6		
	Inferred	525.3	0.57	2,999.2		
Domain	Class	Tonnes (Mt)	Pd (g/t)	Pd Content (koz)	Pt (g/t)	Pt Content (koz)
Total Grade	Mea+Ind	724.1	0.017	387.6	0.009	217.7
	Inferred	525.3	0.013	219.5	0.009	156.4

Table 1.7: Summary of the Pit-Constrained Updated East Zone Mineral Resource Estimate

Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
Higher-Grade	Measured	212.5	0.26	555.5	0.013	26.7
	Indicated	242.7	0.26	630.4	0.013	31.3
	Mea+Ind	455.1	0.26	1,185.9	0.013	58.1
	Inferred	104.6	0.26	271.7	0.013	13.3
Lower-Grade	Measured	72.9	0.19	135.8	0.013	9.2
	Indicated	172.9	0.18	313.4	0.013	21.9
	Mea+Ind	245.8	0.18	449.2	0.013	31.1
	Inferred	40.2	0.19	75.3	0.013	5.1
Domain	Class	Tonnes (Mt)	Fe (%)	Fe Content (Mt)	S (%)	S Content (kt)
Higher-Grade	Measured	212.5	5.99	12.7	0.06	122.3
	Indicated	242.7	6.13	14.9	0.08	188.2
	Mea+Ind	455.1	6.06	27.6	0.07	310.5
	Inferred	104.6	6.16	6.4	0.06	66.1
Lower-Grade	Measured	72.9	7.15	5.2	0.03	25.1
	Indicated	172.9	7.35	12.7	0.03	48.5
	Mea+Ind	245.8	7.29	17.9	0.03	73.6
	Inferred	40.2	7.32	2.9	0.03	11.1
Domain	Class	Tonnes (Mt)	Cr (%)	Cr Content (kt)		
Higher-Grade	Measured	212.5	0.64	1,368.1		
	Indicated	242.7	0.64	1,551.1		
	Mea+Ind	455.1	0.64	2,919.2		
	Inferred	104.6	0.64	668.3		
Lower-Grade	Measured	72.9	0.57	412.8		
	Indicated	172.9	0.54	939.4		
	Mea+Ind	245.8	0.55	1,352.2		
	Inferred	40.2	0.55	221.9		
Domain	Class	Tonnes (Mt)	Pd (g/t)	Pd Content (koz)	Pt (g/t)	Pt Content (koz)
Higher-Grade	Measured	212.5	0.014	93.3	0.009	59.2
	Indicated	242.7	0.015	116.8	0.009	68.6
	Mea+Ind	455.1	0.014	210.1	0.009	127.7
	Inferred	104.6	0.014	45.9	0.008	25.7
Lower-Grade	Measured	72.9	0.008	18.4	0.006	14.9
	Indicated	172.9	0.007	41.1	0.006	34.8
	Mea+Ind	245.8	0.008	59.4	0.006	49.8
	Inferred	40.2	0.008	9.9	0.006	7.7
SUMMARY						
Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
Total Grade	Mea+Ind	701.0	0.23	1,635.1	0.013	89.1
	Inferred	144.8	0.24	347.0	0.013	18.4
Domain	Class	Tonnes (Mt)	Fe (%)	Fe Content (Mt)	S (%)	S Content (kt)
Total Grade	Mea+Ind	701.0	6.49	45.5	0.05	384.0
	Inferred	144.8	6.48	9.4	0.05	77.3
Domain	Class	Tonnes (Mt)	Cr (%)	Cr Content (kt)		
Total Grade	Mea+Ind	701.0	0.61	4,271.4		
	Inferred	144.8	0.61	890.1		
Domain	Class	Tonnes (Mt)	Pd (g/t)	Pd Content (koz)	Pt (g/t)	Pt Content (koz)
Total Grade	Mea+Ind	701.0	0.012	269.6	0.008	177.5
	Inferred	144.8	0.012	55.8	0.007	33.4

It is the opinion of the QPs that both the updated Main Zone and updated East Zone mineral resource estimates, completed in accordance with the requirements of the NI 43-101, reasonably reflect the mineralization that is currently known on the Crawford Nickel Sulphide Project and that there are reasonable prospects for future economic extraction, likely using open pit and/or bulk underground mining methods.

It is QP David Penswick's opinion that the calculated cut-off grade of approximately 0.10% Ni from pit optimization is relevant to the grade distribution of this project and that the mineralization exhibits sufficient continuity for economic extraction under this cut-off value. For purposes of the constraining the resource model, it was decided to conservatively adjust this value upwards to 0.15% Ni.

The mineral resources are not mineral reserves as they do not have demonstrated economic viability. The estimate is categorized as inferred, indicated and measured resources based on drill hole data density, geological and grade continuity, variogram range criteria and specific interpolation parameters.

1.12 Mineral Reserve Estimation & Statement

Not applicable.

1.13 Mining Methods

The Crawford mine plan design results in the mining of 2,794 Mt over a period of 27 years, including two years of pre-stripping before the process plant is commissioned. Of this total, 907 Mt was designed to feed to the mill, for a strip ratio of 2.1. The plan currently includes three discrete open pits. Largest and the first to be mined is the Main Zone (MZ), comprising 80% of feed to the mill and 85% of the recoverable nickel. The smaller East Zone-West (EZ-W) and East Zone-East (EZ-E) will be mined following depletion of the MZ, starting in Year 17 of process operations.

All waste generated while the MZ pit is active, including tailings, will be impounded on surface. The MZ will be mined at a rate faster than is required to satisfy the mill, allowing for accelerated delivery of higher value material to the mill while lower value material will be temporarily stockpiled in one of three low-grade stockpiles. Approximately 25% of total mill feed will be stockpiled. At the peak, the aggregate tonnage contained in the three stockpiles was designed to a total 157 Mt. While some of the higher value stockpiles material will be reclaimed while the MZ is active, the bulk will be reclaimed concurrent with mining from the EZ. As soon as operations in the MZ cease, all waste from the EZ and tailings from the mill will be impounded in the mined out MZ pit.

The pits will be mined with a mixed fleet of equipment. Approximately 58% of clay and gravel overburden will be loaded with small backhoes excavators into 40t articulated trucks or by medium-sized face shovel excavators into 90 t haul trucks. The remaining overburden and all rock, which in aggregate represent 95% of total ex-pit material, will be loaded by large, electrically-powered face shovel excavators into 290 t trucks. No drilling and blasting of overburden will be required. Approximately 85% of rock is sufficiently homogenous as to allow bulk blasting on 15 m benches using 311 mm holes. The remaining 15% of rock must be blasted more selectively, on 7.5 m flitches using 229 mm holes.

Crawford will make extensive use of technology to maximize the productivity of equipment and minimize costs. Technologies that will be employed include:

- autonomous drill systems (ADS) that eliminate the requirement of on-board operators for blast hole drills.
- autonomous haulage systems (AHS) that eliminate the requirement of on-board operators for 290 t haul trucks.
- trolley-assisted truck haulage that reduces the energy consumed while increasing the speed of 290 t haul trucks on uphill loaded hauls.

Over the life of mine, the complement will average 416 full-time equivalent personnel (FTE), reaching a peak in Year 11 of 613.

1.14 Recovery Methods

The process plant and associated service facilities will process ROM ore delivered to primary crushers to produce nickel concentrate and tailings. The proposed process encompasses:

- crushing and grinding of the ROM ore.
- desliming via hydrocycloning.
- slimes rougher and cleaning flotation.
- coarse rougher, scavenger and cleaning flotation.
- regrind of the coarse flotation circuit final concentrate.
- high-grade material flotation.
- magnetic recovery of coarse scavenger tailings.
- regrinding of magnetic concentrate, scavenger concentrate and coarse first cleaner tailings.
- fines rougher and cleaning flotation of the reground material.
- magnetic recovery of fines rougher and first cleaner tailings.

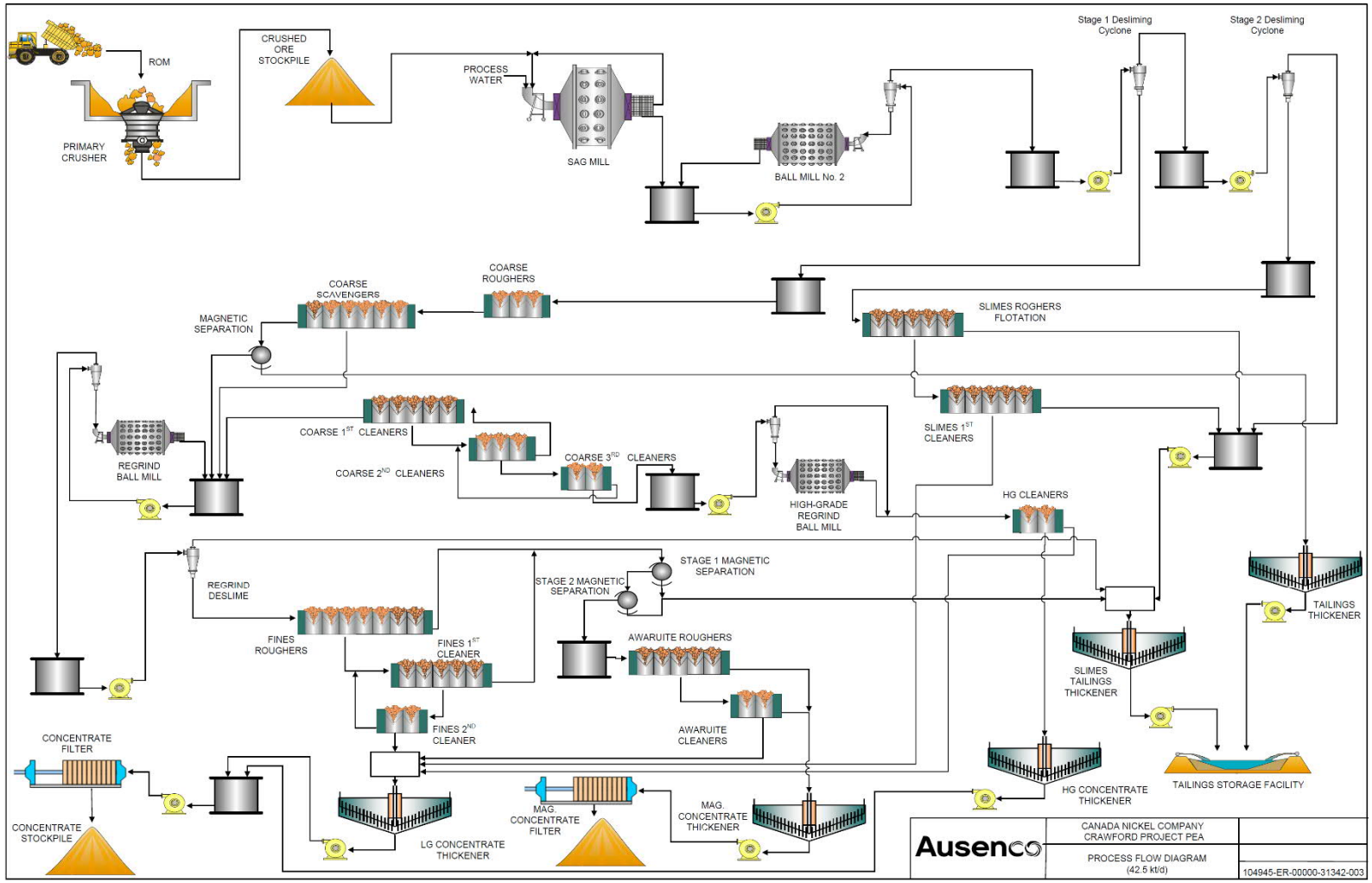
Nickel and magnetite concentrates will be thickened, filtered, and stockpiled on site prior to being loaded onto railcars or trucks for transport to third-party smelters. The slimes flotation tailings, nickel sulphide magnetic separation tailings, and magnetite recovery circuit tailings will be combined and thickened before placement in the tailings storage facility (TSF). The process plant will be built in three phases. Initially, the plant will be designed to process 42.5 kt/d with allowances for a duplicate process expansion to increase plant capacity to 85 kt/d. The ultimate expansion consists of both concentrators expanded to achieve 120 kt/a. Common facilities will include concentrate handling and sulphuric acid off-loading and containment.

The key criteria selected for the base and expansion plant designs are:

- Phase 1 nominal base plant treatment rate of 42.5 kt/d.
- Phase 2 nominal base plant treatment rate of 85 kt/d (duplication of the Phase 1 processing train).
- Phase 3 nominal expansion plant treatment rate of each processing train from 42.5 kt/d to 60 kt/d for a combined 120 kt/d treatment rate.
- design availability of 92% (after ramp-up), which equates to 8,059 operating hours per year, with standby equipment in critical areas.
- sufficient plant design flexibility for treatment of all ore types at design throughput.

A schematic of the process plant (only 42.5 kt/d plant shown) is provided in Figure 1.3.

Figure 1.3: Crawford Process Plant Schematic



Source: Ausenco, 2021.

1.15 Project Infrastructure

The project site is well-serviced with respect to other infrastructure, as shown in Figure 1-4. This includes the following:

- Road – Provincial Highway 655 runs along the western boundary of the property. Approximately 13.2 kilometres of the highway will need to be rerouted as it currently runs through the pit envelope.

- Rail – A rail spur that services the process plant is proposed for the project. The total length of the rail spur is approximately 20 km and will connect to the Glencore Kidd Operations rail line.

- Power – The provincial utility, Hydro One, has indicated that it would be feasible to provide electrical power to the mine site for Phases 1 and 2 via a 2.5 km overhead powerline tied into the existing H6T and H7T located northeast of the property. Phase 3 requires a new 38 km long 230 kV overhead powerline to be constructed, which would be connected as a tee-off to an existing line located at Timmins. The line will enter the property from the south and run to the process plant main 120 kV substation.

- Water – Water for start-up will be provided by surface water storage at the sedimentation pond and, possibly, local groundwater wells. During operations, water demand will largely be met by recycling water from the TSF. Make-up water and freshwater requirements will be provided by the sedimentation pond. A water treatment plant will be constructed to treat excess water from the TSF prior to its discharge to the local catchment area.

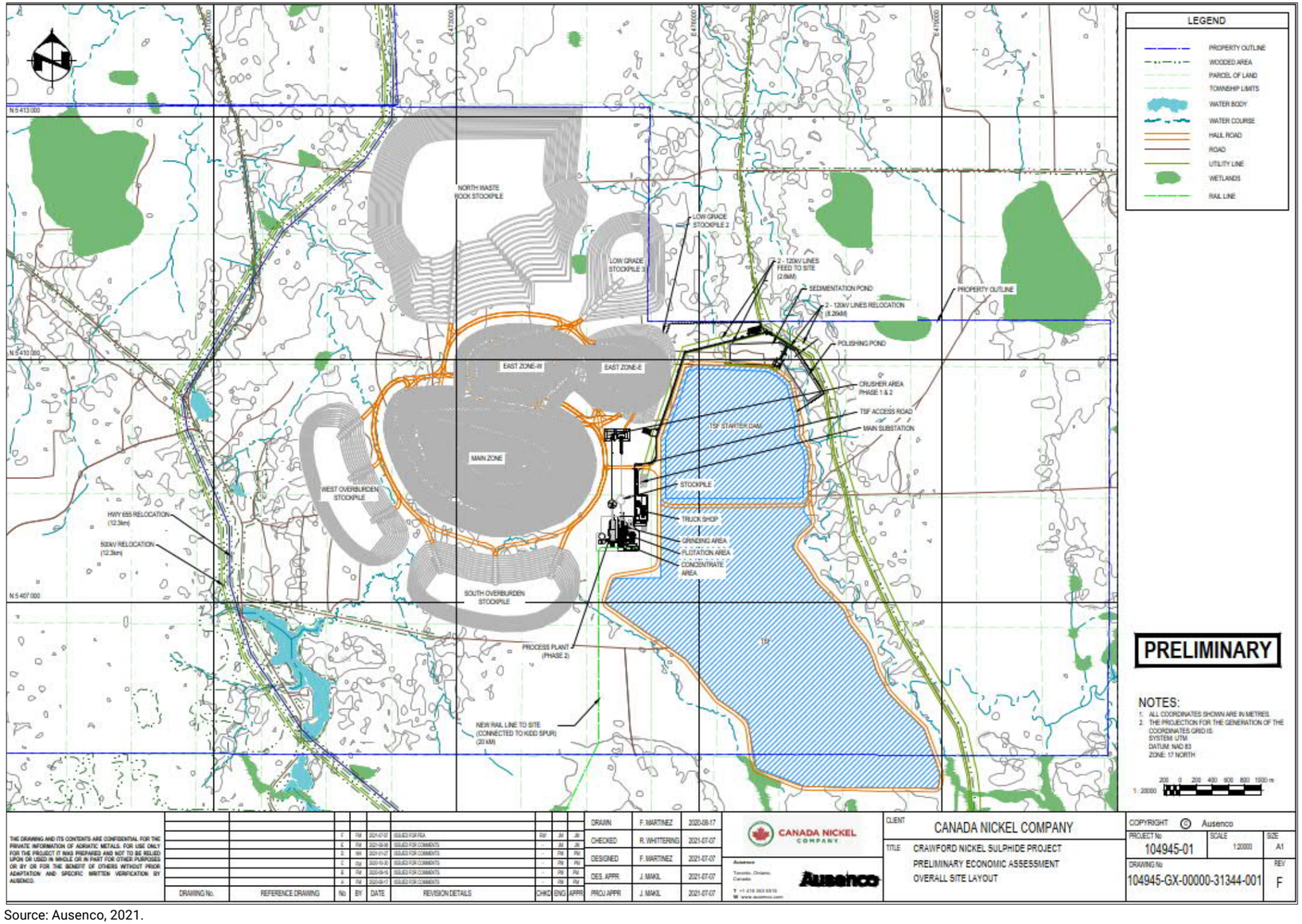
Both the initial and expansion phases of the project will require dual 120:13.8 kV 60/80 MVA main transformers. The new 120 kV substation and six main transformers will be installed near the SAG mill feed conveyor. The 13.8 kV medium-voltage network will be used for the primary electrical distribution and for feeding large loads such as the SAG mill and ball mills.

The process plant area consists of the crushing facility, covered stockpile and process plant building. The overall process plant enclosed structure is approximately 350 m long, and consists of four connected buildings: grinding, flotation, magnetic separation, and concentrate thickening.

The TSF is designed to be situated approximately 200 m east of the process plant and consists of two cells. Cell 1 will be constructed initially, followed by Cell 2 during Year 5 of operations.

The TSF is designed to store approximately 540 Mt of tailings produced over a period of approximately 17 years while the Main Zone pit is active. For the remaining 8 years of project life, mill feed will be sourced from the East Zone and reclaimed stockpiles. Tailings from this material, totalling approximately 317 Mt, will be impounded in the mined out Main Zone pit.

Figure 1.4: Overall Site Layout



Source: Ausenco, 2021.

1.16 Social and Environmental Considerations

1.16.1 Social Considerations

As CNC is moving the project towards a feasibility study and impact assessment, the intention of the company is to fully engage with local Indigenous Nations and stakeholders in a comprehensive consultation process aimed at identifying and addressing significant challenges associated with the development of the Crawford Nickel Sulphide Project, but also seeking to underline and maximize potential social, environmental, and economic benefits of the project. Although the range of stakeholders is expected to evolve to reflect various levels of interest and issues over time, CNC already identified a number of local stakeholders who have or could demonstrate an interest in the project. CNC is also actively engaging local Indigenous communities, including establishment of memoranda of understanding to support future engagement and participation in the project, as appropriate.

In parallel with the consultation process, CNC is 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 adapted to the needs of the communities.

1.16.2 Environmental Considerations

Environment baseline studies have been initiated as needed to support anticipated environmental approvals requirements, as well as engineering design. To date there have been no findings that were in conflict with the proposed design. 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 area and at the site, based on experience with other mining developments in northern Ontario. As the project design and environmental studies progress, this assumption will be confirmed.

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. It is expected that completion of a Federal Impact Assessment process will be required based on the current project design, as well as potentially one or more Provincial Environmental Assessments. The federal and provincial governments typically coordinate processes to gain efficiencies. Upon completion of these processes, a number of federal and provincial authorizations, approvals and permits will be required to construct and operate the mine. These processes are well understood.

1.16.3 Closure and Reclamation Considerations

The project will be required to complete a regulatory Closure Plan prior to commencement of construction activities. Financial assurance is required to be submitted to the province in order to provide sufficient funds for the Crown to undertake the closure and reclamation activities described in the closure plan.

A detailed estimate of the costs associated with closure has not been completed at this stage of the project. Rather, costs have been assumed to be similar to those reported for other similar scale existing operations and prospective projects in the Abitibi region. The financial evaluation makes provision for total closure costs of US$34.4 million.

1.17 Markets and Contracts

Pricing assumptions were developed for nickel, iron and chrome that are recovered in the nickel and magnetite concentrates produced at Crawford and are effective as of May 2021. The long-term price assumptions selected are US$7.75/lb for nickel, consistent with the average of industry analysts who follow the company and a leading independent nickel industry analyst, US$290/t for iron, and US$1.04/lb for chrome. As the iron and chrome content of the magnetite concentrate was being utilized in the production of stainless steel and nickel pig iron, the appropriate iron and chrome pricing basis (U.S. No. 1 heavy melting scrap for iron and U.S. ferrochromium for chrome), the company relied on the 10-year U.S. average No. 1 heavy melting composite price for iron price and the 10-year average ferrochromium price ($/lb chromium content) as tracked by the U.S. Geological Survey and a leading independent industry analyst for chrome prices.

Crawford is anticipated to produce a high-grade (35% nickel) concentrate, a standard-grade concentrate (12% nickel), and a magnetite concentrate containing an average of 48% iron and 3% chrome. The combined Crawford concentrates, with a relatively high average nickel content of 20% nickel over the life of project, and substantial quantities of iron and chrome is suited for use as a high-quality raw material feed for use in stainless steel production, ferronickel, or nickel pig iron producers by roasting the sulphide concentrates. This is followed by production of stainless steel and nickel pig iron using standard rotary kiln/electric furnace and then refining in an AOD (RKEF/AOD). The use of this RKEF/AOD technology to produce nickel pig iron and stainless steel has become the dominant production path for nickel and stainless steel and is in widespread use. The high nickel content of the concentrate means that lower amount of power, reductants (coke), and energy are required for processing resulting in lower costs and higher payabilities compared to traditional smelting and refining.

CRU, a leading, provider of analysis, prices and consulting in the mining, metals and fertilizer markets, prepared a value-in-use study and market analysis that looked at the value that a third-party local producer may be willing to pay for the feed utilizing standard RKEF/AOD technology widely used in China and Indonesia to produce 304 stainless steel and 1.6% nickel pig iron. It found that net payabilities utilizing this approach for all could achieve 91% payability for contained nickel, 71% payability for contained iron, and 43% payability for contained chromium in the concentrates.

1.18 Capital Cost Estimates

The estimate is considered to carry an accuracy of ±50% per AACE Class 5 estimating standards. Table 1.8 provides a summary of the capital cost estimate, including initial capital, and expansion capital. Table 1.9 summarizes the sustaining capital. The costs are expressed in real, Q1 2021 US dollars.

Table 1.8: Summary of Project Capital Costs (US$M)

US$M Initial and Expansion	Initial 42.5 kt/d	85 kt/d	120 kt/d	Life of Mine Years 1-25
Mining	201	-	-	201
Process Plant	294	294	98	685
Site and Services	157	132	4	293
Infrastructure	149	15	25	189
Indirects	108	31	22	161
Owners' Costs	29	-	-	29
Contingency	250	71	45	366
Total	**1,188**	**543**	**194**	**1,925**

Table 1.9: Sustaining Capital Costs (US$M)

US$M Sustaining and Closure	Pre-Production	Phase 1 Years 1-3.5	Phase 2 Years 3.5-7	Phase 3 Years 8-18	Life of Mine Years 1-25
Mining	-	187	195	285	711
Site and Services	-	34	44	136	214
Infrastructure	-	8	15	91	132
Closure	26	9	-	-	34
Total	**26**	**238**	**254**	**512**	**1,091**

Items that would be denominated in foreign currency take account of the forecast exchange rate at the time of purchase. Indirect costs include first fills of consumable items for the initial and expansion estimates, and the release of these under the sustaining estimate.

1.19 Operating Cost Estimates

A summary of average life-of-mine operating costs is provided in Table 1.10.

Table 1.10: Operating Cost Summary

Cost Area	Phase 1 Years 1-3.5		Phase 2 Years 3.5-7		Phase 3 Years 8-18		Life of Mine Years 1-25	
Operating Cost/Tonne Milled	**US$**	**C$**	**US$**	**C$**	**US$**	**C$**	**US$**	**C$**
Labour	2.39	3.19	1.49	1.98	1.20	1.60	1.26	1.68
Consumables	2.49	3.31	2.36	3.14	2.30	3.07	2.25	3.00
Maintenance	1.70	2.27	1.47	1.96	1.69	2.25	1.54	2.05
Diesel	1.02	1.36	0.78	1.04	0.78	1.04	0.72	0.96
Power	2.45	3.26	2.40	3.20	2.35	3.13	2.25	3.00
Other	0.95	1.27	0.52	0.70	0.40	0.53	0.43	0.58
Total	**11.00**	**14.66**	**9.02**	**12.02**	**8.72**	**11.62**	**8.45**	**11.27**

1.20 Economic Analysis

The currently defined resource is expected to produce 1.7 billion pounds of payable nickel (or 2.4 billion pounds of payable nickel equivalent, including by-product chromium and iron) over a 25-year life. The summarized costs and economic returns presented in Table 1.11 assume the following average long-term prices:

- Nickel = $7.75/lb.
- Chromium = $1.04/lb.
- Iron = $290/t.
- CAD $1.00 = $0.75.

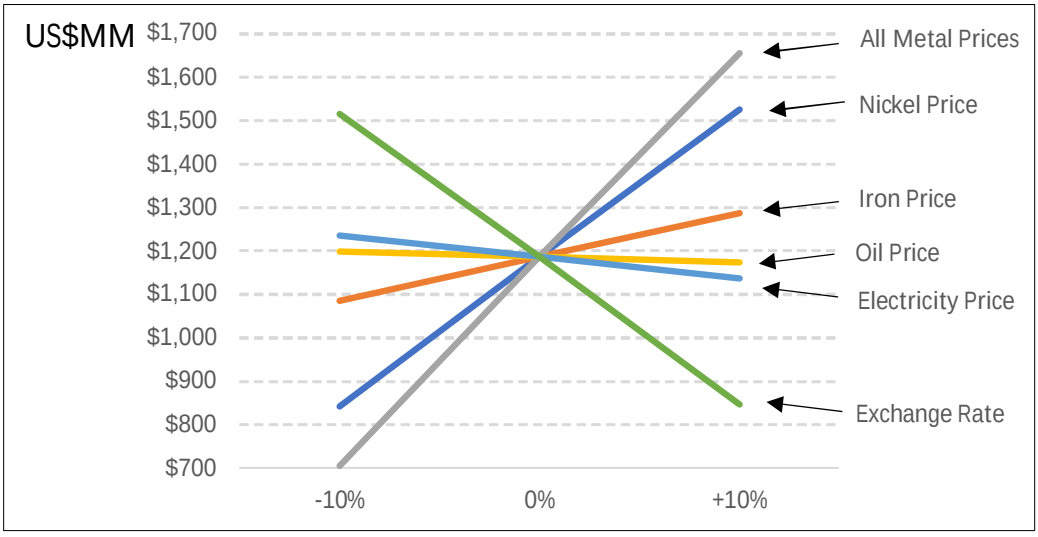
Table 1.11: PEA Summary Metrics

Item	Units	Value	Comment
Mill Feed	Mt	907	
Payable Ni	Mlbs	1,689	
Payable NiEq	Mlbs	2,441	Includes by-product Cr & Fe
Net Smelter Return 1	US$/t mill feed	$20.86	Revenues assumed paid on an FOB mine gate basis
Site Operating Costs	US$/t mill feed	$8.45	
Net C1 Costs	US$/lb Ni	$1.09	Includes site operating costs and net of by-product credits
EBITDA	US$/t mill feed	$11.99	
Peak Funding Requirement	US$ MM	$1,201	Cumulative unlevered investment prior to positive cash flow
Total Investment	US$ MM	$3,016	Includes all capital and closure expenditures
Net AISC	US$/lb Ni	$1.94	Includes C1 costs, royalties, sustaining capital and closure
Pre-Tax NPV0%	US$ MM	$7,855	
Pre-Tax NPV8%	US$ MM	$1,856	
Post-Tax NPV8%	**US$ MM**	**$1,187**	
Post-Tax IRR		**15.8%**	

1.21 Sensitivity Analysis

Figure 1-5 illustrates that economic returns are most sensitive to variance in factors influencing revenue in Canadian dollar terms. A 10% variance in either the nickel price, metallurgical recovery of nickel or CAD f/x rate causes a variance in NPV of approximately 29% or nearly 3.0x. The impact of variance in iron prices or metallurgical recovery of iron is also significant, with a 10% variance in either factor having an impact on NPV of approximately 9% or 0.9x. Use of trolley assist reduces the reliance on diesel so that the impact of variance in oil prices is low at 0.05x.

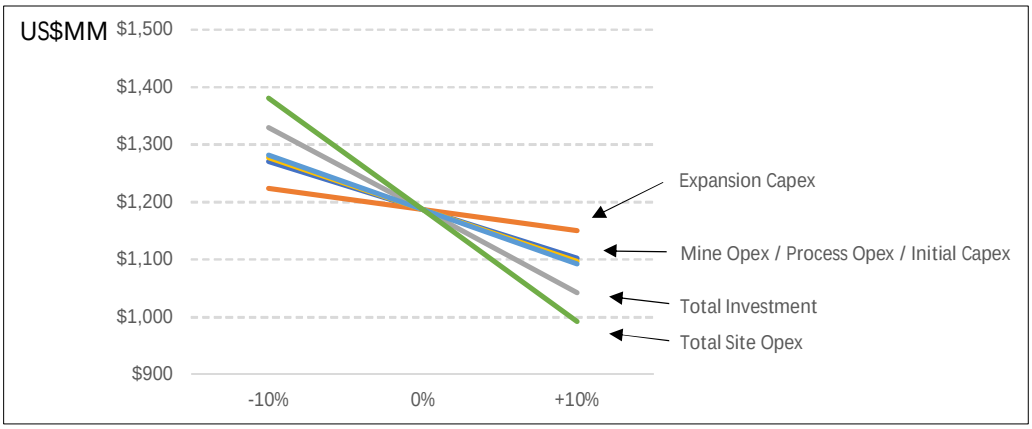
Figure 1.5: NPV8% Sensitivity to Macro-Economic Assumptions



Source: Penswick, 2021.

Figure 1-6 illustrates that NPV is more sensitive to operating costs, with a 10% variance in total site costs having a 16% impact on NPV compared to the same variance in capital costs having a 12% impact.

Figure 1.6: NPV8% Sensitivity to Operating and Capital Costs



Source: Penswick, 2021.

1.22 Other Relevant Information

There is a potential opportunity to use and/or modify Glencore's spare processing capacity at the Kidd concentrator and metallurgical site approximately 40 km south of the Crawford property to process mined ore early in the mine development, prior to the construction completion of the major processing and infrastructure assets.

This opportunity was assessed, and a potential processing configuration exists to treat a feed throughput of 2.75 kt/d, acting as a smaller-scale operation in a shorter ramp-up time frame with a forecasted low capital outlay (refer to Section 24). This opportunity will be further investigated in subsequent study phases.

1.23 Interpretation and Conclusions

The Main Zone Mineral Resource Estimate as of May 19, 2022, contains a higher-grade zone (core) with Measured and Indicated resources of approximately 295 Mt at 0.30% Ni, 0.013% Co, 6.42% Fe, 0.58% Cr, and 0.036 g/t Pd + Pt, and Inferred resources of about 129 Mt at 0.28% Ni, 0.013% Co, 6.78% Fe, 0.55% Cr, and 0.028 g/t Pd + Pt. The Main Zone also contains two lower-grade zones (north and south of the higher-grade zone), with Measured and Indicated resources of approximately 430 Mt at 0.22% Ni, 0.013% Co, 6.85% Fe, 0.59% Cr and 0.019 g/t Pd + Pt, and inferred resources of about 397 Mt at 0.21% Ni, 0.013% Co, 7.01% Fe, 0.58% Cr and 0.021 g/t Pd + Pt.

The East Zone Mineral Resource Estimate as of May 19, 2022, contains a higher-grade zone (core) with Measured and Indicated resources of approximately 455 Mt at 0.26% Ni, 0.013% Co, 6.06% Fe, 0.64% Cr, and 0.023 g/t Pd + Pt, and Inferred resources of about 105 Mt at 0.26% Ni, 0.013% Co, 6.16% Fe, 0.64% Cr, and 0.022 g/t Pd + Pt. The East Zone also contains two lower-grade zones (north and south of the higher-grade zone), with Measured and Indicated resources of approximately 246 Mt at 0.18% Ni, 0.013% Co, 7.29% Fe, 0.55% Cr and 0.014 g/t Pd + Pt, and Inferred resources of about 40 Mt at 0.19% Ni, 0.013% Co, 7.32% Fe, 0.55% Cr and 0.014 g/t Pd + Pt.

All mineral resources are constrained by conceptual pit envelopes in order to demonstrate reasonable prospects for eventual economic extraction. At the Main Zone, a cut-off of 0.25% nickel was used to separate and model the higher-grade and lower-grade domains. At the East Zone, a combination of aluminum, magnesium and density cut-offs were used to

separate two dunite types: A highly serpentinized "pure" dunite with higher grades (equivalent to the Main Zone's higher-grade domain) and a moderately serpentinized "transitional" dunite with lower grades. The latter, together with adjacent lower-grade peridotites, form the lower-grade domain.

The Crawford Nickel Sulphide Project will be a conventional open pit mine/mill operation powered by zero-carbon electricity and utilizing mainly trolley-assist trucks and electric excavators to minimize its carbon footprint through reduced diesel consumption. The project will produce three products: (1) a high-grade concentrate estimated at 35% nickel; (2) a standard grade concentrate estimated at 12% nickel; and (3) a magnetite concentrate estimated at 48% iron and 3% chromium. All of the products are assumed to be sold based on the nickel, iron, and chromium content of the concentrates on terms which provide sufficient incentive for the construction of a co-located stainless steel mill using the same RKEF-AOD approach utilized successfully in China and Indonesia.

The process plant will utilize a conventional milling operation consisting of crushing, grinding, desliming and flotation operations consistent with other ultramafic nickel operations. The process plant will be constructed in three phases. Phase 1 is designed to have a steady-state throughput of 42,500 t/d using a single 36 x 24 ft semi-autogenous grinding mill and a 26.5 x 44 ft ball mill grinding circuit. Phase 2 will double throughput starting in year four, by mirroring the first line, bringing total production to 85,000 t/d. Phase 3 will raise production to the ultimate rate of 120,000 t/d through the addition of secondary crushing, a third ball mill and additional downstream capacity.

The initial capital cost for Phase 1 of the project is estimated at US$1,188 million. Including the subsequent Phase 2 and Phase 3 expansions, project capital costs are estimated to total US$1,925 million. Sustaining capital expenditures over the 25-year life of mine are estimated to total a further US$1,091 million. The life of mine average operating cost is estimated to be US$8.45/t milled.

The PEA has presented robust project economics showing an after-tax NPV 8% of US$1.2 billion and an after-tax IRR of 16% using an average sales price for nickel concentrate of US$7.75/lb, a chromium price of US$1.04/lb, and an iron price of US$290 per tonne.

Simple payback of all invested capital on a post-tax basis is projected to be achieved 7.4 years after the start of commercial production. Recent changes to Canada's tax legislation result in the payment of cash income taxes during the payback period. On a pre-tax basis, the payback period is 9 months sooner at 6.7 years. Over the current 25-year life of mine, the project is forecast to generate an average of US$347 million operating cash flow and US$274 million free cash flow annually.

1.24 Recommendations

1.24.1 Introduction

Considering the positive outcome of this Report, it is recommended to develop the project through additional studies, as outlined below. Table 26.1 summarizes the proposed budget to advance the project through the next study stage.

Figure 1.6: Estimated Cost of Next Study Phase

Description	Cost C$
Geology	$15,000,000
Mining	$2,000,000
Geotechnical	$2,500,000
Metallurgy	$3,000,000
Infrastructure	$500,000

Description	Cost C$
Environmental	$1,500,000
FS Study budget	$1,000,000
Total Recommended Study Budget:	**$25,500,000**

1.24.2 Mineral Resources

Based on the work completed for the mineral resource estimate in the Main Zone, which has increased its Measured and Indicated resources within the higher-grade though still has largely Indicated and Inferred resources within the lower-grade, it is recommended that in-fill drilling programs be aimed at upgrading the remaining Inferred resources within the higher-grade but also to upgrade the sizeable Inferred and Indicated resources within the lower-grade.

Based on the work completed for the mineral resource estimate in the East Zone, which has notably converted most of its Inferred resources to Indicated and Measured, especially in its western region, it is recommended that in-fill drilling programs be aimed at upgrading its Inferred and Indicated resources in the eastern region at moderate to high depths.

Both zones might also benefit from deeper (>1 km) exploration drilling, as it can be reasonably presumed that the deposits are open in depth.

1.24.3 Mining

The following recommendations are made with regard to future mining studies:

- Updates to the resource model resulting from drilling conducted subsequent to the current study should be incorporated.
- A geotechnical program should be executed that includes the following:
    - accurately identifying the interface between the various lithologies, including clay, gravel and rock
    - testing of each of the lithologies in order to determine slope angles for both the pit and various impoundments
- execution of a hydrogeological work program to determine the impact of ground water on slope angles and other elements of the mine design; this work should additionally determine dewatering requirements for the operation
- Condemnation drilling of the footprints identified for various surface infrastructure, including waste impoundments, should be carried out.
- Further work should be conducted to identify the optimal SMU size for various zones of mineralization; focus should be directed to the zones of cobalt and PGE mineralization that have not been included in the current study.
- Vendors of drilling and blasting equipment and services should be invited to test the various rock types in order to determine the optimal hole sizes and powder factors to be used.
- Potential mining contractors and equipment OEMs should continue to be engaged in order to obtain updated quotations.

- The design and schedule for the open pit mines and waste impoundments should be updated using inputs from the various work programs outlined above.

1.24.4 Metallurgical Testwork

The next phase of flowsheet development work should focus on optimizing the recoveries of nickel, iron chromium, cobalt, and platinum group metals (PGM) in the mineral processing flowsheet. The metallurgical testwork should include:

- optimization of the coarse rougher-scavenger flotation circuit
- effect of the secondary regrind size and the secondary deslime on fine circuit flotation performance
- optimization of the fines flotation circuit
- optimization of the slimes cleaning process
- evaluation of alternative strategies to recover nickel that is locked in silicates from the mineral processing tailings
- reduction in reagent consumption across the flowsheet and substitution of costly reagents such as CMC with less expensive alternatives
- impact of regrinding the feed to the magnetic circuit on the quality of the magnetic concentrate (a trade-off study will be done to understand the cost-benefit analysis of further upgrading of the magnetic concentrate on flux requirements in downstream process steps)
- thorough assessment of the size-by-size deportment of nickel in a plant flowsheet with particular reference to hydrocyclone operation which tends to classify acicular particles and particles of different specific gravities differently.
- thorough assessment of the energy sensitivity in flotation with particular focus on the selection of plant equipment
- thorough assessment of the dewatering properties of the tailings and the resulting storage and water balance factors

The results of the flowsheet development work listed above will be tested at the pilot scale and used to lock in a final flowsheet for the feasibility study. The estimated cost for the flowsheet development work that will be completed for the start of the feasibility study, including contingency, is C$1M.

1.24.5 Site Infrastructure

The following activities are recommended to support the design of the site infrastructure into the next phase of the project:

- Geotechnical site investigations should be carried out at the most optimal surface infrastructure site location to characterize the foundation requirements associated with the proposed surface infrastructure facilities.
- The final location of the TMF and mine dumps should be optimized to minimize the relocation of existing power lines and access roads.

1.24.6 Water Management

The goal of water management is to recycle the water from the TSF and feed the process plant with the required water quantity. The volume of the water in the TSF should be sufficient to support the ore processing in both dry seasons and winter conditions. Water from the slurry deposition and precipitation runoff will be collected in a small pond within the TSF. Water will be pumped to a recycle water pond located downstream of the TSF.

A detailed hydrologic/hydrogeological analyses and water balance must be completed to determine water intake and discharge requirements.

1.24.7 Tailings Management Facility

A preliminary geotechnical investigation at the TSF is required within the footprint of the dams to provide a better understating of the subsurface soil and groundwater conditions. The geotechnical investigation for the dams should be boreholes drilled along the proposed alignment of the dams. The next level of design should use the geotechnical investigation results to determine the stability and design of the TSF dams. Tests should be completed to determine properties of the deposited tailings as well.

The feasibility design configuration of the dams can be established once physical and mechanical soil properties are available following sampling and field and laboratory testing. A preliminary geotechnical investigation at the TSF is required within the footprint of the dams to provide a better understating of the subsurface soil and groundwater conditions. The geotechnical investigation for the dams should be boreholes drilled along the proposed alignment of the dams. No less than 16 boreholes should be drilled for the initial investigation. Additional geotechnical investigations may be needed based on the findings of the initial investigation and feasibility design in areas of concern or variable conditions. In addition to field tests (SPT, Nilcon Vane, etc.) carried out during the investigations, a number of laboratory tests should be carried out to classify the soils and the physical/mechanical properties (strength and compressibility characteristics, coefficient of permeability, maximum, and minimum dry density, optimum water content values etc.). The list of tests is shown below:

- Field tests:
  - Nilcon Vane.
  - standard penetration test (SPT).
  - cone penetration test (CPT).
  - dynamic cone penetration test (DCPT).
  - falling head permeability and packer testing.
- Laboratory tests:
  - particle size analysis.
  - Hydrometer.
  - Atterberg limit.
  - specific density (specific gravity).

- proctor test (compaction of the fill materials).

- Advanced laboratory tests:
  - constant and falling head permeability tests on natural and on compacted samples.
  - unconfined and triaxial compression tests.
  - direct shear tests.
  - compressibility/consolidation tests.

Bedrock samples from the available core boxes should be taken to the laboratory and their acid generation potential should be evaluated.

The results of all field and laboratory tests should be presented in a geotechnical factual report.

Test pits should be excavated inside the TSF area to evaluate if there are any suitable borrow areas for materials that could potentially be used for dam construction.

Instrumentation should be installed in the TSF dams and its native foundation material and should be monitored during the construction and operation of the TSF. Monitoring instrumentation may include (but not be limited to) vibrating wire piezometers, inclinometers, settlement cells, groundwater monitoring wells and survey monuments. As part of the routine operation of the tailings' facility, extensive monitoring of all aspects of the operation should be undertaken. Complete details of the monitoring program should be included in the operations manual that will be produced for the TSF at the detailed design stage. Monitoring should be carried out throughout all stages of the facility's life including construction, operation, closure, decommissioning, and post-closure.

1.24.8 Environmental, Permitting & Community Relations

CNC should continue to actively engage with stakeholders and local Indigenous communities on the project design elements going forward.

Environmental baseline studies and geochemistry studies should continue to be progressed in order to support a timely environmental approvals process, as well as to support the engineering design. As possible, arrangements should be made with local Indigenous communities to support completion of Traditional Knowledge / Traditional Land Use studies. CNC has already initiated discussions concerning these arrangements with the Wabun Tribal Council and Taykwa Tagamou Nation.

CNC should enter into the planning stage during 2021 for an Impact Assessment by preparing and submitting an Initial Project Description in the required format. This same document or a similar document, could be used to consult with the Provincial Ministries and obtain clarification on Provincial Environmental Assessment processes required.

2 INTRODUCTION

Some sections, text and figures have been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021, and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

2.1 Introduction

This report (the "Report") was prepared by Caracle Creek International Consulting Inc (Caracle Creek), with support from Ausenco Engineering Canada Inc. (Ausenco), and Wood Environment & Infrastructure Solutions Canada Limited (Wood), for Canada Nickel Company Inc. (CNC) to describe work completed on the Crawford Nickel Sulphide Project (the "Project", the "Property", or "Crawford") and to support disclosures by CNC in a news release dated 6 July 2022 titled, "Updated Mineral Resource Estimate Doubles Measured & Indicated Resources at Canada Nickel's Crawford Nickel Sulphide Project".

This Report and 2021 PEA were prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects. Readers are cautioned that the 2021 PEA is preliminary in nature. and in accordance with the requirements of Form 43-101 F1.

The responsibilities of the engineering and geological consultants are as follows:

- Caracle Creek International Consulting Inc. was commissioned by CNC to manage and coordinate the work related to the report and to complete the mineral resource estimates, inclusive of a review of the drilling campaigns to date, sample preparation, and analyses for use in the mineral resource estimate.
- Ausenco was commissioned to lead the development of the PEA-level design and cost estimating of the process plant and surface infrastructure.
- Wood Environment & Infrastructure Solutions Canada Limited (previously Wood Canada Limited) was commissioned to support environmental planning, assessment, licensing, permitting, and conceptual design of the tailings management facility (TMF) and complete the layout and sizing of water management structure.
- David Penswick, an independent mining consultant, was commissioned to design the open pit mine plan, mine production schedule, mine capital and operating costs and economic evaluation.

2.2 Terms of Reference

This Report supports disclosures by CNC in a news release dated July 6, 2022, titled, "Updated Mineral Resource Estimate Doubles Measured & Indicated Resources at Canada Nickel's Crawford Nickel Sulphide Project". The Report was prepared for CNC by Caracle Creek, Ausenco and Wood, to provide CNC with sufficient information to support their ongoing disclosure requirements with respect to the Crawford Nickel Sulphide Project including updated mineral resource estimates.

The Crawford consists of an open pit mine and an associated processing facility along with on-site and off-site infrastructure to support the operation. The operation is designed to have an open pit mine with an ultimate plant throughput potential of 120,000 tonnes per day (t/d).

The PEA prepared by Ausenco demonstrates the potential to develop a phased conventional nickel sulphide concentrator, producing nickel concentrates and magnetite concentrate.

2.3 Qualified Persons

2.3.1 Responsibilities

Table 2.1 lists the qualified persons (QP) for this Report as defined in NI 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1.

Table 2.1: List of Qualified Persons and Report Responsibilities

Qualified Person	Professional Designation	Organization	Independent of CNC?	Report Section Responsibility
Paul Staples	P.Eng.	Ausenco	Yes	1.2, 1.3, 1.14, 1.15, 1.17, 1.18, 1.19, 1.22, 1.24.1, 1.24.5; 2.3.1; 3.1, 3.3; 5.5; 17; 18.1, 18.2, 18.3, 18.4, 18.5, 18.6, 18.8.7, 18.8.8, 18.10, 18.11, 18.14, 18.15, 18.16, 18.17; 19; 21 (except 21.3.1, 21.3.3, 21.4.6, 21.4.6, 21.4.9, 21.5.3); 24; 25.10, 25.11, 25.13, 25.14, 25.16; 26.1, 26.5; 27
Greg Lane	FAusIMM	Ausenco	Yes	1.10, 1.24.4; 13, 25.5; 26.4
Scott Jobin-Bevans	P.Geo.	Caracle Creek	Yes	1.1, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.11, 1.12, 1.23, 1.24.2; 2; 4.1, 4.2, 4.3, 4.4, 4.7; 6.1, 6.2, 6.3; 7; 8; 9; 10; 12; 14 (except 14.11); 15; 23; 25.1, 25.2, 25.3, 25.4, 25.6, 25.7 25.8, 25.17; 26.1, 26.2; 27
John Siriunas	P. Eng.	Caracle Creek	Yes	2; 6.4; 11; 12
David Penswick	P. Eng.	Independent Consultant	Yes	1.11, 1.13, 1.16.3, 1.20, 1.21, 1.24.3; 14.11; 16; 18.7, 18.9, 18.12, 18.13; 21.3.1, 21.3.3, 21.4.6, 21.4.9, 21.5.3; 22; 25.9, 25.15; 26.3
Sheila Daniel	P.Geo.	Wood	Yes	1.16.1, 1.16.2, 1.24.8; 3.2; 4.5, 4.6; 5.1, 5.2, 5.3, 5.4, 5.6; 20; 25.12; 26.8
Karel Van Zyl	P. Eng.	Wood	Yes	1.24.6, 1.24.7; 18.8.1 to 18.8.6; 26.6, 26.7

2.3.2 Site Visits and Scope of Personal Inspection

The qualified persons listed below have contributed to this Report as specified:

- Scott Jobin-Bevans of Caracle Creek for geology, exploration, mineral resource estimation, drilling, and data verification. Scott has not visited the Property.

- John Siriunas of Caracle Creek for QA/QC and data verification. John visited the Project site on October 12, 2019 (one day), on February 3-4, 2020 (two days), on September 10-11, 2020 (two days), and last on August 11-12, 2022 (two days). Travel from the City of Timmins, Ontario to the Project area takes approximately 30 minutes. Visits to observe the general property conditions and access and to verify the locations of some of the historical drill hole collars and work progress were made in the field. The focus of the September 2020 visit was to confirm the drilling in the East Zone and the progress of drilling in the Main Zone since the previous visit made in February 2020. The most recent site visit (August 2022) was to review the latest drilling used in the updated mineral resource estimate.

  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. In 2019 the secure storage and logging facility at 3700 Highway 101 West in Timmins was visited; in 2020, a visit to the larger facilities at 170 Jaguar Drive, Timmins was made.

  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.

- Paul Staples of Ausenco for recovery methods, plant and infrastructure capital and operating costs and study coordination. Paul has not visited Property.

- Greg Lane of Ausenco for mineral processing and metallurgical testing. Greg has not visited the Property.

- Sheila Ellen Daniel of Wood Environment & Infrastructure Solutions Canada Limited for environmental considerations, geochemistry and permitting. Sheila did not visit the property; however, she has relied upon other Wood staff who conducted environmental field baseline investigations during 2021.

- Karel Van Zyl of Wood Environment & Infrastructure Solutions Canada Limited for geotechnical, and tailings storage facility design. Karel has not visited the Property.

- David Penswick, independent mining consultant, for waste rock and overburden dump design, and low-grade ore stockpile design mining, mine capital, financial model and mine operating costs. David last visited the Property on 30 March 2021.

2.4 Effective Date

This Report has an Effective Date of May 19, 2022.

2.5 Previous Technical Reports

This Report replaces the following previous technical reports and is the current NI 43-101 technical report for the Crawford Project:

Ausenco (2021): Crawford Nickel Sulphide Project – NI 43-101 Technical Report & Preliminary Economic Assessment, Ontario, Canada. Prepared by: Ausenco Engineering Canada Inc. (Staples, Lane, Jobin-Bevans, Siriunas, Penswick, Daniel, Van Zyl). Report Date: July 9, 2021; Effective Date May 21, 2021, 384p.

Caracle Creek (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. Prepared for: Canada nickel Company Inc., Prepare by: Caracle Creek International Consulting Inc. (Jobin-Bevans, Siriunas,

Penswick). Report Effective Date: December 12, 2020; Mineral Resource Estimate Effective Date: 11 December 2020; Original Report Date: 4 December 2020; Amended Report Date: 31 December 2020, 221p.

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., 9 April 2020, 147p.

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. and Noble Mineral Exploration Inc. by Caracle Creek International Consulting Inc., 27 January 2020, 196p.

2.6 Abbreviations

A list of abbreviations used in the Report is provided in Table 2.2; the list is not exhaustive.

Table 2.2: Abbreviations used in the Report.

Abbreviation	Meaning	Abbreviation	Meaning
µm	micron	km	kilometre
°C	degree Celsius	km^2	square kilometre
°F	degree Fahrenheit	L	Litre
°	degrees (Az/dip)	m	metre
µg	microgram	M	mega (million)
a	Annum (year)	Mt	million tonnes
Au	gold	m^2	square metre
C$ or CAD	Canadian Dollar	m^3	cubic metre
cal	calorie	min	Minute
cm	centimetre	mASL	metres above sea level
d	day	mm	millimetre
ft	foot or feet	oz/t, oz/st	ounce per short ton
g	gram	oz	Troy ounce (31.1035 g)
G	giga (billion)	ppb	parts per billion
g/L	gram per litre	ppm	part per million
g/t	gram per tonne	s	Second
ha	hectare	ton, st	short ton
hp	horsepower	t, tonne	metric tonne
in	inch or inches	US$ or USD	United States Dollar
kg	kilogram	y	year

3 RELIANCE ON OTHER EXPERTS

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

3.1 Introduction

The QPs have relied upon the following other expert reports, which provided information regarding sections of this Report.

3.2 Environmental, Permitting, Closure, and Social and Community Impacts

The QPs have fully relied upon, and disclaim responsibility for, information supplied by CNC and Pierre-Philippe Dupont, Vice President of Sustainability, Canada Nickel Company; in particular, for information related to social considerations. This information is used in Section 20.

3.3 Markets

The QPs have not independently reviewed the marketing, iron or nickel unit value pricing and payability information. The QPs have fully relied upon, and disclaim responsibility for, information derived from CNC and experts retained by CNC for this information through the following documents:

- CRU Group, May 15, 2021: Stainless and NPI markets and value in use: A report prepared for Canada Nickel Co., 11 pages.
- KPM, May 15, 2021: Memo: Preliminary Review of Canada Ni's Latest Stainless Steel Scenario, Prepared for Arthur Stokreef and Mark Selby, date, 11 pages.

This information is used in Sections 19, 22, and 25.13 of the Report.

Metals market pricing is a specialized business requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive global database that is outside of the purview of a QP.

The QPs consider it reasonable to rely upon CRU International Limited and Kingston Process Metallurgy Inc. as the company provides up-to-date, in-depth insight and analysis into all facets of the metals industry, including production supply and costs as well as consumption demand, and metal price forecasts.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Crawford Nickel Sulphide Project is situated within the Timmins-Cochrane Mining Camp in Northeastern, Ontario, Canada, a region with a strong mining history (gold, nickel, zinc, lead etc.), and a Canadian province 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 Nickel Sulphide Project, located mostly in Crawford and Lucas townships with a small portion in Carnegie Township, 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 (Figure 4-2). The approximate center 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).

4.3 Mineral Tenure

The Project comprises approximately 11,433.22 ha (114.33 km^2), consisting of a combination of patented lands (Crown patents) and unpatented mining claims (staked claims), summarized in Tables 4.1 and 4.2 and shown in Figure 4.2.

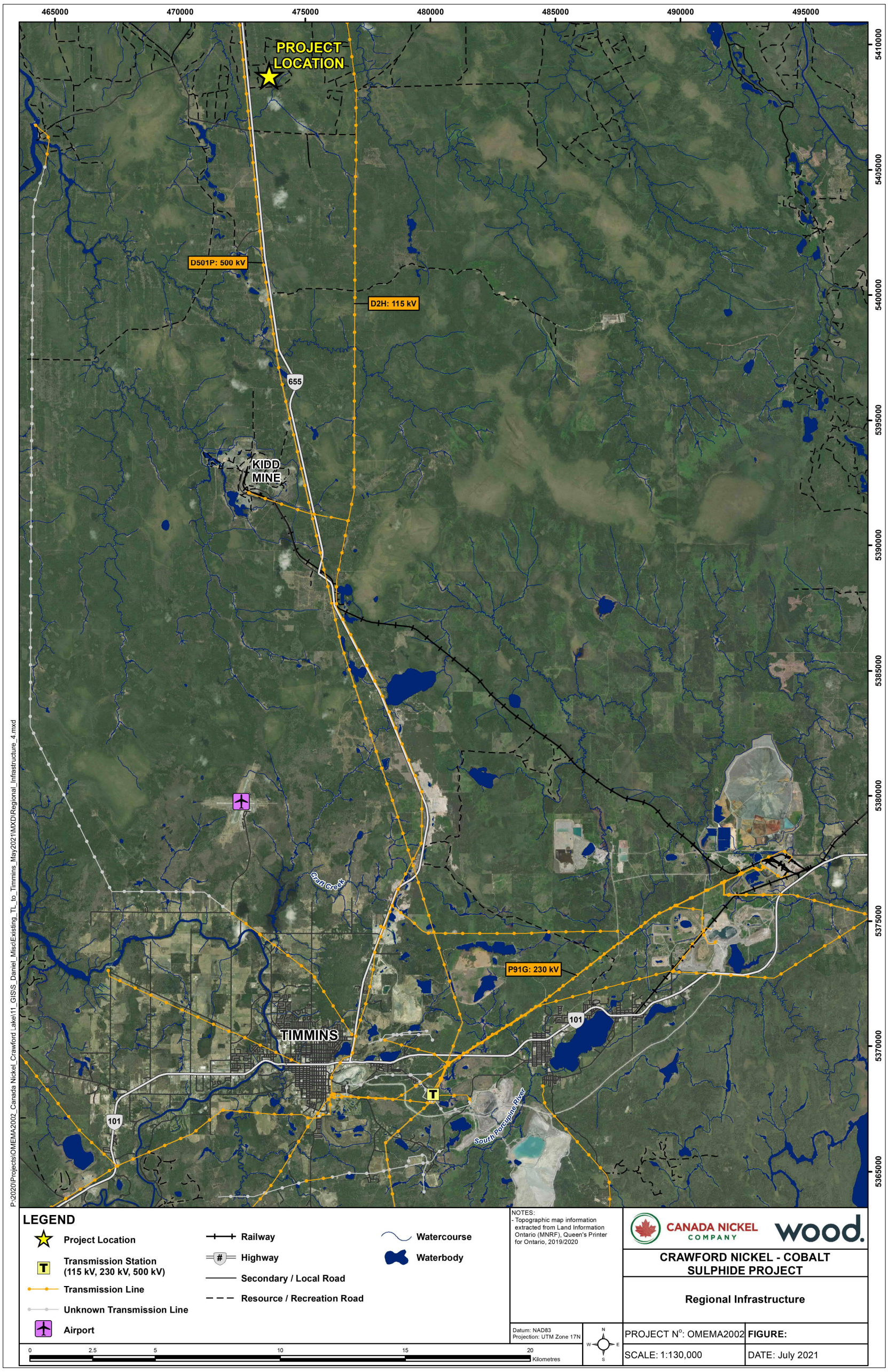
Specifically, the property comprises 140 Crown patents (freehold patented lands) in Crawford and Lucas townships that cover approximately 9389.32 ha and 173 single cell mining claims (SCMC), and 2 multi-cell mining claims (MCMC) in Crawford Township covering approximately 2043.9 ha. In this region of Ontario, one SCMC averages approximately 21.22 hectares.

The 140 Crown patents in Crawford and Lucas townships (Table 4.2) are mineral rights only (CNC does not control the surface rights), 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 (mineral and surface rights) within the boundary of the Project area that is not currently owned or optioned by CNC (Figure 4.2).

The *Ontario Mining Act* (2010) grants surface access to an unpatented mineral claim without owning the surface rights and given proper consultation with appropriate stakeholders. 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.

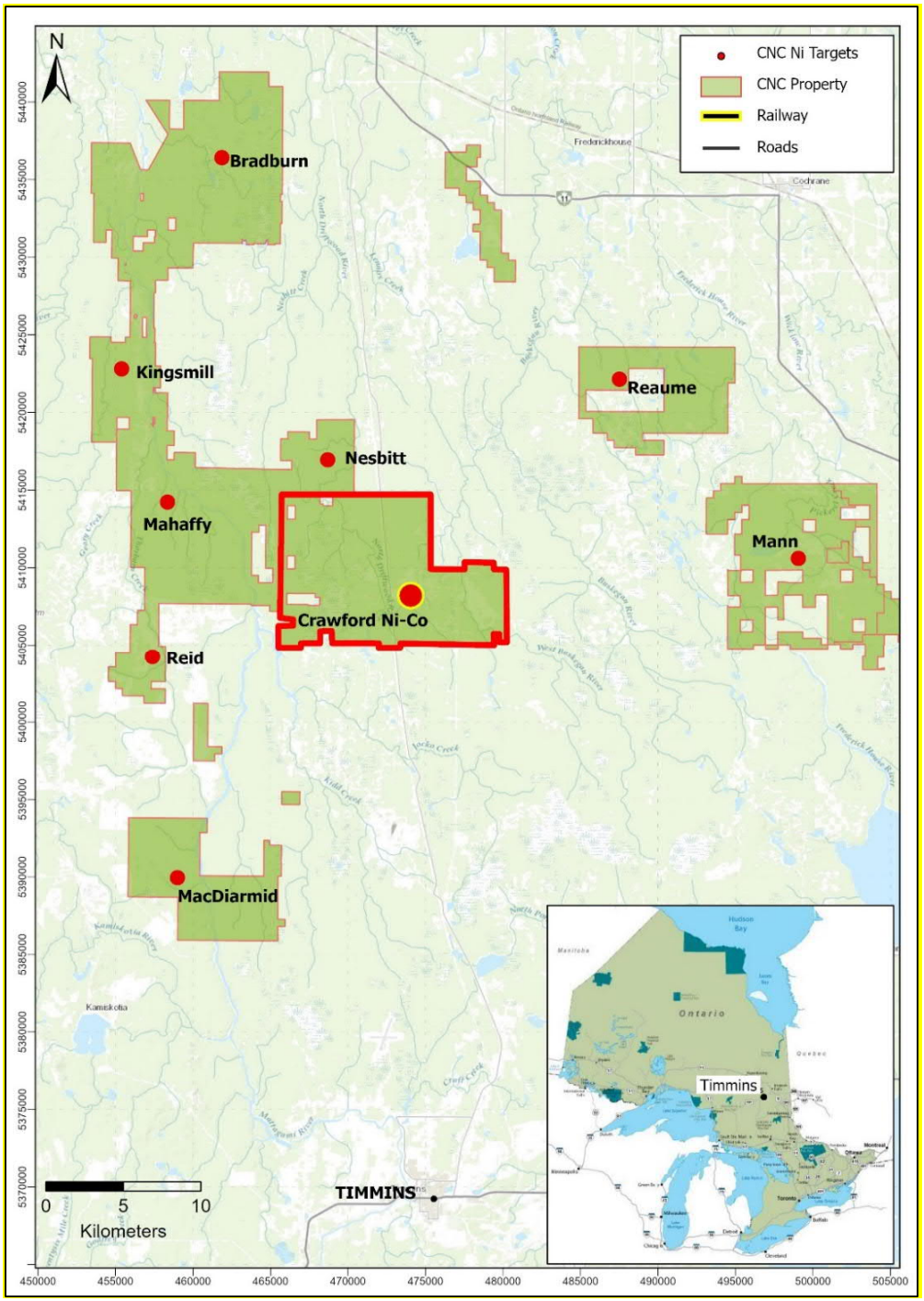
Annual holding costs for the 140 patented lands (mining tax) total approximately $37,557 and the required annual assessment work for the unpatented lands is approximately $40,200.

Figure 4.1: Township-Scale Location of the Crawford Nickel Sulphide Project (red area) in Crawford and Lucas



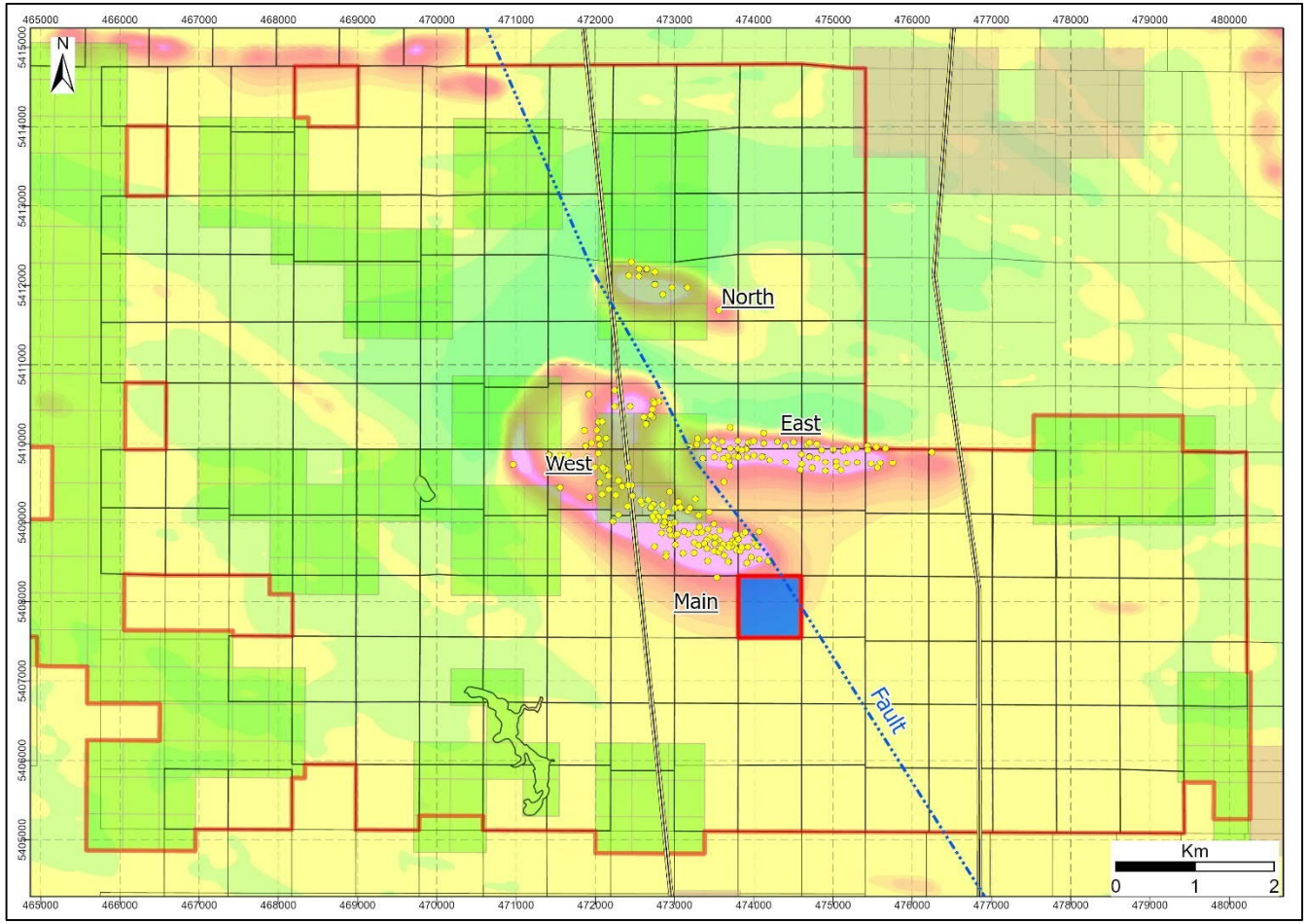
Source: Wood Environment & Infrastructure Solutions Canada Limited, 2021.

Figure 4.2: Township-Scale Location of the Crawford Nickel Sulphide Project (bold red outline) in Crawford and Lucas Townships, Timmins-Cochrane Area of Ontario, Canada.



Note: Locations of all of the properties controlled by CNC are shown. The City of Timmins is located in the lower part of the map and the lower right inset map shows the general location of Timmins and the Project in Ontario. Source: CNC, 2022.

Figure 4.3: Land Tenure for the Crawford Property Superimposed on Total Magnetic Intensity (TMI)



Note: With the boundary of the Crawford Nickel Sulphide Project (red outline) covering patented lands, unpatented mining claims, and Legacy Mining Claims (see Table 4.1 and Table 4.2). The blue area inside of the Project boundary is a patent held by a third party. The Main, East, West, and North zones of the Crawford Ultramafic Complex, the trace (dashed blue) of the main northwest trending regional fault and drill hole collar locations (yellow dots) from 2018 to 2022 diamond drilling are also shown. Geophysics source: MegaTEM, 2002. Source: CNC, 2022.

Patented lands have an annual due date of approximately March 30 for payment of the mining land tax and related holding costs (payment due 60 days from invoicing which is generally the end of January).

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 (see CNC news release dated March 4, 2020), and a 2% NSR on the patented lands *(see* Noble new release dated December 3, 2019, and December 19, 2019).

On the basis of the information provided by CNC and from what is available in the public domain, the Authors confirm that all of the unpatented and patented mining lands which comprise the Crawford Nickel Sulphide Project are in good standing.

Table 4.1: Unpatented Mining Claims (SCMCs) in Crawford and Lucas Townships, Ontario

Legacy Claim	Tenure ID	Type	Anniversary	Holder (%)	Work Req/Year	Description	Area (ha)
4252734	269339	SCMC	2022-10-04	CNC (100%)	$400	S 1/2 Lot 4 Con 1	61.2
	269338	SCMC	2022-10-04	CNC (100%)	$200		
	250662	SCMC	2022-10-04	CNC (100%)	$200		
	225503	SCMC	2022-10-04	CNC (100%)	$200		
	332284	SCMC	2022-10-04	CNC (100%)	$200		
	332283	SCMC	2022-10-04	CNC (100%)	$200		
	316508	SCMC	2022-10-04	CNC (100%)	$200		
	195379	SCMC	2022-10-04	CNC (100%)	$200		
	130662	SCMC	2022-10-04	CNC (100%)	$200		
4259542	333029	SCMC	2022-10-04	CNC (100%)	$200	S 1/2 Lot 4 Con 5	58.9
	271941	SCMC	2022-10-04	CNC (100%)	$200		
	205923	SCMC	2022-10-04	CNC (100%)	$200		
	205922	SCMC	2022-10-04	CNC (100%)	$400		
	167982	SCMC	2022-10-04	CNC (100%)	$200		
	111361	SCMC	2022-10-04	CNC (100%)	$200		
4263066	334714	SCMC	2022-10-05	CNC (100%)	$200	N 1/2 Lot 4 Con 3	67.7
	325300	SCMC	2022-10-05	CNC (100%)	$200		
	312574	SCMC	2022-10-05	CNC (100%)	$200		
	305769	SCMC	2022-10-05	CNC (100%)	$200		
	256605	SCMC	2022-10-05	CNC (100%)	$200		
	256604	SCMC	2022-10-05	CNC (100%)	$200		
	222029	SCMC	2022-10-05	CNC (100%)	$200		
	171996	SCMC	2022-10-05	CNC (100%)	$200		
	171995	SCMC	2022-10-05	CNC (100%)	$400		
4267380	316442	SCMC	2023-09-29	CNC (100%)	$200	N 1/2 Lot 4 Con 5	99.9
	309735	SCMC	2022-10-04	CNC (100%)	$200		
	309734	SCMC	2023-09-29	CNC (100%)	$200		
	309733	SCMC	2023-09-29	CNC (100%)	$400		
	250456	SCMC	2022-10-04	CNC (100%)	$400		
	242401	SCMC	2022-10-04	CNC (100%)	$200		
	213242	SCMC	2023-09-29	CNC (100%)	$400		
	193796	SCMC	2023-09-29	CNC (100%)	$200		
	147108	SCMC	2023-09-29	CNC (100%)	$200		
	130535	SCMC	2023-09-29	CNC (100%)	$200		
4267381	327922	SCMC	2022-10-04	CNC (100%)	$200	S 1/2 Lot 6 Con 6	62.2

Legacy Claim	Tenure ID	Type	Anniversary	Holder (%)	Work Req/Year	Description	Area (ha)
	307890	SCMC	2022-10-04	CNC (100%)	$200		
	230779	SCMC	2022-10-04	CNC (100%)	$200		
	224133	SCMC	2022-10-04	CNC (100%)	$200		
	224132	SCMC	2022-10-04	CNC (100%)	$200		
	164725	SCMC	2022-10-04	CNC (100%)	$200		
	139910	SCMC	2022-10-04	CNC (100%)	$400		
	139909	SCMC	2022-10-04	CNC (100%)	$200		
	113475	SCMC	2022-10-04	CNC (100%)	$200		
4267382	337123	SCMC	2022-10-04	CNC (100%)	$200	S 1/2 Lot 6 Con 4	63.0
	212249	SCMC	2022-10-04	CNC (100%)	$200		
	129457	SCMC	2022-10-04	CNC (100%)	$200		
	129456	SCMC	2022-10-04	CNC (100%)	$200		
	109669	SCMC	2022-10-04	CNC (100%)	$200		
	109668	SCMC	2022-10-04	CNC (100%)	$200		
4267383	332101	SCMC	2022-10-05	CNC (100%)	$200	S 1/2 Lot 6 Con 3	60.8
	328599	SCMC	2022-10-05	CNC (100%)	$200		
	315319	SCMC	2022-10-05	CNC (100%)	$200		
	308592	SCMC	2022-10-05	CNC (100%)	$200		
	260736	SCMC	2022-10-05	CNC (100%)	$200		
	249993	SCMC	2022-10-05	CNC (100%)	$200		
	249992	SCMC	2022-10-05	CNC (100%)	$200		
	212747	SCMC	2022-10-05	CNC (100%)	$400		
	160092	SCMC	2022-10-05	CNC (100%)	$200		
4267384	331720	SCMC	2022-10-05	CNC (100%)	$200	S 1/2 Lot 8 Con 5	60.9
	331719	SCMC	2022-10-05	CNC (100%)	$200		
	247901	SCMC	2022-10-05	CNC (100%)	$200		
	247900	SCMC	2022-10-05	CNC (100%)	$200		
	203341	SCMC	2022-10-05	CNC (100%)	$200		
	164004	SCMC	2022-10-05	CNC (100%)	$200		
	164003	SCMC	2022-10-05	CNC (100%)	$400		
	158708	SCMC	2022-10-05	CNC (100%)	$200		
4267385	313693	SCMC	2022-10-05	CNC (100%)	$200	N 1/2 Lot 8 Con 3	67.8
	275856	SCMC	2022-10-05	CNC (100%)	$200		
	275855	SCMC	2022-10-05	CNC (100%)	$200		
	203181	SCMC	2022-10-05	CNC (100%)	$200		
	158482	SCMC	2022-10-05	CNC (100%)	$200		
4267386	321076	SCMC	2022-10-05	CNC (100%)	$200	S 1/2 Lot 9 Con 3	61.1
	254717	SCMC	2022-10-05	CNC (100%)	$200		
	254716	SCMC	2022-10-05	CNC (100%)	$200		
	254715	SCMC	2022-10-05	CNC (100%)	$200		
	182533	SCMC	2022-10-05	CNC (100%)	$200		
	169076	SCMC	2022-10-05	CNC (100%)	$200		
	169075	SCMC	2022-10-05	CNC (100%)	$200		
	169074	SCMC	2022-10-05	CNC (100%)	$200		
4267387	332656	SCMC	2022-10-05	CNC (100%)	$200	N 1/2 Lot 10 Con 3	67.1
	318442	SCMC	2022-10-05	CNC (100%)	$200		
	271200	SCMC	2022-10-05	CNC (100%)	$200		
	252439	SCMC	2022-10-05	CNC (100%)	$200		

Legacy Claim	Tenure ID	Type	Anniversary	Holder (%)	Work Req/Year	Description	Area (ha)
	167269	SCMC	2022-10-05	CNC (100%)	$200		
	136045	SCMC	2022-10-05	CNC (100%)	$200		
4267388	164776	SCMC	2022-10-05	CNC (100%)	$200	S 1/2 Lot 12 Con 2 & S 1/2 Lot 11 Con 2	126.1
	194046	SCMC	2022-10-05	CNC (100%)	$200		
	307924	SCMC	2022-10-05	CNC (100%)	$200		
	260101	SCMC	2022-10-05	CNC (100%)	$200		
	248653	SCMC	2022-10-05	CNC (100%)	$200		
	224160	SCMC	2022-10-05	CNC (100%)	$200		
	164775	SCMC	2022-10-05	CNC (100%)	$200		
4269603	331930	SCMC	2022-11-03	CNC (100%)	$200	S 1/2 Lot 10 Con 6	64.1
	314635	SCMC	2022-11-03	CNC (100%)	$200		
	307898	SCMC	2022-11-03	CNC (100%)	$200		
	307897	SCMC	2022-11-03	CNC (100%)	$400		
	248624	SCMC	2022-11-03	CNC (100%)	$200		
	230783	SCMC	2022-11-03	CNC (100%)	$200		
	194029	SCMC	2022-11-03	CNC (100%)	$200		
	159418	SCMC	2022-11-03	CNC (100%)	$200		
4269604	321103	SCMC	2022-11-03	CNC (100%)	$200	N 1/2 Lot 9 Con 5	67.0
	321102	SCMC	2022-11-03	CNC (100%)	$200		
	291950	SCMC	2022-10-05	CNC (100%)	$200		
	254736	SCMC	2022-11-03	CNC (100%)	$200		
	105177	SCMC	2022-11-03	CNC (100%)	$200		
	105176	SCMC	2022-11-03	CNC (100%)	$200		
4269605	328041	SCMC	2022-10-05	CNC (100%)	$200	S 1/2 Lot 11 Con 1 & Lot 10 Con 1	137.5
	328040	SCMC	2022-10-05	CNC (100%)	$200		
	308512	SCMC	2022-10-05	CNC (100%)	$200		
	260679	SCMC	2022-10-05	CNC (100%)	$200		
	249915	SCMC	2022-10-05	CNC (100%)	$200		
	231389	SCMC	2022-10-05	CNC (100%)	$200		
	230810	SCMC	2022-10-05	CNC (100%)	$200		
	165360	SCMC	2022-10-05	CNC (100%)	$200		
	145978	SCMC	2022-10-05	CNC (100%)	$200		
	136018	SCMC	2022-10-05	CNC (100%)	$200		
	113580	SCMC	2022-10-05	CNC (100%)	$200		
4269606	169058	SCMC	2022-10-05	CNC (100%)	$200	S 1/2 Lot 12 Con 1	61.0
	152437	SCMC	2022-10-05	CNC (100%)	$200		
	331109	SCMC	2022-10-05	CNC (100%)	$200		
	254689	SCMC	2022-10-05	CNC (100%)	$200		
	304588	SCMC	2022-10-05	CNC (100%)	$200		
	283834	SCMC	2022-10-05	CNC (100%)	$200		
	217852	SCMC	2022-10-05	CNC (100%)	$400		
	124556	SCMC	2022-10-05	CNC (100%)	$200		
	590970	MCMC	2023-05-20	CNC (100%)	$800	S 1/2 Lot 4 Con 6	42.5
	585127	SCMC	2023-04-19	CNC (100%)	$400	N 1/2 Lot 7 Con 1	63.8
	585126	SCMC	2023-04-19	CNC (100%)	$400	N 1/2 Lot 7 Con 1	21.3
	585125	SCMC	2023-04-19	CNC (100%)	$400	S 1/2 Lot 7 Con 1	21.3
	585124	SCMC	2023-04-19	CNC (100%)	$400	S 1/2 Lot 7 Con 1	21.3
	585122	SCMC	2023-04-19	CNC (100%)	$400	S 1/2 Lot 6 Con 1	21.3

Legacy Claim	Tenure ID	Type	Anniversary	Holder (%)	Work Req/Year	Description	Area (ha)
	585121	SCMC	2023-04-19	CNC (100%)	$400	N 1/2 Lot 6 Con 1	21.3
	585120	SCMC	2023-04-19	CNC (100%)	$400	N 1/2 Lot 6 Con 1	21.3
	561936	MCMC	2022-10-15	CNC (100%)	$1,200	N 1/2 Lot 6 Con 1	63.8
	535134	SCMC	2022-11-17	CNC (100%)	$200	S 1/2 Lot 12 Con 3	21.3
	535135	SCMC	2022-11-17	CNC (100%)	$200	N 1/2 Lot 12 Con 3	21.3
	585128	SCMC	2023-04-19	CNC (100%)	$200	S 1/2 Lot 7 Con 1	21.3
	585129	SCMC	2023-04-19	CNC (100%)	$200	S 1/2 Lot 7 Con 1	21.3
	535143	SCMC	2022-11-17	CNC (100%)	$200	N 1/2 Lot 12 Con 2	21.3
	535150	SCMC	2022-11-17	CNC (100%)	$200	N 1/2 Lot 12 Con 2	21.3
	535151	SCMC	2022-11-17	CNC (100%)	$200	N 1/2 Lot 12 Con 3	21.3
	535156	SCMC	2022-11-17	CNC (100%)	$200	S 1/2 Lot 12 Con 4	21.3
	527470	SCMC	2023-08-17	CNC (100%)	$200	S 1/2 Lot 12 Con 6	21.3
	527443	SCMC	2023-08-17	CNC (100%)	$200	N 1/2 Lot 12 Con 6	21.3
	527457	SCMC	2023-08-17	CNC (100%)	$200	N 1/2 Lot 12 Con 6	21.3
	527501	SCMC	2023-08-17	CNC (100%)	$200	N 1/2 Lot 12 Con 5	21.3
	527502	SCMC	2023-08-17	CNC (100%)	$200	S 1/2 Lot 12 Con 6	21.3
	527513	SCMC	2023-08-17	CNC (100%)	$200	N 1/2 Lot 12 Con 4	21.3
	527522	SCMC	2023-08-17	CNC (100%)	$200	S 1/2 Lot 12 Con 4	21.3
	527523	SCMC	2023-08-17	CNC (100%)	$200	N 1/2 Lot 12 Con 4	21.3
	527524	SCMC	2023-08-17	CNC (100%)	$200	S 1/2 Lot 12 Con 5	21.3
	527525	SCMC	2023-08-17	CNC (100%)	$200	N 1/2 Lot 12 Con 5	21.3
4266698	232691	SCMC	2022-12-29	CNC (100%)	$200	N 1/2 Lot 8 Con 3 & N 1/2 Lot 9 Con 3	67.2
	195474	SCMC	2022-12-29	CNC (100%)	$200		
	329367	SCMC	2022-12-29	CNC (100%)	$200		
	332360	SCMC	2022-12-29	CNC (100%)	$200		
	114230	SCMC	2022-12-29	CNC (100%)	$200		
	329368	SCMC	2022-12-29	CNC (100%)	$200		
	232693	SCMC	2022-12-29	CNC (100%)	$200		
	316575	SCMC	2022-12-29	CNC (100%)	$200		
	232692	SCMC	2022-12-29	CNC (100%)	$200		
	316574	SCMC	2022-12-29	CNC (100%)	$200		
	232694	SCMC	2022-12-29	CNC (100%)	$200		
	249218	SCMC	2022-12-29	CNC (100%)	$200		
	329369	SCMC	2022-12-29	CNC (100%)	$200		
	215509	SCMC	2022-12-29	CNC (100%)	$200		
	309848	SCMC	2022-12-29	CNC (100%)	$200		
4269025	188334	SCMC	2024-05-08	CNC (100%)	$200	N 1/2 Lot 7 Con 1	66.3
	188333	SCMC	2024-05-08	CNC (100%)	$200		
	200516	SCMC	2024-05-08	CNC (100%)	$200		
	311051	SCMC	2024-05-08	CNC (100%)	$200		
	207842	SCMC	2024-05-08	CNC (100%)	$200		
	213045	SCMC	2024-05-08	CNC (100%)	$200		
	147709	SCMC	2024-05-08	CNC (100%)	$200	S 1/2 Lot 7 Con 1	21.3
	176814	SCMC	2024-05-08	CNC (100%)	$200	S 1/2 Lot 7 Con 1	21.3
				TOTALS:	**$40,200**		**2043.9**

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-49285		65320-0150(LT)	64.75	$259.00
Lucas	PAT-49287		65320-0138(LT)	64.75	$259.00
Lucas	PAT-49288		65320-0148(LT)	64.75	$259.00
Lucas	PAT-49290		65320-0136(LT)	64.75	$259.00
Lucas	PAT-49291		65320-0146(LT)	64.75	$259.00
Lucas	PAT-49292		65320-0134(LT)	64.75	$259.00
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-49312		65320-0184(LT)	64.34	$257.38
Lucas	PAT-49314	610SND	65320-0164(LT)	64.34	$257.38
Lucas	PAT-49318	616SND	65320-0162(LT)	64.34	$257.38
Lucas	PAT-49319	544SND	65320-0180(LT)	64.55	$258.19
Lucas	PAT-49529	648SND	65320-0178(LT)	64.75	$259.00
Lucas	PAT-49530	593SND	65320-0158(LT)	64.75	$259.00
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-49538	1322SND	65320-0202(LT)	65.15	$260.62
Lucas	PAT-49539	4627SWS	65320-0210(LT)	65.15	$260.62
Lucas	PAT-49540	1320SND	65320-0212(LT)	65.15	$260.62
Lucas	PAT-49541	592SND	65320-0214(LT)	65.15	$260.62
Lucas	PAT-49542	591SND	65320-0200(LT)	64.95	$259.81
Lucas	PAT-49543	518SND	65320-0216(LT)	64.95	$259.81
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-49878	4500NEC	65321-0142(LT)	66.37	$265.47
Crawford	PAT-49879	4485NEC	65321-0144(LT)	66.17	$264.66
Crawford	PAT-49880	4512NEC	65321-0146(LT)	66.17	$264.66
Crawford	PAT-49881	4513NEC	65321-0148(LT)	68.19	$272.76
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

Township	Mining Right Number	Parcel	PIN	Area (ha)	Tax
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

Township	Mining Right Number	Parcel	PIN	Area (ha)	Tax
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
Lucas	PAT-50002	558SND	65320-0182(LT)	64.34	$257.38
Lucas	PAT-50003	481SND	65320-0160(LT)	64.55	$258.19
Crawford	PAT-599335		65321-0280(LT)	64.34	$257.38
			TOTALS:	**9389.32**	**$37,557.26**

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 is 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 is permitted 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-hectare claim unit. The prescribed work must be completed as no payments in lieu of work can be made. 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.

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 the 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 of 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 (LUP) from the MNR. An LUP is considered to be 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. LUPs are often obtained when the land is to be used for the purposes of an exploration camp. When an LUP is issued, the MNR retains future options for the subject lands and controls its use. LUPs 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 hectare 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 *(see* 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 (see Noble news release dated December 3, 2019) and which comprise part of the current Project.

4.5 Permitting Considerations

The Crawford Nickel Sulphide Project is expected to require completion of a federal impact assessment pursuant to the *Impact Assessment Act*, as well as a provincial environmental assessment in accordance with the *Environmental Assessment Act*. An initial project description is required to formally start the federal process, to be followed by a detailed project description from which a determination will be made by the Impact Assessment Agency as to whether a federal impact assessment is required. It is expected that these same documents will be used by the Provincial Ministries to assess provincial environmental assessment needs.

On completion of these processes, permits and approvals for construction and operation of the project can be obtained from the relevant agencies / ministries. Although the Crawford Nickel Sulphide Project is a large-scale project, there are currently no known indicators that these processes cannot be completed successfully.

4.6 Environmental Considerations

The Crawford property is undeveloped with no known environmental liabilities. The primary disturbance to date is related to exploration activities and engineering investigations.

Recognizing the importance of environmental consideration to both project design and approvals, CNC has initiated collection of environmental baseline data using experienced consultants. Early investigations were started in the early spring of 2021 and are expected to continue through 2021 and 2022. There have been no findings from the limited environmental investigations completed to date that pose a concern for project development. The site is located in the southern-most portion of the Kesagami Caribou Range; however, the presence of Woodland Caribou (a species at risk) is considered unlikely based on available monitoring data for the region.

4.7 Social License Considerations

CNC acknowledges the importance of social license to development of modern mines, and for this reason hired a director of communications and community relations in early 2020 and a vice-president of sustainability at the end of that year. Although most of the communication efforts to date focused on governments and socio-economic organizations, CNC engaged in formal discussions with three First Nations communities: Mattagami First Nation and Matachewan First Nation, both member of the Wabun Tribal Council, and the Taykwa Tagamou Nation, leading to the signature of Memorandum of Understanding with both groups. CNC also had early-stage discussions with the Métis Nation of Ontario.

As CNC moves the Project forward, the intention of the company is to fully engage with local Indigenous Nations and stakeholders in a comprehensive consultation process aimed at identifying and addressing significant challenges associated with the development of the Crawford Nickel Sulphide Project, but also seeking to underline and maximize potential social, environmental, and economic benefits of the Project.

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

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

5.1 Accessibility

The Crawford Nickel Sulphide Project is located approximately 42 km north of Timmins, Ontario. The property is highly accessible to the regional infrastructure network, and is bisected by Highway 655, a paved highway offering year-round accessibility. 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 both Timmins and South Porcupine from the south, well as the communities of Smooth Rock Falls and Cochrane, Ontario to the north on Highway 11.

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.

Freight rail access is available in Timmins and Cochrane.

5.2 Climate

The closest regional climate monitoring station to the Project site is located at the Timmins Victor M. Power station (VPA). The station has a continuous record from 1955. A detailed in Table 5.1, 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. Most precipitation occurs in the summer and fall months and the 1981 to 2010 Canadian climate normals show extreme precipitation events of 87.6 mm of rainfall over a 24-hour period for the VPA station.

Table 5.1: Mean Monthly Climate Data from Climate Normals, VPA

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Temperature (°C)	-16.8	-14.0	-7.4	1.8	9.6	14.9	17.5	16.0	11.1	4.4	-3.4	-11.9	1.8
Rainfall (mm)	3.2	1.7	14.1	30.1	62.3	83.2	90.9	81.6	83.7	68.1	30.9	8.5	558.3
Snowfall (cm)	57.8	45.9	44.8	27.2	5.0	0.2	0.0	0.0	1.0	15.1	49.0	65.2	311.3
Precipitation (mm)	51.8	41.3	54.5	56.2	67.4	83.4	90.9	81.6	84.7	82.5	75.9	64.5	834.6

The regional design storm (Timmins Storm) occurred during August 31 to September 1, 1961. At the areal center of the storm, the 6, 12, 18, 24 and 30/36 hour durations produced accumulated precipitation values of approximately 102, 156, 175, 187 and 201 mm, respectively (Environment Canada. 1961).

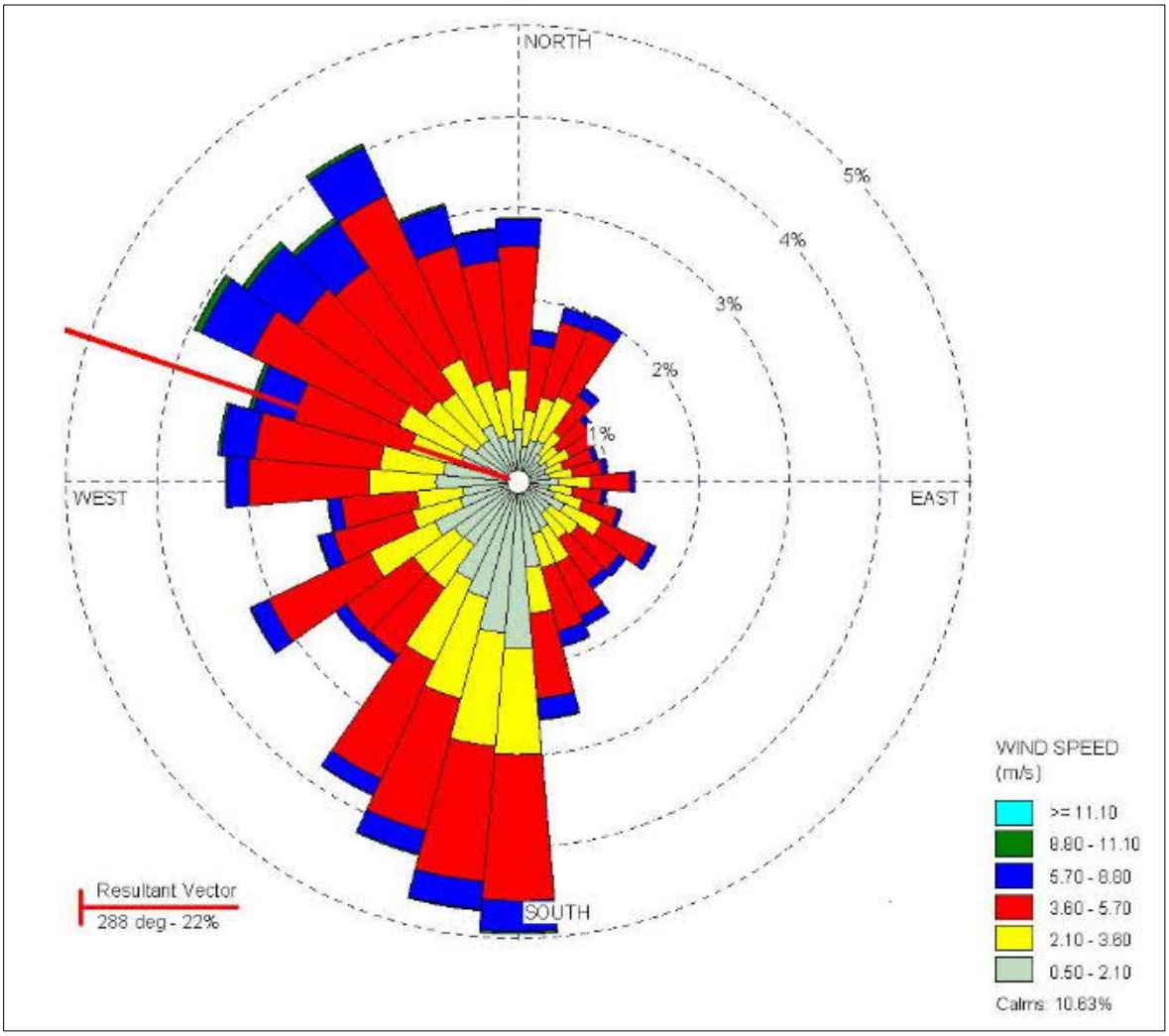
The probable maximum precipitation value for the Timmins area is estimated at approximately 450 mm (Ministry of Natural Resources. 2006).

The Hydrological Atlas of Canada (Natural Resources Canada, 1978) estimates that over most recent years assessed (1957 to 1966), the Crawford Project region experiences 400 to 500 mm/year of lake evaporation and approximately 400 mm/year of evapotranspiration.

The VPA station is the nearest Environment and Climate Change Canada station with reported wind climate normal data. The average annual wind speed was 3.28 m/s, with highest average wind speeds in February, May, and November. Maximum monthly wind gusts ranged from 85 km/h (July) to 158 km/h (June), and were generally blowing from a westerly direction.

Figure 5-1 shows the 25-year wind rose with complete data for Timmins from 1987 to 2011. The wind rose indicates a predominantly westerly wind direction and a resultant vector from the west-northwest.

Figure 5.1: Annual Wind Rose Diagram for Timmins (1987 to 2011)



Source: Generated from Environment and Climate Change Canada (2017), Climate Normals data.

5.3 Local Resources and Infrastructure

The Crawford Nickel Sulphide Project is readily accessible from the City of Timmins by Highway 655, a two-lane paved provincial highway.

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, and a 115 kV transmission located primarily east of the Project site that transmits power from the north.

Approximately 17 km south of the Project site there is a private rail spur that connects the Kidd Mine with Hoyle, Ontario.

The City of Timmins is the fourth largest city in northeastern Ontario in area and serves as a regional service and distribution center. Based on the 2016 Census, the population of Timmins was 41,788, a decrease of 3.2% since 2011. Its economy is based on natural resource extraction and is supported by industries related to the mining of metals (gold, zinc, copper, nickel and silver) and diamonds, and forestry.

The local service industry is well set up to supply the needs of the mining industry. There are well over 100 mining contractors and suppliers within Timmins 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

The area is typical of glaciated terrain of the Canadian Shield. The topography is gently rolling, averaging about 15 m in relief, and is fully vegetated. Elevations within the Project area generally between 265 and 290 mASL. The higher ground usually has a thinner veneer of glacial till over bedrock. There is only a small proportion of outcrop exposure, mostly confined to higher ground, with thicker overburden present in the low-lying areas. Low-lying ground is covered by deeper glacial till and muskeg/swamps. Overburden, which is predominantly glacial till consisting of sand, clay, loose gravel and boulders, varies from less than 10 m to as much as 85 m and has an average thickness of about 50 m.

The Project site and local area is covered by a mix of wetland, mixed deciduous and boreal forest complexes with many rivers and few small lakes throughout. 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 a number of minor waterbodies and tributaries to larger rivers. The Project site is located primarily between the North Driftwood River and the West Buskegau 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 5 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 Nickel Sulphide Project site are summarized in Table 5.2.

Table 5.2: 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/s^2). 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.

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.

5.6 Comments on Accessibility, Climate, Local Resources, Infrastructure and Physiography

Accessibility, climate, local resources, infrastructure and physiography are not expected to be limiting to the Project. Depending on the final development scenarios, in the longer term some of the infrastructure present close to the site (Highway 655 and one or more transmission lines) may 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 back to 1964. The International Nickel Company of Canada led the way in exploring the township during the 1960s with multiple drill holes testing numerous geophysical 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 drill holes reported in Lucas Township which comprise five diamond drill holes and 21 reverse circulation (RC) drill holes (ODHD, 2020). These drill holes 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 (see Section 23, Adjacent Properties).

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, plus 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 measured 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. (TSXV: AIMM) 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 hectares 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-cobalt 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 9 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 being the C-cluster (clustering for classification) algorithm 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 drill hole 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 hectares.

On August 27, 2018, Noble announced that 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 Nickel-Cobalt Sulphide Property. Twelve samples of drill core were selected from 1.5-metre 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 were 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 which will own a consolidated 100% interest in the Crawford property. A definitive agreement was entered into on November 14, 2019, with details provided in a 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 drill holes, 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 hectares 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). Section 7.2.1 and Section 7.2.2 provide a review of the airborne geophysical work completed in 2017 and 2018 and the delineation of the Crawford Ultramafic Complex.

On November 15, 2018, Spruce Ridge (and Noble) announced that it had begun a 2,000-metre program of diamond drilling on the Crawford Township Property. The target of the drilling program was a 3,000-metre long, magnetic anomaly interpreted to be a differentiated ultramafic to mafic intrusive complex, the Crawford Ultramafic Complex. On March 1, 2019, Spruce Ridge (and Noble on March 4) announced the results of its 2018 winter drilling program which totalled 1,818 metres in four drill holes.

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 Nickel Property. The phase 2 drilling program was planned to comprise approximately 4,000 metres of drilling in eight holes. Planned drill holes include infill drilling between the four drill holes put down in the winter of 2018, as well as step-out drilling to the northwest and southeast.

On October 1, 2019, Spruce Ridge announced that it had agreed to sell its interest in the Crawford property to the private company Canada Nickel Company, which was 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 (CNC).

6.2 Historical Drilling

In Crawford Township, between 1964 and 2018, at least 147 drill holes (diamond core and reverse circulation), totalling more than 14,600 metres, were completed. This drilling tested numerous geophysical anomalies, targeting base metals, gold and nickel sulphides in volcanic and mafic-ultramafic rocks (Orix Geoscience, 2019). Reported overburden intervals are drill hole 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 Crawford Ultramafic Complex, was by INCO Canada Ltd. in 1965. A total of eight drill holes 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 metres) of mafic-ultramafic rocks, largely serpentinized peridotite and/or serpentinized dunite. Overburden intervals (drill hole casing length) ranged from 34.75 to 86.87 metres.

Table 6.1: Drill Holes 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 drill holes targeting a magnetic anomaly referred to as "Anomaly 3N" (Table 6.2). The drilling intersected broad intervals (*e.g.*, 153.11 metres) of mafic-ultramafic rocks, largely serpentinized peridotite and/or serpentinized dunite. Overburden intervals (drill hole casing length) ranged from 27.43 to 97.54 metres.

Table 6.2: Drill Hole Summary with Significant Assays, McIntyre Porcupine Mines, Anomaly 3N

Year	Drill Hole	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 (Table 6.3; Spruce Ridge news release March 1, 2019). All four drill hole 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°.

Table 6.3: Summary of Drill Holes Completed by Spruce Ridge Resources in Winter 2018

Summary of Intervals Passing 0.25% Ni cut-off										
Drill Hole	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										
Drill Hole	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										
Drill Hole	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.

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

With the exception of drill hole CR18-02, which was terminated early (216 m) and in dunite, drill core assays show increasing nickel concentrations down the holes. Drill hole 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 drill holes 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.

6.2.3.2.1 Specific Gravity

Drill core from the 2018 drilling 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 drill holes completed in 2018. On the basis of 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 (see Noble news release dated December 9, 2019) and Spruce Ridge (see CNC news release dated December 10, 2019) in December 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 Anomaly 3N – Sulphide Flotation Tests, 1973

In 1973, McIntyre Porcupine Mines Ltd. drill-tested magnetic anomaly "3N" (interpreted to be serpentinized ultramafic rocks), located in the northeast part of Crawford Township (drill hole collar in N1/2 Lot 7, Concession 6). Drill hole 904-73-4 (134.5 m) required 61 m of casing before intersecting mafic-felsic volcanic rocks (61-100 m) before intersecting diorite (100-121 m) and then serpentinized peridotite (121-134.5 m) which had an overall low sulphide content; the drill hole ended in peridotite. A summary of core assays is provided in Table 6.4.

Table 6.4: Summary of Drill Core Assays from Historical Drill Hole 904-73-4

Sample	From (m)	To (m)	Int (m)	Ag (oz/t)	Cu (%)	Ni (%)
12670	97.23	98.45	1.22	0.03	0.04	-
12671	116.77	116.92	0.15	-	0.06	nil
12672	118.26	118.48	0.22	-	0.05	nil
12673	120.85	122.38	1.53	-	0.06	0.35
12674	129.24	130.00	0.76	-	0.05	0.43
12675	132.89	134.42	1.53	-	0.08	0.21

A 6.8 kg composite sample of this drill core (samples 12673, 74, and 75) was sent into the McIntyre Mine laboratory for sulphide flotation tests. The sample was described as serpentinized peridotite with about 5% sulphide mineralization and an assayed head grade that averaged 0.44% Ni and 0.04% Cu.

Three separate flotation tests were completed on the drill core from hole 904-73-4, showing an average 61.7% recovery of nickel (not optimized). Copper recoveries varied from 30% to 50%, but was inconclusive due to the very low copper head grade. The final nickel concentrate averaged 1.80% Ni and 0.24% Cu.

This ultramafic body is not located on the current property and as such the results are not necessarily indicative of results we might expect from similar rocks in the CUC.

6.3.2 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 (see 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 metre 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). The pentlandite, which dominates the nickel-bearing mineral assemblage, is considered most promising for economic nickel extraction. 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.

6.3.3 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 6.5 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 (see Noble news release dated June 11, 2019).

Table 6.5: Comparison between Peroxide Fusion and Aqua Regia Analyses for Cobalt and Nickel

Drill Hole	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%**	

All drill core samples had been 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% (see Table 6.5).

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 drill holes prior to the work done by Spruce Ridge, so in the field it is not possible to confirm the location of historical, pre-2018 drill holes. The following information comes from a review of Spruce Ridge's completed 2018 program and the ongoing 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 drill hole 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 was delivered to the side of Highway 655 by the drilling contractor as the drilling progressed. CNC personnel transported the core to the core shack from that location. Casing has been left in the completed drill holes with the casing being capped and marked with a metal flag.

6.4.2 Core Logging and Sampling

CNC used a rented core shack in Timmins (3700 Highway 101 West), a driving distance of approximately 50 km from the Project area access point. Once the core boxes arrive at the logging facility in Timmins, the boxes are laid out on the logging table in order and the lids removed. Core is stored sequentially, hole by hole, in racks for logging. Core logging consists 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 are 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 metres. The entire intercept of ultramafic rocks was sampled in each drill hole. Magnetic susceptibility was measured every metre. Relative density of core samples was calculated at a variable interval of three to six metres.

Samples are identified by inserting three identical pre-fabricated, sequentially-numbered, weather-resistant sample tags at the end of each sample interval. Once the core is logged, photographed and the samples are marked, the core boxes are transferred to the cutting room for sampling. In general, the core recovery for the diamond drill holes 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 was cut in half with a diamond saw; a separate cutting room is located adjacent to the logging area. Once the core was cut in half it was returned to the core box. A geotechnician prepared the sample tags, selecting half of the core in each interval, placing said core in a sample bag and sealing the bag with a cable tie. The boxes containing the remaining half core are 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 then the larger bag 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 drill hole 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, the 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.

6.4.3.1 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 also 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 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 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 QA/QC Data verification

6.4.4.1 Blank Material

All analyses performed on blank material are considered to be acceptable as the majority of results were reported to be 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.

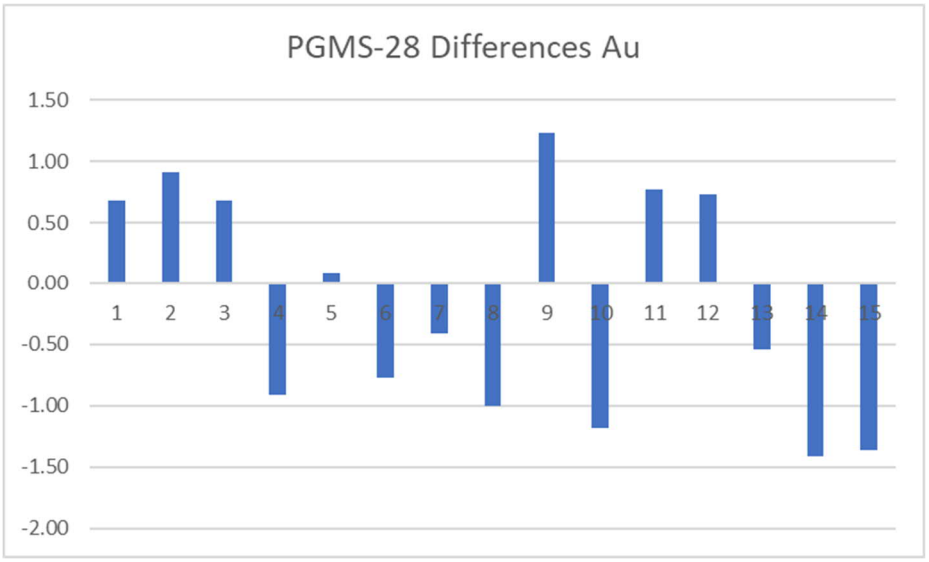
6.4.4.2 Certified Reference Material

Certified reference materials (CRM) are used by Actlabs to internally monitor the accuracy of their analyses. A number of different reference materials for different combinations of elements were used during the course of 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, we have focused 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.

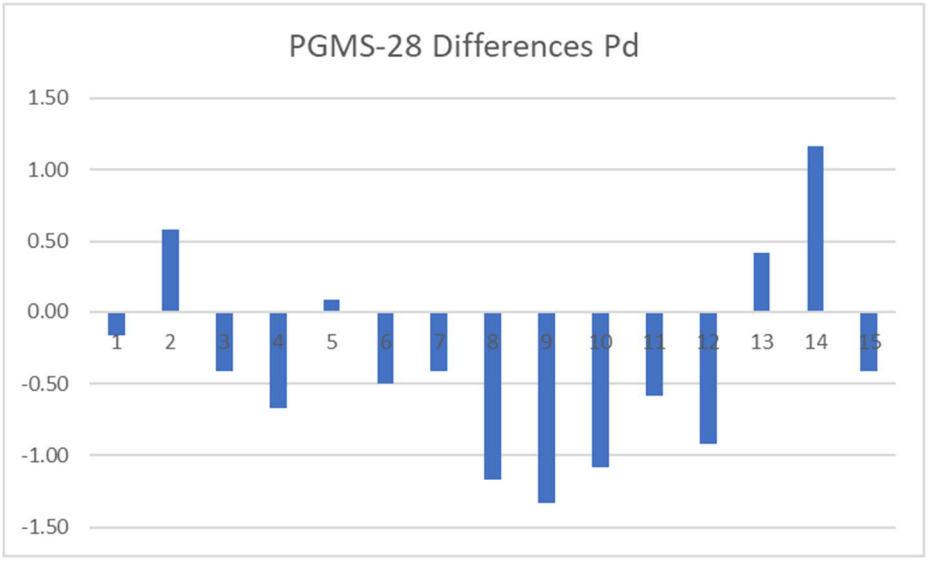
It is observed that all the certified reference material examined in detail averaged within two standard deviations of the certified concentrations over the span of the laboratory work (Figures 6.1, 6.2 and 6.3). That all analyses of certified reference material, over time, averaged close to their certified concentration gives reason that the accuracy of the analyses be considered as acceptable.

Figure 6.1: CRM CDN-PGMS-28 – Standard Deviations of Difference for Au Analysis



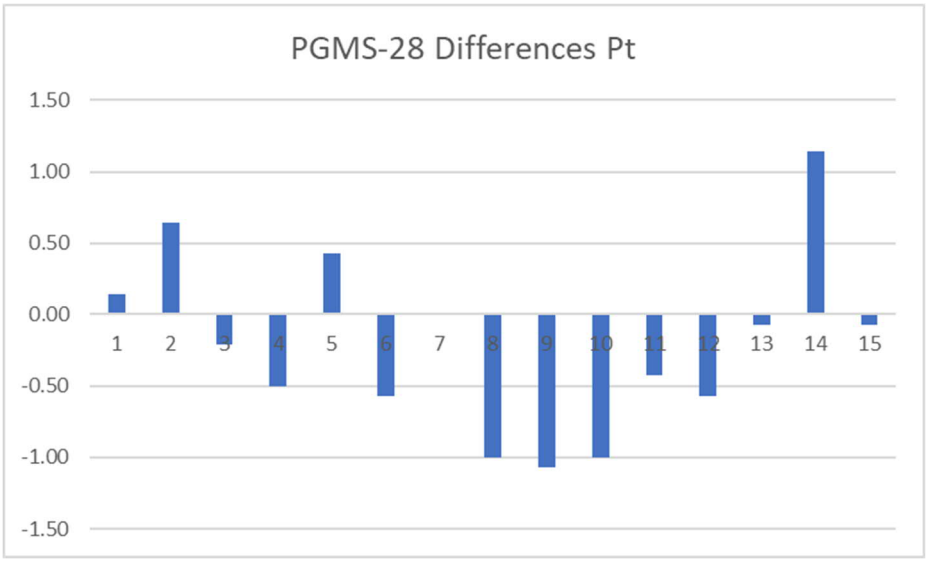
Source: Caracle Creek, 2020.

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

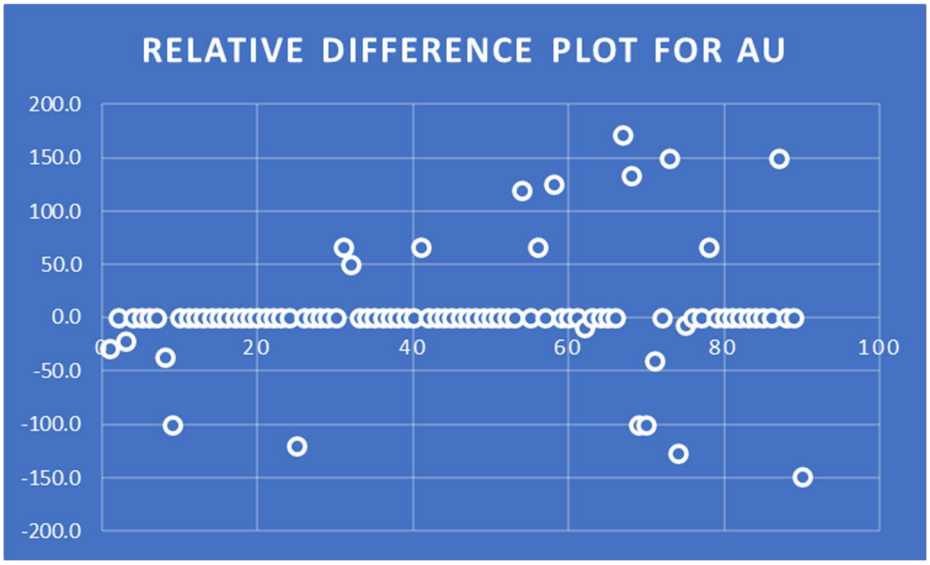


Source: Caracle Creek, 2020.

6.4.5 Duplicate Samples

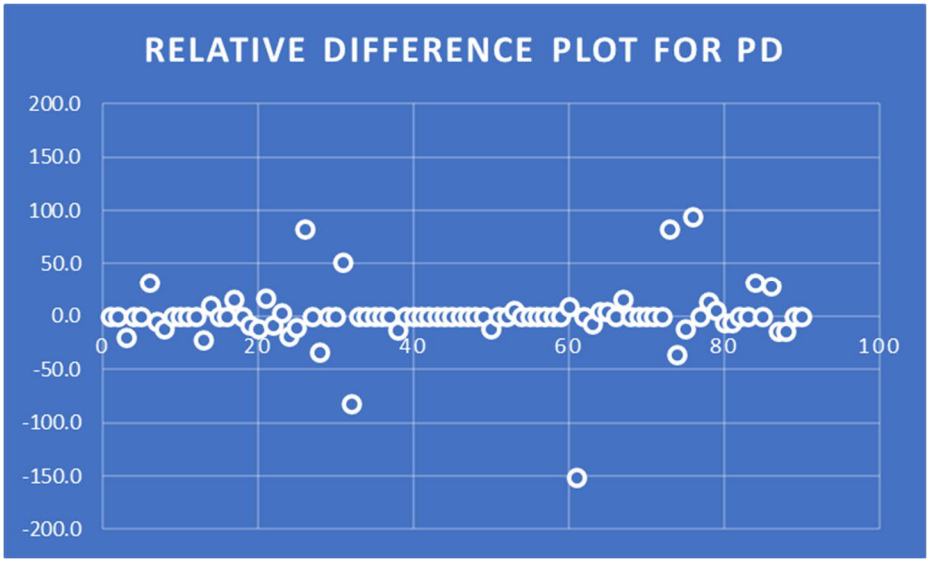
In general, the duplicate material for the precious metal analyses has indicated good reproducibility of the assays (Figures 6.4, 6.5 and 6.6). Where relative differences of over 100% are observed, sample pairs generally exhibit low absolute concentrations of the precious metals and the order of magnitude difference at those levels is not considered to be of importance.

Figure 6.4: Relative Percent Difference of Pairs of Duplicate Samples Analyzed for Au



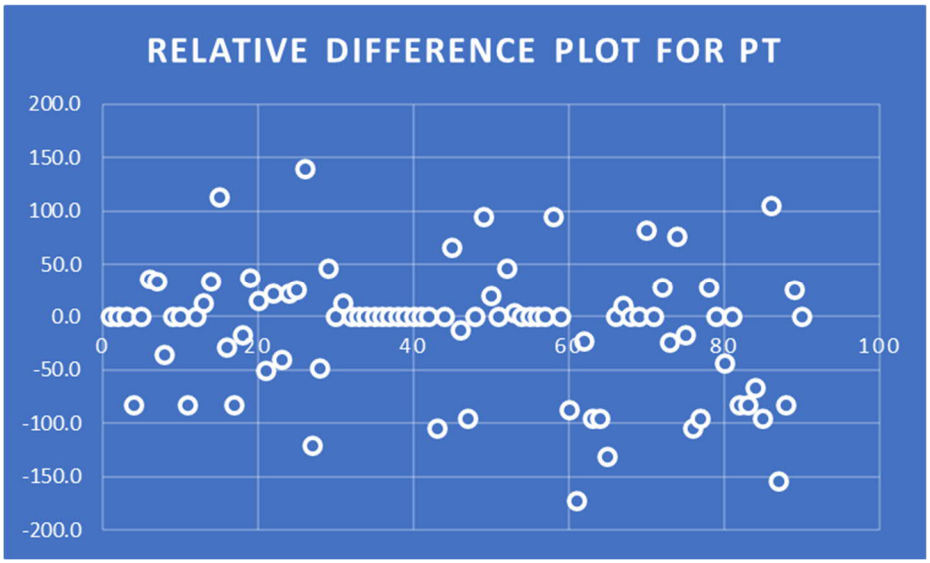
Source: Caracle Creek, 2020.

Figure 6.5: Relative Percent Difference of Pairs of Duplicate Samples Analyzed for Pd



Source: Caracle Creek, 2020.

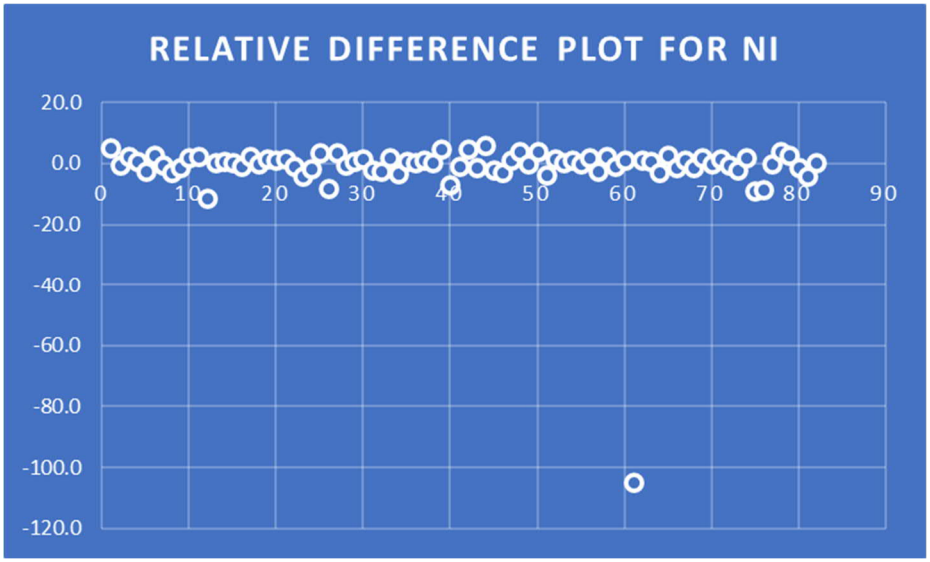
Figure 6.6: Relative Percent Difference of Pairs of Duplicate Samples Analyzed for Pt



Source: Caracle Creek, 2020.

The relative differences for Co and Ni were under 20% with the exception of one sample, 701330, where the relative difference between the pair of Ni analyses was over 100% (Figure 6.7). Again, 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 Nickel Sulphide 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 of approximately 700 km from the southeast to northwest and 350 km from north to south and comprising 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 of incredible importance as 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. At the same time this also creates an opportunity 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 for the most part 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).

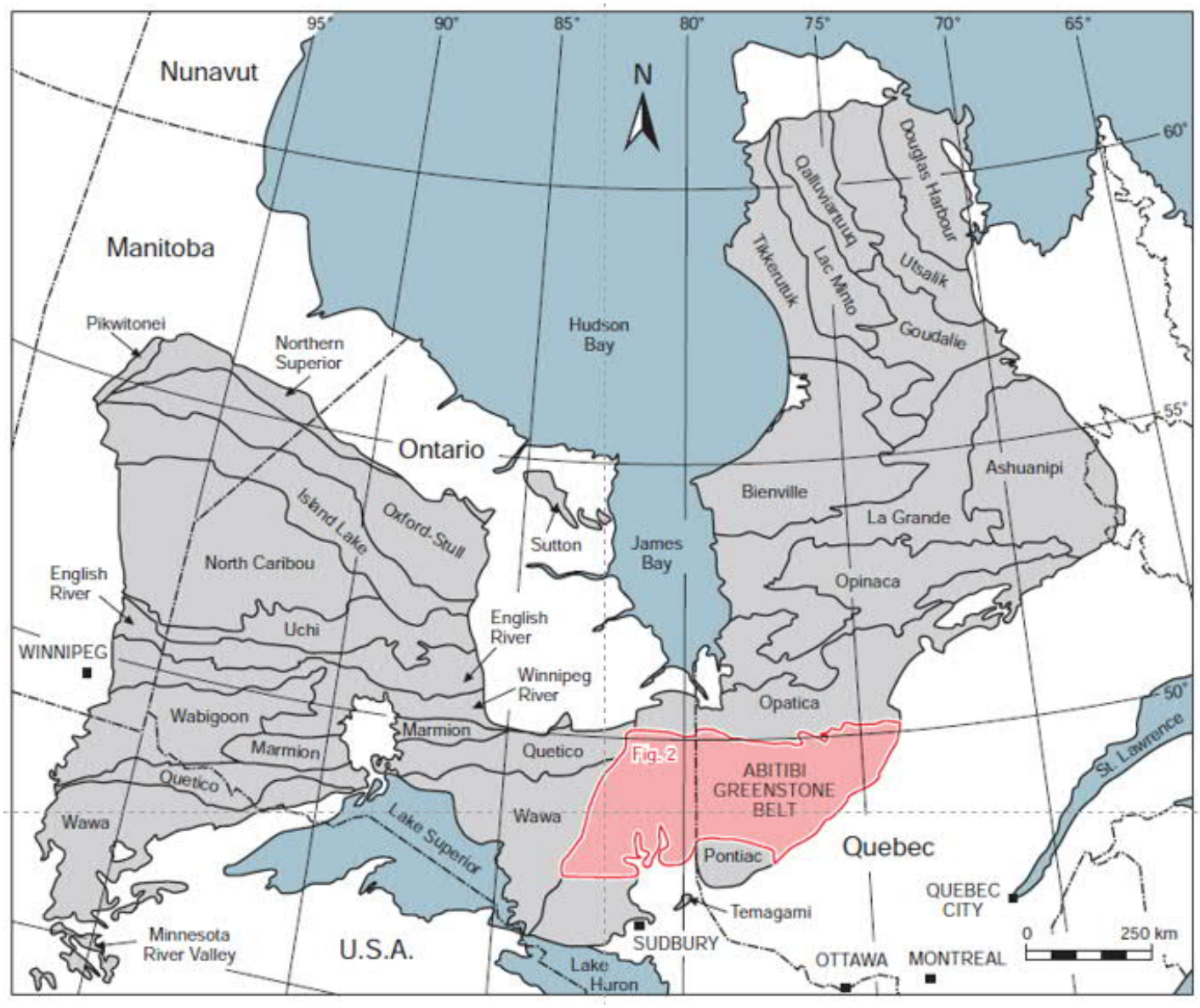
Proterozoic dikes of the Matachewan Dyke Swarm and the Abitibi Dyke Swarm intrude all of the rock in the region. Matachewan dikes generally trend north-northwest while the younger Abitibi Dyke Swarm trends northeast.

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 the 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):

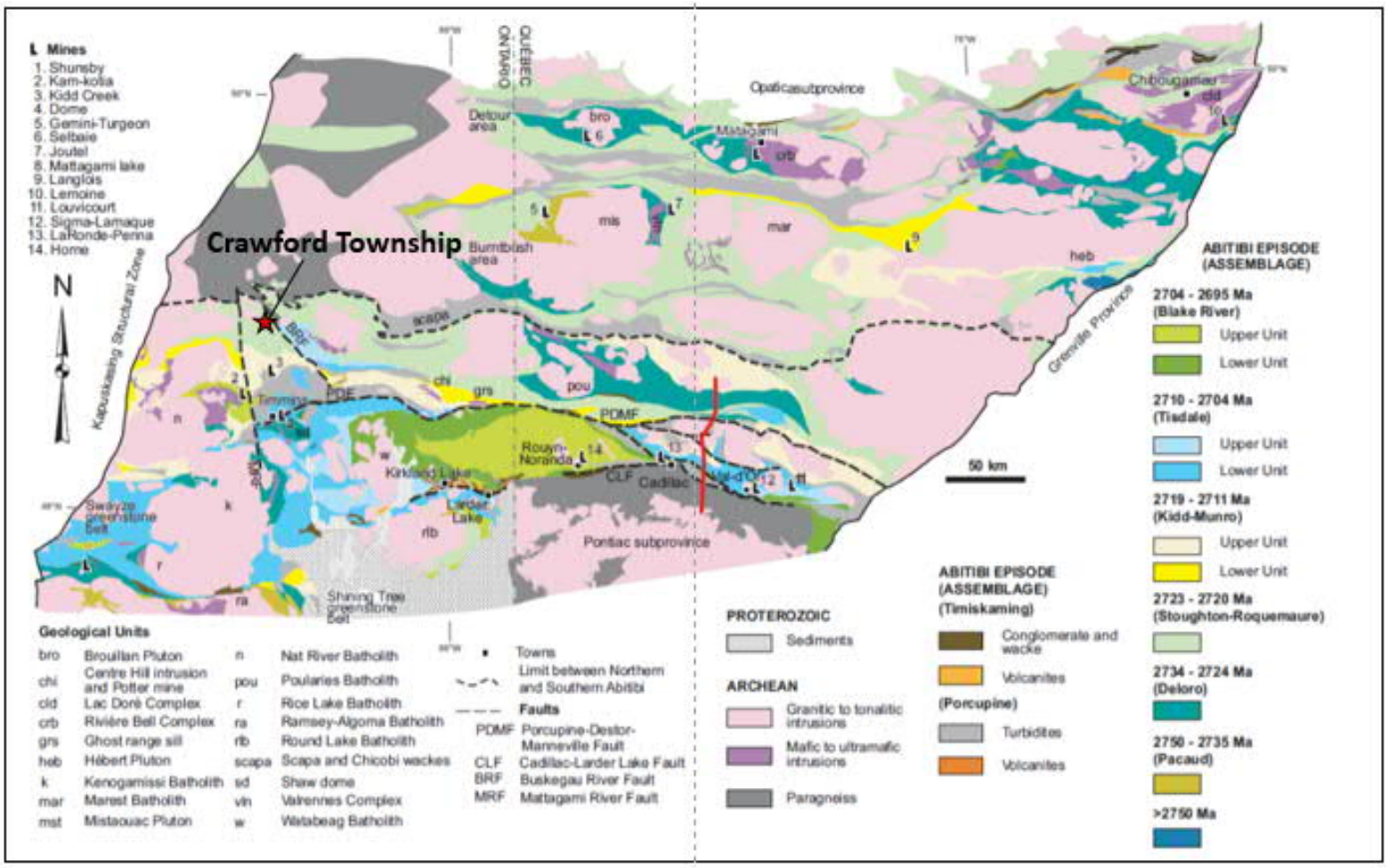
- Pacaud Assemblage (2750-2735 Ma).
- Stoughton-Roquemaure Assemblage (2723-2720 Ma).
- Kidd-Munro Assemblage (2719-2711 Ma).
- Tisdale Assemblage (2710-2704 Ma).

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 and Figure 7.3). 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 sufficient work to verify the information, and mineralization hosted on adjacent or nearby properties is not necessarily indicative of mineralization hosted on CNC's 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 volcanogenic massive sulphide (VMS) deposit 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

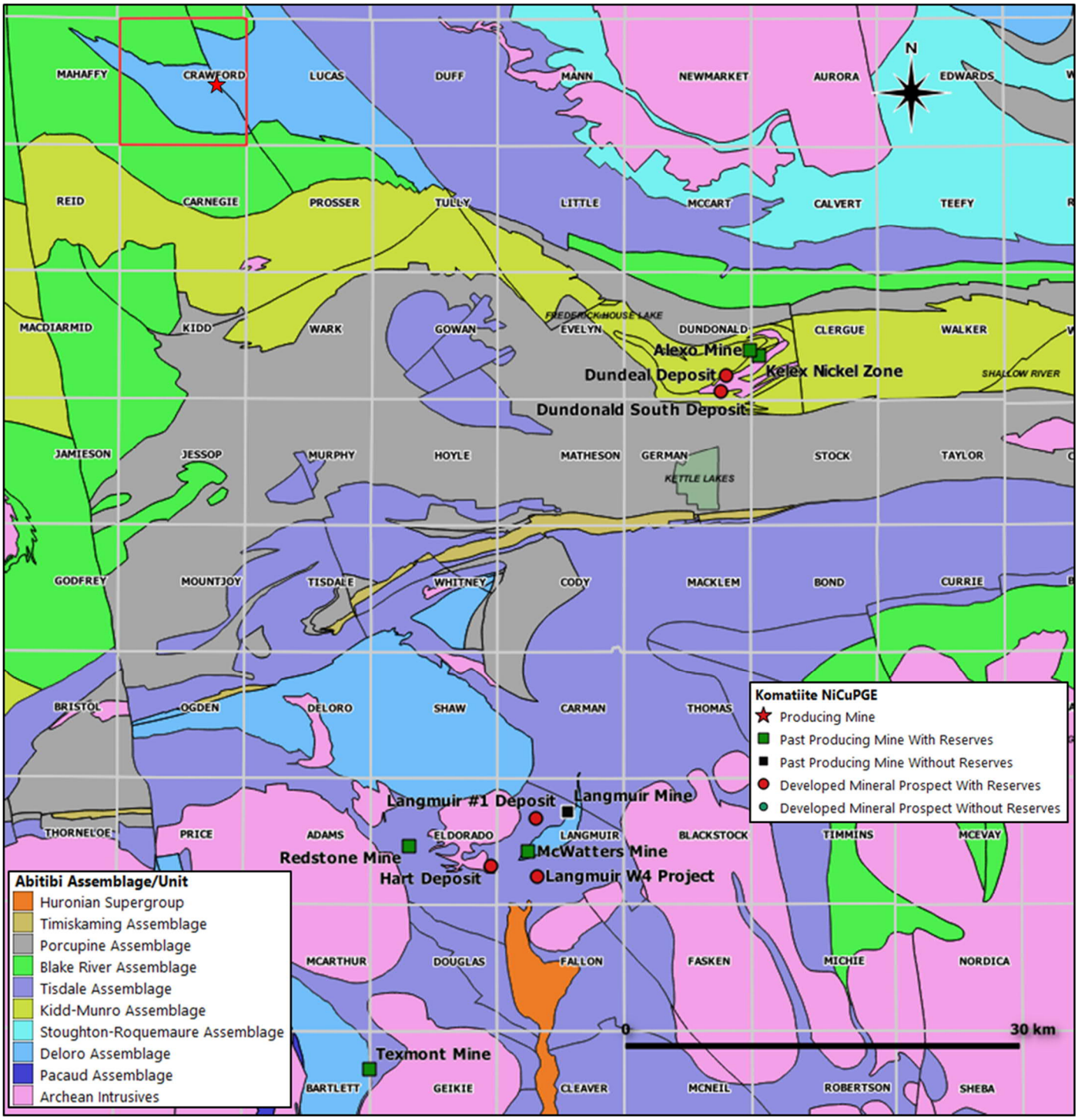
The Greenstone Architecture Project (2003-2005) and Discover Abitibi Initiative (2001-2012), led by the Ontario Geological Survey (OGS), resulted in reclassification of the lithological assemblages in the southern AGB (Ontario portion) by using detailed U/Pb geochronology, and updated geological and 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 VMS exploration, especially for gold-rich VMS deposits (Ross et al., 2009).

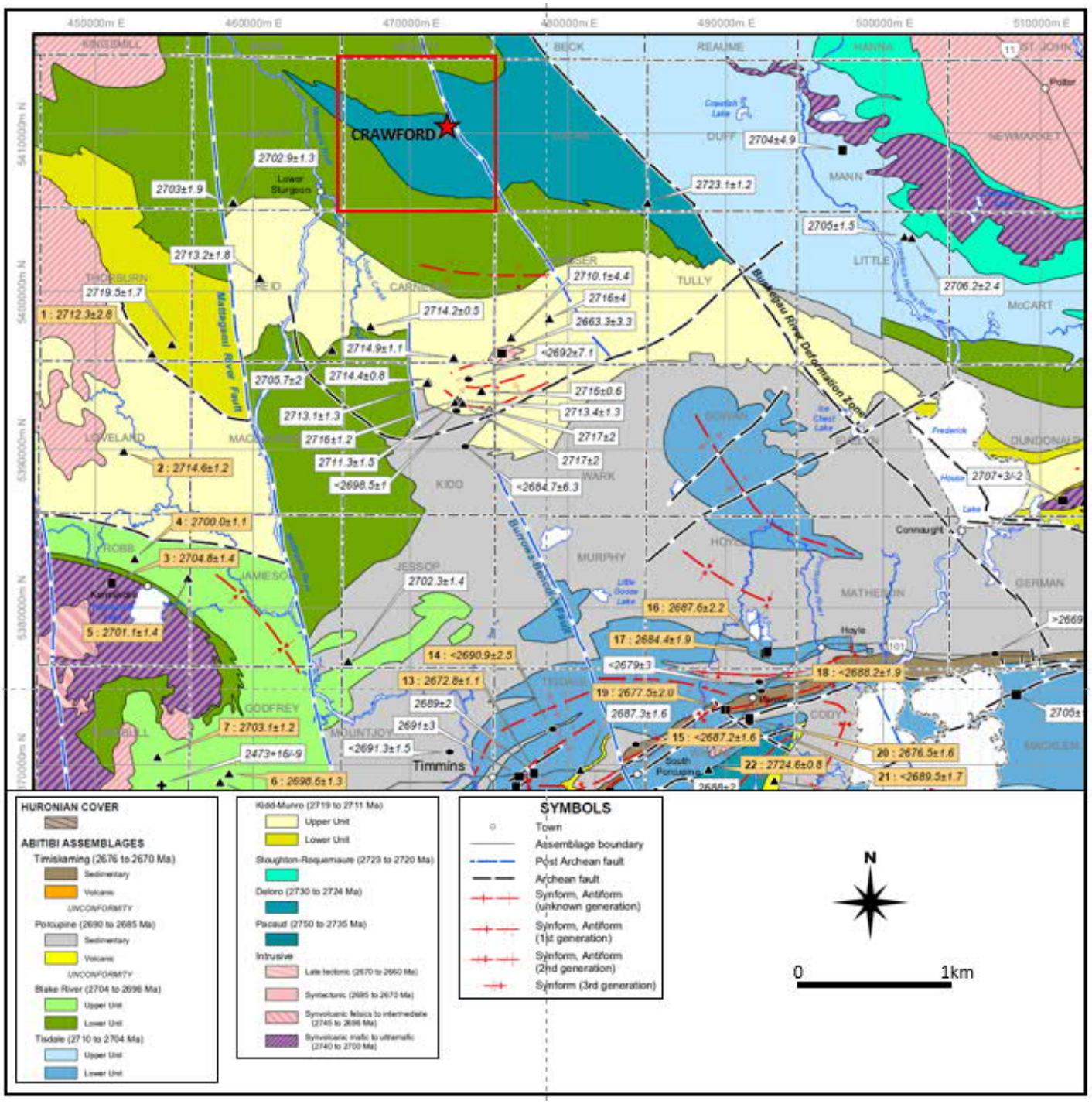
The rocks have undergone greenschist facies metamorphism with widespread carbonate, chlorite and sericite alteration in volcanic rocks and serpentinization in ultramafic rocks (*i.e.*, dunite, peridotite).

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 Crawford Ultramafic Complex (red star). Source: 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



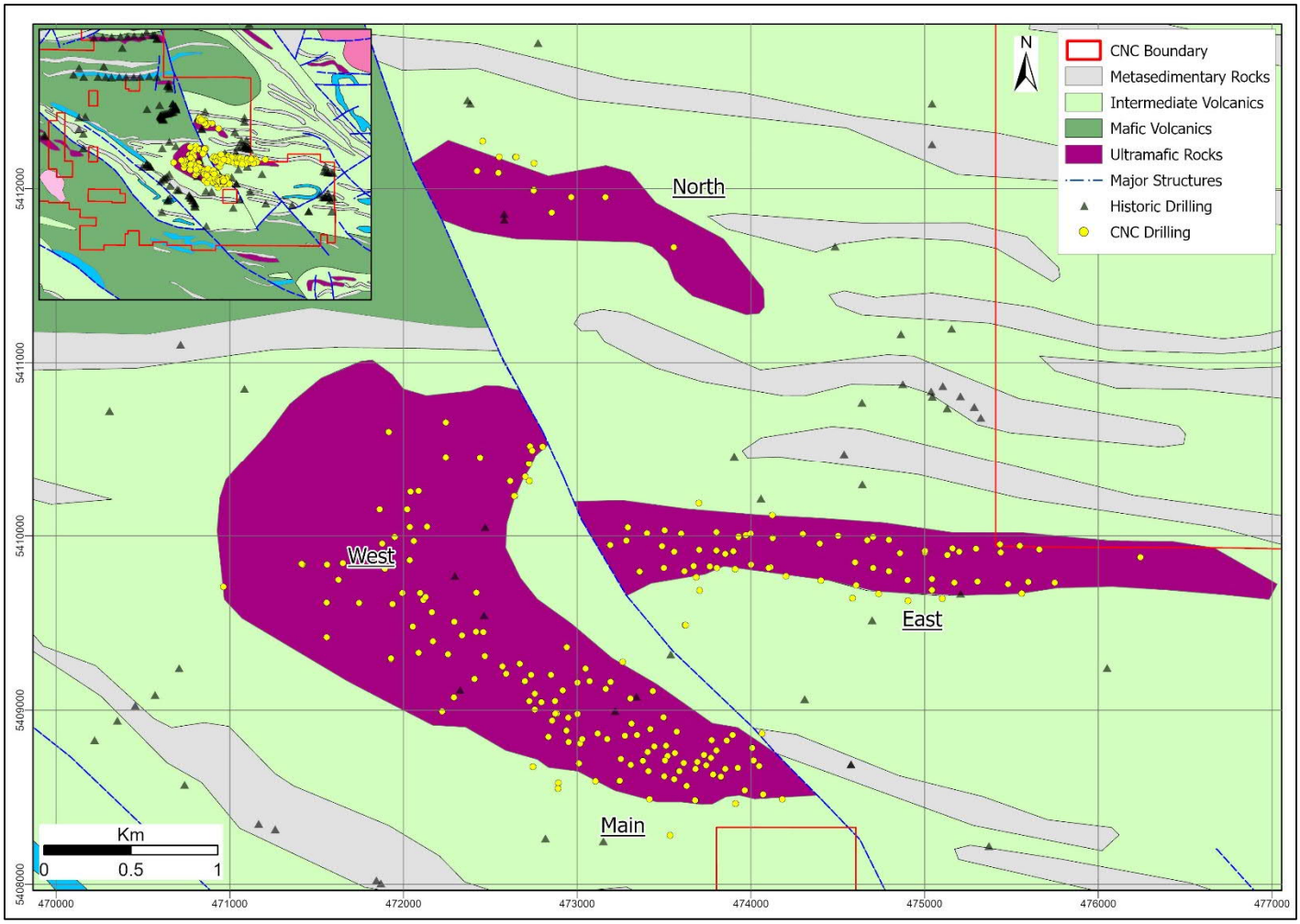
Note: Location of the property in Crawford Township (red square) and the approximate location of the Crawford Ultramafic Complex (red star). Also shown are the age dates from U/Pb geochronology samples taken from various Abitibi assemblages. Source: Ayer et al., 2005.

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 recent historical diamond drill holes (Spruce Ridge Resources, 2018) and the current extensive drilling program by Canada Nickel. 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 f historical and current drill hole collars.



Source: CNC, 2022

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).

8 DEPOSIT TYPES

8.1 Deposit Model

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: 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).

2. 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).

Sulphide nickel-copper-cobalt-PGE mineralization in the Crawford Ultramafic Complex is interpreted as most similar to Mt. Keith-style.

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). Disseminated nickel sulphide mineralization strikes for 2 km, is 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).

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.

8.2.1 Komatiite Volcanic Facies

The five major volcanic facies that are common constituents of komatiitic flow fields (Barnes et al., 2004) are listed below. For more information refer to Table 8-1.

- thin differentiated flows (TDF)
- compound sheet flows with internal pathways (CSF)
- dunitic compound sheet flows (DCSF)
- dunitic sheet flows (DSF)
- layered lava lakes or sills (LLLS)

DCFS and CSF facies represent high-flow magma pathways characterized by olivine cumulates and can be identified by their elevated Ni/Ti and Ni/Cr ratios and low Cr contents (Barnes et al., 2004). Although only DCFS and CSF facies are known to host economic nickel sulphide mineralization (Burley and Barnes, 2019), it does not discount the prospectivity of the other facies, particularly the thick sheets and/or sills associated with the DSF and LLLS types.

The geophysical expression and diamond drilling to date suggests that the CUC is at least 8 km in cumulative strike length and averages about 2 km in width. Both the Main Zone and the East Zone contain several hundred metres (thickness) of dunite-peridotite cumulates and include fractionated upper (northern) zones dominated by pyroxenite and gabbro (and their associated PGE reefs). The CUC is interpreted to be most similar to DCSF komatiite volcanic flow facies.

Table 8.1: Features of Komatiite Volcanic Flow Facies

Facies	Description	Type Examples
Thin differentiated flows (TDF)	Multiple compound spinifex-textured flows; generally less than 10 m thick, with internal differentiation into spinifex and cumulate zones.	Munro Township (Pyke et al., 1973)
Compound sheet flows with internal pathways (CSF)	Compound sheet flows with internal pathways (CSF) Compound thick cumulate-rich flows, with central olivine-rich lava pathways flanked by multiple thin differentiated units, from tens of metres to ~200 m maximum thickness.	Silver Lake Member at Kambalda (Lesher et al., 1984)
Dunitic compound sheet flows (DCSF)	Thick olivine-rich sheeted units with central lenticular bodies of olivine adcumulates, up to several hundred metres thick and 2 km wide, flanked by laterally extensive thinner orthocumulate-dominated sequences with minor spinifex. CSF and DCSF correspond to 'Flood Flow Facies' of Hill et al. (1995).	Perseverance and Mount Keith (Hill et al., 1995)
Dunitic sheet flows (DSF)	Thick, laterally extensive, unfractionated sheet-like bodies of olivine adcumulates and mesocumulates, in some cases laterally equivalent to layered lava lake bodies.	Southern section of the Walter Williams Formation (Gole and Hill, 1990; Hill et al., 1995)
Layered lava lakes and/or sills (LLLS)	Thick, sheeted bodies of olivine mesocumulates and adcumulates with lateral extents of tens of kilometres, with fractionated upper zones including pyroxenite and gabbro, up to several hundred metres in total thickness.	Kurrajong Formation (Gole and Hill, 1990; Hill et al., 1995)

Note: From Barnes et al., 2004.

8.3 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 (the "Dumont") of Dumont Nickel by Magneto Investments L.P., previously Royal Nickel Corporation (RNC), located 220 km to the east of Crawford Township (Figure 8.1). The Archean Dumont Sill, first reported in 1925, is located about 60 km northeast of Rouyn-Noranda and 25 km by road, northwest of the city of Amos, and within the Abitibi Greenstone Belt (Abitibi Region), 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).

Figure 8.1: Simplified Regional Geological Setting of the Abitibi Greenstone Belt (Abitibi Assemblages) and Dumont Sill Location

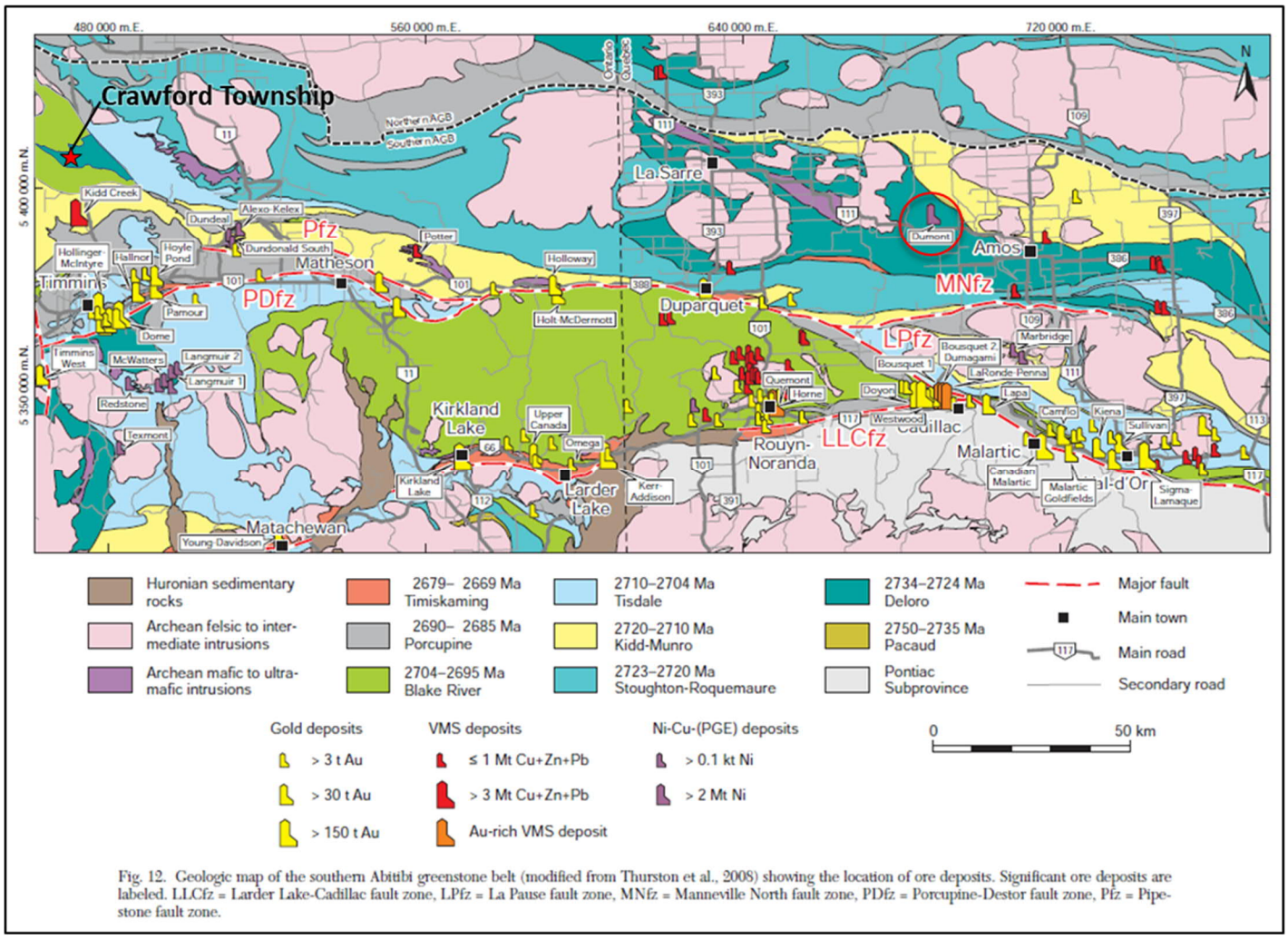


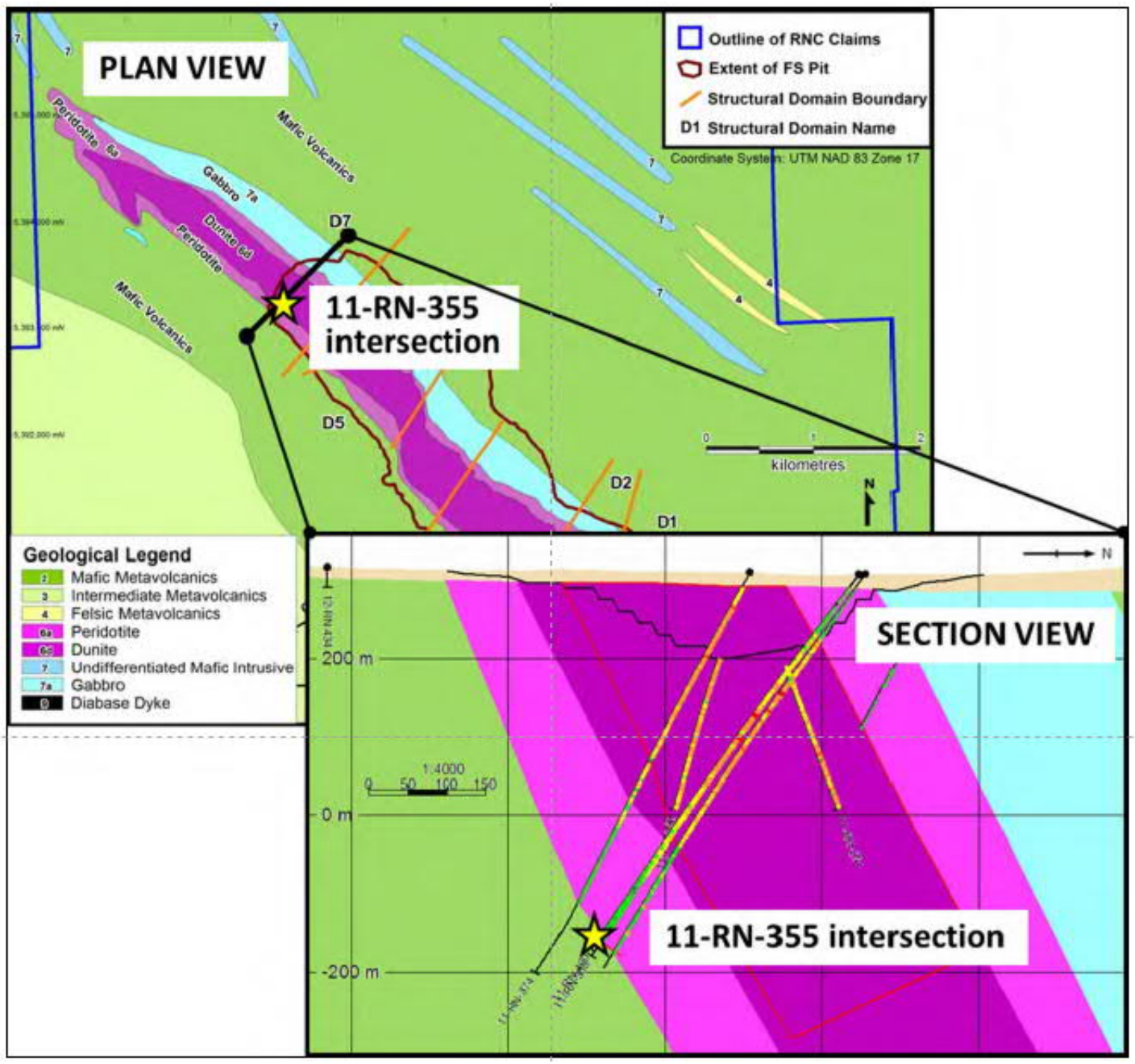
Fig. 12. Geologic map of the southern Abitibi greenstone belt (modified from Thurston et al., 2008) showing the location of ore deposits. Significant ore deposits are labeled. LLCfz = Larder Lake-Cadillac fault zone, LPfz = La Pause fault zone, MNfz = Manneville North fault zone, PDfz = Porcupine-Destor fault zone, Pfz = Pipestone fault zone.

Note: Location of the Dumont Sill, Quebec is indicated with a red circle, and the approximate location of Crawford and Lucas townships and the Crawford Ultramafic Complex Ontario is indicated by a red star. Both the Crawford Ultramafic Complex and the Dumont Sill are interpreted to be hosted by Deloro Assemblage rocks. Source: Modified after Monecke 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 (Figure 8.2). 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).

The Dumont is usually categorized with its most similar counterpart, the Mt. Keith nickel deposit located in the Agnew-Wiluna Greenstone Belt, in the Archean Yilgarn craton of Western Australia (Naldrett, 1989). Although the Dumont and CUC 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.

Figure 8.2: Plan View and Cross-section View (looking northwest) of the Dumont Nickel Deposit showing the Outline of the Proposed Open Pit



Note: Section is from a massive sulphide interval in drill hole 11-RN-355. Source: Ausenco, 2019.

In addition to its lower grade, the Dumont is differentiated from the Mt. Keith nickel deposit by the abundance of the nickel-iron alloy awaruite and by the restricted extent of talc-carbonate alteration, which is limited to the basal contact of the ultramafic body and occurs outside the resource envelope. In addition, the Dumont and CUC have not been subjected to the extensive supergene weathering alteration present at the Mt. Keith deposit.

Both the Dumont and Mt. Keith deposits have undergone pervasive serpentinization and local talc-carbonate alteration due to metamorphism to mid-upper greenschist facies. The observed mineralogy of the Dumont is a result of the serpentinization of a dunite protolith (>90% olivine), which locally hosted a primary, disseminated (intercumulus) magmatic sulphide assemblage and contained "trapped" nickel within the unaltered olivine. The pervasive serpentinization process, whereby olivine reacts with water to produce serpentine, magnetite and brucite, creates a strongly reducing environment where the nickel released from the decomposition of olivine is partitioned into low-sulphur sulphides and newly formed awaruite (see Section 6.3.2, SEM/BEI Mineralogical Study). The final mineral assemblage and texture of the disseminated nickel mineralization in the Dumont deposit and the variability has been controlled primarily by the variable degree of serpentinization that the host dunite has undergone.

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.

Metallurgical testwork by RNC has yielded concentrates with over 29% Ni and 1% Co. The high concentrate grade is a function of the very low sulphur content of the rock, so that most of the recoverable nickel is in low-sulphur minerals like heazlewoodite, or sulphur-free minerals like awaruite, a nickel-iron alloy.

Mineralization hosted by the Dumont Nickel Project is not necessarily indicative of mineralization hosted on CNC's Crawford Nickel Sulphide Project.

9 EXPLORATION

Other than diamond drilling (see Section 10.0), CNC has not completed any other exploration work on the Project as of the Effective Date of May 19, 2022.

10 DRILLING

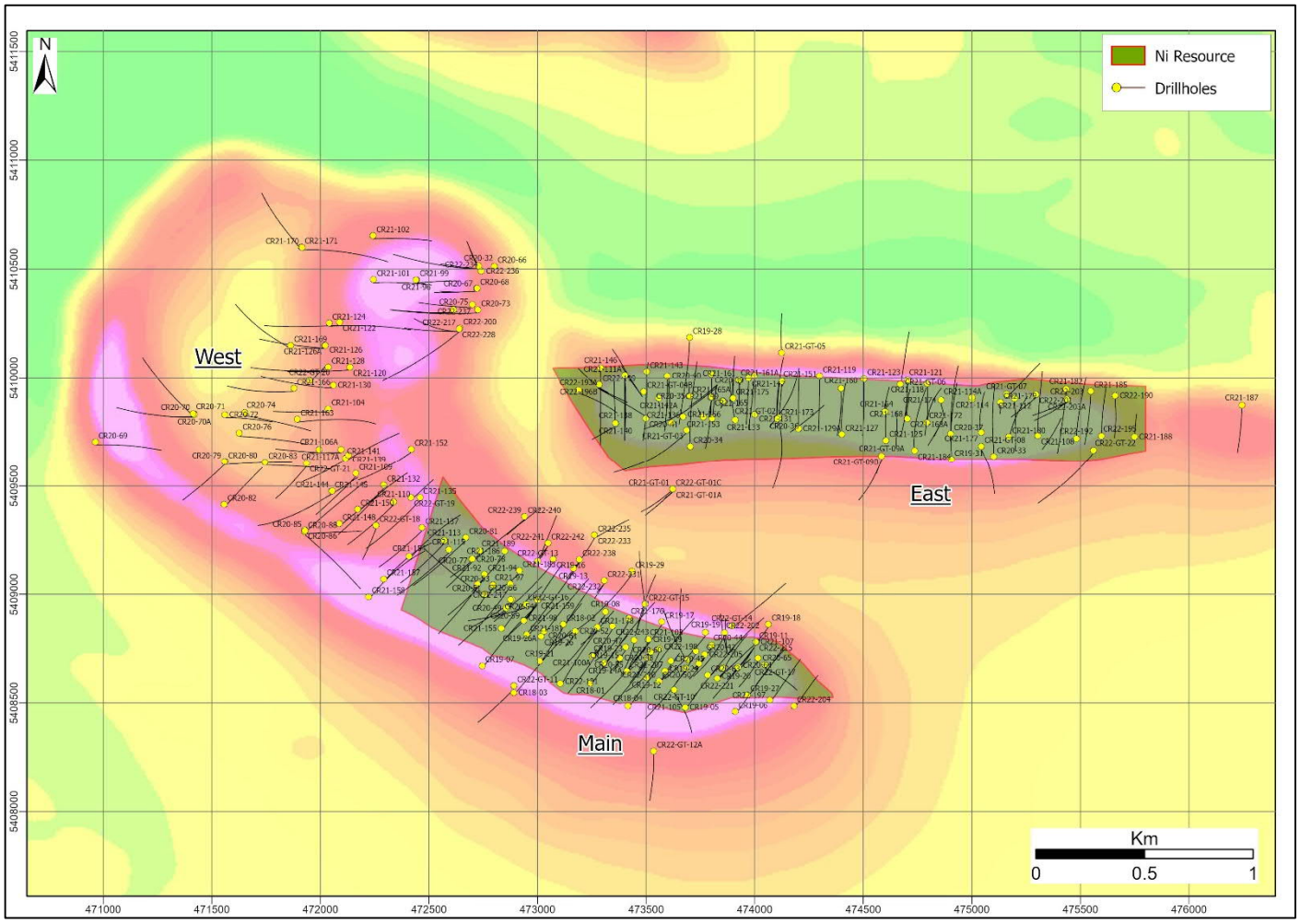
10.1 Introduction

The current drilling program is 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 Nickel Sulphide Project, management and control of the drilling program shifted from Spruce Ridge to CNC.

Results from the initial four drill holes 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 (*see* Noble news release dated March 4, 2019), results from CNC's first nine drill holes (CR19-05 to 13), which totalled 5,280 m, were announced by Noble on December 9, 2019.

As of the Effective Date of the Crawford Mineral Resource Estimate (May 19, 2022), a total of 291 drill holes totalling approximately 123,617.68 metres (up to hole CR22-248), have been completed by Canada Nickel and previous companies (Figure 10.1; Table 10.1). This includes drilling metres (7,564.7 m) from 36 abandoned holes. Additionally, 55 of the 76 drill holes were HQ size, completed for geotechnical and metallurgical testwork, whereas the remaining 236 drill holes used NQ size.

Figure 10.1: Plan View of Diamond Drill Hole Traces from 2018-2022 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, 2022.

10.2 Drill Hole Collar Surveys

The majority of the drill hole collar locations were determined through a differential GPS (DGPS) survey; the remaining locations were determined using an azimuth pointing system (APS). Collar inclination was measured using a manual inclinometer (Table 10.1). DGPS drill hole collar surveys were carried out by Talbot Surveys Inc. of Timmins, Ontario.

Table 10.1: Summary of Parameters for CR18, CR19, CR20, CR21, CR22, CR21-GT, and CR22-GT Series Diamond Drill Holes

Drill Hole	UTM (mN)	UTM (mE)	Elev (m)	Depth (m)	Collar Az	Collar Dip	Survey	Zone
CR18-01	5408577.5	473244.0	273.6	594.00	33	-60	DGPS	Main
CR18-02	5408847.9	473117.7	273.1	216.00	37	-51	DGPS	Main
CR18-03	5408533.0	472890.1	271.5	606.00	36	-51	DGPS	Main
CR18-04	5408472.7	473416.1	272.9	402.00	35	-51	DGPS	Main

Drill Hole	UTM (mN)	UTM (mE)	Elev (m)	Depth (m)	Collar Az	Collar Dip	Survey	Zone
CR19-05	5408466.2	473679.6	273.1	582.00	35	-50	DGPS	Main
CR19-06	5408446.2	473911.0	273.4	576.00	35	-50	DGPS	Main
CR19-07	5408657.6	472744.8	275.9	621.00	35	-50	DGPS	Main
CR19-08	5408905.6	473311.9	274.6	606.00	215	-50	DGPS	Main
CR19-09	5408778.2	473511.0	273.9	603.00	215	-50	DGPS	Main
CR19-10	5408725.2	473729.3	273.8	588.00	215	-50	DGPS	Main
CR19-11	5408767.5	474006.3	274.4	519.00	215	-50	DGPS	Main
CR19-12	5408602.6	473501.6	273.2	576.00	35	-50	DGPS	Main
CR19-13	5409104.9	473164.7	274.9	609.00	215	-50	DGPS	Main
CR19-14	5408633.5	473410.3	273.6	64.50	35	-82	DGPS	Main
CR19-14A	5408633.5	473410.3	273.6	944.20	35	-82	DGPS	Main
CR19-15	5408669.7	473307.9	273.6	600.00	35	-50	DGPS	Main
CR19-16	5409148.6	473072.0	273.3	642.00	215	-50	DGPS	Main
CR19-17	5408859.5	473572.0	274.1	501.00	215	-55	DGPS	Main
CR19-18	5408848.6	474063.5	274.6	507.00	215	-50	DGPS	Main
CR19-19	5408810.9	473773.9	274.3	723.00	215	-65	DGPS	Main
CR19-20	5408599.7	473827.3	273.6	702.00	35	-82	DGPS	Main
CR19-21	5408678.0	473009.8	271.8	702.00	35	-50	DGPS	Main
CR19-22	5409034.9	472873.3	271.1	570.00	215	-50	DGPS	Main
CR19-23	5408702.9	473250.9	273.3	705.00	35	-82	DGPS	Main
CR19-24	5408633.1	473586.5	273.4	702.00	35	-82	DGPS	Main
CR19-25	5409035.2	472723.7	270.4	453.00	215	-50	DGPS	Main
CR19-26	5408792.1	473016.9	271.8	74.60	35	-82	DGPS	Main
CR19-26A	5408792.1	473016.9	271.8	146.20	35	-82	DGPS	Main
CR19-27	5408523.5	473965.6	273.4	444.00	35	-50	DGPS	Main
CR19-28	5410172.1	473700.0	277.8	573.00	180	-50	DGPS	East
CR19-29	5409092.7	473435.3	275.0	750.00	215	-50	DGPS	Main
CR19-31	5409610.1	474905.4	276.1	552.00	360	-50	DGPS	East
CR20-32	5410500.8	472728.3	271.6	633.00	270	-50	DGPS	West
CR20-33	5409622.9	475100.7	275.6	549.00	360	-50	DGPS	East
CR20-34	5409670.8	473703.4	277.6	636.00	360	-50	DGPS	East
CR20-35	5409902.8	473701.3	277.9	390.00	360	-82	DGPS	East
CR20-36	5409798.7	474102.7	277.1	483.00	360	-50	DGPS	East
CR20-37	5409728.2	474902.8	276.2	492.00	360	-50	DGPS	East
CR20-38	5409893.2	473900.0	278.0	327.00	360	-50	DGPS	East
CR20-39	5410003.7	473800.9	278.3	522.00	180	-50	DGPS	East
CR20-40	5409994.9	473597.5	278.0	378.00	180	-50	DGPS	East
CR20-41	5409799.8	473499.3	277.4	426.00	360	-50	DGPS	East
CR20-42	5408708.5	473769.8	274.0	405.00	215	-80	DGPS	Main
CR20-43	5408666.4	473743.4	273.8	402.00	215	-80	DGPS	Main
CR20-44	5408751.9	473802.3	274.1	402.00	215	-80	DGPS	Main
CR20-45	5408678.7	473613.4	273.6	408.00	35	-80	DGPS	Main
CR20-46	5408720.6	473520.7	273.7	411.00	215	-80	DGPS	Main
CR20-47	5408742.4	473404.9	273.9	402.00	215	-80	DGPS	Main
CR20-48	5408693.3	473378.5	273.5	402.00	215	-80	DGPS	Main
CR20-49	5408922.1	472855.7	270.9	402.00	215	-80	DGPS	Main

Drill Hole	UTM (mN)	UTM (mE)	Elev (m)	Depth (m)	Collar Az	Collar Dip	Survey	Zone
CR20-50	5408586.4	473559.2	273.1	402.00	35	-80	DGPS	Main
CR20-51	5408985.4	472755.5	270.4	405.00	215	-80	DGPS	Main
CR20-52	5408816.6	473173.1	273.5	402.00	211	-80	DGPS	Main
CR20-53	5409028.6	472794.1	270.9	402.00	212	-80	DGPS	Main
CR20-54	5408963.3	472882.9	270.9	402.00	218	-80	DGPS	Main
CR20-55	5408633.4	473410.6	273.4	240.00	35	-82	DGPS	Main
CR20-56	5408987.4	472757.0	270.7	585.00	305	-50	DGPS	Main
CR20-57	5409105.4	473165.5	274.8	174.00	215.5	-50	DGPS	Main
CR20-58	5408711.1	473770.4	273.9	405.00	215.3	-80	DGPS	Main
CR20-59	5408865.9	472937.3	271.4	513.00	35	-50	DGPS	Main
CR20-60	5408694.5	473505.2	273.6	402.00	215	-50	DGPS	Main
CR20-61	5408816.0	473030.0	272.0	504.00	35	-50	DGPS	Main
CR20-62	5408695.0	473505.6	273.6	402.00	215	-80	DGPS	Main
CR20-63	5408614.1	473783.9	273.4	402.00	35	-80	DGPS	Main
CR20-64	5408650.8	473923.9	273.9	402.00	35	-80	DGPS	Main
CR20-65	5408693.2	474015.0	274.2	402.00	215	-65	DGPS	Main
CR20-66	5410498.8	472801.1	272.1	402.00	270	-50	DGPS	West
CR20-67	5410397.8	472721.2	271.5	300.00	270	-45	DGPS	West
CR20-68	5410397.9	472720.6	271.6	402.00	270	-60	DGPS	West
CR20-69	5409691.5	470963.4	268.2	501.00	90	-50	DGPS	West
CR20-70	5409822.4	471414.3	267.6	156.00	270	-50	DGPS	West
CR20-70A	5409819.1	471419.0	267.5	576.00	269	-50	DGPS	West
CR20-71	5409820.8	471419.6	267.6	594.08	314	-53	DGPS	West
CR20-72	5409817.3	471559.3	267.2	372.12	90	-50	DGPS	West
CR20-73	5410298.8	472725.0	271.4	525.00	270	-45	DGPS	West
CR20-74	5409826.8	471651.9	266.6	504.00	90	-50	DGPS	West
CR20-75	5410299.2	472613.8	271.0	450.25	270	-50	DGPS	West
CR20-76	5409731.3	471626.5	266.9	501.34	93	-52	DGPS	West
CR20-77	5409150.1	472699.3	270.5	405.00	223	-66	DGPS	Main
CR20-78	5409147.4	472696.6	270.3	592.52	52	-51	DGPS	Main
CR20-79	5409599.3	471558.3	267.9	395.18	225	-52	DGPS	West
CR20-80	5409599.9	471560.9	267.8	501.00	90	-52	DGPS	West
CR20-81	5409248.0	472669.5	270.4	501.00	226	-50	DGPS	Main
CR20-82	5409400.5	471557.3	268.6	421.79	45	-50	DGPS	West
CR20-83	5409597.4	471745.1	266.8	493.37	90	-50	DGPS	West
CR20-85	5409280.1	471929.1	267.5	501.32	90	-50	DGPS	Main
CR20-86	5409282.5	471927.5	267.4	552.50	45	-50	DGPS	West
CR20-88	5409275.4	471927.7	267.3	531.00	135	-50	DGPS	West
CR21-100	5408704.8	473252.6	273.2	57.00	35	-82	DGPS	Main
CR21-100A	5408703.6	473251.9	273.4	250.00	35	-82	DGPS	Main
CR21-101	5410437.7	472244.6	269.6	450.00	90	-50	DGPS	West
CR21-102	5410639.0	472242.8	270.3	432.00	90	-50	DGPS	West
CR21-103	5408777.4	473509.2	273.7	252.00	215	-50	DGPS	Main
CR21-104	5409843.2	472036.2	267.7	402.00	270	-50	DGPS	West
CR21-105	5408465.9	473680.8	273.0	150.00	35	-50	DGPS	Main
CR21-106	5409655.6	471994.3	267.9	67.00	270	-50	DGPS	West

Drill Hole	UTM (mN)	UTM (mE)	Elev (m)	Depth (m)	Collar Az	Collar Dip	Survey	Zone
CR21-106A	5409655.7	471993.9	267.9	303.00	270	-50	DGPS	West
CR21-107	5408767.1	474005.5	274.5	201.00	215	-50	DGPS	Main
CR21-108	5409721.0	475303.5	274.7	501.00	360	-50	DGPS	East
CR21-109	5409544.1	472163.2	267.6	397.50	225	-50	DGPS	West
CR21-110	5409412.6	472337.7	268.8	426.00	225	-50	DGPS	West
CR21-111	5409996.5	473401.8	277.5	115.00	180	-50	DGPS	East
CR21-111A	5409997.1	473401.8	277.6	444.00	180	-50	DGPS	East
CR21-112	5409891.7	475198.0	274.9	435.00	180	-60	DGPS	East
CR21-113	5409234.4	472570.3	269.6	423.00	225	-50	DGPS	Main
CR21-114	5409895.5	475001.7	275.8	227.00	180	-60	DGPS	East
CR21-114A	5409888.1	475001.0	275.7	468.00	180	-60	DGPS	East
CR21-115	5409191.3	472590.5	269.8	450.00	225	-50	DGPS	Main
CR21-116	5409996.9	473998.6	277.7	510.00	180	-50	DGPS	East
CR21-117	5409654.4	472091.1	267.6	99.00	270	-50	DGPS	West
CR21-117A	5409655.0	472095.0	270.0	153.00	270	-50	APS	West
CR21-117B	5409656.0	472097.0	270.0	98.00	270	-55	APS	West
CR21-117C	5409656.4	472096.3	267.6	57.00	270	-55	DGPS	West
CR21-118	5409960.8	474795.1	276.1	504.00	180	-60	DGPS	East
CR21-119	5409994.7	474298.8	276.2	479.00	180	-60	DGPS	East
CR21-120	5410035.8	472135.3	268.7	447.00	270	-50	DGPS	West
CR21-121	5409977.3	474704.2	276.2	507.00	180	-60	DGPS	East
CR21-122	5410242.3	472087.8	268.9	501.00	270	-50	DGPS	East
CR21-123	5409982.6	474503.2	276.2	507.00	180	-60	DGPS	East
CR21-124	5410237.8	472040.2	268.6	557.00	90	-50	DGPS	West
CR21-125	5409698.9	474605.2	276.3	375.60	360	-60	DGPS	East
CR21-126	5410135.3	472021.5	268.3	169.00	270	-50	DGPS	West
CR21-126A	5410135.4	472021.1	268.5	546.00	270	-55	DGPS	West
CR21-127	5409726.2	474401.9	276.7	411.00	360	-50	DGPS	East
CR21-128	5410034.7	472036.3	268.1	549.00	270	-50	DGPS	West
CR21-129	5409752.6	474201.0	276.9	63.00	360	-50	DGPS	East
CR21-129A	5409753.0	474200.9	277.1	411.00	360	-50	DGPS	East
CR21-130	5409952.6	472058.8	267.7	561.00	270	-50	DGPS	West
CR21-131	5409818.0	474000.0	277.3	402.00	5	-65	DGPS	East
CR21-132	5409489.5	472292.0	268.5	489.00	225	-50	DGPS	Main
CR21-133	5409793.3	473910.4	277.8	372.00	355	-60	DGPS	East
CR21-134	5409799.0	473803.8	278.8	66.00	5	-60	DGPS	East
CR21-134A	5409799.3	473803.8	278.6	351.00	5	-60	DGPS	East
CR21-135	5409431.5	472458.8	269.4	564.00	225	-50	DGPS	Main
CR21-136	5409779.7	473617.3	277.6	375.00	355	-55	DGPS	East
CR21-137	5409292.8	472468.4	269.2	468.00	225	-55	DGPS	Main
CR21-138	5409778.2	473359.1	277.1	420.00	360	-60	DGPS	East
CR21-139	5409630.9	472128.4	267.9	465.50	235	-55	DGPS	Main
CR21-140	5409776.6	473359.8	277.2	267.00	245	-60	DGPS	East
CR21-141	5409615.7	472115.5	268.0	588.00	60	-55	DGPS	Main
CR21-142	5409893.2	473557.4	277.5	69.00	178	-86	DGPS	East
CR21-142A	5409893.2	473557.4	277.5	624.00	178	-86	DGPS	East

Drill Hole	UTM (mN)	UTM (mE)	Elev (m)	Depth (m)	Collar Az	Collar Dip	Survey	Zone
CR21-143	5410014.1	473502.2	277.8	501.00	180	-50	DGPS	East
CR21-144	5409462.9	472053.7	267.4	369.00	60	-65	DGPS	Main
CR21-145	5409461.6	472050.9	267.2	675.00	235	-55	DGPS	Main
CR21-146	5410031.5	473290.8	276.7	525.00	180	-50	DGPS	East
CR21-147	5409986.5	473972.0	277.9	351.00	185	-50	DGPS	East
CR21-148	5409311.8	472086.5	268.6	552.00	50	-55	DGPS	Main
CR21-149	5409880.3	473853.6	278.0	474.00	178	-87	DGPS	East
CR21-150	5409378.1	472170.6	267.8	528.00	50	-50	DGPS	Main
CR21-151	5409970.0	474125.0	277.1	309.00	185	-50	DGPS	East
CR21-152	5409657.9	472419.0	269.5	535.50	235	-55	DGPS	Main
CR21-153	5409806.7	473763.5	278.1	669.00	360	-86	DGPS	East
CR21-154	5409162.2	472408.7	269.1	636.00	45	-55	DGPS	Main
CR21-155	5408828.8	472834.0	271.0	579.00	35	-62	DGPS	Main
CR21-156	5409808.9	473669.7	277.8	658.00	0	-86	DGPS	East
CR21-157	5409057.0	472290.7	268.0	549.00	45	-55	DGPS	Main
CR21-158	5408975.3	472222.1	267.8	699.00	45	-55	DGPS	Main
CR21-159	5408960.4	473001.5	272.7	366.00	215	-55	DGPS	Main
CR21-160	5409939.4	474397.3	276.3	549.00	180	-70	DGPS	East
CR21-161	5409975.9	473928.9	278.0	231.00	175	-79	DGPS	East
CR21-161A	5409974.3	473928.9	278.0	432.00	180	-77	DGPS	East
CR21-161B	5409977.0	473931.0	277.0	33.00	180	-76.5		East
CR21-161C	5409974.0	473928.9	277.9	126.00	80	-76.5	DGPS	East
CR21-162	5408960.9	473001.9	272.7	396.00	215	-75	DGPS	Main
CR21-163	5409795.7	471893.3	267.3	675.00	81	-52	DGPS	West
CR21-164	5409830.2	474599.8	276.5	327.00	0	-70	DGPS	East
CR21-165	5409897.2	473801.3	278.6	555.00	180	-86	DGPS	East
CR21-165A	5409895.4	473801.3	278.5	735.00	185	-84	DGPS	East
CR21-166	5409937.8	471877.8	266.7	583.00	265	-50	DGPS	Main
CR21-167	5408576.6	473243.6	273.5	207.00	35	-60	DGPS	Main
CR21-168	5409798.7	474702.6	277.0	339.00	360	-70	APS	East
CR21-168A	5409800.0	474701.0	277.0	420.00	360	-70	APS	East
CR21-169	5410134.8	471863.2	266.9	615.00	300	-50	DGPS	West
CR21-170	5410585.1	471916.4	268.1	531.00	315	-50	DGPS	West
CR21-171	5410587.2	471912.8	268.1	639.00	90	-50	DGPS	West
CR21-172	5409780.0	474795.0	274.0	681.00	360	-75	APS	East
CR21-173	5409798.0	474100.9	277.4	312.00	360	-50	DGPS	East
CR21-174	5409884.0	474858.0	275.0	630.00	168	-85	APS	East
CR21-175	5409893.4	473898.7	278.9	234.00	360	-50	DGPS	East
CR21-176	5408875.0	473421.1	274.2	741.00	210	-70	DGPS	Main
CR21-177	5409735.0	475043.0	276.0	567.00	360	-70	APS	East
CR21-178	5408835.1	473275.0	274.2	504.00	220	-78	DGPS	Main
CR21-179	5409874.0	475131.0	276.0	416.00	180	-85	APS	East
CR21-180	5409715.0	475171.0	278.0	501.00	10	-65	APS	East
CR21-181	5408800.0	472950.0	272.0	606.00	35	-65	APS	Main
CR21-182	5409935.0	475434.0	272.0	489.00	180	-55	APS	East
CR21-183	5409095.8	472916.1	271.3	596.00	220	-75	DGPS	Main

Drill Hole	UTM (mN)	UTM (mE)	Elev (m)	Depth (m)	Collar Az	Collar Dip	Survey	Zone
CR21-184	5409650.0	474736.0	276.0	444.00	360	-50	APS	East
CR21-185	5409925.0	475548.0	273.0	552.00	180	-64	APS	East
CR21-186	5409184.6	472848.8	271.0	621.00	215	-72	DGPS	Main
CR21-187	5409860.0	476243.0	272.0	327.00	180	-50	APS	East
CR21-188	5409715.0	475749.0	272.0	258.00	360	-62	APS	Main
CR21-189	5409186.0	472735.1	270.4	663.00	220	-76	DGPS	Main
CR21-92	5409078.1	472756.1	270.7	600.00	225	-80	DGPS	Main
CR21-94	5409078.0	472756.1	270.6	402.00	225	-50	DGPS	Main
CR21-96	5410437.2	472441.2	270.2	312.00	90	-50	DGPS	West
CR21-97	5409030.1	472795.0	270.7	240.00	215	-80	DGPS	Main
CR21-98	5408866.4	472935.9	271.3	162.00	35	-50	DGPS	Main
CR21-99	5410437.3	472442.0	270.3	417.00	90	-82	DGPS	West
CR21-GT-01	5409471.0	473622.8	272.5	297.00	230	-55	APS	East
CR21-GT-01A	5409472.5	473622.4	277.2	114.80	230	-65	DGPS	East
CR21-GT-02	5409804.8	474109.8	277.4	625.50	170	-60	DGPS	East
CR21-GT-03	5409746.3	473686.0	278.2	618.00	230	-55	DGPS	East
CR21-GT-04	5409924.0	473489.0	273.0	113.00	325	-65		East
CR21-GT-04A	5409924.0	473489.0	272.0	396.00	325	-65	APS	East
CR21-GT-04B	5409914.6	473482.1	277.7	600.00	325	-65	DGPS	East
CR21-GT-05	5410104.5	474125.0	276.7	450.00	30	-65	DGPS	East
CR21-GT-06	5409957.6	474671.1	276.2	573.00	10	-65	DGPS	East
CR21-GT-07	5409909.6	475160.8	275.0	525.00	45	-65	DGPS	East
CR21-GT-08	5409670.1	475043.0	275.9	525.00	150	-65	DGPS	East
CR21-GT-09	5409625.0	474586.0	276.0	40.00	200	-65		East
CR21-GT-09A	5409625.0	474585.0	276.0	170.00	200	-65	APS	East
CR21-GT-09B	5409622.5	474584.8	276.5	525.00	200	-65	DGPS	East
CR22-190	5409905.0	475660.0	271.0	402.00	180	-62	APS	East
CR22-191	5408575.0	473105.0	272.0	675.00	30	-50	APS	Main
CR22-192	5409706.0	475481.0	275.0	491.00	360	-55	APS	East
CR22-193	5409930.0	473189.0	276.0	52.00	110	-70		East
CR22-193A	5409930.0	473189.0	276.0	662.50	110	-70	APS	East
CR22-194	5408775.4	473444.9	274.0	777.00	203	-76	DGPS	Main
CR22-195	5409718.3	475597.5	273.0	423.00	0	-55	APS	East
CR22-196	5409952.3	473280.8	277.7	78.00	118	-70	DGPS	East
CR22-196A	5409954.3	473278.9	277.6	60.00	120	-68	DGPS	East
CR22-196B	5409951.2	473281.5	277.7	762.00	120	-68	DGPS	East
CR22-197	5408499.7	474069.8	274.4	384.00	35	-50	DGPS	Main
CR22-198	5408735.5	473561.5	274.3	1,044.00	205	-75	DGPS	Main
CR22-199	5409955.0	473284.0	273.0	276.00	310	-60	APS	East
CR22-200	5410214.3	472642.7	271.8	633.00	230	-50	DGPS	West
CR22-201	5409910.0	475296.0	275.0	435.00	230	-70	APS	East
CR22-202	5408808.7	473860.5	274.8	675.00	205	-68	DGPS	Main
CR22-203	5409888.0	475438.0	275.0	54.00	245	-70		East
CR22-203A	5409887.0	475438.0	275.0	33.00	238	-69	APS	East
CR22-203B	5409887.0	475438.0	275.0	477.00	238	-68	APS	East
CR22-203BW	5409887.0	475438.2	275.0	456.00	238	-69		East

Drill Hole	UTM (mN)	UTM (mE)	Elev (m)	Depth (m)	Collar Az	Collar Dip	Survey	Zone
CR22-204	5408472.8	474181.0	275.5	369.00	35	-60	DGPS	Main
CR22-205	5408709.2	473769.6	274.4	354.00	215	-80	DGPS	Main
CR22-206	5409880.7	473853.5	278.5	480.40	205	-88	DGPS	East
CR22-207	5408633.0	473410.0	273.0	351.00	35	-82	APS	Main
CR22-208	5408693.0	473378.0	274.0	249.00	215	-80	APS	Main
CR22-210	5408586.0	473559.0	272.0	369.00	35	-80	APS	Main
CR22-211	5409955.0	473284.0	275.0	80.00	120	-68		East
CR22-211A	5409955.0	473284.0	275.0	130.10	120	-68		East
CR22-211B	5409952.0	473281.0	275.0	402.00	117	-68		East
CR22-212	5408752.0	473802.0	274.0	240.00	215	-80	APS	Main
CR22-215	5408767.0	474006.0	275.0	351.00	215	-50	APS	Main
CR22-217	5410212.0	472638.0	272.0	594.00	273	-70	APS	West
CR22-219	5408600.0	473827.0	274.0	237.00	35	-82	APS	Main
CR22-222	5409968.0	473828.0	277.0	48.00	230	-70		East
CR22-222A	5409968.0	473828.0	277.0	69.00	230	-70		East
CR22-222B	5409968.0	473827.0	277.0	42.00	230	-70		East
CR22-222C	5409967.0	473827.0	277.0	456.00	230	-70		East
CR22-221	5408599.0	473828.0	273.5	237.00	32	-82	APS	Main
CR22-224	5408614.0	473783.0	275.0	252.00	40	-80	APS	Main
CR22-226	5408633.0	473587.0	273.0	264.00	35	-82	APS	Main
CR22-227	5408694.0	473505.0	274.0	200.00	215	-50	APS	Main
CR22-228	5410212.0	472638.0	272.0	393.00	273	-52	APS	West
CR22-229	5408646.0	473682.0	274.0	204.00	218	-69	APS	Main
CR22-230	5409953.0	473822.0	277.0	1155.00	230	-70		East
CR22-231	5409049.0	473307.0	273.0	171.00	210	-70	APS	Main
CR22-232	5409049.0	473307.0	273.0	132.00	210	-50	APS	Main
CR22-233	5409258.8	473262.3	274.0	291.00	60	-50	APS	Main
CR22-234	5410477.0	472741.0	270.0	402.00	336	-68	APS	Main
CR22-235	5409258.8	473262.4	274.0	363.00	210	-50	APS	Main
CR22-236	5410477.0	472741.0	270.0	336.00	336	-58	APS	Main
CR22-237	5410323.0	472700.0	269.0	432.00	275	-55	APS	Main
CR22-238	5409146.0	473192.0	275.0	213.00	210	-65	APS	Main
CR22-239	5409345.0	472940.0	271.0	150.00	228	-50	APS	Main
CR22-240	5409344.0	472940.0	271.0	243.01	228	-65	APS	Main
CR22-241	5409222.0	473048.0	274.0	252.00	212	-75	APS	Main
CR22-242	5409222.0	473048.0	274.0	219.00	212	-60	APS	Main
CR22-243	5408840.0	473345.0	272.0	523.00	212	-45	APS	Main
CR22-244	5408686.0	473692.0	274.0	384.00	328	-45	APS	Main
CR22-245	5409816.0	474868.0	275.0	178.30	205	-60		East
CR22-245A	5409817.0	474868.5	275.0	300.00	205	-60		East
CR22-246	5408646.0	473852.0	272.0	387.00	28	-55	APS	Main
CR22-247	5408962.0	472875.0	272.0	390.00	233	-55	APS	Main
CR22-248	5409845.0	474653.0	277.0	310.00	178	-60		East
CR22-GT-01C	5409479.2	473620.3	278.0	356.00	230	-60	DGPS	East
CR22-GT-10	5408547.2	473629.4	273.8	575.00	150	-70	DGPS	Main
CR22-GT-11	5408564.9	472890.8	271.9	471.00	220	-65	DGPS	Main

Drill Hole	UTM (mN)	UTM (mE)	Elev (m)	Depth (m)	Collar Az	Collar Dip	Survey	Zone
CR22-GT-12A	5408264.6	473534.4	272.5	450.00	180	-65	DGPS	Main
CR22-GT-13	5409139.9	473001.2	273.1	501.00	20	-60	DGPS	Main
CR22-GT-14	5408840.6	473894.4	275.0	550.00	50	-55	DGPS	Main
CR22-GT-15	5408941.0	473496.3	275.2	600.00	350	-65	DGPS	Main
CR22-GT-16	5408939.2	472947.8	271.7	576.00	250	-60	DGPS	Main
CR22-GT-17	5408661.5	474047.5	274.8	552.00	100	-65	DGPS	Main
CR22-GT-18	5409304.0	472256.3	268.3	399.00	230	-60	DGPS	West
CR22-GT-19	5409431.8	472416.3	269.2	351.00	45	-65	DGPS	West
CR22-GT-20	5409976.2	471949.7	267.3	327.00	40	-65	DGPS	West
CR22-GT-21	5409590.7	471936.3	267.5	426.00	220	-60	DGPS	West
CR22-GT-22	5409652.3	475558.6	272.5	594.00	220	-60	DGPS	East
			TOTAL:	**123,617.68**				

Notes: "GT" drill holes are geotechnical holes used for lithology in the modelling but do not have any assays. Source: CNC, 2022.

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 ±60 cm or better and ±2.5 m accuracy when SBAS is not available. The handheld GPS unit provided location accuracy of approximately ±3 metres.

In general, drill hole surveys were initiated immediately following the casing and then every 50 m afterward using a Reflex gyrocompass system. If the hole survey was completed after the drill hole was finished and the rods removed, then the survey was taken approximately every 10 metres.

10.3 Diamond Drill Core Assay Results

As of the effective date of the mineral resource estimate, diamond drilling has been completed at the Main, East, and West zones (see Figure 10.1), with 177 of the current 291 drill holes used in the updated Main Zone Mineral Resource Estimate and 81 used in the initial East Zone Mineral Resource Estimate (see Section 14).

10.3.1 Main Zone Drilling

The focus of the ongoing 2019-2022 drilling is to extend along strike mineralization encountered in the original historical 2018 series drill holes (Spruce Ridge) and subsequent drilling campaigns, to test the east-northeastern and west-southwestern extents of mineralization (*i.e.*, the contacts), to test 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.

Table 10.2: Main Zone: Selective Drill Core Assays, CR19, CR20, and CR21 Series Diamond Drill Holes

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	-

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.	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	-

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 (%)
CR19-17	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	-
	166.5	385.5	219.0	143.7	0.28	0.004	0.004	0.003	0.007	0.04	-
	82.5	91.5	9.0	5.9	0.51	0.035	0.320	0.123	0.443	0.30	-
	82.5	84.0	1.5	1.0	0.65	0.094	0.632	0.263	0.895	0.53	-
	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

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.	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

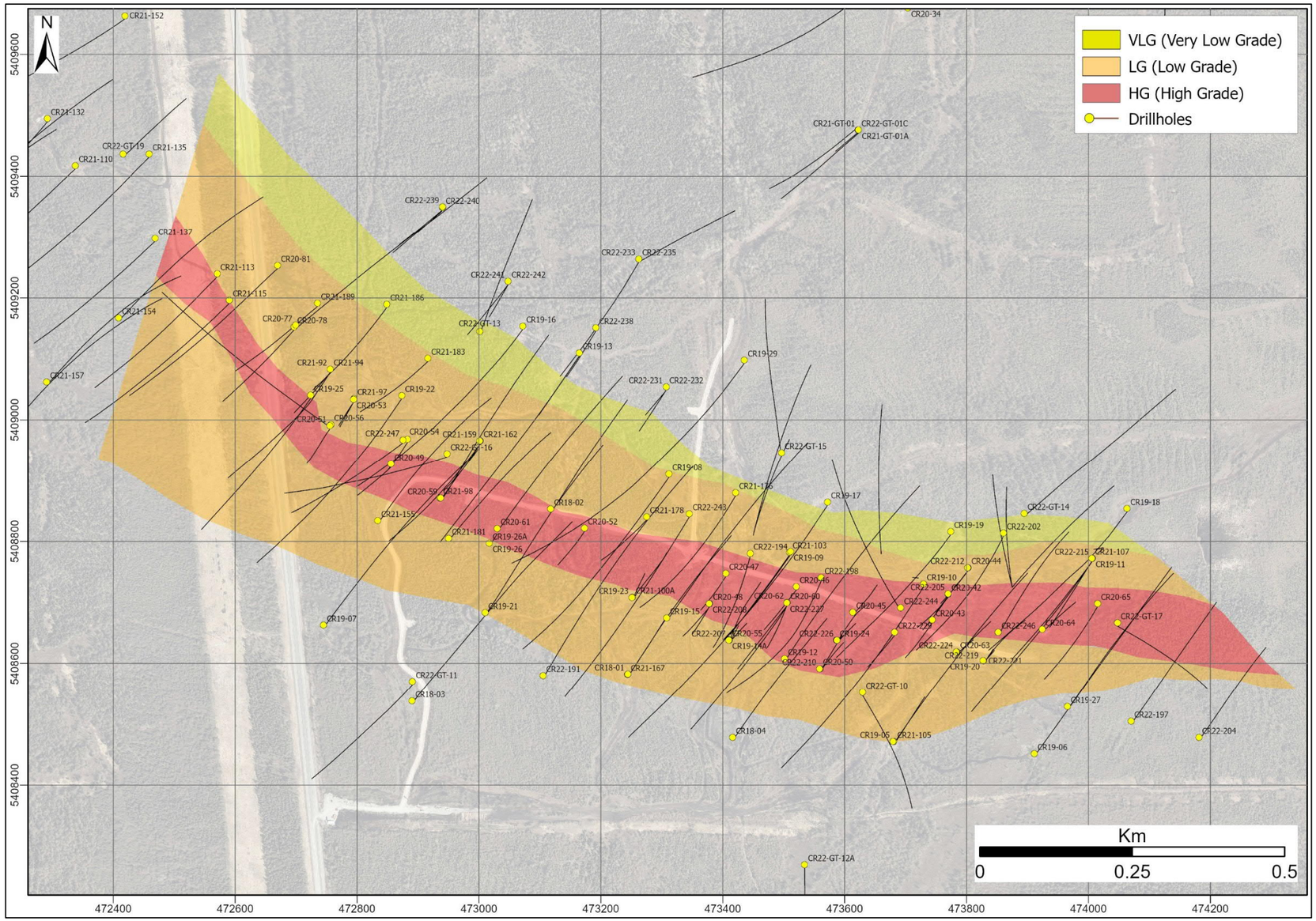
Drillhole	From (m)	To (m)	Interval (m)	Estimated True Width (m)	Ni (%)	Co (%)	Pd (g/t)	Pt (g/t)	Pd+Pt (g/t)	S (%)	Fe (%)
AND	460.5	469.5	9.0	5.9	0.04	0.010	0.600	0.900	1.500	-	-
incl.	465.0	468.0	3.0	2.0	0.05	0.013	0.600	1.200	1.800	-	-
CR20-60	51.0	352.5	301.5	-	0.24	0.013	0.016	0.009	0.025	0.10	6.94
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
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

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.	465.0	741.0	276.0	94.4	0.32	0.013	0.017	0.006	0.023	0.08	7.08

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. Canada Nickel 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 a number of factors. Source: Caracle Creek, 2020.

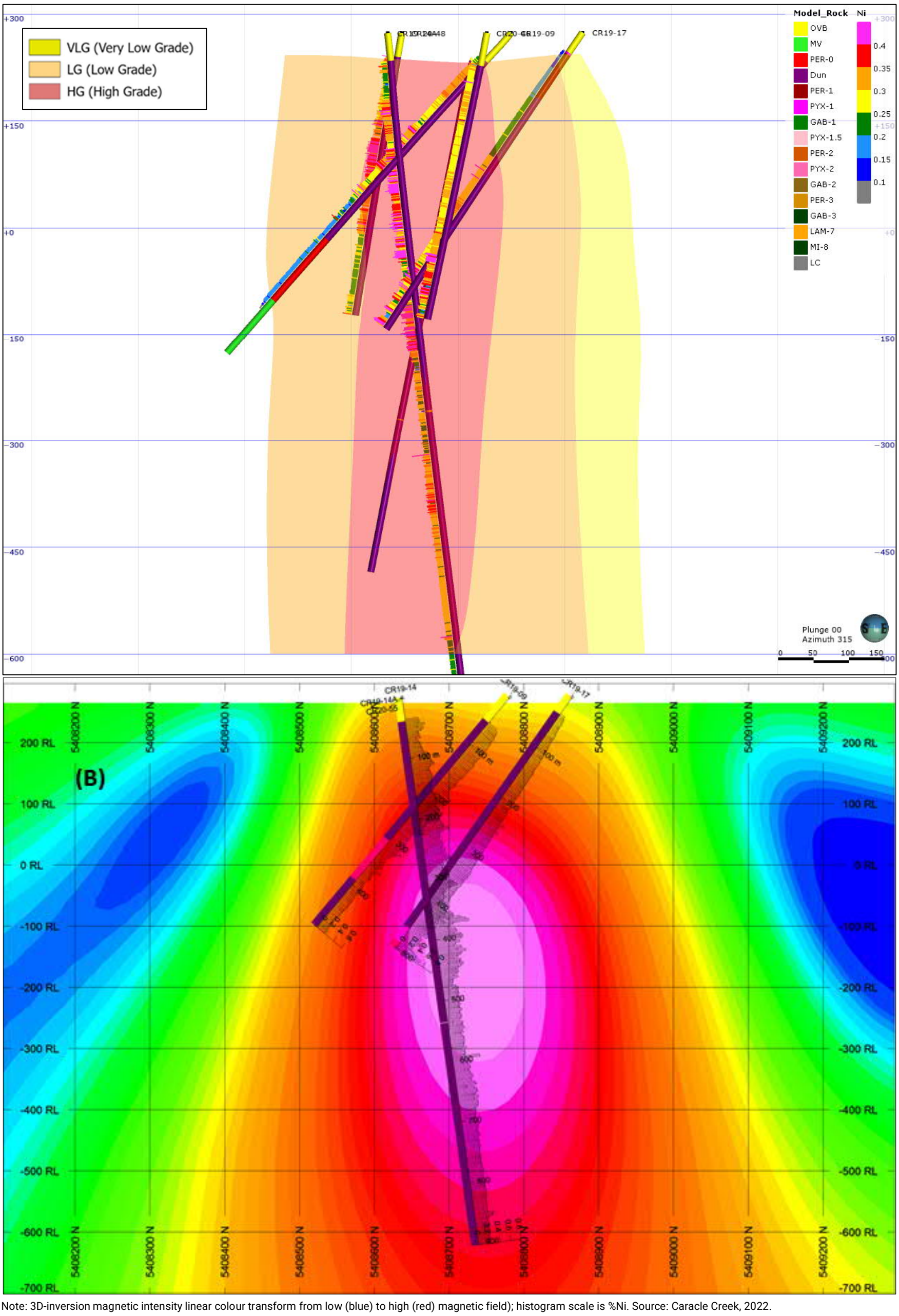
To date, diamond drilling has outlined a west-northwest trending (~285-315Az) ultramafic body (largely dunite-peridotite) that is at least 1.9 km in strike length, 280 to 580 m in width, and more than 800 metres deep (Figures 10.2 and 10.3). Mineralization remains open along strike to the northwest, and at depth. A north-northwest trending regional sinistral, strike-slip fault terminates the ultramafic body along its southeastern extent (see Figure 7.5). A 3D-inversion magnetic anomaly, nearly 1 km deep, has been only partially tested at depth (Figure 10.3).

Figure 10.2: Plan View of Diamond Drill Hole Traces from 2018-2022 drilling at the Main Zone



Note: Drilling is superimposed on the outline of the mineral resource zones. Source: CNC, 2022.

Figure 10.3: Oblique Sections looking Northwest from Main Zone showing Estimation Domains and 3D Inversion Magnetic Intensity



Note: 3D-inversion magnetic intensity linear colour transform from low (blue) to high (red) magnetic field); histogram scale is %Ni. Source: Caracle Creek, 2022.

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 (modeled 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 and internal regions of higher-grade nickel remain open along strike to the west-northwest and extend to a depth of at least 650 m (see Figures 10.2 and 10.3).

10.3.1.2 Main Zone – PGE Reef

The Main Zone PGE reef, located within the northern margin of the ultramafic to mafic body, is associated with a contact between an ultramafic (pyroxenite) unit to the south and a gabbroic unit to the north, reflected in seven drill hole intercepts (see Table 10.3; Figure 10-3). Additional drill holes will be required to better define the PGE reef and as such the PGE reef was restricted to the central region of the modelling area.

Table 10.3: True Width Intercepts for Drill Holes into the Main Zone PGE Reef

BHID	True Width (m)	Pd (ppm)	Pt (ppm)	PGE (ppm)	Ni (%)	Co (%)	Fe (%)	S (%)
CR19-12	7.70	0.315	0.493	0.807	0.064	0.013	7.379	0.039
CR19-13	4.90	0.735	1.012	1.747	0.053	0.012	7.000	0.030
CR19-15	0.90	0.298	0.058	0.356	0.035	0.007	4.870	0.010
CR19-16	5.00	0.772	0.958	1.730	0.060	0.013	7.098	0.044
CR19-29	2.80	0.349	0.484	0.834	0.052	0.011	4.807	0.157
CR20-59	6.60	0.540	0.772	1.313	0.041	0.010	5.857	0.082
CR20-61	0.90	0.127	0.200	0.327	0.080	0.016	7.700	0.030

10.3.2 East Zone Drilling

Located about 1.2 km northeast of the Main Zone (see Figure 7.5), CNC began to drill-test the East Zone in late 2019 and into 2022. with relatively wide-spaced drill hole sections (Figure 10.4). Selective drill core assays from the East Zone are summarized in Table 10.4 (Figure 10.5).

Figure 10.4: Plan View of Diamond Drill Hole Traces from 2018-2022 drilling at the East Zone

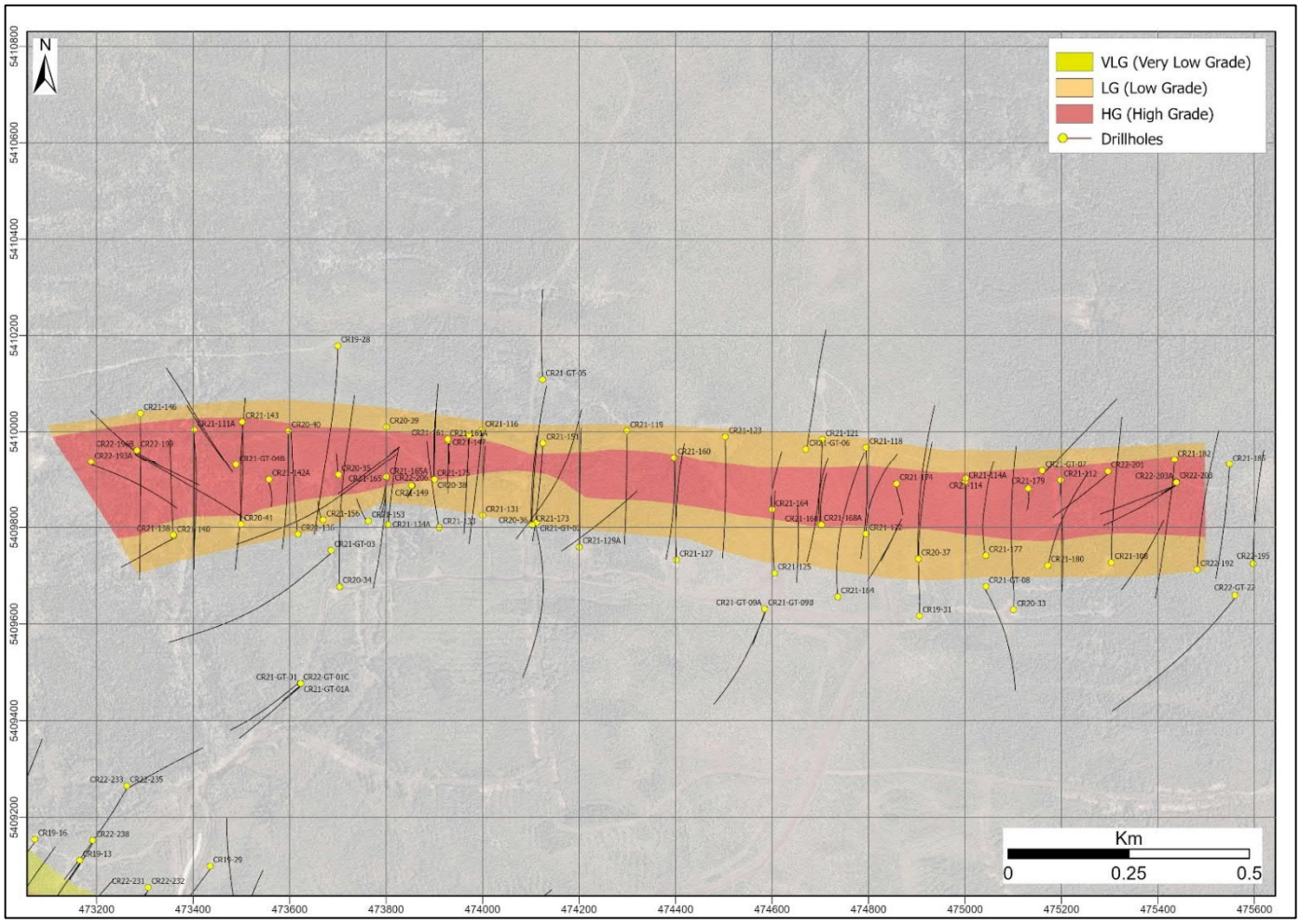
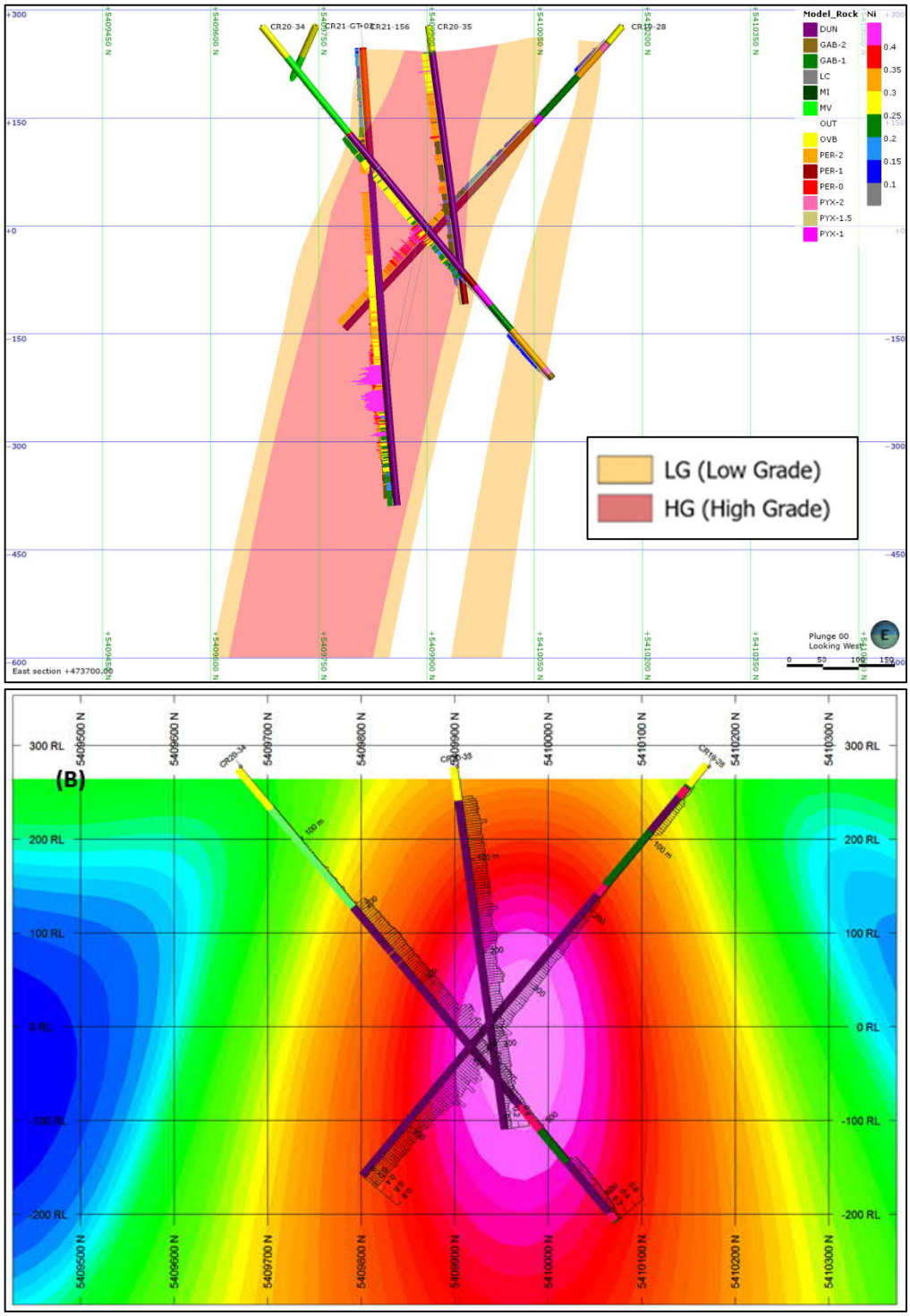


Figure 10.5: Section (473700N) looking West from East Zone showing Estimation Domains and 3D Inversion Magnetic Intensity



Note: 3D-inversion magnetic intensity linear colour transform from low (blue) to high (red) magnetic field); histogram scale is %Ni. Source: Caracle Creek, 2022.

Table 10.4: East Zone: Selective Drill Core Assays, CR19, CR20, and CR21 Series of Diamond Drill Holes

Drill Hole	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	-	-

Drill Hole	From (m)	To (m)	Interval (m)	Estimated True Width (m)	Ni (%)	Co (%)	Pd (g/t)	Pt (g/t)	Pd+Pt (g/t)	S (%)	Fe (%)
CR20-39	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

Drill Hole	From (m)	To (m)	Interval (m)	Estimated True Width (m)	Ni (%)	Co (%)	Pd (g/t)	Pt (g/t)	Pd+Pt (g/t)	S (%)	Fe (%)
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
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

Drill Hole	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.	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

Note: where not estimated, core intervals are not true widths. Canada Nickel 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. Source: CNC, 2022.

10.3.3 East Zone – PGE Reefs

Within the layered ultramafic unit of the East Zone, two domains can be differentiated: (1) a high-nickel, PGE-poor domain to the south, comprising mainly dunite and peridotite, and (2) a low (to barren) nickel domain, comprising peridotite and pyroxenite, with major PGE occurrences interpreted as horizons or "reefs" proximal too the northern margin of the ultramafic body. Nine of the 11 drill holes in the East Zone intersected one or both of the two PGE reefs, with five holes intersecting both the south reef (PGE-1) and north reef (PGE-2) (Table 10.5).

Table 10.5: True Width Intercepts for Drill Holes into the Main Zone PGE Reef

Drill Hole	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	0.028	0.009	5.640	0.122
CR19-31	1.70	0.737	0.849	1.586	0.032	0.009	6.510	0.015
CR20-33	1.50	0.538	0.504	1.042	0.026	0.008	5.700	0.170
CR20-34	4.30	0.865	0.891	1.755	0.059	0.014	8.180	0.060
CR20-35	1.80	0.552	1.130	1.682	0.056	0.011	6.850	0.030
CR20-36	2.10	0.037	0.065	0.101	0.045	0.010	6.300	0.055
CR20-37	2.30	0.685	0.807	1.492	0.056	0.013	7.560	0.047
CR20-38	3.30	0.836	0.988	1.824	0.034	0.008	6.010	0.167
CR20-41	4.70	0.742	0.779	1.520	0.024	0.008	6.070	0.066
CR19-28	4.00	0.150	0.245	0.395	0.043	0.006	5.410	0.015
CR20-34	3.40	0.171	0.275	0.445	0.035	0.006	5.670	0.037
CR20-36	3.10	0.145	0.226	0.371	0.039	0.006	5.770	0.012
CR20-37	2.70	0.162	0.253	0.415	0.043	0.006	5.740	0.010
CR20-38	3.40	0.153	0.259	0.412	0.039	0.006	5.550	0.017
CR21-129A	4.8	0.621	0.660	1.28	0.03	0.009	5.81	0.07

Drill Hole	Estimated True Width (m)	Pd (g/t)	Pt (g/t)	Pd+Pt (g/t)	Ni (%)	Co (%)	Fe (%)	S (%)
including	1.9	1.240	1.280	2.52	0.02	0.008	5.48	0.00
CR21-131	1.3	0.463	0.480	0.94	0.02	0.007	5.64	0.05
CR21-133	5.3	0.673	0.693	1.37	0.02	0.007	5.81	0.12
including	1.5	1.395	1.375	2.77	0.02	0.007	5.91	0.10
CR21-134A	3.0	0.599	0.685	1.28	0.02	0.007	5.60	0.06
including	0.8	1.100	1.270	2.37	0.03	0.007	5.76	0.05
and	2.0	0.384	0.501	0.89	0.05	0.012	7.00	0.03

Source: CNC, 2022.

10.3.4 West Zone Drilling

In October 2020, CNC reported the discovery of previously unknown mineralization in four drill holes from the West Zone, with the first step out hole located about 850 m northwest of the Main Zone (see Figure 7.5). Selective drill core assay results from the West Zone are summarized in Table 10.6. What was previously referred to the as the Thumb Zone and The Knuckle Zone (*e.g.*, Ausenco, 2021) are now part of the West Zone.

Table 10.6: West Zone: Selective Drill Core Assays, CR19 and CR21 Series Diamond Drill Holes

Drill Hole	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		0.2	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	237.2	0.23	0.013	0.007	0.011	0.018	0.02	7.39
incl.	57.0	160.5	103.5	66.5	0.29	0.012	0.018	0.024	0.042	0.03	7.21
CR21-120	111.0	447.0	336.0	216.0	0.23	0.012	0.003	0.004	0.007	0.05	6.99
incl.	255.5	417.5	162.0	104.1	0.27	0.012	0.003	0.004	0.007	0.04	6.80
CR21-122	42.3	501.0	458.7	294.8	0.24	0.012	0.005	0.004	0.009	0.02	6.64
incl.	231.0	501.0	270.0	173.6	0.28	0.012	0.006	0.005	0.011	0.02	6.15
CR21-126A	39.0	546.0	507.0	290.8	0.24	0.013	0.007	0.006	0.013	0.04	7.03
incl.	186.0	510.0	324.0	185.8	0.27	0.012	0.006	0.005	0.011	0.04	6.94
CR21-128	40.9	549.0	508.1	326.6	0.24	0.013	0.017	0.012	0.029	0.05	7.27
incl.	171.5	312.5	141.0	90.6	0.26	0.012	0.005	0.004	0.009	0.05	6.99
CR21-130	32.8	558.0	525.2	337.6	0.24	0.013	0.016	0.010	0.026	0.06	7.04
incl.	322.5	355.5	33.0	21.2	0.34	0.016	0.035	0.019	0.054	0.15	7.64
CR21-132	67.0	489.0	422.0	271.3	0.24	0.014	0.016	0.015	0.031	0.02	7.38
incl.	67.0	169.5	102.5	65.9	0.29	0.013	0.052	0.033	0.085	0.07	6.96
incl.	129.0	165.0	36.0	23.1	0.36	0.013	0.125	0.077	0.202	0.12	6.97
incl.	135.0	151.5	16.5	10.6	0.43	0.013	0.104	0.018	0.122	0.15	7.16
CR21-135	67.0	564.0	497.0	319.5	0.24	0.012	0.012	0.011	0.023	0.03	7.14
incl.	67.0	275.5	208.5	134.0	0.27	0.011	0.024	0.015	0.039	0.04	6.74
CR21-137	61.0	468.0	407.0	233.4	0.21	0.013	0.003	0.006	0.009	0.02	7.18
incl.	61.0	109.5	48.5	27.8	0.25	0.013	0.003	0.005	0.008	0.02	7.15
CR21-139	46.5	465.5	419.0	240.3	0.25	0.014	0.021	0.014	0.035	0.04	7.38

Drill Hole	From (m)	To (m)	Interval (m)		Ni (%)	Co (%)	Pd (g/t)	Pt (g/t)	Pd+Pt (g/t)	S (%)	Fe (%)
incl.	46.5	208.0	161.5	92.6	0.27	0.014	0.014	0.008	0.022	0.05	7.43
CR21-141	46.5	586.5	540.0	309.7	0.22	0.012	0.003	0.005	0.008	0.04	6.79
incl.	46.5	218.5	172.0	98.7	0.25	0.012	0.004	0.005	0.009	0.05	6.30
CR21-144	60.0	369.0	309.0	130.6	0.29	0.015	0.034	0.011	0.045	0.10	7.14
incl.	152.5	369.0	216.5	91.5	0.33	0.015	0.026	0.010	0.036	0.12	7.26
incl.	152.5	222.0	69.5	29.4	0.43	0.016	0.040	0.012	0.052	0.15	7.31
CR21-145	82.0	667.5	585.5	335.8	0.21	0.013	0.009	0.010	0.019	0.06	7.29
CR21-148	49.0	505.0	456.0	261.6	0.25	0.014	0.015	0.014	0.029	0.04	7.42
incl.	328.0	396.0	68.0	39.0	0.31	0.014	0.061	0.035	0.096	0.06	7.53
AND	482.5	502.0	19.5	11.2	0.33	0.014	0.015	0.009	0.024	0.05	7.81
AND	373.0	391.0	18.0	10.3	0.42	0.015	0.018	0.010	0.028	0.13	7.49
CR21-152	63.9	535.5	471.6	270.5	0.24	0.012	0.006	0.005	0.011	0.05	7.04
incl.	262.5	477.0	214.5	123.0	0.28	0.013	0.009	0.005	0.014	0.04	7.31

Note: Core intervals are not true widths. Canada Nickel 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 a number of factors. Source: CNC, 2022.

The four holes intersected mineralized dunite (three of four holes both collared and ended in dunite), consistent with mineralization seen in the Main Zone, across a width of 800 m and strike length of 425 metres. The final 21 metres in the fourth hole intersected disseminated mineralization with sulphide blebs approximately 850 metres along strike from the westernmost portion of the Main Zone's higher-grade zone (see Figure 10.5).

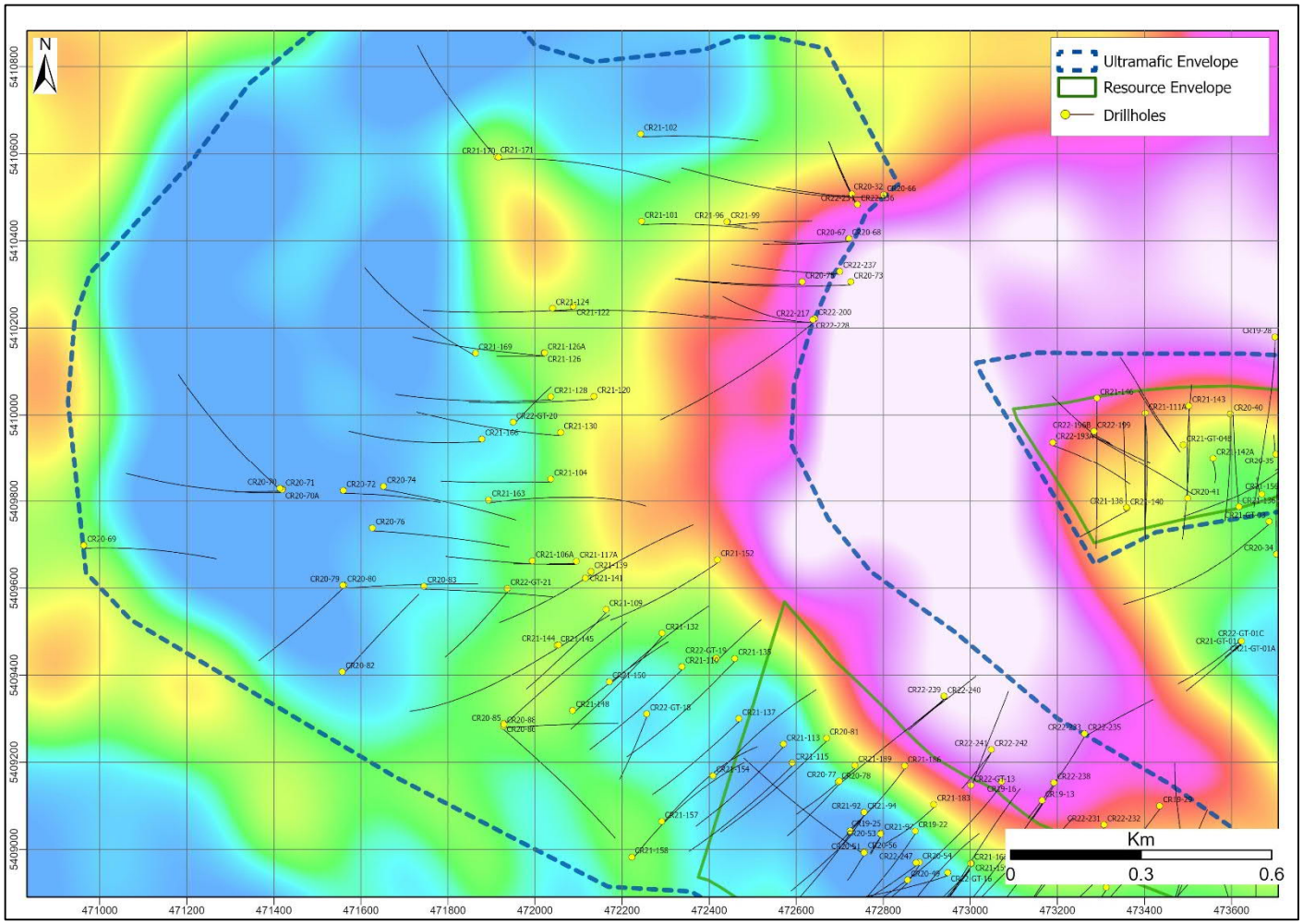
In 2019 and 2020, CNC reported six drill holes (one abandoned) from the West Zone, the interpreted northwestern extension of the Main Zone, located about 825 m west-northwest of the East Zone and about 1 km north of the west end of the Main Zone (see Figure 7.5). Selective drill core assay results from the northern West Zone are summarized in Table 10.7.

Table 10.7: Northern West Zone: Selective Drill Core Assays, CR19 Series Diamond Drill Holes

DDH	From (m)	To (m)	Interval (m)	Estimated True Width (m)	Ni (%)	Co (%)	Pd (g/t)	Pt (g/t)	Pd+Pt (g/t)	S (%)	Fe (%)
CR19-32	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. Canada Nickel 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 a number of factors. Source: CNC, 2022.

Figure 10.6: West Zone drill hole traces for 2020 to 2020 diamond drilling. The eastern extents of the Main Zone and East Zone mineral resource envelopes are also shown.



Note: basemap geophysics is Total Magnetic Intensity. Source: CNC, 2022.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Introduction

Mr. Steve Balch (P.Geo.), a qualified person as defined by NI 43-101, is responsible for the ongoing drilling and sampling program, including quality assurance (QA) and quality control (QC), together QA/QC. Since drilling began in 2019 the protocols followed by company personnel have changed slightly and are described below, current as of the current effective date of the Mineral Resource Estimate (May 19, 2022).

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 at the recommended rate in each batch of 20 samples (previously 35 samples was the threshold). 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, ON. In general, the core recovery for the diamond drill holes 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 a preliminary economic assessment.

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 drill hole 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 to the side of Highway 655 by the drilling contractor as the drilling progressed. CNC personnel transport the core to the core shack from that location. Casing is being left in the completed drill holes with the casing capped 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 that 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, 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 drill hole 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. The logging geologist determines if additional SG measurements need to be made. The geotechnician writes the SG

measurement directly on the core that was measured. Core is stored sequentially, hole by hole, in racks ahead of the logging process.

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 MS Excel spreadsheet. Currently, core logging is done directly into an MS Access based logging system and the geotechnical logging into MS Excel then uploaded into the MS Access database. 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 drill hole.

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. 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 then the larger bag secured with a cable tie. CNC personnel are responsible for transporting the samples to the Actlabs Timmins analytical facility, a driving distance of approximately 3 km from the core current shack location or for loading the transport truck.

11.4 Analytical

Activation Laboratories Ltd., a geochemical services company accredited to international standards, with assay lab ISO 17025 certification, certification to ISO 9001:2008 and CAN-P-1579 (Mineral Analysis), was used for the majority 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., 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 lab or "GQ" for the lab in Lima.

Actlabs and SGS are both independent of Canada Nickel, Noble and Spruce Ridge.

Platinum group elements (PGEs) 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 labs 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.

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 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).

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-2O2	0.01	%
Pt	FA-ICP	5	ng/g (ppb)	Mg	FUS-Na-2O2	0.01	%
Pd	FA-ICP	5	ng/g (ppb)	Mn	FUS-Na-2O2	0.01	%
Al	FUS-Na-2O2	0.01	%	Ni	FUS-Na-2O2	0.005	%
As	FUS-Na-2O2	0.01	%	Pb	FUS-Na-2O2	0.01	%
Be	FUS-Na-2O2	0.001	%	S	FUS-Na-2O2	0.01	%
Ca	FUS-Na-2O2	0.01	%	Sb	FUS-Na-2O2	0.01	%
Co	FUS-Na-2O2	0.002	%	Si	FUS-Na-2O2	0.01	%
Cr	FUS-Na-2O2	0.01	%	Ti	FUS-Na-2O2	0.01	%
Cu	FUS-Na-2O2	0.005	%	W	FUS-Na-2O2	0.005	%
Fe	FUS-Na-2O2	0.05	%	Zn	FUS-Na-2O2	0.01	%
K	FUS-Na-2O2	0.1	%				

Notes: FA-ICP=fire assay with ICP-OES finish. FUS-Na_2O_2=sodium peroxide fusion digestion with ICP-OES finish. %= per cent 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-2O2	0.001	%
Pt	FA-ICP	10	ng/g (ppb)	Mg	FUS-Na-2O2	0.01	%
Pd	FA-ICP	5	ng/g (ppb)	Mn	FUS-Na-2O2	0.001	%
Al	FUS-Na-2O2	0.01	%	Ni	FUS-Na-2O2	0.001	%
As	FUS-Na-2O2	0.003	%	Pb	FUS-Na-2O2	0.002	%
Be	FUS-Na-2O2	0.0005	%	S	FUS-Na-2O2	0.01	%
Ca	FUS-Na-2O2	0.1	%	S	IR	0.005	%
Co	FUS-Na-2O2	0.001	%	Sb	FUS-Na-2O2	0.005	%
Cr	FUS-Na-2O2	0.001	%	Si	FUS-Na-2O2	0.1	%
Cu	FUS-Na-2O2	0.001	%	Ti	FUS-Na-2O2	0.01	%
Fe	FUS-Na-2O2	0.01	%	W	FUS-Na-2O2	0.005	%
K	FUS-Na-2O2	0.1	%	Zn	FUS-Na-2O2	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. %= per cent by weight

1.5 QA/QC – Control Samples

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

A total of 52,452 samples have been submitted for analysis by CNC since the start of the Project. This includes 38,409 samples associated with the Main and West Zones (MWZ) of which 5,527 (14.2%) were for QA/QC purposes and 14,043 samples associated with the East Zone (EZ) of which 2,563 (18.3%) were for QA/QC purposes. CNC originally (as previously stated, starting with drill hole CR19-11) included QA/QC samples at the approximate rate of three samples per batch of 35 samples shipped to the lab (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 also carry out duplicate and replicate ("preparation split") analyses within each sample batch as part of their 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 (6) different samples of CRM into the sample stream: OREAS 683 (PGE ore; MWZ 9 samples, EZ 6 samples), OREAS 70b (nickel sulphide ore; MWZ 523 samples, EZ 377 samples), OREAS 70p (altered ultramafic; MWZ 482 samples, EZ 88 samples), OREAS 72a (nickel sulphide ore; MWZ 288 samples, EZ 93 samples), OREAS 72b (nickel sulphide ore; MWZ 54 samples, EZ 67 samples) ,and OREAS 74a (nickel sulphide ore; EZ 9 samples).

CNC introduced 2,021 samples of blank material (variously "blank gravel", "blank FP", "blank sand' or "blank silica") into the sample stream; 1,383 were related to the MWZ sample submissions and 638 were related to the EZ.

Although CNC did not quarter core sample intervals to generate "sampling" or "field" duplicates or "replicate" samples in order to evaluate the reproducibility of the sampling procedures, CNC did submit 320 core pulp samples to a referee lab, Bureau Veritas Commodities Canada Ltd. (see Section 11.3.2.4).

11.5 QA/QC - Data Verification

11.5.1 Certified Reference Material

Certified reference materials are used by CNC to monitor the accuracy of the analyses performed by Actlabs and SGS. A number of different reference materials for different combinations of elements were used during the course of 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 and OREAS 72a; they report 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. There is no appreciable difference for samples associated with MWZ compared to those associated with EZ. Examples of the CRM responses are shown in Figures 11-1 to 11-32.

Figure 11.1: CRM OREAS 70b – Number of Standard Deviations Difference for Ni Analysis from the Certified Value for Various Analytical Runs – Main and West Zones

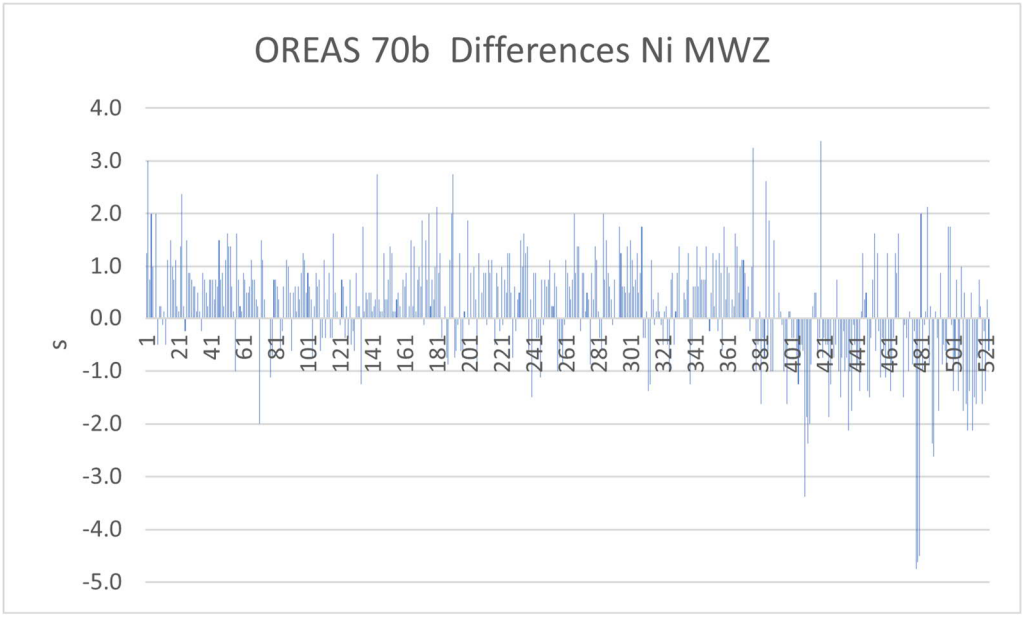


Figure 11.2: CRM OREAS 70b – Distribution of Standard Deviations Difference for Ni Analysis from the Certified Value – Main and West Zones

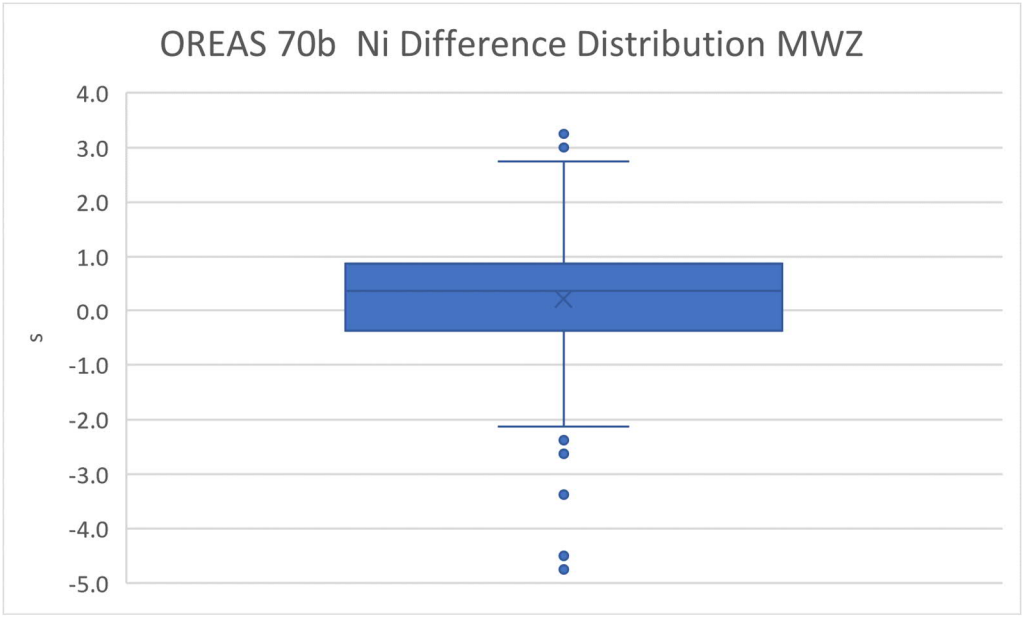


Figure 11.3: CRM OREAS 70b – Number of Standard Deviations Difference for Ni Analysis from the Certified Value for Various Analytical Runs – East Zone

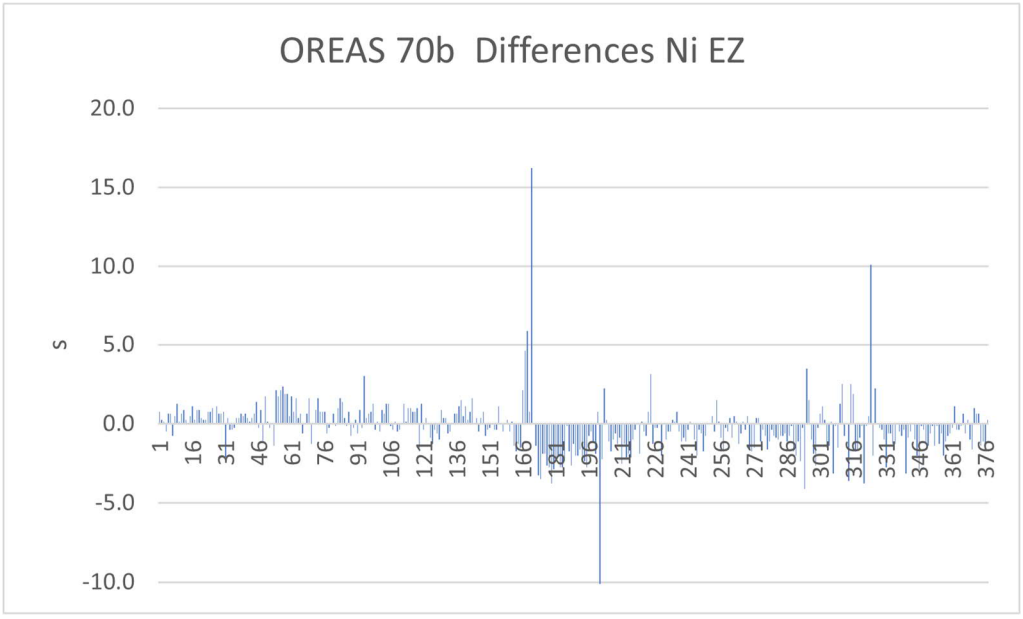


Figure 11.4: CRM OREAS 70b – Distribution of Standard Deviations Difference for Ni Analysis from the Certified Value – East Zone

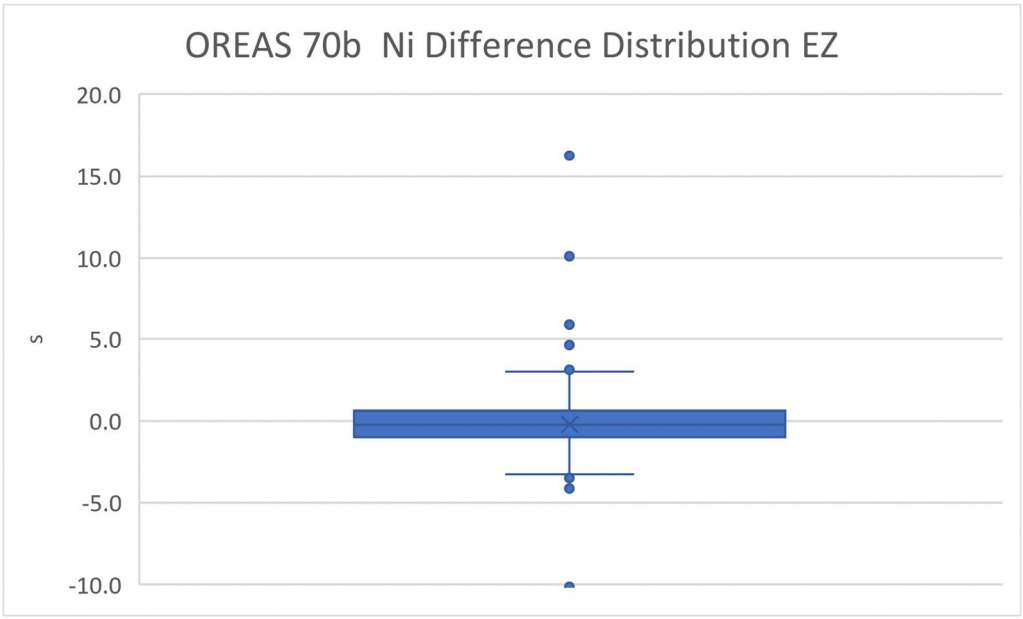


Figure 11.5: CRM OREAS 70b – Number of Standard Deviations Difference for Co Analysis from the Certified Value for Various Analytical Runs – Main and West Zones

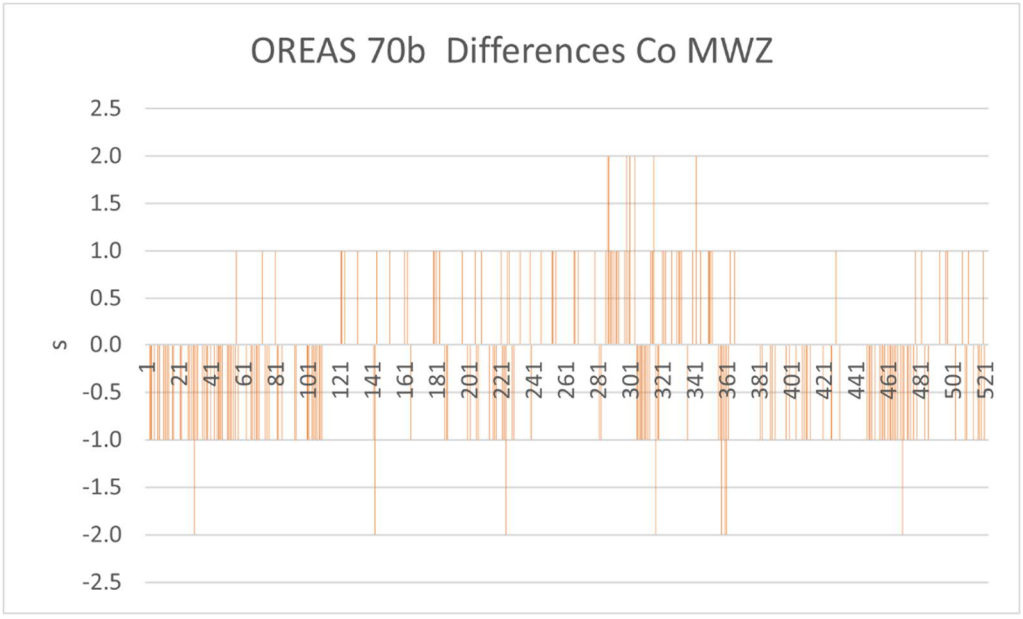


Figure 11.6: CRM OREAS 70b – Distribution of Standard Deviations Difference for Co Analysis from the Certified Value – Main and West Zones

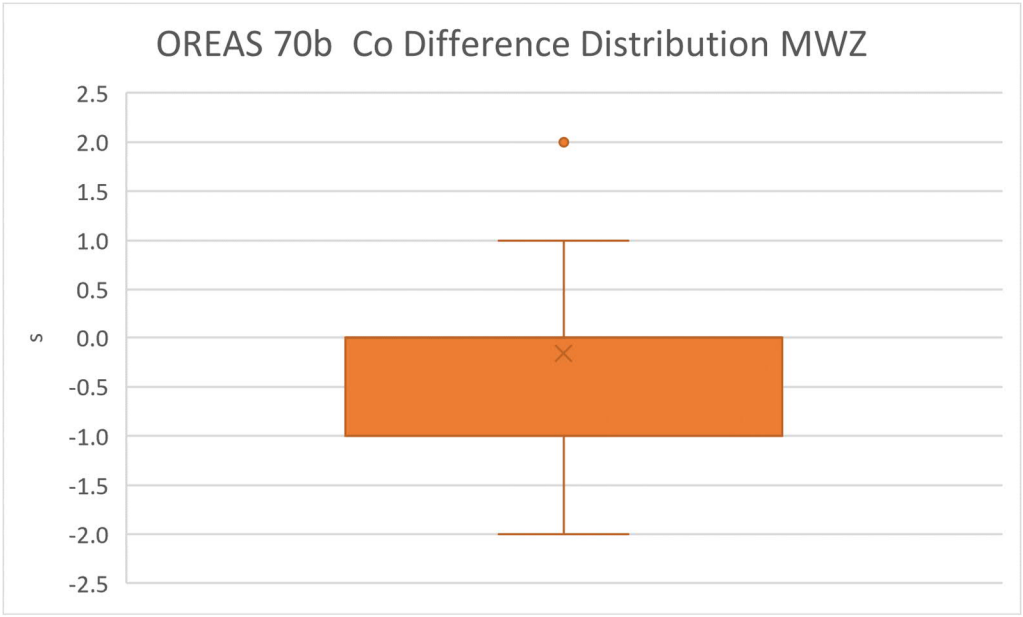


Figure 11.7: CRM OREAS 70b – Number of Standard Deviations Difference for Co Analysis from the Certified Value for Various Analytical Runs – East Zone

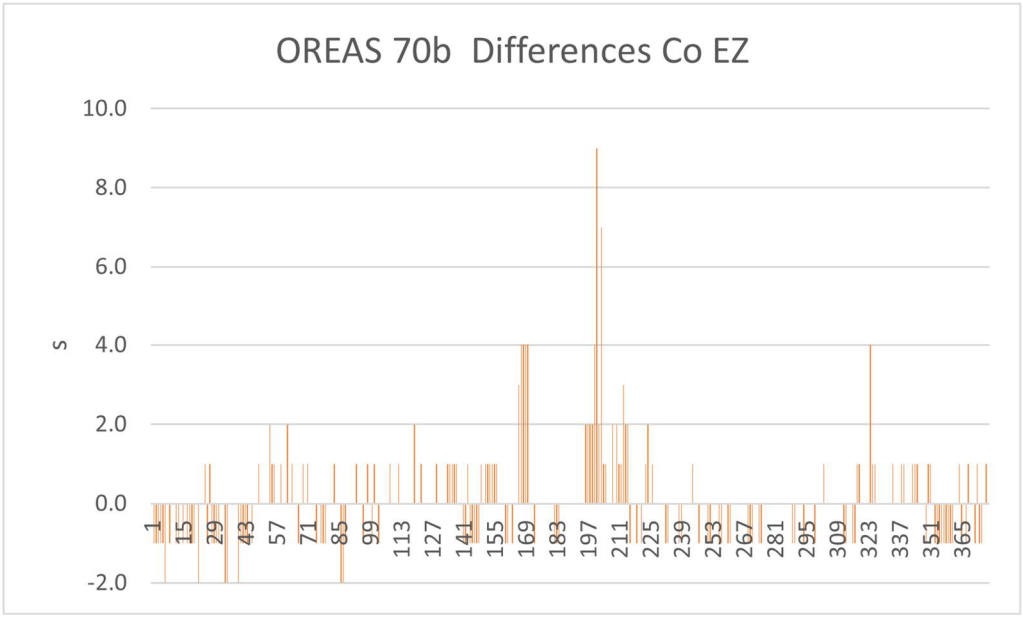


Figure 11.8: CRM OREAS 70b – Distribution of Standard Deviations Difference for Co Analysis from the Certified Value – East Zone

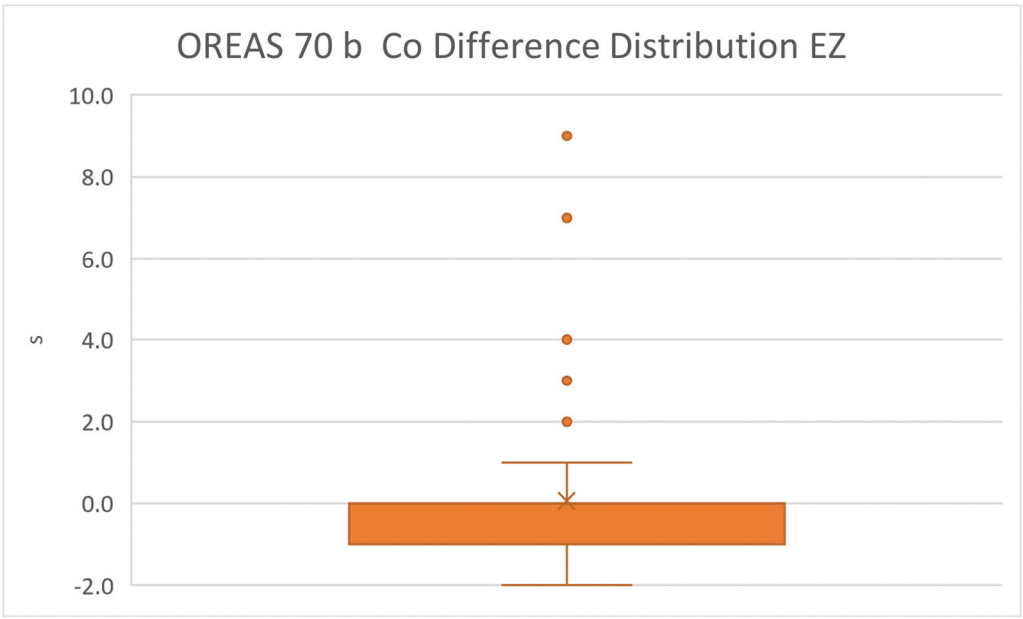


Figure 11.9: CRM OREAS 70b – Distribution of Standard Deviations Difference for Cr Analysis from the Certified Value – Main and West Zones

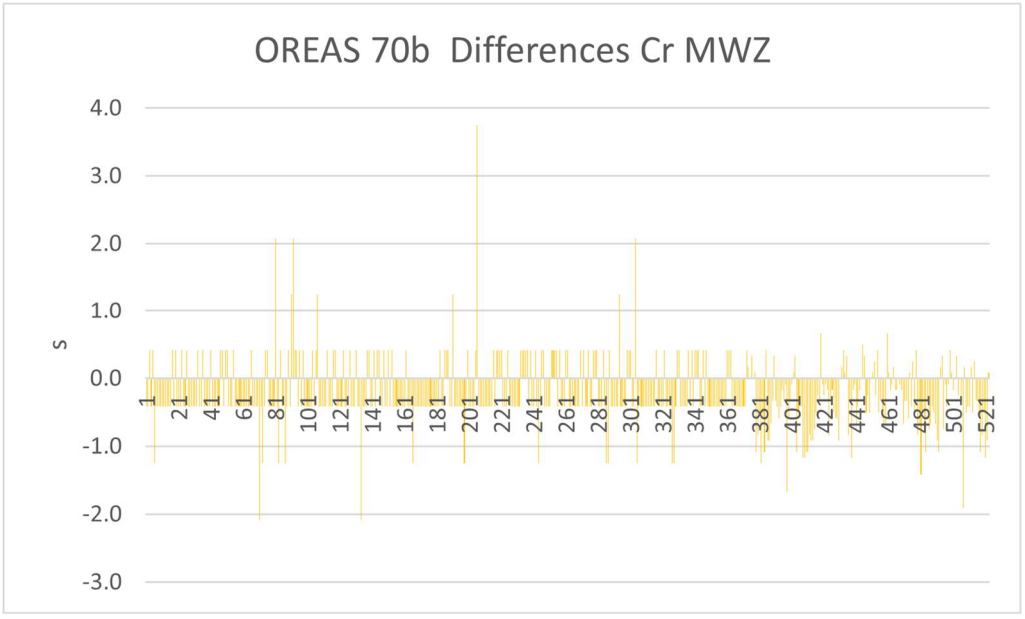


Figure 11.10: CRM OREAS 70b – Distribution of Standard Deviations Difference for Cr Analysis from the Certified Value – Main and West Zones

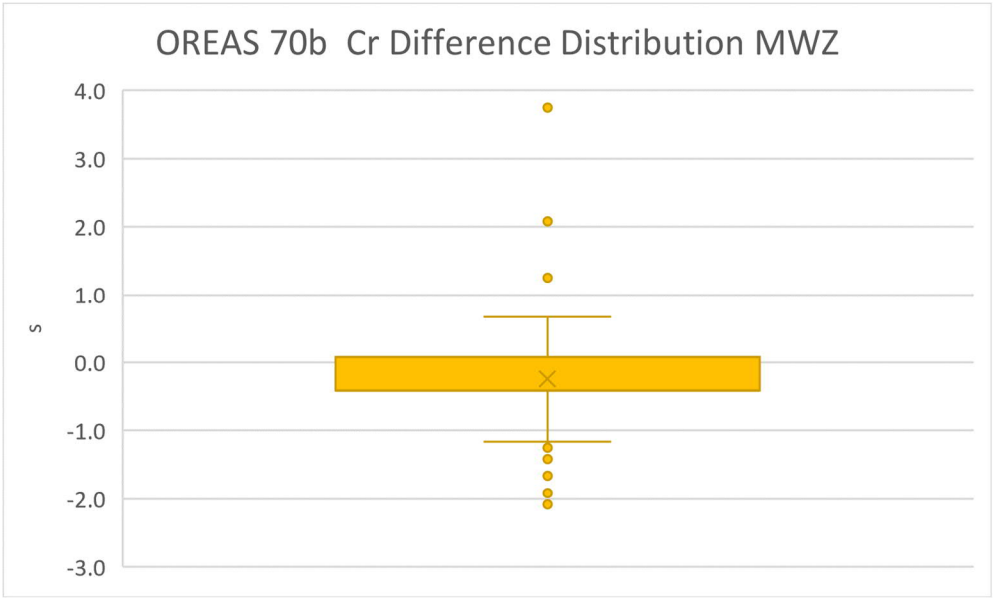


Figure 11.11: CRM OREAS 70b – Distribution of Standard Deviations Difference for Cr Analysis from the Certified Value – East Zone

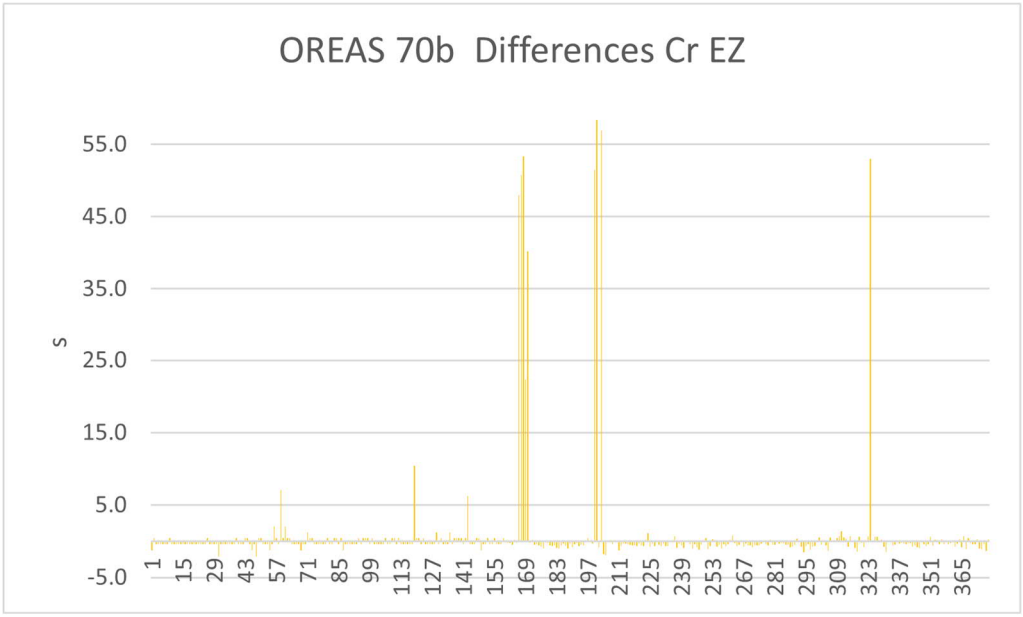


Figure 11.11: CRM OREAS 70b – Distribution of Standard Deviations Difference for Cr Analysis from the Certified Value – East Zone

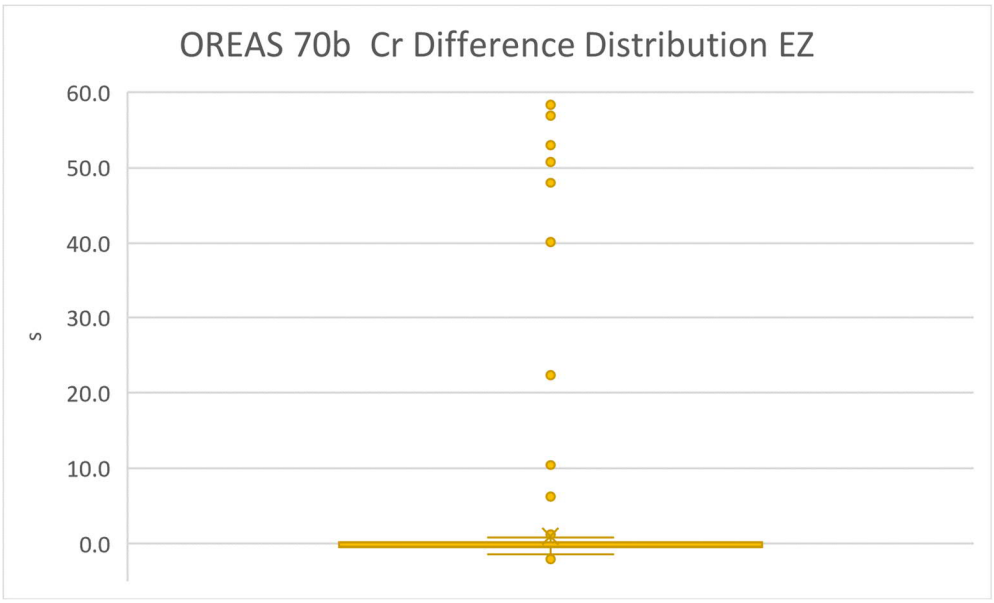


Figure 11.123: CRM OREAS 70b – Number of Standard Deviations Difference for S Analysis from the Certified Value for Various Analytical Runs – Main and West Zones

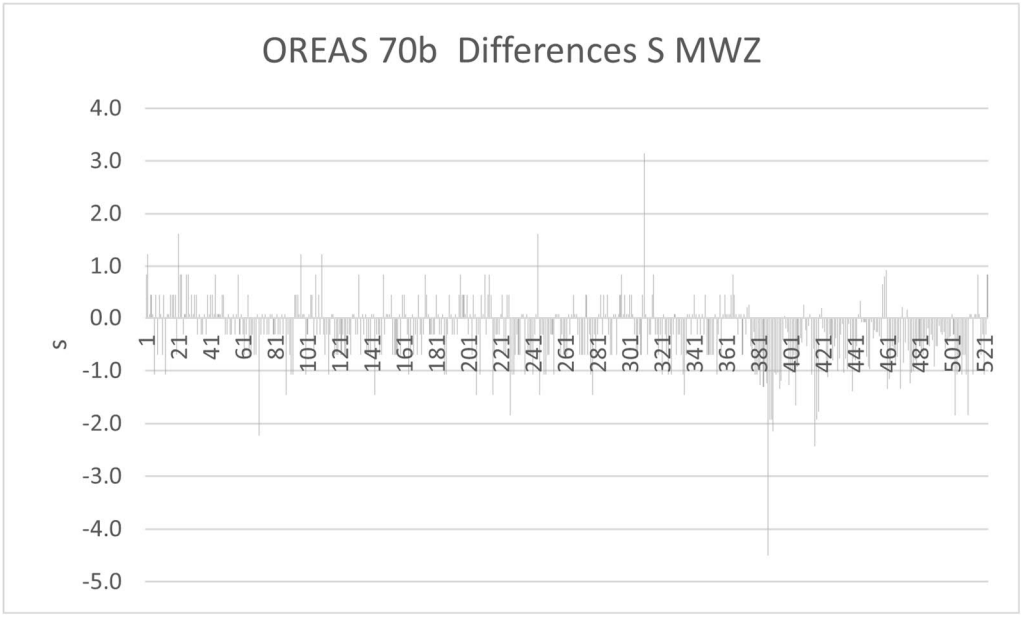


Figure 11.14: CRM OREAS 70b – Distribution of Standard Deviations Difference for S Analysis from the Certified Value – Main and West Zones

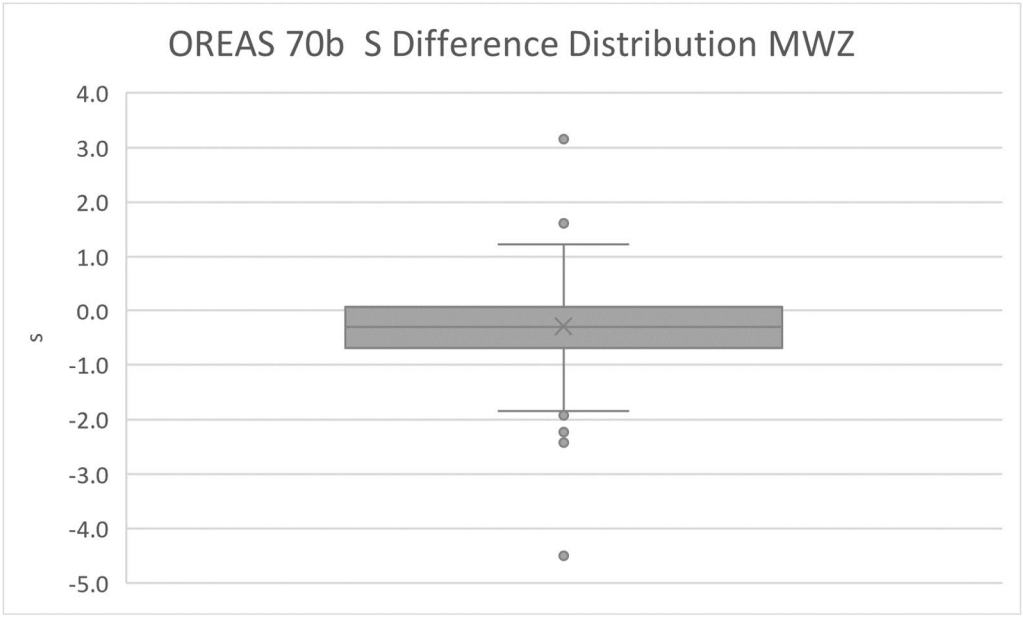


Figure 11.15: CRM OREAS 70b – Number of Standard Deviations Difference for S Analysis from the Certified Value for Various Analytical Runs – East Zone

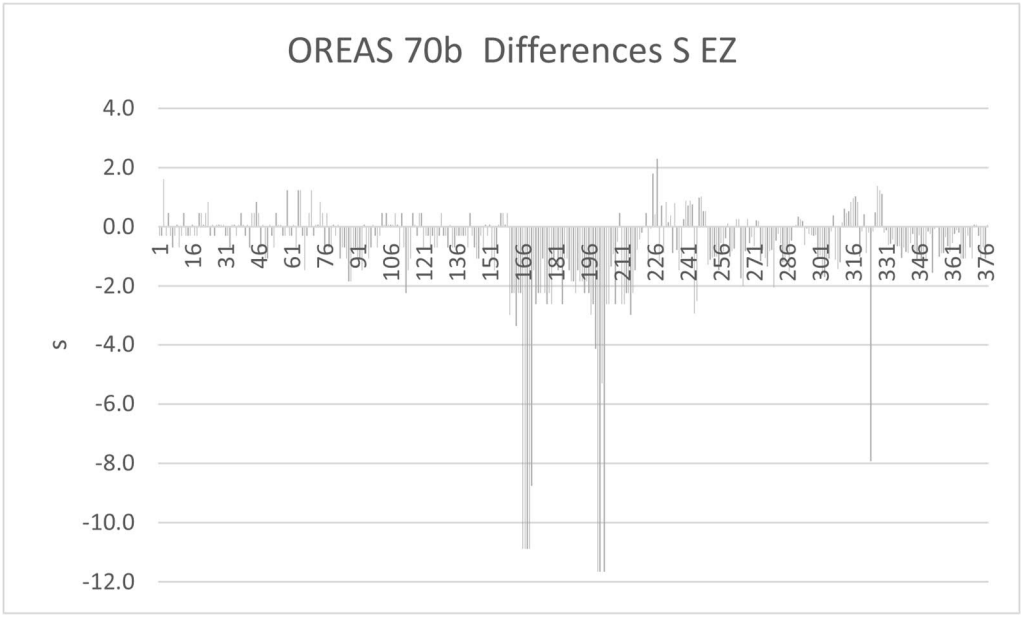


Figure 11.16: CRM OREAS 70b – Distribution of Standard Deviations Difference for S Analysis from the Certified Value – East Zone

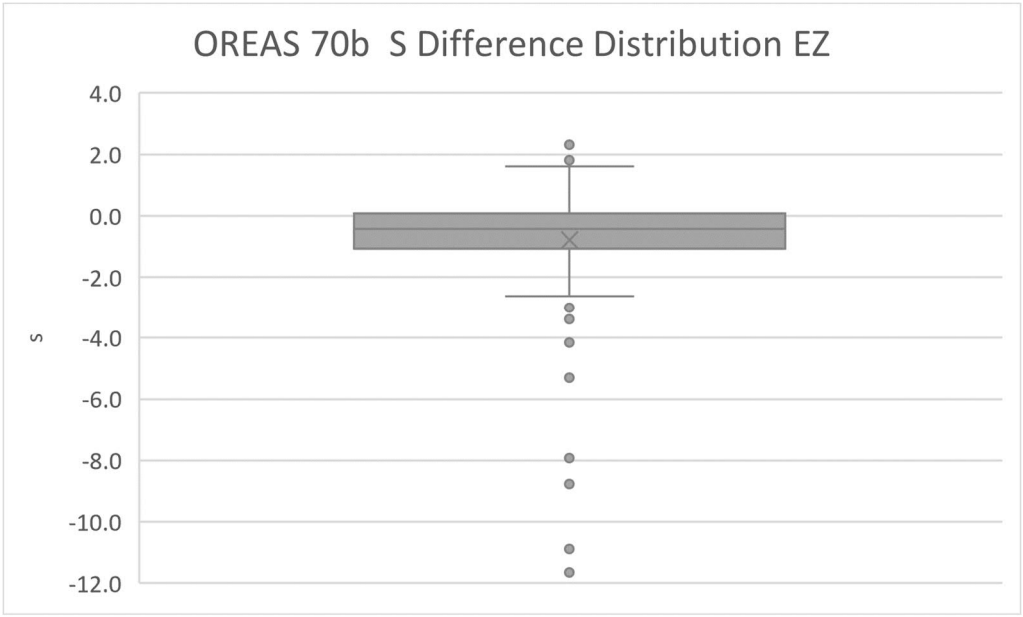


Figure 11.137: CRM OREAS 70b – Number of Standard Deviations Difference for Fe Analysis from the Certified Value for Various Analytical Runs – Main and West Zones

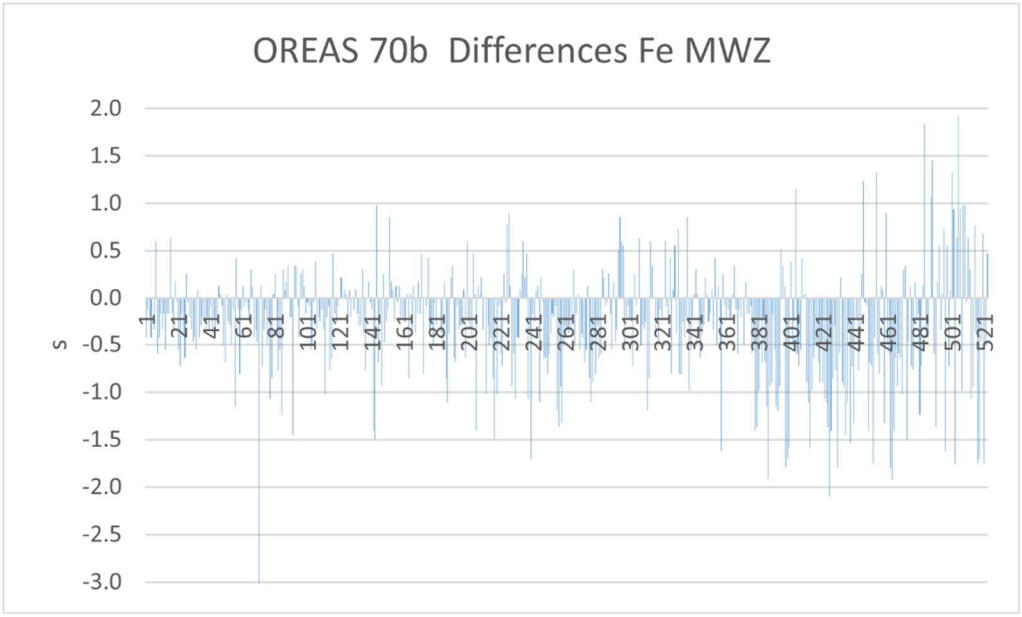


Figure 11.18: CRM OREAS 70b – Distribution of Standard Deviations Difference for Fe Analysis from the Certified Value – Main and West Zones

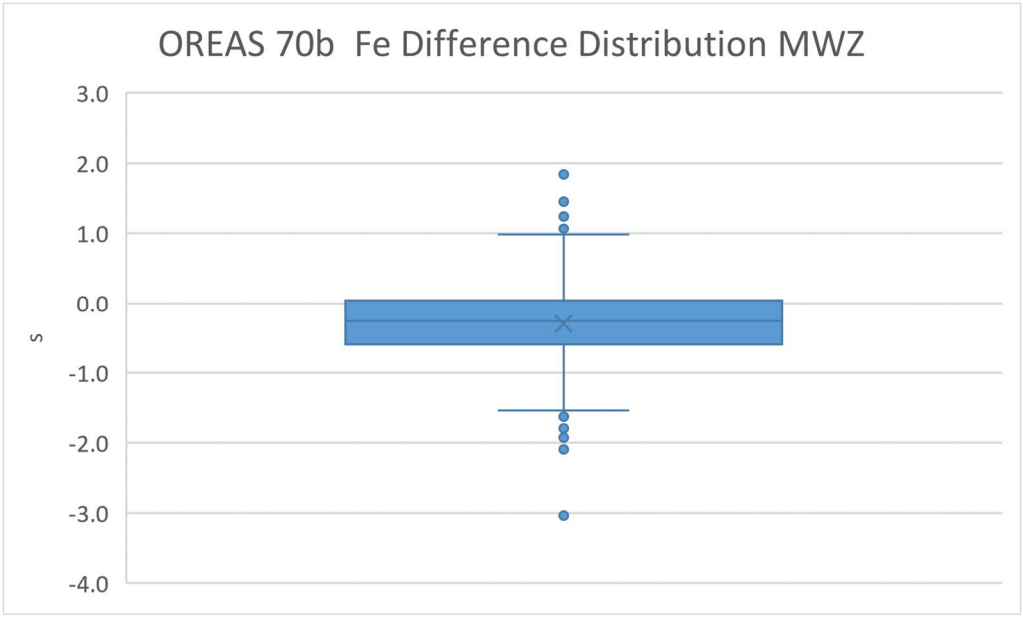


Figure 11.19: CRM OREAS 70b – Number of Standard Deviations Difference for Fe Analysis from the Certified Value for Various Analytical Runs – East Zone

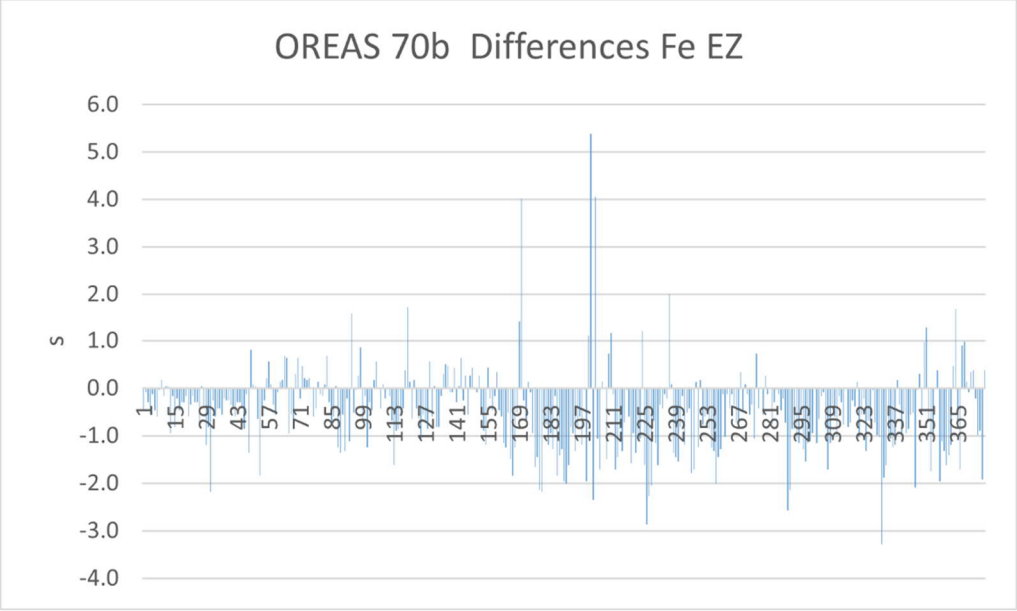


Figure 11.20: CRM OREAS 70b – Distribution of Standard Deviations Difference for Fe Analysis from the Certified Value – East Zone

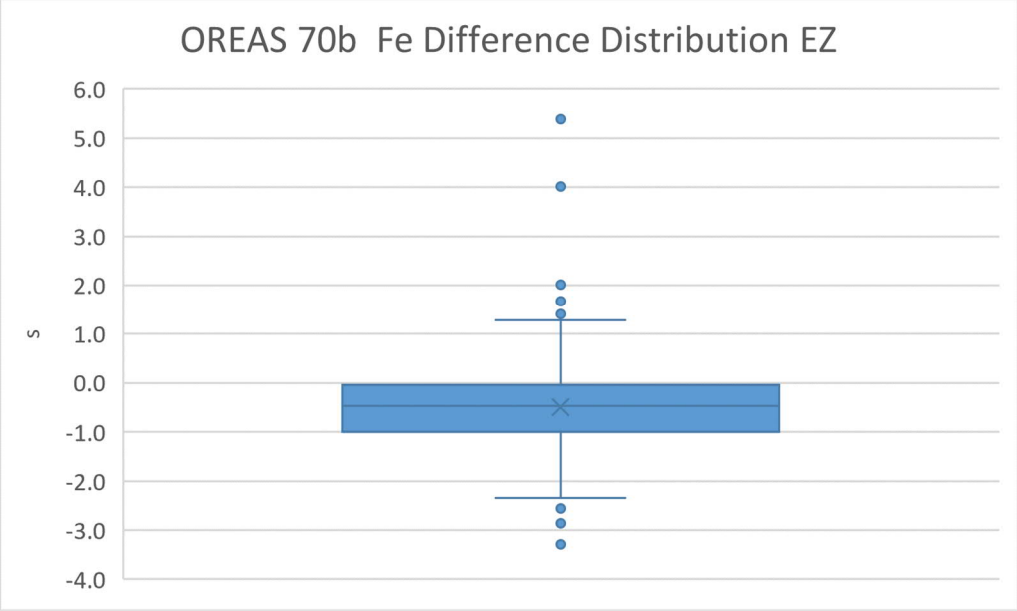


Figure 11.21: CRM OREAS 72a – Number of Standard Deviations Difference for Au Analysis from the Certified Value for Various Analytical Runs – Main and West Zones

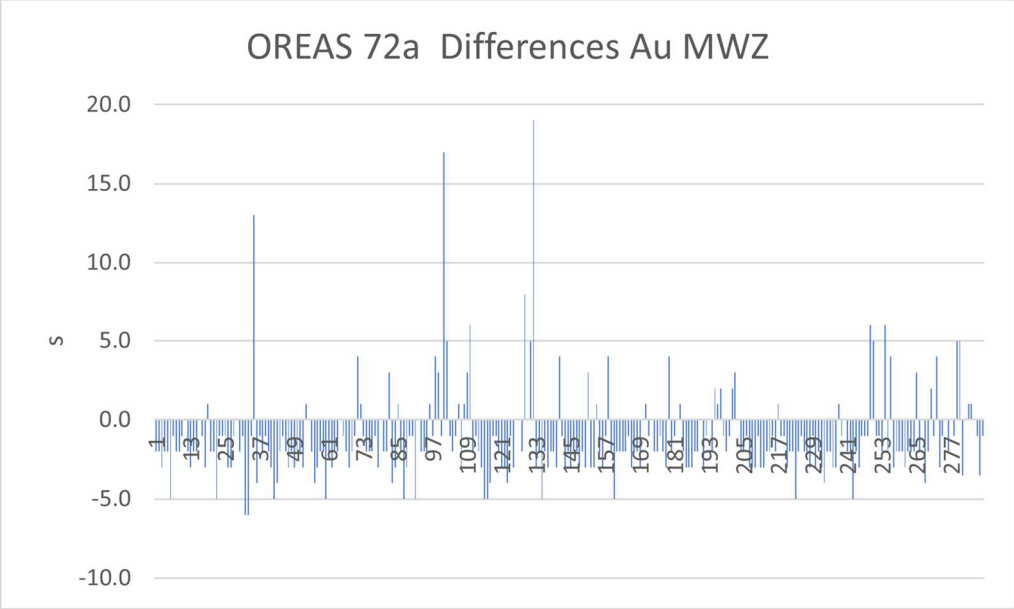


Figure 11.22: CRM OREAS 72a – Distribution of Standard Deviations Difference for Au Analysis from the Certified Value – Main and West Zones

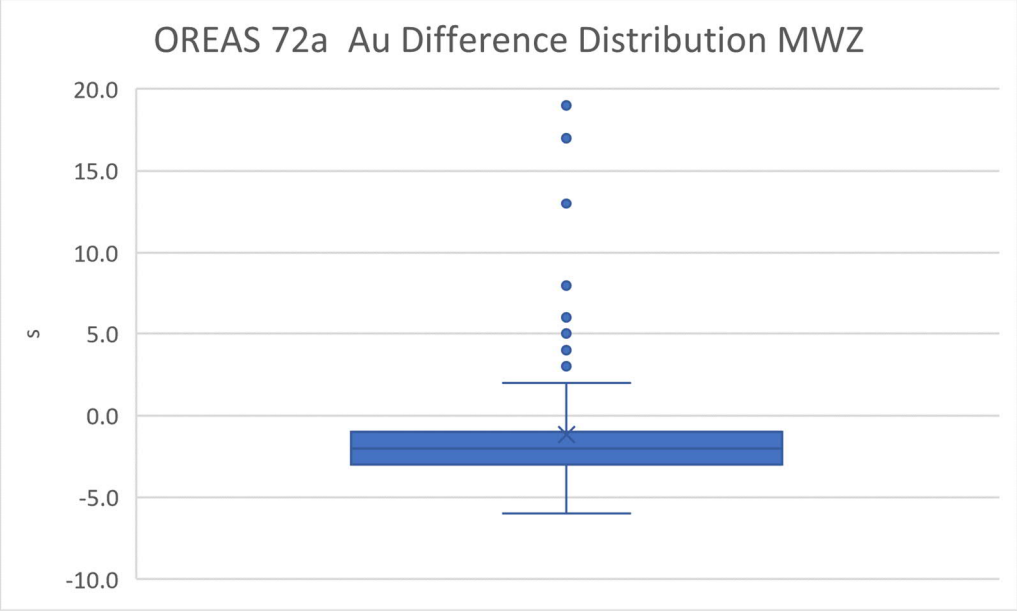


Figure 11.23: CRM OREAS 72a – Number of Standard Deviations Difference for Au Analysis from the Certified Value for Various Analytical Runs – East Zone

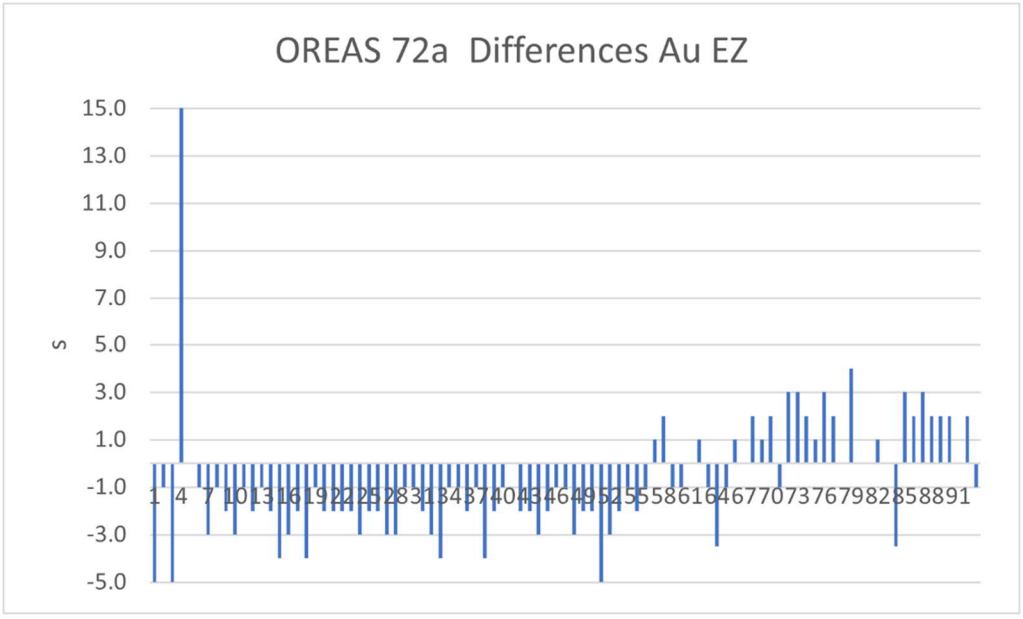


Figure 11.24: CRM OREAS 72a – Distribution of Standard Deviations Difference for Au Analysis from the Certified Value – East Zone

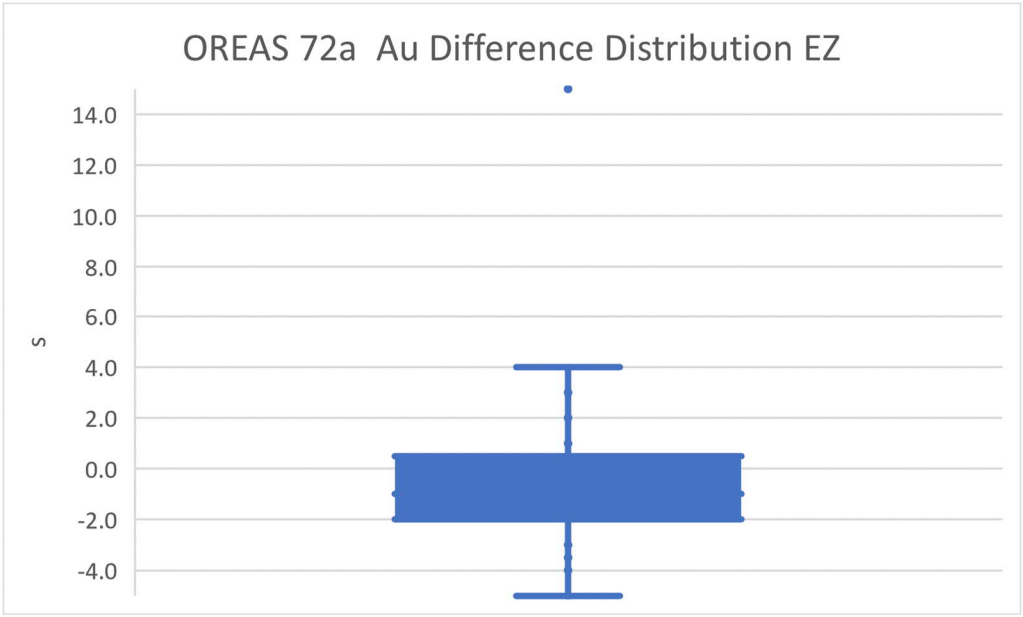


Figure 11.25: CRM OREAS 72ab – Number of Standard Deviations Difference for Pd Analysis from the Certified Value for Various Analytical Runs – Main and West Zones

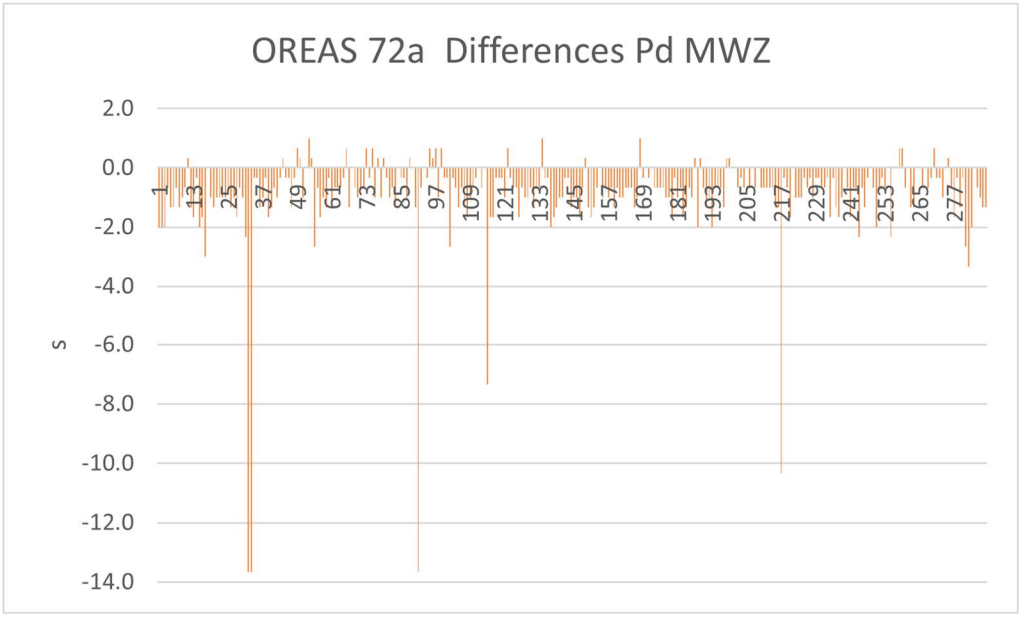


Figure 11.26: CRM OREAS 72a – Distribution of Standard Deviations Difference for Pd Analysis from the Certified Value – Main and West Zones

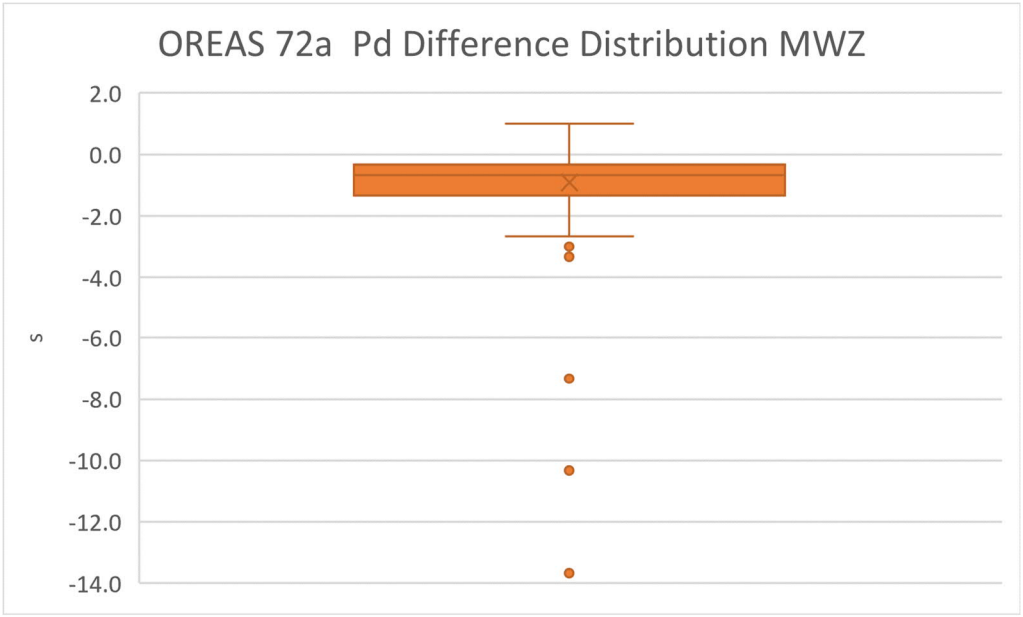


Figure 11.27: CRM OREAS 72a – Number of Standard Deviations Difference for Pd Analysis from the Certified Value for Various Analytical Runs – East Zone

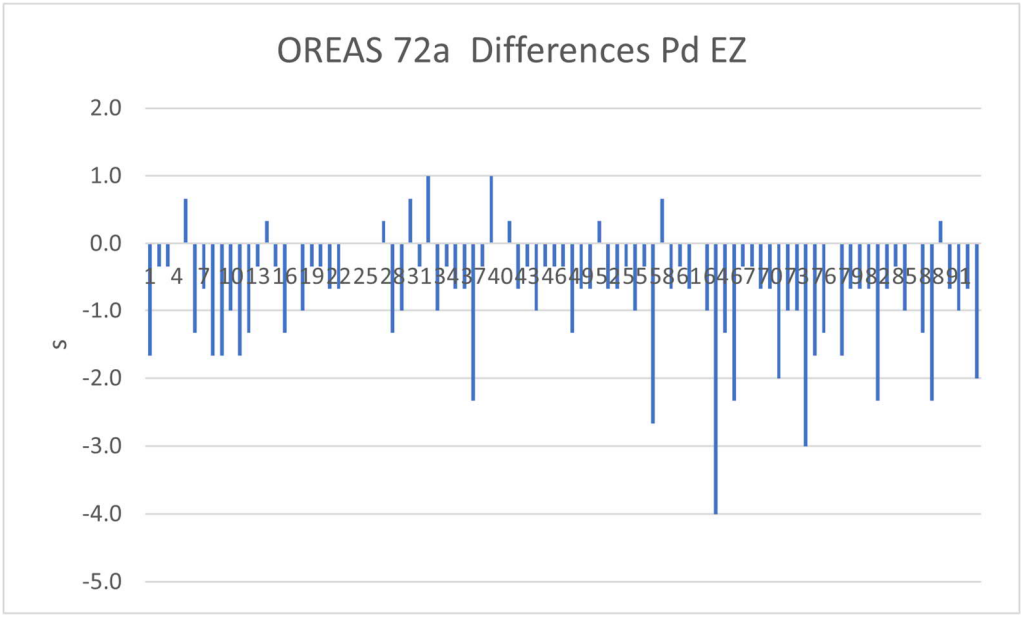


Figure 11.28: CRM OREAS 72a – Distribution of Standard Deviations Difference for Pd Analysis from the Certified Value – East Zone

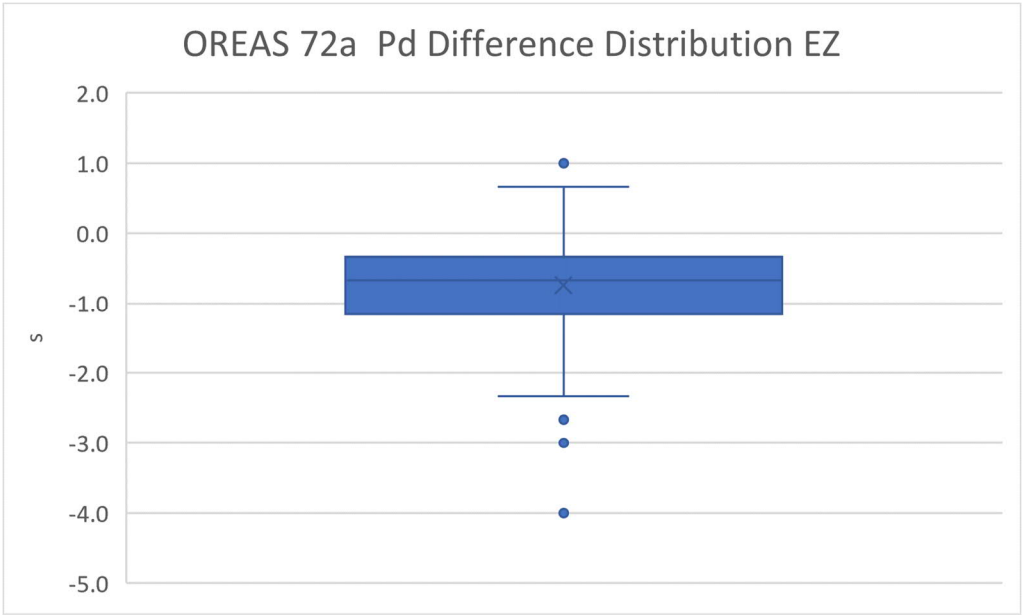


Figure 11.29: CRM OREAS 72a – Number of Standard Deviations Difference for Pt Analysis from the Certified Value for Various Analytical Runs – Main and West Zones

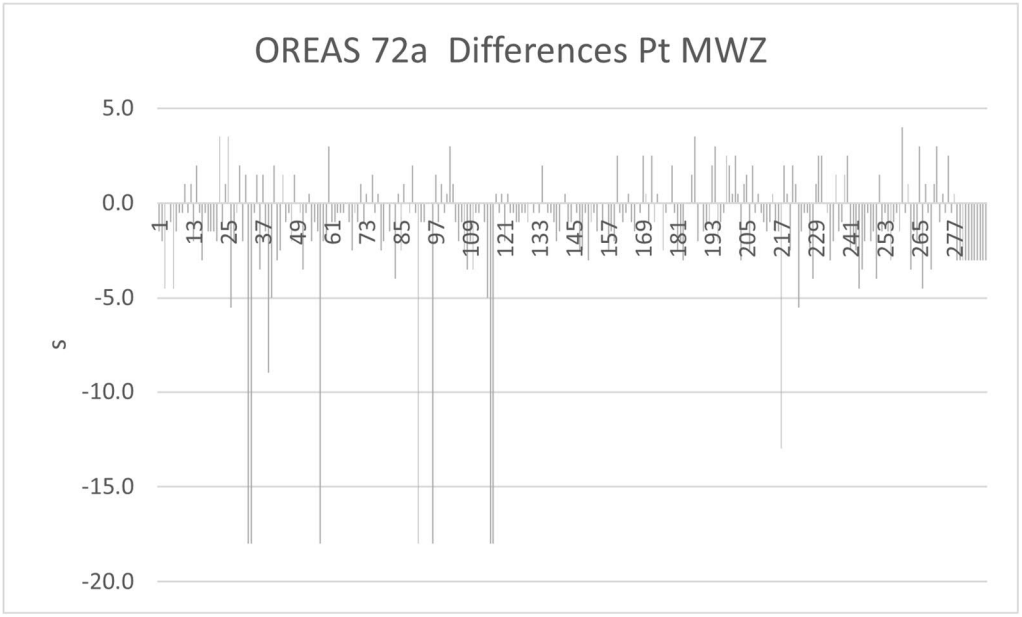


Figure 11.30: CRM OREAS 72a – Distribution of Standard Deviations Difference for Pt Analysis from the Certified Value – Main and West Zones

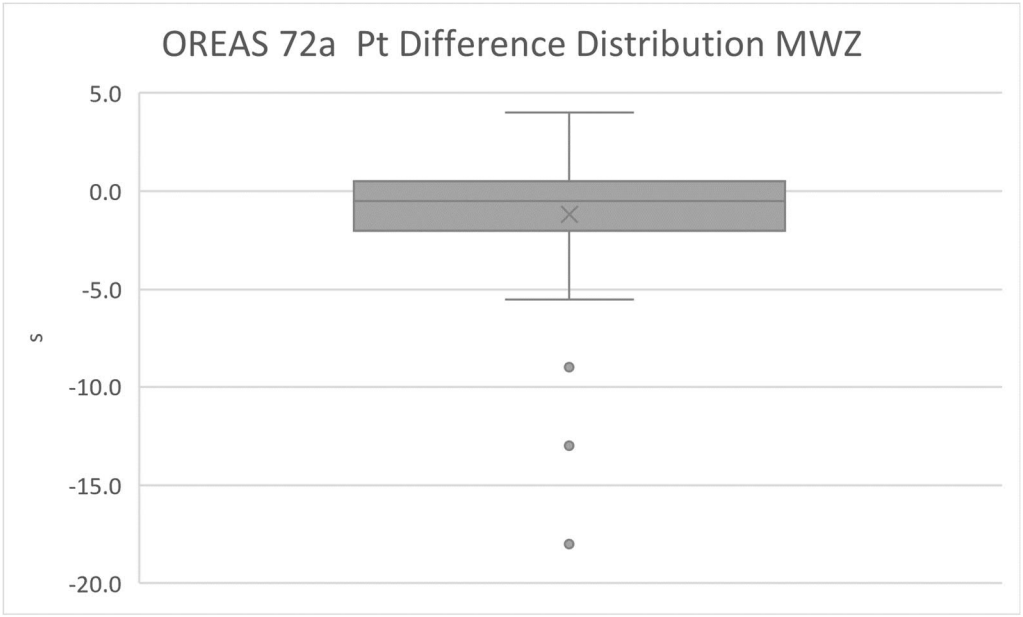


Figure 11.31: CRM OREAS 72a – Number of Standard Deviations Difference for Pt Analysis from the Certified Value for Various Analytical Runs – East Zone

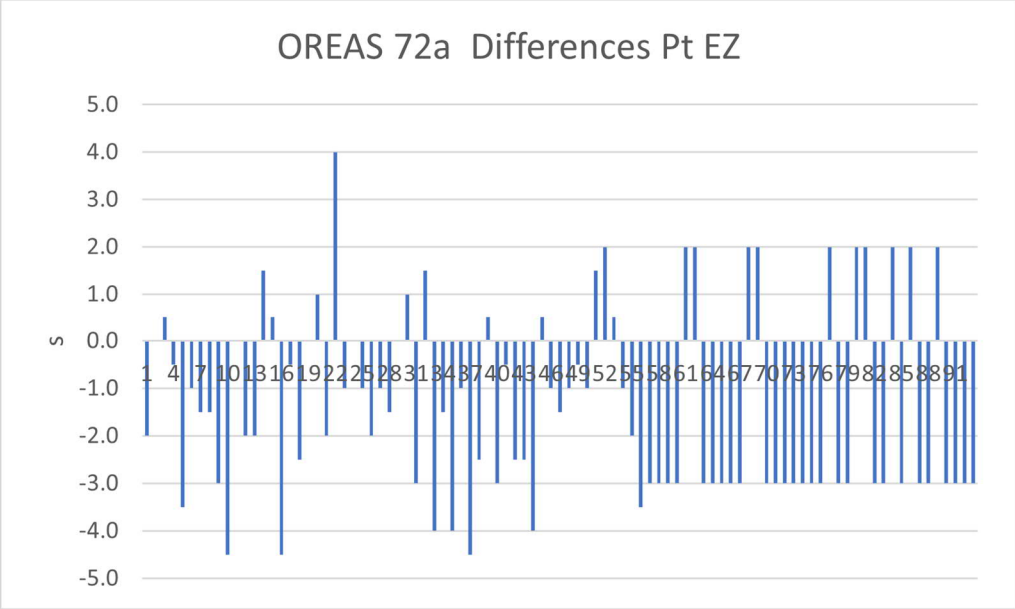
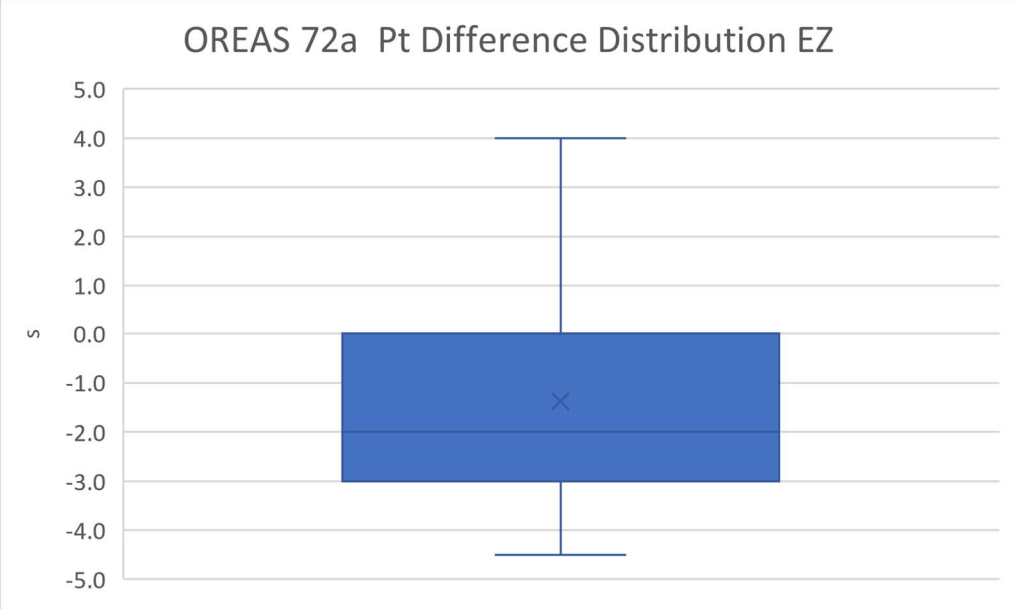


Figure 11.32: CRM OREAS 72a – Distribution of Standard Deviations Difference for Pt Analysis from the Certified Value – East Zone



11.5.2 Duplicate Samples – Analytical Duplicates

CNC did not regularly add any duplicate material to the sample stream. A re-analysis of a prepared pulp was requested at a frequency of one in 20. In general, the duplicate material for the platinum group metals analyses has indicated good reproducibility of the assays as demonstrated by project examples in Figures 11-33 to 11-40.

Figure 11.14 Relative % Difference of Pairs of Duplicate Samples Analyzed for Pd

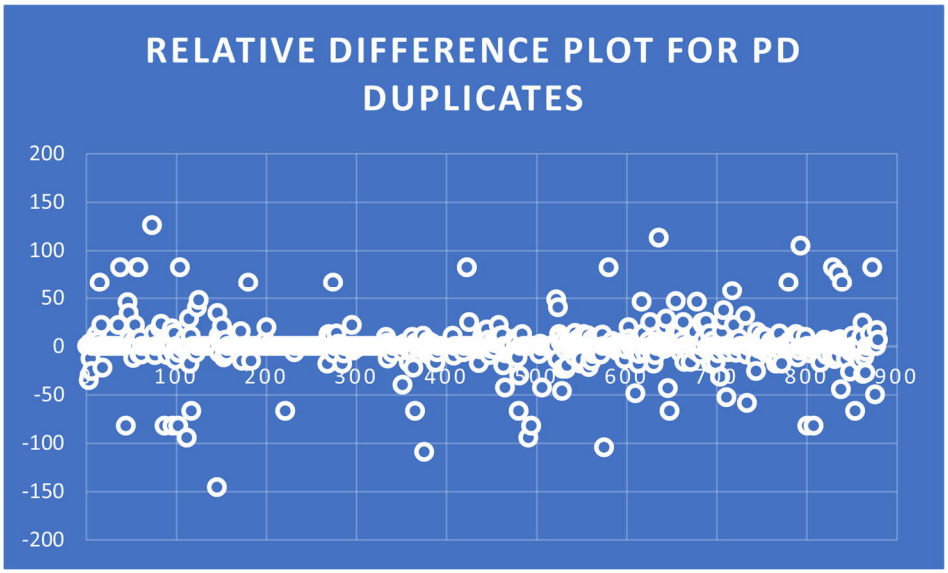


Figure 11.34: Relative % Difference of Pairs of Duplicate Samples Analyzed for Pd vs. the Absolute Concentration of the Original Analysis

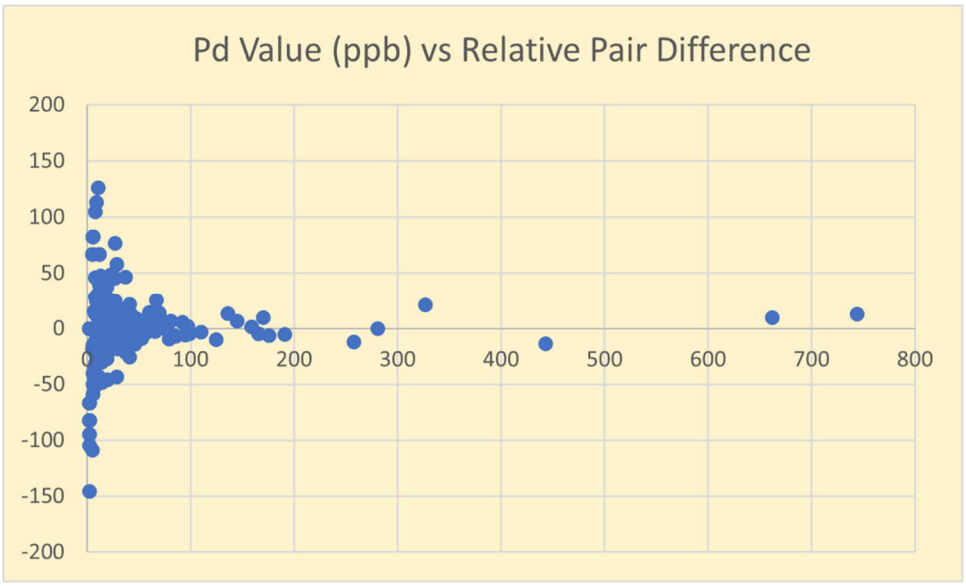


Figure 11.35: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Pd

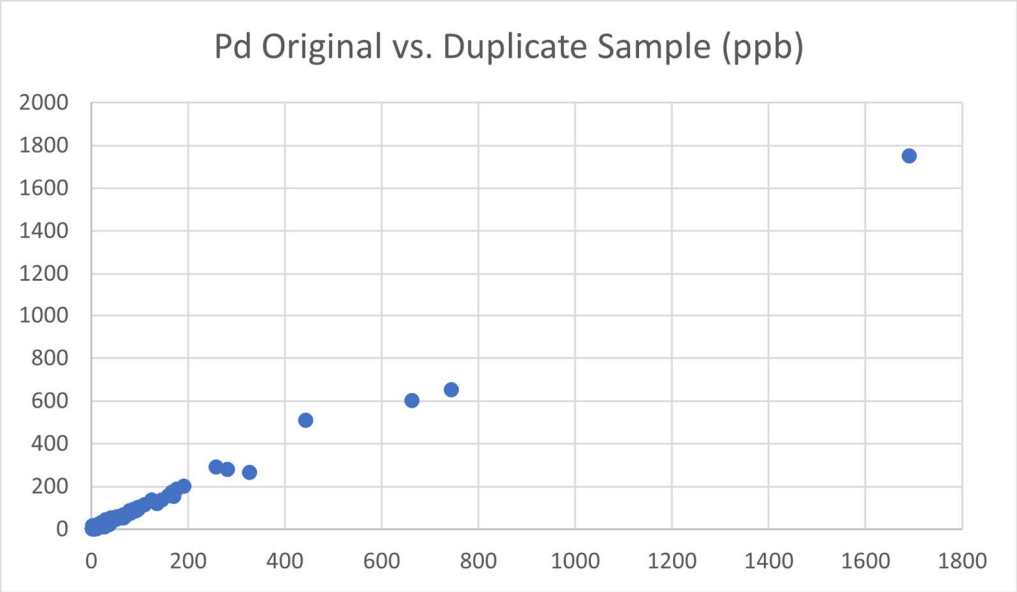


Figure 11.36: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Pd to a Maximum of 800 ppb

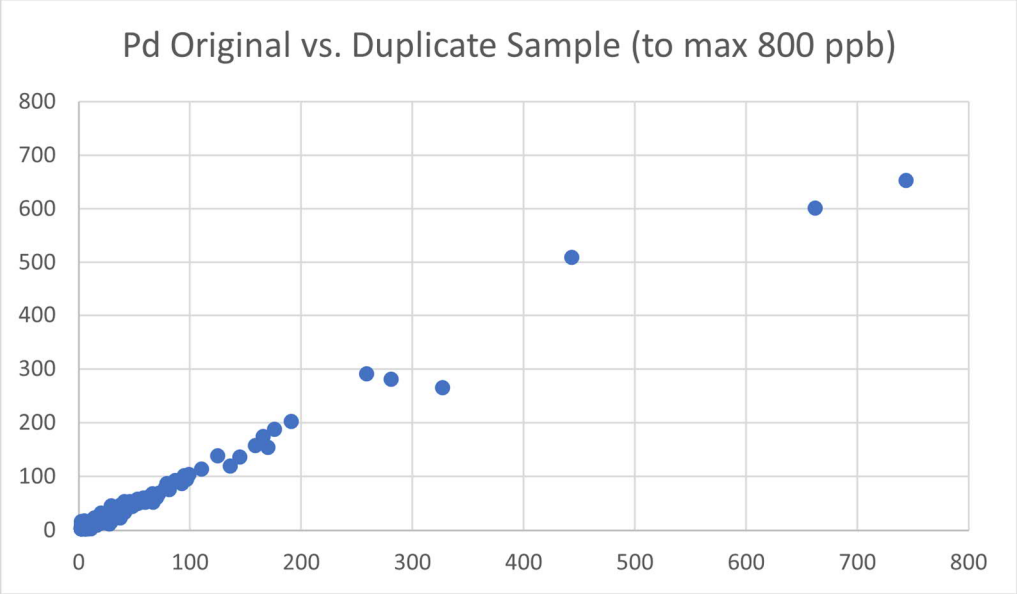


Figure 11.37: Relative % Difference of Pairs of Duplicate Samples Analyzed for Pt

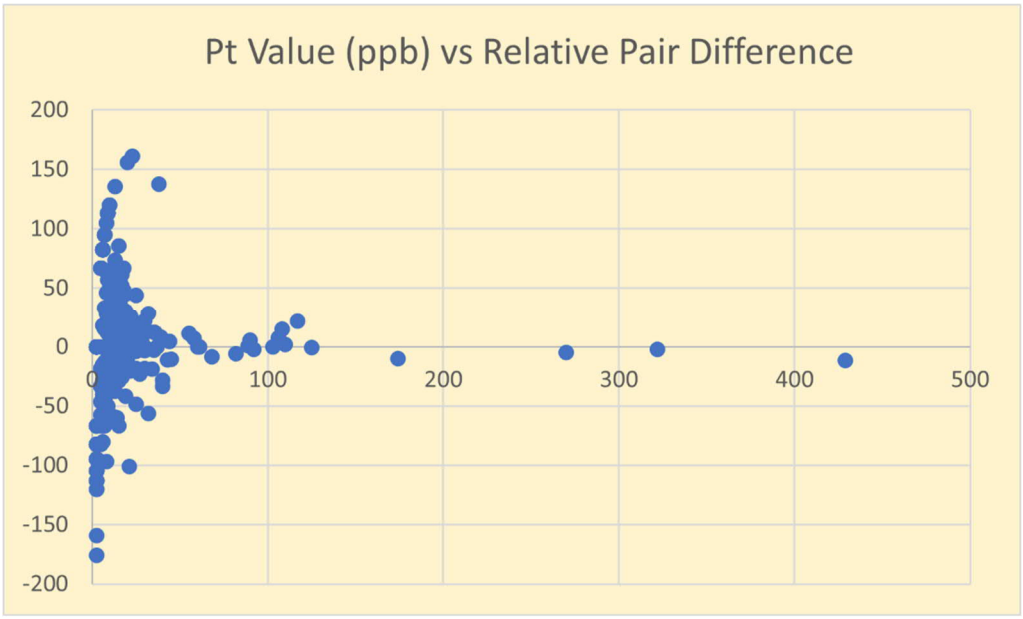


Figure 11.38: Relative % Difference of Pairs of Duplicate Samples Analyzed for Pt vs. the Absolute Concentration of the Original Analysis

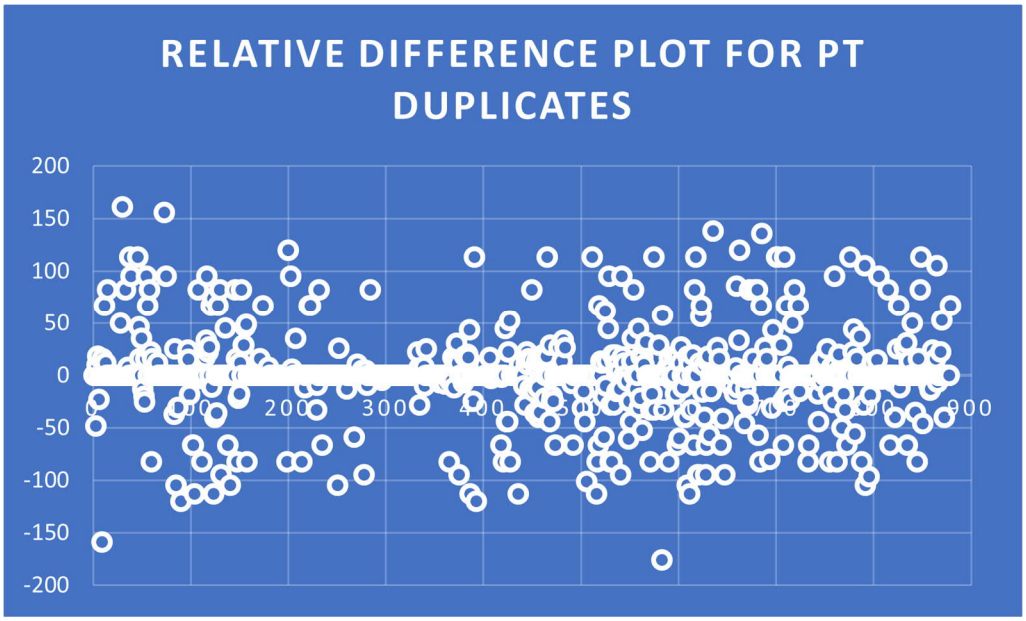


Figure 11.39: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Pt

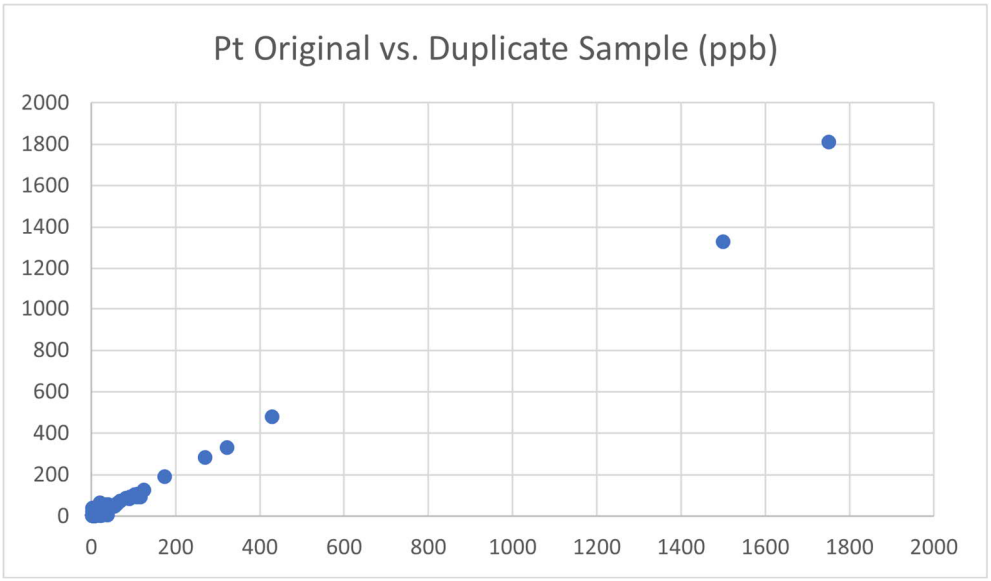
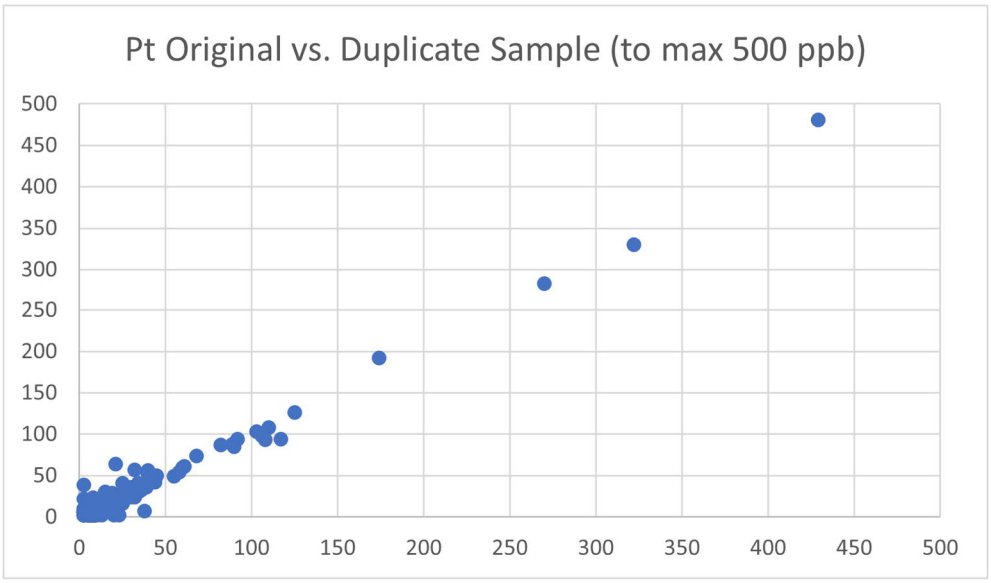


Figure 11.40: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Pt to a Maximum of 500 ppb



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 (see examples in Figures 11-41 to 11-46). 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 similar to 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).

Figure 11.41: Relative % Difference of Pairs of Duplicate Samples Analyzed for Ni

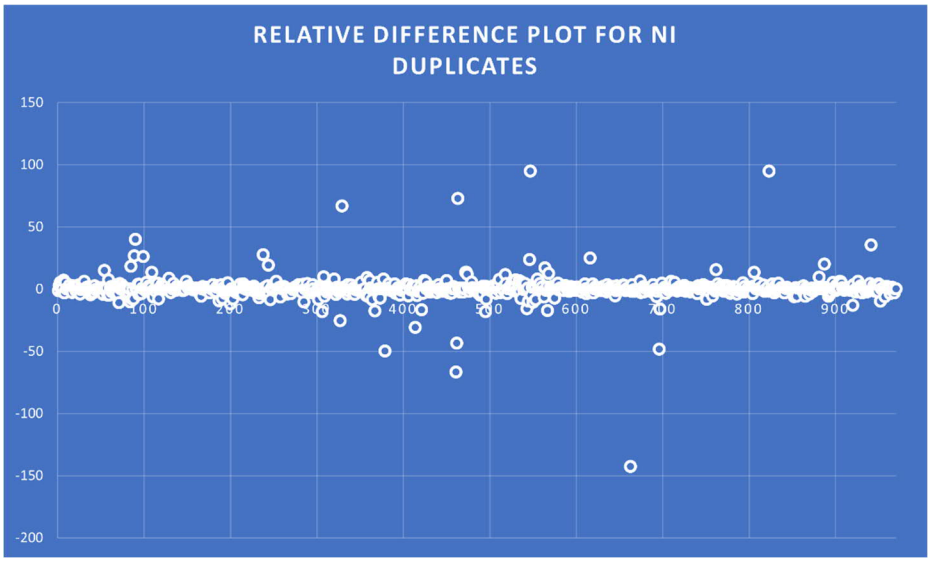


Figure 11.42: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Ni

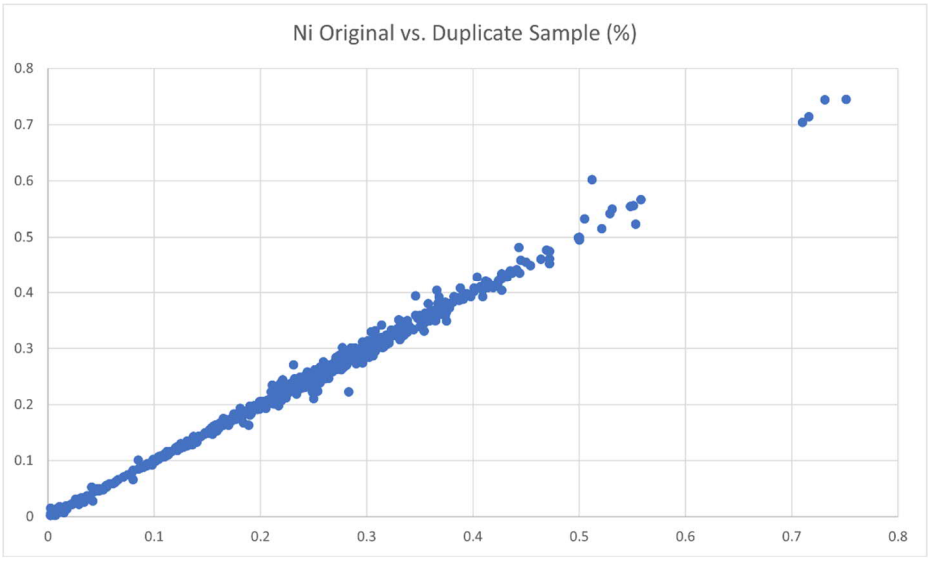


Figure 11.43: Relative % Difference of Pairs of Duplicate Samples Analyzed for Co

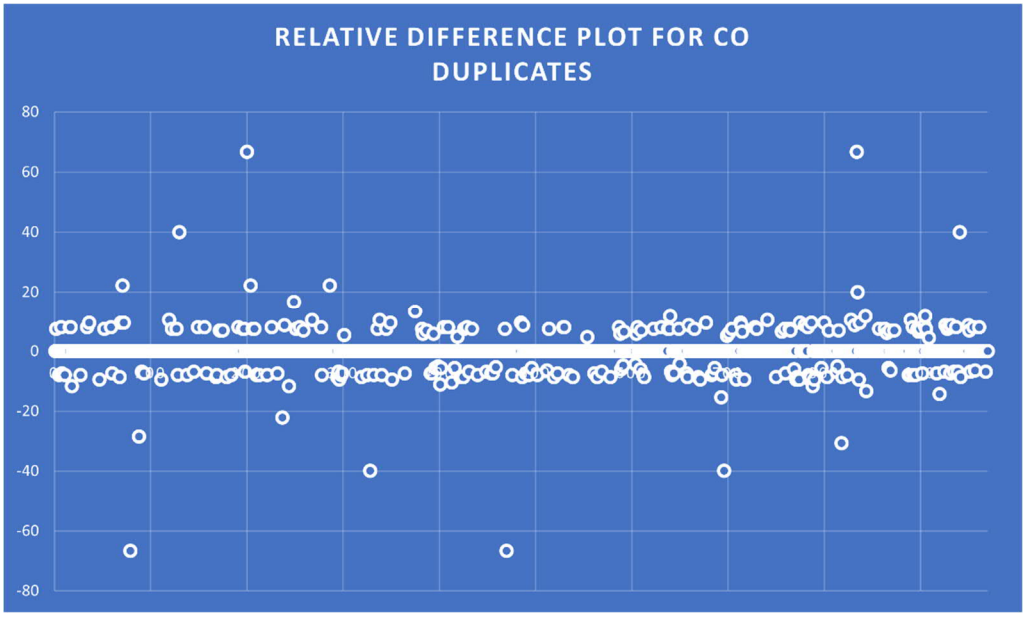


Figure 11.44: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for Co

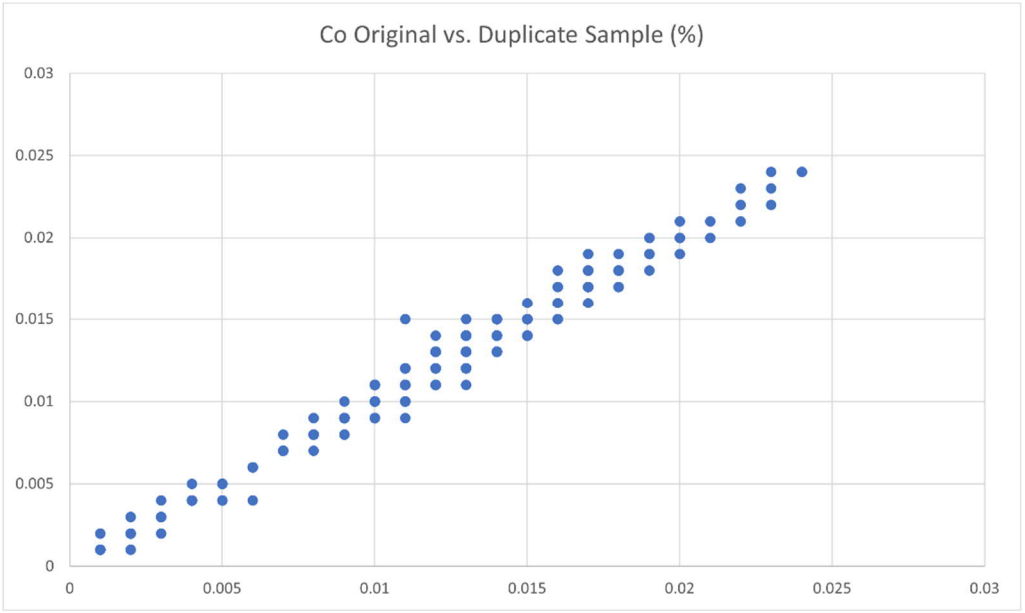
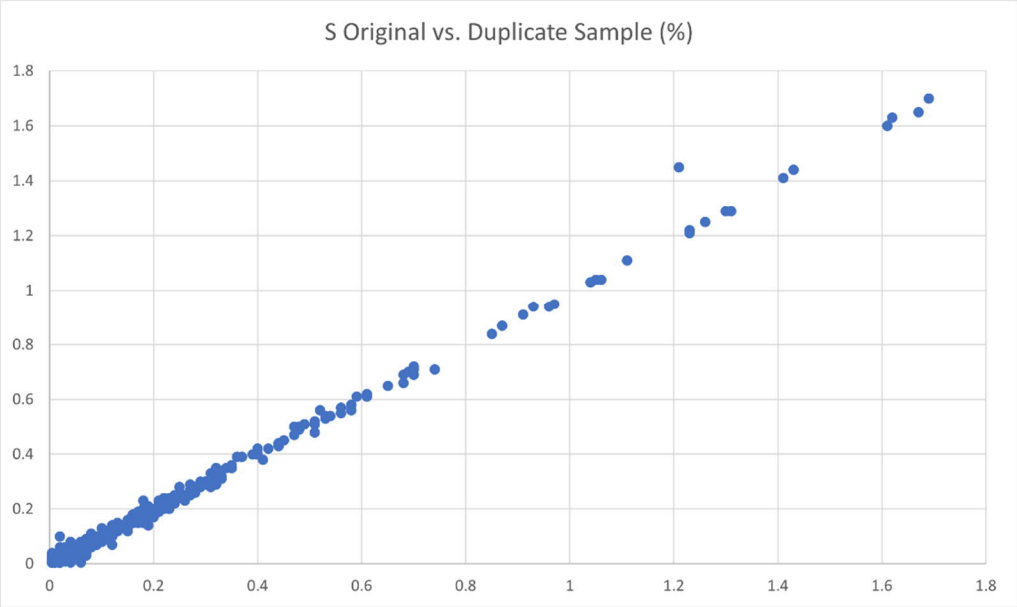
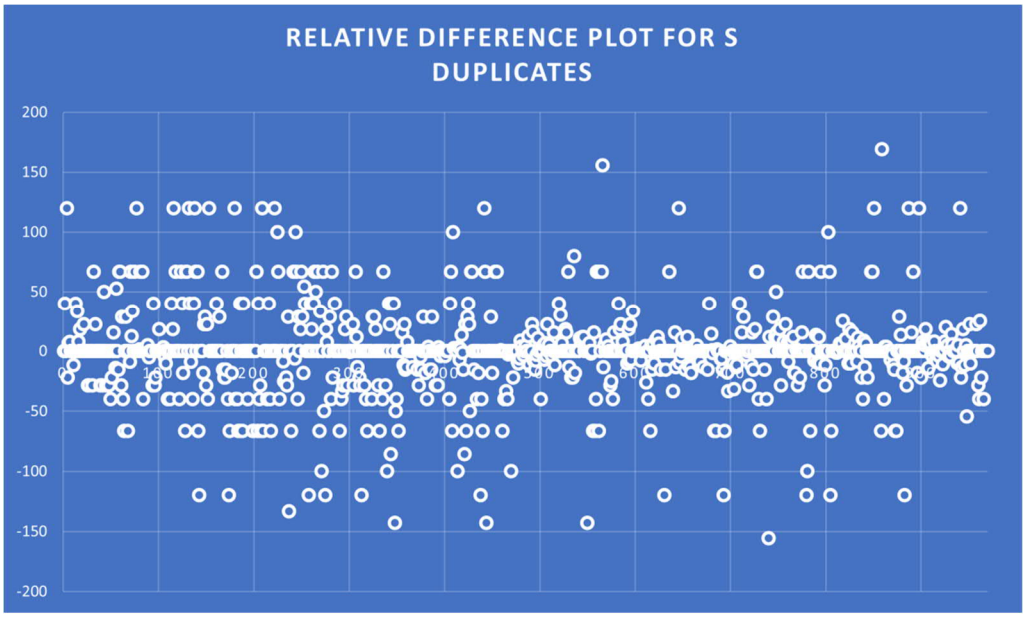


Figure 11.45: Relative % Difference of Pairs of Duplicate Samples Analyzed for S



Source: Caracle Creek, 2020.

Figure 11.46: Plot of Absolute Concentrations of Pairs of Duplicate Samples Analyzed for S



11.5.3 Duplicate Samples – Referee Analyses

Canada Nickel had a total of 320 sample pulps re-analyzed at an alternate laboratory ("Referee Lab"), 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.

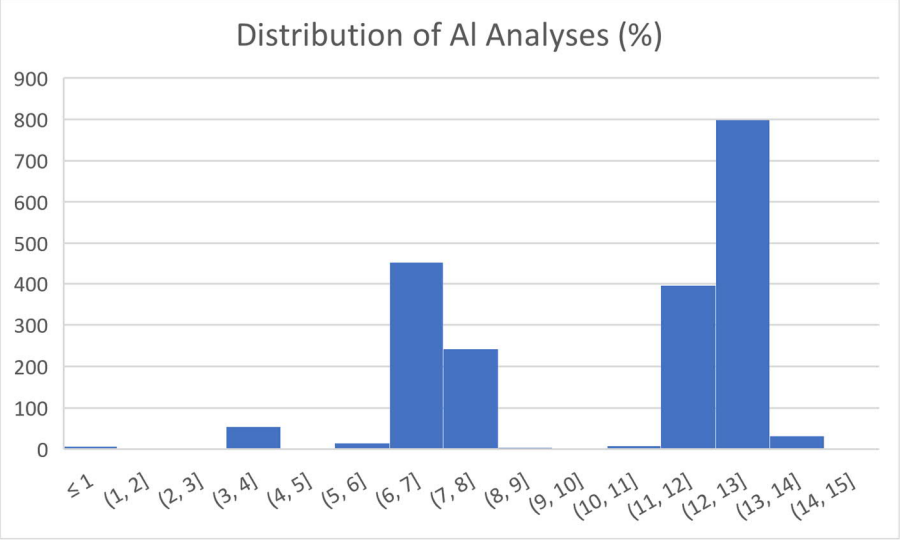
The relative differences for Ni, Cr and S were all under 20% (and mostly under 10%) indicating very good reproducibility of the original analyses.

11.5.4 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 (see Figure 11.47). Analytical results are considered to be acceptable as the results were observed to report low or negligible variance for each element examined.

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.

Figure 111.47: Distribution of Aluminum (Al) Analyses in "Blank" Samples



Note: Aluminum was used to crudely differentiate the rock-type populations.

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 preliminary economic assessment.

11.6 Sample Security and Sample Storage

CNC uses a secure storage and logging facility, which includes office space for the professional and technical staff, located at 170 Jaguar Drive, Timmins, Ontario. 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 lab.

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 out of doors.

12 DATA VERIFICATION

12.1 Internal-External Data Verification

The Authors 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 2021) that had complete databases and documentation (*e.g.*, assay certificates) 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, the Authors have no reason to doubt the adequacy of the historical sample preparation, security and analytical procedures and have complete confidence in all historical information and data that was reviewed.

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 carried out during field visits to the Crawford Township project location. Locations and orientation of drill holes was always found to be consistent with those reported in the drill hole database files.

12.3 Comments on Data Verification

It is the Authors' 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.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The objective of testwork during the preliminary economic assessment was to:

- understand the metallurgical response of the Crawford ultramafic nickel mineralization.
- provide guidance for process plant design.
- provide inputs for operating cost estimation.

The program was designed to develop the parameters for process design criteria for ore flow characteristics, comminution, desliming and flotation in the processing plant. The metallurgical program was performed on comminution samples and metallurgical variability samples. The samples were selected to represent the spatial distribution, ore grade and mineralization with a focus on the material processed during the initial phases of the mine plan.

Metallurgical testwork was completed at two labs: XPS in Sudbury, Ontario and COREM in Quebec City, Quebec, leveraging work completed on other ultramafic deposits such as Dumont and Mt. Keith. Flowsheet development work was done between August 2020 and March 2021 to formulate a standardized flowsheet. The standardized flowsheet was then used to evaluate the metallurgical response of material sampled over a range of resource variables and across the breadth of the deposit (metallurgical variability testing). Within the metallurgical variability testing:

- 14 open circuit tests were competed using a standardized flowsheet.
- 7 locked cycle tests were completed to confirm the results of flowsheet development work and metallurgical variability results.

Comminution circuit testwork was completed at SGS Lakefield on six samples from across the breadth of the Crawford property. These samples were subjected to a suite of tests including a JK drop weight test on one sample, Bond low-energy impact test (CWi) on one sample, SMC tests on five samples, Bond rod work index (RWi) tests on six samples, Bond ball work index (BWi) tests on six samples and Bond abrasion (Ai) tests on six samples.

Test work completed subsequent to March 2021 for the feasibility study was not used in the resource statement update and is not reported herein due to the incomplete nature of the work.

13.2 Sample Selection

13.2.1 Comminution Samples

Three whole HQ core samples and three half NQ core samples were selected for comminution testwork for a total of six samples:

Table 13.1 summarizes the samples that were selected for comminution testwork and used in the development of criteria for the comminution circuit design. CR20-55 Comm was sampled from the high-grade core— it was well serpentinized and had a S/Ni ratio that aligned with the average mill feed over the first six years of operation. The remaining five samples were selected to test for variability in material characteristics.

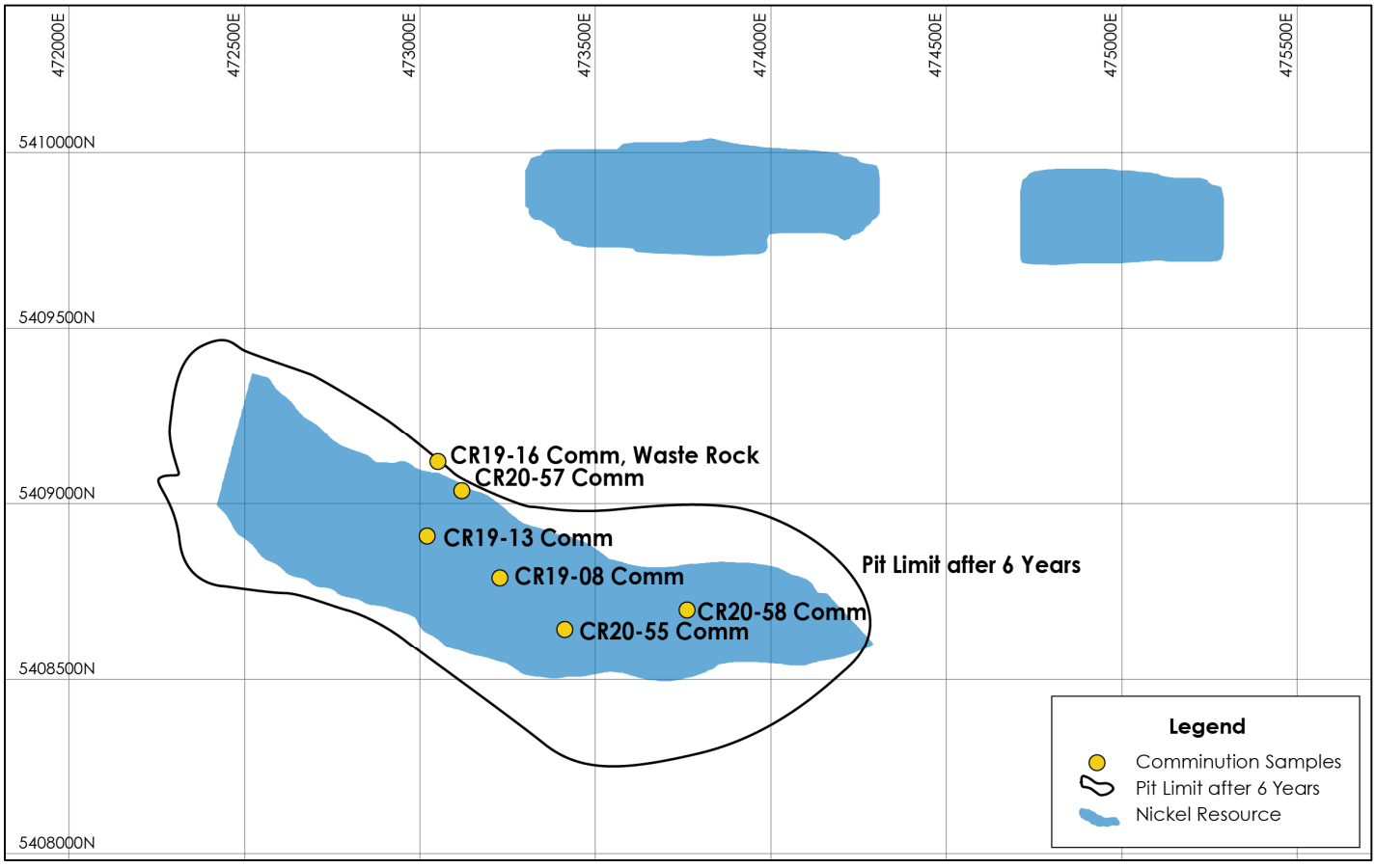
Table 13.1: Comminution Sample Summary

Sample ID	Hole #	From	To	Core Type	Wt (kg)	Mine Domain	Ni	S	S/Ni
CR20-55 Comm	CR20-55	103.1	119.5	HQ - Full	150	Higher-Grade Core	0.35	0.23	0.66
CR20-58 Comm	CR20-58	57	73	HQ - Full	135	Higher-Grade Core	0.38	0.74	1.9
CR19-08 Comm	CR19-08	214.8	246.5	NQ - Half	75	Higher-Grade Core	0.30	0.15	0.50
CR19-13 Comm	CR19-13	372	397.5	NQ - Half	60	Higher-Grade Core - Peridotite	0.28	0.05	0.18
CR20-57 Comm	CR20-57	124.5	132	HQ - Full	65	Northern Lower-Grade	0.11	0.04	0.36
CR19-16 Comm	CR19-16	43.5	69	NQ - Half	60	Waste Rock	0.06	0.08	1.3

Source: CNC, 2021.

Figure 13-1 shows the location of comminution samples. Because CR19-16 Comm lies outside of the resource it was excluded from the discussion and reported results.

Figure 13.1: Location of Comminution Samples



Source: CNC, 2021.

13.2.2 Metallurgical Variability Samples

Fourteen metallurgical variability samples were selected from HQ and half NQ cores across the breadth of the Crawford property. The samples were selected to represent the spatial distribution of the deposit, each of the mine domains (high-grade core, north low-grade and south low-grade) and a range of compositions:

- Nickel grades ranged from 0.16% to 0.43% with an average grade of 0.31%.
- Sulphur grades ranged from 0.02% to 0.63% with an average grade of 0.21%.
- S/Ni ratios ranged from 0.10 to 1.6 with an average ratio of 0.60.

All of the samples except one were taken as distinct samples from continuous sections of core. Comp3 was blended from several samples for the purpose of a locked cycle test.

Samples tested as part of the metallurgical variability program are summarized in Table 13.2. Twelve samples, including two from the East Zone, were selected from the high-grade core domain (HGC), two samples were selected from the north low-grade domain (NLG) and one sample was selected from the south low-grade domain (SLG). The grades and mineralogy of metallurgical samples are summarized in Section 13.5, Metallurgical Testwork (refer to Table 13.12).

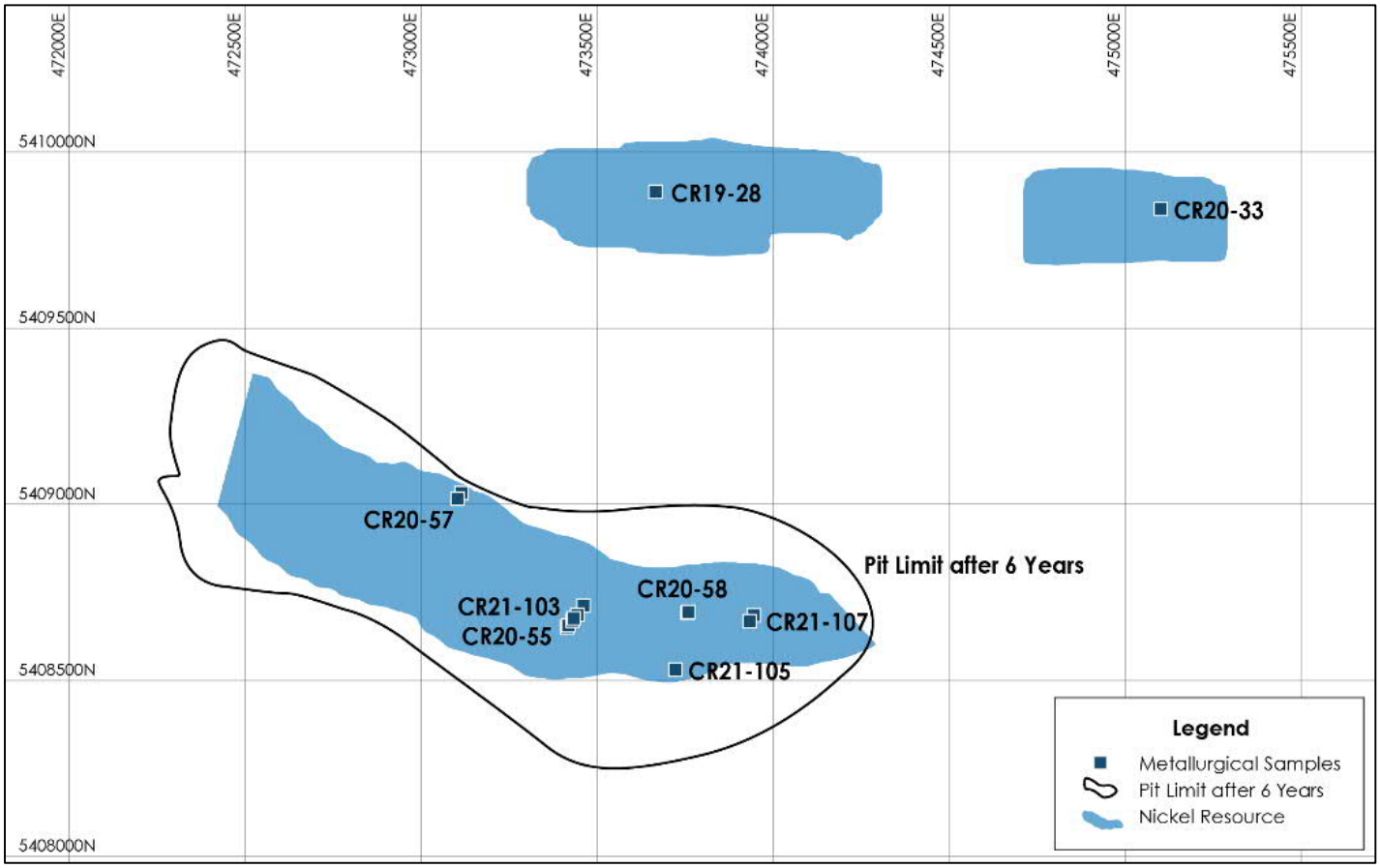
Table 13.2: Metallurgical Variability Samples

Sample ID	Sampled Hole	From	To	Wt (kg)	Domain[1]	Ni	S	S/Ni
57 XPS AW	CR20-57	167.4	174	60	NLG	0.16	0.02	0.10
CR20-57 V2	CR20-57	136.6	142.3	50	NLG	0.2	0.02	0.11
105-V7-Mar21	CR21-105	116.6	127.4	100	SLG	0.23	0.05	0.22
103-V1-Mar21	CR21-103	179.9	194.3	120	HGC	0.26	0.11	0.42
103-V13-Mar21	CR21-103	194.3	205.4	90	HGC	0.26	0.12	0.46
Comp3	Blended[2]	-	-	50	HGC	0.27	0.12	0.44
103-V11-Mar21	CR21-103	162.7	174.2	100	HGC	0.28	0.08	0.29
103-V10-Mar21	CR21-103	115.6	127.2	100	HGC	0.29	0.08	0.28
107-V12-Mar21	CR21-107	191.8	201	80	HGC	0.31	0.33	1.1
107-V8-Mar21	CR21-107	160.7	171.5	90	HGC	0.32	0.36	1.1
XPS - PN Master	CR20-58	109.6	126.7	140	HGC	0.34	0.53	1.5
EZ-33-V15	CR20-33	327.4	349.2	50	HGC	0.36	0.08	0.23
XPS 55-M1	CR20-55	199	221.2	190	HGC	0.36	0.20	0.56
COREM - PN Master	CR20-58	89.8	106.7	140	HGC	0.40	0.63	1.6
CR20-55 LCT1B	CR20-55	138.5	199	500	HGC	0.43	0.20	0.47
EZ-28-V14	CR19-28	411.5	432.1	60	HGC	0.43	0.27	0.63

Notes: 1. NLG = Northern Lower Grade, SLG = Southern Lower Grade, HGC = Higher Grade Core. 2. Comp3 was a blend of material from CR20-55 and CR19-07. Source: CNC, 2021.

Figure 13-2 shows the location of the metallurgical variability samples across the Crawford resource.

Figure 13.2: Location of Metallurgical Variability Samples across the Deposit



Source: CNC, 2021.

13.3 Comminution Circuit Characterization Testwork

Comminution testwork was completed to test the competency, hardness, and abrasion of the Crawford material. Six samples of drill core were shipped to the SGS Lakefield site in August 2020.

"CR20-55 Comm" was selected as the main sample for testing as it represents the typical mill feed in terms of mineralogy particularly in the early years of operation. CR20-55 Comm was submitted for the following suite of tests:

- JK drop weight test (JK DWT).
- Bond low-energy impact test (CWi).
- Bond rod mill work index test (RWi).
- Bond ball mill work index test (BWi).
- Bond abrasion test (Ai).

The other five samples were selected to provide a preliminary indication of variability in the grindability properties of Crawford material. These samples were submitted for the same suite of tests except for the JK drop weight test and the Bond low-energy impact test. The JK drop weight test was substituted with the SMC test. CR19-16 Comm was not reported as it represents waste grade material.

13.3.1 Grindability Testwork Results

Table 13.3 summarizes the grindability test results for the comminution samples. The following discussion is a summary of the results from the SGS grindability report (Davies & Imeson, 2021).

Table 13.3: Summary of JKTech and Work Index Statistics

Sample ID	JKTech Parameters				Work Indices			Ai	Ni
	Axb DWT	Axb SMC	DWI kWh/m³	Rel. Density	CWi kWh/t	RWi kWh/t	BWi kWh/t	g	(%)
CR20-55 Comm	60.7	-	-	2.63	9.5	14.6	20.7	0.003	0.35
CR20-58 Comm		46.4	5.6	2.61	-	15.0	21.1	0.002	0.38
CR19-08 Comm		58.9	4.5	2.63	-	13.1	14.4	0.009	0.30
CR19-13 Comm[1]		44.9	6.1	2.72	-	15.3	18.2	0.07	0.28
CR20-57 Comm		49.0	5.3	2.61	-	13.8	17.9	0.03	0.11

Note: Sample was hosted in peridotite. Source: CNC, 2021.

The key metrics from this testwork that were used for the plant design basis are the Axb and DWT parameters from JK drop weight and SMC tests and the Bond ball mill work indices. The flowsheet utilizes a SAG mill for primary grinding and a ball mill for further size reduction. The Axb parameter from the JK drop weight test was 60.7 which is characterized as moderate competency. The variability samples tested using the SMC test were more competent and were classified as moderate to medium competency. The results of this investigation indicate that material from the Crawford deposit is amenable to SAG/AG milling and the results were similar to those from other low grade ultramafic nickel deposits.

Bond ball mill work index tests on the 5 samples indicate that the ore is hard to very hard. However, these results are impacted by the acicular nature of the ultramafic mineralization. The screening process in the laboratory Bond ball mill work index test results in a different classification function to that in a process plant hydrocyclone resulting in a lower plant operating work index for the milling duties. Ausenco has data from pilot and plant operation to benchmark this effect in ultramafic nickel ore processing.

13.3.2 JK Drop Weight Test & SMC Test

The JK drop weight test and SMC test are used to characterize ore specific parameters for sizing SAG/AG mills. The ore specific parameter is the Axb parameter which is a measure of an ore's resistance to impact breakage.

The JK DWT was performed on one sample (CR20-55 Comm) from the high-grade core domain and the data was interpreted by Contract Support Services (CSS), the North American Agent for JKTech. This sample was characterized as moderately soft with respect to resistance to impact breakage, with an Axb parameter of 60.7.

The SMC test is an abbreviated version of the standard JK drop weight test that is performed on rocks of a single size fraction (-31.5 / + 26.5 mm was the size range in this case). SMC testing was completed at the University of Queensland

on five samples. Samples tested using the SMC method were found to be more competent than the CR20-55 Comm sample with respect to resistance to impact breakage, with Axb parameters in the range of 58.9 to 44.9. The five individual samples are characterized as moderate to medium competency. The relative density of the samples was in the range of 2.61 to 2.72.

13.3.3 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. Twenty rocks in the size range of 2-3" were tested from the CR20-55 Comm sample. The crusher work index from this sample was 9.5 kWh/t which is characterized as moderate crushability.

13.3.4 Bond Rod Mill Work Index Test

The Bond rod mill work index tests were performed at 14 mesh of grind (1, 180 µm) on the six samples. For samples within the resource, the RWi ranged from 13.1 to 15.3 kWh/t, which is characterized as moderately soft to medium hard in the SGS database. CR20-55 Comm had a RWi of 14.6 kWh/t which fell within this range and close to the average RWi of 14.4 kWh/t.

13.3.5 Bond Ball Mill Work Index Test

The Bond ball mill work index tests were performed with a closing screen size of 80 mesh (180 µm). The target P_{80} for these tests was 150 µm and the actual P_{80} for each test is reported in Table 13.4. The BWi for Crawford material ranged from 14.4 to 21.1 kWh/t with an average of 18.5 kWh/t. The material was characterized as hard to very hard with respect to the SGS database. The BWi was significantly harder than the RWi.

As commented above, the Bond method outcomes for acicular mineralogy needs to be interpreted to adjust for the nature of the classifier. This will reduce the operating ball mill work index by at least 15%.

Table 13.4: Measured P_{80} Particle Sizes from Bond Ball Mill Work Index Tests

Sample ID	BWi (kWh/t)	P_{80} (µm)
CR20-55 Comm	20.7	148
CR20-58 Comm	21.1	148
CR19-08 Comm	14.4	138
CR19-13 Comm	18.2	143
CR20-57 Comm	17.9	145

Source: CNC, 2021.

13.3.6 Bond Abrasion Test

Each of the five samples were submitted for Bond abrasion testing. Table 13.5 summarizes the average abrasion index of material within the higher-grade core for dunite and peridotite lithologies. The abrasiveness of the dunite is low, which is typical of other ultramafic deposits and will result in low liner wear rates and low grinding media consumption. Material that is hosted in peridotite is slightly more abrasive than the dunite, with an abrasion index of 0.07g. The peridotite is still classified as a non-abrasive material.

Table 13.5: Bond Abrasion Test Results according to the Domain (Average)

Domain	No. of Data Points	Ai (g)	Ni (%)
Higher Grade Core – Dunite	3	0.005	0.34
Higher Grade Core – Peridotite	1	0.07	0.28

Source: CNC, 2021.

13.4 Mineralogy

For ultramafic nickel deposits, the mineralogy is a critical part of establishing the resource estimate. Nickel can exist in recoverable form as minerals such as heazlewoodite, pentlandite, awaruite and millerite, or it 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.

Mineralogy is also important for understanding the deposit geology. The mineralogy can be used to infer the degree to which the host rock has been serpentinized. As the host rock becomes serpentinized, iron and nickel are reduced out of the silicate matrix into distinct mineral assemblages such as magnetite, heazlewoodite, pentlandite, awaruite or millerite. The nickel mineral assemblages present in the deposit depend on the availability of sulphur and the extent of the serpentinization process. As serpentinization progresses, serpentine minerals see an increase in the weight percent of magnesium within the mineral. For this reason, serpentine minerals have been differentiated into two types within the Crawford mineralogy program: magnesium serpentine, which has less than 5% iron, and iron serpentine which has greater than 5% iron.

At Crawford, 999 distinct samples across 26 drill holes were submitted for mineralogical characterization. Of these samples, 969 or 97% fell within the PEA resource envelope. Samples were selected for mineralogy from drill holes on a specified interval. In the first phase of mineralogy, one sample in every five was analyzed for mineralogy, and in subsequent phases, one sample in every ten samples were analyzed for mineralogy (one sample in every five equates to a sample spacing of 7.5 m). Coarse assay rejects from 1.5-metre segments of NQ were stage crushed to 150 µm, mounted in a polished section and analyzed using QEMSCAN. Table 13.6 summarizes the number of samples characterized for mineralogy by phase.

Table 13.6: Summary of Mineralogy Work

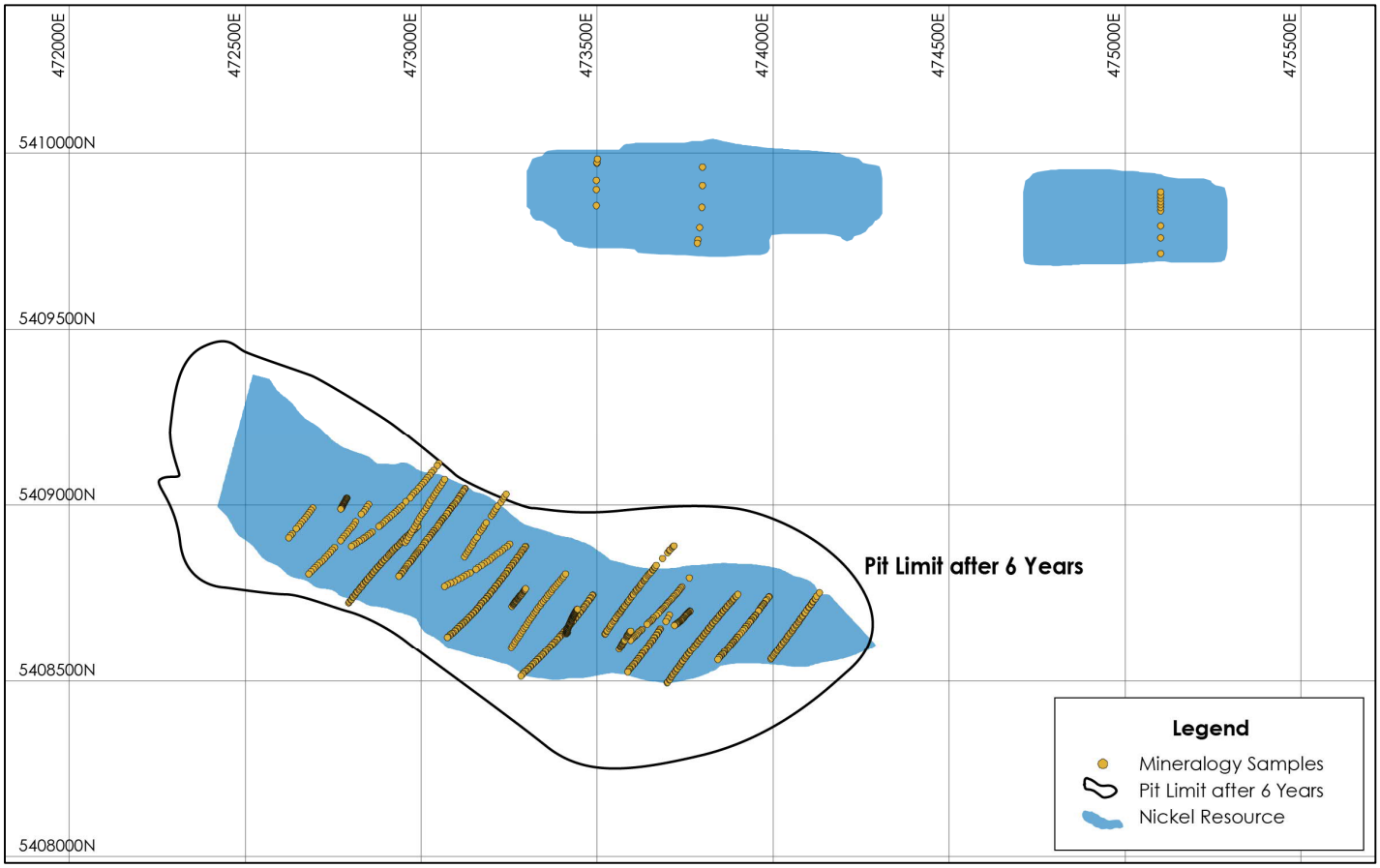
Phase	No. of Drill Holes	No. of Samples	Sample Spacing (m)	Analytical Lab
1	11	650	7.5	SGS, XPS
2	4	136	15	SGS
3	11	213	15	SGS
Total	**26**	**999**	-	-

Source: CNC, 2021.

The goal of the Phase 1 mineralogy program was to cover off the Main Zone of Crawford with respect to the east-west and north-south axes. Holes tested include CR19-05 through to CR19-14A, which are spaced approximately 200 m apart from each other within the initial geophysical target. In this first phase, one in every five samples was analyzed to understand variability within the deposit. From the Phase 1 results, it was concluded that analyzing one sample in every ten was sufficient which formed the basis for Phase 2 and Phase 3. The goals of Phases 2 and 3 for the mineralogy program were

to fill in the Main Zone and to begin characterizing the East Zone of Crawford. Figure 13-3 shows the location of mineralogy samples within the Crawford resource.

Figure 13.3: Location of Mineralogy Samples within Crawford



Source: CNC, 2021.

13.4.1 Mineralogy Results

Tables 13.7 to 13.10 summarize the key minerals in Crawford for the Main and East zones. For the Main Zone, the mineralogy is broken out according to the domain (higher-grade core, northern lower-grade and southern lower-grade) while the East Zone data are summarized collectively. The following observations have been made to support these tables:

- Across Crawford, 478 samples are from the higher-grade core domain, 161 samples are from the northern lower-grade domain and 285 samples are from the southern lower-grade domain. 45 samples are considered waste material. Thirty samples submitted for mineralogy fell outside of the resource envelope and are excluded from the discussion.

- The East Zone statistics are combined for the northern lower-grade, southern lower-grade and higher-grade core domains.

- Magnesium serpentine is the main silicate mineral in Crawford which shows that the deposit is well serpentinized. There appears to be some areas with higher proportions of iron serpentine in the Main Zone. In these areas, there may be a greater proportion of nickel hosted within the silicate matrix.

- The East Zone mineralogy shows that it is well serpentinized with high proportions of magnesium serpentine. Heazlewoodite appears to be the main nickel-bearing mineral in the East Zone. The next phase of mineralogy will focus on generating additional data for the East Zone to confirm these observations across the breadth of this zone.

- In the Main Zone, the higher-grade core is the most serpentinized, followed by the northern lower-grade and lastly the south low-grade domains. This observation stems from the distribution of Mg and Fe serpentine in these domains.

- Crawford has low levels of pyrrhotite across the Main and East zones. The highest concentrations of pyrrhotite are in the higher-grade core of the Main Zone. The 75th percentile of pyrrhotite content in the higher-grade core is 0.07%, which is low. Because of the low pyrrhotite content, the ability to make higher grade pentlandite concentrates from Crawford material has been demonstrated in metallurgical testwork.

- Nickel mineralization in the Main Zone is a mix of pentlandite, heazlewoodite and awaruite. The high-grade core is a mixture of each of these three minerals. In the low-grade domains, nickel minerals present primarily as heazlewoodite and awaruite with lesser amounts of pentlandite.

- Nickel mineralization in the East Zone is primarily heazlewoodite, with lower proportions of awaruite. There does not appear to be as much pentlandite in the East Zone which is likely due to lower sulphur availability during the serpentinization process. More data are required to extrapolate these observations across the deposit, which will be a focus for the next phase of the mineralogy program.

- The East Zone has the highest average concentration of magnetite (6.9%) followed by the lower-grade domains in the Main Zone and finally the Main Zone higher-grade core. The average magnetite concentration is in the range 5.3% to 6.9% across the domains. The average concentration of Cr Spinel is in the range 2.3% to 3.3%. Both magnetite and chrome spinel are magnetic minerals and are recovered to the magnetic concentrate within the Crawford flowsheet.

- Brucite is an important mineral for potential CO_2 sequestration. It is present across the Crawford mine domains with the average assay ranging between 1.2% to 2.5%. A focus of the Net Zero initiative will be to maximize the conversion of brucite to carbonate minerals in the tailings storage facility.

Table 13.7: Mineralogy Summary – Main Zone, Higher-Grade Core Mine Domain (463 Data Points)

Statistic	Serp (Mg) (%)	Serp (Fe) (%)	Pyrrhotite (%)	Pentlandite (%)	Heazlewoodite (%)	Awaurite (%)	Magnetite (%)	Cr-Spinel (%)	Brucite (%)
Average	**69**	**12**	**0.06**	**0.46**	**0.11**	**0.03**	**5.3**	**2.7**	**2.5**
St. Dev.	21	15	0.13	0.46	0.12	0.05	3.1	1.2	2.0
Minimum	0.48	0.07	0.00	0.00	0.00	0.00	0.0	0.3	0.03
25 percentile	63	1.8	0.00	0.09	0.01	0.00	3.1	1.7	1.0
Median	**79**	**4.7**	**0.01**	**0.32**	**0.07**	**0.01**	**5.3**	**2.6**	**2.2**
75 percentile	82	17	0.07	0.74	0.15	0.02	7.4	3.4	3.5
Maximum	**88**	**72**	**1.7**	**2.2**	**0.63**	**0.27**	**20**	**6.4**	**14**

Source: CNC, 2021.

Table 13.8: Mineralogy Summary – Main Zone, Northern Lower-Grade Mine Domain (158 Data Points)

Statistic	Serp (Mg) (%)	Serp (Fe) (%)	Pyrrhotite (%)	Pentlandite (%)	Heazlewoodite (%)	Awaurite (%)	Magnetite (%)	Cr-Spinel (%)	Brucite (%)
Average	**72**	**10**	**0.00**	**0.08**	**0.08**	**0.05**	**5.6**	**3.3**	**1.7**
St. Dev.	16	13	0.01	0.12	0.09	0.05	2.7	1.4	1.3
Minimum	14	0.11	0.00	0.00	0.00	0.00	0.17	1.0	0.02
25 percentile	71	2.9	0.00	0.01	0.02	0.01	4.1	2.3	0.80
Median	**78**	**4.8**	**0.00**	**0.02**	**0.05**	**0.02**	**5.9**	**3.0**	**1.3**
75 percentile	83	11	0.00	0.10	0.12	0.07	7.6	4.1	2.4
Maximum	**88**	**66**	**0.15**	**0.60**	**0.36**	**0.24**	**13**	**9.0**	**6.1**

Source: CNC, 2021.

Table 13.9: Mineralogy Summary – Main Zone, Southern Lower-Grade Mine Domain (282 Data Points)

Statistic	Serp (Mg) (%)	Serp (Fe) (%)	Pyrrhotite (%)	Pentlandite (%)	Heazlewoodite (%)	Awaurite (%)	Magnetite (%)	Cr-Spinel (%)	Brucite (%)
Average	**57**	**15**	**0.00**	**0.08**	**0.09**	**0.02**	**5.8**	**2.3**	**1.2**
St. Dev.	30	19	0.01	0.12	0.10	0.03	3.0	0.9	1.4
Minimum	0.3	0.05	0.00	0.00	0.00	0.00	0.01	0.7	0.00
25 percentile	31	1.5	0.00	0.01	0.01	0.00	3.7	1.6	0.15
Median	**74**	**6.4**	**0.00**	**0.03**	**0.05**	**0.00**	**6.0**	**2.2**	**0.6**
75 percentile	81	26	0.00	0.11	0.12	0.02	7.6	2.9	1.8
Maximum	**90**	**78**	**0.15**	**1.00**	**0.68**	**0.16**	**16**	**4.9**	**8.1**

Source: CNC, 2021.

Table 13.10: Mineralogy Summary – East Zone (22 Data Points)

Statistic	Serp (Mg) (%)	Serp (Fe) (%)	Pyrrhotite (%)	Pentlandite (%)	Heazlewoodite (%)	Awaurite (%)	Magnetite (%)	Cr-Spinel (%)	Brucite (%)
Average	**73**	**7**	**0.00**	**0.00**	**0.14**	**0.02**	**6.9**	**2.5**	**1.8**
St. Dev.	17	6	0.00	0.01	0.14	0.02	2.4	1.5	2.0
Minimum	10	2.09	0.00	0.00	0.00	0.00	0.55	0.8	0.00
25 percentile	70	3.6	0.00	0.00	0.04	0.00	6.3	1.6	0.16
Median	**79**	**5.3**	**0.00**	**0.00**	**0.10**	**0.01**	**7.1**	**1.9**	**0.8**
75 percentile	82	9	0.00	0.00	0.20	0.03	7.9	2.7	3.7
Maximum	**87**	**30**	**0.00**	**0.02**	**0.55**	**0.07**	**11**	**6.0**	**5.1**

Source: CNC, 2021.

13.4.2 Microprobe Analysis

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 for the mineralogy program, as the microprobe analyses are used to estimate the nickel deportment within a sample.

Microprobe analysis was done on polished sections at two labs: XPS and Queen's University. A total of 41 samples were analyzed across ten holes to align with the Phase 1 mineralogy program (Holes CR19-05 through CR19-14A). Three to six samples from each of these holes were analyzed. 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	No. of Pts.	Ni		Fe		Cr		Co	
		Avg. (%)	St. Dev (%)	Avg. (%)	St. Dev (%)	Avg. (%)	St. Dev (%)	Avg. (%)	St. Dev (%)
Awaruite	36	74.0	1.9	23.3	0.3	-	-	1.7	0.6
Heazlewoodite	241	72.6	0.6	1.1	0.7	-	-	0.07	0.20
Pentlandite	231	33.2	2.6	28.0	5.0	-	-	5.9	6.0
Magnetite	404	0.16	0.16	70.9	1.0	0.43	0.65	0.02	0.02
Chrome Spinel	380	0.10	0.03	18.4	4.2	32.4	2.0	0.04	0.02
Serpentine	418	0.08	0.05	2.1	1.4	0.10	0.07	0.01	0.00

Source: CNC, 2021.

Mineral processing testwork has produced nickel concentrates with grades in excess of 35% nickel due to the high nickel tenor of heazlewoodite and awaruite. In Crawford, the average heazlewoodite nickel tenor is 72.6% and the average awaruite nickel tenor is 74.0%. Heazlewoodite is recovered in the flotation circuits while awaruite is recovered in the magnetic circuit.

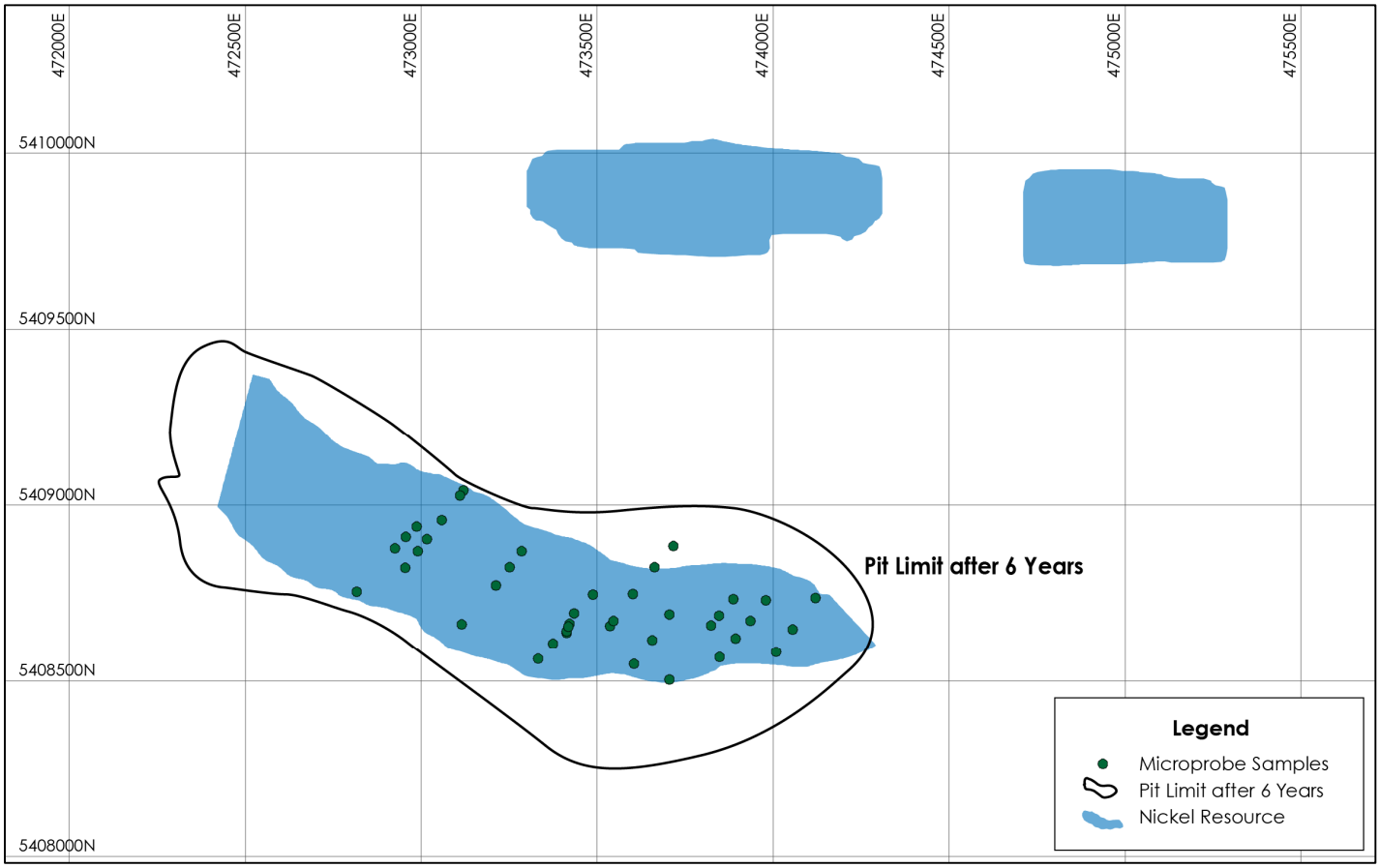
The average nickel tenor of pentlandite across the breadth of the Crawford Main Zone is 33.2% with a standard deviation of 2.7%. Pentlandite will largely report to the lower-grade concentrate at Crawford. Within Crawford, the pyrrhotite levels are low, which means less grade dilution from iron sulphides will occur particularly in the lower grade, pentlandite concentrate.

Magnetite and chrome spinel are magnetic minerals that are recovered to the magnetic concentrate.

Serpentine is the main gangue mineral in the deposit. Microprobe analysis shows that the average nickel tenor of serpentine is 0.08% with a standard deviation of 0.05%. This nickel is hosted within the mineral matrix and is unrecoverable, except through entrainment to the various products in the mineral processing flowsheet.

Figure 13-4 shows the location of samples selected for microprobe analysis within Crawford. The samples were taken from holes CR19-05 – CR19-14A. The samples cover off the Eastern and Central parts of the Crawford Main Zone, which aligns with the mining sequence.

Figure 13.4: Location of Samples Selected for Microprobe Analysis



Source: CNC, 2021.

13.5 Metallurgical Testwork

Metallurgical testwork was completed with two objectives:

- flowsheet development work to define the flowsheet.
- metallurgical variability testwork to evaluate the metallurgical response of material with the standardized flowsheet.

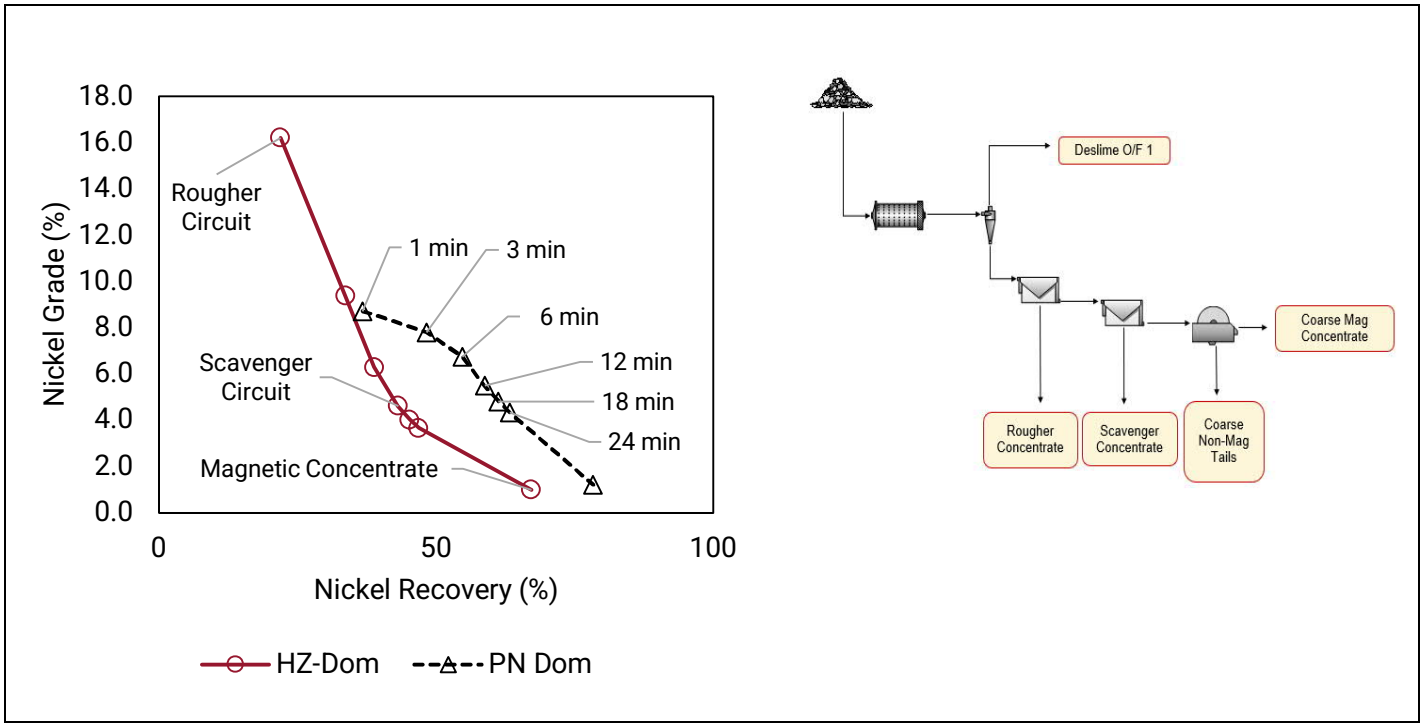
Testwork was completed in parallel at both XPS and COREM, leveraging work completed on other ultramafic nickel deposits such as Mt. Keith and Dumont.

13.5.1 Initial Flowsheet Development Testwork

Initial flowsheet development started with kinetic testwork in the coarse circuit rougher-scavenger flotation cells and was done on a heazlewoodite dominant sample, CR20-55 LCT1B (S/Ni = 0.47), and a pentlandite dominant sample, COREM PN-Master (S/Ni = 1.6). The tests were completed under common conditions using a primary grind size of 80% passing 180 µm.

Figure 13-5 shows the resulting grade-recovery curves for the two samples (left) and the flowsheet used for testing (right). From these initial phases of work, a wide range of flowsheet parameters was tested through 175 tests over the course of 10 months on more than 11 samples containing more than 1100 kg of material.

Figure 13.5: Coarse Circuit Grade Recovery Curves Compared Between Heazlewoodite and Pentlandite Dominant Samples



Source: COREM, 2021.

The following observations were made regarding the flotation kinetics of CR20-55-LCT1B and COREM PN-Master:

- The recovery for the pentlandite sample was higher than that of the heazlewoodite sample, likely due to the higher availability of sulphur. This observation supports the design of the recovery equations, which are also a function of the feed S/Ni ratio.
- The majority of the nickel is recovered in the rougher flotation stage.
- The heazlewoodite sample produced higher grade concentrate than the pentlandite sample in the early part of the rougher float. The first particles to float were likely liberated or sub-liberated heazlewoodite particles which is why high grades were achieved. In the later stages of the float, recovery of slower to float composite particles occurred which lowered the grade. When liberated, heazlewoodite has fast flotation kinetics. This is the basis for the regrind on the coarse third cleaner concentrate, which separates heazlewoodite and pentlandite from each other into a high- and low-grade concentrate, respectively.

- For both samples, the rougher concentrate grade after six minutes of flotation was greater than 6% Ni, which will be suitable feed for the cleaning circuit. The combined rougher-scavenger concentrate for the two samples was in the range of 3.6% to 4.3% Ni. There may be potential to increase the recovery to the rougher-scavenger concentrate by extending the float time or adjusting the reagents.

13.5.2 Metallurgical Variability Testing

Variability samples were selected from the Crawford deposit for recovery characterization using a standardized flowsheet. The tests were completed at two labs in parallel: XPS in Sudbury, Ontario and COREM in Quebec City, Quebec. Fourteen samples were tested using the metallurgical variability test and three locked cycle tests were completed to confirm the results under the final conditions.

Table 13.12 summarizes the head grade and mineralogical characterization of the metallurgical variability samples. The mineralogical characterization is summarized for the five samples that had the results available at the time of reporting.

Table 13.12: Metallurgical Variability Sample Head Grades and Mineralogy

Sample ID	Ni %	S %	Fe %	Cr %	HZ[1] %	AW[1] %	PN[1] %	MT[1] %	Cr Spinel[1] %	Mg Serp[1] %	S/Ni ratio
57 XPS AW	0.16	0.02	7.0	0.47	0.01	0.05	0.07	9.4	1.9	77	0.10
CR20-57 V2[2]	0.18	0.02	5.3	0.43	-	-	-	-	-	-	0.11
105-V7-Mar21[4]	0.23	0.05	6.8	0.58							0.22
EZ-33-V15	0.36	0.08	5.5	0.75	0.37	0.00	0.03	4.6	4.3	81	0.23
103-V10-Mar21[4]	0.29	0.08	6.1	0.72							0.28
103-V11-Mar21[4]	0.28	0.08	5.1	0.70							0.29
103-V1-Mar21[4]	0.26	0.11	6.0	0.68							0.42
Comp3[2]	0.27	0.12	7.8	0.71	-	-	-	-	-	-	0.44
103-V13-Mar21[4]	0.26	0.12	6.1	0.47							0.46
CR20-55 LCT1B[3]	0.43	0.20	7.3	0.43	0.41	0.01	0.27	7.5	1.4	81	0.47
XPS 55-M1	0.36	0.20	7.0	0.26	0.33	0.00	0.41	8.6	1.2	82	0.56
EZ-28-V14	0.43	0.27	6.3	0.65	0.23	0.00	0.48	5.7	2.3	83	0.63
107-V12-Mar21[4]	0.31	0.33	4.8	0.68							1.1
107-V8-Mar21[4]	0.32	0.36	5.6	0.79							1.1
XPS - PN Master	0.34	0.53	5.9	0.84	0.00	0.00	0.82	5.3	3.2	80	1.5
COREM - PN Master[3]	0.40	0.63	5.7	0.84	0.00	0.04	1.2	4.6	2.5	81	1.6

Notes: 1. HZ = heazlewoodite, AW = awaruite, PN = pentlandite, MT = magnetite, Cr Spinel = chrome spinel, Mg Serp = magnesium serpentine. 2. Mineralogical characterization was not completed. 3. Mineralogical characterization was done on a single size fraction. 4. Mineralogical characterizations completed but the results were not available at the time of reporting. Source: CNC, 2021.

13.5.2.1 Standard Variability Test

Figure 13-6 shows the standardized flowsheet that was used for metallurgical variability testing. The flowsheet consists of three main circuits: coarse flotation, fine flotation and the magnetic circuit. The commercial flowsheet would also include a slimes cleaning circuit. However, slimes flotation was not included in the metallurgical variability flowsheet as the contribution to the overall recovery is small relative to the amount of work required to generate the results. Two of the seven locked cycle tests on Samples CR20-55-LCT1B and Comp3 included contribution to nickel recovery based on flowsheet

development tests run on one of the samples. The impact on iron and chromium recoveries from these samples were negligible.

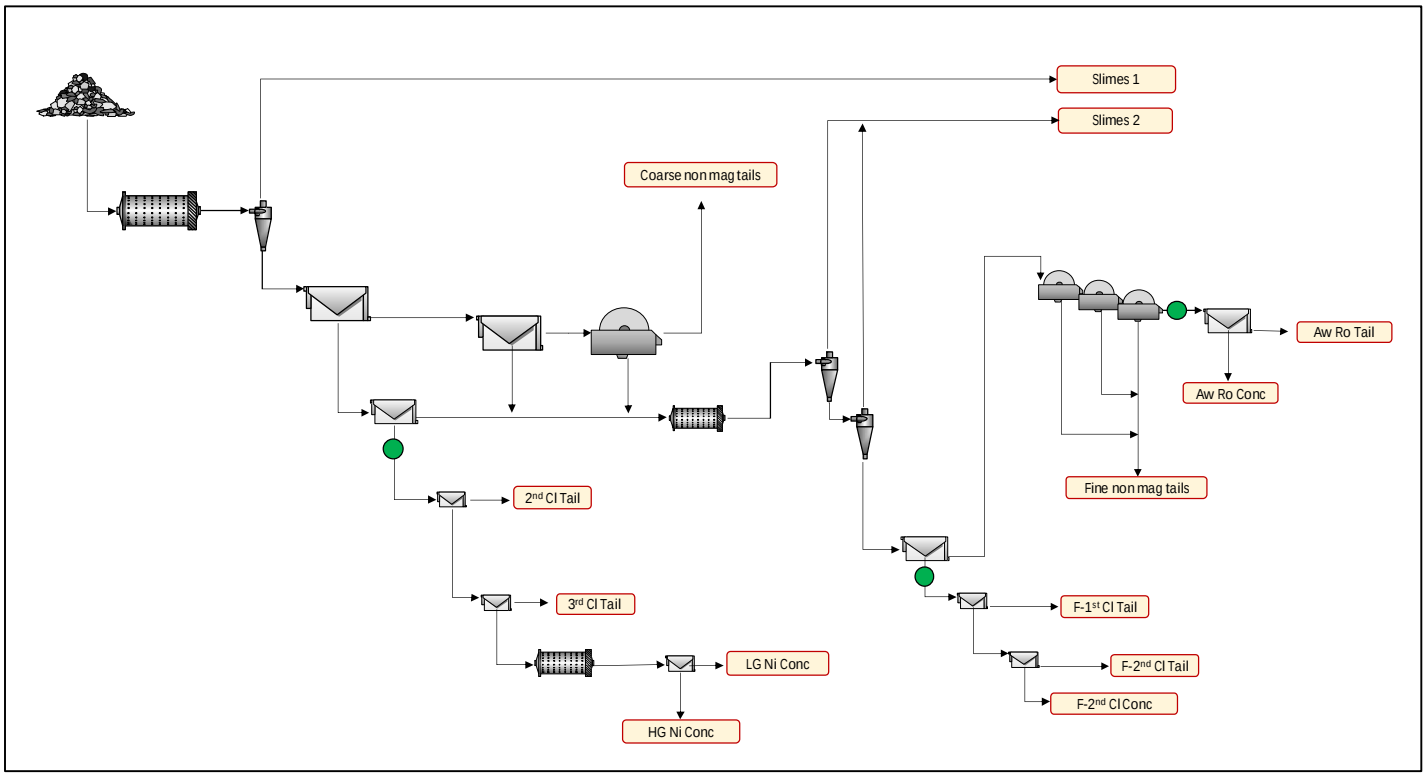
Concentrate from the coarse circuit is only reground prior to production of high- and low-grade nickel concentrates.

Middlings from the coarse circuit are combined with scavenger flotation concentrate and magnetic concentrate and fed to a regrind circuit prior to cleaner flotation (fine flotation). Tailings from the fine flotation circuit feed the magnetic circuit.

To minimize the impact of ultrafines on flotation circuit performance, desliming is done after each grinding stage. There are two desliming steps in the flowsheet: a single stage primary deslime for the primary mill discharge, and a two-stage secondary deslime after the regrind mill. The deslime cyclones are crucial to the Crawford flowsheet because they remove ultrafine serpentine particles and fibers generated in the comminution process. Approximately 20% of the feed nickel reports to the slimes fraction, some of which may be recovered in a slimes circuit (an opportunity for increased Ni recovery).

The green circles in Figure 13-6 identify the streams that were used to define recovery for the open circuit tests. For this metallurgical variability program, nickel recovery was calculated as the sum of nickel recoveries to the coarse cleaner 1 concentrate, the fines rougher concentrate and the magnetic concentrate. Recoveries were defined this way based on a comparison of open circuit and locked cycle test results, for which recoveries were similar using this definition. A comparison of open circuit and locked cycle test results for the same samples is provided in Section 13.6.

Figure 13.6: Standard Open Circuit Flowsheet used for the Metallurgical Variability Testing



Source: CNC, 2021.

Concentrate grade for the flotation product reflects the combined coarse first cleaner concentrate and the fine rougher concentrate. The grade of the magnetic concentrate is reported separately as it is a by-product. See Table 13.15 in Section 13.5 for a summary of the flotation and magnetic concentrate grades as calculated using this approach.

Procedures used as part of the standard variability tests 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 3x 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 XPS or SGS Lakefield for mineralogical characterization to confirm nickel deportment, mineralogy and liberation on a size-by-size basis.
- Grind 2 x 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 approximately 15% to 20% reporting to the overflow.
- Perform flotation separation on the hydrocyclone underflow.
- Grind the coarse third cleaner concentrate to 80% passing 25 to 35 µm and treat with flotation separation to make the low- and high-grade final concentrate products.
- Conduct magnetic separation on the scavenger tails.
- Combine the cleaner 1 tails, scavenger concentrate and magnetic fraction of the scavenger tails and regrind to 80% passing 45 µm.
- Treat the regrind product with a hydrocyclone to deslime the pulp with approximately 35% to 40% of the feed mass reporting to the overflow.
- Perform fine flotation separation on the deslime cyclone underflow.
- Perform three-stage magnetic separation on the fine rougher tails.
- Separate awaruite from magnetite by performing flotation separation on the magnetic concentrate.
- Record weights for all products.
- Send products for analysis using peroxide fusion ICP-OES for Ni, Co Fe, Cr, Mg and S. Leco analysis was used to confirm the sulphur assay.

13.5.2.2 Standard Test – Reagent Usage from Open Circuit Tests

Table 13.13 summarizes the reagent usage according to the unit operation for the metallurgical variability testing with the following comments:

- Carboxy methyl cellulose (CMC) was used as a slime dispersant in flotation.
- Potassium amyl xanthate (KAX51) was added as a collector in flotation.
- Acid was added to the coarse rougher, coarse scavenger and awaruite flotation cells. In the coarse rougher-scavenger, acid addition rates were independent of pH, while in the awaruite flotation cell, acid addition was adjusted to achieve pH 3.5 in the cell.
- Two types of frothers were used in this test program: MIBC and Aerofroth-65 (A-65). MIBC was used in the coarse rougher, coarse scavenger and fine rougher flotation cells at fixed rates. In the coarse and fine cleaning circuits, A-65 was mixed with MIBC in a 1:2 ratio. For cleaning circuits, the average frother addition rate is reported.

Table 13.13: Standard Reagent used in the Metallurgical Variability Tests

Stage	Reagents (g/t)				Time (min)	
	H_2SO_4 (98%)	KAX51	CMC	Frother	Cond.	Froth
Primary Grind						
Deslime						
Rougher	1000	50	100	10.2	10.5	6
Scavenger	666	30		5.1	2	12
Cleaner 1		10	15	1.5	6	10
Cleaner 2		2	1.25	0.7	2	5
Cleaner 3				0.8	0.5	3
Coarse Cleaner Regrind						
Cleaner 4		10		0.7	2	4
Magnetic Separation on Scavenger Tails						
Regrind						
Deslime						
Fine Rougher		20	30	6.4	6.5	6
Fine Cleaner 1		2			6	4
Fine Cleaner 2		1	1.25			3
Magnetic Separation						
Awaruite (AW) Flotation	1800	20		20	6.5	6
Total	**3466**	**145**	**148**	**45**	**42**	**59**

Source: CNC, 2021.

Table 13.4 summarizes the typical cell type, size and agitation speed used for the coarse rougher, coarse scavenger and fine rougher flotation cells. Cleaning cells were typically 2-liter cells or smaller due to the smaller mass feed rates to these stages of flotation. The agitation speed on the cleaning cells was adjusted according to the sample.

Table 13.14: Flotation Cell Type, Size and Agitation Rate used in the Metallurgical Variability Tests

	Coarse Rougher	Coarse Scavenger	Fine Rougher
Cell Type	Denver	Denver	Denver
Cell Size (L)	28	28	10
Agitation Speed (rpm)	933	933	1200

Source: CNC, 2021.

13.5.2.3 Metallurgical Variability Testing Results

Samples were processed using the standardized procedure described above to assess the variability in metallurgical response to the flowsheet. Table 13.15 summarizes the results of 14 open circuit tests over a range of Ni grades, S grades and S/Ni ratios. Samples are sorted according to the S/Ni ratio and domain.

Table 13.15: Metallurgical Variability Test Results Summary

	Recovery			Grade		Feed Characteristics		
Sample ID	Total Rec. (%)	Flotation Conc. (%)	Magnetic Conc. (%)	Flotation Conc. (% Ni)	Magnetic Conc (% Ni)	Ni (%)	S/Ni ratio	Domain[1]
EZ-33-V15[2]	64	58	5.5	5.0	0.39	0.36	0.23	HGC
103-V10-Mar21	38	28	10	9.5	0.31	0.29	0.28	HGC
103-V11-Mar21	30	29	1.4	5.9	0.11	0.28	0.29	HGC
103-V1-Mar21	44	43	0.7	11	0.04	0.26	0.42	HGC
103-V13-Mar21	57	55	2.2	10	0.09	0.26	0.46	HGC
CR20-55 LCT1B[3]	56	54	2.5	2.3	0.20	0.43	0.47	HGC
XPS 55-M1[4]	51	48	3.4	6.9	0.35	0.36	0.56	HGC
EZ-28-V14[5]	56	54	2.1	7.5	0.17	0.43	0.63	HGC
107-V12-Mar21	56	54	1.7	8.7	0.23	0.31	1.1	HGC
107-V8-Mar21	58	56	1.8	8.8	0.19	0.32	1.1	HGC
COREM - PN Master[6]	52	49	3	8.5	0.37	0.40	1.6	HGC
CR20-57 V2	28	15	13	3.0	0.64	0.20	0.10	NLG
57 XPS AW[7]	29	7.8	21	0.7	0.53	0.16	0.10	NLG
105-V7-Mar21	43	42	0.7	6.3	0.11	0.23	0.22	SLG

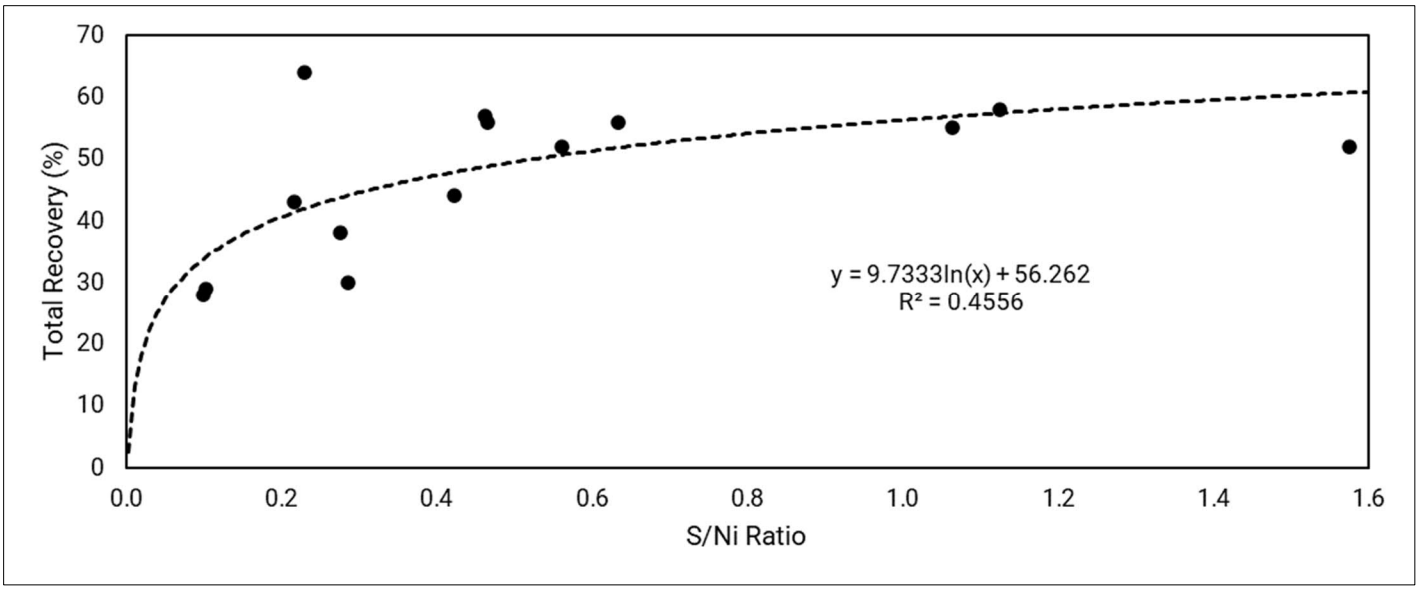
Notes: 1. HGC = high-grade core, NLG = north low-grade, SLG = south low-grade. 2. Test used more MIBC in the coarse rougher (25 g/t total) and coarse scavenger (15 g/t total) flotation cells. Fines circuit flotation times were longer than in the standardized conditions (10-minute fines rougher float and 8-minute fines first cleaner float). 3. Test used an older, preliminary flowsheet. Scavenger concentrate was sent to the coarse first cleaner. 500 g/t acid was added to the fines rougher. It is assumed that these recoveries will be maintained/exceeded using the standardized flowsheet and reagent scheme outlined above. 4. Test used an older, preliminary flowsheet. Scavenger concentrate was sent to the coarse first cleaner. Fines circuit flotation times were longer than in the standardized conditions (10-minute fines rougher float and 8-minute fines first cleaner float). 10 g/t CMC was added to the fines first cleaner. It is assumed that these recoveries will be maintained/exceeded using the standardized flowsheet and reagent scheme outlined above. 5. Fines circuit flotation times were longer than in the standardized conditions (10-minute fines rougher float and 8-minute fines first cleaner float). 10 g/t CMC was added to the fines first cleaner. It is assumed that these recoveries will be maintained/exceeded using the standardized flowsheet and reagent scheme outlined above. 6. Test used an older preliminary flowsheet. Scavenger concentrate was sent to the coarse first cleaner. 500 g/t acid was added to the fines rougher. It is assumed that these recoveries will be maintained/exceeded using the standardized flowsheet and reagent scheme outlined above. 7. Test used a condensed flowsheet due to alloy mineralization. The condensed flowsheet included a primary deslime, coarse rougher, coarse scavenger, regrind, magnetic separation circuit and awaruite circuit all in series. It is assumed that these recoveries will be maintained/exceeded using the standardized flowsheet and reagent scheme outlined above. Source: CNC, 2021.

The following observations support the results summary in Table 13.15:

- Eleven samples were tested from the high-grade core domain with samples that had Ni grades in the range of 0.26 to 0.43 and S/Ni ratios in the range of 0.23 to 1.6. This includes EZ-28-V14 and EZ-33-V15, which were sampled from the East Zone of Crawford.
- Three samples were tested from the low-grade domains: two from the north low-grade domain and one from the south low-grade domain.
- Total recovery was defined as the sum of recoveries to the coarse cleaner 1 concentrate, fine rougher concentrate, and the magnetic concentrate.
- The average total nickel recovery for samples taken from the high-grade core was 51%. The average grade of flotation and magnetic concentrates for samples taken from the high-grade core was 6.2% Ni and 0.27% Ni, respectively. The high-grade core will be mined first to capitalize on the higher net smelter return of this material and minimize the project payback period.
- Samples with a lower S/Ni ratio may see a greater proportion of the nickel reporting to the magnetic concentrate because of the awaruite mineralization, which is a magnetic mineral.

Figure 13-7 shows the total recovery from open circuit tests as a function of the feed S/Ni ratio. Total nickel recovery generally shows a positive correlation with the S/Ni ratio, which is typical of other deposits. A logarithmic curve was used to fit the data, with an R^2 of 0.46. Within the dataset, Samples EZ-33-V15 and 103-V11-Mar21 are outliers. If these two data points are removed, the updated trendline has an R^2 of 0.75.

Figure 13.7: Total Recovery as a Function of the S/Ni Ratio



Source: CNC, 2021.

Some discussion follows regarding the outliers, samples EZ-33-V15 and 103-V11-Mar21:

- The open circuit test on sample EZ-33-V15 had a total nickel recovery of 64% (flotation concentrate grade = 5.0%, magnetic concentrate grade = 0.39%). This test is an outlier with the highest recovery across the fourteen tests. The East Zone may represent a different metallurgical domain for Crawford due the high degree of serpentinization (refer to Table 13.10 in Section 13.4, Mineralogy). The other sample from the East zone, EZ-28-V14, also performed well in the open circuit test with a total recovery of 56% (flotation grade = 7.5% and magnetic concentrate grade = 0.17%), which may support this theory. Further testwork and mineralogical characterization of the East Zone is required to confirm this.

- Sample 103-V11-Mar21 had a lower total nickel recovery of 30% (flotation grade = 5.9% and magnetic concentrate grade = 0.11%). This data point lies below the trendline. A sample taken nearby to 103-V11-Mar21 (nearby sample was 103-V10-Mar21) also exhibited a lower-than-expected recovery. This may indicate a different degree of serpentinization and/or some further alteration of the ore in this localized area, which is impacting recovery. This will be examined when the mineralogy results are received.

Samples which have either heazlewoodite or awaruite in them can produce high grade concentrates due to the nickel tenor of these minerals, which is more than 70% nickel. For each of the open circuit metallurgical variability tests, the nickel split between the high- and low-grade concentrate was calculated using the following equations:

$$\%\ of\ recovered\ Ni\ in\ the\ HG\ Conc. = \frac{Mass\ of\ Ni\ in\ the\ HG\ Conc.}{Mass\ of\ Ni\ in\ the\ HG\ Conc. + Mass\ of\ Ni\ in\ the\ LG\ Conc.} \cdot 100\%$$

$$\%\ of\ recovered\ Ni\ in\ the\ LG\ Conc. = 1 - (\%\ of\ recovered\ Ni\ in\ the\ HG\ Conc.)$$

To calculate these metrics, a grade was assumed to define the high- and low-grade concentrates. The low-grade concentrate was defined as a concentrate with 10% nickel grade. The high-grade concentrate was defined as a concentrate with 35% nickel grade. Samples 103-V11-Mar21, XPS-55-M1, EZ-28-V14, CR20-57 V2 and 105-V7-Mar21 produced concentrates in the high-grade stream that came close to the 35% but were under. For these tests, the high-grade concentrate was still used to calculate the split.

Table 13.16 summarizes the nickel split between the high grade and low-grade concentrates for each of the metallurgical variability tests. The data is sorted by S/Ni ratio and domain.

Figure 13-8 shows the percentage of nickel recovered to the high-grade concentrate as a function of the feed S/Ni ratio. The split in nickel recovery to the high- and low-grade concentrates seems to correlate with the S/Ni ratio, which is a proxy for the mineralogy. For samples with a lower S/Ni ratio where there was a split between the high- and low-grade concentrates, there seems to be two different clusters of data. As the S/Ni ratio increases, the amount of pentlandite in the feed also increases resulting in less nickel being recovered in the high-grade concentrate.

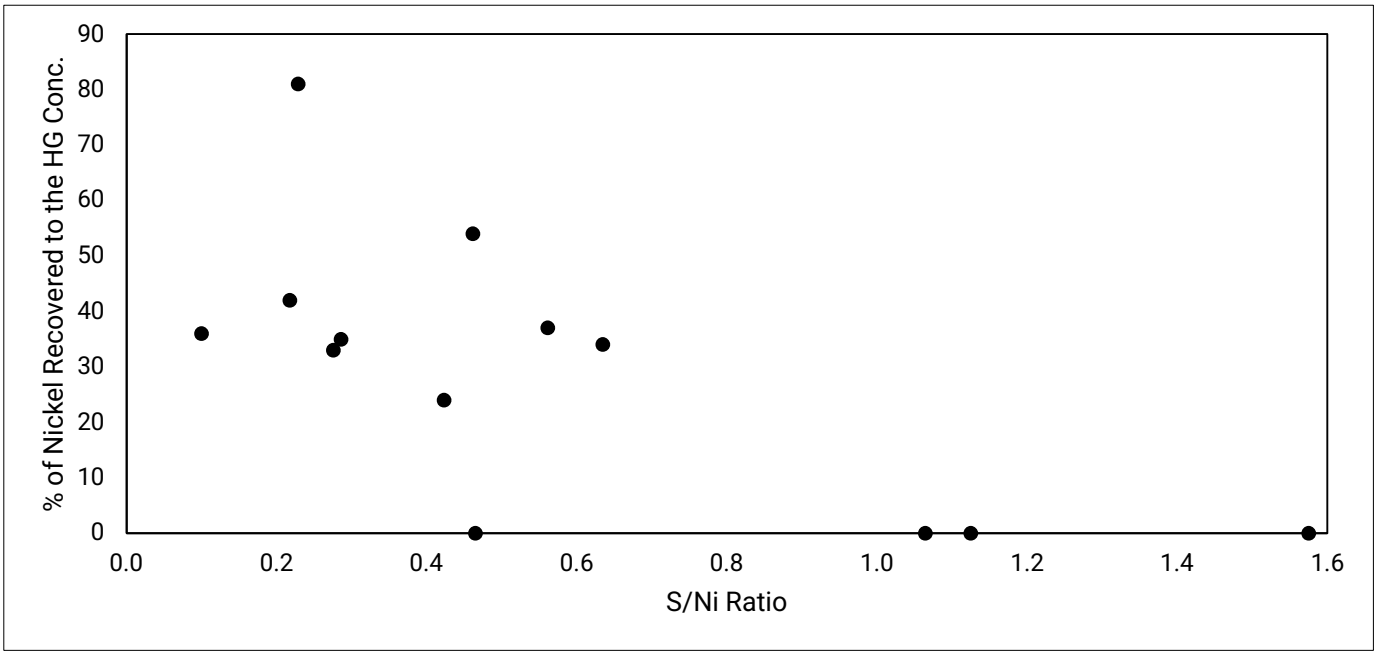
Sample CR20-55 LCT1B is an outlier on Figure 13-8. This sample has a S/Ni ratio of 0.47 and in the open circuit test, all of the nickel was recovered to the low-grade concentrate. This test utilized old conditions that did not promote good cleaning. Using the most up to date flowsheet, it is expected that this sample would yield a fraction of the nickel in a high-grade concentrate. This is supported by the mineralogy which indicates that there is heazlewoodite in this sample (refer to Table 13.12).

Table 13.16: Metallurgical Variability Test Results Summary – Ni Split between High- and Low-Grade Concentrates

Sample ID	Total Recovery (%)	% of recovered Ni in the HG Conc.	% of recovered Ni in the LG Conc.	HG Conc Grade	LG Conc Grade	Ni	S/Ni	Domain
EZ-33-V15[2]	64	81	19	35	10	0.36	0.23	HGC
103-V10-Mar21	38	33	67	35	10	0.29	0.28	HGC
103-V11-Mar21	30	35	65	32	11	0.28	0.29	HGC
103-V1-Mar21	44	24	76	35	10	0.26	0.42	HGC
103-V13-Mar21	57	54	46	35	10	0.26	0.46	HGC
CR20-55 LCT1B[2]	56	0	100	-	-	0.43	0.47	HGC
XPS 55-M1[2]	52	37	63	33	10	0.36	0.56	HGC
EZ-28-V14[2]	56	34	66	33	10	0.43	0.63	HGC
107-V12-Mar21	55	0	100	-	-	0.31	1.06	HGC
107-V8-Mar21	58	0	100	-	-	0.32	1.13	HGC
COREM - PN Master[2]	52	0	100	-	-	0.40	1.58	HGC
CR20-57 V2	28	36	64	32	10	0.20	0.10	NLG
57 XPS AW[2]	29	-	-	-	-	0.16	0.10	NLG
105-V7-Mar21	43	42	58	28	10	0.23	0.22	SLG

Notes: 1. HGC = high grade core, NLG = Northern lower grade, SLG = Southern lower grade. 2. Test completed with an old flowsheet and/or reagent scheme – Refer to footnotes on Table 13.15. It is assumed that these recoveries will be maintained/exceeded using the standardized flowsheet and reagent scheme outlined above. Source: CNC, 2021.

Figure 13.8: Percentage of Nickel Recovered to the High-Grade Concentrate as a Function of the S/Ni Ratio



Source: CNC, 2021.

13.5.3 Locked Cycle Tests

Locked cycle testing was completed to confirm the results of flowsheet development work as well as results generated from the metallurgical variability program. Seven locked cycle tests were completed between August 2020 to May 2021, of which three were done using the standardized flowsheet from the metallurgical variability program.

Tables 13.17 to 13.20 summarize the results of seven locked cycle tests, for which three of the tests were done with the final metallurgical variability flowsheet at a primary grind size of 80% passing 180 µm and final reagents. All samples tested in locked cycle were from the high-grade core domain.

Table 13.17: Locked Cycle Test Results Summary – Nickel

Sample ID	Nickel Recovery			Nickel Grade		Feed Characteristics		
	Total (%)	Flotation Conc. (%)	Magnetic Conc. (%)	Flotation Conc. (% Ni)	Magnetic Conc (% Ni)	Ni (%)	S/Ni	Primary Grind (µm)
COREM - PN Master[1]	53	50	2.9	15	0.29	0.40	1.6	187
107-V12-Mar21[1]	58	56	2.1	15	0.23	0.31	1.1	178
103-V13-Mar21[1]	50	48	2	28	0.09	0.26	0.46	172
XPS - PN Master	47	37	9.8	14	0.82	0.34	1.5	178
XPS 55-M1	55	49	6.3	20	0.31	0.36	0.56	148
CR20-55-LCT1B[2]	53	51	2	14	0.14	0.43	0.47	134
Comp3[2]	52	46	6	15	0.19	0.27	0.44	133

Notes: 1. Test utilized the most up to date flowsheet and a primary grind size of 80% passing 180 µm. 2. In sample CR20-55-LCT1B, an additional 1.9% of nickel recovery at a 10% concentrate grade was added to the flotation recovery to capture recovery from the slimes circuit which was completed on this test in open circuit. An additional 1.8% nickel recovery to 15% nickel grade was completed on Comp3 utilizing the cleaning performance achieved in CR20-55-LCT1B. Source: CNC, 2021.

Table 13.18: Locked Cycle Test Results Summary – Iron

Sample ID	Iron Recovery			Iron Grade		Primary Grind (µm)
	Total (%)	Flotation Conc. (%)	Magnetic Conc. (%)	Flotation Conc. (% Fe)	Magnetic Conc (% Fe)	
COREM - PN Master[1]	40	9.2	31	43	48	187
107-V12-Mar21[1]	37	7.1	30	32	53	178
103-V13-Mar21[1]	47	1	46	14	49	172
XPS - PN Master	36	4.6	31	29	44	178
XPS 55-M1	47	2.9	44	24	45	148
CR20-55-LCT1B	45	4.1	41	21	55	134
Comp3	55	2.1	53	19	46	133

Note: Test utilized the most up to date flowsheet and a primary grind size of 80% passing 180 µm. Source: CNC, 2021.

Table 13.19: Locked Cycle Test Results Summary – Chromium

Sample ID	Chromium Recovery			Chromium Grade		Primary Grind (µm)
	Total (%)	Flotation Conc. (%)	Magnetic Conc. (%)	Flotation Conc. (% Cr)	Magnetic Conc (% Cr)	
COREM - PN Master[1]	21	0.52	20	0.32	4.3	187
107-V12-Mar21[1]	22	0.96	21	0.61	5.4	178
103-V13-Mar21[1]	21	0.24	21	0.24	1.6	172
XPS - PN Master	26	0.55	25	0.47	4.8	178
XPS 55-M1	37	0.5	36	0.16	1.4	148
CR20-55-LCT1B	24	1.1	23	0.33	1.8	134
Comp3	36	0.52	35	0.44	2.7	133

Note: Test utilized the most up to date flowsheet and a primary grind size of 80% passing 180 µm. Source: CNC, 2021.

Table 13.20: Locked Cycle Test Results Summary – Distribution of Nickel between the High- and Low-Grade Concentrates

Sample ID	Total Recovery (%)	% of recovered Ni in the HG Conc.	% of recovered Ni in the LG Conc.	HG Conc Grade (% Ni)	LG Conc Grade (% Ni)	Ni	S/Ni
COREM - PN Master[1]	53	0	100	-	15	0.40	1.58
107-V12-Mar21[1]	58	0	100	-	15	0.31	1.06
103-V13-Mar21[1]	50	75	25	40	15	0.26	0.46
XPS - PN Master	47	0	100	-	14	0.34	1.54
XPS 55-M1	55	51	49	35	12	0.36	0.56
CR20-55-LCT1B	53	17	83	30	12	0.43	0.47
Comp3	52	18	82	30	10	0.27	0.44

Note: Test utilized the most up to date flowsheet and a primary grind size of 80% passing 180 µm. Source: CNC, 2021.

From the above tables, the following observations were made:

- The samples tested capture pentlandite dominant, mixed pentlandite-heazlewoodite and heazlewoodite dominant mineralization.
- The differences in grind size between locked cycle tests illustrates the evolution of the flowsheet, which first utilized a P_{80} of 135 µm. The primary grind size was increased to 80% passing 180 µm with no material loss in recovery and a substantial reduction in energy and reagent requirements.
- Recoveries for nickel, iron and chromium are reported as total recoveries to the flotation concentrates and the magnetic concentrate. The majority of the nickel is recovered to the flotation concentrate while the majority of the iron and chromium are recovered to the magnetic concentrate.
- The nickel recovery to the flotation concentrate was in the range of 37% to 56% with the grades ranging from 14% to 28% nickel. Total recoveries across the locked cycle tests including the flotation and magnetic concentrates were in the range 47% to 58%.

- The iron recovery to the magnetic concentrate was in the range of 30% to 53% with grades ranging from 44% to 55%. Total iron recoveries across the locked cycle tests were in the range of 36% to 55%.

- The chromium recovery to the magnetic concentrate was in the range of 20% to 36% with grades ranging from 1.4% to 5.4%. Total chromium recoveries across the locked cycle tests were in the range 21% to 37%.

- Recoveries in locked cycle confirm the recoveries from the metallurgical variability tests. Samples XPS-55 M1, 107-V12-Mar21 and the COREM PN-Master exceeded the recoveries in open circuit testing, while 103-V13-Mar21 and CR20-55-LCT1B were slightly lower than what was achieved in open circuit. The similar recoveries between open circuit and locked cycle tests formed the basis for defining the open circuit recovery (coarse cleaner 1 concentrate + fine rougher concentrate + magnetic concentrate). This is discussed further in Section 13.6.

- The distribution of recovered nickel between low- and high-grade concentrates is a function of the S/Ni ratio and the mineralogy. For samples with a S/Ni > 1, all nickel was recovered to the low-grade concentrate, as the primary mineral was pentlandite. Samples that had a S/Ni < 1 produced both a low- and high-grade concentrate due to the mixed pentlandite–heazlewoodite mineralization.

Table 13.21 summarizes the nickel accountability for each of the seven locked cycle tests. The nickel accountability was calculated relative to the fresh feed mass to understand the stability of the locked cycle tests. Only the cycles that were used in the recovery calculation were used to calculate the nickel accountability.

Table 13.21: Nickel Accountability and Balance in Locked Cycle Tests

Sample ID	Nickel in Feed (%)	Nickel in Flotation + Mag Concentrates (%)	Nickel in Tails (%)	Nickel Accountability (%)
COREM - PN Master[1, 2]	100	53	45	98
107-V12-Mar21[1, 2]	100	56	40	96
103-V13-Mar21[1, 2]	100	51	50	101
XPS - PN Master	100	45	51	96
XPS 55-M1	100	56	45	101
CR20-55-LCT1B[2]	100	47	42	89
Comp3[2]	100	47	43	90

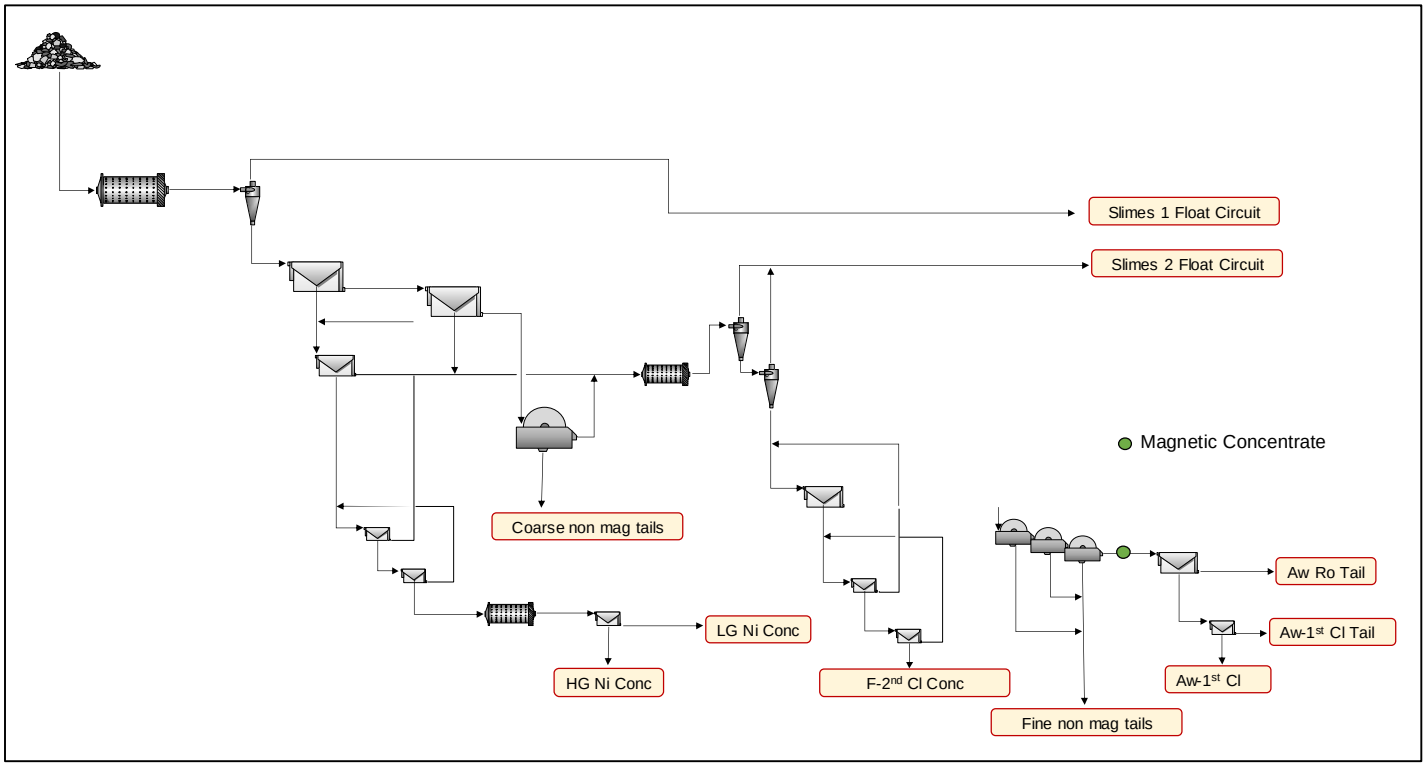
Notes: 1. Test utilized the most up to date flowsheet and a primary grind size of 80% passing 180 µm. 2. Fresh feed mass for each cycle was not measured. The target of 10 kg / cycle was assumed as the fresh feed mass for each cycle. Source: CNC, 2021.

For the locked cycle tests completed with a target P_{80} of 180 µm, the nickel accountability was in the range of 96% to 101%. This suggests that the locked cycle tests had stabilized as the nickel in the fresh feed was close to the nickel contained in the locked cycle test products. Across the seven tests, the nickel accountability was in the range of 89% to 101%.

13.5.3.1 Locked Cycle Test Flowsheet

The locked cycle tests were completed using a series of different flowsheets. Figure 13-9 shows the final flowsheet that was used in the variability program. For other locked cycle tests, the flowsheet layout was similar. However, reagent strategies were different to accommodate differences in the primary grind size.

Figure 13.9: Locked Cycle Test Flowsheet



Source: CNC, 2021.

In the locked cycle tests different strategies were tested on the slimes fractions. Because of the variability in the slimes processing techniques, that part of the flowsheet is not included in Figure 13-9.

13.5.3.2 Locked Cycle Test Reagent Usage

Table 13.22 summarizes the reagents used in each stage of the locked cycle tests that were completed at a primary grind size of 80% passing 180 µm, with the following comments:

- Reagent usage is calculated as averages of the following locked cycle tests: 103-V13-Mar21, 107-V12-Mar21 and COREM PN Master. These samples were selected as they utilized the reagent scheme and flowsheet that was consistent with the metallurgical variability testwork.
- CMC was used as a slime dispersant.
- KAX51 was added as a collector.
- Acid was added to the coarse rougher, coarse scavenger and awaruite flotation cells. In the coarse rougher-scavenger, acid addition rates independent of pH, while in the awaruite flotation cell, acid addition was adjusted to achieve pH 3.5 in the cell.

- Two types of frothers were used in this test program: MIBC and Aerofroth-65 (A-65). MIBC was used in the coarse rougher, coarse scavenger and fine rougher flotation cells. In the cleaning cells, either a 1:2 blend of Aerofroth-65 (A-65) and MIBC, or pure MIBC was added as the frother.

Table 13.22: Locked Cycle Test Reagent Usage

Stage	Reagents (g/t)				Time (min)	
	H_2SO_4 (98%)	KAX51[1]	CMC	Frother	Cond.	Froth
Primary Grind						
Deslime						
Rougher	1000	50	100	11	10.5	6
Scavenger	666	30	0	5.1	2	12
Cleaner 1		10	15	0.8	6	10
Cleaner 2		2	1.3	0.5	2	5
Cleaner 3		0	0	0.2	0.5	3
Coarse Cleaner Regrind						
Cleaner 4		13	0	0.5	2.7	4
Magnetic Separation on Scavenger Tails						
Regrind						
Secondary Deslime						
Fine Rougher		20	30	5.2	6.5	6[2]
Fine Cleaner 1		2	0.8	1.4	4.7	4
Fine Cleaner 2		1	0.4	2.4	0.7	3
Mag Sep						
AW Flotation	1983	16		16	6.5	6
Total	**3649**	**144**	**147**	**43**	**42**	**59**

Notes: 1. KAX51 = Potassium amyl xanthate. 2. Locked cycle test on sample 103-V13-Mar21 used an 8-minute float time in the fine rougher scavenger.
Source: CNC, 2021.

13.5.4 Metallurgical Optimization Work

Metallurgical testwork was completed between August 2020 and March 2021 before starting the metallurgical variability testwork. Development work used samples from holes CR20-55 and CR20-58 to test and optimize the response of material with different mineralization types to the Crawford flowsheet. Heazlewoodite and mixed heazlewoodite-pentlandite samples were sourced from hole CR20-55 (S/Ni < 1) and pentlandite dominant material was sourced from hole CR20-58 (S/Ni > 1).

Table 13.23 highlights the main changes that were made to the Crawford flowsheet as a result of the development work.

Table 13.23: Metallurgical Optimization – Summary of Main Changes to the Crawford Flowsheet

	Primary Grind Size	Coarse Circuit Acid Addition	Rougher CMC	Scavenger Float Time
Initial Flowsheet	135 µm	0 g/t	300 g/t	6 min
Final Flowsheet	180 µm	1666 g/t	100 g/t	12 min

Source: CNC, 2021.

The metallurgical variability test program used a primary grind 80% passing size (P_{80}) of 180 µm. However, the early flowsheets utilized a primary grind P_{80} of 135 µm. It was found that with the use of acid in the coarse rougher scavenger circuit, recoveries could be sustained using a primary grind P_{80} of 180 µm. The acid helps promote the flotation of liberated and composite sulphide particles by removing slime coatings on the mineral surface. By increasing the primary grind size and utilizing acid in the coarse circuit, increased throughput and lower energy costs for grinding were realized.

Increasing the primary grind size also had a positive impact on the reagent consumption. CMC consumption in the coarse circuit was reduced from 300 g/t at 135 µm primary grind size to 100 g/t at 180 µm primary grind size. This is explained by the production of less slimes in the comminution process, which requires less CMC.

The three main nickel-bearing minerals—pentlandite, heazlewoodite and awaruite—all behave differently in the flowsheet. Awaruite is an alloy mineral which is recovered through magnetic circuits while pentlandite and heazlewoodite are both sulphide minerals that float with different kinetics. It was found that recovery could be improved by extending the scavenger float time from 6 to 12 minutes. Increasing the recovery in the scavenger flotation cell reduced the likelihood of losing the nickel to the coarse non-magnetic tailings.

13.5.4.1 Slimes Flotation

Various strategies were tested on the slimes fraction to increase nickel recovery including column cell flotation, Denver cell flotation, treating the primary and secondary deslime products as a combined feed, treating the primary and secondary deslime products separately, and using a hydrocyclone before the slimes float to eliminate fibers.

The flowsheet that produced the best result utilized a column flotation cell, 3000 g/t of acid, 75 g/t of CMC and 15 g/t of potassium amyl xanthate and was completed by COREM. This test was completed in an open cycle test on sample CR20-55-LCT1B at a primary grind size of 80% passing 127 µm on the primary deslime overflow. This sample was also utilized for one of the seven locked cycle tests. Only two of the seven locked cycle tests included any slimes recovery.

The results from this test are discussed:

- The unit cell nickel recovery was 9% in this test.
- The slimes product was upgraded from 0.3% Ni in the column cell feed to a 10% Ni slimes concentrate.

For locked cycle tests completed on sample CR20-55-LCT1B and Comp3, a slimes rougher concentrate was produced; however, no cleaning was done. Based on the cleaning results from the same sample in open cycle test discussed above, 1.9% nickel recovery to a slimes concentrate with a grade of 10% nickel was calculated and incorporated into the recovery result for CR20-55-LCT1B. For the locked cycle test completed on sample Comp3, 1.8% nickel recovery to a slimes concentrate with a 15% nickel grade was calculated based on the slimes rougher concentrate produced during the locked cycle test and applying the previously achieved cleaning performance from the other test.

There has been very little optimization work done around the slimes flotation circuit and represents an opportunity for potential improvement in nickel recovery from the Crawford flowsheet.

13.6 Recovery Equations

The recovery equations were designed using both open circuit and locked cycle test results outlined in Sections 13.5.2 and 13.5.3.

Locked cycle tests simulate a continuous circuit with recirculating loads and indicate how material will behave in a plant setting with a given flowsheet and reagent scheme. The results of locked cycle testing thus provide an indication of the recoveries that would be expected in a commercial plant.

Open circuit tests are single batch tests that do not have recirculating loads. In this metallurgical test program, the recovery definition in open circuit tests was defined as the sum of the recoveries to the coarse cleaner 1 concentrate, fine rougher concentrate and the magnetic concentrate. The coarse cleaner 1 and fine rougher concentrates are the feeds into the cleaning circuits and provide an indication of what might be expected to be recovered in locked cycle test cleaning circuits. The magnetic concentrate is a final product and does not rely on recirculating loads. The rationale behind using this open circuit definition of recovery to define the recovery equations is justified by the results summarized in Table 13.24, which compares the results for locked cycle and open circuit tests completed on the same sample.

Table 13.24: Comparison of Open Circuit and Locked Cycle Test Results – Nickel

Sample ID	Open Circuit Tests			Locked Cycle Tests			Difference in Total Recovery (LCT - OCT) (%)
	Total Ni Rec. (%)	Flotation Rec. (%)	Flotation Conc. Grade (% Ni)	Total Ni Rec. (%)	Flotation Rec. (%)	Flotation Conc. Grade (% Ni)	
COREM - PN Master[1]	52	49	8.5	53	51	15	1
107-V12-Mar21[1]	55	54	8.7	58	56	15	3
103-V13-Mar21[1]	57	55	10	50	48	28	-7
CR20-55-LCT1B[2]	56	54	2.3	53	51	14	-3
XPS 55-M1[2]	52	48	6.9	55	49	20	3

Notes: 1. Test utilized the most up to date flowsheet and a primary grind size of 80% passing 180 µm. The slimes cleaning circuit was not the proven flowsheet. Recovery from the slimes fraction is not included in the locked cycle test result for these tests. 2. Test completed using an old flowsheet at a finer primary grind size. Slimes cleaning circuit used the proven flowsheet and is thus included in the locked cycle test recovery result. Source: CNC, 2021.

The following observations support Table 13.24:

- Recoveries in locked cycle tests were greater than in open cycle tests for three of the five tests summarized in Table 13.24. The average difference in total recovery between locked cycle and open cycle tests was -1%, which shows that this definition of open circuit recovery does provide an indication of the recoveries that would be expected in locked cycle.

- The flotation concentrate grade increased in locked cycle as compared to open cycle tests because the concentrate was a final product from the flowsheet rather than an intermediate product (coarse cleaner 1 and fine rougher concentrate for the open circuit tests). The flotation concentrate grade for locked cycle tests completed at a primary grind size of 180 µm averaged 19% nickel.

The recovery equations were defined based on 14 open circuit and 7 locked cycle tests using samples that were selected to cover the breadth of Crawford. Samples were selected from eight drill holes targeted primarily within the Main Zone high-grade core domain. The high-grade core is the primary domain that will be processed during the project payback period.

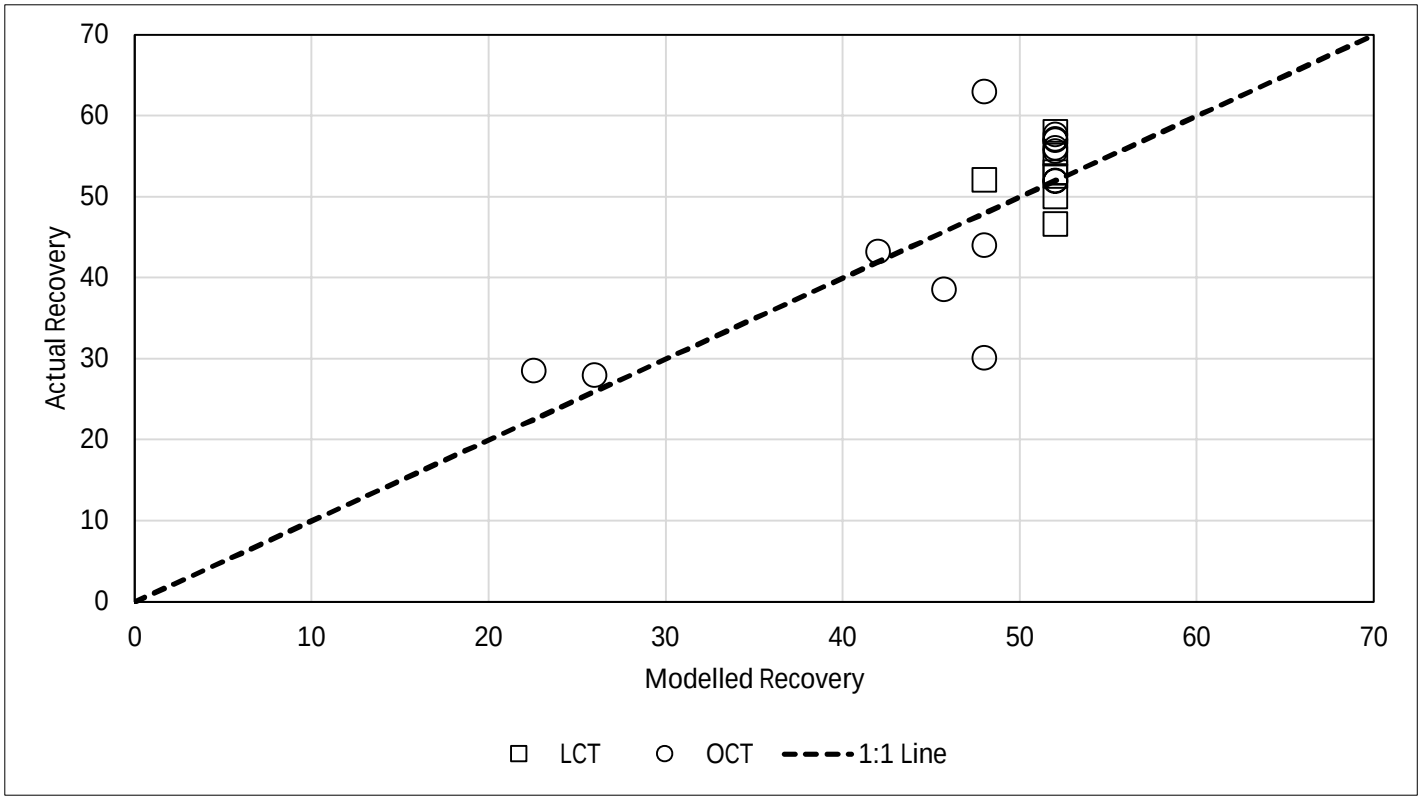
This section outlines the recovery equations for nickel, as well as the by-product streams for iron, chromium, and cobalt.

13.6.1 Nickel

Nickel recovery equations were developed for each of the three domains. These domains were created to represent the degree of serpentinization of the material, as well as the sulphur content, as both variables have an impact on nickel recovery. Based on the mineralogy, the north low-grade domain appears to be more serpentinized than the south low-grade domain, which is expected to result in higher nickel recoveries.

Figure 13-10 compares the actual recoveries to the modelled recoveries for the 14 open circuit tests and 7 locked cycle tests.

Figure 13.10: Nickel Recovery Reconciliation – Actual versus Modeled[1]



Note: When assigning the S/Ni domain to the sample, the reconciled assays specific to that test sample were used instead of the original head assays to determine the modelled recovery. Source: CNC, 2021.

From Figure 13-10, the following observations were made:

- Of the seven locked cycle tests, five achieved higher recoveries than were modelled. The average nickel grade of the flotation concentrate across the seven locked cycle tests was 17.0%.
- Of the 14 open circuit tests in the metallurgical variability program, ten achieved recoveries higher than modelled. The average nickel grade across the 14 open circuit tests was 6.7% (combined grade of the coarse cleaner 1 and fine rougher concentrate). Although this product does not represent the final product from the flowsheet, the grade of the intermediate concentrate in open circuit is suitable for further cleaning into a saleable product.

Within each domain, metallurgical domains were created using the S/Ni ratio. The S/Ni ratio is a good proxy for the mineralogy present in the resource whereby awaruite and heazlewoodite typically coexist in material with lower S/Ni ratios and pentlandite predominates in material with high S/Ni ratios (>1). There is also a transition zone for S/Ni ratios between about 0.5 to 1.0, where mixed heazlewoodite + pentlandite mineralization can be expected. Across all the samples used to design the recovery equations, the S/Ni ratio fell in the range of 0.1 to 1.6.

The higher-grade core, northern lower-grade and southern lower-grade domains have distinct recovery equations when the S/Ni is less than 0.3. If the S/Ni ratio is ≥ 0.3, the recovery was modelled as a constant. Table 13.25 summarizes the nickel recovery equations according to the mine and metallurgical domains.

Table 13.25: Ni Recovery Equations for the Metallurgical Domain S/Ni < 0.3

Metallurgical Domain	Ore Domain	Recovery Equation
S/Ni < 0.3	High Grade Core	Ni Recovery (%) = 90 · (S/Ni) + 20
	North Low Grade	Ni Recovery (%) = 77 · (S/Ni) + 18
	South Low Grade	Ni Recovery (%) = 77 · (S/Ni) + 15
0.3 ≥ S/Ni < 0.5	High Grade Core	Ni Recovery (%) = 48
	North Low Grade	Ni Recovery (%) = 45
	South Low Grade	Ni Recovery (%) = 42
S/Ni ≥ 0.5	All Domains	Ni Recovery (%) = 52

Source: CNC, 2021.

Using the results of open circuit and locked cycle tests, equations to model the recovery of nickel to the high- and low-grade concentrates were developed. The equations that model the distribution of nickel between high- and low-grade concentrates are summarized in Table 13.26 according to the S/Ni ratio of the feed. The equations in Table 13.26 are independent of the mine domain (high grade core, Northern lower grade and Southern lower grade) and are constant values. Based on the path to market, there is no impact on the net nickel value according to where the nickel reports to (high grade versus low grade concentrate).

Table 13.26: Modelled Equations for the Distribution of Nickel Between High- and Low-Grade Concentrates

Metallurgical Domain	% of Nickel Recovered to High Grade Concentrate	% of Nickel Recovered to Low Grade Concentrate
S/Ni < 0.1	30	70
0.1 ≥ S/Ni < 0.5	40	60
0.5 ≥ S/Ni < 1	50	50
S/Ni ≥ 1	0	100

Source: CNC, 2021.

13.6.2 By-product Recoveries

In addition to nickel, Crawford will produce iron, chromium, and cobalt by-products. Recovery equations were developed to model each of these by-product streams. The metallurgical variability tests and locked cycle tests were used to validate the recovery equations.

13.6.2.1 Iron

The Crawford deposit contains magnetite which is recovered as an iron by-product in the magnetic circuit. Iron recovery is measured for the combined nickel sulphide concentrate and the magnetic concentrate. Iron recovery was modelled as the sum of the flotation and magnetic concentrates because the products from the mineral processing flowsheet will all be used for stainless steel production. Thus, the iron units from the sulphide, as well as the magnetic concentrate would be realized in the downstream processing facilities.

The grades and recoveries achieved in locked cycle tests for iron are summarized in Table 13.18 in Section 13.5.3. The grades produced in open circuit testwork were comparable to that which was achieved in locked cycle.

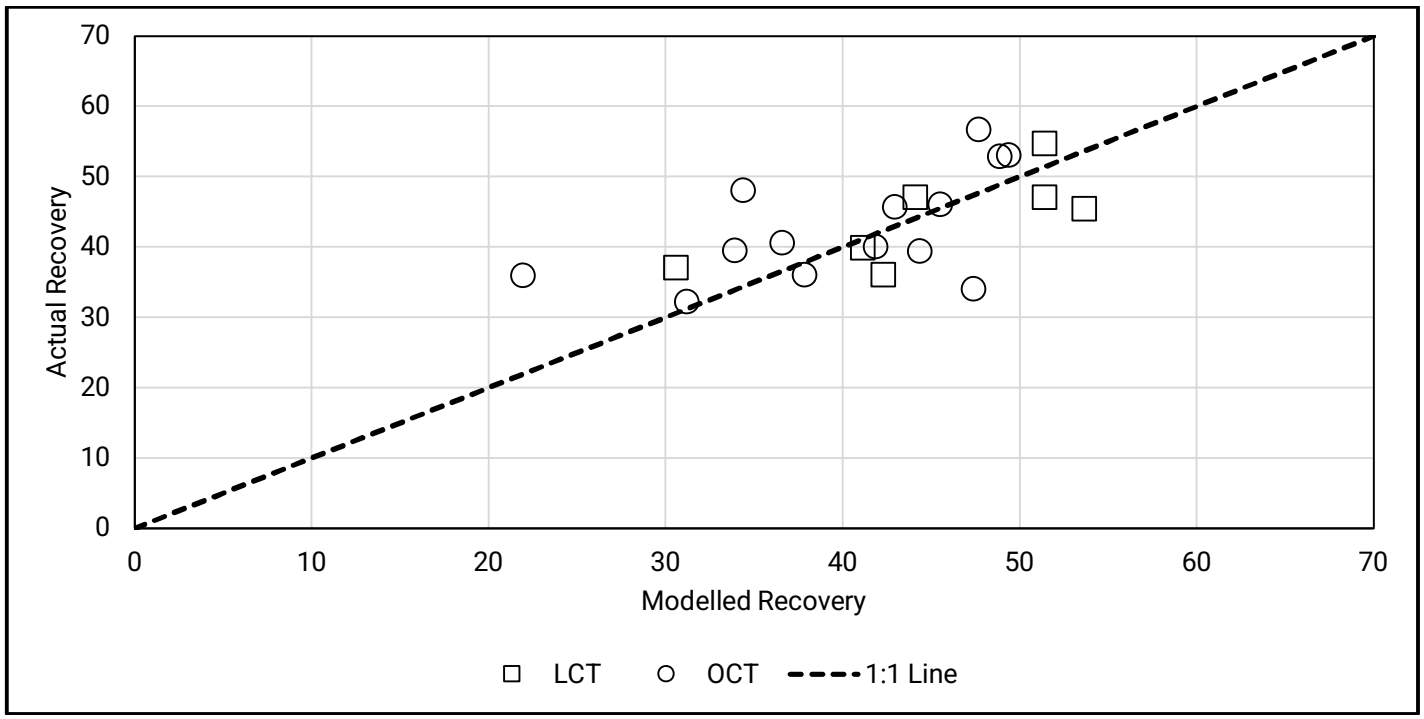
For iron, a single recovery equation was applied across the mine domains and no metallurgical domains were created. Iron recovery was capped at 70% of the magnetite within a given mining block. The equation used to model iron recovery was:

$$Fe\ Recovery\ (\%) = 45.214 \cdot LN\cdot(\%\ Fe\ in\ Feed) - 38.401$$

This relationship was established mid-way through the metallurgical variability test program and thus, the samples used to build this relationship are not all captured within this Report.

Figure 13-11 compares the actual and modelled recoveries for the 14 open circuit and seven locked cycle tests.

Figure 13.11: Iron Recovery Reconciliation – Actual versus Modelled [1]



Note[1]: The reconciled head Fe assay was used to calculate the modelled recovery rather than the measured head assay. Source: CNC, 2021.

From Figure 13-11, the following observations were made:

- Three of the seven locked cycle tests had iron recoveries that were greater than what were modelled. Two of three tests achieved recoveries greater than what was modelled were completed at a primary grind size of 180 µm using the most up to date flowsheet. Across the seven locked cycle tests, the average difference between modelled and actual iron recovery was -1%, which suggests that the modelled recoveries are close to what is being achieved in the metallurgical testwork.

- Ten of the 14 open circuit tests had iron recoveries that were greater than what were modelled. In open circuit testing, the actual iron recovery was on average 3% higher than modelled.

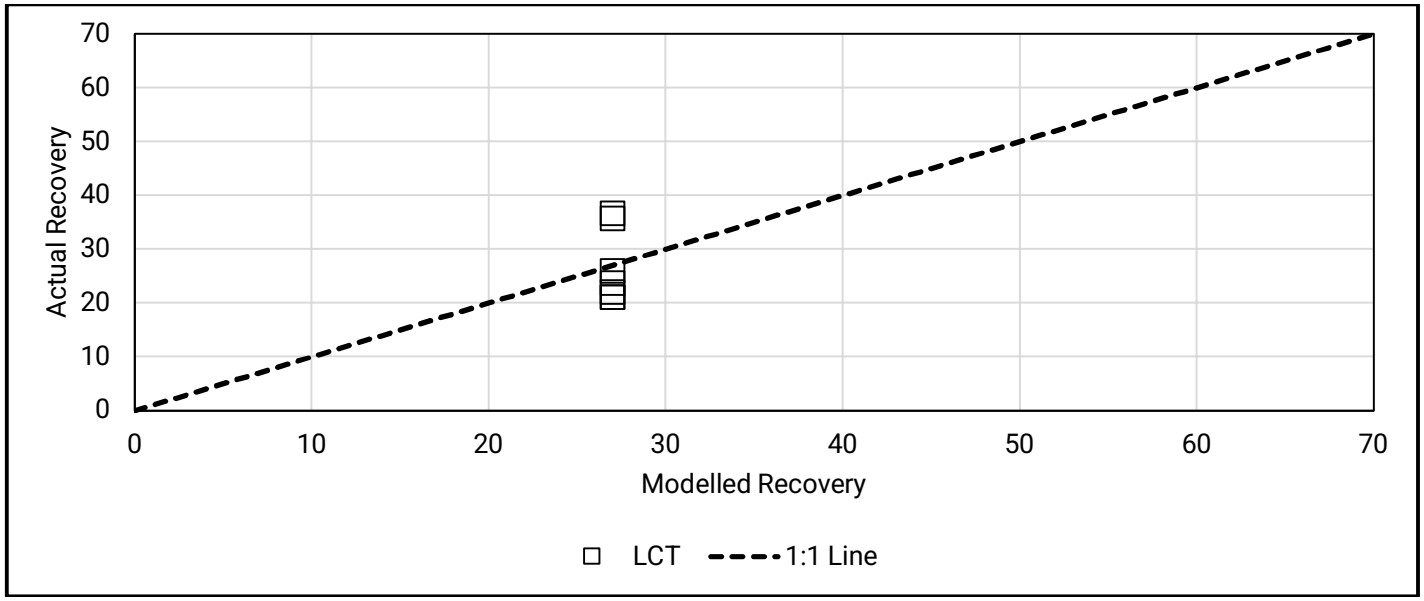
Flowsheet development efforts to date have focused on maximizing nickel recovery through the flotation circuit. The iron recoveries that have been achieved have been incurred with little to no optimization work on that part of the flowsheet. In the next phase of flowsheet development work, efforts will be taken to further increase the iron recovery.

13.6.2.2 Chromium

The chromium recovery across the deposit was assumed to be 27%, which is the average chromium recovery from the locked cycle tests. Two of seven of the locked cycle tests had chromium recoveries that were greater than what was modelled (Figure 13.12). Four of 14 of the open circuit tests had chromium recoveries that were greater than what was modelled (Figure 13.13). Samples tested in locked cycle had chromium grades in the range of 0.26% to 0.84%, which covers the 9th to 98th percentile of chromium grades within the Crawford resource.

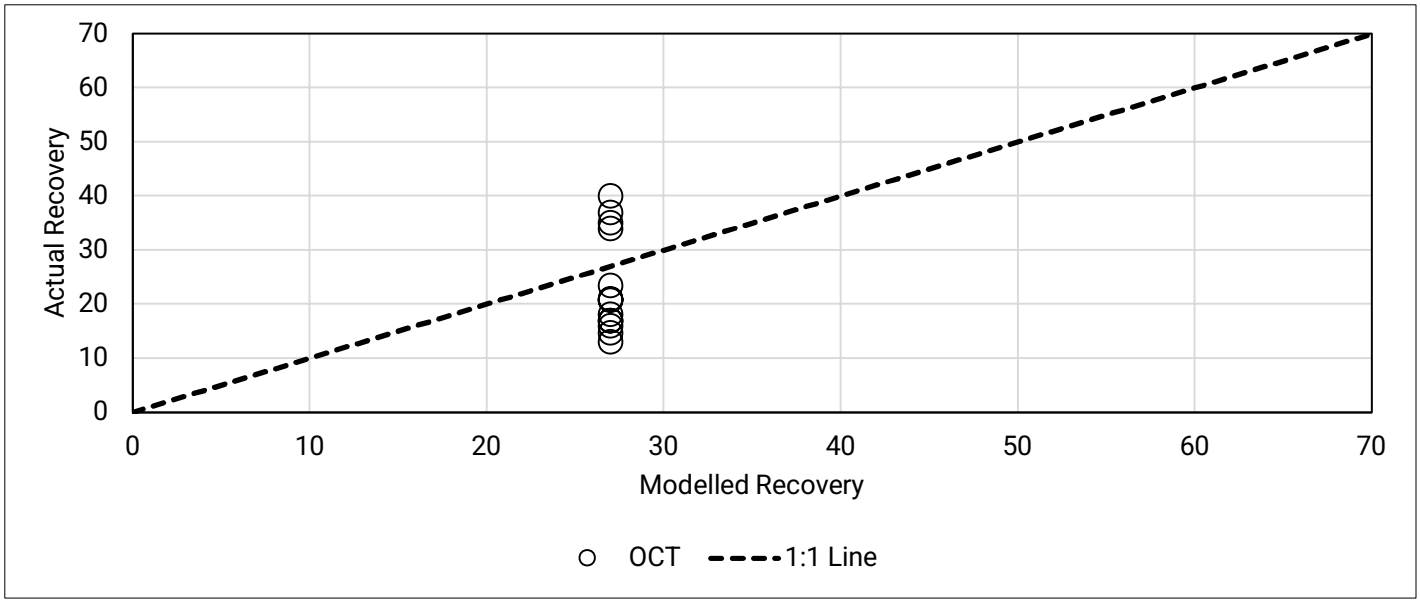
The chromium recoveries achieved to date have been incurred without any optimization work. In the next phase of flowsheet development work, efforts will be taken to increase the chromium recovery further, and to develop a more robust model for estimating its recovery.

Figure 13.12: Chromium Recovery Reconciliation – Actual vs. Modeled for Locked Cycle Tests



Source: CNC, 2021.

Figure 13.13: Chromium Recovery Reconciliation – Actual vs. Modeled for Open Circuit Tests



Source: CNC, 2021.

13.6.2.3 Cobalt

Cobalt recovery was modelled for two metallurgical domains which are defined using the S/Ni ratio. These metallurgical domains were chosen to capture different mineral assemblages in the resource. When the S/Ni ratio is greater than 1, the primary nickel-bearing mineral is pentlandite. For material with a S/Ni ratio less than 1, nickel minerals on average present as mixed heazlewoodite + awaruite mineralization and mixed heazlewoodite + pentlandite mineralization. As the S/Ni ratio increases up to the limit of 1, the proportion of pentlandite in the material increases on average. It is important to differentiate material with and without pentlandite because cobalt seems to be most associated with pentlandite.

Table 13.27 summarizes the cobalt recovery equations according to the S/Ni domain. Irrespective of the S/Ni domain, cobalt recovery was capped at 40%. The recovery was defined as that which reports to the sulphide concentrate streams. For the purposes of the PEA, the cobalt was assigned a payability of 0 because it would not be recovered in the assumed downstream processing facility.

Table 13.27: Cobalt Recovery Equations

Metallurgical Domain	Domain	Recovery Equation
S/Ni < 1	All Domains	Cobalt Recovery (%) = 89 · (S/Ni) - 28[1]
S/Ni ≥ 1	All Domains	Ni Recovery (%) = 40

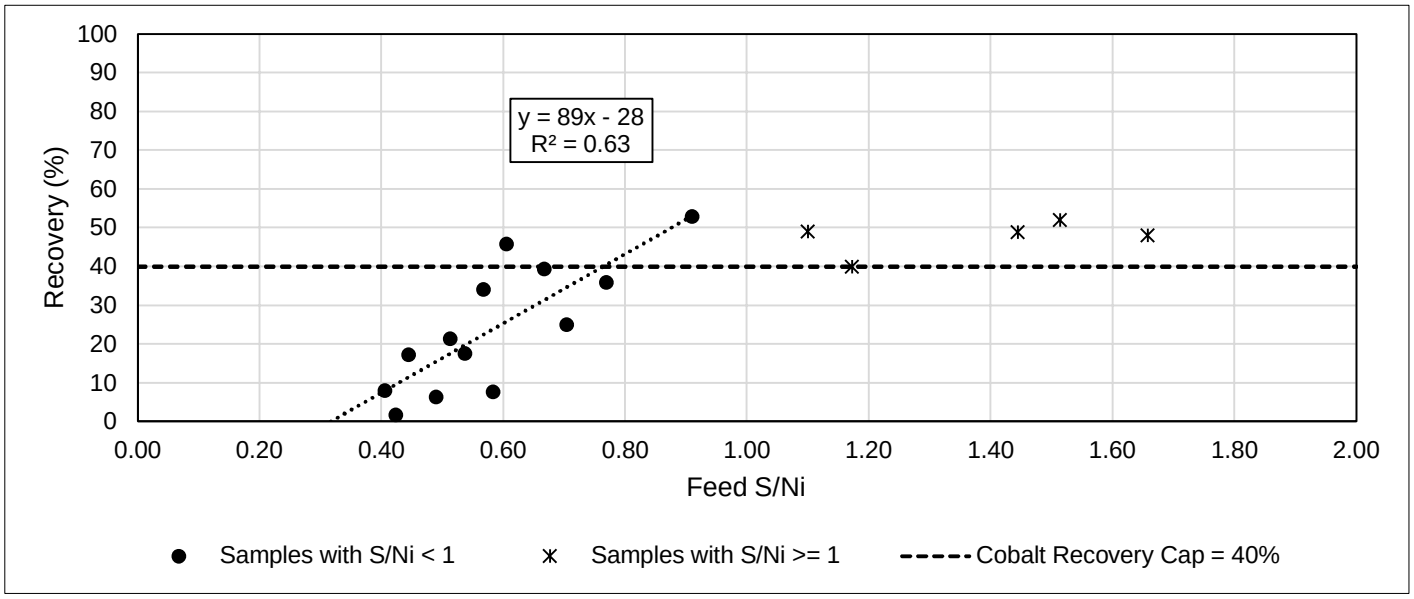
Note: Cobalt recovery is capped at 40% for material that has S/Ni ratio < 1. If a negative recovery is returned, recovery is set to 0. Source: CNC, 2021.

Figure 13-14 shows the relationship between the cobalt recovery and sample feed S/Ni, which was used to generate the cobalt recovery equations. This relationship was established mid-way through the metallurgical variability test program and thus, the samples used to build this relationship are not all captured within this Report.

The following observations are made based on Figure 13-14:

- For samples with a feed S/Ni < 1, cobalt recovery increases with the S/Ni ratio.
- Samples that had a low S/Ni ratio (<0.4) had low cobalt recoveries because cobalt tends to associate with pentlandite, and samples with lower S/Ni ratios tend to have less pentlandite in them.
- For samples with a feed S/Ni > 1, the cobalt recovery was in excess of 40%.

Figure 13.14: Cobalt Recovery as a function of the feed S/Ni



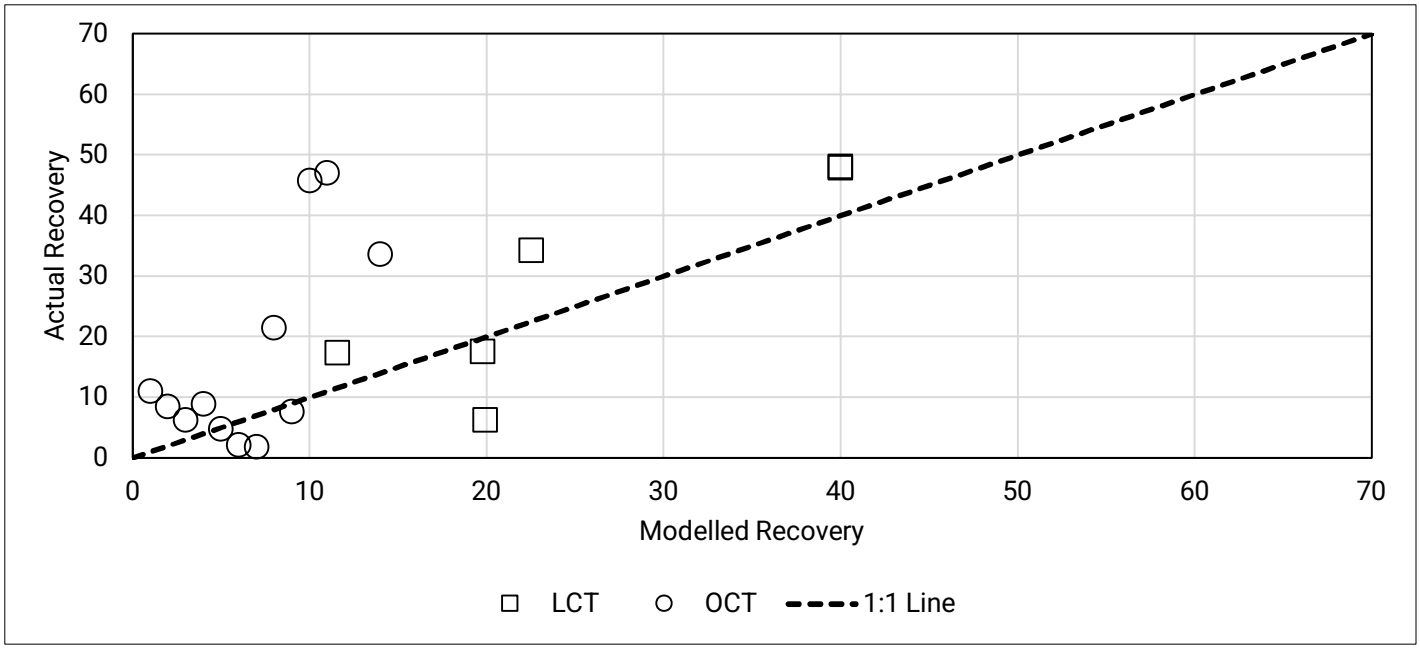
Source: CNC, 2021.

Figure 13-15 compares the actual and modelled recoveries for cobalt for open circuit and locked cycle tests with the following observations:

- Two of six locked cycle tests had cobalt recoveries that were greater than what was modelled. For one of the locked cycle tests, cobalt recoveries were deemed erroneous due to the head assay being below the detection limit.
- Seven of 12 open circuit tests had cobalt recoveries that were greater than what was modelled. For two of the open circuit tests, cobalt recoveries were deemed erroneous due to the head assay being below the detection limit.

Table 13.28 shows the cobalt deportment according to mineral using results of the microprobe work completed at XPS and Queen's University. The average cobalt tenor in pentlandite is 5.8% which falls in the range of 1.3% to 25.0%. Awaruite also shows associations with cobalt, with an average cobalt tenor of 1.7%. Heazlewoodite and serpentine show low cobalt tenors.

Figure 13.15: Cobalt Recovery Reconciliation – Actual versus Modelled[1]



Note[1]: When assigning the S/Ni domain to the sample, the reconciled assays specific to that test sample were used instead of the original head assays to determine the modelled recovery. Source: CNC, 2021.

Table 13.28: Cobalt Deportment by Mineral

Mineral	Number of Points	Average (% Co)	Standard Deviation
Pentlandite	231	5.8	6.0
Awaruite	36	1.7	0.6
Heazlewoodite	241	0.07	0.20
Serpentine	418	0.009	0.004

Source: CNC, 2021.

13.6.2.4 Platinum Group Elements

Understanding the recovery of platinum group metals was not a focus of the preliminary economic assessment and the PGE portion of the resource has been assigned a payability of 0. Concentrates produced from four locked cycle tests and one open circuit test were analyzed for platinum group metals to develop a preliminary understanding of how these metals might accumulate in the final products. However, further investigations are warranted to understand the behaviour of platinum group metals across the breadth of the deposit. Ranges of assays measured in the final products are presented below:

- Platinum assays of final products were in the range of 0.112 to 7.69 g/t Pt.
- Palladium assays of final products were in the range of 0.331 to 23.8 g/t Pd.
- Rhodium assays of final products were in the range of 0.21 to 1.73 g/t Rh.

13.6.3 Concentrate Quality

13.6.3.1 Composition

The concentrate composition from locked cycle tests executed at a primary grind P_{80} of 180 µm and using the standardized flowsheet are summarized in Tables 13.29, 13.30 and 13.31 according to the product stream (high-grade concentrate, low-grade concentrate and magnetic concentrate, respectively). Samples that are included in this summary include 103-V13-Mar21, 107-V12-Mar21 and COREM PN-Master. These samples come from the eastern part of the Main Zone, where mining will start, and capture two metallurgical domains: an intermediate S/Ni domain for the sample 103-V13-Mar21 and the high S/Ni domain for samples 107-V12-Mar21 and COREM PN-Master. The concentrates from other locked cycle tests are not summarized because they either had a finer primary grind size or utilized a preliminary flowsheet that impacted upgrading in the cleaning circuits.

Table 13.29: Quality of the High-Grade Ni Concentrate

Sample ID	% of Rec. Ni in HG Conc.	S/Ni	Ni %	Co %	Fe %	Cr %	S %	Mg %
103-V13-Mar21	75	0.46	40	0.3	11	0.2	16	6.6

Source: CNC, 2021.

Table 13.30: Quality of the Low-Grade Ni Concentrate

Sample ID	% of Rec. Ni in LG Conc.	S/Ni	Ni %	Co %	Fe %	Cr %	S %	Mg %
103-V13-Mar21	25	0.46	15	0.2	18	0.3	6.5	13
107-V12-Mar21	100	1.1	15	0.8	32	0.6	16	6.7
COREM PN-Master	100	1.6	15	0.8	42	0.3	19	3.8

Source: CNC, 2021.

Table 13.31: Quality of the Magnetic Concentrate

Sample ID	S/Ni	Ni %	Co %	Fe %	Cr %	S %	Mg %
103-V13-Mar21	0.46	0.1	BDL[1]	49	1.6	0.1	7.0
107-V12-Mar21	1.1	0.2	BDL[1]	53	5.4	0.4	3.9
COREM PN-Master	1.6	0.3	BDL[1]	48	4.3	0.8	7.3

Note: BDL = below detection limit. Source: CNC, 2021.

Sample 103-V13-Mar21 was a heazlewoodite sample that produced a high-grade concentrate with a nickel grade of 40% and a low-grade concentrate with a nickel grade of 15%.

Samples 107-V12-Mar21 and COREM PN-Master are pentlandite dominant samples for which all the nickel reported to the low-grade concentrate as expected. The nickel grade of the final concentrate produced from both samples was 15%. There was no heazlewoodite in either of these samples which is why no high-grade concentrate was produced.

The flotation and magnetite concentrates have a high MgO content which will affect the flux requirements in downstream processing. Recoveries of pay metals in the downstream flowsheet reflect the additional flux requirements as well as the impact of MgO on the downstream recoveries.

The results summarized above highlight the ability to reject magnesium-rich gangue and produce a saleable concentrate for market.

The high-grade concentrate from sample 103-V13-Mar21 had a cobalt assay of 0.3%. Within the low-grade concentrates, the cobalt grade in the final concentrate was in the range of 0.2% to 0.8% and seems to increase with the S/Ni ratio.

The magnetic concentrates across these three locked cycle tests had an average grade of 50% Fe and 3.8% chromium. There has been very little optimization work done on the magnetic circuit. Optimization of the magnetic circuit will be a focus in the next stages of flowsheet development work.

13.6.3.2 Minor Elements

Locked cycle tests completed on samples CR20-55-LCT1B and Comp3 were assayed for an extended list of components. It was found that the follow elements were either very close to or below the assay detection limit in the in the final products: Ca, K, Ti, P, Cu, Pb, and Zn.

13.7 Next Steps in Flowsheet Development Work

The next phase of flowsheet development work will focus on maximizing the recoveries of nickel, iron chromium, cobalt, and PGM in the mineral processing flowsheet. Opportunities that have been identified to aide in this next phase of work include:

- optimization of the coarse rougher-scavenger flotation circuit.
- effect of the secondary regrind size and the secondary deslime on fine circuit flotation performance.
- optimization of the fines flotation circuit.
- optimization of the slimes cleaning process.
- evaluation of alternative strategies to recover nickel that is locked in silicates from the mineral processing tailings.
- reduction in reagent consumption across the flowsheet and substitution of costly reagents such as CMC with less expensive alternatives.
- impact of regrinding the feed to the magnetic circuit on the quality of the magnetic concentrate; a trade off study will be done to understand the cost-benefit analysis of further upgrading of the magnetic concentrate on flux requirements in downstream process steps.
- thorough assessment of the size-by-size deportment of nickel in a plant flowsheet with particular reference to hydrocyclone operation which tends to classify acicular particles and particles of different specific gravities differently.
- thorough assessment of the energy sensitivity in flotation with particular focus on the selection of plant equipment.

- thorough assessment of the dewatering properties of the tailings and the resulting storage and water balance factors.

The results of the flowsheet development work listed above will be tested at the pilot scale and used to lock in a final flowsheet for the feasibility study. The estimated costs for the flowsheet development work that will be completed before the start of the feasibility study, including contingency, is $850,000 The expected cost of a 25-tonne pilot plant is $175,000, which is included in the $850,000 estimate.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

Caracle Creek was retained by CNC to prepare two updated mineral resource estimates (MREs) in accordance with NI 43-101 and supported by one technical report for the Crawford Nickel Sulphide Project, which incorporates all current diamond drilling for which the drill hole data could be confidently confirmed. Drill hole information up to May 19, 2022, was utilized in the preparation of the two updated mineral resource estimates.

The updated MRE for the Main Zone and the MRE for the East Zone, disclosed herein, were prepared by Miguel Vera (B.Sc., Geology) and this work was reviewed and signed off by qualified person Scott Jobin-Bevans (P.Geo.).

The deposit type being considered for nickel mineralization discovered to date in the Crawford Ultramafic Complex, komatiite-hosted Ni-Cu-Co-(PGE), is comparable to the Dumont nickel deposit, located in Quebec, Canada. The host Archean Dumont Sill is about 7 km long, up to 1 km in width, and like the Crawford ultramafic deposit is located within the Abitibi Greenstone Belt.

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 Crawford Ultramafic Complex. 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, likely using open pit and bulk underground mining methods.

14.2 Resource Database

The drill hole and project database provided by CNC was initially filtered by zones of interest (Main 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 Zone

The working database for the Main Zone, with complementary numbers in parenthesis for the aggregate database of Main Zone plus the new "West Zone" (given that the two were treated as a single entity), contains the following:

- Collar: 111 (177) holes amounting to 49,661.68 m (80,934.63 m), with an approximate mean depth of 450 m. Included are 8 (12) geotechnical holes (with geologs only) and 19 metallurgical holes (twin holes with complete logs but no samples).
- Survey: 110 (174) measured holes, with the remaining hole estimated from its planned direction.
- Lithology: 109 (172) holes with 23 unique rock codes, grouped into 14 codes for modelling purposes (see Section 14.4).
- Assays: 65 (116) holes with 23,270 (34,276) core samples of 1.5 m average length; 34 elements reported.
- Magnetic Susceptibility: 101 (161) holes with 24,324 (44,328) handheld magnetic susceptibility measurements on drill core, taken every 3 m during the initial campaigns, and later every 1 metre.

- Specific Gravity: 102 (162) holes with 6,176 (8,977) density measurements from drill core, taken every 7 m on average.

Secondary data sources include alteration, mineralization, and structural drill hole logs, QEMSCAN data, historical geophysical surveys (magnetic susceptibility, EM, and gravity), geological maps and various work reports.

14.2.2 East Zone

The working database for the East Zone contains the following:

- Collar: 81 holes amounting to 36,452.34 m, with an approximate mean depth of 450 m. Included are 13 geotechnical holes (with geologs only) and 3 metallurgical holes (twin holes with complete logs but no samples).
- Survey: 79 measured holes, with the remaining 2 holes estimated from their planned direction.
- Lithology: 77 holes with 19 unique rock codes, grouped into 13 codes for modelling purposes (see Section 14.4).
- Assays: 47 holes with 12,101 core samples of 1.5 m average length; 34 elements reported.
- Magnetic Susceptibility: 67 holes with 24,148 handheld magnetic susceptibility measurements on drill core, taken every 3 m during the initial campaigns, and later every 1 m.
- Specific Gravity: 68 holes with 3,166 density measurements from drill core, taken every 8 m on average.

Secondary data sources include alteration, mineralization and structural drill hole logs, QEMSCAN data, historical geophysical surveys (magnetic susceptibility, EM and gravity), geological maps and various work reports.

14.3 Methodology

The nickel resource area in the Main Zone measures approximately 1.9 km along strike, 280 to 580 m in width, and 800 m deep, while the nickel resource in the East Zone is approximately 2.6 km along strike, 200 to 350 m in width, and 800 m deep. Estimates are based on a compilation of a few historical and numerous recent diamond drill holes, along with mineralized zones prepared by Caracle Creek.

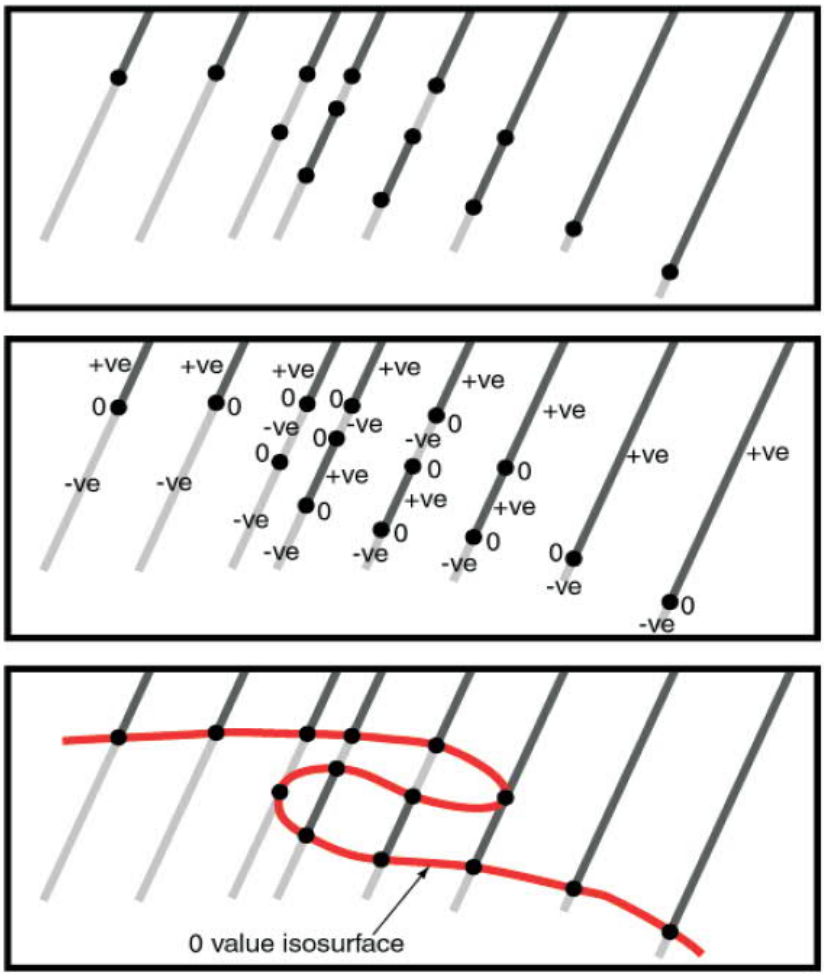
The main steps in the resource estimation methodology were as follows:

- CNC drill hole database compilation and preparation of the working database to be used in the resource estimate.
- Modelling of 3D geological units and mineralized domains based on revised lithological codes, densities, magnetic susceptibility and assay grades.
- Generation of drill hole intercepts for each mineralized zone in all relevant data tables.
- Exploratory data analysis (EDA), capping and compositing of assay grades.
- Variogram modelling and cross-validation.
- Grade interpolations (kriging, IDW, NN) and validations (visual, statistical, swath plots).
- Resource classification.

The mineral resource estimates detailed in the report were prepared using Leapfrog Geo 2021.2 (3D modelling) and Isatis.neo 2022.04 (geostatistics) software packages. The estimation used 3D block modelling, applying the ordinary kriging (OK) and inverse distance weighting (IDW) interpolation methods, depending on the domains and variables.

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



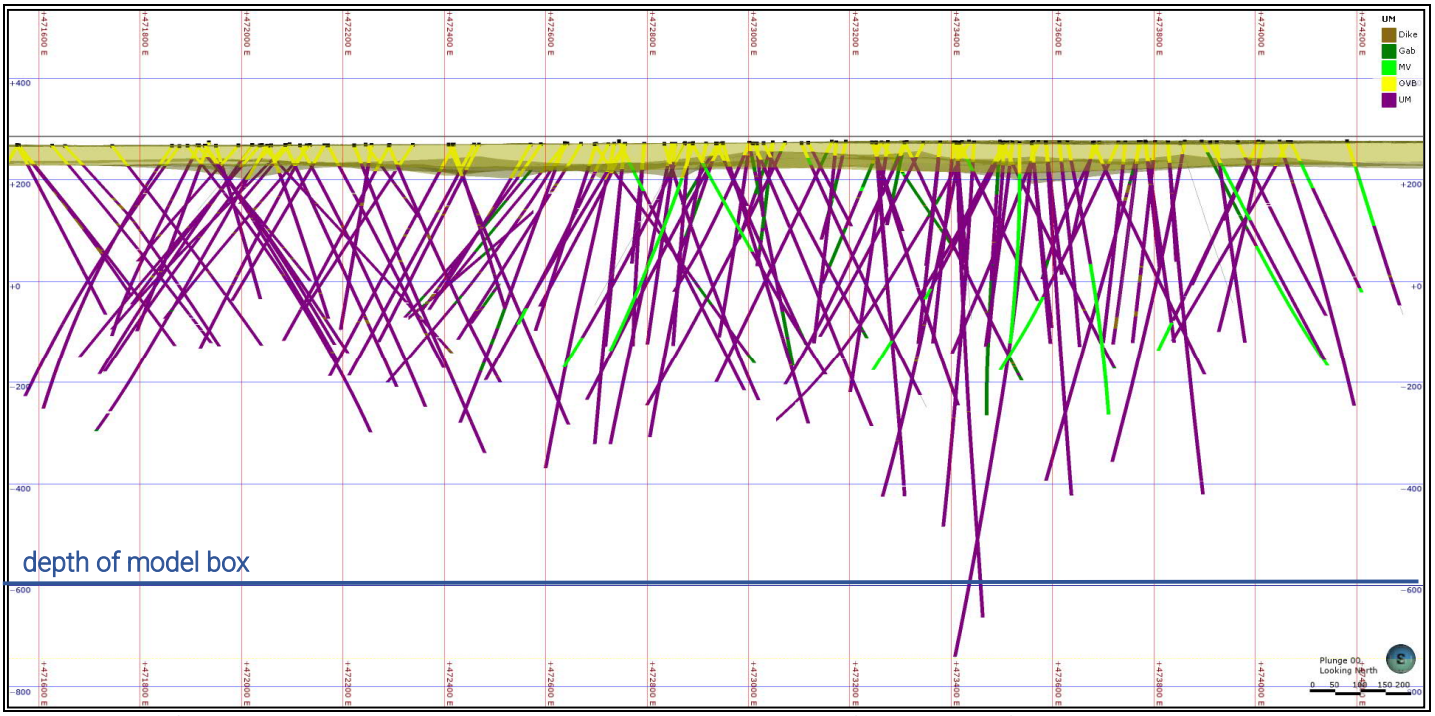
Note: Two sets of intervals (upper panel) converted into positive ("+ve" or inside) and negative ("-ve" or outside) distance points (middle panel) and the resulting interpolation through zero distance ("0" or contact) value points (lower panel). Source: Modified from Cowan et al. (2003).

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 of over 40 m on average, considered overburden. 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" drill hole intervals plus additional test holes outside of the project area.

Figure 14.2: Main Zone Longitudinal View (Looking North) Showing Drill Holes and the OVB Volume (Transparent Yellow)



Note: The ultramafic unit, colored purple in the drill hole 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, 2022.

14.4.2 Lithology

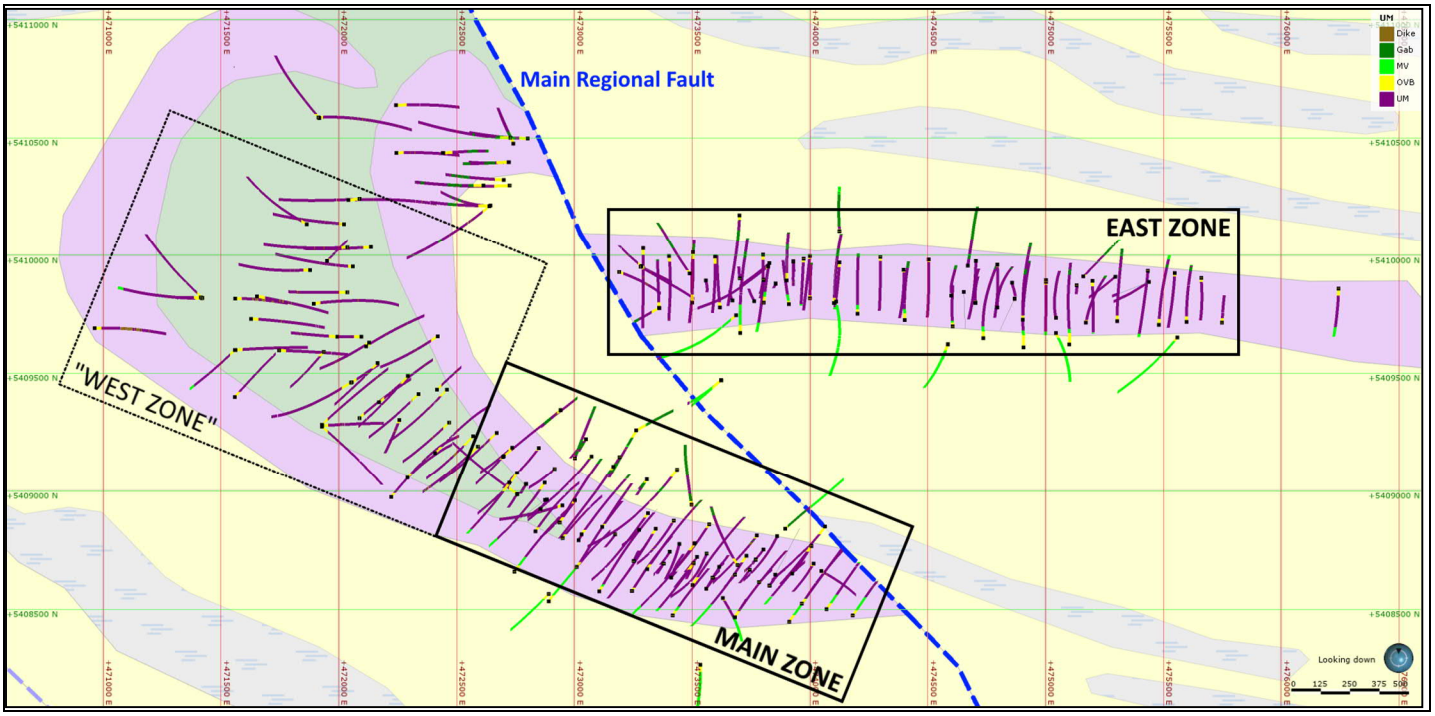
Located at either side of a main regional fault and 1 km apart, with complementary geometries, the CUC's Main and East zones are interpreted as once being part of the same body, displaced by this northwest trending structure (Figure 14.3). As such, lithologies identified in both zones share common features:

- A combination of consistently ordered, sub-vertical 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 Zone and increasingly towards the new West Zone, while in the East Zone they are few and far between.

Figure 14.3: General View of the Main and East Zones, as well as the New "West Zone" Extension in Development



Note: Background geology from Ontario Geological Survey (MRD126), shown against the ultramafic unit, colored purple in the drill hole traces. Source: Caracle Creek, 2022.

Using available core logging information, as well as aluminum/magnesium ratios for rock type validation, lithologies from the Main and East zones were initially generalized and grouped into broader categories (Table 14.1), in order to facilitate correlation between major units. Subsequently, individual horizons from each category were selected 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 sub-vertical fashion. Horizons were numbered relative to their location, from south to north:

- Metavolcanics (MV), the southernmost unit.
- Pyroxenite (PYX-0), very rarely present (likely altered to talc), 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 northernmost modelled unit.

A number of dikes diabase and lamprophyre dikes were also identified through interval correlations and codified, almost all of them in the Main Zone, except for one in the East Zone.

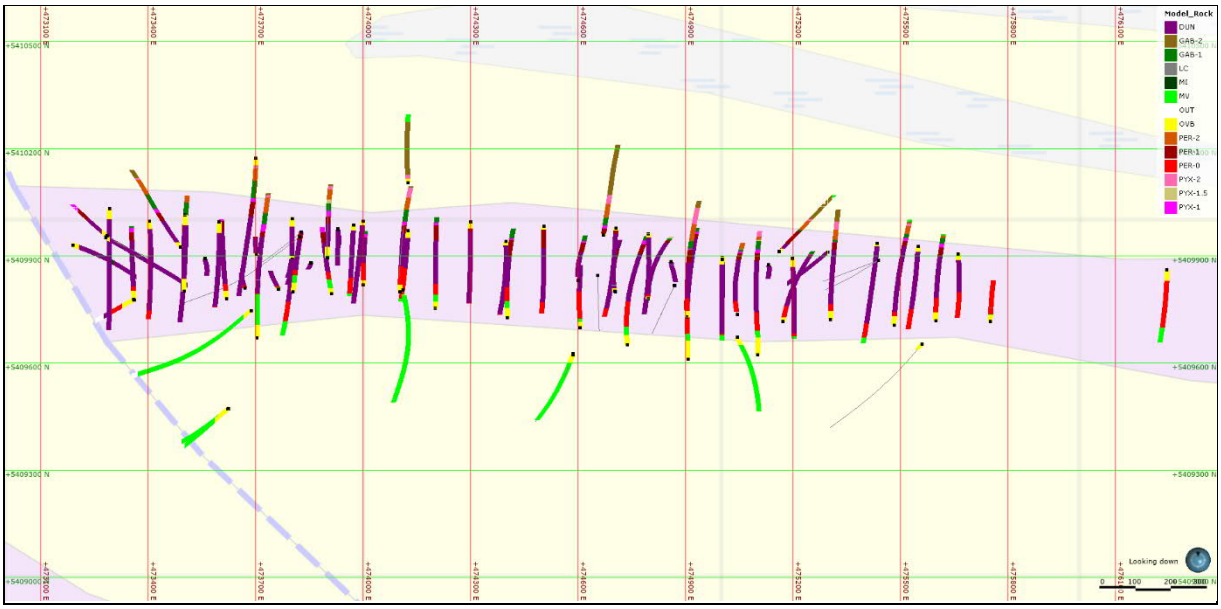
Complementary datasets (i.e., assays, density, hole mag-sus) facilitated verification when lithological boundaries were unclear. For example, nickel and iron grades often drop noticeably outside of the ultramafic unit, density differences between ultramafic and gabbroic rocks are apparent, while magnetic susceptibility is better suited 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 ultramafic unit, as well as confirmation of its overall shape and dimensions.

Table 14.1: Lithologies with Respective Original and Model Rock Codes

LITHOLOGY	MODEL CODE	
Overburden	OVB	By SRK
Lamprophyre Dyke	LAM	Only in MZ
Diabase	MI	
Mafic Intrusive		
Gabbro	GAB	
Dunite	DUN	UM Unit
Bleached Dunite		
Carbonatized Dunite		
Laminated Dunite		
Serpentinite		
Talcose Ultramafics		
Peridotite	PER	
Ultramafic Metavolcanics		
Poikilitic Ultramafic		
Pyroxenite	PYX	
Intermediate Metavolcanics	MV	
Mafic Metavolcanics		
Porphyry		
Felsic Metavolcanics		
Lost Core	LC	
Fault		
Anorthosite	Not modelled, few intervals merged within other units	
Felsic Intrusive		
Intermediate Intrusive		

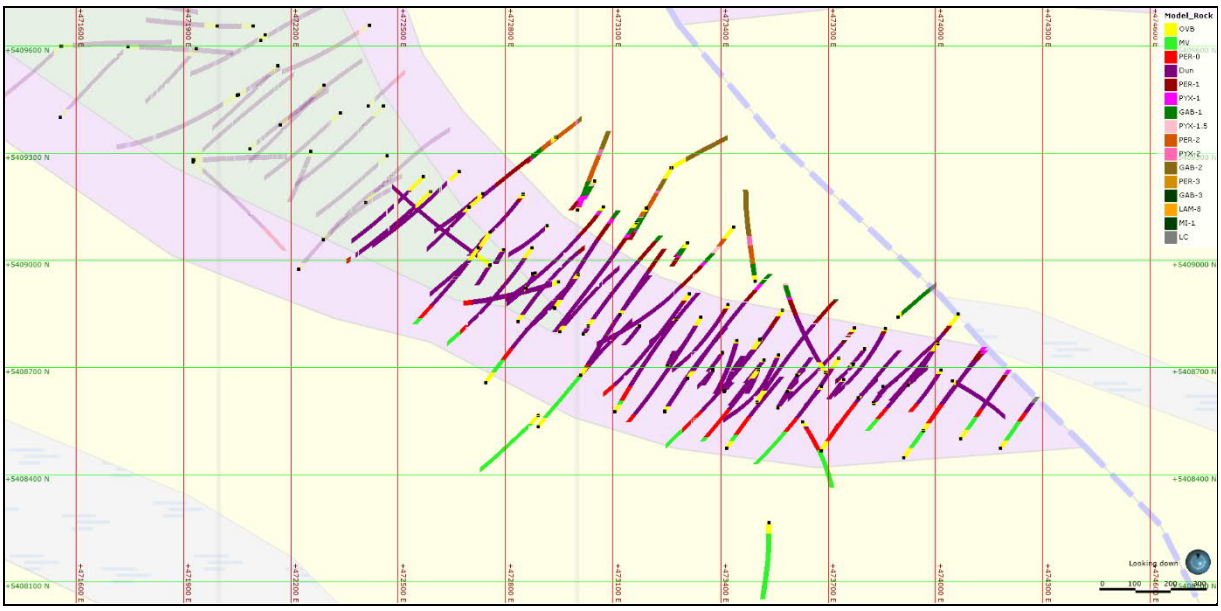
Source: Caracle Creek, 2022.

Figure 14.4: East Zone Plan View of Drill Hole Intercepts showing Grouped Lithologies



Note: Background geology from Ontario Geological Survey (MRD126). Source: Caracle Creek, 2022.

Figure 14.5: Main Zone Plan View of Drill Hole Intercepts showing Grouped Lithologies



Note: Background geology from Ontario Geological Survey (MRD126). West Zone drill holes made transparent to emphasize Main Zone data. Source: Caracle Creek, 2022.

The modelling area, covering both zones, is 7.5 km long (from 469500E to 477000E) by 5 km wide (from 5407000N to 5412000N). The modelling depth was increased from 650 m to approximately 800 m below overburden (at -600 m

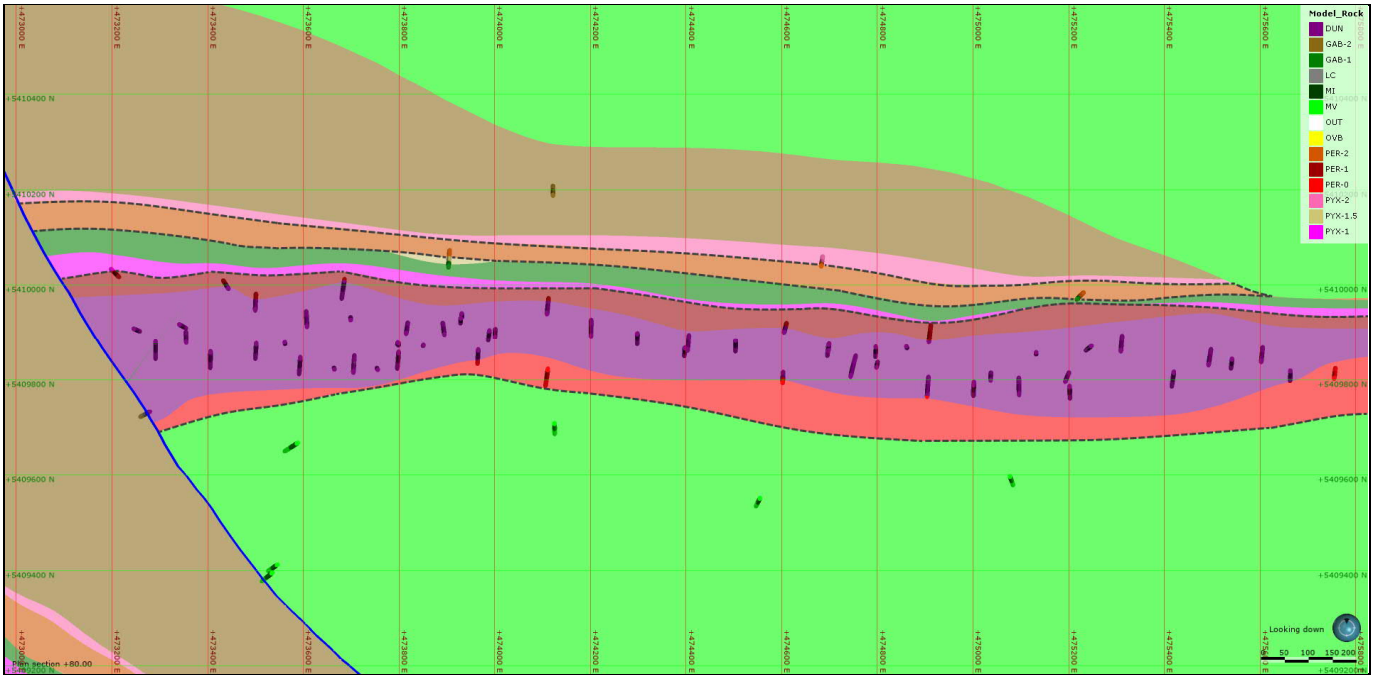
elevation). Despite this bottom limit, both zones can be considered open at depth, with at least two drill holes in the Main Zone extending past the limit with intercepts containing mostly medium to low nickel grades with a few >0.25% Ni intercepts. The East Zone is currently being tested below this limit too.

In order to model each zone separately, a fault surface was first interpolated following the interpreted trace found in the Ontario Geological Survey map, along with several fault intercepts found in the drill hole logs, and then used to generate eastern and western domains, where the East and Main zone would be modelled, respectively. Note that, even though the Main Zone only covers a limited area within its corresponding domain, it cannot be separated from its western extension (West Zone), which was also completely modelled (and estimated) along with it, even if not being considered in this MRE.

The newly codified database served as the base for both models, with second pass verifications done during estimation domain modelling, using complementary datasets. Given the relatively simple nature of the geological sequence, no sections were generated previous to 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, compensate the lack of information in some areas, especially at great depths.

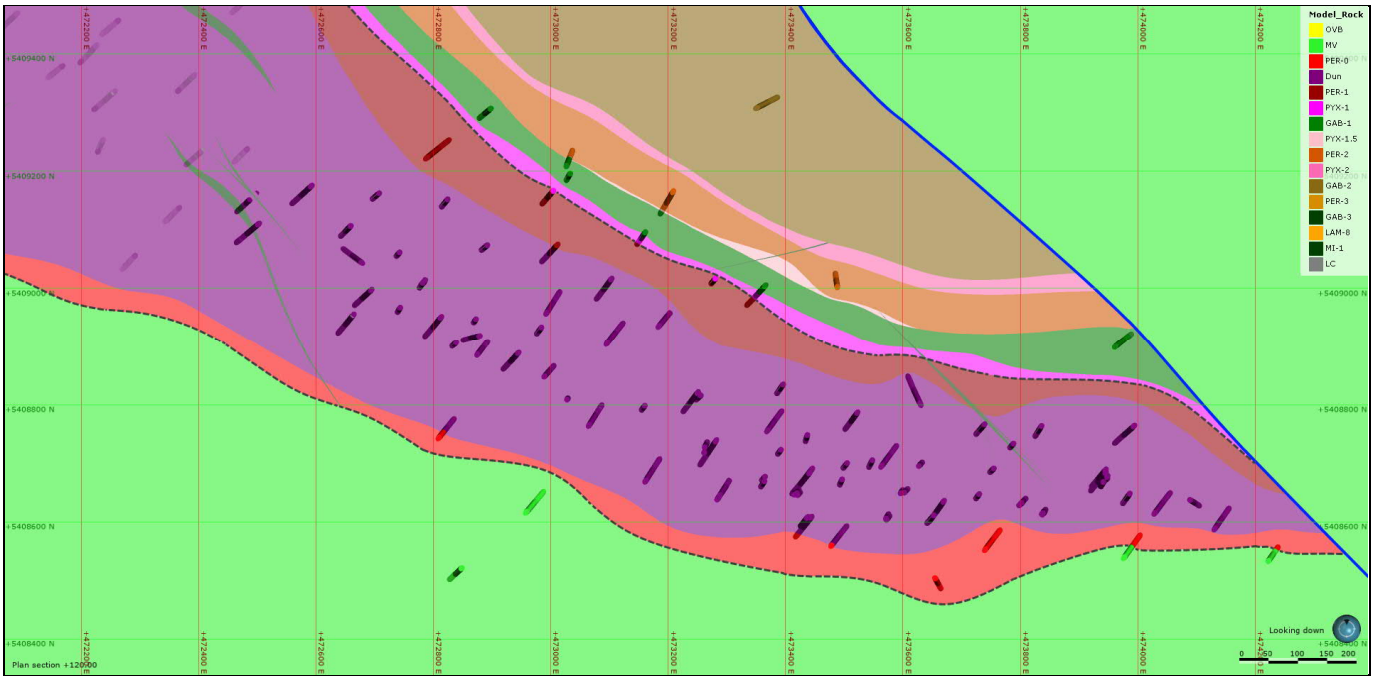
Dikes were also modelled individually, then merged and cut against the previously modelled lithologies, with diabase structures following a sub-vertical trend and lamprophyre structures with a more sub-horizontal orientation.

Figure 14.6: East Zone Plan Section (80 m) showing the Drill Hole Traces and the Final Lithology Wireframes



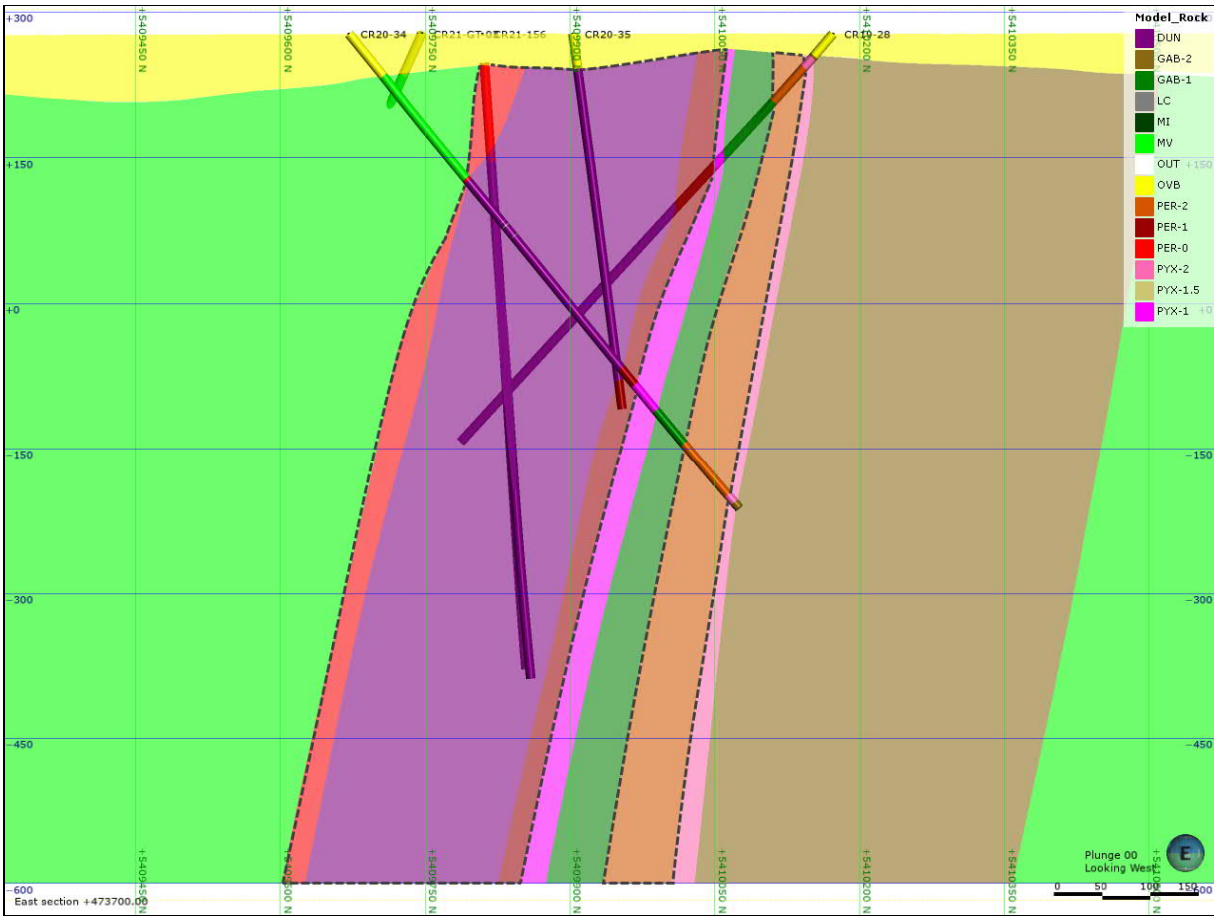
Note: Mineralization envelopes are marked within dashed lines. Source: Caracle Creek, 2022.

Figure 14.7: Main Zone Plan Section (120 m) showing Drill Hole Traces and Final Lithology Wireframes



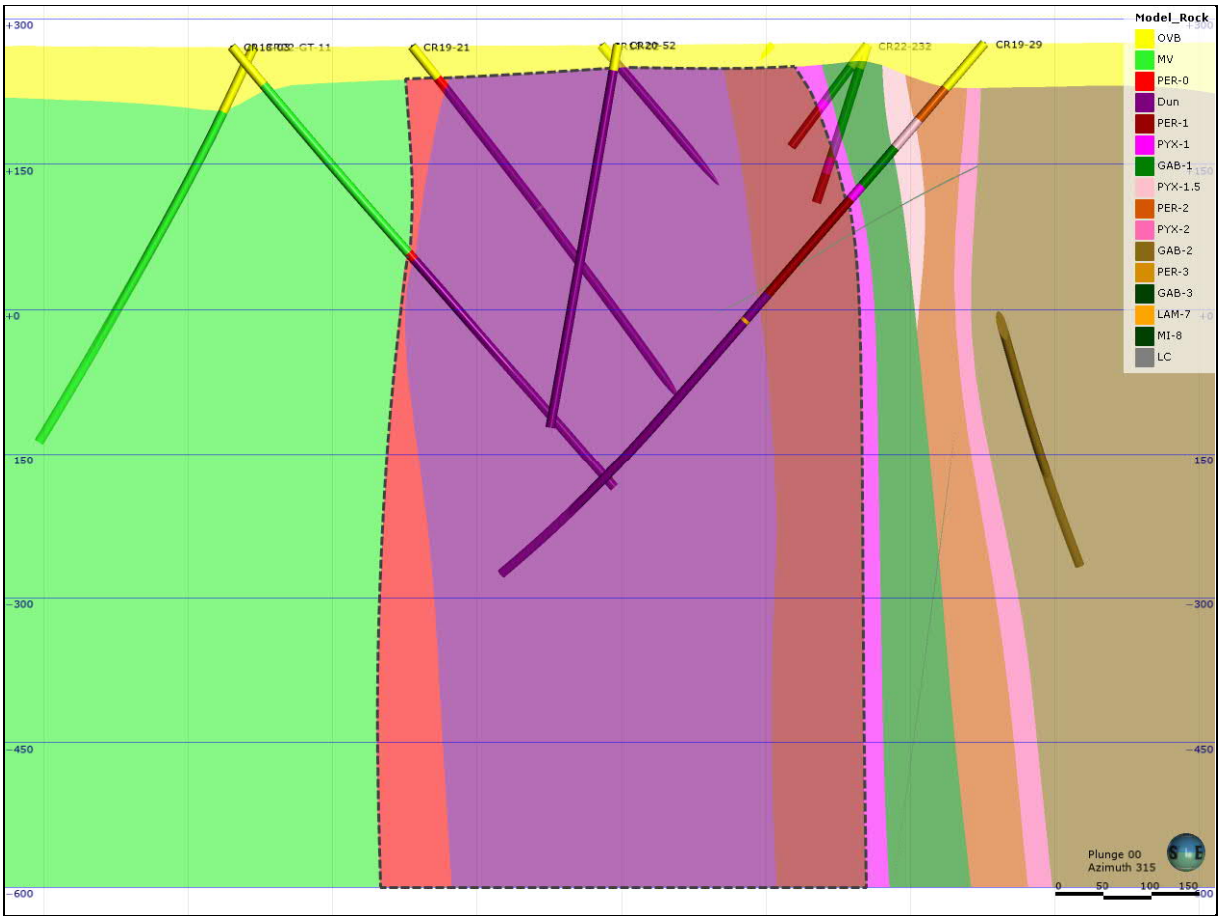
Note: West Zone drill holes made transparent to emphasize Main Zone data. Mineralization envelopes are marked within dashed lines. Source: Caracle Creek, 2022.

Figure 14.8: East Zone Section (473700N) looking West showing the Drill Hole Traces and the Final Lithology Wireframes



Note: Mineralization envelopes are marked within dashed lines. Source: Caracle Creek, 2022.

Figure 14.9: Main Zone Oblique Section looking West showing the Drill Hole Traces and the Final Lithology Wireframes



Note: Mineralization envelopes are marked within dashed lines. Source: Caracle Creek, 2022.

The geological models for the Main 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

Alteration does not seem to affect mineralization in a significant way, though it does pose an influence in how density (representative of serpentinization degree) and magnetic susceptibility (representative of magnetite content) 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 and most relevant alteration in the Main and East zones is serpentinization, given the majority of ultramafic type rocks, with talc-carbonation as a secondary occurrence. Other alteration types (silicification, sericification, albitization, among others) are only found in minor quantities and affecting very limited areas so as to become relevant for study.

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%

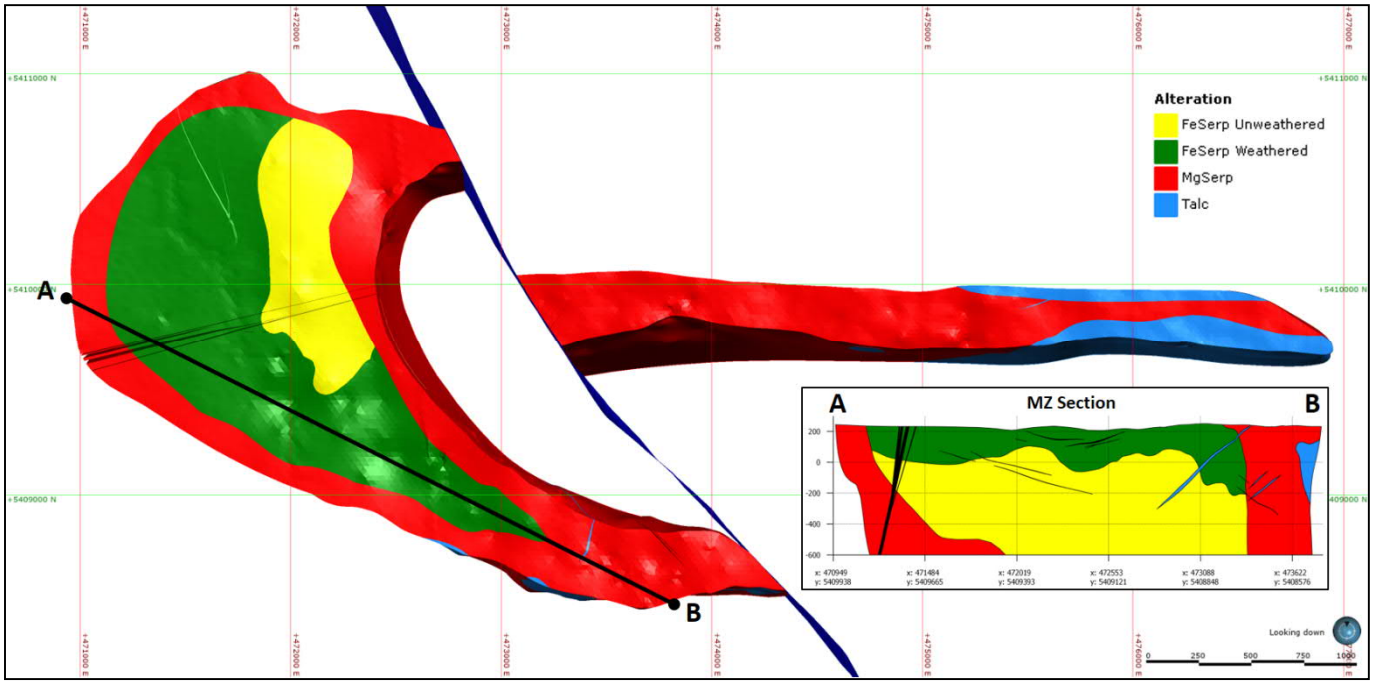
serpentinization), forming Mg-rich (Fe-poor) serpentine/brucite with secondary magnetite. The first phase is metastable, meaning it's susceptible to oxidation (ie. weathering), which is an ongoing process in the project. As such, weathering was included in the model as a third alteration type.

The alteration modelling area was restricted to the main mineralized ultramafic horizons (PER-0, DUN, PER-1). As with the lithology model, the Main Zone alteration cannot be studied without considering its western extension (West Zone).

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 Zone, while the East Zone only shows MgSerp.
- Weathered/Unweathered FeSerp: Weathering was identified in the Main 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 ultramafics" lithology logs, complemented with density values.

Figure 14.10: Plan View of the Alteration Model for Main and East zones



Note: Inset shows a vertical section cutting through the Main and West Zone (see A-B trace in map). Source: Caracle Creek, 2022.

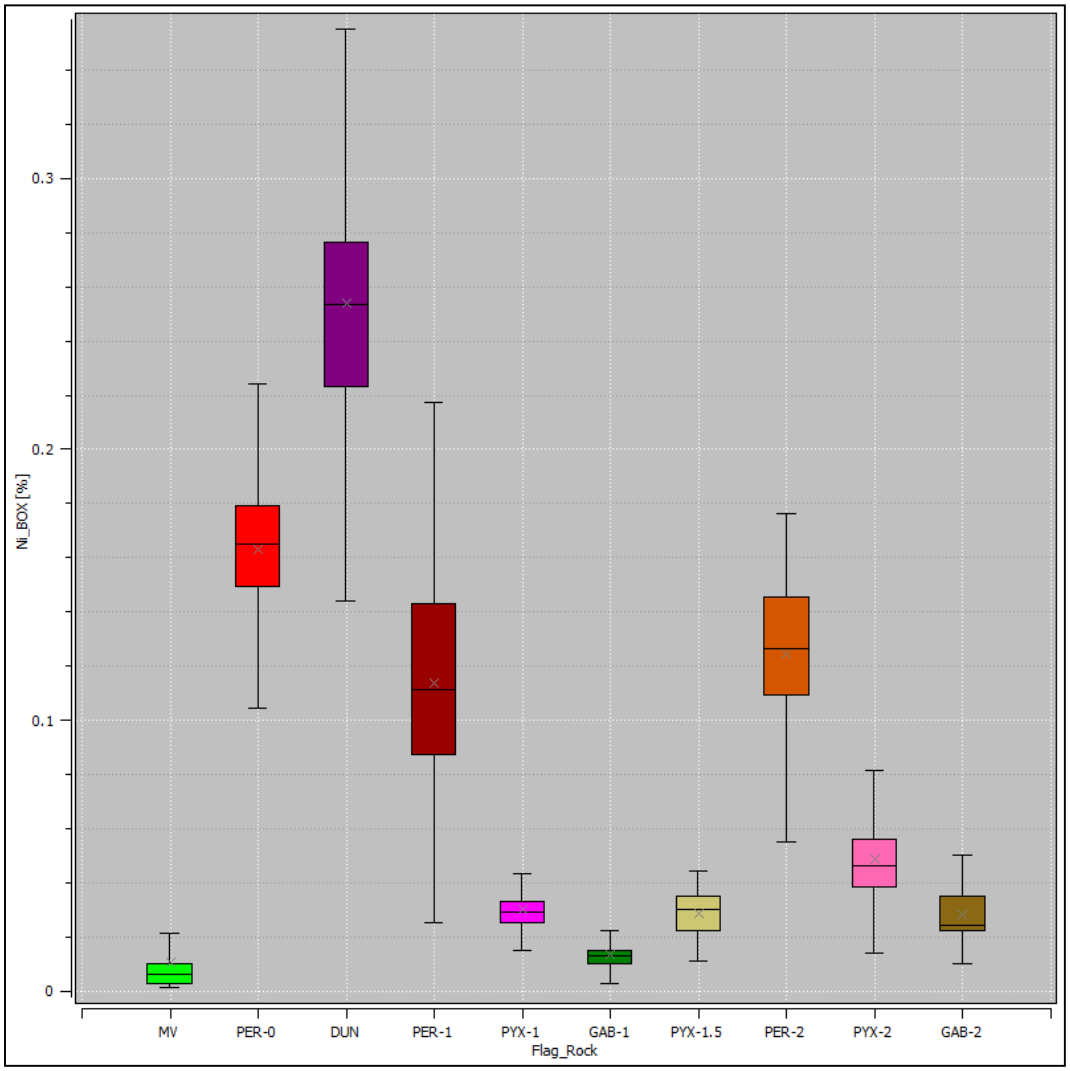
The alteration models for the Main and East Zones developed by Caracle Creek constitute the basis for density and magnetic susceptibility estimation.

14.5 Data Analysis and Estimation Domains

14.5.1 East Zone: Exploratory Data Analysis (EDA)

The East Zone assay database comprises 12,101 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 (10,921 samples), while the last horizon (PER-2) is a secondary target (EST-2) isolated north of the main target and poorly tested (317 samples). Further analyses in this section will be focused on the EST-1 domain.

Figure 14.11: East Zone Box-Plot Showing the Distribution of Nickel Grades according to Lithological Units

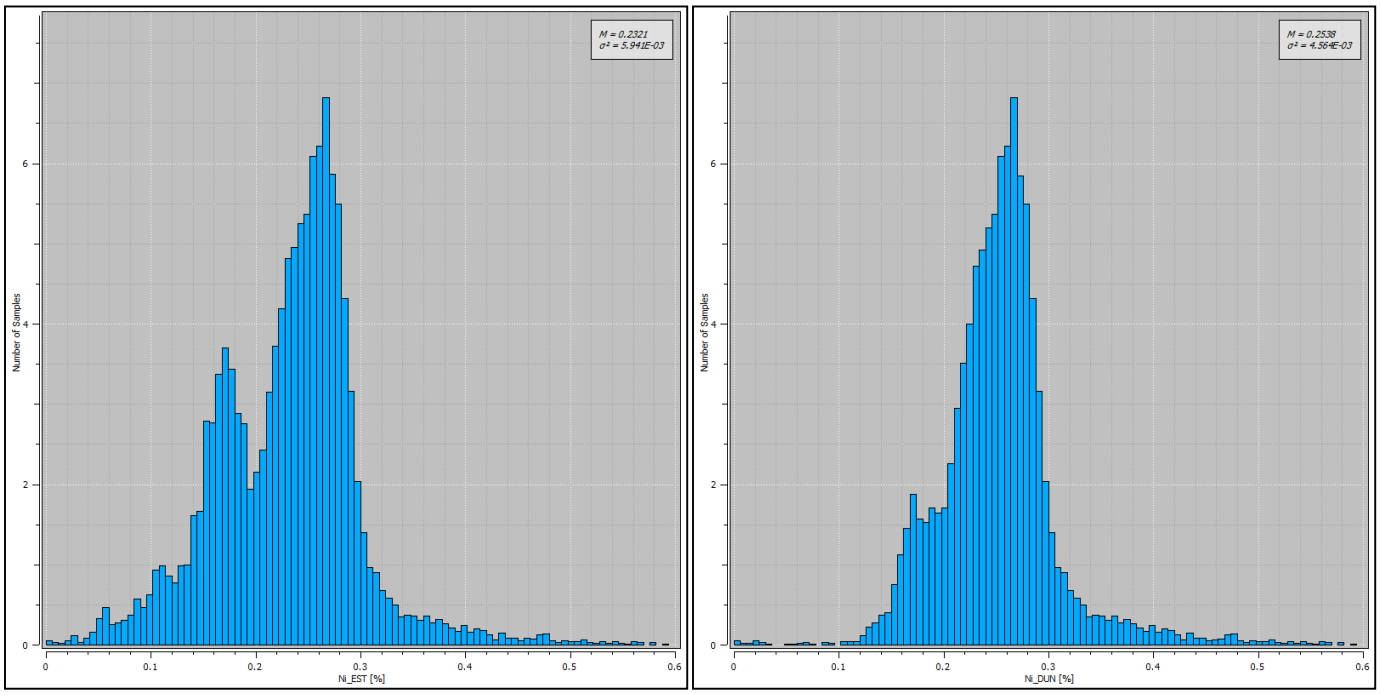


Source: Caracle Creek, 2022.

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 which encompasses most of the moderate to high grades, grading to two lateral ones, north and south, which share the lower grade population, 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)

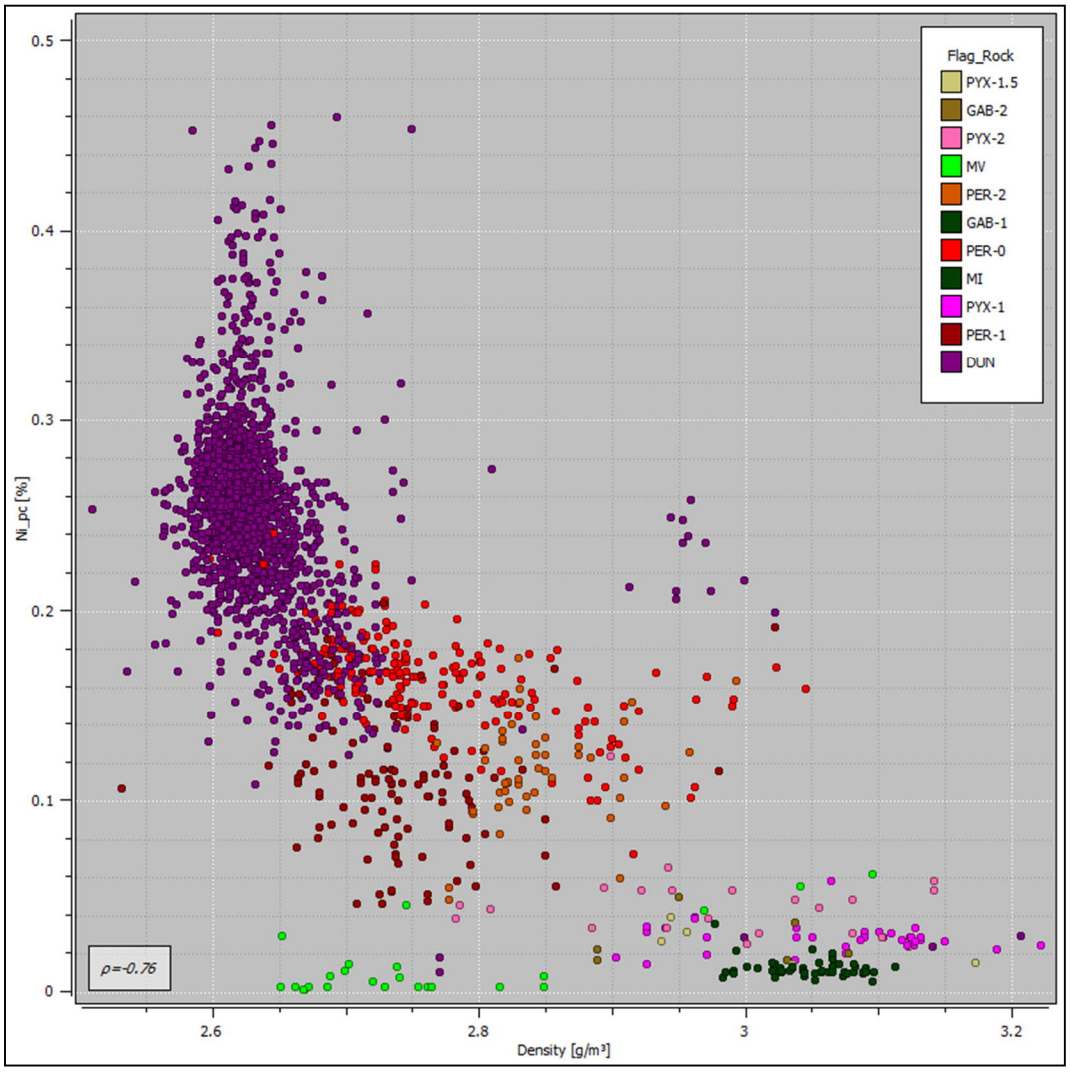


Source: Caracle Creek, 2022.

It's 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.

In order to completely remove this effect and effectively separate both populations, a combination of aluminum and magnesium cut-offs (~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.

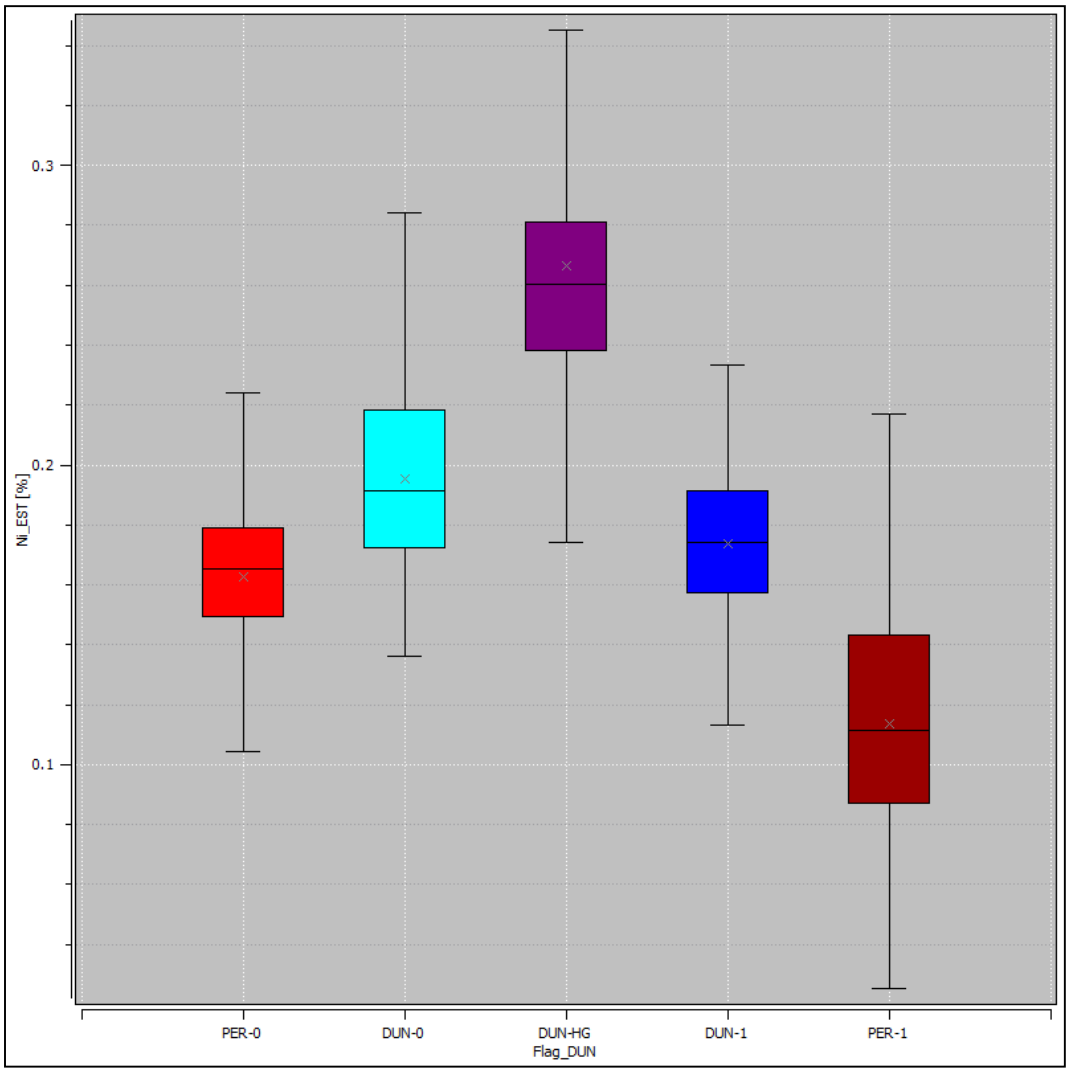
Figure 14.13: East Zone Cross-Plot Showing the Correlation of Density Values to Nickel Grades according to Lithological Units



Source: Caracle Creek, 2022.

The end result, after careful 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 serpentized and higher-grade central dunite, and a secondary "transitional", moderately serpentinized and lower-grade dunite, located at both the northern and southern dunite/peridotite contacts. Statistical analysis of these subdivisions confirms that the transitional dunite units (DUN-0, DUN-1) are more similar to their peridotite counterparts (PER-0, PER-1) than to the central dunite unit (Figure 14.14).

Figure 14.14: East Zone Box-Plot Showing the Distribution of Nickel Grades according to the Dunite Unit Subdivision and adjacent Peridotite Units



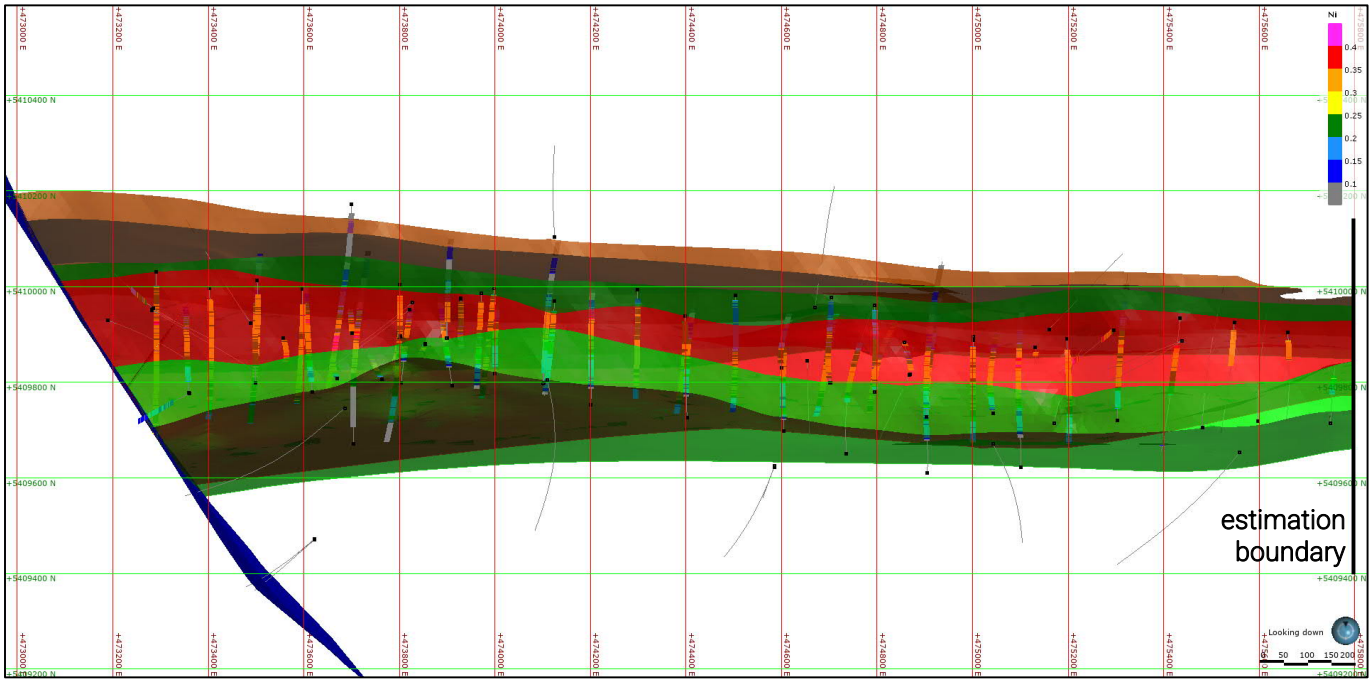
Source: Caracle Creek, 2022.

14.5.2 East Zone: Estimation Domains (Grade Shells)

The two nickel grade populations identified in the EDA are thus proven to correspond to a central higher-grade dunite unit and a combination of transitional dunite and peridotite units, both to the north and south. These domains were generated within the estimation boundary 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 with 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 until 50 m beyond the second to last easternmost drill hole (estimation boundary), given that the last is an exploratory hole located almost 500 m away from the previous hole.

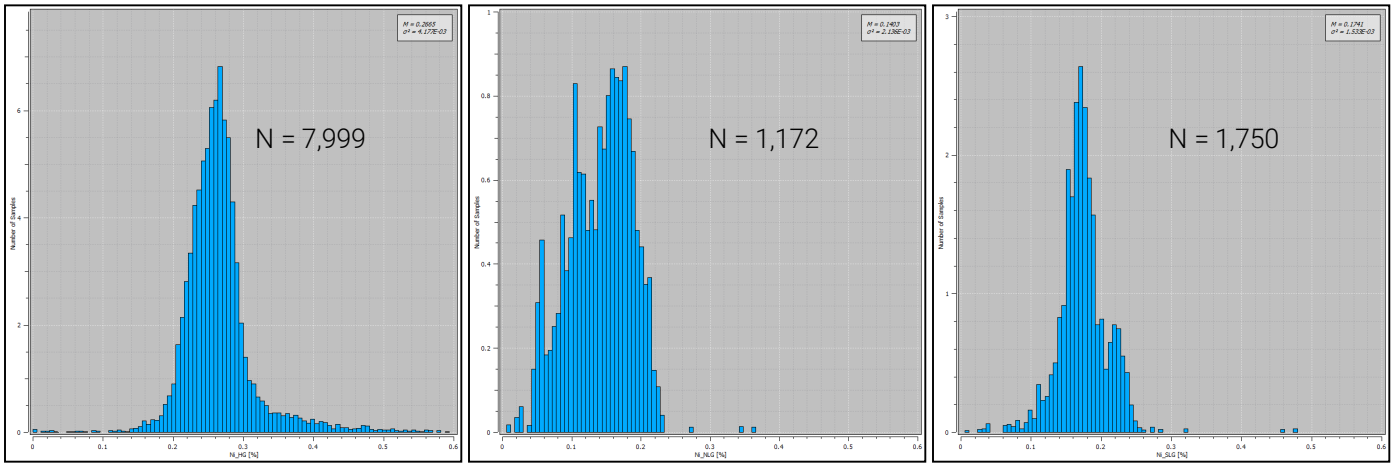
Figure 14.15: East Zone Plan View with Nickel Grade Drill Hole Traces and Estimation Domains



Note: Shows HG (red), SLG (light green), NLG (dark green) and EST-2 (brown) estimation domains. Source: Caracle Creek, 2022.

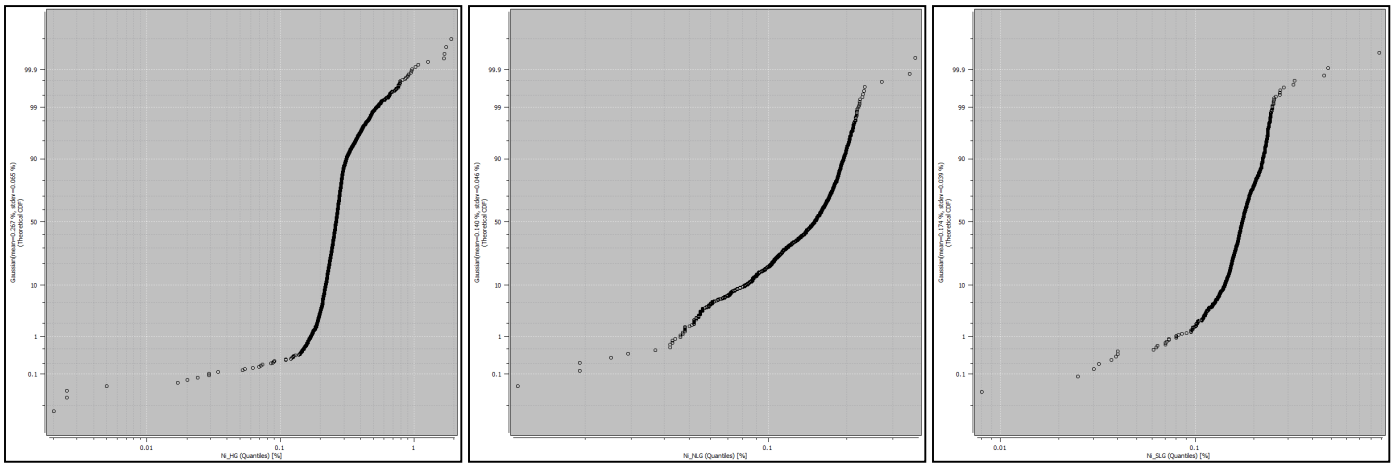
Statistical analysis within the HG/LG domains shows an appropriate separation of the two nickel grade populations, evidenced by adequate distributions and statistical parameters for each estimation domain, though with lower sample numbers in the case of the lower-grade domains (Figures 14.16 and 14.17). Further analyses for the other main elements and density suggest no need for additional domaining, meaning that, either individually or together as the EST-1 domain, these domains will be the base for all the East Zone estimation work.

Figure 14.16: East Zone Nickel Grade Histograms with Sample Amounts for the HG, NLG and SLG Estimation Domains



Source: Caracle Creek, 2022.

Figure 14.17: East Zone Nickel Grade Probability Plots for the HG, NLG and SLG Estimation Domains



Source: Caracle Creek, 2022.

14.5.3 East Zone: Compositing and Capping

Considering that over 99% of the drill hole samples are 1.5 m in length, the 15.0 m height of the blocks (see Section 14.8, Block Modelling) and the sample database size, composites of 7.5 m were selected for the estimation domains. Composites were generated within the general EST-1 domain (meaning that HG/LG boundaries were not taken into account) and EST-2 domain for the 7 studied elements (Ni, Co, Fe, Cr, S, Pd, Pt).

The density database consists of 2,827 data points for the EST-1 domain and 46 data points for the EST-2 domain, a majority of them taken every 4 to 10 m, with an average of 8 m. Given that density measurements are essentially punctual, as opposed to continuous interval samples, they cannot not be composited.

Capping was applied before compositing and only for "true" outliers (values out of context, eg. 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, 1.4% for chromium, 1.0% for sulphur, 0.8 ppm for palladium and none for iron, platinum or density. The latter, however, did receive a bottom cap of 2.4 g/cm^3. Anomalous zones that transition into very high values weren't 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 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 (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 Drill Hole Samples					7.5 m Composites (except Density)				
		Count	Mean	Std Dev	CV	Med	Count	Mean	Std Dev	CV	Med
HG	Ni %	7,999	0.27	0.06	0.24	0.26	1,593	0.27	0.05	0.19	0.26
	Fe %	7,999	5.90	0.88	0.15	5.77	1,593	5.90	0.67	0.11	5.80
	Cr %	7,999	0.65	0.12	0.19	0.67	1,593	0.65	0.10	0.16	0.67
	Density	2,053	2.62	0.05	0.02	2.62	2,044	2.62	0.04	0.01	2.62
NLG	Ni %	1,172	0.14	0.05	0.34	0.15	235	0.14	0.05	0.34	0.14
	Fe %	1,172	7.27	0.85	0.12	7.04	235	7.29	0.64	0.09	7.40
	Cr %	1,172	0.55	0.19	0.35	0.51	235	0.54	0.14	0.25	0.53
	Density	318	2.71	0.08	0.03	2.70	306	2.71	0.06	0.02	2.70
SLG	Ni %	1,750	0.17	0.04	0.22	0.17	347	0.17	0.03	0.17	0.17
	Cr %	1,750	7.45	0.71	0.09	7.50	347	7.46	0.53	0.07	7.54
	Fe %	1,750	0.52	0.11	0.21	0.51	347	0.52	0.09	0.17	0.51
	Density	456	2.77	0.12	0.04	2.73	390	2.74	0.09	0.03	2.71
EST-1	Co %	10,921	0.013	0.007	0.564	0.012	2,175	0.013	0.002	0.128	0.012
	S %	10,921	0.06	0.11	1.79	0.03	2,175	0.06	0.10	1.70	0.03
	Pd ppm	10,921	0.020	0.080	5.224	0	2,162	0.012	0.033	2.750	0.003
	Pt ppm	10,921	0.010	0.050	4.870	0	2,162	0.008	0.016	1.931	0.005
EST-2	Ni %	260	0.12	0.02	0.19	0.13	51	0.12	0.02	0.17	0.12
	Co %	260	0.013	0.002	0.157	0.012	51	0.014	0.002	0.114	0.013
	Fe %	260	9.75	1.37	0.14	7.20	51	9.92	1.15	0.12	10.15
	Cr %	260	0.38	0.12	0.31	0.38	51	0.35	0.08	0.23	0.34
	S %	260	0.05	0.03	0.57	0.05	51	0.06	0.02	0.42	0.05
	Pd ppm	260	0.010	0.050	3.350	0.010	51	0.016	0.030	1.943	0.011
	Pt ppm	260	0.010	0.050	6.172	0	51	0.010	0.033	3.394	0.004
	Density	46	2.86	0.07	0.02	2.83					

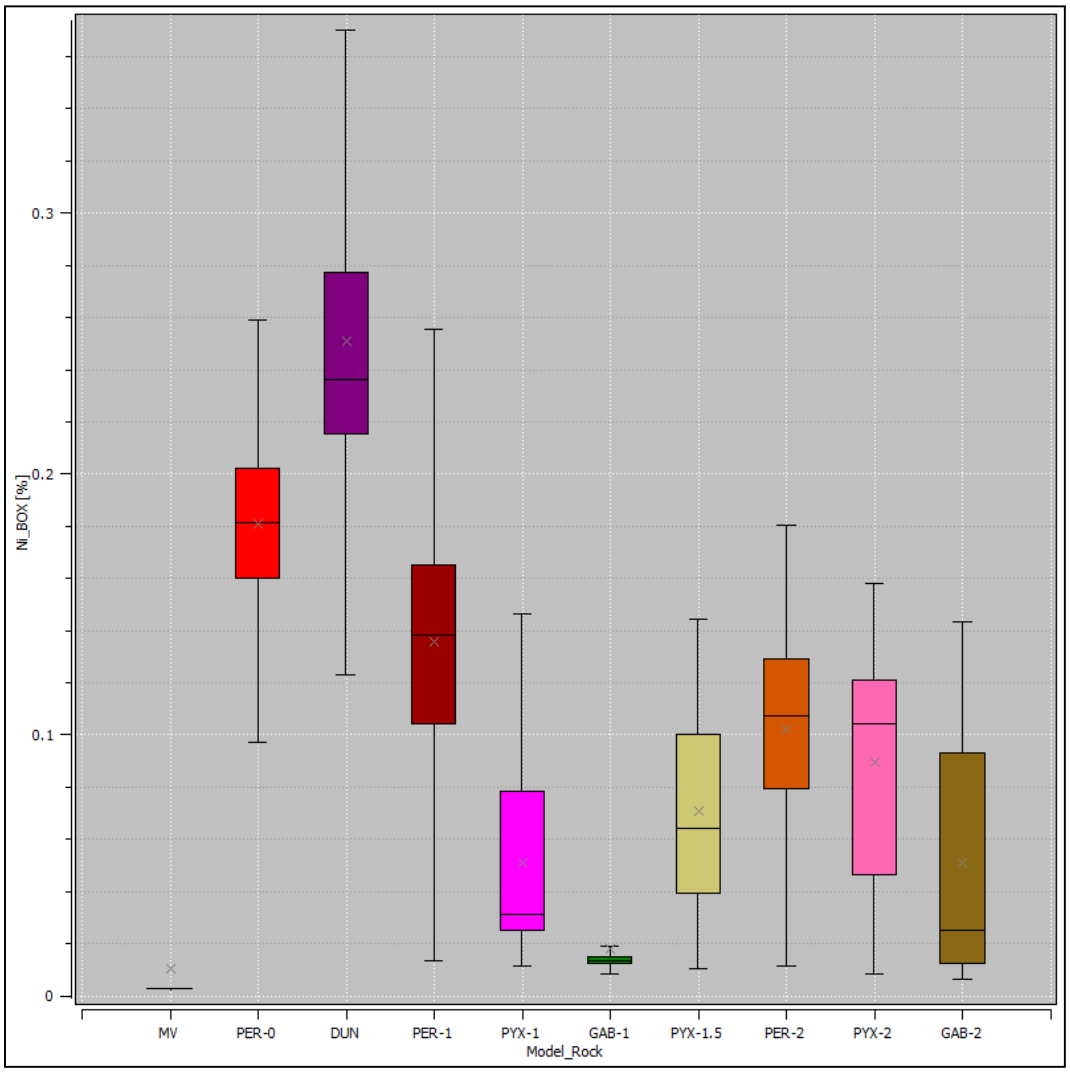
Note: Composites for PGE elements (Pd, Pt) have a lower count due to the exclusion of very high-grades located at the northern contact with pyroxenite, corresponding to a high-grade structure or "PGE reef" that will be evaluated in a future estimate. Density sample count is slightly lower at the composite stage given that some found no pair to a nickel composite for co-kriging (see Section 14.8.1). Source: Caracle Creek, 2022.

14.5.4 Main Zone: Exploratory Data Analysis (EDA)

The Main Zone assay database comprises 18,957 results from drill core samples. However, as mentioned in previous models, the evident mineralization continuity between the Main and West zones makes it impractical to study one without the other, which means that the working database comprises 34,276 samples. Having cleared this, further sample figures and calculations will only consider the Main Zone data.

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 (17,903 samples). There seems to be some potential in horizon PER-2 as a secondary target, like the East Zone, though in this case it is not as well sampled (244 samples) or spatially constrained.

Figure 14.18: Main Zone Box-Plot Showing the Distribution of Nickel Grades according to Lithological Units

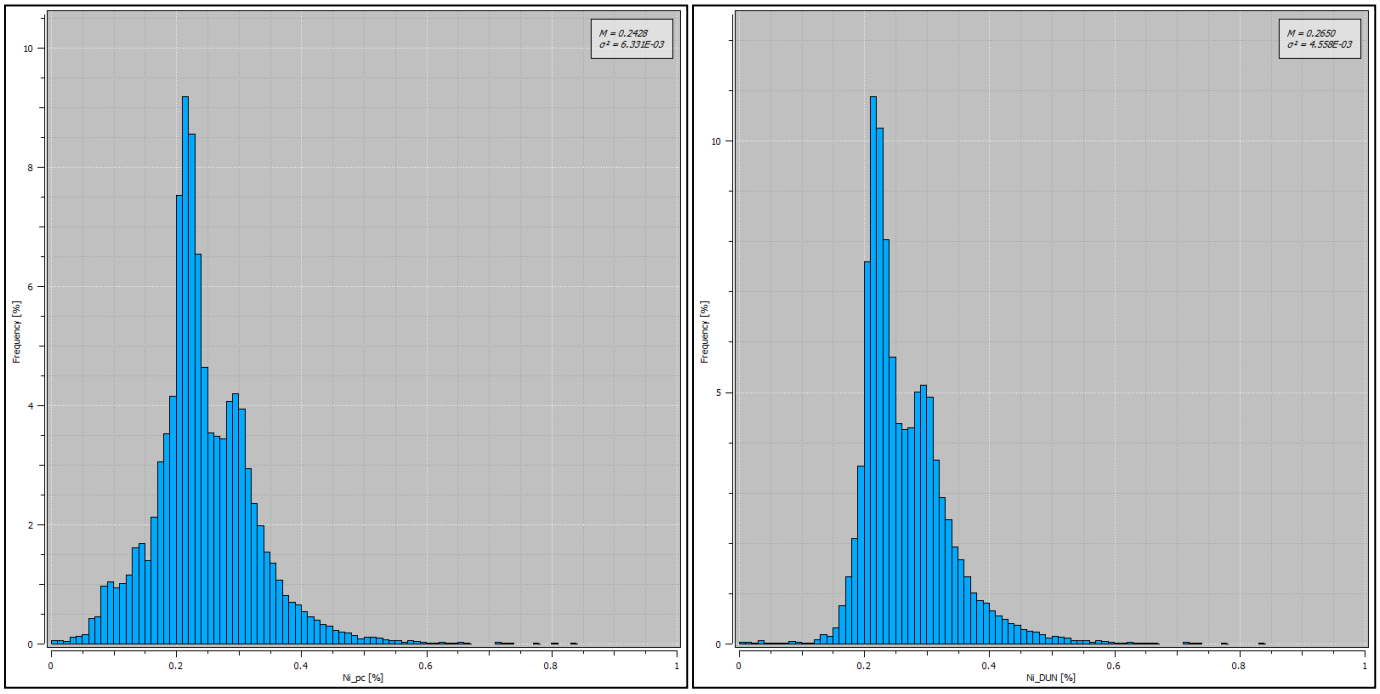


Source: Caracle Creek, 2022.

A histogram of nickel grades within the 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 which encompasses most of the high grades, grading to two lateral ones, north and south, which share the moderate to low grade population. Moreover, it is possible to establish 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).

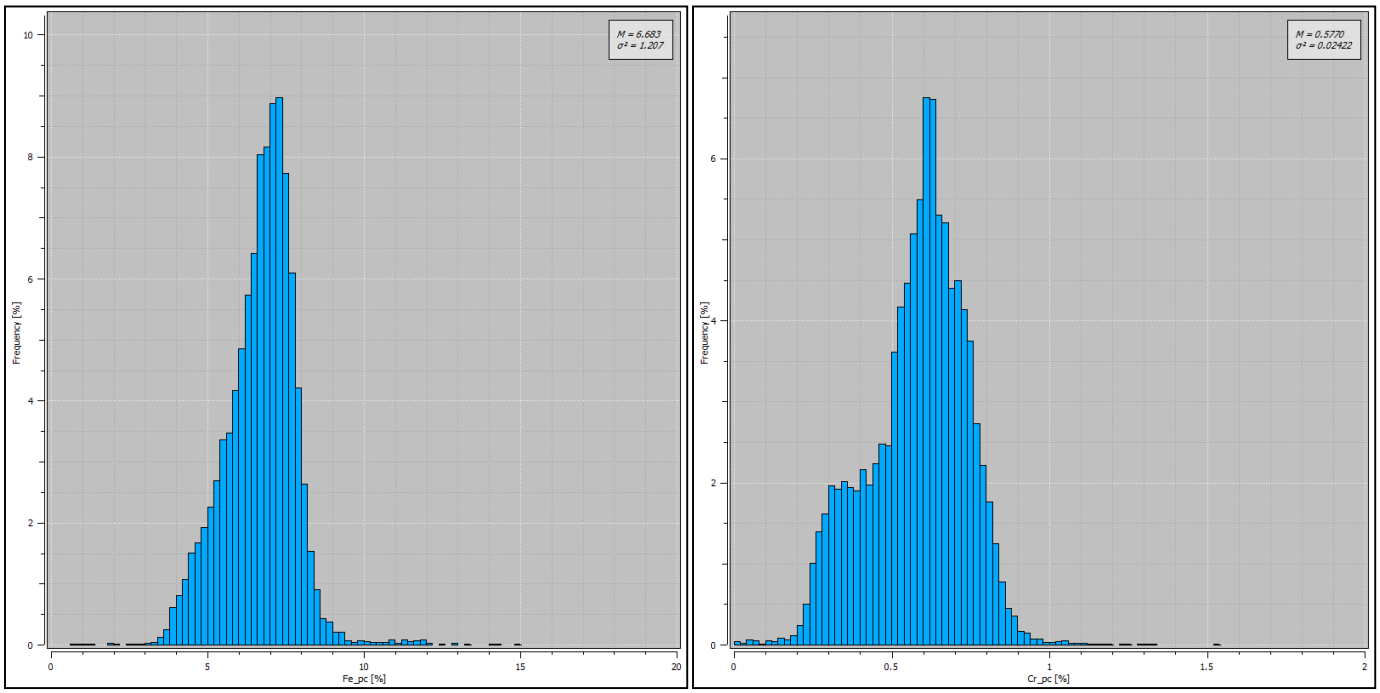
Figure 14.19: Main Zone Histograms Showing the Bimodal Distribution of Nickel Grades within the EST Domain (left) and its Subordinate DUN Unit (right)



Source: Caracle Creek, 2022.

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 3.0-6.0% Fe and 0.2-0.5% Cr within a central horizon, grading to two northern and southern horizons with higher-grade populations of 6.0-10.0% Fe and 0.5-1.0% Cr (Figure 14.20). In the case of chromium, and like nickel, it is possible to establish a separate northernmost lower-grade (<0.5% Cr) population running parallel to the previous ones.

Figure 14.20: Main Zone Histograms Showing the Bimodal Distribution of Iron Grades (left) and Chromium Grades (right) within the EST Domain



Source: Caracle Creek, 2022.

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 center-west of the Main Zone (see Section 14.4.3 and Figure 14.10) generates a major modification in the density arrangement (losing its correlation with nickel grades) which prevents from sorting transitional dunite areas between high and low nickel grades, thus negating the method applied in the East Zone to separate dunite types and, consequently, to generate estimation domains.

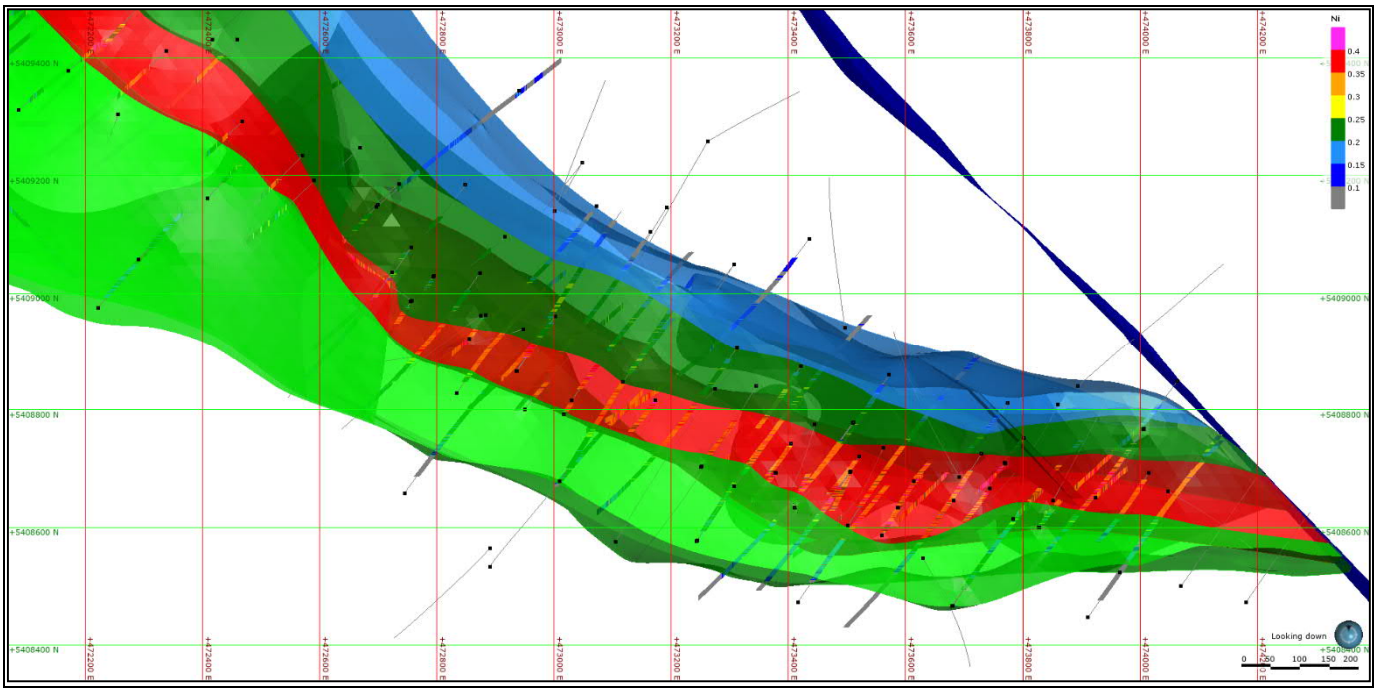
Additionally, the variety and lack of consistency between grade distributions of different elements means that each one requires separate treatment, which calls for the use of grade cutoffs as the most practical way of separating bimodal populations. This is also an appropriate method given that grade populations manifest as reasonably noticeable and correlatable horizons following the lithology trend, allowing to individualize them as one would do with mineral veins.

14.5.5 Main Zone: Estimation Domains (Grade Shells)

Having settled for grade cutoffs to separate bimodal populations means that, in practice, estimation domains had to be based on grade shells. These domains 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 high-grade (SHFE, NHFE) domains for iron (Figure 14.22) and a lower-grade (LCR) domain with two adjacent high-grade (SHCR, NHCR) domains and a northernmost low grade (VLCR) domain for chromium (Figure 14.23).

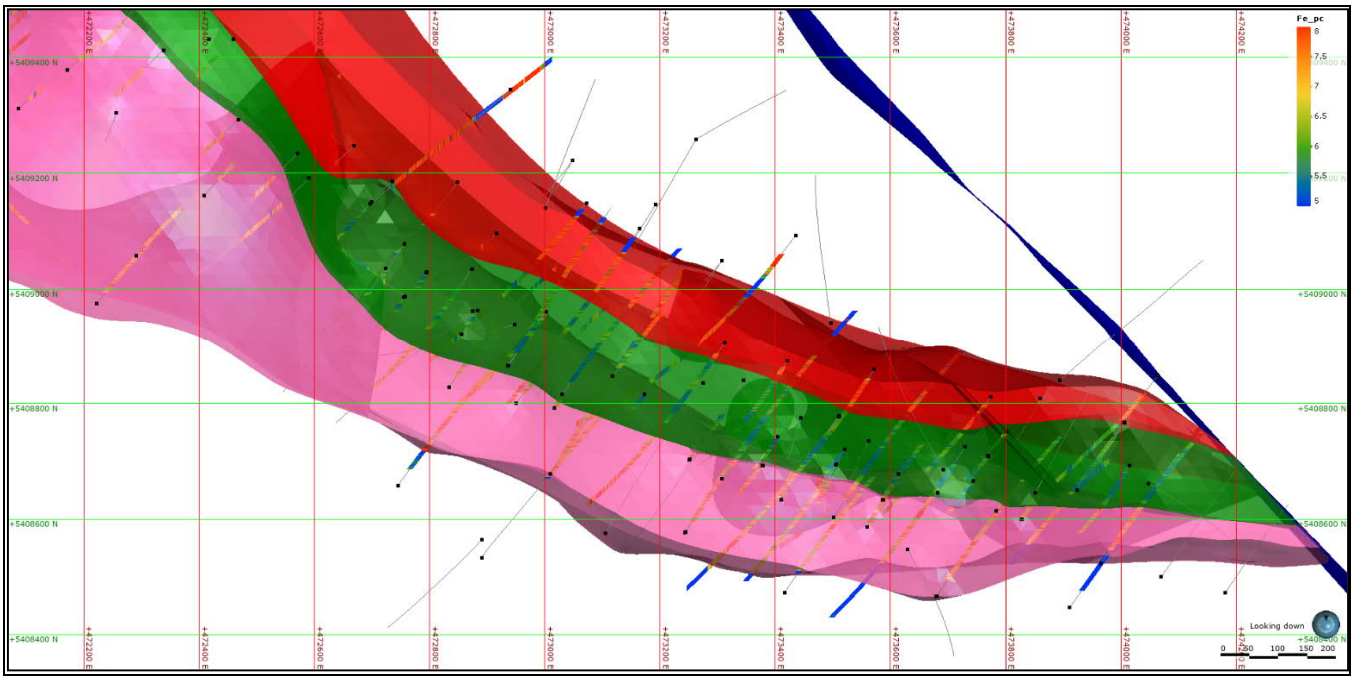
The estimation domains were limited to the lithologies of interest (PER-0, DUN, PER-1) within the Main Zone. The extended estimation domains (including the West Zone) were limited up 50m beyond the second to last northernmost hole drilled within the DUN unit, given that the last is an exploratory hole located over 350 m away from the previous hole.

Figure 14.21: Main Zone Plan View with Nickel Grade Drill Hole Traces and Estimation Domains



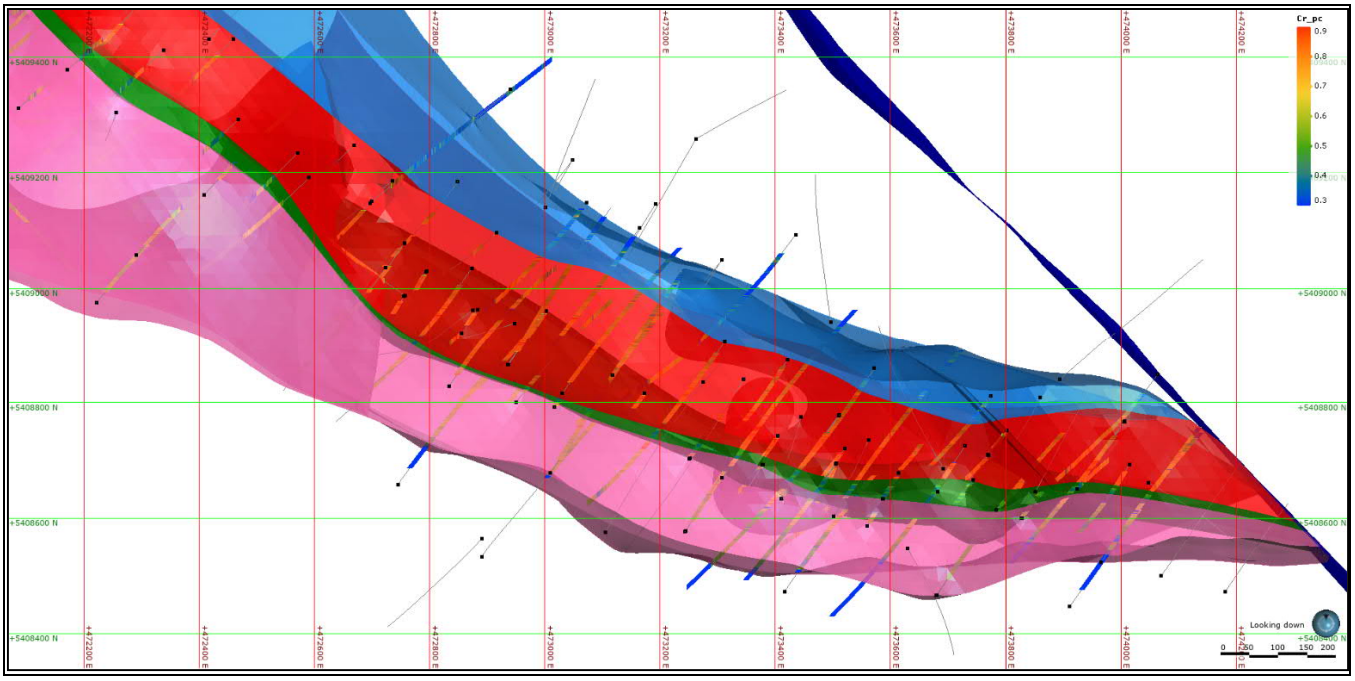
Note: Shows HG (red), SLG (light green), NLG (dark green) and VLG (light blue) estimation domains. Source: Caracle Creek, 2022.

Figure 14.22: Main Zone Plan View with Iron Grade Drill Hole Traces and Estimation Domains



Note: Shows LFE (dark green), SHFE (pink) and NHFE (red) estimation domains. Source: Caracle Creek, 2022.

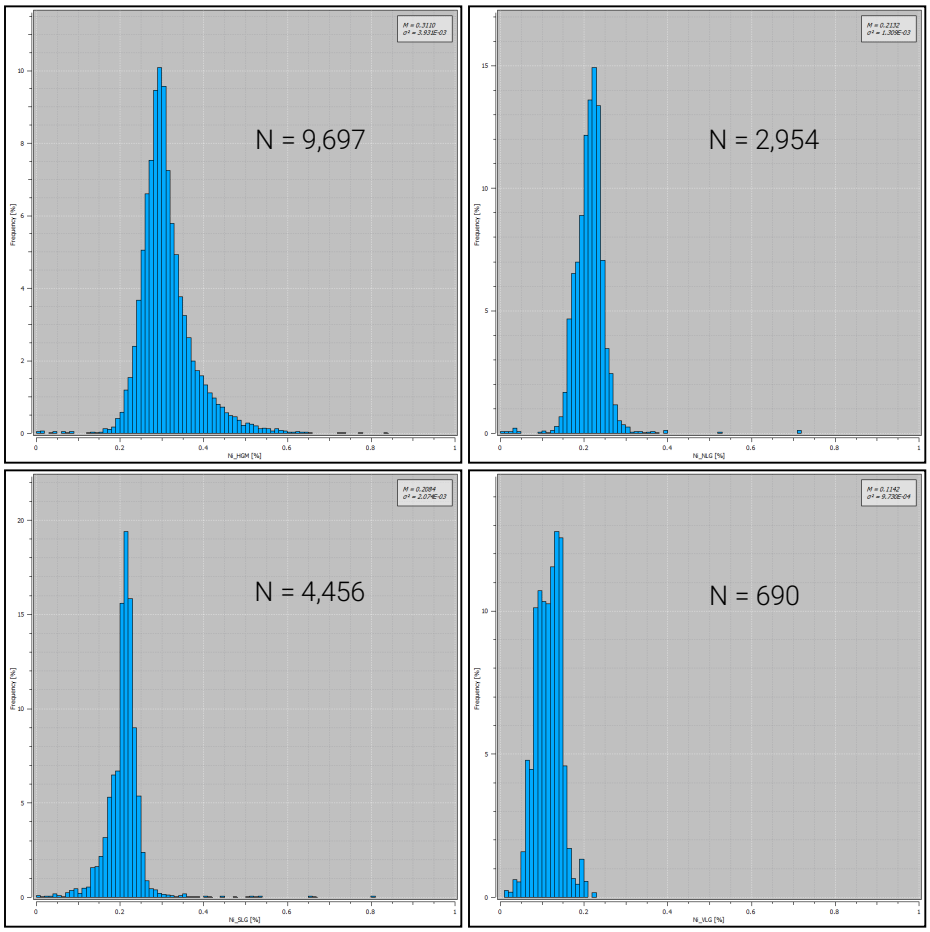
Figure 14.23: Main Zone Plan View with Chromium Grade Drill Hole Traces and Estimation Domains



Note: Shows LCR (dark green), SHCR (pink), NHCR (red) and VLCR (light blue) estimation domains. Source: Caracle Creek, 2022.

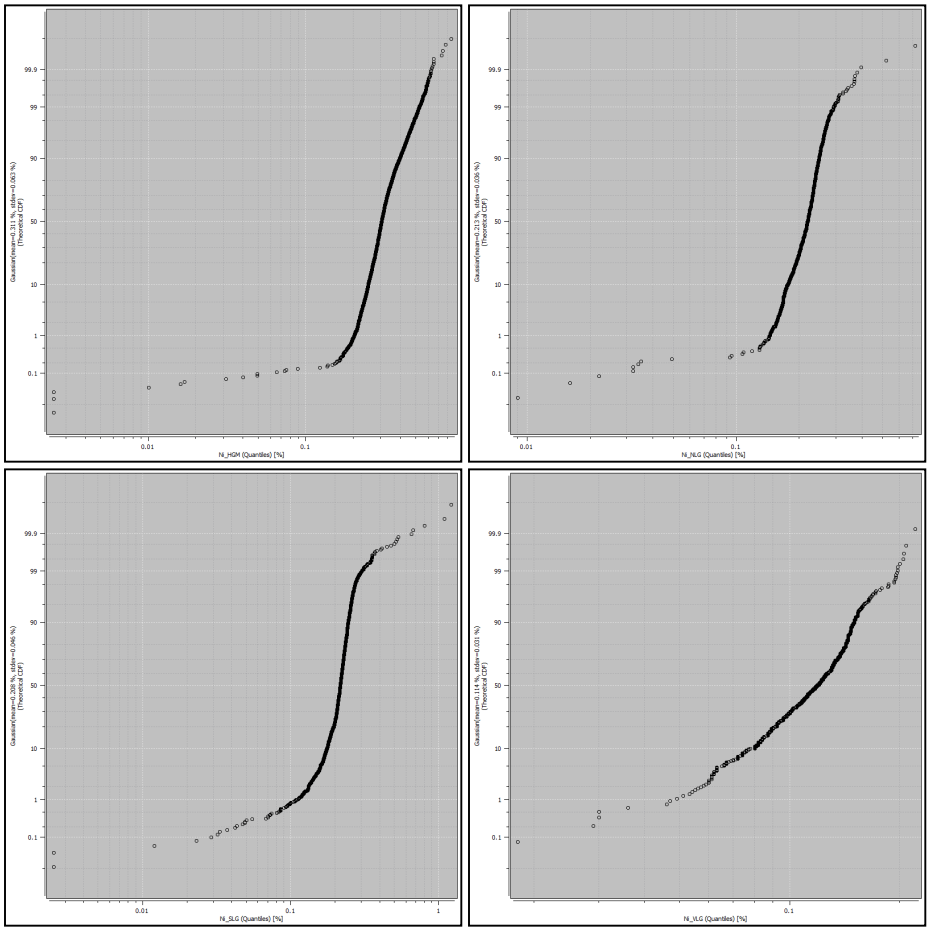
Statistical analysis within the set of estimation domains for each element shows an appropriate separation of the grade populations, evidenced by adequate distributions and statistical parameters (Figures 14.24 and 14.25). Further analyses for the other main elements and density suggest no need for additional domaining, meaning that, either individually or together as the EST domain, these domains will be the base for all the Main Zone estimation work.

Figure 14.24: Main Zone Nickel Grade Histograms with Sample Amounts for the HG, NLG, SLG and VLG Estimation Domains



Note: 106 samples were excluded as they correspond to non-mineralized dikes. Source: Caracle Creek, 2022.

Figure 14.25: Main Zone Nickel Grade Probability Plots for the HG, NLG and SLG Estimation Domains



Source: Caracle Creek, 2022.

14.5.6 Main Zone: Compositing and Capping

Considering that over 99% of the drill hole samples are 1.5 m in length, the 15.0 m height of the blocks (see Section 14.8, Block Modelling) and the sample database size, composites of 7.5 m were selected for the estimation domains. Composites were generated within the general EST domain (meaning that estimation domain boundaries were not taken into account) for the 7 studied elements (Ni, Co, Fe, Cr, S, Pd, Pt).

The density database consists of 5,543 data points for the EST domain, a majority of them taken every 3 to 12 m, with an average of 6 m. Given that density measurements are essentially punctual, as opposed to continuous interval samples, they cannot not be composited.

Capping was applied before compositing and only for "true" outliers (values out of context, eg. 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 paladium, platinum or density. The latter, however, did receive a bottom cap of 2.3 g/cm^3. Anomalous zones that transition into very

high values weren't 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 ordinary kriging (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 Zone Capping Values and Summary Statistics of Declustered Samples and Composites by Element and their Selected Estimation Domain

Domain	Element	1.5 m Drill Hole Samples						7.5 m Composites (except Density)				
		Count	Mean	Std Dev	CV	Med		Count	Mean	Std Dev	CV	Med
HG	Ni %	9,803	0.31	0.06	0.20	0.30		1,965	0.31	0.05	0.16	0.30
	S %	9,803	0.17	0.20	1.22	0.10		1,965	0.17	0.20	1.18	0.10
SLG	Ni %	4,456	0.21	0.05	0.22	0.21		891	0.21	0.03	0.15	0.21
	S %	4,456	0.04	0.05	1.26	0.03		891	0.04	0.04	1.07	0.03
NLG	Ni %	2,954	0.21	0.04	0.17	0.22		591	0.21	0.03	0.13	0.22
	S %	2,954	0.04	0.04	1.17	0.02		591	0.04	0.04	1.12	0.02
VLG	Ni %	690	0.11	0.03	0.27	0.12		138	0.12	0.03	0.25	0.12
	S %	690	0.03	0.03	0.80	0.03		138	0.03	0.02	0.64	0.03
LFE	Fe %	6,811	5.63	0.86	0.15	5.68		1,363	5.62	0.73	0.13	5.70
SHFE		9,591	7.22	0.67	0.09	7.21		1,921	7.22	0.50	0.07	7.24
NHFE		1,501	7.15	1.09	0.15	7.10		301	7.15	0.91	0.13	7.13
LCR	Cr %	3,434	0.38	0.12	0.30	0.36		687	0.38	0.10	0.26	0.36
SHCR		5,763	0.62	0.12	0.19	0.62		1,155	0.63	0.10	0.17	0.63
NHCR		7,938	0.64	0.12	0.18	0.64		1,590	0.65	0.10	0.15	0.65
VLCR		765	0.41	0.12	0.30	0.39		153	0.41	0.07	0.17	0.41
EST	Co %	17,903	0.013	0.002	0.157	0.013		3,585	0.013	0.002	0.128	0.013
	Pd ppm	17,903	0.018	0.062	3.430	0.003		3,569	0.016	0.028	1.756	0.007
	Pt ppm	17,903	0.012	0.062	5.183	0.003		3,569	0.009	0.012	1.289	0.006
	Density	5,543	2.68	0.13	0.05	2.65						

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. Source: Caracle Creek, 2022.

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 and East zones, with a possibility of reducing it to 10 m x 10 m x 7.5 m in the future.

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.

In order to avoid centroid mismatches between the models in areas where they inevitably overlap, both were sourced from a larger, non-rotated 40 m x 40 m x 15 m parent block model, and as result they also share a unique block ID.

Table 14.4: Block Model Properties

East Zone Block Model	X	Y	Z
Minimum Centroid Coordinates	472,010	5,408,530	-592.5
Box Extents	5,000	2,680	885
Block Size	20	20	15
Number of Blocks	250	134	59
Rotation	-	-	-

Main Zone Block Model	X	Y	Z
Minimum Centroid Coordinates	470,010	5,407,450	-592.5
Box Extents	5,360	4,260	885
Block Size	20	20	15
Number of Blocks	268	213	59
Rotation	-	-	-

Source: Caracle Creek, 2022.

14.7 Estimation Strategy

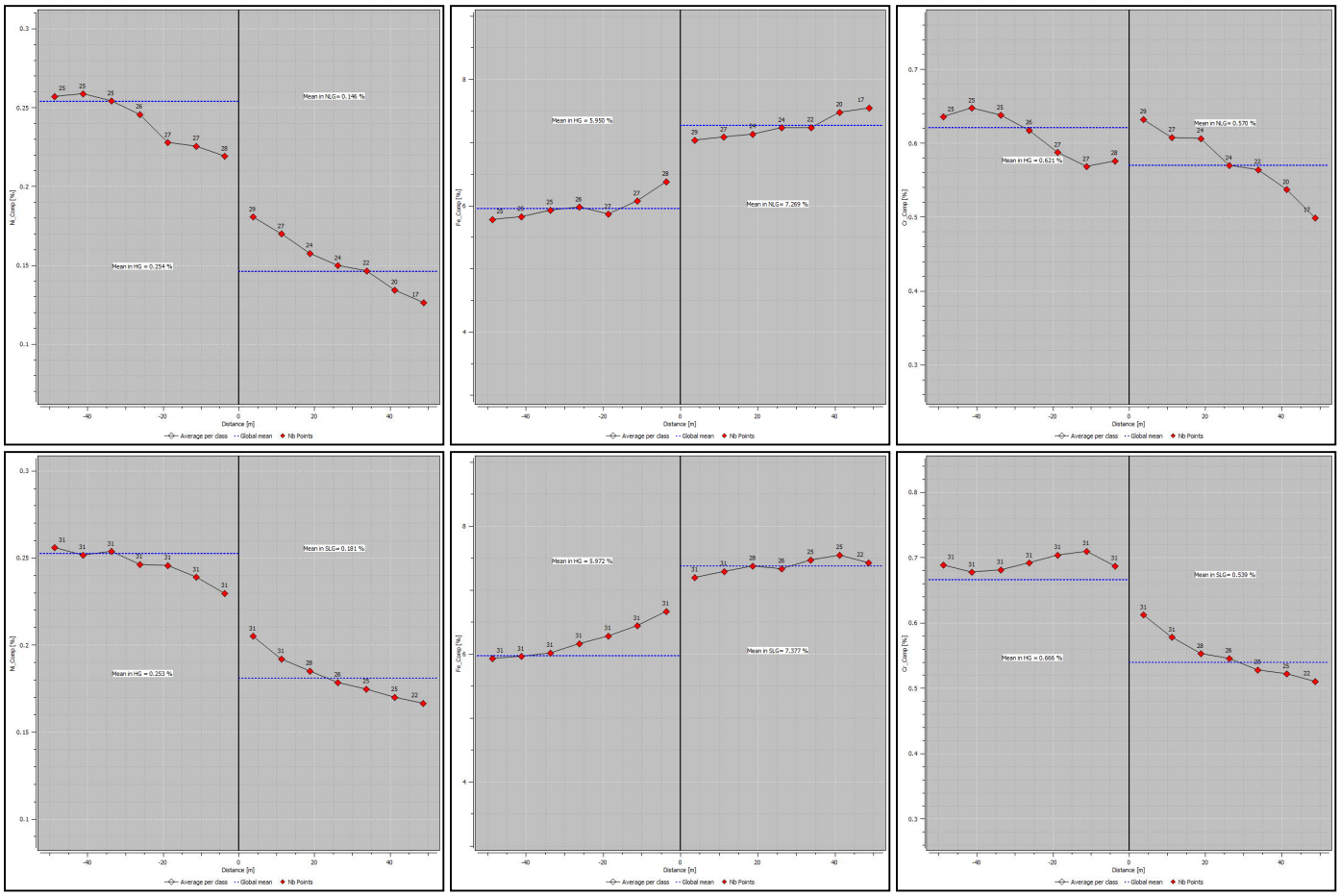
14.7.1 East Zone: Estimation Methodology

The East Zone presents special conditions that allow for an improved nickel estimation strategy. It comprises a single alteration type (MgSerp, see Section 14.4.3) across most of its extent (some talc occurrences appear at the eastern limit), meaning that the density arrangement is fairly uniform and predictable, which in turn favors a good correlation between density and nickel (see Figure 14.13). Thus, a co-kriging approach between both variables was devised, as density values are almost immediately available for every drill hole, providing potential support for an incomplete assay database that depends on laboratory times. To make it possible, given that specific gravity measurements are not collected at the same location or with the same regularity of assay samples, density data points were matched and relocated to the position of the closest nickel composite, with a maximum displacement distance of 3.75m (half a composite) along drill hole traces. These displacements were deemed a reasonable compromise considering the selected block size and should not affect the density estimate significantly.

Due to the noticeable grade differences between the HG, NLG and SLG domains (see Figure 14.26), composites for nickel/density, iron and chromium were filtered for separate estimation within each one. However, given that contacts are transitional (semi-soft boundaries), the filters were required 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 that were to be estimated within subdomains. Other elements such as cobalt, sulphur, palladium and platinum did not show relevant grade variations between subdomains, and were considered for estimation within the general EST-1 domain, with some conditions:

- Palladium and platinum composites with very high grades at the northern contact were excluded as they're part of a PGE "reef" structure (considered for a future estimate), and as such could considerably bias the estimate.
- Sulphur grades were transformed through anamorfosis to a Gaussian distribution (and back-converted after estimation), due to a narrow, very high-grade zone that would cause a large bias with a regular estimation approach.

Figure 14.26: East Zone Contact Analysis Plots of Nickel (left column), Iron (middle colum) and Chromium (right column) for the HG/NLG (top) and HG/SLG (bottom) Domain Transitions



Source: Caracle Creek, 2022.

As previously stated, the EST-2 domain contains only a few composites, which isn't ideal for an OK estimation approach, instead opting for IDW estimation. Similarly, density estimation for the talc alteration subdomains within the EST-1 and EST-2 domains was restricted to IDW due to poor data point numbers.

14.7.2 East Zone: Estimation Parameters

Neighborhood 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 2x4x4 ratio. A three-pass strategy was implemented, each with successively larger neighborhood search radii and more relaxed parameters. A 4th pass with an "infinite" range was considered for any remaining block that did not meet previous criteria.

Table 14.5: East Zone Neighborhood Search and Estimation Parameters for all Estimation Domains

Parameter	Neighborhood			
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 Drill Hole	4	4	4	4
Minimum Points per Drill Hole	-	-	-	-
Minimum Drill Holes	2	1	1	1
Search Radius Directions	182° Az / 75-80° Dip / 0° Pitch			
Search Radius Axis 1	300	2x	3x	∞
Search Radius Axis 2	200	2x	4x	∞
Search Radius Axis 3	50	2x	4x	∞

Source: Caracle Creek, 2022.

14.7.3 Main Zone: Estimation Methodology

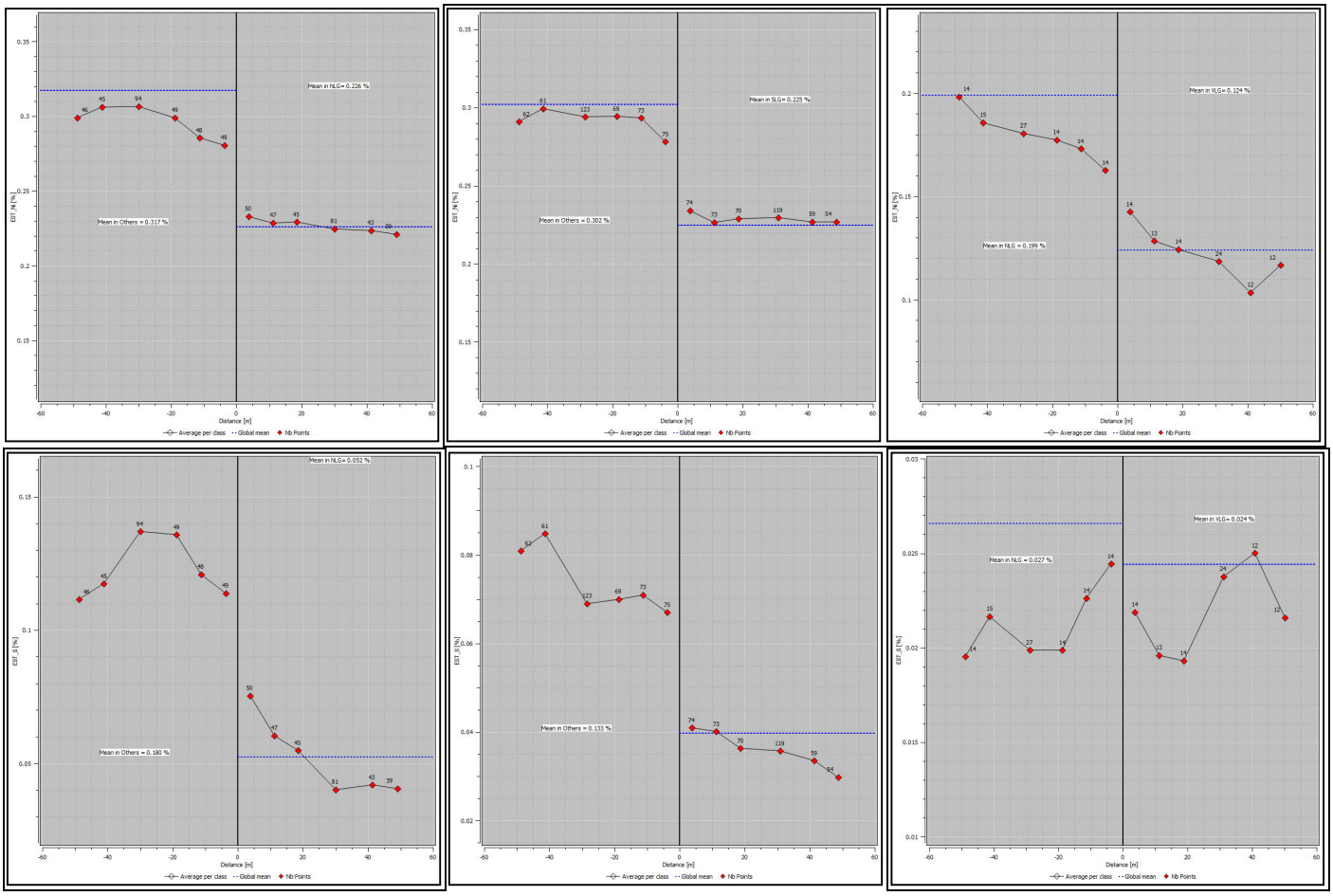
The Main Zone alteration conditions, with MgSerp transitioning to FeSerp towards the west, and the variety of grade distributions among the different variables prevent the application of the East Zone nickel/density co-kriging strategy, as previously discussed (see Section 14.5.4). Hence, densities will be estimated separately within the corresponding alteration domains (with talc restricted to IDW estimation due to poor data point numbers), while nickel and other elements will have individual treatments instead of the more unified approach of the East Zone.

The use of grade shells to separate populations of nickel, iron and chromium guaranteed that estimation domains would display noticeable grade differences between them (see Figures 14.27, 14.28 and 14.29), justifying composite filtering for separate 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 estimation within the nickel subdomains, with the exception of the VLG domain, which showed no transition or grade difference to the NLG domain (see Figure 14.27), meaning that they could be treated as a single domain.

As seen in the East Zone, contacts between 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 that were to be estimated within subdomains, with the exception of 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.

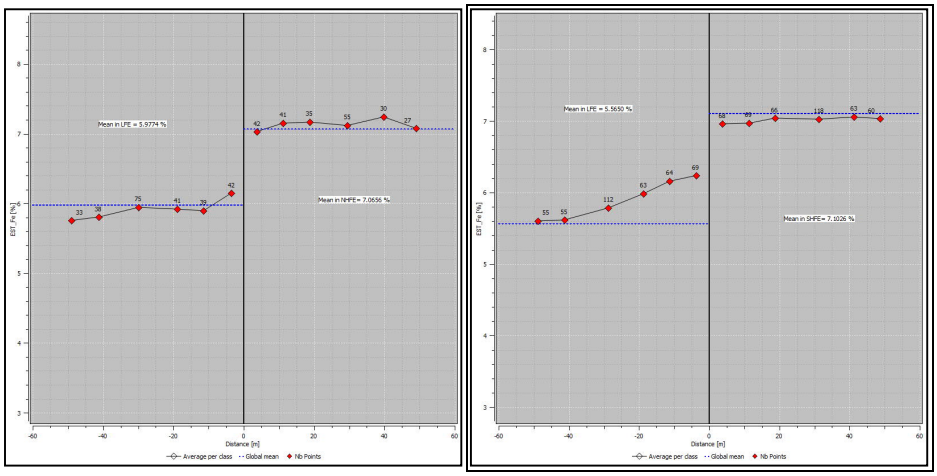
Other elements such as cobalt, palladium and platinum did not show relevant grade variations between subdomains, and were considered for estimation within the general EST domain. Palladium and platinum composites with very high grades at the northern contact were excluded as they're part of a PGE "reef" structure (considered for a future estimate), and as such could considerably bias the estimate. Palladium also showed a slight bimodal distribution that could be resolved by estimating within the nickel subdomains, but this was disregarded in order to maintain consistency with platinum, which did not display this bimodality and is otherwise highly correlated to palladium.

Figure 14.27: Main 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



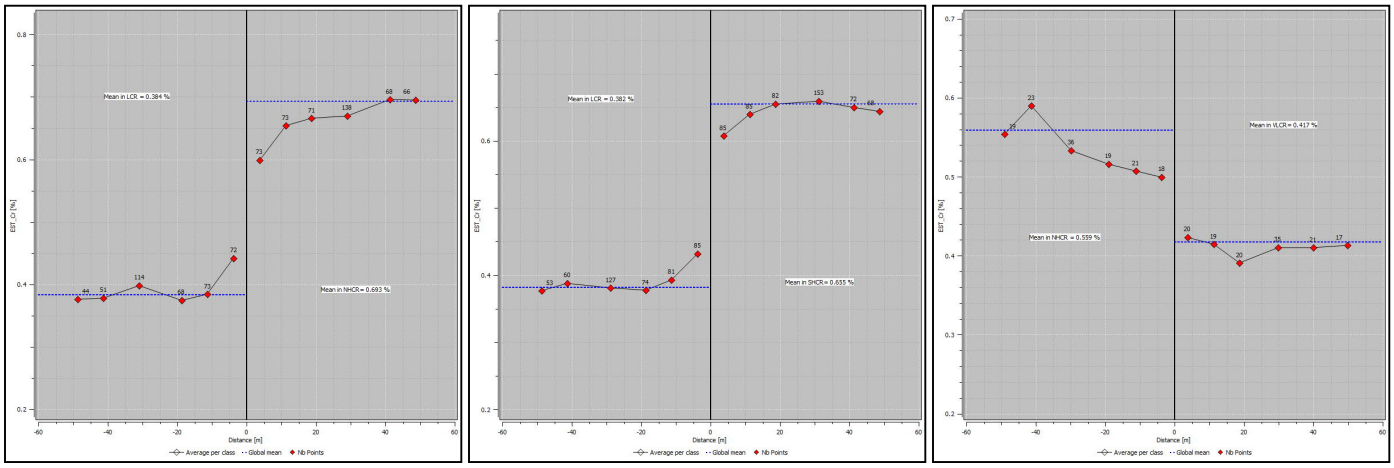
Source: Caracle Creek, 2022.

Figure 14.28: Main Zone Contact Analysis Plots of Iron for the LFE/NHFE (left) and LFE/SHFE (right) Domain Transitions



Source: Caracle Creek, 2022.

Figure 14.29: Main 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, 2022.

14.7.4 Main Zone: Estimation Parameters

Neighborhood 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 the Main Zone deposit, with a mineralization trend that starts in an approximate east-west direction (like in the East 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 generated from the general trends of the contact surfaces for the EST domain and its subdomains.

Blocks were discretized to a 2x4x4 ratio. A three-pass strategy was implemented, each with successively larger neighborhood search radii and more relaxed parameters. A 4th pass with an "infinite" range was considered for any remaining block that did not meet previous criteria.

Table 14.6: Main Zone Neighborhood Search and Estimation Parameters for all Estimation Domains

Parameter	Neighborhood			
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 Drill Hole	4	4	4	4
Minimum Points per Drill Hole	-	-	-	-
Minimum Drill Holes	2	1	1	1
Search Radius Directions	Local Anisotropies			
Search Radius Axis 1 (range)	100-300	2x	4x	∞
Search Radius Axis 2 (range)	50-250	2x	4x	∞
Search Radius Axis 3 (range)	40-150	2x	4x	∞

Source: Caracle Creek, 2022.

14.8 Variography

14.8.1 East Zone

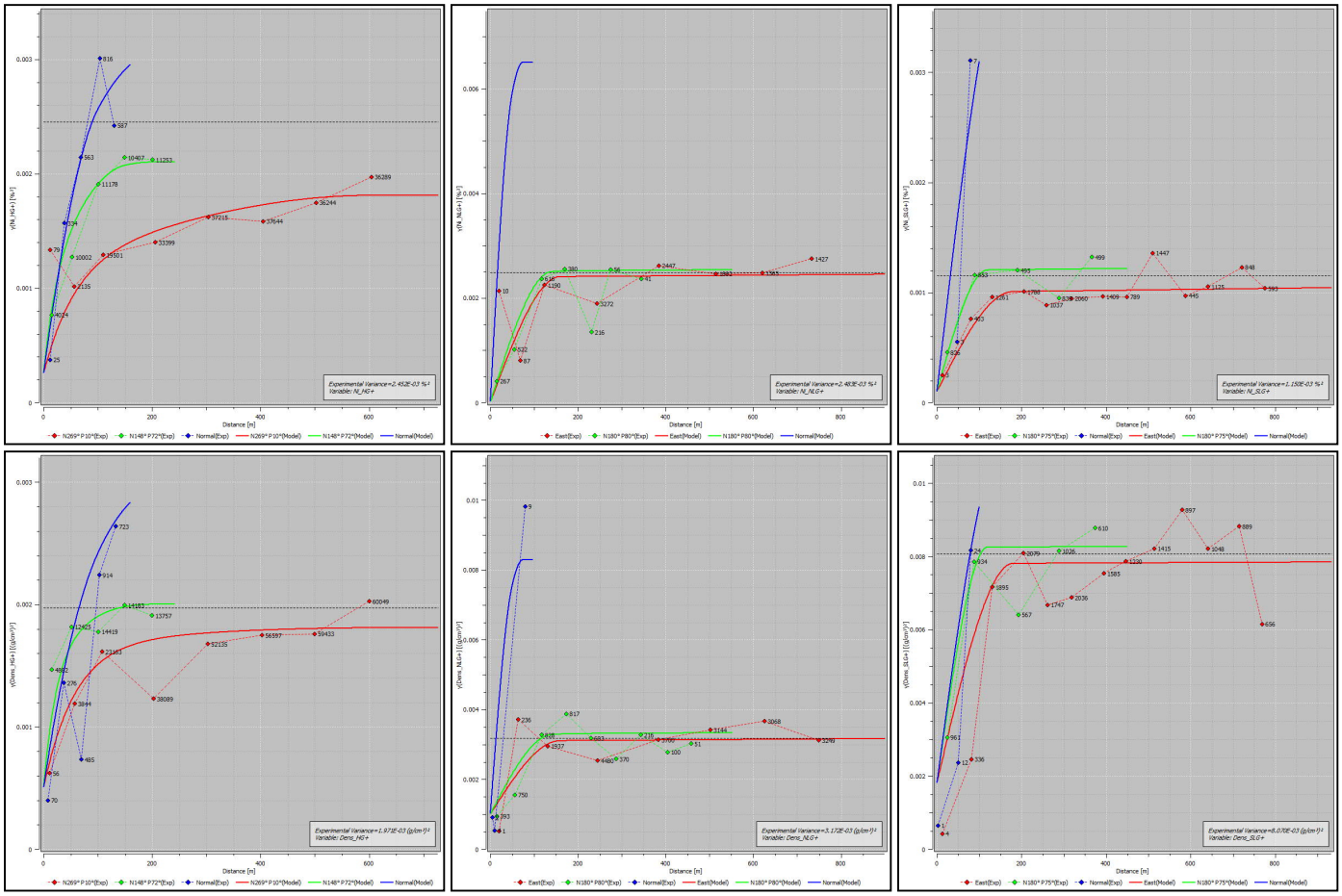
Variography was carried out for the 7 studied elements and density within the EST-1 domain or its subdomains, depending on the spatial distribution and variability of their grades. The EST-2 domain did not require variography. A comprehensive list of variables and their estimation domains is presented next:

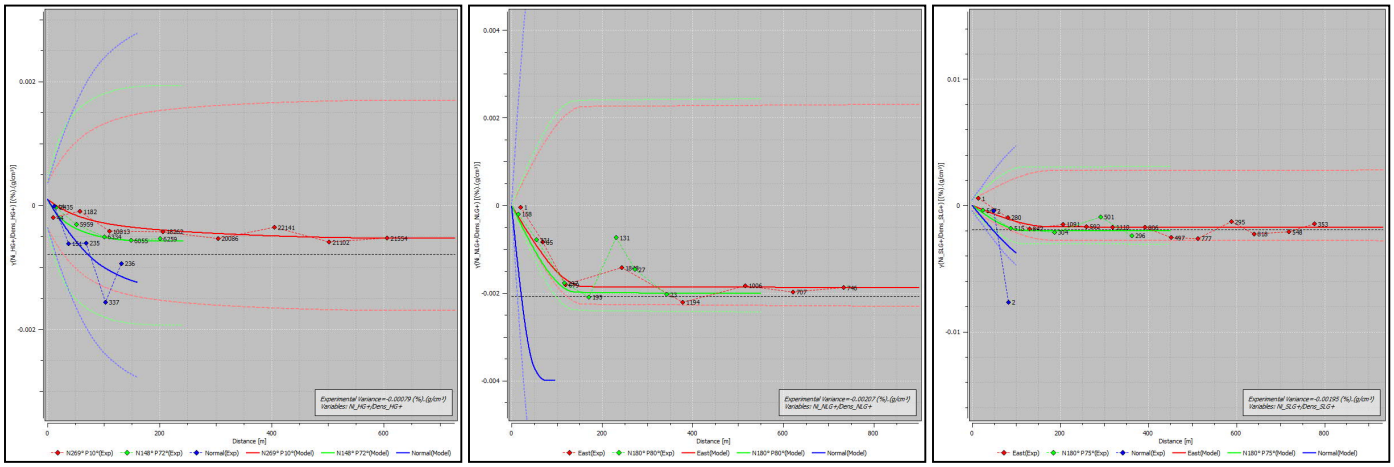
- Nickel/Density (direct and cross variography): HG, NLG and SLG domains (Figure 14.30), with densities limited to the MgSerp alteration domain. The talc alteration domain did not require density variography. Additional nickel variogram within the general EST-1 domain for resource classification (see Figure 14.44).
- Cobalt: EST-1 domain (Figure 14.31).
- Iron: HG, NLG and SLG domains (Figure 14.32).
- Chromium: HG, NLG and SLG domains.
- Sulphur (gaussian transformed variography): EST-1 domain (Figure 14.31).
- Palladium: EST-1 domain.
- Platinum: EST-1 domain.

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 92° azimuth and dips ranging between 70-80°S.

Multidirectional variograms were modelled (and cross-variograms in the case of nickel and density) using declustering weights and considering zonal anisotropies due to the significant variability differences between directions. Some notable grade outliers were excluded in some 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 to evaluate variogram robustness and served for recalibration of the nugget value to achieve better validation results.

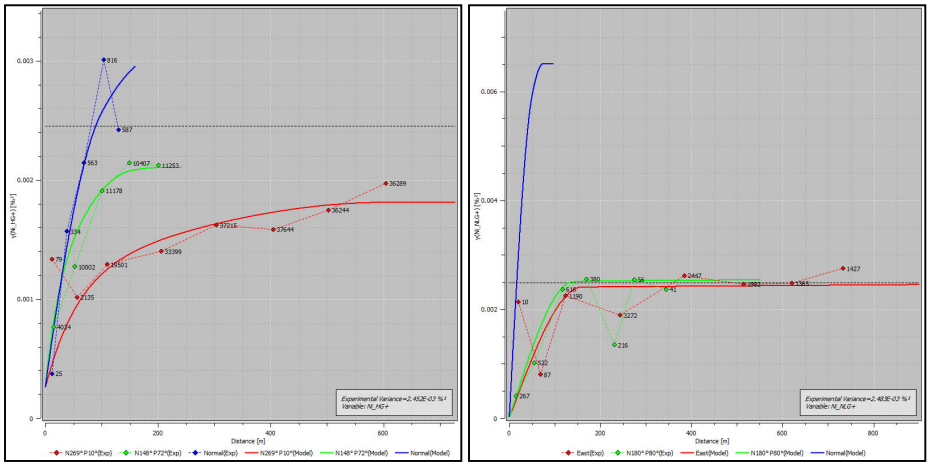
Figure 14.30: East Zone Variograms of Nickel (top row), Density (middle row) and their Cross-Variograms (bottom row) for the HG (left column), NLG (middle column) and SLG (right column) Estimation Domains





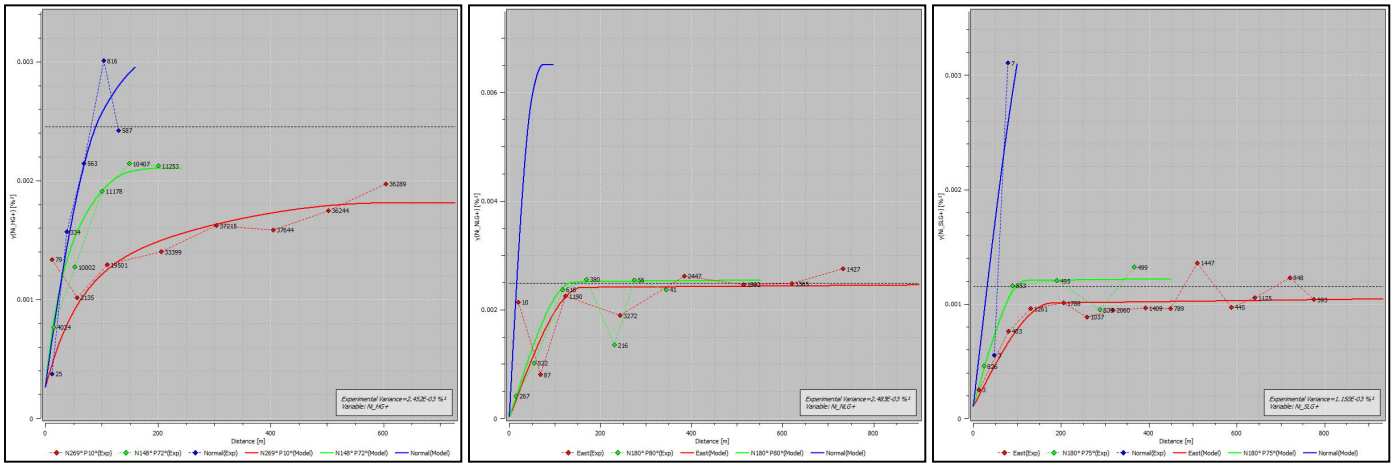
Source: Caracle Creek, 2022.

Figure 14.31: East Zone Variograms of Cobalt (Left) and Sulphur (Right) for the EST-1 Estimation Domain



Source: Caracle Creek, 2022.

Figure 14.32: East Zone Variograms of Iron for the HG (left), NLG (middle) and SLG (right) Estimation Domains



Source: Caracle Creek, 2022.

14.8.2 Main Zone

Variography was carried out for the 7 studied elements and density within the EST domain or its subdomains, depending on the spatial distribution and variability of their grades. As previously noted, these variograms comprise composites of the Main and West zones, which were estimated as a single deposit given the mineralization continuity between them. A comprehensive list of variables and their estimation domains is presented next:

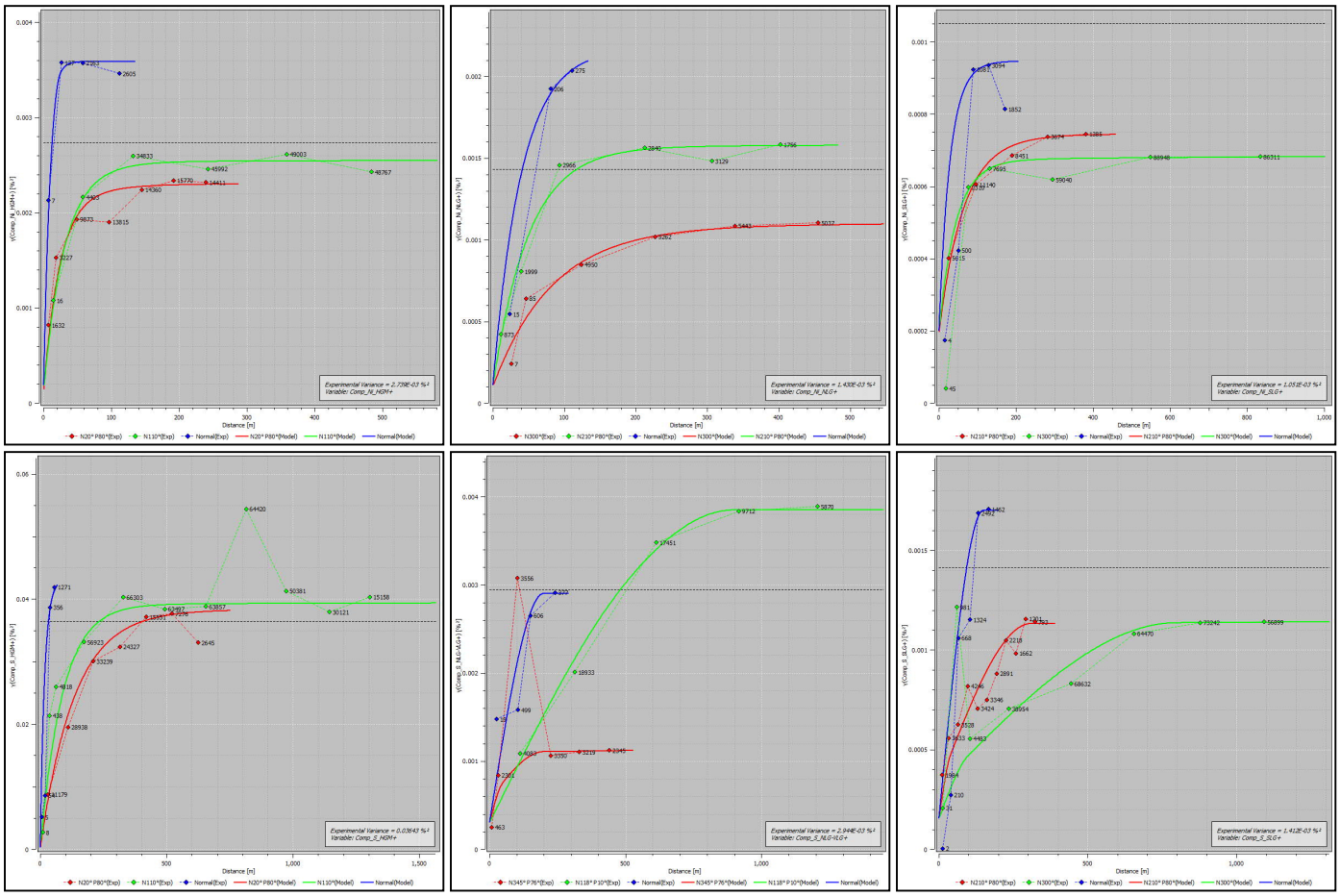
- Nickel: HG, NLG, SLG and VLG domains (Figure 14.33). Additional nickel variogram within the general EST domain for resource classification (see Figure 14.46).
- Cobalt: EST domain.
- Iron: LFE, NHFE and SHFE domains (see Figure 14.34).
- Chromium: LCR, NHCR, SHCR and VLCR domains (Figure 14.35).
- Sulphur: HG, NLG, SLG+VLG (combined) domains (Figure 14.33).
- Palladium: EST domain.
- Platinum: EST domain.
- Density: MgSerp, FeSerp Unweathered and FeSerp Weathered alteration domains (see Section 14.4.3). The talc alteration 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, with the exception of the sub-horizontal density trend within FeSerp domains.

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 some instances to reduce noise. Down-the-hole variograms were also modelled in each case for an initial approach to the nugget value. Cross-

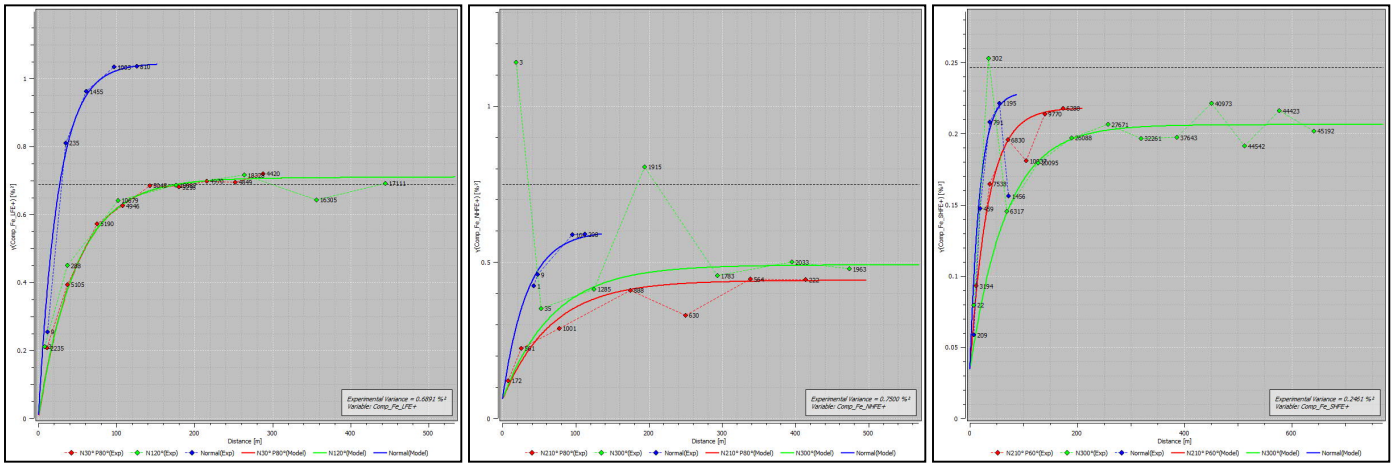
validation was carried out to evaluate variogram robustness and served for recalibration of the nugget value to achieve better validation results.

Figure 14.33: Main Zone Variograms of Nickel (top row), and Sulphur (bottom row) for the HG (left column), NLG (middle column) and SLG (right column) Estimation Domains



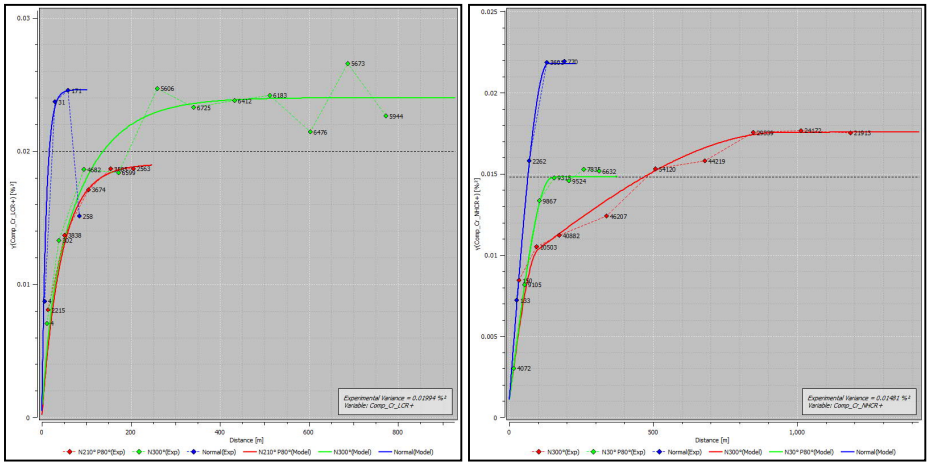
Note: The sulphur variogram for the NLG domain also includes the VLG domain, as they were combined for this element. The nickel variogram for the VLG estimation domain was not included for practical purposes. Source: Caracle Creek, 2022.

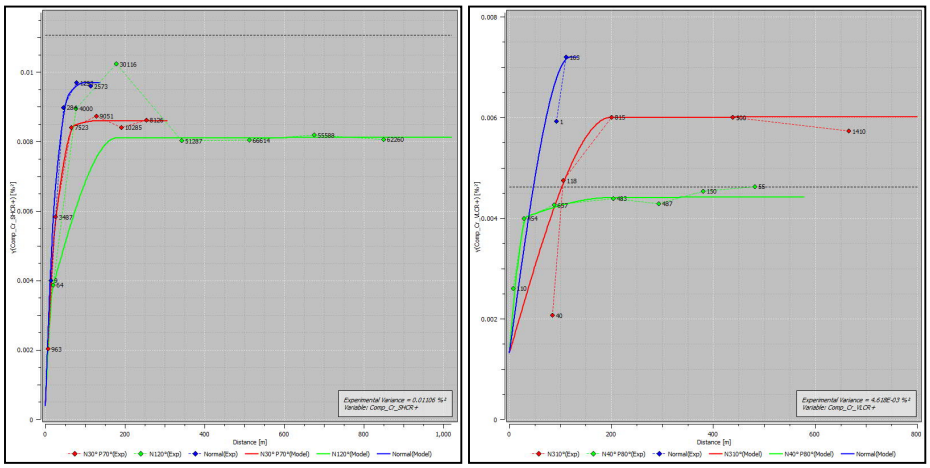
Figure 14.34: Main Zone Variograms of Iron for the LFE (left), NHFE (middle) and SHFE (right) Estimation Domains



Source: Caracle Creek, 2022.

Figure 14.35: Main Zone Variograms of Chromium for the LCR (top left), NHCR (top right), SHCR (bottom left) and VLCR (bottom right) Estimation Domains





Source: Caracle Creek, 2022.

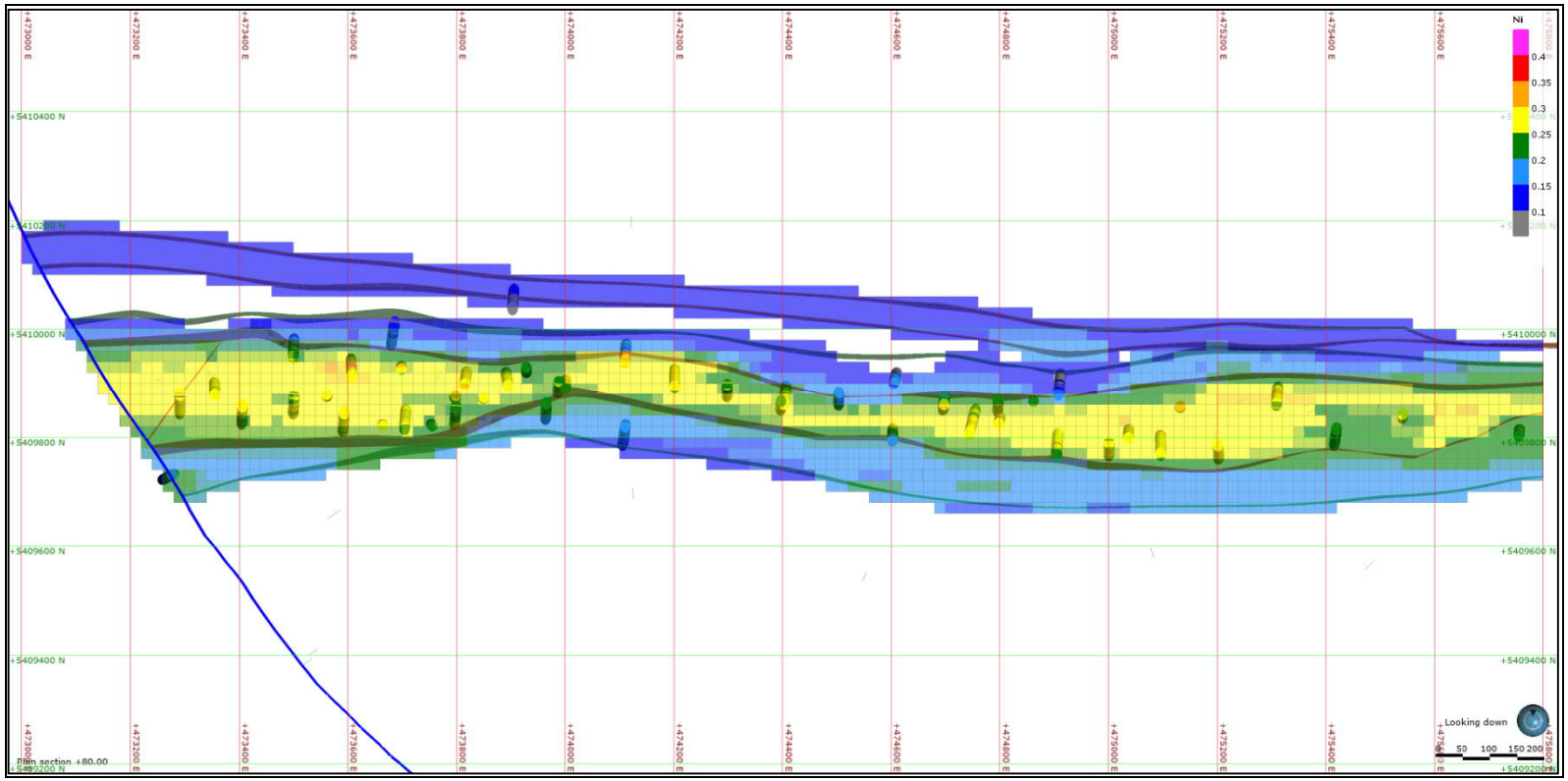
14.9 Block Model Validation

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 only when possible, for other elements.

14.9.1 Visual Validation

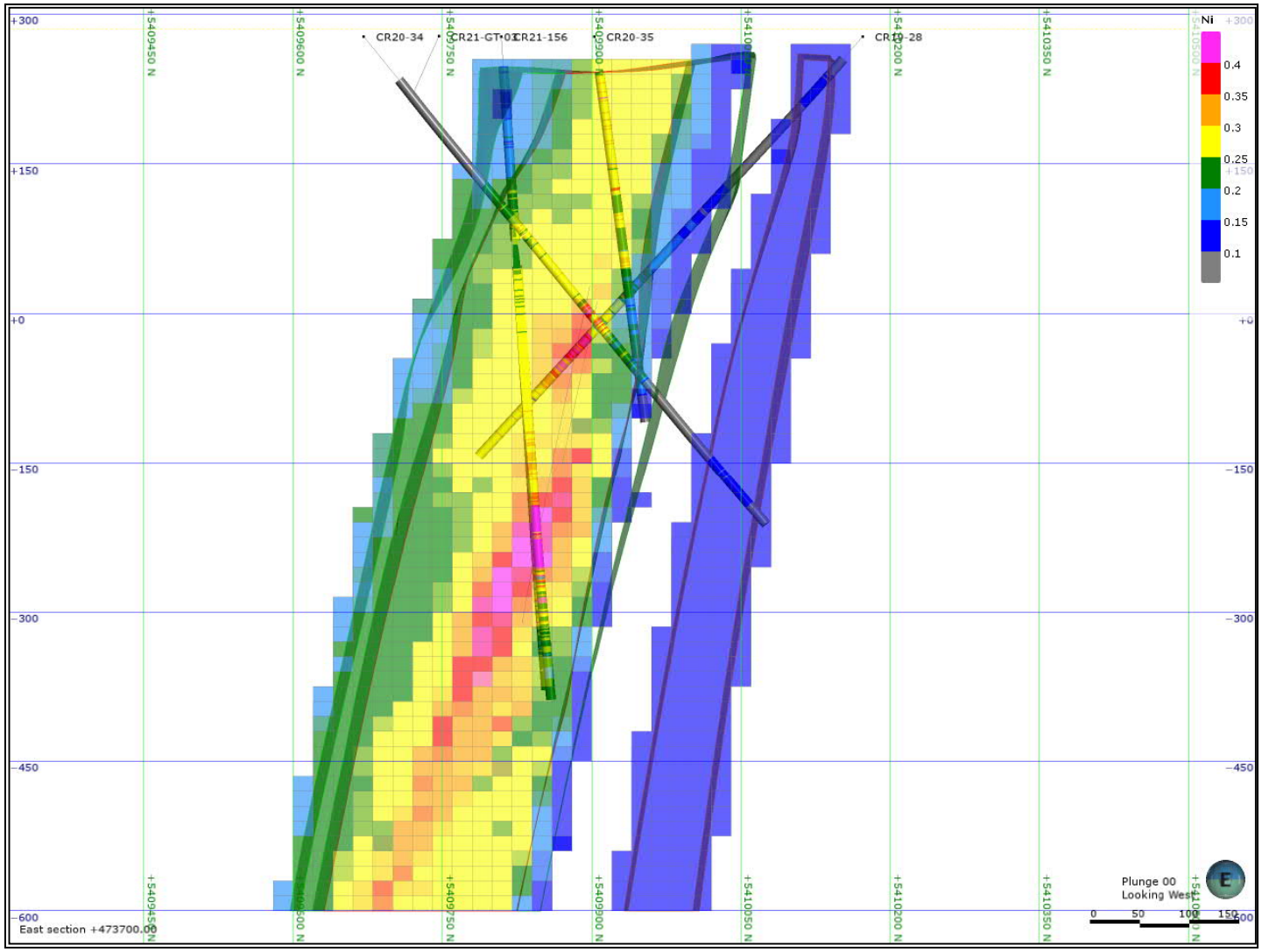
Predefined sections (Figures 14.37 and 14.39), based on drill hole direction and location, were used for visual comparison of block models and composites, as well as plan views (Figures 14.36 and 14.38). These show generally good consistency between estimates and composites.

Figure 14.36: East Zone Plan Section (80 m) comparing Block Model Grades against Nickel Composites for all Estimation Domains



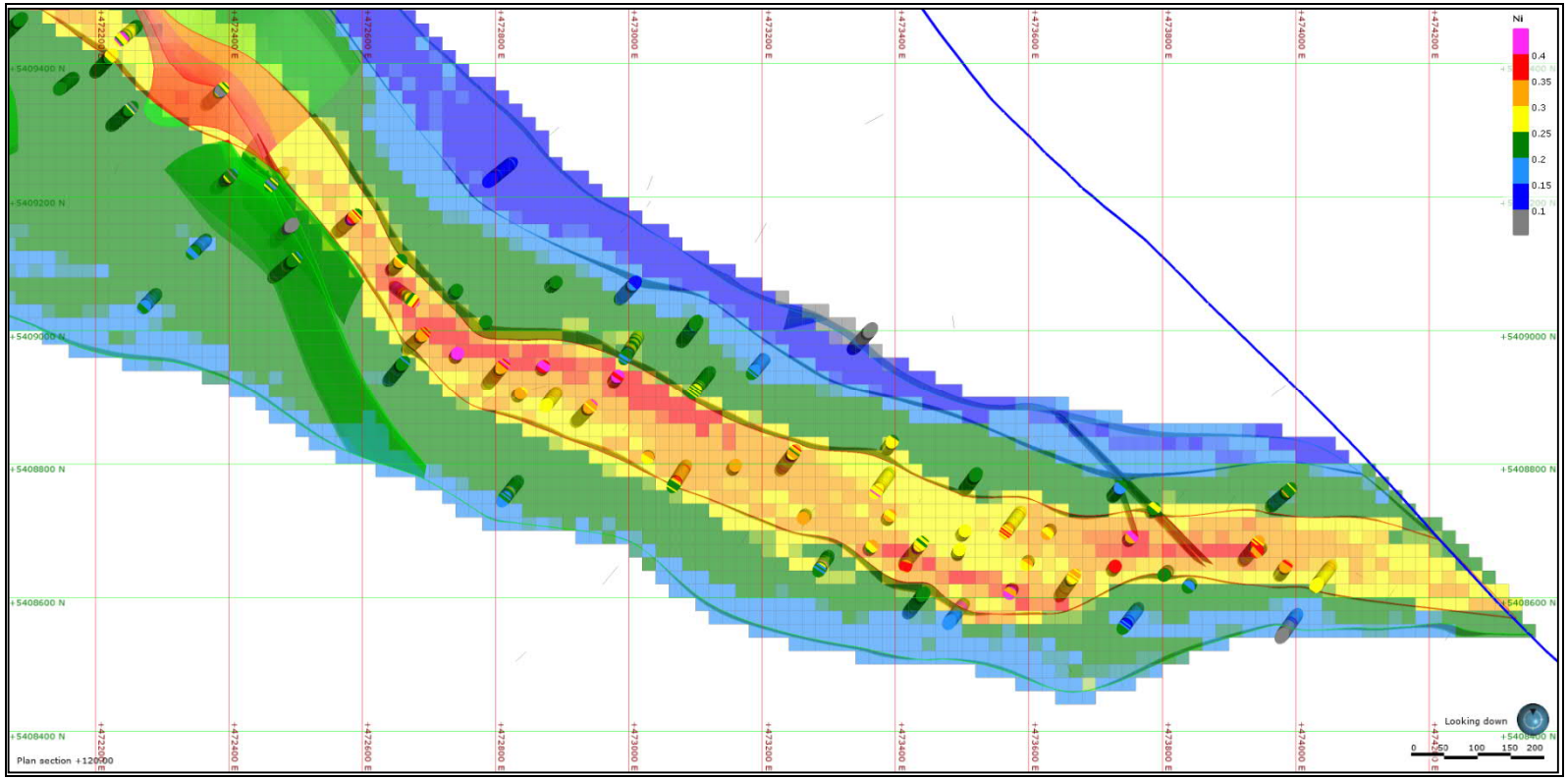
Source: Caracle Creek, 2022.

Figure 14.37: East Zone Section (473700N) looking West comparing Block Model Grades against Nickel Composites for all Estimation Domains



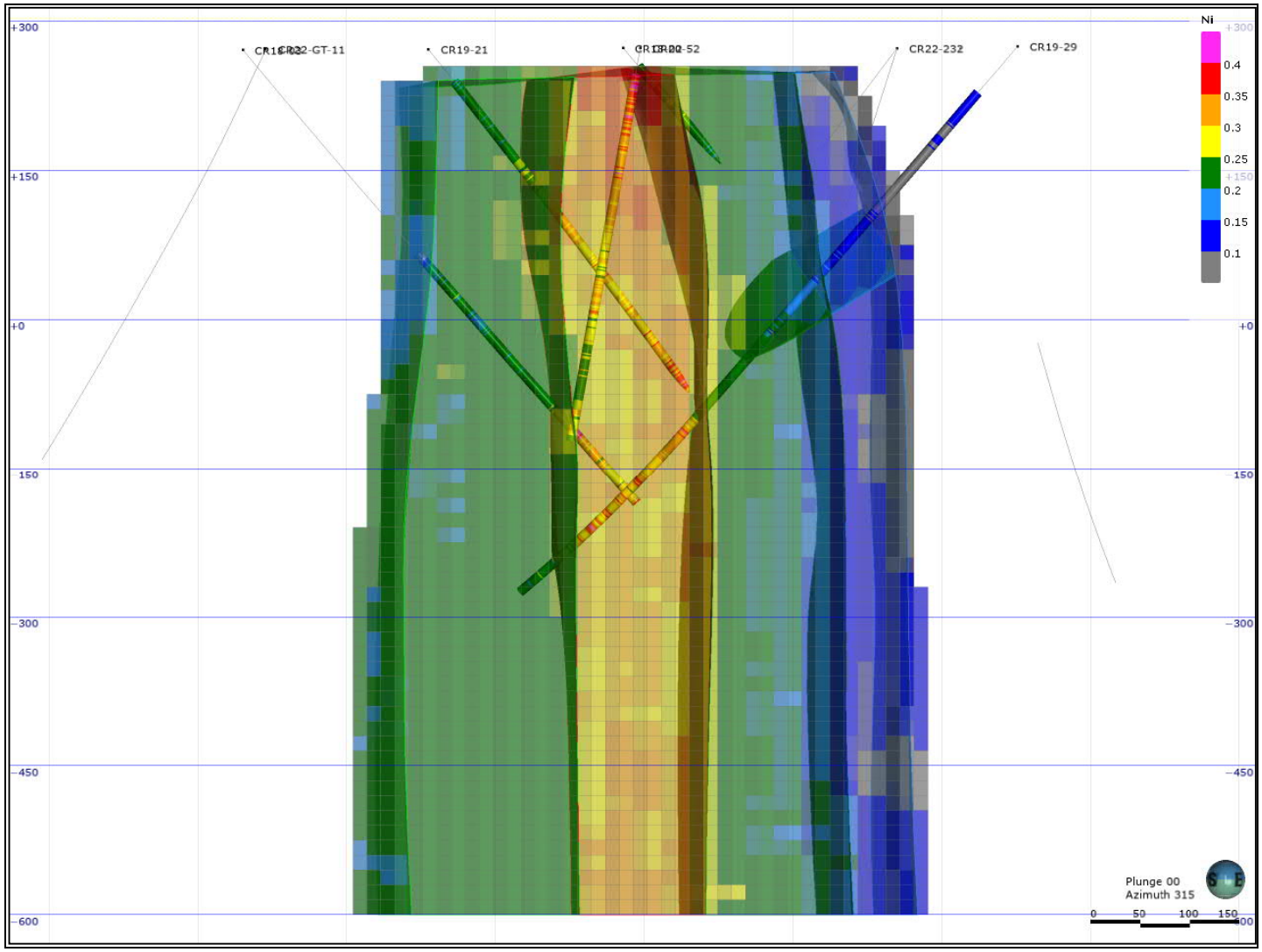
Source: Caracle Creek, 2022.

Figure 14.38: Main Zone Plan Section (120 m) comparing Block Model Grades against Nickel Composites for all Estimation Domains



Source: Caracle Creek, 2022.

Figure 14.39: Main Zone Oblique Section looking West comparing Block Model Grades against Nickel Composites for all Estimation Domains



Source: Caracle Creek, 2022.

14.9.2 Statistical Validation

Global bias measures the percentage difference between 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.

Most variables within the East Zone show generally good consistency, with sulphur and PGE estimations as notable exceptions, all showing apparent underestimation. These results are not unexpected given the high coefficient of variance of these variables. PGE estimations are mostly a case of low mean values (*e.g.*, 0.0121 ppm Pd for composites vs 0.011 ppm Pd for OK and 0.0083 ppm Pt for composites vs 0.0076 ppm Pt for OK) which, as previously stated, tend to produce large percentage differences for small variations. Sulphur, however, is not just a case of low mean values, but an indication that alternative estimation strategies should be investigated.

Table 14.7: East Zone Global Statistical Comparisons between Estimates and Composites in Nickel Domains

Element	Domain	Data	Mean	Bias	Std Dev	CV
Ni %	HG	Composites	0.26	-	0.0030	0.21
		OK	0.25	-2.12%	0.0012	0.14
		IDW	0.26	-1.65%	0.0013	0.14
		NN	0.25	-4.84%	0.0026	0.21
	NLG	Composites	0.15	-	0.0027	0.34
		OK	0.15	-1.99%	0.0014	0.25
		IDW	0.15	-3.33%	0.0013	0.25
		NN	0.14	-10.70%	0.0034	0.43
	SLG	Composites	0.18	-	0.0012	0.19
		OK	0.19	4.08%	0.0007	0.14
		IDW	0.19	3.42%	0.0005	0.11
		NN	0.18	0.79%	0.0010	0.17
Co %	EST	Composites	0.013	-	0.000003	0.128
		OK	0.013	0.959%	0.000001	0.079
		IDW	0.013	1.283%	0.000001	0.082
		NN	0.013	0.012%	0.000002	0.115
Fe %	HG	Composites	6.01	-	0.57	0.13
		OK	6.11	1.68%	0.21	0.08
		IDW	6.06	0.89%	0.22	0.08
		NN	6.05	0.83%	0.80	0.15
	NLG	Composites	7.07	-	0.61	0.11
		OK	7.15	1.25%	0.25	0.07
		IDW	7.08	0.20%	0.32	0.08
		NN	6.98	-1.24%	0.58	0.11
	SLG	Composites	7.32	-	0.41	0.09
		OK	7.42	1.32%	0.12	0.05
		IDW	7.45	1.82%	0.13	0.05
		NN	7.38	0.82%	0.36	0.08
Cr %	HG	Composites	0.65	-	0.011	0.16
		OK	0.64	-1.20%	0.002	0.08

Element	Domain	Data	Mean	Bias	Std Dev	CV
		IDW	0.65	-0.02%	0.003	0.09
		NN	0.63	-1.76%	0.009	0.15
	NLG	Composites	0.55	-	0.016	0.23
		OK	0.51	-6.65%	0.005	0.14
		IDW	0.52	-5.97%	0.005	0.14
		NN	0.49	-9.75%	0.011	0.21
	SLG	Composites	0.54	-	0.011	0.19
		OK	0.55	1.84%	0.003	0.11
		IDW	0.57	4.54%	0.004	0.12
		NN	0.54	-0.91%	0.008	0.16
S %	EST	Composites	0.06	-	0.010	1.70
		OK	0.05	-17.36%	0.005	1.45
		IDW	0.06	2.32%	0.006	1.31
		NN	0.05	-13.90%	0.007	1.65
Pd ppm	EST	Composites	0.012	-	0.00111	2.747
		OK	0.011	-9.658%	0.00014	1.073
		IDW	0.011	-10.085%	0.00018	1.221
		NN	0.012	-3.890%	0.00088	2.540
Pt ppm	EST	Composites	0.008	-	0.00026	1.925
		OK	0.008	-8.706%	0.00004	0.807
		IDW	0.008	-7.123%	0.00005	0.931
		NN	0.011	26.631%	0.00072	2.544
Density (MgSerp)	HG	Composites	2.62	-	0.0018	0.02
		OK	2.63	0.22%	0.0004	0.01
		IDW	2.63	0.10%	0.0004	0.01
		NN	2.64	0.41%	0.0018	0.02
	NLG	Composites	2.69	-	0.0043	0.02
		OK	2.70	0.50%	0.0011	0.01
		IDW	2.70	0.47%	0.0008	0.01
		NN	2.71	0.98%	0.0042	0.02
	SLG	Composites	2.72	-	0.0089	0.03
		OK	2.71	-0.54%	0.0029	0.02
		IDW	2.71	-0.59%	0.0029	0.02
		NN	2.72	-0.17%	0.0081	0.03

Source: Caracle Creek, 2022.

Similarly, most variables within the Main Zone show generally good consistency, with low-grade sulphur and palladium estimations as notable exceptions, 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 14.8: Main Zone Global Statistical Comparisons between Estimates and Composites in Nickel Domains

Element	Domain	Data	Mean	Bias	Std Dev	CV
Ni %	HG	Composites	0.30	-	0.003	0.17

Element	Domain	Data	Mean	Bias	Std Dev	CV
		OK	0.30	-2.00%	0.001	0.10
		IDW	0.29	-2.16%	0.001	0.10
		NN	0.30	-1.82%	0.003	0.18
	NLG	Composites	0.22	-	0.001	0.17
		OK	0.22	2.12%	0.001	0.10
		IDW	0.22	2.28%	0.000	0.09
		NN	0.21	-2.20%	0.002	0.20
	SLG	Composites	0.22	-	0.001	0.15
		OK	0.21	-2.42%	0.000	0.10
		IDW	0.22	0.72%	0.000	0.09
		NN	0.22	1.41%	0.001	0.15
	VLG	Composites	0.12	-	0.001	0.28
		OK	0.11	-6.00%	0.001	0.25
		IDW	0.11	-9.95%	0.001	0.25
		NN	0.11	-5.45%	0.001	0.32
Co %	EST	Composites	0.013	-	0.000002	0.117
		OK	0.013	0.879%	0.000001	0.074
		IDW	0.013	0.871%	0.000001	0.078
		NN	0.013	0.413%	0.000002	0.119
Fe %	LFE	Composites	5.96	-	0.689	0.14
		OK	6.24	4.82%	0.256	0.08
		IDW	6.21	4.30%	0.276	0.08
		NN	6.30	5.74%	0.620	0.13
	NHFE	Composites	7.03	-	0.750	0.12
		OK	7.02	-0.14%	0.310	0.08
		IDW	7.09	0.88%	0.372	0.09
		NN	7.11	1.19%	1.105	0.15
	SHFE	Composites	7.26	-	0.246	0.07
		OK	7.39	1.72%	0.073	0.04
		IDW	7.36	1.36%	0.093	0.04
		NN	7.36	1.39%	0.213	0.06
Cr %	LCR	Composites	0.46	-	0.021	0.32
		OK	0.46	0.17%	0.004	0.13
		IDW	0.46	1.32%	0.005	0.15
		NN	0.50	8.35%	0.021	0.29
	NHCR	Composites	0.60	-	0.018	0.23
		OK	0.59	-1.19%	0.006	0.13
		IDW	0.59	-1.63%	0.006	0.14
		NN	0.57	-3.70%	0.019	0.24
	SHCR	Composites	0.60	-	0.011	0.18
		OK	0.59	-1.05%	0.004	0.10
		IDW	0.59	-1.16%	0.005	0.12
		NN	0.57	-4.32%	0.012	0.19
	VLCR	Composites	0.42	-	0.005	0.17
		OK	0.42	-0.56%	0.001	0.09
		IDW	0.42	-1.11%	0.002	0.10
		NN	0.42	-0.11%	0.006	0.18

Element	Domain	Data	Mean	Bias	Std Dev	CV
S %	HG	Composites	0.14	-	0.031	1.29
		OK	0.13	-3.75%	0.016	0.95
		IDW	0.13	-8.66%	0.016	1.03
		NN	0.13	-4.56%	0.032	1.37
	NLG+VLG	Composites	0.04	-	0.003	1.36
		OK	0.04	-12.71%	0.001	1.02
		IDW	0.03	-15.54%	0.001	1.06
		NN	0.04	-9.71%	0.003	1.42
	SLG	Composites	0.04	-	0.001	1.09
		OK	0.03	-15.33%	0.001	0.79
		IDW	0.03	-9.29%	0.001	0.73
		NN	0.03	-8.90%	0.001	1.11
Pd ppm	EST	Composites	0.016	-	0.00167	2.632
		OK	0.011	-29.140%	0.00017	1.184
		IDW	0.012	-24.319%	0.00018	1.131
		NN	0.014	-10.409%	0.00077	1.994
Pt ppm	EST	Composites	0.010	-	0.00033	1.839
		OK	0.009	-11.305%	0.00004	0.722
		IDW	0.009	-7.097%	0.00004	0.699
		NN	0.010	3.098%	0.00020	1.375
Density	MgSerp	Composites	2.67	-	0.008	0.03
		OK	2.68	0.27%	0.003	0.02
		IDW	2.68	0.49%	0.003	0.02
		NN	2.69	0.89%	0.008	0.03
	FeSerp Unweathered	Composites	2.77	-	0.016	0.05
		OK	2.77	0.17%	0.007	0.03
		IDW	2.79	0.79%	0.008	0.03
		NN	2.82	1.76%	0.021	0.05
	FeSerp Weathered	Composites	2.57	-	0.008	0.04
		OK	2.58	0.39%	0.003	0.02
		IDW	2.56	-0.21%	0.002	0.02

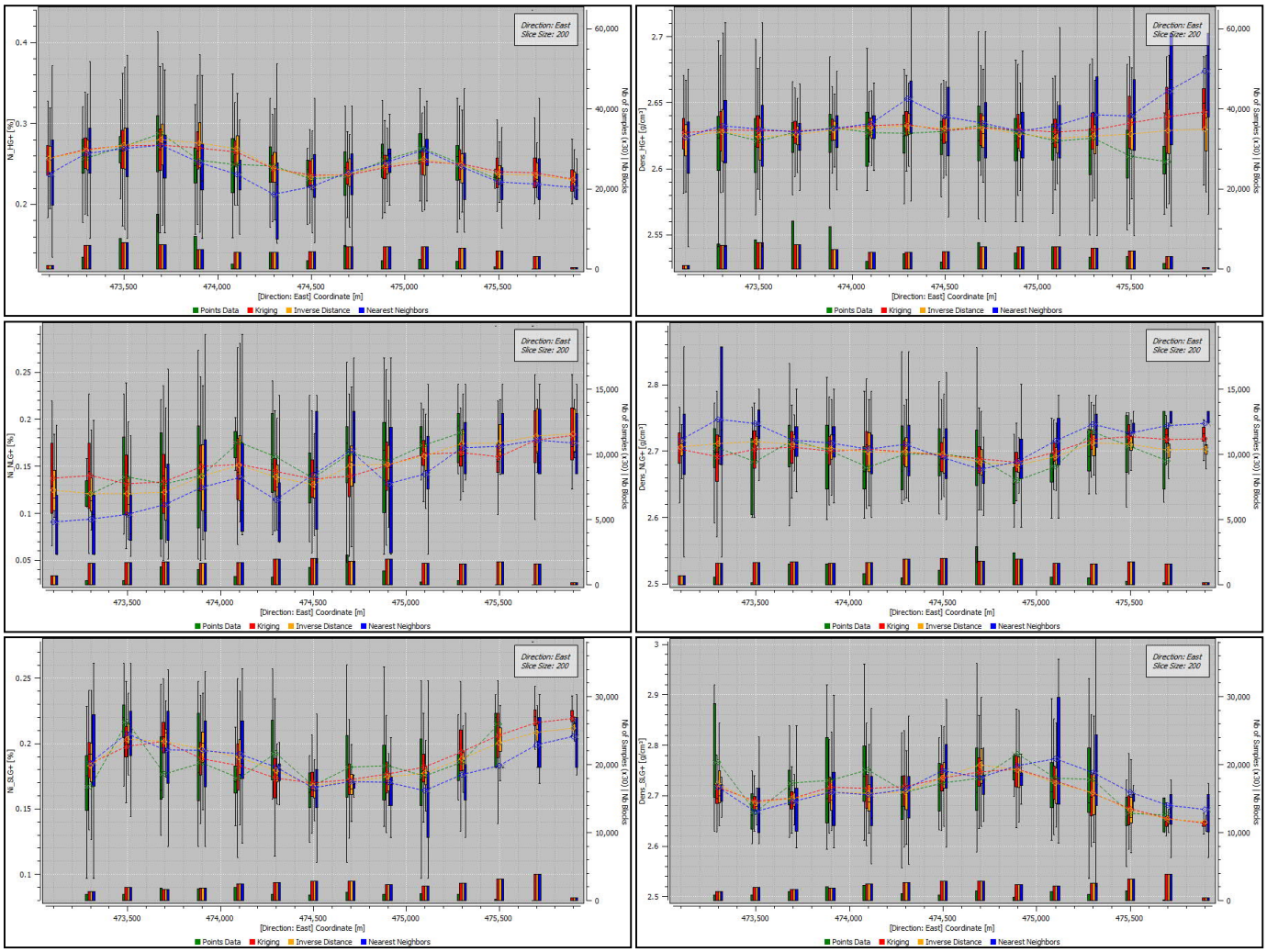
Source: Caracle Creek, 2022.

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.

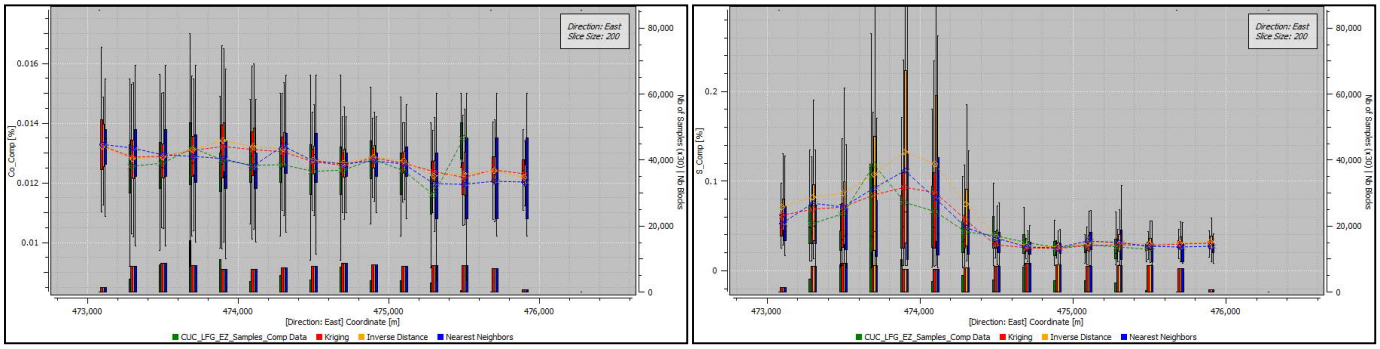
All variables show generally good consistency between estimates and composites, especially in windows with high composite counts. Some notable deviations between composites/estimates and NN interpolation manifest in some cases, an occurrence also observed in statistical validations and which is currently being investigated, even if it does not really affect the original estimation results.

Figure 14.40: East Zone 200-m Spaced East-West Swath Plots of Nickel (left column) and Density (right column) for the HG (top row), NLG (middle row) and SLG (bottom row) Estimation Domains



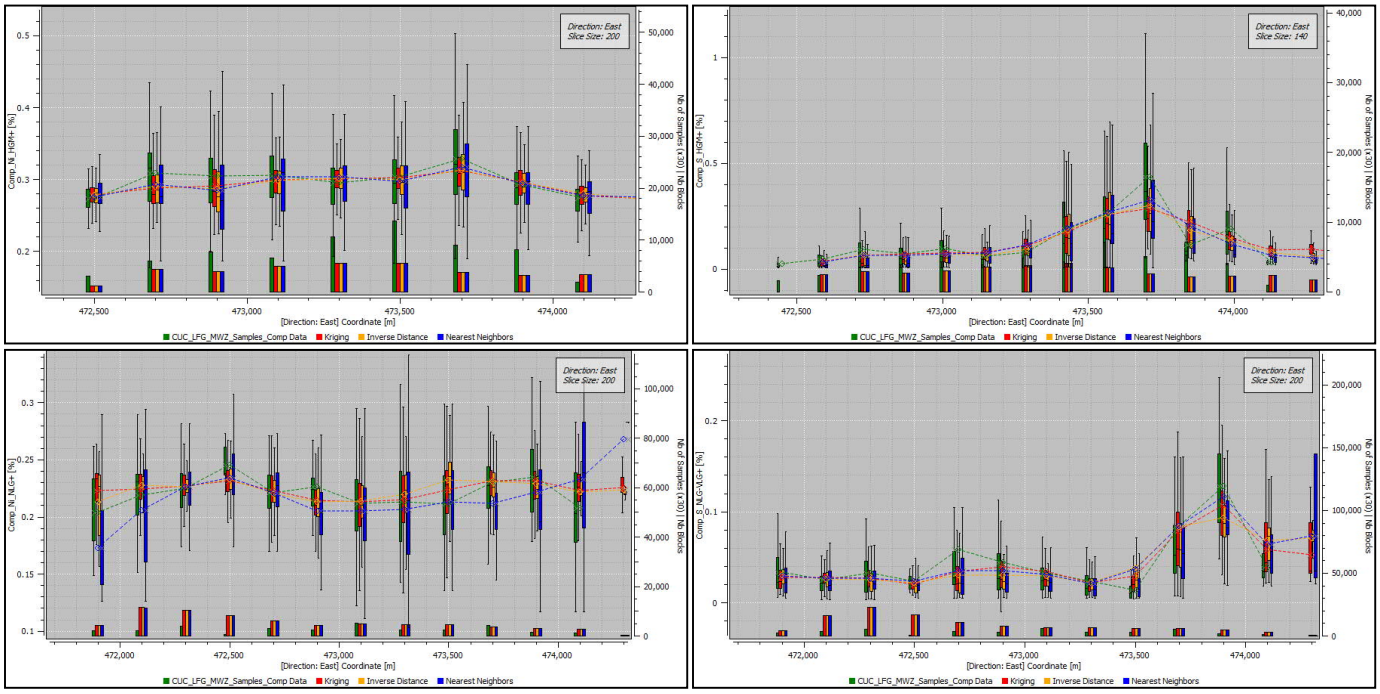
Source: Caracle Creek, 2022.

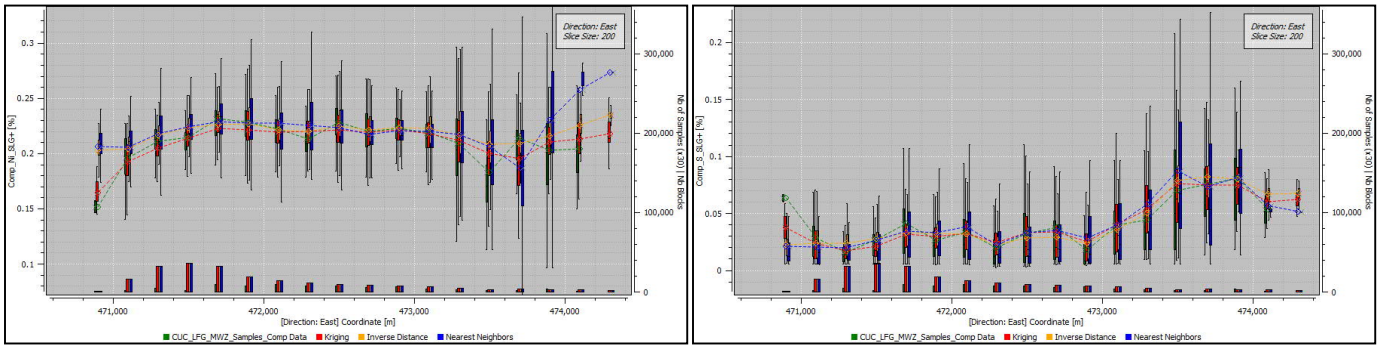
Figure 14.41: East Zone 200-m Spaced East-West Swath Plots of Cobalt (left) and Sulphur (right) for the EST-1 Estimation Domain



Source: Caracle Creek, 2022.

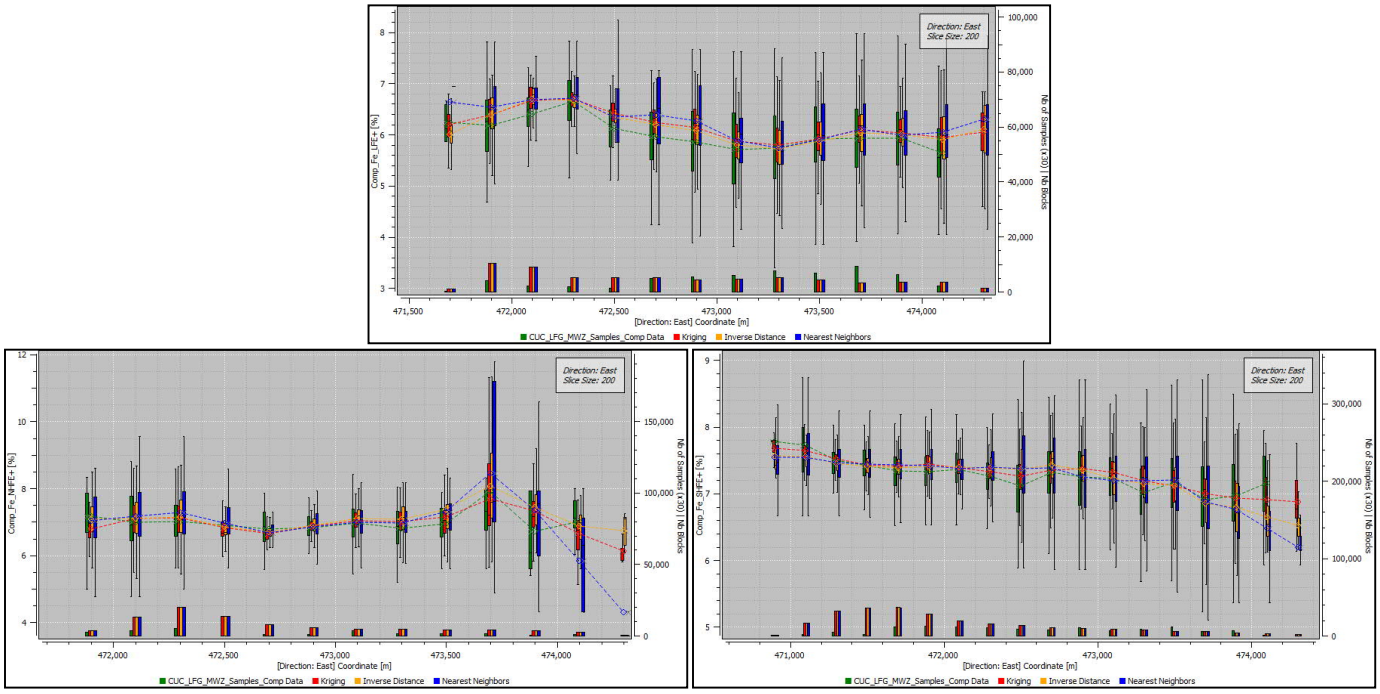
Figure 14.42: Main Zone 200-m Spaced East-West Swath Plots of Nickel (left column), and Sulphur (right column) for the HG (top row), NLG (middle row) and SLG (bottom row) Estimation Domains





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, 2022.

Figure 14.43: East 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, 2022.

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 on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

- 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 drill holes 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 all nickel composites within the general EST-1 domain (Figure 14.44), and the block evaluation 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 drill holes to achieve Measured status and two for Indicated.

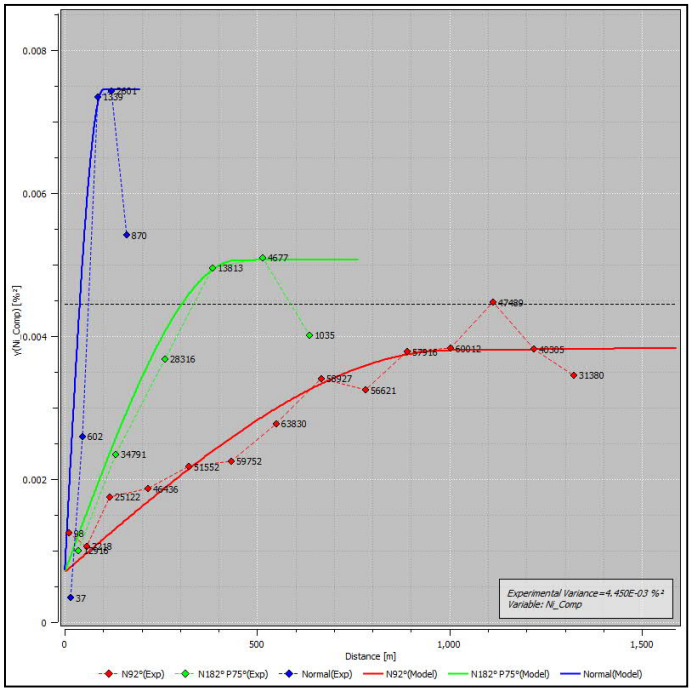
The result from the block evaluation was reviewed for any isolated blocks or group of blocks, which were assigned to the surrounding class if deemed appropriate. Smoothing was not carried out since the resulting classification was sufficiently coherent (Figure 14.45), though it is a decision that will be revisited in the future.

Table 14.9: East Zone Neighborhood Search and Estimation Parameters for Resource Classification

Parameter	Neighborhood			
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 Drill Hole	4	4	4	4
Minimum Points per Drill Hole	-	-	-	-
Minimum Drill Holes	3	2	1	1
Search Radius Directions	182° 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	∞

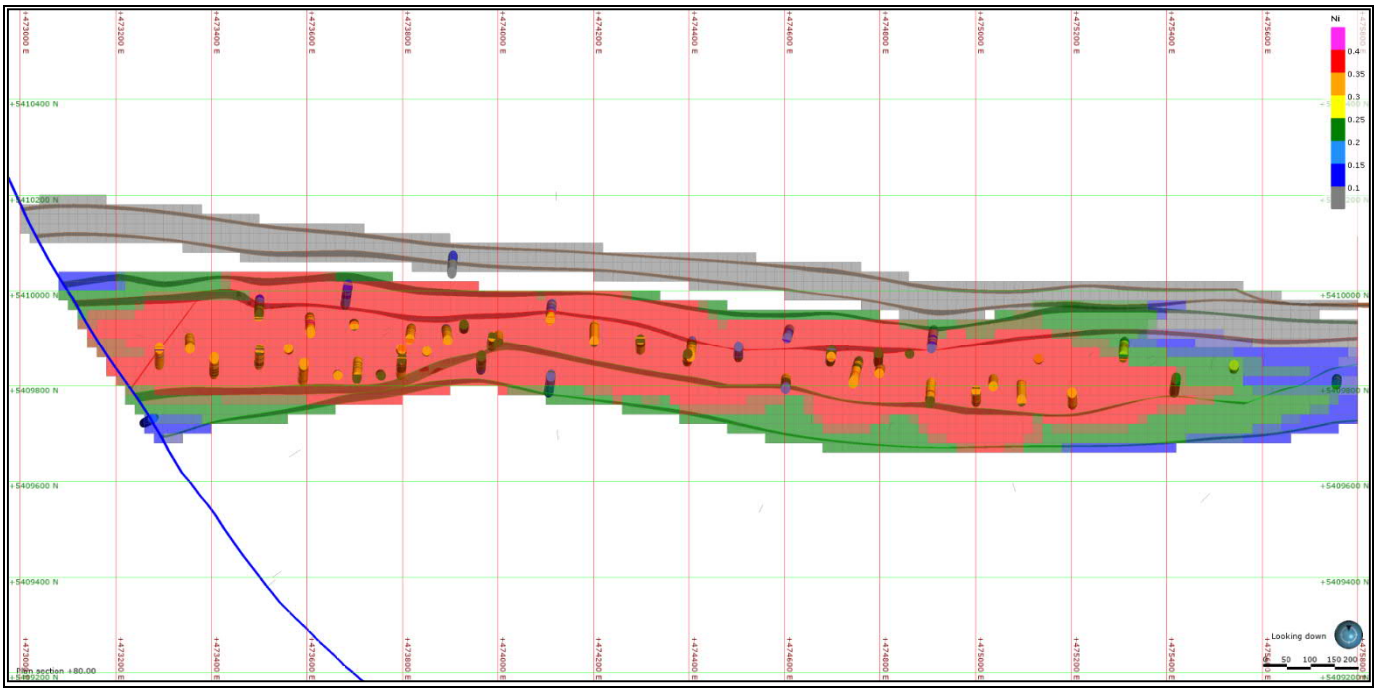
Source: Caracle Creek, 2022.

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, 2022.

14.10.2 Main Zone: Mineral Resource Classification

Similar to the East Zone, resource classification in the Main Zone consisted in evaluating block proximity to drill holes 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 all nickel composites within the general EST domain (Figure 14.46), and the block evaluation 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 drill holes to achieve Measured status and three “incomplete” drill holes for Indicated.

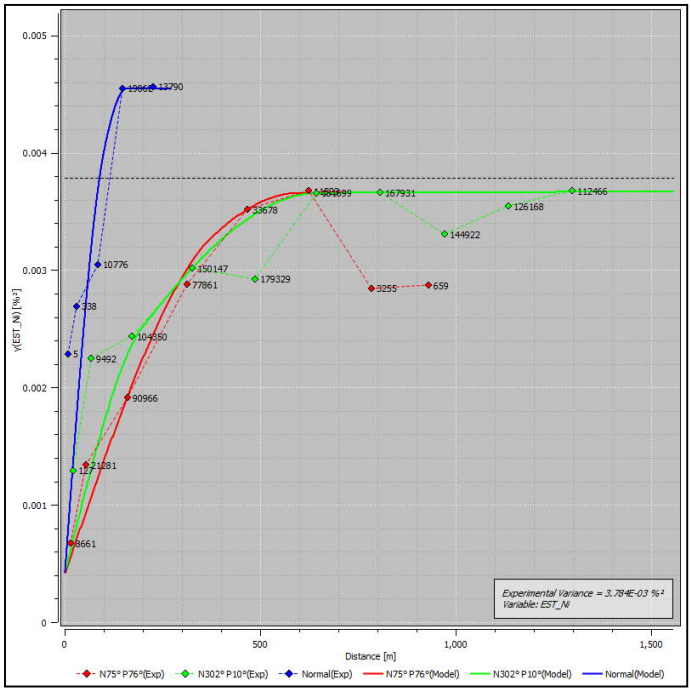
The result from the block evaluation was reviewed for any isolated blocks or group of blocks, which were assigned to the surrounding class if deemed appropriate. Smoothing was not carried out since the resulting classification was sufficiently coherent (Figure 14.47), though it is a decision that will be revisited in the future.

Table 14.10: Main Zone Neighborhood Search and Estimation Parameters for Resource Classification

Parameter	Neighborhood			
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 Drill Hole	4	4	4	4
Minimum Points per Drill Hole	-	-	-	-
Minimum Drill Holes	3	2.5	2	1
Search Radius Directions	30° 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	∞

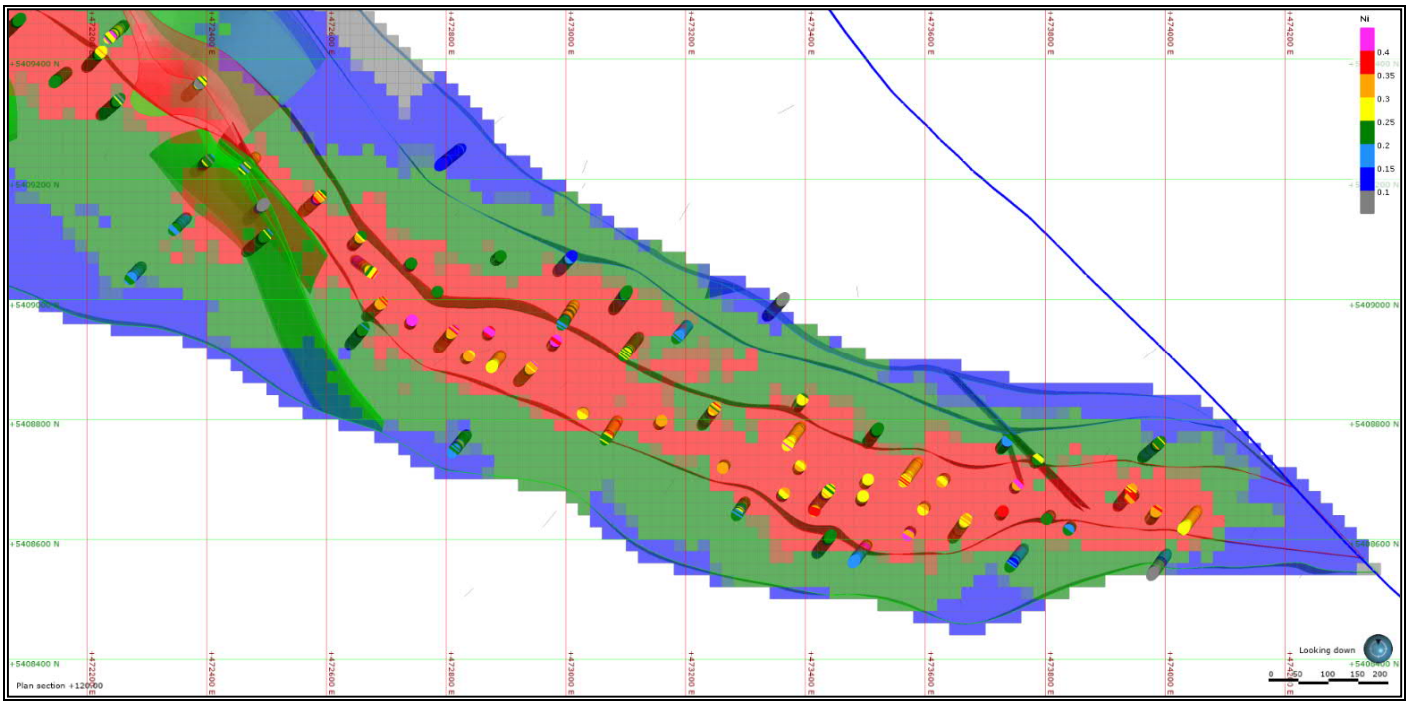
Source: Caracle Creek, 2022.

Figure 14.46: Main Zone EST-1 Domain Nickel Variogram for Resource Classification



Source: Caracle Creek, 2022.

Figure 14.47: Main Zone Plan Cross-section of Resource Classification, looking Down at 0 Elevation



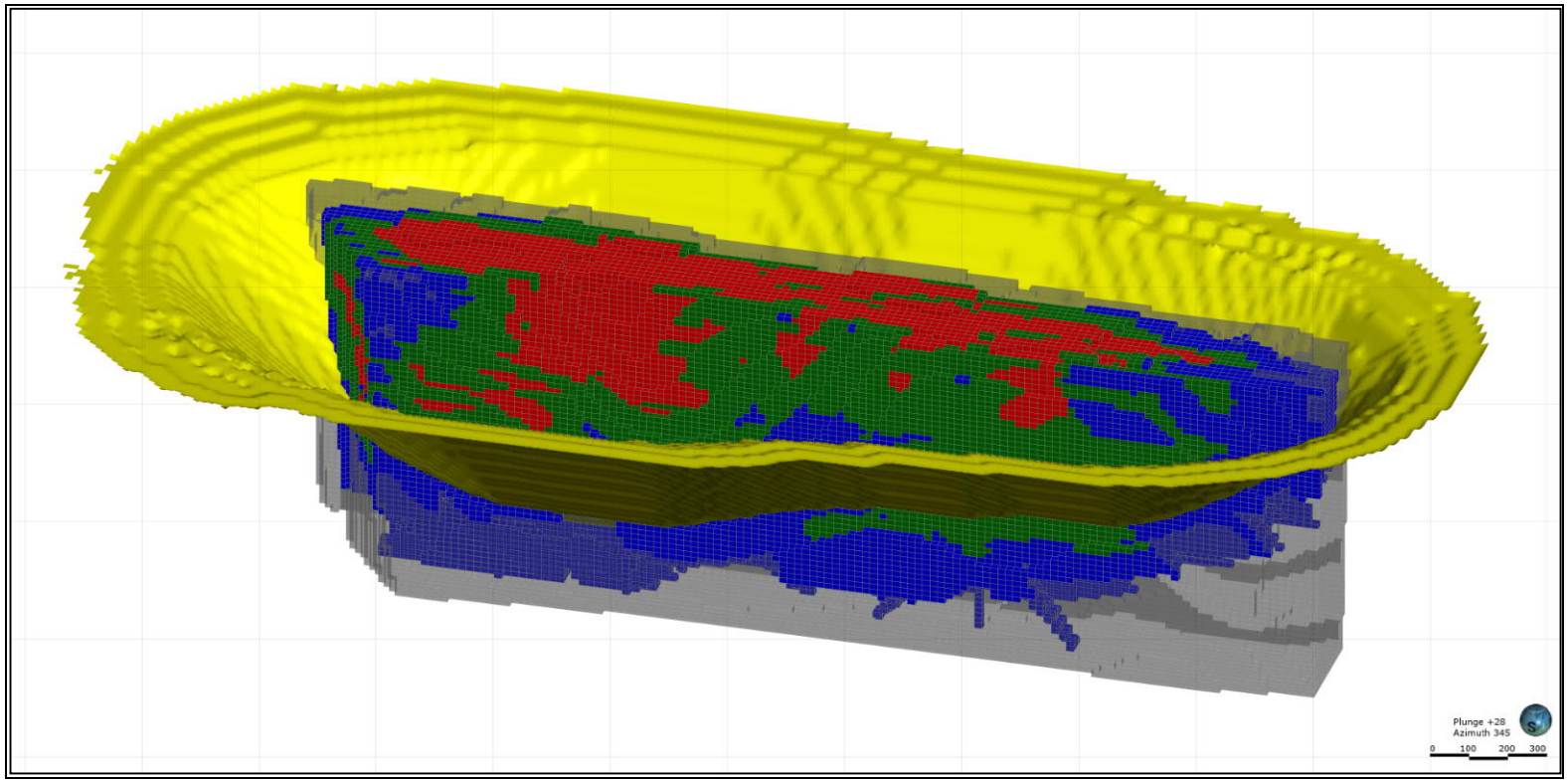
Note: Shows measured (red), indicated (green) and inferred (blue) blocks for all estimation domains. The UM wireframe is shown as a transparent purple outer shell, along with the lower- and higher-grade domains as dark blue and dark red inner shells. Source: Caracle Creek, 2022.

14.11 Pit Optimization and Cut-off Grade

According to CIM (2014) and CIM (2019), in order for a mineral deposit to be considered a mineral resource it must be shown that there are "reasonable prospects for eventual economic extraction". This requirement implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recoveries. In order to determine the quantity of mineralization that shows a "reasonable prospect for eventual economic extraction" using open pit mining methods, Independent Consultant and QP David Penswick (P.Eng.), supervised a pit optimization using Datamine NPVS, which employs the Lerchs-Grossmann (LG) algorithm. This algorithm uses the net value of every block in the model to determine the ultimate extent of an open pit that maximizes overall project value.

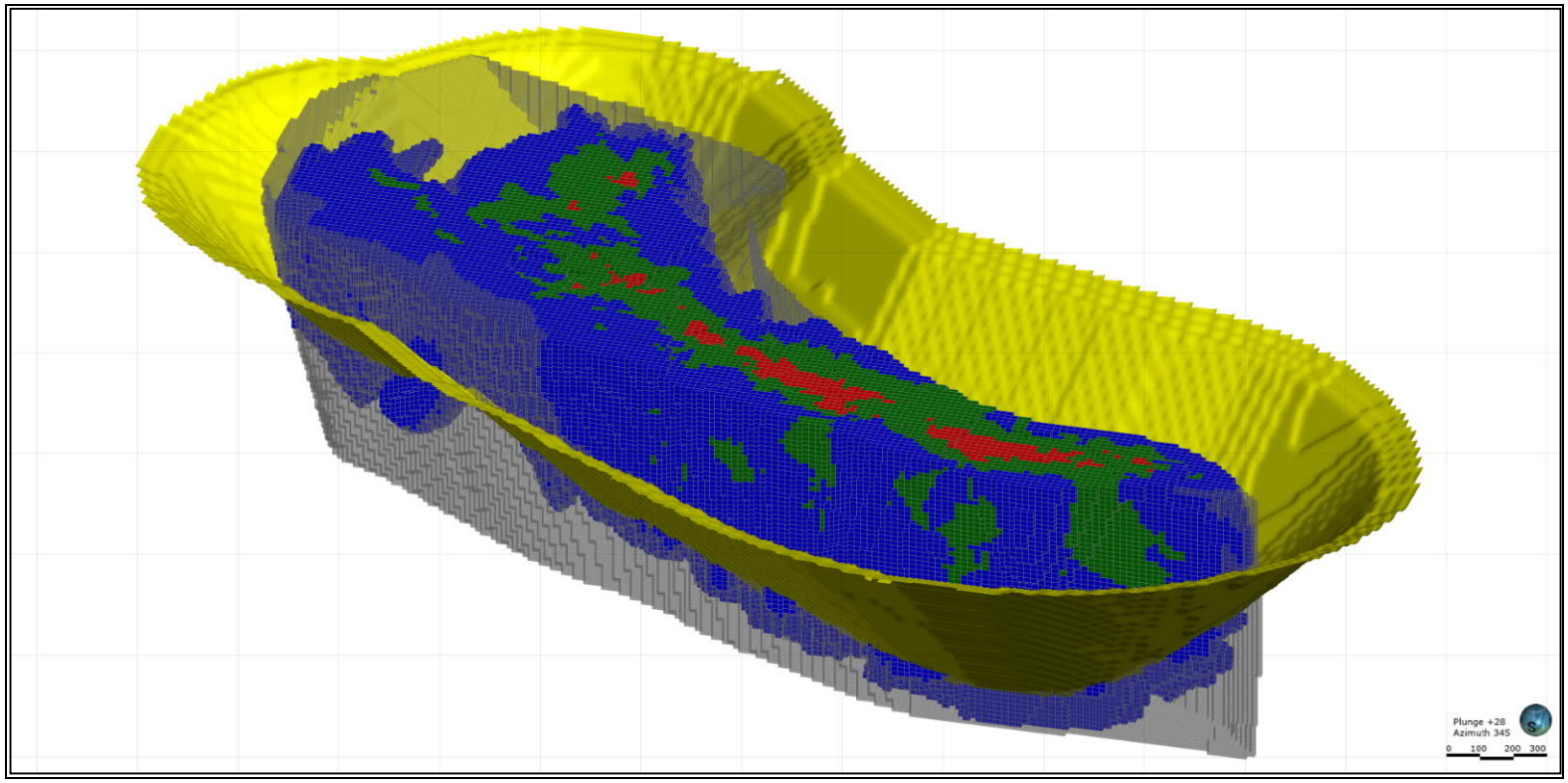
It is important to note that the results from the pit optimization exercise are used solely for testing the "reasonable prospects for eventual economic extraction" by open pit mining methods and do not represent an economic study. Figures 14.48 and 14.49 show 3D views of the generated pits around the East Zone and Main Zone Mineral Resource Estimates.

Figure 14.48: East Zone 3D View looking North-Northwest



Note: The theoretical pit surfaces are shown against measured (red), indicated (green) and inferred (blue) blocks for all estimation domains. The semi-transparent grey area represents potential resources. Source: Caracle Creek, 2022.

Figure 14.49: Main Zone 3D View Looking North-Northwest



Note: The theoretical pit surfaces are shown against measured (red), indicated (green) and inferred (blue) blocks for all estimation domains. The semi-transparent grey area represents potential resources. Source: Caracle Creek, 2022.

14.11.1 Techno-Economic Parameters

Parameters used in the pit optimization were based on consensus forecasts at the time of the optimization (for macro-economic parameters) along with the reported or forecast performance of peer operations and projects (for slope angles and unit costs).

The mineral resource estimates have been revised to include a conceptual pit envelope constraint that was developed using the following optimization parameters. Metal prices used were US$7.75/lb nickel, US$15/lb cobalt, US$200/tonne Fe recovered to magnetite, US$1,600/oz Pd, and US$800/oz Pt. Different pit slopes were used for each layer (in degrees): 9.5 in clay, 21.8 in gravel, and 45 in rock. Exchange rate utilized was US$/C$ at $0.75. Mining costs utilized different values for overburden (clay, gravel), selective mining, and bulk mining, ranging from C$1.75 to C$3.15/t mined. Processing costs and general and administration costs for a 100 kt/d operation (similar to Dumont) were C$6.18/t. Selling costs are expected to be C$0.85/lb nickel.

14.11.2 Conceptual Mining Process

As discussed in Section 16, three fleets of mining equipment ae currently contemplated:

- The first fleet would comprise small backhoe excavators matched with 40-tonne articulated trucks and be used to strip clay where the clay – gravel contact is greater than 7.5 m below surface.
- Shallower clay and a portion of the underlying gravel would be mined on 7.5m benches using medium-sized face shovel excavators matched with 90-tonne trucks.
- Deeper gravel and all rock would be mined using large, electrically powered face shovel excavators matched with autonomous and trolley-assisted 290-tonne trucks.

The cost structure for the fleets described above has been estimated as per the parameters outlined above, in Section 14.13.1.

14.11.3 Processing and Recovery Assumptions

The metallurgical recovery for Crawford has been estimated on initial mineralogy and metallurgy work based on the reported and forecast performance of benchmarks. Based on the range of grade and ratio of sulphur to nickel at Crawford, recovery could be expected to range from 10% to 60%. Given the current terms for treatment and refining, it has currently been assumed material would be roasted, resulting in higher payables for Ni but no contribution from by-products.

It may prove economically viable to smelt the concentrate produced from higher Co and PGE-content mineralization, particularly the peripheral PGE reef deposits. This would allow for a contribution from by-product non-ferrous metals.

Based on initial mineral processing work underway, and performance of benchmarks, it has been assumed that 30% to 40% of total iron would be recovered to a saleable magnetite concentrate.

14.11.4 Cost Assumptions

The estimated cost structure for mining takes account of the pits extending to a maximum ultimate depth of approximately 600 m and the anticipated ultimate scale of approximately 300 kt/d mined. The estimated cost structure for processing takes account of the difference in electricity rates for Quebec (that would be applied to Dumont) and Ontario, along with the anticipated ultimate scale of Crawford of approximately 100 kt/d milled. The estimated cost structure for G&A takes into account the total complement for, and the local taxes and insurance liabilities associated with, an operation of the scale outlined above.

14.11.5 Cut-off Grade: Calculated

Based on the techno-economic parameters described above, the marginal cut-off can be achieved with less than 1 lb of payable nickel per tonne of ore processed.

With the current techno-economic parameters, the calculated cut-off approximates 0.10% Ni and is thus lower than the 0.15% Ni cut-off that has been used as the basis for the Mineral Resource Statement (see Section 14.14). The 0.15% Ni cut-off was selected as a conservative cut-off given the early stage of the project's development.

It is QP David Penswick's opinion that the calculated cut-off grade of approximately 0.10% Ni from pit optimization is relevant to the grade distribution of this project and that the mineralization exhibits sufficient continuity for economic extraction under this cut-off value. For purposes of constraining the resource model, it was decided to conservatively adjust this value upwards to 0.15% Ni.

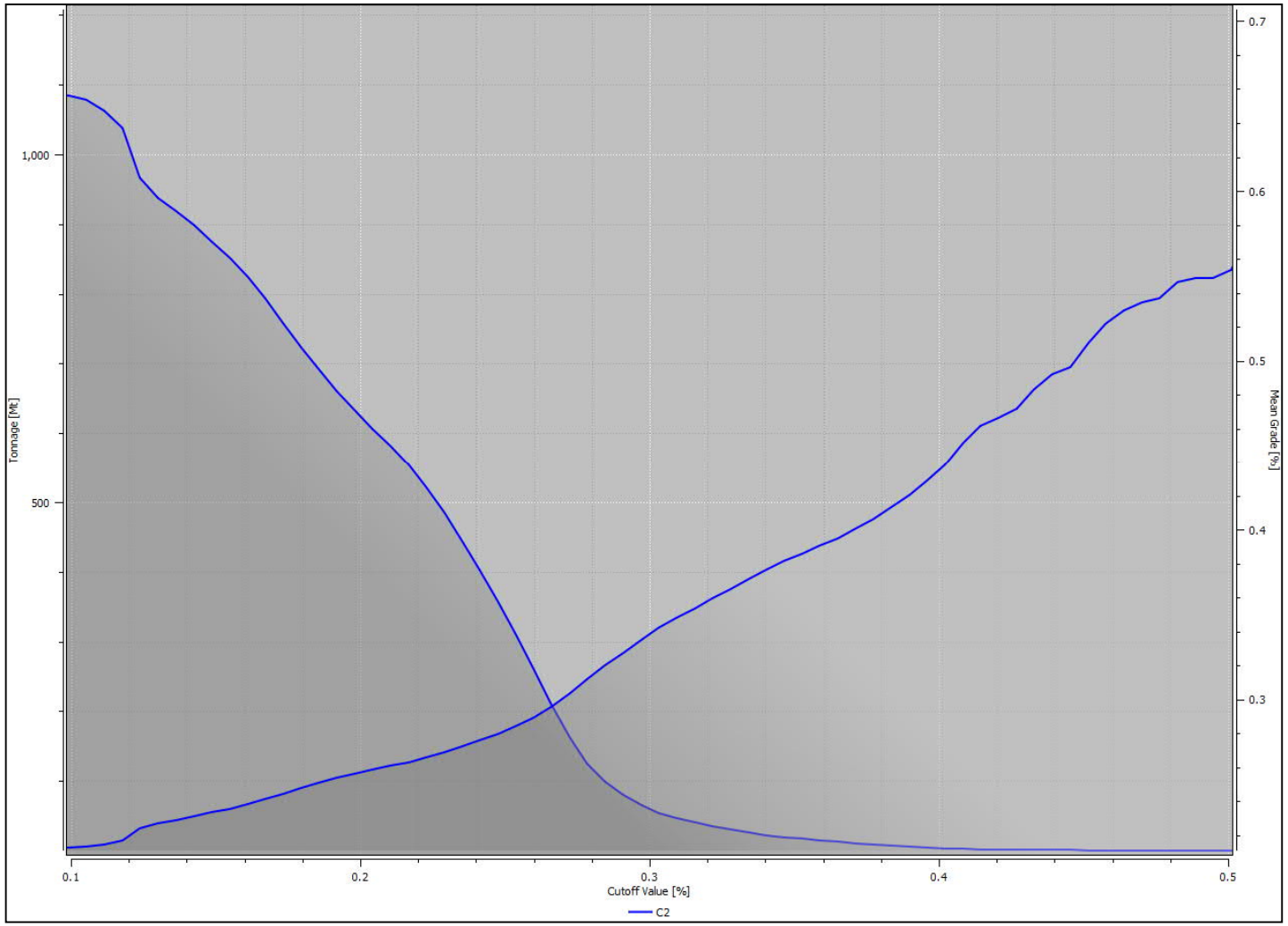
14.11.6 Grade Sensitivity Analysis: East Zone

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 14.50), marking a nickel cut-off grade of 0.266% Ni, included as a data point in the grade sensitivity analysis (Table 14.11). The reader is cautioned that the numbers presented in the following figure and table should not be misconstrued with a mineral resource statement (see Section 14.14).

14.11.7 Grade Sensitivity Analysis: Main Zone

Based on the combined block model from Section 14.10.2 (Mineral Resource Classification) and constrained by the RF100 pit, a grade-tonnage curve was calculated for the nickel domains (Figure 14.51), marking a nickel cut-off grade of 0.253% Ni, included as a data point in the grade sensitivity analysis (Table 14.12). The reader is cautioned that the numbers presented in the following figure and table should not be misconstrued with a mineral resource statement (see Section 14.14).

Figure 14.50: East Zone Grade-Tonnage Curve for Nickel Grades



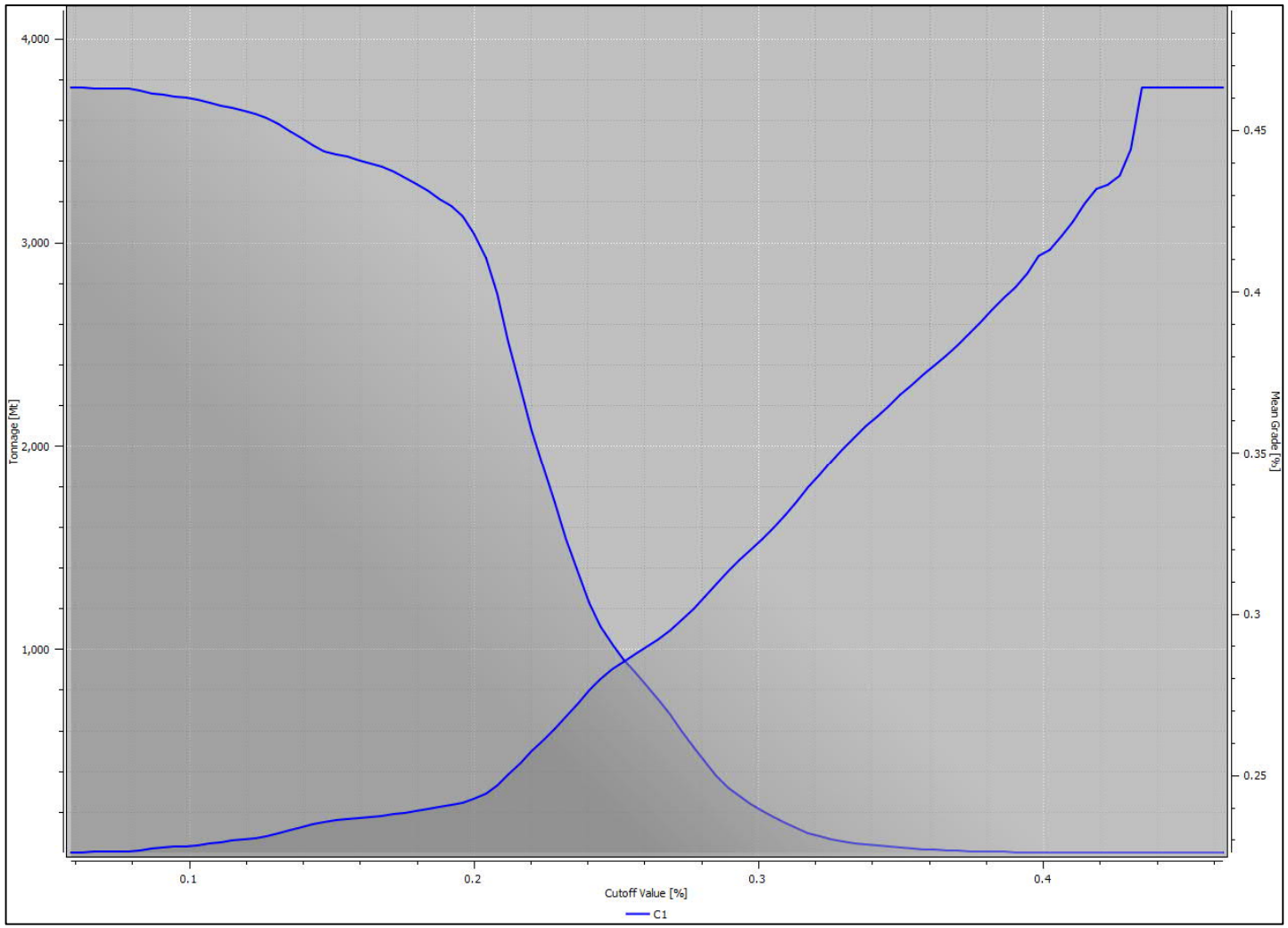
Source: Caracle Creek, 2022.

Table 14.11: East Zone Grade Sensitivity Analysis for Nickel

Cut-off	Tonnage	Ni (%)	Metal Content (t)
0.1	1,083,290,000	0.21	2,304,460
0.12	1,019,000,000	0.22	2,231,430
0.14	906,691,000	0.23	2,089,000
0.16	828,627,000	0.24	1,971,570
0.18	720,445,000	0.25	1,787,290
0.2	623,720,000	0.26	1,603,790
0.22	538,538,000	0.26	1,424,590
0.24	412,657,000	0.27	1,134,650
0.26	260,816,000	0.29	754,695
0.266	210,505,000	0.30	622,378
0.28	117,058,000	0.31	367,847
0.3	59,877,700	0.34	202,799
0.32	36,349,300	0.36	130,144
0.34	22,341,500	0.38	84,019
0.36	14,066,800	0.39	55,105
0.38	7,732,920	0.41	31,658
0.4	3,477,820	0.43	15,108

Source: Caracle Creek, 2022

Figure 14.51: Main Zone Grade-Tonnage Curve for Nickel Grades



Source: Caracle Creek, 2022.

Table 14.12: Main Zone Grade Sensitivity Analysis for Nickel

Cut-off	Tonnage	Ni (%)	Metal Content (t)
0.1	3,708,960,000	0.23	8,460,080
0.12	3,641,640,000	0.23	8,385,780
0.14	3,507,730,000	0.23	8,209,460
0.16	3,401,590,000	0.24	8,052,360
0.18	3,288,690,000	0.24	7,858,890
0.2	3,048,330,000	0.24	7,397,770
0.22	2,098,800,000	0.26	5,394,130
0.24	1,249,370,000	0.28	3,444,120
0.253	944,533,000	0.29	2,695,070
0.26	834,244,000	0.29	2,412,300
0.28	469,369,000	0.30	1,426,710
0.3	217,195,000	0.32	699,279
0.32	86,745,100	0.34	296,226
0.34	36,317,800	0.36	130,709
0.36	14,843,200	0.38	55,783
0.38	4,398,730	0.39	17,246
0.4	791,311	0.41	3,259

Source: Caracle Creek, 2022

14.12 Mineral Resource Statement

The measured ("Mea"), indicated ("Ind") and inferred ("Inf") mineral resource estimates presented herein are constrained within pit shells developed for the Main and East zones from the pit optimization analysis discussed above. The effective date of the mineral resource estimate is May 19, 2022.

14.12.1 East Zone: Mineral Resource Estimate

Pit constrained, class-characterized mineral resources with respect to the higher and lower-grade nickel estimation domains within the East Zone are presented for all elements studied in Table 14.13.

14.12.2 Main Zone: Mineral Resource Estimate

Pit constrained, class-characterized mineral resources with respect to the higher and lower-grade nickel estimation domains within the Main Zone are presented for all elements studied in Table 14.14.

Table 14.13: Summary of the Pit-Constrained Updated East Zone Mineral Resource Estimate

Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
Higher-Grade	Measured	212.5	0.26	555.5	0.013	26.7
	Indicated	242.7	0.26	630.4	0.013	31.3
	Mea+Ind	455.1	0.26	1,185.9	0.013	58.1
	Inferred	104.6	0.26	271.7	0.013	13.3
Lower-Grade	Measured	72.9	0.19	135.8	0.013	9.2
	Indicated	172.9	0.18	313.4	0.013	21.9
	Mea+Ind	245.8	0.18	449.2	0.013	31.1
	Inferred	40.2	0.19	75.3	0.013	5.1
Domain	Class	Tonnes (Mt)	Fe (%)	Fe Content (Mt)	S (%)	S Content (kt)
Higher-Grade	Measured	212.5	5.99	12.7	0.06	122.3
	Indicated	242.7	6.13	14.9	0.08	188.2
	Mea+Ind	455.1	6.06	27.6	0.07	310.5
	Inferred	104.6	6.16	6.4	0.06	66.1
Lower-Grade	Measured	72.9	7.15	5.2	0.03	25.1
	Indicated	172.9	7.35	12.7	0.03	48.5
	Mea+Ind	245.8	7.29	17.9	0.03	73.6
	Inferred	40.2	7.32	2.9	0.03	11.1
Domain	Class	Tonnes (Mt)	Cr (%)	Cr Content (kt)		
Higher-Grade	Measured	212.5	0.64	1,368.1		
	Indicated	242.7	0.64	1,551.1		
	Mea+Ind	455.1	0.64	2,919.2		
	Inferred	104.6	0.64	668.3		
Lower-Grade	Measured	72.9	0.57	412.8		
	Indicated	172.9	0.54	939.4		
	Mea+Ind	245.8	0.55	1,352.2		
	Inferred	40.2	0.55	221.9		
Domain	Class	Tonnes (Mt)	Pd (g/t)	Pd Content (koz)	Pt (g/t)	Pt Content (koz)
Higher-Grade	Measured	212.5	0.014	93.3	0.009	59.2
	Indicated	242.7	0.015	116.8	0.009	68.6
	Mea+Ind	455.1	0.014	210.1	0.009	127.7
	Inferred	104.6	0.014	45.9	0.008	25.7

Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
Lower-Grade	Measured	72.9	0.008	18.4	0.006	14.9
	Indicated	172.9	0.007	41.1	0.006	34.8
	Mea+Ind	245.8	0.008	59.4	0.006	49.8
	Inferred	40.2	0.008	9.9	0.006	7.7
SUMMARY						
Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
Total Grade	Mea+Ind	701.0	0.23	1,635.1	0.013	89.1
	Inferred	144.8	0.24	347.0	0.013	18.4
Domain	Class	Tonnes (Mt)	Fe (%)	Fe Content (Mt)	S (%)	S Content (kt)
Total Grade	Mea+Ind	701.0	6.49	45.5	0.05	384.0
	Inferred	144.8	6.48	9.4	0.05	77.3
Domain	Class	Tonnes (Mt)	Cr (%)	Cr Content (kt)		
Total Grade	Mea+Ind	701.0	0.61	4,271.4		
	Inferred	144.8	0.61	890.1		
Domain	Class	Tonnes (Mt)	Pd (g/t)	Pd Content (koz)	Pt (g/t)	Pt Content (koz)
Total Grade	Mea+Ind	701.0	0.012	269.6	0.008	177.5
	Inferred	144.8	0.012	55.8	0.007	33.4

Source: Caracle Creek, 2022

Table 14.14: Summary of the Pit-Constrained Updated Main Zone Mineral Resource Estimate

Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
Higher-Grade	Measured	159.0	0.31	493.5	0.013	20.6
	Indicated	135.6	0.30	402.3	0.013	17.6
	Mea+Ind	294.5	0.30	895.8	0.013	38.2
	Inferred	128.7	0.28	366.9	0.013	17.0
Lower-Grade	Measured	92.1	0.23	213.2	0.013	12.0
	Indicated	337.5	0.22	732.2	0.013	43.9
	Mea+Ind	429.6	0.22	945.3	0.013	56.0
	Inferred	396.5	0.21	839.3	0.013	51.3
Domain	Class	Tonnes (Mt)	Fe (%)	Fe Content (Mt)	S (%)	S Content (kt)
Higher-Grade	Measured	159.0	6.31	10.0	0.17	265.8
	Indicated	135.6	6.55	8.9	0.13	182.9
	Mea+Ind	294.5	6.42	18.9	0.15	448.7
	Inferred	128.7	6.78	8.7	0.10	134.0
Lower-Grade	Measured	92.1	6.78	6.2	0.05	42.7
	Indicated	337.5	6.87	23.2	0.04	142.1
	Mea+Ind	429.6	6.85	29.4	0.04	184.8
	Inferred	396.5	7.01	27.8	0.05	190.7
Domain	Class	Tonnes (Mt)	Cr (%)	Cr Content (kt)		
Higher-Grade	Measured	159.0	0.59	942.9		
	Indicated	135.6	0.57	771.7		
	Mea+Ind	294.5	0.58	1,714.6		
	Inferred	128.7	0.55	708.5		
Lower-Grade	Measured	92.1	0.61	564.2		
	Indicated	337.5	0.59	1,990.9		
	Mea+Ind	429.6	0.59	2,555.0		
	Inferred	396.5	0.58	2,290.7		
Domain	Class	Tonnes (Mt)	Pd (g/t)	Pd Content (koz)	Pt (g/t)	Pt Content (koz)
Higher-Grade	Measured	159.0	0.027	136.0	0.011	56.8
	Indicated	135.6	0.024	103.1	0.012	50.6

Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
	Mea+Ind	294.5	0.025	239.1	0.011	107.4
	Inferred	128.7	0.017	70.1	0.011	46.7
Lower-Grade	Measured	92.1	0.012	35.5	0.009	26.7
	Indicated	337.5	0.010	113.0	0.008	83.6
	Mea+Ind	429.6	0.011	148.5	0.008	110.3
	Inferred	396.5	0.012	149.4	0.009	109.7
SUMMARY						
Domain	Class	Tonnes (Mt)	Ni (%)	Ni Content (kt)	Co (%)	Co Content (kt)
Total Grade	Mea+Ind	724.1	0.25	1,841.2	0.013	94.1
	Inferred	525.3	0.23	1,206.2	0.013	68.3
Domain	Class	Tonnes (Mt)	Fe (%)	Fe Content (Mt)	S (%)	S Content (kt)
Total Grade	Mea+Ind	724.1	6.68	48.4	0.09	633.5
	Inferred	525.3	6.95	36.5	0.06	324.8
Domain	Class	Tonnes (Mt)	Cr (%)	Cr Content (kt)		
Total Grade	Mea+Ind	724.1	0.59	4,269.6		
	Inferred	525.3	0.57	2,999.2		
Domain	Class	Tonnes (Mt)	Pd (g/t)	Pd Content (koz)	Pt (g/t)	Pt Content (koz)
Total Grade	Mea+Ind	724.1	0.017	387.6	0.009	217.7
	Inferred	525.3	0.013	219.5	0.009	156.4

Source: Caracle Creek, 2022

14.13 Exploration Targets

Despite having been quantified by the same methodologies used for classified resources, as has been thoroughly described in the report, tonnages and grades of exploration targets are conceptual in nature. Insufficient geological and sampling data prevents the definition of a mineral resource, and as such it is uncertain if further exploration will confirm the calculations presented in this section, or if the targets will be effectively delineated as mineral resources.

Tonnage and grade ranges for PGE (palladium and platinum) exploration targets from the Main Zone PGE reef domain are presented in Table 14.15 and from the East Zone's two PGE reef domains in Table 14.16. It's important to note that these figures were not updated in this MRE, and instead come from a previous calculation.

Table 14.15: Ranges for Exploration Targets in the Main Zone PGE Reef Domain

Domain	Tonnes (Mt)	Pd (g/t)	Pt (g/t)	PGE (g/t)
PGE	5 – 6	0.4 – 0.5	0.5 – 0.6	1.0 – 1.1

Source: Caracle Creek, 2020

Table 14.16: Ranges for Exploration Targets in the Two East Zone PGE Reef Domains

Domain	Tonnes (Mt)	Pd (g/t)	Pt (g/t)	PGE (g/t)
PGE-1	8 – 12	0.5 – 0.6	0.6 – 0.7	1.1 – 1.3
PGE-2	9 – 13	0.1 – 0.2	0.2 – 0.3	0.3 – 0.5
Total	17 – 25	0.3 – 0.4	0.4 – 0.5	0.7 – 0.9

Source: Caracle Creek, 2020

15 MINERAL RESERVE ESTIMATES

This section is not applicable.

16 MINING METHODS

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

16.1 Hydrology and Hydrogeological Considerations

The proposed mine development will cover an area that is drained by West Buskegau River and North Driftwood River, which are located to the east and west of the site, respectively. All streams and rivers in the area are part of the Hudson's Bay watershed, which they reach via the Abitibi River and Moose River drainage basins.

There are no major waterbodies in or around the site footprint. A surface water management system will be constructed to direct flows around the main surface infrastructure to avoid interaction between surface runoff and contact water, including mine water and stockpile seepage. The water management system will also be designed to redirect surface water away from the open pits.

Contact water associated with the mining process will be contained within a closed system and re-used or treated prior to discharge.

A groundwater model has not yet been constructed. 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

A geotechnical analysis of the various lithologies that will be encountered at Crawford has yet to be completed and design slopes for the pit and various impoundments are based on that achieved and/or planned at similar type operations and projects in the Abitibi Region. Slopes planned for the pit include:

- Pit walls located in clay will have an overall angle of 6H:1V (9.5°).
- Pit walls located in gravel will have an overall angle of 2.5H:1V (21.8°).
- Pit walls located in rock will have an overall angle of 1H:1V (45°).

Slopes planned for impoundments are described below.

- The overburden impoundments, comprised of mixed clay and gravel, will be constructed on 5 m lifts. Slopes facing the pit will have an overall slope of 6H:1V (9.5°) while the slope of other faces will be 3H:1V (18.4°).
- The rock impoundments, including both waste rock and the temporary low-grade stockpiles, will be constructed on 10 m lifts. Slopes facing the pit will have an overall slope of 6H:1V (9.5°) while the slope of other faces will be 3H:1V (18.4°)

16.3 Open Pit Mine Plan

16.3.1 Introduction

The Crawford mine plan currently includes three discrete open pits. The largest and first to be mined is the Main Zone, comprising 77% of the total mined tonnage, 80% of feed to the mill and associated contained nickel, and 85% of the recoverable nickel. Immediately north of the Main Zone is the East Zone, which is divided by a saddle into East Zone-West (EZ-W) and East Zone-East (EZ-E). EZ-W is the larger of the two, comprising 16% of the total tonnage to be mined, 14% of feed to the mill and associated contained nickel, and 11% of the recoverable nickel. Mining of the EZ-W will begin in the last year of Main Zone operation. EZ-E will be mined last, starting 33 months after EZ-W.

All waste generated while the Main Zone pit is active will be impounded on the surface. Tailings from the material processed through the mill will be stored in the tailings storage facility (TSF) located to the east of the pits. This facility will be constructed in two phases. The starter phase will use a downstream construction method to its ultimate 36 m height, utilizing waste rock and gravel from the open pit. The much larger ultimate phase will also use the downstream construction method to a height of 49 m before converting to an upstream method utilizing mainly rehandled tailings for the final 24 m. Gravel that is not used to construct the TSF as well as clay will be impounded in two overburden storage facilities (OBS-S and OBS-E) to the south and east of the Main Zone. Waste rock not used in constructing the TSF will be impounded in a waste rock storage facility to the north of the East Zone (WRS-N). The Main Zone will be mined at a rate faster than is required to satisfy the mill, allowing for accelerated delivery of higher value material to the mill while lower value material will be temporarily stockpiled in one of three low-grade stockpiles. LG1 will be comprised of highest value material and located closest to the mill. Next highest value material will be stored in LG2. Both LG1 and LG2 will be located within the footprint of the future EZ-E pit. At the time that EZ-E starts up, both these stockpiles will have been completely reclaimed, with only LG3 to the north of the East Zone remaining. Waste from the EZ-W will initially be impounded in surface facilities. As soon as operations in the Main Zone cease, all remaining waste from EZ-W and EZ-E (including tailings produced by the mill) will be impounded in the mined-out Main Zone pit. Ultimately, the Main Zone void will be 55% filled by the various waste stored within it.

The pits will be mined with a mixed fleet of equipment. A portion of the deepest clays will be loaded with small backhoe excavators and hauled using 40 t articulated trucks. The remaining clay and a portion of the underlying gravel will be loaded with medium-sized face shovel excavators and hauled using 90 t manually operated haul trucks. The remaining gravel and all the underlying waste rock will be loaded using large-face shovel excavators and 290 t haul trucks that utilize an autonomous haulage system (AHS) and have no onboard operator. Starting in Year 2 of operations, trolley assist will be employed to reduce the diesel consumed while increasing the speed of trucks on uphill hauls.

None of the clay or gravel will require drilling and blasting. Rock will be drilled and blasted as follows:

- Where material is homogenous (*i.e.*, adjacent blocks are either completely waste rock or lower value mineralization or higher value mineralization with no interstitial blocks of a different type) it will be blasted in bulk, using blast holes measuring 311 mm on 15 m benches.

- Where material is heterogenous—with one or more SMU blocks of waste rock mixed with mineralization or lower value mineralization mixed with higher value—it will be blasted selectively, using 227 mm holes on 7.5 m flitches.

The mine will have an electrical supply system to feed the electrical equipment, including the large excavators, drills, trolley-assisted haul trucks and dewatering pumps.

The following steps were performed to achieve the current design and schedule:

- 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 ($NPV_{8\%}$), although internal rate of return (IRR) and cash-flow index (CFI; the ratio of NPV to maximum at-risk investment) were also considered
- The Lerchs-Grossmann (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 inter-ramp angles and ramps of sufficient width for 290 t haul trucks using trolley-assist.
- A generic engineered schedule was developed for the optimal sequence and optimized using the techno-economic model.
- A cut-off value was calculated based on the forecast costs of the operation and confirmed through testing of alternatives.

Mining recovery and planned dilution are based on the difference between the inventory of mineralization above cut-off within the LG optimized pit and the engineered design. Each of the above steps is discussed in more detail in the following sections.

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
- 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 more than 10,000 unique schedules for which key production, cost and valuation metrics were recorded. These were then ranked, allowing the optimal scenario to be identified.

16.3.3 Pit Optimization

LG optimization is an industry-standard tool used to define the limits of and sequence for mining an open pit. The design process is initiated by calculating the net value of each block in the model, by subtracting estimated costs for mining, processing, and G&A from the net smelter return (NSR) of each block. Waste blocks with no NSR value have a negative net value. Table 16.1 summarizes key cost assumptions used in generating the net value of blocks.

Table 16.1: Cost Assumptions used in Crawford LG Optimization

Item	units	Base[1]	MCAF[2]	Total
Clay Mining	C$/t mined	C$2.85	C$0.10	
Gravel Mining	C$/t mined	C$1.94	C$0.08	
Selective Rock Mining	C$/t mined	C$1.82	C$0.06	
Bulk Rock Mining	C$/t mined	C$1.16	C$0.02	
Processing	C$/t milled			C$5.67
G&A	C$/t milled			C$0.51

Notes: 1. Base Cost of mining at surface. 2 Mining Cost Adjustment Factor for each 15m bench below surface.

Selective mining of rock only applies to the Main Zone, where there is a higher density of resource drilling that allowed use of a 10 m (X) x 10 m (Y) x 5 m (Z) SMU. These would normally be aggregated into 20 m (X) x 20 m (Y) x 15 m (Z) 'bulk blocks'. However, where one or more of the 12 individual cells combining to form a bulk block was different from the remainder (*i.e.*, waste vs. low-value mill feed or low-value mill feed vs. high-value mill feed), then the cells would be mined selectively (to avoid dilution) using smaller and higher unit cost equipment. The concept for equipment at the time the LG optimization was performed was to use the same 300 t excavators and 90 t trucks planned for mining gravel for selective rock, while bulk rock would be mined with rope shovels and 290 t trucks that utilized AHS and trolley assist. This concept explains the vast difference in costs between selective and bulk mining. That mining concept since evolved to the current, which is to use 700 t excavators and 290 t trucks for both bulk and selective material. The only material difference between the two is the drilling equipment, cost of explosives accessories, and bench heights that would be employed. As a result, the mining costs used in the LG optimization are more conservative than that of the current design.

Table 16.2 summarizes revenue assumptions used in the LG optimization. At this stage, it has been assumed that Crawford will produce two concentrates, as follows:

- Nickel will be recovered to concentrates with low grades of byproduct cobalt and platinum group elements (PGE), including palladium and platinum. The most economic treatment route for this concentrate is roasting. With roasting, no byproduct credits are realized.

- Iron and chromium will be recovered to an iron concentrate. At the time of the LG optimization, chromium had not been included in the block model and the values for this concentrate are based solely on the iron content.

Table 16.2: Revenue Assumptions used in Crawford LG Optimization

Metal	Price (US$)	Payability
Nickel	$7.75/lb	91.5%
Cobalt	$15.00/lb	0.0%
Palladium	$1,600/oz	0.0%
Platinum	$800/oz	0.0%
Iron	$200/t	100%
Chrome	n/a	n/a

Notes:1 Payability for iron units within magnetite concentrate (*i.e.*, $100/t for concentrate grading 50% Fe).

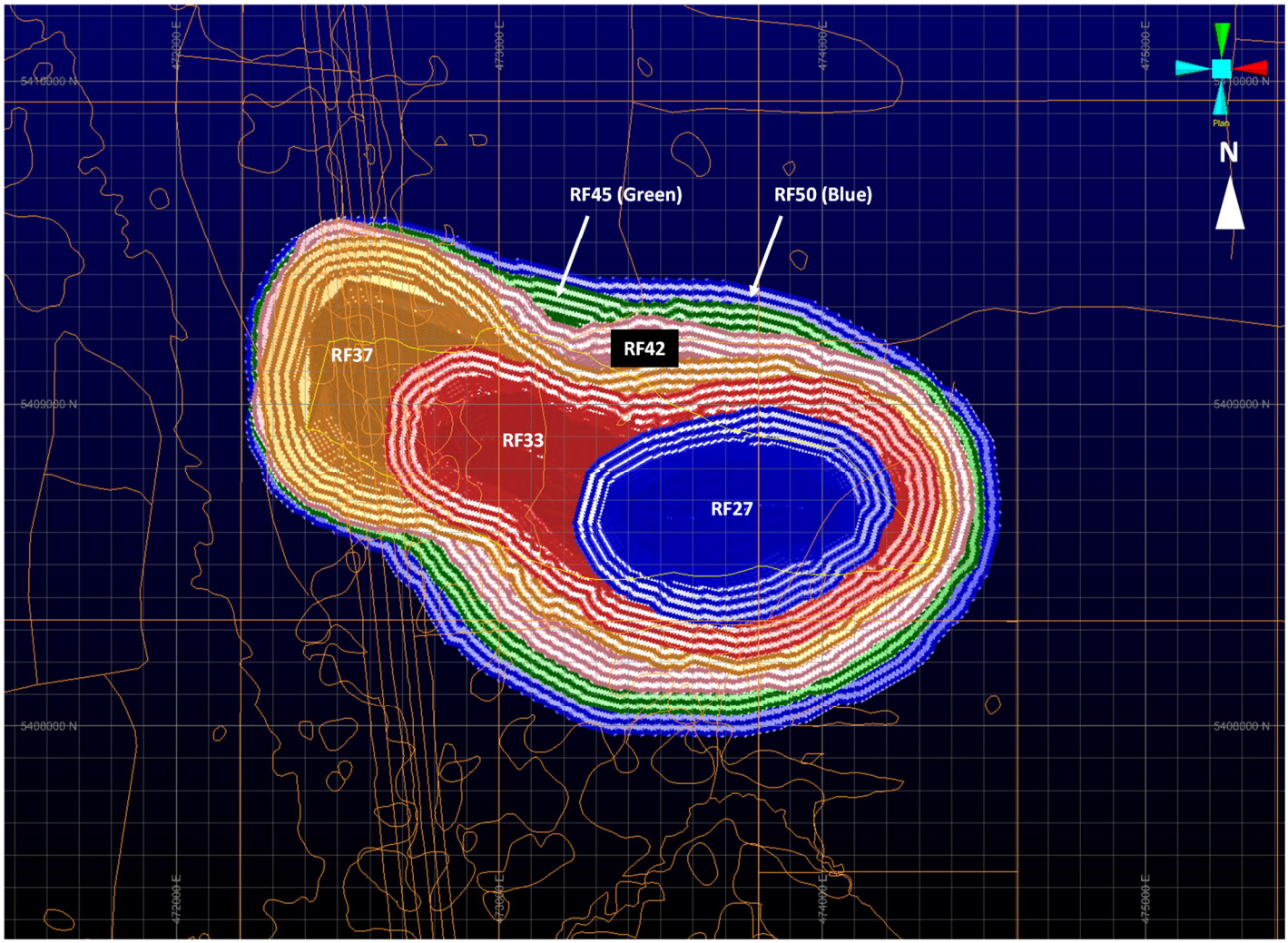
It should be noted that there are discrete zones of higher PGE grades (~1 g/t 2E PGE) in both the Main Zone and East Zone. In the event this material were milled separately, likely through a second much smaller process facility, it could yield a nickel

concentrate with sufficiently high by-product credits as to justify conventional treatment through a smelter and refinery. This option has not been considered in this PEA but will be evaluated during the next stage of study.

Slope angles used in the LG optimization have been reported previously, in Section 16.2 (6H:1V for clay, 2.5H:1V for gravel and 45° for rock).

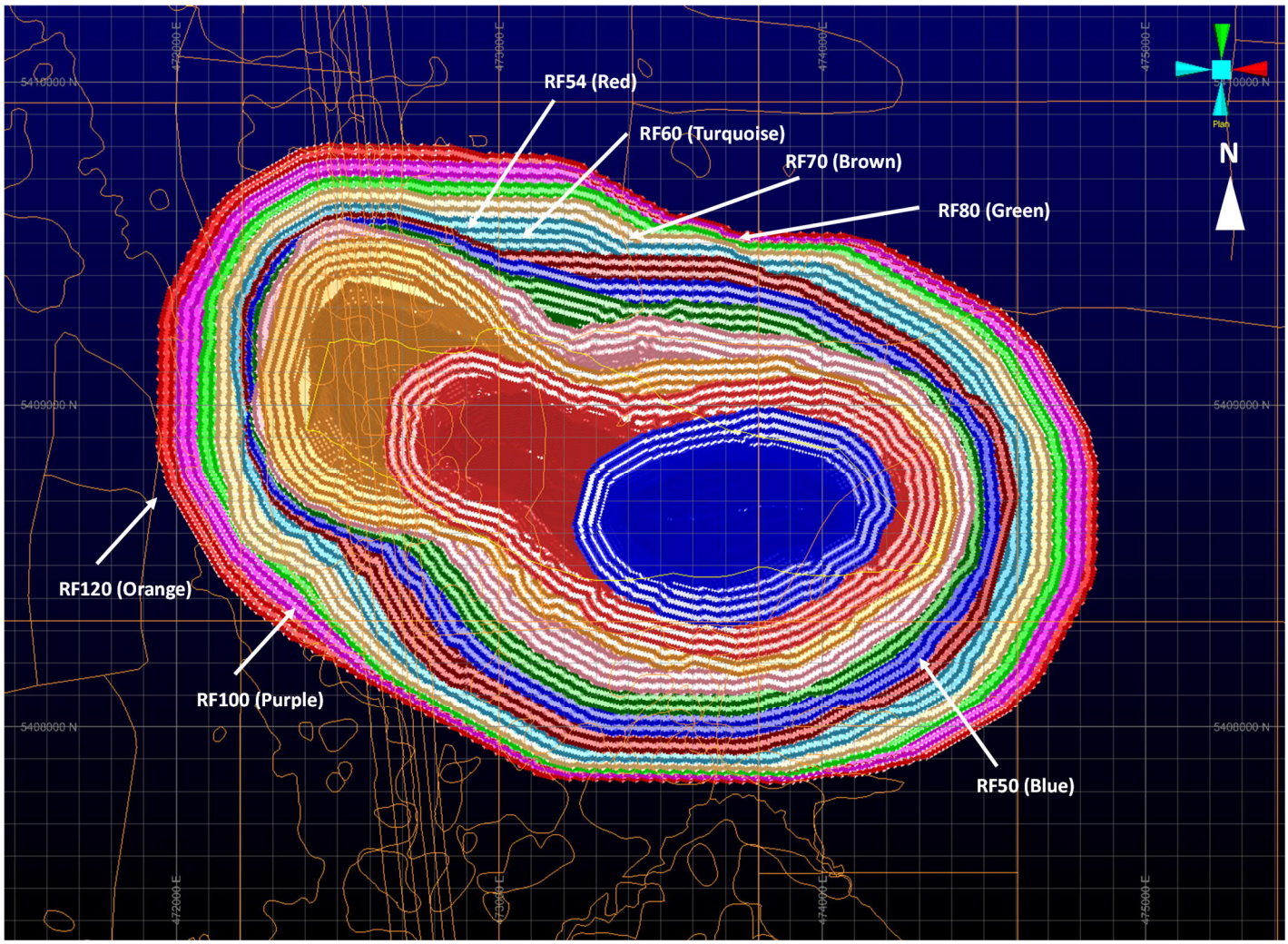
To maximize NPV, a "cone" of ore and associated waste stripping was generated. Varying the metal price generated higher-value nested cones, enabling identification of the optimal development sequence. The smallest shell in the Main Zone was generated with a revenue factor (RF) of 27% or $2.09/lb Ni. Nested shells were generated for each subsequent 1% increase in revenue factor. As it was possible that the cost structure for the resultant mine design would be less than that assumed for the LG optimization, and/or that the evaluation price would be revised upwards, revenue factors up to RF120 were considered. As can be seen in Figures 16.1 and 16.2, the Main Zone initially progresses in a westerly manner from the starter pit located centrally in the east. Thereafter, the pit expands concentrically.

Figure 16.1: Main Zone LG Optimization Progression (Pt. 1)



Source: Penswick, 2021.

Figure 16.2: Main Zone LG Optimization Progression (Pt. 2)

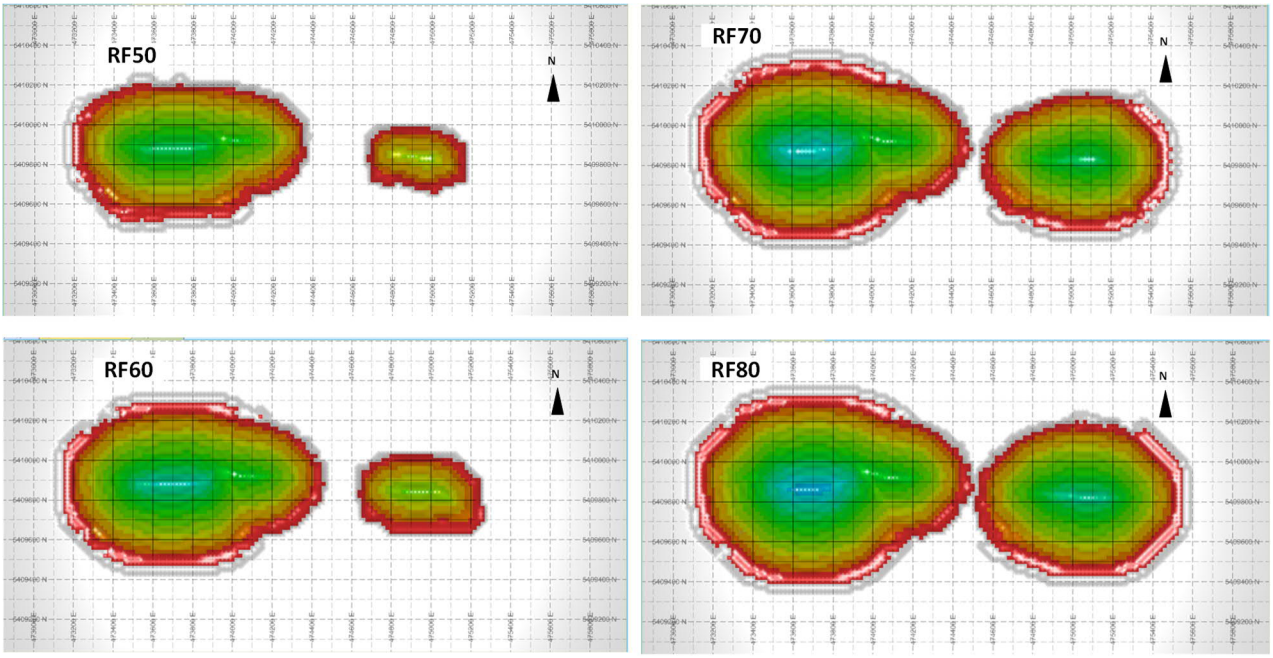


Source: Penswick, 2021.

Figure 16-3 illustrates the manner in which the East Zone progresses, with the two lobes expanding concentrically. Note that the lobes ultimately overlap at RF120.

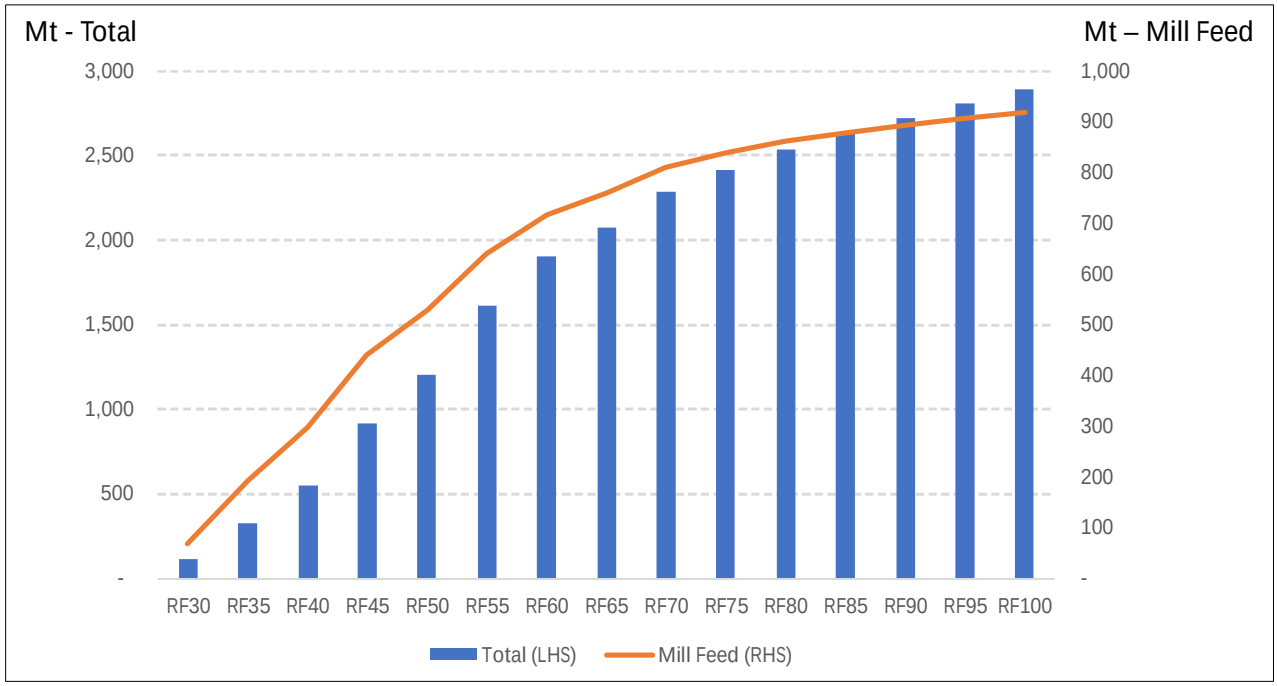
Figures 16.4 and 16.5 summarize the total tonnage mined and tonnage of mill feed by revenue factor increment for the Main Zone and East Zone, respectively. By RF65, the Main Zone captures 83% of total mill feed in 72% of the total tonnage to be mined. Thereafter, increases in mill feed become progressively more marginal as required total mining increases non-linearly. For the East Zone, the 80% threshold of total mill feed is achieved later, at RF70.

Figure 16.3: East Zone LG Optimization Progression



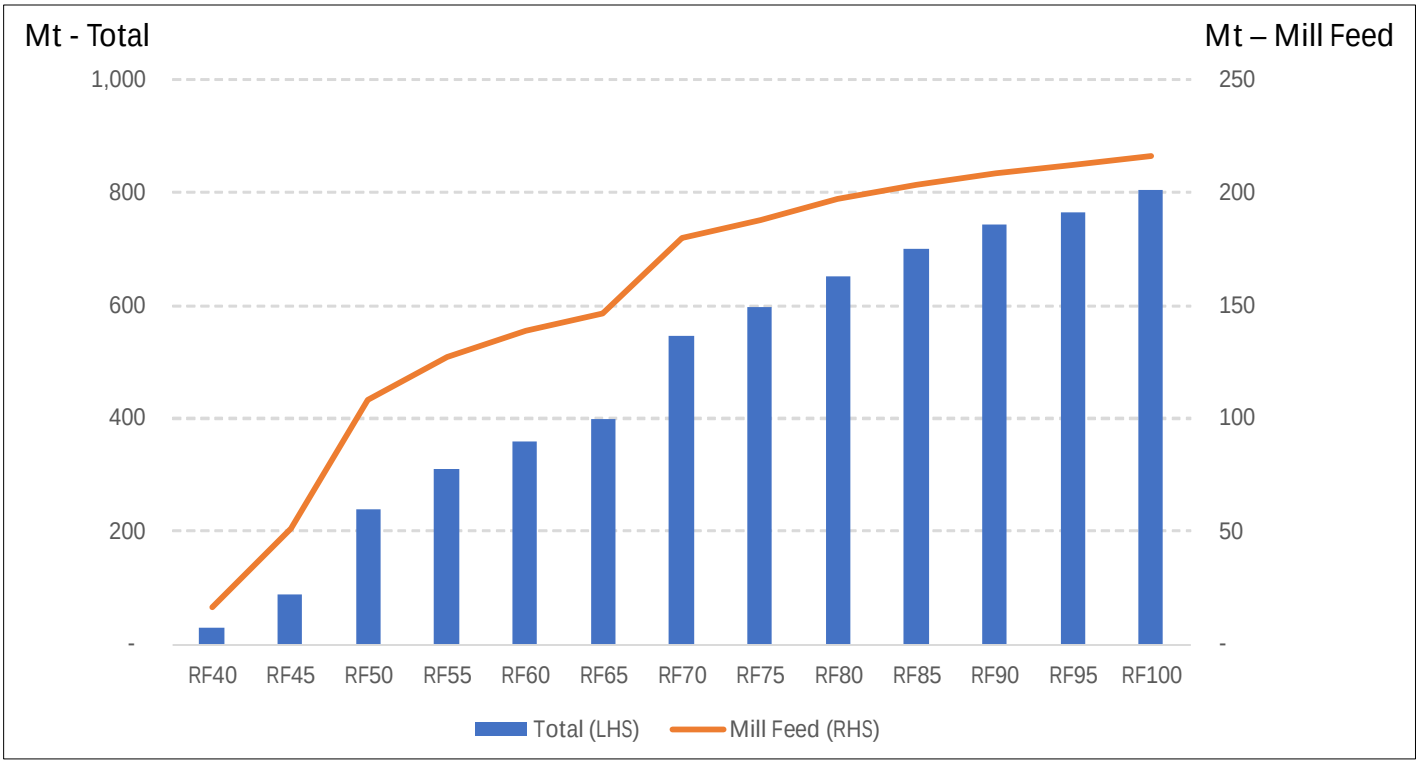
Source: Penswick, 2021.

Figure 16.4: Main Zone LG Optimization – Total Mined and Mill Feed by RF



Source: Penswick, 2021.

Figure 16.5: East Zone LG Optimization – Total Mined and Mill Feed by RF



Source: Penswick, 2021.

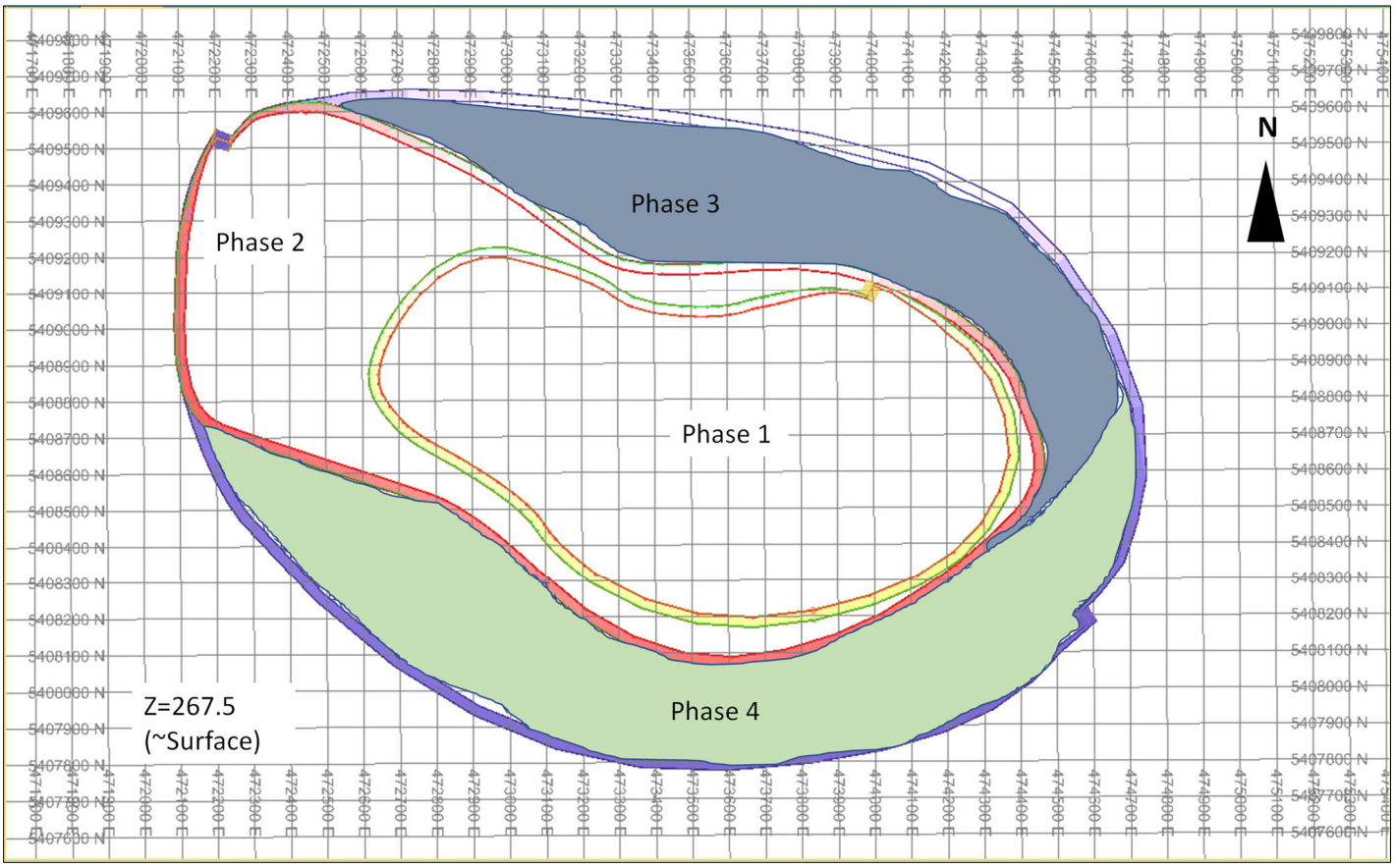
16.3.4 Phase Design

Nested shells generated by the LG optimization were aggregated to form stages. These were then tested iteratively with the techno-economic model. It was found that returns were maximized by limiting the Main Zone pit to RF60 while following the sequence described below and shown in Figure 16-6:

- Phase 1 targets the highest value mineralization, located centrally and to the east.
- Phase 2 extends to the western limits of the zone.
- Phase 3 extends along the entire strike length to the northern limits.
- Phase 3 is a final pushback along the entire strike length to the southern limits.

For the East Zone, returns were maximized by limiting the pits to RF70. Each of the discrete lobes will be mined in a single phase, with EZ-W preceding EZ-E.

Figure 16.6: Plan View of Main Zone Phases



Source: Penswick, 2021.

16.3.5 Pit Design

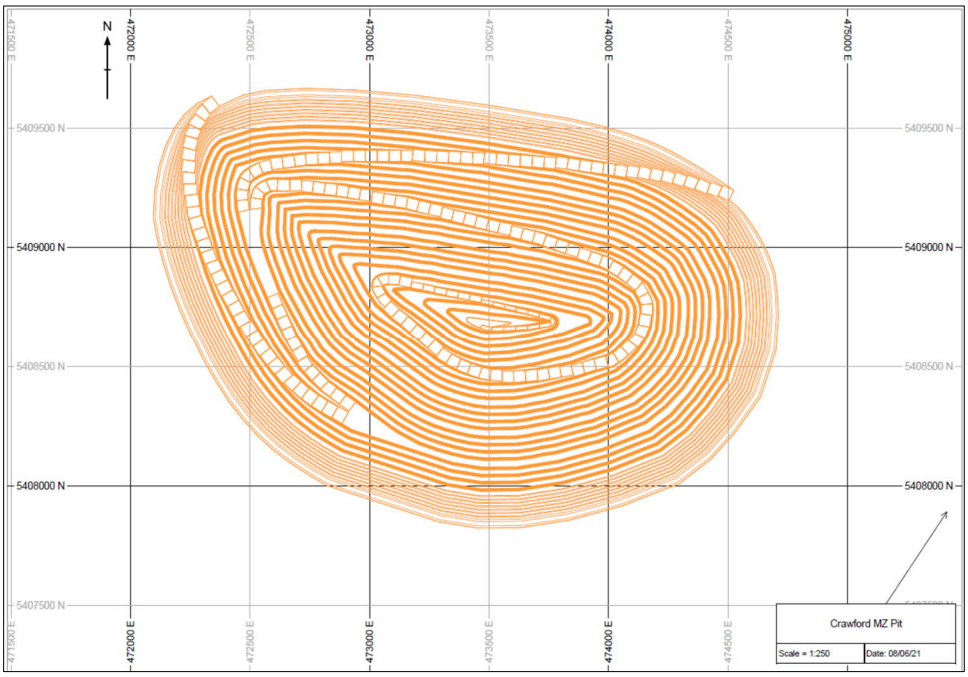
Pit shells generated using the LG algorithm represent a theoretical design and, while the final walls honour a best estimate of slopes taking account of ramps, the shell cannot be considered practical because ramps have not been located. The engineered design provided for ramps measuring up to 54 m (refer to Section 16.5.2 for details).

The engineered design also translates the overall slope angles used by LG into bench face angles and berms as follows:

- Clay (6H:1V / 9.5°): 5 m bench height with a 30° face angle and 25 m berm
- Gravel (2.5H:1 / 22°): 5 m bench height with a 60° face angle and 10 m berm
- Rock (1H:1 / 22°): 30 m bench height (double benching) with a 70° face angle and 19 m berm

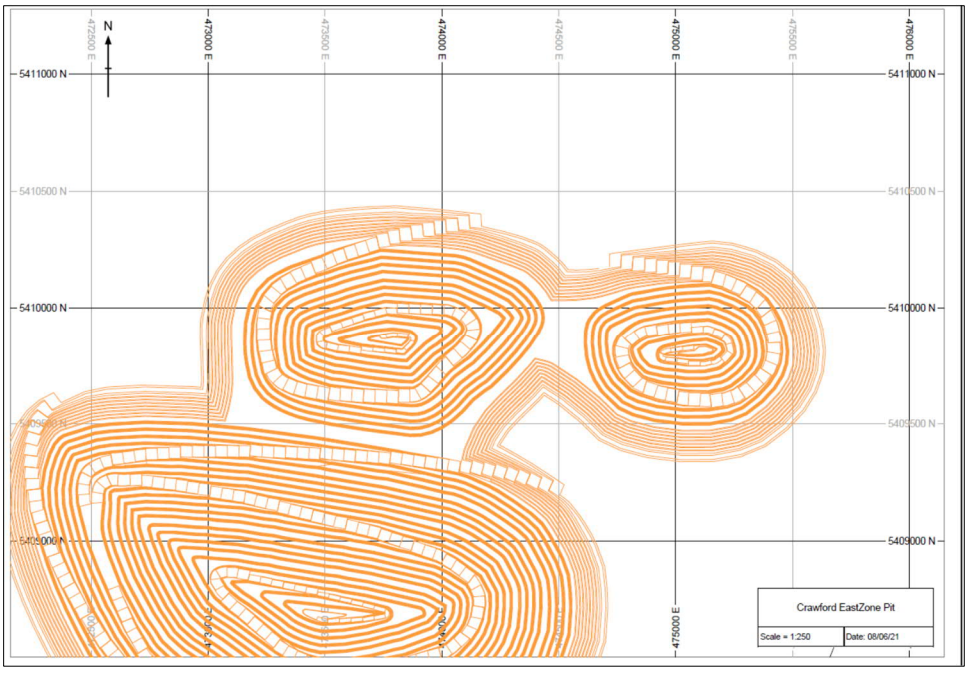
Figures 16.7 and 16.8 provide isometric views of the engineered Main Zone and East Zone ultimate pits, while Table 16.3 compares the content of the engineered designs with the LG shells on which they are based.

Figure 16.7: Engineered Main Zone Ultimate Pit



Source: Penswick, 2021.

Figure 16.8: Engineered East Zone Ultimate Pit



Source: Penswick, 2021.

Table 16.3: Comparison of LG and Engineered Designs

Pit	Design	Mt		Grades			Con'd Metal		
		Waste	Mill Feed	% Ni	% Fe	% NiEq	Ni (Mlbs)	Fe (kt)	NiEq (Mlbs)
Main Zone	LG (RF60)	1,184	716	0.255	6.60	0.333	4,035	47,269	5,255
	Engineered	1,428	723	0.251	6.50	0.327	3,995	46,998	5,208
	% Recovery						99%	99%	99%
	dilution		2%						
East Zone	LG (RF70)	365	180	0.248	6.44	0.323	983	11,590	1,282
	Engineered	459	184	0.242	6.53	0.318	980	12,008	1,290
	% Recovery						100%	104%	101%
	dilution		2%						
Combined	LG	1,548	896	0.254	6.57	0.331	5,018	58,859	6,536
	Engineered	1,887	907	0.249	6.51	0.325	4,975	59,006	6,498
	% Recovery						99%	100%	99%
	dilution		2%						

It can be seen that the engineered designs honour mill feed contained within the LG shells, with an aggregate planned mining recovery of 99% and dilution of just 2%. However, this was achieved at the expense of a 22% increase in waste stripping, as a result of the inclusion of ramps in the design while adopting a conservative approach of maintaining a 45° inter-ramp slope angle in rock.

During the feasibility study, recommended inter-ramp slope angles will be provided from geotechnical analysis. For the LG optimization, these recommended slopes will be flattened to account for the impact of ramps.

16.3.6 Mining Schedule

A generic schedule was developed using mine planning software and was based on the annual targets that had been found the yield optimal results to date. This schedule was then refined using the techno-economic model, by increasing or relaxing the targets for specific periods. A key element of the scheduling process was the allocation of mill feed to one of ten bins based on NSR value as follows (in Canadian dollars):

- Bin1 = ≥$7 and <$12 and totals 16.7 Mt.
- Bin2 = ≥$12 and <$16 and totals 29.0 Mt.
- Bin3 = ≥$16 and <$20 and totals 98.7 Mt.
- Bin4 = ≥$20 and <$23 and totals 172.4 Mt.

- Bin5 = ≥$23 and <$26 and totals 170.9 Mt.
- Bin6 = ≥$26 and <$28 and totals 60.4 Mt.
- Bin7 = ≥$28 and <$30 and totals 54.9 Mt.
- Bin8 = ≥$30 and <$35 and totals 105.4 Mt.
- Bin9 = ≥$35 and <$40 and totals 86.8 Mt.
- Bin10 = ≥$40 and totals 111.8 Mt.

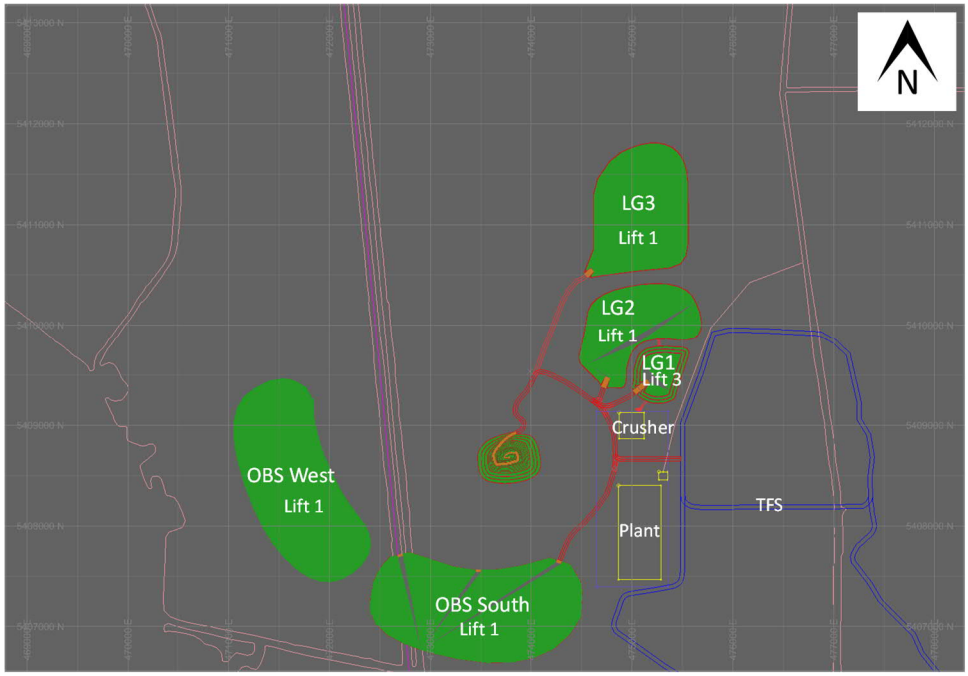
Bin1 to Bin4 will be directed to the lowest grade stockpile, LG3, when the mill is adequately fed from other higher value bins. Bin5 to Bin8 will be directed to LG2, and Bin9 to Bin10 to LG1. The stockpiles will be reclaimed ahead of lower value ex-pit material (*i.e.*, material in LG1 would be fed to the mill in priority of Bin8 coming from the pit). Consequently, in some periods, material from LG1 or LG2 will be reclaimed at the same time as ex-pit material is delivered to lower value stockpiles.

Key constraints that were honoured during refinements to the schedule included the following:

- A minimum pre-strip of 40 Mt was applied to ensure sufficient waste rock and gravel for construction of infrastructure such as roads and the TSF. While increasing the tonnage of pre-trip beyond 40 Mt would increase the average value of material milled in the initial year of mill operation (as additional higher value material will have been mined and stockpiled during the pre-strip period), the benefits of this did not offset the associated increase in initial capital costs.
- A maximum of 450 Mm^3 tailings can be accommodated within the surficial TSF capital. Thereafter, the Main Zone pit must be depleted and available to accept the impoundment of all subsequent tailings. Timing of depletion of the Main Zone is a function of its mining rate relative to the mill throughput; increasing the rate at which the Main Zone is mined accelerates delivery of higher value material to the mill along with making in-pit impoundment becoming available sooner (and costs associated with TSF construction reduced), but at the cost of additional rehandle and the capital cost of additional fleet. The optimal mining rate for Main Zone was found to result in the delivery of 409 Mm^3 to the surficial TSF.
- Start-up of EZ-E must wait until the higher-grade stockpiles LG1 and LG2 are depleted. This can be achieved by either delaying the start of the zone and/or adjusting the bins of material directed to LG1 and LG2. The optimal approach was found to be the bins described above and start-up of EZ-E 33 months after the start-up of EZ-W.

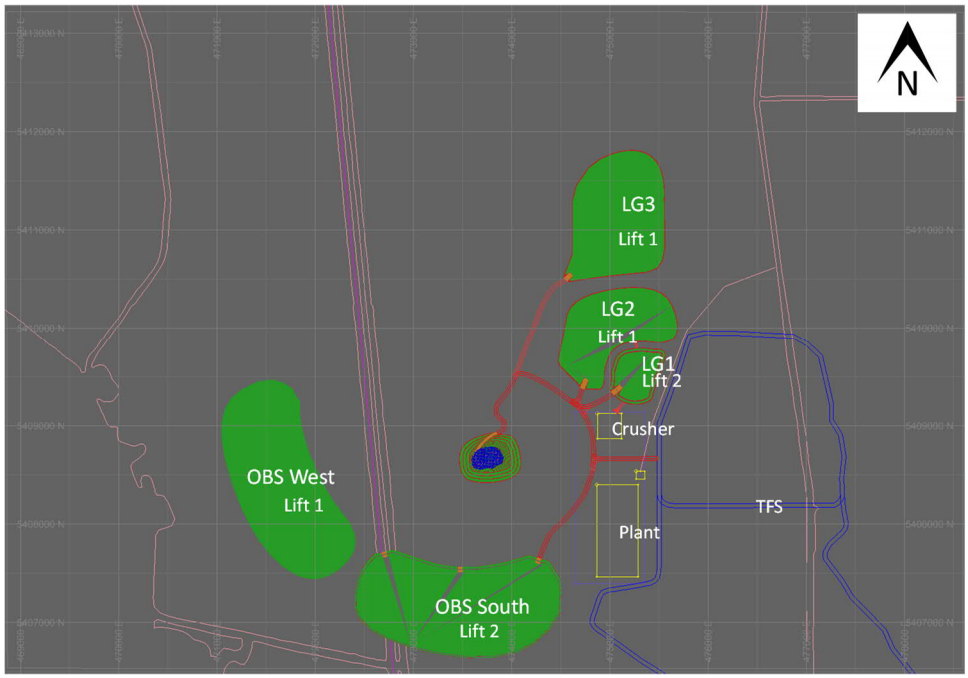
Figures 16.9 to 16.18 depict stages of the mine development, including the associated face positions of the various impoundments. Figure 16-19 illustrates the LOM production.

Figure 16.9: End of Pre-Strip



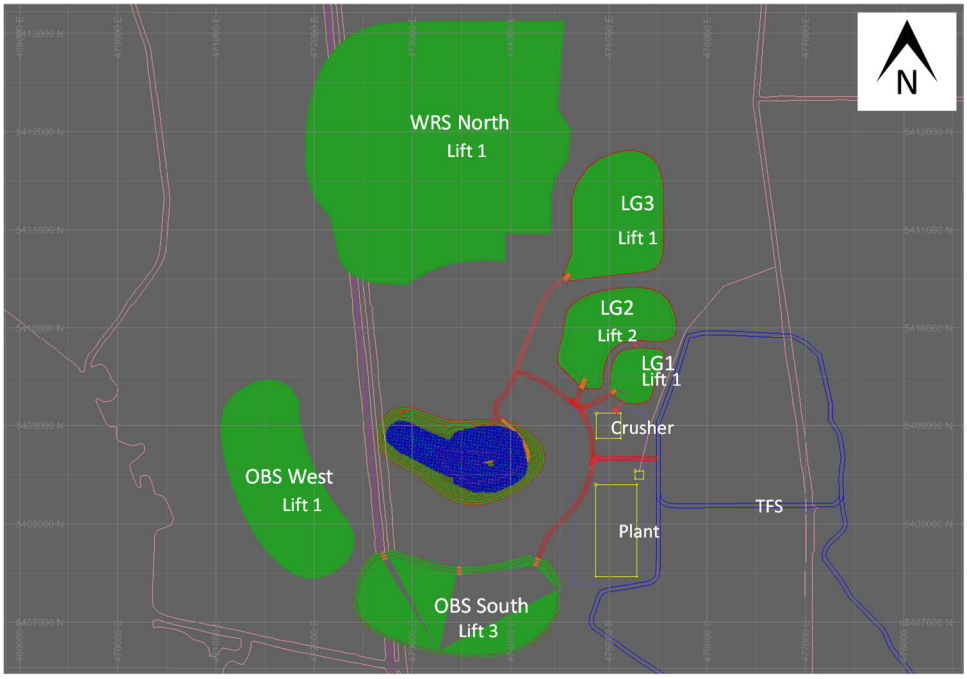
Source: Penswick, 2021.

Figure 16.10: End of Year 1



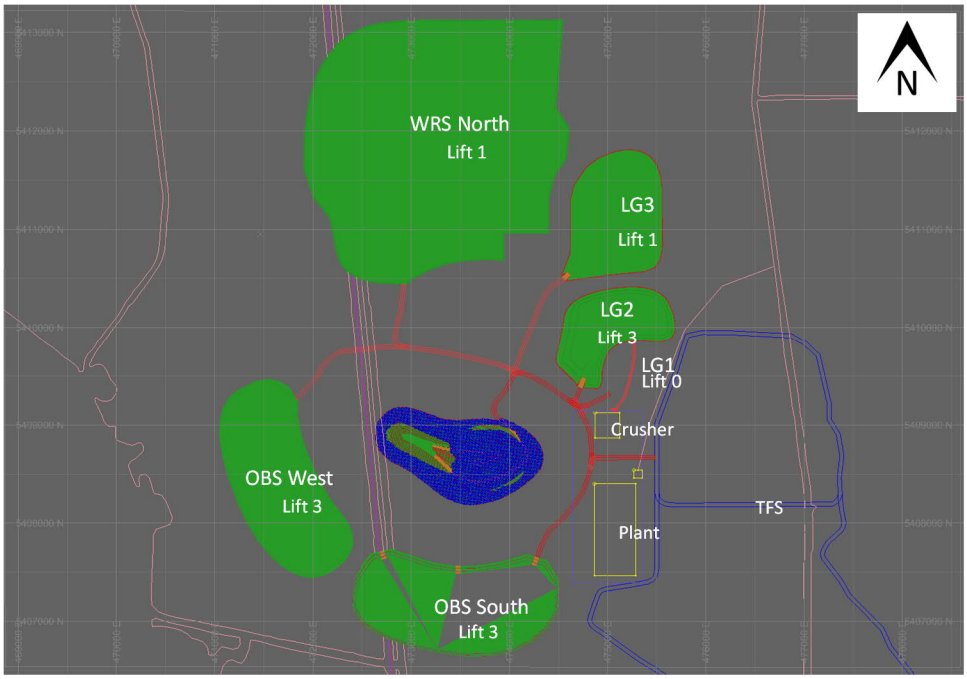
Source: Penswick, 2021.

Figure 16.11: End of Year 2



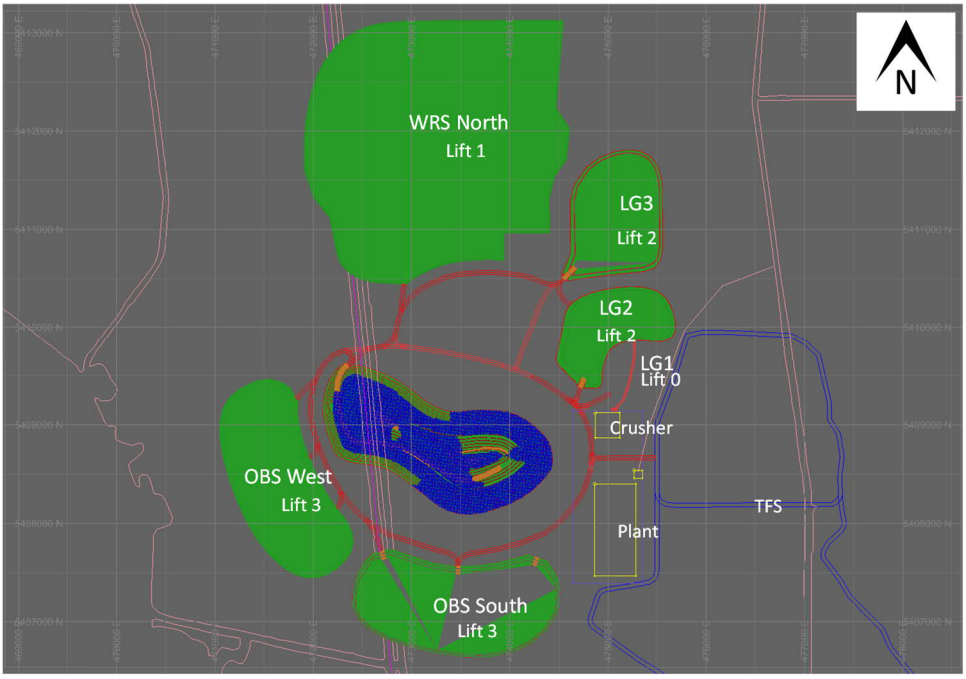
Source: Penswick, 2021.

Figure 16.12: End of Year 3



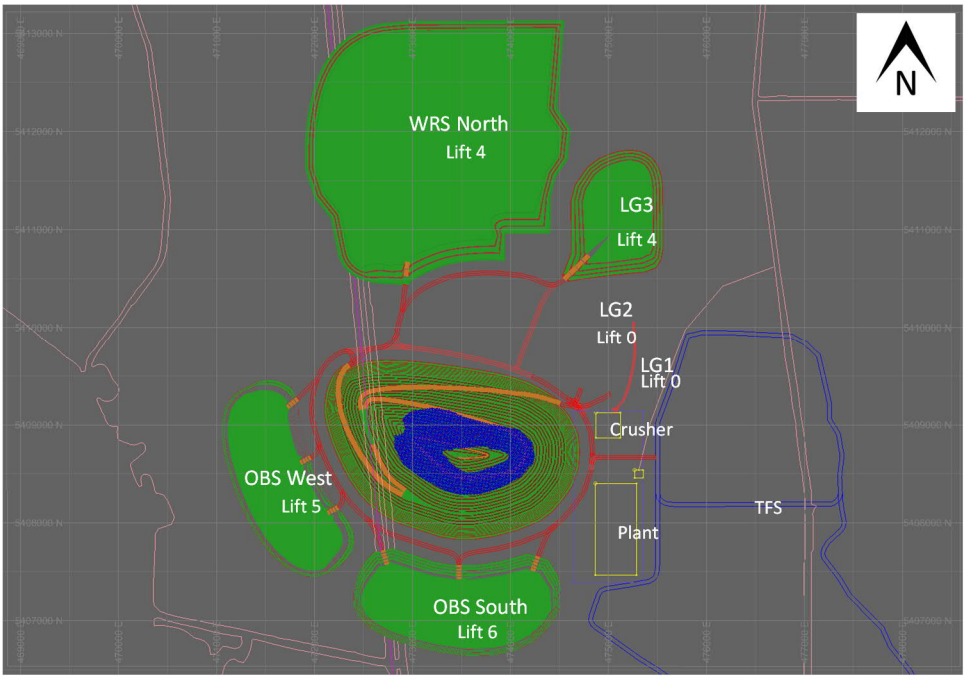
Source: Penswick, 2021.

Figure 16.13: End of Year 5



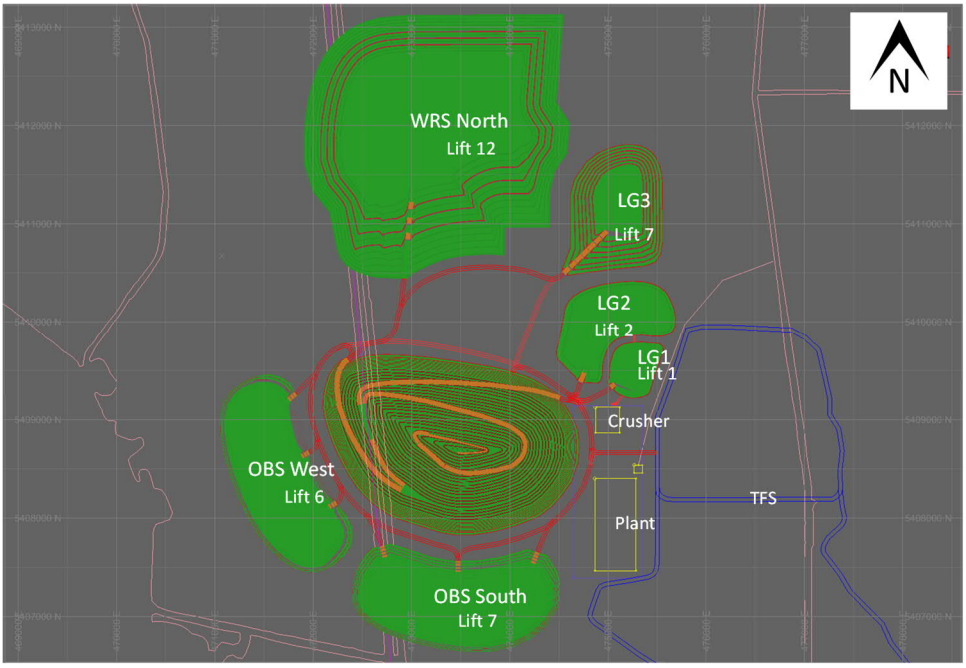
Source: Penswick, 2021.

Figure 16.14: End of Year 10



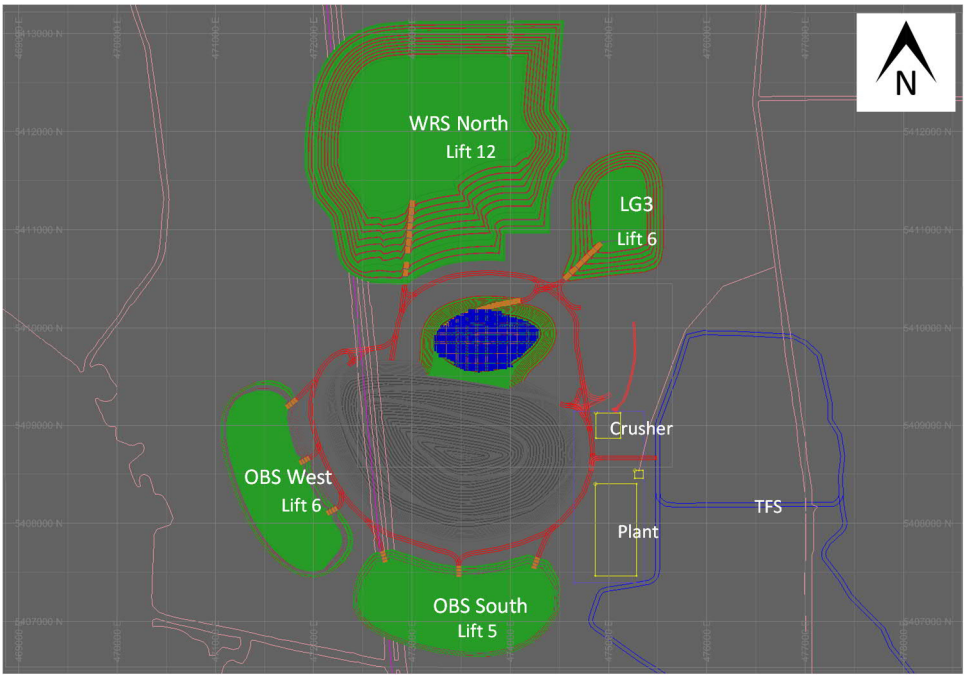
Source: Penswick, 2021.

Figure 16.15: Main Zone Immediately before Start of East Zone (mid-Year 16)



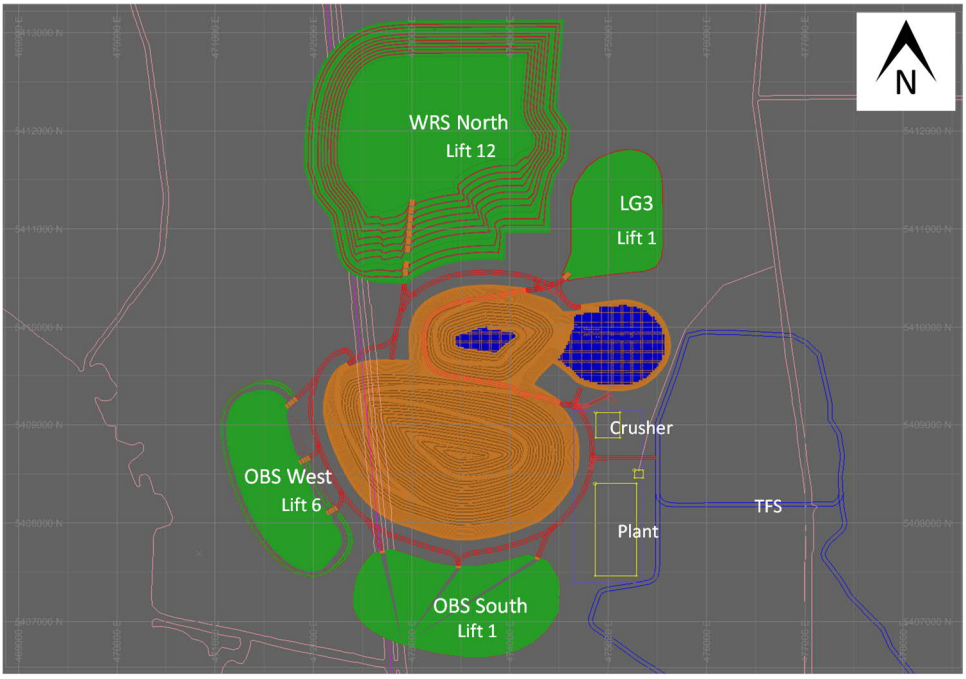
Source: Penswick, 2021.

Figure 16.16: EZ-W before Start of EZ-E (End of Year 18)



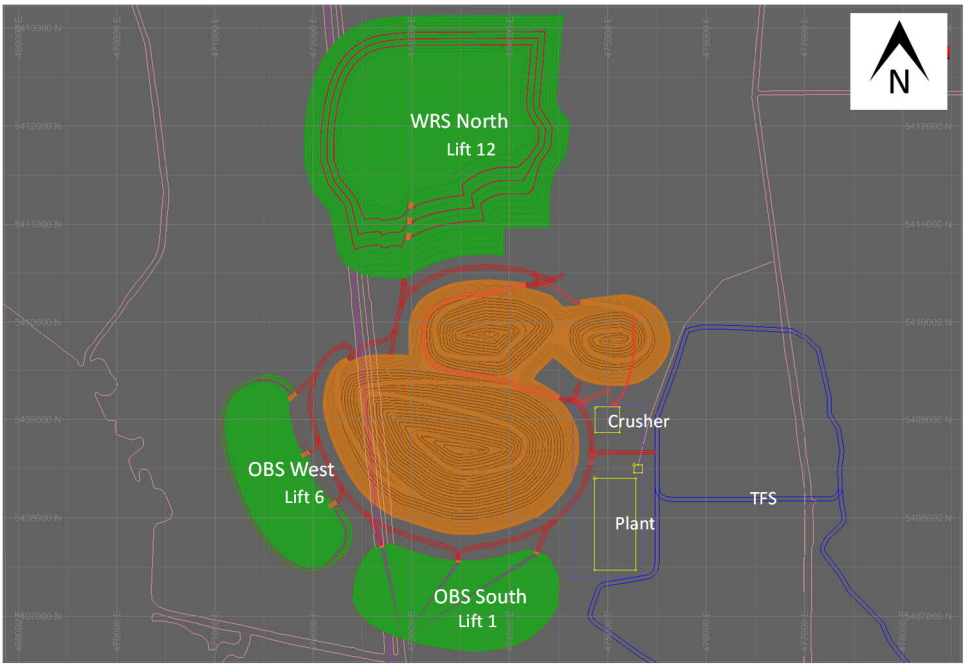
Source: Penswick, 2021.

Figure 16.17: East Zone before Depletion of LG3 (End of Year 22)



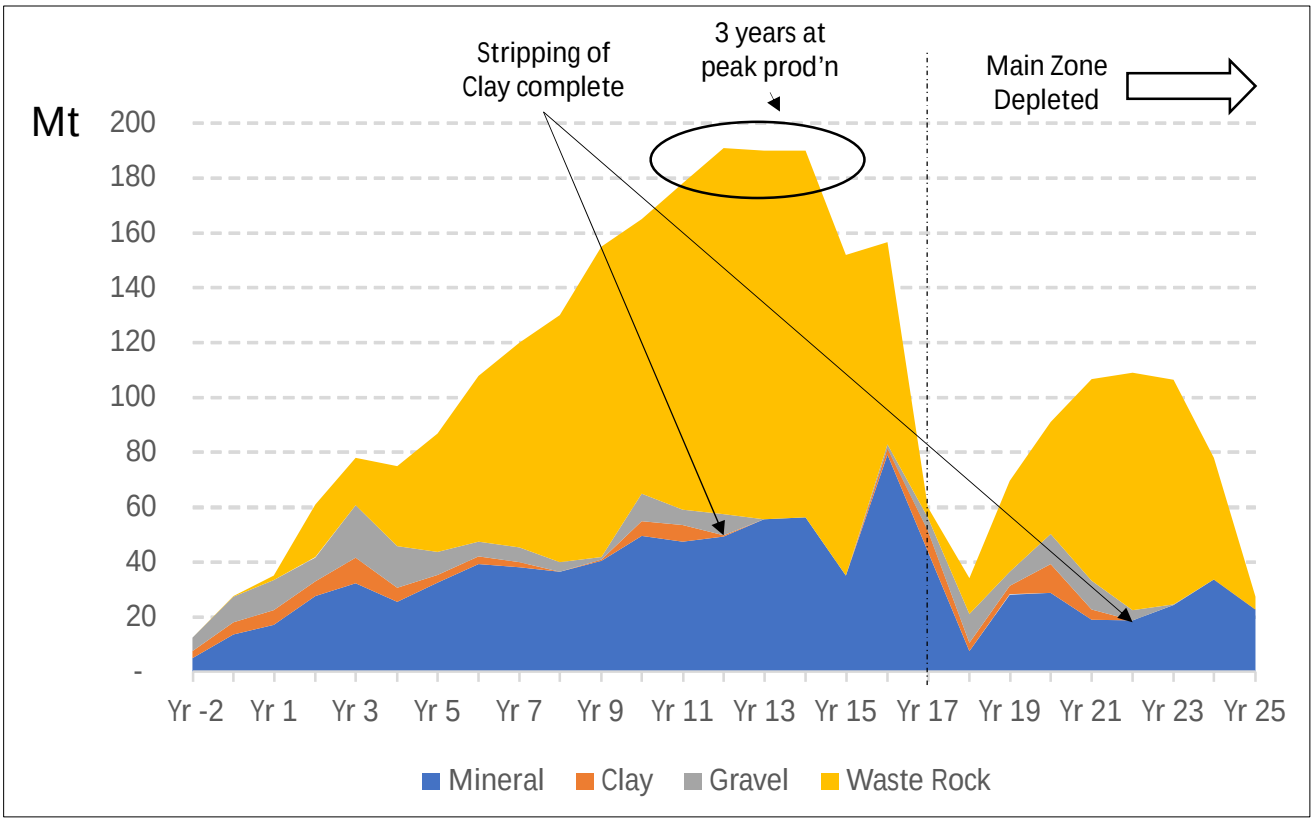
Source: Penswick, 2021.

Figure 16.18: End of Life (End of Year 25)



Source: Penswick, 2021.

Figure 16.19: LOM Production Schedule



Source: Penswick, 2021.

Attention is directed to the following:

- The Main Zone operates for 6 years following the completion of clay stripping, while the East Zone operates for 4 years. With more detailed scheduling it may prove possible to defer some clay stripping.
- The Main Zone ramps up to a peak production rate of 190 Mt/a (520 kt/d). The mining fleet peaks at this time, at seven 700 t excavators and forty-nine 290 t trucks. While the duration that peak production is maintained is relatively short, immediately afterwards the initial units of fleet reach their economic lives and are retired and no replacement truck purchases are required (two excavators are replaced over the life of mine). The open pit thus achieves a relatively high level of capital efficiency.
- At the end of Main Zone life, the combined inventory of material in higher grade stockpiles LG1 and LG2 totals 53 Mt with an average NSR value of C$30/t, which is 25% higher than the average value of all East Zone material. Start-up of the East Zone is thus deferred until these stockpiles are largely depleted, leading to the dip in ex-pit production in Year 17.

Table 16.4 summarizes mine production as well as feed by source (ex-pit and stockpiles) to the mill. Over the life of mine, 25% of total mill feed is temporarily stockpiled. The average residence time is 14 months in LG1, 33 months in LG2, and 128 months in LG3.

Table 16.4: LOM Production Schedule

Item	units	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y4	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y10-18	Y19+
MZ: Mineral	Mt	724	5	14	17	28	32	26	32	39	38	37	40	50	48	49	56	56	35	79	44	-	-	-	-	-	-	-	-	-	416	-
EZ-W: Mineral	Mt	129	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0	0	8	28	28	15	10	10	16	13	-	8	121
EZ-E: Mineral	Mt	55	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0	0	1	4	8	14	18	9	-	0	55
Mineral	**Mt**	**907**	**5**	**14**	**17**	**28**	**32**	**26**	**32**	**39**	**38**	**37**	**40**	**50**	**48**	**49**	**56**	**56**	**35**	**79**	**44**	**8**	**28**	**29**	**19**	**19**	**24**	**34**	**23**	**-**	**424**	**176**
MZ: Clay	Mt	51	2	4	5	5	9	5	3	3	2	-	1	5	6	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12	-
EZ-W: Clay	Mt	20	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2	8	2	0	5	2	0	-	0	0	-	12	7
EZ-E: Clay	Mt	11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	3	5	2	-	-	-	-	-	1	10
Clay	**Mt**	**80**	**2**	**4**	**5**	**5**	**9**	**5**	**3**	**3**	**2**	**-**	**1**	**5**	**6**	**0**	**-**	**-**	**-**	**2**	**8**	**3**	**3**	**10**	**4**	**0**	**-**	**0**	**0**	**-**	**24**	**17**
MZ: Gravel	Mt	115	5	9	11	9	19	15	8	6	5	4	1	10	6	8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	23	-
EZ-W: Gravel	Mt	30	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	5	10	3	6	4	1	0	0	0	-	16	14
EZ-E: Gravel	Mt	17	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0	2	5	7	3	0	-	-	-	0	17
Gravel	**Mt**	**162**	**5**	**9**	**11**	**9**	**19**	**15**	**8**	**6**	**5**	**4**	**1**	**10**	**6**	**8**	**-**	**-**	**-**	**1**	**5**	**10**	**5**	**11**	**10**	**4**	**0**	**0**	**0**	**-**	**40**	**31**
MZ: Waste Rock	Mt	1,260	0	0	2	19	17	29	43	60	75	90	113	100	119	134	134	134	117	73	1	-	-	-	-	-	-	-	-	-	811	-
EZ-W: Waste Rock	Mt	258	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	3	13	29	29	46	52	52	30	3	-	17	241
EZ-E: Waste Rock	Mt	124	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	4	12	28	34	30	15	1	-	1	123
Waste Rock	**Mt**	**1,641**	**0**	**0**	**2**	**19**	**17**	**29**	**43**	**60**	**75**	**90**	**113**	**100**	**119**	**134**	**134**	**134**	**117**	**74**	**4**	**13**	**33**	**41**	**74**	**87**	**82**	**44**	**4**	**-**	**828**	**365**
Total Ex-pit	**Mt**	**2,794**	**12**	**28**	**35**	**61**	**78**	**75**	**87**	**108**	**120**	**130**	**155**	**165**	**178**	**191**	**190**	**190**	**152**	**157**	**60**	**34**	**70**	**91**	**107**	**109**	**107**	**78**	**27**	**-**	**1,317**	**588**
ROM Mineral to Mill	Mt	679	-	-	8	11	16	16	22	25	25	28	40	44	44	44	44	44	35	37	15	7	28	29	19	19	24	34	23	-	312	176
ROM Mineral to LGS/Pile	Mt	228	5	14	9	16	17	9	11	14	13	8	-	6	4	6	12	12	-	42	29	1	-	-	-	-	-	-	-	-	111	-
Reclaimed LGS/Pile	Mt	228	-	-	5	4	-	5	9	6	6	13	3	-	-	-	-	-	9	7	29	37	16	15	25	25	14	-	-	-	82	94
Total Mill Feed	**Mt**	**907**	**-**	**-**	**13**	**16**	**16**	**21**	**30**	**31**	**31**	**42**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**38**	**34**	**23**	**-**	**394**	**270**
Grade Ni	%Ni	0.249	-	-	0.335	0.315	0.316	0.279	0.258	0.266	0.271	0.258	0.250	0.235	0.241	0.266	0.247	0.244	0.224	0.258	0.275	0.233	0.228	0.229	0.222	0.221	0.224	0.237	0.270	-	0.247	0.230
Grade Cr	%Cr	0.598	-	-	0.649	0.617	0.612	0.623	0.634	0.637	0.624	0.605	0.592	0.598	0.604	0.611	0.591	0.579	0.568	0.531	0.540	0.587	0.617	0.618	0.597	0.586	0.589	0.619	0.650	-	0.579	0.608
Grade Fe	%Fe	6.51	-	-	5.78	6.06	6.12	6.29	6.47	6.49	6.47	6.49	6.44	6.54	6.50	6.61	6.71	6.69	6.64	6.57	6.62	6.55	6.36	6.40	6.47	6.54	6.65	6.66	6.36	-	6.60	6.49

16.3.7 Cut-off Grade

Crawford is a polymetallic deposit, with the distribution of total NSR by metal as follows (payable production):

- Nickel: 1,689 Mlbs = 69% of total NSR.
- Chromium: 1,388 Mlbs = 8% of total NSR.
- Iron: 15,120 kt = 23% of total NSR.

As a result of the multi-element nature, cut-off is expressed as an NSR value rather than grade of nickel or nickel equivalent (NiEq).

As is normal for open pit mines, the calculation for cut-off ignores mining costs and includes only the marginal costs listed below.

- The incremental haulage costs incurred for re-handling low-grade material (as material at the marginal cut-off grade would initially be stockpiled) are included. The LOM average is low (C$0.40/t), as the low-grade stockpiles are depleted while the East Zone remains active. Fixed overhead costs are thus not allocated to the cost of rehandle.
- Milling costs, including sustaining capital that would be effectively expended on a per-tonne basis, are included. The LOM average is C$5.78/t, which includes C$0.19/t of sustaining capital.
- General and administration costs, which average C$0.57/t over the life of mine, are included.

Marginal costs total C$6.75/t. Note that this calculation excludes costs associated with the terrestrial TSF as this facility will have been decommissioned and tailings will be pumped into the mined-out pit by the time the lowest value material is reclaimed from the stockpile.

The theoretically calculated cut-off was also tested iteratively, by adjusting the cut-off value upwards and rescheduling the mine plan. NPV was maximized with the C$7/t cut-off, thus proving the validity of this value as a cut-off. It should be noted that there is only a limited tonnage of mineralization contained within the C$7 to C$12/t cut-off bin (as reported above, 16.7 Mt or approximately 2% of total planned mill feed). The financial impact of adjusting the cut-off upward to the C$12 to C$16/t bin would be negligible.

Note that at the average recovery for nickel of 37.3%, the C$7/t cut-off equates to a cut-off grade of 0.08% Ni.

16.3.8 Dilution and Mining Recovery

The drill hole sample compositing and block grade interpolation process used to construct the deposit block model is believed to incorporate sufficient dilution. Hence, no additional factors were applied.

Planned dilution and mining recovery can be expressed as the difference in grade and contained metal between the LG optimization on which the engineered design was based and the actual engineered design. These have been summarized previously in Figure 16-3 and are 2% and 99%, respectively.

Unplanned dilution and mining losses could occur on the contact between mineralization and waste. At this stage, that contact has not been measured and unplanned dilution and mining losses have not been included. A calculation of these will be included during the feasibility study. Operations with similar geometry and mining methods as Crawford typically report values of <0.5% for unplanned dilution and >99.5% for mining recovery, including unplanned losses.

16.4 Mining Description

16.4.1 Mining Fleet General

Ex-pit mining operations at Crawford will be conducted by the following fleets of production mining equipment (in order of lithology that will be mined— Figures 16.20 and 16.21):

- Areas where the depth of clay exceeds 7.5 m will be mined using 200 t backhoes loading 40 t articulated trucks. The backhoes will load from on top of the clay, requiring the surface to be armoured with crushed rock to prevent sinking. No drilling and blasting will be required for the overburden. The nominal bench height will be 5 m.

- Areas where the depth of clay is ≤ 7.5 m will be mined using a 300 t hydraulic excavator operating in face shovel configuration. The excavator will load from the underlying gravel footwall and deliver all material into 90 t rigid body haul trucks. No drilling and blasting will be required for the overburden. The nominal bench height will be 7.5 m.

- Below the lowest horizon of the clay/gravel interface, the horizon of mixed gravel and rock will be mined using a 700 t, electrically-powered hydraulic excavator operating in face shovel configuration. The excavator will deliver material into 290 t haul trucks using AHS to replace on-board operators. From Year 2 onwards, trucks will also be equipped to use trolley assist on equipped uphill hauls. No drilling and blasting will be required for the overburden, while rock will be drilled using rotary blast hole drills with a nominal hole diameter of 229 mm. Drills will be equipped with ADS to replace on-board operators. The nominal bench height in mixed gravel and rock will remain at 7.5 m.

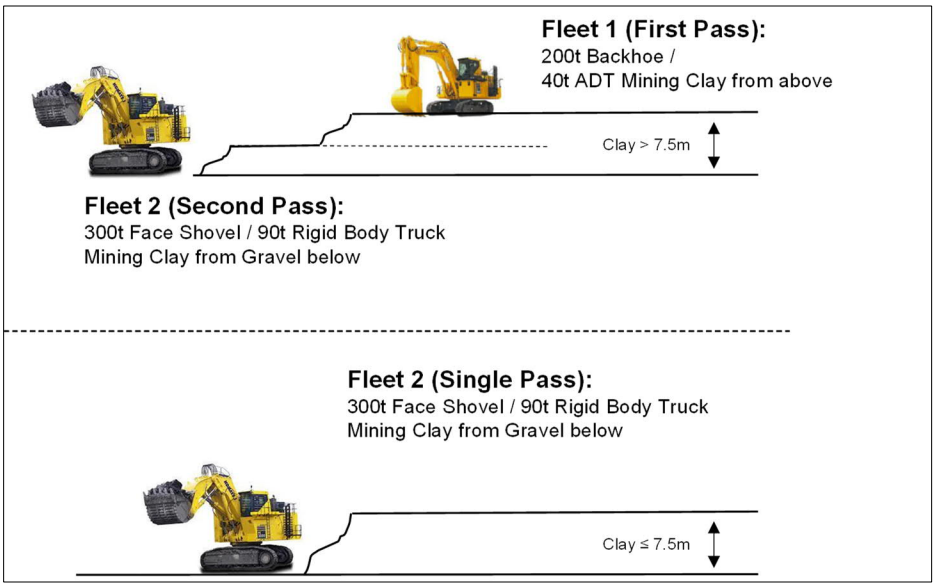
- Below the lowest horizon of the gravel/rock interface, the bench height will be increased to 15 m with the bulk of material blasted using holes measuring 311 mm in diameter. Where selective mining is required, each bench will be divided into two 7.5 m flitches that will be drilled using 229 mm holes. Selective mining will be conducted using the same 700 t excavators and 290 t AHS and trolley-assist equipped haul trucks as will be used for bulk mining.

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.

It has been assumed that a mining contractor would be employed in the initial year of pre-stripping. Thereafter, all mining fleet would be purchased and operated by the Owner.

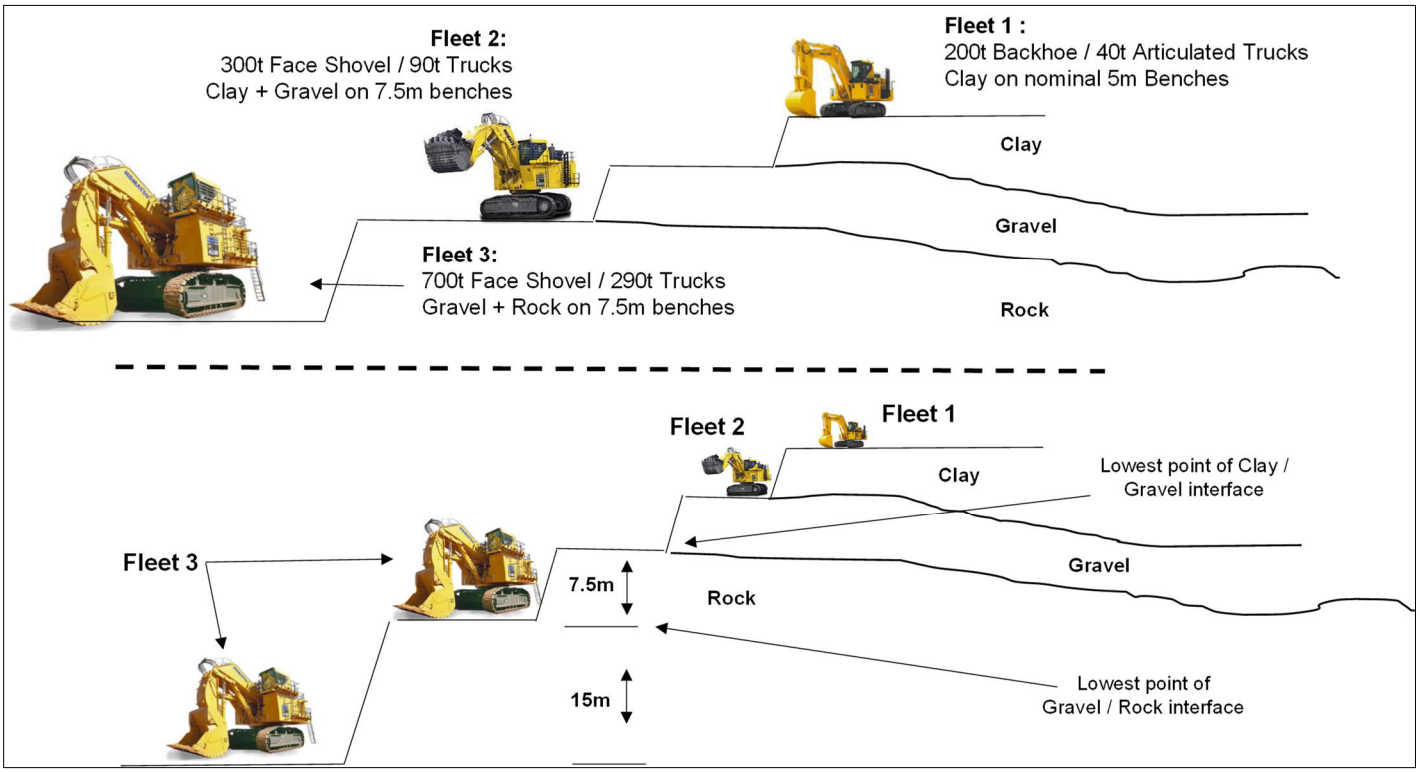
The duty cycle for production units was estimated by first principles, based on the production plan and the assumption that the mine will operate 24 hours per day, 365 days per year. Mechanical availability and utilization of equipment will vary according to the particular unit of equipment. Table 16.5 summarizes these values, along with the annual engine hours each would achieve (product of availability and utilization) for the main pieces of equipment.

Figure 16.20: Crawford Mining Fleets – Clay (not to scale)



Source: Penswick, 2021.

Figure 16.21: Crawford Mining Fleets – Below the Clay Horizon (not to scale)



Source: Penswick, 2021.

Table 16.5: Mining Fleet Availability and Utilization

Category	Unit	Example	Availability	Utilization	Engine Hours
Production Fleet	ADS Selective Blasthole Drill	Sandvik DR410 / Epiroc PV 235	86%	93%	7,006
	ADS Bulk Blasthole Drill	Sandvik DR412 / Epiroc PV 271	86%	93%	7,006
	200t Excavator	Cat 6018 / Komatsu PC 2000	83%	80%	5,817
	300t Excavator	Cat 6030 / Komatsu PC 3000	83%	80%	5,817
	700t Excavator	Cat 6060 / Komatsu PC 7000	83%	85%	6,180
	40t Articulated Truck	Cat 745 / Komatsu HM 400	85%	80%	5,957
	90t Haul Truck	Cat 777 / Komatsu HD 785	85%	80%	5,957
	290t AHS Haul Truck	Cat 794 / Komatsu 930e	86%	85%	6,404
	Small Front-End Loader	Cat 988 / Komatsu WA600	80%	71%	5,000
	Medium Front-End Loader	Cat 992 / Komatsu WA900	80%	71%	5,000
Support Fleet	Small Tracked Dozer	Cat D8 / Komatsu D155	75%	50%	3,000
	Medium Tracked Dozer	Cat D9 / Komatsu D375	75%	50%	3,000
	Large Tracked Dozer	Cat D11 / Komatsu D475	75%	50%	3,000
	Wheel Dozer	Cat 844	75%	50%	3,000
	Small Grader	Cat 14M	75%	50%	3,000
	Large Grader	Cat 18M	75%	50%	3,000
	130t Water Tanker	Cat 785 / Komatsu HD 1500	80%	75%	5,256
	Utility Excavator	Cat 390 / Komatsu PC 1250	75%	15%	1,000

The use of automation technologies is expected to impact utilization of the following units:

- ADS-equipped blasthole drills, where operator delays will be largely eliminated. Based on recommendations from OEMs with ADS units operating elsewhere, these units are expected to achieve average engine hours of approximately 7,000 per annum.
- AHS-equipped 290 t haul trucks, where operator delays will also be largely eliminated. Based on recommendations from OEMs with AHS units operating elsewhere, these units are expected to achieve average engine hours of approximately 6,400 per annum, or 8% higher than planned for manually operated units at Crawford.
- The 700 t excavators loading AHS trucks. To take advantage of the increased utilization of AHS trucks, some operations relieve the excavator operators with dozer operators to ensure the unit can load through shift change and other operator breaks. It has been assumed this practice would be employed at Crawford with the dozer and excavator operators placed at the same pay scale.

Table 16.6 provides a summary of the mining equipment fleet by year.

Table 16.6: Mining Fleet by Year

Item	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10-18	Y 19 +	Max
ADS Selective Blasthole Drill	0	1	1	1	2	2	1	2	2	2	2	2	1	3
ADS Bulk Blasthole Drill	0	1	1	1	1	1	2	3	3	3	4	4	2	5
200 t Excavator	1	2	2	2	2	2	2	2	2	1	1	2	1	2
300 t Excavator	1	3	3	2	3	2	2	2	2	1	2	2	1	3
700 t Excavator	0	1	2	3	3	4	4	5	6	6	7	6	4	7
40 t Articulated Truck	5	4	5	5	8	5	3	2	2	1	1	6	4	15
90 t Haul Truck	0	6	9	9	16	10	10	11	10	8	9	11	7	16
290 t AHS Haul Truck	0	5	7	12	14	14	18	24	30	37	42	39	18	49
Small-Tracked Dozer	1	2	2	3	4	2	2	2	3	3	4	3	1	6
Medium-Tracked Dozer	0	1	1	1	2	1	1	1	1	0	0	1	1	2
Large-Tracked Dozer	1	2	3	4	4	5	5	6	7	7	8	6	4	8
Wheel Dozer	0	1	1	2	2	2	2	3	3	3	4	3	2	4
Small Grader	0	1	1	1	1	1	1	1	1	0	0	1	1	1
Large Grader	0	1	1	2	2	2	2	3	3	3	4	4	2	5
Water Tanker	1	1	1	2	2	2	2	2	2	2	2	2	1	2
Small Front-End Loader	0	1	1	1	1	1	1	1	1	0	0	1	1	1
Medium Front-End Loader	0	0	0	1	1	1	1	1	2	2	2	1	1	2
Utility Excavator	0	2	2	2	2	2	2	2	2	2	2	2	2	2

16.4.2 Drilling & Blasting

Geotechnical testing of the various lithologies encountered at Crawford will be undertaken in the next stage of study. In the meantime, parameters for the various rocks are assumed to be similar to those for the same rock types encountered elsewhere in the Abitibi Region. To date, seven lithologies of rock have been identified, which can be broadly categorized as:

- "Soft", including the main mineral-bearing rocks dunite and peridotite, where the expected UCS would be 80 to100 MPa.
- "Hard", including pyroxenite, which does carry some mineral, and waste rocks which include gabbro and various metavolcanics, where the expected UCS would be 150 to 180 MPa.

It was assumed that both categories of rock could be adequately fragmented for productive loading by a 700 t excavator (equipped with 38 m^3 dippers) with a powder factor of 0.25 kg/t. This would require patterns with a hole burden and spacing of approximately 7 m for selective blasts (on 7.5 m flitches), increasing to 10 m for the bulk blasts on 15 m benches. Based on OEM recommendations, the following drill penetration rates were assumed:

- Soft rocks: 45 m/h for both the selective (229 mm) holes and bulk (311 mm) holes
- Hard rocks: 13 m/h for the selective holes increasing to 20 m/h for the bulk holes.

Delays for moving between holes and patterns were estimated to range between 10% and 12% of the total utilized time. With the use of ADS technology, other delays would be minimized.

Rock would be blasted using emulsion explosives manufactured at site by the explosives supplier. The explosives supplier would also take responsibility for charging holes and maintaining all explosives-related plant structures and equipment.

It has been assumed that the final walls would be pre-split. A detailed pre-split design will be generated during the next phase of study. In the meantime, based on costs reported elsewhere, the cost associated with pre-splitting was assumed to be approximately 5% of the cost of normal blasthole drilling and blasting.

16.4.3 Loading & Hauling

Multiple fleets of load and haul equipment will be employed to ensure the various lithologies at Crawford will be mined most productively. Areas where the clay thickness exceeds 7.5 m will be initially stripped using 200 t backhoe excavators. Where clay is shallower and on benches with mixed clay and gravel, 300 t excavators in face shovel configuration will be used. Both the smaller excavators will be diesel powered. They will load a combined 5% of the 2,794 Mt ex-pit tonnage. The 700 t excavators will operate on both 7.5 m flitches (where selective loading of heterogenous materials is required) and 15 m benches (where materials are homogenous and can be loaded in bulk). In addition to 2,645 Mt ex-pit loading, these units will also load 228 Mt from low-grade stockpiles. Criteria used to calculate the productivity of various loading units is provided in Table 16.7. Design criteria for the various sizes of truck that will be employed are listed in Table 16.8.

Table 16.7: Loading Design Criteria

Item	Units	Excavator Size		
		200 t	300 t	700 t
Bucket Capacity	m^3	12.0	16.0	38.0
Bucket Factor	tonnes	18.4	27.4	65.0
Truck Factor	tonnes	37	87	286
Passes to fill truck[1]	number	3.25	3.45	4.65
Cycle Time	minutes	0.5	0.5	0.5
Spot Time	minutes	0.5	0.5	0.5
Time to load truck	minutes	2.13	2.23	2.83
Engine Hours	annual hours	5,817	5,817	6,180
Non-Productive time[2]	annual hours	1,745	1,745	1,854
Nominal productivity	annual kt loaded	4,231	9,565	26,307

Notes: 1. Includes allowance for in-efficiency. 2. Includes blast delays, waiting for trucks and other operator delays

Table 16.8: Hauling Design Criteria

Item	Units	Truck Size		
		40 t	90 t	290 t
Payload	tonne	37	87	286
Loading Time	minutes	2.13	2.23	2.83
Dumping Time	minutes	2.00	1.00	1.00
Queuing Time	minutes per cycle	2.00	2.00	2.00
Speed - in-pit flat (empty & full)	km/h	19	25	30
Speed - ex-pit (empty & full)	km/h	26	35	50
Speed - uphill loaded conventional	km/h	10.0	11.7	11.0
Speed - uphill loaded trolley	km/h			22.0
Speed - downhill empty	km/h	23	40	59
Average Cycle Time	**km/h**	**37.6**	**31.3**	**24.5**
Average Fuel Burn	**liters/h**	**29.6**	**54.8**	**165.7**
Average Power Consumed	**MW**			**0.9**

The following should be noted:

- Mines that employ manually-operated trucks typically impose speed limits for safety reasons. When AHS systems are used, it is more common to let trucks operate unconstrained, at their rimpull limits. Note that every person or machine entering the area where AHS trucks are active will be electronically tagged so as to be visible to the trucks. Any person or vehicle crossing the path of an AHS unit will automatically cause the autonomous unit to be stopped.

- In the event AHS and trolley assist were not employed, the average cycle time for the 290 t trucks would increase 31% while fuel burn would increase 80%. In this scenario, no power would be consumed.

16.4.4 Support Equipment

Open-pit haul roads and working faces would be maintained with a fleet of support equipment that includes the following:

- Tracked dozers, for ripping footwalls and for heavy construction work. The fleet requirements were estimated based on the empirical relationships of 0.5 operating small dozers for every operating 200 t excavator and 1.0 operating medium and large dozers for every 300 t and 700 t excavator, respectively.
- Wheeled dozers, for lighter construction and general clean-up. A single size of unit would be employed, with fleet requirements estimated based on the empirical relationship of 0.5 operating dozers for every operating 700 t excavator.
- Graders, with two sizes of units employed. Graders with a 14 ft blade would maintain roads used by the 40 t articulated trucks while units with an 18 ft blade would maintain other roads. The fleet requirements were estimated based on the empirical relationship of one grader for every 13 trucks.
- Water tankers, for spraying roads to mitigate dust. The fleet requirements were based on the annual surface area of operating roads, reported average annual evaporation for the region of 0.38 m and the assumption that 5% of this evaporation would be experienced in each of the first and fourth quarters (*i.e.*, fall and winter) while 45% would be experienced in each of the second and third quarters (spring and summer).
- Front-end loaders (FEL) for construction and clean-up activities, including the loading of roadstone into trucks. A small FEL with bucket capacity of 10 t would be employed to support operations in clay, while a larger unit with a 20 t payload would support other operations. This unit would also be used to feed the roadstone crusher and load crushed roadstone onto 90 t trucks. The maneuverability and size of fleet of the 700 t excavators is such that it will not be necessary to provide a large FEL for contingent production loading capacity.
- Utility excavators would be used to scale highwalls, dig ditches and perform other construction work as required.

16.4.5 Technology

Fleet productivity will be maximized and operating costs minimized by the extensive use of technologies including the following:

- ADS for blasthole drills, which is offered as a factory-installed option by drill equipment OEMs. A key technological building block of ADS is high-precision GPS, which eliminates the requirement for staking the location of holes while ensuring more accurate determination of collar and toe elevations. Another key building block is rock recognition, which ensures more consistent drill performance, particularly when lithologies change down the hole.
- AHS for haul trucks. This technology is offered as a factory-installed option by Komatsu and Caterpillar.
- A fleet management system (FMS). An FMS, such as Modular Mining's Dispatch, assigns, tracks and monitors the mobile equipment fleet. Assignments are continually updated to take account of the status of all equipment and thus achieve the mine plan with minimal resources and operating expenditure. For example, the FMS may identify a section of road where haul trucks are forced to reduce speed. A dozer and/or grader will then be automatically dispatched to the location where cutting and/or filling is required to return the road to grade.

- High-precision GPS (HP GPS) for loading units. These systems help achieve the planned X-Y-Z limits for composites, thereby minimizing dilution, maximizing mining recovery and improving pit floor conditions. Improved floor conditions, in turn, will facilitate faster truck speeds and extended tire life.

- HP GPS on dozers and graders, to ensure pit floor and ramp conditions are maintained to the highest standard. Improved floor and ramp conditions will facilitate faster truck speeds and extended tire life.

- Payload monitoring on loading units. This provides operators with the dipper-by-dipper mass delivered to a haul truck to minimize over- or under-loading. It maximizes haul-truck utilization while minimizing wear on truck frames, suspension, and tires.

- Tire monitoring systems. These use sensors within the tires on haul trucks, FELs and wheel dozers that report real-time pressure and heat data which in turn can be used for reassignments that increase safety and tire life. For example, a haul truck with tires approaching the limit that could lead to a heat separation can be re-routed to duties at lower speeds, allowing the tires to cool.

16.5 Mining Infrastructure

16.5.1 Impoundments

Materials will be impounded in the following facilities (see Figures 16.9 to16.18 previously for the arrangement of the various impoundments). Note that other than the TSF (which is discussed in Section 18), all impoundments will be constructed using the slope angles discussed previously (6H:1V facing the pit and 3H:1V elsewhere).

- Three low-grade stockpiles (LG1 to LG3). A total of 228 Mt will be impounded over the LOM, with the maximum tonnage at any one time being 157 Mt. These will be constructed in 10 m lifts.

- Two overburden stockpiles (OBS-South and OBS-West). These will contain clay, gravel and rock, with clay being tipped within cells constructed from gravel and rock. These will be constructed in 5 m lifts and will be accessed by multiple ramps to minimize haulage distances.

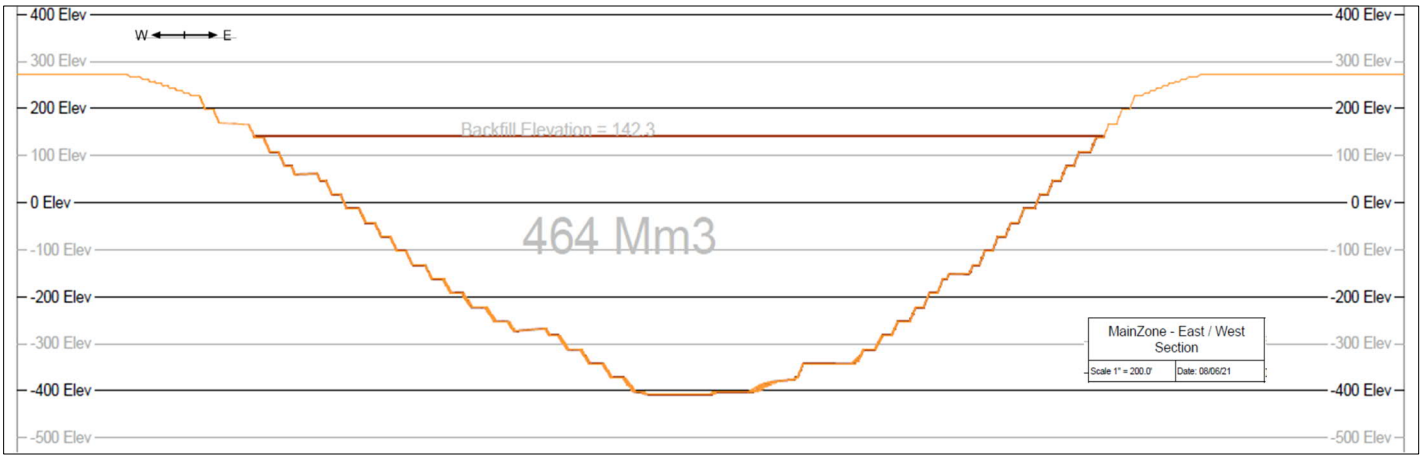
- A waste rock stockpile (WRS-North). This will contain only rock and will be constructed in 10 m lifts. The facility will be accessed using a single ramp equipped with trolley assist.

- Tailings will be stored in the TSF, which will be constructed in two phases. The starter dam will be located so as to minimize the haul distance for trucks delivering construction materials. It will extend to a height of 36 m and provide capacity for 47 Mm^3 tailings, equivalent to the initial four years of mill production. Thereafter, the facility will be extended to its ultimate footprint and raised to a height of 73 m, impounding the remaining 362 Mm^3 tailings that will be produced before the Main Zone is depleted.

- Following depletion of the Main Zone, the mined-out pit will act as an impoundment for all tailings, overburden and rock produced thereafter. An aggregate of 464 Mm^3 of the various waste products will be impounded within the pit, representing 55% of its total volume. Over time, the remaining 128 m to surface will fill with water (Figure 16-22).

Salient metrics for the various impoundments are given in Table 16.9.

Figure 16.22: Waste Impounded in Main Zone Pit



Source: Penswick, 2021.

Table 16.9: Impoundments

Impoundment	Mass (Mt)	Volume (Mm^3)	Height (m)
LG1	9	4	30
LG2	45	22	40
LG3	107	53	80
OBS-South	83	51	35
OBS-West	68	42	35
WRS-North	981	469	120
Starter TSF	48	27	36
Ultimate TSF	266	131	73

The following should be noted:

- The mass, volume and height for the three low-grade stockpiles reflects the maximum amount of material impounded at any one time. At the end of life, the stockpiles will all be completely depleted.
- The mass and volume for OBS-South reflects the total material delivered over the LOM, and "height" is the maximum height reached. Some gravel impounded in this facility is then rehandled and used for TSF construction. At the end of mine life, the facility will contain 14 Mt within a single 5 m lift.
- The tonnage and volume for the TSF reflect only the materials used for construction.

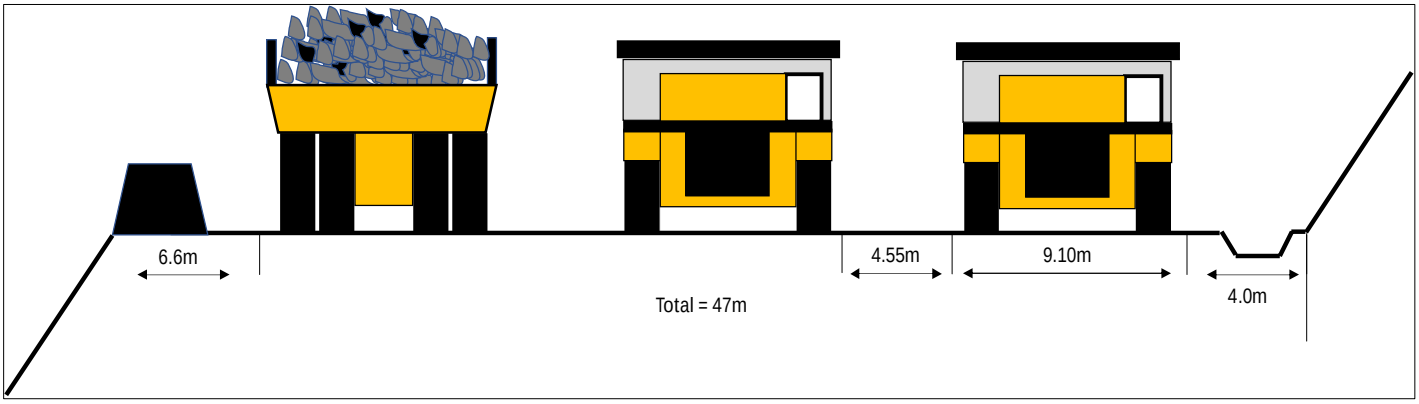
16.5.2 Roads

In-pit ramps have been sized to allow the following:

- three-lane traffic, which allows overtaking of a stopped truck (lanes are spaced at half a truck width)
- a safety berm on the in-pit side of the ramp
- a drainage ditched on the highwall side of the ramp

For the 290 haul trucks planned for use, this necessitates a 47 m ramp width (Figure 16-23).

Figure 16.23: Crawford In-pit Ramp Design (Excludes Trolley)



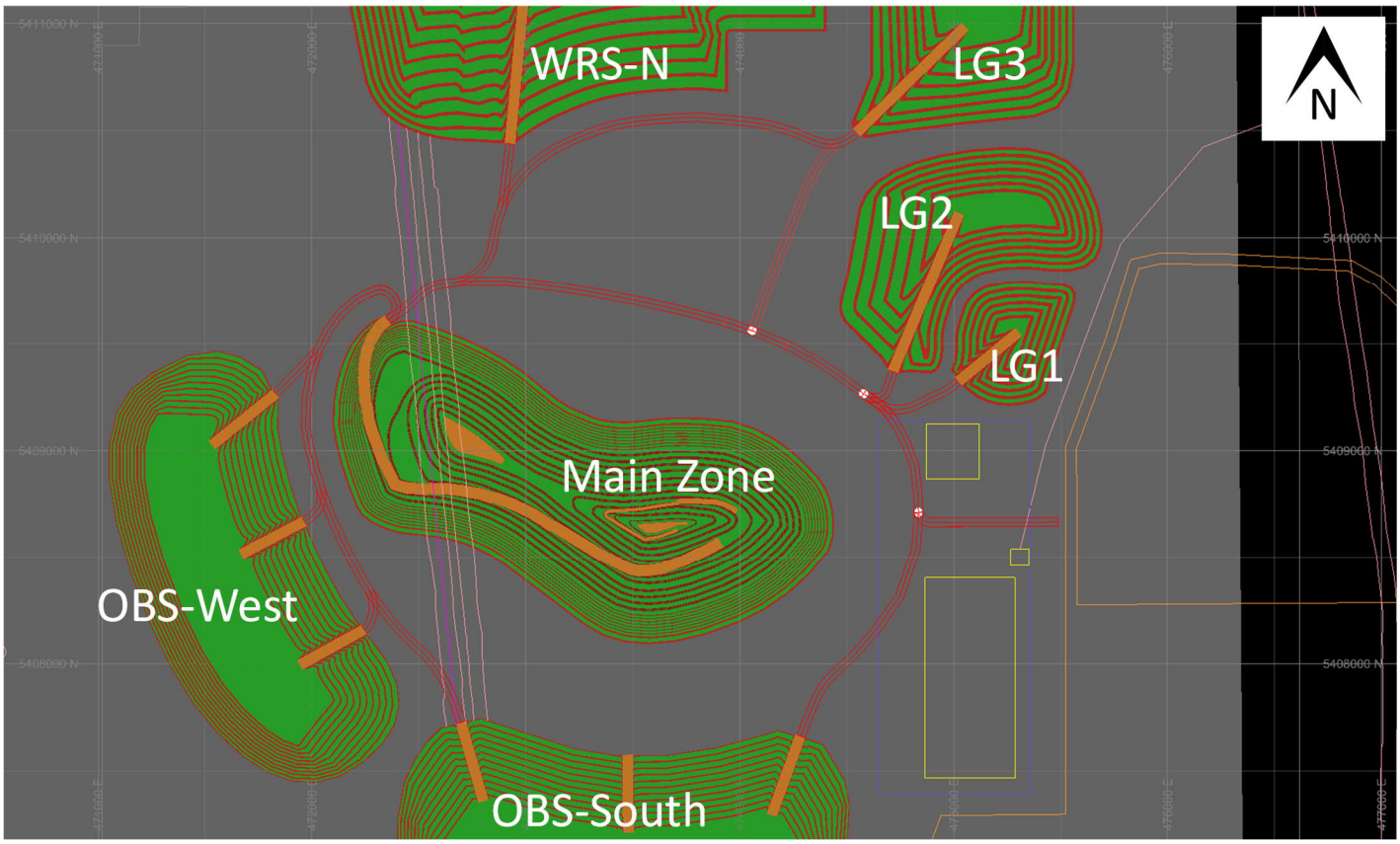
Source: Penswick, 2021.

On ramps that will be equipped with trolley assist, an additional 7 m has been allowed to accommodate substations and poles, for a total width of 54 m.

Figure 16-24 illustrates the surface road network at the end of Year 4, when the bulk of permanent roads totalling 34.5 km have been constructed.

The cost of constructing roads assumes that waste materials (rock and gravel) from the Crawford pit would be used and the average thickness of material would be 2 m. Subsequent extensions to provide access for the East Zone and final TSF footprint add another 7.6 km of road, with the ultimate network totalling 42.1 km.

Figure 16.24: Crawford Surface Road Network at Year 4



Source: Penswick, 2021.

16.5.3 Workshop

A workshop and associated warehouse would be provided to maintain the fleet of equipment. The size of this workshop was based on the empirical factor of one bay for every five 290 t haul trucks or ten 90 t haul trucks. At start-up, the workshop will comprise four bays. As the fleet expands in response to increased production rate and longer hauls, the workshop will be progressively expanded to 11 bays.

16.5.4 Fuel Farm

Fuel consumption has been estimated from first principles based on the burn rate for the various pieces of equipment that would be operated and specific duty cycle. Over the life of mine, approximately 1.1 Mm^3 diesel will be consumed with the average daily consumption rate being 110 m^3. The peak rate will be 234 m^3/d. Fuel would be trucked to site on a daily basis and storage has been based on an assumed inventory of three days, as recommended by fuel suppliers.

16.5.5 Explosives Plant

Rock will be blasted using emulsion explosives. These will be manufactured at site, from inert raw materials that would be trucked to site. Over the life of mine, 661 kt explosives will be consumed at an average rate of 70 t/d. Peak consumption will be 135 t/d.

The explosives manufacture facility will use intellectual property owned by the explosives supplier. In line with North American practices, the facility would thus be owned and operated by the explosives supplier. Based on budgetary quotations provided by suppliers, the financial model assumes that the capital cost associated with all equipment and facilities will be borne by the explosives OEM and recovered by way of a service charge applied once the project begins generating cashflow.

16.5.6 Roadstone Crusher

To ensure the truck fleet achieves high productivity, including an average life of 7,200 hours for tires on the 290 t AHS-equipped haul trucks and the utilization of the trolley system discussed in the next section, roads would be continually re-surfaced with crushed waste rock. Rock would be crushed to a nominal size of 18 mm through a two-stage plant (primary jaw and secondary cone crusher).

The calculated duty-cycle of this plant is based on all trolley-equipped ramps will receive 35 mm (approximately 1 ft) of crushed material annually while non-trolley-equipped roads would receive 50% of this treatment. The roadstone crusher would also crush all stemming used in blast holes. The life-of-mine requirement for roadstone product has been estimated at 54 Mt. Ex-pit waste would be temporarily stockpiled in close proximity to the crusher, then fed using a front-end loader. The same front-end loader would then load 90 t trucks that would distribute material to the various roads.

16.5.7 Trolley Assist

16.5.7.1 Background

Trolley assist is an established technology. The first open pit commercial scale application of a trolley system of any kind was Lac Jeannine (Quebec, iron ore) in the 1970s. The initial commercial application using the pantograph technology employed globally today was Palabora (South Africa, copper) in 1980. Trolley assist has since been successfully employed at a number of mines in Africa, Europe, North America, and Latin America.

A diesel-electric haul truck normally utilizes a diesel engine to drive an alternator, which produces the electricity used to drive the wheel motors. The voltage and amperage of power is conditioned in order for the motors to provide the desired speed and torque, much as a transmission does in a mechanical drive truck. The speed on grade is limited by the kilowatt output of the diesel engine.

With trolley assist, two pantographs mounted on the truck collect electric power from an overhead line. Electricity is fed to the line from a direct current (DC) substation, requiring that the alternating current (AC) supply to the mine be first rectified. As there is no need for the electricity provided by the on-board alternator, speed of the diesel engine drops to idle. Speed of the truck is no longer controlled by the output of the engine but rather the capabilities of its traction wheelmotors.

Savings from trolley assist can be categorized as follows:

- Energy Cost Savings – which occur as power is supplied to wheelmotors 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. For the current Crawford design, use of the trolley would save 700 ML of diesel costing C$640 million, at the expense of 3,400 MWh of electricity costing C$380 million.

- Productivity Savings – which result from the increased speed of haul trucks traveling uphill on trolley, with a doubling of speed possible for the class of truck planned for use at Crawford. This allows the mine plan to be achieved with fewer trucks and an associated reduction in labour. The reduction in truck fleet has additional benefits by reducing congestion associated with 'bunching' of units following shift change and other stoppages. For the current Crawford design, the use of trolleys would reduce truck fleet usage by 690 thousand hours or 16%, leading to a reduction in truck purchases of 12 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 has significant environmental benefits, resulting from the reduction in particulate matter and greenhouse gases associated with generating energy from hydro-carbons.

The savings associated with trolley-assist are partially offset by costs associated with operating the system, including 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.

- an increase of 7 m in the width of equipped ramps to accommodate trolley-assist infrastructure (primarily the substations), which necessitates flatter overall slopes and increased waste stripping.

16.5.7.2 Crawford Trolley Assist Design

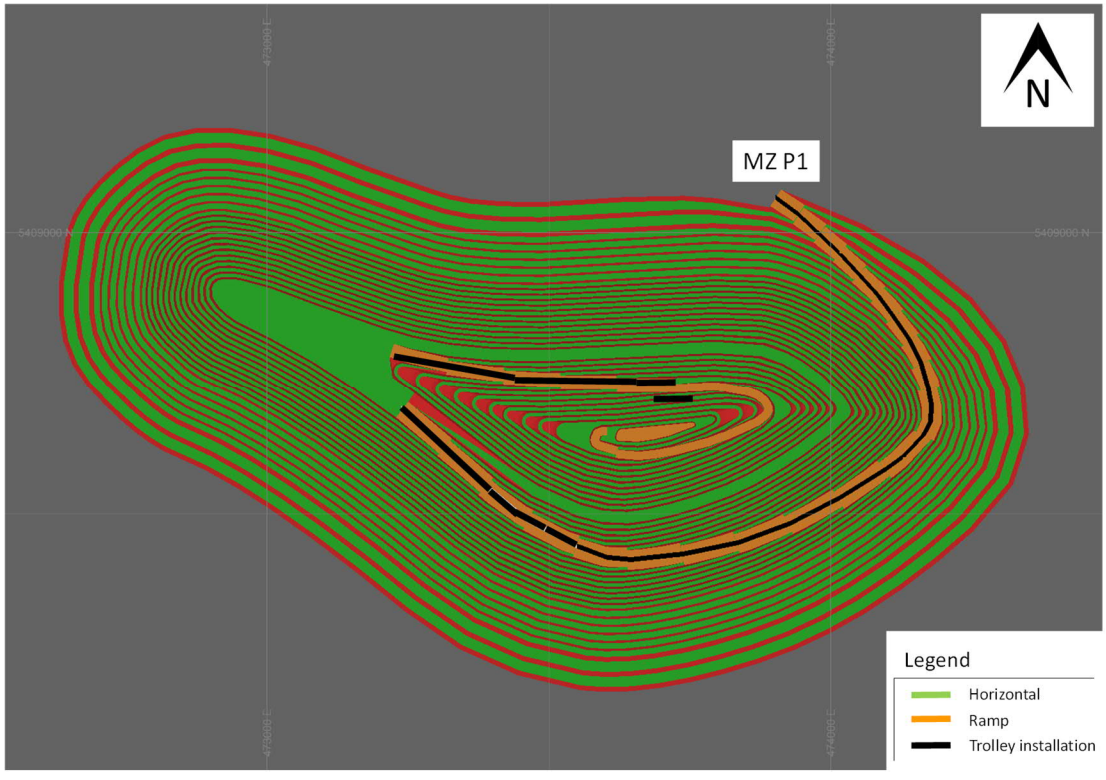
The Crawford design was optimized for use of trolley assist by reducing the number of ramp systems, both in the pit and on waste dumps. This ensured each ramp would achieve the critical mass of traffic necessary to justify installation of infrastructure. While this will lead to longer horizontal hauls in some instances, it will enable all permanent ramps to be equipped for trolley technology. Note that the large Main Zone pit maintains two ramp systems for the bulk of its life, ensuring dual redundancy to mitigate geotechnical risk. Over the life of mine, seven ramp systems will be equipped (Figures 16-25 to 16-28 and Table 16.10), as follows:

- The initial system equipped will be the Main Zone Phase 1, starting in Year 2. The trolley infrastructure on this ramp will ultimately extend 2.3 km and handle 2% of all uphill loaded hauls. The system will be decommissioned after Year 8, with infrastructure re-used.

- Main Zone Phase 2 will be equipped also starting in Year 2. Infrastructure on this ramp will ultimately extend 3.5 km and handle 7% of all uphill loaded hauls. The system will be decommissioned after Year 10, with infrastructure re-used.
- Main Zone Phase 3/4 waste will be equipped starting in Year 4. The upper 825 m of infrastructure will already be in place (from Phase 2). After 1.5 km, this ramp ties into Main Zone Phase 3/4 ore at depth. The infrastructure on this upper 1.5 km will handle 14% of all uphill loaded hauls. The system will be decommissioned after Year 13, with infrastructure re-used.
- Main Zone Phase 3/4 ore will be equipped starting in Year 6. Infrastructure on this ramp will extend 6.8 km, the lower 5.3 km being shared by trucks using the Main Zone Phase 3/4 waste exit. This is the highest usage ramp, with infrastructure handling 44% all uphill loaded hauls. The system will be decommissioned in Year 17, with 3.8 km (55%) of the infrastructure subsequently re-used.
- EZ-W will be equipped starting in Year 16 and will handle 8% of all loaded uphill hauls.
- EZ-E will be equipped starting in Year 18 and will handle 3% of all loaded uphill hauls.
- The waste rock storage facility will be equipped starting in Year 2 and will handle 7% of all loaded uphill hauls.

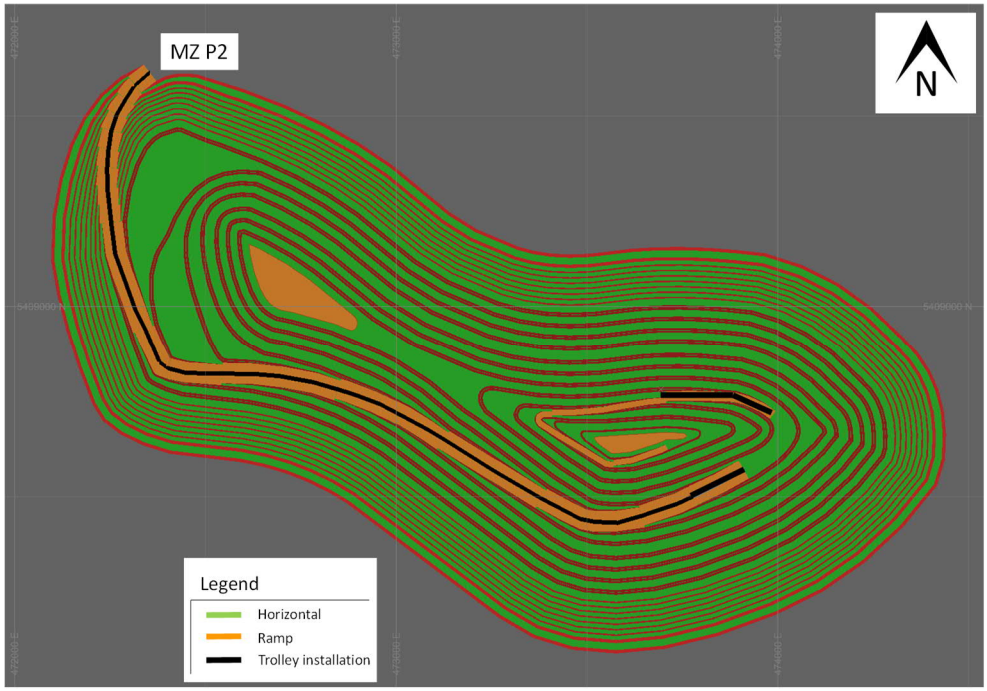
In aggregate, the system will achieve a very high utilization of 84% of all loaded uphill hauls for the 290 t haul trucks.

Figure 16.25: Main Zone Phase 1 Ramp System Trolley Installations



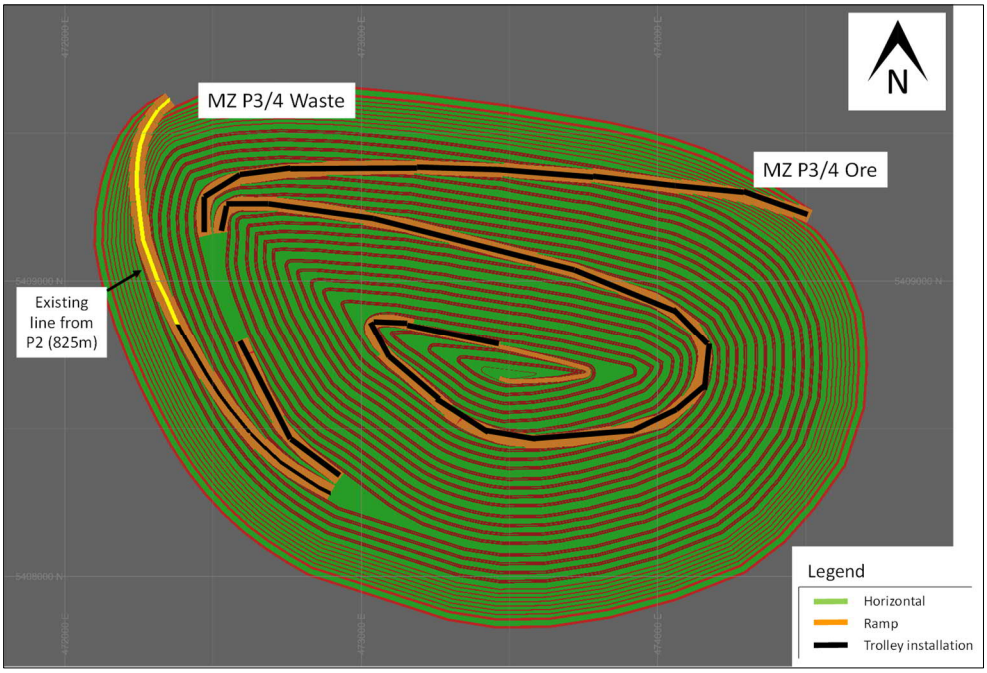
Source: Penswick, 2021.

Figure 16.26: Main Zone Phase 2 Ramp System Trolley Installations



Source: Penswick, 2021.

Figure 16.27: Main Zone Phase 3/4 Ramp System Trolley Installations



Source: Penswick, 2021.

Figure 16.28: East Zone and WRS-N Trolley Installations



Source: Penswick, 2021.

Table 16.10: Utilization of Trolley Ramps

Ramp	Maximum Length (m)	kilotonnes x kilometres travelled[1]	Utilization[2]
MZ Ph1	2,325	11,260,983	2%
MZ Ph2	3,525	38,363,079	7%
MZ Ph3 W	1,500	79,712,689	14%
MZ Ph4 O	6,825	251,290,202	44%
EZ-W	3,675	45,702,763	8%
EZ-E	2,925	15,383,422	3%
WRS	1,200	42,181,003	7%
Subtotal Trolley	**21,975**	**483,894,140**	**84%**
Total Mine	**577,424,177**		

Notes: 1. Uphill loaded hauls for 290 t trucks only. 2. Percentage of total 290 t tuck uphill loaded kilometres.

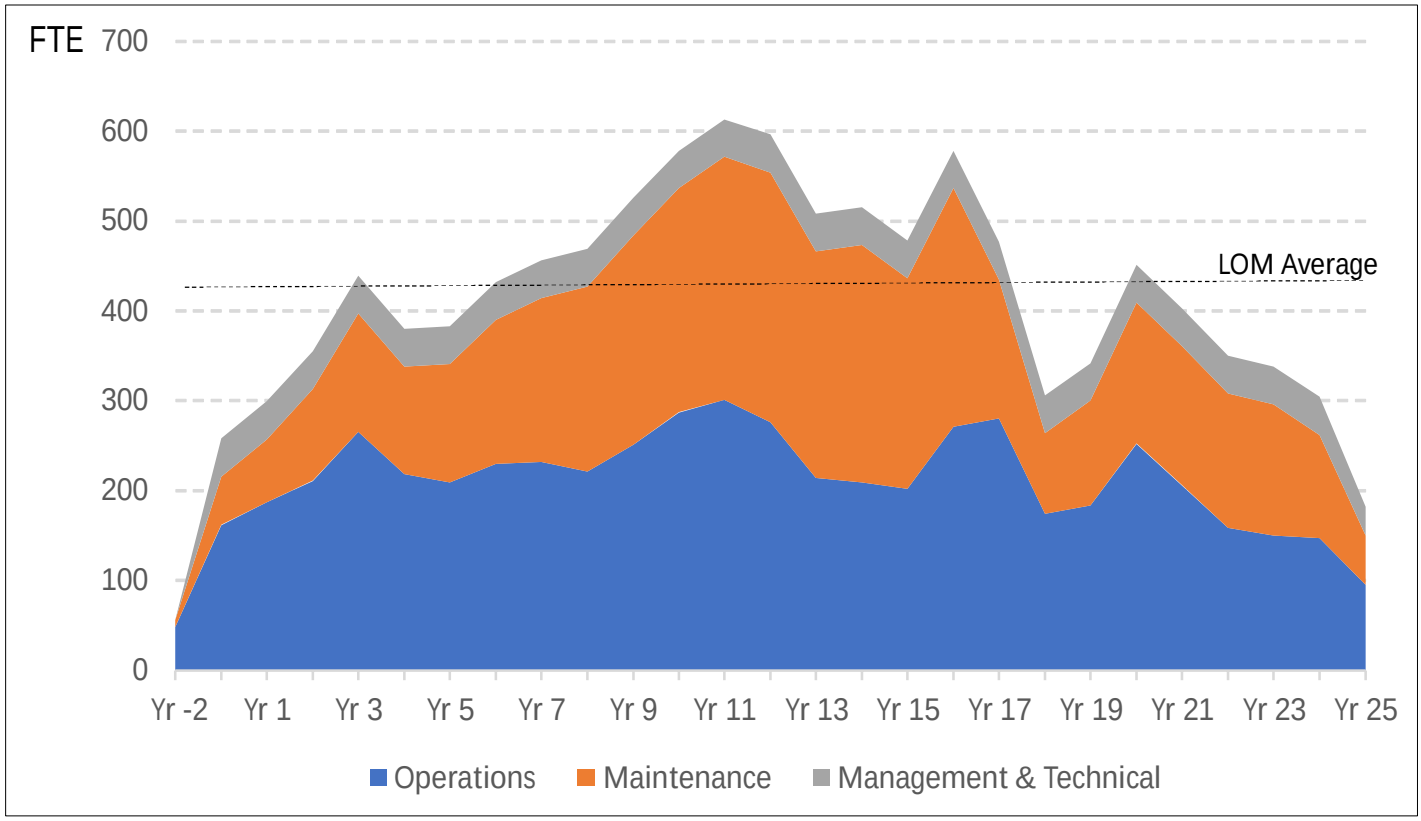
16.6 Labour

The open pit will operate continuously, with two 12-hour shifts daily, 365 days per year. This will be achieved by four crews each working an average of 42 hours per week; 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 (Figure 16-29 and Table 16.11) 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 personnel associated with construction of the TSF.

Over the life of mine, including the period of pre-stripping, the complement will average 416 full-time equivalent personnel (FTE), reaching a peak in Year 11 of 613. In the event that trolley assist and automation technologies (AHS and ADS) are not employed, the average complement would be 30% higher (at 541 FTE) while peak numbers would be 43% higher (at 878 FTE).

Figure 16.29: Mining Labour



Source: Penswick, 2021.

Table 16.11: Mining Labour

Area	Position	LOM Average	
		Complement	Cost to CNC (C$k/a)
Operations	Drill Operators	9	96
	Contract Blasting Labour	12	148
	Excavator Operators	29	91
	FEL Operators	5	91
	Truck Drivers	50	88
	Contract AHS Support	9	394
	Support Equipment Operators	75	85
	Roadstone Operators	3	91
	Dispatch Operators	12	110
	Subtotal / Average	**204**	**106**
Maintenance	Mechanics / Electricians	143	95
	Maintenance Attendants	19	77
	Maintenance Planners	2	110
	Subtotal	**164**	**93**
Management & Technical	Manager	1	250
	Superintendent	2	208
	General Foremen / Chief Engineer	3	179
	Operations Supervisors	15	119
	Maintenance Supervisors	4	151
	Engineers	7	155
	Geologists	4	144
	Survey	3	124
	Technicians	4	131
	Clerical	5	77
	Subtotal	**48**	**136**
Total		**416**	**105**

17 RECOVERY METHODS

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

17.1 General

The process plant and associated service facilities will process ROM ore delivered to primary crushers to produce nickel concentrate and tailings. The proposed process encompasses:

- crushing and grinding of the ROM ore.
- desliming via hydrocycloning.
- slimes rougher and cleaning flotation.
- coarse rougher, scavenger and cleaning flotation.
- regrind of the coarse flotation circuit final concentrate.
- high-grade material flotation.
- magnetic recovery of coarse scavenger tailings.
- regrinding of magnetic concentrate, scavenger concentrate and coarse first cleaner tailings.
- fines rougher and cleaning flotation of the reground material.
- magnetic recovery of fines rougher and first cleaner tailings.

Concentrate will be thickened, filtered and stockpiled on site prior to being loaded onto railcars or trucks for transport to third-party processing facilities.

The coarse scavenger magnetic separation tailings will be fed to the coarse tailing's thickener. All other tailings will be combined to feed a slimes thickener. After thickening, all tailings will be sent to the tailings storage facility (TSF).

The process plant will be built constructed in three phases. Phase 1 will have a steady-state throughput of 42,500 t/d. Phase 2 will double throughput starting in year four, by mirroring the first line. Phase 3 will raise production to the ultimate rate of 120,000 t/d through the addition of secondary crushing, a third ball mill, and additional downstream capacity.

17.2 Plant Design Basis

The key criteria selected for the plant design are as follows:

- Phase 1 nominal base plant treatment rate of 42.5 kt/d.
- Phase 2 nominal base plant treatment rate of 85 kt/d (duplication of the Phase 1 processing train).
- Phase 3 nominal expansion plant treatment rate of each processing train from 42.5 kt/d to 60 kt/d for a combined 120 kt/d treatment rate.
- design availability of 92% (after ramp-up), which equates to 8,059 operating hours per year, with standby equipment in critical areas.
- sufficient plant design flexibility for treatment of all ore types at design throughput.

17.3 Design Criteria Summary

The overall approach was to design a robust processing facility that could treat a wide range of ore variability and operating conditions. The key project and ore-specific criteria for the plant design and operating costs are provided (Table 17.1).

The process plant will utilize a conventional milling operation consisting of crushing, grinding, desliming, and flotation operations consistent with other ultramafic nickel operations.

17.4 Processing Strategy

Selection and sizing of the crushing and grinding circuits was determined through comminution testwork performed at SGS-Lakefield. Testwork provided a crusher work index, Bond ball and rod mill indices, Bond abrasion index, SMC and JK Axb values for the selected samples. Ausenco elected to use the 75th percentile of each of these values in the design.

17.5 Head Grade

Each plant is designed to treat ore with a head grade of 0.32% Ni. The Phase 3 plant design will need to be revisited once Phase 1 is in operation to determine the optimum design point.

17.6 Flowsheet Development & Equipment Sizing

The process plant flowsheet design for the Crawford circuit was conceptually based on those of comparable large flotation plants and then confirmed or altered based on trade-off studies and metallurgical testwork. Figure 17.1 shows a process schematic for the Crawford plant (only 42.5 kt/d plant shown).

Details of the flowsheet design and the selection of major equipment for the process plants are discussed in the following sections.

Table 17.1: Summary of Process Plant Design Criteria

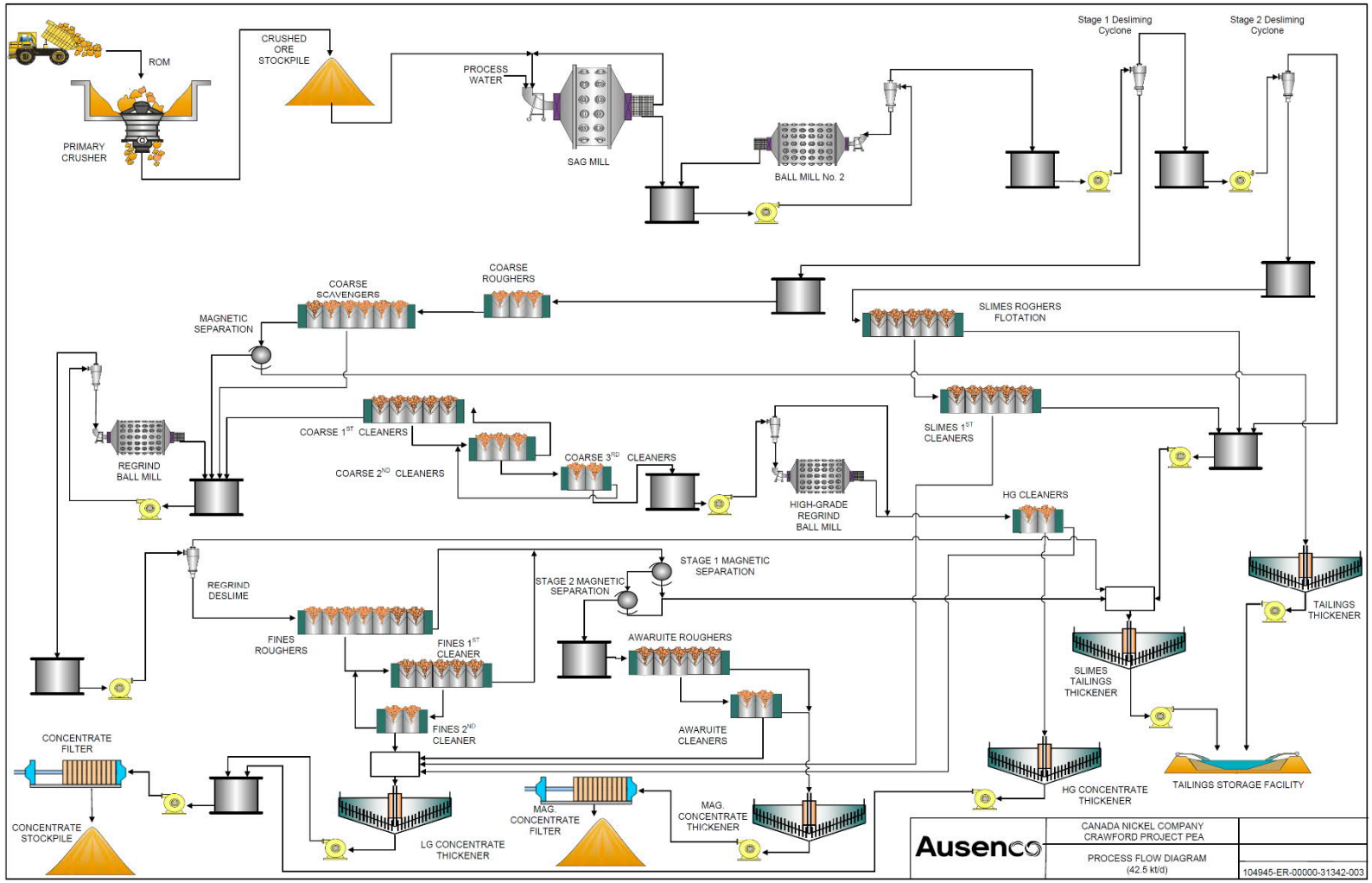
Criteria		Units	Phase 1	Phase 2	Phase 3
Period		year	1 to 3.5	3.5 to 7	8 to 18
Plant Throughput, per concentrator		kt/d	42.5	42.5	60
Plant Throughput, total		kt/d	42.5	85	120
		Mt/a	15.5	31.0	43.8
Crusher Availability		%	75	75	75
Crusher Throughput		t/h	2,361	4,722	6,667
Mill/Flotation Availability		%	92	92	92
Mill Throughput		t/h	1,925	3,850	5,435
Physical Characteristics (Design Values)	BWi	kWh/t	20.7	20.7	20.7
	RWi	kWh/t	14.6	14.6	14.6
	CWi	kWh/t	11.5	11.5	11.5
	JK Axb	-	60.7	60.7	60.7
	Specific Gravity	t/m³	2.57	2.57	2.57
Grind Size	P_{80}	µm	200	200	200
Head Grade		% Ni	0.32	0.26	0.25
		% Cr	0.62	0.63	0.58
		% Fe	6.02	6.46	6.58
Metal Recovery	Overall Nickel	%	50	44	39
	Magnetite	%	38	31	33
Concentrate Grade	High-grade Nickel	%	35	35	35
	Low-grade Nickel	%	12	12	12
	Magnetite	%	48	48	48
Ni Concentrate Filtration Rate		kg/m²/h	210	210	210
Ni Concentrate Thickening Flux		t/m²/h	0.25	0.25	0.25
Magnetite Concentrate Filtration Rate		kg/m²/h	210	210	210
Magnetite Concentrate Thickening Flux		t/m²/h	0.25	0.25	0.25
Tailings Thickening Flux	Slimes	kg/m²/h	0.35	0.35	0.35
	Coarse	kg/m²/h	1.0	1.0	1.0
Tailings Thickener Underflow Density	Slimes	% w/w	40	40	40
	Coarse	% w/w	55	55	55
PAX Consumption		g/t	149	149	149
MIBC Consumption		g/t	25	25	25
CMC Consumption		g/t	150	150	150
Sulphuric Acid Consumption (H_2SO_4)		g/t	1,666	1,666	1,666
Flocculant Consumption	Concentrate	g/t	10	10	10
	Slimes	g/t	60	60	60
	Coarse	g/t	30	30	30
SAG Mill Media Consumption		t/a	738	1,476	1,780
Ball Mill Media Consumption		t/a	916	1,833	2,484
Regrind Mill Media Consumption		t/a	427	855	1,200
HG Regrind Mill Media Consumption		t/a	9	17	25

17.7 Unit Process Selection

The process plant design is based on a flowsheet with unit process operations that are well proven in the minerals processing industry. The Crawford flowsheet incorporates the following unit process operations for each 42.5 kt/d plant:

- Ore from the open pit is crushed using a primary gyratory crusher (assisted with a rock breaker) to a crushed product size of nominally 80% passing (P_{80}) 68 mm. Crushed ore is fed onto the sacrificial conveyor, which then feeds the covered stockpile feed conveyor.
- A covered conical stockpile of crushed ore with a live capacity of 17 h, with three apron feeders, each capable of feeding 60% of the full mill throughput.
- A single 17.5 MW SAG mill, 11.0 m diameter (36 ft) with 7.3 m effective grinding length (EGL) (24 ft), utilizing trommel screens for classification and oversize recirculation.
- A single 17.5 MW ball mill, 8.1 m diameter (26.5 ft) with 13.4 m EGL (44 ft), in closed circuit with hydrocyclones, grinding to a product size of nominally 80% passing (P_{80}) 200 µm.
- Two-stage desliming circuit via hydrocyclones, targeting an overall mass split to slimes of about 25%, with the first stage to split mass according to an overflow particle size (P_{80}) of approximately 45 µm. Second stage to split mass to obtain an overflow with a P_{80} of 25 µm. The hydrocyclone sizes for each stage are 800 and 250 mm, respectively.
- Slimes rougher flotation consisting of five 100 m³ forced air tank flotation cells to provide 16.5 minutes of retention time.
- Slimes first cleaner flotation consisting of five 30 m³ forced air tank flotation cells to provide 23.1 minutes residence time.
- Coarse rougher flotation consisting of two trains of three (6 total cells) 300 m³ forced air tank flotation cells per train to provide 18 minutes of retention time.
- Coarse first cleaner, second cleaner, and third cleaner flotation consisting of four 130 m³, three 50 m³ and two 10 m³ forced air tank flotation cells to provide 30 minutes, 15 minutes, and 9 minutes of retention time, respectively.
- Coarse scavenger flotation consisting of two trains of four (8 total cells) 300 m³ forced air tank flotation cells to provide 36 minutes of retention time.
- Magnetic separation (first stage) on coarse scavenger flotation tailings, consisting of eight 3.6 m long low intensity magnetic separators (LIMS) for a nominal mass recovery of 11% of coarse rougher flotation plant feed.
- Magnetic concentrate regrind stage in an 8 MW ball mill, 6.7 m diameter (22.0 ft) with 9.6 m EGL (31.5 ft), operating in closed circuit with hydrocyclones, grinding to a product size of nominally 80% passing (P_{80}) of 45 µm.
- Fines rougher flotation consisting of five 160 m³ forced air tank flotation cells to provide 30 minutes of retention time.
- Fines first cleaner and second cleaner flotation consisting of five 50 m³ and two 10 m³ forced air tank flotation cells to provide 25.5 minutes and 16.5 minutes of retention time, respectively.

Figure 17.1: Process Plant Schematic (42.5 kt/d)

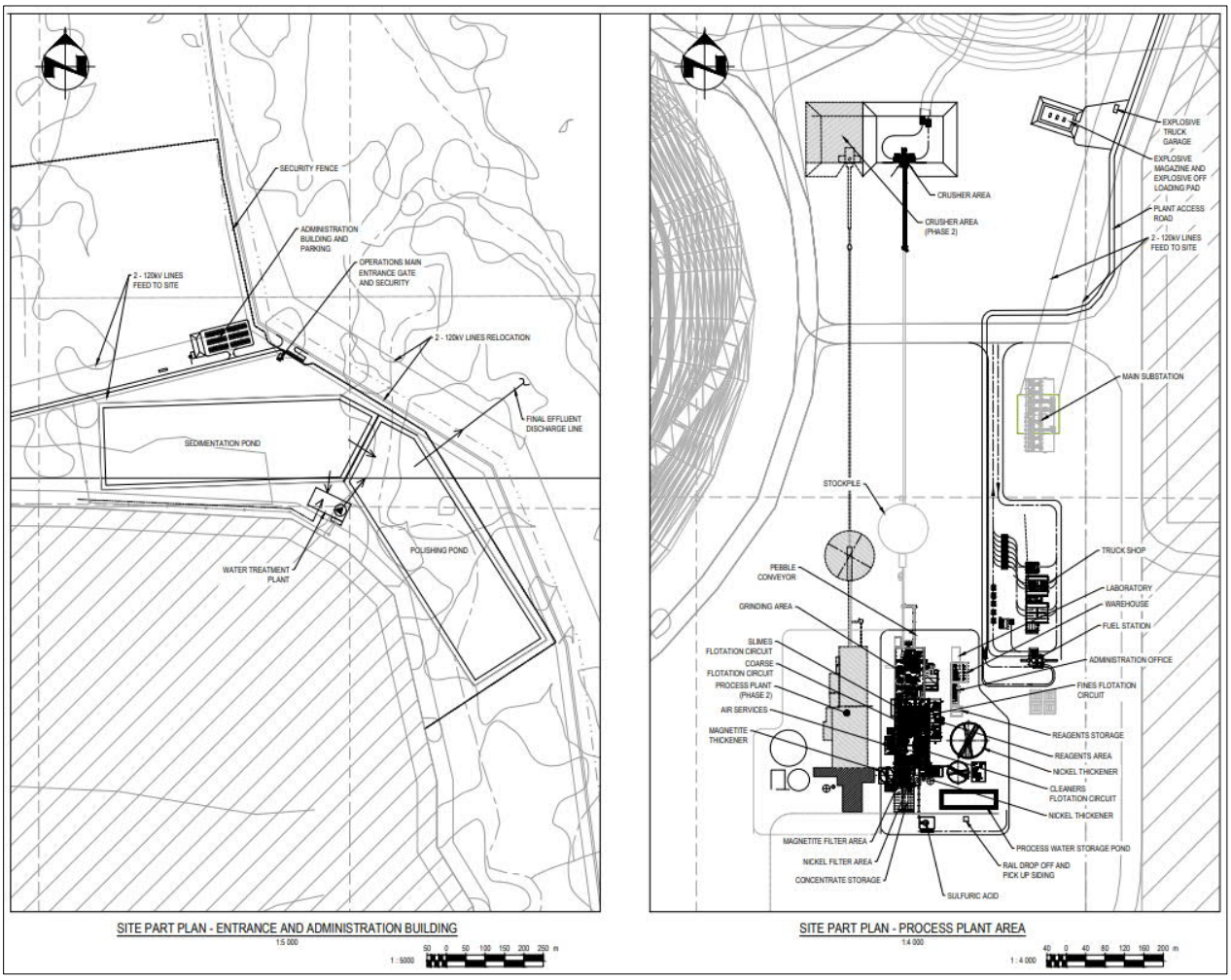


Source: Ausenco, 2021.

- Two stages of magnetic separation (second and third stage) on fines flotation tailings, consisting of five 3.6 m long LIMS magnetic separators for the second stage and an additional five 3.6 m long LIMS magnetic separators for the third stage, for a nominal stage mass recovery of 5.4% on plant feed.
- High-grade cleaner flotation consisting of two 4.3 m³ forced air flotation cells to provide 10 minutes of retention time.
- High-grade concentrate thickening in a 4 m diameter high-rate thickener.
- Low-grade concentrate (the combination of the slimes cleaner concentrate, the high-grade flotation tailings, and the fines cleaner concentrate) thickening in an 8 m diameter high-rate thickener.
- Thickening of the slimes tailings, dosed with a small portion of the thickened coarse tailings to improve settling properties, in two 75 m diameter high-rate thickeners.
- Thickening of the coarse tailings in a single 45 m diameter high-rate thickener.
- TSF for process tailings deposition that will impound tailings for the first 19 years of operation. Thickened coarse tailings and thickened slimes tailings are fed to the TSF using dedicated pipelines.
- Reagent mixing facilities for PAX (collector), MIBC (frother), CMC (depressant) and flocculants.
- Reagent off-loading facilities for MIBC (frother) and sulphuric acid.
- Process water and distribution system for reticulation of process water throughout the plant as required. Process water is collected in a process water pond that is predominantly supplied from the thickener overflows and tailings storage facility. Other sources include pit de-watering operations.
- Potable water is generated by treatment water from the freshwater tank in a reverse osmosis (RO) unit at the site. Potable water is distributed to the plant and for miscellaneous purposes around the site.
- Raw water, filtered using sand filters, distribution services to supply cooling water, gland water, reagent mixing water, firewater, etc.
- Plant, instrument and flotation air services and associated infrastructure.

Layouts of the process plant area and process plant are shown in Figures 17.2.

Figure 17.2: Layout of Process Plant Area



Source: Ausenco, 2021.

17.8 Comminution Circuit

17.8.1 Primary Crushing

Based on the design throughput and moderate competency ore characteristics, a single 60 x 89 gyratory crusher is considered the most suitable for the primary crushing duty for the initial plant and Phase 2 expansion. For the 120 kt/d expansion, two crushers will be required. The primary crushers will be located at the edge of the ROM pad. A partially buried crusher design has been selected to reduce ROM pad elevation (reduce mine haulage costs) without major excavation being needed.

Trucks will dump from both sides of a 323 m³ capacity live hopper above the crushers. Alternatively, ore can be rehandled and fed with a front-end loader (FEL).

Primary crushing will be installed inside an enclosed building. This will help minimize dust emissions and reduce noise. An overhead crane will be installed in the building for maintenance of the crushers. Auxiliary crusher equipment includes a mobile rock breaker and dust suppression system. Water sprays are used to minimize dust at the crusher bin, crusher discharge and crusher belt feeder. The gyratory crushers will crush ore to a product size of 80% passing 68 mm.

17.8.2 Secondary Crushing

For the 120 kt/d expansion only, secondary crushing will be required. The secondary crushing stage will be an open circuit, with a classification screen after primary crushing. The screen oversize feeds the three secondary cone crushers, model MP 1250 or similar, whose products report to the crushed ore stockpile. Screen undersize also reports to the crushed ore stockpile.

Secondary crushing will be installed in an enclosed building. An overhead crane will be installed in the building for maintenance of the crushers. Auxiliary crusher equipment includes a mobile rock breaker and dust suppression system. Water sprays are used to minimize dust at the crusher bin, crusher discharge and crusher belt feeder. The cone crushers will crush ore to a product size of 80% passing 46 mm.

17.8.3 Crushed Ore Stockpile

Crusher product will be conveyed from the crusher discharge vault by the variable-speed primary crusher discharge belt feeder and discharged onto the sacrificial conveyor. Ore is transferred via the stockpile feed conveyor to the crushed ore stockpile. A weightometer will be installed on the sacrificial conveyor to provide production rate data for the crushing circuit. A cross belt self-cleaning electromagnet, followed by a metal detector, is fitted over the sacrificial conveyor to detect and remove tramp steel prior to discharge onto the stockpile feed conveyor.

The stockpile will provide a minimum of 17 hours' live capacity at the nominal 42.5 kt/d SAG mill fresh feed rate. The live capacity reduces to 12 hours once the plant is expanded to the 60 kt/d throughput. In the event of the crushing circuit being out of operation for extended periods, a bulldozer can be used to reclaim the dead material in the stockpile to provide emergency feed to the milling circuit. Three apron feeders have been selected to reclaim ore from the stockpile, each able to deliver 60% of the nominal mill feed rate.

The stockpile will be enclosed to minimize fugitive dust emissions. The cover will be a dome of galvanized structural steel construction and cladding.

17.8.4 Grinding Design Criteria

The grinding circuit was designed to be capable of processing the required tonnage of 42.5 kt/d and will double throughput to 85 kt/d by mirroring the first line and thereafter will raise production to the ultimate rate of 120,000 t/d through the addition of secondary crushing and a third ball mill.

Flotation testwork and mineralogy have indicated that Crawford ores are relatively insensitive to grind sizes (P_{80}) up to about 180 µm in the laboratory. In order to achieve the specified design recovery, Ausenco has nominated a primary grind size target of P_{80} of 200 µm. This P_{80} has been selected because it is typical for sulphides to be found in finer size fractions under plant conditions when compared to laboratory testwork (as screens are used for sizing in the laboratory compared to hydrocyclones in the process plant).

Ausenco uses a power-based approach to determine grinding circuit power requirements. The approach is based on empirically derived models developed from a database of actual plant operation data and associated bench-scale testwork. Critical input parameters to the model are ore competency (measured by either JK drop weight Axb or SMC DWi values) and Bond work indices (CWi, RWi and BWi). Ausenco's power-based model predicts the milling efficiency of the various circuits based on the JK drop weight/SMC data, which is a measure of ore competency. The approach also considers the impact of ultramafic ores on the Bond BWi test results.

Specific energy and mill sizing determined using Ausenco's in-house method for the major ore types is shown in Table 17.2.

Table 17.2: Mill Design Criteria

Criteria		Units	Design 42.5 kt/d	Design 85 kt/d	Design 120 kt/d
Throughput (Nominal)		t/h	1,925	3,850	4,755
Mill Type			SAG	SAG	SAG
Shell Power Required		kW	13,086	26,172	34,714
No. of Mills			1	2	2
Mill Speed		% Nc	75	75	75
Ball Charge Volume	Nominal	% vol	12	12	15
	Maximum (design)	% vol	18	18	18
Total Charge Volume	Nominal	% vol	26	26	26
	Maximum (design)	% vol	35	35	35
Mill Diameter	Inside shell	M	11.0	11.0	11.0
Mill Length	EGL	M	6.7	6.7	6.7
Installed Motor Power		kW	17,500	17,500	35,000
Mill Type			Ball	Ball	Ball
Grind Size	P_{80}	µm	200	200	200
Pinion Power Required		kW	32,486	32,486	44,027
Number of Mills			2	2	3
Mill Speed		% Nc	75	75	75
Ball Charge Volume	Nominal	% vol	30.3	30.3	28.5
	Maximum (design)	% vol	33	33	33
Mill Diameter	Inside shell	m	8.1	8.1	8.1
Mill Length	EGL	m	12.7	12.7	12.7
Installed Motor Power		kW	17,500	17,500	17,500

The installed ball mill power of 17,500 kW incorporates allowances for drive train losses as well as a design contingency to account for the accuracy of the models, calculations and testwork used to determine the expected average pinion power.

The installed motor power for the SAG mills incorporates similar allowances, as well as an additional contingency to allow adjustment in the mill operating conditions to handle ore variability. These allowances and contingencies require an installed power of 17,500 kW per mill.

17.8.5 Reclaim, SAG & Ball Mill Circuit

The crushed ore will be reclaimed from the ore stockpile by three apron feeders onto the SAG mill feed conveyor. The feeders will be equipped with variable speed drives.

A SAG mill feed weightometer will be installed on each SAG mill feed conveyor to provide feed rate data for control of the reclaim feeders. The reclaimed crushed ore will be fed at a controlled rate to the SAG mill.

Discharge from the SAG mill will gravitate through a trommel screen. Oversized pebbles from the trommel screen (scats) will be recycled back onto the mill feed conveyor. A cross belt self-cleaning electromagnet removes broken and worn mill balls and other tramp steel from the scats stream. Pebbles will be reintroduced onto the mill feed conveyor via the recycle pebble conveyors. Undersize from the SAG trommel screen will gravity flow into the cyclone feed hopper.

A pebble circulating load of 18% to 22 % of the fresh feed rate has been assumed in the design, based on typical industry experience with ores of similar competency. The conveyors are designed to handle peak loads of up to 30% of fresh feed.

The SAG mill discharge slurry will be pumped via dedicated cyclone feed pumps to the ball mill cyclone cluster in Phase 1 and three clusters in Phase 3, each operating in a closed-circuit configuration with a single ball mill. Water is added to the cyclone feed hopper as needed to achieve the required cyclone feed pulp density.

Hydrocyclone underflow from each cluster will gravity flow to a dedicated 17.5 MW twin-pinion ball mill (two 8.75 MW motors operating in parallel). Discharge from each ball mill will gravity flow through a trommel screen, into the cyclone feed hopper for reclassification. Cyclone overflow will gravity flow to the first stage deslime cyclone feed hopper.

17.8.6 Mill Circuit Classification

The classification circuit has been designed for a nominal circulating load of 300%. This is a typical design value for material of similar characteristics and target grind size and is widely used in the industry for SAB circuits. To avoid damage to the cyclone feed pumps and cyclone clusters, the SAG mill discharge slurry first passes through a trommel screen to remove pebbles and grinding media; the undersize flows into the hydrocyclone feed hopper.

SAG and ball mill discharge slurries will be combined in the ball mill hydrocyclone feed hopper and then pumped to a hydrocyclone cluster to a target overflow P_{80} of 200 µm.

Hydrocyclone overflow will report as feed to a desliming circuit prior to slimes flotation, while the coarse hydrocyclone underflow from each of the two clusters will report to a dedicated ball mill for further grinding.

17.8.7 Deslime Circuit

A two-stage desliming circuit will deslime the hydrocyclone overflow from ball mills to remove fine fibrous particles. This is critical to achieve optimal flotation kinetics in the coarse flotation circuit. The deslime circuit accomplishes this with two-stage hydrocyclone clusters. The two-stage desliming circuit targets an overall mass split to slimes of about 20%. The hydrocyclone sizes for each stage are 800 and 250 mm.

Overflow from the first stage of desliming passes through a horizontal trash screen to remove large particles that could potentially block the smaller cyclone in the second stage. Trash screen underflow is feed for the second desliming stage.

The underflows from the first stage feeds the coarse rougher flotation circuit. The stage 2 hydrocyclone overflow flows by gravity to the slimes flotation circuit and its underflow feeds the slimes flotation circuit.

17.9 Flotation Circuit Design

Mineralogical examination and lab-scale testwork has revealed that a majority of the nickel sulphide in the ore is recoverable, with adequate concentrate grades, through flotation at a P_{80} of 200 µm. However, the use of magnetic recovery stages and regrind has shown that additional nickel sulphide and magnetite recovery is achievable.

The magnetic recovery stage has the main purpose of recovering magnetite; however, nickel that is in various alloy forms, predominately awaruite which is a naturally occurring nickel/iron alloy, could be recovered via a dedicated fines flotation recovery circuit.

A subsequent regrind stage and magnetic recovery is required to increase liberation of the nickel and magnetite that allows for higher rates of gangue rejection. The addition of a flotation stage on the regrind circuit product allows for recovery of additional nickel sulphides and allows for higher rates of gangue rejection.

17.9.1 Circuit Type & Size

The flotation circuit selected to concentrate Crawford ore consists of the following:

- slimes rougher and one-stage cleaner flotation.
- coarse rougher, scavenger and three-stage cleaner flotation.
- high-grade regrind and cleaning of coarse flotation concentrate.
- magnetic separation of coarse flotation tailings.
- regrind and fines rougher and two-stage cleaner flotation.
- secondary and tertiary magnetic separation.

Slimes, high-grade flotation tailings, and fines flotation concentrates are all combined as final low-grade concentrate. The high-grade concentrate is the high-grade flotation concentrate. Coarse flotation tailings report to the tailings thickener while the other tailings report to the slimes thickener. The residence times for the nickel flotation circuit have been based on the testwork performed on Crawford ore.

The testwork flotation and design residence times are summarized in Table 17.3.

Table 17.3: Summary of Flotation Residence Times

Flotation Stage	Locked-Cycle Test Time (min)	Scale Factor	Specified Design Time (min)
Slimes Roughers	5	3.3	16.5
Slimes First Cleaners	7	3.3	23.1
Coarse Roughers	6	3.0	18
Coarse Scavengers	12	3.0	36
Coarse First Cleaners	10	3.0	30
Coarse Second Cleaners	5	3.0	15
Coarse Third Cleaners	3	3.0	9
Fines Roughers	10	3.0	30
Fines First Cleaners	8	3.2	25.5
Fines Second Cleaners	5	3.3	16.5
High-Grade Cleaner	3	3.3	9.9

17.10 Flotation Circuit Configuration

17.10.1 Slimes Flotation

Stage 2 deslime hydrocyclone underflow will be fed by gravity to the slimes rougher conditioning tank, where flotation reagents will be added. The slimes flotation circuit consists of five 100 m³ tank flotation cells for Phase 1 and seven cells for Phase 2. The cells will be in a single cell arrangement with an elevation change between each cell. Additional dosing points for the flotation reagents will be located along the slimes flotation banks.

Concentrate from the slimes flotation cells will be fed to the slimes cleaning circuit. Slimes rougher tailings will be pumped to the slimes tailings mixing tank.

The slimes first cleaner stage consists of five 30 m³ tank flotation cells for Phase 1 and six cells for Phase 2, operating in an open circuit configuration. Slimes first cleaner flotation tailings are pumped to slimes tailings mixing tank and the concentrate is fed to the low-grade concentrate thickener.

The reagents added will consist of a combination of PAX (collector), MIBC (frother) and CMC (depressant).

17.10.2 Coarse Rougher Flotation

Stage 1 deslime hydrocyclone underflow is fed via gravity to two parallel rougher conditioning tanks, where flotation reagents will be added. The conditioning tanks will gravity flow to the coarse rougher flotation cells, which are connected in series. Two trains of three 300 m³ forced-air tank flotation cells have been selected to provide the required residence time for the rougher flotation. An additional two 300 m³ will be added in Phase 2. The cells will be in a single cell configuration with an elevation change and level control between each cell. Additional dosing points for flotation reagents will be located along the rougher banks.

Concentrate from each train of the rougher cells will feed the coarse cleaning circuit. Rougher tailings from each train will flow by gravity to the coarse scavenger stage.

The reagents added will consist of a combination of PAX (collector), MIBC (frother), CMC (depressant) and sulphuric acid.

17.10.3 Coarse Scavenger Flotation

The coarse scavenger flotation stage consists of eight 300 m³ tank flotation cells arranged in two trains. An additional eight cells will be added for the Phase 2 expansion. The cells will be in a single cell configuration with an elevation change and level control between each cell. Coarse scavenger tailings will be fed to magnetic separation, while the concentrate will feed the magnetic concentrate regrind stage.

The reagents added will consist of a combination of PAX (collector), MIBC (frother), CMC (depressant), and sulphuric acid.

17.10.4 Coarse Cleaner Flotation

Two different streams feed the coarse first cleaner flotation circuit:

- coarse rougher flotation concentrate.
- coarse second cleaner flotation tailings.

The coarse first cleaner stage consists of four 130 m³ tank flotation cells for Phase 1 and five for Phase 2. Coarse first cleaner flotation tailings are pumped to the regrind circuit described in Section 17.10.6, and concentrate feeds the coarse second cleaner flotation stage.

The coarse second cleaner stage consists of three 50 m³ tank flotation cells with a single additional cell for Phase 3. The coarse third cleaner stage consists of two 10 m³ tank flotation cells and is also sufficient for Phase 2. The second and third stages are configured such that third cleaner flotation tailings flow by gravity back to the head of the second cleaner stage.

The coarse second and third cleaner stage tailings are recirculated to the previous cleaner stage. The final concentrate from the coarse third cleaner stage reports to the high-grade regrind stage.

The reagents added will consist of a combination of PAX (collector), MIBC (frother) and CMC (depressant).

17.10.5 1st Stage Magnetic Separation

A low-intensity magnetic separation (LIMS) circuit is used to treat the tailings of the coarse scavenger stage. The function of this circuit is to recover nickel contained in magnetic alloys (primarily awaruite) that can be found in the sulphide rougher and cleaner tails.

Testwork performed using magnetic separation established mass recovery and approximate concentrate nickel grade design criteria. Other parameters are based on benchmarking and vendor recommendation.

The selected design criteria are summarized in Table 17.4.

Table 17.4: Summary of Magnetic Concentrate Recovery Circuit Design Loadings

Magnetic Separation Stage Feeds	Magnet Strength Gauss	Magnet Linear Loading t/h/(m drum)	Magnet Volumetric Loading m³/h/(m drum)	Magnet Configuration
First Stage: nickel sulphide rougher & first cleaner tailings	1,000	26	80	Counter-current
Second/Third Stages: magnetic sulphide scavenger tailings	1,000	26	80	Counter-current

Nickel sulphide rougher and cleaner flotation tailings will be pumped to two trains, each consisting of four single drum magnetic separators (3.6 m long and 1.2 m diameter) per train via a feed distributor. An additional two magnetic separators will be installed for Phase 3 per concentrator. The number of separators selected will allow for variations in throughput and magnetic recoveries.

Magnetic concentrate will gravity flow to a common hopper and be pumped to the regrind circuit, specifically the regrind mill cyclone feed hopper.

The first stage magnetic tailings will gravity flow to a common hopper for all three stages of magnetic tailings (non-mags) which will be pumped to the coarse tailings thickener.

17.10.6 Magnetic Concentrate Regrind

A closed regrind circuit is used to grind the combined coarse first cleaner flotation tailings, coarse scavenger flotation concentrate and first magnetic separation stage concentrate streams.

The regrind mill discharge first passes through a trommel screen to remove scats. The underflow is fed to the regrind mill cyclone feed hopper and then pumped to a cluster of twenty-one (nineteen operating and two standby) 250 mm hydrocyclones to achieve a product size of nominally 80% passing (P_{80}) of 45 µm. Hydrocyclone overflow will report as flotation feed to magnetic sulphide scavenger flotation, while the underflow will report to the regrind ball mill for further grinding.

The regrind mill fresh feed rate is 482 t/h. A single 6.7 m (22 ft) diameter, 9.6 m (31.5 ft) long 8 MW regrind ball mill was sized using a design BWI of 16.6 kWh/t. An identical regrind circuit will be installed for Phase 2 and the combined regrind mill circuits can process the total 120 kt/d throughput for Phase 3.

17.10.7 Fines Flotation

The fines flotation circuit consists of a rougher flotation stage fed by the regrind deslime underflow stream and two stages of cleaning.

The fines rougher stage consists of five 160 m³ tank flotation cells, with an additional cell for Phase 3. Fines rougher flotation tailings feed the two magnetic separations stages described in section 17.10.7, and concentrate feeds the fines first cleaner flotation stage.

The fines first cleaner stage consists of five 50 m³ tank flotation cells in Phase 1 and two cells in Phase 3. Fines first cleaner flotation tailings are combined with the fines rougher stage tailings to feed the two magnetic separation stages. The concentrate feeds the fines second cleaner flotation stage.

The fines second cleaner stage consists of two 10 m³ tank flotation cells sufficiently sized for both Phase 1 and 3. The fines second cleaner stage tailings feed the fines first cleaner flotation stage, forming a closed circuit, and the concentrate feeds the HG regrind mill.

The reagents added will consist of a combination of PAX (collector), MIBC (frother) and CMC (depressant).

17.10.8 Second and Third Stage Magnetic Separation

Two stages of low-intensity magnetic separation cleaning circuits are used to treat the tailings of the fines rougher and first cleaner flotation stages. The function of this circuit is to recover magnetite and further reject gangue material.

The only design criterion that has been established at this stage is approximate mass recovery and an approximate concentrate iron grade. Other parameters are based on benchmarking and vendor recommendation.

The fines sulphide rougher flotation tailings will be first pumped to a train of four single-drum magnetic separators (3.6 m long and 1.2 m diameter) via a feed distributor (second stage). The recovered concentrate is then pumped to a second train of two single-drum magnetic separators (3.6 m long and 1.2 m diameter) via a feed distributor (third stage). The number of magnetic separators selected will allow for variations in throughput and magnetic mass recoveries.

Magnetic third stage concentrate will be pumped to the magnetite concentrate thickener. Magnetic separation tailings (non-mags) from both stages will feed the slimes thickener.

17.10.9 High-Grade Regrind

An open regrind circuit is used to grind the combined coarse third cleaner flotation concentrate. It is composed of one 3 m (10 ft) diameter, 5.3 m (17.4 ft) long 650 kW regrind ball mill and one 250 mm cyclone cluster for both Phase 1 and Phase 3. The material first feeds a scalping cyclone cluster, whose cut size is 25 µm. Cyclone underflow feeds the regrind mill. The regrind mill discharge first passes through a trommel screen to remove scats. The undersize reports to the cyclone feed hopper. Hydrocyclone overflow will report to the high-grade flotation circuit, while the underflow will report to the regrind ball mill for further grinding.

17.10.10 High-Grade Cleaner Flotation

The high-grade regrind stage product feeds a high-grade cleaner flotation circuit.

The high-grade cleaner stage consists of two 4.3 m³ flotation cells subtility sized for both Phase 1 and Phase 3. The tailings are directed to the low-grade concentrate thickener and the concentrate is the only product feeding the high-grade concentrate thickener.

The reagents added will consist of a combination of PAX (collector), MIBC (frother) and CMC (depressant).

17.11 Nickel Concentrate Thickening, Storage & Filtration

Larger thickeners have been selected to accommodate the additional concentrate produced for a plant throughput from 42.5 to 60 kt/d (*i.e.*, equivalent to 120 kt/d through both concentrators).

High-grade concentrate will be thickened to approximately 60% w/w solids in a 4 m diameter above-ground high-rate thickener. A settling rate of 0.25 t/m²/h has been selected as the basis of design for nickel concentrate thickener based on the testwork results.

Low-grade concentrate will be thickened to approximately 60% w/w solids in an 8 m diameter above-ground high-rate thickener. A settling rate of 0.25 t/m²/h has been selected as the basis of design for nickel concentrate thickener based on the testwork results.

Magnetic concentrate will be thickened to approximately 60% w/w solids in a 30 m diameter above-ground high-rate thickener. A settling rate of 0.25 t/m²/h has been selected as the basis of design for nickel concentrate thickener based on the testwork results.

The concentrate storage tanks prior to filtration will have been sized based on 24-hours of residence time for the 42.5 kt/d plant. The concentrate storage allows for routine maintenance of the concentrate filter.

A single horizontal recessed-plate diaphragm pressure filter will batch filter both the low-grade and high-grade nickel concentrates. A single vertical recessed-plate diaphragm pressure filter will filter the magnetite concentrate.

Filter cake is stockpiled and loaded onto rail cars via a front-end loader.

Expansion to a plant throughput of 120 kt/d will require additional horizontal recessed-plates to be installed within the existing concentrate filters within both concentrators.

17.12 Tailings Disposal

The design basis chosen for this level of study includes a coarse tailings thickener and a slimes tailings thickener. Disposal of thickened tailings from each thickener will be fed to the TSF in dedicated pipelines. Water will be recovered from the TSF surface and recycled to the plant as reclaim water.

The coarse tailings thickener design has been sized on a settling rate of 1.0 t/m²/h and results in the selection of a 45 m high-rate thickener considering total throughput for Phase 3. The feed slurry to this thickener consists of the magnetic separation tailings. The slurry will be thickened to a target density of 55% w/w solids.

The coarse tailings thickener underflow will be split into two streams. One pump will send a small portion, approximately 10% w/w, to the slimes tailings mixing tank. The majority of the underflow will be sent to the TSF. There will be two sets of two pumps in series for the first year of operation and will increase to two sets of four pumps in series for the subsequent years of operation (total of eight). The Phase 3 coarse tailings thickener will require all eight pumps at start-up. The thickened coarse tailings slurry will be pumped to the TSF via a 7.5 km pipeline pipe. Process water from the coarse thickener overflow will flow by gravity to the plant process water storage pond.

Slimes rougher and cleaner flotation tailings, magnetic separation stage 1 and 2 tailings and fines flotation tailings will feed the slimes tailings thickeners, which have been sized using a settling rate of 0.35 t/m²/h and the selection of two 75 m high-rate thickeners. The slurry will be thickened to a target density of 35% w/w solids. Process water from the slimes thickeners overflow will flow by gravity to the plant process water storage pond.

For Phases 2 and 3, additional pipelines will be required for both coarse and slimes tailings. Reclaim water from the TSF will be pumped back to the process water pond via barge pumps and a 4 km HDPE water pipeline.

17.13 Reagents

Reagents for the project are listed below.

Collector – Potassium amyl xanthate (PAX) is a sulphide mineral collector and will be supplied in 1000 kg bulk bags as a dry reagent. PAX will be shipped by road to site and offloaded by forklift. PAX will be stored in the reagent's storage area of the warehouse facility and delivered to the PAX mixing area. PAX bulk bags will be lifted by the common reagents area overhead crane and loaded into the mixing tank by way of a bag splitter. Water is added to the agitated tank to produce a solution concentration of 20% w/w. The diluted mix is transferred to the PAX storage tank by way of pump. The PAX solution is stored in a day tank, where it is reticulated around the plant in a ring main system using the ring main pumps (duty/standby arrangement).

Frother 1 – Methyl isobutyl carbinol (MIBC) will be supplied by bulk tankers and off-loaded via pump into a storage tank. The storage tank will have capacity for several days of consumption at design flow rates. The frother will be distributed to the flotation circuit dosing points by dedicated metering pumps.

Depressant – Carboxy methyl cellulose (CMC) is used as a gangue depressant in this flotation circuit and will be supplied in 1000 kg bulk bags as a dry reagent. CMC will be shipped by road to site and offloaded by forklift. CMC will be stored in the reagents storage area of the warehouse facility and delivered to the CMC mixing area. CMC bulk bags will be lifted by the common reagents area overhead crane and loaded into the storage hopper by way of a bag splitter. Loose CMC is transported via screw feeder to the CMC mixing tank. Water is added to the agitated tank to produce a solution concentration of 0.5% w/w. The diluted mix is transferred to the CMC storage tank by way of pump. The depressant will be distributed to the flotation circuit dosing points by dedicated metering pumps.

pH Modifier – Sulphuric Acid (H_2SO_4) will be supplied by bulk tankers and off-loaded into a storage tank; expansion will require an additional storage tank. The sulphuric acid will be distributed to the flotation circuit dosing points by multiple centrifugal pumps.

Flocculant – A flocculant mixing, storage and dosing system located in the reagent preparation area will be provided to facilitate concentrate thickening. Flocculant will be supplied in 25 kg bulk bags and will be shipped as a dry reagent. The flocculant will be manually loaded into the concentrate flocculant storage hopper and fed via screw feeder to the concentrate flocculant mixing tank where it is diluted to 0.25% w/w. The diluted mix is transferred to the concentrate flocculant storage tank by way of a pump. The flocculant will be pumped via dosing pumps to an inline mixer where the solution is further diluted to 0.025% w/w and fed to the concentrate thickener.

Grinding Media – Forged carbon steel grinding media will be delivered to site in 20-tonne containers. The balls will be unloaded into a storage bin via a vendor-supplied, hydraulically-operated container unloader. Overhead cranes in the primary milling and regrind areas will be used to load steel balls into the SAG mill, ball mill and regrind mill.

17.14 Air Services

17.14.1 Process Air

The flotation blowers will supply low-pressure process air to the flotation cells. The blowers will generate air at the highest pressure required by the flotation cells. Pressure reducers will be used to step-down the pressure to the flotation cells requiring lower pressures. Multiple-stage, centrifugal type blowers will be used with a "blow-off" arrangement to adapt to fluctuations in flotation air demand.

The blowers will be housed inside their own room to reduce plant noise to an acceptable level. The room will have ventilation for cooling.

17.14.2 Plant and Instrument Air

Three rotary screw air compressors will provide intermediate pressure compressed air for plant and instrument air requirements. There will be two duty and one standby compressor operating in lead-lag mode. Plant air will be stored in the plant air receivers to account for variations in demand prior to being distributed throughout the plant.

A fourth air compressor is dedicated to the concentrate filter press requirements. It is equipped with its own dedicated air dryer and receivers.

Valving will allow the standby plant compressor to be used as a standby filter compressor as required.

18 PROJECT INFRASTRUCTURE

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

18.1 Introduction

The overall site plan (Figure 18.1) shows the major Project facilities, including the open pit mines, TSF, waste rock facilities, polishing pond, mine services, access road, process and non-process buildings.

The Project process area consists of crushing, stockpile conveyor, coarse ore stockpile and reclaim, SAG and ball mill grinding circuit, nickel flotation circuit including regrind, nickel and magnetite concentrate thickening, filtration and storage, rail car/truck loadout, tailings thickening facility, reagents, and ancillary services.

The layout of the plant and all associated facilities will require the relocation of the Highway 655 and 500 kV power transmission line located on the west of the mine site. The two 120 kV power transmission power lines will be relocated on the east side of the mine site.

The site layout considers the site topography and limits imposed by the locations of the pit, stockpiles and waste dumps subject to the above constraints. The grinding area of the process plant is located on low-bearing capacity ground and will require deep foundation civil design and take advantage of gravity flow where possible.

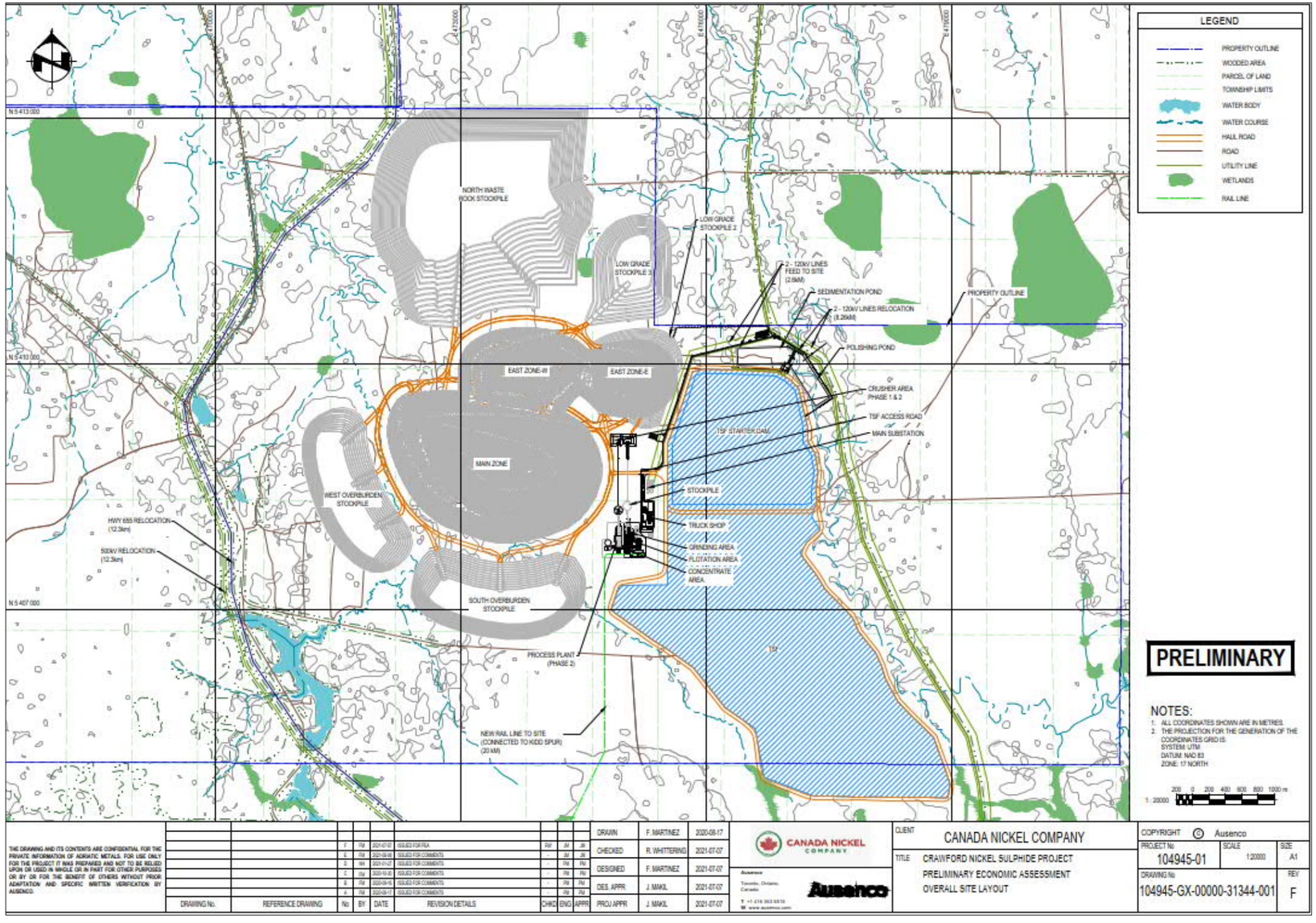
18.2 Site Power Supply

Hydro One Networks Inc. (HONI) will provide electrical power to the mine site.

- Phases 1 and 2 will require the following:
    - a twin 120 kV, 2.5 km long overhead powerline to be constructed, that would be connected as a tee-off to H6T and H7T Lines northeast of the property and runs up to the process plant main 120 kV substation.
    - a section of about 8.6 km of those two lines will need to be relocated further east to leave room for the TSF.
    - a section of 12.3 km of the 500 kV line that runs parallel to Highway 655 will need to be relocated further west of the property as it currently runs through the pit outline.
- Phase 3 will require a new 38.0 km, 230 kV line from Timmins to accommodate 250 MW of load.

To accommodate the different load profiles and changes for the expansion, the initial phases will have dual voltage transformers (double tap) 120/230:13.8 kV transformers. The new 120 kV/230 kV substation and six main transformers will be installed near the SAG mill feed conveyor. The 13.8 kV medium voltage network will be used for the primary electrical distribution and for feeding large loads such as the SAG mill and ball mills.

Figure 18.1: Overall Site Layout



Source: Ausenco, 2021.

The 13.8 kV distribution circuits run from the main electrical room E1 (located adjacent to 120 kV outdoor substation) to secondary electrical rooms located close to the areas served. In these secondary electrical rooms, the 13.8 kV distribution voltage will be converted to 4.16 kV and 600 V using 13.8-4.16 kV and 13.8-0.6 kV indoor dry-type transformers. For the mine circuit, an isolation transformer 13.8-13.8 kV will be used to separate the neutral grounding circuit of the portable substations from the main plant ground system.

In case of power failure, two 13.8 kV emergency diesel generators will automatically start and supply power for all essential plant loads. The generators will be located at the main electrical room E1 conveniently located for distribution of power throughout the plant using the 13.8 kV network. Uninterruptable power supplies (UPS) and DC battery systems will be provided in the various electrical rooms for essential protection and control equipment.

Power factor correction equipment and harmonic filters will be located near the main electrical room and connected to the main 13.8 kV switchgear to ensure that the electrical load, as seen by HONI, meets their requirements.

18.3 Propane Gas

The use of propane gas is considered for heating buildings (process plant and workshops) in this study and is included as part of the operating cost estimate. Deliveries will be by tanker truck.

18.4 Rail Spur

A rail spur that services the process plant is proposed for the Project. The total length of the rail spur is approximately 20.0 km and will connect to the Glencore Kidd Operations rail line. The rail spur consists of a fuel delivery track and a freight delivery siding south of the process plant.

18.5 Roadways

Access to Crawford site will be along Highway 655. Approximately 13.2 kilometres of the highway will need to be rerouted as it currently runs through the pit envelope.

The Crawford on-site roads will be constructed of crushed waste rock available from site and naturally available materials. As discussed in Section 16.5.6 previously, a roadstone crusher will be utilized to provide aggregate for resurfacing haul roads.

18.6 Process Plant

The process plant area consists of the crushing facility, covered stockpile, and process plant building. The overall process plant enclosed structure is approximately 350 m long, and consists of four connected buildings: grinding, flotation and magnetic separation, and concentrate thickening. These are described below.

The primary crushing facility is closely located to the open pit, to the east. The crushed ore is conveyed to a covered stockpile that is approximately 40 m high x 96 m diameter. The crushed ore conveyor from the crushed ore tunnel to the crushed ore transfer station is approximately 200 m long. The stockpile feed conveyor extends a further 800 m from the transfer station to the stockpile.

From the covered stockpile, the ore is conveyed via apron feeders through a reclaim tunnel and via a 280 m long SAG mill feed conveyor into the grinding area. The feed to the SAG mill is at 90°, which helps reduce the size of the grinding building. The grinding building consists of a SAG mill, a ball mill, a regrind mill, desliming cyclones and an overhead crane. It is 100 m

long x 81 m wide x 47 m high. The grinding area electrical room E3 is connected at the east side. The plant control room will be located at an elevated position adjacent to the hydrocyclone cluster and will have aluminum-framed windows for viewing into the process plant. In particular, the grinding and flotation areas will be easily viewable from the control room. The lunchroom is located below the control room.

The slimes and nickel flotation building is located south of the grinding circuit. It contains the slimes flotation, nickel roughers and cleaner cells, magnetic separators, two overhead cranes, and is 138 m long x 74 m wide x 29 m high. The reagents mixing area is connected to the east of the nickel flotation building. The process air blowers, plant air compressors, and electrical rooms E4 and E5 are connected to the west side of the building.

To the south of the slimes and nickel flotation building is the concentrate thickening building, which is 42 m long x 35 m wide x 19 m high. The water pumphouse and process water pond are east of the concentrate thickening building. The tailings thickeners are adjacent to the process water pond, on the east side.

During the plant expansion in Year 4 of operations, a second train of the crushing facility, stockpile, grinding building, slimes and nickel flotation building, concentrate thickening building, process water pond, and tailings thickeners will be duplicated and built to the west of the original process plant. The ultimate plant expansion to 120 kt/d will require secondary crushing and a third ball mill which will be installed in separate buildings.

18.7 Impoundments

Materials excavated from the pit will be impounded in the following facilities:

- three separate low-grade stockpiles.
- two separate overburden stockpiles.
- a single waste rock stockpile.
- a co-disposal in-pit storage facility using the mined-out Main Zone pit.

Details regarding each of these impoundments are provided in Section 16.

18.8 Tailings Storage Facility

18.8.1 Tailings Storage Facility (TSF) Introduction

The area of the TSF is approximately 12 km^2 and lies to the east of the open pit. The layout of the TSF is shown in Figure 18-1 and is constrained by the open pit, the mill complex and a creek on the west, and a creek on the east. On the north and south the TSF is constrained by the property boundaries. Tailings will be pumped from the mill into the TSF area as slurry and deposited on long beaches. Excess and runoff water will be collected in a small pond within the TSF and pumped to a water recycle pond before being pumped back to the process plant for reuse.

18.8.2 Conceptual TSF Design

The TSF dams will be constructed from gravel and waste rock from the open pit. During later dam raises recovered tailings will also be used as fill material.

The TSF starter dam will be constructed at the north of the area shown in Figure 18-1. The footprint of this dam has been selected in order to minimize haulage distances from the pit. This dam will reach a final height of 36 m, providing storage for 47 Mm^3 tailings, which is equivalent to 51 months production.

The ultimate dam encompasses the starter dam and rises to a final height of 73 m, providing 362 Mm^3 storage capacity. The entire starter dam and initial 49 m of the ultimate dam have been designed using the downstream method. Currently, the slope designs are conservative at 8H:1V on the downstream and 3.5H:1V on the upstream sides. The crest will be 15 m wide, to provide adequate space for the vehicles, safety berms and the slurry deposition bench. With these parameters, construction of the starter dam will require 26 Mm^3 gravel and rockfill while the ultimate dam will require a further 125 Mm^3 for a total of 152 Mm^3.

The final 24 m of the ultimate dam have been planned, using the upstream construction method, utilizing mainly coarse recovered tailings. The selected construction method will be dependent on the ability to create a coarser tailings stream that will act in a drained manner.

At the start of construction unsuitable materials will be removed from the dam's footprint as required by the design. The footprint maintains a safe distance from the shoulder of the adjacent river valley.

The low permeability dam element will either be a clay core or low-permeability liners (geosynthetic clay liner (GCL), high-density polyethylene (HDPE), or bituminous geomembrane (BGM)) on the upstream slope.

The TSF is located in a natural flat area. The design will be adjusted in the next design phase based on updated survey information. Final configuration of the dams will be established once dam fill and dam foundation soil properties are available following sampling and field and laboratory testing.

The design of the TSF should be based on governmental guidelines for the design of such facilities and generally accepted good practice. In particular, the design of the TSF should be carried out with reference to the following documents:

- Canadian Dam Association (CDA) Dam Safety Guidelines (CDA 2013)
- Ministry of Natural Resources Guidelines (LRIA, 2011)
- Global Tailings Standard

An engineer of record (EOR) for the dams should be appointed. Yearly dam safety inspections should be conducted and dam safety reviews at intervals suitable to the hazard potential.

18.8.3 Tailing Deposition

Slurry deposition of tailings has been assumed at this stage of design. Tailings should be deposited from all perimeter dams to create upstream tailings beaches to push the pond further to the upstream of the TSF perimeter dams to improve the stability of the dams. A small pond will be maintained within the TSF to collect excess water and precipitation. Water will be pumped to a recycle pond constructed downstream of the TSF. There will be two tailing streams from the mill (coarse and fine tailings) in later stages of production. Coarse cyclone generated tailings will be deposited to provide a coarse tailings beach that in support of centerline or upstream construction at some locations. Coarse tailings will be 20% to 30% of the total production at that time.

The starter tailings dams will store approximately 47 Mm^3 and the final TSF dams will store approximately 362 Mm^3 of tailings. Early testing has indicated that the tailings will be non-acid-generating (NAG). The specific gravity of the waste rock is 2.7 to 3.0.

18.8.4 TSF Closure Requirements

After mine closure, the plan is to vegetate the downstream slopes and top surface of TSF and allow drainage from the top of the TSF through a wide natural drainage channel with no pond.

Reclamation and closure of the TSF should be based on the general objectives below:

- Reclamation objectives will be incorporated into all aspects and stages of facility development including, the design, planning, construction, and operation of the facility, throughout its operating life.
- Progressive reclamation will be implemented where possible.

Upon cessation of operations, the TSF should be decommissioned and rehabilitated to:

- allow for future land use consistent with regional objectives or other uses as approved by the relevant authorities.
- ensure that long-term physical and chemical stability is provided.
- manage surface water and drainage paths.
- protect public health and safety.
- minimize adverse environmental impacts.
- meet all legal requirements and comply with all relevant operating licenses, legislation, policies, codes of practice and commitments to stakeholders.

The primary objective of the reclamation and closure activities will be to transform the TSF area to its pre-mining usage and capability, as much as is reasonably practicable. Groundwater monitoring wells should be retained for use as long-term monitoring devices. Post-closure requirements will include the annual inspection of the TSF by a geotechnical engineer and periodic dam safety reviews.

18.8.5 Water Supply

The initial tailings production is expected to be approximately 42,500 t/d. The daily water requirement for this volume of tailings was estimated to be 131,750 m^3. The mill is expected to start a couple years after the start of mining. This provides the opportunity to store water for the initial year of tailings production since little recycle water can be recovered during the first year of tailings deposition. Sources of water for the recycle pond and mill is the TSF, open pit dewatering, stockpile drainage ditches and the adjacent on-site creeks. The recycle water pond should also have capacity to provide water to the mill during the winter months when there is little recovery of water from the TSF due to freezing conditions. The recycle pond should also have capacity to handle the spring melt within the TSF before excess water can be treated and released to the environment. The starter dam recycle water pond should be located south of the starter dam of the TSF.

18.8.6 Water Management

The goal of water management is to recycle the water in the TSF and feed the mill with the required water quantity. Best management practices should be adopted to ensure that all options for control of seepage and surface runoff are utilized. The volume of the water in the TSF should be sufficient to support the mill in both dry seasons and winter conditions.

Water from the slurry deposition and precipitation runoff will be collected in a small pond within the TSF. Water will be pumped to a recycle water pond downstream of the TSF.

A decant pond area should be constructed inside the TSF using rockfill dams. The purpose of this area is to separate the fine portion of the tailings from the water that will be pumped to the recycle pond and then back to the mill. The decant dams will be raised with the centerline method to maintain the same pond size. Geotextile may be used for the decant dam construction if the quantity of fines in the rockfill do not meet the filter criteria.

Adequate free board and an emergency spillway will be designed to accommodate the environmental design flood (EDF) and inflow design flood (IDF) events for the TSF dams and recycle pond dams. Should there be excess water in the TSF due to precipitation, the excess water will have to be treated before being released to the environment.

Seepage collection ditches will be constructed at the perimeter of the TSF and recycle pond dams to collect the seepage water and pump it back to the TSF or recycle pond. The seepage collection ditches will be lined with riprap to protect against erosion.

During operations, coarse tailings will be deposited around the perimeter of the dam using a conventional spigot method to develop a tailings beach to keep the supernatant pond away from the perimeter dams. The mixed tailings stream will be deposited in the center of the TSF as mentioned in Section 18.8.1.

Water for recycle will be obtained using a floating barge, pipeline and siphon system.

18.8.7 Tailings Delivery System

The tailings delivery system will transport separated coarse and slime tailings from the processing plant to the TSF. The tailings delivery system is described below.

- The delivery system will be sized initially on the basis of a 42.5 kt/d operation. The coarse tailings will be discharged along the perimeter of the TSF to build the dykes. The coarse tailings pipeline will initially consist of a DN630 HDPE (high-density polyethylene) pipeline, approximately 4 km long.

- At Year 2, this pipeline will be expanded in two 7.5 km long branches built with DN550 DN630 HDPE. A carbon steel section will be added at the pump discharge when the pressure caused by the dyke height is too high for HDPE (around Year 10). This pipeline will initially transport 2,041 m^3/h of coarse tailings to the TSF.

- A second line of the same size and length will be installed adjacent to this pipeline in order to meet the expansion to 85 kt/d during the fourth year of operation. The two pipelines will transport a combined total of 4,082 m^3/h of coarse tailings to the TSF.

- The slimes tailings will be discharged in the center of the TSF. The initial slimes tailings pipeline will consist of a DN710 HDPE pipe approximately 4.5 km long. In Year 2 a second branch approximately 3 km long consisting of a DN710 HDPE pipe will be installed. This pipeline will initially transport 1,656 m^3/h of slimes tailings to the TSF.

- A second line of the same size and length than will be installed adjacent to this pipeline in order to meet the expansion to 85 kt/d during Year 4 of operation. The two pipelines will transport a combined total of 3,312 m^3/h of slimes tailings to the TSF. Both tailings thickener underflow pumps will be on emergency power to prevent the lines from freezing in case of a power loss.

- Additional coarse and slimes pumps will be installed for the ultimate expansion to 120 kt/d.

- The slimes and coarse pipelines will be progressively extended after the 85 kt/d expansion, as included in the sustaining capital cost.

- Slimes and coarse tailings thickener underflow pumps will be on emergency power to prevent the lines from freezing in case of a power loss.

18.8.8 Return Water Delivery System

The return water delivery system for recycle water from the TSF has been sized on the basis of 1,886 m^3/h of water being pumped from the TSF or recycle water basin to the process water pond, for the initial 42.5 kt/d operation. This system will consist of barge pumps and a DN600 HDPE pipeline, approximately 4 km long, adjacent to the tailings pipeline. A second line of the same size and length will be installed adjacent to this pipeline to meet the expansion to 85 kt/d during Year 4 of operation. The two water return lines will transport a combined total of 3,772 m^3/h. The pipelines will be heat traced at low points to prevent freezing.

18.9 Truck Shop & Warehouse Facilities

A truck maintenance facility that will service the mining fleet is located west of the open pit and northwest of the process plant. For the initial 42.5 kt/d operation, only four truck bays will be required. As the fleet expands in response to increased production rate and longer hauls, the workshop will be progressively expanded to 11 bays in Year 11. The building type will be structural steel and covered in architectural cladding. The tire yard is located beside the truck shop.

The warehouse will house mechanical, electrical, instrumentation, and general items. The warehouse structure will be contiguous to the plant maintenance workshop. Internal offices will be supplied adjacent to the warehouse for warehouse and maintenance staff.

18.10 Ancillary Buildings and Services

18.10.1 Assay Laboratory

An area has been set aside for trailers or/a building to be supplied by the analytical services provider. A proposed location to the south of the office and warehouse buildings has been reserved. The construction of a building or rental of trailer(s) costs have not been included. The building/trailer costs are captured in the operating costs for sample processing by the analytical service provider. The laboratory will process samples from the mining and exploration operations, as well as the process plant.

18.10.2 Administration Office Complex

A single-storey administration building is located near the main site entrance gate. The building will have a reception area, offices, meeting rooms, a main conference room, medical clinic, kitchenette, and washrooms. The offices will be for

managers, engineers, geologists, and clerks. A parking lot and transport and pick-up turnaround area are located adjacent to the administration building.

18.10.3 Sewage Treatment

The sewage treatment plant is located approximately 150 m southwest of the main administration building. The sewage sludge builds up at the bottom of the clarifier tank and is removed by a vacuum truck every six to nine months when full. The sludge is then transported and deposited into the municipal garbage dump landfill.

Treated sewer effluent is pumped to the process water storage pond.

18.11 Water Supply & Distribution

The process water storage pond (Figure 18-1) lies south of the tailings thickener and supplies the process plant with the majority of its water. The process water pond is fed from overflow from the tailings thickener and concentrate thickener, as well as from return water from the TSF or the Pit (during the in-pit tailings disposal phase). The water return HDPE pipeline feeding the process water pond is 24 inches in diameter and approximately 4 km long.

The process water pond is designed for a volume of approximately 20,000 m^3 and a two-hour retention time for the 42.5 kt/d case. For expansion to 85 kt/d, a second process water pond of the same size is added.

18.11.1 Raw Water

Raw water is retrieved mainly from the TSF and sedimentation pond or from the pit (during in-pit tailings disposal phase) and pumped to the raw water storage tank located adjacent to the tailing's thickener. From the raw water storage tank, the raw water is pumped to various users throughout the process plant, including the reagent area and all pump gland seals.

18.11.2 Potable Water

Fresh water will be supplied by local wells and will be treated with a reverse osmosis unit to produce potable water for drinking, cooking, and showers. It will also be used for emergency shower and eyewash stations throughout the plant. The reverse osmosis concentrate (brine retentate) is pumped to a local area sump and periodically pumped back into the process circuit.

18.11.3 Fire Water

Fire water is contained in the raw water storage tank. The total volume of the tank is 2,500 m^3, of which 1,000 m^3 is designated for fire water and 1,500 m^3 for raw water distribution. Level controls will assure that the level of the tank does not fall below the 1,000 m^3 volume mark.

During the Phase 2 expansion to 85 kt/d, a second, smaller raw water storage tank will be added, providing an additional 1,500 m^3 in volume.

18.12 Fuel Supply, Storage & Distribution

The initial 42.5 kt/d maximum diesel fuel consumption will be 50,000 L/d and increases steadily to 122,000 L/d at the time of expansion. The fuel farm has been sized to allow for surge. It is recommended that approximately three days of storage

be provided for a total of 854,000 L. The diesel fuel is required primarily for the mining fleet. A single diesel fuel tank delivery volume is 150,000 L; therefore, three tanks will be required at start-up, then four tanks after year 8, and thereafter, five tanks will be required from year 10 onwards.

The diesel fuel tanks will be above-ground, horizontal, cylindrical tanks inside a rectangular secondary containment casing. The tankers can be unloaded and loaded three times each week, as per the rail schedule at the fuel delivery track. The fuel tanks and fuel dispensing pumps are located adjacent to the truck maintenance facility for easy access to the mining fleet.

In addition, there is one 35,000 L regular gasoline double-walled storage tank for cars, pickup trucks, and other site vehicles.

18.13 Transportation & Shipping

The concentrate loadout area is located at the south end of the process plant. A capacity of two days of concentrate production can be stockpiled in the filter discharge area building under cover, prior to being loaded on railcars on the concentrate loadout rail spur adjacent to the concentrate handling building. The nickel and magnetite concentrates are loaded onto rail cars using a front-end loader (FEL). Fiberglass rail car covers are easily removed with a mobile crane and placed south of the rail spur during loading procedures, and quickly bolted back into place on completion. These will be loaded at the plant site or a transfer facility from a stockpile with FELs. It has been assumed that concentrates would be shipped to a treatment facility that would be constructed in the region. Alternatively, concentrates could be shipped through the Port of Quebec to existing facilities in either Europe or Asia.

A drive-through truck loading bay to the south end of the concentrate loading building complete with rolling doors at the east entrance and west exit has also been provided. Trucks can be loaded by front-end loader while remaining under the cover of the building.

Nickel concentrate initial peak throughput will be 149 kt/a, peak production of 211 kt/a in Phase 2, and peak production of 267 kt/a in Phase 3. The iron concentrate peak production includes 704 kt/a in Phase 1, 1,326 kt/a in Phase 2, and 2,153 kt/a in Phase 3.

A trade-off study was conducted to compare the costs of transporting nickel and magnetite concentrate by truck and by rail. It was decided to include a rail spur, with the key deciding factors being its impact on delivery of fuel, reagents and consumables, and explosives supplies.

18.14 Construction Camps

A permanent mining camp will not be required. All labour can be sourced or housed in the surrounding communities (Timmins, Cochrane, Smooth Rock Falls).

18.15 Site Security

All entrants to the mine and plant site must pass through the security guardhouse located at the front gate. The entrance to the site, separating the plant site from Highway 655, is fenced with approximately 5.5 km of chain-link security fencing. The explosives area and emulsion plant (built during the expansion phase) are also fenced for security. The plant site is not fenced on the western, eastern, and southern sides.

18.16 Communications

18.16.1 Enterprise Ethernet Networking

The Enterprise Ethernet Network system will include all the necessary cabling, router, firewall and accessories required to transmit data within the plant, as well as provide communication with the external links.

IT rooms in the administrative building will contain equipment for off-site communication. Other equipment, such as a patch panel and repeater, will be located in remote electrical rooms or in local communication cabinets.

Restricted access to the IT room will be enforced by means of access control cards and video monitoring.

Firewalls and routers will allow communication within the different systems and users within the premises, while preventing intrusion to sensible data from outside. System servers will be used to collect and save data from the different systems.

The administrative network, by means of dedicated fiber optic and Cat6 cables, will service all major buildings to support telephone, intercom, process CCTV, and access systems, as well as providing a link from the process network to the external internet.

The process network, by means of redundant dedicated fiber-optic cables and copper cabling, will service all the buildings where process control equipment is located.

18.16.2 Process Control System

The process control system will consist of a redundant operation station located in the main control room. Other non-redundant control stations will be located in each electrical room.

Process controllers, input/output (I/O) cabinets and human-machine interface (HMI) will be located in electrical rooms or control cabinets as part of the equipment package (*e.g.*, crusher, blower and air compressor systems).

Communication between the processor and remote I/O cabinet will be redundant; communication with other equipment (*e.g.*, package controller, MCC and switchgear) will be non-redundant.

18.16.3 Telephone & Intercom System

The telephone and intercom system will allow direct communication between different areas and buildings throughout the plant.

The intercom or public announcement equipment will be installed in noisy areas or outside of buildings, where a telephone set is not practical.

The telephone and intercom systems will use IP addressing. The telephone management system will provide functions such as call directory, forwarding, messaging, usage statistics, call transferring, etc.

18.17 Surface Water Management System

18.17.1 Water Management Plan

The water management plan must facilitate the operation of the mine development through a wide range of climatic conditions, while at the same time protecting the environment. The prime objectives of the water management plan are as follows:

- provide a reliable water supply to the concentrator, maximizing the use of recycled water from TSF.
- facilitate mining of the ore deposit by limiting inflows to the open pit and by timely removal of groundwater discharges and precipitation falling on the incremental catchment of the open pit.
- provide sediment control.
- collect and treat contact water that could otherwise impair water quality of receiving streams.
- protect mine infrastructure during extreme flood events.

The water management plan changes throughout the mine life, according to five main project phases:

- Phase 1 – Construction.
- Phase 2 – Low Ore Production.
- Phase 3 – High Ore Production.
- Phase 4 – Milling Low-grade Ore Stockpiles.
- Phase 5 – Closure.

Each phase incorporates diversion structures, ditches, sump and pump systems, sedimentation ponds, and reservoirs that manage contact water and impacted contact water (water released from tailings) separately as the overall surface area or footprint of the mine expands.

18.17.2 Contact Water Diversions

Surface water runoff that comes in contact with disturbed areas, other than tailings, is considered to be contact water. The contact water will require removal of high suspended sediment as a treatment procedure. This water includes runoff from the waste rock, overburden or low-grade ore stockpiles, and water pumped from the open pit.

Development of the ore deposit will require the diversions of several creeks around the ultimate footprint of the open pit. The diversions implemented could be either a pump and pipeline system or an open channel.

Sumps will be situated in low elevation areas throughout the catchment area of the western and southern branches of the creeks, where water flow by gravity conveyance is not feasible. Each sump collects a combination of surface water runoff from various site facilities, seepage from the tailing's dams, from direct precipitation, and will be implemented at various times throughout the development of the mine site.

The contact water collected in the southeast reservoir can be pumped to the concentrator as reclaim or as a raw water source. The open pit will also pump water to the southeast reservoir through an oil separator.

Starting in Year 19, all water will be diverted to the main pit to accelerate filling and reclamation of the excavation.

18.17.3 Impacted Contact Water Diversions

Water released from tailings and surface water runoff that comes into contact with tailings is considered "impacted contact water". This water includes runoff and seepage from the tailings dams collected by a network of channels and sumps situated around the TSF, which will be pumped back into the TSF. The concentrator will reclaim as much of the TSF water as possible. Excess impacted contact water will be pumped to the water treatment plant, separate from the contact water which is directed to the sedimentation pond, for treatment and discharge to the environment.

18.17.4 Sedimentation Pond

The sedimentation pond is located north of the TSF and, for modelling purposes, is assumed to have a capacity of approximately 1 Mm³. Excess water from this pond reports to the water treatment plant (if additional treatment is required). The sedimentation pond will be in place early in the construction phase to capture and treat runoff for high suspended solids (TSS) throughout from the disturbed areas during the construction phase.

The sedimentation pond also receives excess water from the open pit area during construction, the southeast reservoir prior to the start of low-grade ore (LGO) stockpile milling, water from sump 9 during construction, local runoff, and direct precipitation. The pond allows high TSS (total suspended solids) to settle to acceptable concentrations.

The sedimentation pond was sized for the 1:10-year return period flows and for a sediment threshold value of 0.01 mm. In order to minimize their footprint within the lower mine area, the depth of the sedimentation pond has been set at 6 m, and the length to the width ratio is 3 to 1, respectively. The pond is situated east of the open pit, and southwest of the administration building.

18.17.5 Tailings Management Facility (TSF and RWB)

The tailings management facility (TMF), which includes the TSF and the recycle water basin (RWB), will serve three key roles in the management of water at the mine. Firstly, runoff generated within the catchment of the TSF, available water within the tailings slurry, and the associated TSF seepage collection system will provide an important source of water for the concentrator. Secondly, the RWB will serve as the live water storage during operations to meet the demand of the concentrator (*i.e.*, temporarily store water during wet periods for subsequent use in the concentrator during the winter). Finally, the TMF will provide enough storage capacity to manage the environmental design flood or excess water that exceeds the treatment rate capacity, and that will be sent to the WTP between April and November.

The largest inflow to the TSF will be the water released from tailings; the largest withdrawal will be outputs of reclaimed water for the concentrator and excess water to be treated at WTP. Water inflows to the TSF will include local runoff, pumped flows from the surrounding sumps, and direct precipitation. Outflows will include evaporation, seepage to groundwater, and loss of water to tailings voids.

TSF will operate with a supernatant pond maintained as low as possible to ensure the settlement of tailings particles. It is assumed for modelling purposes that the minimal required volume for settlement is 1 Mm³, which will ensure approximately five days' residence time based on expected flow of water released from tailings.

Operational procedures will be implemented to ensure that:

- Prior to spring freshet, water storage in the TSF and RWB will be maintained at the minimal to manage the freshet.
- Prior winter freeze-up, sufficient water will be stored in the RWB to compensate for unavailable tailings water during the winter and meet the recycle water demand for the concentrator.

The TSF and RWB will both maintain a minimum freeboard of 1.5 m between environmental design flood water levels and the dam crest. Excess water from the TSF and RWB will be pumped to the water treatment plant prior to discharge to the catchment area via polishing pond.

18.17.6 Collection System for Waste Rock Dump Runoff

Preliminary geochemical analyses and water quality modelling (SGS Lakefield, 2021) indicate that the waste rock and the low-grade ore stockpiles will not be acid generating and that their runoff will not require treatment prior to discharge to the environment. Channels will be constructed along the outer limits of the stockpiles to capture sediment-laden runoff water and route surface water flows to a network of sumps and reservoirs.

East of the waste rock and overburden piles, two channels will route runoff to the southeast reservoir, the north waste dump channel and the south waste dump channel. West of the waste rock and overburden piles, the east pit channel will also route runoff to the southeast reservoir.

The low-grade ore (LGO) stockpiles will evolve in size and shape over time across the ground surfaces that are north of the pit. Sumps will collect runoff from around these areas and will discharge water to the east pit channel, and eventually to the southeast reservoir (SER).

It is assumed that infiltration into the stockpiles from rainfall will eventually report to the southeast reservoir (SER) and excess water from the southeast reservoir will be pumped towards the sedimentation pond.

18.17.7 Water Treatment

Excess impacted contact water from TSF will be directed to the water treatment plant (WTP). Treatment will be required for possible elevated metal concentrations. The WTP will operate from April to November. The WTP will provide management of the TSF pond elevations to maintain the minimal operational target levels as well as manage a design flood. The treatment rate was optimized with the water balance model to prevent any uncontrolled discharge of untreated impacted contact water to the local water catchment area.

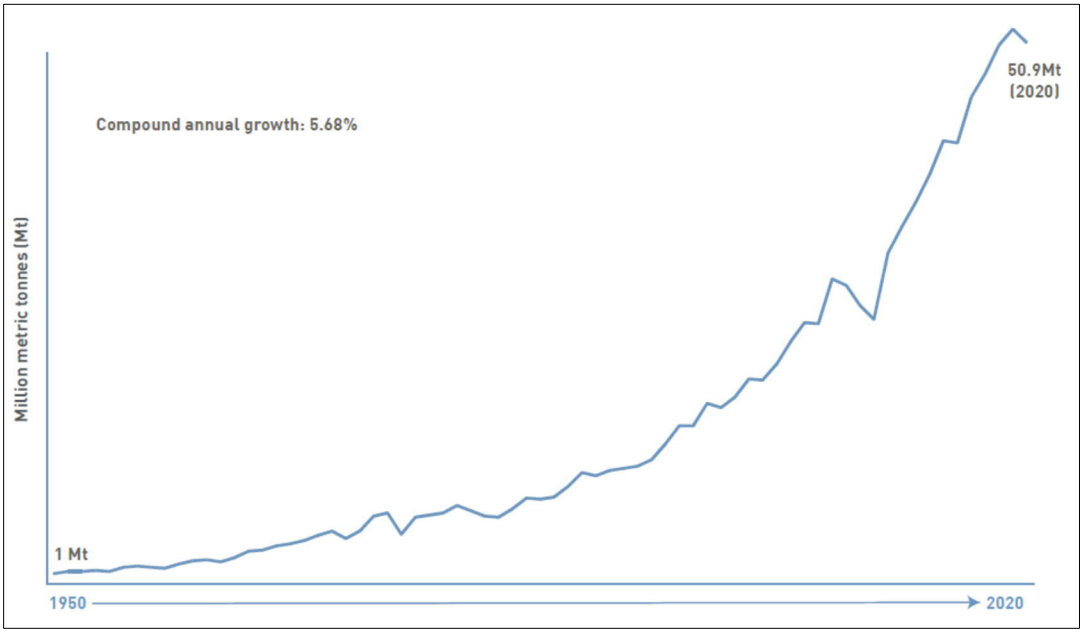
19 MARKET STUDIES AND CONTRACTS

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

19.1 Market Studies

Global nickel consumption is expected to increase to 9.2 Mt by 2050 versus 2.5 Mt in 2019 according to a recent forecast published in December 2020 by leading western nickel producer Glencore. This represents an average annual growth rate of 4.3% over the period. This growth rate is consistent with long-term historical growth rates of 4% to 5% (supported by long-term growth in stainless steel of 5.7%), which has represented approximately two-thirds of nickel demand in recent years. This greater than three-fold increase in expected demand will result from continued strong growth in stainless steel demand in conjunction with substantial new demand from electric vehicles (EVs) which utilize nickel as a key component in their batteries.

Figure 19.1: Global Stainless Steel Production 1950-2020 (slab/ingot equivalent)

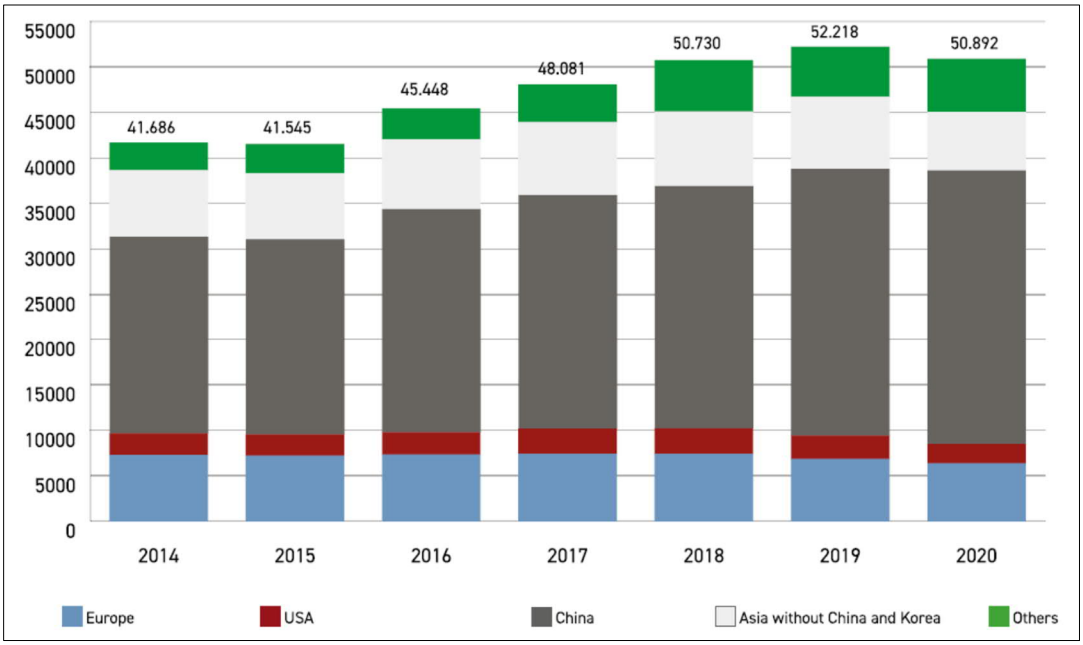


Source: ISSF, “Stainless Steel in Figures”, 2021.

EVs are still a small portion of the vehicle market and in 2020 accounted for approximately 3% of global vehicle sales, but with consensus trend growth rates of 25% to 35% per year in sales, the share of EVs will grow steadily with forecasts of the global market share of electric vehicles being 10% to 20% by 2025 and 30% to 50% by 2030.

As with most commodities, China has been a substantial driver in both production and consumption of stainless steel. Despite China's increasing dominance, the US and Europe continue to be important producers and have consistently produced 9.5 to 10 Mt/a of stainless steel prior to the pandemic in 2020 (Figure 19.2) and have maintained a substantial market price differential vs. China (Figure 19.3), providing substantial benefits for local producers.

Figure 19.2: Stainless Steel Production by Region 2014-2020 (kilotonne slab/ingot equivalent)



Source: ISSF, "Stainless Steel in Figures", 2021.

Figure 19.3: CRU Stainless Steel Prices (2015 to June 2021) – U.S., Europe, China (304 2mm, US$/t)



Source: CRU, 2021.

19.2 Commodity Price Projections

Pricing assumptions were developed for nickel, iron and chrome that are recovered in the nickel and magnetite concentrates produced at Crawford. The price forecasts are effective as of May 2021. As the timing for the Project and its expected 25-year mine life falls within the long-term forecast of those forecasts, a single metal price was utilized for all production years for each commodity.

Table 19.1: Long-term Pricing Assumptions

Metal	Unit	Price (USD)
Nickel	US$/lb	$7.75
Iron	US$/t	$290
Chrome	US$/lb	$1.04

A long-term nickel price assumption of $7.75 per pound was utilized in the study which is consistent with the average of industry analysts who follow the company and a leading independent nickel industry analyst.

As the iron and chrome content of the magnetite concentrate was being utilized in the production of stainless steel and nickel pig iron, the appropriate iron and chrome pricing basis (U.S. No. 1 heavy melting scrap for iron and U.S. ferrochromium for chrome) for those products was utilized in the analysis; however, U.S. scrap iron prices and U.S. chrome prices are not commonly forecast. To derive an appropriate long-term price to be utilized in the study for a project with a 25-year mine life, the company relied on the 10-year U.S. average No. 1 heavy melting composite price for iron price and the 10-year

average ferrochromium price ($/lb chromium content) as tracked by the U.S. Geological Survey and a leading independent industry analyst for chrome prices.

19.3 Concentrate Marketing

Crawford will produce a high-grade (35% nickel) concentrate, a standard-grade concentrate (12% nickel), and a magnetite concentrate containing an average of 48% iron and 3% chrome. The combined Crawford concentrates, with a relatively high average nickel content of 20% nickel over the life of project, and substantial quantities of iron and chrome is ideally suited for use as a high-quality raw material feed for use in stainless steel production, ferronickel, or nickel pig iron producers by roasting the sulphide concentrates. This is followed by the production of stainless steel and nickel pig iron using standard rotary kiln/electric furnace and then refining in an AOD (RKEF/AOD). The use of this RKEF/AOD technology to produce nickel pig iron and stainless steel has become the dominant production path for nickel and stainless steel and is in widespread use. The high nickel content of the concentrate means that lower amount of power, reductants (coke), and energy are required for processing resulting in lower costs and higher payabilities compared to traditional smelting and refining.

CRU, a leading, provider of analysis, prices and consulting in the mining, metals and fertilizer markets, prepared a value-in-use study and market analysis that looked at the value that a third-party local producer would be willing to pay for the feed utilizing standard RKEF/AOD technology widely used in China and Indonesia to produce 304 stainless steel and 1.6% nickel pig iron. It found that net payabilities utilizing this approach for all should be able to achieve 91% payability for contained nickel, 71% payability for contained iron, and 43% payability for contained chromium in the concentrates. This approach was chosen and these figures utilized in the PEA because these payabilities are higher than what is available in the market from traditional smelters and refiners.

CRU developed these payabilities utilizing a scenario under which a third party would construct a local processing facility utilizing standard RKEF/AOD technology widely used in China and Indonesia. The purpose is to produce a blend of 304 stainless steel and 1.6% nickel pig iron to utilize the nickel and iron content of the feeds and be prepared to pay a price for the feeds which would still provide sufficient financial incentive for the construction of the local facilities.

NPI pricing and production costs were generated by CRU using CRU's Nickel Cost Model based on the estimated operating costs at typical production facilities and the specific mass balance and power, reductants, and energy based on scoping study work done by Kingston Process Metallurgy (KPM). CRU utilized stainless operating and capital costs provided by Steel and Metals Market Research (SMR) utilizing their stainless steel production benchmarking adjusted for the specific mass balance and power, flux, reductants, and energy based on scoping study work completed by KPM. The technical viability of using roasted nickel sulphide concentrate has been successfully demonstrated in the production of NPI and stainless steel. After successfully initially demonstrating the potential of roasted nickel concentrate as a more valuable alternative to traditional smelting and refining in 2011, RNC worked with the Tsingshan Group (Tsingshan), beginning in 2012, to validate the concept. In 2014, Tsingshan began construction of 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. Additionally, Tsingshan signed an offtake agreement with Western Areas Ltd. In late 2016.

Tsingshan is now the world’s largest nickel and stainless steel producer.

19.4 Comments on Market Studies and Contracts

With roasting, no payment will be realized for the cobalt and PGM contained in concentrate. At higher prices for cobalt and/or PGM, it could be more economic to treat the concentrate or a portion of the concentrate via conventional smelting and refining or by alternate processes to allow the nickel and cobalt to be utilized by the battery industry. Canada Nickel continues to evaluate and discuss with potential partners a range of market alternatives for concentrate treatment.

There are various nickel smelters globally. Brief profiles of the most likely smelters and other concentrate processing options are provided in the subsections below.

19.4.1 Glencore

The Glencore smelter located in Falconbridge (a suburb of Sudbury) currently treats concentrates produced by Xstrata's operations located in the Sudbury basin (the bulk coming from the Nickel Rim South mine) and in Quebec (Raglan), as well as from third parties. The smelter uses electric furnace technology, which is more suitable for treating concentrates containing elevated levels of MgO. The average MgO content of feed is understood to be higher than the MgO level of feed treated at Vale's Copper Cliff facility. As such, it should be possible to treat Crawford concentrate at the smelter in Falconbridge without exceeding MgO limits. Matte produced by the Falconbridge smelter is shipped to the Nikkelverk refinery in Norway. Overall cobalt recovery through the smelter and refinery is approximately 70%.

19.4.2 Vale

Vale's main smelter is located at Copper Cliff, which is another suburb of Sudbury. The smelter uses flash smelting technology, which is less suitable for treating concentrates containing elevated levels of MgO. However, the large capacity of the facility coupled with the high nickel grade of Dumont concentrate would result in concentrates from Dumont representing a small portion of the total feed tonnage. Furthermore, Vale's own Sudbury basin mines typically produce concentrates with low MgO. As a result, it should be possible to treat Crawford concentrate at Copper Cliff without exceeding MgO limits.

19.4.3 Boliden/Norilsk

Boliden currently operates the Harjavalta flash smelter in Finland. Harjavalta is part of a polymetallic complex that treats separate copper and nickel concentrates. Output from the smelter is refined at the adjacent Harjavalta Refinery, which is owned by Norilsk. The Harjavalta smelter has a capacity of ~40 kt/a of contained nickel and is understood to be operating at significantly less than design levels. It would thus have capacity for a significant percentage of Crawford concentrate. It is understood that the smelter can be expanded by converting the copper processing to nickel processing for a relatively minimal capital investment. The smelter can accommodate some quantity of MgO bearing concentrates. The Harjavalta refinery owned by Norilsk has a capacity of ~65 kt/a and is beginning to receive direct intermediate feeds from Talvivaara. The complex achieves high recoveries for cobalt.

19.4.4 Jinchuan

Jinchuan operates an integrated smelting and refining facility in Gansu Province, China. The smelter currently has a capacity of ~120 kt/a contained nickel, while the refinery has a capacity of ~150 kt/a contained nickel. Over 40% of the concentrate feed to the Jinchuan smelter currently comes from third-party sources. Given our understanding of their mine production profile, Jinchuan will have the capability to take MgO-bearing feeds and will continue to need third-party concentrates to fill its smelting and refining capacity.

19.4.5 Blackstone/EcoPro Joint Venture

Ecopro, a leading high-nickel battery precursor producer, has signed a joint venture agreement with Blackstone Minerals which is advancing towards a production facility that will utilize nickel concentrate feeds to produce battery precursor materials in a single facility. CNC believes these kinds of integrated facilities located in close proximity to nickel sources is

an emerging model of nickel concentrate processing and could become an attractive opportunity for CNC to process its nickel concentrates.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

20.1 Social and Community Considerations

20.1.1 Community Relations

As described earlier in this document, CNC was created at the end of 2019 and listed on TSXV in early 2020 to advance the development of the Crawford Nickel Sulphide 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 the company 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. At this early stage of development, CNC focused on generating efficient communication channels with the relevant local and provincial authorities, such as the City of Timmins, the Ministry of the Environment, Conservation and Parks (MECP), the Ministry of Energy, Mines and Northern Development (ENDM) and Ministry of Indigenous Affairs and the Ministry of Innovation, as well as relevant socio-economic groups including the Timmins Chamber of Commerce, the Timmins Economic Development Corporation and the Mining Association of Canada.

CNC set the basis of its stakeholder engagement strategy by hiring a director of communications and community relations in early 2020 and by adding a vice-president sustainability at the end of that year. Although most of the communication efforts to date focused on governments and socio-economic organizations, CNC engaged in formal discussions with three First Nations communities: Mattagami First Nation and Matachewan First Nation, both member of the Wabun Tribal Council, and the Taykwa Tagamou Nation, leading to the signature of Memorandum of Understanding (MOU) with both groups. CNC also had early-stage discussions with the Métis Nation of Ontario.

The COVID-19 pandemic situation significantly reduced CNC's capacity to efficiently engage with Indigenous Nations and the project's stakeholders. In order to reduce the risk of transmission, CNC's internal policy focused on reducing non-essential human contacts. As a result, CNC's staff optimized the use of technological tools to maintain efficient communication channels with a number of key groups and individuals, such as using videoconferences wherever possible. The upcoming communication and consultation strategy for 2021 and beyond will be further adapted to overcome challenges related with this new social environment.

As CNC is moving the Project towards a feasibility study and impact assessment, the intention of the company is to fully engage with local Indigenous Nations and stakeholders in a comprehensive consultation process aimed at identifying and addressing significant challenges associated with the development of the Crawford Nickel Sulphide Project, but also seeking to underline and maximize potential social, environmental, and economic benefits of the Project. The output of this collaborative effort is intended to be integrated in the Project's Impact Assessment and is meant to influence the upcoming feasibility study. Although the range of stakeholders is expected to evolve to reflect various levels of interest and issues over time, CNC already identified a number of local stakeholders who have or could demonstrate an interest in the Project (Table 20.1). 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 adapted to the needs of the communities.

Table 20.1: Preliminary list of Indigenous Groups, Stakeholders, and Governmental Organizations to be Consulted by CNC

Indigenous Nations / Groups
Wabun Tribal Council
Matachewan First Nation
Mattagami First Nation
Taykwa Tagamou Nation
Metis Nation of Ontario – Region 3
Surface Rights Holders
City of Timmins
City of Cochrane
City of Iroquois Falls
Provincial Crown
Federal Crown
General Public
Community Groups / Organizations
Cochrane District Social Planning Council
Cochrane District Social Services Administration Board
Timmins Community Development Committee
Environmental Groups / Organizations
Friends of the Porcupine River Watershed
Economic Groups / Organizations
Timmins Chamber of Commerce
Timmins Economic Development Corporation
Porcupine Prospectors and Developers Association
Iroquois Falls and District Chamber of Commerce
Cochrane Economic Steering Board
Abitibi Institute
Educational Groups / Organizations
Northern College
Far Northeast Training Board / Commission de Formation du Nord-Est
Recreational Groups / Organizations
Ontario Federation of Anglers and Hunters (Northeastern)
Timmins Snowmobile Club
Government
Provincial / Local Authorities
Ministry of Energy, Northern Development and Mines (ENDM) – Sudbury Office
Ministry of Environment, Conservation and Parks (MECP) – Timmins Office
Ministry of Natural Resources and Forestry (MNRF) – Timmins Office
Ministry of Economic Development, Job Creation and Trade
Ministry of Transportation – North Bay Office
Mattagami Region Source Protection Committee
Northern Claybelt Complex Conservation Reserve
Porcupine Health Unit
Federal Authorities
Aboriginal Affairs and Northern Development Canada
Innovation, Science and Economic Development Canada
Environment and Climate Change Canada
Impact Assessment Agency of Canada

20.1.2 Indigenous Consultations

20.1.2.1 Indigenous Consultations Prior to CNC

As described in Section 6, the Crawford property was previously owned by Noble Mineral Exploration Inc. (Noble). CNC now owns 100% interest in the Crawford Nickel Sulphide Project. On 17 January 2012, Noble announced the signing of a MOU with Mattagami First Nation and Matachewan First Nation in relation to exploration to be conducted on its Project 81 (including the Crawford property), in the Timmins area, Northeastern Ontario. The legally binding MOU, dated January 9, 2012, was approved by the Venture Stock Exchange (TSXV) on 3 February 2012. Under the exploration agreement, Noble and the First Nations have agreed to terms that underline each party's mutual respect for the land and a responsible approach to exploring in their traditional territory. The agreement was to remain in effect during the initial program and until such time as Noble and the First Nations enter into an Impact and Benefit Agreement (IBA).

An exploration agreement was also endorsed between Noble and the Taykwa Tagamou Nation (TTN) on May 20, 2013. This agreement was aimed at acknowledging the exploration activities pursued by Noble on TTN traditional land and also at setting the basis of the negotiation of an impact and benefits agreement if Noble intends to increase its activities beyond grassroot exploration.

20.1.2.2 First Nations Consultations by CNC

Following the acquisition of the Crawford Nickel Sulphide Project, CNC renewed its exploration agreement with Mattagami and Matachewan First Nations, both members of the Wabun Tribal Council. This MOU—signed on 29 February 2020, amended on 4 June 2020 to account for TTN's land claim, and refined on 20 August 2020—is meant to set a clear and constructive framework for the exploration and project development pursued on Mattagami and Matachewan First Nations traditional land. This agreement sets the basis of the negotiation of an IBA for the Crawford mine operations while setting interim measures covering business opportunities, employment and training, and financial aspects. IBA negotiations with the Wabun Tribal council officially started in January 2021 and are ongoing.

Discussions with TTN took a different path than the traditional IBA approach. While development of the project was fast-tracked by the creation of CNC, TTN sought the opportunity to fully participate in the project's development by developing and owning the electrical transmission asset that will supply the project and by participating in the financing of the heavy equipment fleet.

A first MOU, signed on 3 October 2020, underlines the intention of TTN to raise and deploy capital to own and develop the electrical transmission assets that would supply the Crawford Nickel Sulphide Project under best commercial practices to ensure fair market value. Subject into entering definitive agreements based on this non-binding MOU, CNC would rent these assets from TTN at a fair market rate over the life of the mine or 20 years (whichever comes first) and TTN would be granted an option to acquire a direct minority interest in CNC at fair commercial terms.

Another non-binding MOU, signed on October 16, 2020, contemplates the opportunity for TTN to participate in the financing of all or some of the heavy mining equipment fleet for the operation. This MOU also underlines the will for both parties to evaluate the training and associated employment opportunities that could be available to TTN where specialized maintenance and operation is required for the equipment and where that equipment is financed or owned in whole or in part by TTN.

Notwithstanding these agreements and as mentioned in the previous section, CNC intends to fully integrate Indigenous Nations into the upcoming consultation process with the purpose of identifying opportunities as well as potential issues in order to minimize the project's environmental and social impact while maximizing its shared value for all stakeholders This

consultation process will directly feed the Impact Assessment and is intended to support and influence the project's technical studies and future development plans.

20.1.2.3 Métis Nation of Ontario Consultations.

The Métis people are considered part of Canada's Indigenous groups, along with First Nations and Inuit. Although the Métis Nation of Ontario is not associated with any fixed settlement, both Federal and Provincial Governments recognize its right for self-governance as well as Aboriginal harvesting rights for food under Section 35 of the *Constitution Act of 1982*.

CNC has clearly demonstrated its intention to consult the various stakeholders and rightsholders of the Crawford Nickel Sulphide Project. It is in this context that preliminary discussions took place between the MNO and CNC in May 2021. The two groups showed a clear intention to establish an effective channel of communication associated with the development of the project.

20.2 Regulatory Framework

20.2.1 Impact / Environmental Assessment

Most mining projects in Canada are reviewed under one or more environmental assessment (EA) / impact assessment (IA) 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. Entities involved in the review process normally include government agencies, Indigenous Nations, various interested parties, and the general public.

Normally, there is an opportunity to coordinate the Federal and Provincial processes to reduce duplication of effort in both document preparation and consultation efforts, and it is encouraged by both levels of government. A cooperation agreement is in place between the Province of Ontario and Government of Canada: the *Canada-Ontario Agreement on Environmental Assessment Cooperation (2004)*, which will facilitate this approach to completing the different EA requirements (Government of Canada, 2004).

20.2.1.1 Federal Requirements

A Federal Impact Assessment for the project could potentially be required under one of two scenarios:

- If the project meets the requirements under the *Impact Assessment Act*, or
- If the project is designated by the Federal Minister of Environment and Climate Change Canada (ECCC) as requiring an Impact Assessment

If the following conditions of the Physical Activities Regulations (SOR/2019-285) pursuant to the *Impact Assessment Act* are met (or others not listed below), documentation must be provided to the Impact Assessment Agency to assess whether an Impact Assessment is required for the Crawford Nickel Sulphide Project.

The conditions of the regulation below may apply to the project based on the current preliminary project design.

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 (tpd) or more

(d) a new metal mill, other than a uranium mill, with an ore input capacity of 5,000 tpd or more.

It is not expected that the following condition would apply based on the current project design:

60 The construction, operation, decommissioning and abandonment of a new structure for the diversion of 10,000,000 cubic metres per year (m^3/y) or more of water from a natural water body into another natural water body.

The maximum rate of ore mining and processing at the project is expected to be substantially more than 5,000 t/d. The project therefore is expected to meet the conditions of the Physical Activities Regulations and is required to initiate the planning stage for an Impact Assessment with the Impact Assessment Agency. It is expected that the Impact Assessment Agency will determine that a Federal Impact Assessment is required for the Crawford Nickel Sulphide Project.

20.2.1.2 Provincial Environmental Assessment Requirements

Ontario mining projects are also required to meet the requirements of the Ontario *Environmental Assessment Act*, regardless of whether a Federal Impact Assessment is needed. Provincial Environmental Assessment processes that could potentially apply to the project include the following:

- Ministry of Northern Development, Mines, Natural Resources and Forestry (MNRF) Class EA for Resource Stewardship and Facility Development Projects, for facilities or infrastructure that are to be constructed on Crown land (including typically below the high-water mark of waterbodies / watercourses) or other use of Crown resources (such as aggregates). The Class EA may be carried out at one of four levels, depending on the expected level of impact. The determination of category level is made by the local MNRF office through application of screening criteria.
    - Category A:
        - Potential for low negative environmental effects and/or public or agency concern.
    - Category B:
        - Low to medium potential for significant net negative environmental effects, usually with a high degree of certainty.
        - Medium potential for some concern is anticipated.
    - Category C:
        - Medium to high potential for significant net negative effects, with some uncertainty with predictions of effects, requiring additional research and/or evaluation.
        - Effects require mitigation techniques tailored to the project.
    - Category D (Individual EA):

  - Concern likely to be high, with potential for adverse reaction, based on experience or previous consultation.
  - If the project is determined to projects may be outside the scope of Categories A, B, or C, and should instead be subject to the requirements for an individual EA under Part II of the *Environmental Assessment Act*.

- Electricity Projects Regulation EA requirements for power supply that could be required to support construction / operations:
  - Class EA Screening Level required for total name plate diesel power generation of 1 megawatt or greater, but less than 5 MW (≥ 1 MW and < 5 MW).
  - Individual EA required for ≥5 MW of diesel power generation.
  - Hydro One Networks Inc. (HONI) Class EA for Minor Transmission Facilities required for a 115 kV transmission line longer than 2 km, a 115 kV+ transformer station, and/or >115 kV and <500 kV transmission line (*i.e.*, 230 kV) longer than 2 km but shorter than 50 km.
  - Individual EA required for a >115 kV and <500 kV transmission line of 50 km or longer length.
- Class EA Screening Level required for natural gas power generation greater or equal to 5 MW.
- Ministry of Transportation (MTO) Class EA for Provincial Transportation Facilities for the relocation of Highway 655, if applicable. While having an environmental component, this process is much more strongly engineering-driven.

20.2.2 Environmental Approvals

20.2.2.1 Federal Environmental Approvals

Environmental approvals related to the *Fisheries Act*, *Canadian Navigable Waters Act* and *Explosives Act* are anticipated to be required for the Crawford Nickel Sulphide Project. Fisheries and Oceans Canada (DFO), Environment and Climate Change Canada (ECCC), Transport Canada (TC) and Natural Resources Canada (NRCan) are the agencies primarily involved with approvals under these statutes.

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 passage is not interfered with. NRCan is responsible for authorizing the manufacturing and storage of explosives through licenses and issues permits for the transportation of explosives.

It is expected 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*.

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.

Table 20.2 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). Other approvals may arise through consultation with Federal agencies or with changes to the design of the project.

Table 20.2: Anticipated Federal Environmental Approvals

Approval / Instrument	Responsible Agency	Description
Fisheries Act Authorization *Fisheries Act*	DFO	Based on an assessment of impacts to fish and fish habitat, authorization may be required for an undertaking or activity that may result in the death of fish or the harmful alteration, disruption or destruction of fish habitat. This may include: Establishment of the stockpile(s) and tailings storage facility In-water structures such as for fresh water taking Watercourse diversions if applicable Mine dewatering groundwater effects that would cause fish disruption to watercourses supporting fisheries.
Schedule 2 Listing MDMER pursuant to the *Fisheries Act*	ECCC	Overprinting of water frequented by fish by tailings and mine rock stockpiles (or other deleterious material) could require a listing under Schedule 2 of the MDMER. Potential areas of impact include TMF and waste rock stockpile(s).
Land-use clearance	NAV CANADA	Construction of tall structures, use of cranes, transmission line towers.
Manufacturing, storage, and transportation of explosives *Explosives Act*	NRCan	Explosives magazine, manufacturing facility and transportation require a Federal permit, pursuant to Section 7. If facility is owned by licensed explosives contractor, permit will be issued to them.
Works in Navigable Waters *Canadian Navigable Waters Act*	TC	Alteration of navigable waters and crossing of navigable waters with infrastructure.
Aeronautical obstruction clearance *Canadian Aviation Regulations (SOR/96-433) of the Aeronautics Act*	TC	Marking and lighting for structures that could interfere with aeronautical navigation.

20.2.2.2 Provincial Environmental Approvals

It is anticipated that the following four primary provincial ministries will be involved in environmental approvals for the project:

- Ministry of Energy Northern Development and Mines (ENDM) 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 project aspects that are related to water quality and quantity, air quality and noise, and most recently, 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 the ENDM.
- Ministry of Transportation (MTO) manages the provincial highway system amongst other aspects, including Highway 655 which bisects the site.

Additional agencies that may be involved in permitting include:

- Ontario Energy Board (OEB)
- Ministry of Heritage, Sport, Tourism and Culture Industries (MHSTCI)

Provincial environmental approvals that are expected to be required to construct and operate the project include what are shown in the preliminary list in Table 20.3.

Table 20.3: Anticipated Provincial Environmental Approvals

Approval	Responsible Ministry	Description
Closure Plan *Mining Act*	ENDM	Mine construction / production and closure, including financial assurance. Offline (above the high-water mark of local waterbodies / watercourse) dams such as for the TMF.
Environmental Compliance Approval(s) – Air and Noise *Environmental Protection Act*	MECP	Release of air emissions and noise, such as from haul trucks (road dust), and structures including the ore processing plant, refueling / oil / lubrication area, administration complex, on-site laboratory and other buildings.
Environmental Compliance Approval – Waste Disposal Site / Waste Management *Environmental Protection Act*	MECP	For operation of a landfill and/or waste transfer site. For the transportation of waste materials offsite and onto Provincial highways.
Environmental Compliance Approval – 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.
Environmental Compliance Approval(s) – Industrial Sewage Works *Ontario Water Resources Act*	MECP	Construction of a mine / mill water treatment system(s) discharging to the environment, such as for the TMF, ponds, mine water, site storm water, stockpile runoff and refueling and oil/lubrication areas.
Overall Benefits Permit *Endangered Species Act*	MECP	Any activity that could adversely affect species or their habitat identified as Endangered or Threatened in the various schedules of the Act. The project site is within the Kesagami home range of Woodland Caribou and as a minimum a screening will be required.

Approval	Responsible Ministry	Description
Permit(s) to Take Water *Ontario Water Resources Act*	MECP	For taking of groundwater or surface water (in excess of 50,000 liters per day), such as for supplementing the process plant water balance, pit dewatering, potable needs, and potential ice bridges during transmission line construction. During construction, a permit(s) may be required for dam and/or mill construction to keep excavations dry. A permit may be required for realignments and dam construction at project site waterbodies / watercourses.
Clearance Letter *Heritage Act*	MHSTCI	Confirmation that appropriate archeological studies and mitigation, if required have been completed for the project.
Aggregate Resource License *Aggregate Resource Act*	MNRF	Extraction of aggregate for construction purposes.
Forest Resource License(s) *Crown Forest Sustainability Act*	MNRF	For cutting of Crown merchantable timber, such 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	To obtain tenure for long-term facilities on Crown land, such as for a transmission line, or shoreline structures (dock, pumphouse and pipeline). 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. Consultation with MNRF planned regarding tenure for lake bottom where main pit is located.
Scientific collection permits (including for fish collection, and salvage) *Fish and Wildlife Conservation Act*	MNRF	Baseline fish / wildlife studies. Initial fish relocation from watercourses and waterbodies to be overprinted or removed. Also potential for destruction of beaver dams.
Work Permit(s) *Lakes and Rivers Improvement Act, Public Lands Act*	MNRF	For work / construction on Crown land. Could be required as part of construction of a transmission line and public infrastructure re-routing. Multiple permits could be required. 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.
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.

20.3 Environmental Studies

The project site is located on the northern edge of the Canadian Shield in northeastern Ontario. The site is generally in a natural condition with the exception of existing provincial infrastructure (highway, transmission lines) that cross portions of the lands, and prior impacts as a result of mineral exploration and forestry operations. Vegetation communities comprise a mix of deciduous and coniferous forests, and low-lying wetland areas / muskeg.

CNC has initiated environmental baseline studies at the project site, in acknowledgment that programs will be required to meet the anticipated baseline environmental data needs of the:

- Federal *Impact Assessment Act.*
- Provincial *Environmental Assessment Act.*
- Future environmental approval applications, such as requirements under the *Fisheries Act* and the *Environmental Protection Act.*

A winter large mammal survey was completed during March 2021. Moose and stick nests were identified within the survey area which extends beyond the boundaries of the site. No Species at Risk, such as Woodland Caribou, were identified as being present.

The Crawford Nickel Sulphide Project is located along the southern boundary of the Kesagami Caribou Range for Woodland Caribou. The northern portion of this 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 this range, where the project occurs, is highly impacted by 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 Ministry of Environment Climate and Parks for restoration of habitat for Woodland Caribou as habitat function has become significantly degraded.

With the assumption that a Federal Impact Assessment will be required, studies on the following are expected to be initiated or will continue through 2021 and 2022:

- atmospheric (climate/meteorological, air quality, greenhouse gas emissions, light and noise).
- aquatic resources (fish species, fish compositions, benthos and habitat).
- hydrology (surface water flows and bathymetry).
- hydrogeology (groundwater paths).
- water quality (surface water and groundwater).
- terrestrial resources (species and habitat).
- geochemistry (mineral wastes).
- human environment (socio-economics and land use).
- archaeology and heritage resources.
- Indigenous Traditional Knowledge / traditional land use.

20.4 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 area and at the site, based on experience with other mining developments in northern Ontario. As the project design and environmental studies progress, this assumption will be confirmed.

20.5 Preliminary Geochemical Assessment

CNC has initiated geochemical assessments of anticipated mineral wastes from the project, including both mine rock and tailings. Early results from the limited static and kinetic testwork completed to date are favourable, suggesting that acid generation may not be a significant concern.

20.6 Preliminary Reclamation Approach

The project will be required to complete a regulatory Closure Plan as per the requirements of Ontario Regulation 240/00: Mine Development and Closure Under Part VII of the *Mining Act* in Ontario, prior to commencement of construction activities. The Ontario *Mining Act* requires that a closure plan be filed for any mining project before the construction of the project is initiated. It also requires that financial assurance be submitted to the province in order to provide sufficient funds for the Crown to undertake the closure and reclamation activities described in the closure plan. Table 20.3 summarizes a preliminary closure approach for the project. Reclamation is proposed to be completed progressively during operations as reasonable, which is considered an industry best management practice.

Table 20.4: Preliminary Approach to Project Reclamation

Project Element	Preliminary Approach
Open Pits	Remove all infrastructure and equipment within the open pit. Revegetate overburden slopes to a stable condition (regrading to stable slopes to be carried out as an operational safety measure). Allow pit to fill naturally by means of seepage and runoff inputs from the local area. Ramps will be barricaded and a safety berm established around the perimeter. Channels will be constructed from open pits for future passive overflow if needed.
Buildings, Machinery, Equipment and Infrastructure	Salvageable machinery, equipment and other materials will be dismantled and taken off site (sale or reuse; financial assurance does not account for salvage value of equipment/machinery). Remaining items managed according to regulatory requirements at the time, either within an approved landfill at the project site or at a licensed facility off site. Above grade concrete structures will be broken and reduced to near grade with rebar to be cut flush with the surface. Concrete structures will be infilled with clean mine rock, if needed.
Chemicals	All petroleum products, chemicals and explosives / explosive components remaining will be removed from the site and transported to a licensed facility for disposal. Soil found to exceed acceptable criteria will be handled otherwise according to regulatory requirements.
Onsite Infrastructure	Roads not needed in the longer term will be scarified and seeded. Onsite power lines and associated materials without value will be dismantled and deposited in an approved landfill.
TMF and Ponds	With the assumption subject to confirmation that the tailings will be non-potentially acid generating, the surface will be covered with growth material and seeded. Overflow spillways within the dams will be deepened, if needed. Water pond will be breached and stabilized to allow for free drainage.
Overburden / Mine Rock Stockpiles	Stockpiled overburden will be used for reclamation on site, as needed, and subsequently reshaped and revegetated. With the assumption subject to confirmation that the mine rock is non-potentially acid generating, flat surfaces will be covered with growth material and seeded.

Project Element	Preliminary Approach
Offsite Relocated Infrastructure	Assumes to remain in relocated position.

20.6.1 Final Reclamation Costs

A detailed estimate of the costs associated with closure has not been completed at this stage of the project. Rather, costs have been assumed to be similar to those reported for other similar scale existing operations and prospective projects in the Abitibi region. The financial evaluation makes provision for total closure costs of US$34.4 million.

21 CAPITAL AND OPERATING COSTS

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

21.1 Capital Cost Estimate Input

The capital cost of the project, including the Phase 1 production rate of 42.5 kt/d, Phase 2 expansion to 85 kt/d, and Phase 3 expansion to 120 kt/d. Sustaining expenditures over the 25-year life, has been estimated based on the scope of work defined in the sections below. The parties below have contributed to the preparation of the capital cost estimate in specific areas, as listed:

Ausenco:

- Crushing.
- Process.
- Loadout.
- Mine office, truckshop and washbay.
- Utilities.
- On-site infrastructure.
- Off-site infrastructure.
- Indirect costs.
- Contingency.

Wood:

- Tailings storage facility (excluding dam earthworks).
- Waste dumps.
- Channel design and sumps.

Dave Penswick:

- Site preparation (clearing and grubbing).
- Open pit (OP) mine development (by both Owner and Contractor).
- OP mobile equipment.
- OP Ancillary equipment.
- Tailings dam earthworks.

- Owner's costs.

All amounts expressed are in US Dollars unless stated otherwise.

21.2 Capital Cost Estimate Summary

The estimate provided is classified as an AACE Class 5 estimate with an accuracy of ±50%. Tables 21.1 provides a summary of the capital cost estimate, including initial capital, and expansion capital. Table 21.2 summarizes the sustaining capital. The costs are expressed in real Q1 2021 US dollars, unless stated otherwise.

Items that would be denominated in foreign currency take into account the forecast foreign exchange rate (forex) at the time of purchase. Indirect costs include first fills of consumable items for the initial and expansion estimates

Table 21.1: Summary of Capital Costs (US$M)

US$ millions Initial and Expansion	Initial 42.5 kt/d	85 kt/d	120 kt/d	Life of Mine Years 1-25
Mining	201	-	-	201
Process Plant	294	294	98	685
Site and Services	157	132	4	293
Infrastructure	149	15	25	189
Indirects	108	31	22	161
Owners' Costs	29	-	-	29
Contingency	250	71	45	366
Total	**1,188**	**543**	**194**	**1,925**

Table 21.2: Sustaining Capital Costs (US$M)

US$ millions Sustaining and Closure	Pre-Production	Phase 1 Years 1-3.5	Phase 2 Years 3.5-7	Phase 3 Years 8-18	Life of Mine Years 1-25
Mining	-	187	195	285	711
Site and Services	-	34	44	136	214
Infrastructure	-	8	15	91	132
Closure	26	9	-	-	34
Total	**26**	**238**	**254**	**512**	**1,091**

The estimate is based on an EPCM execution approach according to the following parameters and qualifications:

- The estimate is based on Q1 2021 prices and costs.
- Financing related charges (*e.g.*, fees, consultants, etc.) are excluded.
- There is no escalation added to the estimate, other than the contingency.

Data for these estimates have been obtained from numerous sources, including:

- conceptual level engineering design.
- mine schedules.
- topographical information obtained from site survey.
- budgetary equipment quotes from multiple potential OEMs.
- budgetary unit costs from local contractors for civil, concrete, steel, electrical and mechanical works.
- data from recently completed similar studies and projects.
- information provided by CNC and Wood.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs and Owner's costs) were identified and analyzed. In each of these categories, a percentage of contingency was allocated based on the accuracy of the data, to derive the overall contingency amount.

21.3 Capital Cost Estimate

21.3.1 Mining

Mine capital costs have been estimated by Dave Penswick, who is an independent consultant. Table 21.3 summarizes elements of the mining capital estimates for the initial and sustaining phases of expenditure. Note that the strategy of accelerated mining rates and stockpiling of lower value mineralization (discussed in Section 16) decouples the mine plan from that of the mill. For this reason, all expenditures following the initial phase of pre-stripping have been classified as sustaining.

Table 21.3: Summary of Mining Capital Costs

WBS Area	Units	Initial	Sustaining	Total
Site Preparation	US$M	$14	$30	$44
Contractor Stripping	US$M	$54	$0	$54
Owner Stripping	US$M	$44	$0	$44
Mining Fleet	US$M	$45	$516	$561
Ancillary Equipment	US$M	$14	$6	$20
Technology	US$M	$22	$37	$59
Infrastructure	US$M	$7	$21	$28
Trolley Assist	US$M	$0	$100	$100
Mining Total	**US$M**	**$201**	**$711**	**$912**

Sources of the estimates presented in Table 21.3 are described below.

Site Preparation – Costs include the following:

- Clearing and grubbing costs of $1,684/hectare and for a total area cleared of approximately 1,800 hectares.

- The cost of lining the ROM pad and LG stockpiles of $3.14/m^2 with a total footprint of approximately 2.2 Mm^2.
- Construction of pioneer roads totalling $1.6 million.
- Construction of 42 km of surface mine haul roads at a rate of $784/m.

Contractor Stripping – The estimate is based on the quantity of mining that would be allocated to the contractor and a quotation received from a contractor with experience working in the area.

Owner Stripping – The estimate is based on the quantity of mining that would be performed by the Owner and a zero-based model of mining costs

Mining Fleet and Ancillary Equipment – The zero-based model includes a derivation of the mobile equipment that would be required to achieve the planned mining schedule. Unit costs for specific units were based on budgetary estimates provided by dealers representing the major original equipment manufacturers (OEMs), including Caterpillar and Komatsu. Estimates included not only the cost of machines, but also the associated cost of transport to site and assembly. It was also assumed the fleet would be purchased under a lease-to-own contract from the OEM with indicative rates of a 25% down payment, five-year term and 5% interest rate.

Technology – Crawford will make extensive use of technologies that will improve productivity and/or lower unit costs. These technologies are described in Section 16 and include the following:

- An autonomous haulage system (AHS) that eliminates the need for an on-board operator for the 290 t haul trucks.
- An autonomous drilling system (ADS) that eliminates the need for an on-board operator for the blast hole drills.
- A fleet management system to automatically dispatch equipment in such a manner as to improve efficiency (*i.e.*, minimal queuing) and effectiveness (*e.g.*, dozers repairing roads where trucks are having to slow down)
- High-precision GPS (HPGPS) guidance and monitoring for drills, excavators, graders and dozers. HPGPS will improve the accuracy of work being performed, including drilling to the planned depth of a hole, loading to the boundary of a composite or cutting a road to grade. Benefits will include reduced dilution, improved mining recovery and faster speeds for trucks.
- Payload monitoring for excavators and trucks to ensure that trucks are optimally loaded.
- Excavator dipper tooth monitoring to ensure tramp steel is not delivered to the primary crushers, as this would cause blockages and lead to lengthy interruptions to production.
- Haul truck tire temperature and pressure monitoring, to maximize tire life.

Infrastructure – Elements included in the estimate are as follows:

- A truck workshop that is sized at one bay per ten x 90 t trucks or five x 290 t haul trucks. The workshop includes bays initially and is ultimately expanded to 11 bays.
- An equipment fueling station. Storage has been based on three days consumption. The station is initially equipped with three x 150 kL tanks, and ultimately expanded to five tanks.

- A roadstone crusher that is commissioned ahead of the trolley-assist system to ensure well-maintained roads that will lead to maximum trolley utilization.

- Pit dewatering equipment. Pumps and pipeline are added as the pit deepens.

The cost of the magazine and facilities for manufacture and storage of explosives will be borne by the explosives supplier and recovered as an operating expense over a period of five years once the operation is generating cash flow.

Trolley Assist – Elements included in the estimate are follows:

- An AC mine sub-station sized to handle maximum demand from the pit that will peak at approximately 90 MW.

- The installation of trolley infrastructure that has been priced at an average unit rate that makes provision for DC substations spaced at 1 km distance.

- The removal of trolley infrastructure as the pit walls are pushed back and ramps are relocated.

- Removed infrastructure is available to be re-used. Over the life of mine, a total 21,875 m of infrastructure are installed, with 11,000 m being new and the remaining 10,875 m being re-installations.

- Pantographs for the trolley-equipped haul trucks. Over the life of mine, a total of 49 trucks are equipped.

Note that the initial segment of trolley line will be commissioned in Year 2 of operations and therefore the system is not included in the initial capital cost.

21.3.2 Process Plant

The nickel and magnetite recovery plant and associated facility estimates have been prepared on a commodity basis (*i.e.*, divided into earthworks, concrete, structural, etc.) and reported by area (*i.e.*, crushing, milling, etc.). The estimate is based on the purchase of new mechanical equipment, and quantities have been factored from first principles.

The estimate assumes the majority of the work will be carried out under fixed price or unit price contracts under a normal development schedule. No allowance is included for contracts on a cost plus or fast-track accelerated schedule basis. The erection of tankage, structural, mechanical, piping, electrical, instrumentation, and civil works will be performed by experienced contractors, using local labour.

21.3.3 Tailings Storage Facility

The initial capital estimate makes provision for constructing the TSF starter dam for to a height of 12 m. This will be sufficient to store an initial 18 months of tailings production. The dam would be constructed from a combination of waste rock and gravel stripped from the open pit.

Construction would take place continually on a 12-month basis and the bulk of material would therefore be provided on a ROM basis. For this material, the cost of loading, dumping and haulage to a pit exit is excluded from the capital estimate as it is provided for in the mining cost and only the cost associated with the incremental hauling distance to the TSF is included. Over the life of mine, the incremental one-way haulage distance for material used in TSF construction is approximately 5.6 km, in addition to the average 5.5 km one-way haul for all ex-pit material.

In the early years of pit development, some stockpiling of construction materials will be required to ensure the dam is raised at a continuous rate. For this material the cost associated with rehandle loading is included in the capital cost. After material is delivered, some will be placed using smaller scale equipment and the costs associated with operating these machines has also been included in the capital cost. All the equipment used in the delivery and placement of construction material will also be used in the open pit and has been costed under "mining fleet". The capital estimate for TSF earthworks consequently does not include the purchase of any mining equipment.

The starter dam will continue to be raised at an annual rate of approximately 8 m until reaching its ultimate height of 36 m. Thereafter, construction of the larger ultimate dam will commence. This dam, which will initially be constructed using the same downstream method as the starter dam, will be lifted to a height of 49 m at an annual rate of approximately 6 m. At this point, further raises will be achieved using the upstream method with coarse tailings and thus require little additional construction material from the open pit. The ultimate height of the dam will be 73 m.

Over the life of mine, approximately 302 Mt of waste rock and gravel from the open pit will be used in construction of the TSF. The unit cost of delivering and placing this material is estimated at approximately US$0.63/t.

21.3.4 On-Site Infrastructure

The following administration buildings will be built:

- main administration building with medical center and training room.
- mine dry.
- security office.
- security gatehouse.
- assay lab.

In addition, the process plant buildings listed below will be built. The capital cost for these buildings is included in the process plant cost estimate.

- primary crushing facility.
- process building (includes grinding, flotation, magnetic separation, cleaning and scavenging).
- crushed ore stockpile cover.
- plant workshop (part of process plant building) and warehouse reagent storage (part of process plant building).
- explosives manufacturing facility.
- mine truck maintenance facility.

The cost also includes the supply of the electrics, fittings, and furnishing for the buildings, but excludes earthworks. The cost to supply power and water services to the buildings form part of the process plant cost.

21.3.5 Rail Spur

A rail spur, approximately 20 km long, services the process plant and will connect to the Glencore Kidd Operations rail line. The rail spur consists of a fuel delivery track and a freight delivery siding south of the process plant.

21.4 Basis of Estimate

Direct costs are quantity based and include all permanent equipment, bulk materials, freight (inland and ocean), subcontracts, labour, contractor indirects, and growth allowance associated with the physical construction of the facilities.

The same estimate build-up and philosophy was used for the 42.5 kt/d, 85 kt/d and the 120 kt/d expansion case, taking into account that the scope of work was different in certain areas.

21.4.1 Commodity Take-offs

Bulk material take-offs to a PEA level were developed from arrangement drawings. Rates were obtained from quotes obtained from local contractors. These rates include the appropriate gang rate for the commodity and the actual cost of the permanent materials. Local freight associated with contractor-supplied material is included in the unit rates.

No imported fill is required. Aggregate material is available via an on-site crushing plant purchased by the Owner. The crushing plant will be operated by the mine, and aggregate will be provided "free-issued" at the site of the crusher (*i.e.*, the cost associated with production of aggregate has been provided for in the mining cost). Excess cut material can be stockpiled on site or consumed for future construction.

21.4.2 Labour rates

Labour rates have been built up from first principles for different trades (welders, boilermakers, roofers, pipefitters, millwrights, storeman, crane operator, etc.). These rates have been based on the Ontario labour collective agreement (industrial sector).

These labour rates include the following:

- Base hourly rate.
- Contribution rate from collective agreement – industrial sector:
  - vacation, holiday and sick leave pay.
  - premiums.
  - safety, health and welfare.
  - compensation for safety clothing and equipment.
  - social benefits and funds.
- Contribution rate from the Government of Ontario:

- Ontario Pension Plan.
- Ontario Parental Insurance Plan.
- Health Services Fund.
- Labour Standards Commission.
- Workforce Skills Development and Recognition Fund.
- Compensation Tax.

The work week is 50 hours, which consists of 40 regular hours and 10 overtime hours. The 10 overtime hours are calculated as 4 hours x 1.5 (the regular rate) and 6 hours x 2.0. This is based on working Monday to Friday at 8 h/d regular pay; Monday to Thursday at 4 hours at time and-a-half, and 6 hours on Saturday at double time.

Contractor indirect costs for structural, mechanical, piping, electrical and instrumentation have been developed with the assistance of well-established local construction contractors within Ontario; earthworks and concrete has been based on unit rates from contractors within Ontario. Distributable costs have been allocated by percentage in the estimate on a manhour basis and are inclusive of the following:

- salaries, salary burden, allowances and benefits for the contractor's indirect labour, supervisory and management staff.
- staff recruitment and travel expenses.
- living out allowances.
- mobilization and demobilization.
- temporary buildings and facilities at site specifically for and used by the contractor.
- workshop equipment and supplies.
- vehicles and equipment used by staff during construction.
- construction equipment including cranes up to 100 tonnes.
- temporary construction power (diesel generator sets).
- small tools and consumables.
- site office overheads, such as stationery, communications, light and power, first aid, security, etc.
- head office costs/contribution.
- financing charges.
- insurances.
- advertising.

- profit.

The total structural steel-mechanical-platework and piping (SMPP) contractor all-in install labour rate obtained was C$195/h which includes the SMPP base crew rate of C$120/h and the addition of costs associated with the items listed above. These rates were confirmed by local SMPP contractors from northern Ontario.

21.4.3 Equipment Costs

Costs were sourced from Ausenco's database and include all the mechanical equipment and mobile equipment.

21.4.4 Freight

All bulk materials, plant and equipment items within the direct costs are based on delivery to site. Where possible, plant and equipment costs have been obtained from historical quotes inclusive of the freight component; if not, percentage allowances have been included, where applicable. For mechanical equipment, 4% of the equipment supply cost has been included for inland freight and 12% for ocean freight for items not sourced in North America. These percentages represent averages for projects executed in Canada.

21.4.4.1 Duties & Taxes

Where duties are applicable on imported items of supply, the appropriate allocation of funds has been included in the estimate. All taxes are excluded unless otherwise stated.

21.4.5 Indirect Costs & Owner's Costs

Indirect costs include items that are necessary for project completion, but not related to the direct construction cost. These items are summarized in the subsections below.

21.4.5.1 Temporary Facilities & Services

Temporary facilities and services are items that are not directly attributable to the construction of specific physical facilities of the plant or associated infrastructure, but which are required to be provided during the construction period to support construction.

These costs include:

- EPCM office complex, HS&E services, security services, site vehicles, refueling, bus transportation, recurring project costs, maintenance services, provision of temporary roads, temporary power, water, effluent disposal and other facilities as required. For the expansion phase, power for construction work is to be provided by the Owner.
- Heavy lift construction cranes. These are larger cranes (>100 tonne capacity) than what the construction contractor provides.

21.4.5.2 EPCM

The engineering, procurement, project and construction management (EPCM) budget has been compiled by identifying resources over a defined schedule. A detailed assessment of consultants and project general expenses are also included in EPCM costs. The EPCM estimate includes the following:

- corporate services.
- project services.
- engineering.
- drafting.
- construction.
- travel expenses.
- home office expenses.
- site office expenses.
- consultants (geotechnician, shipping logistics specialist, surveys, soils and compaction testing, concrete testing, fire and safety).

21.4.5.3 Vendor Representatives

Allowances for vendor representatives, for both installation supervision and for the commissioning component, are included and are based on vendors recommended support that were provided in the quotations. These have been incorporated where applicable. Where these were not provided in the quotation but still required, a 3% of equipment supply cost was included.

21.4.5.4 Construction Camp

There is no requirement for a construction camp. All labour can be sourced from the Timmins region.

21.4.5.5 Spares

Spares include capital spares, one-year operational spares, and commissioning/start-up spares. A spares list was developed from the budget quotations provided. Where spares were not priced in the quotation, a percentage of the equipment cost was applied to each spare's category.

An increase in spares inventory is allowed for in the expansion phase.

21.4.5.6 Commissioning Support

The direct installation hours do not include construction labour to assist the EPCM commissioning team. Costs for these are based on two crews consisting each of one electrical technician, two fitters and one trade's assistant for the duration of four months. For the expansion phase, only two months are included.

21.4.5.7 First Fills

An estimate for first fills for the following reagents has been included: PAX, MIBC, CMC, H_2SO_4, $CuSO_4$, and flocculant. A 100% charge for 12 mm, 38 mm, 65 mm and 100 mm grinding balls is also included.

An allowance has also been made for oils, lubricants, hydraulics, and greases.

21.4.5.8 Modification Squad (Mod Squad)

The direct installation hours do not include post construction modifications to facilitate handover and acceptance by the Owner. An allowance was included in the estimate in the form of a "mod squad" and are based on a crew of two fitters, three boilermakers, two trade assistants, and one electrical technician.

21.4.6 Owner (Corporate) Capital Costs

Owner's costs make provision for the following:

- Owner's Team logistics over the duration of the project.
- capitalized process operating costs, incurred prior to commissioning of the mill; following mill commissioning, all costs have been expensed.
- capitalized G&A operating costs.
- recruitment, orientation and training for the operating team.
- construction insurance.

Escalation is excluded from this estimate.

21.4.7 Estimate Growth & Contingency

21.4.7.1 Growth Allowance

From the time the estimate is prepared to the time the facility is completely constructed, a number of detail variations that are not scope changes are expected to occur. Allowances have been included in the direct cost section of the estimate and are specified against line items.

The growth categories assigned to each line item are dependent upon what level of definition was obtained. In this case, the growth allowance for both the initial and expansion capital cost was calculated to between 12% to 15% for the process plant. Nil growth has been applied to the mining, winter works, and indirect costs.

21.4.7.2 Estimate Contingency

An estimate contingency allowance has been included and is money that is expected to be spent. It is meant to cover additional costs that will be incurred as a result of final detailed design and investigation to provide a holistic estimate of the defined scope. It is not intended to be a provision for changes in scope and standards.

The value of the construction cost and estimate contingency represent an estimated project scope value of 100%. In this case the estimate total contingency is assessed at 30% for the initial capital cost, and 15% for the 85 kt/d expansion, and 30% for the 120 kt/d expansion, based on an analytical method addressing the elements of the estimate and assessing the estimate for scope, cost and confidence.

21.4.8 Exclusions

- project finance and interest charges.
- foreign exchange hedging.
- residual value of temporary equipment and facilities.
- residual value of any redundant equipment.
- cost to Owner of any downtime.
- currency fluctuations.
- escalation
- impact caused by modifications directed by government authorities, including schedule.
- increased costs due to early works (*e.g.*, concrete requirements before there is a batch plant on site).
- removal, remediation, or disposal of hazardous/contaminated materials encountered during construction.
- costs of any special requirement due to the participation of outside financing sources.
- costs to identify, locate, remove or relocate existing underground obstructions or utilities.

21.4.9 Project Deferred & Sustaining Capital

Ongoing capital requirement for the mine production period totals $1,091 million over the mine life. Items covered under sustaining capital include the following:

- ongoing clearing of land prior to pushbacks of the pit or extension of waste dumps .
- purchase of new production and auxiliary fleet for the mine (in response to longer hauls as the pit deepens) and replacement fleet (as the initial generation of equipment reaches the end of its economic life).
- expansion of the workshop that will be required as the fleet expands; the initial workshop of four bays will ultimately be expanded to 11 bays.
- ongoing expansion of the TSF.
- general plant and infrastructure replacements.

21.5 Operating Cost Estimate

21.5.1 Summary

This section details the estimated operating costs for mining, process plant and general and administration (G&A) for the Crawford Nickel Sulphide project. The costs are presented in real Q1 2021 US dollars, unless stated otherwise, such as in Table 21.7. The estimate is considered to carry an accuracy of ±50% per AACE Class 5 estimating standards.

Operating costs were estimated in the following manner:

- 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 commodities and labour rates for the Timmins region.
- 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.
- The operating cost estimate for the concentrator includes those costs associated with operating the TSF.
- G&A costs were based on the level of support required for the operation.
- Costs for treatment and refining of concentrate were based on the commercial terms discussed in Section 18 and the scheduled production of concentrate.
- Processing operating costs were calculated exclusive of variability from design throughputs (*e.g.*, neglects ramp-up period, etc.).

A summary of life-of-mine operating costs is provided in Table 21.4.

Table 21.4: Operating Cost Summary

	Phase 1 Years 1-3.5		Phase 2 Years 3.5-7		Phase 3 Years 8-18		Life of Mine Years 1-25	
Operating Cost/Tonne Milled	**US$/t**	**C$/t**	**US$/t**	**C$/t**	**US$/t**	**C$/t**	**US$/t**	**C$/t**
Labour	2.39	3.19	1.49	1.98	1.20	1.60	1.26	1.68
Consumables	2.49	3.31	2.36	3.14	2.30	3.07	2.25	3.00
Maintenance	1.70	2.27	1.47	1.96	1.69	2.25	1.54	2.05
Diesel	1.02	1.36	0.78	1.04	0.78	1.04	0.72	0.96
Power	2.45	3.26	2.40	3.20	2.35	3.13	2.25	3.00
Other	0.95	1.27	0.52	0.70	0.40	0.53	0.43	0.58
Total	**11.00**	**14.66**	**9.02**	**12.02**	**8.72**	**11.62**	**8.45**	**11.27**

21.5.2 Key Assumptions

Key assumptions used in generating the operating cost estimates are given below:

- Prices in Canadian dollars for goods and services obtained prior to the cost basis date of Q1 2021 have been escalated to this date using average Canadian consumer price index (CPI) for the period January 2019 to January 2021 of 2.22% per annum.

- Labour costs were estimated based on the organizational structure developed for each area and the rates of pay are based on wages and benefits at existing mining operations in the Timmins region of Ontario.

- Based on the costs reported by similar scale consumers of electricity in the region, it has been assumed the blended average price paid for power, inclusive of transmission, consumption and demand charges, would equate to C$71.42/MWh. The forecast long-term diesel price of C$0.92/L is based on forecast long-term oil prices of US$60/bbl and a CAD exchange rate of US$0.75.

21.5.3 Mine Operating Costs

A summary of mining operating costs by area and activity is given in Tables 21.5 and 21.6. Note that these tables exclude $181 million of expenditures related to the delivery and placement of material in the TSF that has been capitalized.

The forecast mining costs for the project are low relative to some existing large-scale Canadian open pit hard rock mines, but can be explained by the following factors:

- The use of automation technologies, including AHS and ADS, to reduce labour and increase the utilization of equipment. In the event that equipment was manually operated, costs would increase an estimated $0.10/t.

- The use of trolley-assisted haulage to reduce net energy costs and increase haul truck travel speeds. In the event that trolley-assist were not used, costs would increase an estimated $0.10/t.

- The geometry of mineralization allows for highly productive, bulk mining. This is in contrast to gold mines where irregular mineralization necessitates selective mining, with more units of smaller capacity.

- The mechanical properties of approximately 42% of rocks at the Crawford site include very low abrasion indices, which will result in lower consumption of ground engaging tools (GET). It will also be possible to blast rock with a relatively low powder factor that will allow for widely spaced blast holes, leading to low drilling and blasting costs.

Table 21.5: Mining Operating Costs by Area

Area	Units	Total	Capitalized Stripping	Expensed	Expensed Rate
Labour Number	FTE pa	10,371	239	10,132	272 kt/FTE
Labour Cost	US$M	$817	$20	$797	$0.29/t mined
Consumables	US$M	$558	$5	$552	$0.20/t mined
Maintenance	US$M	$1,101	$9	$1,093	$0.40/t mined
Diesel	US$M	$640	$7	$633	$0.23/t mined
Power	US$M	$354	$1	$354	$0.13/t mined
Contracts & Other	US$M	$108	$56	$52	$0.02/t mined
TOTAL	US$M	**$3,579**	**$98**	**$3,481**	**$1.26/t mined**
Total Mined	Mt	2,794	40	2,754	
Total Mill Feed	Mt	907	0	907	
Unit Cost	US$/t mined	$1.28	$2.45	$1.26	
	US$/t milled	$3.95	$0.00	$3.84	

Table 21.6: Mining Operating Costs by Activity

Area	Units	Total	Capitalized Stripping	Expensed	Expensed
Contract Mining	US$M	$54	$54	$0	$0.00/t mined
Drill & Blast	US$M	$557	$6	$551	$0.20/t mined
Ex-pit Loading	US$M	$375	$5	$370	$0.13/t mined
Ex-pit Hauling	US$M	$1,487	$16	$1,471	$0.53/t mined
Stockpile Rehandle	US$M	$69	$0	$69	$0.03/t mined
Support Equipment	US$M	$465	$5	$461	$0.17/t mined
Maintenance Labour	US$M	$332	$5	$327	$0.12/t mined
Management & Technical	US$M	$238	$6	$232	$0.08/t mined
Total	US$M	**$3,579**	**$98**	**$3,481**	**$1.26/t mined**

21.5.4 Process Operating Costs

The processing plant operating costs are based on the flowsheets described in Section 17. The battery limits for the determination of process operating costs begins with the crushing facilities and end with the TSF, and include plant services

21.5.4.1 Basis of Estimate

The process plant operating costs were determined from first principles using input from a variety of sources, including the following:

- process design criteria.

- reagent and equipment supplier quotations.
- staffing levels for processing plant estimated by Ausenco.
- personnel salaries and overheads based on information from similar projects in the region.
- client recommendations.
- previous study assessments.

21.5.4.2 Inclusions

The process plant operating cost estimate includes all direct costs associated with the production of nickel concentrate. Included in the Ausenco operating cost estimate are the following:

- labour for supervision, management, and reporting of on-site organizational and technical activities directly associated with the processing plant.
- labour for operating and maintaining plant mobile equipment and light vehicles, process plant, and supporting infrastructure.
- labour and operational costs for the laboratory.
- costs associated with direct operation of the processing plant, including all reagents, consumables, and maintenance materials used in operating and maintaining the mobile equipment and light vehicles.
- cost of power supplied to the process plant from the power grid.
- operational costs of the wastewater treatment facilities.
- general operations associated costs including consultants, training and general supplies.

21.5.4.3 Exclusions

The plant operating costs exclude the following:

- corporate overheads.
- escalation or exchange rate fluctuations.
- mine operating costs other than grade control assays.
- exploration labour and operating costs.
- environmental permits.
- contingency.

- import duty and taxes.
- sustaining capital.
- interest and financing charges.
- mine or plant closure/rehabilitation activities.

21.5.4.4 Process Plant Operating Costs Summary

The plant is designed for an initial ore throughput of 42.5 kt/d followed by expansions to 85 kt/d and 120 kt/d, all at an availability of 92.0%. Processing costs include labour, power, maintenance materials, reagents and consumables, mobile equipment, and ongoing metallurgical testing. The estimated overall operating cost for the 42.5 kt/a is $6.29/t of ore milled, decreasing to $6.01/t ore milled when processing 85 kt/a, and decreasing again to $5.51/t after expansion to 120 kt/a.

The processing plant costs are summarized in Table 21.7 in Q1 2021 Canadian dollars (C$ and CAD).

Table 21.7: Process Plant Cost Summary

Cost Centre	42.5 kt/d		85 kt/d		120 kt/d	
	C$M/a	C$/t	C$M/a	C$/t	C$M/a	C$/t
Operation Labour	5.1	0.33	9.2	0.30	9.2	0.21
Maintenance Labour	3.5	0.23	6.3	0.20	6.3	0.14
Power	42.0	2.71	82.9	2.67	108.4	2.47
Reagent & Consumables	35.9	2.31	68.30	2.20	93.8	2.14
Maintenance	7.8	0.50	14.71	0.47	17.7	0.40
Vehicle & Mobile Equipment	0.9	0.06	1.3	0.04	1.7	0.04
Laboratory & Assays (by 3rd party)	2.5	0.16	3.8	0.12	4.5	0.10
Total	**97.6**	**6.30**	**186.6**	**6.01**	**241.6**	**5.50**

21.5.5 General and Administrative (G&A)

The estimated cost for G&A expenses is based upon the level of service required for the size of Crawford's operation and takes into account existing local services. The costs summarized in Table 21.8 are almost entirely fixed in nature, with the resulting unit costs (per tonne milled) at 120 kt/d being approximately one-third of the unit costs at the lower milling rate of 42.5 kt/d. Note that the ex-pit production rate decreases in later years as mining transitions to the East Zone, leading to a decrease in administrative costs associated with the total labour complement.

Table 21.8: G&A Operating Costs by Area

Area	Unit	42.5 kt/d		85 kt/d		120 kt/d	
Material Milled	Mt/a	14.7		30.2		42.2	
Labour	FTE	39		39		39	
Area	Unit	US$M/a	$/t	US$M/a	$/t	US$M/a	US$M/a
Labour Cost	US$	$3.1	$0.21	$3.1	$0.10	$2.7	$0.06
Consumables	US$	$0.5	$0.04	$0.7	$0.02	$0.7	$0.02
Maintenance	US$	$0.1	$0.00	$0.1	$0.00	$0.1	$0.00
Diesel	US$	$0.3	$0.02	$0.3	$0.01	$0.2	$0.01
Power	US$	$0.6	$0.04	$0.6	$0.02	$0.6	$0.01
Contracts & Other	US$	$9.9	$0.67	$11.4	$0.38	$10.8	$0.26
Total	**US$**	**$14.5**	**$0.99**	**$16.2**	**$0.54**	**$15.1**	**$0.36**

21.5.6 Contingency

Contingency is not included in the operating cost estimate.

22 ECONOMIC ANALYSIS

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

22.1 Summary

The scope of the Crawford Nickel Sulphide Project that was evaluated includes the following:

- Two separate open pits will be mined. Mining will commence in the Main Zone, which contains approximately 80% of the total feed to the concentrator.
- The initial 12 months of pre-stripping will be conducted by contractors. Thereafter, blast hole charging will continue to be performed by a third party, but all other mining activities will be performed by the Owner.
- The bulk of overburden overlying the deposit will be mined by smaller equipment, including small hydraulic excavators, 45 t articulated trucks, and 90 t haul trucks. However, in excess of 95% of the mining will be performed using large, electrically-powered, hydraulic excavators and 290 t haul trucks. The 290 t haul trucks and blast hole drills will both be operated autonomously while the 290 t haul trucks will also be equipped for trolley-assist operation on uphill hauls.
- Following a 24-month construction period, the concentrator will initially produce 42.5 kt/d for 42 months, after which an expansion to 85 kt/d will be commissioned. A final expansion to 120 kt/d will be commissioned after 84 months of cumulative operation.
- Production rates from the Main Zone will initially exceed mill requirements, so processing of higher value material will be prioritized to reduce the payback period. Lower-value material will be temporarily stockpiled. Approximately 25% of the total material fed to the mill will be rehandled and material stored in stockpiles will peak at 157 Mt. In order to ensure the mill is always fed with the highest value material available, approximately 33% of stockpiled material will be reclaimed while the Main Zone is still active. The remainder will supplement feed from the smaller East Zone after the Main Zone is depleted. Following its depletion, the mined-out Main Zone will be used to impound tailings, thus reducing the surface footprint of the TSF.

Salient metrics for the project are summarized in Table 22.1.

Table 22.1: PEA Summary Metrics

Item	Units	Value	Comment
Mill Feed	Mt	907	
Payable Ni	Mlbs	1,689	
Payable NiEq	Mlbs	2,441	Includes byproduct Cr & Fe
Net Smelter Return	US$/t mill feed	$20.86	Revenues assumed paid on an FOB mine gate basis
Site Operating Costs	US$/t mill feed	$8.45	
Net C1 Costs	US$/lb Ni	$1.09	Includes site operating costs net of byproduct credits
EBITDA	US$/t mill feed	$11.99	
Peak Funding Requirement	US$ MM	$1,201	Cumulative unlevered investment prior to positive cash flow
Total Investment	US$ MM	$3,016	Includes all capital and closure expenditures
Net AISC	US$/lb Ni	$1.94	includes C1 costs, royalties, sustaining capital and closure
Pre-Tax $NPV_{0\%}$	US$ M	$7,855	
Pre-Tax $NPV_{8\%}$	US$ M	$1,856	
Post-Tax $NPV_{8\%}$	**US$ M**	**$1,187**	
Post-Tax IRR		**15.8%**	

22.2 Assumptions

Key price assumptions used in the evaluation are as follows:

- Forecast long-term prices for nickel, chromium and iron of US$7.75/lb, US$1.04/lb and US$290/t, respectively, are projections provided by the company based on the views of selected industry analysts.
- Payability has been assumed to be 91% for nickel, 43% for chromium, and 71% for contained iron on a FOB basis at the mine gate, which requires no costs associated with concentrate transport or treatment. These estimates are based on an analysis by CRU utilizing scoping study work completed by Kingston Process Metallurgy Inc. (KPM) and Steel and Metals Market Research (SMR). These rates of payability are believed to provide sufficient incentive for the construction of a local stainless steel mill that would also produce additional nickel pig iron products based on the nickel/iron mix of the feeds.
- A forecast long-term exchange rate of C$1.00 to US$0.75 was assumed and a forecast long-term price for Brent oil of US$60/bbl has been taken from consensus projections of North American equity analysts. Based on the current relationship between the prices of oil and the rack rate for diesel at Toronto, along with current transportation charges, the expected volume discount and posted carbon taxes for 2022, this oil price translates to a delivered cost of C$0.92/L diesel.
- The weighted average LOM electricity price is forecast to be $75/MWh and is based on Crawford's expected demand profile along with published rates for large users in the area.

Working capital has been calculated as follows:

- Contractual terms for the sale of concentrate would include payment for 90% of concentrate value within 30 days and the remaining 10% within 60 days.

- Accounts payable would be settled within 30 days.
- First fills for the mine and G&A areas have been calculated based on a warehouse inventory of one month for all consumable items with the exception of tires (four months), diesel (five days), and electricity (no holding). It was assumed that no advance purchase for 50% of mine maintenance items would be required as these would be held on a consignment basis. First fills for the process plant have been calculated by Ausenco using first principles.

The evaluation was conducted on both a pre-tax and post-tax basis. The post-tax evaluation incorporates the following features of the Canadian income, Ontario income, and mining tax codes:

- The Ontario Mining Tax (OMT) is applied at a rate of 10% on resource profits, which are normal profits less a further annual allowance up to 7.8% of the cumulative expenditure on processing assets. Income taxes allow deduction of the OMT from taxable income.
- Income tax rates of 15% (federally) and 10% (provincially), 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%, 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%. The accelerated rate of 100% for pre-production expenditures (Canadian Exploration Expenses (CEE)) were phased out in 2017.
- Net operating losses (NOL) can be pooled for up to 20 years and deducted in their entirety in a single year.

22.3 Base Case Results

The total life of project can be subdivided into the following periods:

- Phase 1 production at a concentrator throughput rate of 42.5 kt/d for 42 months (3.5 years).
- Phase 2 production at a concentrator throughput rate of 85 kt/d for 42 months (3.5 years, to the end of Year 7).
- Phase 3 production at a concentrator throughput rate of 120 kt/d for 132 months (11 years, to the end of Year 18). During Phase 3, the concentrator will be fed at the target rate with mostly ROM material, with only a small percentage of total feed coming from the higher-grade stockpiles.
- Residual production commences when the two higher-grade stockpiles have been depleted. Initially the concentrator continues to operate at a rate of 120 kt/d, fed from ROM material and the lowest-grade stockpile. After the stockpile is depleted, the production rate decreases. The residual period totals 81 months (7.75 years, to Q4 of Year 25).

Summary metrics for each of the periods are presented in Table 22.2. Note that project capital expenditures for each of the phases occurs in the preceding period (*e.g.*, project capital for Phase 2 occurs in the period that Phase 1 is operational). Also note that production metrics for Phase 1 include mining during the pre-stripping period.

Table 22.2: Summary of Metrics by Phase

Item	Units	Phase 1	Phase 2	Phase 3	Residual	Total
Mill Feed Mined	Mt	109	122	501	176	907
Total Mined	Mt	252	353	1,602	588	2,794
Stripping Ratio	waste : mill feed	1.34	1.90	2.20	2.34	2.08
Mill Feed	Mt	52	106	480	270	907
Ni Grade	% Ni	0.32	0.26	0.25	0.23	0.25
Equivalent Ni[1] Grade	% NiEq	0.40	0.36	0.34	0.32	0.34
Nickel Recovery	% of con'd Ni	50%	44%	39%	28%	37%
Equivalent Ni[1] Recovery	% of con'd NiEq	46%	39%	37%	30%	36%
Payable Ni	Mlbs Ni	165	245	935	344	1,689
Payable NiEq [1]	Mlbs NiEq	207	327	1,330	577	2,441
Net C1 Cash Costs [2]	US$/lb Ni	$1.46	$1.32	$1.20	$0.45	$1.09
Project Capital	US$ M	$1,188	$543	$194	$0	$1,925
Sustaining Capital	US$ M	$264	$255	$512	$60	$1,091
Net AISC [2]	US$/lb Ni	$3.09	$2.57	$1.97	$0.89	$1.94
Post-Tax $NPV_{8\%}$	**US$ M**					**$1,187**

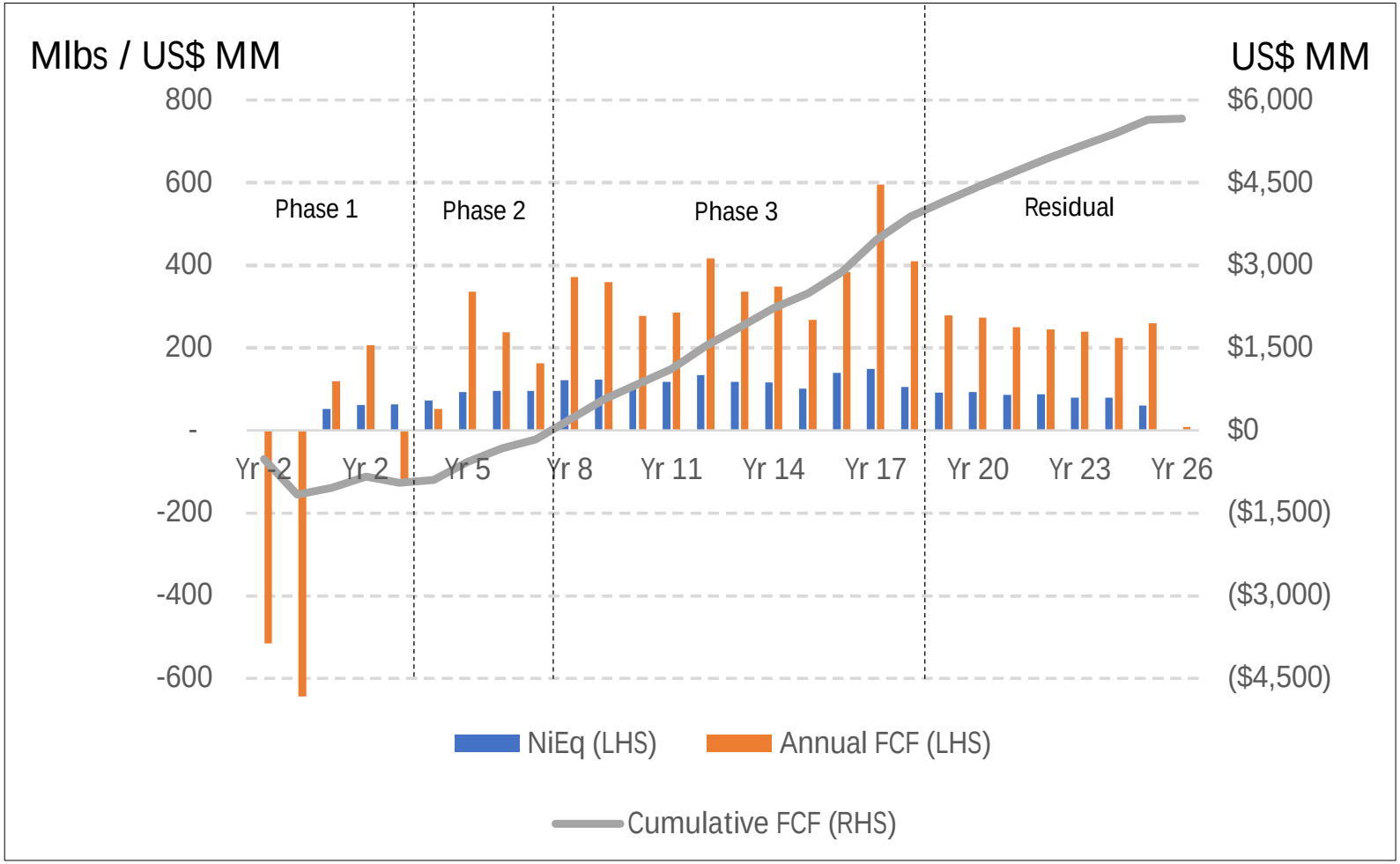
Notes: 1. Chrome and Iron converted to equivalent Nickel using ratio of prices. 2. Net of by-product credits

Figure 22.1 provides a graph of cash flow for the life of the project. The following information is highlighted:

- The peak funding requirement of US$1,201 million is reached months after the start-up of commercial operations. The operation is forecast to generate positive free cash flows from the second quarter of operation.
- Simple payback of all invested capital on a post-tax basis is achieved 7.4 years after the start of commercial production. With the elimination of accelerated depreciation, projects now pay cash income taxes during the payback period. On a pre-tax basis, the payback period is nine months sooner at 6.7 years.
- Over the current 25-year life of mine, the project is forecast to generate an average of US$347 million operating cash flow and US$274 million free cash flow annually.

Table 22.3 provides detailed metrics for the life-of-mine production and unit costs, while Table 22.4 provides financial statements.

Figure 22.1: Production and Cash Flow over the Project Life



Source: Penswick, 2021.

Table 22.3: Detailed Production and Unit Costs

Item	Units	Total	Pre-production	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10-18	Year 19+
Ex-pit Mined	Mt	2,794	40	35	61	78	75	87	108	120	130	155	1,317	588
Mill Feed Mined	Mt	907	19	17	28	32	26	32	39	38	37	40	424	175
Y End Stockpile	Mt		19	23	35	52	56	58	67	74	69	65	94	-
Mill Feed	Mt	907	-	13	16	16	21	30	31	31	42	44	394	270
Grade Ni	% Ni	0.25	-	0.33	0.31	0.32	0.28	0.26	0.27	0.27	0.26	0.25	0.25	0.23
Grade Cr	% Cr	0.60	-	0.65	0.62	0.61	0.62	0.63	0.64	0.62	0.60	0.59	0.58	0.61
Grade Fe	% Fe	6.51	-	5.78	6.06	6.12	6.29	6.47	6.49	6.47	6.49	6.44	6.60	6.49
Con'd Ni	Mlb	4,975		94	108	108	130	173	182	185	237	241	2,147	1,370
Con'd Cr	Mlb	11,952		183	211	209	290	426	436	427	554	571	5,028	3,616
Con'd Fe	Kt	59,006		740	940	950	1,330	1,969	2,013	2,006	2,695	2,819	26,025	17,518
Con'd NiEq	Mlb	6,784		116	135	136	170	235	244	246	319	328	2,935	1,920
Recovery Ni	% Ni	37%		49%	50%	51%	46%	43%	44%	44%	41%	39%	39%	28%
Recovery Cr	% Cr	27%		27%	27%	27%	27%	27%	27%	27%	27%	27%	27.0%	27%
Recovery Fe	% Fe	36%		32%	39%	41%	39%	36%	30%	29%	34%	38%	36%	38%
Rec'd Ni	Mlb	1,856		47	54	55	59	75	80	81	98	95	834	378
Rec'd Cr	Mlb	3,227		49	57	57	78	115	118	115	150	154	1,358	976
Rec'd Fe	Kt	21,296		239	362	385	515	701	598	575	915	1,071	9,265	6,671
Rec'd NiEq	Mlbs	3,086		62	75	77	89	117	118	118	152	155	1,363	759
Payable Ni	Mlb	1,689	-	43	49	50	54	69	73	74	89	86	759	344
Payable Cr	Mlb	1,388	-	21	25	24	34	49	51	50	64	66	584	420
Payable Fe	kt	15,120	-	170	257	273	366	498	424	408	649	760	6,578	4,736
Payable NiEq	Mlb	2,441	-	52	62	63	72	94	95	95	122	123	1,084	577
Mine Operating Cost	US$/t mined	$1.26		$1.44	$1.29	$1.21	$1.26	$1.23	$1.16	$1.17	$1.20	$1.14	$1.29	$1.30
	US$/t milled	$3.84		3.94	5.09	6.08	4.48	3.52	4.04	4.52	3.76	4.05	4.29	2.83
Mill Operating Cost	US$/t milled	$4.19		$4.95	$4.71	$4.71	$4.68	$4.53	$4.51	$4.51	$4.17	$4.13	$4.11	$4.08
G&A Operating Cost	US$/t milled	$0.42		$1.08	$0.92	$0.97	$0.70	$0.50	$0.50	$0.51	$0.38	$0.38	$0.37	$0.37
Total Site Costs	US$/t milled	$8.45		$9.97	$10.72	$11.76	$9.87	$8.55	$9.05	$9.54	$8.31	$8.55	$8.77	$7.28

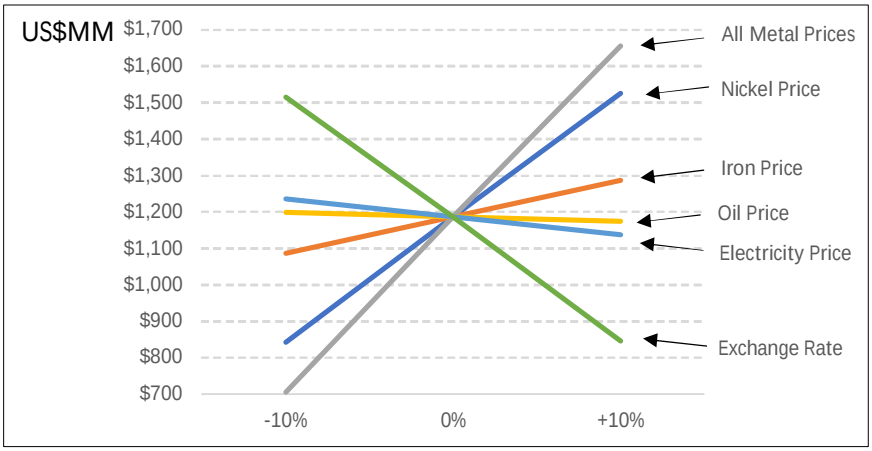
Table 22.4: Financial Statements

Income Statement	Units	Total	Prod'n	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10-18	Y 19+
Revenue Ni	US$M	13,088		329	382	386	417	531	563	570	692	667	5,885	2,665
Revenue Cr	US$M	1,443		22	25	25	35	51	53	52	67	69	607	437
Revenue Fe	US$M	4,385		49	75	79	106	144	123	118	188	220	1,908	1,374
Net Smelter Return	US$M	18,916		401	482	491	558	727	738	740	947	956	8,400	4,475
Mining Opex	US$M	3,481		50	79	94	95	107	125	140	156	177	1,693	763
Processing Opex	US$M	3,801		63	73	73	99	138	140	140	173	181	1,619	1,101
G&A Opex	US$M	385		14	14	15	15	15	16	16	16	16	147	101
Total Operating Costs	US$M	7,666		128	166	182	209	260	281	296	345	374	3,459	1,965
Net C1 Cash Costs	US$/lb Ni	1.09		1.32	1.34	1.56	1.25	0.94	1.45	1.72	1.01	0.99	1.24	0.45
Royalties	US$M	378		8	10	10	11	15	15	15	19	19	168	90
EBITDA	US$M	10,871		265	306	298	339	452	443	429	583	563	4,773	2,421
Cash Flow Statement	**Units**	**Total**	**Pre-Prod'n**	**Y 1**	**Y 2**	**Y 3**	**Y 4**	**Y 5**	**Y 6**	**Y 7**	**Y 8**	**Y 9**	**Y 10-18**	**Y 19+**
EBITDA	US$M	10,871	0	265	306	298	339	452	443	429	583	563	4,773	2,421
Cash Income Taxes	US$M	1,886	0	0	0	0	0	49	56	52	100	100	981	547
Cash Mining Taxes	US$M	313	0	0	15	10	9	9	8	7	10	8	157	80
Post-Tax OCF	US$M	8,673	0	265	291	288	330	394	378	370	472	455	3,636	1,794
Project Capital	US$M	1,925	1,189	0	0	362	181	0	65	129	0	0	0	0
Sustaining Capital	US$M	1,057	0	0	46	86	76	51	56	88	82	74	427	73
Working Capital & Closure	US$M	34	(30)	100	(0)	(30)	47	3	(12)	(3)	28	(8)	(14)	(45)
Total Investment	US$M	3,016	1,159	100	46	417	303	53	108	214	109	65	413	28
Net AISC	US$/lb Ni	1.94		2.80	3.28	3.28	2.40	1.97	2.86	3.03	2.05	2.43	1.90	0.89
Pre-Tax Free Cash Flow	US$M	7,855	(1,159)	165	260	(119)	35	399	335	215	474	497	4,360	2,392
Post-Tax Free Cash Flow	**US$M**	**5,657**	**(1,159)**	**165**	**245**	**(129)**	**26**	**341**	**270**	**156**	**363**	**390**	**3,223**	**1,766**

22.4 Sensitivity Analysis

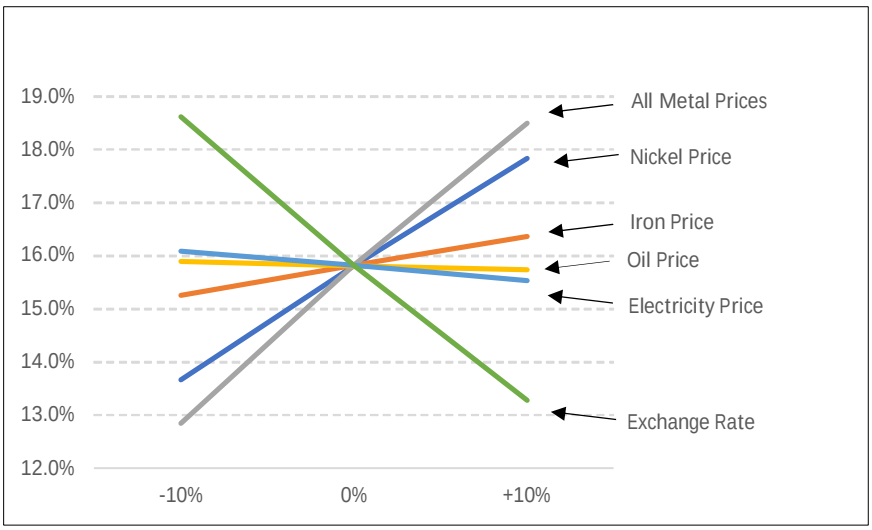
Figures 22-2 and 22-3 illustrate the sensitivity of NPV and IRR to a variety of macro-economic inputs. The single element with the greatest sensitivity is exchange rate, with a 10% variance (± US$0.075) having an impact of +28% / -29% on NPV and +18% / -16% on IRR. The NPV sensitivity to nickel prices (or nickel recovery and payability) is similarly ±29%, while the IRR sensitivity is lower at +13% / -14%. The sensitivity to iron prices is about one-third that of nickel, at +8% / -9% for NPV and +3% / -4% for IRR. Given the use of trolley-assist to reduce total diesel consumption in the pit by 40%, the project is relatively insensitive to oil price.

Figure 22.2: NPV8% Sensitivity to Macro-Economic Assumptions



Source: Penswick, 2021.

Figure 22.3: IRR Sensitivity to Macro-Economic Assumptions

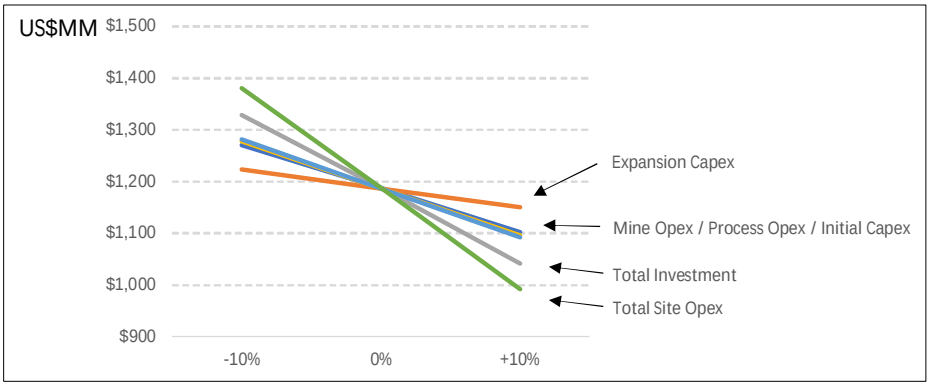


Source: Penswick, 2021.

Note that the base case analysis assumes carbon taxes would be applied to the price for diesel, with these taxes representing approximately 14% of the delivered cost for diesel. It has been reported by fuel suppliers that it may be possible to obtain an exemption from payment of the carbon taxes, in which case the NPV would increase by 2% to US$1,216 million and the IRR would increase by 1% to 16.0%.

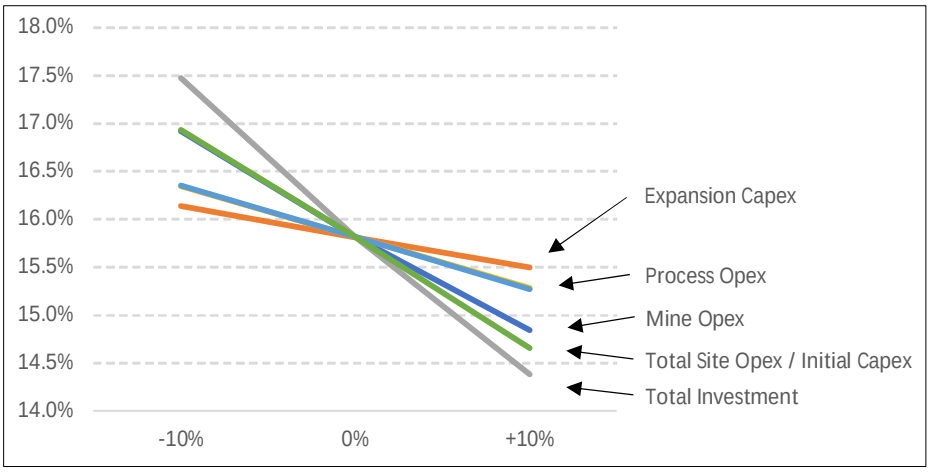
Figures 22.4 and 22.5 illustrate the sensitivity of NPV and IRR to capital and operating costs. NPV is more sensitive to operating costs, with a 10% variance in total site operating costs (± US$0.85/t milled) having an impact of ±16%, or a third higher than of total investment at ±12%. NPV is equally sensitive to mine operating costs, process operating costs and initial capital costs at ±8%, and relatively insensitive to expansion capital costs at ±3%. The ranking of capital and operating sensitivity reverses for IRR, with total investment at ±10% is 50% higher than the impact of total site operating costs at 7%. Mine operating costs and process operating costs have an equal impact at ±3% or 50% higher than expansion capital costs at 2%.

Figure 22.4: NPV8% Sensitivity to Operating and Capital Costs



Source: Penswick, 2021.

Figure 22.5: IRR Sensitivity to Operating and Capital Costs



Source: Penswick, 2021.

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 the Report.

24 OTHER RELEVANT DATA AND INFORMATION

This section has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

The Project discussed in this Report includes 24 months of pre-stripping and construction before start-up of the Phase 1 concentrator at the Crawford site at a steady-state throughput of 42.5 kt/d. However, there is an opportunity to start up a simpler and smaller scale operation in a much shorter time frame and for a lower capital cost. This smaller scale operation is referred to as “Phase 0”.

In January 2021, it was announced that CNC had entered into a non-binding memorandum of understanding (MOU) with Glencore Canada Corporation (Glencore) regarding the potential use and/or modification of Glencore’s spare processing capacity at the Kidd concentrator and metallurgical site (Met Site) approximately 40 km south of the Crawford. In the event of a positive outcome to the MOU discussions, the Phase 0 concentrator could be located at the Met Site. Its location at an active operation, coupled with a low planned throughput of 2.75 kt/d, would greatly simplify the permitting process.

A conceptual layout was prepared for the proposed metallurgical flowsheet based on a site visit by Ausenco in August 2020.

In the event that Phase 0 proceeds, there is further opportunity to utilize the processing infrastructure in the longer term for the higher value / higher PGE mineralization. In turn, the tonnage of higher value / higher PGE mineralization may justify a dedicated rail line that would allow for significantly lower transportation costs.

In the event that Phase 0 does not proceed, the 3.4 Mt of mill feed and 9 Mt stripping are included in the first year of mine production for the base case plan.

25 INTERPRETATION AND CONCLUSIONS

Section 25 (except 25.1 to 25.4, 25.6 to 25.7) has been taken directly from the technical report titled, "Crawford Nickel Sulphide Project: NI 43-101 Technical Report & Preliminary Economic Assessment", dated July 9, 2021 and with an effective date of May 21, 2021 (the "2021 PEA"). The mineral resource estimate update, as described in this Report, does not have a negative impact on or otherwise adversely affect the mineral resource inventory that formed the basis of the 2021 PEA. The results and conclusions of the 2021 PEA are still considered current and therefore have been carried-over for this Report.

25.1 Introduction

This Report was prepared and compiled by Caracle Creek and Ausenco under the supervision of the QPs at the request of CNC. This report has been prepared in accordance with the provisions of NI 43-101 Standards of Disclosure for Mineral Projects. The QPs note 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, Water Rights, Royalties and Agreements

The project comprises approximately 11,433.22 ha (~114.3 km^2), consisting of a combination of patented lands (Crown patents) and unpatented mining claims (staked claims), summarized in Tables 4.1 and 4.2 and shown in Figure 4-2.

Specifically, the property comprises 140 Crown patents (freehold patented lands) in Crawford and Lucas townships that cover approximately 9389.32 ha, 173 single cell mining claims (SCMC) and 2 multi-cell mining claims (MCMC) in Crawford Township covering approximately 2043.9 hectares. In this region of Ontario, one SCMC averages approximately 21.22 ha. The 140 Crown patents in Crawford and Lucas townships are mineral rights only (CNC does not control the surface rights), and are registered with the Land Registry Office, District of Cochrane (LRO 06).

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 (see CNC news release dated March 4, 2020), and a 2% NSR on patented lands (see Noble new release dated December 3, 2019, and December 19, 2019).

Annual holding costs for the 140 patented lands (mining tax) total approximately $37,557 and the required annual assessment work for the unpatented lands is approximately $40,200.

On the basis of the information provided by CNC and from what is available in the public domain, the Authors confirm that all of the unpatented and patented mining lands which comprise the Crawford Nickel Sulphide Project are in good standing.

25.3 Geology and Mineralization

25.3.1 Crawford Ultramafic Complex

The main target on the property is the Archean-age Crawford Ultramafic Complex, a differentiated ultramafic to mafic komatiitic flow (sill) that is hosted by the Deloro Assemblage of the AGB and comprises mainly dunite (+90% olivine) and peridotite (+40% olivine), which have undergone extensive serpentinization, along with minor gabbro and pegmatite which have all been cut by late felsic (aplite) and mafic dikes. The CUC is completely covered and as such is currently mainly defined by its geophysical signature (strong magnetic highs), a few historical diamond drill holes dating back to 1964, the more recent 2018 drilling by Spruce Ridge, and the current 2019-2020 (ongoing) drilling by CNC.

The ultramafic rocks (peridotite-dunite) from the CUC intersected in drill core have, for the most part, undergone intense serpentinization resulting in a substantial volume increase and the liberation of nickel and iron. This pervasive serpentinization process creates a strongly reducing environment where the nickel released from the decomposition of olivine is partitioned into low-sulphur sulphides like heazlewoodite and into the nickel-iron alloy, awaruite.

25.3.2 Deposit Model

Sulphide mineralization discovered to date on the Crawford property can be characterized as komatiite-hosted Ni-Cu-Co-(PGE) deposit type and most similar to the sub-type Mt. Keith style (Lesher and Keays, 2002). Of the five major volcanic facies for komatiitic flow fields suggested by Barnes et al (2004), the CUC is interpreted to be most similar to the dunitic compound sheet flow (DCSF), the same flow field facies interpreted for Mt. Keith. The DCSF facies represent high-flow volume magma pathways characterized by thick olivine-rich cumulates. Ultramafic rocks in the CUC are komatiitic, having magnesium oxide contents that range from 18.43 to 46.81 wt% MgO (determined by ICP Peroxide Fusion) and average 39.3 wt% MgO (937 samples).

The geological analogue for the CUC is the Dumont nickel deposit, hosted by the Dumont Sill in the Abitibi Greenstone Belt of north-central Quebec (Duke, 1986), 220 km east of the CUC. Stratigraphic studies in the AGB suggest that the host rocks of the Dumont Sill are correlative with the Deloro Assemblage, the same stratigraphy in which the CUC is hosted. Like the CUC, the thick olivine-rich cumulates in the Dumont are suggestive of DCSF komatiite volcanic flow facies.

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The 2018 diamond drilling program (Spruce Ridge) marked the first work on the CUC target since the 1960s. The first four drill holes completed in late 2018, were reported by Spruce Ridge and Noble (March 2019) to have broad intersections of pervasively serpentinized, low-sulphur dunite and peridotite with nickel concentrations ranging from 0.224% to 0.339% Ni over intervals ranging from 130.5 to 558 m.

As of the effective date of the Crawford Mineral Resource Estimate (May 19, 2022), a total of 291 drill holes totalling approximately 123,617.68 metres (up to hole CR22-248), have been completed by Canada Nickel and previous companies (Figure 10.1; Table 10.1). This includes drilling metres (7,564.7 m) from 36 abandoned holes. Additionally, 55 of the 76 drill holes were HQ size, completed for geotechnical and metallurgical testwork, whereas the remaining 236 drill holes used NQ size.

At the Main Zone, drilling to date has defined ultramafic hosted nickel mineralization within an ultramafic body with a minimum strike length of 1.9 km, 280 to 580 m wide, and at least 800 m deep. Mineralization remains open along strike to the northwest and at depth. In addition, a PGE reef (approximately 860 m in strike length) has begun to be defined within the northern margin of the ultramafic body.

At the East Zone, drilling to date has defined ultramafic hosted nickel mineralization within an ultramafic body with a minimum strike length of 2.6 km, 200 to 350 m wide, and at least 800 m deep. Mineralization remains open along strike to the east and west and at depth. In addition, two distinct PGE reefs have begun to be defined within the northern margin of the ultramafic body.

Initial drilling at the West Zone, about 850 m west-northwest of the western end of the Main Zone, has intersected nickel sulphide mineralization hosted by olivine-rich rocks, similar to that found at the Main and East zones. Drilling at the northern West Zone, the interpreted northern extension of the Main Zone, located about 825 m west-northwest of the East Zone and about 1 km north of the west end of the Main Zone, has intersected nickel sulphide mineralization hosted by olivine-rich rocks, similar to that found at the other zones.

25.5 Metallurgical Testwork

The metallurgical testwork program focused on:

- defining the mineralogy across the orebodies.
- developing a flowsheet that is robust for the treatment of the range of feed types.
- defining the potential for marketable concentrates.
- confirming the metallurgy performance with locked cycle tests.
- developing metallurgical recovery models for the value metals.
- assessing the risks and opportunities to be addressed for the foreshadowed feasibility study,

The orebody contains recoverable nickel in pentlandite and heazlewoodite to flotation concentrates, awaruite to magnetic concentrate and non-recoverable nickel in serpentine. Magnetite and chrome spinel are recoverable by magnetic separation as potentially marketable products.

Brucite is problematic for flotation and is separated to a fines stream to improve the differential flotation of the deslimed stream. However, brucite offers some potential for carbon capture in tailings storage.

The testwork program has allowed the development of recovery models that facilitated development of preliminary economic models and directed focus for testwork to manage potential risks and opportunities in future work.

Marketable products are a high-grade nickel concentrate (>35% Ni), a low-grade nickel concentrate (approximately 12% Ni) and an iron (48%)/chrome (3%)/nickel (0.2%) concentrate.

Metal recoveries to the concentrates vary with the feed mineralogy. Nickel and cobalt recoveries are principally modeled as a function of the S/Ni ratio. Iron recovery is a function of iron head grade. Chromium recovery requires further work and chromium recovery was fixed at 27% for all ore types for this Report.

Test work completed subsequent to March 2021 for the feasibility study was not used in the resource statement update and is not reported herein due to the incomplete nature of the work.

25.6 Resource Database

The updated Main Zone MRE and initial East Zone MRE are supported by a database that consists of 258 surface drill holes (111 in the Main Zone plus 66 in its western extension, and 81 in the East Zone) with a total of 52,431 core assay samples (including 6,054 QA/QC samples).

The QPs have reviewed the drilling, logging and sampling, quality assurance-quality control, analytical and security procedures for the 2018 to 2022 drilling programs and concluded that the observed failure rates are within acceptable ranges and that no significant assay biases or issues are present.

The QPs are of the opinion that the protocols in place are adequate and in general, to industry standards. The database for the Crawford Nickel Sulphide Project is of good overall quality and is appropriate for the purposes of mineral resource estimation. The measured density (specific gravity) of the host ultramafic rock units and sampling density allows for a

reliable estimate to be made of the size, tonnage and grade of the mineralization in accordance with the level of confidence established by the mineral resource categories in the CIM Definition Standards (CIM, 2014).

25.7 Mineral Resource Estimates

Current mineral resources for the project are constrained by theoretical open pit models. An updated Main Zone Mineral Resource Estimate (Main Zone deposit) and East Zone Mineral Resource Estimate (East Zone deposit) have been completed using all available information and data (see Tables 14.14 and 14.13). The mineral resources for the project were classified in accordance with the most current CIM Definition Standards (CIM, 2014).

In order to determine the quantity of mineralization that shows a "reasonable prospect for eventual economic extraction" using open pit mining methods, independent consultant David Penswick (P.Eng.), carried out a pit optimization using Datamine NPVS, which employs the Lerchs-Grossmann algorithm. This algorithm uses the net value of every block in the model to determine the ultimate extent of an open pit that maximizes overall project value.

The mineral resource estimates have been revised to include a conceptual pit envelope constraint that was developed using the following optimization parameters. Metal prices used were US$7.75/lb nickel, US$15/lb cobalt, US$200/tonne Fe recovered to magnetite, US$1,600/oz Pd, US$800/oz Pt. Different pit slopes were used for each layer (in degrees): 9.5 in clay, 21.8 in gravel, and 45 in rock. The exchange rate utilized was USD/CAD at $0.75. Mining costs utilized different values for overburden (clay, gravel), selective mining, and bulk mining, ranging from C$1.75 to C$3.15/t mined. Processing costs and general and administration costs for a 100 kt/d operation (similar to Dumont) were C$6.18/t. Selling costs are expected to be C$0.85/lb nickel.

It is important to note that the results from the 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.

It is the opinion of the QPs that both the updated Main Zone (Table 14.14) and East Zone (Table 14.13) mineral resource estimates, completed in accordance with the requirements of the NI 43-101, reasonably reflect the mineralization that is currently known on the Crawford Nickel Sulphide Project and that there are reasonable prospects for future economic extraction, likely using open pit and/or bulk underground mining methods.

The mineral resources are not mineral reserves as they do not have demonstrated economic viability. The estimate is categorized as inferred, indicated and measured resources based on drill hole data density, geological and grade continuity, variogram range criteria and specific interpolation parameters. The effective date of the mineral resource estimates is May 19, 2022, based on the drill hole data compilation status, cut-off grade parameters and pit optimization.

25.8 Exploration Targets

Despite having been quantified by the same methodologies used for classified resources, as has been thoroughly described in the report, tonnages and grades of exploration targets are conceptual in nature. Insufficient geological and sampling data prevents the definition of a mineral resource, and as such it is uncertain if further exploration will confirm the calculations presented in this section, or if the targets will be effectively delineated as mineral resources.

Tonnage and grade ranges for PGE (palladium and platinum) exploration targets from the Main Zone PGE reef domain are presented in Table 25.1 and from the East Zone's two PGE reef domains in Table 25.2. It's important to note that these figures were not updated in this MRE, and instead come from a previous calculation.

Table 25.1: Ranges for Exploration Targets in the Main Zone PGE Reef Domain

Domain	Tonnes (Mt)	Pd (g/t)	Pd (g/t)	PGE (g/t)
PGE	5 – 6	0.4 - 0.5	0.5 - 0.6	1.0 - 1.1

Source: Caracle Creek, 2020

Table 25.2: Ranges for Exploration Targets in the two East Zone PGE Reef Domains

Domain	Tonnes (Mt)	Pd (g/t)	Pd (g/t)	PGE (g/t)
PGE-1	8 -12	0.5 – 0.6	0.6 - 0.7	1.1 – 1.3
PGE-2	9 - 13	0.1 – 0.2	0.2 - 0.3	0.3 – 0.5
TOTAL	17 - 25	0.3 – 0.4	0.4 - 0.5	0.7 – 0.9

Source: Caracle Creek, 2020

25.9 Mine Plan

The mine plan, including both the production schedule and associated estimates of capital and operating costs, is considered to be achievable. A key element is use of autonomous mining equipment and trolley-assisted haulage. Both these technologies are considered proven, as they are employed at numerous sites globally on a commercial scale.

The mine plan underpins the cash flow model and financial statements developed for the PEA.

25.10 Process Recovery

A staged development approach has been adopted to mitigate technical and financial risk during the initial years of operation. The processing plant will initially be comprised of a single line with a nameplate throughput of 42.5 kt/d. Phase 2 will be expanded to two lines with a nameplate throughput of 85 kt/d after Year 4. Phase 3 will raise production to the ultimate rate of 120,000 t/d through the addition of secondary crushing and a third ball mill and additional downstream capacity.

The testwork proved that the Crawford material could be processed in a conventional wet grinding circuit followed by hydrocyclone desliming, nickel flotation, and magnetic recovery. The cleaning circuit is a multiple stage circuit with a regrind on the magnetic concentrate and cleaner tails.

Flotation testwork indicates that nickel recovery is relatively insensitive to grind sizes (P_{80}) up to about 180 µm. The most effective unit operation for improving flotation performance is an aggressive desliming stage to remove the fine particles that cause viscosity problems in the rougher stage. The production of both a low-grade and high-grade nickel concentrates, and a magnetite concentrate delivers a robust product more adaptable to market changes.

25.11 Infrastructure

The infrastructure for the project consists of open pit mines, a phased processing plant design, TSF, mine services, roads, and a wastewater treatment plant. In addition, off-site infrastructure will include the relocation of the Highway 655, as well as a HV powerline, and rail spur from Timmins.

A trade-off study was conducted to compare the costs of transporting nickel and magnetite concentrate by truck and by rail. It was decided to include a rail spur, although the cheapest option was a truck-rail combination, where concentrate is trucked to an existing transfer facility in Timmins for transport by rail to Port of Quebec. The key deciding factors were the fact that the rail spur will be utilized to deliver fuel, reagents and consumables, and explosives supplies.

The starter dams will confine an area to store approximately 47 Mm^3 of tailings and the final TSF dams will facilitate the storage of approximately 407 Mm^3 of tailings. The tailings dams will be constructed using rockfill with an upstream liner. The TSF dams will be raised in stages to optimize tailings storage and water management. Initial raises of the TSF dams will use the downstream construction method. In later stages of production coarse cyclone generated tailings would be deposited to provide a coarse tailings beach in support of centerline or upstream construction.

25.12 Environmental, Permitting and Social Considerations

CNC is actively engaging with stakeholders and local Indigenous communities as is a best practice at this stage of mine development. CNC are also establishing memoranda of understanding as appropriate with local First Nations to support future engagement and participation in the project.

There were no findings from the environmental studies or review that were in conflict with the proposed design. Additional studies are underway as needed to support completion of the anticipated impact / environmental assessment processes, as well as obtaining the necessary environmental approvals for construction and operations. The environmental effects from project are expected to be consistent with the scale of the mining operations in a northern Ontario setting. Reclamation on eventual closure of the mine will return the site to a more naturalized condition.

25.13 Markets and Contracts

The long-term price assumptions selected are US$ 7.75/lb for nickel (this is consistent with the average of industry analysts who follow the company and a leading independent nickel industry analyst), US$290/tonne for iron, and $US1.04/lb as of May 2021. As the iron and chrome content of the magnetite concentrate was being utilized in the production of stainless steel and nickel pig iron, the appropriate iron and chrome pricing basis (U.S. No. 1 heavy melting scrap for iron and U.S. ferrochromium for chrome), the company relied on the 10-year U.S. average No. 1 heavy melting composite price for iron price and the 10-year average ferrochromium price ($/lb chromium content) as tracked by the U.S. Geological Survey and a leading independent industry analyst for chrome prices.

Crawford will produce a high-grade (35% nickel) concentrate, a standard-grade concentrate (12% nickel), and a magnetite concentrate containing an average of 48% iron and 3% chrome. The combined Crawford concentrates, with a relatively high average nickel content of 20% nickel over the life of project, and substantial quantities of iron and chrome, is ideally suited for use as a high-quality raw material feed for use in stainless steel production, ferronickel, or nickel pig iron producers by roasting the sulphide concentrates. This is followed by production of stainless steel and nickel pig iron using standard rotary kiln/electric furnace and then refining in an AOD (RKEF/AOD).

CRU, a leading, provider of analysis, prices and consulting in the mining, metals and fertilizer markets, prepared a value-in-use study and market analysis that looked at the value that a third-party local producer would be willing to pay for the feed utilizing standard RKEF/AOD technology widely used in China and Indonesia to produce 304 stainless steel and 1.6% nickel pig iron. It found that net payabilities utilizing this approach achieve 91% payability for contained nickel, 71% payability for contained iron, and 43% payability for contained chromium in the concentrates.

No contracts are currently in place for any production from the project. It is expected that the sale of concentrate will include a mixture of long-term and spot contracts. It is recommended to continue to investigate metal price forecast in the next study phase.

25.14 Capital & Operating Costs

The estimate provided is classified as an AACE Class 5 estimate with an accuracy of ±50%, based on vendor quotations and consultant's experience for the project as described in the project.

The capital cost is evaluated at US$1,189 million for Phase I, and a total capital cost of US $1,925 million. The sustaining capital cost is evaluated at US $264 million for Phase 1 and total sustaining capital cost of US $1,091 million over the life of mine. The life of mine average operating cost is evaluated at US$8.45/t.

25.15 Economic Analysis

Crawford has been evaluated using forecast long-term prices of US$7.75/lb for nickel, US$1.04/lb for chromite, $290/t for iron and a Canadian exchange rate of US$0.75. Over a 25-year life, planned payable production from the project will include 1,689 Mlbs nickel, 1,388 Mlbs chromite and 15 Mt iron at a net all-in sustaining cost of US$1.96/lb Ni.

This plan results in an after-tax NPV 8% of US$ 1,187 million, an IRR of 15.8% and a payback period of 7.4 years.

Returns are most sensitive to the exchange rate, followed by factors impacting revenue; such as metal prices and recovery. Returns are more sensitive to variance in planned operating costs than to capital costs.

25.16 Risks and Uncertainties

Ausenco, and in consultation with both CNC and the projects team consultants are not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or relevant issues could be expected to materially affect the reliability or confidence in the current project development information and mineral resource discussed herein or the right or ability to perform future work on the Crawford Nickel Sulphide Project.

External risks are, to a certain extent, beyond the control of the project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the project's region, adequate surface rights, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource estimates.

As with all mineral exploration projects, there is an inherent risk associated with mineral exploration. Many of these risks are based on a lack of detailed knowledge and can be managed as more sampling, testing, design, and engineering are conducted at each of the next study stages. The mineral resources may be affected by a future conceptual study assessment of mining, processing, environmental, permitting, taxation, socio-economic, and other factors.

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. The CUC offers good potential for developing a low-grade, large tonnage nickel (Co, Pt, Pd, Fe) resource and should be investigated further. Its analogue, the Dumont nickel deposit (Dumont Sill) in Quebec, shares many similarities to the CUC and extensive exploration work at Dumont, largely diamond drilling, has resulted in the delineation of large tonnage, low-grade nickel resources and the delivery of a positive feasibility study (Ausenco, 2019).

Whether an economic size and grade of deposit can be developed from the CUC will be predicated largely on the success of metallurgical testwork and the price of nickel and other recoverable metals. The Crawford Nickel Sulphide Project is still at an early stage, but initial metallurgical work and mineralogical studies have shown that the nickel contained within the serpentinized ultramafic rocks of the CUC can be liberated. Critical to the success of this project is completing further thorough metallurgical test work to determine if the nickel could be economically extracted.

Given that the CUC is completely covered by 10 m (or more) of overburden, and with some 20 historical drill holes (including CR18-01 to 04) and 72 current drill holes (CR19-05 to CR20-73) targeting the CUC itself, there is much additional sampling

(*i.e.,* diamond drilling) required in order to understand the geology, mineralization, geochemistry, and geometry of the ultramafic body.

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.

25.17 Interpretation and Conclusions

It is the opinion of the Authors that additional exploration and development expenditures are warranted on the Crawford Nickel Sulphide Project.

Considering the positive outcome of this PEA report, it is recommended to pursue the next phase of the project. Next steps in the development of the project should focus on expansion drilling to extend current mineral resource limits and in-fill drilling to upgrade confidence in the categorization of current mineral resources.

26 RECOMMENDATIONS

26.1 Introduction

Considering the positive outcome of this Report, it is recommended to continue developing the project through additional studies, as outlined below. Table 26.1 summarizes the proposed budget to advance the project through the next study stage.

Table 26.1: Proposed Budget Summary

Description	Cost C$
Geology	15,000,000
Mining	2,000,000
Geotechnical	2,500,000
Metallurgy	3,000,000
Infrastructure	500,000
Environmental	1,500,000
FS Study budget	1,000,000
Total Recommended Study Budget:	**25,500,000**

26.2 Mineral Resources

Based on the work completed for the mineral resource estimate, which has outlined largely Indicated and Inferred categories of resources, it is recommended that programs aimed at in-fill drilling be undertaken in order to upgrade current resources to the measured category.

26.3 Mining

The following recommendations are made with regard to future mining studies:

- Updates to the resource model resulting from drilling conducted subsequent to the current study should be incorporated.
- A geotechnical program should be executed that includes the following:
  - accurately identifying the interface between the various lithologies, including clay, gravel and rock.
  - testing of each of the lithologies in order to determine slope angles for both the pit and various impoundments.
- execution of a hydrogeological work program to determine the impact of ground water on slope angles and other elements of the mine design; this work should additionally determine dewatering requirements for the operation.
- Condemnation drilling of the footprints identified for various surface infrastructure, including waste impoundments, should be carried out.
- Further work should be conducted to identify the optimal SMU size for various zones of mineralization; focus should be directed to the zones of cobalt and PGE mineralization that have not been included in the current study.

- Vendors of drilling and blasting equipment and services should be invited to test the various rock types in order to determine the optimal hole sizes and powder factors to be used.
- Potential mining contractors and equipment OEMs should continue to be engaged in order to obtain updated quotations.
- The design and schedule for the open pit mines and waste impoundments should be updated using inputs from the various work programs outlined above.

26.4 Metallurgical Testwork

The next phase of flowsheet development work should focus on optimizing the recoveries of nickel, iron chromium, cobalt, and PGM in the mineral processing flowsheet. The metallurgical testwork should include:

- optimization of the coarse rougher-scavenger flotation circuit.
- effect of the secondary regrind size and the secondary deslime on fine circuit flotation performance.
- optimization of the fines flotation circuit.
- optimization of the slimes cleaning process.
- evaluation of alternative strategies to recover nickel that is locked in silicates from the mineral processing tailings
- reduction in reagent consumption across the flowsheet and substitution of costly reagents such as CMC with less expensive alternatives.
- impact of regrinding the feed to the magnetic circuit on the quality of the magnetic concentrate (a trade-off study will be done to understand the cost-benefit analysis of further upgrading of the magnetic concentrate on flux requirements in downstream process steps).
- thorough assessment of the size-by-size deportment of nickel in a plant flowsheet with particular reference to hydrocyclone operation which tends to classify acicular particles and particles of different specific gravities differently.
- thorough assessment of the energy sensitivity in flotation with particular focus on the selection of plant equipment.
- thorough assessment of the dewatering properties of the tailings and the resulting storage and water balance factors.

The results of the flowsheet development work listed above will be tested at the pilot scale and used to lock in a final flowsheet for the feasibility study. The estimated cost for the flowsheet development work is C$1M.

26.5 Site Infrastructure

The following activities are recommended to support the design of the site infrastructure into the next phase of the project:

- Geotechnical site investigations should be carried out at the most optimal surface infrastructure site location to characterize the foundation requirements associated with the proposed surface infrastructure facilities.

- The final location of the TMF and mine dumps should be optimized to minimize the relocation of existing power lines and access roads.

26.6 Water Management

The goal of water management is to recycle the water from the TSF and feed the process plant with the required water quantity. The volume of the water in the TSF should be sufficient to support the ore processing in both dry seasons and winter conditions. Water from the slurry deposition and precipitation runoff will be collected in a small pond within the TSF. Water will be pumped to a recycle water pond located downstream of the TSF.

A detailed hydrologic/hydrogeological analyses and water balance must be completed to determine water intake and discharge requirements.

26.7 Tailings Management Facility

A preliminary geotechnical investigation at the TSF is required within the footprint of the dams to provide a better understating of the subsurface soil and groundwater conditions. The geotechnical investigation for the dams should be boreholes drilled along the proposed alignment of the dams. The next level of design should use the geotechnical investigation results to determine the stability and design of the TSF dams. Tests should be completed to determine properties of the deposited tailings as well.

26.7.1 Proposed Geotechnical Investigation

The feasibility design configuration of the dams can be established once physical and mechanical soil properties are available following sampling and field and laboratory testing. A preliminary geotechnical investigation at the TSF is required within the footprint of the dams to provide a better understating of the subsurface soil and groundwater conditions. The geotechnical investigation for the dams should be boreholes drilled along the proposed alignment of the dams. No less than 16 boreholes should be drilled for the initial investigation. Additional geotechnical investigations may be needed based on the findings of the initial investigation and feasibility design in areas of concern or variable conditions. In addition to field tests (SPT, Nilcon vane, etc.) carried out during the investigations, a number of laboratory tests should be carried out to classify the soils and the physical/mechanical properties (strength and compressibility characteristics, coefficient of permeability, maximum, and minimum dry density, optimum water content values etc.). The list of tests is shown below:

- Field tests:
    - Nilcon Vane.
    - standard penetration test (SPT).
    - cone penetration test (CPT).
    - dynamic cone penetration test (DCPT).
    - falling head permeability and packer testing.

- Laboratory tests:
  - particle size analysis.
  - hydrometer.
  - Atterberg limit.
  - specific density (specific gravity).
  - proctor test (compaction of the fill materials).
- Advanced laboratory tests:
  - constant and falling head permeability tests on natural and on compacted samples.
  - unconfined and triaxial compression tests.
  - direct shear tests.
  - compressibility/consolidation tests.

Bedrock samples from the available core boxes should be taken to the laboratory and their acid generation potential should be evaluated.

The results of all field and laboratory tests should be presented in a geotechnical factual report.

Test pits should be excavated inside the TSF area to evaluate if there are any suitable borrow areas for materials that could potentially be used for dam construction.

Instrumentation should be installed in the TSF dams and its native foundation material and should be monitored during the construction and operation of the TSF. Monitoring instrumentation may include (but not be limited to) vibrating wire piezometers, inclinometers, settlement cells, groundwater monitoring wells and survey monuments. As part of the routine operation of the tailings' facility, extensive monitoring of all aspects of the operation should be undertaken. Complete details of the monitoring program should be included in the operations manual that will be produced for the TSF at the detailed design stage. Monitoring should be carried out throughout all stages of the facility's life including construction, operation, closure, decommissioning, and post-closure.

26.8 Environmental, Permitting & Community Relations

CNC should continue to actively engage with stakeholders and local Indigenous communities on the project design elements going forward.

Environmental baseline studies and geochemistry studies should continue to be progressed in order to support a timely environmental approvals process, as well as to support the engineering design. As possible, arrangements should be made with local Indigenous communities to support completion of Traditional Knowledge / Traditional Land Use studies. CNC has already initiated discussions concerning these arrangements with the Wabun Tribal Council and Taykwa Tagamou Nation.

CNC should enter into the planning stage during 2021 for an Impact Assessment by preparing and submitting an Initial Project Description in the required format. This same document or a similar document, could be used to consult with the Provincial Ministries and obtain clarification on Provincial Environmental Assessment processes required.

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