Project # 18000-01

NI 43-101 TECHNICAL REPORT FEASIBILITY STUDY, ELK CREEK SUPERALLOY MATERIALS PROJECT, NEBRASKA

REPORT TO:

NioCorp Developments Ltd.

EFFECTIVE DATE:

April 16, 2019

NI 43-101 TECHNICAL REPORT

FEASIBILITY STUDY

ELK CREEK SUPERALLOY MATERIALS PROJECT NEBRASKA

Prepared for:

NioCorp Developments Ltd.

Project # 18000-01

Issue Date: May 29, 2019

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REVISION HISTORY

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24	May 23, 2019	Nordmin	QP's		Draft for review and consent.
25	May 29, 2019	Nordmin	Nordmin	QP's	FINAL

FORWARD-LOOKING STATEMENT

This document includes certain "forward-looking statements" which are not comprised of historical facts. Forward-looking statements include estimates and statements that describe the Company's future plans, objectives or goals, including words to the effect that the Company or management expects a stated condition or result to occur. Forward-looking statements may be identified by such terms as "believes", "anticipates", "expects", "estimates", "may", "could", "would", "if", "yet", "potential", "undetermined", "objective", or "plan". Since forward-looking statements are based on assumptions and address future events and conditions, by their very nature they involve inherent risks and uncertainties. Although these statements are based on information currently available to the Company, the Company provides no assurance that actual results will meet management's expectations. Risks, uncertainties and other factors involved with forward-looking information could cause actual events, results, performance, prospects and opportunities to differ materially from those expressed or implied by such forward-looking information. Forward looking information in this news release includes, but is not limited to, the Company's objectives, goals or future plans, statements, exploration results, potential mineralization, the estimation of Mineral Resources, exploration and mine development plans, the timing of the commencement of operations and estimates of market conditions. Factors that could cause actual results to differ materially from such forward-looking information include, but are not limited to the failure to identify Mineral Resources, failure to convert estimated Mineral Resources to reserves, the inability to complete a feasibility study which recommends a production decision, the preliminary nature of metallurgical test results, delays in obtaining or failures to obtain required governmental, environmental or other project approvals, political risks, inability to fulfill the duty to accommodate First Nations and other indigenous peoples, uncertainties relating to the availability and costs of financing needed in the future, changes in equity markets, inflation, changes in exchange rates, fluctuations in commodity prices, delays in the development of projects, capital and operating costs varying significantly from estimates and the other risks involved in the mineral exploration and development industry, and those risks set out in the Company's public documents filed on SEDAR. Although the Company believes that the assumptions and factors used in preparing the forward-looking information in this news release are reasonable, undue reliance should not be placed on such information, which only applies as of the date of this news release, and no assurance can be given that such events will occur in the disclosed time frames or at all. The Company disclaims any intention or obligation to update or revise any forward-looking information, whether as a result of new information, future events or otherwise, other than as required by law.

CAUTIONARY NOTE TO U.S. READERS REGARDING ESTIMATES OF MEASURED, INDICATED AND INFERRED RESOURCES

This Technical Report uses the terms “Measured” and “Indicated” Mineral Resources and “Inferred” Mineral Resources. The Company advises U.S. investors that while these terms are recognized and required by Canadian securities administrators, they are not recognized by the SEC.

The estimation of “Measured” and “Indicated” Mineral Resources involves greater uncertainty as to their existence and economic feasibility than the estimation of Proven and Probable reserves. The estimation of “Inferred” resources involves far greater uncertainty as to their existence and economic viability than the estimation of other categories of resources. It cannot be assumed that all or any part of a “Measured,” “Inferred” or “Indicated” Mineral Resource will ever be upgraded to a higher category.

Under Canadian rules, estimates of “Inferred Mineral Resources” may not form the basis of feasibility studies, pre-feasibility studies or other economic studies, except in prescribed cases, such as in a preliminary economic assessment under certain circumstances. The SEC normally only permits issuers to report mineralization that does not constitute “reserves” as in-place tonnage and grade without reference to unit measures. Under U.S. standards, mineralization may not be classified as a “reserve” unless the determination has been made that the mineralization could be economically and legally produced or extracted at the time the reserve determination is made. U.S. investors are cautioned not to assume that any part or all of a “Measured,” “Indicated” or “Inferred” Mineral Resource exists or is economically or legally mineable. Information concerning descriptions of mineralization and resources contained herein may not be comparable to information made public by U.S. companies subject to the reporting and disclosure requirements of the SEC.

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1. SUMMARY

Nordmin Engineering Ltd. (“Nordmin”), Optimize Group (“Optimize”), SRK Consulting (U.S.), Inc. (“SRK”), Tetra Tech (“Tetra Tech”), Adrian Brown Consulting (“Adrian Brown”), Zachry Engineering Corporation (“Zachry”), Metallurgy Concept Solutions (“MCS”) and SMH Process Innovation LLC (“SMH”) were retained by NioCorp Developments Ltd. “(NioCorp” or “the Company”) to complete a feasibility-level Canadian National Instrument 43-101 (“NI 43-101”) Technical Report (“Technical Report”) for the Elk Creek Superalloy Materials Project (“Elk Creek” or “the Project”) located in southeast Nebraska.

1.1 Principal Outcomes

The 2019 Technical Report is based on an assumption of processing of 36,313 (kt) over a 36-year life of mine (LOM) to produce 168,861 tonnes of Nb in the form of ferroniobium, 3,410 tonnes of Sc_2O_3 and 418,841 tonnes of TiO_2.

Initial capital costs are estimated at US$ 1,143 million. The initial capital estimate is partially offset by a Gross Pre-production Revenue Credit of US$ 265 million (generated by pre-production product sales) to net to a cost of US$ 879 million.

Total capital costs, inclusive of sustaining, closure/reclamation and contingency costs US$ 1,565 million.

Total LOM operating costs are estimated to be US$ 7,132 million.

On a pre-tax basis, the NPV (8% discount) is US$ 2,564 million, the IRR is 27.3%, and the assumed payback period is within 2.85 years.

On a post-tax basis, the NPV (8% discount) is US$ 2,098 million, the IRR is 25.8%, and the assumed payback period is within 2.86 years.

1.2 Terms of Reference

The Report was prepared to support disclosures in the NioCorp news release dated April 16, 2019, entitled “New Mine Design Expected to Deliver Higher NPV, Stronger Investment Returns, Accelerated Cash Flows, Longer Mine Life, Lower Risk and a further reduction of Environmental Impacts to NioCorp’s Elk Creek Project.”

The Report uses Canadian English and metric units unless otherwise indicated. Monetary units are in United States Dollars (US$).

1.3 Property Description and Ownership

The Elk Creek Project is a greenfield exploration project located in southeast Nebraska, USA. It is located approximately 105 km (65 miles) southeast of Lincoln, Nebraska (the state capital), and 129 km (80 miles) south of Omaha, Nebraska. The mineralization is centred about 40°16'0.3.5" N latitude and 96°11'08.5" E longitude. The area is well developed with direct access to roads, rail, supply and distribution companies, and a local workforce including heavy equipment operators. Geologists can be sourced from local universities. An experienced mining related workforce can be found in Denver, Colorado (eight-hour drive west of the Project). The deposit is located within the U.S. Geological Survey (USGS) Tecumseh Quadrangle Nebraska SE (7.5 minute series) mapsheet in Sections 1-6, 9-11; Township 3N; Range 11 and Sections 19-23, 25-36; Township 4N, Range 11.

The Property consists of 21 option agreements covering approximately 1,635 hectares (ha). Option agreements are between NioCorp's subsidiary Elk Creek Resources Corp. (ECRC) and the individual landowners. ECRC is a Nebraska based wholly owned subsidiary of NioCorp. NioCorp retains 100% of the mineral rights to the Project and is the operator. The agreements are in the form of five-year pre-paid Exploration Lease Agreement (ELA), with an Option to Purchase (OTP) the mineral rights and/or the surface rights at any time during the term of the agreement. The individual landowners have title to the surface and subsurface rights, and the agreements are primarily concerned with only the mineral and surface interest of each property. The agreements convey to the Company adequate surface rights to access the land and to complete mineral exploration work. The options agreements that the company currently holds include all the Indicated and Inferred resources described in this Technical Report.

The leases covering the Project are 100% owned by NioCorp and, apart from a 2% NSR royalty attached with the OTPs that include the mineral rights, have no other outstanding royalties, agreements or encumbrances.

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Property is easily accessible year-round as it is situated approximately 75 km southeast of Lincoln (State Capital), Nebraska and approximately 110 km south of Omaha, Nebraska. Access to the site can be completed via road or from one of the regional airports. There are several regular flights to both Lincoln and Omaha; however, the Project is most easily accessible from Lincoln.

Southeast Nebraska is situated in a Humid Continental Climate (Dfa) on the Köppen climate classification system. In eastern Nebraska, this climate is generally characterized by hot, humid summers and cold winters. Average winter temperatures vary between -10.4°C to 1.6°C. Average summer temperatures vary between 18°C to 32°C. Exploration and mining related activities may be conducted all year round.

1.5 History

Exploration activities at the Project prior to NioCorp ownership were conducted by the following companies: USGS, Cominco American, Molycorp and Quantum. These activities consisted of airborne magnetic and gravity surveys, geochemical sampling, Reverse Circulation (RC) drilling, core drilling and Mineral Resource Estimates.

Since 2014, NioCorp has completed metallurgical testing, core drilling, mineral resource updates in 2014, Mineral Resource update in 2015, two Preliminary Economic Assessments in 2015, and a Feasibility Study in 2017.

1.6 Geological Setting and Mineralization

The Project includes the Elk Creek Carbonatite (the Carbonatite) that intruded older Precambrian granitic and low to medium grade metamorphic basement rocks. Both the Carbonatite and Precambrian rocks are interpreted to be unconformably overlain by approximately 200 meters (m) of Paleozoic marine sedimentary rocks of Pennsylvanian age. As a result of this thick cover, there is no surface outcrop within the Project area of the Carbonatite, which was identified and targeted through magnetic surveys and confirmed through subsequent drilling. The available magnetic data indicates dominant northeast, west-northwest striking lineaments, and secondary northwest and north oriented features that mimic the position of regional faults parallel and/or perpendicular to the Nemaha Uplift.

The Carbonatite hosts significant niobium (reported as Nb_2O_5), titanium (reported as TiO_2) and scandium (reported as Sc) and is composed predominantly of dolomite, calcite and ankerite, with lesser chlorite, barite, phlogopite, pyrochlore, serpentine, fluorite, sulphides and quartz. Niobium is contained primarily within the mineral pyrochlore, and rare earth element (REE) mineralization is reported to occur as bastnäsite, parisite, synchysite and monazite.

The Elk Creek Deposit (the Deposit), as defined in the Mineral Resource Estimate, consists of niobium, titanium, and scandium mineralization that is primarily hosted within a magnetic (hematite) dolomite carbonatite (MCarb). There are three mineralized domains within the MCarb which have been modelled and defined as:

- High grade Nb_2O_5 /TiO_2
  - The current known extents of the high grade niobium, titanium, and scandium are approximately 750 m along strike, 400 m wide, and 800 m in dip extent below the unconformity.
  - The high grade Nb_2O_5/TiO_2 domain consists of sixteen individual zones (wireframes) with each zone having a slightly different orientation to one another but generally follow a trend of azimuth 100°- 130° and dip of 32°- 52°. Each wireframe used a minimum of ~1% Nb_2O_5 over 10 m cut-off grade; however, zones 9, 10, 13, 17, 18, and 19 used a minimum modelling criterion of 0.5% Nb_2O_5 over 10 m.
- High grade Sc
  - The high grade Sc domain consists of seven individual zones (wireframes) with each zone having a slightly different orientation to one another but generally follow a trend of azimuth 105°- 125° and dip of 41°- 47°. Each wireframe used a minimum of ~70 ppm Sc over 9 m cut-off; however, zones 3 and 4 used a minimum modelling criterion of 60 ppm over 9 m.
- Low grade
  - The current known extents of the low grade niobium, titanium, and scandium are approximately 830 m along strike, 500 m wide, and 850 m in dip extent below the unconformity.
  - The low grade Nb_2O_5, TiO_2, and Sc consists of one zone (wireframe) having an azimuth of 120° and dip of 74°. The low grade zone used ~0.3% Nb_2O_5 cut-off to create the boundary.

1.7 Drilling

Drilling at the Project was conducted in three phases. The first was during the 1970s and 1980s by the Molybdenum Company of America (Molycorp), the second in 2011 by Quantum, and the third and latest program in 2014 by NioCorp. To date, 129 diamond core holes have been completed for a total of 64,981 m over the entire geological complex. Of these, a total of 48 holes (33,909 m) have been completed to date in the mineralized area and used in the current Mineral Resource Estimate.

Since the completion of the April 28, 2015, Mineral Resource Estimate, additional drilling was completed. An additional five drill holes, totalling 3,353.1 m, were completed. This drilling was for the purpose of hydrogeological and geotechnical studies. These holes were used to assist in the geological modelling for the updated resource estimate.

All drilling has been completed using a combination of Tricone, Reverse Circulation (RC) or Diamond Drilling (DDH) in the upper portion of the hole within the Pennsylvanian sediments. All drilling within the underlying Carbonatite has been completed using DDH methods.

Before the core was split for sampling, depth markers were checked, the core was carefully reconstructed, washed, geotechnically and geologically logged for lithologies, alteration, structures, and mineralization, rock quality rating (RQD), photographed, and marked for sampling. A sampling of the holes for assay was guided by the observed geology. Logging and sampling information was entered into a database template on a computer which was integrated into the Project master digital database on a daily basis. The database provided to Nordmin was in Microsoft Excel® .csv spreadsheets containing collar locations surveyed in UTM coordinates, downhole deviation surveys, assay intervals with elemental analyses, geologic intervals with rock types, alteration and key structures.

Core recovery at the Project has allowed for representative samples to be taken and accurate analyses to be performed. All NioCorp drill hole collars have been surveyed using a Sokkia GS2700 IS GPS, which has 10 mm horizontal and 20 mm vertical accuracy. The trajectory of all drill holes was determined during drilling with a Devico DeviFlex survey tool, which is a nonmagnetic, electronic, multi-shot tool or a Reflex Gyro survey tool. Data points were collected at either 3.05 m or 6.1 m intervals. The 2014/15 program also used an ATV acoustic Teleview (ATV) downhole equipment to collect various geological features.

Nordmin is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. In Nordmin's opinion, the drilling, core handling, logging, and sampling procedures meet or exceed industry standards and are adequate for the purpose of Mineral Resource Estimation.

The quality assurance and quality control (QA/QC) program used at the Project included the insertion of standard reference materials (SRMs), blanks, and duplicates into the sample stream. Results from the QA/QC samples were continually tracked by NioCorp as certificates for each sample batch were received. If QA/QC samples of a sample batch passed within acceptable limits, the results of the sample batch were imported into the master database.

Only the contractor and NioCorp geological staff were authorized to be at drill sites and in the core processing facility. After logging, sampling and shipment preparation, samples were transported directly from the Project site to the lab by NioCorp staff.

Results of the QA/QC program were well documented by NioCorp. Nordmin has relied on documentation provided by NioCorp in addition to the audit of the QA/QC data. Nordmin considers the QA/QC protocols in place for the Project to be acceptable and in line with standard industry practice. Based on the data validation and the results of the standard, blank, and duplicate analyses, Nordmin is of the opinion that the assay and bulk specific density databases are of sufficient quality for Mineral Resource Estimation at the Elk Creek Deposit.

1.8 Data Verification

Nordmin's data verification steps included site visits during which Nordmin personnel reviewed core handling, logging, sample preparation and analytical protocols, density measurement system, and storage procedures. Nordmin also reviewed the geological interpretation, reviewed how the drill hole collar locations are defined and inspected multiple diamond drill hole collar locations, observed the data management system, obtained a copy of the master database and obtained laboratory certificates for all drilling assays. A review of the database indicated no significant issues. Nordmin has identified several further recommendations to NioCorp to ensure the continuation of a robust QA/QC program but has noted that there are no material concerns with the geological or analytical procedures used or the quality of the resulting data. Nordmin considers the resource database reliable and appropriate to support a Mineral Resource Estimate.

1.9 Mineral Processing and Metallurgical Testing

Mineral Processing

The feasibility-level comminution test work was completed in two stages at SGS Canada Inc. (SGS) in Lakefield, Ontario.

The test work results indicate that the Project ore is categorized as soft to moderately hard in terms of ore hardness, and amenable to standard grinding as well as a high-pressure grinding roll (HPGR) operation.

Hydrometallurgical Testing (Hydromet)

Metallurgical test work was conducted at SGS, Hazen Research and Kingston Process Metallurgy (KPM) throughout 2014, 2015, 2016 and 2017 to properly design the required process units for the conversion of the ore into a niobium concentrate suitable for further treatment into ferroniobium (FeNb) as well as marketable products of titanium dioxide and scandium trioxide. Test work consisted of multiple bench and pilot scale hydrometallurgical test programs aimed at further refining the final flowsheet using different reagents and technologies. Pilot test programs showed that high recovery rates of the niobium, scandium and titanium could be achieved, and that recycling and regeneration of reagents was also possible; thus, minimizing fresh reagent input and waste generation. Recoveries of 85.8% Nb_2O_5 and 93.1% Sc_2O_3 have been demonstrated while achieving 40.3% recovery of TiO_2.

Pyrometallurgical Processing (Pyromet)

Pyrometallurgical test work has been carried out at Kingston Process Metallurgy (KPM) in Kingston, Ontario, Canada. The testing performed has demonstrated that:

- The aluminothermic reduction of Nb_2O_5 precipitate to produce FeNb alloy regardless of the high level of TiO_2 in the precipitate;
- 96 % niobium recovery from the hydromet precipitate;
- Hematite powder (Fe_2O_3) can be used as the iron source for the aluminothermic reduction.

1.10 Mineral Resource Estimation

Nordmin was supplied with an individual drill hole database from NioCorp in text format. The database included 129 drill holes, and 41 were used in the estimation for they were within the Elk Creek Deposit while the remaining drill holes were drilled within the region but not specifically within the Elk Creek Deposit.

Nordmin examined and modelled the lithological and geochemical correlations between rock types, geochemistry, and mineralization. These correlations demonstrated that higher grade Nb_2O_5, TiO_2 was associated with increasing Fe_2O_3, U, and Th while higher grade Sc grades are strongly associated with higher concentrations of CaO, MgO, U, and Th.

Geological interpretations supporting the estimate were generated by Nordmin and reviewed by NioCorp. In total, three domains consisting of 24 wireframes, were constructed to support high grade Nb_2O_5/TiO_2, Scandium and low grade domains. Wireframes were initially created on 25 m sections and then adjusted on plan views to edit and smooth each wireframe where required. The wireframes terminate at plunge and depth due to lack of drilling. No wireframe overlapping exists within a given domain, but wireframes of different high grade domains do locally overlap. Due to the contrasts in the physical characteristics between the different mineral phases, Nordmin elected to create hard boundaries to separate the high grade mineralization from the low grade mineralization for each zone within each domain. This approach has the advantage of being able to interpret the mineralization in context with the deposit geology and associated geochemistry using explicit modelling. It is Nordmin's opinion that the explicit modelling approach minimizes risks compared to using implicit modelling for resource estimation within this deposit.

Samples were composited to 2 m lengths. High grade outlier assay values were capped, and non assayed intervals were assigned a minimum detection default grade. Variograms were constructed and were used to support search ellipsoid anisotropy, linear trends observed in the data, and Mineral Resource classification decisions.

A total of 2,043 specific gravity (SG) measurements were provided from onsite drill core measurements, taken mainly during 2010 and 2014. Measurements were taken from NQ, and HQ sized core using the weight in air versus the weight in water method (Archimedes). Nordmin determined that the required amount of SG measurements did not exist to estimate the entire block model directly. Additionally, it was determined that the lithology code was not an accurate indicator of changes to the specific gravity within the mineralized areas. More accurately, the changes to the SG are directly proportional to the changing Nb_2O_5, TiO_2 and Fe_2O_3 grades. The SG measurements increase with increasing Nb_2O_5, TiO_2, and Fe_2O_3 grades. Therefore, mean SG values were assigned to the block model based upon the ordinary kriged estimated Nb_2O_5 grade.

Interpolation methods included Ordinary Kriging (OK), Inverse Distance weighting to the second power (ID2) and Nearest Neighbour (NN). Zonal controls were used to constrain the grade estimates to within each low and high grade wireframe. These controls prevented samples from separate low or high grade wireframes from influencing the block grades of one another, acting as a "hard boundary" between the zones. A total of three nested, searches were performed on all zones. The search distances were based upon the variogram ranges outlined in Section 14.6.2. The search radius of the first search for the low and high grade Nb_2O_5, TiO_2, and Sc was based upon the first structure of the variogram, the second search being two times the first structure and the third search on the maximum of the second structure within the variogram. Search strategies for each domain used an elliptical search with a minimum of three samples and a maximum of twelve samples from a minimum of two holes in the first, second and third passes.

Nordmin validated the block model using swath plots, volumetric comparison of blocks versus wireframes, visual inspection, a parallel secondary estimation using Nearest Neighbour (NN) and inverse distance weighting to the second power (ID2) and statistical comparison of block grades with assay and composite grades. Nordmin found grade continuity to be reasonable and confirmed that the block grades were reasonably consistent with local drill hole composite grades.

Indicated and Inferred categories were assigned based on the following parameters:

- Indicated Mineral Resources: defined by 50 m by 75 m drill hole spacing demonstrating strong geological continuity between drill hole intercepts.
- Inferred Mineral Resources: defined by a drill hole spacing that is >50 m by 150 m drill hole spacing demonstrating reasonable continuity assumed between holes.

To fulfill the requirement to meet "reasonable prospects for eventual economic extraction," Nordmin estimated a potential underground mining cut-off grade using assumptions from previous technical studies and on known operating costs for underground (UG) mines operating in the region. Major portions of the project are amenable for underground extraction with a processing method to recover FeNb (as the saleable product of Nb_2O_5), TiO_2 and Sc_2O_3 products.

The breakeven NSR cut-off grade (CoG) was estimated by Nordmin using input costs for mining, processing and general and administrative as outlined in Section 14.10.

The Mineral Resource Estimate for the Elk Creek Deposit is reported in accordance with NI 43-101 and has been estimated in conformity with current CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. This estimate includes the estimated Mineral Resources for Nb_2O_5, TiO_2, and Sc (see Table 1-1). The effective date of the Mineral Resource Estimate is February 19, 2019.

In Nordmin's opinion, the estimation methodology is consistent with standard industry practice, and the Indicated and Inferred Mineral Resource Estimate for the Elk Creek Deposit are considered to be reasonable and acceptable.

Factors that may affect the resource estimate include: product price assumptions; changes in local interpretations of mineralization geometry and continuity of mineralization zones; changes to geotechnical, hydrogeological, and metallurgical recovery assumptions; input factors used to assess reasonable prospects for eventual economic extraction; assumptions as to social, permitting and environmental conditions; and additional infill or step out drilling or results obtained from future drilling programs.

Nordmin is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate that is not discussed in this Technical Report.

Table 1-1: Mineral Resource Estimate for Elk Creek, Effective Date February 19, 2019

Classification	Cut-off NSR (DIL) (US$/t)	Tonnage (t)	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (t)	TiO_2 Grade (%)	Contained TiO_2 (t)	Sc Grade (ppm)	Contained Sc (t)
Indicated	180	183,185,498	0.54	981,092	2.15	3,940,419	57.65	10,562
Inferred	180	103,992,535	0.48	498,864	1.81	1,886,181	47.38	4,928

Source: Nordmin, 2019. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.

- Mineral Resources are reported inclusive of the Mineral Reserve. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Nordmin does not consider them to be material.
- The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves (May 10, 2014) as required by NI 43-101.
- CIM definition standards for Mineral Resources and Mineral Reserves (May 2014) defines a Mineral Resource as:

- o "(A) concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".

- Historical samples have been validated via re-assay programs, and all drilling completed by NioCorp has been subjected to QA/QC. All composites have been capped and then composited where appropriate, and estimates completed used ordinary kriging. The concession is wholly owned by, and exploration is operated by NioCorp Developments Ltd.
- The project is amenable to underground longhole open stoping mining methods. Using results from metallurgical test work, suitable underground mining and processing costs, and forecast product pricing Nordmin has reported the Mineral Resource at an NSR cut-off of US$ 180/tonne.
- Economic Assumptions Used to Define Mineral Resource Cut-Off Value:

$$\text{Diluted NSR (US\$)} = \frac{\text{Revenue per block } Nb_2O_5 \text{ (diluted)} + \text{Revenue per block } TiO_2 \text{ (diluted)} + \text{Revenue per block Sc (diluted)}}{\text{Diluted tonnes per block}}$$

Parameter	Value	Unit
Mining Cost	50.00	US$/t mined
Processing	125.00	US$/t mined
General and Administrative	5.00	US$/t mined
Total Cost	**180.00**	**US$/t mined**
Nb_2O_5 to Niobium conversion	69.60	%
Niobium Process Recovery	82.36	%
Niobium Price	39.60	US$/kg
TiO_2 Process Recovery	40.31	%
TiO_2 Price	0.88	US$/kg
Sc Process Recovery	93.14	%
Sc to Sc_2O_3 conversion	153.40	%
Sc Price	3,675.00	US$/kg
Calculated CoG NSR diluted 6 %	180.00	US$/t

- Price assumptions for FeNb, Sc_2O_3, and TiO_2 are based upon independent market analyses for each product.
- Price and cost assumptions are based on the pricing of products at the "mine-gate," with no additional down-stream costs required. The assumed products are a ferroniobium product (in metal form, approximately 65% Nb and 35% Fe), a titanium dioxide product in powder form, and scandium trioxide in powder form.
- The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate CoG, considering extraction scenarios and processing recoveries. Based on this requirement, Nordmin considers that major portions of the project are amenable for underground extraction with a processing method to recover FeNb (as the saleable product of Nb_2O_5), TiO_2, and Sc_2O_3 products.
- The result of positive indications from the company's metallurgical testing and development program, titanium (TiO_2) and scandium (Sc) were added to the Mineral Resource Statement in February 2015. Both metals can be recovered with simple additions to the existing process flowsheet and will possibly provide additional revenue streams that may complement the planned production of ferroniobium.
- Nordmin has provided reasonable estimates of the expected costs based on the knowledge of the style of mining (underground) and potential processing methods.
- Nordmin completed a site inspection of the deposit by Glen Kuntz, BSc, P.Geo., Consulting Specialist - Geology/Mining, an appropriate "independent qualified person" as this term is defined in NI 43-101.

1.11 Mineral Reserve Estimation

The Project is currently in the exploration phase and has not been developed. Based on geotechnical information and mineralization geometry, an underground longhole stoping method (LHS) has been determined to be suitable for the deposit. Paste backfill will be used to allow for a high recovery of ore material.

The stopes dimensions are 15 m wide, and stope length varies based on Nb_2O_5mineralization grade to a maximum of 25 m per panel with a level spacing of 40 m. The variation on stope length allowed for optimization of the Nb_2O_5 grade with a minimal increase to operating costs. The level spacing of 40 m was beneficial to operating and sustaining capital costs. Each block is mined with a bottom-up sequence. A partial sill pillar level is designed to be left between these two mining fronts/blocks. The extraction of ore from the partial sill pillar level is expected to be 62.5% using production up-holes through 25 m of the 40 m thick sill pillar and is accounted for within the reserves. This methodology will allow partial mining of ore on the sill pillar level, while at the same time allowing the development of the lower mining block and establishing an early start to the mining of the upper mining block. Using this approach minimizes the impact on initial capital investment. The backfill was designed to have an adequate strength to allow for mining adjacent to filled stopes, thus eliminating the need for rib pillars.

There will be two shafts, which will minimize the amount of development through water-bearing horizons located in the first 200 m from surface. Both shafts will be excavated at the same time using conventional shaft sinking methods in conjunction with a freezing process through the first 200 m from the surface. The production shaft will facilitate main access and egress, material hoisting, fresh air intake, and material logistics. The ventilation shaft will serve as the mine exhaust system as well as a second means of mechanical egress. Mined ore will be transported from the stopes to the main production shaft hoisting system by underground LHD's, trucks, ore passes, crusher and conveyor circuit.

A 3D mine design has been created representing the reserve areas. The underground mine design process results in mine plan resources of 36.3 Mt (diluted) with an average grade of 0.81% Nb_2O_5, 2.86% TiO_2, and 65.7 ppm Sc. This estimate is based on a mine design using elevated CoGs and applying the US$ 180/t NSR CoG to capture all potential mineral reserves within the design. These numbers include a 95% mining ore recovery to the designed wireframes in addition to applying approximately 6% unplanned dilution.

Mineral Reserves were classified using the 2014 CIM Definition standards. Table 1-2 summarizes the underground Mineral Reserves. This Mineral Reserve Estimate is as of February 19, 2019.

Table 1-2: Underground Mineral Reserves Estimate for Elk Creek, Effective Date February 19, 2019

Classification	Tonnage (x1000 t)	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (t)	Payable Nb (t)	TiO_2 Grade (%)	Contained TiO_2 (t)	Payable TiO_2 (t)	Sc Grade (ppm)	Contained Sc (t)	Payable Sc_2O_3 (t)
Proven	-	-	-	-	-	-	-	-	-	-
Probable	36,313	0.81	293,321	168,861	2.86	1,039,050	418,841	65.7	2,387	3,410
Total	36,313	0.81	293,321	168,861	2.86	1,039,050	418,841	65.7	2,387	3,410

Source: Nordmin, 2019. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.

- The Mineral Reserve is based on the mine design, mine plan, and cash-flow model utilizing an average cut-off grade of 0.788% Nb_2O_5 with an NSR of US$ 500/mt.
- Nordmin considers that the Mineral Reserve is amenable for underground extraction with a processing method to recover FeNb (as the saleable product of Nb_2O_5), TiO_2, and Sc_2O_3 products.
- The economic assumptions used to define Mineral Reserve cut-off grade are as follows:
  - Annual life of mine (LOM) production rate of ~7,220 tonnes of FeNb/annum,
  - Initial elevated five-year production rate ~ 7,351 tonnes of FeNb/annum
  - Mining dilution of ~6% was applied to all stopes and development, based on 3% for the primary stopes, 9% for the secondary stopes, and 5% for ore development.
  - Mining recoveries of 95% were applied.
  - Price assumptions for FeNb, Sc_2O_3, and TiO_2 are based upon independent market analyses for each product.

- o Price and cost assumptions are based on the pricing of products at the "mine-gate," with no additional down-stream costs required. The assumed products are a ferroniobium product (metallic alloy shots consisting of 65%Nb and 35% Fe), a titanium dioxide product in powder form, and scandium trioxide in powder form.

Parameter	Value	Unit
Mining Cost	43.55	US$/t mined
Processing	108.16	US$/t mined
Water Management and Infrastructure	13.71	US$/t mined
Tailings Management	1.35	US$/t mined
Other Infrastructure	6.96	US$/t mined
General and Administrative	8.65	US$/t mined
Royalties/Annual Bond Premium	7.53	US$/t mined
Total Cost	**189.91**	**US$/t mined**
Nb_2O_5 to Niobium conversion	69.60	%
Niobium Process Recovery	82.36	%
Niobium Price	39.60	US$/kg
TiO_2 Process Recovery	40.31	%
TiO_2 Price	0.88	US$/kg
Sc Process Recovery	93.14	%
Sc to Sc_2O_3 conversion	153.40	%
Sc Price	3,675.00	US$/kg

- The Mineral Reserve has an average LOM NSR of US$ 538.63 /tonne.
- Nordmin has provided detailed estimates of the expected costs based on the knowledge of the style of mining (underground) and potential processing methods (by 3rd party Qualified Persons).
- Mineral reserve effective date February 19, 2019. The financial model was run post-February 2019, which reflects a total cost per tonne of US$ 196.41 versus US$ 189.91 (February 19, 2019, Mineral Reserve Details Table above). Nordmin does not consider this a material change.
- Price variances for commodities are based on updated independent market studies versus earlier projected pricing. The updated independent market studies do not have a negative effect on the reserve.
- Nordmin completed a site inspection of the deposit by Jean- Francois St-Onge, P.Eng., Associate Consulting Specialist – Mining, an appropriate "independent qualified person" as this term is defined in NI 43-101.

1.12 Mining Methods

Geotechnical

A geotechnical field characterization program has been undertaken to assess the expected rock quality. This program included logging core, laboratory strength testing, in situ stress measurements and oriented core logging of jointing. The results of this program have provided adequate quantity and quality data for the feasibility-level design of the underground workings.

A geotechnical assessment of the orebody shape and ground conditions has determined that longhole open stoping mining is an appropriate mining method. Stopes have been sized to maintain stability once mucked empty. A primary/secondary extraction sequence with tight backfilling allows optimization of ore recovery while maintaining ground stability. Primary stopes will be backfilled with cemented pastefill, while secondary stopes will be backfilled with low-cement pastefill and uncemented waste rock from underground development activities.

The design has been laid out using empirical design methods based on similar case histories. The stability of the 2017 feasibility study mine design has been checked with 3D numerical stress-strain models of the working, which included consideration for mine-scale faulting. The modelling results confirm that stopes and access drifts are predicted to remain stable during active mining, including areas adjacent to pastefilled primary stopes. The revised stope dimensions have been reverified using empirical design methods. The current design has not been reverified using numerical analyses, but this reverification is recommended as the mine design is advanced to the final design.

Ground support requirements have been based on empirical ground support methods and have considered variable levels of required ground support.

The location of underground infrastructure (i.e., shafts, ventilation raises, shops, etc.) have been situated to minimize the adverse impact of encountering geologic structures (i.e., weaker faults and shear zones).

Mine Design

Potential mining areas were identified using stope optimization within Datamine Minable Shape Optimizer software using a minimum mining stope width of 15 m, a stope height of 40 m, and a variable stope length perpendicular to strike to a maximum of 25 m. The variable stope length perpendicular to the strike allowed for optimization of the Nb_2O_5 grade. The stope optimizer output was reviewed on a level by level basis, and a 3D mine design was generated. Generally, stopes would be selected based on the minimum CoG or CoNSR. As the CoNSR value is much lower than the resulting average NSR stope value, the CoNSR was not the decisive factor in the stope optimization process. Rather than using a minimum CoG or CoNSR, the mine design targeted higher annual ferroniobium production during the first five years of ore delivery, which resulted in an averaged annual production rate of 7,351 tonnes per year over this period. The steady-state average annual ferroniobium production was 7,220 tonnes annually. This strategy results in a LOM NSR average value of US$ 538.63/t. Note that two mining blocks are included in the mining design. There is a partial sill pillar separating the upper and lower blocks. A portion of the ore within the partial sill pillar level is planned to be extracted (62.5%) and is accounted for within the mineral reserves. There is additional mineralization potential below the lower mining block. The mining blocks identified provide an approximate 36-year LOM; therefore, the additional design below the lower mining block was not completed at this time. The design includes stopes, development accesses, and necessary infrastructure.

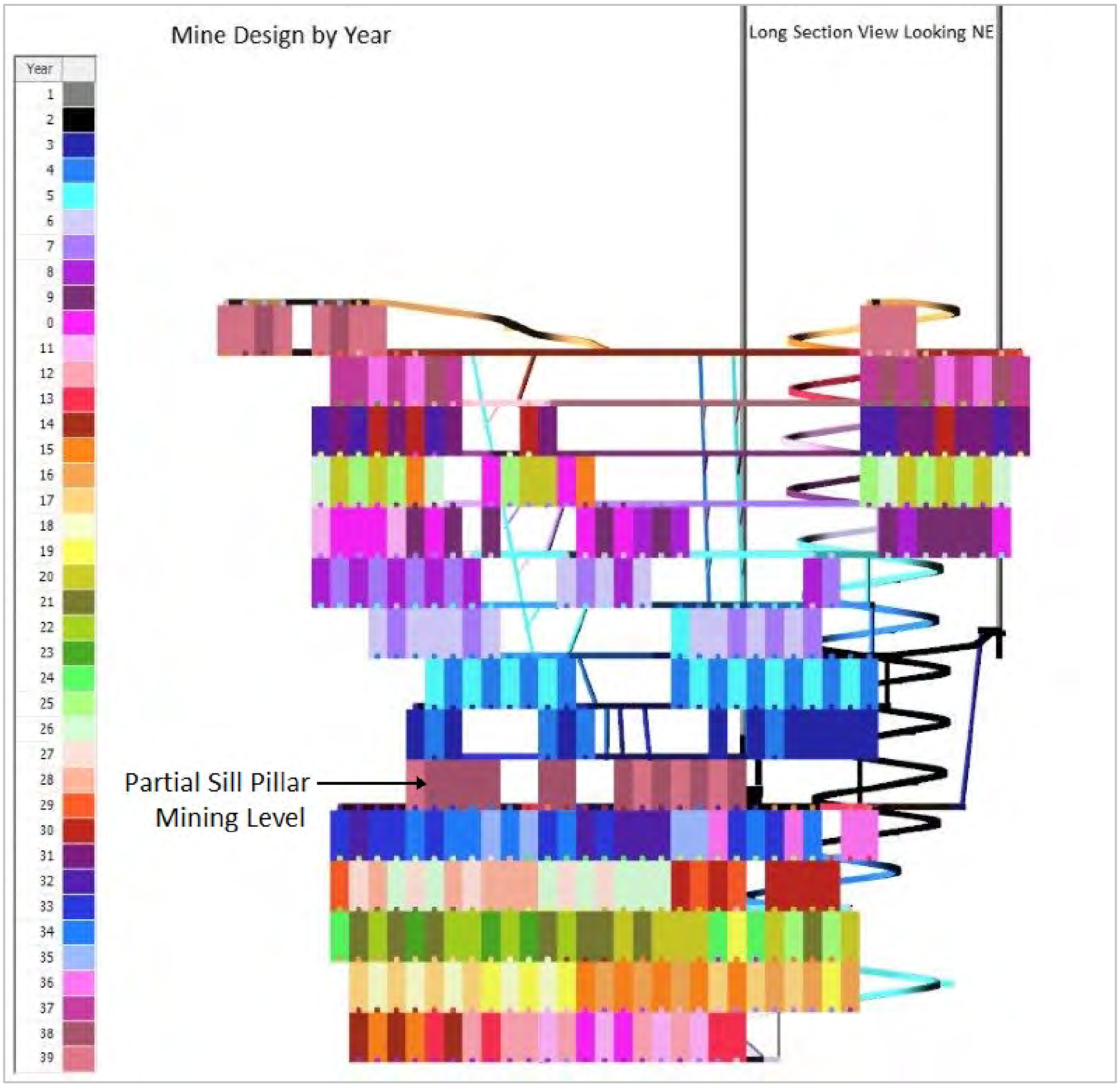
Stope optimizer shapes were used as a basis for the design work. Each stope has a 4.3 m x 4.0 m access located at the bottom of the stope. Top accesses are designed to give access to stopes on the next level and to allow for backfilling. The stopes are drilled from the top and rings are blasted from the end of a stope toward the footwall access. The blasted material is remotely mucked from the stope access. A primary/secondary stoping sequence will be used. The stope accesses are connected to a level access located in waste or low grade material. The level accesses connect to the main ramp which is offset at least 75 m from the footwall of the stopes. Each level is connected to an intake and exhaust ventilation system.

There is a significant increase in the overall depth and vertical extent of the mined ore zone from the previous SRK 2017 feasibility study. The designed vertical extent is 600 m with a bottom elevation of -495 m, versus the previous vertical extent of 450 m with a bottom elevation of -375 m. To efficiently develop the increase in depth and vertical extent, the production shaft and exhaust system (ventilation shaft), were excavated to lower depths. The ventilation shaft is designed to a 530 m depth versus the previous ventilation raise depth of 386 m. The production shaft is designed to a 755 m depth versus the previous depth of 440 m. The deeper production shaft and related crushing and conveying system is complemented with an ore pass and waste pass system that results in an overall material handling system that has suitable ore storage above and below the crusher station, fewer haulage trucks, and less ventilation requirements. The decrease in ventilation requirements in turn allowed for a 6.0 m diameter production shaft versus the 7.5 m diameter production shaft in the previous SRK 2017 feasibility study.

The underground mine will be accessed through a 6.0 m diameter production shaft system that will be excavated using conventional shaft sinking methods in conjunction with a freezing process through the first 200 m from the surface down to the potential water-bearing contact between the Pennsylvanian sediments and carbonatite unit, (reference Figure 7-5). This method, unlike a raisebore method of excavation, allows control of potential water inflows. Upon completion of the first 200 m section, the shaft sinking continues, but freezing is no longer required to reach the bottom elevation. The production shaft is excavated to a lower elevation than in the previous 2017 SRK feasibility study. This allows earlier access to higher grade ore in the central portion of the mine and to also access higher grade ore in the lower mining block.

A 6.0 m diameter ventilation/exhaust shaft is excavated concurrently using the same method as the production shaft and is outfitted with an emergency mechanical egress system. In addition, the sinking of the ventilation shaft, which is not as deep as the production shaft, allows for an earlier start to key lateral development via the ventilation shaft.

The production and development schedules were completed using Deswik scheduling software. A delay of 28 days was used prior to driving on paste fill or mining adjacent to a paste filled stope. A production rate of 2,764 t/d was targeted with ramp-up to full production as quickly as possible. Production shaft and ventilation shaft sinking preparation begins eight months after the commencement of detailed engineering with actual sinking beginning five months later and subsequent lateral mine development beginning nine months later. Production stoping begins 16 months after the start of lateral development, with a production ramp-up period through the next six months, after which the mine and plant are operating at full capacity. Figure 1-1 shows the mine production schedule coloured by year.



Source: Nordmin, 2019

Figure 1-1: Mine Production Schedule – Coloured by Year

The mine utilizes drill jumbos for lateral development equipped with tophammer drills. Production stope drilling utilizes down-the-hole drills. Rock bolters are used for ground support, and probe holes will be used to support mine grouting where required. The mine will operate a fleet of 40-tonne haul trucks being loaded by 6.2 m^3 (14 t) LHDs. The ore is fed through grizzlys with rock breakers into an underground crusher and via a material handling system to the surface. The mine has full infrastructure underground including ventilation, pumping system, electrical substation and distribution system, warehousing, explosives storage, communications system, and maintenance garage. The mine will have a staff of approximately 216 people at the peak of production.

1.13 Recovery Methods

Mineral Processing

The Mineral Processing building will house all of its equipment within a single large building. This building will be an engineered steel structure with dimensions approximately 61.9 m x 46.6 m (203 ft x 153 ft) with a 31.7 m (104 ft) eave height.

The primary driver of the comminution circuit design is the dry processing of ore. The process design relies upon two things; receiving a primary crusher product with a characteristic particle size of (P_{80}) 115 mm at the comminution circuit feed bin and producing feed material for the downstream hydrometallurgical processing at a characteristic particle size of (P_{80}) 1.1 mm.

The primary crusher product will be fed to the secondary cone crusher system, operating in closed circuit with a double deck screen. The screen undersize from the cone crusher system will be fed to an HPGR unit, operating in closed circuit with another double deck screen. The HPGR screen undersize is the comminution product that will report to the hydrometallurgical process.

Hydrometallurgical Processing (Hydromet)

The Hydromet Plant building is a very large multi-level engineered steel structure with dimensions approximately 167.6 m x 61 m (550 ft x 200 ft) with a 30.5 m (100 ft) eave height. The building will house the equipment on two levels for the 15 individual processes required to separate the three recoverable minerals. The Hydromet Plant is supported by a Hydrochloric Acid Regeneration (HCl) plant and a Sulphuric Acid Plant.

The Acid Regeneration (HCl) plant is composed of a large open air tank farm-type equipment area with containment capability situated next to a single large building. The open air tank farm area will contain the reactors, agitated tanks, heat exchangers and absorber columns needed for the process.

The Sulphuric Acid Plant serves as the environmental control device for scrubbing of SO_2 gas produced by calciners in the Hydromet area. This SO_2 gas will be scrubbed and regenerated to sulfuric acid which will be used in the Hydromet and other areas of the surface plant as a reagent. It is composed of a large open equipment area with dimensions approximately 82 m x 72.2 m (269 ft x 237 ft). Within this general area is a containment area that contains the absorber and drying towers, acid coolers, acid dryers and pumps. The remaining open area will contain the heat exchangers, superheater, converter and blowers used for the regeneration of sulphuric acid.

Pyrometallurgical Processing (Pyromet)

The Pyromet building will house most of its equipment within a single building. This building will be an engineered steel structure with dimensions approximately 45.7 m x 45.7 m (150 ft x 150 ft) with a 22.9 m (75 ft) eave height.

The purpose of the pyrometallurgical (Pyromet) plant is to reduce the niobium pentoxide coming from the hydromet feed by converting it into a saleable ferroniobium (FeNb) metal. Pyromet enhances the purification of the FeNb by eliminating the Ti co-precipitated with the Nb.

The aluminothermic reduction has been selected as the technology to convert the hydrometallurgical Nb_2O_5 precipitate into a FeNb metal. Aluminum shots and iron oxide pellets will be introduced on a continuous basis along with the fluxing agents to initiate and complete the exothermic chemical reduction of the Nb_2O_5. This reduction is performed in a single electrical arc furnace with a continuous feed of precipitate, additives and fluxes to produce a saleable a FeNb metal alloy.

1.14 Project Infrastructure

There are several local communities near the Project, including Elk Creek, Tecumseh and Lewiston that are intended to provide local housing for the Project construction and operating staff. There are several other communities within driving distance, and the large cities of Lincoln and Omaha are within reasonable driving distance. Both cities have substantial regional airports.

Presently, the site has no existing infrastructure except for access via the Nebraska state highway 50 and County Road 721. The Project will be accessed from County Road 721 through a guarded gate house into the Project property.

The Project will incorporate surface and underground infrastructure, as well as surface tailings and salt storage facilities. The offsite infrastructure includes a new high voltage transmission line constructed by the local utility company and providing power to an on-site primary sub-station and 45 km (28 mile) natural gas pipeline built by the owner of the interstate pipeline.

On-site power will include a 44 kV transmission line between the primary substation and the mine substation, along with a 13.8 kV on-site power distribution network. Telecommunications service will be provided by the local telecom supplier with on-site telecommunications distribution consisting of a combination of hardwire and fiber optics systems.

The on-site surface infrastructure will include:

- an electrical substation and distribution system;
- on-site telecommunications;
- fuel storage and dispensing system for above ground vehicles;
- fuel storage and dispensing system for diesel storage and pipeline transmission to underground mine fuel storage;
- truck scale;
- process water treatment center;
- potable water/fire water system including tankage, distribution and hydrants;
- sanitary wastewater collection system with lift stations discharging into above grade tanks for transport/disposal to a local municipal wastewater plant;
- natural gas distribution to site loads; and
- access roads to the site with parking, fencing and security.

Infrastructure building facilities will include the following: leased modular trailers for the administration building and security gate house, assay laboratory, combination warehouse and maintenance shop, modular warehouse/maintenance shop offices, process water treatment plant building, and the mine change building.

The mining related facilities will include a lined mine waste rock and ore storage area, surface water control facilities, and the tailings and salt impoundment. The mine surface facilities include two headframes and their associated hoisthouses, mine substations, temporary power generation system, paste backfill and cement plant, a multi-use facility comprised of a warehouse, maintenance shops, mine dry, and the administration building. The underground will be serviced by the production and ventilation shafts.

The underground facilities will include a shop, warehouse, fuel storage and filling area, offices, explosives storage areas, electrical distribution system, water pumping and discharge system, process water distribution, compressed air distribution, and the backfill distribution system. The underground material handling system includes a grizzly, feeder, crusher, storage bins, conveyors, and a skip loading system that loads skips in order to hoist the mined material to the surface facility.

A temporary contractor constructed and operated freeze plant will be located adjacent to the headframes in order to service both shafts simultaneously. The freeze plant will be operated during the early phases of shaft sinking and will draw power from the temporary power generation system until permanent site power is available. The temporary plant will be removed from the site during the final stages of shaft sinking.

1.14.1 Tailings

The tailings produced by the process plant will consist of filtered water leach residue, calcined excess oxide, and slag tailings. The four tailings storage facilities (TSF) will be constructed to contain the tailings; the first three cells in the Plant Site area, later in the mine life Area 7. The TSFs will be located in the Plant Area for the first 18 years of operation, and in Area 7 from Year 19 through Year 36. The Plant Site TSFs would contain approximately 7.8 Mt of tailings. The Area 7 TSF would contain an additional approximately 6.7 Mt of tailings. The TSFs have been designed to incorporate two independent areas: a composite-lined tailings solids storage area; and an area with double lined containment including a leak collection and recovery system for management of stormwater runoff and drainage from the tailings solids. The TSFs will store predominantly dry (i.e., not in a slurry consistency) tailings from the plant with embankment construction based on a "downstream" construction method. Facility closure is considered in the design.

1.15 Markets and Contracts

Market studies for niobium, titanium dioxide and scandium trioxide are an important part of the proposed Elk Creek Mine. These products, especially niobium and scandium trioxide (scandium), are thinly traded without an established publicly available price discovery mechanism.

Marketing studies and product price assumptions are based on research and forecasts for the following products:

- Ferroniobium (FeNb): Roskill’s Global Industry, Markets and Outlook 2018 (Roskill, 2019)
- Scandium Trioxide (Sc_2O_3): OnG Commodities LLC (OnG, 2019) - specializes in the scandium alloys and scandium markets.
- Titanium Dioxide (TiO_2): USGS Commodity Market Summaries (Bedinger, 2019) and Adroit Market Research (Androit, 2019).

NioCorp is considering selling ferroniobium, scandium trioxide and titanium dioxide products from the Project through all avenues, which include entering into long-term contracts with buyers.

At the time of this report, NioCorp had entered into three off-take agreements covering ferroniobium and scandium trioxide production from the Project.

No off-take agreements have been executed at the time of the report for the titanium dioxide product from the Project. It is assumed this product and all other material not covered by an off-take agreement will be sold on a spot price, ex-mine gate basis.

1.16 Environmental Studies, Permitting and Social or Community Impact

NioCorp has developed information and conducted a number of environmental studies related to baseline characterization for the Project. These include:

- Soils
- Climate/Meteorology/Air Quality
- Cultural and Archeological Resources
- Vegetation
- Wildlife
- Threatened, Endangered, and Special Status Species
- Land Use
- Hydrogeology (Groundwater)
- Hydrology (Surface Water)
- Wetlands/Riparian Zones
- Geochemistry

The geochemistry and characterization/classification of the ore and waste materials (including the final process waste streams making up the bulk of the tailings mass and the crystallized RO water treatment salts), directly influences the management of these materials given the presence of naturally occurring radioactive materials (NORMs) (i.e., uranium and thorium) and the potential for limited reaction to contact with water. These materials currently classify as non-hazardous based on regulatory testing. Site-wide management of non-contact and contact stormwater will be essential to the Project compliance. Given the presence of low levels of NORMs and this potential reactivity, NioCorp will take the conservative approach of placing this material in a double-lined containment facility from which any surface water runoff or seepage can be controlled and managed. It is not anticipated that any of mine development or waste rock would remain exposed on the surface post closure.

Characterization of the various tailings materials has included both the Toxicity Characteristic Leaching Procedure (TCLP) and the Synthetic Precipitation Leach Procedure (SPLP), which are designed to determine the mobility of both organic and inorganic analytes present in the liquid, solid, and multiphasic wastes, and assist in the proper classification of waste materials. The most recent tailings material testing showed negligible mobility of regulated constituents (indicating a non¬hazardous classification), although the pH of the TCLP/SPLP extracts remained high. While the calcined tailings are likely to produce heat when exposed to atmospheric moisture and precipitation (i.e., exothermic hydration), this reaction is not "violent" as defined under 40 CFR § 261.23(2) Characteristic of reactivity [for hazardous wastes] (adopted by the State of Nebraska under Title 128 - Nebraska Hazardous Waste Regulations). Given the limited quantities of ore available for testing, further characterization of these materials is recommended in order to establish representativeness of the deposit as a whole with respect to waste classification.

There are currently no known environmental issues that could materially impact NioCorp's ability to extract the Mineral Resources or Mineral Reserves at Elk Creek. However, there are several key permitting risks and uncertainties that could affect the Project financing and schedule. These are outlined below.

Overburden developed during mine construction will be excavated, crushed and used as a construction material. Small quantities of waste rock will be temporarily stored on the surface (on a lined pad) prior to final disposal underground or within the lined tailings impoundment. The water leach residue tailings will be returned to the underground mine as backfill, along with a portion of the oxide tailings. The remaining oxide tailings and slag will be deposited within double-lined, surface storage facilities.

For the first years of construction during the advancement of the shafts and underground development, there will be a requirement for subsurface dewatering. Dewatering will continue throughout the LOM. The initial volumes of saline water from testing of the dewatering wells will be stored on the surface and treated/used. During operations, clean water produced from the RO water treatment system(s) will be used in the process plant, and the reject concentrate will be evaporated and crystallized for disposal as solid waste material.

Upon cessation of mining, the facilities will be closed in accordance with state requirements and best industry practice. Until such time that the final TSF closure cover can be constructed, and any residual water or seepage eliminated, the TSF contact water will require active management.

Engagement of local, state and federal regulators has commenced. Initiation of the formal permitting program for the Project is dependent upon the completion of the mine plan and surface facilities being developed as part of this technical document, as well as additional characterization of the waste materials and potential worker exposures under the jurisdiction of the Nebraska Department of Health and Human Services (DHHS) and U.S. Department of Labor — Mine Safety and Health Administration (MSHA), both of whom will have primary oversight of worker safety and monitoring programs with respect to the presence of NORMs in the ore and waste rock. A comprehensive table of requisite permits and authorizations is presented in Section 20.3.

Stakeholder engagement has been undertaken in parallel with field operations in Nebraska and has included town hall discussions and individual meetings. Some early communications have occurred between NioCorp and Johnson, Pawnee, Nemaha and Richardson County representatives (including the county commissioners) as well as the Southeast Nebraska Development District.

Without specific hardrock mining regulations, there are limited obligatory requirements for reclamation and closure of mining properties in Nebraska. There are provisions, however, within the applicable regulatory framework that is likely to be applied to the Project during the permit and licensing processes, specifically those associated with the TSF and mineral processing facilities. This will include the provision of financial surety for proper closure and reclamation of the site. The currently estimate direct costs for closure and reclamation of the Project, plus financial assurance premiums, is US$ 50.2 million.

Overall, the Project appears to be sufficiently advanced to initiate the submission of formal permit applications which will govern the construction, operation, and closure of the mine.

1.17 Capital Cost Estimate

The estimate meets the classification standard for a Class 3 estimate as defined by AACE International and has an intended accuracy of ± 15%. The estimate is reported in Q1 2019 U.S. constant dollars. The capital cost estimate reflects a detailed bottom-up approach that is based on key engineering deliverables that define the project scope. This scope was described and quantified within material take-offs (MTO's) in a series of line items. Capital costs are divided among the areas of underground mining, processing, infrastructure, water management, tailings management, mining indirects, mining and processing commissioning and contingency. Sustaining capital costs are related to underground mining fixed equipment and development, process plant, infrastructure maintenance, tailings management, mine closure and contingency.

The mining, processing and infrastructure capital costs were developed, including a combination of vendor and contractor quotations, first principles buildup, allowances, and historical database costs. The estimates include labour, materials, equipment purchase and operation cost, rental equipment, supplies, freight, and energy. The costs developed include direct and indirect costs and included separate contingencies on both. Equipment purchase includes freight, an allowance for transporting underground, initial training and commissioning. The tailings and water management capital costs were based on contractor estimates for earthworks and liner installation. The estimates were developed from recent and relevant costs on other projects or developed from first principles.

Table 1-3 shows the breakout in LOM initial and sustaining capital estimates, which total US$ 1,609 million. An overall 9.67% contingency factor has been applied to the initial capital estimate, while a smaller 6.25% contingency was applied to the sustaining capital estimate. The pre-production period is defined from April 2019 to the end of construction in June 2022 plus a six-month ramp-up period through the end of December 2022. Commercial production is then to be declared on January 1, 2023. The initial capital estimate of US$ 1,143 million will be partially offset by a Gross Pre-production Revenue Credit of US$ 265 million, (generated by pre-production product sales) which equates to a net cost of US$ 879 million.

Table 1-3: Capital Costs Summary (US$ 000's)

Description	Initial	Sustaining	Total
Capitalized Pre-production Expenses	82,531	-	82,531
Site Preparation and Infrastructure	40,569	15,007	55,576
Processing Plant	367,439	96,448	463,886
Water Management & Treatment	73,756	23,613	97,369
Mining Infrastructure	256,731	180,438	437,170
Tailings Management	21,423	78,855	100,277
Site Wide Indirects	7,368		7,368
Processing Indirects	96,028		96,028
Mining Indirects	39,766		39,766
Process Commissioning	13,350		13,350
Mining Commissioning	1,444		1,444
Owner's Costs	33,619		33,619
Mine Water Management Indirects	8,520		8,520
Closure and Reclamation		44,267	44,267
Subtotal	**1,042,542**	**438,628**	**1,481,171**
Contingency	100,797	27,429	128,227
Total Capital Costs	**1,143,340**	**466,058**	**1,609,397**
Pre-production Revenue Credit	(264,747)		(264,747)
Net Project Total	**878,593**	**466,058**	**1,344,651**

Source: Nordmin, 2019

1.18 Operating Cost Estimate

Operating cost estimates were developed to show monthly and annual costs for production. All unit costs are expressed as US$/tonne processed and are based on Q1 2019 US$.

Operating cost metrics in the technical economic model are reported on a LOM basis meaning that all of these unit rates are stated on a LOM basis where the costs are estimated from the beginning of construction to the end of mine life. LOM operating costs include the pre-production and first/last years of production.

The total LOM operating cost unit rate of US$ 196.41/t processed is summarized in Table 1-4.

Table 1-4: LOM Operating Cost Unit Rate Summary

Description	LOM US$/t ore
Mining Cost	43.04
Process Cost	106.70
Water Management Cost	16.78
Tailings Management Cost	1.99
Other Infrastructure	5.47
Site G&A Cost	8.29
Other Expenses	6.30
Subtotal	**188.56**
Royalties/Annual Bond Premium	7.84
Total LOM Operating Costs	**196.41**

Source: Nordmin, 2019

1.19 Economic Analysis

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this Technical Report include, but are not limited to, statements with respect to future niobium, scandium and titanium prices, the estimation of Mineral Resources and Mineral Reserves, the estimated mine production and niobium, scandium and titanium recovered, the estimated capital and operating costs, and the estimated cash flows generated from the planned mine production.

Actual results may be affected by:

- Differences in estimated initial capital costs and development time from what has been assumed in this Technical Report.
- Unexpected variations in the quantity of ore, grade or recovery rates, or presence of deleterious elements that would affect the process plant or waste disposal.
- Unexpected geotechnical and hydrogeological conditions from what was assumed in the mine designs, including water management during construction, mine operations, and post mine closure.
- Differences in the timing and amount of estimated future niobium, titanium dioxide and scandium trioxide production, costs of future niobium, titanium dioxide and scandium trioxide production, sustaining capital requirements, future operating costs, assumed currency exchange rate, requirements for additional capital, unexpected failure of plant, equipment or processes not operating as anticipated.
- Changes in government regulation of mining operations, environment, and taxes.
- Unexpected social risks, higher closure costs and unanticipated closure requirements, mineral title disputes or delays to obtaining surface access to the property.

The production schedules and financial analysis annualized cash flow tables are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Technical Report and may result in changes to the calendar timelines presented and the information and statements contained in this Technical Report.

The technical, economic model metrics are prepared on an annual pre-tax and after-tax basis, the results of which are summarized in Table 1-5. Based on current assumptions and design listed in this report, the Project returns a pre-tax NPV 8% of US$ 2,564 million and an IRR of 27.3% along with an after-tax NPV 8% of US$ 2,098 million and IRR of 25.8%.

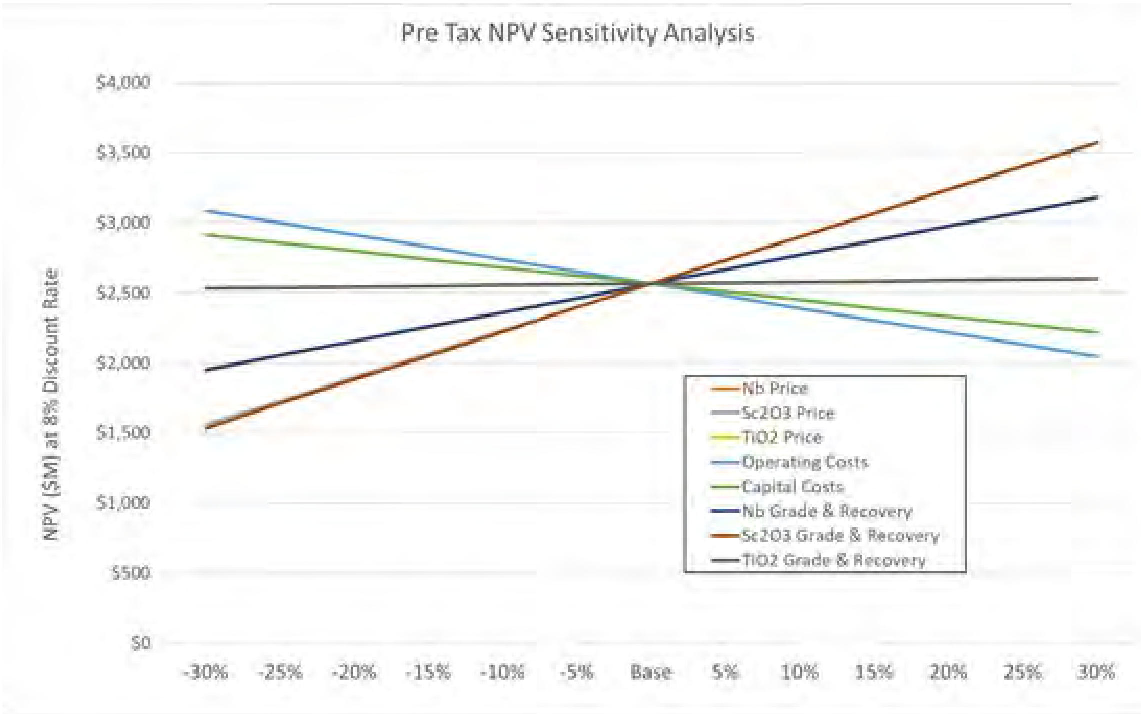
Table 1-5: Project Metrics Summary

Description	Value
Gross Revenue	US$ 20,807,083
Operating Costs	(US$ 6,717,578)
Operating Margin (EBITDA)	US$ 13,675,041
Income Tax	(US$ 2,319,660)
Working Capital	0
Operating Cash Flow	US$ 11,354,694
Initial Capital	(US$ 1,143,340)
Sustaining Capital	(US$ 421,791)
Reclamation/Salvage Capital	(US$ 44,267)
Total Capital	(US$ 1,609,397)
Pre-tax Free Cash Flow	US$ 12,064,956
Pre-tax NPV @ 8%	US$ 2,564,433
Pre-tax IRR	27.3%
Pre-tax Undiscounted PB from Start of CP (Years)	2.85
After-tax Free Cash Flow	US$ 9,745,296
After-tax NPV @ 8%	US$ 2,098,167
After-tax IRR	25.8%
After-tax Undiscounted PB from Start of CP (Years)	2.86

Source: Nordmin, 2019

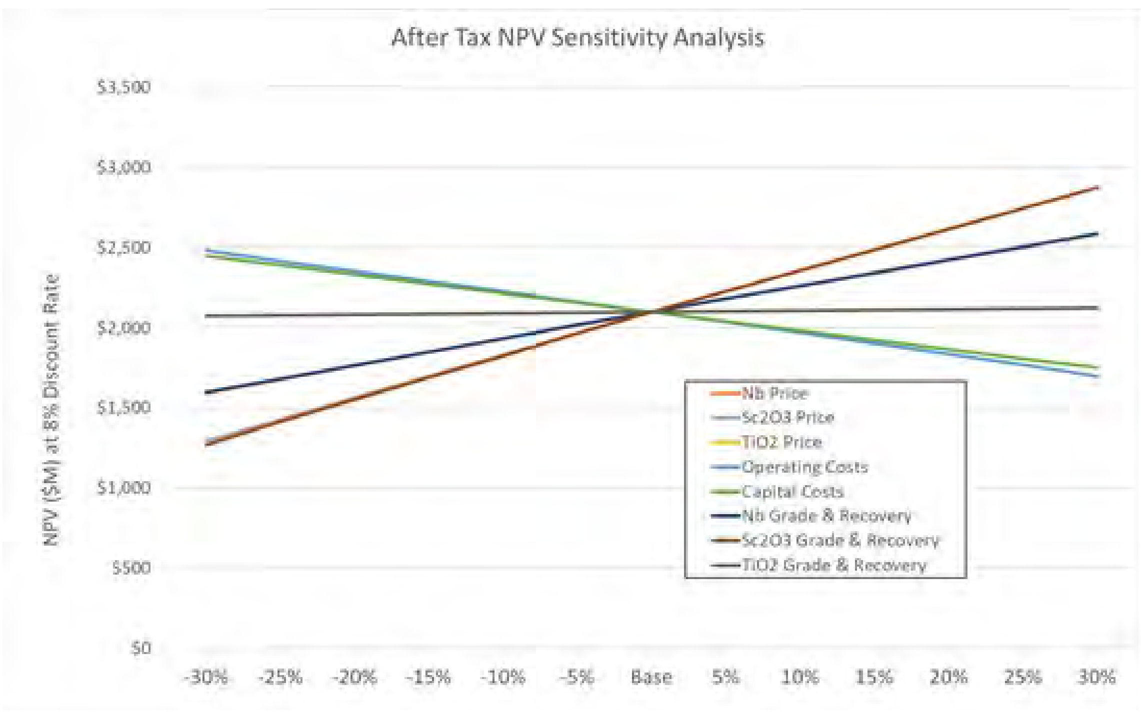
The cash flow model was tested for sensitivity to variances in milled tonnes, head grades (Nb, Sc, and Ti), process recoveries (Nb, Sc, Ti), metal prices, initial/sustaining capital expenditure and operating costs (mining, processing, water management, tailings management, site G&A and royalties).

Figure 1-2 and Figure 1-3 illustrate the results of pre/post tax basis with respect to four of the operational parameters and product prices along with recovery and head grades. The anticipated project cash flow is sensitive to the price of scandium and niobium compared to capital and operating costs, which were both quite similar.



Source: NioCorp, 2019

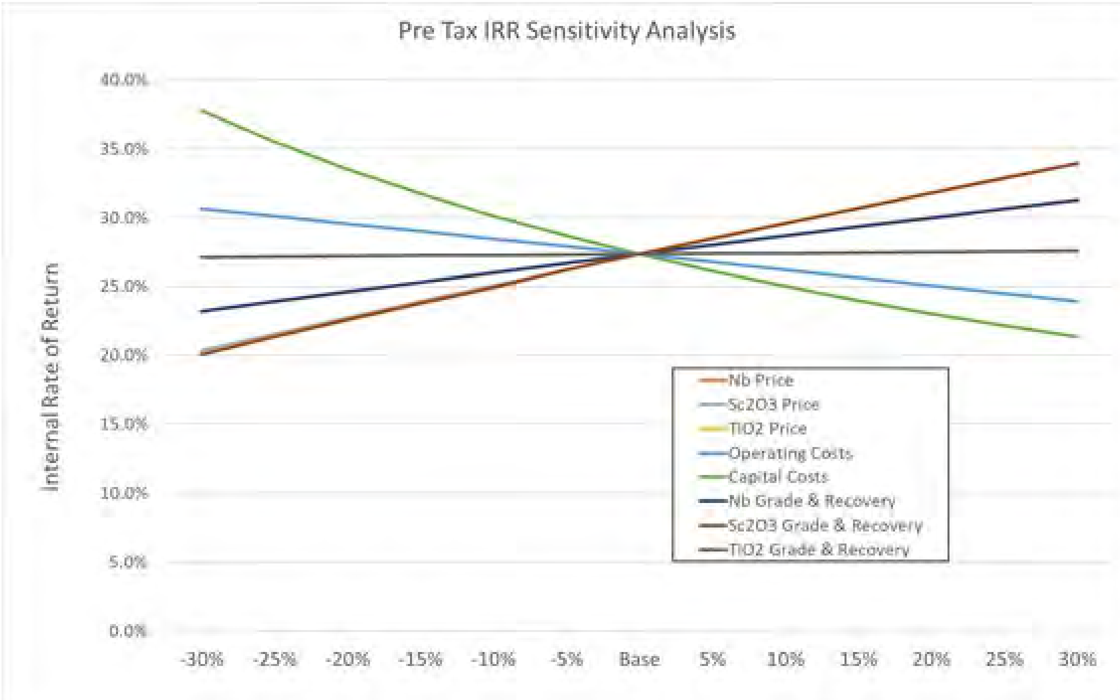
Figure 1-2: Pre-Tax NPV 8% Sensitivity Graph



Source: NioCorp, 2019

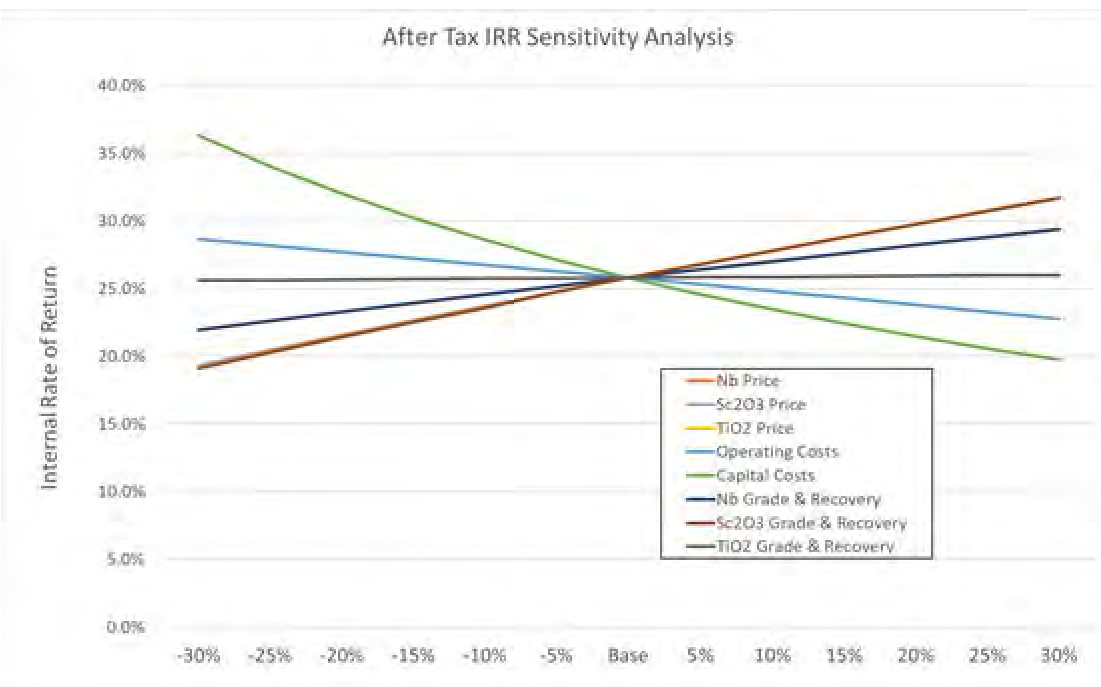
Figure 1-3: After-Tax NPV 8% Sensitivity Graph

Sensitivity graphs in Figure 1-4 and Figure 1-5 demonstrate the Project IRR is sensitive to changes in Sc_2O_3 and Nb prices on both a pre-tax and after-tax basis, but capital costs clearly have a greater effect than operating costs.



Source: NioCorp, 2019

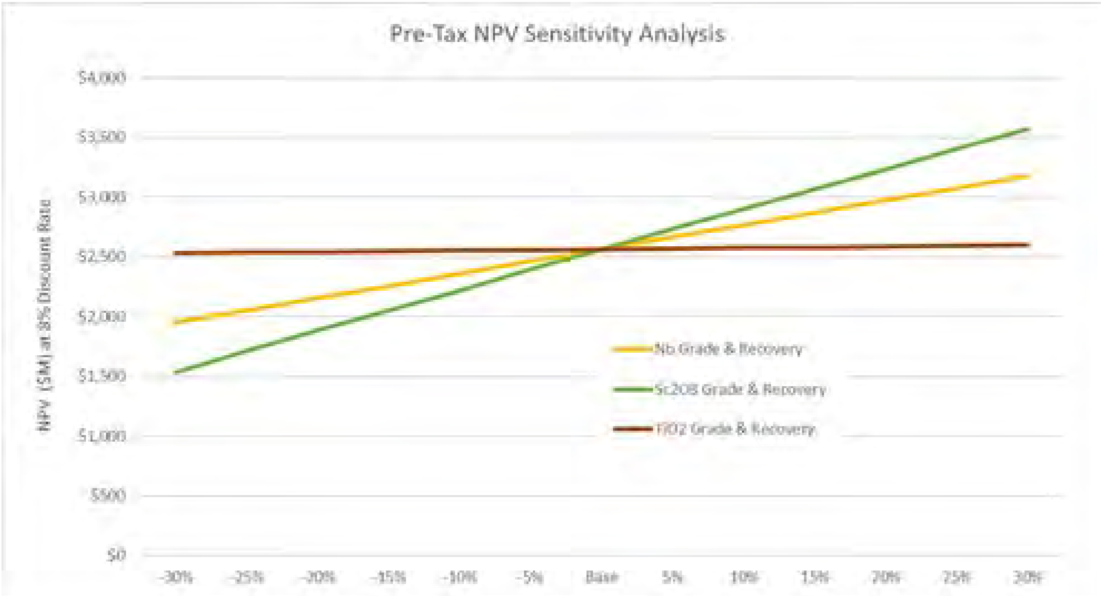
Figure 1-4: Pre-Tax IRR Sensitivity Graph



Source: NioCorp, 2019

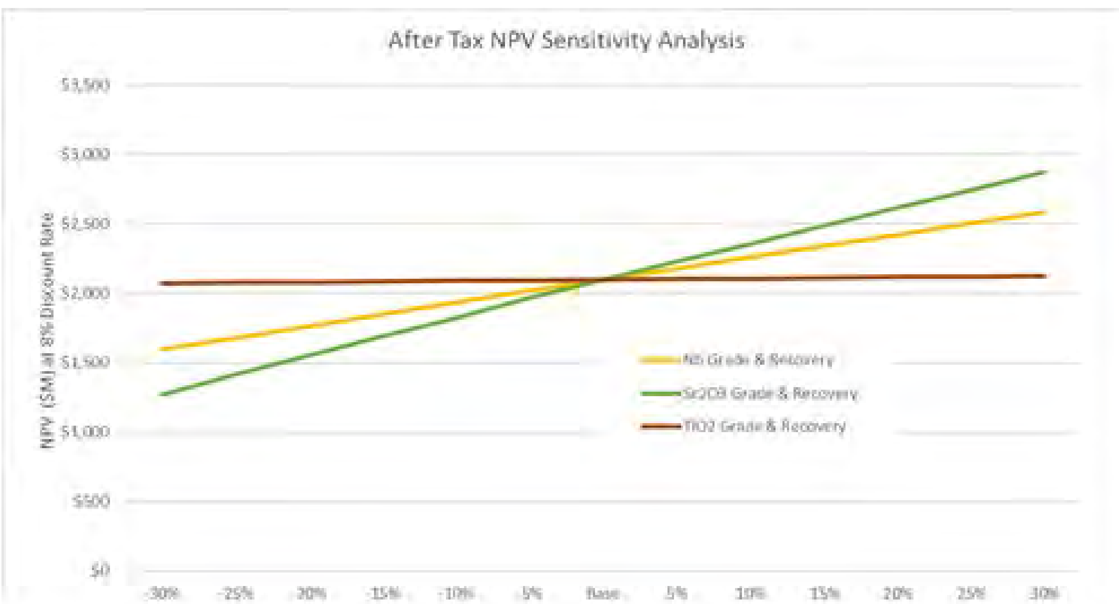
Figure 1-5: After-Tax IRR Sensitivity Graph

Figure 1-6 and Figure 1-7 illustrates the results of pre/post tax basis with respect to head grades and process recoveries of the three products. Not surprisingly, the impact of a head grade reduction is exactly equivalent to a process recovery reduction for each of the products.



Source: NioCorp, 2019

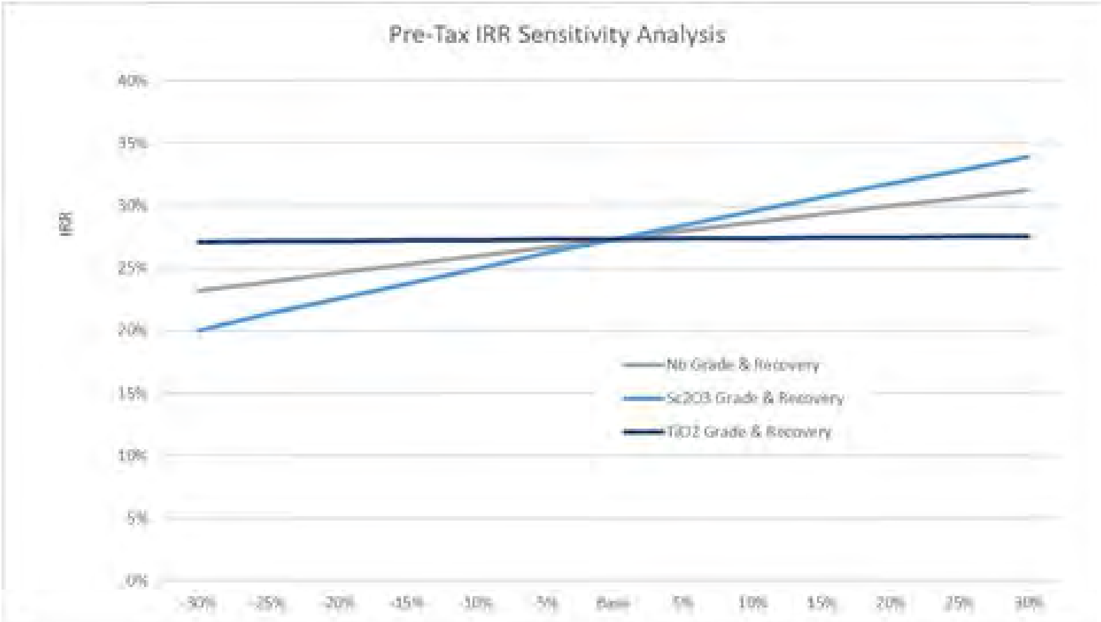
Figure 1-6: Pre-Tax NPV 8% Sensitivity Graph (Grade & Recovery)



Source: NioCorp, 2019

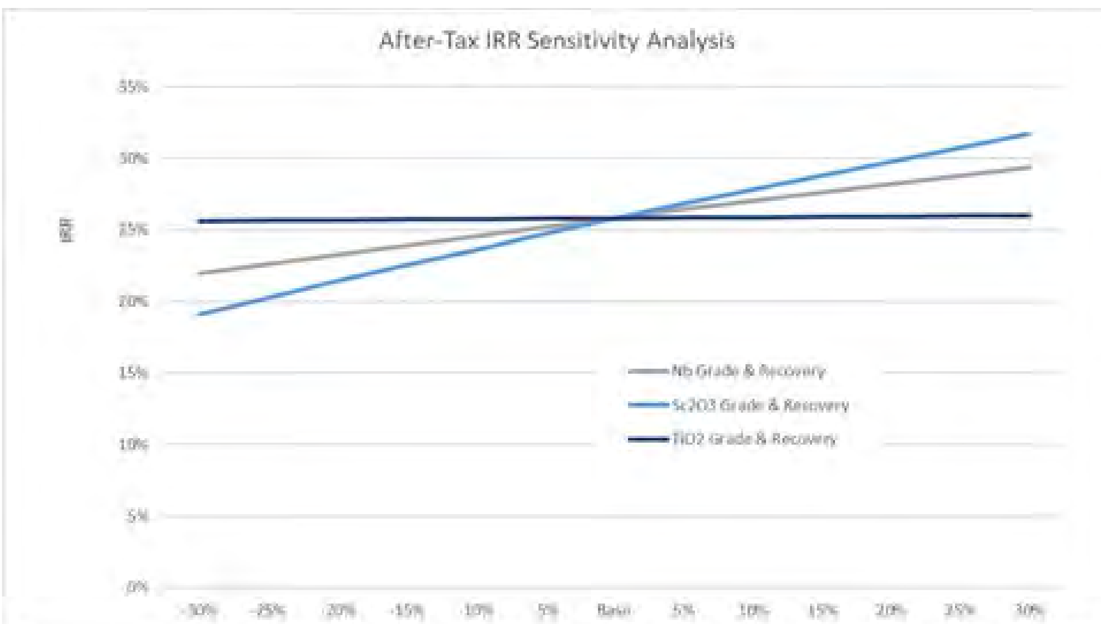
Figure 1-7: After-Tax NPV 8% Sensitivity Graph (Grade & Recovery)

Sensitivity graphs in Figure 1-8 and Figure 1-9 demonstrate the Project IRR is sensitive to changes in Sc_2O_3 and Nb head grade and recovery on both a pre-tax and after-tax basis, but with limited to no impact from TiO_2.



Source: NioCorp, 2019

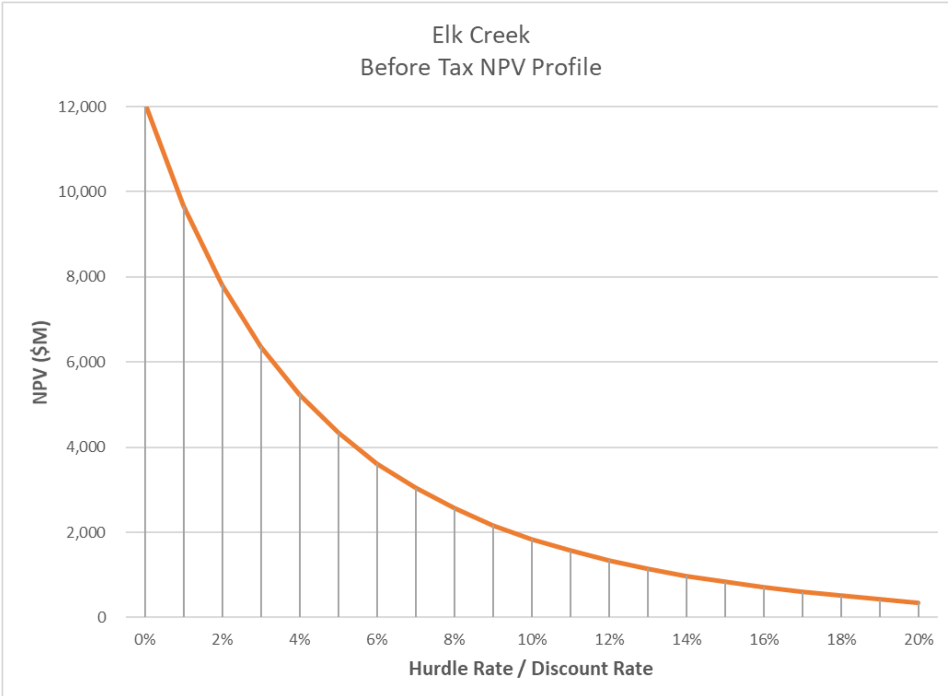
Figure 1-8: Pre-Tax IRR Sensitivity Graph (Grade & Recovery)



Source: NioCorp, 2019

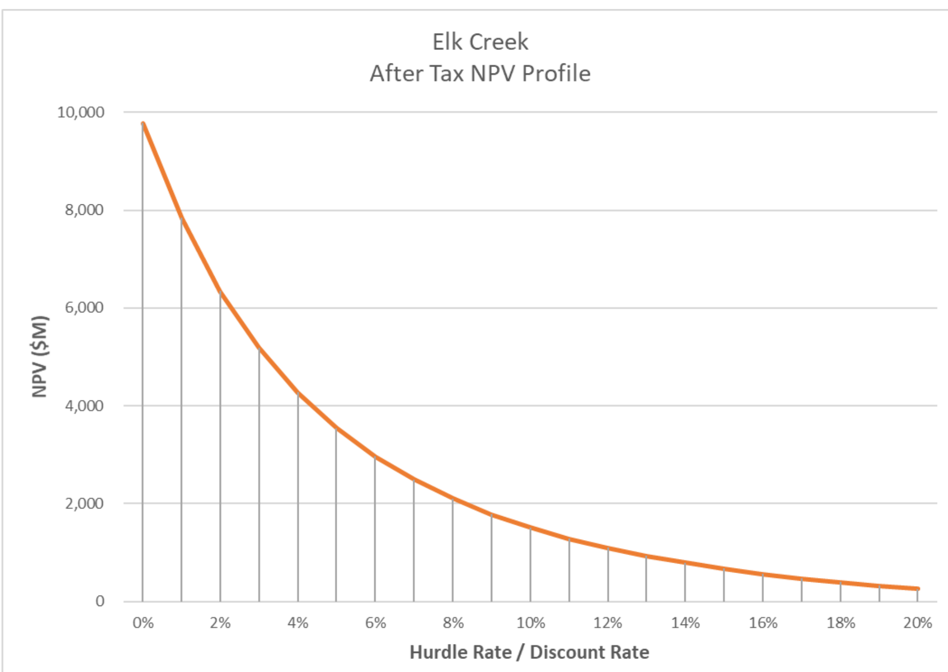
Figure 1-9: After-Tax IRR Sensitivity Graph (Grade & Recovery)

Discount rate sensitivity is always important in a Project valuation, and with respect to this Project, there is a complex process plant flow sheet and market uncertainty to account for. NPV profile charts are presented in Figure 1-10 and Figure 1-11, which shows pre and after-tax NPV results for 100 basis point increments between 0% and 20%. It should be noted that with current assumptions, the Project breaks even at a ~20% hurdle rate on an after-tax basis.



Source: Nordmin, 2019

Figure 1-10: Before-Tax NPV Profile



Source: Nordmin, 2019

Figure 1-11: After-Tax NPV Profile

1.20 Adjacent Properties

There are no significant properties adjacent to the Elk Creek Project.

1.21 Other Relevant Data and Information

1.21.1 Project Implementation Plan

A Project Implementation Plan (PIP) communicates the overall execution strategy for project construction. The project objective is to reach 100% of production capacity within a total budget of US$ 879 million (includes gross pre-production revenue credit). The Project timeline is based on First Metal 42 months after Authorization to Proceed, plus an additional four months of ramp-up to 100% of production capacity for a total project schedule lasting 46 months.

The schedule highlights are as follows:

- The total duration of the project is 46 months from Authorization to Proceed to the end of the ramp-up period.
- A four-month ramp-up period (included in the overall schedule) is allotted to increase the site throughput to 100% of nameplate rating
- The Project timeline is linked to both the mining-related activities and the surface operations in both sequencing and duration. The construction of the main surface plant buildings and supporting infrastructure is not on the critical path.
- The critical path sequence of activities follows:
    - Completion of the air permit.
    - Completion of drilling, sampling and final hydrogeological investigation.
    - Engineering and procurement for shaft sinking and mining components.
    - Construction of temporary power plant for shaft sinking and construction activities.
    - Construction of the temporary freeze plant for shaft sinking activities.
    - Establishing commercial natural gas and electricity service to the Project site.
    - Construction of the backfill plant.
    - Sinking both the production and ventilation shafts.
    - Underground mine infrastructure.
    - Completion of commissioning of the processing plant up to First Metal.
    - Ramp up of processing plant to full production capabilities.

The PIP execution is based on the use of two main EPCM contractors. One contractor with responsibilities for all mining related work and a second contractor responsible for all other site-wide related work. Certain portions of the site-wide work will be performed with EPC sub-contracts awarded to companies that specialize in process and technology related packages, such as the acid plant. The approach is reflected in the capital cost estimate for the Project.

Key early works for the Project include finalizing the contracting approach and contracting the key contractors; optimization metallurgical studies; completing key permitting activities for early works; conduct hydrogeological confirmation testing conduct confirmatory geotechnical drilling for shafts; detailed and procurement of long lead time equipment for shaft, headframe, and mine substation;

detailed engineering, site preparation for early works activities; and finalize contract and commence construction on the third-party natural gas pipeline and electric power supply.

The PIP further identifies and defines project and document control systems and responsibilities, engineering activities, supply chain and procurement activities, construction management activities and responsibilities, and the commissioning and operational readiness strategy.

1.21.2 Opportunities and Risks

An opportunity and risk analysis was completed by Nordmin in conjunction with Optimize, SRK, Tetra Tech, Adrian Brown, Zachry, MCS, SMH and NioCorp at the project level, reviewing both opportunities and risks that were identified both during the previous 2017 feasibility study and the 2019 feasibility study by Nordmin.

1.21.2.1 Opportunities

Opportunities recognized during the analysis included:

Mine Operations

- Optimizing the mine plan based upon market conditions. At present, the production stopes are dictated by their niobium content. There are existing areas within the footwall zone that have high concentrations of scandium, but they have been dismissed as ore due to their lower content of niobium. If the scandium market demand remains intact and the processing plant can increase scandium throughput possibly through a separate circuit, then there would be additional ore within the existing vertical extent of the present mine design.
    - i.e. The current resource model has many resource blocks that have an NSR greater than $500/tonne that are currently not in the mine plan for they do not meet the niobium head grade requirements but do consist of high grade Scandium. As such, if market conditions change, there is an opportunity for the operation to adjust to meet the market needs.
- After completion of additional diamond drilling underground and development within the ore zone, there could be a reason to increase the width of the stopes from 15 m wide to 20 m wide, if geotechnical factors allow. This would decrease ore drive development by 25%, which is the predominant development activity.
- There could be an opportunity to replace the mining contractor after approximately three years of steady-state production. After this period of time, the full requirements to obtain sustainable production levels would be understood, and the owner could replace the contractor with their own workforce. The resulting operating cost should decrease; this would be partially offset with the purchase and sustainable capital for mobile equipment.

Ventilation

- The decrease in ventilation requirements. With present-day equipment manufacturing capabilities, it would be unreasonable to expect a mining contractor to equip themselves with an electric powered mucking and hauling fleet. It is reasonable to transition the diesel-powered haulage fleet to electric power possibly three years into full production. This change over to an electric powered fleet would have a substantial decrease in demand for ventilation underground, although part of the savings related to this would be offset with higher haulage costs to cover the more expensive equipment.

Resource/Reserve Expansion Potential

- The current deposit is open in the hanging wall, foot wall and at depth and along strike. Further drilling during the infill definition, drill programs can be used to determine if the ore body can be expanded.

Cost Estimating

- Use the Hydromet and Mineral Plant buildings for tanks fabricated on site.
- Consider surface-based stormwater drainage.

EPCM Phase

- Mineral Processing and Pyromet buildings: Stick built building vs prefab building.
- Hydromet building: Stick built first story and prefab second.
- Quality: Specialized contractor for installation of the liner in tailing and active dewatering pond.
- Environmental protection: Environmental barrier at ground level during construction.
- Update Paste Backfill Cement Content based on additional testing and evaluate lower cement content.

1.21.2.2 Risks

The risk analysis defined 66 risks and their associated potential mitigation strategies (see Appendix D).

- 24 risks were considered as a pre-response consequence of moderate, major or severe and a likelihood of likely or almost certain.
    - If the action plan is initiated, the post response consequence for these high-risk items reduces to 6 risks.
- 35 risks were considered as a pre-response consequence of minor or moderate and a likelihood of unlikely or possible.
    - If the action plan is initiated, the post response consequence for these moderate risk items reduces to 14 risks.

The major group of risks identified and have an action planned assigned in Appendix D are the following:

Mine Operational Risks

- Shaft Location - Drilling pilot holes for shaft locations to determine local geological, geotechnical and hydrological characteristics and conditions that would be encountered during shaft sinking.
- Resource/Reserve and Mine Design - Significant infill definition drilling is required during construction and operations phases to determine local geological, geotechnical and hydrological characteristics and conditions in conjunction.
- Grade Control - A daily grade control monitoring program is required to maximize the value of ore mined and fed to the surface plant. The grade control process involves the predictive delineation of the tonnes and grade of ore that will be recovered by the mining team. The

program will involve incorporating the results from the infill drilling program in conjunction with an underground chip sampling program to define the boundaries of mineable ore blocks and determine the daily/weekly feed grades to the plant.

- UG Ground Support/Hydrogeology – an ongoing probe hole drill program/grout program needs to be established to support mining activities and not create significant production delays. The need to develop and deploy a high-pressure grout injection system is required to protect the mine from excess inflow to safeguard the project from injury, property damage and loss of life or equipment.

Hydrometallurgical Process Risks

A summary of the recommended test work is presented below to reduce further the risks associated with the Hydromet process design. It is expected that the work would proceed in parallel with detailed engineering for the project and would take an estimated 4 months to complete. Two commercial labs, one in the US and one in Canada, have been identified with the capabilities to complete this test work.

HCl Leach

- Optimize leaching of iron (Fe) to correlate with optimum niobium (Nb) precipitation and Fe/Nb ratios– aiming for the highest recovery of Nb while preventing titanium (Ti) co-precipitation.
- Validate the method used in the aging of the HCl Leach liquor prior to scandium (Sc) Solvent Extraction.

Acid Bake – Water Leach

- Perform vendor testing and optimization of Acid Bake operations and equipment.
- Validate process control and equipment capabilities - optimizing mixing time, temperature, acid to residue ratio.
- Optimize water to residue ratio in Water Leach.

Iron Reduction

- Verify reaction kinetics and the use of briquettes.

Nb Precipitation

- Optimize FeNb ratio.
- Optimize Precipitant (dilution water) acidity to maximize Nb precipitation and Ti selectivity.
- Optimize Final Free Acid (FAT) to maximize selectivity against Ti.

Ti Precipitation

- Further test work required to maximize the removal of uranium and thorium from the Titanium dioxide product to increase its value.

Sc Precipitation

- Optimize the H_3PO_4 addition.
- Optimize the Fe addition.
- Perform locked cycle tests on the Calcium loop.

Sc Refining

- Optimize and further evaluate Zr/Nb removal using mixed organics – stripping acid.
- Optimize conditions to minimize Sc losses.

Sc oxalate Precipitation

- Verify precipitation using solid oxalic acid – optimal amount for optimal recovery.
- Optimize acidity, temperature, and g/l with solid oxalic acid.
- Optimize the washing of Sc oxalate for calcining equipment integrity.

Acid Regeneration

- Optimize the filtration – evaluate equipment and filtration media.

Sulfate Calcining

- Optimize residence time.
- Vendor testing of different equipment and assembly.

General

- Equipment selection, material of construction and vendor guarantee testing.
- Consider a fully integrated pilot testing to be operated onsite during construction of a full-size plant to make final adjustments and equipment selection.
- Further, perform process engineering during the detailed design phase.
- Perform process simulation of the yearly or monthly elemental feed composition using the METSIM model and the compositions from the mine plan.

Scandium Market Risks and Sales Plan

At the time of this report, NioCorp had entered into one offtake agreements covering scandium trioxide production from the Project.

The scandium trioxide offtake agreement is structured similarly to the Niobium contracts. The agreement has a ten-year term and a minimum of 12 t/y. At that rate, approximately 10 - 15% of the projected annual production is contracted. Further, the customer may elect to take more material in any given year above the prescribed minimum quantity.

NioCorp is also working with other potential customers at the time of writing and discussions with these potential customers are proceeding under the provisions of Non-Disclosure Agreements (NDAs).

1.22 Interpretation and Conclusions

Under the assumptions presented in this Technical Report, the Project shows positive economics. The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Technical Report.

1.22.1 Geology & Mineralization

Nordmin constructed mineralization models for the deposit based on all available drilling data. As summarized in Section 14, block modelling was completed in Datamine through explicit modelling of each domain (high grade Nb_2O_5/TiO_2, high grade Sc, and low grade). Structural and mineralization trends were used in the interpretation and for selection of modelling parameters. A block model was built by estimating and combining block models for each domain, and the final block model has been fully validated with no material bias identified.

Nordmin has classified the Mineral Resource into Indicated and Inferred resource categories based on geological and grade continuity as well as drill hole spacing. The Mineral Resource Estimate has been defined based on NSR cut-off grade to reflect processing methodology and assumed revenue streams from Nb_2O_5, TiO_2, and Sc for the deposit. The updated Mineral Resource represents an increase in contained metal.

Additional material exists in the geological model, which has not been classified as Indicated or Inferred resource. The deposit remains open along strike in both directions and at depth, and there exists significant resource expansion potential based both on these factors as well as areas of the block model that require improved definition through diamond drilling.

1.22.2 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

Exploration completed to date has resulted in the delineation of the Elk Creek Deposit and a number of exploration targets.

Nordmin is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. In Nordmin's opinion, the drilling, core handling, logging, and sampling procedures meet or exceed industry standards and are adequate for the purpose of Mineral Resource Estimation.

Nordmin considers the QA/QC protocols in place for the Project to be acceptable and in line with standard industry practice. Based on the data validation and the results of the standard, blank, and duplicate analyses, Nordmin is of the opinion that the assay and bulk density databases are of sufficient quality for Mineral Resource Estimation at the Elk Creek Deposit.

No limitations were placed on Nordmin's data verification process. Nordmin considers the resource database reliable and appropriate to support a Mineral Resource Estimate.

1.22.3 Processing and Metallurgical Testing

Mineral Processing

The Feasibility-level comminution test work was completed in two stages at SGS. The primary stage was conducted on six composite samples and 13 variability samples and included the determination of standard comminution parameters (SGS 2016a). The second stage of comminution test work was conducted on a single composite sample, using a LABWAL HPGR semi-pilot scale test work program

(SGS 2016b). The test work results indicate that the Project ore is categorized as soft to moderately hard in terms of ore hardness, and amenable to standard grinding as well as an HPGR operation.

Hydrometallurgical Plant

Pilot test programs showed that high recovery rates of the niobium, scandium and titanium could be achieved, and that recycling and regeneration of reagents was also possible; thus, minimizing fresh reagent input and waste generation. Recoveries of 85.8% Nb_2O_5 and 93.1% Sc_2O_3 have been demonstrated while achieving 40.3% recovery of TiO_2.

Further understanding of the Process was achieved with respect to the kinetics of each unit operation, which suggested that the design be adjusted from what was initially shown in the 2015 PEA. Among the changes, the following are of interest:

- The temperature of the HCl Leach was adjusted to control leaching of the iron. The Fe to Nb ratio in the Leach residue has an impact on the precipitation of Nb and the co-precipitation of titanium.
- Acid Bake total mixing and reaction time was reduced to 2.5 hours.
- Iron Reduction step was optimized based on actual reduction of Fe3+, which resulted in an improved iron consumption.
- Dilution ratio in the Niobium Precipitation was reduced from 5:1 to 0.6:1, thus reducing reagent consumption and equipment size. This, however, comes at the expense of a slight reduction in Nb recovery and an increase in Ti co-precipitation.

  Secondary scandium recovery from the barren sulphate solution was developed. Selective precipitation of the scandium over impurities was achieved. Scandium precipitated in this section is combined and recovered in the Sc Solvent Extraction.
- A scandium purification step was added that provided a 99.9% scandium product (as Sc_2O_3).
- HCl Acid Regeneration development proved that recovery of chlorides in excess 99% is achievable.
- Further test work and development provided the basis to greatly reduce the need for neutralizing reagents while increasing the recovery of sulphur; therefore, greatly reducing the need for sulphur import.
- Mixed sulphur oxide gas is treated and cleaned prior to being sent to the Acid Plant, therefore, reducing the size and cost of the Acid Plant.

Pyromet

Lab testing has confirmed most of the anticipated findings from the mathematical model that was developed by applying thermodynamic principles:

- The aluminothermic reduction of niobium oxide precipitate and iron oxide has been demonstrated. Ferroniobium particles have been formed, and the chemical proportion of iron and niobium is just what was expected.
- The change to produce a higher TiO_2 content product from the Hydro-Met did not change what was anticipated: Ti content in the FeNb alloy did not increase, and the reduction of Nb_2O_5 did not seem to be affected by this higher amount of titanium oxide.

- Different temperatures have been used in various tests which have provided good reference points on the slag behaviour. The Electric Arc Furnace (EAF) temperature during the operation is expected to correspond to a temperature between 1850°C and 1900°C.

1.22.4 Mining & Mineral Reserve

Geotechnical

A geotechnical field characterization program has been undertaken to assess the expected rock quality. This program included logging core, laboratory strength testing, in situ stress measurements and oriented core logging of jointing. The results of this program have provided adequate quantity and quality data for the feasibility-level design of the underground workings.

A geotechnical assessment of the orebody shape and ground conditions has determined that long-hole open stoping mining is an appropriate mining method. Stopes have been sized to maintain stability once mucked empty. A primary/secondary extraction sequence with tight backfilling allows optimization of ore recovery while maintaining ground stability. Primary stopes will be backfilled with cemented pastefill, while secondary stopes will be backfilled with either light-cement pastefill or uncemented waste rock from development.

The design has been laid out using empirical design methods based on similar case histories. The stability of the 2019 feasibility study mine design has been checked with 3D numerical stress-strain models of the working, which included consideration for mine-scale faulting. The modelling results confirm that stopes and access drifts are predicted to remain stable during active mining, including areas adjacent to pastefilled primary stopes. The revised stope dimensions have been reverified using empirical design methods. The current design has not been reverified using numerical analyses, but this reverification is recommended as the mine design is advanced to the final design.

Ground support requirements have been based on empirical ground support methods and have considered variable levels of required ground support.

The location of underground infrastructure (i.e., shafts, ventilation raises, shops, etc.) have been situated to minimize the adverse impact of encountering geologic structures (i.e., weaker faults and shear zones).

Mine Design

Longhole stoping is seen as the appropriate mining method for the deposit geometry. The large stope sizes minimize the mining cost. The increased dilution due to large stopes sizes is not particularly material to the mine plan as dilution has some grade.

An NSR approach was used focused on targeted amounts of Nb2O5 and takes into account revenue for three elements (Nb_2O_5, TiO_2, and Sc) and generates three separate products (TiO_2, FeNb, and Sc_2O_3). Stope optimization was completed to identify economic mining areas. The 3D mine design was completed on an elevated CoG, which achieved over three times the actual calculated cut-off. Two mining blocks were designed, giving a 36-year LOM, although additional material, classified as indicated, exists below the mine plan presented here.

The underground mine is accessed through a 6.0 m diameter production shaft system. A 6.0 m diameter ventilation/exhaust shaft serves as the mine exhaust, the second means of access, and second mechanical emergency egress. Both shafts are excavated using conventional shaft sinking methods in conjunction with freezing technology to an elevation 200 m below surface.

Tonnage and grades presented in the reserve include dilution and recovery and are benchmarked to other similar operations. Productivities were generated from first principles with inputs from mining contractors, blasting suppliers, and equipment vendors where appropriate. The productivities were also benchmarked to similar operations. Equipment used in this study is standard equipment used worldwide with only standard package/automation features.

A monthly/quarterly/yearly production schedule was generated using Deswik scheduling software. The steady-state mine production schedule of 2,764 t/d ore was based on the processing throughput. The mine design targeted higher annual ferroniobium production during the first five years of ore delivery, which resulted in an averaged annual production rate of 7,351 tonnes per year over this period. The steady-state average annual ferroniobium production was 7,220 tonnes annually.

1.22.5 Recovery Methods

Based upon the ore body samples retained, all bench testing performed, and process analyses completed to date, SMH, MCS and Zachry are confident that the current design will yield the FeNb, TiO_2 and Sc_2O_3 in the quantities expected. While this level of design is feasibility, it is expected that additional design effort during the detail phase will likely yield better results and further improve the efficiency and yields of the processes.

1.22.6 Infrastructure

Onsite and Offsite Infrastructure

Based upon the most current operating and process design information and expectations, the on-site and off-site infrastructure and services will meet each of the required needs of this entire facility.

Infrastructure buildings, office space, locker facilities and showers were sized and designed based upon current workforce projections for the site, as well as a tentative work schedule of 12 hr shifts for shift personnel, and standard 8 hr shifts for non-shift staff. A change in the number of shifts and/or shift durations may have an impact on the requirements of these facilities.

Likewise, both potable water and wastewater distribution systems were sized based upon the above shift criteria. Changes in the number of personnel, and/or changes in numbers of shifts and shift durations may have an impact on the potable and wastewater demands which must be addressed during the detail phase of this design.

Off-site infrastructure in the form of natural gas and electrical power services provided by others are readily available, and well within the current demand requirements of the facility. Potable water sources yielding approximately 4,000 gpm are available from the local municipality (City of Tecumseh), as well as from two private landowners. The public water source would require a service extension from the existing system, while the private sources would require pipelines from the respective owner's wells.

Changes to the process during the detail phase, in particular, changes to the Hydromet process could also have an impact on both potable and process water and an impact on the WWTP (Waste Water Treatment Plant). Additionally, changes to the process could have an impact on the quantities and type of reagents required, which could, in turn, change the size of storage tanks and facilities, as well as the types and materials of construction of these facilities (tanks, totes, bunkers, etc.).

Foundation designs for large loads and structures, as well as roadway designs, were based upon the most current geotechnical report and best engineering practices for the local site conditions.

The most current geotechnical report partially addressed the recommended designs for deep foundations or foundations for large loads; building columns, columns with bridge crane loads, large process equipment or structures. It will be important that the final geotechnical report address these types of loads and provide specific recommendations, but that the final geotechnical site evaluation includes test borings in the final locations of buildings, process equipment and major structures. Recommendations should further include expected settlements, as well as pavement designs with material and compaction recommendations.

1.22.7 Tailings

The tailings storage facilities (TSF) are designed for storage of dry tailings solids in lined facilities permitted under State of Nebraska Industrial Solid Waste regulations. Separate lined “leachate collection ponds” (LCPs) will be used for management of precipitation contacting the tailings solids. Based on the parameters and assumptions outlined in Section 18.12, the Plant Site and Area 7 TSFs have been designed with adequate containment and capacity to manage the planned filtered water leach residue, calcined excess oxide, and slag deposition for a 36-year LOM.

1.22.8 Environmental, Permitting and Social or Community Impact

NioCorp has developed information and conducted a number of environmental studies related to baseline characterization for the Project, the most important of which are the studies related to hydrogeology and geochemistry. The production rate and geochemistry of dewatering water will dictate what is critical to the onsite water balance and any additional management (active or passive) that may be required.

The geochemistry and characterization/classification of the ore and waste materials (including the final process waste streams making up the bulk of the tailings mass and the crystallized RO water treatment salts), directly influences the management of these materials given the presence of naturally occurring radioactive materials (NORMs) (i.e., uranium and thorium) and the potential for limited reaction to contact with water. These materials currently classify as non-hazardous based on regulatory testing. Site-wide management of non-contact and contact stormwater will be essential to Project compliance.

Engagement of local, state and federal regulators has commenced. Initiation of the formal permitting program for the Project is dependent upon the completion of the mine plan and surface facilities being developed as part of this technical document, as well as additional characterization of the waste materials and potential worker exposures under the jurisdiction of the Nebraska Department of Health and Human Services (DHHS) and U.S. Department of Labor — Mine Safety and Health Administration (MSHA), both of whom will have primary oversight of worker safety and monitoring programs with respect to the presence of NORMs in the ore and waste rock.

Without specific hardrock mining regulations, there are limited obligatory requirements for reclamation and closure of mining properties in Nebraska. There are provisions, however, within the applicable regulatory framework that is likely to be applied to the Project during the permit and licensing processes, specifically those associated with the TSF and mineral processing facilities. This will include the provision of financial surety for proper closure and reclamation of the site. The currently estimate costs for closure and reclamation of the Project is US$ 45 million.

Overall, the Project appears to be sufficiently advanced to initiate the submission of formal permit applications which will govern the construction, operation, and closure of the mine. However, given the complexity of the mine design, process operations, accelerated schedule currently envisioned by NioCorp, and the inexperience of the state regulators with this type of mining, one must recognize that risks remain within the permitting process that could slow Project development, even with the overwhelming support that the Project appears to have from the communities and stakeholders.

1.22.9 Market Studies and Contracts

Market studies for niobium, titanium dioxide and scandium trioxide are an important part of the proposed Elk Creek Mine. These products, especially niobium and scandium trioxide (scandium), are thinly traded without an established publicly available price discovery mechanism.

Marketing studies and product price assumptions are based on research and forecasts for the following products:

- Ferroniobium (FeNb): Roskill's Global Industry, Markets and Outlook 2018 (Roskill, 2018)
- Scandium Trioxide (Sc_2O_3): OnG Commodities LLC (OnG, 2019) - specializes in the scandium alloys and scandium markets.
- Titanium Dioxide (TiO_2): USGS Commodity Market Summaries (Bedinger, 2019) and Adroit Market Research (Johnson, 2019).

NioCorp is considering selling ferroniobium, scandium trioxide and titanium dioxide products from the Project through all avenues, which include entering into long-term contracts with buyers.

At the time of this report, NioCorp had entered into three off-take agreements covering ferroniobium and scandium trioxide production from the Project.

No off-take agreements have been executed at the time of the report for the titanium dioxide product from the Project. It is assumed this product and all other material not covered by an off-take agreement will be sold on a spot price, ex-mine gate basis.

1.22.10 Capital and Operating Costs

The estimate meets the classification standard for a Class 3 estimate as defined by AACE international and has an intended accuracy of ± 15%. The estimate is reported in Q1 2019 U.S. constant dollars.

Total LOM capital costs, including initial, sustaining and reclamation costs, are US$ 1,565 million. The initial capital estimate of US$ 1,143 million can be partially offset by a Gross Pre-production Revenue Credit of US$ 265 million (generated by pre-production product sales) to net to a cost of US$ 879 million.

The operating cost estimates were developed to show annual costs for production. Unit costs are expressed as US$ 196.41/tonne processed. LOM operating costs are estimated to be 1,560 million. The operating cost varies by year, by mine location and production. The annual operating cost varies by year but averages approximately US$ 44 million per year over the LOM. The mining operating cost is based on a Q1 2019 cost basis.

1.22.11 Economic Analysis

The 2019 Technical Report is based on an assumption of processing of 36,313 (kt) over a 36-year life of mine (LOM) to produce 168,861 tonnes of Nb in the form of ferroniobium, 3,410 tonnes of Sc_2O_3 and 418,841 tonnes of TiO_2.

On a pre-tax basis, the NPV (8% discount) is US$ 2,564 million, the IRR is 27.3%, and the assumed payback period is within 2.85 years.

On a post-tax basis, the NPV (8% discount) is US$ 2,098 million, the IRR is 25.8%, and the assumed payback period is within 2.86 years.

1.23 Recommendations

No additional work is identified other than that included in the workplan provided for the detailed engineering phase of design, and the Total Installed Cost (TIC) estimate included within the FS capital estimate. The following are some of the key early work recommendations included in the next steps of the Project Execution.

1.23.1 Geology and Resources

No additional work or costs have been identified or recommended beyond the work outlined in the 2019 feasibility study. Two of the key items that Nordmin recommends is:

- Extensive definition drilling as development progresses.
- Multiple pilot holes are drilled for the location of the production shaft and the ventilation shaft development.

1.23.2 Process

Mineral Processing

Further pilot testing should be conducted prior to the detailed engineering stage. The results from further pilot testing will be used to confirm unit energy consumption, optimize the HPGR operating variables, and to confirm the size of HPGR required.

Hydromet

Adequate test work was conducted to support a feasibility-level design for the hydromet plant, and all sections of the process have been successfully tested at the pilot scale required for a feasibility study. However, optimization was not achieved in all areas, and certain areas would benefit from further "post feasibility study" test work, preferably before or during the early portion of the detailed engineering activities begin. Further testing would also be beneficial for vendor guarantee.

Pyrometallurgical Processing

The following items should be addressed during the detailed design stage:

- Perform optimization testing for the processes in the Hydromet plant to enhance the separation concentration ratio of Nb_2O_5/TiO_2.
- Perform larger scale testing to demonstrate the on-going success of the Nb2O5 reduction with a lower Nb_2O_5/TiO_2 ratio of 0.48.
- Perform larger scale reduction test work to clearly demonstrate the efficiency of the EAF and its benefits on the separation of FeNb alloy from the slag.
- Develop the proper composition for the fluxes in order to ensure a long service life of the refractory and not compromise the availability of the pyromet plant.

1.23.3 Geotechnical

To advance to the final mine design, additional characterization data will be required to reduce geotechnical uncertainty. SRK recommends the following characterization activities:

- Drill holes at the final shaft and ventilation raise location to confirm ground conditions for the shaft ground support.
- Drill an additional 4 to 6 geotechnical drill holes in the footwall infrastructure and planned stope mining areas to verify the range of expected ground conditions. The program should include collecting:
    - RMR/Q data;
    - structural orientation data;
    - updating the structural model and geotechnical models; and
    - updating mine design parameters.

Additional geotechnical drill holes are required to characterize ground conditions for the final alignment of the ramps and footwall drives. It is recommended these holes be drilled from underground after the shaft is constructed and the initial access drives are mined.

SRK recommends that the numerical analyses be re-assessed as the project design stage is advanced. The design stope dimensions and mining sequence have been changed since the 2017 feasibility study analyses were conducted, but the results are generally considered applicable to the 2019 feasibility study design because the hydraulic radius and net extraction ratios are quite similar.

1.23.4 Mining & Mineral Reserve

No additional work or costs have been identified or recommended beyond the work outlined in the 2019 feasibility study.

1.23.5 Recovery Methods

Mineral Processing

Based on the metallurgical test work and the process design completed for the feasibility study, Zachry recommends further pilot testing of the HPGR option. The pilot HPGR test work program should be conducted prior to the detailed engineering stage in order to confirm and determine:

- The main operating variables of the HPGR
- Flake strength
- Wear rate:
    - Abrasion Index

The results from the pilot testing will be used to confirm the unit energy consumption, optimize the HPGR operating variables, and confirm the size of the HPGR unit for industrial operation.

The sampling requirement for the pilot testing is estimated to be approximately 1.5 t. The cost of pilot testing will be in the range of US$ 8,500 to US$ 30,000.

Hydrometallurgical Plant

Any additional work required is included in the detailed engineering scope of work and included in the feasibility cost.

Pyromet

Even though the testing has shown good results and is aligned in accordance with the mathematic model developed using thermodynamic calculations, a few minor issues remain to be addressed:

- Optimize the capacity of the Hydromet to increase the proportion of Nb_2O_5 in the precipitate. A target ratio of Nb_2O_5 / TiO_2 of 1 would be suitable.
- Perform large scale testing with an EAF to ensure good separation of slag/metal liquid and ensure the homogeneity of the ferroniobium alloy.
- Develop a flux that will enhance the fluidity of the slag at 1850 °C and 1900°C.
- Select a material for the refractory that will resist the aluminothermic conditions in the Electrical Furnace.

1.23.6 Infrastructure

Due to the short Project schedule of 46 months, the long lead time for the natural gas pipeline to the site (18 months) and power transmission on site (24 months) it is paramount that adequate in-depth planning and scheduling of the site infrastructure take place soon after the Authorization to Proceed is received. Site infrastructure, primarily power transmission and natural gas distribution, have a direct impact on project rental costs.

Temporary power distribution is required throughout the site for the start of major construction activities when permanent power systems would normally be available, namely for shaft freezing, the start of shaft sinking operations, and erection of equipment and structures. Additionally, without a natural gas distribution system, fuel for power generation systems will need to be replenished by a local supplier until the permanent system is available for use. At present, this accounts for nearly one half of the entire project construction schedule.

Appropriate planning with local utilities with the goal of reducing procurement time and total lead time prior to site power and natural gas distribution is paramount to this project.

Additionally, at the forefront of detailed engineering, a review and rationalization of all duplicate use facilities should be performed between the mining and mill operations. Potential synergies may exist, thus reducing the cost for construction and operation.

1.23.7 Environmental, Permitting and Social or Community Impact

With respect to environmental, permitting and social/community issues for the Project, SRK provides the following recommendations to NioCorp:

- Remain engaged and transparent with Bold Nebraska and other stakeholders/non-governmental organizations throughout the permitting process and provide them with an opportunity to participate in any public meetings or town hall discussions. This tends to garner less opposition when it comes time for formal public comments on permit applications.
- Complete more detailed hydrogeological investigations of the orebody to more accurately and precisely define the quantity and long-term quality of dewatering water expectations, and assess the feasibility of RO water treatment brine reinjection.
- Continue characterization work on the mine waste rock, process tailings, and RO water treatment crystallized salt materials in order to define the extent and partitioning of radionuclides more precisely. Assess the potential effects of the exothermic reactions from

the hydration of the calcined tailings materials on the overall TSF facility, worker safety, and surrounding environment, including the potential for rad-containing, fugitive dust generation.

2. INTRODUCTION

2.1 Terms of Reference and Purpose of the Technical Report

This Technical Report was prepared as a feasibility-level Canadian National Instrument 43-101 ("NI 43-101") Technical Report ("Technical Report") for NioCorp Developments Ltd. "(NioCorp" or "the Company") by Nordmin Engineering Ltd. ("Nordmin"), Optimize Group ("Optimize"), SRK Consulting (U.S.), Inc. ("SRK"), Tetra Tech ("Tetra Tech"), Adrian Brown Consulting ("Adrian Brown"), Zachry Engineering Corporation ("Zachry"), Metallurgy Concept Solutions ("MCS") and SMH Process Innovation LLC ("SMH")(collectively referred to as "the Consultants") on the Elk Creek Superalloy Materials Project ("Elk Creek" or "the Project") located in southeast Nebraska.

The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in Nordmin's services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications outlined in this Technical Report.

This Technical Report is intended for use by NioCorp subject to the terms and conditions of its contract with Nordmin and relevant securities legislation. The contract permits NioCorp to file this Technical Report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this Technical Report by any third party is at that party's sole risk. The responsibility for this disclosure remains with NioCorp. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This Technical Report provides Mineral Resource and Mineral Reserve Estimates, and a classification of resources and reserves prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM, 2014).

2.2 Qualifications of Consultants

The Consultants preparing this Technical Report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, underground mining, mining backfill, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, pipeline design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this Technical Report has any beneficial interest in NioCorp. The Consultants are not insiders, associates, or affiliates of NioCorp. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between NioCorp and the Consultants. The Consultants are being paid a fee for their work in accordance with reasonable professional consulting practices.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this Technical Report, and are members in good standing of relevant professional institutions. QP Certificates of Authors are provided in Appendix A. The QP's are responsible for specific sections as follows:

- Adrian Brown, PE (Adrian Brown Consultants, Civil Engineer) is the QP responsible for the hydrogeology design parameters in Section 16.3 and portions of sections 1, 25 and 26 summarized within this Technical Report.

- Chris Dougherty, P.Eng. (Nordmin, Principal, Consulting Specialist and Civil Engineer) is the QP responsible for Paste Backfill Plant and Underground Distribution Section 18.13, Freezing Plant Section 18.14, Taxes, Royalties and Other Interests Section 22.5 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
- Orest Romaniuk, P.Eng. (Zachry Engineering Corporation, Senior Engineer) is the QP responsible for Surface Crushing, Ore Storage and Mineral Processing Plant Sections 13.1, 17.1.1, 17.2.1, 17.3.1 and 17.4.1 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
- Joshua Sames, PE (SRK, Senior Consultant) is the QP responsible for earthworks and tailings management Sections 18.7, 18.8, 18.10, 18.11, 18.12 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
- David Winters, PE, SE, MBA (Tetra Tech Senior Consultant, Civil and Structural Engineer) is the QP responsible for the Infrastructure Sections 17.5.1, 17.6.1, 17.6.2, 17.7 (Intro), 17.7.1, 18.4, 18.5.1 through to 18.5.4, 18.6, 18.9 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
- Eric Larochelle, B.Eng. (SMH Director, Specialty Metals & Hydrometallurgy) is the QP responsible for mineral processing and metallurgical testing Sections 13.2, 17.1.2, 17.1.4, 17.2.2, 17.2.4, 17.3.2, 17.3.4, 17.4.2, 17.4.4, 17.5.2, 17.5.4, 17.6.3, 17.7.2 and 17.7.3, and portions of Sections 1, 25 and 26 summarized within this Technical Report.
- Gregory Menard, P.Eng. (Nordmin, Senior Mechanical Engineer) is the QP responsible for Infrastructure Sections 16.4.5, 16.6.3, 16.8.2 through to 16.8.4, 16.8.7 through to 16.8.9, 16.8.12 through to 16.8.15, 18.1, 18.2, 18.3, 18.12, 24 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
- Glen Kuntz, P. Geo. (Nordmin, Consulting Specialist Geology/Mining) is the QP responsible for property, geology and resource Sections 4 through 12, 14, 16.1, 19, 23 and portions of Sections 1, 21, 22, 25 and 26 summarized within this Technical Report.
- Jean-Francois St-Onge, P.Eng. (Optimize Group, Associate, Consulting Specialist Mine Engineer) is the QP responsible for Mineral Reserve and Mining sections 15, 16.4.1 through to 16.4.4, 16.5, 16.6, 16.7, 168.5, 16.8.6, 16.8.11, 16.8.16, 16.8.17 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
- John Tinucci, Ph.D., PE (SRK Practice Leader/Principal Consultant, Geotechnical Engineer) is the QP responsible for the Geotechnical Section 16.2 and portions of Sections 1, 25 and 26 summarized within this Technical Report.
- Mark Allan Willow, MSc, CEM, SME-RM (SRK Principal Environmental Scientist) is the QP responsible for Environmental, Permitting and Social or Community Impact Section 20 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
- Sylvain Harton, P.Eng. (Metallurgy Concept Solutions, Senior Metallurgist) is the QP responsible for pyrometallurgical plant Sections 13.3, 17.1.3, 17.2.3, 17.3.3, 17.4.3, 17.5.3, 17.6.4 and portions of Sections 1, 25 and 26 summarized within this Technical Report.

2.3 Details of Inspection

A summary of site visit inspections by the Consultants is provided in Table 2-1.

Table 2-1: Site Visit Participants

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
Glen Kuntz	Nordmin	QP Geology Resources/Mining	November 11 -12, 2018	Visited the greenfield site, reviewed drill core, geotechnical, local geology, surrounding region.
Jean Francois St-Onge	Optimize Group	QP Reserves/Mining	November 12, 2018	Visited the greenfield site, reviewed drill core, local ground conditions, local geology, geotechnical, surrounding region.
Eric Larochelle	SMH		October 22, 2014	
Mark Willow	SRK	Environmental and Permitting	June 1 - 3, 2015	Site visit and meeting with Nebraska permitting officials.
Guy Cinq-Mars	Tetra Tech	Mech Design	Nov 21,22, 2016	Visited the greenfield site and surrounding region.
Sylvain Turcotte	Tetra Tech	Mech Design	Nov 21, 22, 2016	Visited the greenfield site and surrounding region.
Pierre Mathieu	Tetra Tech	Project Management	Nov 21, 22, 2016	Visited the greenfield site and surrounding region.

2.4 Effective Dates

This Technical Report has the following effective dates:

- Date of database close-out for Mineral Resource Estimation: December 20, 2018
- Date of Mineral Resource Estimate: February 19, 2019
- Date of Mineral Reserve Estimate: February 19, 2019
- Date of supply of latest information on mineral tenure: April 16, 2019
- Date of supply of latest information on drill programs: December 20, 2018
- Date of financial analysis: April 16, 2019

The overall effective date of the Technical Report is the date of the financial analyses and is April 16, 2019.

2.5 Sources of Information

The sources of information utilized in the preparation of the Technical Report were provided by Scott Honan, M.Sc., SME-RM, Vice President, Business Development of NioCorp and President, Elk Creek Resources Corporation, under the direction of Mark Smith, PE., CEO and Executive Chairman of NioCorp.

This Technical Report has been prepared in accordance with NI 43-101, Form 43-101F1 and Companion Policy 43-101CP.

Historical work conducted in the region has been compiled by NioCorp.

2.6 Acknowledgements

Nordmin and NioCorp would like to thank and acknowledge the following people who have contributed to the preparation of this Technical Report and the underlying studies under the supervision of the Qualified Persons, including Mr. Mark Smith, President, CEO and Executive Chairman, Mr. Scott Honan Vice President, Business Development, Kelton Smith Director of Engineering and Neal Shah, CFO of NioCorp.

2.7 Units of Measure

Unless otherwise noted, the following measurement units, formats and systems are used throughout this Technical Report.

- Measurement Units: all references to measurement units use the System International (SI, or metric) for measurement. The primary linear distance unit, unless otherwise noted, are metres (m).
- General Orientation: all references to orientation and coordinates in this report are presented as UTM.
- Currencies outlined in the Technical Report are stated in U.S. dollars (US$) unless otherwise noted.

The symbols and abbreviations used in this Technical Report are outlined in Section 28.4.

3. RELIANCE ON OTHER EXPERTS

The Consultants' opinion contained herein is based on information provided to the Consultants by NioCorp throughout the course of the investigations. Nordmin has relied upon the work of other consultants in the Project areas in support of this Technical Report.

In each case, the Qualified Person hereby disclaims responsibility for such information to the extent of his/her reliance on such reports, opinions, or statements. This reliance applies to the information provided by NioCorp for Section 4.2 to Section 4.6 and Section 22.

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

These items have not been independently reviewed by Nordmin and Nordmin did not seek an independent legal opinion of these items.

Nordmin

Nordmin and Mr. Glen Kuntz, P.Geo. and QP relied on the following experts to complete his sections of this Technical Report. Mr. Kuntz has reviewed the data supplied by other experts and in his professional judgement, has taken appropriate steps to ensure that the work, information, and advice from the noted experts below are sound for the purpose of this Technical Report.

Sample Preparation, Analysis and Security and Data Verification

- Francine Long, P. Geo., has over 11 years of experience in the global mining industry and mineral sector projects. She has been a part of the geological technical team on over 75 projects focused on advancing greenfield and brownfield mineral exploration projects and works in both underground and open pit production environments.

Mineral Resource Estimate

- Christian Ballard, P. Geo., has over 13 years of experience as a Mine Geologist and several years of geological exploration experience. He is a specialist in mine geology, resource modelling and evaluation. Christian has vast experience in creating and maintaining geological models. He has practical experience in the progression of underground development and calculation of long and short-range ore resources, geostatistics and block modelling and interpretation for the purpose of ongoing official resource refinement. Experience includes wireframe modelling, block modelling estimation methods, and variography.

Nordmin and Mr. Gregory Menard, P.Eng. and QP relied on the following experts to complete his sections of this Technical Report. Mr. Menard has reviewed the data supplied by other experts and in his professional judgement, has taken appropriate steps to ensure that the work, information, and advice from the noted experts below are sound for the purpose of this Technical Report.

Mine and Project Infrastructure & Services

- Power Systems: Tim Ricard, P.Eng. has over 10 years of experience in power systems engineering and support services, primarily in industrial, mining and manufacturing environments. He specializes in power systems analysis and protection and control. He has extensive experience in protective device coordination, arc flash analysis and device protection settings.
- Electrical and Control Systems: Erick Bruce, P. Eng. is a Senior Electrical and Control Engineer. He has a broad range of experience in detailed engineering, purchasing, construction supervision and commissioning of electrical and control systems. He has specialized expertise in PLC and HMI based burner management systems.
- Mine Infrastructure: Stan Emms has over 40 years of experience promoting safety, efficiency and cost control to a wide variety of mining activities both in underground and open pit operations, as well as managing mine related construction projects. He has vast experience with establishing a correct and practical approach towards mine design that benefits the feasibility, construction, operation and closure phases related to an ore deposit.
- Ventilation Systems: Agnes Krawczyk, P. Eng. has over 14 years of experience in a variety of underground and open pit mining environments. She has led activities in all aspects of mine planning, scheduling, surveying, rock mechanics, ventilation, capital and operating budgets.

Scandium Marketing and Pricing

Nordmin relied on Dr. Andrew Matheson of OnG Commodities LLC (OnG) for input on scandium marketing and pricing. Dr. Matheson has extensive experience and expertise in the development and implementation of market assessments across a range of materials and industries over the course of 20+ years. He provides independent, expert judgment of the outlook for scandium markets and products.

The report referenced is titled “Scandium: A Market Assessment” by OnG Commodities LLC dated April 2017. The pricing sheet for the OnG Commodities report was updated for NioCorp in 2019.

Dr. Matheson's expertise includes:

- Education: a Ph.D. from Cambridge University in theoretical physics, and an MBA from Harvard Business School.
- Consulting: management consulting for The Boston Consulting Group, and ten years as an independent consultant providing strategic, operating and market development advice to companies in the US and Asia, in industries including oil and gas, mining and metals processing, electronic materials, automotive materials. He has also served over the past five years as a contractor with Roskill Inc, a widely recognized and respected consulting firm in the field of minor metals.
- Electronics: Dr. Matheson served as general manager of a division of Cabot Corporation manufacturing consumable materials for the semiconductor industry, and also as COO of a UK-based company developing and licensing audio technology to the consumer electronics industry.
- Mining: at Cabot Corporation. Dr. Matheson served for several years in roles overseeing the company's global mineral development efforts in tantalum, as well as Cabot's procurement

efforts in tantalum. Thus he has extensive experience working with smaller-scale and junior miners both as a customer and in a development role. As far back as 1998, he was investigating scandium recovery from tailings in the United States.

- Specialty Metals: Dr. Matheson is the founder and CEO of a company developing technology to produce metal powders that can provide benefits in aerospace and automotive markets. While the materials Dr. Matheson's company is developing are not direct substitutes for competitors to scandium alloys, they are directed to the same major markets (aerospace and automotive). Commercial qualification and adoption paths are common to new materials in these industries, and Dr. Matheson's experience is directly applicable to scandium.

Optimize

Optimize and Mr. Jean-Francois St. Onge, P. Eng. and QP relied on the following experts to complete his sections of this Technical Report. Mr. St. Onge has reviewed the data supplied by other experts and in his professional judgement, has taken appropriate steps to ensure that the work, information, and advice from the noted experts below are sound for the purpose of this Technical Report.

Mineral Reserve Estimation

- Agnes Krawczyk, P. Eng. has over 14 years of experience in a variety of underground and open pit mining environments. She has led activities in all aspects of mine planning, scheduling, surveying, rock mechanics, ventilation, capital and operating budgets.
- Brett Stewart has been a design technician working in Mining Design for 16 years. He has a solid understanding of mining methods and is an expert in several software suites including 3D Mine Planning and Design, Datamine Block Model Import and Evaluation, Mine 2-4D EPS, and AutoCAD as well as the Microsoft Office Programs. Brett brings practical design experience allowing for the establishment of a workable mine design for the lifecycle of the ore body from feasibility through, operation and closure.

Mining Methods

- Stan Emms has over 40 years of experience promoting safety, efficiency and cost control to a wide variety of mining activities both in underground and open pit operations, as well as managing mine related construction projects. He has vast experience with establishing a correct and practical approach towards mine design that benefits the feasibility, construction, operation and closure phases related to an ore deposit.
- Agnes Krawczyk, P. Eng. has over 14 years of experience in a variety of underground and open pit mining environments. She has led activities in all aspects of mine planning, scheduling, surveying, rock mechanics, ventilation, capital and operating budgets.
- Brett Stewart has been a design technician working in Mining Design for 16 years. He has a solid understanding of mining methods and is an expert in several software suites including 3D Mine Planning and Design, Datamine Block Model Import and Evaluation, Mine 2-4D EPS, and AutoCAD as well as the Microsoft Office Programs. Brett brings practical design experience allowing for the establishment of a workable mine design for the lifecycle of the ore body from feasibility through, operation and closure.

SRK

SRK and Mr. Joshua Sames, PE. and QP relied on the following experts to complete his sections of this Technical Report. Mr. Sames has reviewed the data supplied by other experts and in his professional judgement, has taken appropriate steps to ensure that the work, information, and advice from the noted experts below are sound for the purpose of this Technical Report.

Project Infrastructure

- Clara Balasko, PE., is a registered professional engineer with 15 years of experience. She specializes in tailings storage facility design including slurry, paste and dry stack tailings disposal and has worked on numerous projects in the Americas, Australia and Asia. Clara brought experience in tailings storage facility design, development of design criteria, site selection, development of tailings storage facility stage capacity curves and water balance models, design of embankments, environmental containment, dry tailings stacking plan, and closure design for tailings storage facilities. Clara was a significant contributor to site characterization and tailings management system design but has since left SRK.
- Dave Bentel, Pr.Eng, has more than 41 years of experience in the provision of engineering and environmental permitting services for mining facilities, including process fluid and stormwater management, tailings disposal, tailings recovery and re-treatment, heap leach, and open pit and waste rock disposal facilities. Dave has vast experience with establishing practical solutions towards mine design and closure and was involved throughout the design process.

Salt Management Cells

- Breese Burnley, PE., has more than 26 years of experience in engineering design, permitting and closure of facilities for mine water management, tailings disposal, heap leaching, and waste rock disposal. Breese brings experience with establishing a practical and innovative design for mine waste storage facilities thorough feasibility, operation and closure. He has also served as a practice leader with SRK Consulting over the last 6 years and is a recognized professional in his field.

SRK and Mr. Mark Willow, MSc, CEM, SME-RM and QP relied on the following experts to complete his sections of this Technical Report. Mr. Willow has reviewed the data supplied by other experts and in his professional judgement, has taken appropriate steps to ensure that the work, information, and advice from the noted experts below are sound for the purpose of this Technical Report.

Environmental

- Filiz Toprak, MSc, is an SRK mining consultant with over 15 years of experience. She uses her training and background in mining engineering in projects focused on mine reclamation and closure cost estimation. She currently focuses on different types of closure cost estimates to address requirements based on financial assurance, financial reporting, and project planning. Ms. Toprak prepared the closure cost estimate for the Elk Creek Project.

Metallurgy Concept Solutions

Metallurgy Concept Solutions and Mr. Sylvain Harton, Qualified Person, relied on the following experts to complete his Sections of this Technical Report. Mr. Harton has reviewed the data supplied by other experts and in his professional judgement, has taken appropriate steps to ensure that the work, information, and advice from the noted experts below are sound for the purpose of this Technical Report.

- Kingston Process Metallurgy (KPM) – the laboratory that performed the niobium oxide aluminothermic testing.

SMH Process Innovation

SMH Process Innovation and Mr. Eric Larochelle, Qualified Person, relied on the following experts to complete his Sections of this Technical Report. Mr. Larochelle has reviewed the data supplied by other experts and in his professional judgement, has taken appropriate steps to ensure that the work, information, and advice from the noted experts below are sound for the purpose of this Technical Report.

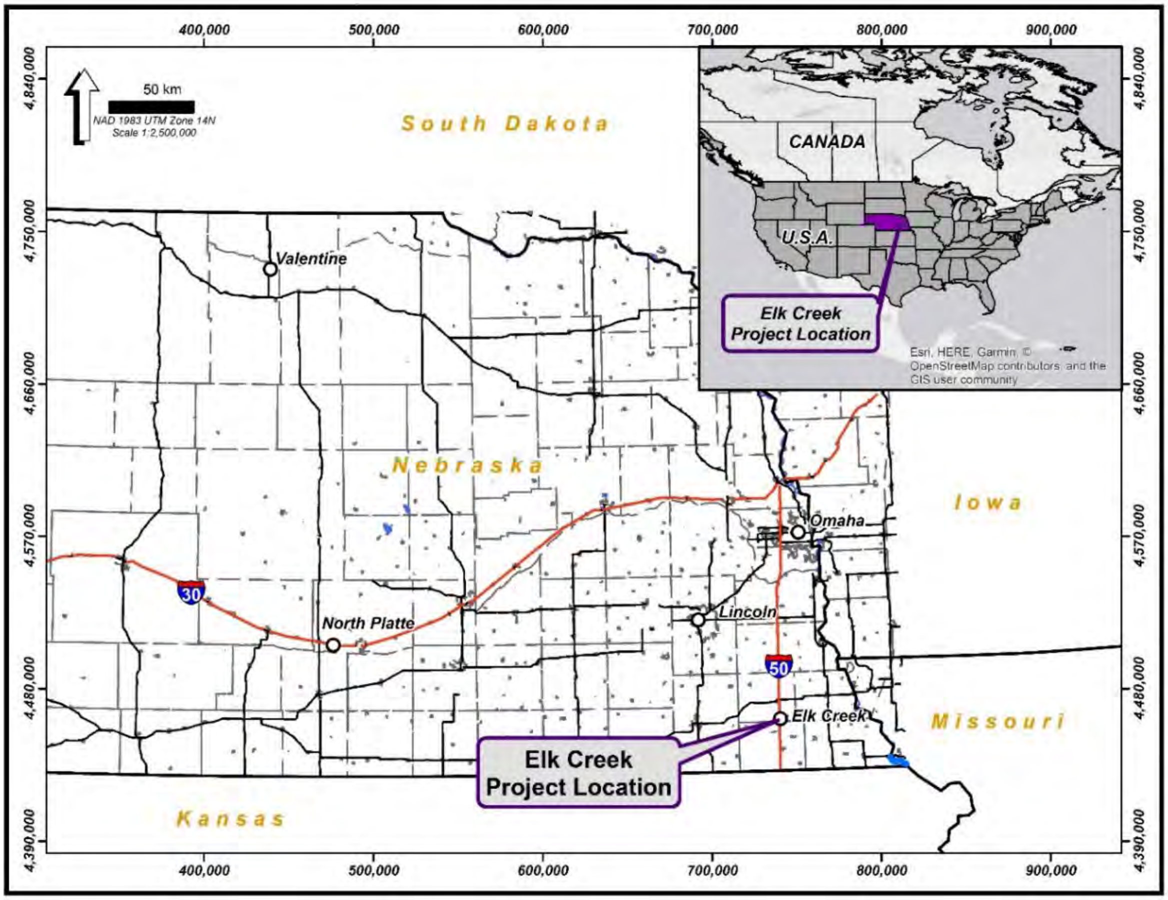
- Hydrochloric Acid Regeneration
    - Mr. K. Michael (Mike) Sessions, PE, Chief Process Engineer, a chemical engineer (M.S., Tennessee Technological University, 1985) with 31 years of experience in process design, process simulation, process scale-up, plant operations, troubleshooting and management, of a variety of chemical processes including pharmaceuticals, foods, commodity and specialty chemicals, as well as in the specification and commissioning of a wide variety of process control instrumentation.
- Sulphuric Acid Plant
    - Mr. Douglas K. Louie, PE, Owner of DKL Engineering, an engineer with over 30 years of experience in process design, process simulation, process scale-up, plant evaluation, and troubleshooting in the Sulphuric Acid industry.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Project is located in southeast Nebraska, USA. The Property is situated, as shown in Figure 4-1 and is located as follows:

- Within United States Geological Survey (USGS) Tecumseh Quadrangle Nebraska SE (7.5-minute series) mapsheet in Sections 1-6, 9-11. Township 3N. Range 11 and Sections 19-23, 25-36. Township 4N, Range 11.
- At approximately 40°16' north and 96°11' west in the State of Nebraska, in the central USA.
- On the border of Johnson and Pawnee counties.
- Approximately 75 km southeast of Lincoln, Nebraska, the state capital of Nebraska.
- Approximately 110 km south of Omaha, Nebraska.
- Approximately 183 km northwest of Kansas City, Kansas and Missouri.
- Approximately 5 km southwest of the town of Elk Creek, Nebraska. the closest municipality to the Deposit.
- Approximately 53 km west of the state border with Missouri.
- Approximately 55 km southwest of the state border with Iowa.
- Approximately 29 km north of the state border with Kansas.
- Approximately 53 km west of the Missouri River, which forms the state border with Missouri and Iowa.
- Approximately 5 km southeast of the Nemaha River, a tributary of the Missouri River.



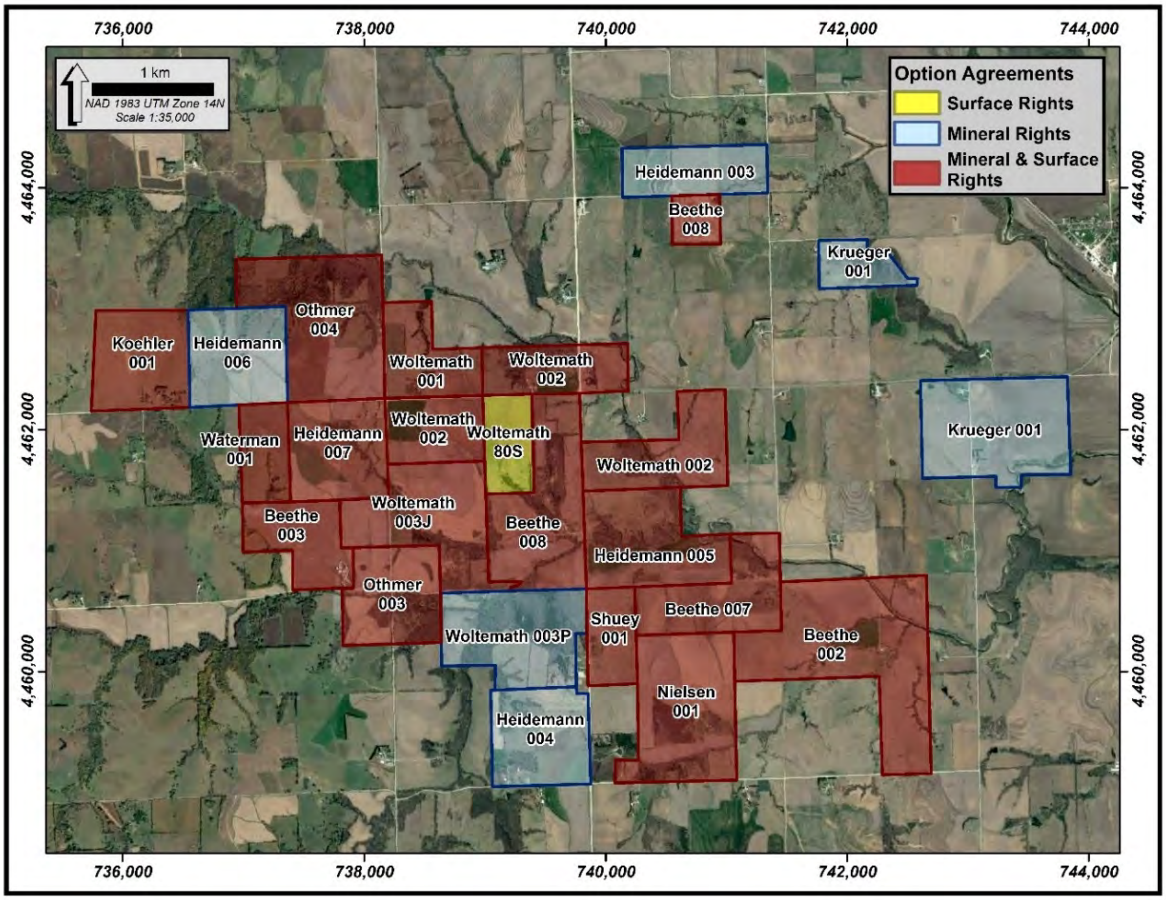
Source: Nordmin, 2019

Figure 4-1: Project Location Map

4.2 Property Description and Land Tenure

The Project is a niobium, scandium and titanium-bearing carbonatite deposit located in Johnson County, southeast Nebraska.

The Property consists of 21 option agreements covering approximately 1,635 hectares (ha). Option agreements are between NioCorp's subsidiary Elk Creek Resources Corp. (ECRC) and the individual landowners (see Figure 4-2). ECRC is a Nebraska based wholly owned subsidiary of NioCorp. NioCorp retains 100% of the mineral rights to the Project and is the operator. The agreements are in the form of five-year pre-paid Exploration Lease Agreement (ELA), with an Option to Purchase (OTP) the mineral rights and/or the surface rights at any time during the term of the agreement. The individual landowners have title to the surface and subsurface rights, and the agreements are primarily concerned with only the mineral and surface interest of each property. The agreements convey to the Company adequate surface rights to access the land and to complete mineral exploration work. The options agreements that the Company currently holds include all the Indicated and Inferred resources described in this Technical Report. Active lease agreements are listed in Table 4-1.



Source: Nordmin, 2019 *Infilled blue polygons are indicating option agreements for minerals only. For the 32.37 hectare parcel north of Beethe008, NioCorp has an option to purchase the surface rights, and negotiations to secure the mineral rights are underway.

Figure 4-2: Land Tenure Map*

Table 4-1: Active Lease Agreements Covering the Project

Agreement Identifier	Hectares	Acres	Agreement Expiry
Beethe008	107.82	266.43	30-Apr-20
Beethe002	146.56	362.16	19-Feb-21
Beethe003	48.69	120.32	24-Jun-20
Beethe007	66.27	163.75	20-Jan-21
Heidemann003	48.56	120.00	17-Mar-20
Heidemann004	62.96	155.58	15-Mar-20
Heidemann005	79.55	196.57	16-Mar-20
Heidemann006	64.75	160.00	26-Mar-20
Heidemann007	64.75	160.00	25-Mar-20
Koehler001	64.75	160.00	12-Jun-20
Krueger001	123.41	304.95	18-Dec-19
Nielsen001	112.81	278.75	25-Jun-20
Othmer003	61.48	151.93	22-Jan-21
Othmer004	113.31	280.00	22-Jan-21
Watermann001	32.37	80.00	6-Sep-21
Woltemath80S	32.37	80.00	4-Dec-19
Woltemath001	48.47	119.77	21-Jan-20
Woltemath002	152.49	376.81	4-Dec-19
Woltemath003J	89.03	220.00	25-Mar-20
Woltemath003P	82.96	205.00	25-Mar-20
Shuey001	32.37	80.00	28-May-20

Source: NioCorp, 2017

The current (2019) Mineral Resource and the Mineral Reserve is wholly contained within parcels Woltemath003J and Beethe008, and agreements covering both properties have been secured. Negotiations for additional lands to support various configurations of the surface operations have been completed.

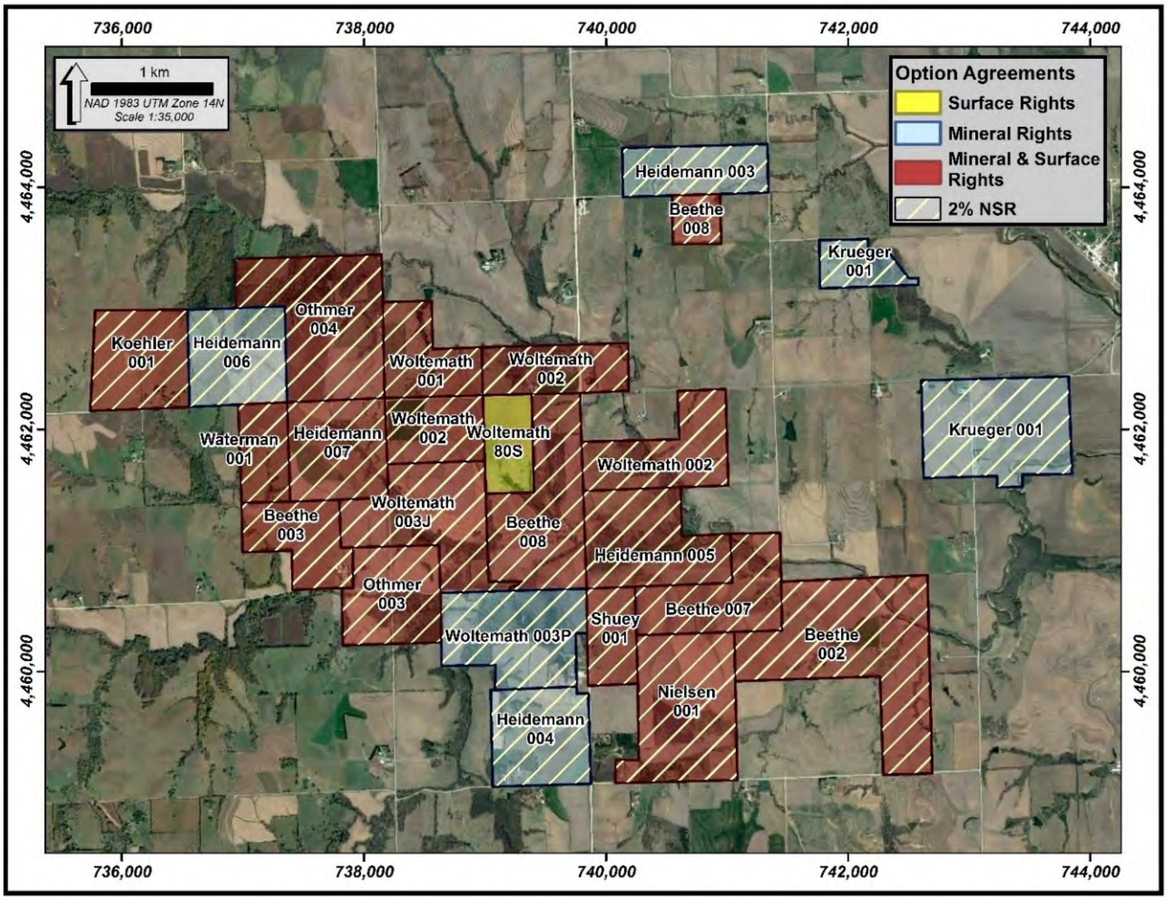
4.2.1 Nature and Extent of Issuer's Interest

As part of the exploration option agreements where required, the Company has also secured surface rights, which allow for access to the land for drilling activities and associated mineral exploration and project development work.

The agreements that involve mineral rights include a 2% Net Smelter Return (NSR) royalty attached with the OTP. The agreements grant the operator an exclusive right to explore and evaluate the property for a period of 60 months, with an OTP the mineral rights, the surface rights or a combination of the mineral and surface rights at any time during the term.

4.3 Royalties, Agreements and Encumbrances

The leases covering the Project are 100% owned by NioCorp and, apart from a 2% NSR royalty attached with the OTPs that include the mineral rights, have no other outstanding royalties, agreements or encumbrances (see Figure 4-3).



Source: Nordmin, 2019

Figure 4-3: Net Smelter Return (NSR) Claim Map

4.3.1 Required Permits and Status

The exploration work conducted to date on the Project has been completed under Exploration Permit NE0211001 issued by the State of Nebraska, Department of Environmental Quality. The permit provides the Company with the right to have ten open boreholes active at the Project at any given time. In addition to the exploration permit, the Company acquired an exemption letter from the Department of Health and Human Services for the use of a handheld held Niton X-Ray Florescence Analyzer (Niton), used in 2014 on drill core for preliminary analysis onsite.

The proposed Project will be held to permitting requirements that are determined to be necessary by Johnson, Pawnee, Richardson and Nemaha Counties, the State of Nebraska, and the U.S. Army Corps of Engineers (USACE).

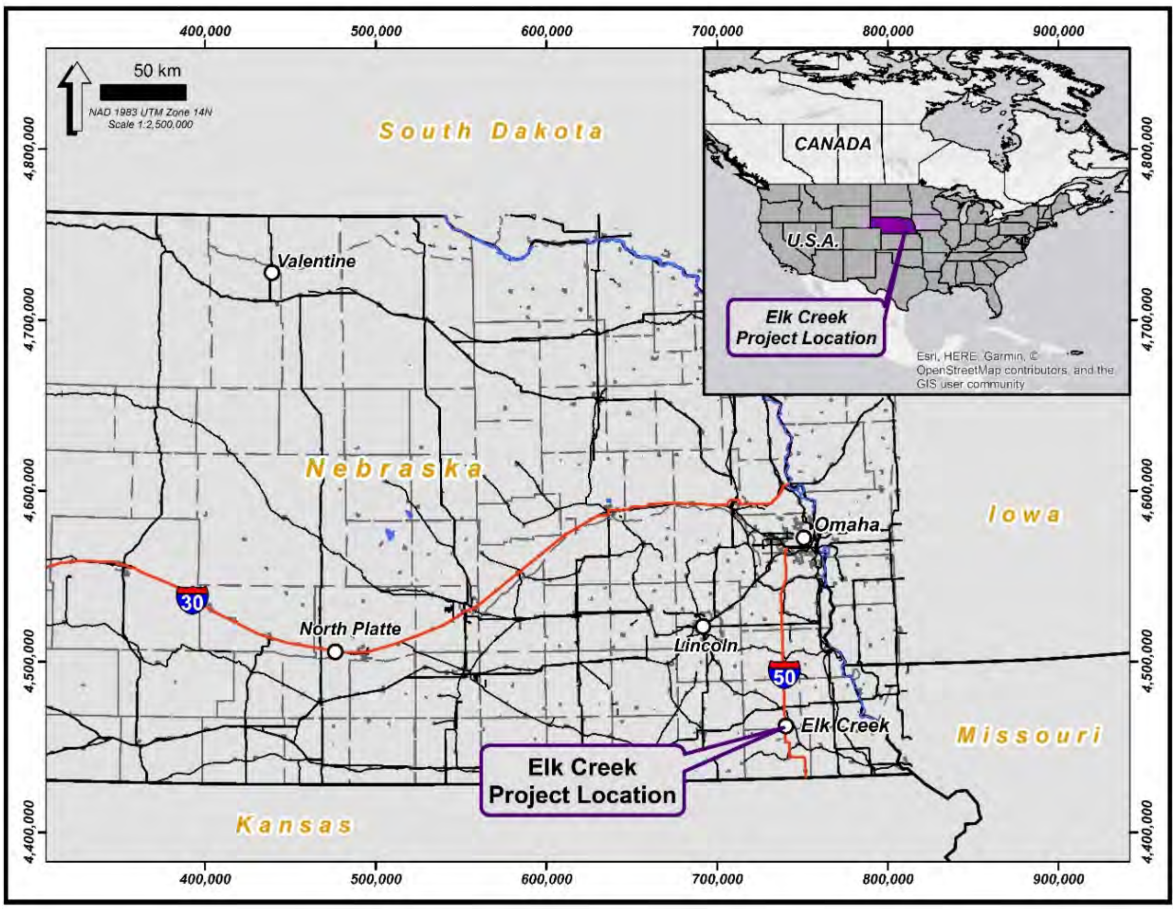
4.4 Other Significant Factors and Risks

There are no known other significant factors or risks which could have a material impact on the ability to affect access, titles or the right to perform exploration work on the property.

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

5.1 Accessibility and Transportation to the Property

The Property is easily accessible year-round as it is situated approximately 75 km southeast of Lincoln (State Capital), Nebraska and approximately 110 km south of Omaha, Nebraska. Access to the site can be completed via road or from one of the regional airports. There are several regular flights to both Lincoln and Omaha; however, the Project is most easily accessible from Lincoln (see Figure 5-1).



Source: Nordmin, 2019

Figure 5-1: Project Location Showing Main Access Routes

From Lincoln Municipal Airport, the Property is accessed via paved roads on the main network and a secondary network of gravel roads by following:

- Interstate Highway 80 south for approximately 3.5 km to the Beatrice exit;
- then join Highway 77 south for approximately 41 km;
- then join Highway 41 south for approximately 47 km; and,
- then join Highway 50 south for approximately 16 km to the approximate center of the Elk Creek Deposit.

The drive from the Lincoln Municipal Airport to the property is typically 1 hour and 15 minutes, and from Omaha's Eppley Airport the drive is approximately 1 hour and 45 minutes.

Technical and trades personnel can be sourced from local colleges and universities. An underground experienced mining related workforce can be found in neighbouring states such as Salt Lake, Utah, South Dakota and Denver Colorado (eight hours drive west of the Project).

5.2 Climate and Length of Operating Season

Southeast Nebraska is situated in a Humid Continental Climate (Dfa) on the Köppen climate classification system. In eastern Nebraska, this climate is generally characterized by hot, humid summers and cold winters. Average winter temperatures vary between -10.4°C to 1.6°C. Average summer temperatures vary between 18°C to 32°C. Exploration and mining related activities may be conducted all year round.

Average monthly precipitation (rain and snowfall) varies between 22 and 127 mm. Average yearly precipitation is between 800 and 850 mm with an average yearly snowfall of approximately 72 cm (see Table 5-1). Nebraska is located within an area known for tornados which run through the central U.S. where thunderstorms are common in the spring and summer months. Tornadoes primarily occur during the spring and summer and may occur into the autumn months.

Table 5-1: Summary of the Project Precipitation Data (4) (5)

Station	Mean Monthly Precipitation	Mean Monthly Pan Evaporation	Mean Monthly Lake Evaporation (5)	Annual Evapotranspiration
	Tecumseh Station (1) (mm)	Sabetha Lake Station (2) (mm)	Sabetha Lake Station (2) (mm)	Rainwater Basin Station (3)
January	21		-	30
February	28	-	-	32
March	49	-	-	66
April	72	131	98	84
May	111	167	126	98
June	117	186	139	98
July	99	210	158	102
August	97	190	142	87
September	89	138	103	86
October	58	103	77	81
November	39	57	43	58
December	26	-	-	29
Annual	**805**	**1,182**	**887**	**851**
7 Year Wet-Cycle Total	**6,662**			
7Year Dry-Cycle Total	**4,318**			

(1) Tecumseh station data (WRCC, DRI) is considered the most representative based on elevation and proximity to the Project site.
(2) Data from Southwest Climate and Environmental Information Collaborative (WRCC, DRI); Sabetha Lake station data is considered the most representative based on elevation and proximity to the Project site.
(3) RAWS Network (DRI), ASCE Standardized Reference ET Calculations.
(4) 5-year average from 2009 through 2013.
(5) Based on Lake Evaporation as 75% of Pan Evaporation.

5.3 Sufficiency of Surface Rights

The Company has negotiated surface rights as needed as part of the ELAs. There is enough suitable land area available within the mineral claims for mine waste disposal, for future tailings disposal, a processing plant, and related mine infrastructure.

5.4 Physiography

The local topography of eastern Nebraska is relatively low-relief with shallow rolling hills intersected by shallow river valleys. Elevation varies from about 325 to 390 metres above sea level (masl). Bedrock outcrop exposure is nonexistent in the Project area.

Much of the Project area is used for cultivation of corn and soybeans, along with uses as grazing land. Native vegetation typical of eastern Nebraska is upland tall-grass, prairie and upland deciduous forests.

6. HISTORY

The following section provides a summary of the history of the Project, and Nordmin has relied upon information provided in the 2017 NI 43-101 Technical Report produced by SRK Consulting, entitled "Revised NI-43-101 Technical Report Feasibility Study Elk Creek Niobium Project Nebraska", with an Effective Date of June 30, 2017.

6.1 Ownership History

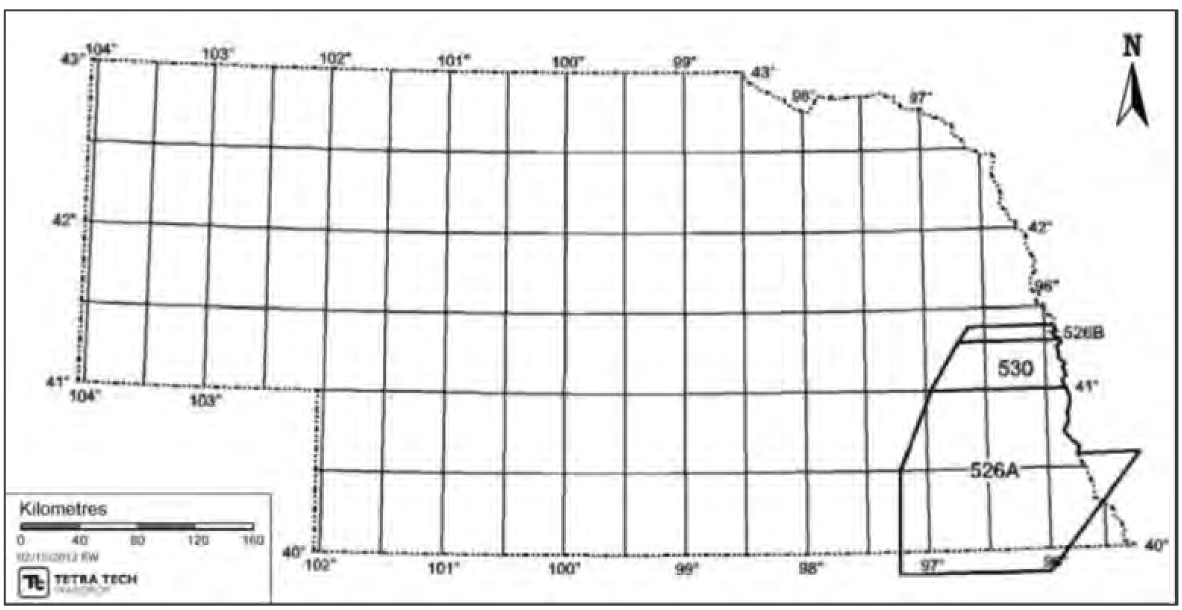
The USGS completed the initial regional geological work. The details of the initial ownership of the complete Project area are not clear, but it is reported that land packages were initially controlled by Cominco American Inc. (Cominco American) and Molycorp during the early 1970s.

The majority of the historical exploration over the Project area was completed by Molycorp before 1984 (ECRC). On May 4, 2010, Quantum announced the acquisition of the mineral rights to the Project. On March 3, 2013, Quantum announced an official name change from Quantum Rare Earth Developments Corp. to NioCorp Developments Ltd. (NioCorp). NioCorp's focus is to develop the Project.

6.2 Exploration History

6.2.1 USGS, 1964

Between November 1963 and January 1964, the USGS flew three airborne magnetic surveys over southeast Nebraska. A total of 6,590 line km was flown (836, 209 and 5,544 line miles, respectively) along an east-west direction at a flight line spacing of 3.2 km (2 miles) and an altitude of 305 m (1,000 ft) above ground (USGS website: OFR 99-0557). Figure 6-1 shows the area covered by the airborne survey.

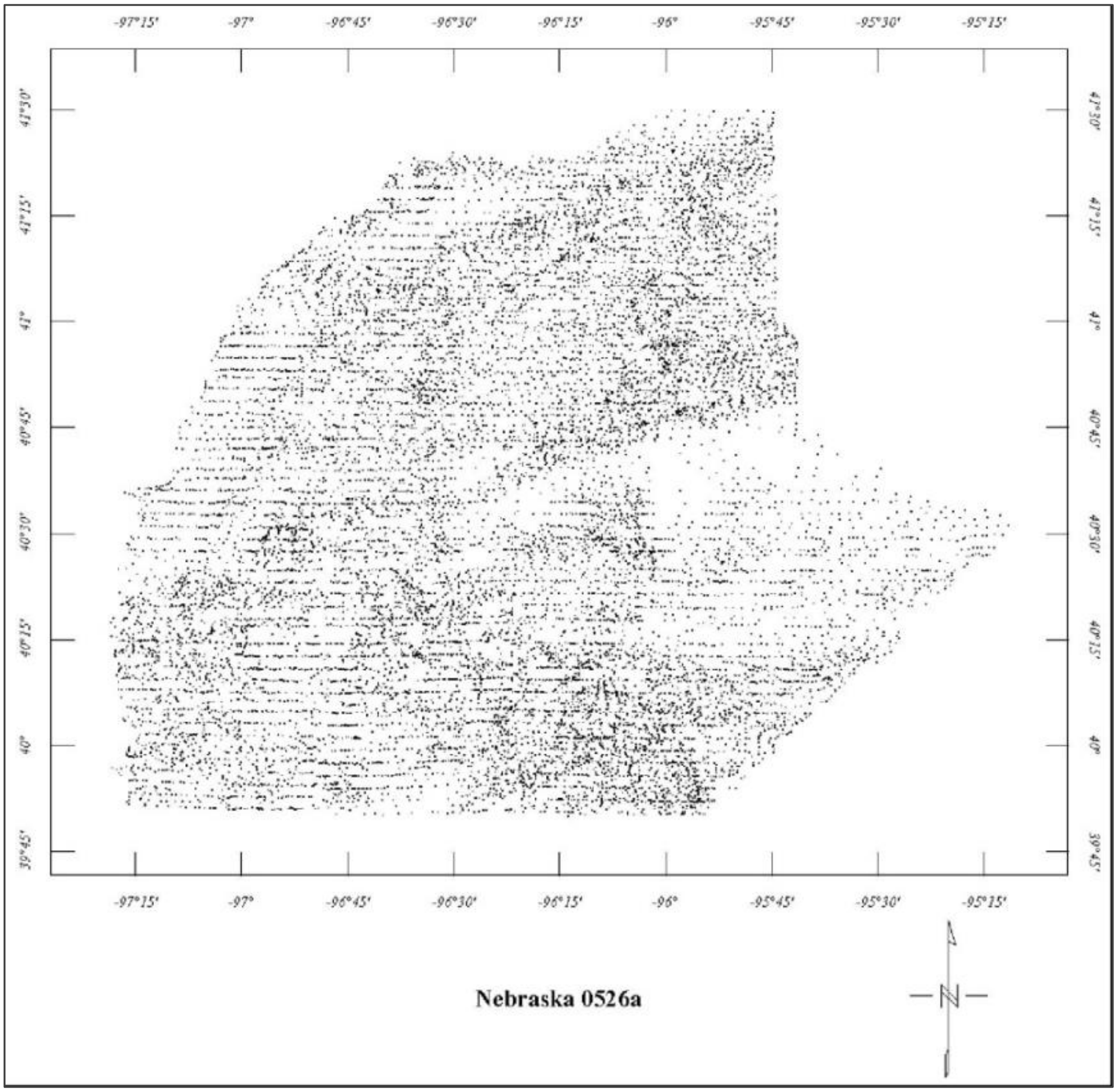


Source: Tetra Tech, 2012 – Modified from USGS, 1964

Figure 6-1: 1964 USGS Aeromagnetic Survey Area Showing Surveys 526A, 526B and 530 Respectively

This wide spacing of the flight lines illustrates only regional features and does not locate local anomalies (i.e., Elk Creek Nb-REE anomaly). Details of the aeromagnetic survey may be found in

USGS Publication 73-297, which was unavailable at the time of writing. Results of the aeromagnetic survey are shown in Figure 6-2.



Source: Tetra Tech, 2012

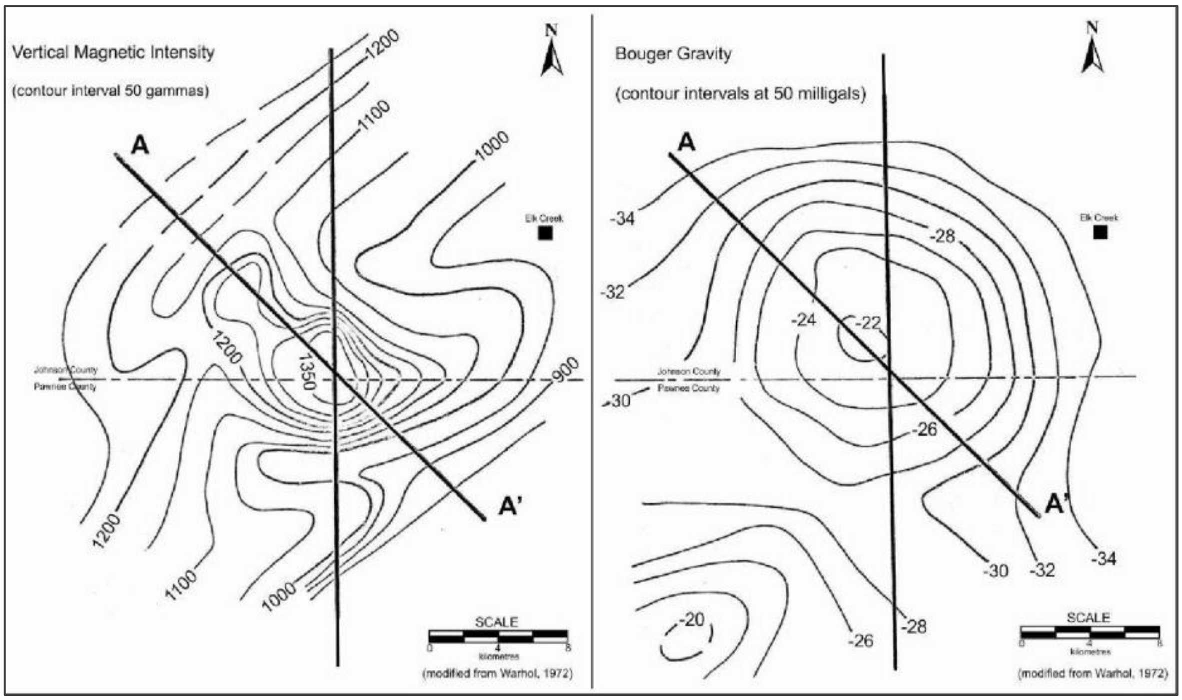
Figure 6-2: 1964 USGS Aeromagnetic Results (Merged 526A, 526B, and 530 Surveys)

6.2.2 Discovery, 1970-1971

Further investigation of the Project was not completed until 1970 when the Elk Creek gravity anomaly was initially identified during a reconnaissance gravity geophysical survey of southeast Nebraska by the Conservation and Survey Division (CSD) of the University of Nebraska-Lincoln (UNL). During the same period the UNL geology department (operating independently), was mapping the magnetic expression of the Nemaha Arch and the Humboldt Fault.

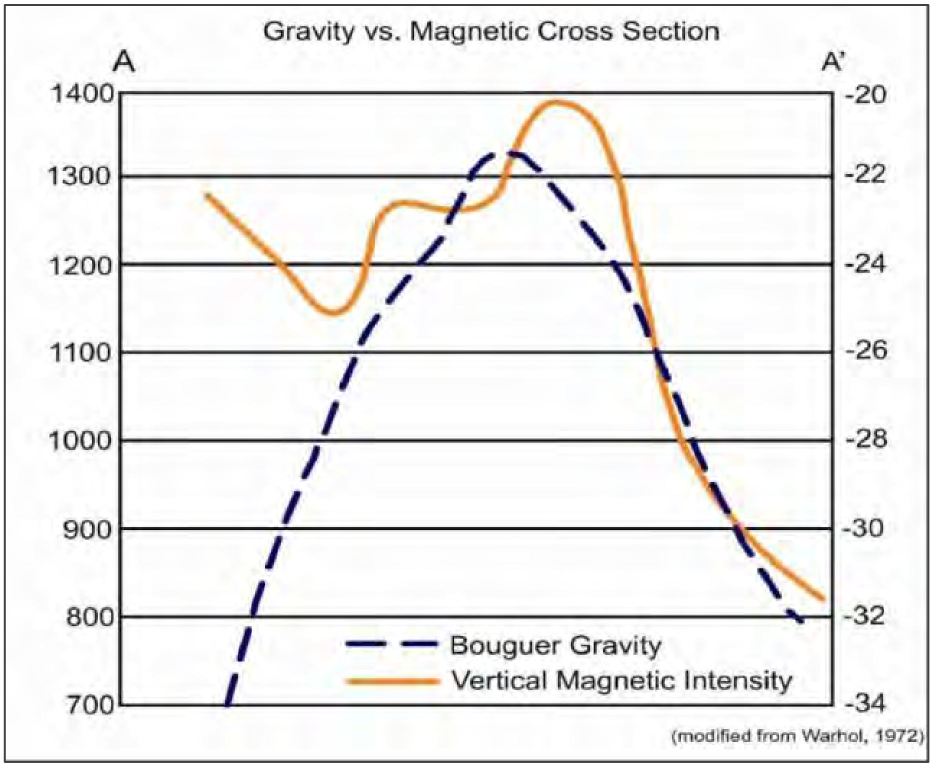
A comparison of the two geophysical survey results showed a positive anomaly that was coincident with a positive gravity anomaly over the area now defined as the Elk Creek gravity anomaly (Anzman, 1976). The geophysical gravity survey outlined a near-circular anomaly, along with a concurrent magnetic anomaly, approximately 7 km in diameter. Analysis of the geophysical data provided a model of a cylindrical mass of indefinite length with a radius of 1,676 m (5,500 ft) (Carlson et al., 2005). Figure 6-3 and Figure 6-4 illustrate the results of the two surveys.

In 1971, the Nebraska Geological Survey (NGS) commissioned a test drill hole 2-B-71 to determine the source of the near circular gravity anomaly. With some support from the United States Bureau of Mines (USBM), the test hole was deepened. The test hole 2-B-71, later renamed NN-1 by Molycorp, encountered 191 m (628 ft) of marine sediments, followed by a carbonate-rich rock (Carbonatite) to the end of the hole at 290 m (952 ft) (Brookins et al., 1975) in what is now referred to as the Elk Creek Carbonatite.



Source: Tetra Tech, 2012

Figure 6-3: Comparison of the 1970 Magnetic and Gravity Geophysical Surveys



Source: Tetra Tech, 2012

Figure 6-4: Cross-section A-A' of the 1970 Gravity and Magnetic Geophysical Surveys

6.2.3 Cominco American, 1974

The earliest known reference to Cominco American operating within the Elk Creek gravity anomaly area is from 1974. It is unclear precisely when Cominco American first acquired the mineral rights in the Elk Creek anomaly area. It is believed that between 1971 and 1973 both Cominco American and Molycorp held mineral rights over selected portions of the Elk Creek gravity anomaly.

In 1974, Cominco American completed five drill holes (CA-1 to CA-5) within the Elk Creek gravity anomaly. Details of the Cominco American drill holes and exploration activities within the property were not available. The information on drilling activities stated here was taken from the Molycorp database. Nordmin has not reviewed or included any information from Cominco American as part of the current study.

6.2.4 Molycorp, 1973-1986

The earliest known reference to Molycorp operating within the Elk Creek gravity anomaly area is from 1973. It is unclear precisely when Molycorp first acquired the mineral rights in the Elk Creek anomaly area. Molycorp completed a number of phases of exploration on the Project during this period including more detailed geophysical surveys, regional drilling (mineralization limits) and focused drilling on the Project area. The exploration program focused on understanding the potential for rare earth elements of economic significance at the Project, with results showing a niobium anomaly at Elk Creek.

Between 1986 and 2011, no further exploration had been recorded on the property.

6.2.5 Geophysical Surveys

In 1973, a detailed aeromagnetic survey was flown by Olympus Aerial Surveys Inc. (Olympic Aerial Surveys), of Salt Lake City, Utah, USA, for Molycorp, with the aim of locating drill sites. Flight lines within the Elk Creek anomaly area were spaced at 200 m, and outside the anomaly at 400 m. A total of 50,764 ha was covered by 2,090 line km (Anzman, 1976). The altitude of the survey was not stated in Anzman 1976.

In 1980, an extensive regional geophysical program was made in southeastern Nebraska including the Elk Creek anomaly. The program consisted of 6,437 line km of aeromagnetics and approximately 4,000 gravity station readings. The aeromagnetic survey was contracted by Olympus Aerial Surveys.

The gravity geophysical survey was conducted by the CSD-UNL, which undertook approximately a quarter of the station readings, and by Molycorp's in-house Geophysical Services Group, which undertook the remaining three-quarters of the gravity station readings.

6.2.6 Drilling

Between 1973 and 1974, Molycorp completed six drill holes: EC-1 to EC-4, targeting the Elk Creek anomaly and two other holes outside the Elk Creek anomaly area (Anzman, 1976). Drill holes were typically carried out by RC drilling through the overlying sedimentary rocks and diamond drilling through the Ordovician-Cambrian basement rocks.

Molycorp continued their drill program from 1977 and, in May 1978, Molycorp made its discovery of the current Project with drill hole EC-11. EC-11 is located on Section 33, Township 4N, and Range 11. The Carbonatite hosting the Project was intersected at a vertical depth of 203.61 m (668 ft).

Molycorp continued its drilling program through to 1984, which mainly centred on the Project within a radius of roughly 2 km. By 1984, Molycorp had completed 57 drill holes within the Elk Creek gravity anomaly area, which included 25 drill holes over the Project area.

From 1984 to 1986, drilling was focused on the Elk Creek gravity anomaly area. The anomaly area is roughly 7 km in diameter and drilling was conducted on a grid pattern of approximately 610 m by 610 m (roughly 2,000 ft by 2,000 ft.) with some closely spaced drill holes in selected areas.

By 1986, a total of 106 drill holes were completed for a total of approximately 46,797 m (153,532 ft). The deepest hole reached a depth of 1,038 m (3,406 ft) and bottomed in carbonatite.

6.2.7 Molycorp Data Verification, 1973-1986

Verification work on the historical database has been completed by Dahrouge Geological Consulting Ltd (Dahrouge), who was contracted by Quantum to compile and verify the historical database between 2010 and 2011. Work included data capture from historical drilling logs, verification drilling and re-analysis of historical samples.

The following excerpt was taken from the Technical Report on the Elk Creek Property, 2010 (McCallum and Cathro, 2010).

"In some of the analytical log sheets available to the Authors, it appears that Molycorp analyzed niobium through their exploration division laboratory at Louviers, Colorado. They also analyzed the same interval at another, unspecified, commercial laboratory. It is unclear to the Authors what material the duplicate analyses were derived from (coarse reject duplicate, pulp duplicate, or % core duplicate).

Molycorp utilized the commercial laboratory, Skyline Labs Inc., of Wheat Ridge, Colorado between 1980 and 1986, with analysis by ICP spectrographic methods and unknown preparation methods.

According to analytical reports and certificates available at UNL, values of lanthanum, cerium, neodymium, barium, sodium, thorium, lead, thorium, uranium, potassium, titanium, zinc, vanadium, niobium, phosphorous, beryllium, zircon, strontium, lithium, yttrium, silver, chromium, copper, iron, manganese, nickel and cobalt were tested. The intervals tested are comprised of commonly 100 ft intervals, presumably composited from the pulverized material of the 10 ft intervals.

In the "Niobium Analytical Standardization" report, dated June 1983, by Sisneros and Yernberg, it was noted that the routine XRF analysis performed by Molycorp's exploration division laboratory at Louviers generated niobium values that were higher than other analytical techniques. This difference in niobium values was concluded not to be a product of preparation techniques, but a result of the standardization errors in the XRF analytical technique. A set of fifteen composites was prepared from Elk Creek drill core samples and analyzed with varying methods including XRF, ICP emission spectrometry and DC plasma emission spectrometry at ten laboratories. It was concluded that the difference was caused by high barium and iron within the matrix of the sample, with the largest deviations found in the coarse-grained material. The deviation of Molycorp's routine analytical method compared to the recommended value ranges from 20% to just below 50% (except for one sample deviating 1%). The recommended value was based on a statistical analysis of the round-robin results.

The correction for the effect of barium and iron on the given Louviers niobium value was calibrated with the XRF instrument at Molycorp's Louviers, Colorado exploration laboratory, and many of the previously analyzed samples were re-tested with the new calibration. The samples that have received the Ba+Fe correction have been noted on the historic Molycorp analytical logs; however, in the later series of holes, it is not identified on the assay log. It is expected that all holes drilled after 1983 were analyzed with the newer calibration."

6.3 Historical Resource Estimates

6.3.1 Molycorp Internal Estimates

During the review of historical documentation and the previous NI 43-101 Technical Report, it has been noted that Molycorp produced an internal estimate of the tonnage and grade within the Project. This estimate is not considered to be compliant with CIM terms and conditions, nor was it documented to a NI 43-101 standard. The estimate is based on assay analysis conducted by Molycorp at its laboratory at Louviers, Colorado, USA and other analytical work at several commercial laboratories.

On February 5, 1986, in an internal Molycorp memo (Cook and Shearer, 1986) from the two principle project geologists (Cook and Shearer) states:

"Niobium Resource Lands (Elk Creek Section 33)

These lands include the Section 33 niobium resource and adjacent untested lands. The resource contains 39.4 million tons of 0.82% Nb_2O_5 and is open to the north, west and at depth."

Tetra Tech commented in the April 2012 NI 43-101 Technical Report that the memo is the only evidence of a historic resource conducted on the property. There are no documents available to explain or support how this resource was estimated. Tetra Tech concluded during its investigation that it was apparent that the historic resource may have been estimated by a polygonal method.

6.3.2 Tetra Tech Wardrop Estimate (April 2012)

In April 2012, Tetra Tech produced a NI 43-101 Technical Report for the Project based on the results of verification work completed by Quantum through Dahrouge. The Tetra Tech Mineral Resource Estimate for the Project was prepared in accordance with CIM Best Practices and disclosed in accordance with NI 43-101, with an Effective Date of March 21, 2012.

The Mineral Resource was estimated by the OK interpolation method using capped grade values. The Mineral Resource for the Project was classified as having Indicated and Inferred resources based on drill hole spacing, drill hole location and sample data population.

The Mineral Resource Estimate for the deposit, at 0.4 Nb_2O_5% cut-off grade (CoG), reported an Indicated resource of 19.3 Mt at 0.67 Nb_2O_5%; and an Inferred resource of 83.3 Mt at 0.63 $Nb_2O_5$5%.

Table 6-1 and Table 6-2 present the Tetra Tech Indicated and Inferred resource estimates for the Project at various Nb_2O_5% cut-offs between 0.35 and 0.70 Nb_2O_5%.

Tetra Tech concluded that the Project warranted further investigation and development.

Table 6-1: Tetra Tech 2012 Indicated Mineral Resource Grade Tonnage Sensitivity for the Project.

Cut-off Nb_2O_5 (%)	Density g/cm^3	Tonnes (000's)	Nb_2O_5 (%)	Contained Metal (000's kg)
0.70	2.96	7,226	0.86	61,940
0.65	2.96	9,113	0.82	74,653
0.60	2.96	11,373	0.78	88,770
0.55	2.96	13,441	0.75	100,722
0.50	2.96	15,844	0.71	113,271
0.45	2.96	17,940	0.69	123,279
0.40	2.96	19,319	0.67	129,182
0.35	2.96	19,632	0.66	130,376

Grey highlight is the cut-off used for quoting the Mineral Resource.
Source Tetra Tech, 2012

Table 6-2: Tetra Tech 2012 Inferred Mineral Resource Grade Tonnage Sensitivity for the Project*.

Cut-off Nb_2O_5 (%)	Density g/cm^3	Tonnes (000's)	Nb_2O_5 (%)	Contained Metal (000's kg)
0.70	2.96	20,984	0.8	167,447
0.65	2.96	32,115	0.76	242,535
0.60	2.96	44,596	0.72	320,521
0.55	2.96	58,803	0.68	402,231
0.50	2.96	71,333	0.66	468,026
0.45	2.96	80,297	0.64	510,904
0.40	2.96	83,288	0.63	523,844
0.35	2.96	83,744	0.63	525,591

Source Tetra Tech, 2012
*Grey highlight is the cut-off used for quoting the Mineral Resource.

6.3.3 SRK Estimates (September 2014 – August 2015)

In September 2014, SRK produced a NI 43-101 Technical Report for the Project based on the historical drill hole information and the results from Phase I of the 2014 NioCorp drilling program. The Mineral Resource Estimate for the Project was prepared in accordance with CIM Best Practices and disclosed in accordance with NI 43-101, with an Effective Date of September 9, 2014.

The Mineral Resource was estimated by the OK interpolation method using capped grade values. The Mineral Resource for the Project was classified as having Indicated and Inferred resources based on drill hole spacing, drill hole location and sample data population.

The Mineral Resource Estimate for the deposit, at 0.3 Nb_2O_5% CoG, reported an Indicated resource of 28.2 Mt at 0.63 Nb_2O_5%; and an Inferred resource of 132.8 Mt at 0.55 Nb_2O_5%.

Table 6-3 provides the Indicated and Inferred resource estimates for the Project, and Table 6-4 shows the grade-tonnage sensitivity at various Nb_2O_5% cut-offs between 0.35 and 0.70 Nb_2O_5%.

SRK concluded that the Project warranted further infill drilling to increase the current level of confidence, and commencement of other technical disciplines investigations such as geotechnical and hydrogeological studies to improve the investigation and development of the project.

Table 6-3: SRK Historical Mineral Resource Statement for the Project, Effective September 9, 2014

Classification	Cut-off (Nb_2O_5%)	Tonnage (000's t)	Grade (Nb_2O_5%)	Contained Nb_2O_5 (000's kg)
Indicated	0.30	28,200	0.63	177,000
Inferred	0.30	132,800	0.55	733,700

(1) Mineral resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material. All composites have been capped where appropriate. The Concession is wholly owned by, and exploration is operated by NioCorp Developments Ltd.

(2) The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves (May 10, 2014) as required by NI 43-101.

(3) SRK assumed the Project was amenable to a variety of underground mining methods. In the absence of definitive pricing for Nb and established rates of metallurgical recovery, SRK reported the Mineral Resource at a cut-off of 0.3% Nb_2O_5. The Company's previous Mineral Resource, dated April 2012, was calculated at a cut-off of 0.4% Nb_2O_5.

(4) SRK completed a site inspection to the deposit by Mr. Martin Pittuck, MSc, C.Eng., MIMMM, an appropriate "independent qualified person" as this term is defined in NI 43-101.

Table 6-4: Grade Tonnage Showing Sensitivity of the Project Mineral Resource (September 2014) to CoG

Classification	Cut-off (Nb_2O_5%)	Tonnage (000's t)	Grade (Nb_2O_5%)	Contained Nb_2O_5 (000's kg)
Indicated	0.60	15,800	0.78	123,700
	0.55	17,400	0.76	132,800
	0.50	19,100	0.74	141,800
	0.45	20,700	0.72	149,600
	0.40	22,600	0.70	157,400
	0.35	25,300	0.66	167,500
	0.30	28,200	0.63	177,200
Inferred	0.60	51,900	0.78	404,900
	0.55	57,300	0.76	435,800
	0.50	63,700	0.74	469,600
	0.45	71,700	0.71	507,700
	0.40	87,000	0.66	573,300
	0.35	111,100	0.60	662,700
	0.30	132,800	0.55	733,700

Source SRK, 2014a

In February 2015, an initial estimate of the Nb_2O_5 Mineral Resource was completed by SRK Consulting on the Elk Creek Deposit (see Table 6-5).

Table 6-5: SRK Historical Mineral Resource Statement - Effective Date February 20, 2015

Classification	Cut-off (Nb_2O_5%)	Tonnage (000's)	Grade (Nb_2O_5%)	Contained Nb_2O_5 (000's kg)
Indicated	0.3	81,200	0.71	578,200
Inferred	0.3	99,800	0.56	557,500

(1) Mineral resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material. All composites have been capped where appropriate. The Concession is wholly owned by, and exploration is operated by NioCorp Developments Ltd.
(2) The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves (May 10, 2014) as required by NI 43-101.
(3) SRK assumed the Elk Creek Deposit to be amenable to a variety of underground mining methods. Using results from initial metallurgical test work, suitable underground mining and processing costs, and forecast niobium price SRK reported the Mineral Resource at a cut-off of 0.3% Nb_2O_5.
(4) SRK completed a site inspection to the deposit by Mr. Martin Pittuck, MSc, C.Eng, MIMMM, an appropriate "independent qualified person" as this term is defined in National Instrument 43-101.

The February 20, 2015 estimate was based on the certified assays for Nb_2O_5 only, with the estimate subsequently updated on receipt of the TiO_2 and Sc ppm assays to produce a Mineral Resource Estimate for the deposit that was disclosed February 23, 2015, with an effective date of April 28, 2015.

The estimated cost information presented here is used as a guide to assist in the preparation of a Mineral Resource Estimate and to select an appropriate resource reporting CoG. The calculated Nb_2O_5 CoG is based on a fixed relationship between Nb_2O_5 and TiO_2 of 3.5 TiO_2:1 Nb_2O_5. Similarly, a Nb_2O_5 and Sc fixed relationship of and 9 Sc: 1 Nb_2O_5 was used for the CoG calculation.

The Mineral Resource was filtered to show all blocks above the mining cut-off to ensure estimates form suitable mining targets. Any isolated blocks of material reporting above cut-off can be removed as they will unlikely warrant the cost of development. No such cases existed at the Project, and all material within the geological wireframes above a cut-off of 0.3 Nb_2O_5 % was considered to have reasonable prospects of being mined via underground methods.

As a result of positive indications from the company's ongoing metallurgical testing and development program, titanium (TiO_2) and scandium (Sc) were added to the Mineral Resource Estimate in February 2015. Both metals can be recovered with simple additions to the existing process flowsheet and will potentially provide additional revenue streams that complement the planned production of ferroniobium.

The Mineral Resource Estimate in Table 6-6 was determined using the economic parameters as defined in the 2015 PEA. The Mineral Resource was disclosed on February 23, 2015.

Table 6-6: SRK Mineral Resource Estimate for the Project, Effective Date April 28, 2015

Classification	Cut-off (Nb_2O_5%)	Tonnage (000's T)	Grade (Nb_2O_5%)	Contained Nb_2O_5 (000's kg)	Grade (TiO_2%)	Contained TiO_2 (000's kg)	Grade (Sc g/t)	Contained Sc (000's kg)
Indicated	0.3	80,500	0.71	572,000	2.68	2,160,000	72	5,800
Inferred	0.3	99,600	0.56	558,000	2.31	2,300,000	63	6,300

(1) Mineral resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material. All composites have been capped where appropriate. The Concession is wholly owned by, and exploration is operated by NioCorp Developments Ltd.

(2) The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves (May 10, 2014) as required by NI 43-101.

(3) SRK assumed the Project was amenable to a variety of underground mining methods. Using results from initial metallurgical test work, suitable underground mining and processing costs, and forecast niobium price SRK has reported the Mineral Resource at a cut-off of 0.3% Nb_2O_5

(4) SRK completed a site inspection of the deposit by Mr. Martin Pittuck, MSc, C.Eng., MIMMM, an appropriate "independent qualified person" as this term is defined in NI 43-101.

6.3.4 SRK Estimates (May 15, 2017)

In May 2017, SRK Consulting completed an updated Mineral Resource Estimate and updated Feasibility Report.

The Mineral Resources utilized the entire assay information from the historical drilling, the NioCorp 2014 drilling program and the 2016 re-assay program. The grade estimation (Nb_2O_5%, TiO_2%, Sc ppm) utilized an Ordinary Kriging (OK) algorithm supported by 5 m sample composites for all the mineralized units, with Inverse Distance Weighting (IDW) to a power of 2, and the nearest neighbour estimate completed as cross-checks. The updated Mineral Resource is based on an additional 203 pulp samples, 374 pulverized pulps, and 90 chip samples re-assayed from historical holes, which had not previously been assayed for TiO_2% or Sc (ppm). The Mineral Resource also accounted for an estimate of the density values, as a relationship has been identified by SRK for higher density values at higher Nb_2O_5, TiO_2 and Fe_2O_3 grades.

The Mineral Resource Estimate in Table 6-7 has been determined using a net smelter return cut-off value based on economic parameters defined as part of the current study. This should be considered the latest estimate for the Project and be used in any future studies.

A summary of the sensitivity of the tonnage and grade to CoG is shown in Table 6-8.

Table 6-7: SRK Mineral Resource Statement for Elk Creek, Effective Date May 15, 2017

Classification	Cut-off NSR (US$/t)	Tonnage (000's t)	Grade Nb_2O_5 (%)	Contained Nb_2O_5 (t)	Grade TiO_2 (%)	Contained TiO_2 (t)	Grade Sc (g/t)	Contained Sc (t)
Indicated	180	90,900	0.66	598,400	2.59	2,353,300	70	6,300
Inferred	180	133,600	0.48	643,800	2.23	2,985,300	59	7,800

(1) Mineral resources are reported inclusive of the Mineral Reserve. Mineral resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be

material. All composites have been capped where appropriate. Historical samples have been validated via re-assay programs, and all drilling completed by NioCorp has been subjected to QA/QC. All composites have been capped where appropriate, and estimates completed using Ordinary Kriging. The Concession is wholly owned by, and exploration is operated by NioCorp Developments Ltd.

(2) The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves (May 10, 2014) as required by NI 43-101.

(3) The Project is amenable to underground longhole open stoping mining methods. Using results from metallurgical test work, suitable underground mining and processing costs, and forecast product pricing SRK has reported the Mineral Resource at an NSR cut-off of US$ 180/t.

(4) NSR uses the following factors:
- Nb_2O_5: 0.699 is a conversion from Nb_2O_5 to Nb, 1,000 is kg conversion, 85.8% is the hydromet plant recovery, 0.96 is the pyromet plant recovery, 100% payability, assuming a US$ 38.5 kg selling price.
- TiO_2: 1,000 is kg conversion, 40.3% is metallurgical recovery, assuming 100% payability, assuming a US$ 0.88/kg is selling price.
- Sc: 93.1% is met recovery, 100% payability, US$ 3,500 kg is selling price per kg of scandium oxide, with a conversion of 0.652 is the amount of Sc in Sc_2O_3
- Price assumptions for FeNb, Sc_2O_3, and TiO_2 are based upon independent market analyses for each product.

(5) SRK completed a site inspection of the deposit by Mr. Martin Pittuck, MSc, C.Eng., MIMMM, an appropriate "independent qualified person" as this term is defined in NI 43-101.

The Mineral Resource presented has been reported following CIM guidelines. Inferred Mineral Resources are not included in the mine plan for this study. Mineral resources that are not Mineral Reserves do not have demonstrated economic viability.

The study includes price and market assumptions concerning an expanded demand in the scandium market. There is no certainty that the prices used will be realized.

Table 6-8: Grade Tonnage Showing Sensitivity of the Mineral Resource to CoG

Classification	Cut-off NSR (US$/t)	Tonnage (000's t)	Grade (Nb_2O_5%)	Contained Nb_2O_5 (t)	Grade (TiO_2%)	Contained TiO_2 (t)	Grade (Sc g/t)	Contained Sc (t)
Indicated	0	91,899	0.65	600,000	2.58	2,366,900	69.13	6,400
	50	91,899	0.65	600,000	2.58	2,366,900	69.13	6,400
	100	91,778	0.65	599,900	2.58	2,366,100	69.20	6,400
	150	91,583	0.65	599,700	2.58	2,364,100	69.31	6,300
	180	90,938	0.66	598,700	2.59	2,354,300	69.62	6,300
	200	90,247	0.66	597,600	2.60	2,343,200	69.92	6,300
	250	88,467	0.67	594,100	2.61	2,311,900	70.61	6,200
	275	87,172	0.68	591,100	2.63	2,290,300	71.05	6,200
	300	85,153	0.69	585,800	2.65	2,254,200	71.67	6,100
	350	79,782	0.71	568,000	2.70	2,150,300	73.19	5,800
Inferred	0	345,322	0.19	652,500	0.87	3,017,700	22.79	7,900
	50	136,311	0.48	650,400	2.21	3,017,700	57.73	7,900
	100	136,275	0.48	650,300	2.21	3,017,300	57.74	7,900
	150	135,174	0.48	647,100	2.22	3,004,500	58.13	7,900
	180	133,550	0.48	643,500	2.23	2,984,200	58.55	7,800
	200	131,414	0.49	638,600	2.25	2,958,500	59.07	7,800
	250	124,544	0.50	621,700	2.30	2,862,000	60.48	7,500
	275	119,198	0.51	607,300	2.33	2,776,100	61.43	7,300
	300	111,489	0.53	585,600	2.37	2,640,500	62.64	7,000
	350	89,420	0.58	516,000	2.49	2,223,900	65.79	5,900

Source: SRK, 2016

There has been no material change to the block estimates in this current update as no additional drilling has been completed. The only two differences between the models have been:

- The addition of TiO_2 and Sc assays which were previously absent; and
- Change in the reporting criteria from using a 0.3% Nb_2O_5 CoG to an NSR basis.

Overall the result has been an increase in the tonnage in both the Indicated and the Inferred categories as more marginal material is considered economical. The Indicated portion of the Mineral Resource has increased by 13% in terms of tonnage, while the Inferred has increased by 34%. In terms of the contained Nb_2O_5 metal, the latest model represents an increase of 5% in the Indicated and 15% in the Inferred respectively, while the TiO_2 and Sc increase approximately 9% in the Indicated, and 29% and 24% in the Inferred respectively.

6.4 Historic Production

There has been no historical production from the Mineral Resource at the Project.

7. GEOLOGICAL SETTING AND MINERALIZATION

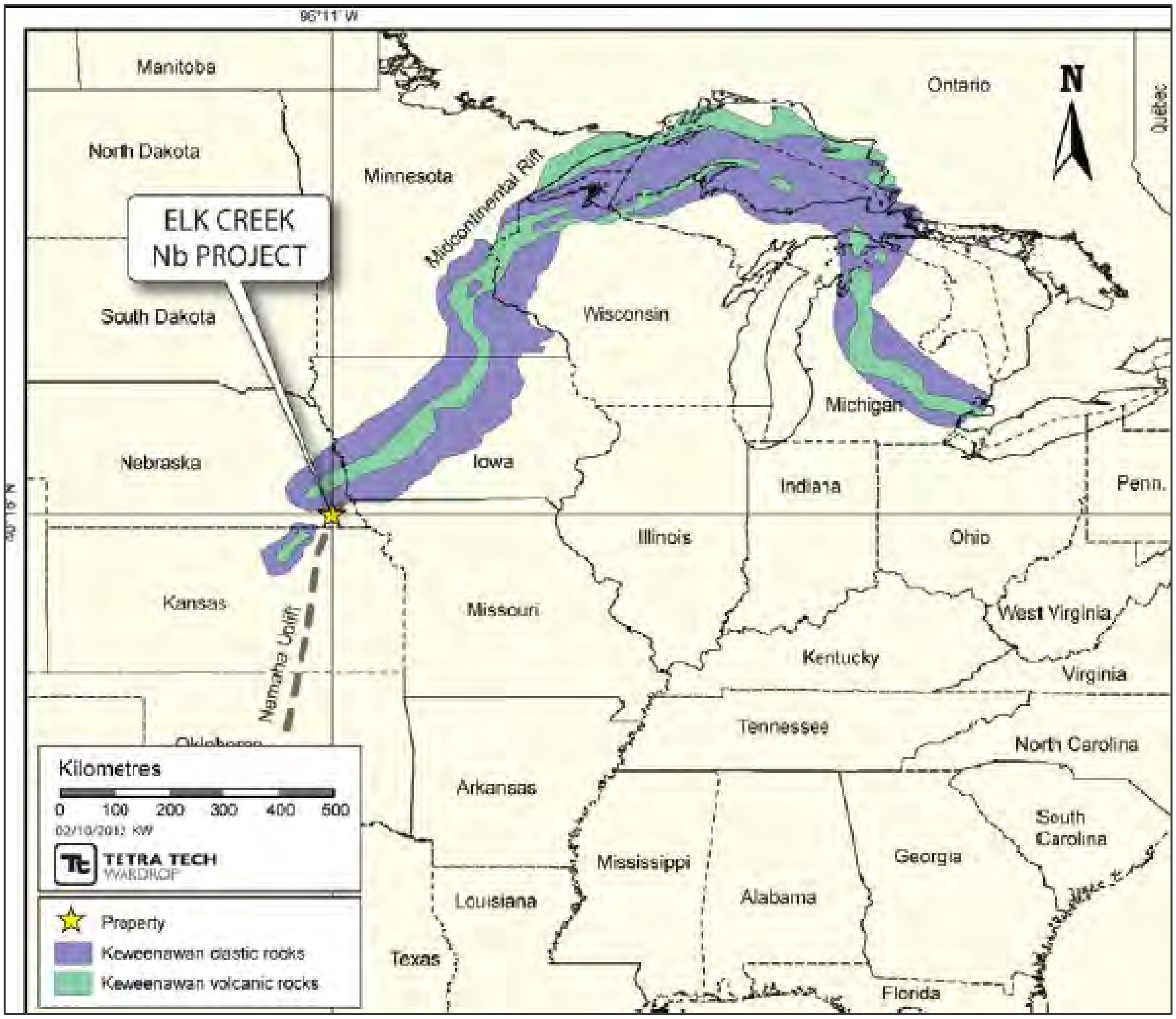
7.1 Regional Geology

The Nebraska Precambrian basement predominantly comprises granite, diorite, basalt, anorthosite, gneiss, schist and clastic sediments. A series of island arcs sutured onto the Archean continent created the basic framework of the area. This suture left a north-trending intervening boundary zone ancestral to the Nemaha Uplift, providing a pre-existing tectonic framework which controlled the trend of the later Midcontinent Rift System (1.0 to 1.2 Ga) (Carlson & Treves, 2005). The carbonatite is located at the northeast extremity of the Nemaha Uplift.

The Midcontinent Rift System, or Keweenawan Rift, comprises mafic igneous rocks and forms a belt over 2,000 km long and 55 km wide that is exposed at the surface in the Lake Superior Region and extends southwards through the states of Michigan, Wisconsin, Minnesota, Iowa, Nebraska and into Kansas (Carlson, 1992). Both basalt and associated red clastic sedimentary rocks are found in the Precambrian basement of southeastern Nebraska. These rocks are very similar to those found in the Lake Superior region and are thus considered to be a product of the Keweenawan rifting (Burchett and Reed, 1967; Treves et al., 1983). Figure 7-1 illustrates the major rock types of the Midcontinental Rift system.

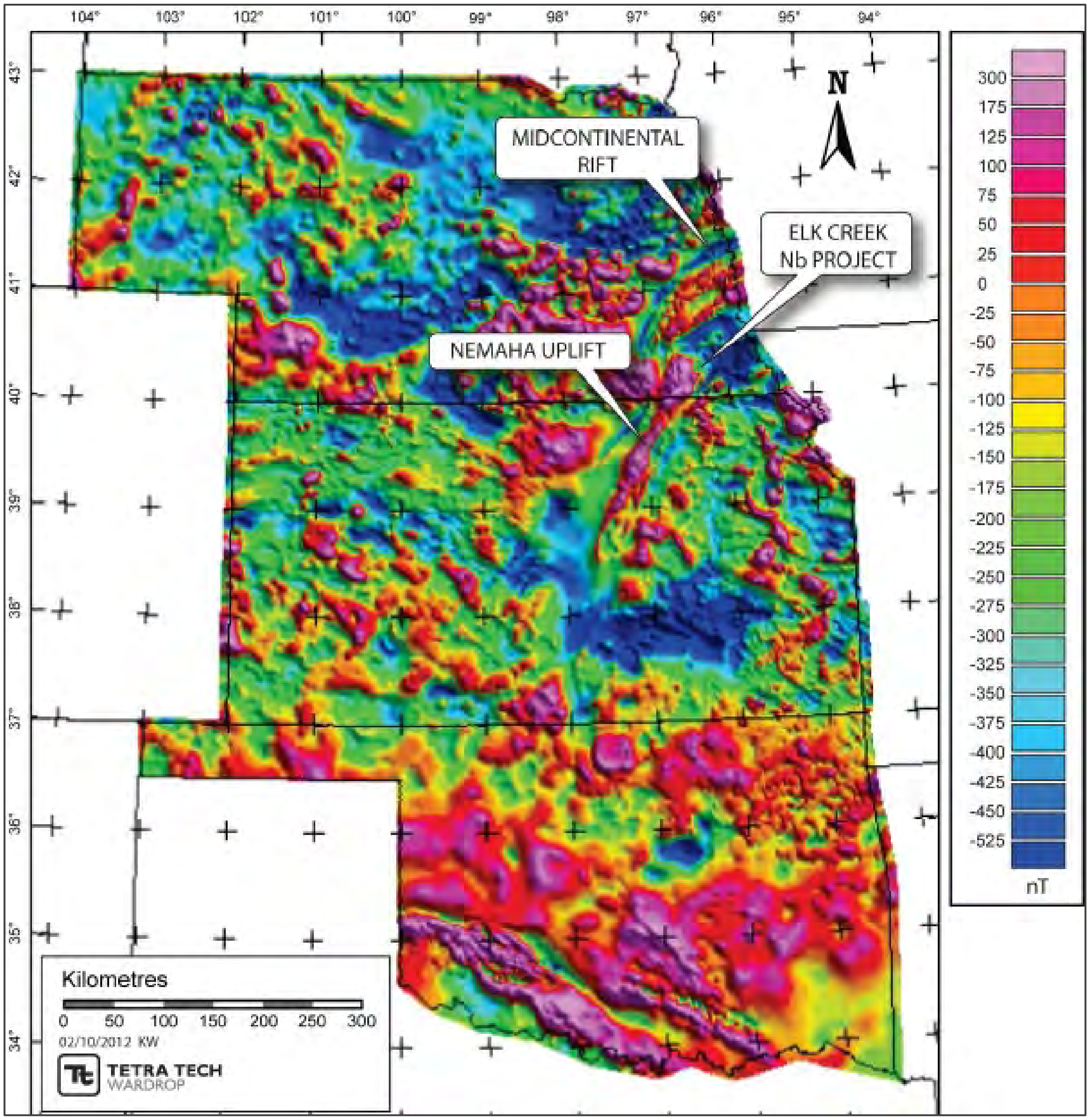
The Nemaha Uplift (300 Ma) extends southward as a narrow belt from southeastern Nebraska across Kansas along the midcontinent rift system (King, 1969) (see Figure 7-1). Along the northern and eastern margins are complex fault zones and steeply dipping units. Regional north-northeast to northeast striking faults are locally transected by northwest trending ones, including the Central Plains mega shear (Central Missouri Fault) to the north and the Oklahoma mega shear to the south (McBee, 2003). The Carbonatite body intruded near to the axis of the Nemaha uplift and has similar dates to a cluster of carbonatites north of Lake Superior that are in the range of 560 to 580 Ma. (Woolley, 1989; Erdosh, 1979). Temporally the carbonatite occurs near the boundary between the Penokean Orogen (approximately 1,840 Ma) and the Dawes terrane (1,780 Ma) of the Central Plains Orogen (Carlson and Treves, 2005).

Figure 7-2 shows a merged airborne magnetic anomaly map of Nebraska, Kansas and Oklahoma states (USGS, 2004) showing the Midcontinent Rift and Nemaha Uplift systems.



Source: Modified from Palacas et al., 1990

Figure 7-1: Regional Geology Map



Source: Modified from USGS 2004, showing the Midcontinental Rift and Nemaha Uplift.

Figure 7-2: Merged Aeromagnetic Anomaly Map of Nebraska, Kansas and Oklahoma States

Regional geophysical data and drilling have confirmed the presence of kimberlitic intrusive bodies in northern Kansas to the southwest of the Carbonatite. These kimberlites were emplaced along the rift system during the Cretaceous time (Berendsen and Weis, 2001).

The Paleozoic rocks overlying the carbonatite region are dominated by approximately 200 m of essentially flat-lying Pennsylvanian marine strata consisting of carbonates, sandstones and shales. The eastern portion of Nebraska was glaciated several times throughout the early Pleistocene (Wayne, 1981), resulting in the deposition of approximately 50 m of unconsolidated till.

7.2 Property Geology

The property includes the Carbonatite that has intruded older Precambrian granitic and low- to medium-grade metamorphic basement rocks. The Carbonatite and Precambrian rocks are believed to be unconformably overlain by approximately 200 m of Paleozoic marine sedimentary rocks of Pennsylvanian age (ca. 299 to 318 Ma).

As a result of this thick cover, there is no surface outcrop within the Project area of the carbonatite, which was identified and targeted through magnetic surveys and confirmed through subsequent drilling. The available magnetic data indicates dominant northeast, west-northwest striking lineaments and secondary northwest and north oriented features that mimic the position of regional faults parallel and/or perpendicular to the Nemaha Uplift.

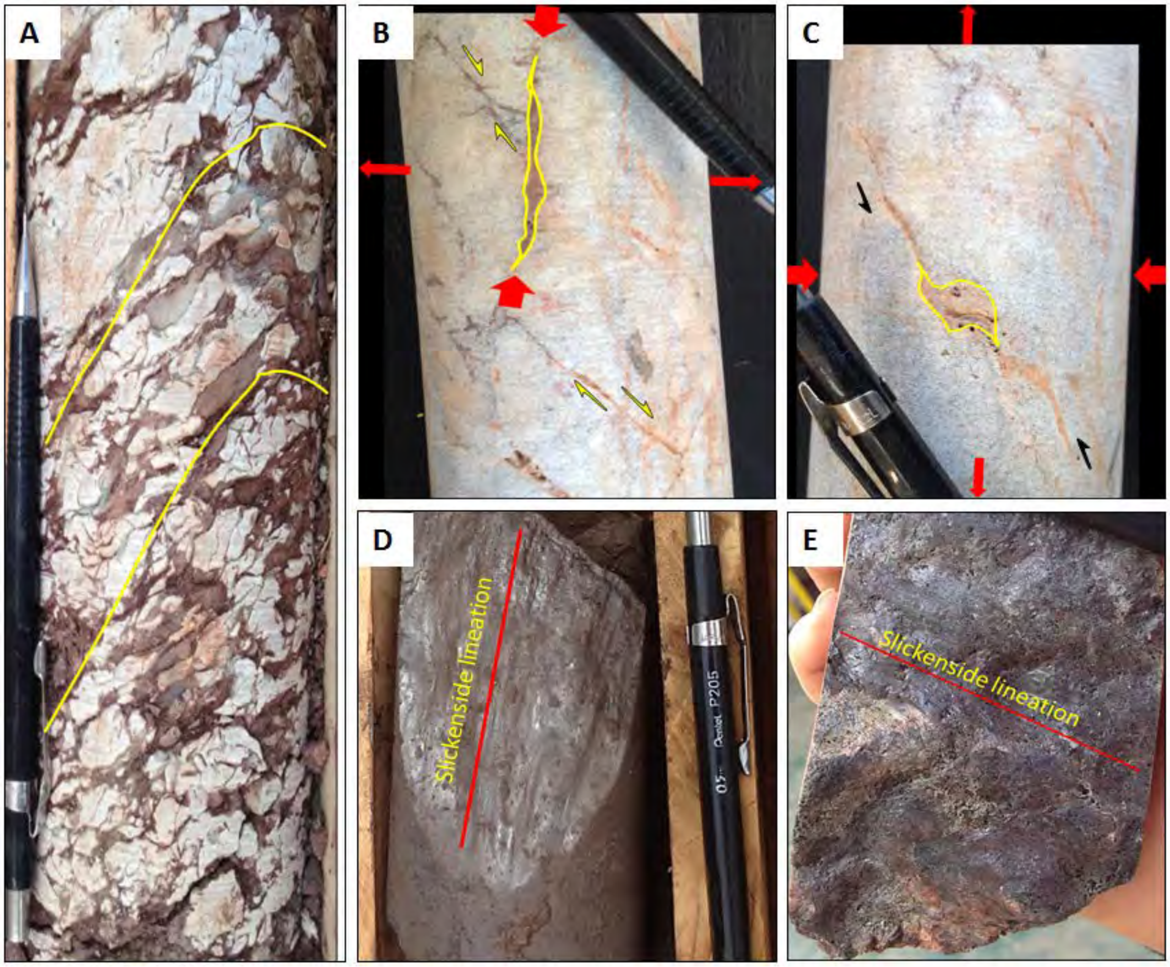
7.3 Elk Creek Carbonatite

The Elk Creek Carbonatite is an elliptical magmatic body with a northwest-trending long axis perpendicular to the strike of the 1.1 Ga Midcontinent Rift System, near the northern part of the Nemaha uplift (Burchett, 1982; Carlson, 1992). The definitive confirmation of carbonatite was completed using Rare Earth Element (REE), P_2O_5 and 87Sr/86Sr isotope analysis (Brookins et al., 1975). The carbonatite has also been compared to the Iron Hill carbonatite stock in Gunnison County, Colorado, based on similar mineralogy (Xu, 1996).

The Carbonatite consists predominantly of dolomite, calcite and ankerite, with lesser chlorite, barite, phlogopite, pyrochlore, serpentine, fluorite, sulphides and quartz (Xu, 1996). The stratigraphic reconstruction based on drill core observation in the area suggests that the carbonatite is unconformably overlain by approximately 200 m of essentially flat-lying Palaeozoic marine sedimentary rocks, including carbonates, sandstones and shales of Pennsylvanian age (ca. 299 to 318 Ma).

Current studies suggest that the Carbonatite was emplaced ca. 500 Ma (Xu, 1996) in response to stress along the Nemaha Uplift boundary predating deposition of the Pennsylvanian sedimentary sequence (ca. 299 to 318 Ma). Observations on drill cores from the Project site show that the contact between the carbonatite body and the Pennsylvanian sediments is a sheared but oxidized contact suggesting that the carbonatite is intrusive in the Pennsylvanian sequence (see Figure 7-3 and Figure 7-4). Furthermore, both rock types appear to have been affected by at least one main brittle-ductile deformation event resulting in the formation of fault structures. Microstructures including sub-vertical and sub-horizontal tension veins, together with related sheared veins and fault planes displaying sub-vertical and sub-horizontal slickensides along drill cores are indications for the presence of extensional and oblique to strike-slip faults (see Figure 7-3 and Figure 7-4). These faults may correspond to the magnetic lineaments present in the area. Investigations aiming to define the location, as well as the orientation and kinematics of these structures are discussed in more detail in Section 7.6.

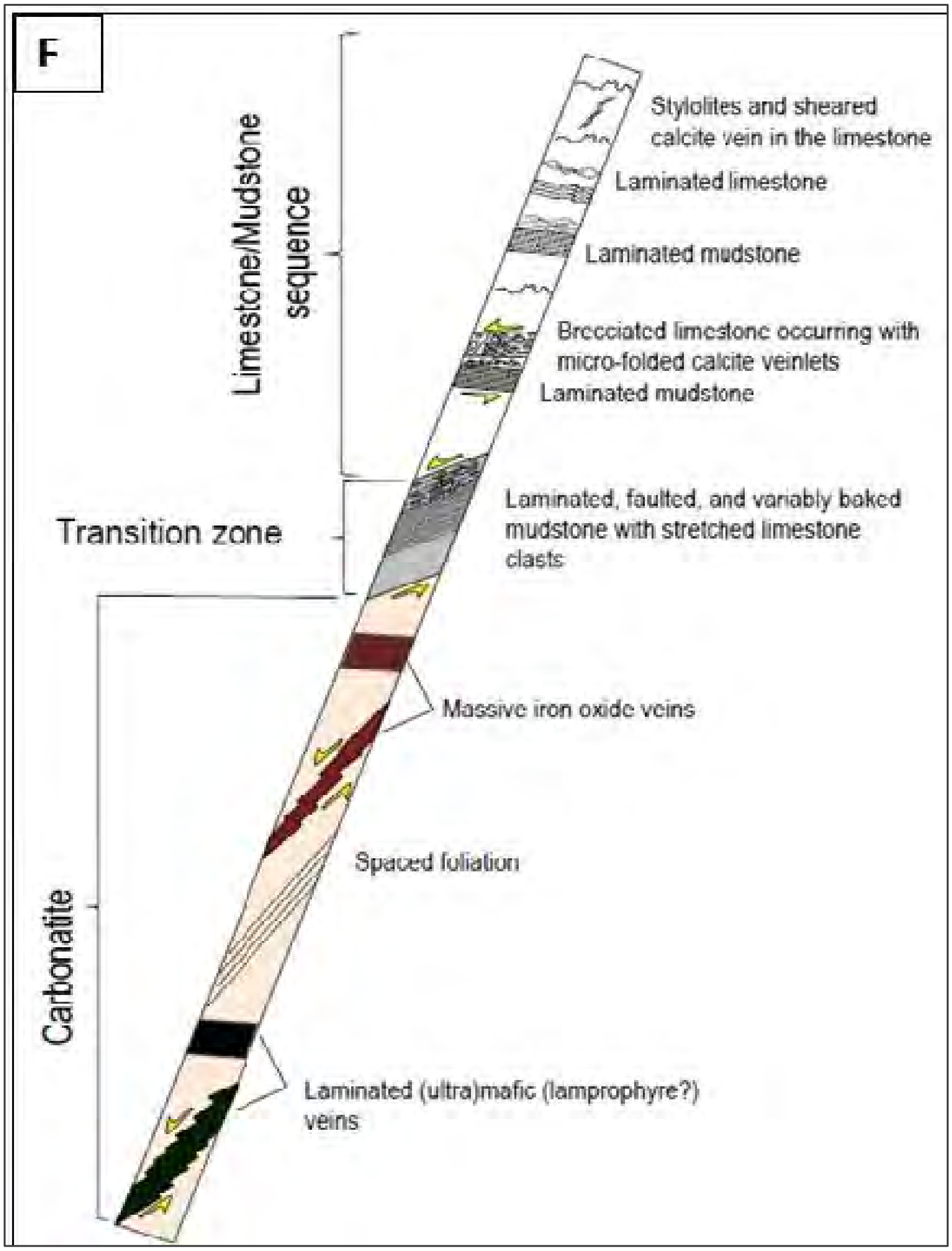
Microstructures presented in Figure 7-3 suggest the presence of extensional and strike-slip to oblique faults in the area as follows: (A) Spaced foliation and breccia in the contact zone between the Carbonatite and the Pennsylvanian sequence; Subvertical (B) and subhorizontal (C) tension veins and associated sheared veins in the Carbonatite; Fault planes showing subvertical (D) and oblique (E) slickensides in the Carbonatite. Note that observations were made on cores from subvertical holes (about 70° plunge).



Source: SRK, 2014a

Figure 7-3: Core Photographs Showing Microstructures

Figure 7-4 presents microstructures along a composite subvertical drill core, suggesting that the Carbonatite is intrusive within the Pennsylvanian rock sequence.



Source: SRK, 2014a. Illustration not to scale.

Figure 7-4: Schematic of Drill Hole Showing the Typical Transition from Pennsylvanian Sediments to Carbonatite Units

Figure 7-5 and Figure 7-6 outline specific intervals within drill hole NEC14-022. These intervals include the contact between the Pennsylvanian sequence and the corresponding next approximate 5 m of carbonatite below the contact with the sediments. NEC14-022 is located on the southeastern extent of the mineralized carbonatite and within approximately 250m of the proposed production and ventilation shaft locations. The faulted and/or fractured mudstone contact is approximately 2 m in thickness and is bound by massive limestone and barite dolomite carbonatite units. The contact has had little to no water movement along or near the contact for the mudstone is relatively fresh and not weathered or significantly stained due to water flow.

Source: Nordmin, 2019

Figure 7-5: Drill Hole NEC14-022, ~ 2m Interval of the Mudstone Contact Between the Pennsylvanian Sediments and the Carbonatite Units

Source: Nordmin, 2019

Figure 7-6: Drill Hole NEC14-022, Relatively M=Massive Dolomitic Carbonatite ~ 3m Below the Contact with the Pennsylvanian Sediments in Figure 7-5

7.3.1 Age Dating

An original hypothesis suggested that the Elk Creek Carbonatite was of Keweenawan age (Treves et al., 1983) or ca. 1,100 Ma. In 1985, Paterman, of the USGS Isotope Laboratory, provided a K-Ar age of 544 (±7) Ma (Cambrian) from biotite within the carbonatite. Two more K-Ar dates were provided by Georgia State University (M. Ghazi (date unknown)) which also provided dates from biotite samples. The ages of 464 (±5) Ma and 484 (±5) Ma, respectively, are Ordovician and thus much younger than the Midcontinent Rift System. While these radiometric dates provide a generalized time range for the carbonatite intrusion, additional age dating is required to establish a more precise date.

7.4 Carbonatite Lithological Unit

The lithological units present in the Carbonatite complex were defined by Molycorp during their drill programs and simplified by Dahrouge for interpretation purposes during each stage of the Project (2011 and 2014). The units in Table 7-1 (youngest at the top) represent the data captured during the 2011 field program. The information was compiled from the drill logs and the corresponding geology reports for each drill hole.

Table 7-1: Project Rock Types as Defined by Molycorp and Dahrouge (2011)

Name (Molycorp)	Code	Name (Dahrouge)	Code
Overlying Lithologies			
Quaternary sediments	Qt	Overburden	Ovb
Pennsylvanian Sediments	Pu	Pennsylvanian Sediments	sed
Elk Creek Complex			
Younger Mafic Rock	ym		mafBc
Barite Beforsite III Barite Beforsite II	bb III bb II	Barite Dolomite Carbonatite	dolCarb
Beforsite Breccia	bbx	Dolomite Carbonatite Breccia	dolCarbBc
Barite Beforsite I	bb I	Barite Dolomite Carbonatite	dolCarb
Apatite Beforsite II Apatite Beforsite I	ab II ab I	Apatite Dolomite Carbonatite Breccia	dolCarb
Older Mafic Rock	om	Mafic dyke, vein or fragment	maf
Magnetite Beforsite	mb	Magnetite Dolomite Carbonatite	mdolCarb
Syenite II Syenite I	sy II sy I	Syenite	sy
Host Rocks			
Granite/Gneiss	pCgg	Granite/Gneiss	gn
Amphibole Biotite — Gneiss	pCbg	Amphibole Biotite — Gneiss	gn

A study of six Molycorp drill holes by Xu (1996) identified two main phases within the area, a carbonate phase and a silicate phase. The study was based on drill holes 2-B-71 (also known as "NN-1"), EC-40, EC-42, EC-50, EC-70 and EC- 82.

The carbonate phase was classified into two main units (defined by texture, massive or brecciated) and several sub-units (defined by mineralogy as presented below).

<u>Massive Carbonatite</u>

- Dolomite carbonatite
- Apatite bearing dolomite carbonatite and pyrochlore-bearing carbonatite
- Apatite dolomite carbonatite
- Hematite dolomite carbonatite
- Magnetite dolomite carbonatite

<u>Brecciated Carbonate</u>

The silicate phase was also classified into several units as follows:

- Altered basalt
- Altered lamprophyre
- Altered syenite

In the 2014 drilling, the Dahrouge geologists split the dolCarb units down into a number of key units using the information of the different phases of carbonatite. The main carbonatite lithologies used are:

- Dolomite Carbonatite — dolCarb
- Dolomite Carbonatite Breccia — dolCarbBc
- Hematite dolomite Carbonatite — hemdolCarb
- Magnetite dolomite Carbonatite — mdolCarb
- Magnetite dolomite Carbonatite Breccia — mdolCarbBc

Nordmin and SRK both consider the more detailed split of the Carbonatite units to be relevant to determining the distribution of different grade populations as supported by statistics (discussed in Section 14.3). The most significant difference is the change in the logging codes between dolCarb and mdolCarb, in terms of the major rock types.

7.5 Marine Sedimentary Rocks

The State of Nebraska-wide test hole database contains information for about 5,500 test holes drilled since 1930 by the CSD, School of Natural resources (SNR), UNL (UNL-CSD/SNR), and cooperating agencies. Test hole location data, as well as lithological descriptions, stratigraphic interpretations and geophysical log records, are included in the database. In addition, UNL-CSD/SNR maintains an extensive collection of geologic samples obtained from the drilling process (UNL-CSD/SNR website).

The overlying sedimentary units on the Project are of Pennsylvanian age. The CSD's 1971 test hole 2-B-71, also labelled NN-1 by Molycorp, intersected several formations of overlying Pennsylvanian strata (see Table 7-2).

Table 7-2: Stratigraphy Overlying the Elk Creek Carbonatite

System	Series	Group	Formation	Member	Depth From (ft)	Depth To (ft)
Quaternary	-	-	-	-	0.00	43.90
Pennsylvanian	Virgilian	Wabaunsee	Zeandale	Wamego	43.90	82.50
Pennsylvanian	Virgilian	Wabaunsee	Emporta	Elmont	82.50	95.00
Pennsylvanian	Virgilian	Wabaunsee	Auburn	-	95.00	113.50
Pennsylvanian	Virgilian	Wabaunsee	Bern	Wakarusa	113.50	138.60
Pennsylvanian	Virgilian	Wabaunsee	Scranton	-	138.60	238.80
Pennsylvanian	Virgilian	Wabaunsee	Howard	-	238.80	243.10
Pennsylvanian	Virgilian	Wabaunsee	Severy	-	243.10	265.50
Pennsylvanian	Virgilian	Shawnee	Topeka	Coal Creek	265.50	292.00
Pennsylvanian	Virgilian	Shawnee	Calhoun	-	292.00	292.80

Pennsylvanian	Virgilian	Shawnee	Deer Creek	Ervine Creek	292.80	331.00
Pennsylvanian	Virgilian	Shawnee	Tecumseh	-	331.00	341.50
Pennsylvanian	Virgilian	Shawnee	Lecompton	Avoca	341.50	369.00
Pennsylvanian	Virgilian	Shawnee	Kanawaka	-	369.00	370.00
Pennsylvanian	Virgilian	Shawnee	Oread	Kereford	370.00	422.30
Pennsylvanian	Virgilian	Douglas	-	-	422.30	478.40
Pennsylvanian	Missourian	Lansing	Stanton	South Bend	478.40	494.70
Pennsylvania	Missourian	Lansing	Stanton	Rock Lake	494.70	500.00
Pennsylvanian	Missourian	Lansing	Stanton	Stoner	500.00	515.10
Pennsylvanian	Missourian	Lansing	Vilas	-	515.10	516.40
Pennsylvanian	Missourian	Lansing	Plattsburgh	-	516.40	523.40
Pennsylvanian	Missourian	Kansas City	Bonner Springs	-	523.40	526.50
Pennsylvanian	Missourian	Kansas City	Wyandotte	Farley	526.50	565.00
Pennsylvanian	Missourian	Kansas City	Lane	-	565.00	567.40
Pennsylvanian	Missourian	Kansas City	Iola	-	567.40	590.00
Pennsylvanian	Missourian	Kansas City	Chanute	-	590.00	594.40
Pennsylvanian	Missourian	Kansas City	Drum	-	594.40	602.50
Pennsylvanian	Missourian	Kansas City	-	-	602.50	628.30
Cambrian	Undifferentiated	-	Elk Creek Carbonatite	-	628.30	952.00

Test Hole 2-B-71 or NN-1
Source: McCallum and Cathro, 2010

There are active limestone quarries, and underground mines within approximately 70 km from the project site that create road materials, lime, back fill and construction materials. These mines are actively mining approximately 2.2 million tons/year from within the Pennsylvanian Limestone units. The Pennsylvanian limestone unit is the same unit that is currently located above the Carbonatite unit at the project site.

7.6 Structural Geology

Based on data provided to carry out this structural study, Nordmin and SRK conclude that the Project contains five main sets of brittle faults variably cutting through the Pennsylvanian rocks and the Carbonatite boundary which appears to be tectonic. The orientations of the faults were determined by comparing Acoustic Televiewer (ATV) logs with specific Nordmin, and SRK customized structural core logging data, and by undertaking a preliminary interpretation of the provided geophysics images.

Nordmin and SRK have used this data to model the fault pattern in 3D for use in further resource estimation and geotechnical studies. The overall fault model included approximately 28 structures with the vicinity of the Project with varying levels of confidence. Based on a review within the mineralization, at least three key northeast trending faults have been identified and used during the geological model process.

The joints and veins define orientation sets comparable to the fault trends. Hematite veins, which may be up to a metre thick, represent the weakest fault and joint infilling material which may be problematic for mining and should, therefore, be given more attention during any future geotechnical studies.

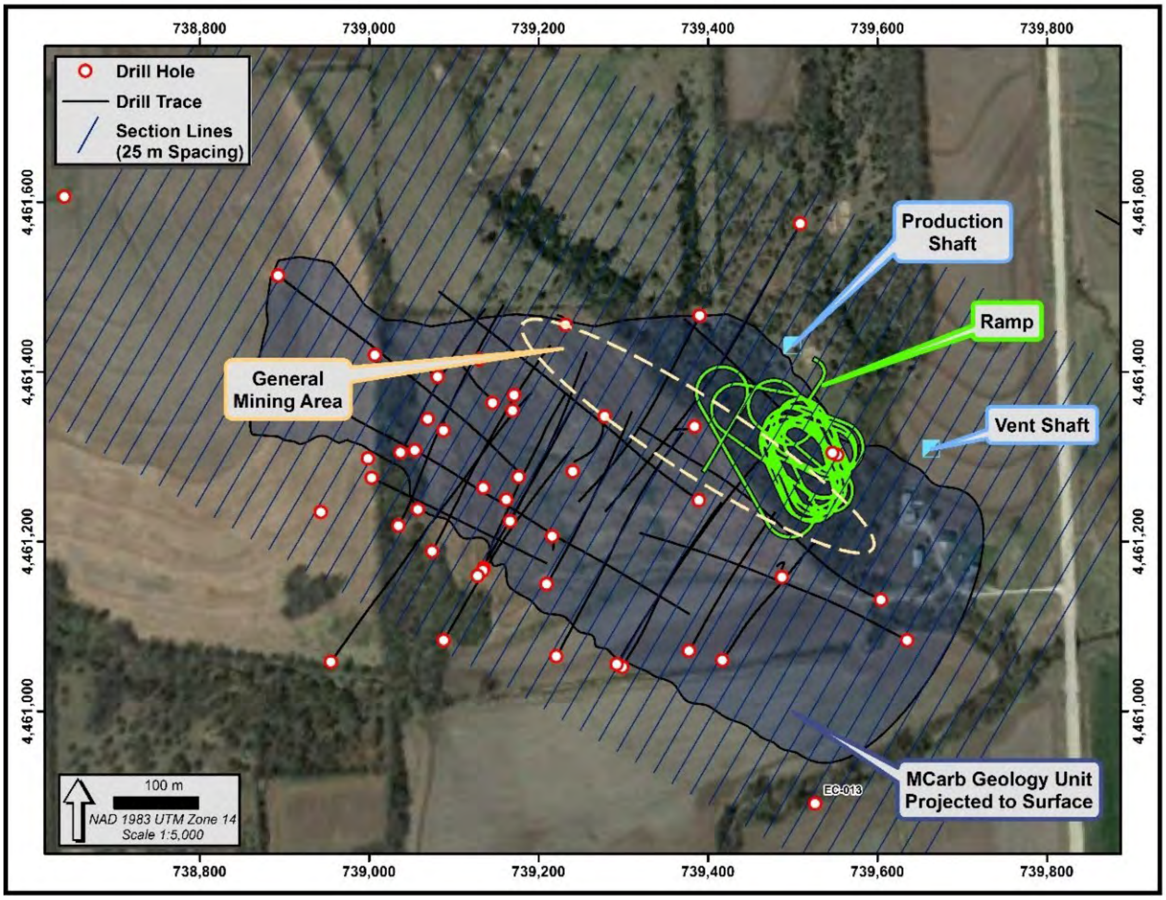
7.7 Mineralization

The property hosts niobium, titanium, and scandium mineralization as well as REE and barium mineralization that occur within the Elk Creek Carbonatite.

The current known extents of the high grade niobium, titanium, and scandium are approximately 750 m along strike, 400 m wide, and 800 m in dip extent below the unconformity.

The current known extents of the low grade niobium, titanium, and scandium are approximately 830 m along strike, 500 m wide, and 850 m in dip extent below the unconformity.

Figure 7-5 demonstrates the mineralization is open in all directions. For this Technical Report, niobium, titanium and scandium are considered the main elements of interests, within the additional background on REE mineralization, included and discussed below.



Source: Nordmin, 2019

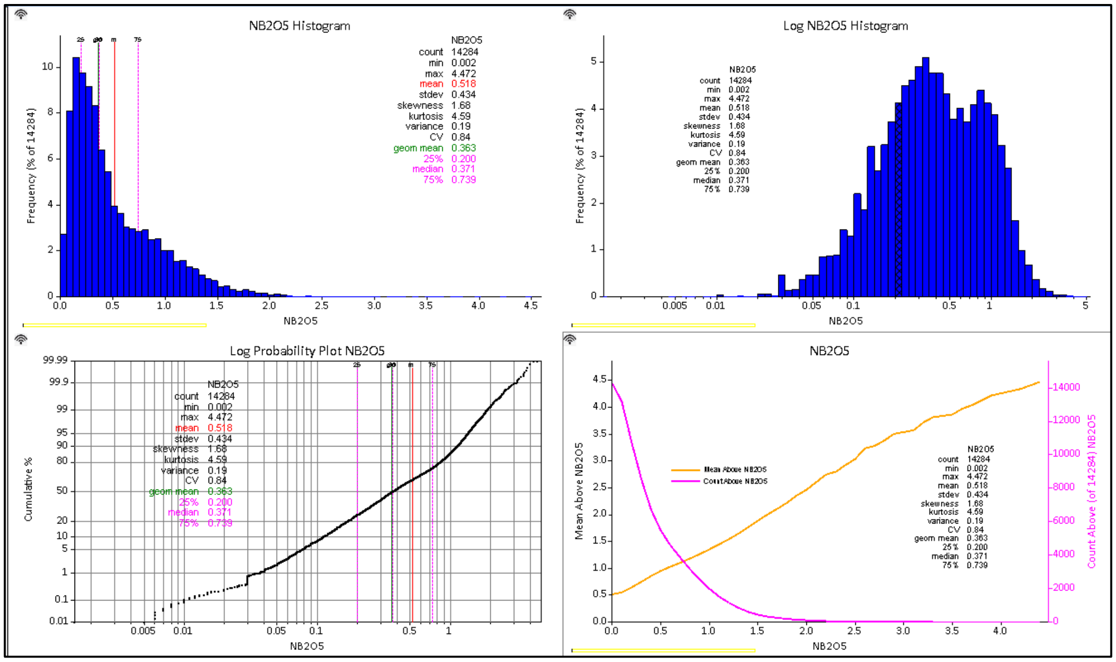
Figure 7-5: Plan View of the Location of the Mineralized Carbonatite (Mcarb)

The initial Molycorp drill hole database contained a separate geological report summarizing rock types, assay results and associated petrographic descriptions identifying niobium and/or REE minerals. Niobium was reported to be hosted in pyrochlore, and REE mineralization was reported to occur as bastnäsite, parisite, synchysite and monazite. During the 2014 NI 43-101, Technical Report SRK highlighted that the level of detail shown in the geological reports had not been transferred to the 2014 SRK resource model. Since 2014, the database was improved by Dahrouge geologists familiar with the current logging codes, conducting a review of the historical logs, reports and available drill core to provide an updated geological database.

7.7.1 Niobium and Titanium Mineralization

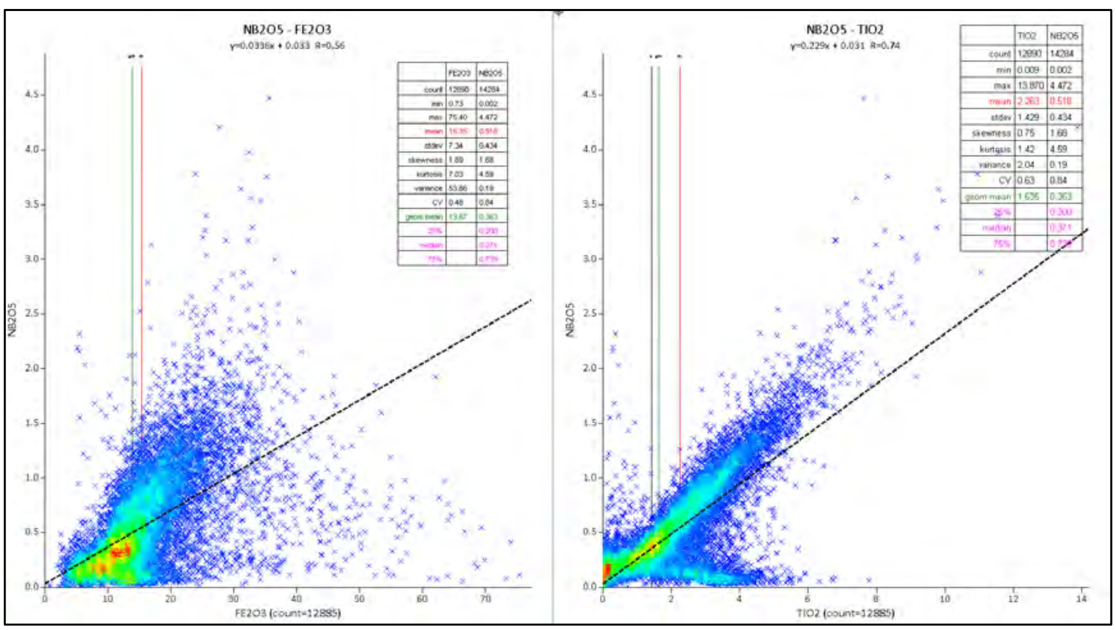
The deposit contains significant concentrations of niobium. Based on the metallurgical test work completed to date at a number of laboratories using QEMSCAN® analysis, the niobium mineralization is known to be fine-grained, and that 77% of the niobium occurs in the mineral pyrochlore, while the balance occurs in an iron-titanium-niobium oxide mineral of varying composition. Within the mineralized Carbonatite, there are 14,284 samples of Nb_2O_5, the maximum Nb_2O_5 grade is 4.472%, and the mean Nb_2O_5 grade is 0.518% (see Figure 7-6).

Figure 7-7 demonstrates that there is a fairly high correlation between increasing Nb_2O_5 grade and Fe_2O_3 and TiO_2 grades.



Source: Nordmin, 2019

Figure 7-6: Basic Statistics of Nb_2O_5 Mineralization



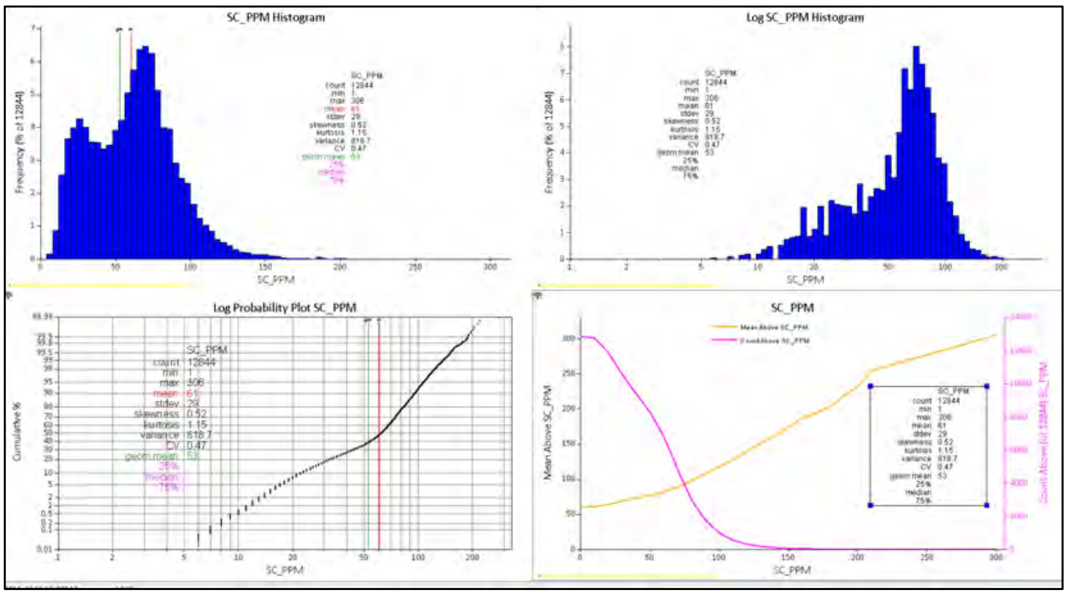
Source: Nordmin, 2019

Figure 7-7: Correlation Statistics of Nb_2O_5 and TiO_2 and Fe_2O_3

7.7.2 Scandium Mineralization

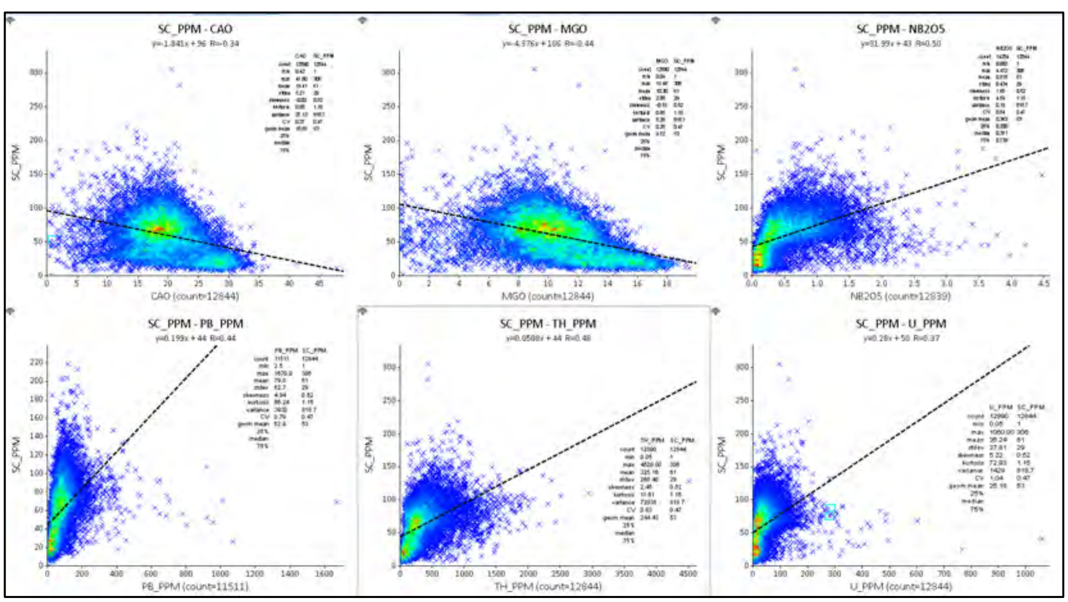
Within the Elk Creek Carbonatite, a host of other elements exist with varying degrees of concentration. The Company has completed both whole rock analysis and multi-element analysis on all samples for the 2014 program, plus re-sampling of selected historical core and/or pulps between 2011 and 2014.

As the metallurgical test work advanced during 2014 and 2015, the ability to obtain a titanium dioxide (TiO_2) and scandium (Sc) product became apparent. TiO_2 is typically found to be related to the niobium grades with a range of 3:1 to 4:1 found within the core of the deposit. The scandium mineralization does not directly correlate to niobium mineralization but does show a grade increase with increasing niobium at low grades and high grade Scandium (>75 ppm) is also associated with higher grade concentrated distributions of CaO, Mgo, Th, U and Pb and As (see Figure 7-8 and Figure 7-9).



Source: Nordmin, 2019

Figure 7-8: Basic Statistics of Sc Mineralization



Source: Nordmin, 2019

Figure 7-9: Correlation Statistics of Sc with Nb_2O_5, CAO, MGO, U, Th and Pb

7.7.3 Rare Earth Element Mineralization

Within the Elk Creek Carbonatite complex, there are several occurrences of REE mineralization, including the Project. REE mineralization within the Carbonatite occurs within the following minerals:

- Bastnäsite ($[Ce,La,Y]CO_3F$)
- Parisite ($Ca[Ce,La]_2[CO_3]3F_2$)
- Synchysite ($Ca(Ce,La)(CO_3)_2 F$)
- Monazite ($[Ce,La]PO_4$)

Quantum's re-sampling program discovered high grade REE mineralization in EC-93 as noted in the Molycorp drill logs excerpt:

> *"Barite beforsite is the predominant lithology from 149.4 to 304.8 m. It contains xenoliths of syenite, older mafic rocks, and apatite beforsite I, and is intruded by younger mafic rocks. Intervals 33 m (100 ft) long contain 2.13% to 2.75% LnO from 149.4 to 274.3 ft. An interval 18.3 m long at 179.8 to 198.1 ft contains 3.89% LnO. The highest grade mineralization intercepted was 3 m at 4.72% LnO at 155.4 to 158.5 m. Lanthanide minerals occur as radial patches and random aggregates of needles, irregular patches and vein-like aggregates. The aggregates occur with and without quartz. The aggregates appear as light-gray patches in reddish-brown, hematite-altered beforsite. Although individual lanthanide mineral grains are in the micrometre size range, aggregates of lanthanide minerals range from 0.23 to 8 mm. in maximum dimension. Monazite and bastnäsite have been identified in the aggregates, and EDX spectra show Ce > La."*

It should be noted that Molycorp term's LnO or rare-earth oxides (REO) incorporates lanthanum, cerium and neodymium along with the other 12 rare earth elements. Present day nomenclature for REE is shown in Table 7-3.

Table 7-3: List of Elements and Oxides Associated REE Mineralization

Element	Element Acronym	Compound	Common Oxides
Associated Elements and Oxides			
Niobium	Nb	Nb_2O_5	
Light Rare Earth Metals and Oxides (LREO)			
Lanthanum	La	La_20_3	
Cerium	Ce	Ce_20_3	
Praseodymium	Pr	Pr_20_3	
Neodymium	Nd	Nd_20_3	
Samarium	Sm	Sm_20_3	
Heavy Rare Earth Metals and Oxides (HREO)			Total REEs
Europium	Eu	Eu_20_3	
Gadolinium	Gd	Gd_20_3	
Terbium	Tb	Tb_20_3	
Dysprosium	Dy	Dy_20_3	
Holmium	Ho	Ho_20_3	
Erbium	Er	Er_20_3	
Thulium	Tm	Tm_20_3	
Ytterbium	Yb	Yb_20_3	
Lutetium	Lu	Lu_20_3	
Yttrium	Y	Y_20_3	

Source: SRK, 2017

8. DEPOSIT TYPES

The Project is hosted within the Elk Creek Carbonatite. By definition, a carbonatite is an igneous rock body with greater than 50% modal carbonate minerals, mainly in the form of calcite, dolomite, ankerite, or sodium- and potassium-bearing carbonates. Carbonatites commonly occur as intrusive bodies, such as isolated sills, dykes, or plugs, although rarely occur as extrusive rocks. Many carbonatites are associated with alkali silicate rocks (for example, syenite, nepheline syenite, ijolite, urtite, pyroxenite, etc.). Carbonatites are usually surrounded by an aureole of metasomatically altered rocks called fenites. Carbonatite-associated deposits can be classified as magmatic or metasomatic types (Richardson and Birkett, 1996).

Carbonatites have been classified based on chemical classification into four classes (Woolley and Kempe, 1989; Wyllie and Lee, 1998), and further subdivided based on mineralogical and textural characteristics:

- Calcio-carbonatite coarse-grained: *sövite*, and finer-grained: *alvikite*
- Magnesio-carbonatite dolomite-rich: *beforsite*, and ankerite-rich: *rauhaugite*
- Ferro-carbonatite (iron-rich carbonates)
- Natro-carbonatite (sodium-potassium-calcium carbonates)

The use of a chemical classification of carbonatites should be used with caution when replacement, or metasomatic, processes have altered the primary composition of the carbonatite rock (Mitchell, 2005).

The majority of carbonatite deposits are located within stable, intra-plate crustal units, although some are linked with orogenic activity or plate separation. It is also important to note that carbonatites tend to occur in clusters, and in many places, there has been a repetition of activity over time (Woolley, 1989).

Worldwide, carbonatite deposits are mined for niobium, REE, iron, copper, phosphate (apatite), vermiculite and fluorite; with barite, zircon/baddeleyite, tantalum and uranium as common by-products (Richardson and Birkett, 1996).

9. EXPLORATION

The carbonatite complex is a 6 km to 8 km diameter, alkaline intrusive complex that is buried under approximately 200 m of Pennsylvanian marine clastic sedimentary rocks in southeastern Nebraska. The carbonatite complex is composed of several lithologies. Apatite dolomite is the volumetrically dominant lithology, followed by undifferentiated mafic rocks, syenite, dolomite breccia, barite, and a small body of magnetite dolomite. The magnetic dolomite is the primary host of the niobium mineralization (see Figure 9-1).

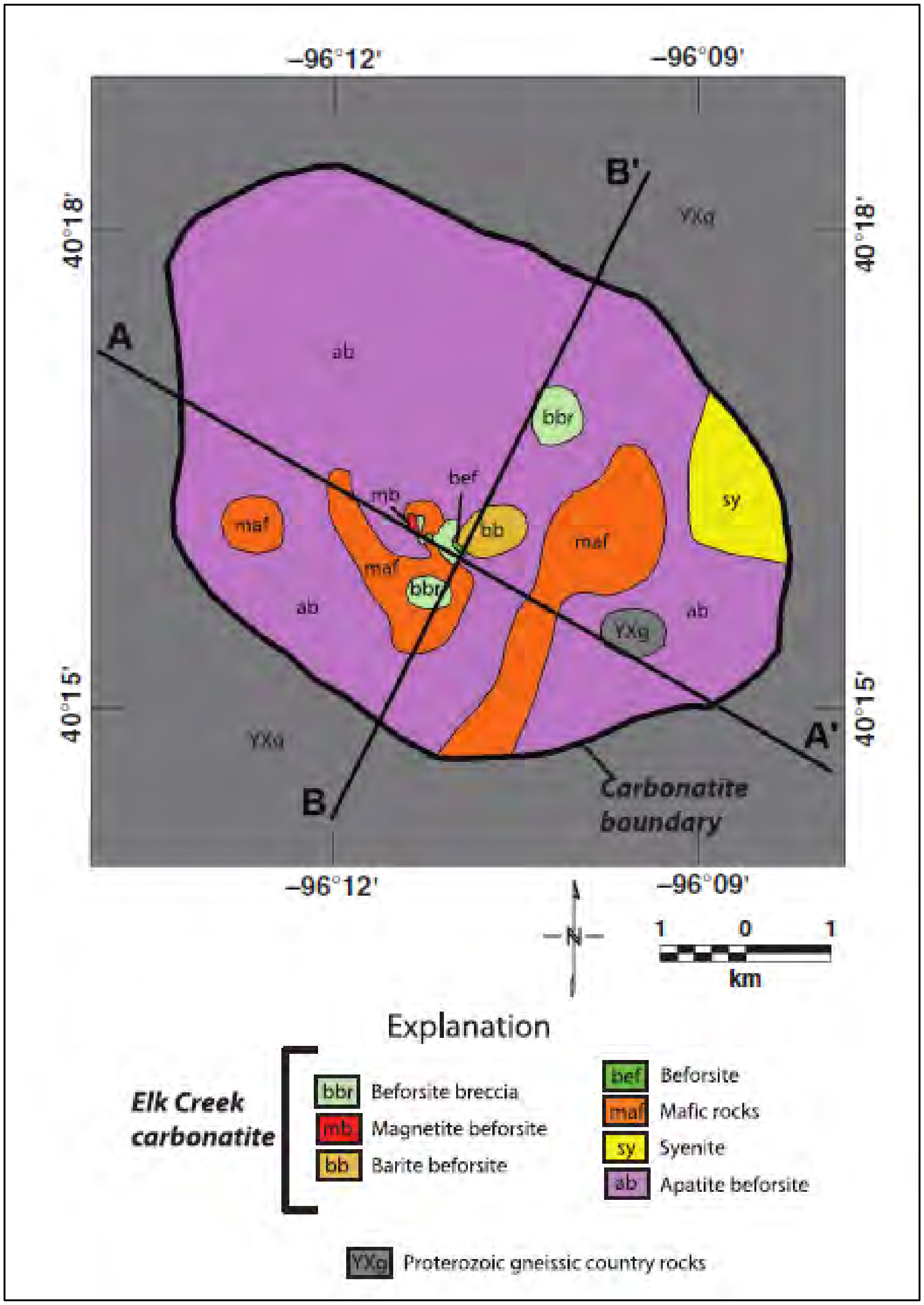
9.1 Significant Results and Interpretation of the Recent Geophysics Interpretation

The geophysical information that was prepared and published by Benjamin Drenth in September 2014 discusses the finding that the AGG data contains many short-wavelength anomalies, and anomalies of interest are often obscured. The measured Gzz anomalies (see Figure 9-2) may be considered as the sum of the effects of crystalline basement rocks (including the carbonatite and surrounding Precambrian rocks), Pennsylvanian sedimentary rocks, and possibly noise. The spectral characteristics of these effects are sufficiently different that the effects of the crystalline rocks may be isolated using matched filtering (Syberg, 1972).

The Gzz data are shown in Figure 9-2 with the short wavelengths unrelated to the crystalline basement rocks removed. The locations of density contrasts, such as those formed by contacts and faults, can be mapped using the HGM of the gravity field (Cordell, 1979; Cordell and Grauch, 1985). This is based on the principle that gravity gradients will reach maximum values over near vertical density contrasts. The HGM of the gravity field is calculated using match filtered horizontal tensor components (not shown) measured during the AGG survey, as opposed to being calculated from the gravity field.

The carbonatite does not produce a notable RTP aeromagnetic anomaly (see Figure 9-3) because most of the carbonatite volume is weakly magnetized. Strong, complex aeromagnetic highs lie over the central area (see Figure 9-3), where two strongly magnetized lithologies are present: mafic rocks and magnetite beforsite. The mafic rocks are far more voluminous, suggesting that they are the primary source for the magnetic anomalies. A large aeromagnetic high in the vicinity of the known location of MB suggests that it may also be an important source despite its small (known) volume.

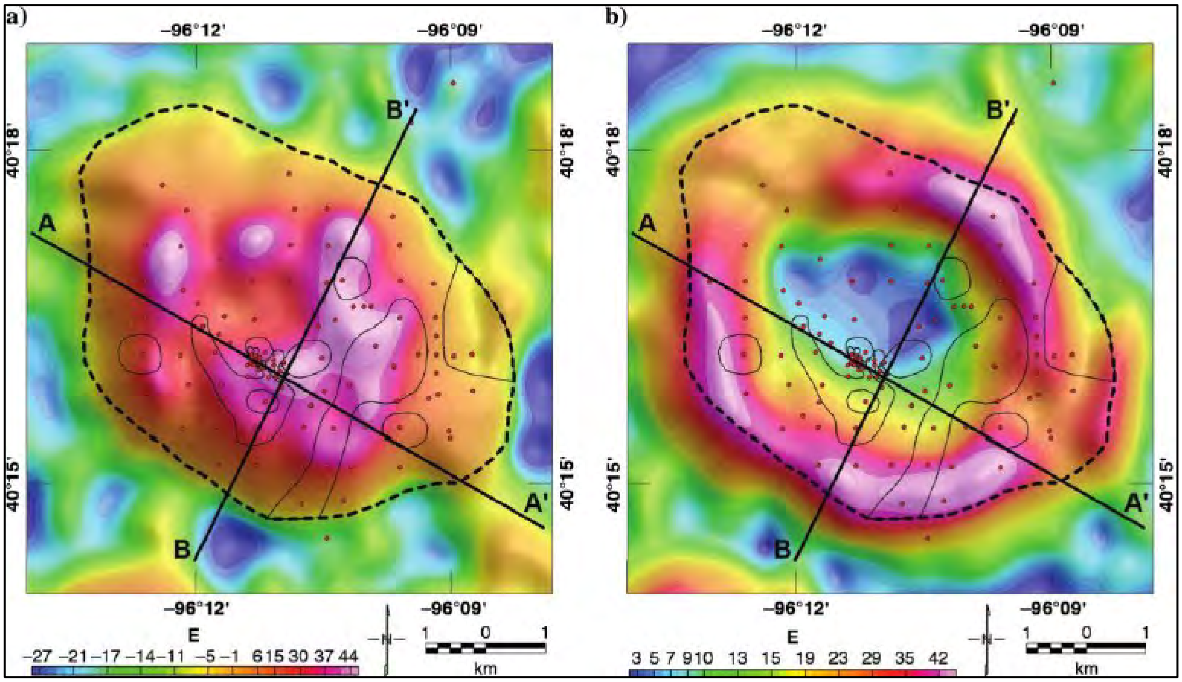
Nordmin considers the exploration programs completed at the Elk Creek Deposit to date to be appropriate for the style of mineralization located within the carbonatite.



Source: Benjamin Drenth, September 2014

Figure 9-1: Geology of the Elk Creek Carbonatite as Expressed in Drill Holes at an Elevation of 120 m Above Sea Level (Roughly 230 m Below Ground Surface)

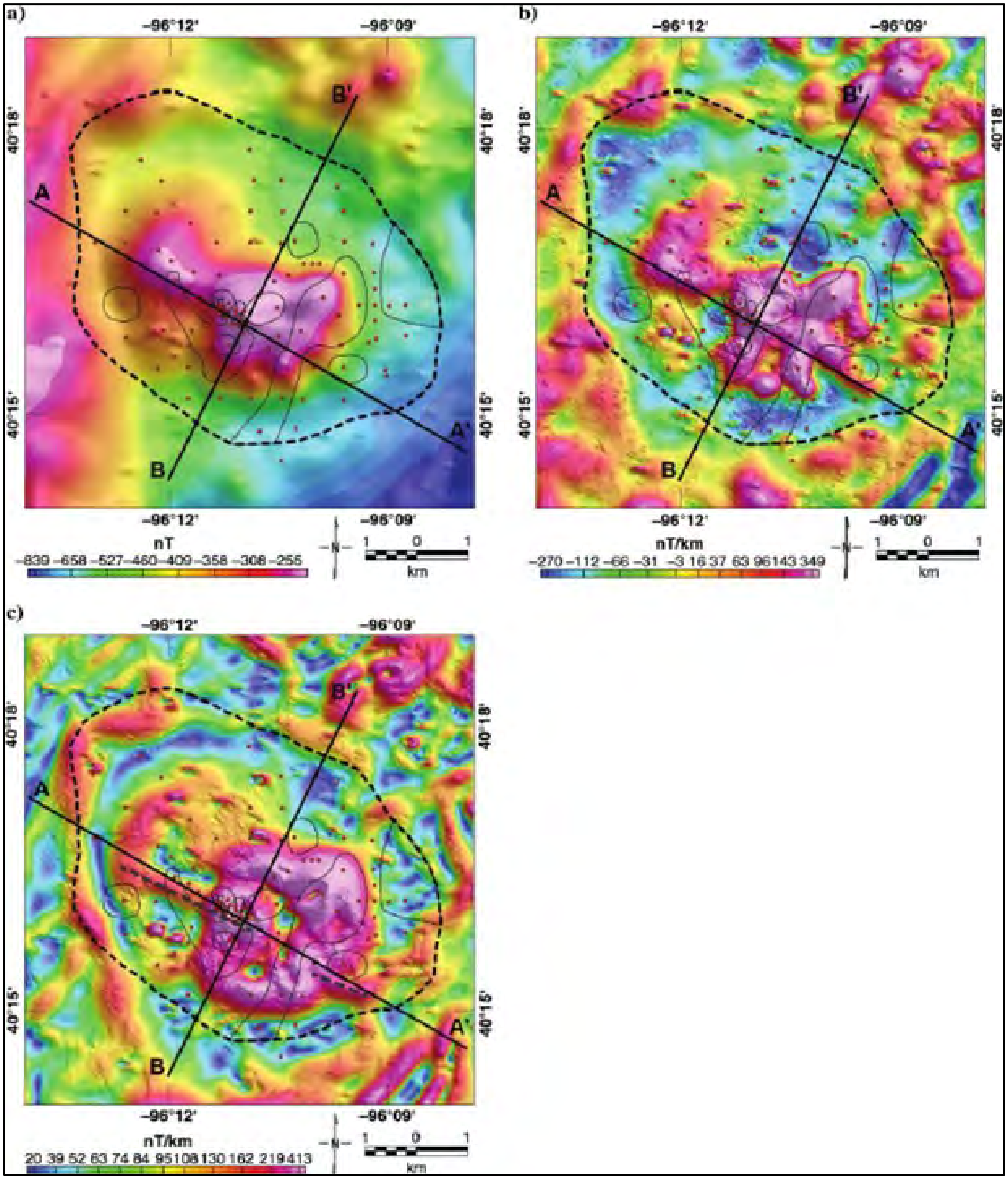
In Figure 9-2, the red dots indicate the drill hole collar locations. In image a) the Gzz data is filtered to remove effects of noise and overlying sedimentary rocks, and in image b) the horizontal gradient magnitude (HGM) of the gravity field is calculated from match filtered tensor geometric data.



Source: Benjamin Drenth September 2014

Figure 9-2: Filtered AGG Data

In Figure 9-3, the red dots indicate the drill hole collar locations. Image a) is reduced to pole total field magnetic anomalies, image b) displays the vertical derivative of reduced to pole total field anomalies and image c) demonstrates the HGM of reduced to pole total field anomalies.



Source: Benjamin Drenth September 2014

Figure 9-3: Aeromagnetic Data

9.2 Quantum, 2010-2011

9.2.1 Data Compilation, 2010-2011

During 2010, the Company contracted Dahrouge to undertake a compilation of all Molycorp hard copy data and digitize all paper files, including drill logs and accompanying drill core geological reports, internal memos and other historical reports.

The historical drill core logs feature almost all the 106 Molycorp drill holes, and four (out of five) Cominco American drill holes. Eight historic Molycorp drill logs were not available in the historical database.

The information gathered by Dahrouge has been compiled into a central database (or Elk Creek Database) using CAE Mining Fusion software.

9.2.2 Quantum Re-Sampling Program, 2010

Commencing in November 2010, the Company contracted Dahrouge to undertake a re-sampling of the historical drill core pulps as part of a QA/QC program to ascertain the reliability of the historic drill core assay results and to obtain a more detailed analysis of the REE content of the samples. The samples were re-analyzed separately by XRF. The Nb_2O_5 assay results were validated and incorporated into the Project database.

SRK has reviewed the results of the program and confirms that it has followed current industry standards in the preparation and correlation of the database.

9.3 Quantum, 2011-2012

9.3.1 Airborne Gravity and Magnetic Geophysical Survey, 2011

In April 2011, Quantum commissioned Fugro of Ottawa, ON, to conduct high-resolution FALCONTM airborne gravity gradiometer (gD) and Total Magnetic Intensity (TMI) geophysical surveys.

10. DRILLING

10.1 Type and Extent

Mineral resource definition drilling at the Project was conducted in three phases. The first was during the 1970s and 1980s by Molycorp, the second in 2011 by Quantum, and the third and latest program in 2014 by NioCorp. To date, 129 diamond core holes have been completed for a total of 64,981 m (see Figure 10-1). All drilling has been completed using a combination of Tricone, Reverse Circulation (RC) or Diamond Drilling (DDH) core in the upper portion of the hole within the Pennsylvanian sediments. All drilling within the carbonatite has been completed using diamond coring methods.

To date, local labour has been used by drilling contractors when preparing the drill hole pads. All drilling has been completed using standardized procedures which are in line with international standards of best practice. The drilling completed by Molycorp was completed by using company-owned equipment and sampling procedures. The drilling companies used by the Company during the 2011 and 2014 drilling programs are detailed below:

- 2011: Black Rock Drilling, LLC (BRD Personnel and Leasing Corp.), 17525 E Euclid Ave, Spokane Valley, WA 99216;
- 2014: Envirotech Drilling LLC, 900 East 4th Street, Winnemucca, NV 89445
- 2014: West-Core Drilling, LLC, 561 W Main Elko, NV 89801 USA; and
- 2014: Idea Drilling, 1997 9th Avenue North, Virginia, MN 55792.

The drilling has been completed using conventional techniques, using experienced drilling contractors. A portion of the 2014 drill holes used RC drilling within the Pennsylvanian sediments, to increase the efficiency in drilling through the cover material, within areas of strong geological confidence.

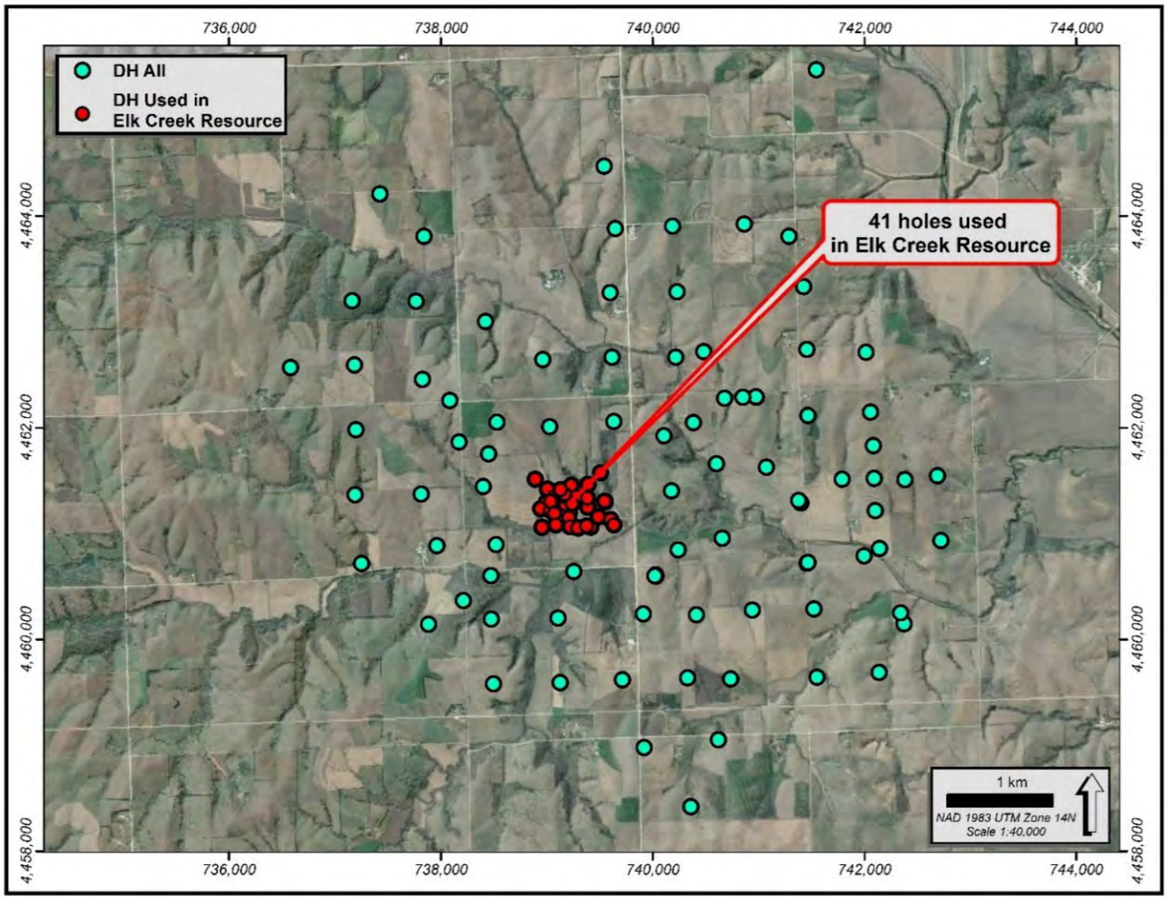
The following sections provide a summary of the resource drilling completed by Molycorp, Quantum and NioCorp (as shown in Table 10-1).

Table 10-1: Summary of Drilling Database within the Geological Complex

Year	Company	Number of Holes	Average Depth (m) Sum Length	(m)
1970-1980	Molycorp	106	434.7	46,078.3
2011	Quantum	5	684.0	3,419.9
2014	NioCorp	18	845.4	15,482.8
Subtotal		**129**	**501.7**	**64,981.0**

Source: Nordmin, 2019

During 2015 five holes, for a total length 3,353.1 m, were drilled and were not included in the Mineral Resource Estimate dated April 28, 2015. This drilling was completed for the hydrogeological and geotechnical studies. The drilling has been completed by Idea Drilling and Envirotech Drilling LLC. A sampling of these holes has not been completed to date, and therefore, these holes have not been considered in the Mineral Resource and are excluded from Table 10-1. Inclusive of these holes, the total drilling on the Project is 134 holes for 68,334.1 m.



Source: Nordmin, 2019

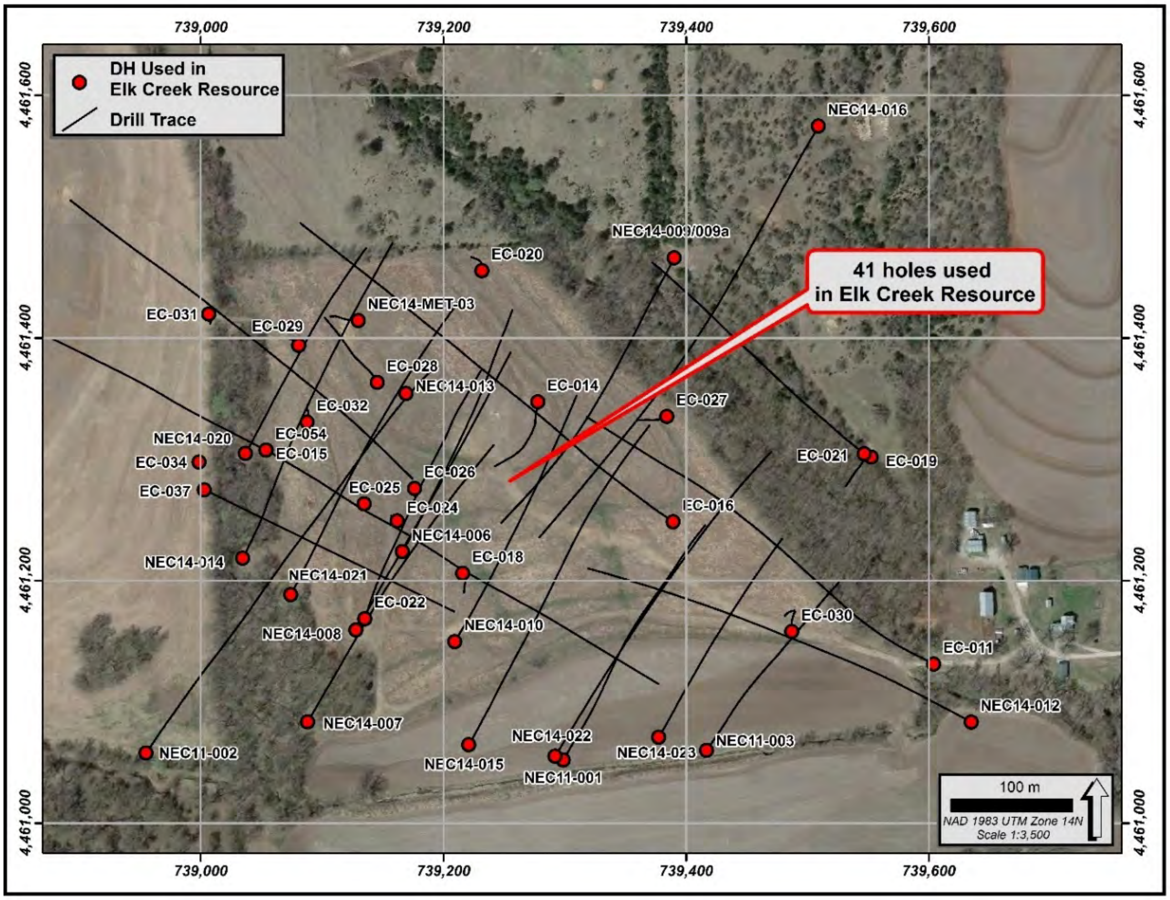
Figure 10-1: Drill Hole Location Map of all Drilling Versus the Topographic Contour

Not all the drill holes within the Project were used in the 2019 Mineral Resource Estimation, as many do not intersect the Nb_2O_5 anomaly and are located at a significant distance away from the Deposit. There are a total of 129 drill holes within the Project, of these 48 drill holes are within the Elk Creek Deposit area (see Figure 10-1). Table 10-2 summarizes the drill holes used for the Mineral Resource Estimation.

Table 10-2: Summary of Drilling Database within Elk Creek Deposit Area

Year	Company	Number of Holes	Average Depth (m) Sum Length	(m)
1970-1980	Molycorp	27	596.6	16,108.2
2011	Quantum	3	772.6	2,317.7
2014	NioCorp	18	845.4	15,482.8
Subtotal		**48**	**700.9**	**33,908.7**

Source: Nordmin, 2019



Source: Nordmin, 2019

Figure 10-2: Location of Drill Holes Used in the 2019 Mineral Resource

10.2 Molycorp, 1973-1986

Between 1973 and 1986, Molycorp completed a regional scale drill program over an approximately 7 km by 7 km gravity anomaly that included the Elk Creek Deposit. The total program consisted of 106 drill holes for a total of approximately 46,078 m. Outside the Elk Creek Deposit area, the regional drill program was conducted on a regular grid of 610 m by 610 m (2,000 ft by 2,000 ft) with some closely spaced holes in selected areas within the gravity anomaly (see Figure 10-3).

Included in this total, approximately 27 holes for 16,108 m were drilled over the deposit. Drilling orientations varied considerably.

The Molycorp drill hole locations centred over the Elk Creek Deposit are presented in Figure 10-1 (shown in red).

10.3 Quantum, 2011

In April 2011, Quantum conducted a preliminary drill program (three holes) on the Elk Creek Deposit along with two initial holes focused on REE enrichment targets (see Figure 10-3). These holes have been excluded from the current Mineral Resource Estimation, as they do not intersect the Nb_2O_5 anomaly and are located to the east. The objectives of the drill program over the Project were to verify the presence of higher-grade niobium mineralization at depth and to infill drill the known niobium deposit to upgrade the resource category of the previous resource estimate and expand the known resource. The drill program was also established to collect sufficient sample material for metallurgical characterization and process development studies of the niobium mineralization.

The 2011 program consisted of five inclined drill holes, totalling 3,420 m of NQ size diameter core. Inclusive of this total, three drill holes, totalling 2,318 m were drilled into the known Elk Creek Deposit. The summary of the 2011 drill program is listed in Table 10-3.

Table 10-3: Summary of the 2011 Drill Program

Drill Hole	UTM Easting	UTM Northing	Elevation (m)	Depth (m)	Bearing (°)	Dip (°)
NEC11-001	739299.0	4461052.0	341.49	900.38	28.1	-72.0
NEC11-002	738955.0	4461058.0	343.88	908.61	33.5	-61.0
NEC11-003	739417.0	4461060.0	340.79	508.71	34.3	-55.9
Outside Elk Creek Deposit; REE Exploration Targets						
NEC11-004	741997.0	4460790.0	333.65	465.73	80.7	-55.6
NEC11-005	740604.0	4461660.0	337.48	636.42	95.7	-56.0
Total				**3,419.85**		

Source: Tetra Tech, 2012

DDH NEC11-001 targeted the eastern portion of the deposit below the historical drill hole EC-11 and between vertical holes EC-27 and EC-30. DDH NEC11-002 was drilled into the northwestern portion of the deposit. DDH NEC11-003 was drilled into the southeastern portion of the deposit. Drill holes NEC11-004 and 005 drilled into regional REE targets and are not subject to this Technical Report and have been excluded from the Mineral Resource Estimate.

The Quantum 2011 drill hole locations centred over the Elk Creek Deposit are presented in Figure 10-1 (shown in green).

Results from the 2011 drilling program provided additional information on areas of the deposit at depth where limited information was previously available. The drill holes confirmed the high grade potential of the niobium mineralization, as indicated by the previous drilling completed by Molycorp.

10.4 NioCorp 2014 Program

The 2014 drilling campaign was conducted using a three-phase program. The campaign was designed to increase the confidence with bringing material up from the Inferred to Indicated resource category, based on the 2012 Mineral Resource Estimate. The program was initially designed for 14 drill holes approximately 12,150 m (referenced in a press release on April 29, 2014) but was expanded to 19 drill holes, to give a total of 15,482.8 m (see Figure 10-3).

The drilling was conducted by both West-Core Drilling and Idea Drilling, both private contractors.

A location map of the drill holes used in the resource is found in Figure 10-2. Except for two drill holes, the drill holes were designed to drill perpendicular to the strike of the ore body, trending both southwest or northeast. Two drill holes, NEC14-011 and NEC14-012, were orientated southeast and northwest, respectively. Locations and survey information of the 2014 drill program can be found at Table 10-4.

Three of the 19 holes drilled were for metallurgical characterization development studies. Two of these drill holes (NEC14-MET-01 and NEC14-MET-02) were not sampled. The third drill hole (NET14-

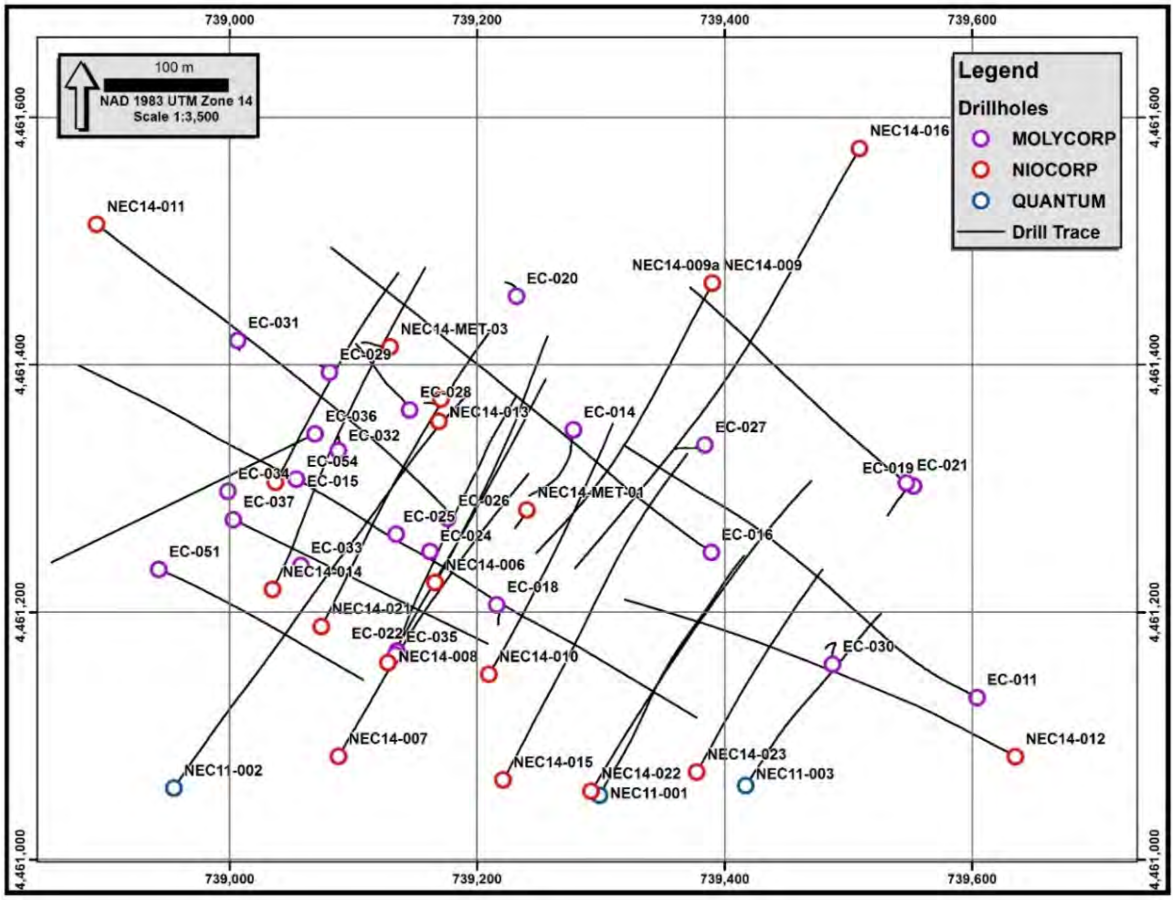
MET-03) was quartered, with one quarter being sent for assays and remainder sent for metallurgical purposes. No further drill holes were drilled since the February 20, 2015, Mineral Resource Estimate.

The NioCorp 2014 drill hole locations (shown in Table 10-4) are presented in Figure 10-2 (shown in red).

Table 10-4: NioCorp 2014 Drill Hole Location

BHID	XCOLLAR	YCOLLAR	ZCOLLAR	LENGTH	AZIMUTH	INCLINATION	COMMENTS
NEC14-006	739166.2	4461224.0	352.0	772.7	29.9	-70.8	
NEC14-007	739088.2	4461083.5	344.8	907.4	29.4	-70.6	
NEC14-008	739128.1	4461159.3	351.2	886.0	30.8	-69.8	
NEC14-009	739390.2	4461466.2	349.3	751.3	208.7	-70.3	
NEC14-009a	739390.2	4461466.2	349.3	411.5	208.7	-70.3	Wedge from 485.41 m
NEC14-010	739209.5	4461149.8	347.8	796.1	30.0	-73.1	
NEC14-011	738892.5	4461513.6	359.7	900.4	125.8	-65.3	
NEC14-012	739635.1	4461083.4	339.9	843.2	299.8	-68.0	
NEC14-013	739169.3	4461354.3	355.2	880.3	149.4	-89.2	
NEC14-014	739034.8	4461218.6	346.1	901.0	28.6	-77.6	
NEC14-015	739221.0	4461064.7	342.4	827.8	29.1	-72.4	
NEC14-016	739509.1	4461574.7	354.7	913.8	210.5	-60.0	
NEC14-020	739037.1	4461305.0	348.4	587.6	28.2	-70.6	
NEC14-021	739074.3	4461188.5	347.1	865.0	29.5	-69.2	
NEC14-022	739292.2	4461055.3	340.3	950.4	31.3	-68.4	
NEC14-023	739377.6	4461071.0	341.5	615.1	30.2	-71.1	
NEC14-MET-01	739240.4	4461282.7	352.8	894.7	302.6	-89.6	
NEC14-MET-02	739171.1	4461372.4	355.8	865.0	88.1	-89.6	
NEC14-MET-03	739129.9	4461414.5	355.4	913.3	249.8	-89.8	
SUBTOTAL				**15,482.8**			

* Note that NEC14-009a started at a depth of 485.51 m, total metres for 2014 drilling program is 19 holes



Source: Nordmin, 2019

Figure 10-3: Elk Creek Drill Hole Location Map by Company

10.5 Procedures (NioCorp 2014 Program)

Detailed descriptions of Molycorp's drilling, sample procedures, analyses and security have not been documented and reviewed by Nordmin. Given Molycorp's position as a leader in the rare earth industry at the time, it is likely Molycorp applied industry best practice for the time period. The 2011 drilling campaign was managed by Dahrouge and SRK under the same quality and procedures used in the current study. The 2014 drilling program includes a quality control program to ensure the results can be used to verify earlier drilling results and add confidence to the overall understanding of the deposit.

For the 2014 drilling program planned drill hole collars were initially located using a handheld GarminTM Global Positioning System (GPS) and marked with wooden stakes. A tracked excavator was used to construct the drill pad and collars were then relocated using the GPS with wooden stakes after pad construction. A geological compass and an Azimuth Pointing System (APS) were used to sight in the drill to the planned azimuth and inclination.

The 2014 core drilling was conducted by both West-Core Drilling and Idea Drilling, both private contractors. West-Core used both an AVD R40 track-mounted core drill and an Atlas Copco CS-14 track-mounted core drill, while Idea used an Atlas Copco CT-20 truck-mounted core drill. Overburden was cased in all drill holes to depths ranging from 18 m to 37 m. The Pennsylvanian limestones and mudstones overlying the target carbonatite were drilled PQ-sized core and HQ-sized core for drill holes NEC14-020 to NEC14-023. The carbonatite was drilled with the HQ-sized core, except for the three metallurgical holes (NEC14-MET-01, NEC14-MET-02 and NEC-14MET-03), which were drilled entirely using PQ-sized core. Core size reduction took place just beneath the Pennsylvanian-carbonatite contact at depths ranging from 206 m to 238 m. The core drilling rigs operated 24 hours/day and 7 days/week, with the typical progress of 40 m/day.

During the drilling operation, the core was retrieved from the core barrel and laid sequentially into wooden core boxes by the drilling contractor. Interval blocks were placed at all run breaks. Once the box was full, the ends and top of the box were labelled with drill hole identification and the sequential box number. Upon completing a box, it was stacked on a pallet or a truck bed at the drill rig. At the end of each drilling shift, the boxes of the core were transported by the drilling contractor in a pickup truck to the NioCorp field office. At this point, the core was in the custody of Dahrouge Geological Consulting Ltd. (Dahrouge). Eight of the 2014 drill holes had piezometers installed in them after drilling was complete. For these drill holes, surface completion consisted of surface casing capped with a locking steel cover, a 1.2 m^2 cement pad around the surface casing and a steel nameplate attached to the casing (see Figure 10-4).

Surface completion for the drill holes that did not have piezometers installed consisted of a steel marker post and attached nameplate. All nameplates include drill hole number, total depth and orientation. Abandonment of the drill holes consisted of cementing from total depth to surface in the non-piezometer drill holes and from total depth to the bottom of the piezometers in the other drill holes with piezometer installations (see Figure 10-5).

Source: Nordmin, 2019

Figure 10-4: Collar Location of NEX 14-009

Source: Nordmin, 2019

Figure 10-5: Collar Location of NEX 14-MET-01

10.5.1 Collar Surveys

All drill hole collars were initially surveyed before drilling using a handheld GPS. On completion of the drill hole an external contractor ESP INC. (Engineering/Surveying/Planning), based in Lincoln, Nebraska, was used to provide a detailed survey of the collar location using a Sokkia GS2700 IS GPS, which has 10 mm horizontal and 20 mm vertical accuracy. Data was provided to SRK in digital format in NAD 1983 UTM Zone 14N grid coordinates.

The location of 24 of the 29 Molycorp drill collars was re-excavated as required to confirm the collar coordinates from 2011 over Elk Creek, by CES Group P.A. Engineers & Surveyors (CES), based in Kansas City, Missouri. All collars were surveyed using the same UTM coordinate system.

10.5.2 Downhole Surveys

Initial collar surveys of dip and azimuth were taken using compass measurements for all holes (RC and DD). Downhole surveying was undertaken on historical Molycorp holes drilled into below the Pennsylvanian sediments at an interval of 30.48 m (100 ft).

The 2011 drilling program was surveyed at 3.05 m (10 ft) intervals, based on the drilling rod lengths used at the time. All drill holes were surveyed immediately after completion of drilling. Downhole deviations, subsurface azimuth and dip, were mapped using a Devico DeviFlex survey tool, which is a nonmagnetic, electronic, multi-shot tool. The DeviFlex tool consists of two independent measuring systems, while three accelerometers and four strain gauges used to calculate inclination and change in azimuth.

The DeviFlex tool communicates with a PDA, and the survey results can be viewed on the PDA immediately after completion of the survey. Dahrouge geologists checked the downloaded data for possible errors, and inconsistencies and some readings were removed for quality control purposes.

The DeviFlex output contains a column for possible tool movement during surveying. In the event, there was a potential tool movement, that specific reading was removed from the dataset.

The DeviFlex tool records changes in azimuth, as opposed to absolute azimuth measurements. Because of this, an initial (surface) survey azimuth is required to calibrate the DeviFlex downhole azimuth readings. CES surveyed all initial drill hole azimuths by surveying the azimuth of the drill rods extruding from the ground during drilling. These initial azimuth readings were used to calibrate the DeviFlex downhole change in azimuth readings and calculate absolute azimuth measurements.

The 2014 drilling program was surveyed at 6.1 m (20 ft) intervals using a Reflex GYRO survey tool. Dahrouge geologists operated the GYRO and collected the surveys. Downhole deviations, subsurface azimuth and dip, were mapped with the GYRO, which utilizes a digital MEMS-gyro non-magnetic assemblage. The Gyro tool was used to mitigate magnetic deviation caused by metal equipment, or naturally occurring minerals such as magnetite and pyrrhotite which occur in the deposit.

These surveys were synchronized electronically with a receiver at the surface, and recordings were collected every 30 seconds after the tool equilibrated. The Reflex GYRO has an integrated Azimuth Pointing System (APS) that is used to orientate the True North azimuth, a GPS position and degree of inclination. Downhole surveys were completed through the drill rods, and location data points were collected every 6.1 m (20 ft).

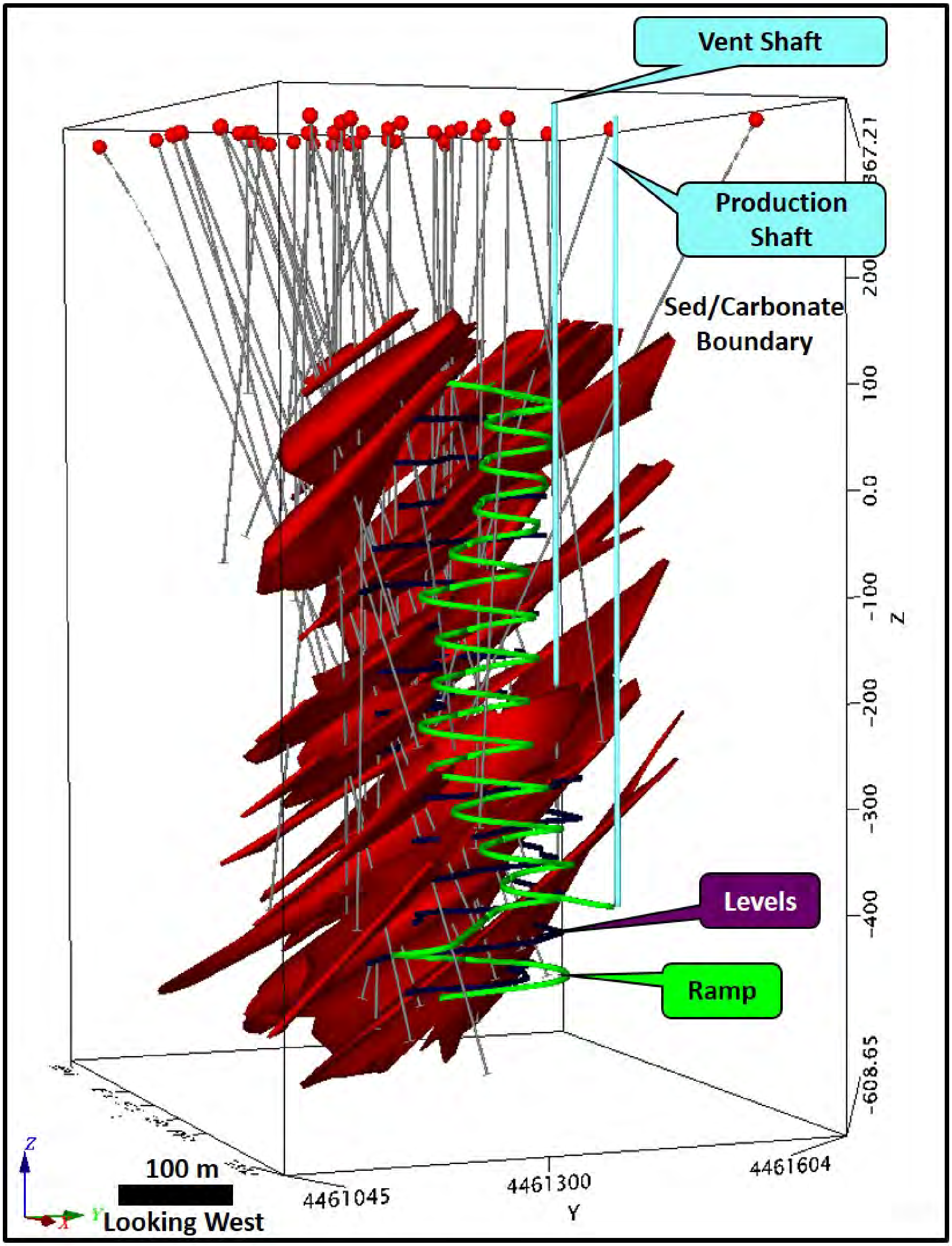
Nordmin considers the methods used for the downhole surveying during the 2011 and 2014 campaigns to be in line with industry standards. Given the longhole lengths of over 700 m, the Company has used suitable techniques to provide a continuous (ranging from 3 to 6 m) measure of the drill hole trace from the base of the hole. The use of a Gyro has avoided any potential issues due to the magnetic nature of the rocks. The confidence in the drill hole location of the Molycorp drilling is considered slightly lower due to their historic nature and the wider measurement spacing and equipment used to complete the surveys.

Overall Nordmin considers the level of confidence in the downhole surveys to be sufficient and adequate for the use in Mineral Resource Estimation practices.

10.6 Interpretation and Relevant Results

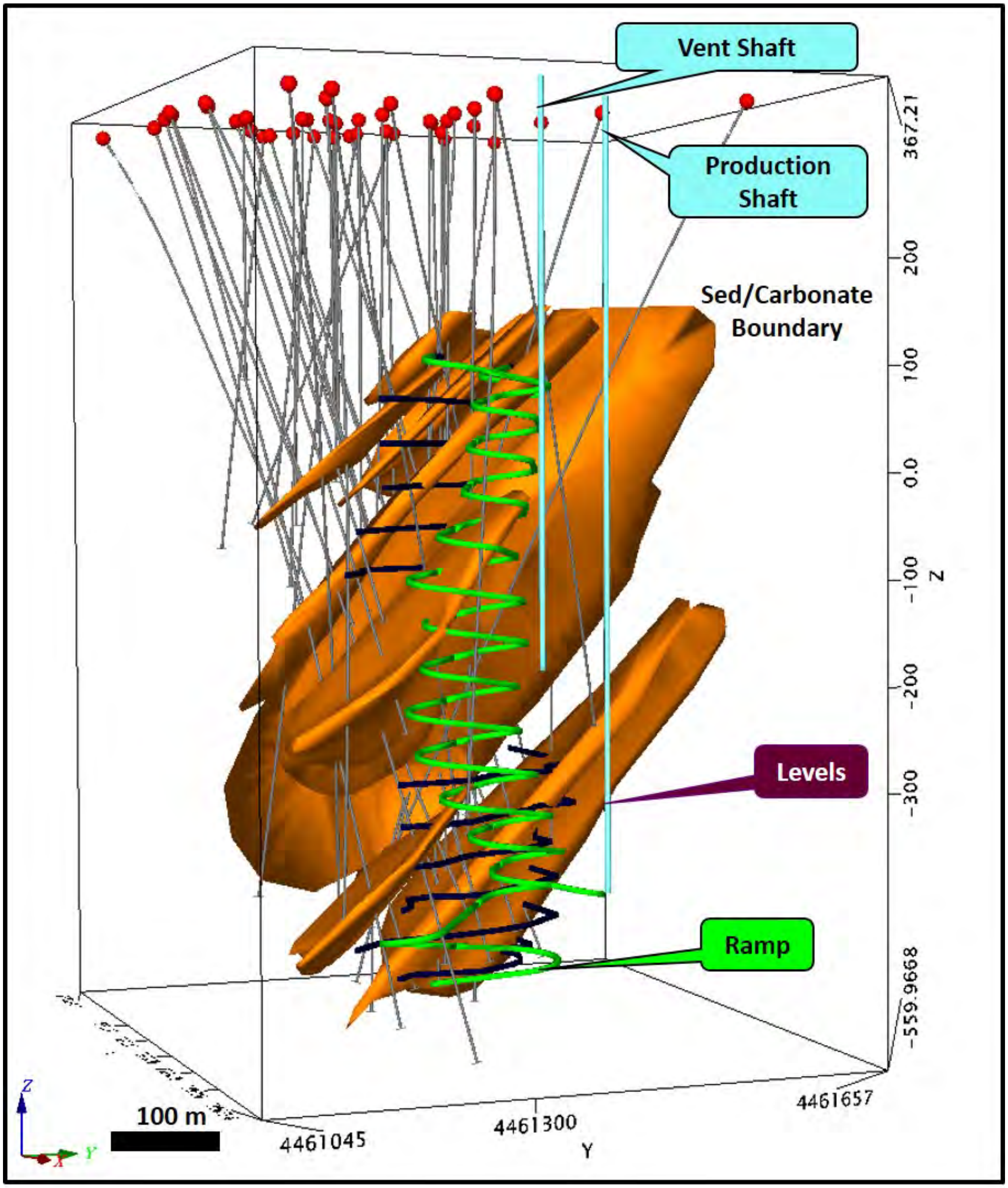
No new drilling has been completed since the previous NI 43-101 Technical Report for the purposes of Mineral Resource definition.

The 2019 updated estimate is based on the results of re-assays of TiO_2 and Sc, and the remodelling of the mineralization that created high grade Nb_2O_5 and TiO_2 structures averaging ~1.0% Nb_2O_5 (see Figure 10-6) and high grade SC structures averaging ~75 ppm Sc (see Figure 10-7) that are surrounded by a lower grade Nb, Sc, and TiO_2 structure.



Source: Nordmin, 2019

Figure 10-6: High Grade Nb_2O_5 and Tio_2 Mineralized Structures



Source: Nordmin, 2019

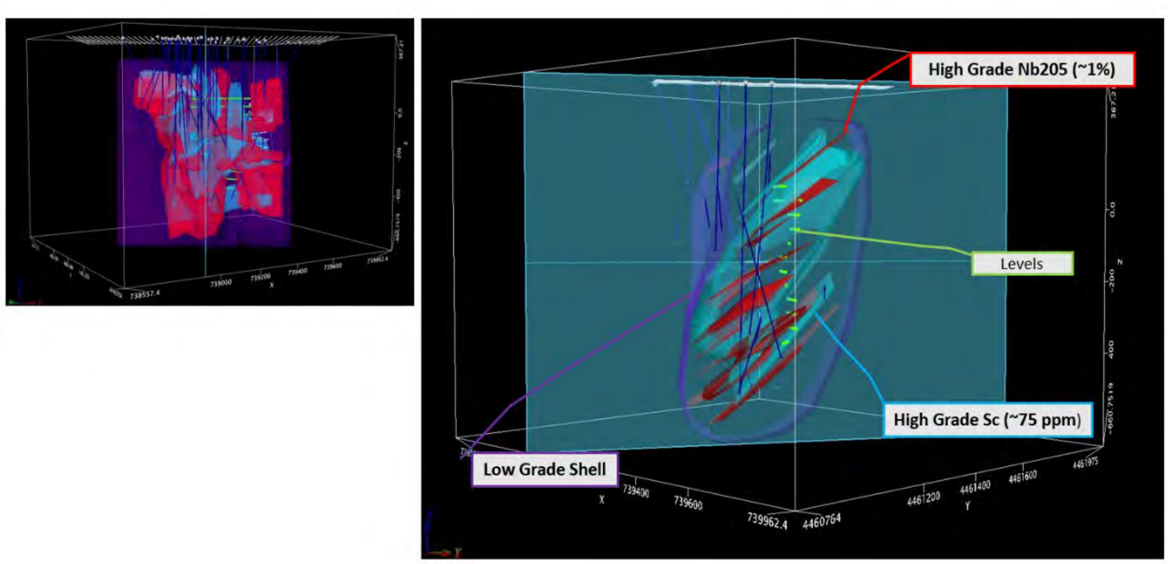
Figure 10-7: High Grade Sc Mineralized Structures

The only significant change is in the database due to the re-assay of 766 samples (inclusive of 98 quality control samples), from the edge of the defined mineralization limits which had not previously been assayed for TiO_2 or Sc.

The drilling was conducted by reputable contractors using industry standard techniques and procedures. This work has confirmed the presence of niobium, titanium and scandium mineralization hosted in dolomite-carbonatite and lamprophyre rocks. In general, the Lamprophyre is niobium depleted, but contacts between lamprophyre and carbonatite may be enriched.

The drill holes within the Deposit and Mineral Resource area have variable drill spacing between 25 m and 225m. The major drilling direction used by NioCorp has been towards the northeast (see Figure 10-8). Two sets of scissor holes were drilled to the southwest on separate drilling lines within the central portion of the deposit, to confirm that there is no directional bias in the selected drill hole orientation.

The majority of the holes have inclinations in the order of 60° to 70°. The use of scissor holes has confirmed the sub-vertical nature of the southwest contact (see Figure 10-8).



Source: Nordmin, 2019

Figure 10-8: Typical Cross-Sections Looking Northwest Showing NioCorp Holes Drilled to the Northeast and Southwest, Defining the High grade Nb_2O_5, Tio_2 and Sc, and Low Grade Zones

10.7 Qualified Person's Opinion

It is the Qualified Person's opinion that the drilling and logging procedures put in place by NioCorp and its subcontractors meet acceptable industry standards and that the information can be used for geological and resource modelling.

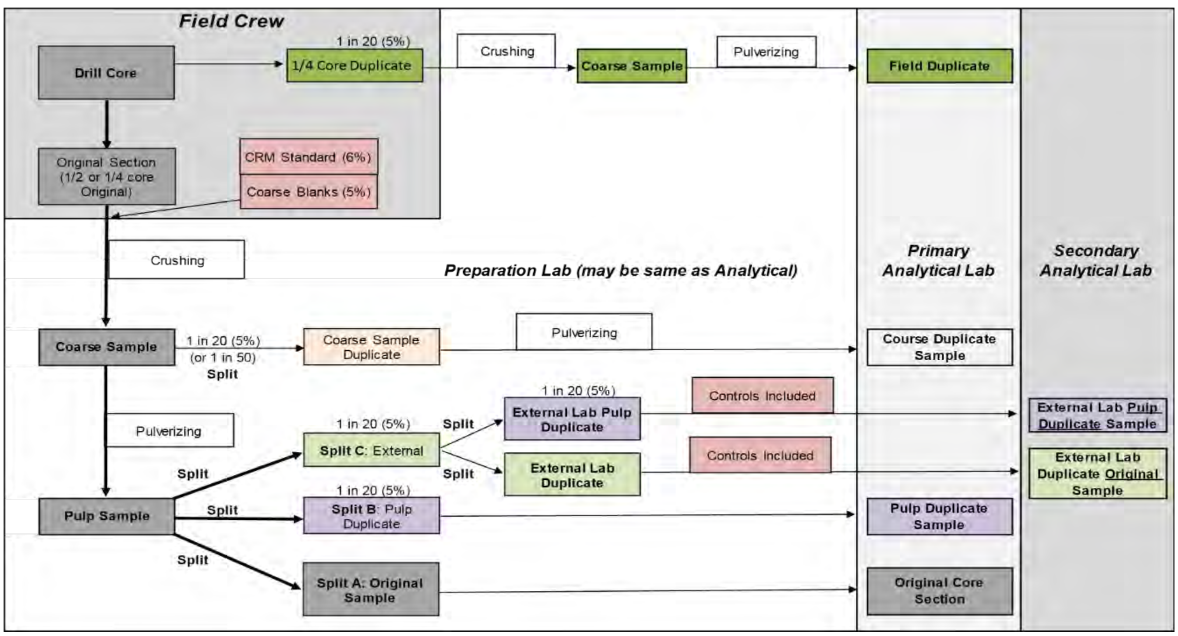
11. SAMPLE PREPARATION, ANALYSES AND SECURITY

The following section summarizes the sampling methodology used by Molycorp and the Company during the historic drilling, the 2011 and the 2014 drill programs.

11.1 Sample Preparation and Security

11.1.1 NioCorp Drilling Program, (2011 - Current)

The sample collection, preparation, and shipment workflow process were strictly monitored to reduce or eliminate the downstream progression of incorrectly identified samples. Dahrouge Geological Consulting managed the drilling QA/QC program, which consisted of the insertion of control samples to monitor each stage of preparation and analysis. These control samples included laboratory-blind certified reference material samples (CRMs), optical-quality quartz blanks, field duplicates, coarse-reject duplicates, pulp duplicates, and external "umpire" lab duplicates. All samples were prepared and analyzed at Activation Laboratories (Actlabs), and select samples were subsequently sent to SGS Labs for a secondary check analysis (see Figure 11-1).



Source: NioCorp, 2019

Figure 11-1: Sample Process Flow Chart

Once the field and control samples were received at Actlabs, they were crushed, pulverized, and split. A pulverization target of 95% passing 200 mesh (-200 mesh) was applied, with a quartz wash between the preparation of each sample. All duplicate sample splits were extracted simultaneously to their parent samples to ensure a sample level of homogenization and handling procedures. The 2011 program did not utilize laboratory blind duplicates.

- Pulps were prepared by Actlabs, which were then subsequently assayed using ICP-MS whole rock analysis, complete elemental packages, and XRF niobium analysis.
- Dahrouge Geological Consulting reviewed the initial results of the ICP-MS whole rock analysis and selected 55 samples were analyzed for FUS-ICP and ICP-MS reporting niobium and whole

rock values. External pulps splits for check analysis were created at the same time as primary pulps.

NioCorp employed rigorous security measures to prevent tampering of the core or samples before and during the transport process. These measures included redundant sample identification, appropriate sample bag closures and the shipment of sample bags inside pails with lids. Nordmin is of the opinion that these measures are consistent with current industry best practices for projects at this scale of exploration.

11.1.1.1 Core Logging Preparation

The core logging, sampling method, and approach were consistent through the 2011 and 2014 drill programs. The core was boxed at the drill site and delivered each day to the project core processing facility where it was logged and split. The diamond drilling programs utilized up to three coring drill rigs which were monitored by two qualified professional geologists, one drill supervisor and an experienced geologist.

Standardized core logging codes and lithology descriptions were created in Datamine's Fusion drill hole database to ensure consistency among logging geologists. A total of twenty-two detailed rock codes were used during the logging, which was subsequently reduced to ten codes under a simplified logging code defined as "MAJOR" in the database (see Table 11-1).

Table 11-1: Summary of Major Rock Codes Used by Dahrouge Geologist

MAJOR	Description
Casing	Drill hole casing
TILL	Till
SEDT	Sediments
CARB	Carbonatite
MCARB	Magnetite Carbonatite
CARB-LAMP	Carbonatite mixed with lamprophyre
MCARB-LAMP	Magnetite Carbonatite mixed with lamprophyre
LAMP	Lamprophyre
MAFIC	Mafic intrusive units
INT	Other intrusive units

Source: Dahrouge, 2015

The sampling procedure used to collect core samples entailed:

- The sampling of the entire carbonatite intersection, including the geologically-logged low grade niobium carbonatite intervals of the footwall or hanging wall, for all holes except NEC14-020 to NEC14-023, where approximately 10 m of the hanging wall was sampled;
- Sample intervals, generally 1 m in length, were marked on the core and recorded in the geological database (Fusion).
- Sample intervals were assigned a unique sample number.
- Specific gravity measurements were performed at approximately 6 m spacing.
- Hand-held Niton-XRF measurements were collected on the core to assist geological and sample divisions.
- Magnetic susceptibility measurements were performed on the core to assist geological and sample divisions.

- Clearly marked sample intervals were split in half by a wet diamond saw.
- Split intervals were cleaned before bagging, and the cutting equipment was regularly cleaned.
- Sampled intervals were placed in durable barcoded sample bags that were clearly labelled and contained back up sample tags within each bag.
- Sample bags containing original core sections and field inserted control samples were barcode-scanned and secured in five-gallon plastic shipping pails.
- Hard copy and digital detailed shipping logs and preparation requests were sent to the primary analytical laboratory.
- Sampled core sections and blind control samples were shipped for analysis in secured pails and transferred using a bonded trucking company.
- Storage of the unsampled half of the core in labelled wooden core boxes at the Project site for reference or further sampling.

The core samples and the core library are securely stored at the project facility work area located in the southeast corner of the Beethe008. This material is stored inside locked metal buildings when the Project is not operating (see Figure 11-2).

Source: Nordmin, 2019

Figure 11-2: Storage Location of Drill Core and Pulps

The drill core within each core box was marked up and split along orientation marks. Cutting was completed using one of three electric-powered, water-cooled diamond-bladed BD 3003E core saws at the Project sample preparation and storage facility. HQ and minor intervals of PQ core were halved for assay. Drill hole NEC14-MET-03, a PQ-sized hole, was quartered with one quarter being assayed, and the remaining core packaged for metallurgical testing.

Infrequent broken or soft sections of the core (typically the iron oxide altered zones) were sampled by the geologists, and an equal sample split was taken from this material. These intervals account for a significantly small portion of the sampled material. Core not used for assaying or metallurgical testing is stored at the project facility work area at the Project site.

Drill core was digitally photographed under natural outdoor or indoor fluorescent lighting before core cutting. All digital photos are of high resolution and stored in a digital archive format. The geological logs included observations of colour, lithology, texture, structure, mineralization, and alteration. All geological information is collected at a sample interval scale and recorded into the Fusion Database, which was the digital core logging and sampling storage software program.

SRK was responsible for the geotechnical logging. Rock quality was determined using the Q-system (Q=(RQD/Jn)*(Jr/Ja)*(Jw/SRF), where RQD = Rock quality designation; Jn = Joint set number; Jr = Roughness of the most unfavorable joint or discontinuity; Ja = Degree of alteration or filling along the weakest joint; Jw = Water inflow; SRF = Stress reduction factor. SRK personnel also recorded hardness and weathering to aid in geotechnical parameters for the future mine design.

Core recovery and RQD were generally competent for the majority of the drill core. Core recovery was recorded in the database and was measured in the field at the drilling rig by the geologist. The borehole name is noted, and the drilling interval was compared to the actual core recovered to back-calculate the recovery. The recovery information was loaded into the sample database.

Nordmin has reviewed the drill core recovery results of some of the holes during the site visit along with various logs. Overall the core recovery within the Nb_2O_5, TiO_2, and Sc-mineralized units had a range of 93%-99% recovery.

11.1.2 Molycorp, 1973-1986 (Historical)

The NQ, as well as some BQ core, was washed, visually logged, and photographed, and was hydraulically split and crushed on site before the samples were sent to the lab. Once crushed, the drill core samples were collected and sent to Molycorp's exploration laboratory at Louviers, Colorado, for niobium and LnO analysis. The analytical methods are described in an internal memo by Sisneros and Yernberg, 1983, where "...Niobium was analyzed by wavelength dispersive XRF on pressed powder pellets, following pulverization to -325 mesh. Molycorp did include a series of quality control methods. Standardization was provided by using a variety of Elk Creek samples, which had been analyzed by alternative methods at other internal Molycorp laboratory facilities. Over the project duration, the number and/or identification of the standards used changed several times. In 1981, the instrumentation changed from a Philips PW1212 to a PW1400." (Sisernos and Yernberg, 1983)

The assay tables from some of the holes (EC-27 and EC-30) indicate a 'tentative test' (XRF) of niobium value from Louviers laboratory and a 'commercial lab test’ (XRF) of niobium values. It is unclear which commercial laboratory conducted these tests, although the 1983 Niobium Analytical Standardization report mentions that the Molycorp exploration department occasionally utilized Bondar-Clegg. Notes on the assay tables indicate that the commercial laboratory utilized one standard (from hole EC-11) for its XRF analysis, whereas Louviers utilized 19 standards from drill hole EC-11.

The drill core crushed (coarse reject), and pulverized material is currently stored at a facility managed by the University of Nebraska-Lincoln (UNL). This facility is located approximately 8.5 km south of the town of Mead, Nebraska, and approximately 63 km northeast of Lincoln, Nebraska. Before the acquisition of the core by UNL in the early 2000s, the core was stored in the steel sheds on the property of Beverly Beethe.

Additional, QA/QC data from the historic drill program was inserted during subsequent re-sampling programs. The re-sampling in 2016 allowed for the insertion of scandium CRM reference material into a portion of the historical data.

11.1.3 Quantum Re-Sampling, 2010

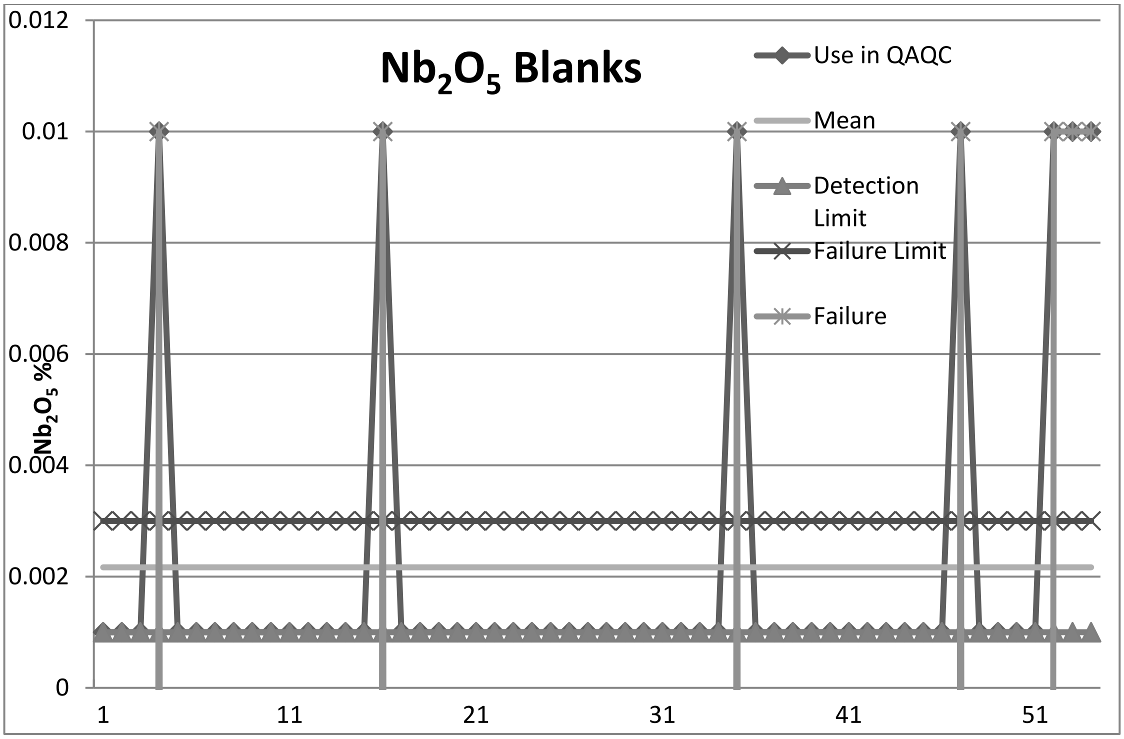
The 2010 re-sampling program involved sending 1,860 samples of pulverized material to ALS testing facility in North Vancouver, B.C., from the Molycorp drill holes that were originally prepared by the analytical division of Molycorp (see Table 11-2 and Figure 11-3). Samples were derived from 1.52 m (5 ft) or 3.05 m (10 ft) intervals of split NQ or HQ diameter core. The samples were selected based on the geological interpretation at the time and in areas of elevated Nb_2O_5 values. The purpose of this program was to have continuous sample selection down each drill hole.

This re-sampling exercise also allowed the opportunity to increase the amount of QA/QC data available over the historical data period. A protocol was implemented, which included the routine insertion of field duplicates, laboratory pulp duplicates, blanks and two niobium-certified reference standards (SX18-01 and SX18-05 see Figure 11-4 and Figure 11-5). Samples were transported to the ALS Chemex (ALS) facility in Reno, Nevada, where they were prepared for analysis prior to being shipped to the ALS testing facility in North Vancouver, B.C. The ALS testing facility, using method XE-XRF10, whereby samples are prepared by pulverizing to 90% passing -70 pm, then decomposed utilizing a lithium borate flux, for analysis by XRF. A portion of niobium results was checked with Hazen of Golden, Colorado (Quantum news release February 22, 2011).

Table 11-2: Summary of 2010 Re-Sampling Re-assay Program QA/QC Submissions

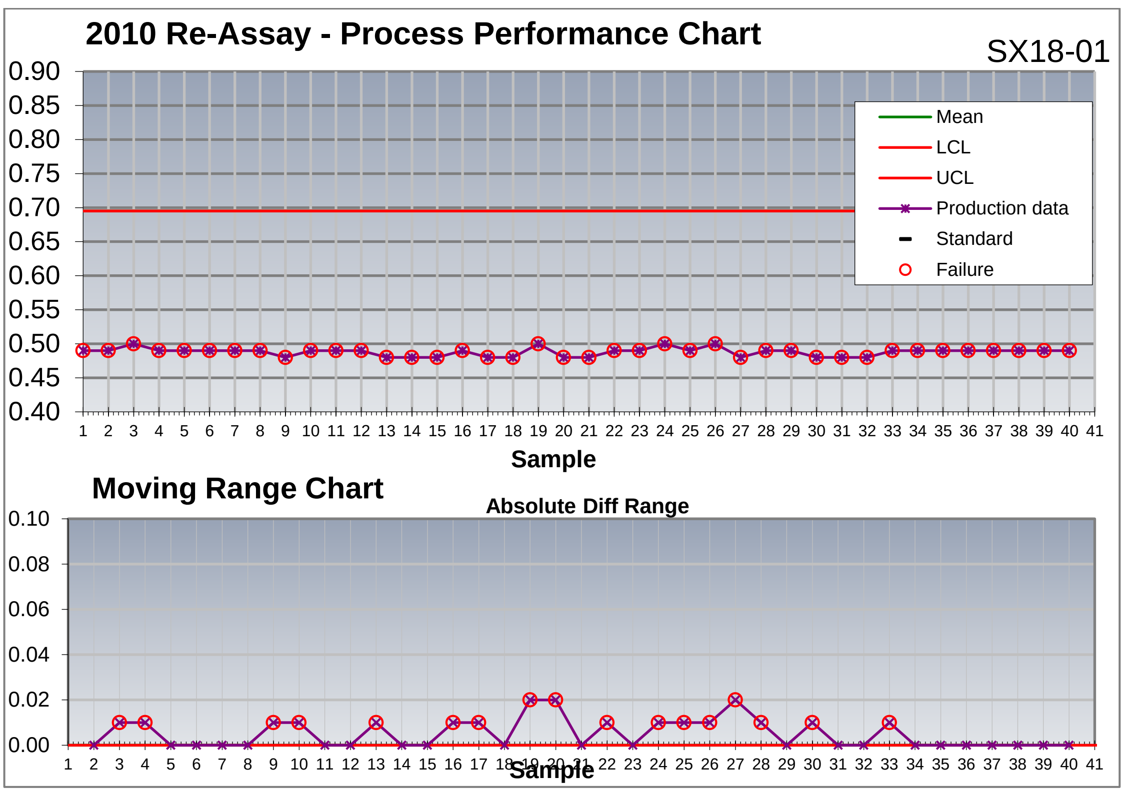
Sample Type	Number of Re-sampled Historical Data
# of Assays Resampled	1,860
Quartz Blanks Inserted	54
Insertion Rate of Blanks	2.9%
Failure Rate of Blank Material	13%
# of Nb_2O_5 CRM SX18-01 Material Sent to ALS Lab (0.695) MEDIUM GRADE	46
# of Nb_2O_5 CRM SX18-05 Material Sent to ALS Lab (0.973) HIGH GRADE	47
Insertion of Nb_2O_5 CRM Material	5%

Source: Nordmin, 2019



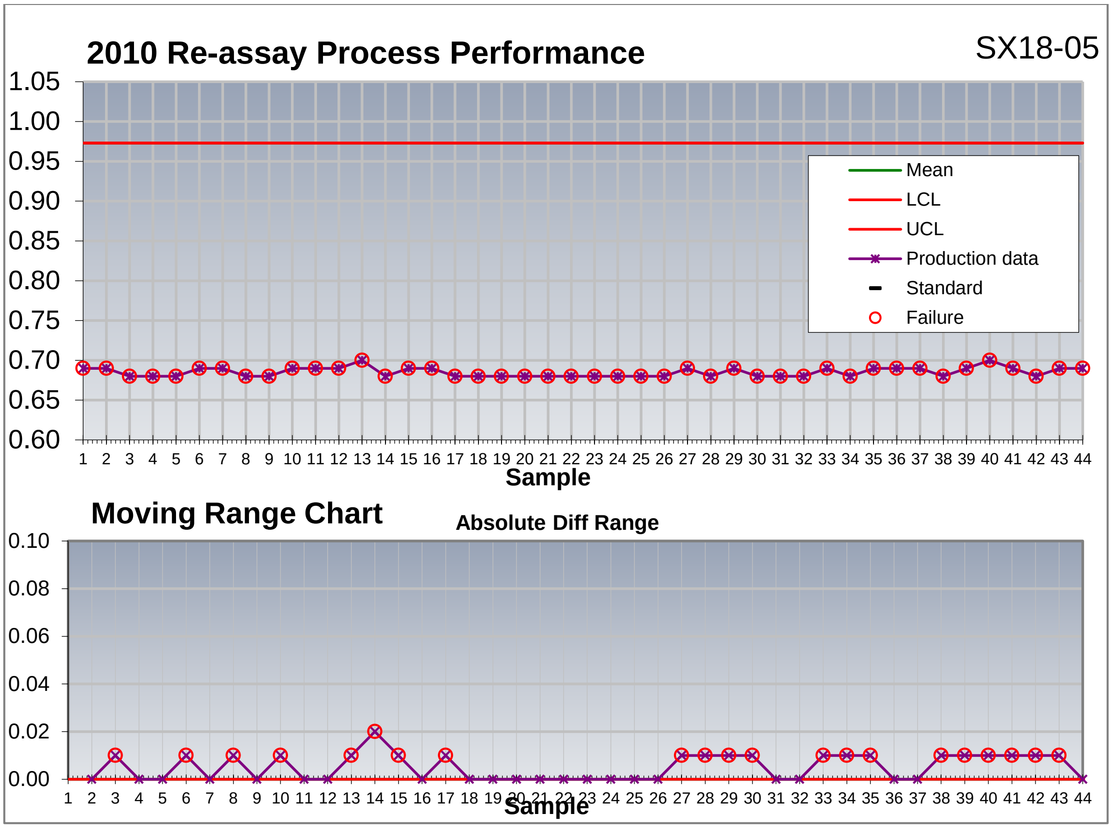
Source: Nordmin, 2019

Figure 11-3: Summary of Blank Control Charts for Nb_2O_5 Submission to Actlabs 2010 Re-Assay Program



Source: Nordmin, 2019

Figure 11-4: Summary of 2010 SX18-01 Nb_2O_5 Control Chart



Source: Nordmin, 2019

Figure 11-5: Summary of 2010 SX18-05 Nb_2O_5 Control Chart

CRM SX18-01 and SX18-05 are both consistently low using the ME-XRF10 methodology compared to the 2011 and 2014 drill programs. The 2011 and 2014 sampling or re-assaying programs did not use this methodology.

11.2 Sample Analysis Procedures, 2011 – Current

The 2011 and 2014 sawn core samples were shipped to Activation Laboratories Ltd. (Actlabs), 1336 Sandhill Drive, Ancaster, Ontario, Canada. Actlabs was the primary laboratory for sample preparation and analysis of the 2011 and 2014 drill core samples. Actlabs regularly participates in proficiency testing and maintains formal approval of CAN-P-1578, CAN-P-1579, CAN-P-1585, CAN-P-4E (ISO/I EC 17025:2005) accreditation from Standards Council of Canada and maintains current certification. Actlabs maintains ISO-17025 standards, which are obtained through experienced peer audits that ensure they conform to recognized analytical standards. Additionally, the accredited method validation verifies several analytical variables designed to ensures that data obtained from these methods are defensible. Actlabs maintain a custom Laboratory Information Management System (LIMS) system for reporting requirements.

NioCorp employed SGS through the 2014 sampling program as their secondary laboratory. SGS is an integrated geochemistry, mineralogy, and metallurgy laboratory in Lakefield, Ontario which has extensive experience with Nb_2O_5 and REE analysis for both exploration and metallurgy projects. SGS Lakefield is IS017025 accredited for the analysis methods used on this project (GO_XRF76V & GE JCP90A).

Core samples were shipped to Actlabs, where they were received, weighed, prepared, and assayed. Sample preparation is completed using Actlabs' RX1 preparation package that has been modified to meet the Project requirements. A summary of the process is detailed below:

- Samples were received and cataloged.
- Collection of as-received sample weight (kg).
- Drying of the whole sample at 60°C for 12 hours, in a customized high air flow drying room.
- Collection of dry sample weight (kg).
- Crushed in a jaw crusher (Boyd crushers) to 90% passing -10 mesh (2 mm), with quartz cleaner between each sample.
- Riffle split (RSD splitters or the option of Jones Riffle split) coarse crushed sample and extract a 250 g sample.
- Pulverization of the 250 g sample using ESSA pulverizers with ring and puck bowls to 95% -200 mesh (75 µm), with quartz cleaner used between each sample.
- Laboratory internal coarse-reject duplicates (1 in 50) and pulp duplicates (1 in 30) are also routinely prepared.
- Quality of the rejects and pulps are routinely monitored to ensure proper preparation procedures are performed.

Core samples were systematically assayed at Actlabs for niobium (Nb_2O_5) and tantalum (Ta_2O_5) by XRF analysis, using a Panalytical Axios-mAX, following a lithium metaborate/tetraborate fusion of a 2 g sample. All XRF analysis followed procedures outlined in Actlabs "8-XRF" package, with selected analytical results provided for Nb_2O_5 and Ta_2O_5. A whole rock and forty-three (43) major elements analyses were completed using ICP and ICP/MS (by a Perkin Elmer Sciex ELAN 6000, 6100, 9000 ICP/MS) finish following a lithium metaborate/tetraborate fusion preparation as defined by analytical Actlabs' "8-REE Major Elements Fusion ICP(WRA)/Trace Elements Fusion ICP/MS(WRA4B2)" package.

Additional analysis was performed for fluoride, using the analytical package "4F-F". Fluoride content is quantified using a fluoride ion electrode to directly measure fluoride-ion activity when a prepared fuseate is dissolved in dilute nitric acid and its ionic strength adjusted in ammonium citrate buffer. Before the analysis, the sample is prepped using a combined fusion with lithium metaborate and lithium tetraborate in an induction furnace. Fluoride analysis was completed for 2014 drill holes, NEC14-006, NEC14-007, and NEC14-008.

All QC data is registered in the LIMS system, and assay results have been returned to NioCorp in an electronic format. Following the QA/QC review by the qualified geologist, the results are loaded into the Fusion database with the batch number and date of assay recorded.

During the preparation procedure, coarse-reject splits and pulp-splits are extracted from the original core sections for primary laboratory and secondary (external) laboratory check analysis. These samples are then inserted into the sampled sequence.

Pulp samples are routinely extracted with inserted CRM samples which were prepared by Actlabs and shipped to SGS (Lakefield), where they were received, evaluated for sample quality and re-homogenized, and assayed. SGS (Lakefield) prepared and re-homogenized samples before analysis using MISC80. During preparation, SGS completed a 10% sieve check (SCR32 package) to ensure 95% sample pulverization passes 200 mesh (75 µm) preparation requirements. Samples were assayed using an XRF analysis for Nb_2O_5 and thirteen major whole rock oxides, following a borate fusion as defined under SGS package "GO XRF76V - ORE GRADE" (see Table 11-3). Scandium analysis has been completed at SGS laboratory using GE JCP90A package, which has a detection limit of 5 ppm.

Table 11-3: Detection Limits for Primary Laboratory (Actlabs)

XRF (%)		Trace Elements ICP & ICP/MS (ppm)					
Oxide	**Detection Limit**	**Element**	**Detection Limit**	**Reported By**	**Element**	**Detection Limit**	**Reported By**
Nb_2O_5	0.003	Ag	0.5	ICP/MS	Nb	1	ICP/MS
Ta_2O_5	0.003	As	5	ICP/MS	Nd	0.1	ICP/MS
4F-F (%)		Ba	3	ICP	Ni	20	ICP/MS
Analysis	**Detection Limit**	Be	1	ICP	Pb	5	ICP/MS
F	0.01	Bi	0.4	ICP/MS	Pr	0.05	ICP/MS
Fusion ICP (%)		Ce	0.1	ICP/MS	Rb	2	ICP/MS
Oxide	**Detection Limit**	Co	1	ICP/MS	Sb	0.5	ICP/MS
SiO_2	0.01	Cr	20	ICP/MS	Sc	1	ICP
Al_2O_3	0.01	Cs	0.5	ICP/MS	Sm	0.1	ICP/MS
Fe_2O_3	0.01	Cu	10	ICP/MS	Sn	1	ICP/MS
MgO	0.01	Dy	0.1	ICP/MS	Sr	2	ICP
MnO	0.001	Er	0.1	ICP/MS	Ta	0.1	ICP/MS
CaO	0.01	Eu	0.05	ICP/MS	Tb	0.1	ICP/MS
TiO_2	0.001	Ga	1	ICP/MS	Th	0.1	ICP/MS
Na_2O	0.01	Gd	0.1	ICP/MS	T	0.1	ICP/MS
K_2O	0.01	Ge	1	ICP/MS	Tm	0.05	ICP/MS
P_2O_5	0.01	Hf	0.2	ICP/MS	U	0.1	ICP/MS
Loss on Ignition	0.01	Ho	0.1	ICP/MS	V	5	ICP
		In	0.2	ICP/MS	W	1	ICP/MS
		La	0.1	ICP/MS	Y	2	ICP
		Lu	0.04	ICP/MS	Yb	0.1	ICP/MS
		Mo	2	ICP/MS	Zn	30	ICP/MS
					Zr	4	ICP

Source: Nordmin, 2019

11.2.1 NioCorp 2015-2016 Re-Assay Program

During the 2015 Mineral Resource Estimate completed by SRK, it was noted that a portion of the database was missing assays grades for TiO_2, and Sc used in the estimate. The samples were sourced from NioCorp's storage facility in Mead and submitted to Actlabs using the same, Code 8-Nb_2O_5 & Ta_2O_5 - XRF Option, and ICP/MS methods described in Section 11.3 of this Technical Report.

The sourced samples were a combination of chips, pulps and pulverized samples from the original 1/2 NQ core samples. The samples were selected based on sample lists of absent values from the 2015 Mineral Resource Estimate and were located under the guidance of NioCorp's geologist. During the re-assay program, NioCorp included QA/QC samples in the form of standards and duplicates. In total, 766 samples were included in the program of which 44 were pulp duplicates, and 55 were standard reference material. A summary of the submissions for the 2016 program is shown in Table 11-4.

Table 11-4: Summary of 2016 Re-Assay Program Submissions

Sample Type	Number of Submissions
Standards	55
Pulp Dups	44
Pulp Samples	203
Pulverized Samples	374
Chip Samples	90
Samples	766

Source: Nordmin, 2019

11.3 Quality Assurance/Quality Control Programs

Quality Control (QC) measures are typically set in place to ensure the reliability and trustworthiness of exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are essential as a safeguard for project data and form the basis for the Quality Assurance (QA) program implemented during exploration.

Analytical QC measures typically involve internal and external laboratory procedures implemented to monitor the precision and accuracy of the sample preparation and assay data. They are also important to identify potential sample sequencing errors and to monitor for contamination of samples.

Sampling and analytical QA/QC protocols typically involve taking duplicate samples and inserting quality control samples (CRMs and blanks) to monitor the reliability of the assay results throughout the drill program. Umpire check assays are typically performed to evaluate the primary lab for bias and involve re-assaying a set proportion of sample rejects and pulps at a secondary umpire laboratory.

11.3.1 Re-Sampling/Verification of Historical Assays

In 2010 to 2011 Dahrouge Geological consulting completed a re-sampling/verification work program concerning the historical assays.

- A total of 1,860 samples (approximately 44% of the original assays) were selected for re-analysis during the program and subjected to the current QA/QC protocols. The selection for re-assay was based on available material and proximity to the mineralization wireframe used during that study.
- In 2015/2016 a re-assay program was completed which consisted of sending 1,433 pulps to Actlabs, which previously were absent for titanium and scandium. This included the insertion of additional QA/QC material. The QA/QC program by NioCorp used a CRM sourced from Geostats (GRE-03 and GRE-04), which contained a certified value for Sc (ppm) and Ti (%). A summary of the additional type of samples, source, and level of insertion for the re-assay program is included in Table 11-5.

Table 11-5: Summary of Actual Submissions per Sample Type within the 2015-2016 Re-Assay Program

Sample Type	Type	Total	Insertion Rate
Original Sections	1/2 NQ core	1,433	NA
Pulp Duplicates	Pulp-split	7	0.5%
CRMs (Ti & Sc)	Pulp	62	5.0%

Source: Nordmin, 2019

11.3.2 NioCorp 2011 - Current

NioCorp integrated a series of routine QA/QC procedures throughout the sampling and analytical analysis for both the 2011 and 2014 drilling programs to ensure the highest level of quality was maintained throughout the process. This included the insertion of duplicate samples taken from various stages of the process, insertion of known control samples (CRMs and blanks) and sending third-party pulps to the secondary lab (SGS).

Sample tickets were assigned initially at the core shed using barcodes with duplicate tickets placed inside and on the outside of the bag. Sample identification was confirmed using barcode labelling and visual sample type comparisons before sample shipment. The use of barcoded samples ensured both shipment forms and analytical labs used accurate information. Multiple types of QC samples were inserted at this stage of the process, which includes the following:

- Field quartz blanks (1 in 20, 5%) were inserted within or immediately after samples collected from mineralized intervals, targeting zones of elevated visual mineralization, where possible.
- CRMs (1 in 20, 5%) were inserted in the field with the sample sequence.
- Field quarter-core duplicates (1 in 20, 5%) were inserted to test mineralization and sampling variability.

These following control measures were used to monitor both the precision and accuracy of sampling, sub-sampling, preparation and assaying. A summary of the designed type of samples, source and level of insertion is included in Table 11-6 and the actual submissions in Table 11-7.

Table 11-6: Summary of Designed Level of Insertion of QC Submissions (2011 and 2014 Drill Program)

Sample Type	Sample Sub-type	Type	Insertion Rate
Blanks	Field Quartz Blanks	Optical Quartz	5.0%
Certified Reference Material	SX18-01 (Dilinger Hutte Lab)	Nb CRM	6.0%
	SX18-02 (Dilinger Hutte Lab)	Nb CRM	
	SX18-04 (Dilinger Hutte Lab)	Nb CRM	
	SX18-05 (Dilinger Hutte Lab)	Nb CRM	
Duplicates	Field quartered core	¼ HQ Core	5.0%
External Lab Checks	Coarse-Rejects	Reject split	3.0%
	Pulp	Pulp split	5.0%
	Field Quartz Blanks	Optical Quartz	(5% of splits)

Source: Dahrouge, 2014

Table 11-7: Summary of Actual Submissions for Nb_2O_5 (2011 and 2014 Drill Program

Sample Type	Type	Total Samples	Insertion Rate
Original Sections	1/2 HQ core, PQ core	9,653	NA
Field Duplicates	1/4 HQ core	419	4.3%
Coarse-Reject Duplicates	Crush-split	260	2.7%
Pulp Duplicates	Pulp-split	468	4.9%
Standards (CRMs)	Pulp	496	5.1%
Field Blanks	Optical Quartz	454	4.7%
External Checks	Pulp	462	4.8%
External Checks Duplicates	Pulp-split	44	9.5%*
External Checks CRMs	Pulp	49	10.6%*

Source: Nordmin, 2019
*Insertion rate is a percentage of total External Check Samples submitted
*Does not include any duplicates for the CRM in 2011

Except for 2016, the QA/QC data was analyzed by the project geologist on a routine basis before entering the data into the central database. When CRMs failed, each result was checked for possible sample swaps or significant failures. A single failure did not guarantee the failure of the entire batch, but multiple failures warrant re-analysis.

Failures were reported directly back to the laboratory, for re-analysis. In the 2011 drill program, no standards were identified as failing with a high enough level of concern to request re-assay. Any revision to the 2011 certificates was either a result of further re-sampling or internal lab revisions. In the 2014 program, when it was determined that a CRM failure was significant, ten samples on either side of the CRM were re-assayed. The re-assay was taken as the final assay result, and the original certificate was overwritten when imported into Fusion. Fusion maintains both certificates and can produce an audit trail to detail which assays have been updated. A limited number of samples were re-sampled throughout the program.

The following section provides details of the types of samples used at each section of the sampling process and followed by a discussion of the QA/QC results.

11.3.3 Quality Assurance/Quality Control Results

11.3.3.1 Field Quartz Blanks

Coarse natural clear quartz blanks (sourced from an optical-quality quartz quarry) were included to identify potential contamination.

Methodology:

- Utilization of the same sample preparation system as project samples.
- Field quartz blanks were indistinguishable from project samples, thus preventing the blanks from being treated differently to the project samples at the applicable laboratory.

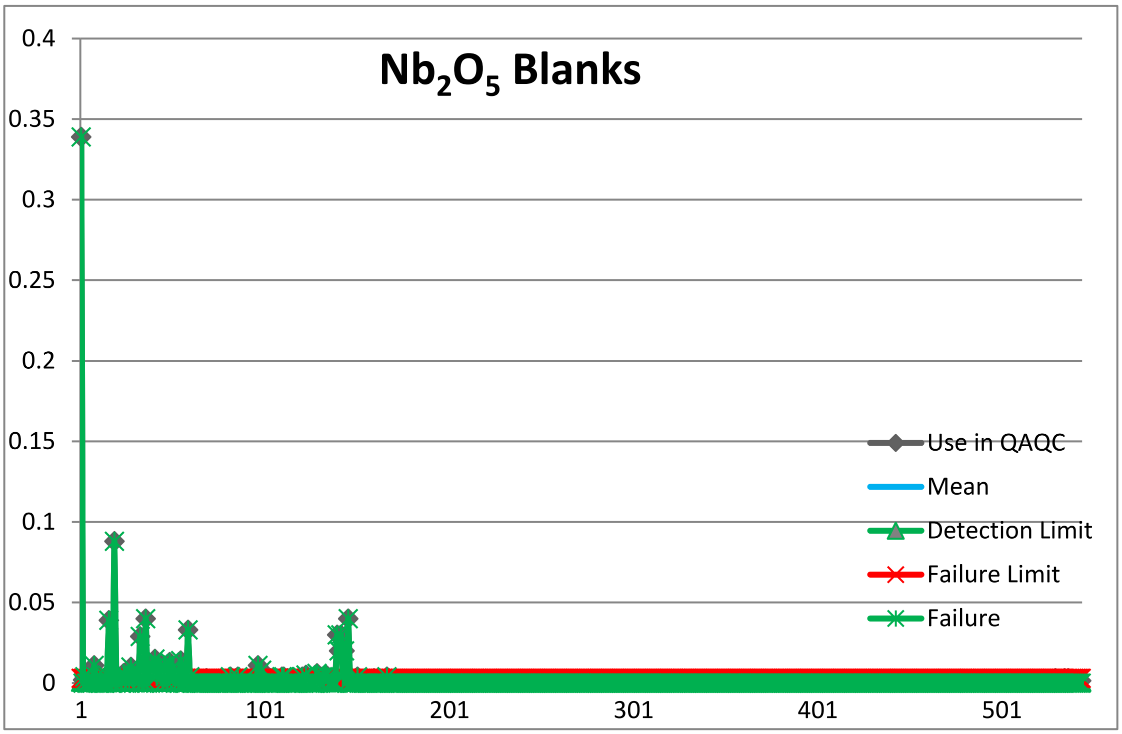
The field quartz blanks used a base detection limit of 2 x XRF detection limits and 20 x ICP-MS detection limits. Results falling above this value are reported to the laboratory as having potential contamination.

Nordmin notes that both sets of controlled blanks used in the two analytical procedures exhibited clusters of contamination at the start of the program. These clusters were significantly reduced as the program advanced; the failing blanks were re-analyzed to differentiate between inaccurate analytical results and potential contamination points. The findings were discussed with the lab to allow for corrective measures to be implemented (see Table 11-8 and Figure 11-6).

Table 11-8: Summary of 2011 and 2014 Drill Program Nb_2O_5 Blank Insertion

Element Nb_2O_5 – Actlabs	Drill Program	
Lab Package: 8-Nb_2O_5 (XRF)	2011	2014
# of Assays sent to Lab	1,800	9,653
# of Field Quartz Blanks Sent to Lab	90	454
Insertion Rate of Blanks	5.0%	4.7%
# of Blank Failure (2x XRF Detection Limit)	35	19
Percentage of Blank Failure Rate	39%	4%

Source: Nordmin, 2019



Source: Nordmin, 2019

Figure 11-6: Summary of Blank Control Charts for Nb_2O_5 Submission to Actlabs 2011 and 2014 Drill Program

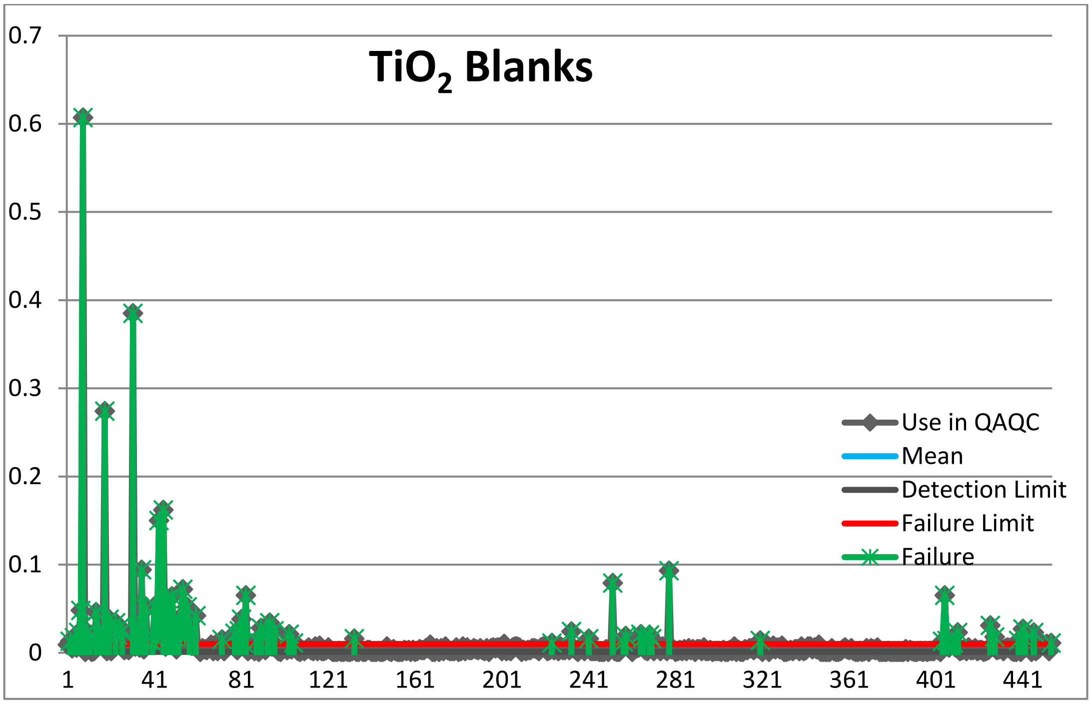
Nordmin has noted the TiO_2 QA/QC data is more variable than the Nb_2O_5 data for the 2011 and 2014 program. Overall, the majority of the 2014 samples are less than 0.01% control line, which is the equivalent of 10x the detection limit, above which potential contamination may be identified.

Overall Nordmin considers that the blank material has acceptable levels of error and there is limited evidence of any major contamination issues at the laboratory since the problem was identified and corrected during the 2014 drill program (see Table 11-9 and Figure 11-7).

Table 11-9: Summary of 2011 and 2014 Drill Program TiO_2 Blank Insertion

Element TiO_2 - Actlabs	Drill Program	
Lab Package: 8-REE (FUS-ICP/FUS-MS)	2011	2014
# of Assays sent to Lab	1,800	9,653
# of Field Quartz Blanks Sent to Lab	91	454
Percentage Insertion of Blanks	5.0%	4.7%
# of Blank Failure (10 ICP-MS Detection Limit)	82	40
Percentage of Blank Failure Rate	75%	9%

Source: Nordmin, 2019



Source: Nordmin, 2019

Figure 11-7: Summary of Blank Control Charts for TiO_2 Submission to Actlabs

11.3.3.2 CRM Material

<u>Nb_2O_5 CRMs</u>

A summary of the defined limits and results for Nb_2O_5 2011 and 2014 drill programs are shown in Table 11-10. The results for Nb_2O_5 CRM are demonstrated in Figure 11-8 to Figure 11-14. The CRM submissions show an insertion rate of between 3.3% and 5.1%, and a relatively low failure rate between 4.2 % and 5.1%, which are within acceptable limits. Nordmin discovered that all four CRMs report a high bias within the various grade ranges, and a moving range further demonstrates the bias. Possible reasons for the continued high bias include:

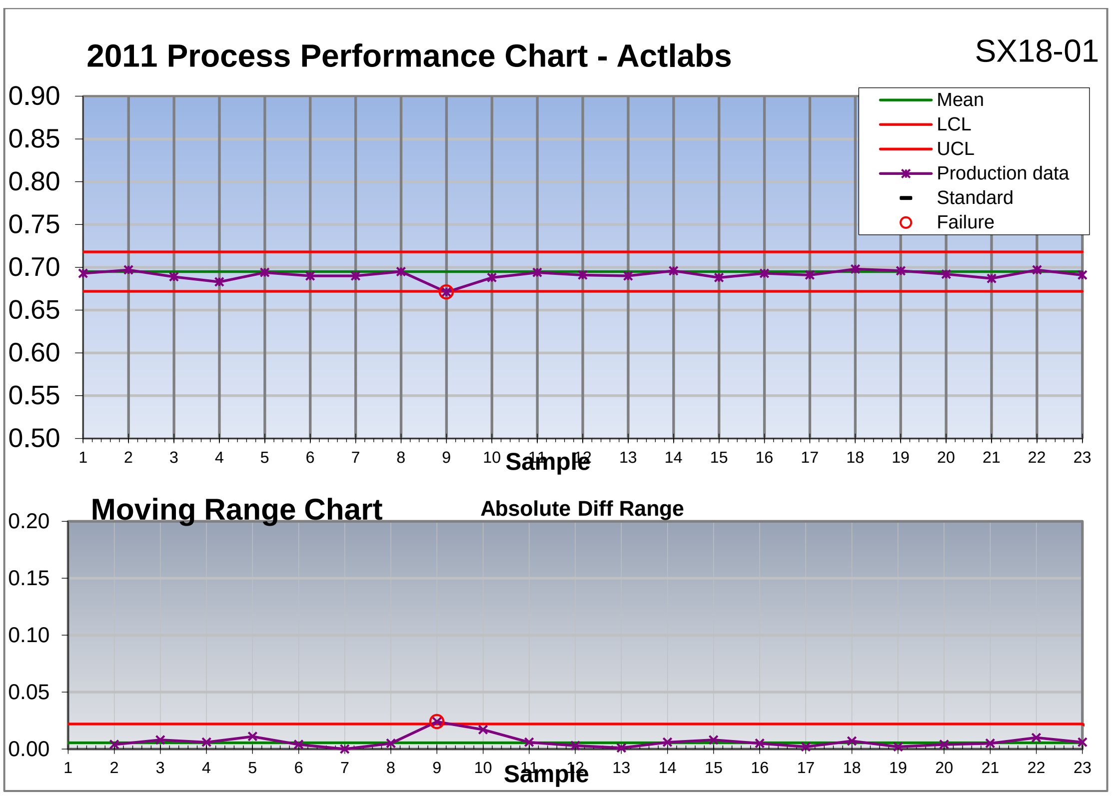
- An error possibly introduced during the manufacture of the CRMs.

- The method or equipment used by Actlabs may not be compatible with the use of a specific CRM. For example, SGS used a borate fusion with an XRF finish, compared to Actlabs' use of a lithium metaborate/tetraborate fusion
- Further follow-up work is required to determine the cause of the bias across all four standards.

Table 11-10: Summary of Nb_2O_5 CRM (Actlabs)

Element Nb_2O_5 - Actlabs	Drill Program	
Lab Package: 8-Nb_2O_5 (XRF)	**2011**	**2014**
# of Assays sent to Lab	1,800	9,653
# of CRM SX18-01 Material Sent to Lab (0.695) MEDIUM GRADE	23	169
# of CRM SX18-02 Material Sent to Lab (0.199) LOW GRADE	0	154
# of CRM SX18-04 Material Sent to Lab (1.32) HIGH GRADE	17	165
# of CRM SX18-05 Material Sent to Lab (0.973) MEDIUM GRADE	19	8
Total Nb_2O_5 CRM sent to the lab	59	496
Total Insertion Rate	3.3%	5.1%
Total Error Rate	5.1%	4.2%

Source: Nordmin, 2019

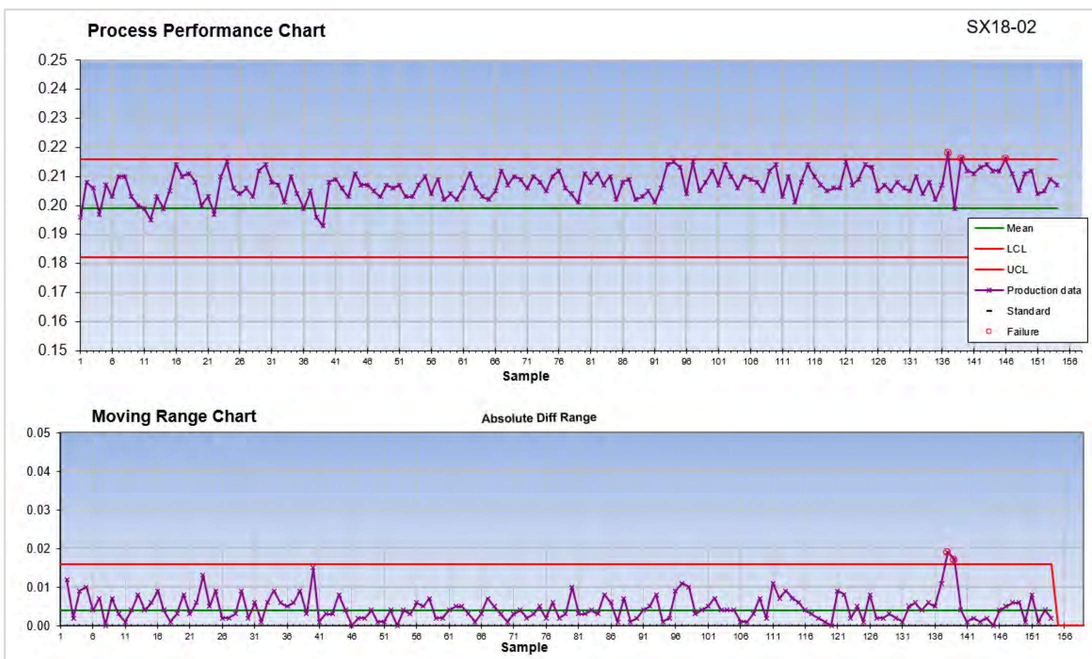


Source: Nordmin, 2019

Figure 11-8: Summary of 2011 SX18-01 Nb_2O_5 Control Chart

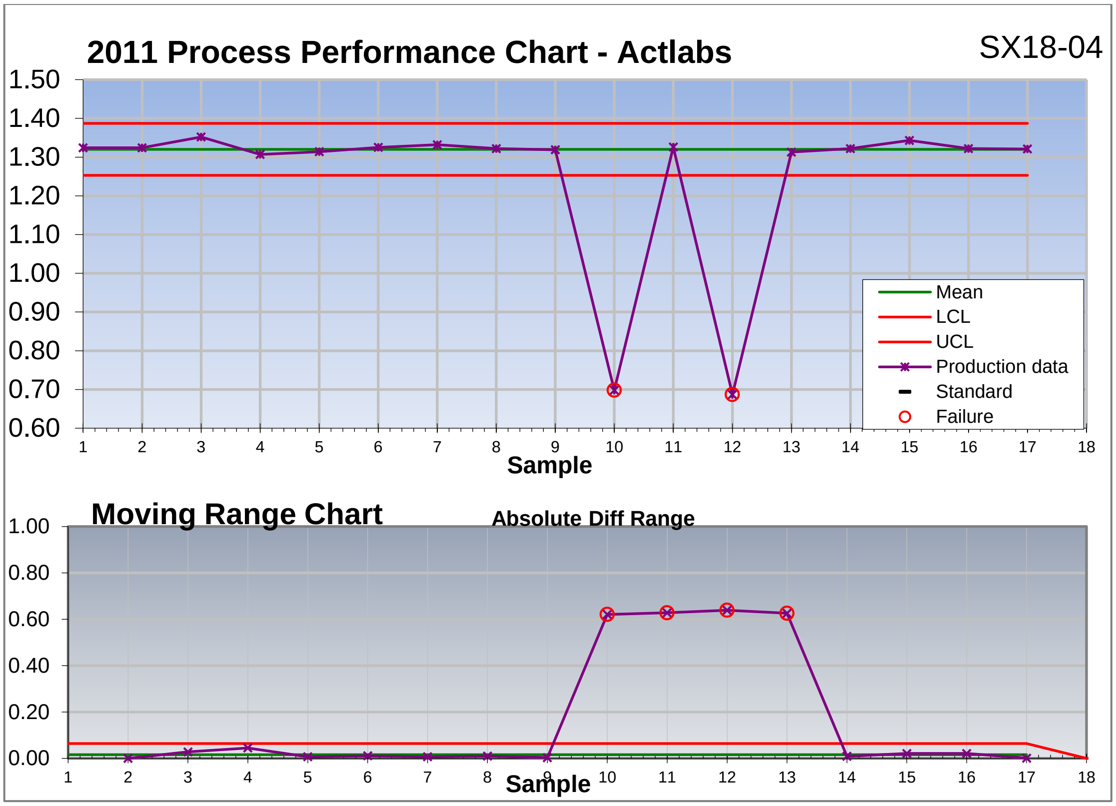
Source: Nordmin, 2019

Figure 11-9: Summary of 2014 SX18-01 Nb_2O_5 Control Chart



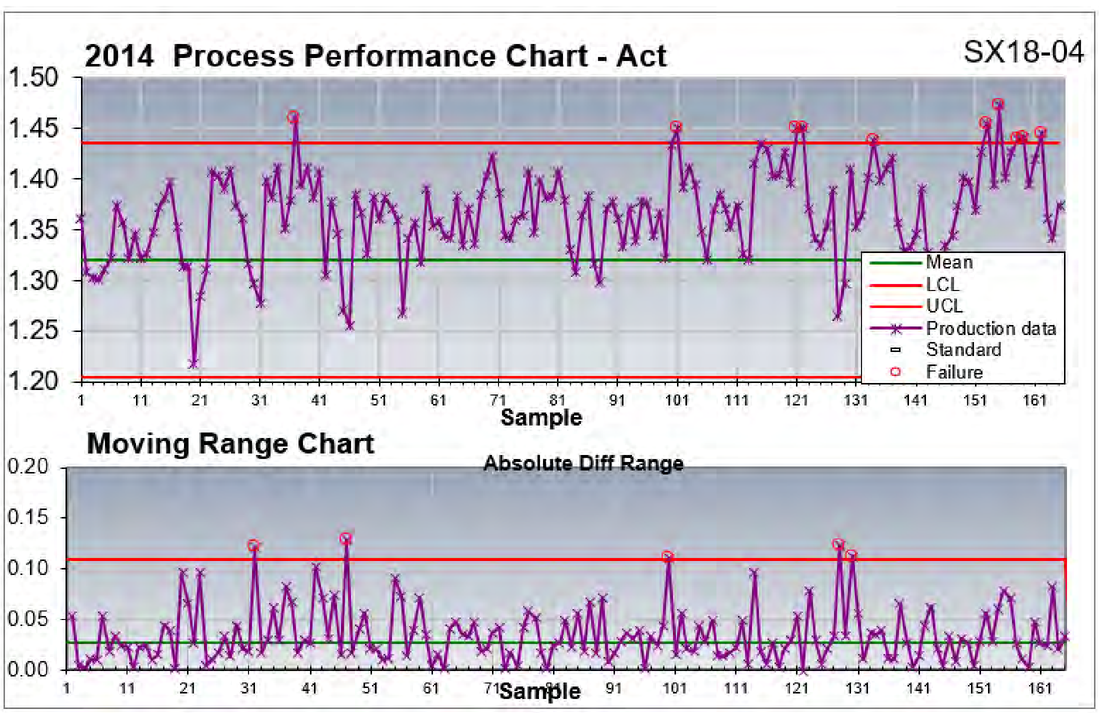
Source: Nordmin, 2019

Figure 11-10: Summary of 2014 SX18-02 Nb_2O_5 Control Chart



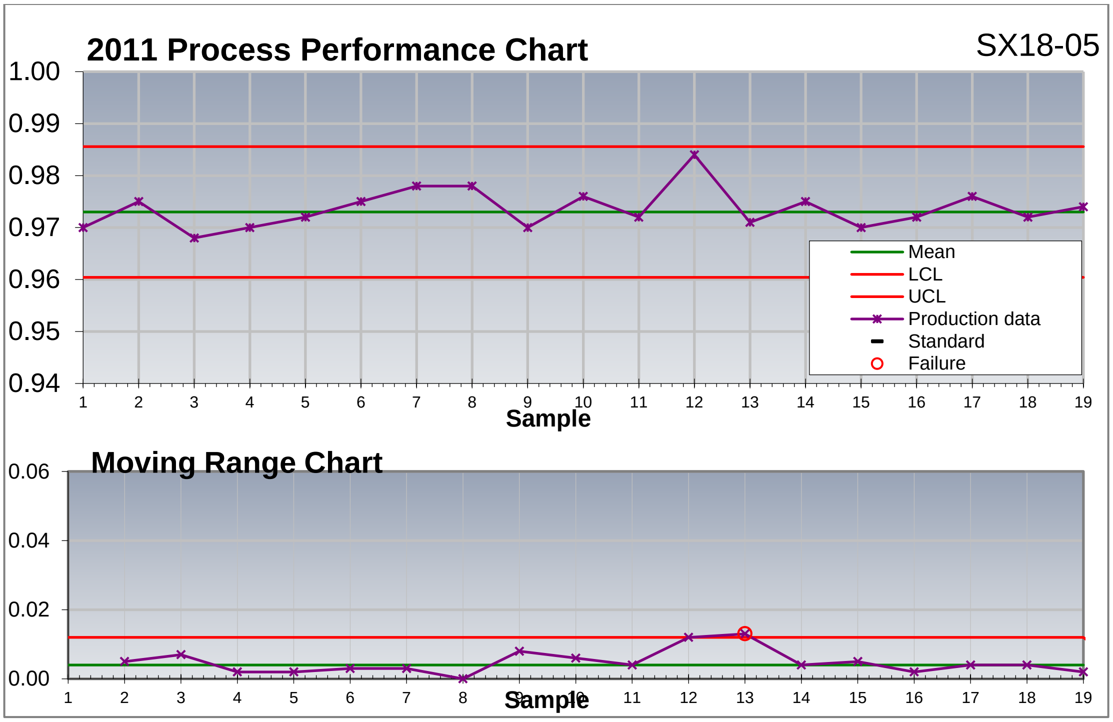
Source: Nordmin, 2019

Figure 11-11: Summary of 2011 SX18-04 Nb_2O_5 Control Chart



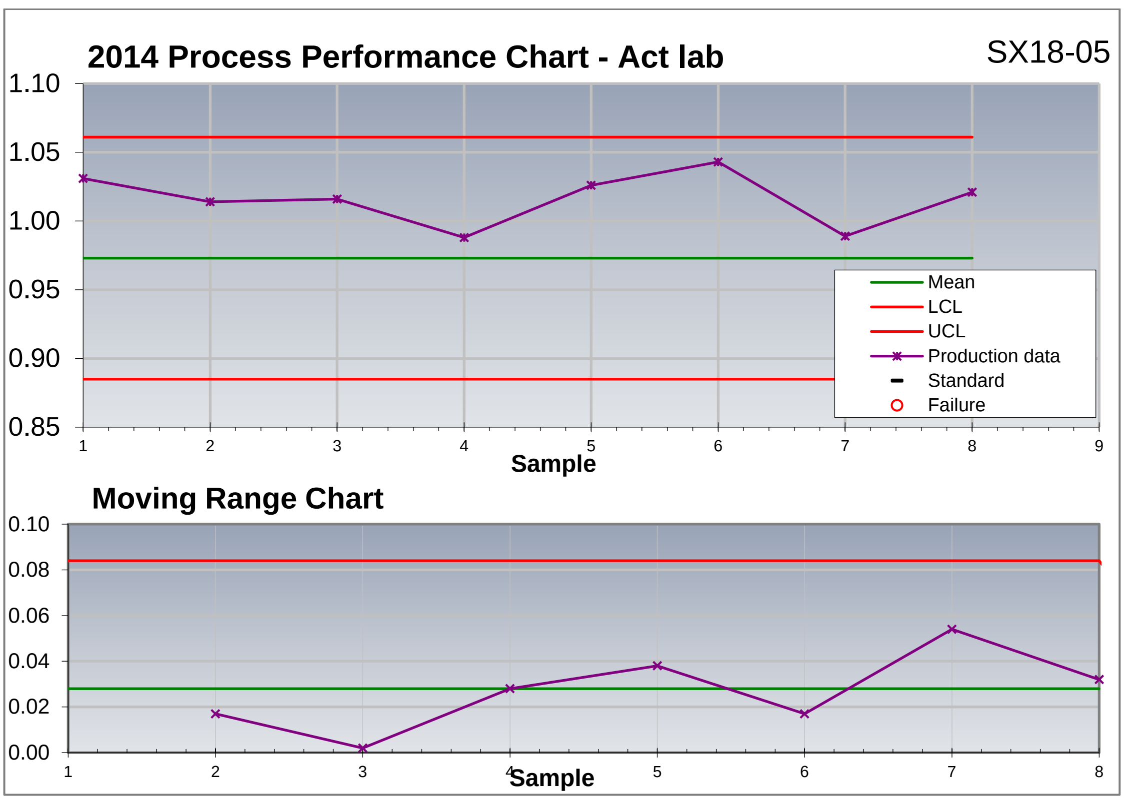
Source: Nordmin, 2019

Figure 11-12: Summary of 2014 SX18-04 Nb_2O_5 Control Chart



Source: Nordmin, 2019

Figure 11-13: Summary of 2011 SX18-05 Nb_2O_5 Control Chart



Source: Nordmin, 2019

Figure 11-14: Summary of 2014 SX18-05 Nb_2O_5 Control Chart

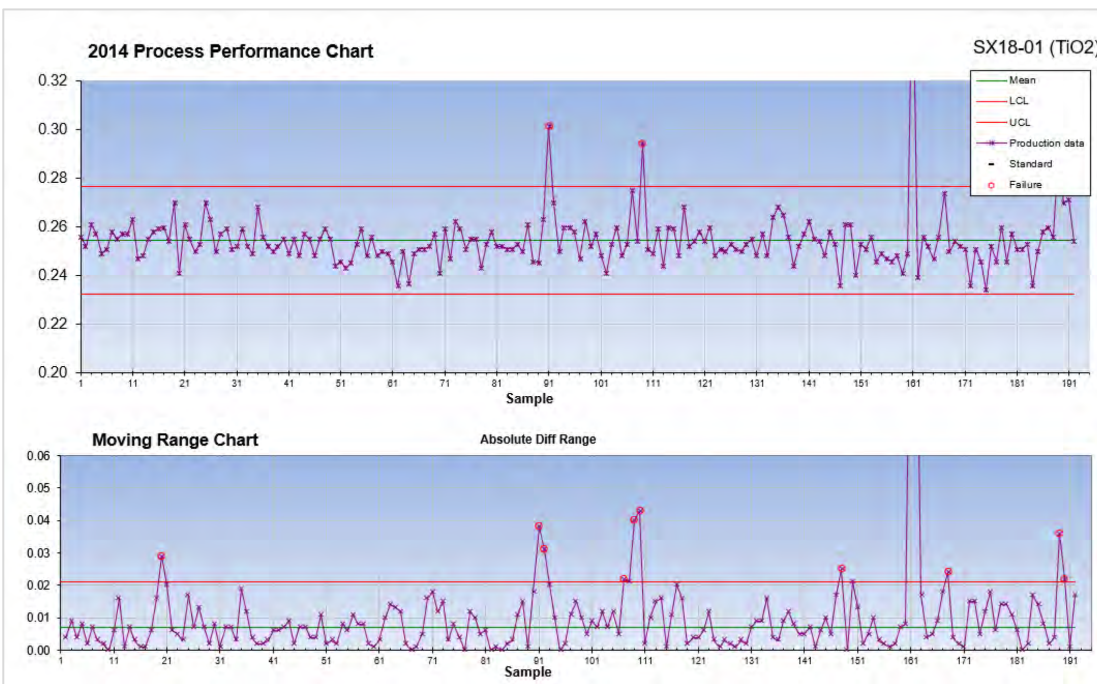
TiO₂ CRMs

A summary of the defined limits and results for TiO_2 during the 2011 and 2014 drill programs are shown in Table 11-11. The results for TiO_2 CRM are demonstrated in Figure 11-15 to Figure 11-18. A total of 557 CRMs were inserted at a rate between 3.3 % and 5.2%, resulting in a failure rate between 2.8% and 6.7%. A significant issue identified was the grade range of the TiO_2 in the CRM is in the order of 0.25% to 0.30%, which is an order of magnitude lower than the typical grade ranges at the Project of 2.0% to 3.5% within the resource model. Given the low grade nature of the assays in the TiO_2 CRMs, Nordmin has relied more heavily on the duplicate assays and external checks by SGS, as well as the results of the 2016 re-assay program.

Table 11-11: Summary of TiO_2 CRM (Actlabs)

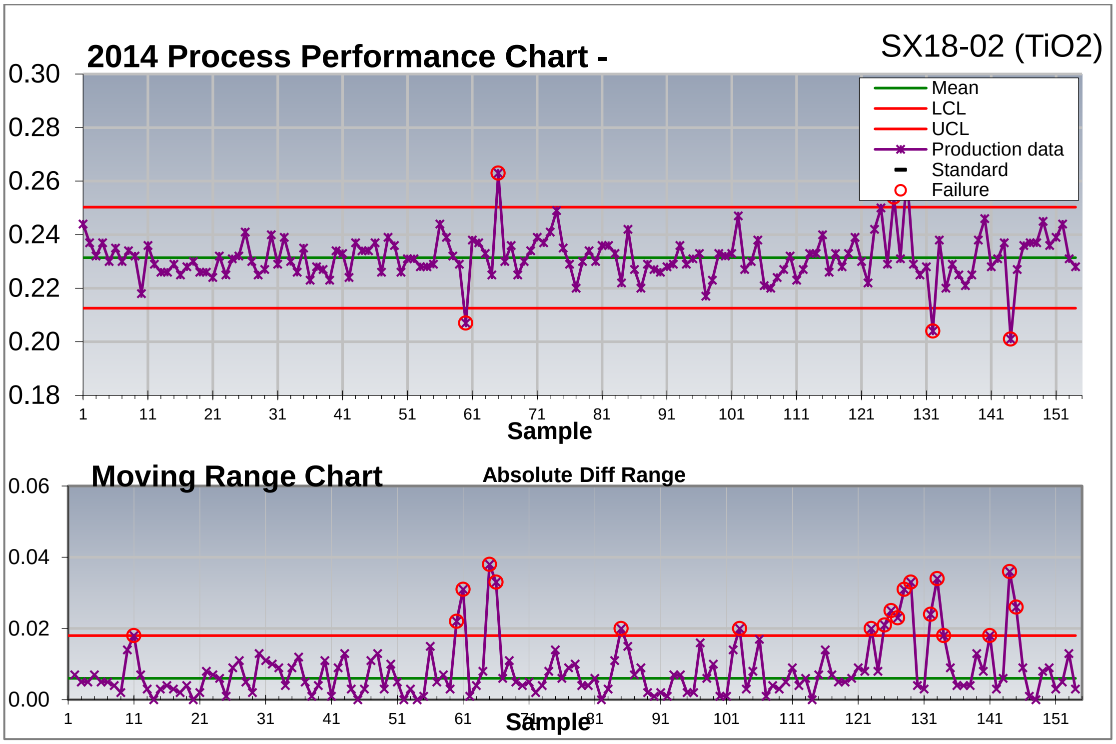
Element TiO_2 - Actlabs	Drill Program	
Lab Package: 8-REE (FUS-ICP/FUS-MS)	**2011**	**2014**
# of Assays sent to Lab	1,800	9,653
# of CRM SX18-01 Material Sent to Lab (0.266) LOW GRADE	23	169
# of CRM SX18-02 Material Sent to Lab (0.237) LOW GRADE	0	154
# of CRM SX18-04 Material Sent to Lab (0.287) LOW GRADE	17	167
# of CRM SX18-05 Material Sent to Lab (0.295) LOW GRADE	19	8
Total CRM sent to the lab (TiO_2)	59	498
Total Insertion Rate	3.30%	5.20%
Total Error Rate	6.7%	2.80%

Source: Nordmin, 2019



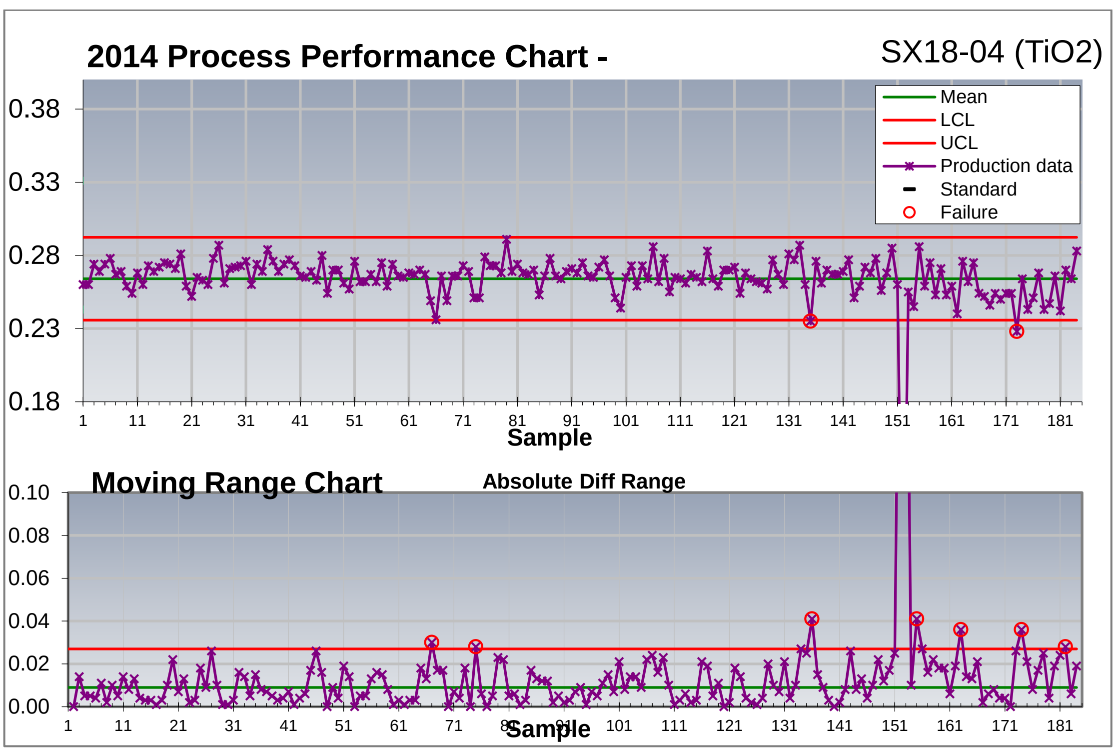
Source: Nordmin, 2019

Figure 11-15: Summary of 2014 SX18-01 TiO_2 Control Chart



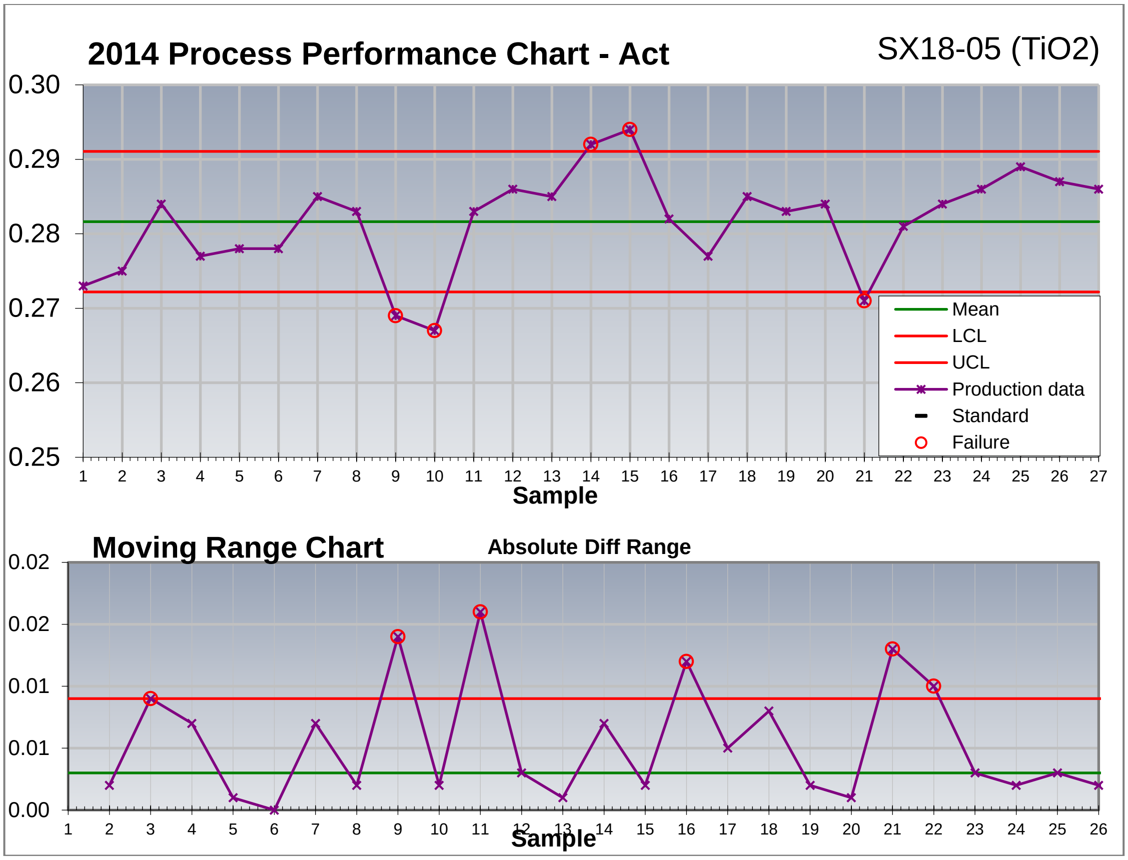
Source: Nordmin, 2019

Figure 11-16: Summary of 2014 SX18-02 TiO_2 Control Chart



Source: Nordmin, 2019

Figure 11-17: Summary of 2014 SX18-04 TiO_2 Control Chart



Source: Nordmin, 2019

Figure 11-18: Summary of 2014 SX18-05 TiO_2 Control Chart

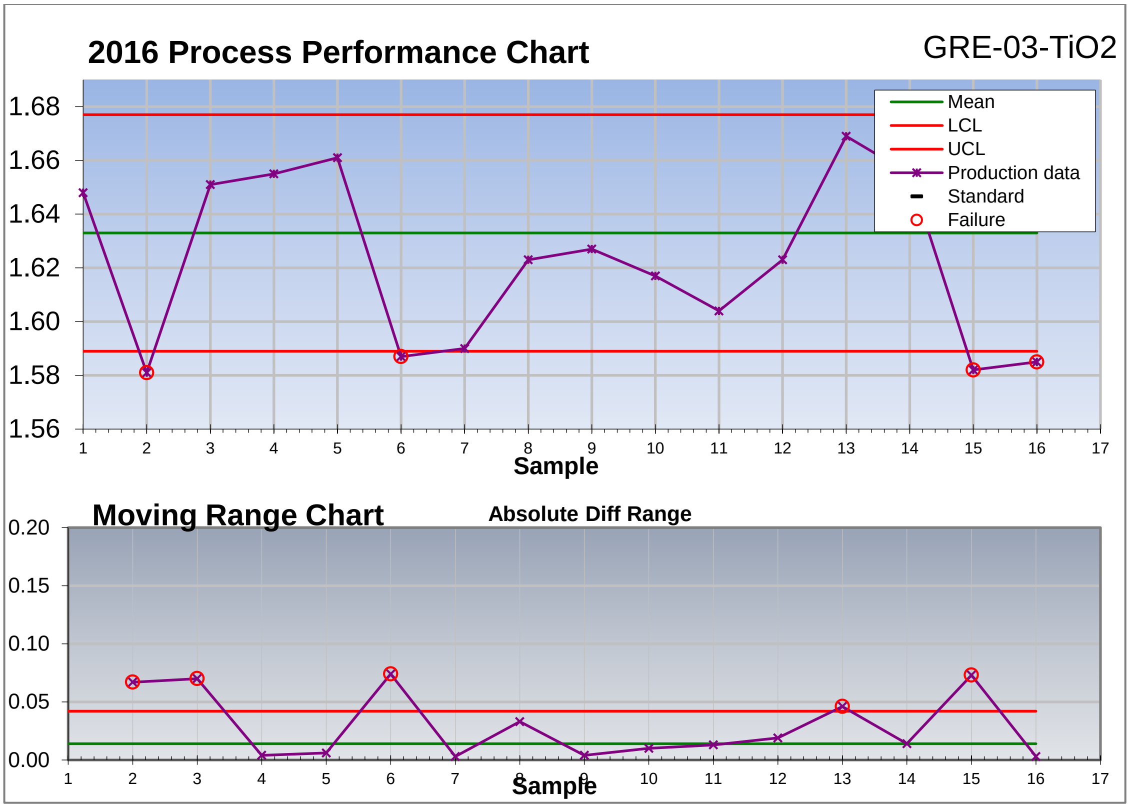
11.3.3.3 TiO_2 CRMs (2015 - 2016 Re-Sampling Program)

A summary of the defined limits and results for TiO_2 2015 - 2016 re-sampling program are shown in Table 11-12. Thirty-one CRMs were inserted, resulting in a failure rate of 16%. The results for TiO_2 CRM are demonstrated in Figure 11-19 and Figure 11-20.

Table 11-12: Summary of TiO_2 CRM (2015 - 2016 Re-Sampling Program)

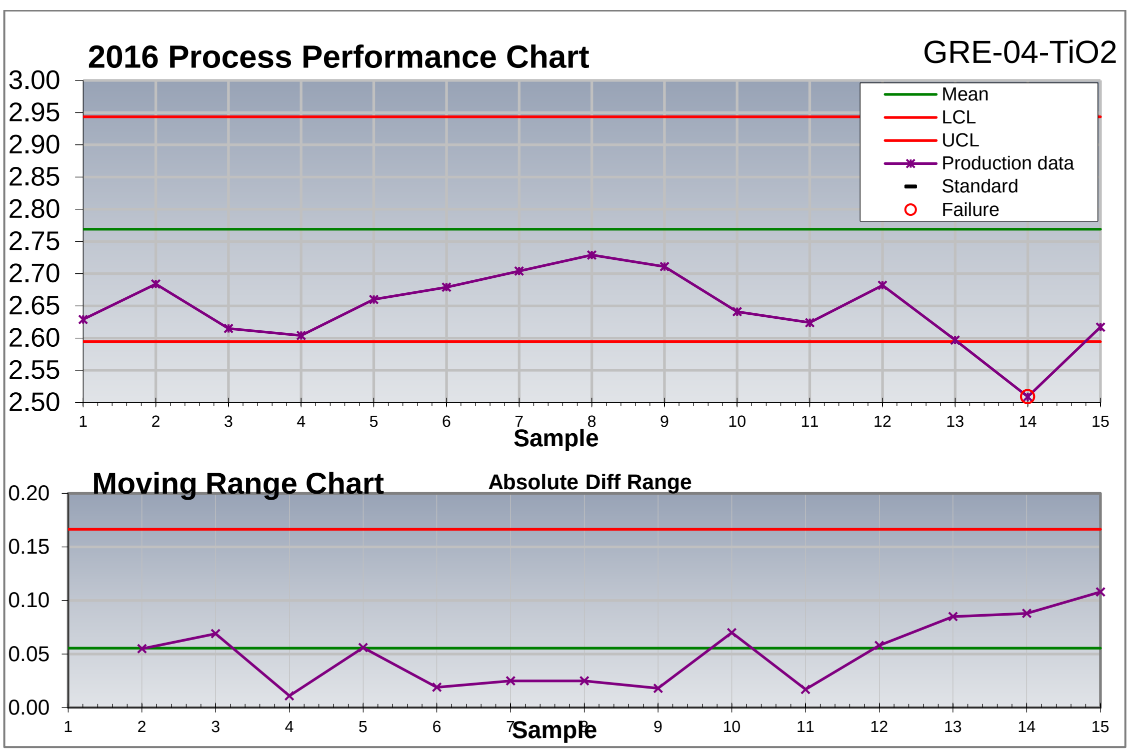
Element TiO_2 - Actlabs 2016	Drill Program
Lab Package: 8-REE (FUS-ICP/FUS-MS)	2016 Re-Sampling Program of Historic Molycorp Holes
# of CRM GRE-03 Material Sent to Lab (1.633) MEDIUM TiO_2 GRADE	16
# of CRM GRE-04 Material Sent to Lab (2.769) HIGH GRADE TiO_2 Grade	15
Total CRM sent to the lab (TiO_2)	31
Total Error Rate	16%

Source: Nordmin, 2019



Source: Nordmin, 2019

Figure 11-19: Summary of 2016 GRE-03-TiO_2 Control Chart



Source: Nordmin, 2019

Figure 11-20: Summary of 2016 GRE-04-TiO_2 Control Chart

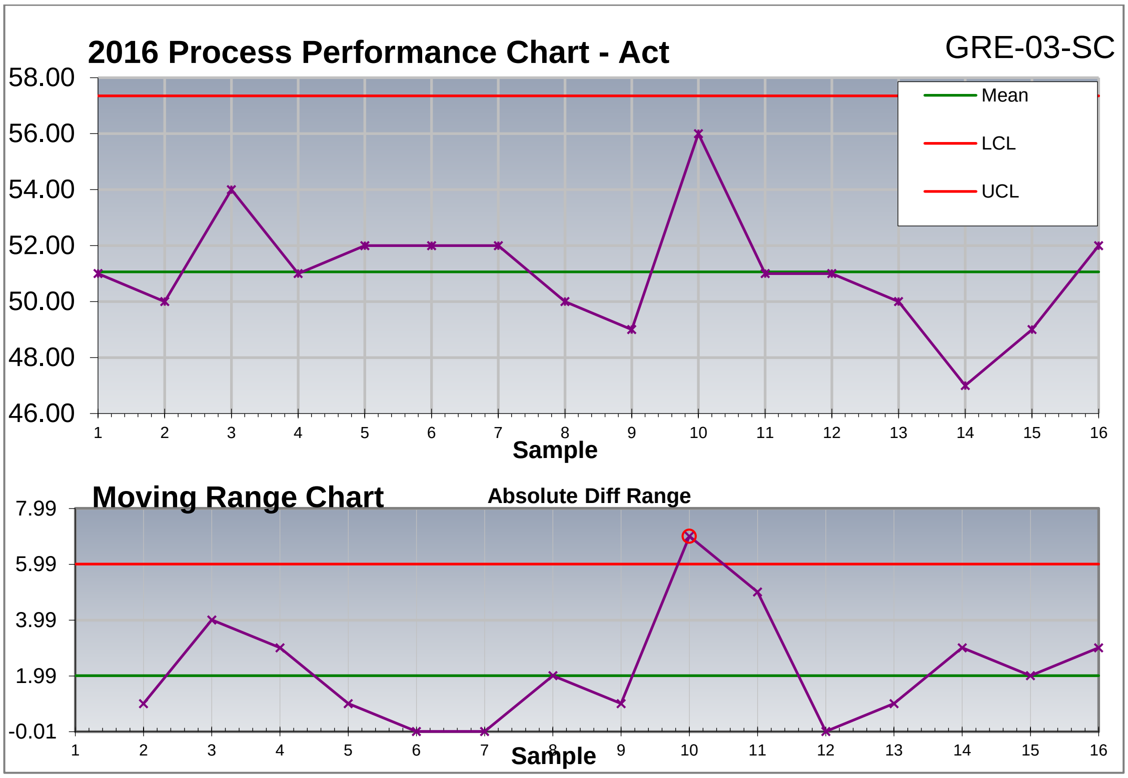
11.3.3.4 Scandium CRMs

The pre-2014 routine sample submissions to Actlabs were primarily focused on Nb_2O_5, which predates the 2016 changes in the metallurgical flowsheet for TiO_2 and Sc recovery. The revised focus on TiO_2 and Sc led to a re-assay program of the 2011 sample pulps which had not previously been analyzed for TiO_2 or Sc (see Table 11-13). The results for Sc CRM are demonstrated in Figure 11-21 and Figure 11-22.

Table 11-13: Summary of SC CRM

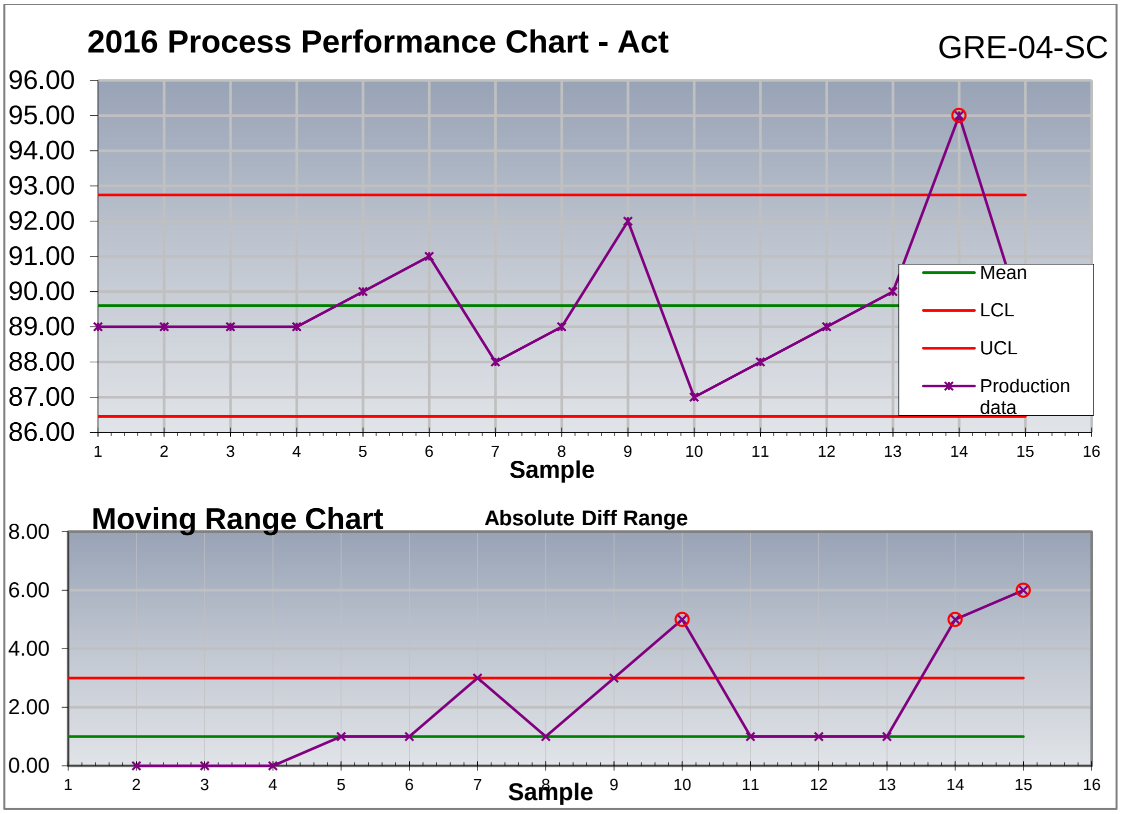
Element Sc - Actlabs 2016	Drill Program
Lab Package: 8-REE (FUS-ICP/FUS-MS)	2016 Re-Sampling of Historic Molycorp Holes
# of CRM GRE-03 Material Sent to Lab (49.85) MEDIUM GRADE	16
# of CRM GRE-04 Material Sent to Lab (88.36) HIGH GRADE	15
Total CRM sent to the lab (TiO_2)	31
Total Error Rate	3.2%

Source: Nordmin, 2019



Source: Nordmin, 2019

Figure 11-21: Summary of 2016 GRE-03-Sc Control Chart



Source: Nordmin, 2019

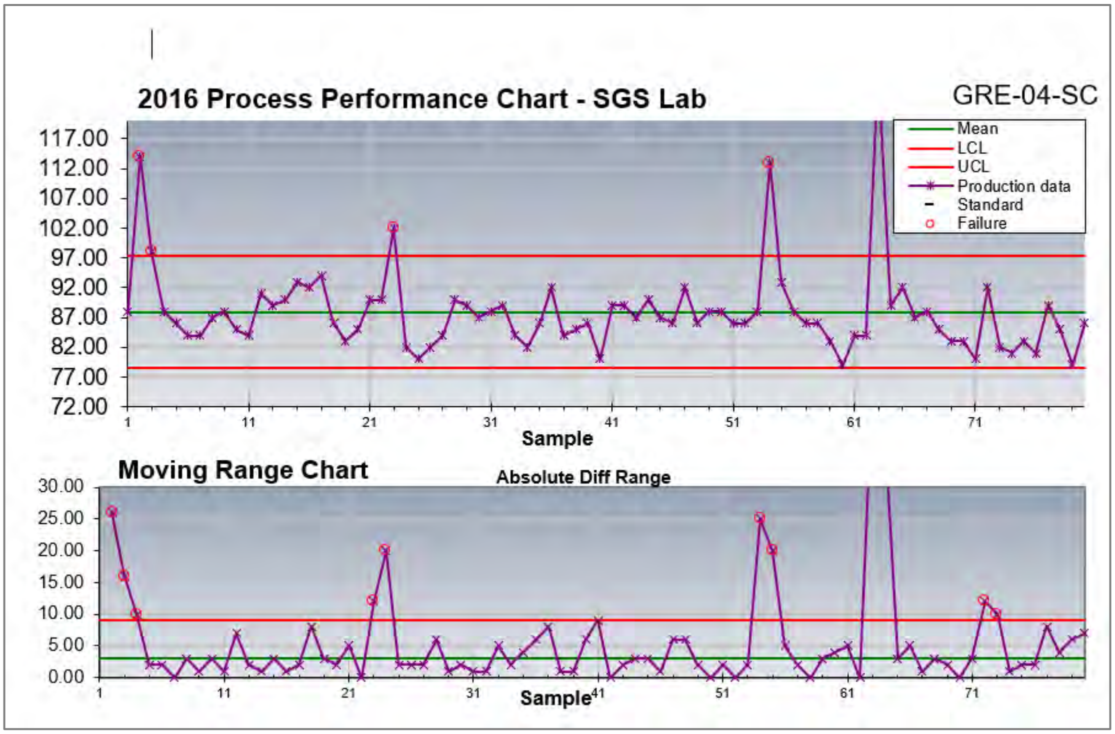
Figure 11-22: Summary of 2016 GRE-04-Sc Control Chart

A total of 80 SC CRM were inserted at SGS (see Table 11-14). The results indicated a good correlation between SGS Labs' scandium values and fell well above and below the expected grade. A total of five samples reported above the guideline line of 3 standard deviations during the study (see Figure 11-23).

Table 11-14: Secondary Lab (SGS) CRM Check for Scandium

Element Sc - SGS Labs	**Drill Program**		
Lab Package: GE JCP90A	**Historic**	**2011**	**2014**
# of Assays sent to Lab	9,016	1,800	9,653
# of CRM GRE-04 Material sent to the lab (88.36 ppm) HIGH GRADE	67	n/a	13

Source: Nordmin, 2019



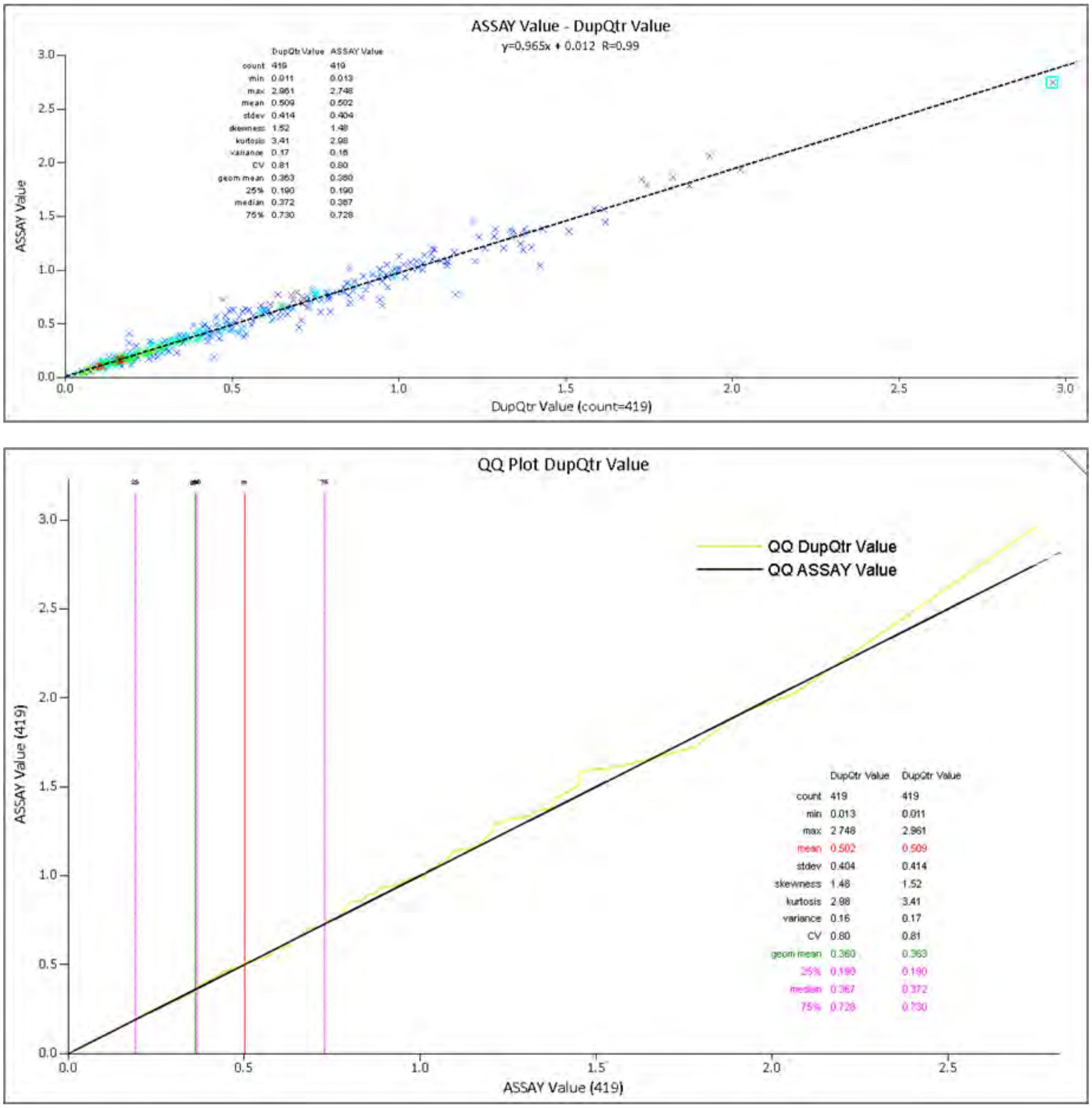
Source: Nordmin, 2019

Figure 11-23: Summary of 2016 GRE-04-Sc Control Chart

11.3.3.5 Field ¼ Core Duplicates

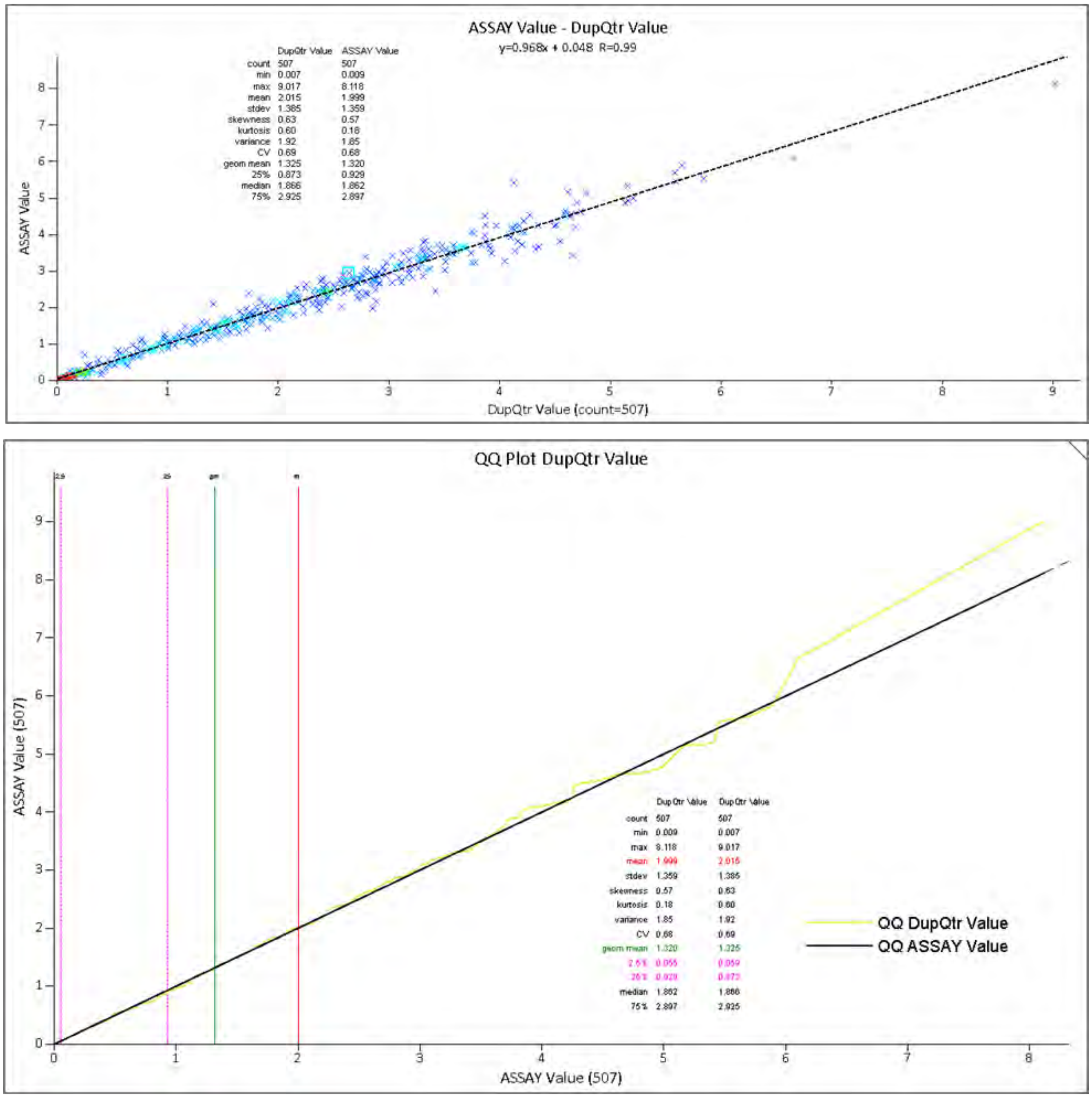
A total of 419 field duplicate samples comprised of 1/4 core were resubmitted to Actlabs as part of the routine sample submission from DDH samples, which represent 4.3% of total sample submissions from the 2014 drilling program.

Nordmin has compared the base statistics for the two datasets and found the difference in the mean grades to be 1.3% for Nb_2O_5, 1.0% for TiO_2, and 0.4% for Sc (ppm), which indicates an acceptable level of precision at the laboratory. The results are shown in Figure 11-24, Figure 11-25 and Figure 11-26, and indicate a reasonable comparison between the original and duplicate assays.



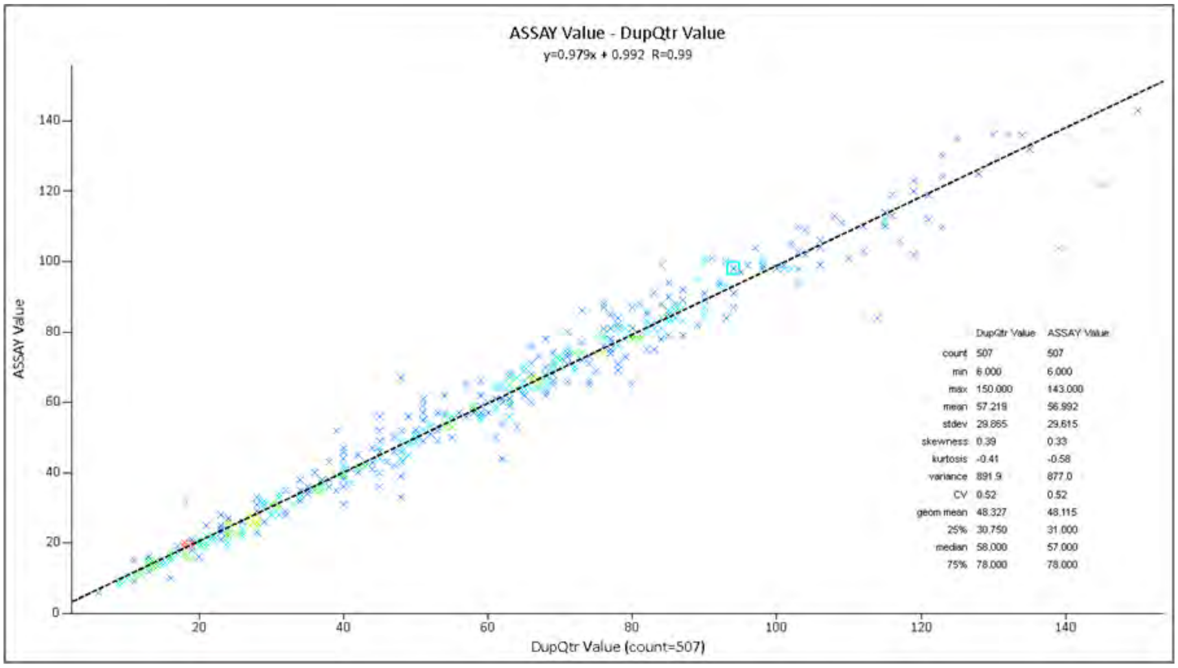
Source: Nordmin, 2019

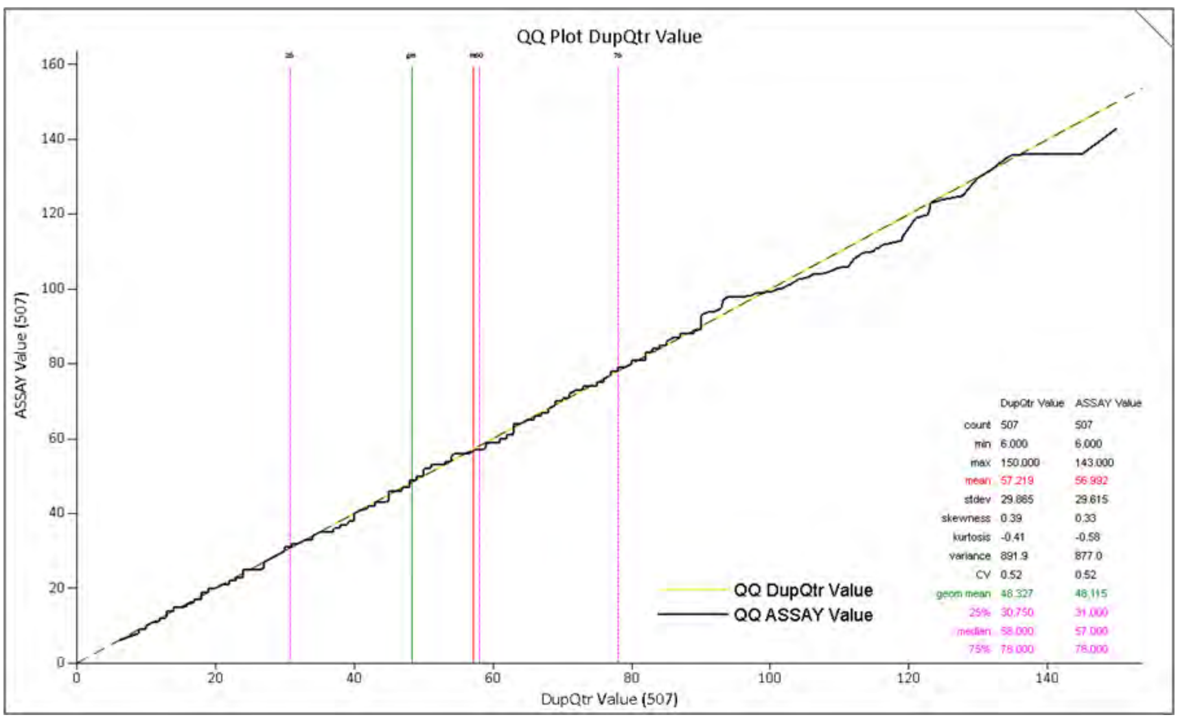
Figure 11-24: XY Scatter and QQ Plot Showing a Comparison of Original Versus Field Duplicate Analysis Nb_2O_5



Source: Nordmin, 2019

Figure 11-25: XY Scatter and QQ Plot Showing a Comparison of Original Versus Field Duplicate Analysis TiO_2



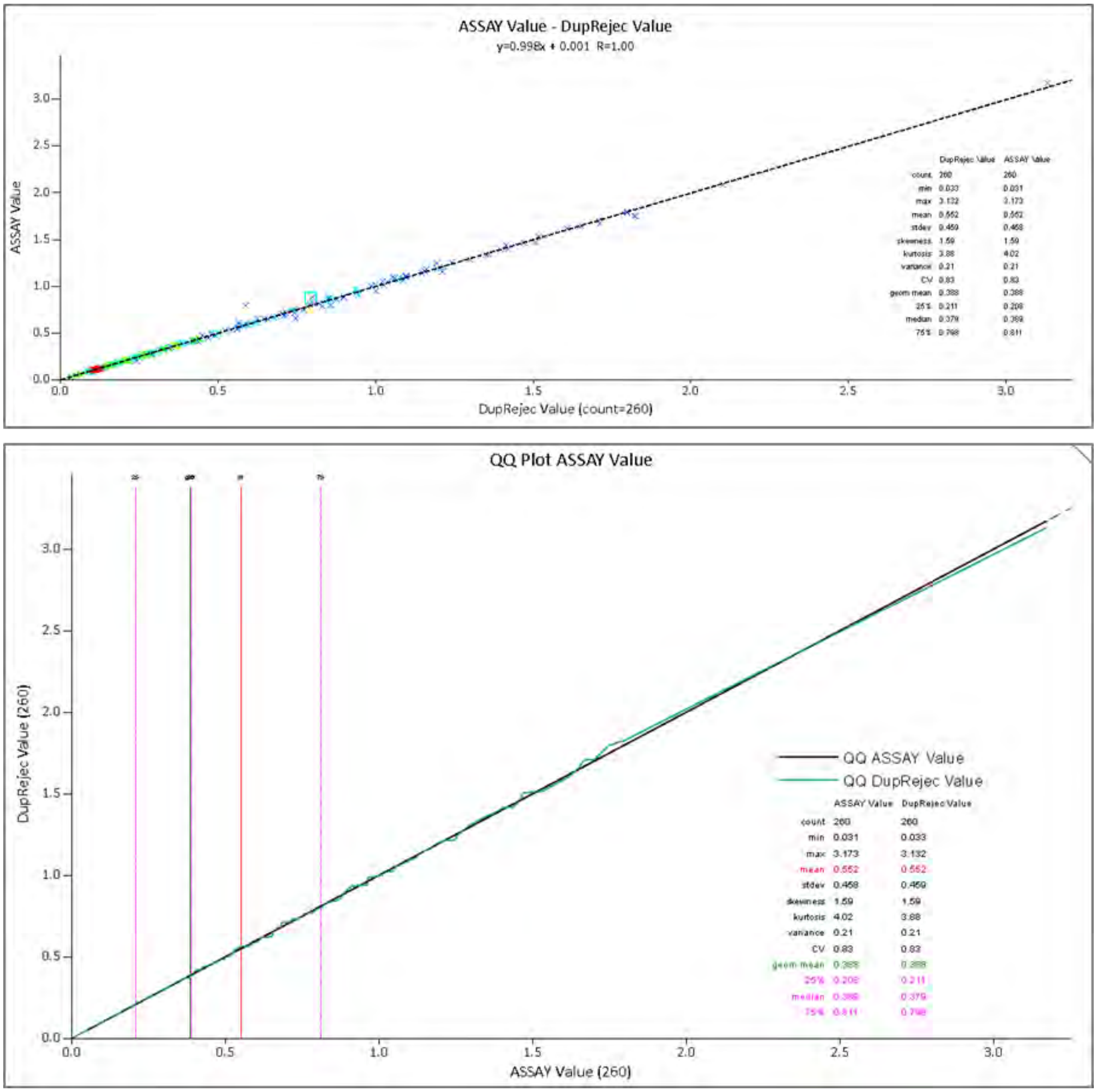


Source: Nordmin, 2019

Figure 11-26: XY Scatter and QQ Plot Showing a Comparison of Original Versus Field Duplicate Analysis Sc (ppm)

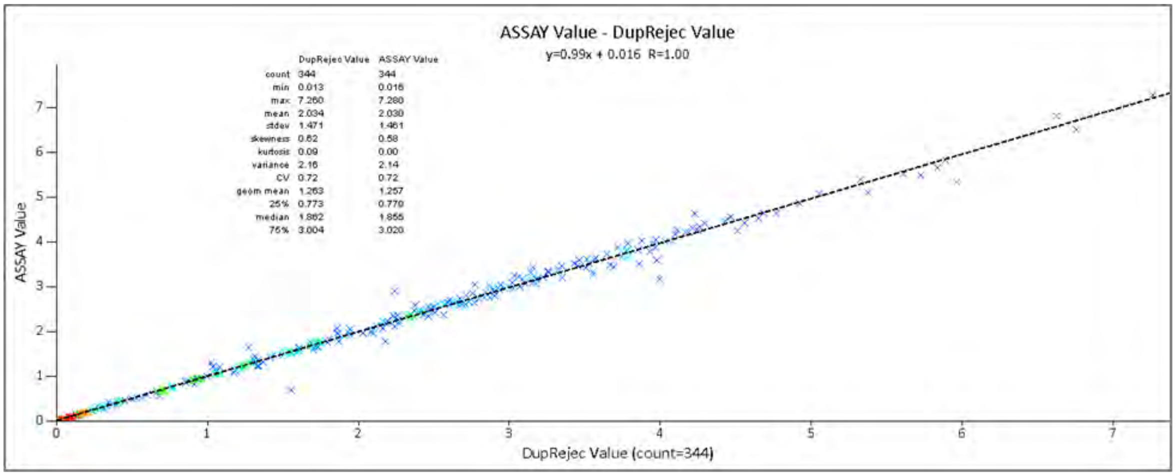
11.3.4 Reject Duplicates

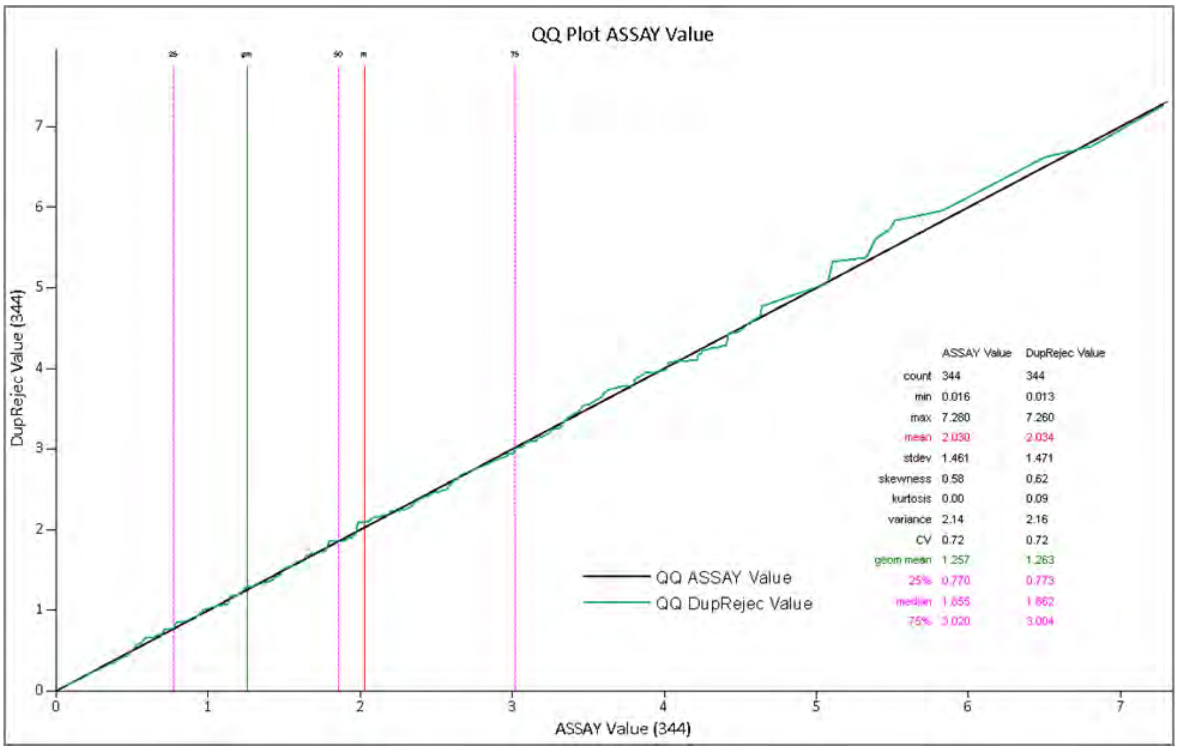
A second riffled split sample of 260 reject duplicate samples, taken after crushing, were submitted to Actlabs for re-analysis (blind) as part of the routine sample submission from diamond drill hole samples, which represent 2.7% of the total sample submissions from the 2014 drilling program. The results are shown in Figure 11-27, Figure 11-28 and Figure 11-29, and indicate a reasonable comparison between the original and duplicate assays. Nordmin compared the base statistics or the two datasets and found the difference in the mean grades to be 0.0% for Nb_2O_5, 0.1% for TiO_2, and 0.4% Sc (ppm), which indicates an acceptable level of precision at the laboratory.



Source: Nordmin, 2019

Figure 11-27: XY Scatter and QQ Plot Showing a Comparison of Original Versus Reject Duplicate (Riffle Split) Analysis Nb_2O_5



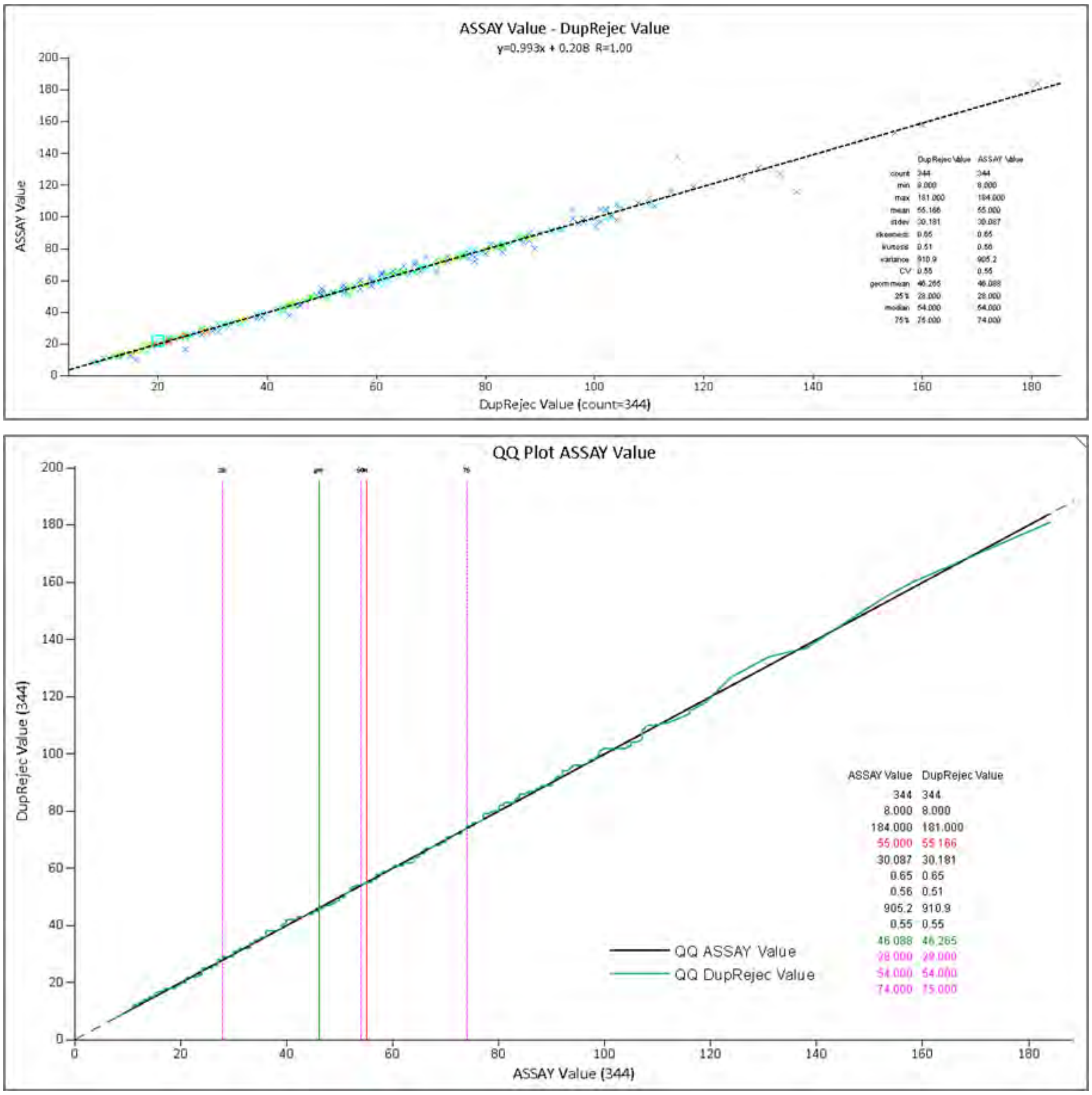


Source: Nordmin, 2019

Figure 11-28: XY Scatter and QQ Plot Showing a Comparison of Original Versus Reject Duplicate (Riffle Split) Analysis TiO_2

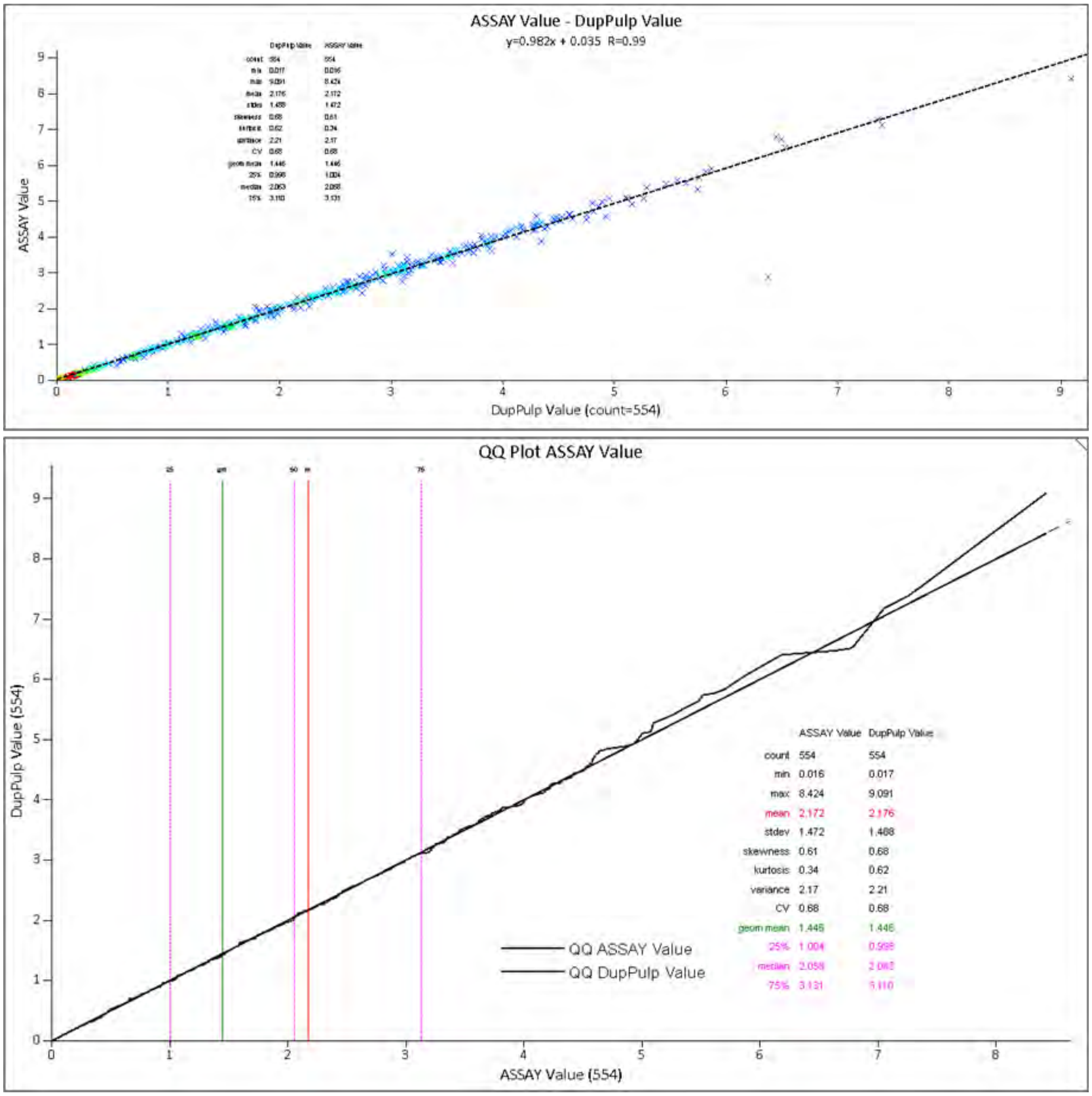
11.3.5 Field Pulp Duplicates

A second riffled sample split of 468 Nb_2O_5 pulp duplicate samples and 554 TiO_2 and Sc pulp duplicate samples comprising, taken after pulverization, were submitted to Actlabs as part of the routine sample submission from diamond drilling samples, which represent ~4.9% of total sample submissions from the 2014 drilling program. The results are shown in Figure 11-29, Figure 11-30 and Figure 11-31, and indicate a reasonable comparison between the original and duplicate assays. Nordmin has also compared the base statistics for the two datasets and found the difference in the mean grades to be 0.4% for Nb_2O_5, 0.2% for TiO_2, and 0.1% Sc (ppm), which indicates an acceptable level of precision at the laboratory.



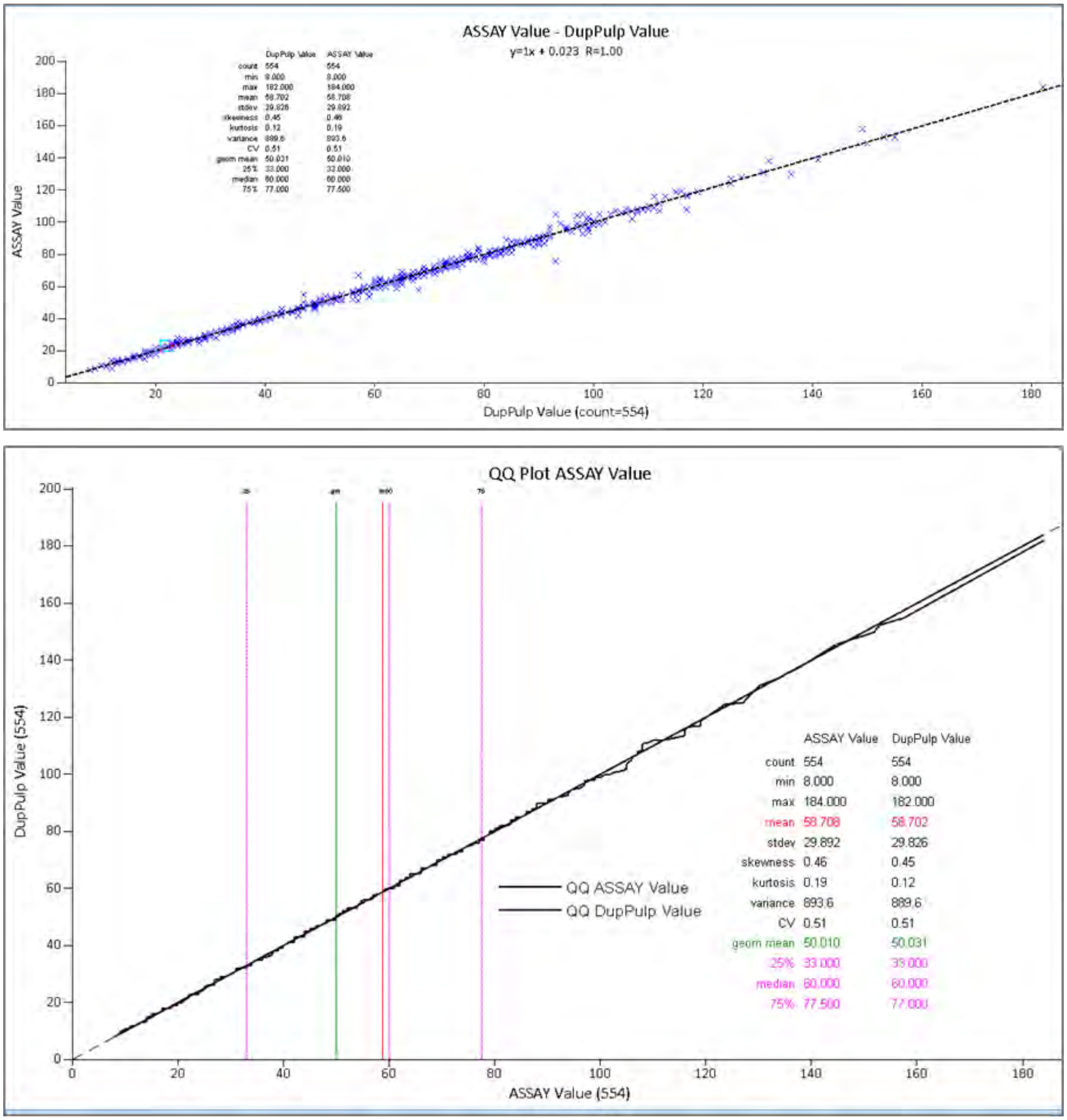
Source: Nordmin, 2019

Figure 11-29: XY Scatter and QQ Plot Showing a Comparison of Original Versus Reject Duplicate (Riffle Split) Analysis Sc (ppm)



Source: Nordmin, 2019

Figure 11-30: XY Scatter and QQ Plot Showing a Comparison of Original Versus Pulp Duplicate Analysis TiO_2



Source: Nordmin, 2019

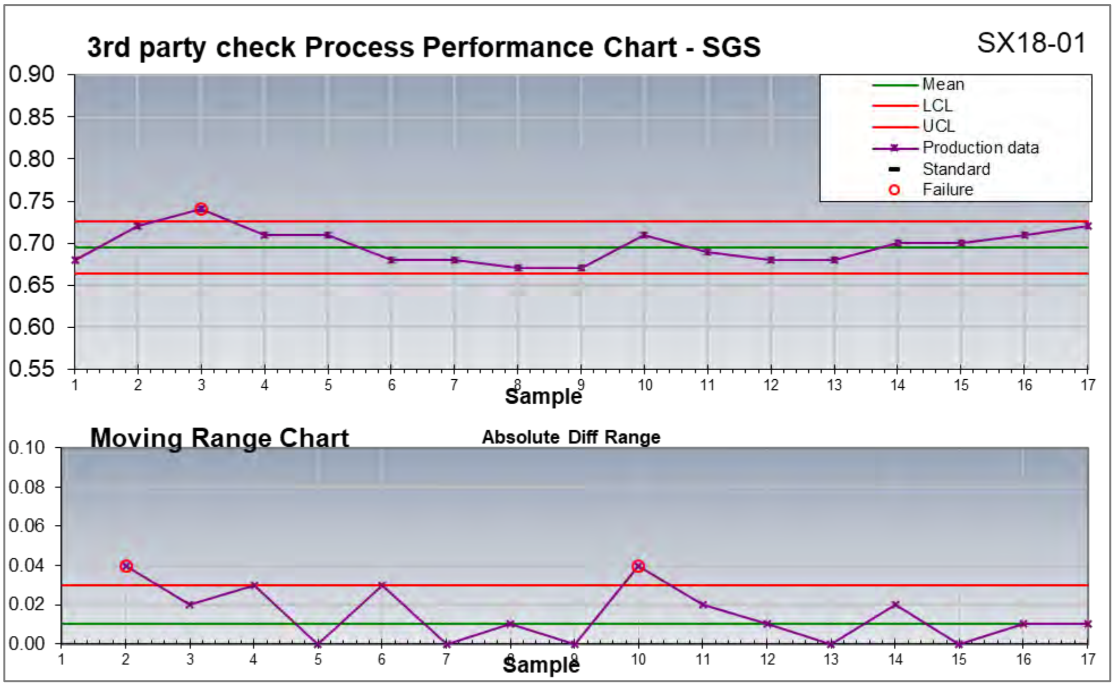
Figure 11-31: XY Scatter and QQ Plot Showing a Comparison of Original Versus Pulp Duplicate Analysis Sc (Ppm)

Nordmin has reviewed all the data available using primarily XY scatter plots and QQ-Plots and concludes that no significant issues regarding the precision exist from the Actlabs assays in the database. All phases of the sample preparation display strong correlations between the original and duplicate assays.

11.3.6 Third-Party Duplicate Check Analysis, SGS Versus Actlabs

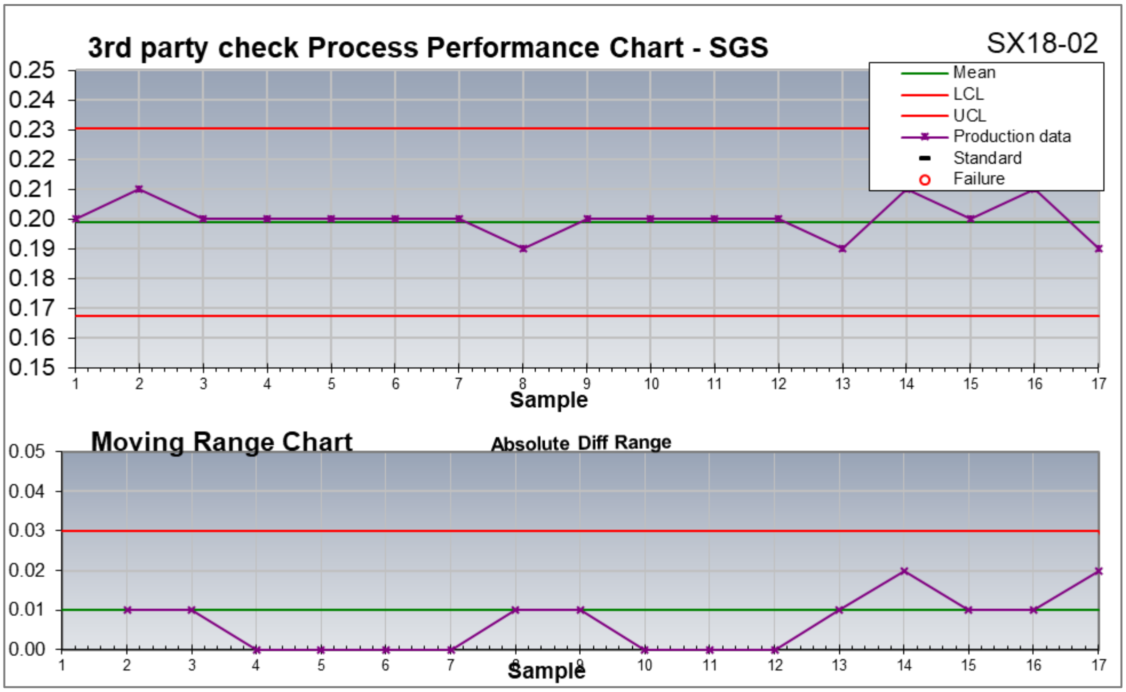
A total of 462 pulp duplicate samples comprising a second riffled sample split of pulverized material, taken at the same time of extraction as the primary pulps, were submitted as part of the routine sample submission to a check laboratory (SGS). The total number of samples represents approximately 5% of the original submissions.

The CRM material submitted to SGS returned assays within the range of CRM values similar to Actlabs for both Nb_2O_5 (see Figure 11-32, Figure 11-33 and Figure 11-34) and TiO_2 (see Figure 11-35). However, the SGS dataset is limited to 49 standard submissions versus 492 Actlabs submissions and the previous bias noted has not yet been resolved. The charts indicate that similar to Actlabs, the CRMs returned values within the upper and lower limits of the assigned grades for Nb_2O_5 and TiO_2 and have a slightly improved distribution between the upper and lower limits. In Nordmin's opinion, both laboratories provide sufficient accuracy for Indicated Mineral Resources.



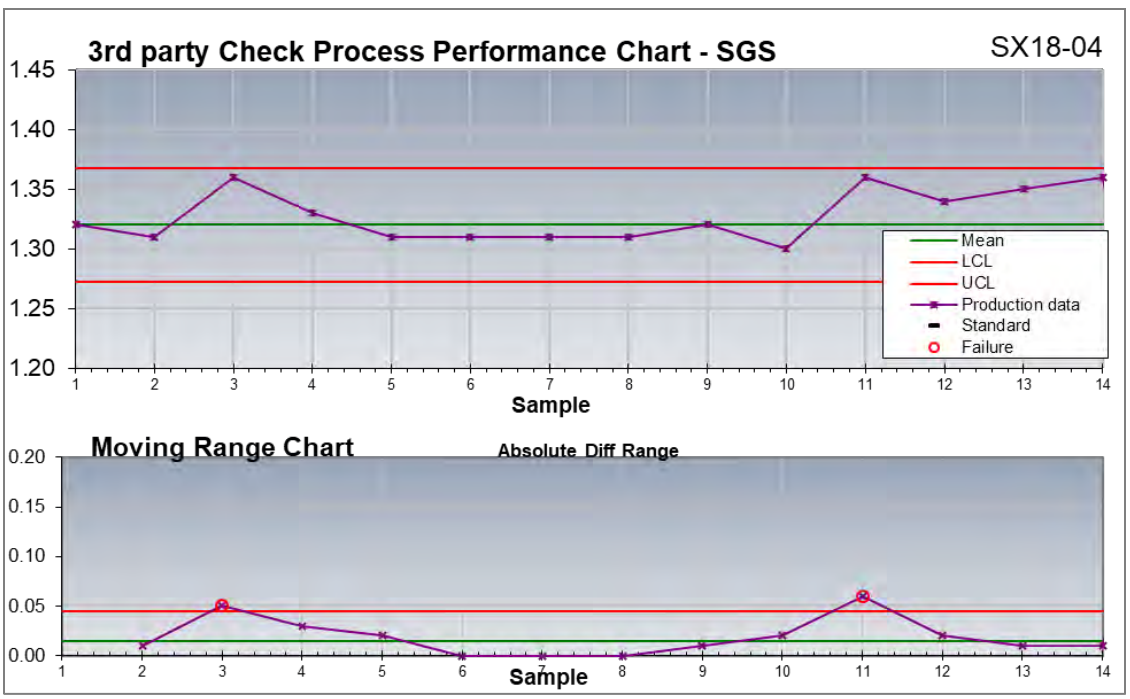
Source: Nordmin, 2019

Figure 11-32: Summary of SX18-01 Nb_2O_5 Control Chart



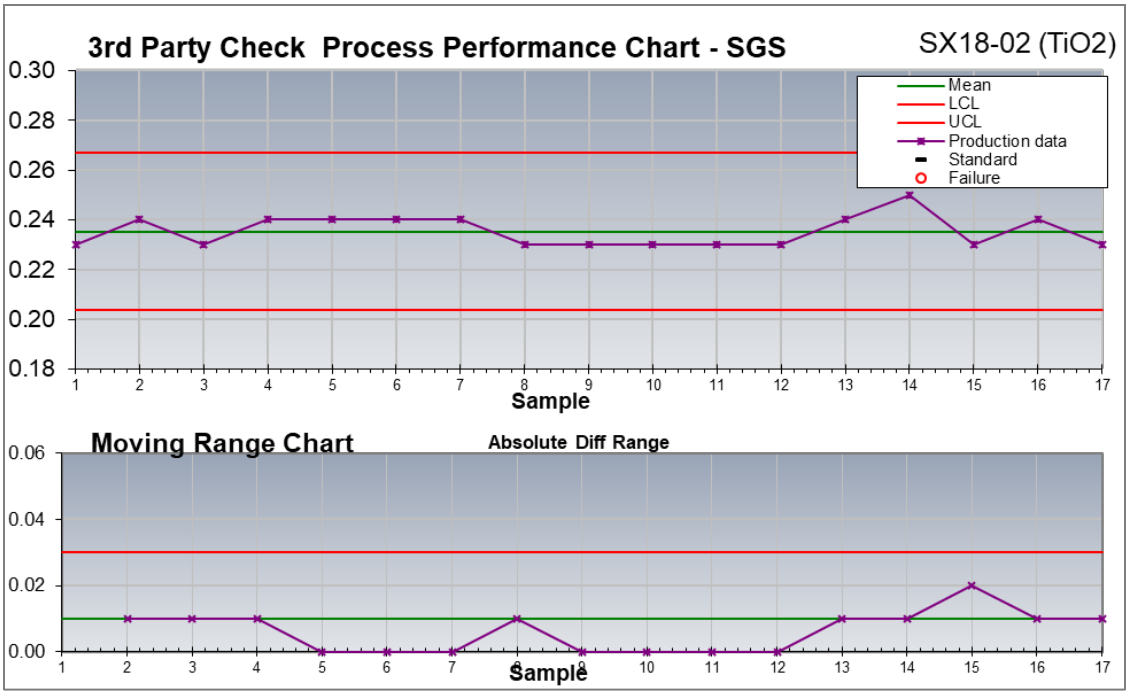
Source: Nordmin, 2019

Figure 11-33: Summary of SX18-02 Nb_2O_5 Control Chart



Source: Nordmin, 2019

Figure 11-34: Summary of SX18-04 Nb_2O_5 Control Chart



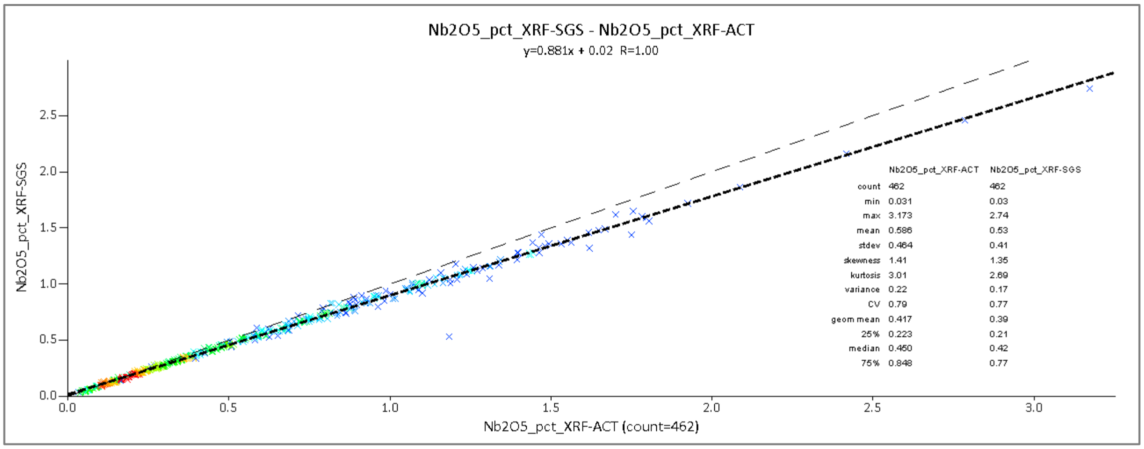
Source: Nordmin, 2019

Figure 11-35: Summary of SX18-02 TiO_2 Control Chart

A review of the XY scatter plot (see Figure 11-36) and box plot (see Figure 11.37) for Nb_2O_5 demonstrates Actlabs reporting consistently higher Nb_2O_5 grades greater than 0.5%. The difference increases slightly with samples that are greater than 1%, but the sample dataset is too small to determine the significance. A review of the duplicate comparisons for Nb_2O_5, TiO_2, and Sc between Actlabs and SGS does not indicate a similar high bias (see Figure 11-38 to Figure 11-40).

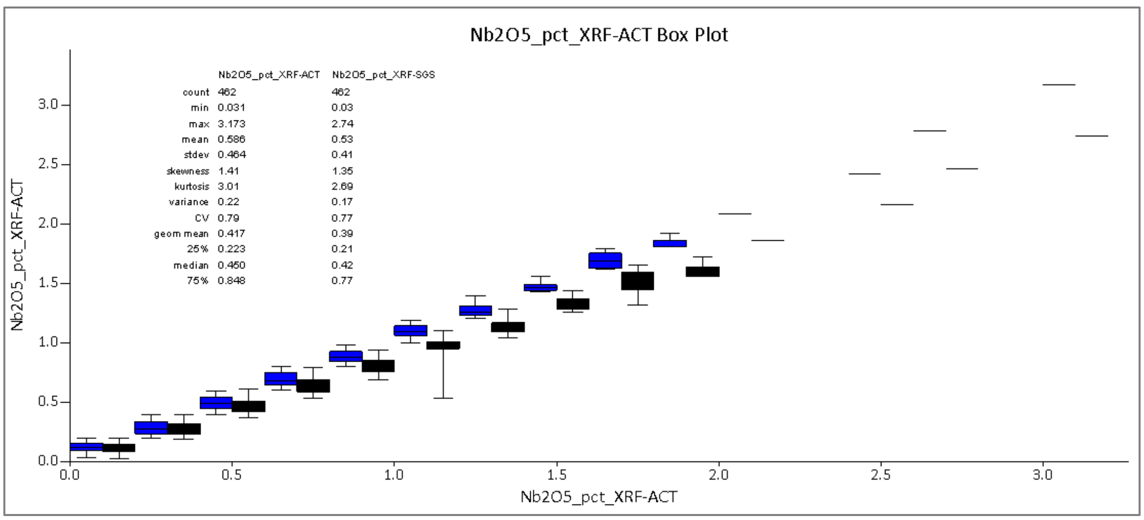
Therefore, Nordmin has assumed the following possible explanations for the continued high bias:

- An error could have been introduced during the manufacture of the CRMs.
- The method or equipment used by Actlabs may not be compatible with the use of a specific CRM. For example, SGS used a borate fusion with an XRF finish, compared to Actlabs' use of a lithium metaborate/tetraborate fusion, which Actlabs state provides improved reliability.
- Further follow-up work is required to determine the cause of the bias as the project continues to develop, and further definition drilling is required.



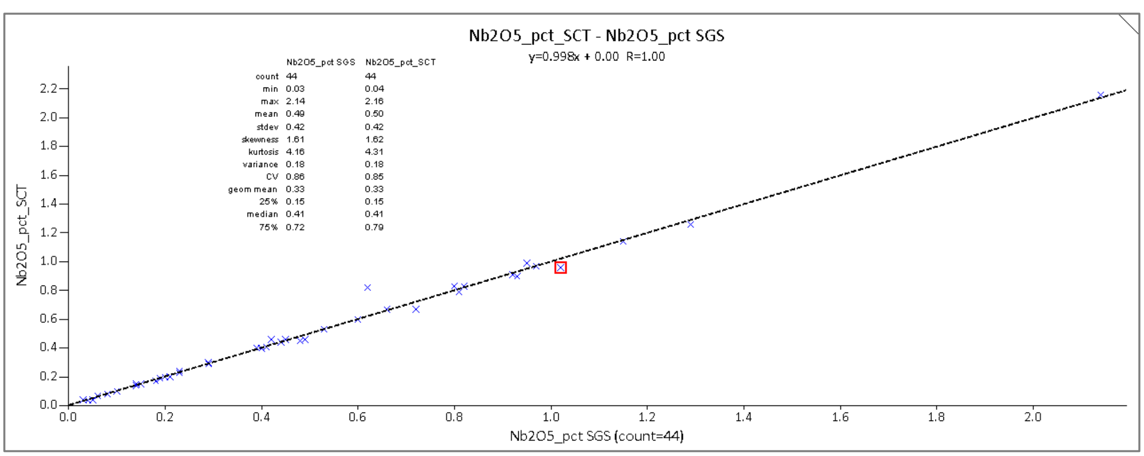
Source: Nordmin, 2019

Figure 11-36: Scatter Plot of Nb_2O_5 Actlabs Versus SGS Labs



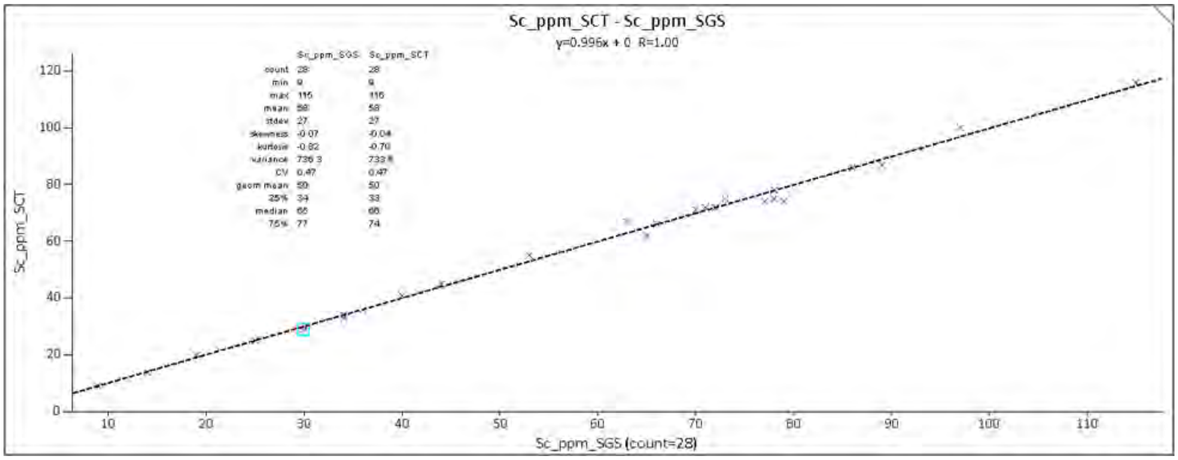
Source: Nordmin, 2019

Figure 11-37: Box Plot of Nb_2O_5 Actlabs Versus SGS Labs



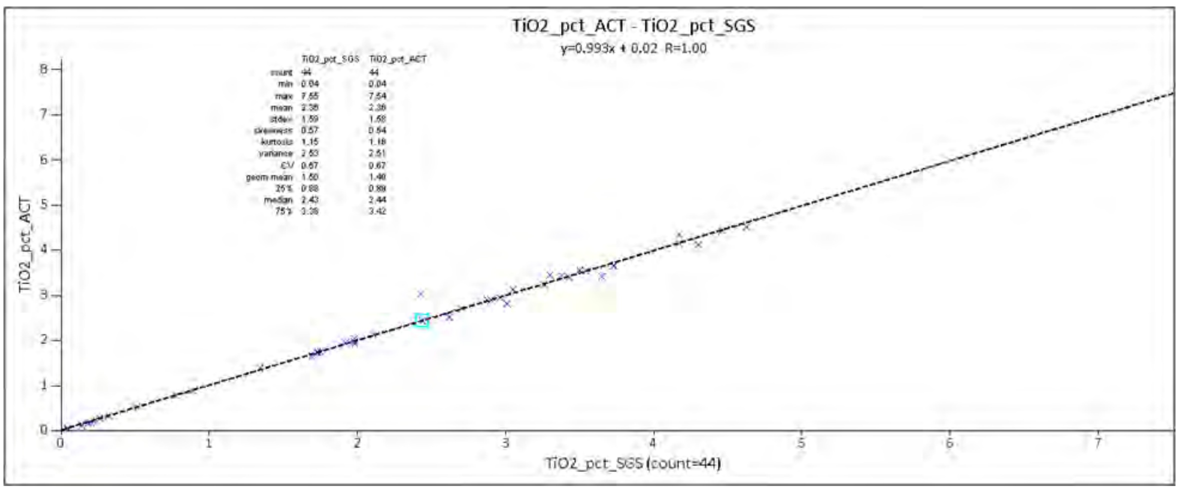
Source: Nordmin, 2019

Figure 11-38: Scatter Plot of Nb_2O_5 Duplicates Actlabs Versus SGS Labs



Source: Nordmin, 2019

Figure 11-39: Scatter Plot of Sc Duplicates Actlabs Versus SGS Labs



Source: Nordmin, 2019

Figure 11-40: Scatter Plot of TiO_2 Duplicates Actlabs Versus SGS Labs

11.3.7 Qualified Person's Opinion on the Adequacy of Sample Preparation, Security and Analytical Procedures

Nordmin has been supplied with all raw QA/QC data and has reviewed and completed an independent check of the results for all NioCorp sampling programs. It is Nordmin's opinion that the sample preparation, security and analytical procedures used by NioCorp are consistent with standard industry practices and that the data is suitable for the 2019 Mineral Resource Estimate. Nordmin identified several further recommendations (see Section 26) to NioCorp to ensure the continuation of a robust QA/QC program but has noted that there are no material concerns with the geological or analytical procedures used or the quality of the resulting data.

12. DATA VERIFICATION

12.1 Nordmin 2019

Nordmin completed several data validation checks throughout the duration of the 2019 Mineral Resource Estimate. The verification process included a two-day site visit to the Elk Creek Project property by the Nordmin Qualified Persons (independent) to review drill core geology, geological procedures, chain of custody of drill core, sample pulps and for the collection of independent samples for metal verification. Data verification included a survey spot check of drill collars, a spot check comparison of Nb_2O_5, TiO_2 and Sc assays from the drill hole database against original assay records (lab certificates), spot check of drill core lithologies recorded in the database versus the core located in the core storage shed and a review of QA/QC performance of the drill programs. Nordmin also completed additional data analysis and validation, as outlined in Section 14.

12.2 Nordmin Site Visit 2018

A site visit to the Elk Creek property was carried out December 11 and 12, 2018 by Glen Kuntz, P.Geo., QP for Mineral Resources and Jean-Francois St-Onge, P.Eng., QP for Mineral Reserves. The site visit included:

- Review of the geological and geographical setting of the Project.
- Review and inspection of the site geology, mineralization and structural controls on mineralization.
- Review of the drilling, logging, sampling, analytical and QA/QC procedures.
- Review of the chain of custody of samples from the field to the assay lab.
- Review of the drill logs, drill core, storage facilities and independent assay verification on selected core samples.
- Confirmation of the drill hole collar locations.
- Assessment of logistical aspects, potential shaft locations and other practicalities relating to the property.
- Review of the structural measurements recorded form the televiewer logs and how these measurements are utilized within the 3D structural model.
- Validation of a portion of the drill hole database.

The Dahrouge geologists completed the initial core logging and sampling associated with the 2014 and 2015 work programs, therefore, Nordmin relied on Dahrouge's database to review the core logging procedures, collection of samples, chain of custody associated with the 2014/15 programs. NioCorp and Dahrouge provided Nordmin with excerpts from the drill database (Datamine Fusion) for the Project and electronic copies of the original logging and assay reports.

Dahrouge employed a rigorous QA/QC protocol including the routine insertion of field duplicates, laboratory pulp duplicates, blanks and niobium certified reference standards. Nordmin was provided with an excerpt from the database for review.

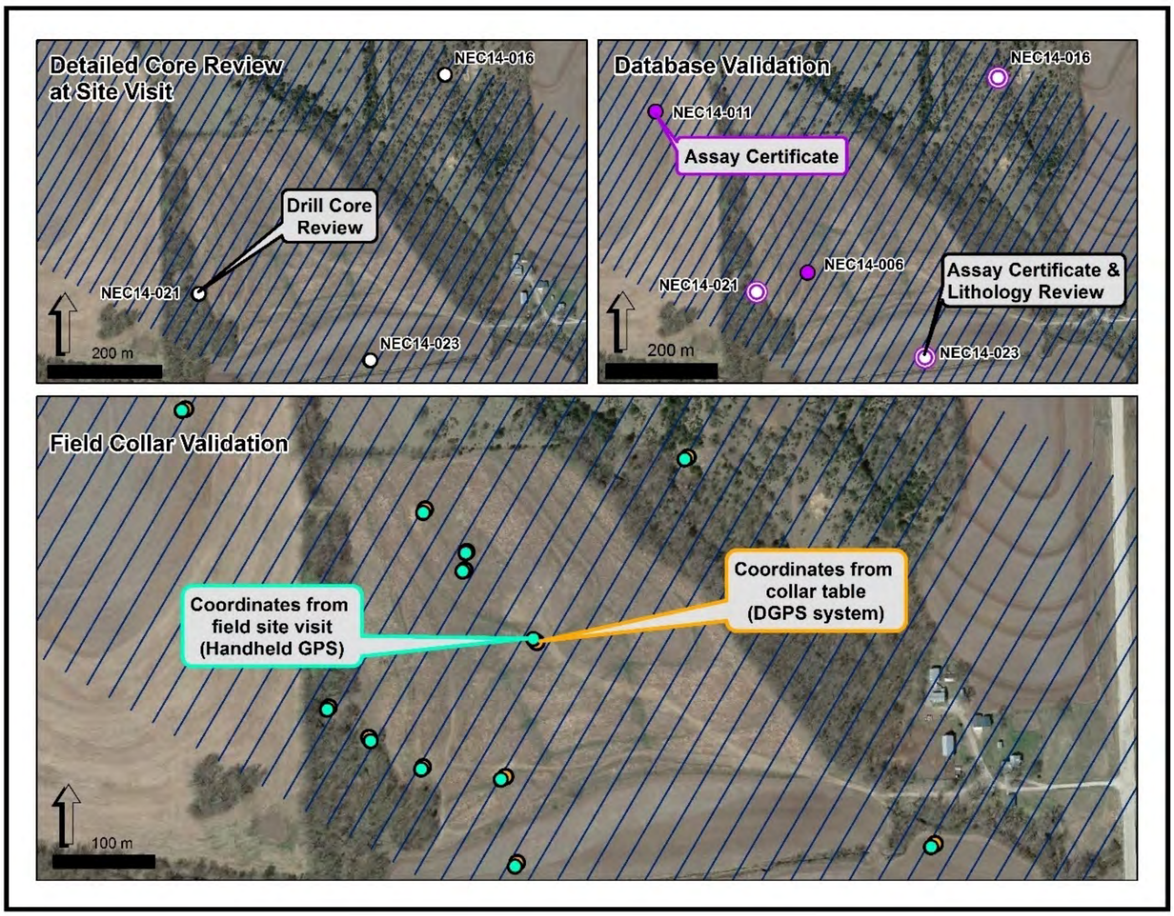
The collection and use of the structural information were reliable, and representative of the structure features being drilled. This was found to be consistent with industry standards and in accordance with NioCorp's internal procedural documentation.

No significant issues were identified during the site visit. Nordmin was accompanied by the Dahrouge geologists who have been involved with the Project since 2014.

- The geological data collection procedures and the chain of custody were found to be consistent with industry standards and in accordance with NioCorp's internal procedural documentation.
- Nordmin was able to verify the quality of geological and sampling information and develop an interpretation of niobium, titanium and scandium grade distributions appropriate to use in the Mineral Resource model.

12.2.1 Core Review Validation

The Qualified Persons completed a detailed review of three drill holes while onsite. Each of these holes was used within the resource estimate (see Figure 12-1). The core for each hole was laid out on a series of wooden horses to allow for reviewing the core from a lithological, structural, alteration and mineralization perspective (see Figure 12-3 and Figure 12-4).



Source: Nordmin, 2019

Figure 12-1: Locations of Drill Holes Validated Via Core Review, Database Validation or Field Collar Corroboration

Source: Nordmin, 2019

Figure 12-2: Detailed Core Review at the Elk Creek Project, Nebraska (A)

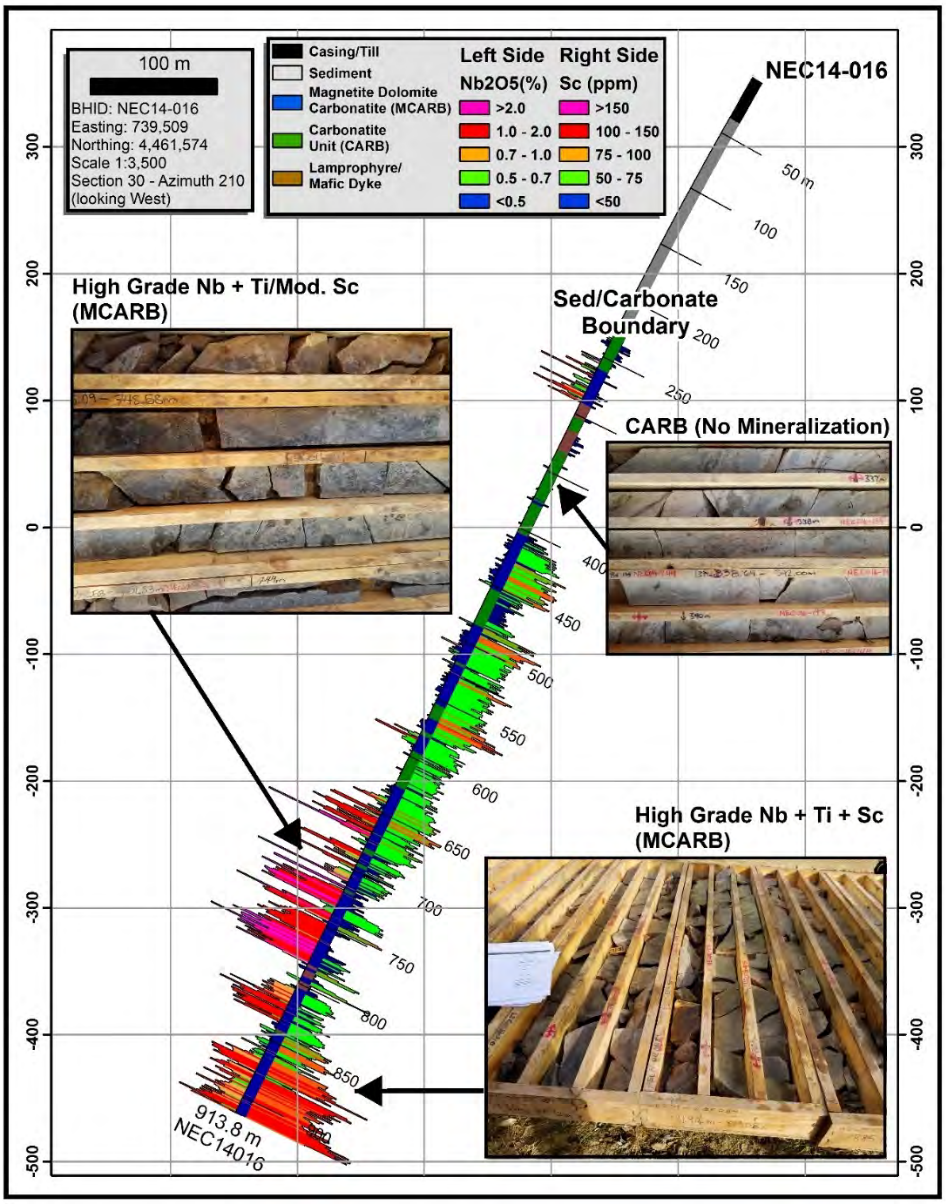
Source: Nordmin, 2019

Figure 12-3: Detailed Core Review at the Elk Creek Project, Nebraska (B)

Figure 12-4 demonstrates the four main lithological, structural and mineralized zones in DDH NEC14-016 observed by Nordmin. The drill holes sections reviewed were:

1. Limestone (sediment)/carbonatite boundary
2. Non-mineralized to low grade carbonatite
3. High grade Nb_2O_5, TiO_2 combined with low grade Sc
4. High grade Nb_2O_5, TiO_2 and Sc

Each of these areas was inspected for any material differences concerning the lithology, structure, alteration and mineralization, through a comparison of the drill log record (Fusion database) to the visual inspection of the core. DDH NEC14-016 compares well to the information recorded in the drill logs.



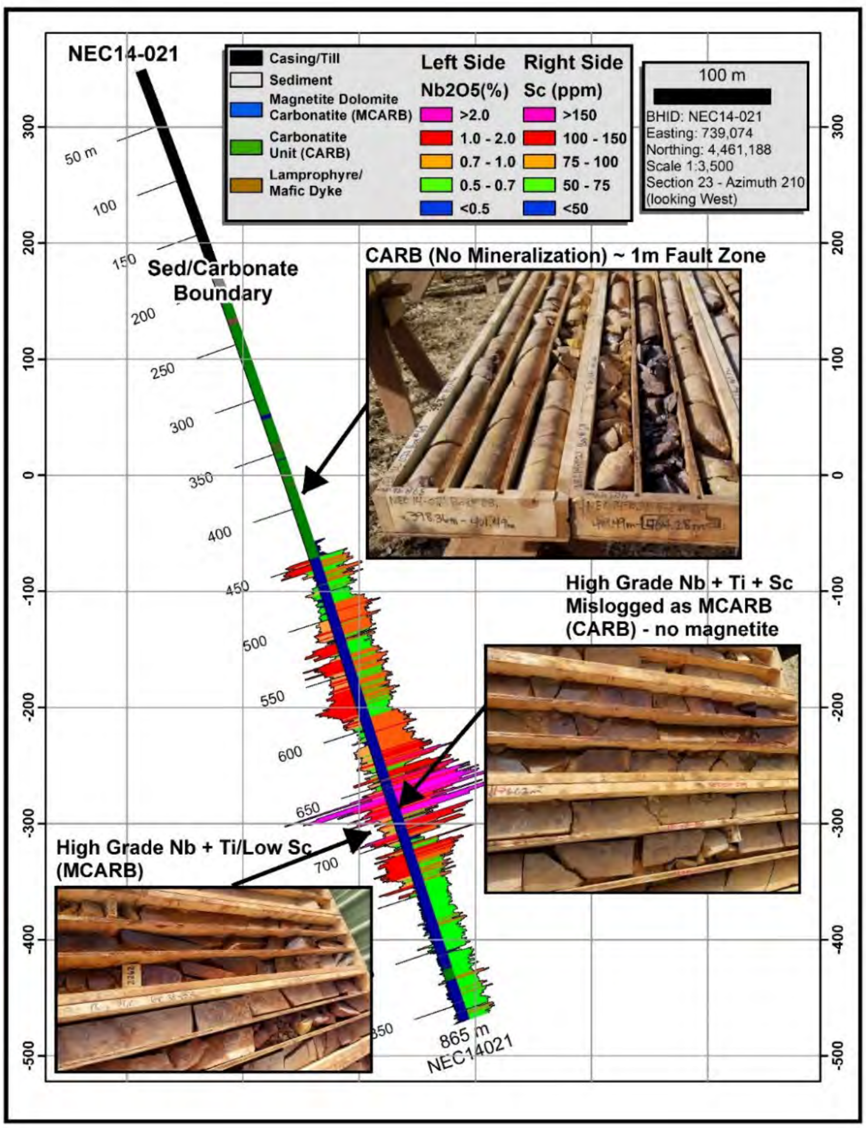
Source: Nordmin, 2019

Figure 12-4: DDH NEC14-016

Figure 12-5 demonstrates the four main lithological, structural and mineralized zones in DDH NEC14-021 observed by Nordmin. The drill holes sections reviewed were:

1. Limestone (sediment)/carbonatite boundary
2. Carbonatite, no mineralization but strongly faulted
3. High grade Nb_2O_5, TiO_2 and Sc
4. High grade Nb_2O_5, TiO_2 and low grade Sc

During the visual inspection of the core for DDH NEC14-021 it was observed that the zone which contains high grade Nb_2O_5, TiO_2 and Sc was not magnetic and therefore was mis-logged in the drill database for a section of approximately 20 m to 40 m of core length. This observation suggests that high grade mineralization can occur in carbonatite that is not magnetic. This may be a localized occurrence; a further investigation will be required once underground definition drilling has been initiated.



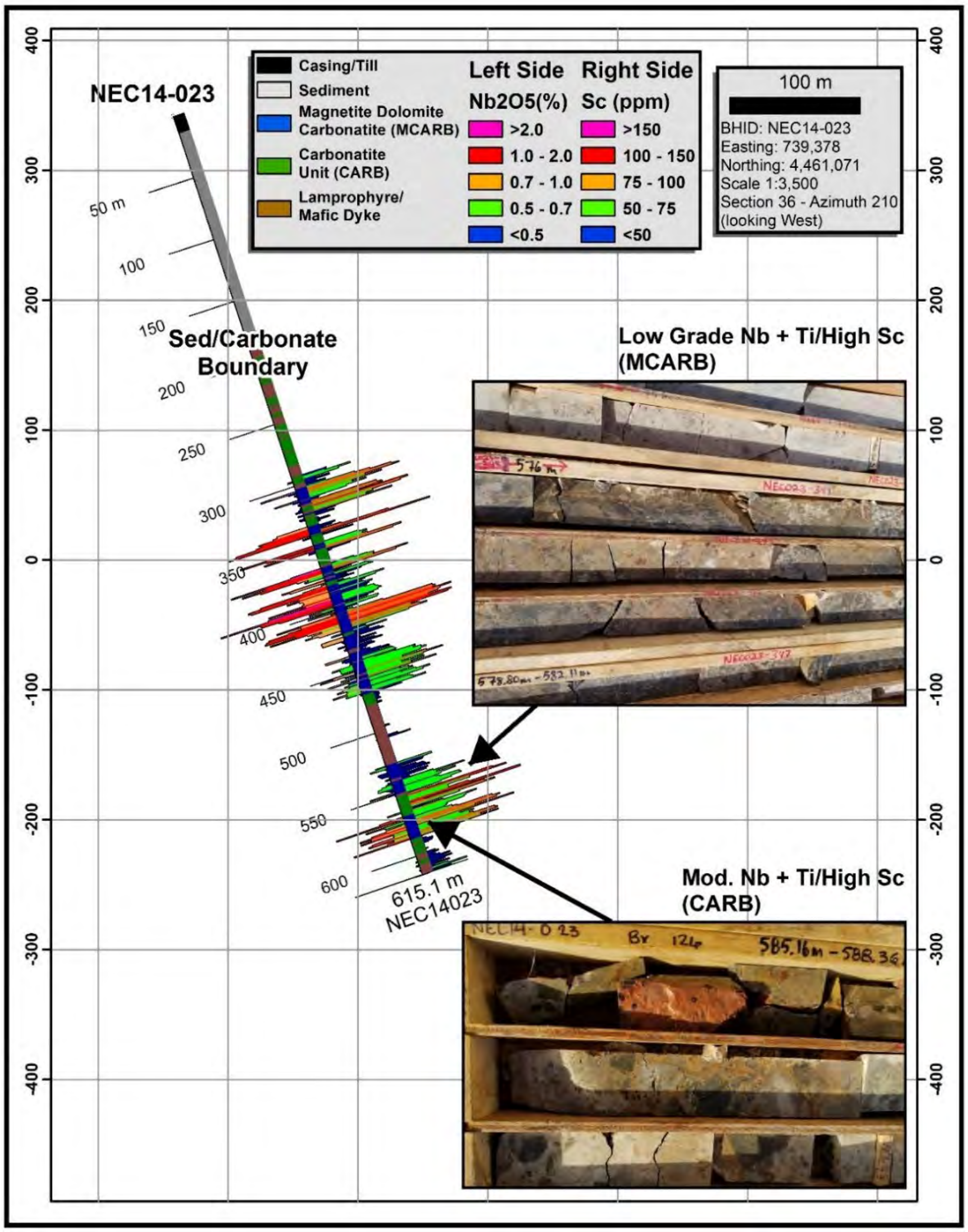
Source: Nordmin, 2019

Figure 12-5: DDH NEC14-021

Figure 12-6 demonstrates the three main lithological, structural and mineralized zones in DDH NEC14-023 observed by Nordmin. The drill holes sections reviewed were:

1. Limestone (sediment)/carbonatite boundary
2. Low grade Nb_2O_5, TiO_2 and relatively high grade Sc
3. Moderate grade Nb_2O_5, TiO_2 and high grade Sc

The drill logs for DDH NEC14-023 compared well to the three areas that were reviewed and demonstrated a consistent approach to the collection of drill hole information.



Source: Nordmin, 2019

Figure 12-6: DDH NEC14-023

12.2.2 Field Collar Validation

The Qualified Person confirmed the collar locations of 12 of 41 drill holes used within the resource estimate. The Qualified Persons collected the collar locations using a Garmin GPSMAP 62 handheld GPS unit versus the differential GPS (sub-centimetre accuracy) used in the NioCorp Database. Approximately 30% of the collar locations were checked, and all the collar locations were located within the acceptable error limit of the GPS unit (see Table 12-1).

Table 12-1: Field Check Comparing DDH Collar Coordinates Using a Handheld GPS Versus Database Coordinates

BHID	EAST	NORTH	ELEVATION	DEPTH	ORIGINAL COLLAR					VERIFICATION				
DATABASE					BHID	EASTING	NORTHING	ELEVATION	DEPTH	BHID	EAST	NORTH	ELEVATION	DEPTH
NEC14-008	739128.12	4461159.35	351.22	817.47	NEC14-008	739126	4461157	348	817.47	NEC14-008	2.12	2.35	3.22	0
NEC14-009a	739390.23	4461466.19	349.27	738.99	NEC14-009a	739388	4461464	349	738.99	NEC14-009a	2.23	2.19	0.27	0
NEC14-010	739209.51	4461149.76	347.78	726.03	NEC14-010	739207	4461148	345	726.03	NEC14-010	2.51	1.76	2.78	0
NEC14-011	738892.54	4461513.62	359.71	886.05	NEC14-011	738890	4461512	357	886.05	NEC14-011	2.54	1.62	2.71	0
NEC14-012	739635.1	4461083.41	339.89	897.03	NEC14-012	739633	4461082	341	897.03	NEC14-012	2.1	1.41	-1.11	0
NEC14-013	739169.32	4461354.33	355.17	796.14	NEC14-013	739167	4461353	353	796.14	NEC14-013	2.32	1.33	2.17	0
NEC14-014	739034.77	4461218.57	346.13	900.38	NEC14-014	739033	4461216	345	900.38	NEC14-014	1.77	2.57	1.13	0
NEC14-MET-03	739129.92	4461414.45	355.37	843.23	NEC14-MET-03	739128	4461412	354	843.23	NEC14-MET-03	1.92	2.45	1.37	0
NEC14-MET-01	739240.41	4461282.7	352.77	880.26	NEC14-MET-01	739238	4461285	351	880.26	NEC14-MET-01	2.41	-2.3	1.77	0
NEC14-MET-02	739171.12	4461372.42	355.81	900.99	NEC14-MET-02	739170	4461371	354	900.99	NEC14-MET-02	1.12	1.42	1.81	0
NEC14-021	739074.34	4461188.48	347.11	913.33	NEC14-021	739076	4461185	346	913.33	NEC14-021	-1.66	3.48	1.11	0
NEC14-015	739221.01	4461064.66	342.44	827.05	NEC14-015	739219	4461061	350	827.05	NEC14-015	2.01	3.66	-7.56	0

Source: Nordmin, 2019

12.2.3 Database Validation

The Nordmin Qualified Person completed a spot check verification on 386 (9.9%) of the lithologies and 2,718 (18.3%) of the assays. A summary of the data validation is listed in Table 12-2.

Table 12-2: Summary Validation of Assays and Lithologies

	Assays in Database Validated	Lithology in Database Validated
Total Number of Holes	41	41
Total Number of Assays/Litho Units	**14,021**	**4,031**
# of Borehole Verified	5	3
# of Assays or Litho Units Verified	2718	386
Verification Rate	19%	10%
Number of Errors	1	1
Error Rate	0.04%	0.26%

Source: Nordmin, 2019

Nordmin found two assay entry errors in the database and did not identify any material issues with the lab certificates.

12.3 Core Storage and Core Logging and Cutting Facilities

DDH holes, pulps and coarse rejects from various zones are archived in a storage facility at the Elk Creek project site in Nebraska (see Figure 12-7 and Figure 12-8).

Source: Nordmin, 2019

Figure 12-7: Core Storage in Secured Area at the Elk Creek Project, Nebraska

Source: Nordmin, 2019

Figure 12-8: Core Logging, Cutting and Storage of Pulps Facility Located at the Elk Creek Project Site, Nebraska

12.4 Independent Sampling

The Nordmin Qualified Person selected intervals from 12 holes from the 2014 drill program for validation sampling as listed in Table 12-3. A total of 61 sample pulps plus three control samples (two CRM standards and one blank) were taken. Nordmin elected to choose a variety of grade ranges from various drill holes.

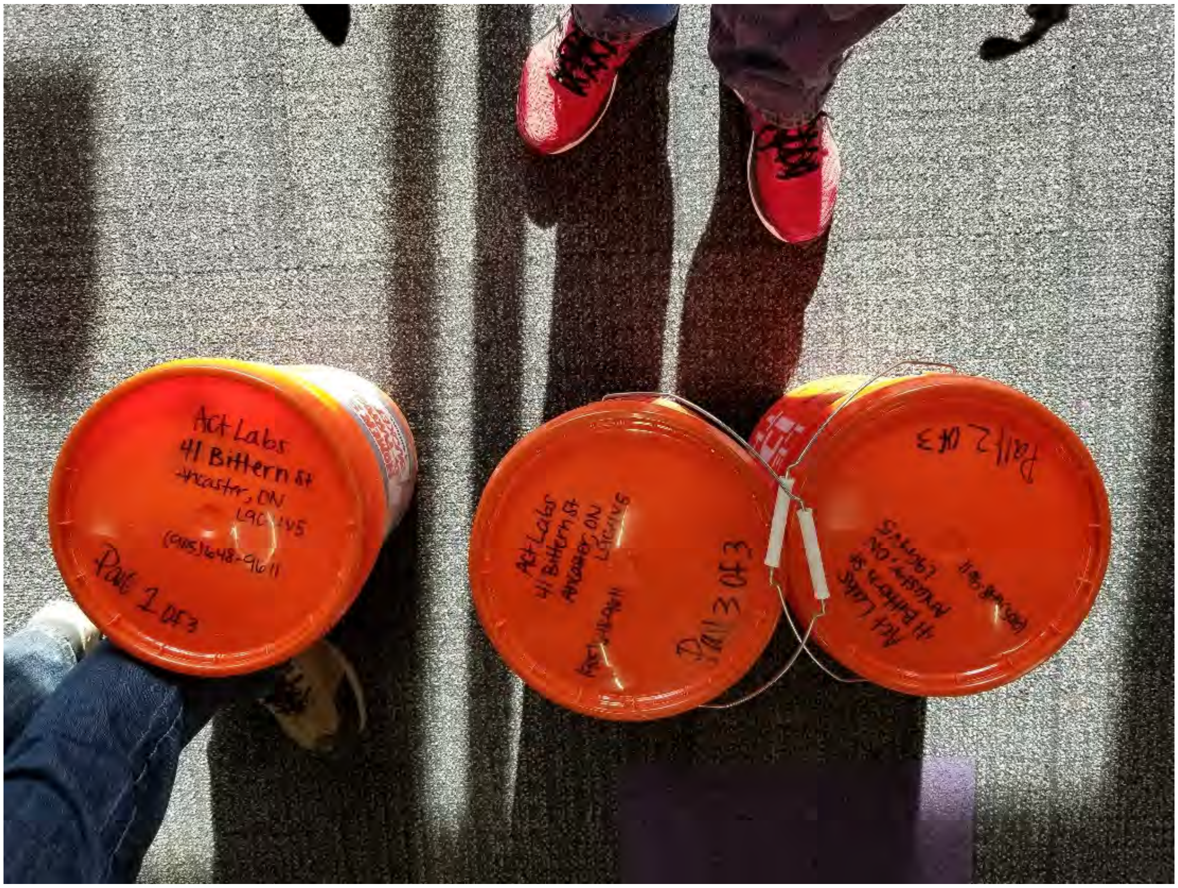
Table 12-3: 2014 Drill Program Intervals Selected for Verification Sampling

BHID	Sample Number	FROM	TO
NEC14-008	NEC008-455	574.98	576.00
	NEC008-456	576.00	577.00
	NEC008-457	577.00	578.00
	NEC008-684	766.54	767.85
	NEC008-685	767.85	769.00
	NEC008-686	769.00	770.00
NEC14-009	NEC009-035	223.50	224.15
	NEC009-055	242.00	243.00
NEC14-009a	NEC009a-074	648.00	649.00
	NEC009a-146	707.00	708.00
	NEC009a-221	766.50	767.23
NEC14-010	NEC010-297	448.00	449.00
	NEC010-298	449.00	450.00
	NEC010-300	451.00	452.00
	NEC010-324	472.00	473.00
	NEC010-328	476.00	477.20
	NEC010-496	618.63	619.79
	NEC010-582	695.00	696.00
	NEC010-584	697.00	698.00
	NEC010-606	716.42	717.00
	NEC010-607	717.00	718.00
	NEC010-608	718.00	718.65
NEC14-011	NEC011-167	359.00	360.00
NEC14-012	NEC012-082	253.64	255.00
	NEC012-135	296.08	297.00
	NEC012-166	323.00	324.00
NEC14-013	NEC013-305	451.00	452.00
	NEC013-341	482.00	483.19
	NEC013-522	629.00	630.00
	NEC013-750	820.00	821.00
	NEC013-783	847.00	848.00
NEC14-014	NEC014-394	523.00	524.00

	NEC014-680	756.00	757.00
NEC14-016	NEC016-550	670.00	671.00
	NEC016-598	709.00	710.00
	NEC016-613	722.00	723.00
	NEC016-635	741.00	742.00
	NEC016-668	767.00	767.62
	NEC016-674	772.00	773.00
NEC14-020	NEC020-005	241.00	242.00
	NEC020-062	286.00	287.00
NEC14-021	NEC021-280	663.00	664.00
	NEC021-283	664.00	665.00
	NEC021-284	665.00	666.00
	NEC021-286	666.00	667.00
	NEC021-287	667.00	668.00
	NEC021-293	672.00	673.00
	NEC021-297	675.00	676.00
	NEC021-298	676.00	677.00
	NEC021-299	677.00	678.00
	NEC021-300	678.00	679.00
	NEC021-301	679.00	680.00
	NEC021-309	686.00	687.00
NEC14-022	NEC022-083	455.05	456.60
	NEC022-119	485.00	486.00
	NEC022-121	486.00	487.00
	NEC022-122	487.00	488.00
	NEC022-123	488.00	489.10
	NEC022-184	539.00	540.00
	NEC022-185	540.00	541.00
	NEC022-300	632.10	633.00

Source: Nordmin, 2019

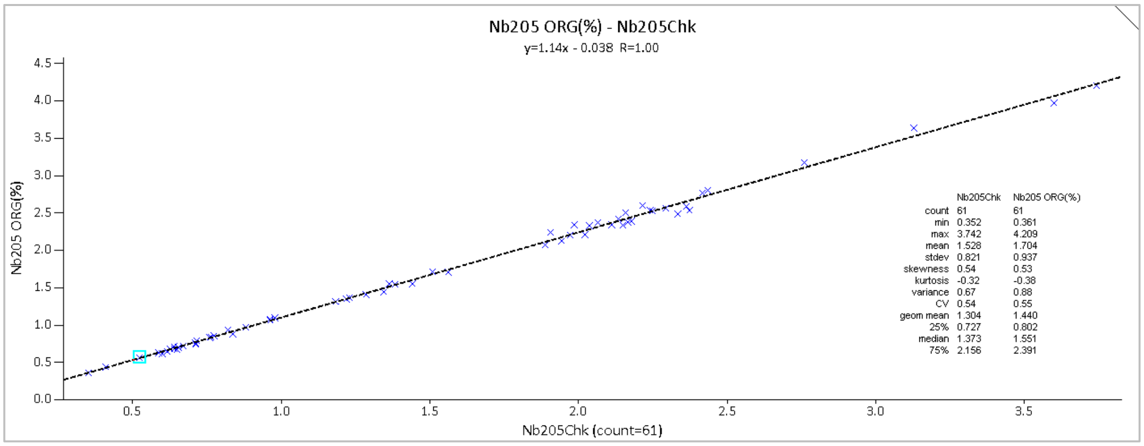
Pulp samples selected by Nordmin for verification analysis were individually placed into plastic sample bags which were combined into larger pails for shipping and dropped off at the FedEx office for delivery (see Figure 12-9). All pulp samples were shipped to the Actlabs facility in Ancaster, Ontario, for analysis using the NioCorp's analytical procedures.



Source: Nordmin, 2019

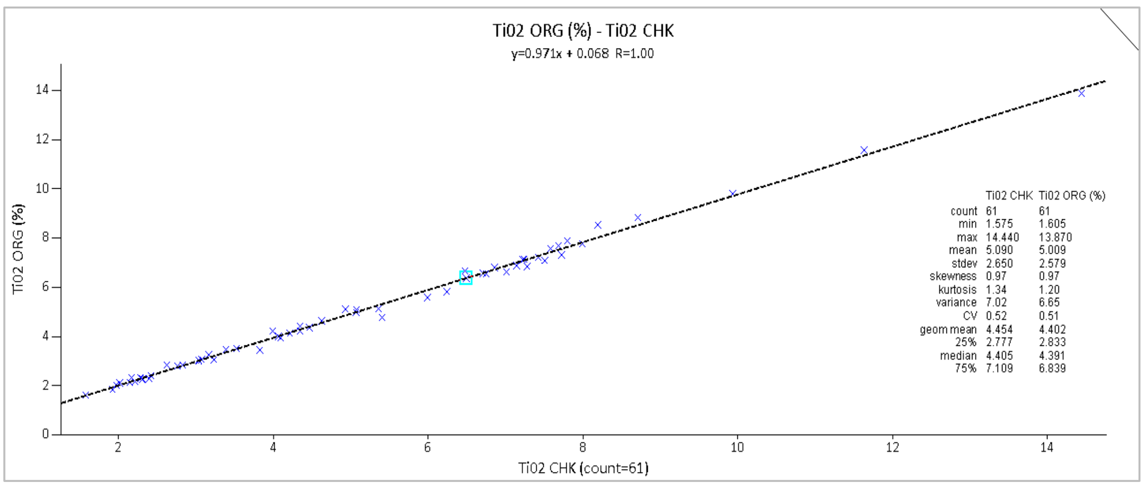
Figure 12-9: Nordmin Verification Samples

The Nordmin assay results were compared to the NioCorp database and summarized in the following scatter plots for Nb_2O_5, TiO_2 and Sc (see Figure 12-10, Figure 12-11 and Figure 12-12). Despite some sample variance, most assays compared within reasonable tolerances for the deposit type and no material bias was evident.



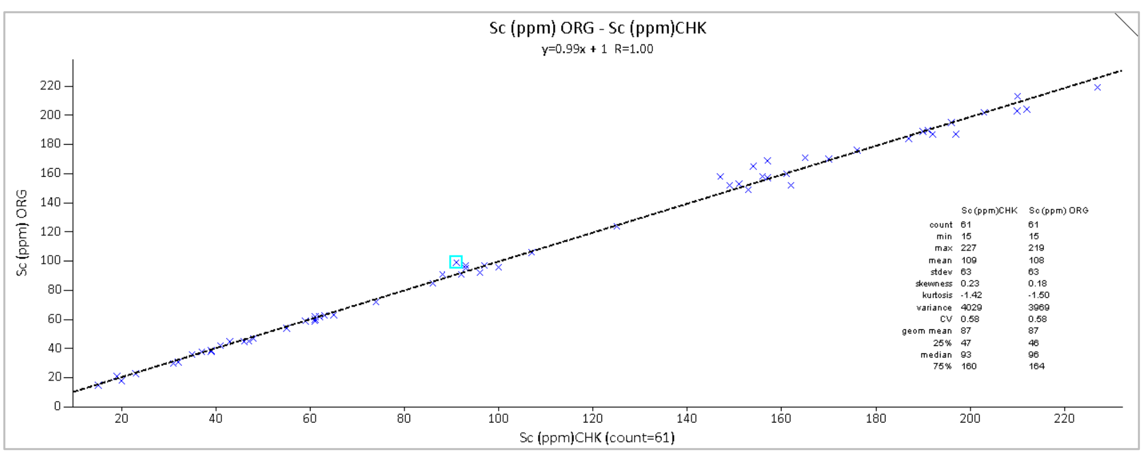
Source: Nordmin, 2019

Figure 12-10: Scatter Plot Comparison of Nb_2O_5 Verification Samples



Source: Nordmin, 2019

Figure 12-11: Scatter Plot Comparison of Tio_2 Verification Samples



Source: Nordmin, 2019

Figure 12-12: Scatter Plot Comparison of Sc Verification Samples

12.5 Summary of Previous Data Validation 2012 - 2016

Both Tetra Tech and SRK completed various data verification analyses between 2012 and 2016.

Tetra Tech reviewed the database of drill holes within the Project area and found:

- The database consisted of 29 drill holes, totalling 18,159.15 m. Twenty-seven of the twenty-nine drill holes, totalling 17,057 m, were used in the interpretation of the Project.
- Tetra Tech performed an internal verification process of the Project database against the original logs, surveyor reports, and laboratory-issued assay certificates.
- The data verification process examined the collars (easting, northing, elevation), lithologies (interval, rock type), and assays (sample number, Nb_2O_5% value).
- No errors were found in the collar, lithology, and assay files.
- A number of holes (EC-25, EC-33, EC-34, EC-35, EC-36 and EC-37) were missing downhole survey information. These holes appear on the southwestern limit of the deposit and were used only in the interpretation of the deposit, not in the resource estimate.
- Quantum's 2010 to 2011 re-sampling data compared to historical values was less than 1% different in all cases except for one where the tolerance was less than 2% (SRK repeated this exercise and reported similar conclusions).
- Completed verification on the lithological logging for potential transcription errors for NEC11-001, NEC11-002, and NEC11-003. It was noted that overburden depths were not recorded in the logs but were entered in the database.
- Sixteen of the historic Molycorp drill holes were attributed with a negative azimuth value in the survey file. All drill holes were vertical, therefore rendering the azimuth insignificant, and all such negative values were corrected to zero.
- Tetra Tech identified a number of cases where minor intervals were logged in the field but were not transcribed into the database.
- All errors were corrected by Tetra Tech before importing into technical software.

Independent check samples were collected during the site visit by Tetra Tech. Four (4) one-quarter (1/4) core samples were collected from the available drill core at the core storage site at Quantum's core logging and sampling facility.

The samples were sent to Actlabs in Ancaster, ON for analysis. The sample preparation was carried out by crushing the sample with the entire sample passing a 10 Mesh (1.7 mm) screen. The sample was then split, and 250 g pulverized with hardened steel to 95% passing a 150 Mesh (106 µm) screen (Actlabs code RX1). Analysis of niobium was conducted using XRF analysis. The Tetra Tech check sample analysis correlated well with Quantum's assay results for the same sample intervals in three of the four cases. Tetra Tech concluded that the analytical results for Nb_2O_5% were confirmed and that the results were adequate for purposes of the 2012 Technical Report.

SRK completed further validation of the historical database, on an on-going basis with input from the Dahrouge geological team. During the initial geological modelling exercises, SRK noted some difficulty in linking the geological interpretation between the NioCorp and historical database. At the request of SRK, a relogging program was completed by the same Dahrouge geologist to ensure consistent logging codes were used.

Historical Assay Information (Adjustments in Molycorp Assays)

During a review of the historical assays against the raw Molycorp database obtained by Dahrouge since SRK's 2014 estimate, an issue was noted whereby a proportion of the Molycorp assay database had been factored (original assays factored by 80%). No clear explanation has been defined within these cases as to the reason for the factored assay results.

The historical database export was provided to SRK and included information for the historical assays broken down into the following categories:

- Nb_2O_5_%_Orig-XRF (Molycorp data, not always reported)
- Nb_2O_5_%_Corr-XRF (Molycorp laboratory corrected data, not always reported)
- Nb_2O_5_%_ALS (2010 re-assay)

Within the 2012 data compilation, the general format has been to adjust any results which contained only the original Molycorp XRF data by the aforementioned 80%.

SRK estimated this was completed for approximately 10% of the assays within the 0.3% grade shell limit and decreasing to <4% within the 0.4% grade shell. The influence on the mean grade is negligible (<0.5%) based on a study of the mean grade using the original versus the adjusted values.

As no defined reason for the adjustment had been noted, SRK used only the original data where no re-assays (due to pulps not being located) had been completed during 2011, 2012 and 2016 verification programs. SRK did not anticipate the use of the factored or unfactored historical assays to have a material impact on the 2017 Mineral Resource Estimate.

During 2016 NioCorp completed a re-assay program of historical holes which were not previously assayed for TiO_2 and Sc (ppm).

Absent TiO_2 and Sc Assays

In the 2015 Mineral Resource Estimate, SRK noted a total of 6.0% and 7.1% of the assays within the mineralized wireframes contain absent values for TiO_2 and Sc respectively. The average Nb_2O_5 grade for the absent values was approximately 0.3% Nb_2O_5.

SRK requested the Company locate the historical samples for these absent assays and complete a re-assay program primarily using ICP/MS to establish valid TiO_2 and Sc grades. A total of 203 pulp samples, 374 pulverized pulp, and 90 chip samples were located at the Mead facility and submitted to Actlabs for analysis.

The re-assays were validated by SRK and the database updated accordingly. SRK completed a statistical analysis on the assays to note the changes prior to estimation. SRK initially compared the results from the historical Nb_2O_5 assays and the latest assays which indicated a slight drop in the mean grade from 0.31% to 0.29%, between the two datasets (7% reduction). SRK used these assays in the 2017 Mineral Resource Estimate to ensure no overestimation occurs within these areas.

12.6 Review of NioCorp QA/QC

NioCorp has a robust QA/QC process in place, as previously described in Section 11. NioCorp and Dahrouge consultants actively monitored the assay results throughout the drill programs and summarized QA/QC results in reports provided to NioCorp for review. A number of failures for standard and blank reference materials were documented, resulting in re-assay of entire sample batches. Most of the certified reference materials performed as expected within tolerances of 2 to

3 standard deviations of the mean grade. Nordmin is satisfied that the QA/QC process is performing as designed to ensure the quality of the assay data.

12.7 Qualified Person's Opinion

Upon completion of the data verification process, it is the Nordmin Qualified Person's opinion that the geological data collection and QA/QC procedures used by NioCorp are consistent with standard industry practices and that the geological database is of suitable quality to support the Mineral Resource and Reserve.

12.7.1 Limitations

Nordmin was not limited in its access to any of the supporting data used for the resource estimation or describing the geology and mineralization in this report.

Database verification is limited to the procedures described above. All Mineral Resource data relies on the industry professionalism and integrity of those who collected and handled the database.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical test work was conducted at SGS Canada Inc. (SGS), Hazen Research (Hazen) and Kingston Process Metallurgy (KPM) throughout 2014, 2015, 2016 and into 2017 to properly design the required process units for the conversion of mined ore into niobium, titanium and scandium products. The preliminary test work was performed on flotation concentrate, which has since been abandoned due to the poor recovery it offered. Test work then focused on whole ore as a feed and consisted of the extensive exploratory bench and pilot scale hydrometallurgical test programs aimed at defining and proving out a final flowsheet using different reagents and technologies. The final process flowsheet was therefore established and proven by test work and piloting performed in all the process units.

13.1 Mineral Processing

The feasibility-level comminution test work was completed in two stages at SGS Canada Inc. (SGS) in Lakefield, Ontario. The primary stage test work (SGS 2016a) was conducted on six composite samples and 13 variability samples and included:

- Bond Rod Mill Work Index (Rwi) testing.
- Bond Ball Mill Work Index (Bwi) testing.
- Bond Abrasion Index (Ai) testing.
- Bond Low-energy Impact (Cwi) testing.
- JK Drop Weight (JKDW) testing.
- Semi-autogenous grinding (SAG) Mill Comminution (SMC) testing.

The second stage of comminution test work (SGS 2016b) was conducted on a single composite sample, using a LABWAL high-pressure grinding roll (HPGR) semi-pilot scale test work program.

The test work results indicate that the Project ore is categorized as soft to moderately hard in terms of ore hardness, and amenable to standard grinding as well as an HPGR operation.

13.2 Hydromet

13.2.1 Testing and Procedures

Sample Representativeness

Three different whole ore samples were received and tested at SGS minerals Lakefield site, for the development of the process described in this report:

- Master Whole Ore Composite Sample (originally on site from previous test work and further described in the previous reporting at the PEA level):
    - Assay reject sample from exploration activity, all passing 10 mesh. These samples were received at SGS Lakefield from the 2014 core drilling program and were used as feed material to test the pre-feasibility in preliminary test work programs. A total of 800 kg of feed samples were processed by SGS Lakefield. A total of ten representative samples representing different areas of the mine that could be reasonably expected during production were combined into a composite sample and used as feed to the hydrometallurgical program.
- 2016 Pilot Plant Ore (received April 2016):

- o Assay reject sample from exploration activity, all passing 10 mesh. Approximately 2,640 kg of coarse reject material was processed by SGS Lakefield.
- Quarter Core (received April 2016). Approximately 1,068 kg of quarter core material was processed by SGS Lakefield.

The received quarter core sample was stage crushed to ¾" and blended. Sub-samples of the quarter core sample were further stage crushed to 100% passing 6 mesh and 10 mesh and used in the test work. Head assays of the three ore samples are summarized in Table 13-1. The assay reject based samples were very similar in composition, while the quarter core sample was slightly lower in niobium, titanium, and scandium content.

Table 13-1: Whole Ore Sample Head Assays

Feed Assays (%)*	Master Hole Ore Composite	2016 Pilot Plant Ore	Quarter Core 6 mesh
Si	4.86	4.72	4.86
Al	1.14	1.09	1.23
Fe	13.5	13.2	13
Mg	5.39	5.45	5.72
Ca	12.7	13.0	13.0
Na	0.30	0.24	0.11
K	1.08	1.3	1.5
Ti	1.98	1.78	1.57
P	0.33	0.34	0.30
Mn	0.51	0.52	0.50
Cr	0.01	0.02	0.02
V	0.03	0.03	0.03
Ba	-	4.73	4.30
Y (g/t)	174	166	151
Sc (g/t)	85	82	73
S	-	1.40	1.10
Nb	0.61	0.53	0.43
LOI	24.4	25.3	26.1

*unless otherwise stated
Source: Tetra Tech, 2017

<u>Hydrochloric Acid Leach (605)</u>

There were a number of preliminary hydrochloric acid leach tests performed at the bench-scale level using different hydrochloric acid concentrations, temperature and residence times in order to confirm the leachability of the Sc and material in the ore. The operating conditions were adjusted to maximize Nb recovery as well as the Nb to Ti selectivity in the processes downstream. A pilot test program, including two pilot campaigns (PP1-013 & PP2-013), was performed.

PP1-013 ran for 80 hours while PP2-013 ran for 88 hours for a total of 168 hours and processed a total of 1,680 kg of ore samples. The objective of the Hydrochloric Acid Leach pilot circuit was to leach out impurities and a large portion of the scandium from the ore while maintaining conditions to maximize Nb recovery as well as Ti selectivity. The pregnant leach solution (PLS) from the whole ore pre leach (WPL) circuit was collected for future test work aimed at the recovery of the leached scandium. The remaining solids were collected for future test work of downstream circuits aimed at recovering niobium, titanium, and unleached scandium.

A summary of the results from PP1-013 and PP2-013 can be found in Table 13-2 and Table 13-3, respectively. A summary of the design conditions and elemental extraction chosen for the feasibility study can be found in Table 13-4.

Table 13-2: PP1-013 Extraction Summary

	Weight Loss %	Fe %	Mg %	Ca %	Ti %	Sc %	Nb %	Th %	U %
Average	74	84	96	99	1	68	0	45	7
Min	70	78	95	99	1	62	0	40	6
Max	77	91	97	99	2	75	1	56	8
Revstd	4	0.06	0.01	0.00	0.61	0.07	0.76	0.14	0.10

Source: SGS 2016 report "Whole Ore Pre-Leaching as Part of the Flowsheet Development for the Elk Creek Deposit Project 14379-013."

Table 13-3: PP2-013 Extraction Summary

	Weight Loss %	Fe %	Mg %	Ca %	Ti %	Sc %	Nb %	Th %	U %
Average	76	77	95	99	1	68	0	41	8
Min	66	69	95	99	1	61	0	30	6
Max	100	87	98	100	3	82	2	57	14
Revstd	0.15	0.06	0.01	0.00	0.61	0.09	1.40	0.20	0.32

Source: SGS 2016 report "Whole Ore Pre-Leaching as Part of the Flowsheet Development for the Elk Creek Deposit Project 14379-013")

From the above results, a set of design conditions were obtained. Table 13-4 shows the hydrochloric acid leach design basis, in terms of extraction to the PLS.

Table 13-4: HCl Leach - Summary Design & Extraction Extent

Si	0	%
Al	19.1	%
Fe	81.9	%
Mg	95.3	%
Ca	99.1	%
Na	20.6	%
K	18	%
Ti	0.8	%
P	77.9	%
Mn	99.4	%
Ba	0.0	%
Sc	62.5	%
Sr	99.1	%
Nb	0.0	%
U	4.3	%
Th	40.5	%
Zr	36.0	%
Cr	6.2	%
V	58.3	%

Source: Tetra Tech, 2017

Acid Bake (610) and Water Leach (615)

The residues from the Hydrochloric Acid Leach testing were used in a series of Acid Bake tests, directed to extracting the niobium after sulphation using sulphuric acid at elevated temperature.

Following the preliminary bench-scale test work, two pilot campaigns were performed on the Acid Bake. A first campaign (PP1-015) was performed using a pug mill followed by a rotary kiln feeding into the Water Leach. Excessive abrasion and corrosion wear on the pug mill due to inappropriate material of construction forced the second campaign (PP2-015) to be operated with batch acid mixing followed by a continuous run of rotary kiln feeding into the Water Leach. Metal extraction is different between a batch and continuous operation. For this reason, PP2-015 results were excluded from the design data of the Acid Bake and Water Leach. PP1-015 ran over the course of 103 hours and produced a total of 159 kg of solids fed to Water Leach. Table 13-5 shows a summary of the results from PP1-015 Acid Bake and Water Leach.

The results from PP1-015 were used to derive the design conditions. A summary of the design conditions and elemental extraction chosen for the feasibility study can be found in Table 13-6.

Table 13-5: PP1-015 Acid Bake and Water Leach Extractions

	Weight Loss %	Si %	Fe %	Mg %	Ca %	Ti %	P %	Sc %	Nb %	Th %	U %
Average	55%	0.12	84	97	89	85	90	90	92	90	97
Min	50%	0.08	77	95	84	80	84	85	88	59	96
Max	61%	0.15	89	99	92	89	92	93	95	96	98
Revstd	7%	0.21	0.04	0.02	0.03	0.03	0.03	0.03	0.02	0.13	0.01

Source: SGS 2017 report "An Investigation into Flowsheet Development for the Elk Creek Flowsheet — Acid Bake Through to Titanium Precipitation for the Elk Creek Deposit SGS Project No.: 14379-015

Table 13-6: Acid Bake and Water Leach - Extraction Results

Hydrochloric Acid Leach Residue		
AB Temperature	300	°C
AB Acid Ratio	0.775	t/t
WL Temperature	95	°C
Fe	79.8	%
Mg	97.4	%
Ca	91.9	%
Ti	87.3	%
Nb	93.8	%
U	97.1	%
Th	95.5	%

Source: Tetra Tech, 2017

<u>Integrated Operation Iron Reduction (620), Niobium Precipitation (625) and Titanium Precipitation (635)</u>

The water leach liquors were processed in a series of preliminary tests in the Iron Reduction, Niobium Precipitation and Titanium Precipitation aimed at producing and confirming the operating conditions to be used in the pilot campaign. An integrated pilot plant (PP3-015) was operated continuously from August 28 to September 2, 2016, providing approximately 111 hours of operation. The campaign processed water leach liquor that was produced in the PP1-015 and PP2-015 campaigns.

Iron Reduction (620)

A summary of the design conditions is provided in Table 13-7.

Table 13-7: Iron Reduction

Temperature	Ambient	°C
Residence Time	1.25	H
Fe addition	1.125	
Ratio of Briquettes and Powder	90:10	

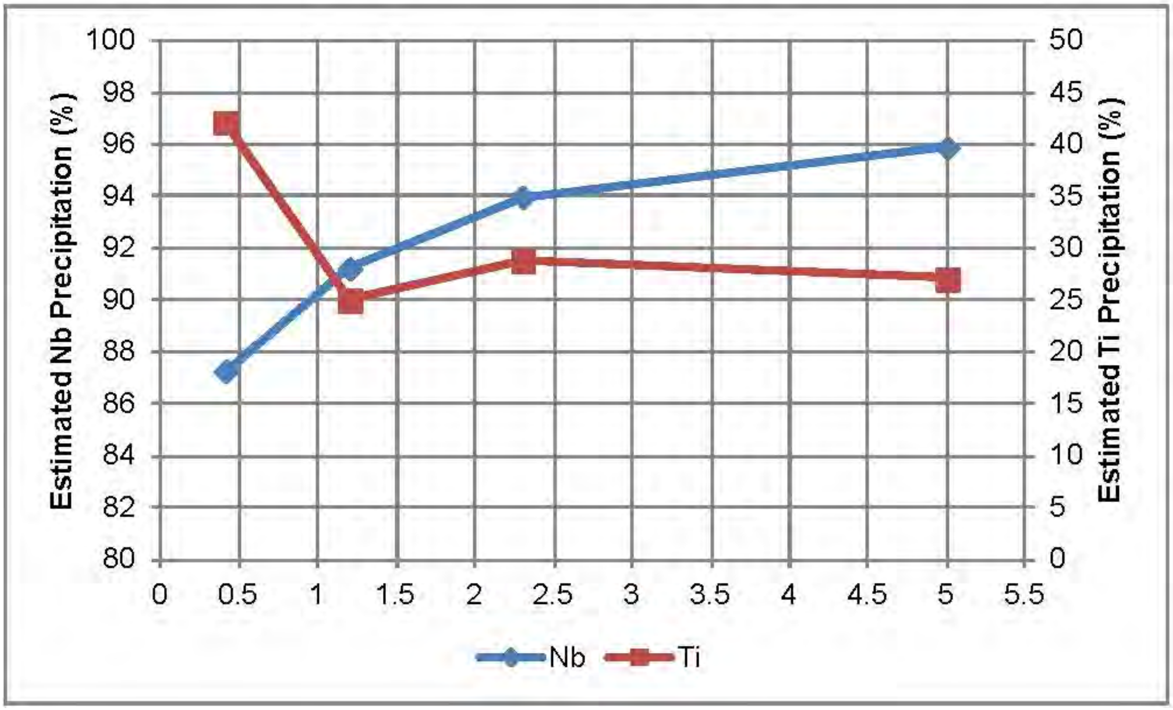
Source: Tetra Tech, 2017

Niobium Precipitation (625)

Additional pilot-scale tests on Iron Reduction (IR) and NbP were conducted in November 2016 to investigate different physical aspects of the process, and follow-up pilot plant campaigns were conducted in December 2016 and March 2017. All test work was carried out under continuous operating conditions.

The feed to the continuous NbP test campaigns was produced by processing Hydrochloric Acid Leach residue from an earlier pilot campaign (PP1-013) which was processed through Acid Bake and Water Leach steps. The Water Leach filtrate was then advanced to continuous testing of the Iron Reduction (IR) and NbP processes. The conditions for IR were left unchanged from the latest bench and pilot work. Based on the established conditions, a full pilot campaign (PP1-018) was conducted over five days in December 2016. The overall dilution ratio was decreased to 2 (effective ratio of ~2.3 based on flow differential) resulting in a drop from 96% to 94% niobium recovery. Following the March 2017 pilot campaign (PP2-018) calculations and investigations confirmed that a 91.51% extraction of Nb could be achieved at a dilution ratio of 0.6:1. To conservatively design the plant, two additional tanks were included to increase the residence time. While being lower than the 96% at 5:1 or the 94% at 2:1, a reduced dilution ratio of 0.6:1 is preferable by greatly reducing the equipment size and the reagent consumption.

Figure 13-1 shows the trends in Nb and Ti recoveries in the NbP as a function of the dilution ratio. The trend shows that the 0.5-0.8 dilution ratio is in the greatest inflection portion of the curve. A regression based on the results provides the required verification that 91.51% recovery can be achieved with a 0.6:1 dilution ratio.



Source: SGS 2017 report "An Investigation into Flowsheet Development for the Elk Creek Flowsheet — Acid Bake Through to Titanium Precipitation for the Elk Creek Deposit SGS Project No.: 14379-015

Figure 13-1: Average Estimated Precipitation Versus Dilution Ratio

The results from the March campaign were used to derive the design conditions. A summary of the design conditions and elemental extractions is provided in Table 13-8.

Table 13-8: Niobium Precipitation - Elemental Extractions

Temperature	100	°C
Residence Time	**4**	**H**
Si	0	%
Al	0.3	%
Fe	0.1	%
Mg	0.3	%
Ca	0.3	%
Na	0.0	%
K	0.7	%
Ti	53.6	%
Mn	5.4	%
Cr	16.1	%
V	12.4	%
Ba	67.6	%
Sc	4.2	%
S	1.2	%
U	37.0	
Th	8.5	
Nb	91.5	%

Source: Tetra Tech, 2017

Caustic Leach — Phosphate Removal

The Niobium Precipitates were used in a series of caustic leach tests, aimed at confirming the process for reducing the phosphate concentration in the final Niobium Precipitate. Ten caustic leach tests investigated a selection of NaOH solutions at various concentrations, temperatures and contact times. A summary of the retained design conditions is presented in Table 13-9.

Table 13-9: Phosphate Removal - Summary Extractions

Si	37.2	%
Al	14.4	%
Mg	0.09	%
Ti	0.04	%
P	95.1	%
Mn	0.2	%
Cr	0.5	%
V	28.2	%
Nb	0.2	%
Sr	100	%
K	57.4	%
S	93.7	%

Source: Tetra Tech, 2017

Titanium Precipitation (635)

A continuous neutralization circuit was operated from August 29 to September 2, 2016, providing approximately 86 hours of operation. The circuit processed filtrate from the NbP circuit. Feed was pumped into the circuit, and as the purpose of the pilot campaign was to test the titanium precipitation, sodium carbonate was added to lower the acidity from approximately 100 g/L sulphuric acid in the feed to between 15 g/L and 20 g/L in the discharge of the circuit with a target of 15 g/L.

The use of sodium carbonate was initially considered for the full-size plant along with magnesium carbonate but was later discarded due to the high cost of the reagent and the difficulties in regenerating. Further testing was performed using lime and limestone that confirmed a significant amount of scandium is being trapped in the gypsum formed by the partial neutralization of the NbP filtrate. To counter the significant scandium losses and the high cost of reagent, a "Calcium Loop" was designed that uses a small amount of fresh lime with recycled lime to partially neutralize the NbP filtrate. A purge back to HCI Acid Leach controls the scandium concentration inside the loop and recovers any elements that may have been trapped. This loop uses a calciner in the same fashion as with the sulphates coming from the Acid Regeneration and the Tailings Neutralization to regenerate and recycle lime out of the gypsum. As these were tested in continuous pilot scale operation, no further piloting was performed on the "Calcium Loop" calciner.

The neutralization was followed by a continuous Titanium Precipitation circuit that was also operated from August 29 to September 2, 2016, providing approximately 81 hours of operation. Table 13-10 shows the design basis of the TiP in terms of elemental extraction.

Table 13-10: Titanium Precipitation - Elemental Extractions

Si	18.9	%
Al	0.9	%
Fe	1.2	%
Mg	0.1	%
Ca	0.6	%
Na	0.0	%
K	0.6	%
Ti	93.5	%
Cr	3.6	%
V	5.2	%
Sc	0.8	%
U	6.4	%
Th	4.3	%
Nb	76.1	%

Source: Tetra Tech, 2017

Scandium Precipitation (628)

The Titanium Precipitation filtrate solution was used in a series of scandium precipitation tests. The limited quantity of scandium contained in the filtrate restricted the number and size of test programs. In all nine bench-scale tests and two bulk campaigns were performed. A total of 659 L of the filtrate were treated, producing a combined 970 g of the precipitate.

A summary of the results can be found in Table 13-11. From these tests and bulk campaigns, design conditions were determined. A summary of the retained design conditions is presented in Table 13-12.

Table 13-11: Scandium Precipitation Summary

Test ID	ScTiP1	ScTiP2	ScTiP3	ScTiP4	ScTiP5	ScTiP6	ScTiP7	ScTiP8	ScTiP9	ScTiP10	ScTiP11
Iron Powder Added	Yes	Yes	No	No	Yes	No	Yes	Yes	Yes	Yes	Yes
Phosphoric Acid Added	Yes	No	Yes	No	Yes	Yes	Yes	Yes	Yes	Yes	Yes
Reagent	$MgCO_3$	$MgCO_3$	$MgCO_3$	$MgCO_3$	$Ca(OH)_2$	$Ca(OH)_2$	$MgCO_3$	$MgCO_3$	$MgCO_3$	$MgCO_3$	$MgCO_3$
Reagent Addition, kg/m^3	17.2	17.8	20.0	18.1	16.6	57.4	14.9	14.9	13.6	13.2	15.6
Iron Powder Addition, kg/m^3	2.2	4.0	-	-	5.1	-	9.1	10.8	9.7	9.4	3.3
Phosphoric Acid Addition, kg/m^3	1.4	-	1.4	-	1.4	3.5	1.4	1.0	0.7	0.3	1.4
Test Temperature, °C	75	75	75	75	75	75	75	75	75	75	75
Final Target Pulp pH	3.35	4.00	4.00	4.00	3.25	3.25	3.25	3.25	3.25	3.25	3.25
Final Filtrate pH	2.88	3.95	3.71	4.00	2.92	3.13	2.83	2.78	2.87	2.71	2.70
Precipitation (%)	ScTiP1	ScTiP2	ScTiP3	ScTiP4	ScTiP5	ScTiP6	ScTiP7	ScTiP8	ScTiP9	ScTiP10	ScTiP11
Sc	93.22	12.09	95.85	-	96.02	97.06	95.66	96.59	96.25	84.92	96.45
Al	70	-	-	59	86	59	77	67	57	19	66
Fe	3	3	18	13	4	4	1	1	2	1	2
Mg	-	-	-	-	1	2	-	-	-	-	-
Ca	1	1	3	2	88	97	1	1	1	1	1
Na	-	-	-	-	-	4	-	-	-	-	-
K	2	-	4	3	5	26	3	2	1	1	3
Ti	100	100	100	100	100	100	100	100	100	100	100
P	53	34	99	31	80	99	61	57	76	65	51
Mn	-	-	1	3	22	43	2	1	1	1	1
Cr	83	88	99	98	95	98	95	76	74	35	92
V	15	97	93	96	34	82	55	17	13	4	19
Th	98	84	100	73	100	100	100	100	100	100	100
Zr	-	-	-	-	-	-	65	59	56	49	95
S	-	-	-	-	-	-	100	-	-	-	-
Nb	-	-	-	-	-	-	100	94	93	91	-

Source: Tetra Tech, 2017

Table 13-12: Scandium Precipitation - Elemental Extractions

Criteria	Value	Unit
Iron Addition	2.9	kg/m^3
Phosphoric Acid Addition	4.5	kg/m^3
$MgCO_3$ Addition	15.3	kg/m^3
Final pH target	3.25	-
Sc	96.4	%
Ti	100.0	%
Zr	100.0	%
Nb	95.0	%
Th	99.7	%

Source: Tetra Tech, 2017

Sulphate Calcining and Mixed Oxides Handling (630)

Sulphate calcining was tested at three facilities all using the Hydrochloric Acid Regeneration solids. Solids were initially calcined at SGS using equipment available. The wet filter cake from the Hydrochloric Acid Regeneration pilot plant was processed through a rotary kiln at a temperature of 1100°C at SGS. The temperature limit was set by the available equipment and not by the process requirement. At such a temperature, it was expected that the Calcium sulphate in the feed material would not be converted, while the free sulphuric acid and any iron sulphate or magnesium sulphate would be converted. In bench tests under similar conditions, typically 70-80% total sulphur removal was noted. The continuous pilot plant operated for 91.5 hours. Overall, the pilot was successful, resulting in 53 kg of calcine produced (from 183 kg of wet filter cake fed). Total sulphur removal was calculated to be 80%, as expected. The calcined product from the pilot at SGS was primarily gypsum with associated iron and magnesium oxides. It was, however, desired to convert the remaining sulphur associated with the gypsum. This led to test work being performed at Hazen Research in Golden, Colorado as well as at Kingston Process Metallurgy Inc. in Kingston Ontario Canada.

Test campaigns were initiated at Hazen using calcined material from the pilot campaign at SGS Lakefield. Conditions were set to provide a reducing atmosphere. This allowed for the better conversion of the gypsum into an oxide mix at a lower temperature than is typically required for gypsum under an oxidizing atmosphere.

Bench-scale tests were initiated followed by a bulk pilot campaign processing a total of 45 kg of calcined material producing 14.5 kg of mixed oxide material and 6 kg of kiln cleanout material containing mixed oxides and a small amount of etched refractory.

Table 13-13 provides a summary of the results of both the bench scale tests and the pilot campaign. The pilot campaign showed that gypsum was essentially wholly converted to oxides with only a trace amount of sulphur left in the mixed oxide material.

Table 13-13: Sulphate Calcining Results Summary

Mineral Identified	Selected Sample Composition, wt%							
	ARC Solids Bulk Shipment 54764	SGS Arc-1 (bucket sample) (reed w/bulk)	Bulk Process Sectional Kiln Average Calcine	Bulk Process Sectional Kiln 3 Larger Aggloms.	Batch Kiln Calcine			
					BK-1	BK-2	BK-3	BK-4 [c/]
Anhydrite ($CaSO_4$)	67	65	< 1	nd	< 1	25	12	< 1
Magnetite (Fe_3O_4)	25	28	8	13	nd	nd	nd	nd
Hematite (Fe_2O_3)	< 1	nd	nd	nd	nd	nd	nd	nd
Periclase (MgO)	7	7	16	6	29	16	18	2
Srebrodolskite ($Ca_2Fe_2O_5$)	nd	nd	42	14	44	44	51	52
Lime (CaO)	nd	nd	3	nd	18	9	11	7
Oldhamite (CaS)	nd	nd	< 1	nd	5	< 1	2	15
Portlandite ($Ca(OH)_2$)	nd	nd	nd	< 1	3	< 1	1	8
Corundum (Al_2O_3)	nd	nd	3	6	nd	nd	nd	nd
Quartz (SiO_2)	nd	nd	< 1	nd	nd	nd	nd	nd
Larnite (Ca_2SiO_4)	nd	nd	22	18	nd	nd	nd	nd
Merwinite ($Ca_3Mg(SiO_4)_2$)	nd	nd	4	17	nd	nd	nd	nd
Jasmundite ($Ca_{10.5}Mg_{0.5}Si_4SO_{16}$)	nd	nd	nd	nd	nd	nd	nd	15
Mulite ($3Al_2O_32SiO_2$)	-	-	-	7	-	-	-	-
Nepheline ($(Na,K)AlSiO_4$)	-	-	-	4	-	-	-	-
Sodium Aluminum Silicate (NaAlSiO4)	-	-	-	6	-	-	-	-
Gehlenite ($Ca_2Al(AlSi)0_7$)	-	-	-	< 1	-	-	-	-
Ilvaite (CaFeSilicate)	-	-	-	6	-	-	-	-
Other Results								
Total Sulphur	16.2	14.8	0.8	0.5	2.2	6.3	4.2	7.7
Neutralization Potential, kg HCl/ton sample [b/]	na	110	na	na	809	502	624	na

Source: Tetra Tech, 2017

Scandium Extraction (640)

The pre-leach liquors were treated in a series of scandium extraction (ScSx) tests and a pilot campaign, aimed at confirming the process for extracting scandium from leach liquors. Within the overall flowsheet, 62.5% of the available scandium is dissolved in the Hydrochloric Acid Leach circuit at design conditions. Scandium that is not dissolved in the HCI Leach circuit is further recovered and eventually is combined in the Pregnant Leach Liquor and fed to the solvent extraction circuit. Two successful and separate solvent extraction pilot plant campaigns (PP2 and PP3) were performed and the circuit operated for a total 215 hours, and approximately 3800 L of PLS was processed, producing 118 g of scandium solids containing on average 42.9% Sc (77.7 g of Sc_2O_3 equivalent content).

The extractant used was Di-(2-ethylhexyl) phosphoric acid (D2EHPA) prepared with tridecanol (as modifier) in Orfom SX80.

Scandium extraction averaged more than 99% throughout the two campaigns with scandium levels in the raffinate consistently below the analytical detection limit (<0.07 mg/L). Thorium and iron extractions averaged 0.1%. Titanium extraction extent averaged 93 and 95%, which showed that no selectivity against titanium took place in the extraction circuit. However, titanium was efficiently removed in the scrub circuit. A single wash stage was included in which more than 55% of the iron and more than 20% of the thorium was removed from the loaded organic. Some of the titanium, 7%, was also removed; less than 0.1% scandium was removed in this circuit. The wash liquor was mixed with the PLS and put back into the extraction circuit.

The scrubbing circuit was designed to remove the remaining iron, thorium and titanium from the washed organic using a solution of H_2SO_4 and H_2O_2. More than 99% of the iron was removed from the washed organic. Thorium in the scrubbed organic was below the detection limit (<2.5 mg/L) throughout the campaigns. Titanium and scandium scrubbing reached 98% and 7%, respectively. The scrubbed scandium is combined with the Ti Precipitation feed liquor, where titanium is recovered as TiO_2 and scandium is subsequently recovered via precipitation from the Ti Precipitation filtrate and brought back into the Scandium SX circuit as impure re-leach liquor.

The strip circuit used a NaOH solution to strip the scandium from the scrubbed organic and at the same time precipitate scandium hydroxide. The aqueous phase was sent to a filter to remove the scandium solids and recycle the strip liquor as strip feed (after adjusting its NaOH concentration).

The overall recovery of scandium to the solids, ranged from 93% to 95% at the beginning of the PP2 campaign and 91% during PP3. Approximately 8% of the total scandium reported to the scrub liquor. This scandium fraction is recovered in the Scandium Precipitation circuit (628). Therefore, overall scandium recovery in the Solvent Extraction (to solids and to the scrub liquor) is greater than 99%. Table 13-14 and Table 13-15 provide a summary of the overall deportment of metals in the Scandium Solvent Extraction unit pilot campaigns PP2-014 and PP3-014.

Table 13-14: PP2 Overall Metal Distribution

Stream	Sc %	Th %	Fe %	Ti %	Ca %	Mg %	Mn %	Al %	Ba %	Be %	Co %	Cr %	K %	Na %	Ni %	P %	Sr %	V %	Zn %
Raffinate	1	100	100	8	100	100	100	100	100	99	99	100	100	77	100	100	100	100	100
Scrub Sol'n	8	0	0	88	0	0	0	0	0	1	1	0	0	0	0	0	0	0	0
Conditioning	0	0	0	0	0	0	0	0	0	0	0	0	0	22	0	0	0	0	0
Strip Solution	91	0	0	4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**
Stream	**Sc**	**Th**	**Fe**	**Ti**	**Ca**	**Mg**	**Mn**	**Al**	**Ba**	**Be**	**Co**	**Cr**	**K**	**Na**	**Ni**	**P**	**Sr**	**V**	**Zn**
Raffinate	1	100	100	8	100	100	100	100	100	99	99	100	100	77	100	100	100	100	100
Scrub Sol'n	8	0	0	88	0	0	0	0	0	1	1	0	0	0	0	0	0	0	0
Conditioning	0	0	0	0	0	0	0	0	0	0	0	0	0	22	0	0	0	0	0
Strip Solution	91	0	0	4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Source: Tetra Tech, 2017

Table 13-15: PP3 Overall Metal Distribution

Stream	Sc %	Th %	Fe %	Ti %	Ca %	Mg %	Mn %	Al %	Ba %	Be %	Co %	Cr %	K %	Na %	Ni %	P %	Sr %	V %	Y %	Zn %
Raffinate	0	100	100	5	100	100	100	100	100	99	99	100	100	75	100	100	100	100	100	100
Scrub Sol'n	8	0	0	91	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0
Conditioning	0	0	0	0	0	0	0	0	0	0	0	0	0	25	0	0	0	0	0	0
Strip Solid	92	0	0	3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Source: Adapted from SGS 2017 report "An Investigation into An Integrated Scandium Solvent Extraction Pilot Plant for The Elk Creek Project 14379-014

Scandium Refining (645)

The scandium hydroxide [$Sc(OH)_3$] produced in the scandium solvent extraction test work was used in the steps to test the scandium refining portion of the process. A number of leaches both in HCl and H_2SO_4 were performed with the intent of testing the removal of impurities such as titanium and niobium from the scandium produced. HCl was initially tested but was rejected due to poor results. H_2SO_4 was retained in combination with a solvent extraction step. A series of $Sc(OH)_3$ re-leaches tests using $H2_SO_4$ were performed in order to provide approximately 5 L of Sc rich solution. Then the solvent extraction was tested in a series of 6 campaigns treating a total of 5 L. Alamine 336 was first used and provided best results for Zr but only partially removed Ti and Nb. Aliquat 336 was then used and provided a very good result for Ti and Nb but only partially removed Zr. A mix of Alamine 336 and Aliquat 336 is finally used to get the best removal of Zr, Ti and Nb.

A summary of the retained design conditions is presented in Table 13-16.

Table 13-16: Scandium Refining - Impurity Extractions

Criteria	Value	Unit	Comment
Zr extraction	98.8	%	
Ti extraction	91.3	%	
Nb extraction	94.9	%	
Sc extraction	99.3	%	Only traces loading

Source: Tetra Tech, 2017

The Sc rich raffinate was then mixed with an oxalic acid solution in a batch wise fashion to form scandium oxalate crystals. A series of 15 tests were performed to crystallize scandium oxalate. The initial solution used came from the initial re-leach with HCI. The solution used was later changed to a sulphate based scandium re-leach following the development in the impurities removal steps. The crystallization proved to be straight forward and provided very good recovery of scandium oxalate well above 98-99%. Scandium oxalate was further filtered, washed and calcined to produce scandium trioxide with a purity of 99.9%. The limited quantity of scandium available prevented large scale testing of filtration or calcining.

Hydrochloric Acid Regeneration (660)

Using scandium solvent extraction raffinate, a five-day continuous pilot plant seeking to regenerate HCI from the raffinate using sulphuric acid (H_2SO_4) was conducted between October 18, 2016, and October 22, 2016. Two separate runs (PR1, PR2) were operated. The process was successful and showed that 99.94% of the chlorides in the feed stream were converted to HCI into the vapour phase. All the Calcium and most of the iron and magnesium were precipitated as insoluble metal sulphates, forming a filter cake of 50-60% moisture under pilot conditions.

When collecting the HCI without any water addition, the HCI content was calculated to exceed the azeotropic point of HCI, reaching 28% HCI by weight prior to the shutdown of the pilot plant. Reaching even higher concentration would have required increased cooling capacity at a lower temperature. This represents a significant opportunity to reduce the upgrading requirement prior to recycling the stream to pre-leach.

The feed solution averaged 40 g/L Fe, 24 g/L Mg, 56 g/L Ca, and 300 g/L CI, while the discharge solution from AR6 averaged 0.9 g/L Fe, 3.5 g/L Mg, 0.2 g/L Ca, and 39 mg/L Cl. This represents a major decrease in the amount of dissolved metal and near complete removal of chloride from the feed. Table 13-17 shows the composite solid discharge assay.

Table 13-17: PP1 Composite Discharge Solids Assays

AR6 D/C Slurry - Solids	Solids Assays (%, unless otherwise noted)							
	Fe	Mg	Ca	Ti	Mn	S	Cl (g/t)	SO4*
Average	8.41	3.84	12.7	0.01	0.38	22.3	190	67
Min	7.97	3.74	11.7	0.01	0.36	22.0	60	66
Max	8.74	3.93	13.2	0.01	0.39	22.7	439	68
Revstd	3%	2%	4%	0%	3%	1%	70%	1%

Source: SGS 2017 report "An Investigation into Acid Regeneration Pilot Plant Elk Creek Deposit Project 14379-016

In total, 290 kg (or 223 L) of Scandium Solvent Extraction raffinate was processed, and the pilot generated -230 kg of wet solids ranging from 40-56% solids, resulting in approximately 106 kg of dry equivalent.

Tailings Neutralization (665)

Two bulk neutralization campaigns were performed on tailings solution. In all 511 L of tailings, the solution was neutralized with limestone locally obtained from a mine in Weeping Water, NE. Neutralization proved to follow theoretical models. Tailings Neutralization solids were also calcined in bulk to provide samples that were used in the sulphate calcining campaigns performed at Hazen Research.

13.2.2 Relevant Results

A number of individual extractions were compiled to define the total recovery of each of the pay metals. A summary of the results and design conditions is shown in Table 13-18.

Table 13-18: Recovery Summary

	Recovery		
	Nb From Test Work	Sc From Test Work	Ti From Test Work
HCl Leach	100.0%	100.0%	100.0%
Acid Bake - Water Leach	93.8%	97.0%	87.3%
Nb Precipitation	91.5%	98.3%	49.4%
Partial Neutralization	100.0%	100.0%	99.9%
Ti Precipitation	-	99.7%	93.5%
Sc Precipitation - Re-Leach	-	98.6%	-
Sc Solvent Extraction	-	100.0%	-
Sc Purification	-	99.3%	-
Overall	85.8%	93.14%	40.3%

13.2.3 Significant Factors

Adequate test work was conducted to support a feasibility-level design for the hydromet plant; however, optimization was not achieved in all areas. Certain areas will certainly benefit from further "post feasibility study" test work, preferably before detailed engineering activities begin. For instance, the test work performed for the feasibility study shows indications that several factors influence the precipitation of niobium as well as the selectivity with respect to titanium. Test work indicates that an increase in Fe/Nb ratio positively affects Nb precipitation while also promoting selectivity against Ti. Precipitant (dilution water) acidity inversely affects Nb precipitation but increases the selectivity against Ti precipitation. Final free acid titration (FAT) has very little effect on the Nb precipitation, but it greatly increases the selectivity against Ti precipitation. The above factors have not been optimized in this study, and further testing will be required to achieve optimal results. Such optimization could also be achieved with performing process simulation of the yearly or monthly elemental feed composition using the METSIM model and the compositions from the mine plan.

13.3 Pyrometallurgy

Pyrometallurgical test work was carried out at Kingston Process Metallurgy (KPM) in Kingston, Ontario, Canada. Since the Hydromet plant testing demonstrated higher levels of TiO_2 at the exit of the NbP section, the purpose of the Pyromet testing was changed slightly from its original scope. The objective of the Pyromet was to produce a saleable FeNb metal, but in addition, it will play a role of purification by eliminating the excess TiO_2 left by the Hydromet. The testing performed at KPM facilities was required to demonstrate the capability of the Pyromet to produce an acceptable quality of FeNb alloy that meets the product specifications of the FeNb producers. The testing performed at the KPM facilities demonstrated several points listed below:

- The aluminothermic reduction of Nb_2O_5 precipitate to produce FeNb alloy was demonstrated regardless of the high level of TiO_2 in the precipitate.
- Niobium recovery from the Hydromet precipitate reached 96% at the exit of the Pyromet.
- Hematite powder (Fe_2O_3) was successfully added as the iron source for the aluminothermic reduction.

14. MINERAL RESOURCE ESTIMATE

14.1 Introduction

The 2019 Mineral Resource Estimate for the Elk Creek Deposit was completed by each of Mr. Glen Kuntz, P.Geo., Consulting Specialist Geology/Mining, Mr. Christian Ballard, P.Geo., Senior Project Geologist and Ms. Francine Long, P.Geo., Senior Project Geologist of Nordmin Engineering Ltd. The effective date of this Mineral Resource Estimate is February 19, 2019.

The Mineral Resource Estimate for the Elk Creek Deposit is based on data provided by NioCorp from surface diamond drill programs completed between the early 1980s and 2014. All data received was in the local project NAD83 UTM coordinate system, and no other data translations were completed for the purpose of this Mineral Resource Estimate.

The deposit mineralization was modelled in three domains and defined as high grade Nb_2O_5/TiO_2, high grade Sc, and low grade Nb_2O_5/TiO_2, and Sc (as described in more detail in Section 14.3). A 3D block model was constructed to estimate Nb, Ti, and Sc along with auxiliary minerals that affect mineral processing.

This section describes Nordmin's Mineral Resource Estimation methodology and summarizes the key assumptions. It is Nordmin's opinion that the Mineral Resource Estimate reported herein is a reasonable representation of the global Nb_2O_5, TiO_2, and Sc Mineral Resources found at the Elk Creek Deposit at the current level of sampling. No additional drilling has been completed since the 2017 Mineral Resource Estimate.

The 2019 Mineral Resource Estimate was reported at a US$ 180 NSR breakeven cut-off grade and has been estimated in conformity with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014). Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into a Mineral Reserve. Density values were assigned to the model based on the average value of increasing concentrations of niobium/titanium/iron grade. The software used for the 2019 Mineral Resource Estimate was Datamine Studio RM collectively referred to as Datamine Studio.

14.2 Source Database

The drill hole database was constructed by Dahrouge from 1) Molycorp data and 2) raw data captured by Dahrouge during the 2011 and 2014 drilling campaigns. The information has been provided to Nordmin in text format and imported into Datamine Studio software. Nordmin determined the data to be of good quality. The database contained collar locations surveyed in UTM coordinates, downhole drill hole deviation surveys, assay intervals with elemental analyses, geologic intervals with lithologies, alteration, specific gravity, and key structures.

14.2.1 Drill Holes

A drill hole database, consisting of 41 holes totalling 18,051 m of core, 14,017 Nb_2O_5 assays, 12,885 TiO_2 assays, and 12,840 Sc assays were made available for modelling. This database includes the regional carbonatite as well as the Elk Creek Deposit. The holes within the regional carbonatite are not included in this Mineral Resource Estimate.

The database was analyzed for interval errors, out of range values and was reviewed in 3D space to validate the hole locations and desurveyed hole traces. A minor number of interval issues were identified and corrected before commencing block modelling and grade estimation. The quality of the drill hole data is supported by NioCorp's QA/QC process as described in Section 11, independent sample verification and check logging as summarized in Section 12. Nordmin did not identify any material flaws in the drill hole data or data collection procedures. Data collection procedures were found to be consistent with current industry practices. Nordmin has determined the drill hole database to be of suitable quality to support the 2019 Mineral Resource Estimate.

14.2.2 Specific Gravity

A total of 2,043 SG measurements were provided from onsite drill core measurements, taken mainly during 2010 and 2014. Measurements were taken from NQ, and HQ sized core using the weight in air versus the weight in water method (Archimedes) applying the following formula:

SG = weight in air / (weight in air – weight in water)

Nordmin determined that the required amount of SG measurements did not exist to estimate the entire block model directly.

Additionally, it was determined that the lithology code was not an accurate indicator of changes to the specific gravity within the mineralized areas. More accurately, the changes to the SG are directly proportional to the changing Nb_2O_5, TiO_2 and Fe2O3 grades. The SG measurements increase with increasing Nb_2O_5, TiO_2, and Fe_2O_3 grades.

Therefore, mean SG values were assigned to the block model based upon the ordinary kriged estimated Nb_2O_5 grade (see Table 14-1). If the Nb_2O_5 grade is less than 0.5%, a default SG of 3.04 was assigned.

Table 14-1: Specific Gravity

Nb_2O_5 Grade	SG Assigned	Number of Blocks
0.0 to <=0.5%	3.04	9,589,915
0.5 to <=0.9%	3.06	7,072,438
>0.9 to <=2.0	3.14	1,873,251
>2.0 to <=2.5	3.24	948
>=2.5	3.37	0

Source: Nordmin, 2019

14.3 Geological Domaining

The diamond drill logs identified four major rock units (nineteen sub lithologies) to support the mineralization within the deposit. The units consist of sediments, carbonatite (general carbonate-rich and magnesium hematitic carbonate-rich), mafic units, and lamprophyre dikes (see Table 14-2). The unconformity contact between the Pennsylvanian sediments and carbonatite was built using a point cloud created from the logged contact wherever available in applicable drill holes.

Nordmin examined and modelled the lithological and geochemical correlations between rock types, geochemistry, and mineralization. These correlations demonstrated that higher grade Nb_2O_5, TiO_2 was associated with increasing Fe_2O_3, U, and Th while higher grade Sc grades are strongly associated with higher concentrations of CaO, MgO, U, and Th.

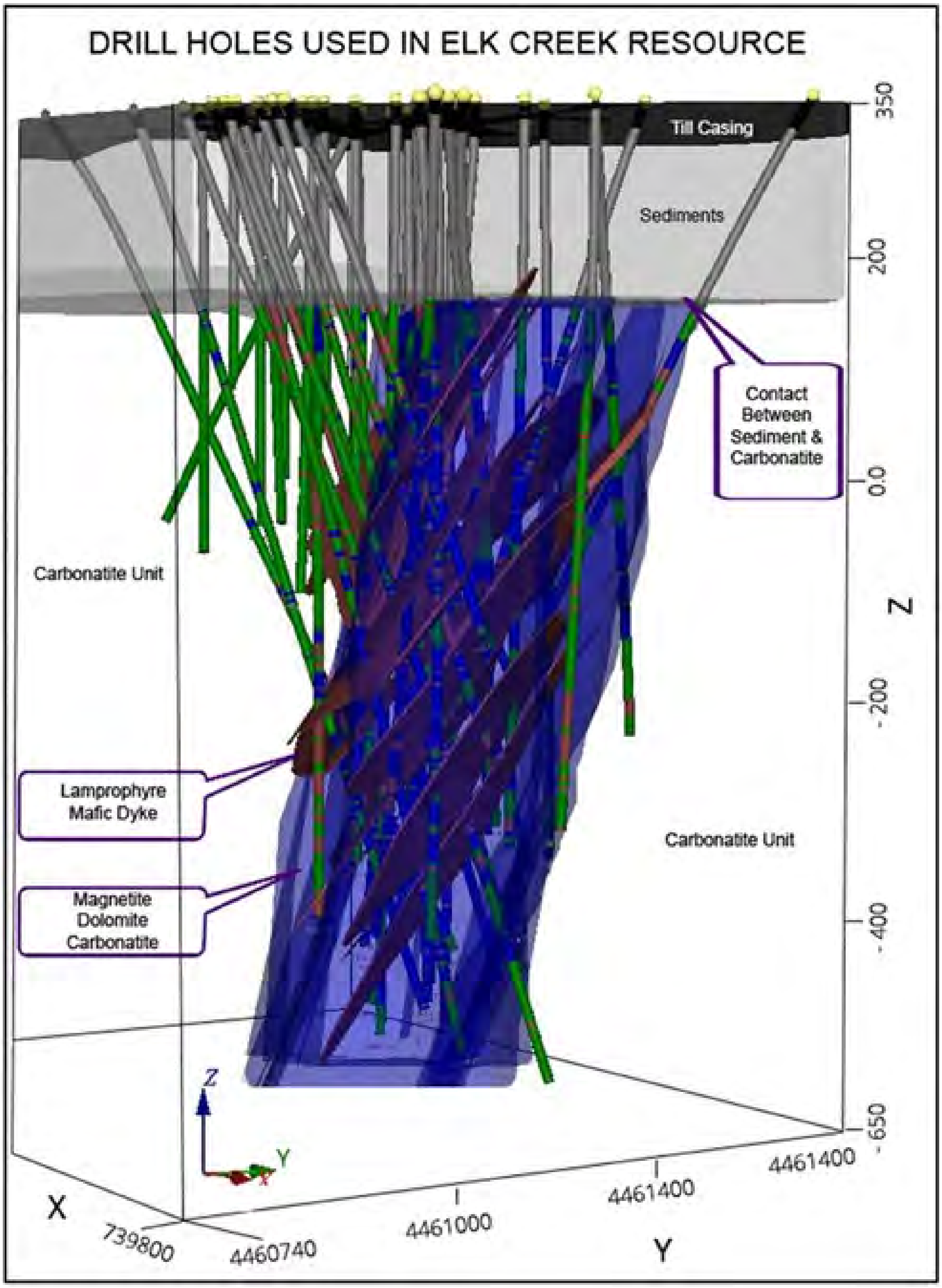
Table 14-2: Major Lithological Units

Major Unit	Description
Casing	Drill hole casing
TILL	Till
SEDT	Sediment
CARB	Carbonatite
MCARB	Magnesium carbonatite
CARB-LAMP	Carbonatite mixed with lamprophyre
MCARB-LAMP	Magnesium carbonatite mixed with lamprophyre
LAMP	Lamprophyre
MAFIC	Mafic intrusives
INT	Intrusives

The niobium, titanium, and scandium mineralization is primarily hosted within a magnetic (hematite) dolomite carbonatite (MCarb) (see Figure 14-1). There are three mineralized domains within the MCarb which have been modelled and defined as:

- High grade Nb_2O_5/TiO_2
- High grade Sc
- Low grade

Nordmin, applying the approach of a large mineable area, created high grade domains and associated zones (wireframes) within each domain based on drill hole intersections and grade, including relevant geological data and structure. All wireframes were clipped to the sediment unconformity. In areas of less defined or scattered mineralization, areas were combined to avoid irregular wireframes. Where wireframes merge or diverge, multiple sections along strike were reviewed to determine if the breaks in mineralization were consistent or just localized patches of low grade mineralization. If a break in the mineralization occurred only over a small strike length or vertically, the break would be consolidated within the surrounding mineralization. In areas where breaks in mineralization are consistent over multiple sections, wireframes merged or diverged accordingly. To avoid overly complex wireframes that merge and frequently diverge with irregular local deviations, as well as areas that could not be extended or replicated, some mineralization was disregarded.



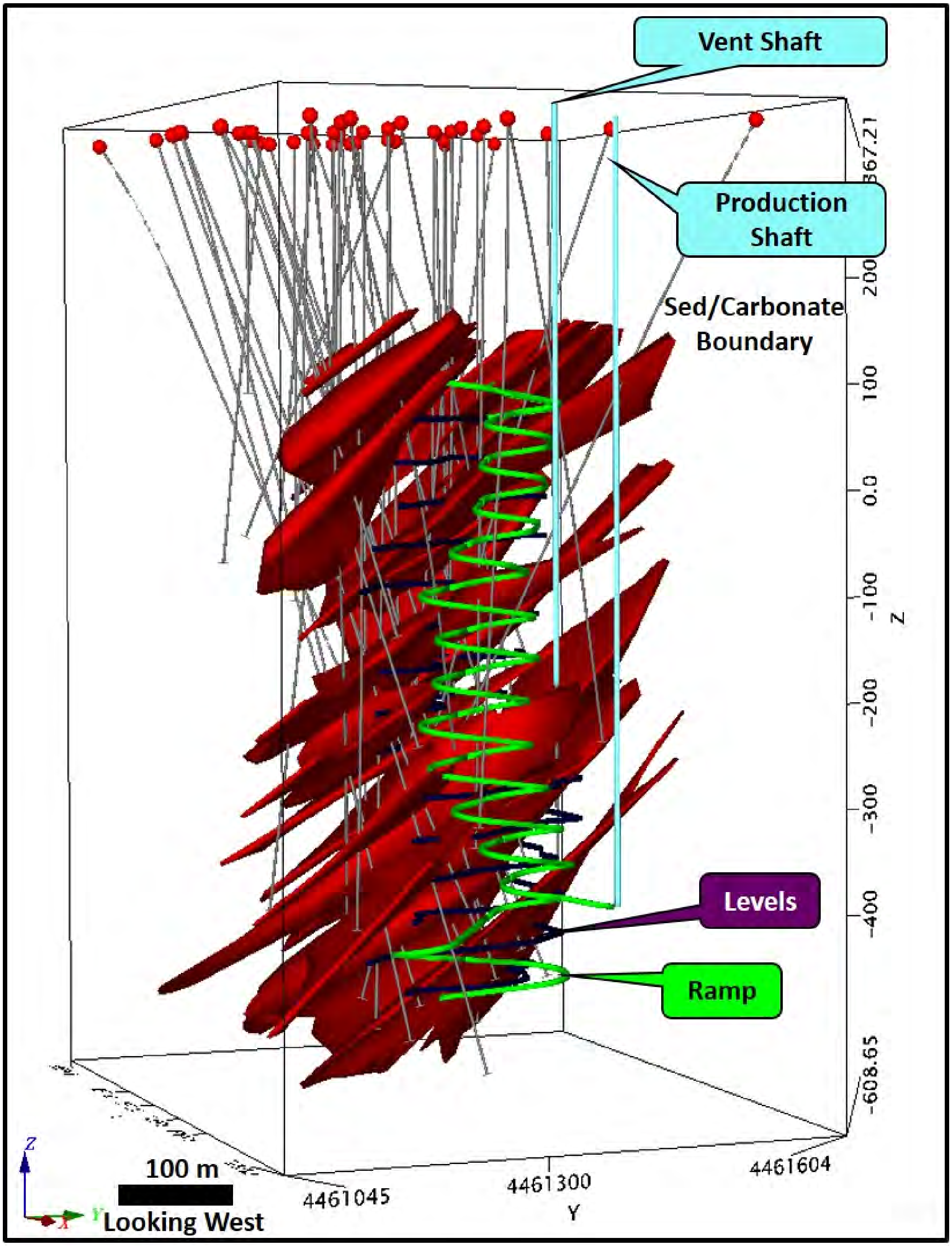
Source: Nordmin, 2019

Figure 14-1: 3D View of Elk Creek Deposit Showing Main Project Area

Summary of Domain Models

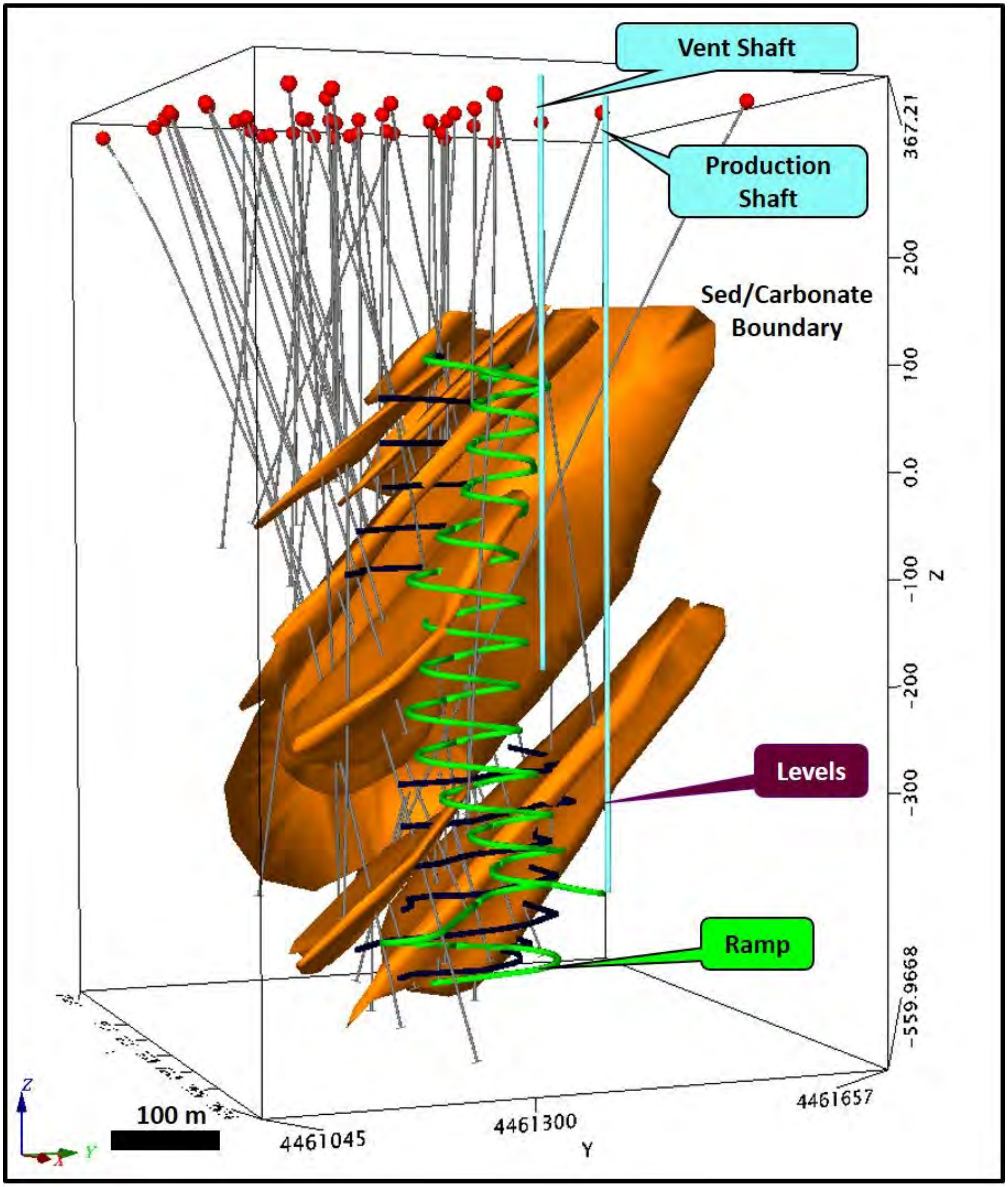
- The high grade Nb_2O_5/TiO_2 domain consists of sixteen individual zones (wireframes) with each zone having a slightly different orientation to one another but generally follow a trend of azimuth 100°- 130° and dip of 32°- 52°. Each wireframe used a minimum of ~1% Nb_2O_5 over 10 m cut-off grade; however, zones 9, 10, 13, 17, 18, and 19 used a minimum modelling criterion of 0.5% Nb_2O_5 over 10 m. This lower grade material was included to honour the geological intervals as defined above. See Figure 14-2, Figure 14-4 and Table 14-3 for details.
- The high grade Sc domain consists of seven individual zones (wireframes) with each zone having a slightly different orientation to one another but generally follow a trend of azimuth 105°- 125° and dip of 41°- 47°. Each wireframe used a minimum of ~70 ppm Sc over 9 m cut-off; however, zones 3 and 4 used a minimum modelling criterion of 60 ppm over 9 m. See Figure 14-3, Figure 14-4 and Table 14-3 for details.
- The low grade Nb_2O_5, TiO_2, and Sc consists of one zone (wireframe) having an azimuth of 120° and dip of 74°. The low grade zone used ~0.3% Nb_2O_5 cut-off to create the boundary. See Figure 14-4, Figure 14-5 and Table 14-3 for details.

Wireframes were initially created on 25 m sections and then adjusted on plan views to edit and smooth each wireframe where required. The wireframes terminate at plunge and depth due to lack of drilling. No wireframe overlapping exists within a given domain, but wireframes of different high grade domains do locally overlap. Due to the contrasts in the physical characteristics between the different mineral phases, Nordmin elected to create hard boundaries to separate the high grade mineralization from the low grade mineralization for each zone within each domain. This approach has the advantage of being able to interpret the mineralization in context with the deposit geology and associated geochemistry using explicit modelling. It is Nordmin's opinion that the explicit modelling approach minimizes risks compared to using implicit modelling for resource estimation within this deposit.



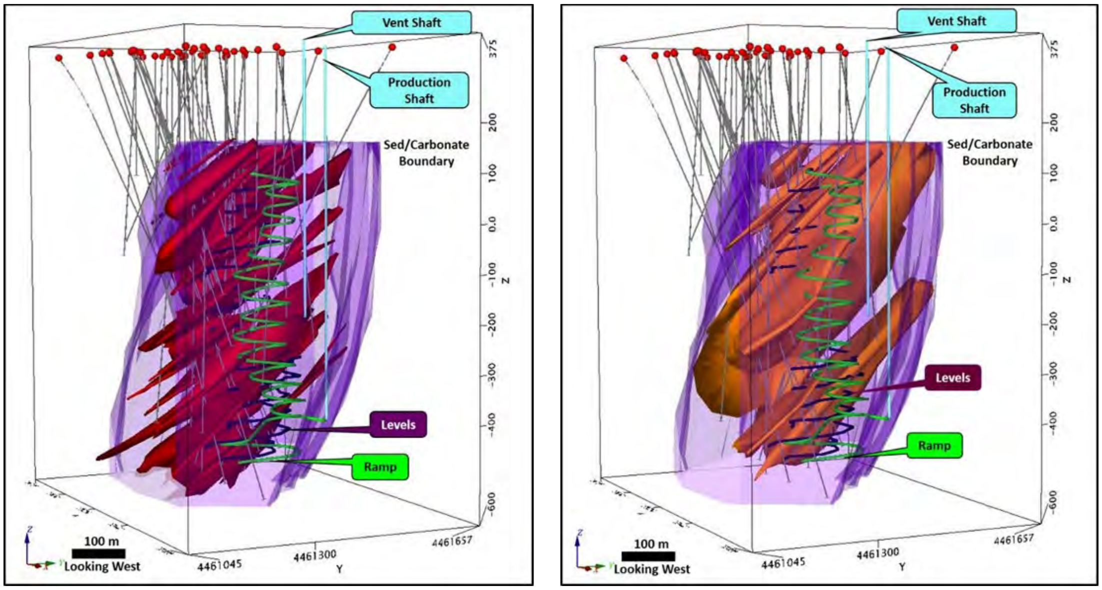
Source: Nordmin, 2019

Figure 14-2: 3D View of High Grade Nb_2O_5/Tio_2 Wireframes with Drill Holes and Planned Infrastructure



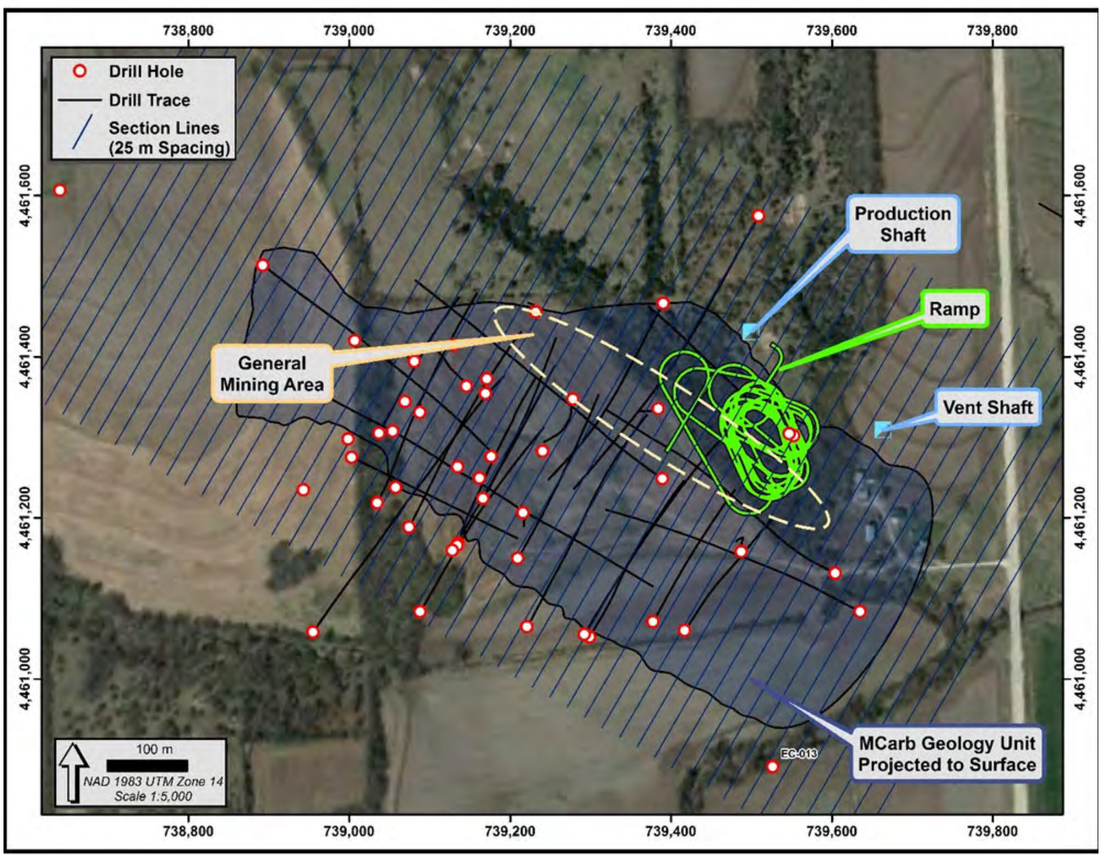
Source: Nordmin, 2019

Figure 14-3: 3D View of High Grade Sc Wireframes with Drill Holes and Planned Infrastructure



Source: Nordmin, 2019

Figure 14-4: 3D View of High Grade Nb_2O_5/TiO_2 Wireframes (Red), High Grade Sc Wireframes (Orange), Low Grade Wireframe (Purple), Drill Holes and Planned Infrastructure



Source: Nordmin, 2019

Figure 14-5: Plan View Showing Low Grade/Mcarb Unit Wireframe, Drill Holes, and some of the Planned Infrastructure

Table 14-3: Wireframe Summary

Domain	Zone	Strike	Dip	Height	Avg. Orientation Strike Length	Avg. Orientation True Width	Maximum Distance Volume	Maximum Distance Tonnage
High Grade Nb_2O_5/TiO_2	2	105°	45°	207 m	120 m	18 m	310,939	976,348
	3	118°	46°	215 m	200 m	6 m	188,823	592,904
	4	115°	46°	270 m	150 m	40 m	785,117	2,465,267
	5	110°	40°	215 m	150 m	25 m	417,098	1,309,688
	7	105°	39°	135 m	75 m	7 m	30,665	96,288
	8	120°	36°	155 m	105 m	8 m	93,426	293,358
	9	118°	40°	320 m	290 m	19 m	623,104	1,956,547
	10	136°	32°	150 m	70 m	8 m	43,014	135,064
	11	099°	42°	250 m	30 m	9 m	12,824	40,267
	12	114°	43°	230 m	260 m	16 m	278,870	875,652
	13	102°	32°	70 m	210 m	27 m	1,052,758	3,305,660
	14	118°	52°	620 m	310 m	80 m	7,187,431	22,568,533
	15	100°	44°	430 m	230 m	66 m	1,937,930	6,085,100
	17	110°	28°	405 m	530 m	21 m	2,454,925	7,708,465
	18	108°	45°	160 m	105 m	21 m	282,449	886,890
	19	120°	35°	340 m	80 m	24 m	161,129	505,945
High Grade Sc	1	125°	41°	230 m	75 m	14 m	94,243	288,384
	2	120°	46°	115 m	300 m	30 m	671,958	2,056,191
	3	110°	41°	280 m	45 m	9 m	99,098	303,240
	4	126°	41°	285 m	150 m	27 m	665,740	2,037,164
	5	108°	46°	500 m	95 m	25 m	689,332	2,109,356
	8	110°	47°	720 m	510 m	200 m	21,740,460	66,525,808
	9	105°	47°	460 m	210 m	50 m	4,554,130	13,935,638
Low Grade		120°	74°	850 m	830 m	430 m	234,547,774	713,025,233

Source: Nordmin, 2019

14.4 Exploratory Data Analysis

The exploratory data analysis was conducted on raw drill hole data selected from within each mineral zone in order to determine the nature of the Nb, Ti and Sc grade distribution, correlation of grades with individual rock units and the identification of high grade outlier samples. Nordmin used a combination of descriptive statistics, histograms, probability plots and XY scatter plots to analyze the grade population data. The findings of the exploratory data analysis were used to help define modelling procedures and parameters used in the Mineral Resource Estimate as further described in this section.

Descriptive statistics were used to analyze the grade distribution of each sample population, determine the presence of outliers and identify correlations between grade and rock types for each mineral zone. Table 14-4 and Table 14-5 provide a summary of the descriptive statistics for the raw sample populations captured from within each mineral zone.

Table 14-4: Summary of Data Available by Zone

Domain	Zone	Number of Drill Holes	Number of Samples
High Grade Nb_2O_5	2	2	23
	3	3	11
	4	5	198
	5	5	88
	7	1	7
	8	1	7
	9	14	150
	10	3	13
	11	3	15
	12	5	53
	13	5	127
	14	20	1,491
	15	10	356
	17	25	514
	18	4	107
	19	3	64
	Total	**35**	**3,224**
High Grade Sc	1	3	31
	2	7	159
	3	2	20
	4	6	79
	5	5	62
	8	29	3,209
	9	9	627
	Total	**35**	**4,187**
Low Grade	**Total**	**40**	**9,707**

Source: Nordmin, 2019

Table 14-5: Descriptive Statistics of Raw Sample Data by Zone

Domain	Zone	Number of Samples	Minimum Grade Nb_2O_5 (%)	Maximum Grade Nb_2O_5 (%)	Mean Grade Nb_2O_5 (%)	Skewness	Standard Deviation	Coefficient of Variation
High Grade Nb_2O_5	2	23	0.24	1.83	1.03	-0.29	0.40	0.38
	3	11	0.61	2.19	1.21	0.97	0.47	0.39
	4	198	0.11	3.78	1.06	0.95	0.48	0.45
	5	88	0.04	2.58	1.02	-0.12	0.56	0.55
	7	7	1.59	2.21	1.89	0.32	0.31	0.16
	8	7	1.30	2.49	1.75	0.88	0.44	0.25
	9	150	0.05	2.09	0.85	0.41	0.47	0.55
	10	13	0.62	1.12	0.86	0.25	0.16	0.18
	11	15	0.14	2.32	1.06	0.39	0.75	0.70
	12	53	0.11	1.84	1.03	-0.15	0.35	0.33
	13	127	0.03	2.79	0.95	0.41	0.64	0.67
	14	1,491	0.02	4.21	0.98	0.84	0.54	0.56
	15	356	0.00	2.88	0.89	0.52	0.52	0.59
	17	514	0.03	3.26	0.89	0.12	0.48	0.54
	18	107	0.18	1.51	0.85	-0.38	0.36	0.42
	19	64	0.08	3.13	0.67	2.10	0.60	0.90
Domain	**Zone**	**Number of Samples**	**Minimum Grade Sc (ppm)**	**Maximum Grade Sc (ppm)**	**Mean Grade Sc (ppm)**	**Skewness**	**Standard Deviation**	**Coefficient of Variation**
High Grade Sc	1	31	47	100	72	0.40	13	0.19
	2	159	0	186	82	0.28	31	0.38
	3	20	25	90	58	-0.09	18	0.32
	4	79	0	138	56	0.24	34	0.61
	5	62	0	185	61	0.67	32	0.52
	8	3,209	0	306	79	0.22	26	0.32
	9	627	17	170	77	0.68	22	0.28
Domain	**Zone**	**Number of Samples**	**Minimum Grade Nb_2O_5 (%)**	**Maximum Grade Nb_2O_5 (%)**	**Mean Nb_2O_5 (%)**	**Skewness**	**Standard Deviation**	**Coefficient of Variation**
Low Grade Nb_2O_5		9707	0.00	4.47	0.42	1.99	0.31	0.74
Domain	**Zone**	**Number of Samples**	**Minimum Grade Sc (ppm)**	**Maximum Grade Sc (ppm)**	**Mean Grade Sc (ppm)**	**Skewness**	**Standard Deviation**	**Coefficient of Variation**
Low Grade Sc		9116	0.00	282	51	0.35	30	0.59

Source: Nordmin, 2019

Analysis of the grade ranges and the coefficient of variation indicated that some of the high grade outlier data might impact the grade estimation if not accounted for. Outlier handling techniques and descriptive statistics (probability plots, histograms, decile plots) were generated for the high grade and low grade Nb_2O_5/TiO_2 and Sc zones in order to analyze the grade distribution and relationship between sample length and grade to help determine a sample capping and compositing strategy.

14.5 Data Preparation

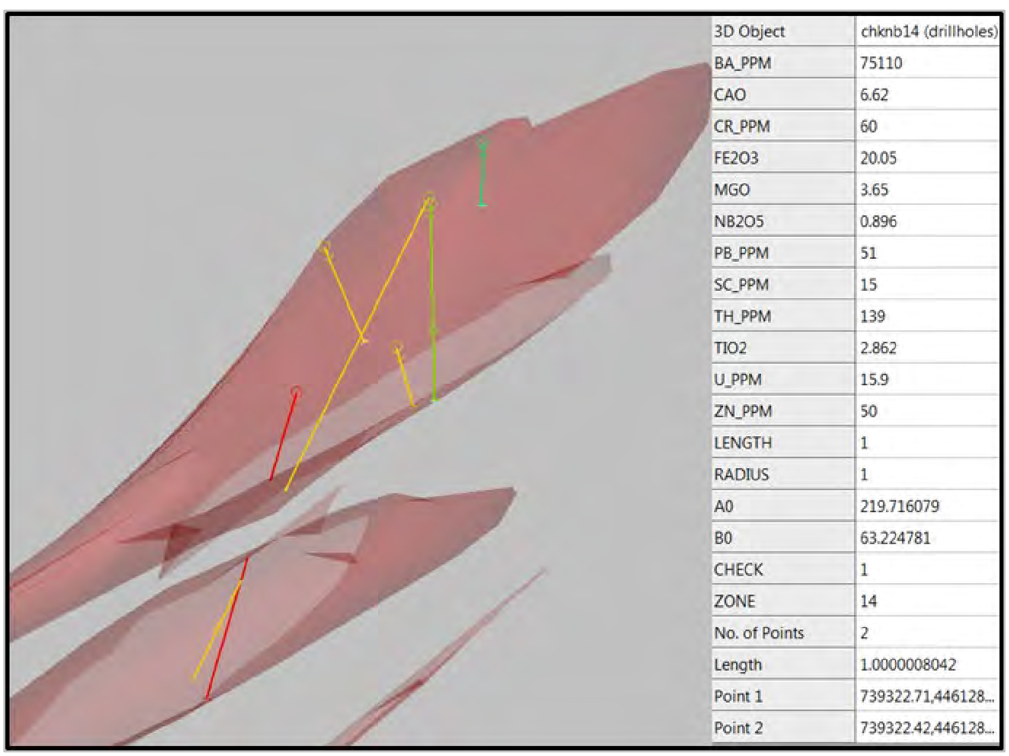
Prior to grade estimation, the data was prepared in the following manner:

- All drill hole samples that intersected and were within a high grade or low grade wireframe were assigned an integer representative of the wireframe number.
- Non-assayed intervals were assigned a minimum detection default grade.
- High grade outlier samples in each domain were top-cut to a maximum value.
- All samples were composited to an average sample length of 2 m.

Before the undertaking of statistical analysis, an outlier analysis was completed, and samples were composited to equal lengths for constant sample volume in order to honour sample support theories.

14.5.1 Assignment of Wireframe Number

All drill hole samples were assigned a wireframe zone integer based upon the modelled high grade or low grade wireframe to assign the samples within each wireframe. The flagged samples are individually reviewed on section and plan to ensure that the correct samples have been assigned to each zone. An example of the DDH capture for wireframe 14 is located below in Figure 14-6. These flagged samples are grouped into their specific domains and used to determine the capping, compositing, and variography analyses. These groupings were used in the grade estimation process to constrain the grade estimates, effectively acting as hard boundaries to prevent samples from one high grade wireframe influencing grade estimates of another.



Source: Nordmin, 2019

Figure 14-6: DDH Capture for High Grade Nb_2O_5 Zone 14

14.5.2 Non-Assayed Sample Intervals

Table 14-6 summarizes the drill holes used in the resource model. Where non-assayed intervals exist for non-payable fields, minimum detection values were substituted to remove bias from the block model.

Table 14-6: Summary of Drilling Database within the Elk Creek Deposit 2019 Mineral Resource Estimate

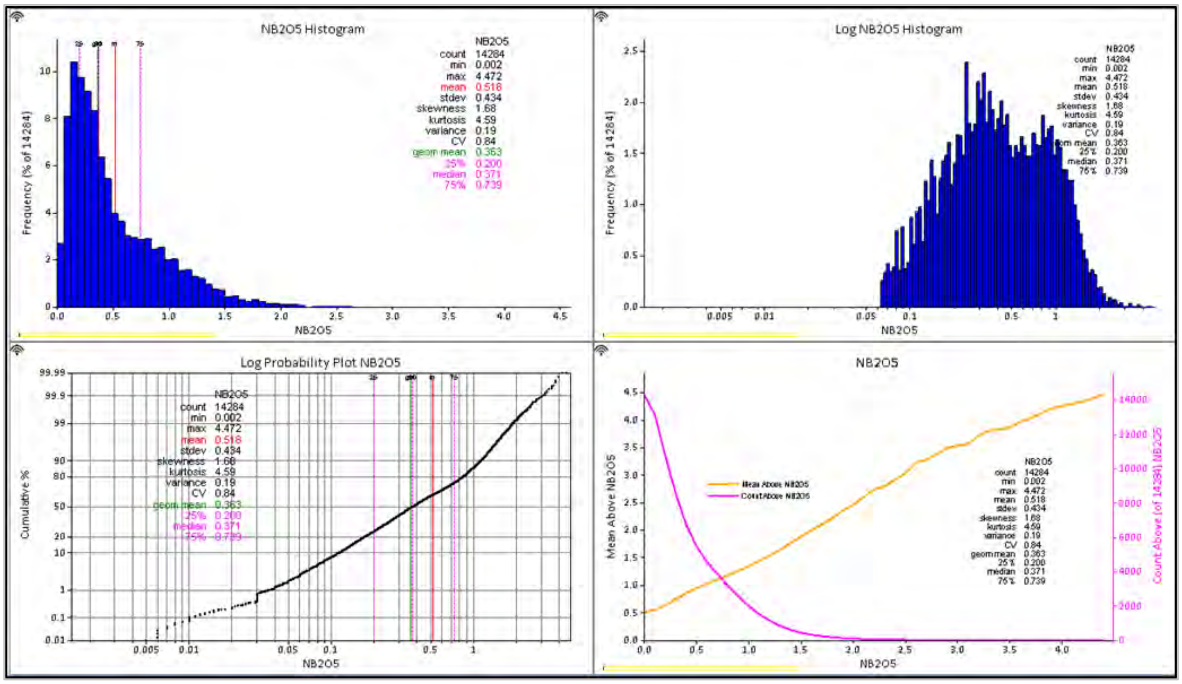
Number of Drill Holes	41			
Number of Survey Records	4,145			
Number of Lithology Records	4,031			
Field	**Count**	**Count at Minimum Detection**	**Total Assay Count**	**% of Minimum Detection**
Ag Assays	11,423	2,308	13,731	17%
As Assays	11,430	2,301	13,731	17%
Ba Assays	12,916	815	13,731	6%
CaO Assays	12,916	815	13,731	6%
Cr Assays	12,916	815	13,731	6%
Fe_2O_3 Assays	12,916	815	13,731	6%
MgO Assays	12,916	815	13,731	6%
Nb_2O_5 Assays	13,719	12	13,731	0%
Pb Assays	11,430	2,301	13,731	17%
Sc Assays	12,871	860	13,731	6%
Th Assays	12,916	815	13,731	6%
TiO_2 Assays	12,916	815	13,731	6%
U Assays	12,916	815	13,731	6%
Zn Assays	11,429	2,302	13,731	17%

Source: Nordmin, 2019

14.5.3 Outlier Capping

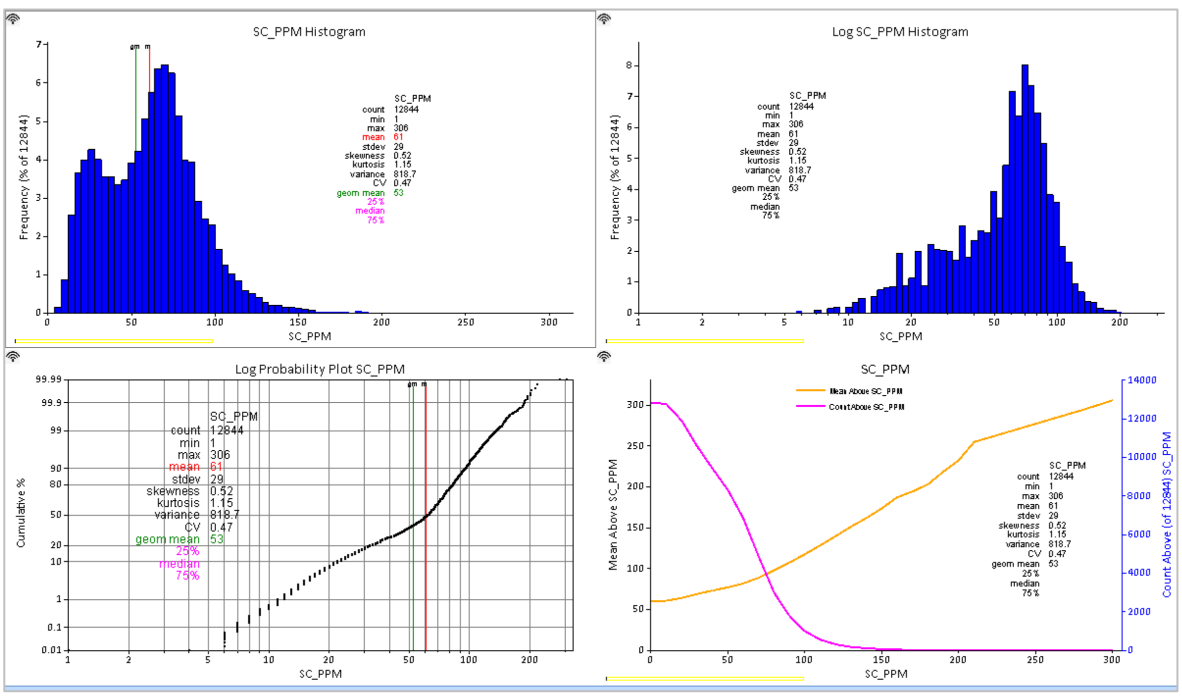
Grade outliers are high grade assay values that are much higher than the general population of samples and have the potential to bias (inflate) the quantity of metal estimated in a block model. XY scatter plots, cumulative probability plots and decile analysis were used to analyze the raw drill hole data for each domain in order to determine threshold limits for outlier values.

Reasonable ranges were first established using the histograms and the probability plots for Nb_2O_5, TiO_2, and Sc. Figure 14-7 displays the histograms and probability plots for the high grade Nb_2O_5 domain and Figure 14-8 for the high grade Sc domain. Raw assay data for the high grade Nb_2O_5/TiO_2, high grade Sc, and low grade material were examined to assess the amount of metal that may be at risk from outlier high grade assays. Statistical analysis was performed using X10 Geo, decile analysis, probability, and histograms to determine cap grades and their overall impact on metal content.



Source: Nordmin, 2019

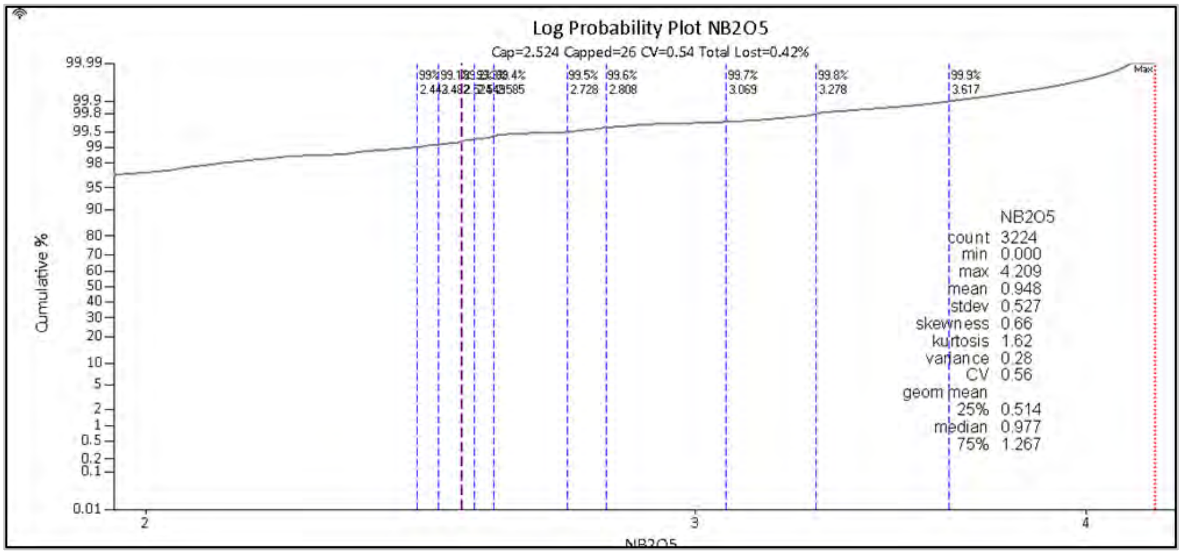
Figure 14-7: Histogram and Probability Plots for Nb_2O_5



Source: Nordmin, 2019

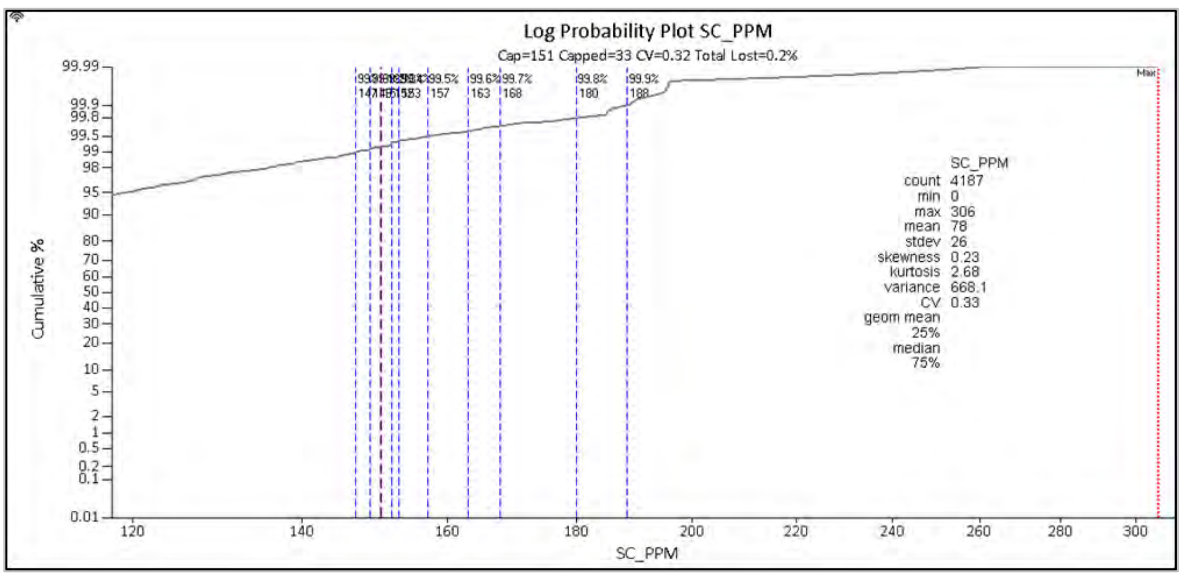
Figure 14-8: Histogram and Probability Plots for Sc

Following the statistical analysis, the selected top-cutting thresholds were evaluated for metal loss using a decile analysis (see Figure 14-9 and Figure 14-10). The term metal loss is a theoretical calculation used to gauge the sensitivity of the selected top-cut value and does not represent actual in situ metal.



Source: Nordmin, 2019

Figure 14-9: Decile Analysis Plot for High Grade Niobium



Source: Nordmin, 2019

Figure 14-10: Decile Analysis Plot for High Grade Scandium

Due to the nature of the mineralization, it was prudent to apply capping to Nb_2O_5, TiO_2, and Sc to each subset of the data. After capping, the resulting change to overall mean grades is insignificant. Cap values were applied as per Table 14-7 below.

Table 14-7: Theoretical Metal Loss

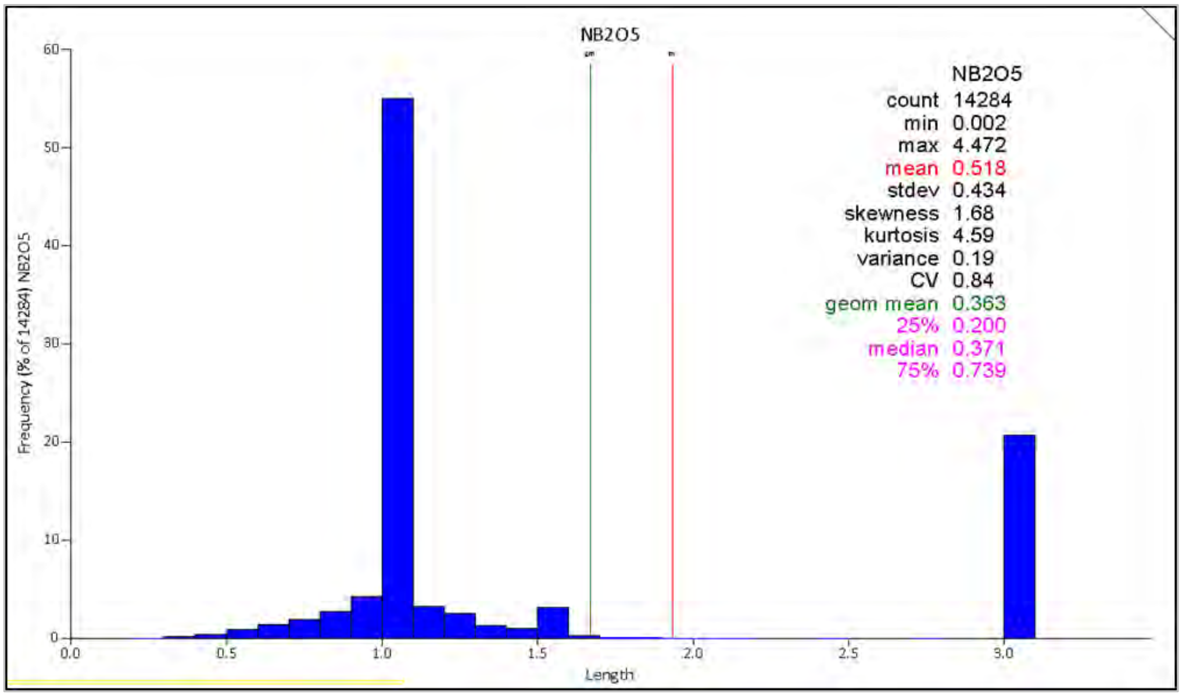
Domain	Field	Number of Uncapped Samples	Number DDH Capped Samples	Uncapped Average	Capped Average	Capped Value	Capped CV	Theoretical Metal loss (%)
High Grade Nb_2O_5/TiO_2	Nb_2O_5 (%)	3224	26	0.948	0.944	2.5	0.54	0.42
	TiO_2 (%)	3224	33	3.299	3.287	7.53	0.45	0.37
High Grade Sc	Sc (ppm)	4187	33	77.99	77.84	150	0.32	0.20
	Nb_2O_5 (%)	9708	97	0.419	0.415	1.4	0.7	1.00
Low Grade	TiO_2 (%)	9708	88	1.834	1.825	5.0	0.67	0.5
	Sc (ppm)	9116	90	50.62	50.37	126	0.57	0.5

Source: Nordmin, 2019

14.5.4 Compositing

Compositing of samples is a technique used to give each sample a relatively equal length in order to reduce the potential for bias due to uneven sample lengths; it prevents the potential loss of sample data and reduces the potential for grade bias due to the possible creation of short and potentially high grade composites that are generally formed along the zone contacts when using a fixed length. The NioCorp raw core sample data was found to have a relatively narrow range of sample lengths due to the width of the mineralization within the larger carbonatite lithological unit. A histogram of raw sample length was generated for each zone in order to determine the most common sample length used, as illustrated in Figure 14-11.

Nordmin elected to use the raw drill hole data as the basis for outlier analysis due to the majority (75%) of the samples having a length between 0.5 m and 1.5 m which includes >50% of the samples having a 1 m length.



Source: Nordmin, 2019

Figure 14-11: Sample Lengths of Raw Nb_2O_5 Samples

Samples captured within all zones were composited to an average length of 2 m based on the observed modal distribution of sample lengths which supports a 5 m x 5 m x 5 m block model. The option to use a variable composite length was chosen to allow for backstitching shorter composites that are located along the edges of the Deposit. All composite samples were generated within each mineral zone with no overlaps along boundaries. The composite samples were validated statistically to ensure there was no loss of data or change to the mean grade of each sample population (see Table 14-8).

Table 14-8: Composite Analysis

Domain	Zone	Number of Drill Holes	Number of Composites	Nb_2O_5 (%)		TiO_2 (%)		Sc (ppm)	
				Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
High Grade Nb_2O_5	2	2	12	1.04	1.04	3.78	3.78	64.56	64.56
	3	3	12	1.09	1.09	4.54	4.54	38.75	38.75
	4	5	118	1.07	1.07	3.93	3.90	57.91	57.91
	5	5	51	0.93	0.93	3.47	3.47	43.30	43.30
	7	1	2	1.90	1.90	5.16	5.16	24.90	24.90
	8	1	4	1.76	1.76	5.17	5.17	64.51	64.51
	9	14	93	0.82	0.82	2.83	2.83	67.32	66.36
	10	3	11	0.88	0.88	2.78	2.78	98.34	98.34
	11	3	23	1.06	1.06	0.46	0.46	45.55	45.55
	12	5	40	1.08	1.08	3.45	3.45	109.44	100.61
	13	5	88	0.78	0.78	2.60	2.59	67.38	67.02
	14	20	890	0.94	0.93	3.39	3.37	70.78	70.53
	15	10	237	0.91	0.91	2.83	2.83	72.16	71.12
	17	25	317	0.87	0.87	3.27	3.27	82.68	81.46
	18	4	62	0.86	0.86	3.22	3.22	71.84	71.84
	19	3	40	0.62	0.62	2.25	2.25	79.01	79.01
	Total	**35**	**2000**	**0.92**	**0.91**	**3.21**	**3.22**	**70.90**	**71.60**
High Grade Sc	1	3	25	1.13	1.09	4.26	4.10	71.86	71.86
	2	7	114	0.73	0.72	2.57	2.55	85.87	84.50
	3	2	10	0.29	0.29	1.23	1.23	57.56	57.56
	4	6	60	0.51	0.49	2.45	2.41	63.39	63.11
	5	5	67	0.61	0.60	2.40	2.37	77.98	77.73
	8	29	2,076	0.75	0.71	2.84	2.75	73.65	73.60
	9	9	256	1.13	1.09	4.26	4.10	71.86	71.86
	Total	**35**	**2,608**	**0.63**	**0.61**	**2.42**	**2.45**	**76.70**	**77.00**
Low Grade	**Total**	**40**	**5,038**	**0.35**	**0.35**	**1.48**	**1.49**	**39.20**	**39.30**

Source: Nordmin, 2019

14.6 Block Model Resource Estimation

14.6.1 Block Model Strategy and Analysis

A series of upfront test modelling was completed to define an estimation methodology to meet the following criteria:

- Representative of the deposit geology and structural model.
- Accounts for the variability of grade and orientations and the continuity of mineralization.
- Controls the smoothing (grade spreading) of grades and influence of outliers between high grade and low grade areas within the deposit.
- Accounts for the majority of mineralization on the property.
- Robust and repeatable within the mineral domains.

Multiple test scenarios were evaluated in order to determine the optimum processes and parameters to use to achieve the stated criteria. Each scenario was based on nearest neighbour

(NN), Inverse Distance Squared (ID2), and Ordinary Kriging (OK) interpolation methods. All test scenarios were evaluated based on global statistical comparisons, visual comparisons of composite samples versus block grades and the assessment of overall smoothing. Based on results of the testing, NioCorp and Nordmin determined that that the final resource estimation methodology would constrain the mineralization by using hard wireframe boundaries to control the spread of high grade and low grade mineralization and would use OK interpolation method to achieve the criteria listed above.

14.6.2 Assessment of Spatial Grade Continuity

Datamine and SAGE 2001 were used to determine the geostatistical relationship of the Elk Creek Deposit. Independent variography was performed on the composite data for each zone (high grade Nb_2O_5/TiO_2, high grade Sc, and low grade Nb_2O_5, TiO_2, and Sc). Experimental grade variograms were calculated from the caped/composited sample Nb_2O_5 and Sc data in order to determine the approximate search ellipse dimensions and orientations.

The analyses considered the following:

- Downhole variograms were created and modelled to define the nugget effect.
- Experimental pairwise-relative correlogram variograms were calculated to determine directional variograms for the strike and down dip orientations.
- Variograms were modelled using an exponential with practical range.
- Directional variograms were modelled using the nugget defined in the downhole variography, and the ranges for the along strike, perpendicular to strike and down-dip directions.
- Variograms outputs were re-oriented to reflect the orientation of the mineralization.

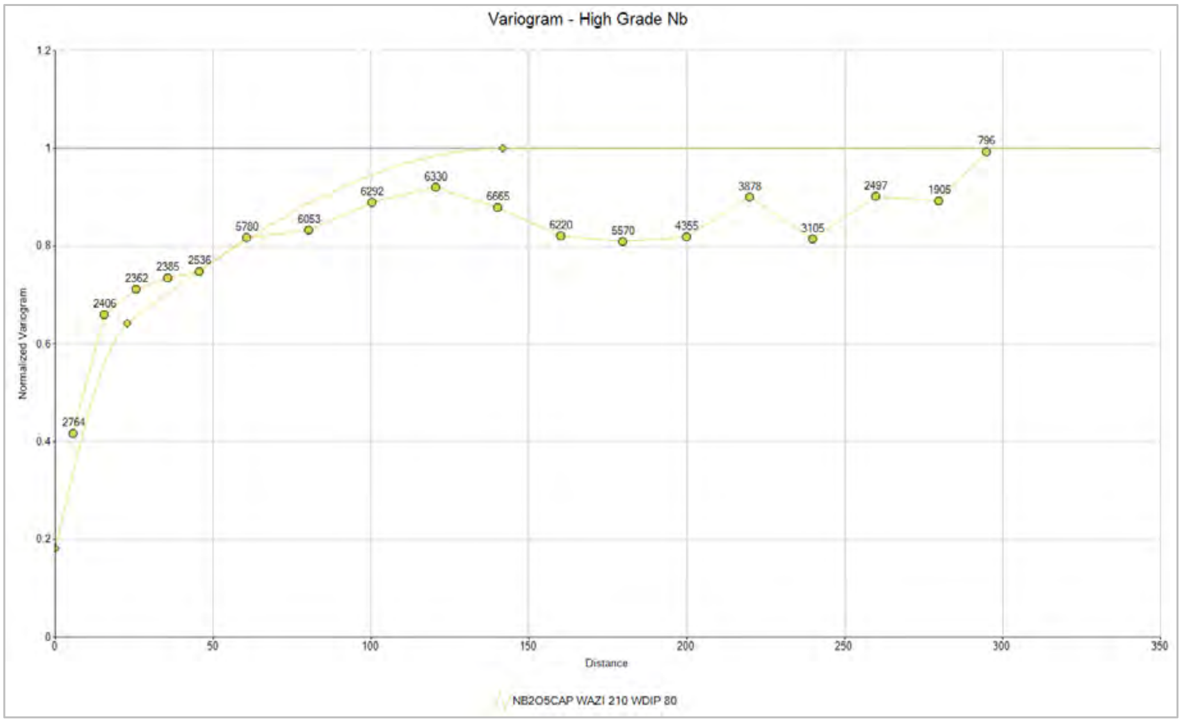
Pairwise correlograms for Nb_2O_5 were generated for the Nb_2O_5 and Sc high grade domains, as well as for the low grade domain, based upon the parameters outlined in Table 14-9.

Table 14-9: Correlogram Parameters

Elements	Low/High Grade Nb_2O_5, Ti and Sc
Lag Distance	20
Number of Lags	15
Minimum Number of Pairs	350
Horizontals Band Width	15
Vertical Band Width	30

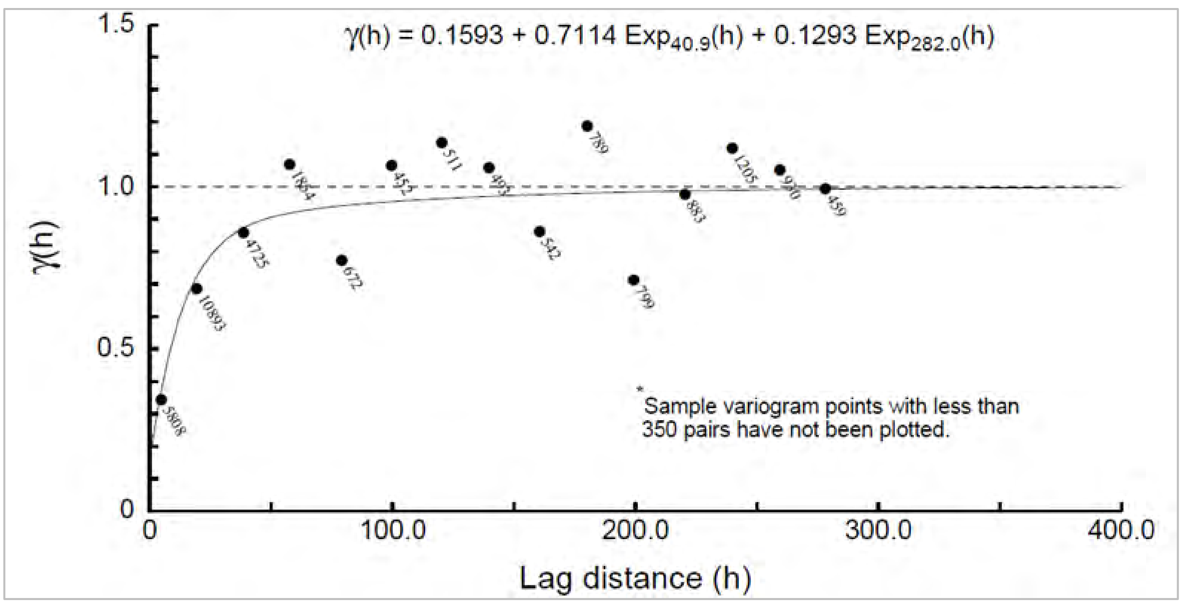
Source: Nordmin, 2019

A set of two structure variograms models were fitted to the experimental variogram data. An example of the variogram model for high grade Nb_2O_5 and high grade Sc domains are provided in Figure 14-12 and Figure 14-13.



Source: Nordmin, 2019

Figure 14-12: Variogram Model for High Grade Nb_2O_5 Domain



Source: Nordmin, 2019

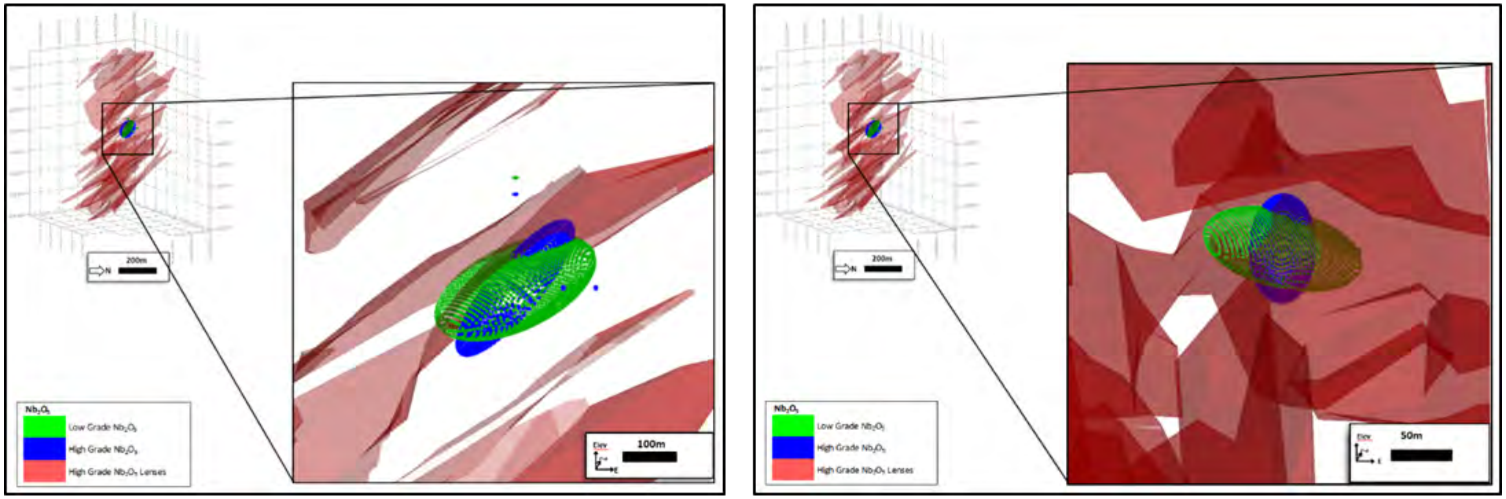
Figure 14-13: Variogram Model for High Grade Sc Domain

The down-dip and along-strike directions of the mineralization were interpreted to be the directions of greatest grade continuity. Search ellipse dimensions and rotation angles per domain are defined in Table 14-10, Figure 14-14, Figure 14-15 and Figure 14-16.

Table 14-10: Search Parameters and Rotation Angles

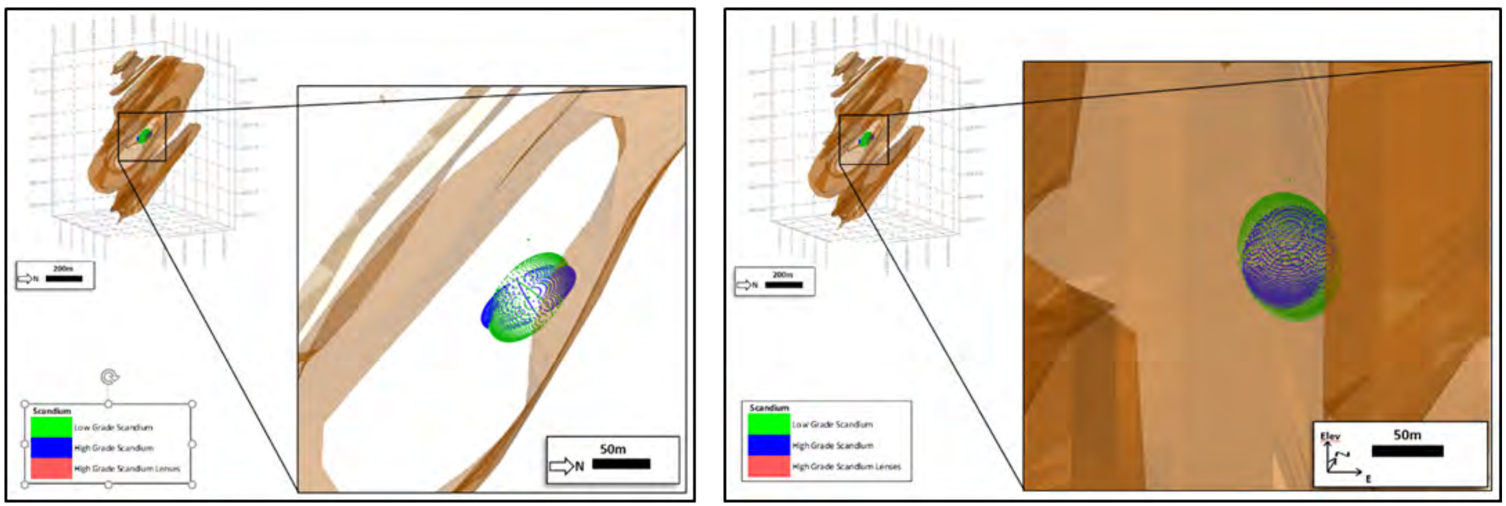
Domain	Field	Pass 1 Dist 1	Pass 1 Dist 2	Pass 1 Dist 3	Pass 2 Dist 1	Pass 2 Dist 2	Pass 2 Dist 3	Pass 3 Dist 1	Pass 3 Dist 2	Pass 3 Dist 3	Rot. Axis 1	Rot. Ang 1	Rot. Axis 2	Rot. Ang 2	Rot. Axis 3	Rot. Ang 3
High Grade Nb_2O_5/ TiO_2	Nb_2O_5	15	24	51	30	48	102	67.5	108	229.5	Z	-55	Y	-40	Z	0
	TiO_2	15	45	40	30	90	80	67.5	180	180	Z	83	Y	35	X	15
High Grade Sc	Sc	15	23	35	30	46	70	67.5	103.5	157.5	Z	-55	Y	-60	Z	0
Low Grade	Nb_2O_5	60	50	20	120	100	40	210	175	70	Z	7	Y	-4	X	-31
	TiO_2	15	45	40	30	90	80	52.5	157.5	140	Z	-62	Y	-30	X	61
	Sc	20	25	35	40	50	70	80	100	140	Z	-70	Y	-41	X	4

Source: Nordmin, 2019



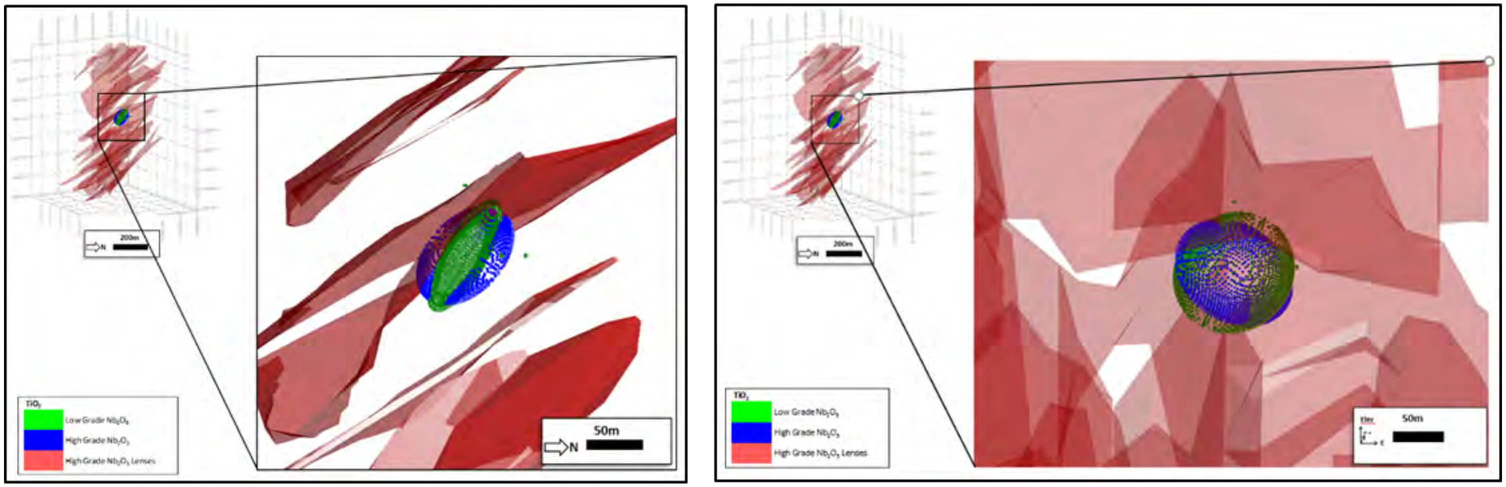
Source: Nordmin, 2019

Figure 14-14: Nb_2O_5 search ellipsoids (green=low grade, blue=high grade, red=high grade Nb_2O_5 lenses).



Source: Nordmin, 2019

Figure 14-15: Scandium Search Ellipsoids (Green=Low Grade, Blue=High Grade, Orange=High Grade Sc Lenses)



Source: Nordmin, 2019

Figure 14-16: TiO_2 Search Ellipsoids (Green=Low Grade, Blue=High Grade, Red=High Grade Nb_2O_5 Lenses)

14.6.3 Block Model Definition

The Elk Creek Mineral Resource block model covers a portion of the carbonatite where there is an increase in niobium, scandium and titanium mineralization. Block model shape and size is typically a function of the geometry of the deposit, the density of sample data, drill hole spacing and the selected mining unit (SMU). On this basis, a parent block size of 5 m x 5 m x 5 m (N-S x E-W x Elevation) was selected. All mineral zone volumes were filled with blocks using the parameters described in Table 14-11. Block volumes were compared to the mineral zone volumes to confirm there were no errors during the process. Block volumes for all zones were found to be within reasonable tolerance limits for all mineral zone volumes. Sub-blocking has been allowed to a threefold factor to a minimum of 0.625 m in all directions to maintain the geological interpretation and accommodate the high and low grade zones (wireframes). The block model was not rotated but was clipped to the sedimentary contact, since the mineralization does not extend past this boundary. The resource estimation was conducted using Datamine Studio RM™ version 1.4.175.0 within the UTM grid (NAD83 Zone 14N).

Table 14-11: Block Model Origin Summary

Item	Origin	Block Dimension	Number of Blocks	Minimum Sub-Block
Easting	738000.0	5 m	400	0.625
Northing	4460000.0	5 m	400	0.625
Elevation	-800.0	5 m	240	0.625

Source: Nordmin, 2019

14.6.4 Interpolation Methods

All block models were generated using Nearest Neighbour (NN), Inverse Distance Squared (ID2), and Ordinary Kriging (OK) interpolation methods to complete global comparisons and validation purposes. Ordinary Kriging was the grade interpolation method used for the 2019 Mineral Resource Estimate. The OK method is a spatial estimation method where the error in variance is minimized through the kriging variance. The OK method was chosen over ID2 and NN to control the smoothing of grades better and attribute more weight to samples that are located in the main orientation of the low and high grade domains.

14.6.5 Search Strategy

Zonal controls were used to constrain the grade estimates to within each low and high grade wireframe. These controls prevented samples from separate low or high grade wireframes from influencing the block grades of one another, acting as a “hard boundary” between the zones. For example, the 890 composites identified within high grade Nb_2O_5 Zone 14 were used to estimate this zone, and all other composites were ignored during the estimate of Zone 14.

General search orientations, defined by dip and dip direction, were estimated into the block model based on the shape of the modelled mineral domains. A total of three nested, searches were performed on all zones. The search distances were based upon the variogram ranges outlined in Section 14.6.2. The search radius of the first search for the low and high grade Nb_2O_5, TiO_2, and Sc was based upon the first structure of the variogram, the second search being two times the first structure and the third search on the maximum of the second structure within the variogram. Search strategies for each domain used an elliptical search with a minimum of three samples and a maximum of twelve samples from a minimum of two holes in the first, second and third passes. Unestimated blocks were left as absent and not reported in the Mineral Resource Estimate. Composite controls are summarized in Table 14-12.

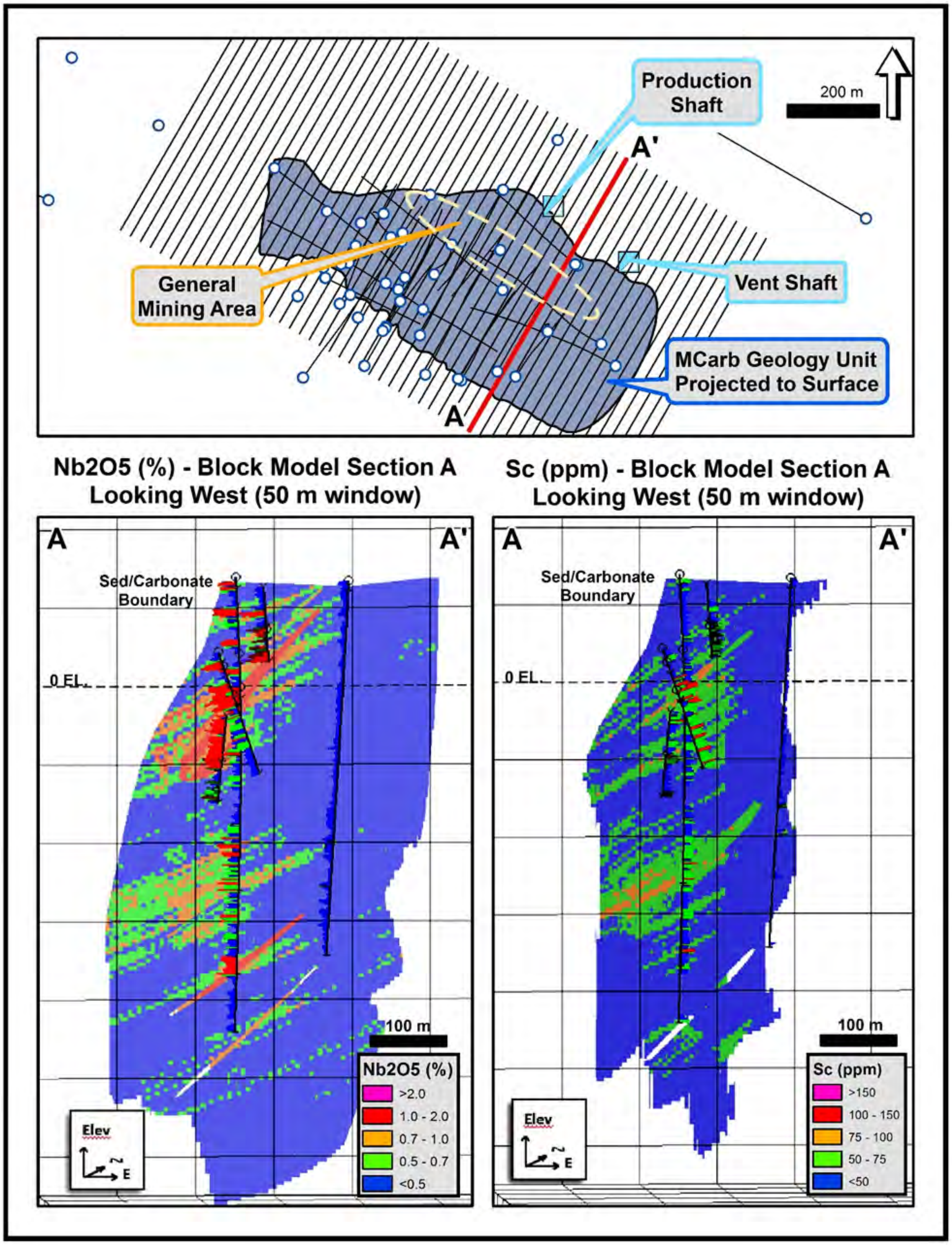
Table 14-12: Summary of Composite Controls for all Zones Within all Domains

Pass 1-3 Item	Min Composites	Max Composites	Min number of Drill holes	Max number composites per hole
All Zones	3	12	2	2

Source: Nordmin, 2019

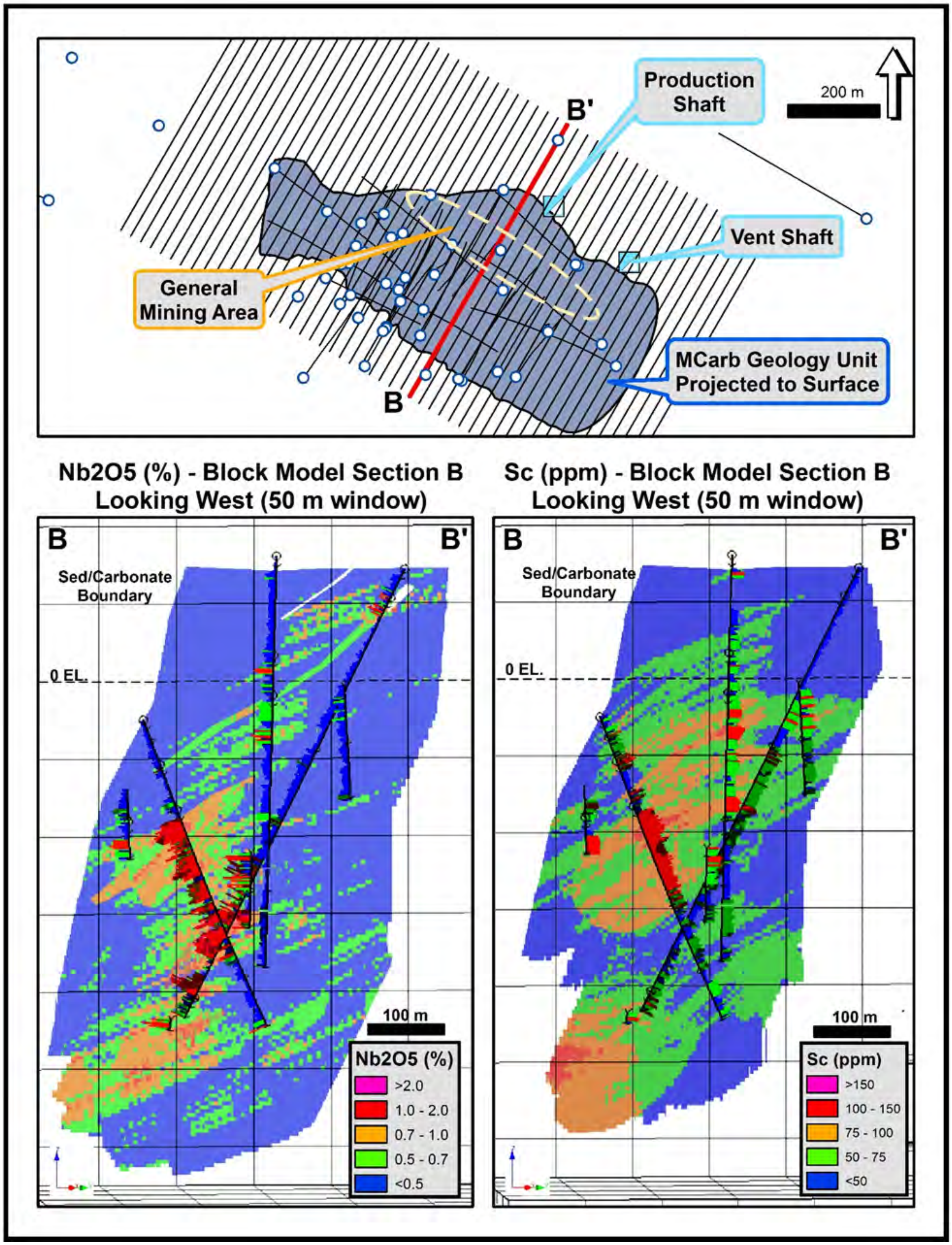
14.7 Model Validation

The block model validation process included visual comparisons between block estimates and composite grades in plan and section, local versus global estimations (NN, ID2, OK) and swath plots. Block estimates were visually compared to the drill hole composite data in all the domains and corresponding zones to ensure agreement. No material grade bias issues were identified, the block grades were identified, and the block grades compared well to the composite data (see Figure 14-17, Figure 14-18 and Figure 14-19).



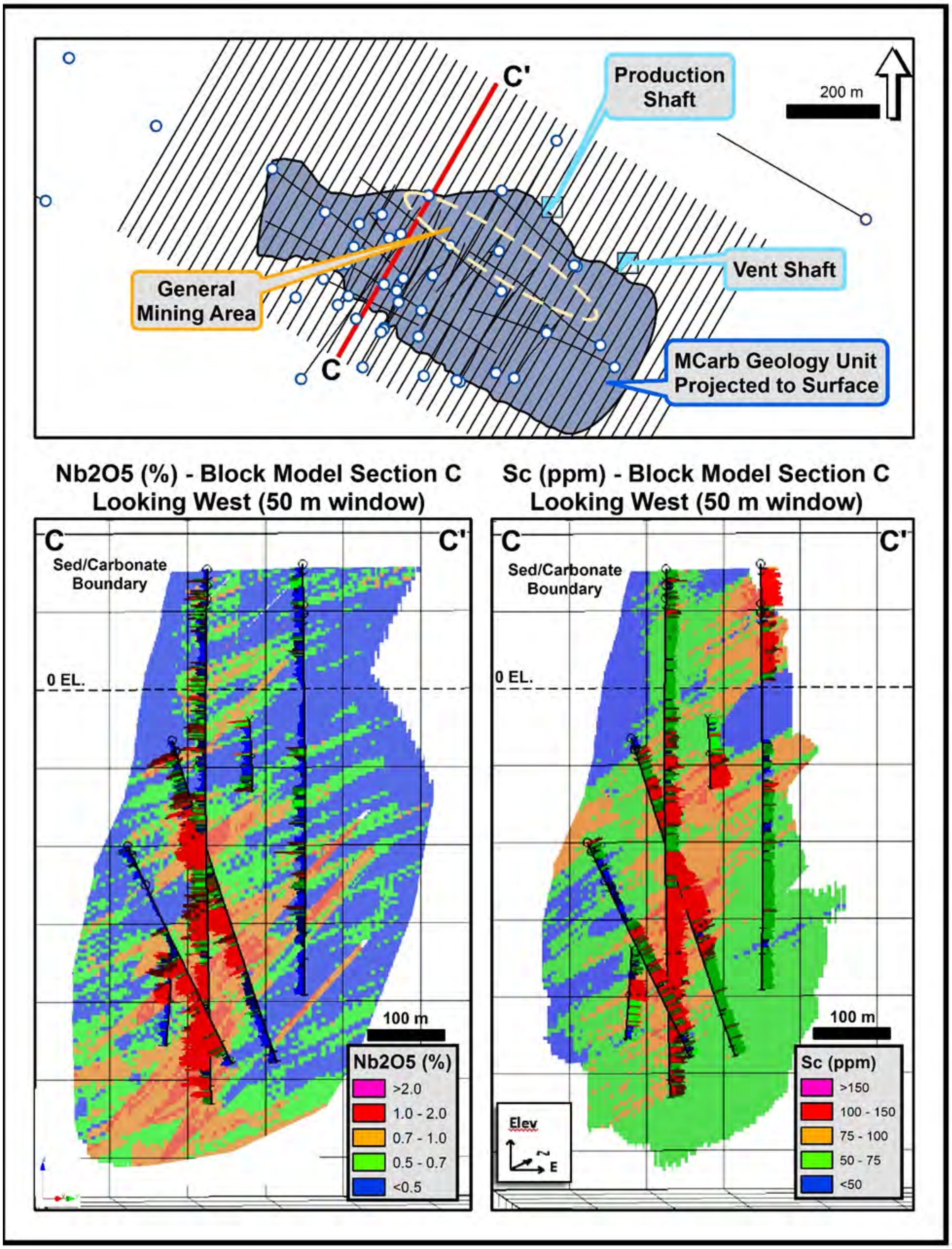
Source: Nordmin, 2019

Figure 14-17: Block Model and Diamond Drill Composites, Cross-Section A



Source: Nordmin, 2019

Figure 14-18: Block Model and Diamond Drill Composites, Cross-Section B



Source: Nordmin, 2019

Figure 14-19: Block Model and Diamond Drill Composites, Cross-Section C

14.7.1 Visual Comparison

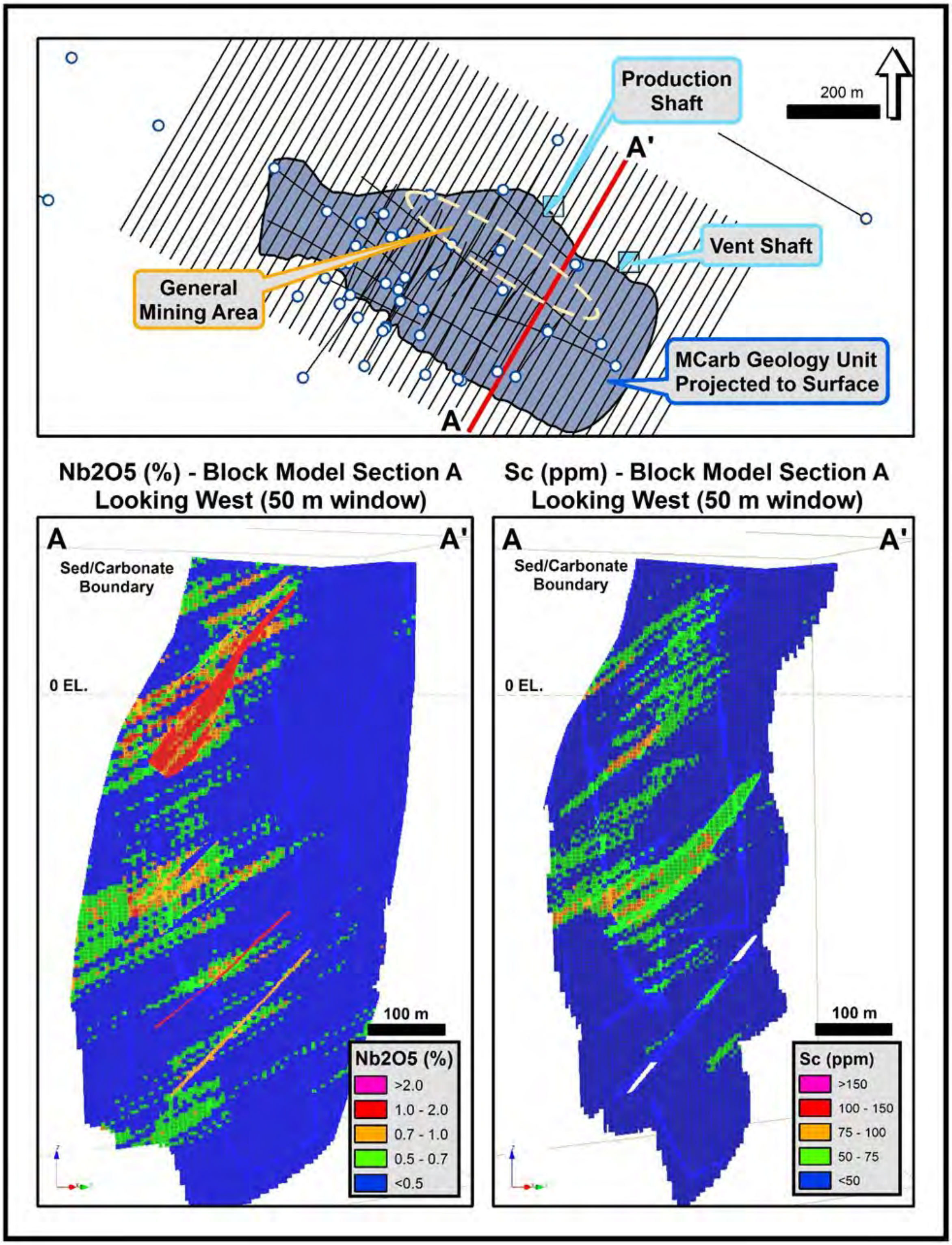
The validation of the interpolated block model was assessed by using visual assessments and validation plots of block grades versus capped assay grades. The review demonstrated a good comparison between local block estimates and nearby samples, without excessive smoothing in the block model (see Figure 14-20, Figure 14-21 and Figure 14-22).

Global statistical comparisons between the composite samples, NN estimates, ID2 estimates and OK for various NSRs were compared to assess global bias, where the NN model estimates represent de-clustered composite data. Clustering of the drill hole data can result in differences between the global means of the composites and NN estimates. Similar global means of the NN, ID2 and OK estimates indicate there is no apparent global grade bias in the model. The results summarized in Table 14-13 indicate no material grade bias was found in the block model.

Table 14-13: Comparison of Mean Estimated Grades for Various NSR Cut-Off Grades

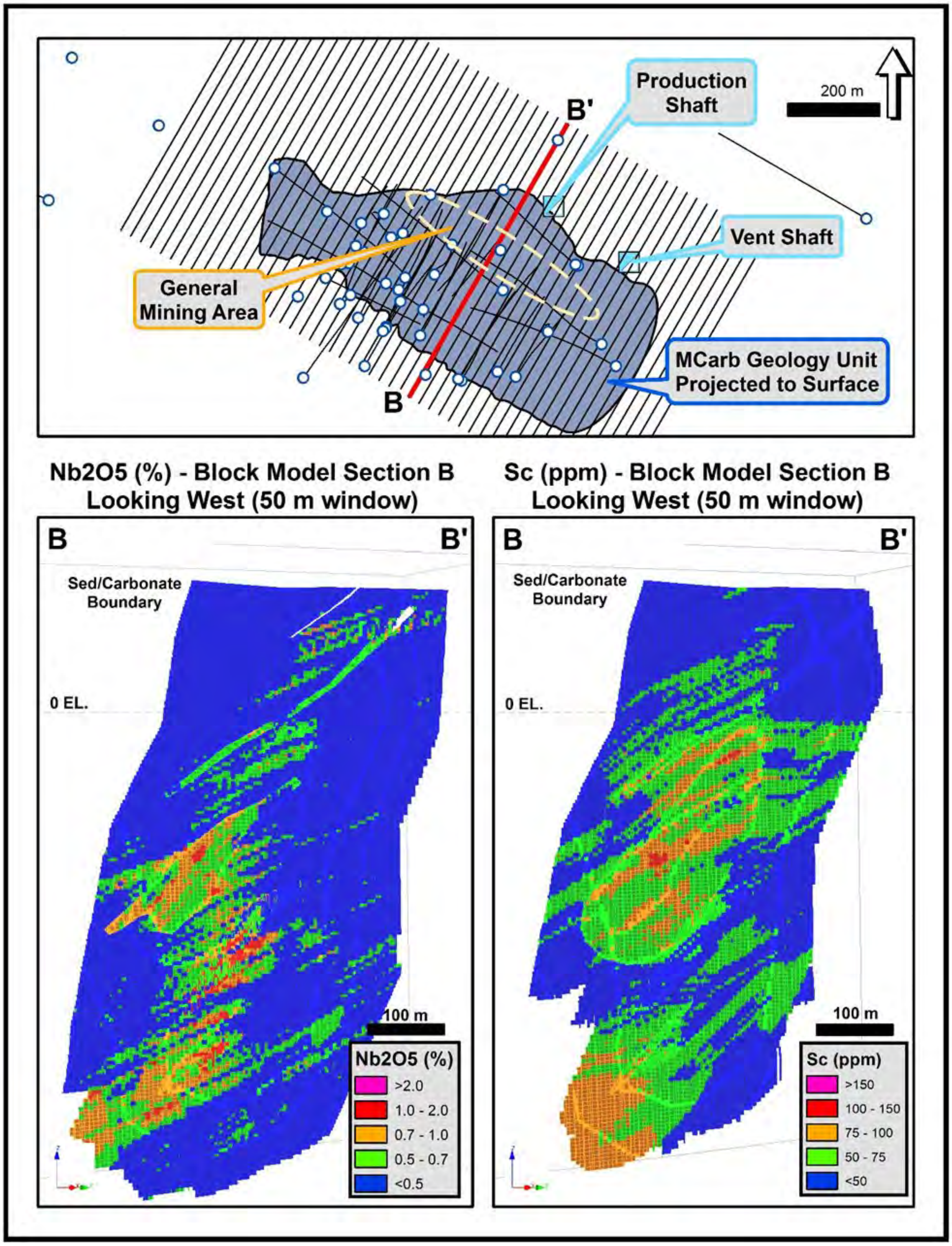
		Indicated			Inferred			Indicated + Inferred		
	NSR cut-off	NN Mean Nb_2O_5 (%)	ID Mean Nb_2O_5 (%)	OK Mean Nb_2O_5 (%)	NN Mean Nb_2O_5 (%)	ID Mean Nb_2O_5 (%)	OK Mean Nb_2O_5 (%)	NN Mean Nb_2O_5 (%)	ID Mean Nb_2O_5 (%)	OK Mean Nb_2O_5 (%)
Nb_2O_5	100	0.53	0.54	0.55	0.42	0.46	0.48	0.48	0.51	0.52
	180	0.55	0.56	0.57	0.46	0.50	0.51	0.52	0.54	0.54
	200	0.56	0.57	0.57	0.47	0.51	0.51	0.52	0.55	0.55
	300	0.60	0.61	0.61	0.53	0.56	0.56	0.58	0.60	0.60
	400	0.67	0.67	0.67	0.59	0.62	0.63	0.65	0.66	0.66
	500	0.77	0.77	0.76	0.71	0.73	0.73	0.76	0.76	0.76
	NSR cut-off	NN Mean TiO_2 (%)	ID Mean TiO_2 (%)	OK Mean TiO_2 (%)	NN Mean TiO_2 (%)	ID Mean TiO_2 (%)	OK Mean TiO_2 (%)	NN Mean TiO_2 (%)	ID Mean TiO_2 (%)	OK Mean TiO_2 (%)
TiO_2	100	2.15	2.18	2.19	1.54	1.73	1.78	1.91	2.00	2.02
	180	2.25	2.27	2.28	1.73	1.89	1.91	2.06	2.13	2.14
	200	2.28	2.30	2.30	1.77	1.92	1.94	2.10	2.16	2.17
	300	2.42	2.43	2.42	1.97	2.08	2.08	2.27	2.32	2.31
	400	2.54	2.54	2.53	2.13	2.22	2.21	2.42	2.45	2.44
	500	2.74	2.73	2.71	2.39	2.47	2.42	2.66	2.67	2.65
	NSR cut-off	NN Mean Sc (ppm)	ID Mean Sc (ppm)	OK Mean Sc (ppm)	NN Mean Sc (ppm)	ID Mean Sc (ppm)	OK Mean Sc (ppm)	NN Mean Sc (ppm)	ID Mean Sc (ppm)	OK Mean Sc (ppm)
Sc	100	57.31	57.78	57.77	38.51	42.08	41.80	49.76	51.47	51.35
	180	60.62	60.84	60.84	46.34	49.85	49.43	55.39	56.82	56.66
	200	61.64	61.76	61.75	48.37	51.63	51.18	56.88	58.13	58.00
	300	67.12	66.76	66.75	58.09	59.69	59.34	64.22	64.49	64.37
	400	73.25	72.62	72.67	66.63	67.77	67.68	71.41	71.27	71.28
	500	80.82	79.76	79.64	75.52	75.86	76.04	79.64	78.89	78.84

Note: For reference purposes, based on global classification for Indicated + Inferred resource categories



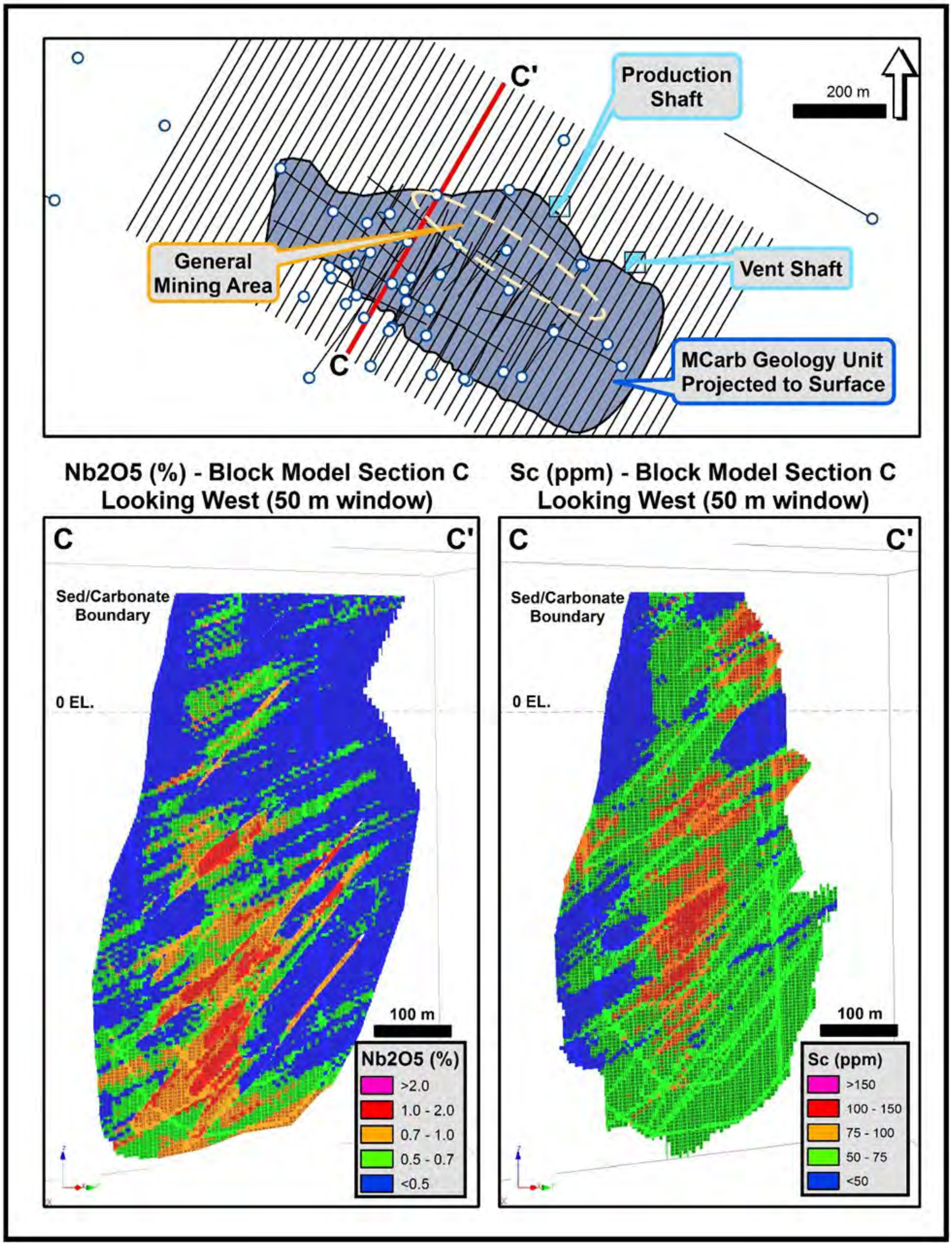
Source: Nordmin, 2019

Figure 14-20: Block Model Cross-Section A



Source: Nordmin, 2019

Figure 14-21: Block Model Cross-Section B

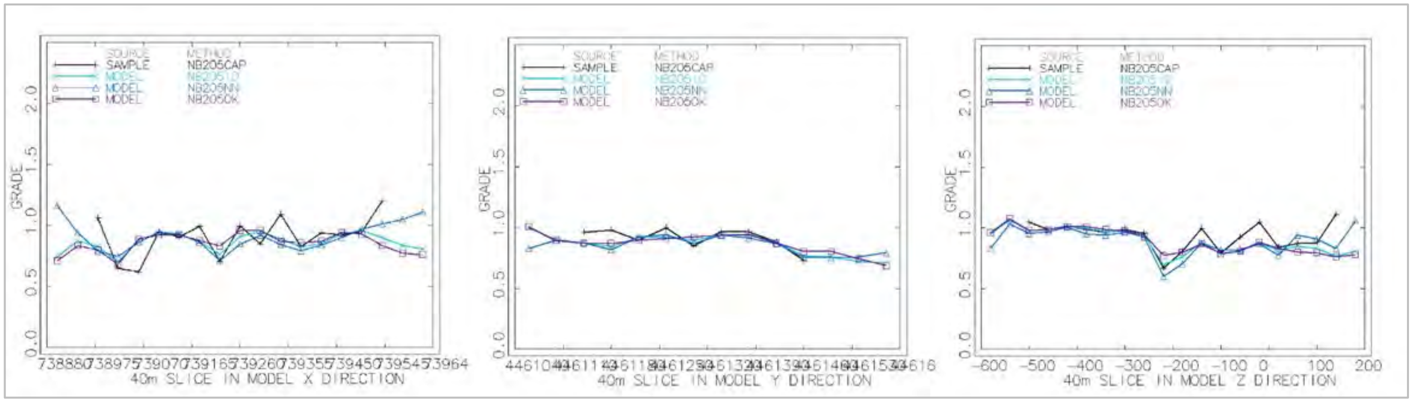


Source: Nordmin, 2019

Figure 14-22: Block Model Cross-Section C

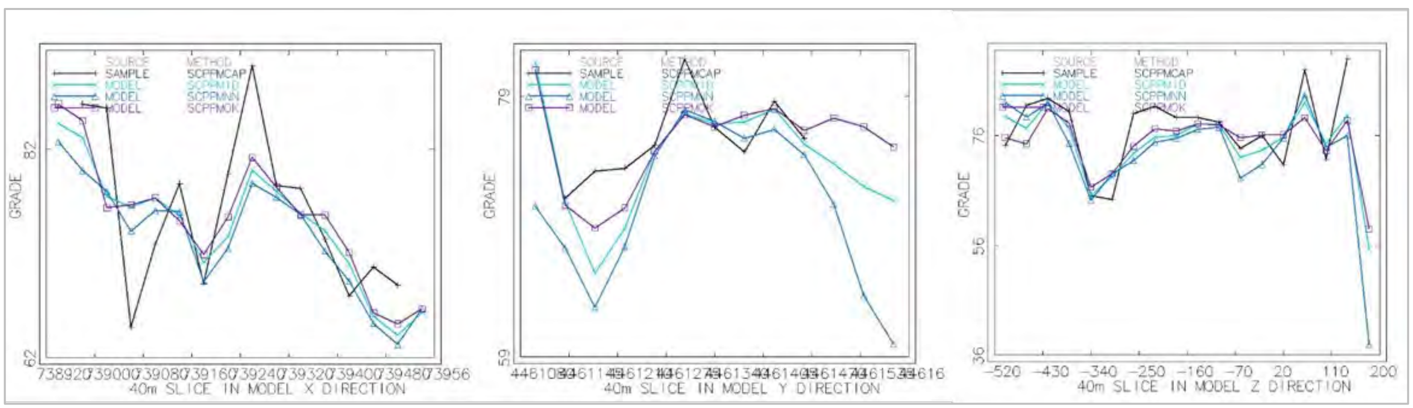
14.7.2 Swath Plots

A series of swath plots were generated for Nb_2O_5, TiO_2, and Sc from slices throughout each domain. They compare the block model grades for NN, ID2, and OK to the drill hole composite grades to evaluate any potential local grade bias. Review of the swath plots did not identify bias in the model that is material to the 2019 Mineral Resource Estimate, as there was a strong overall correlation between the block model grade and the capped composites used in the 2019 Mineral Resource Estimate (see Figure 14-23 to Figure 14-28).



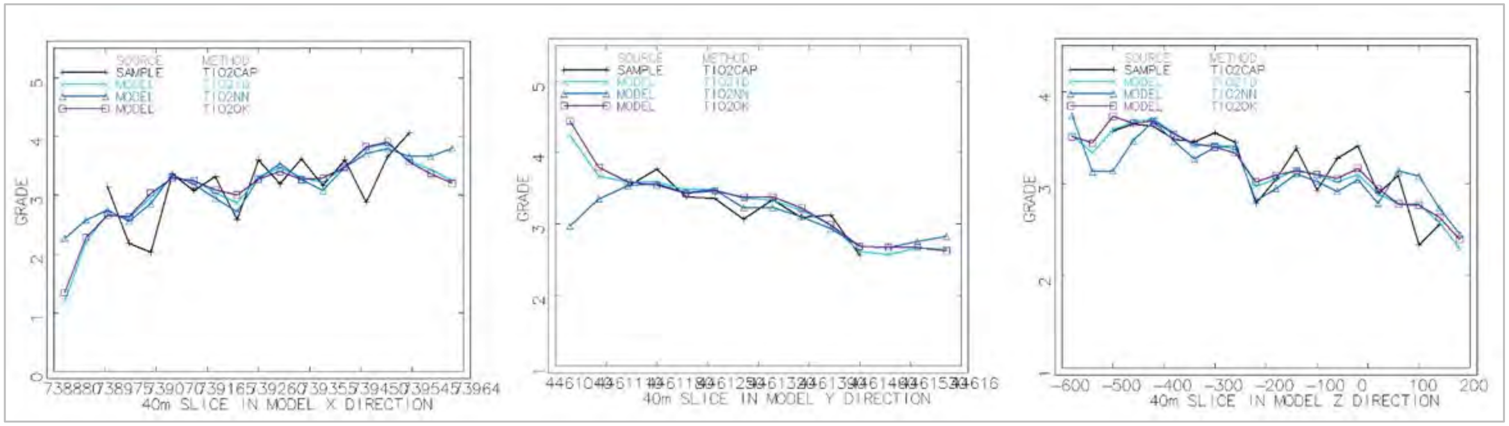
Source: Nordmin, 2019

Figure 14-23: Swath Plots for High Grade Nb_2O_5 with Composite Grades



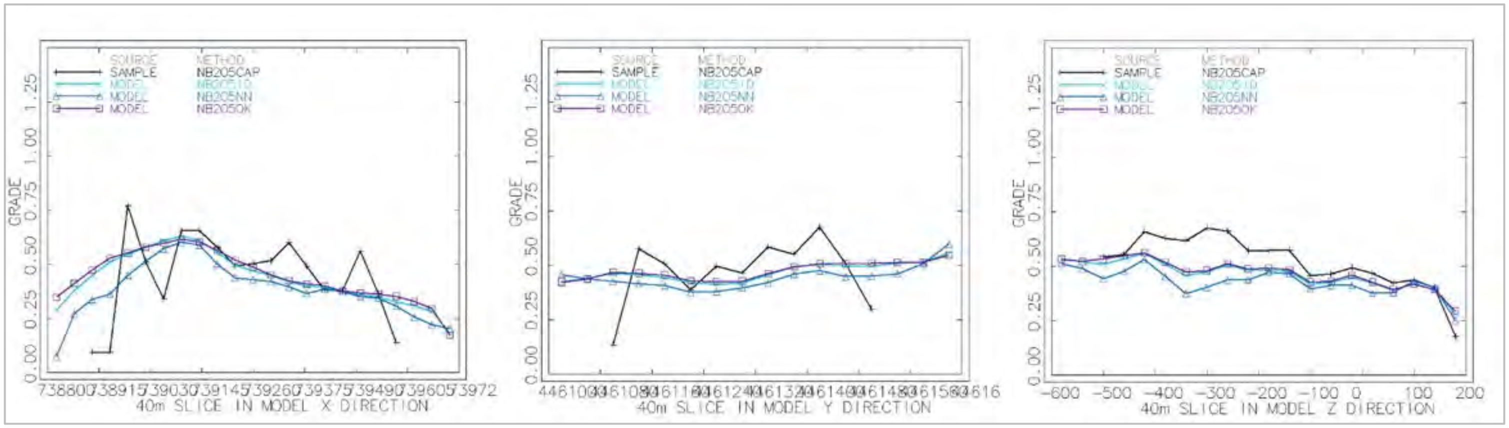
Source: Nordmin, 2019

Figure 14-24: Swath Plots for High Grade Sc with Composite Grades



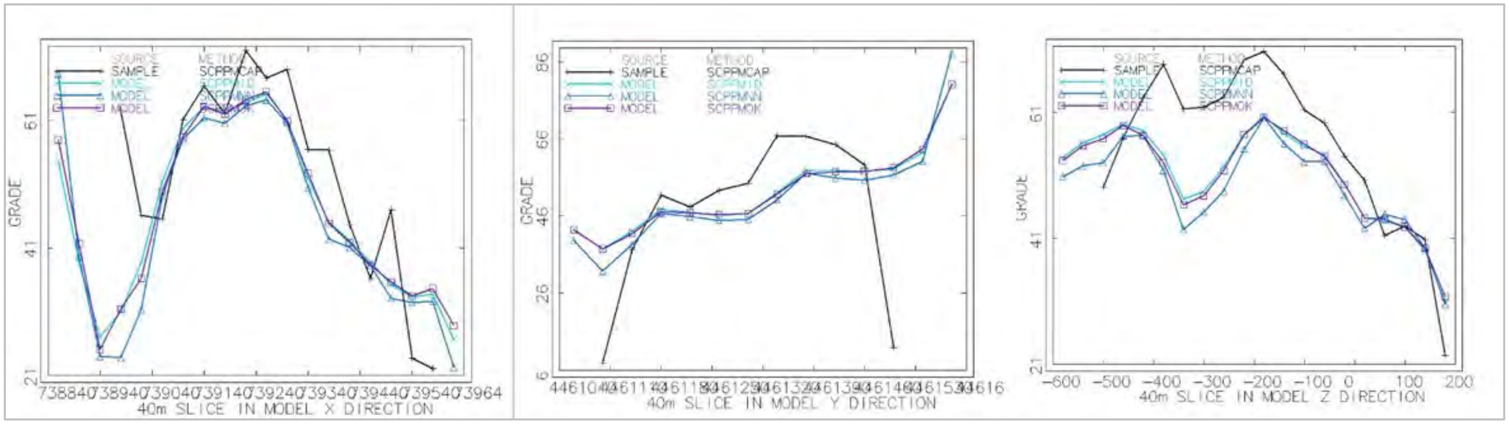
Source: Nordmin, 2019

Figure 14-25: Swath Plots for High Grade Tio$_2$ with Composite Grades



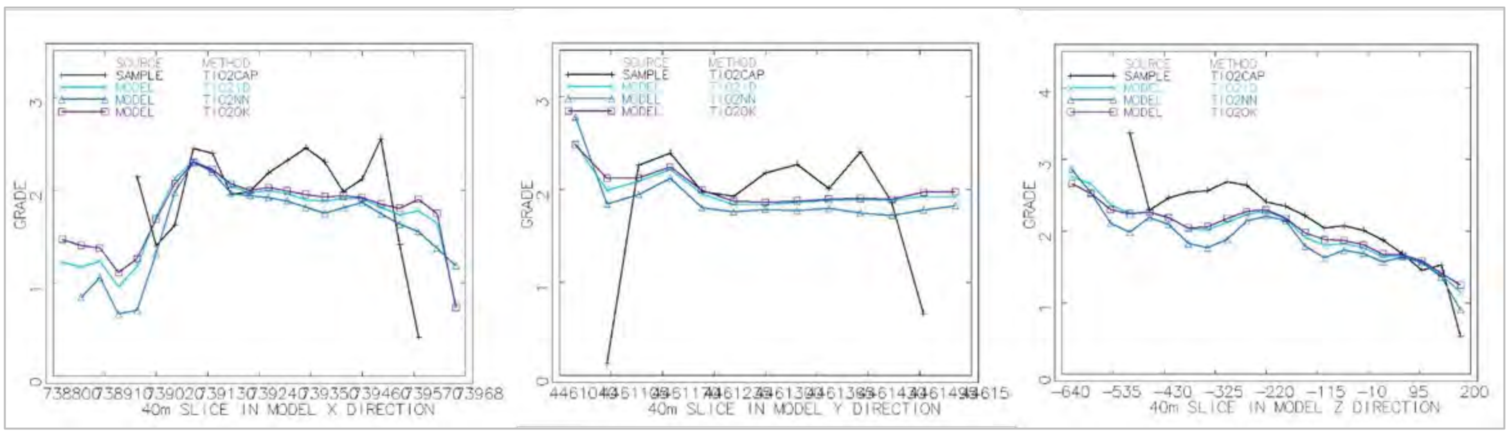
Source: Nordmin, 2019

Figure 14-26: Swath Plots for Low Grade Nb$_2$O$_5$ with Composite Grades



Source: Nordmin, 2019

Figure 14-27: Swath Plots for Low Grade Sc with Composite Grades

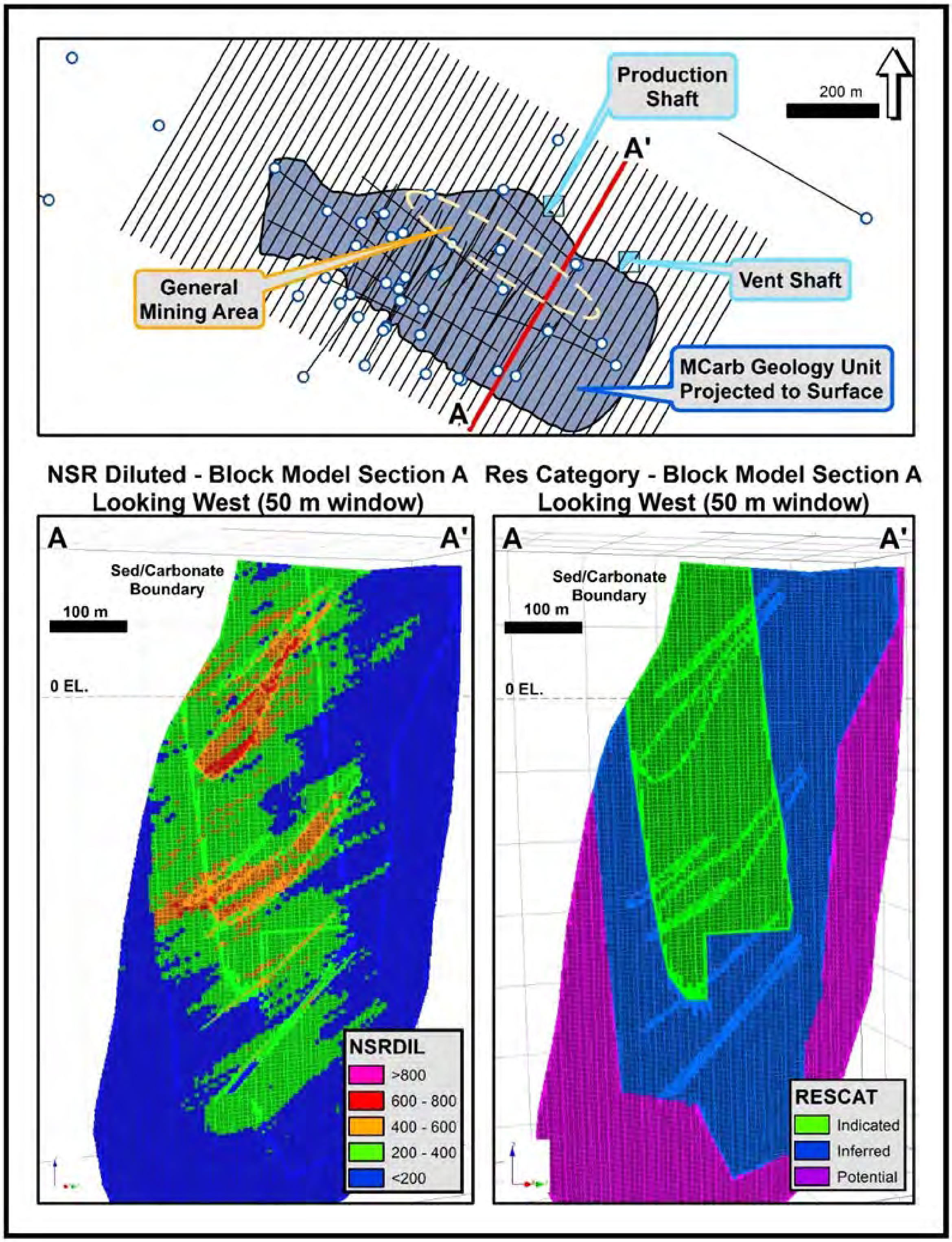


Source: Nordmin, 2019

Figure 14-28: Swath Plots for Low Grade Tio$_2$ with Composite Grades

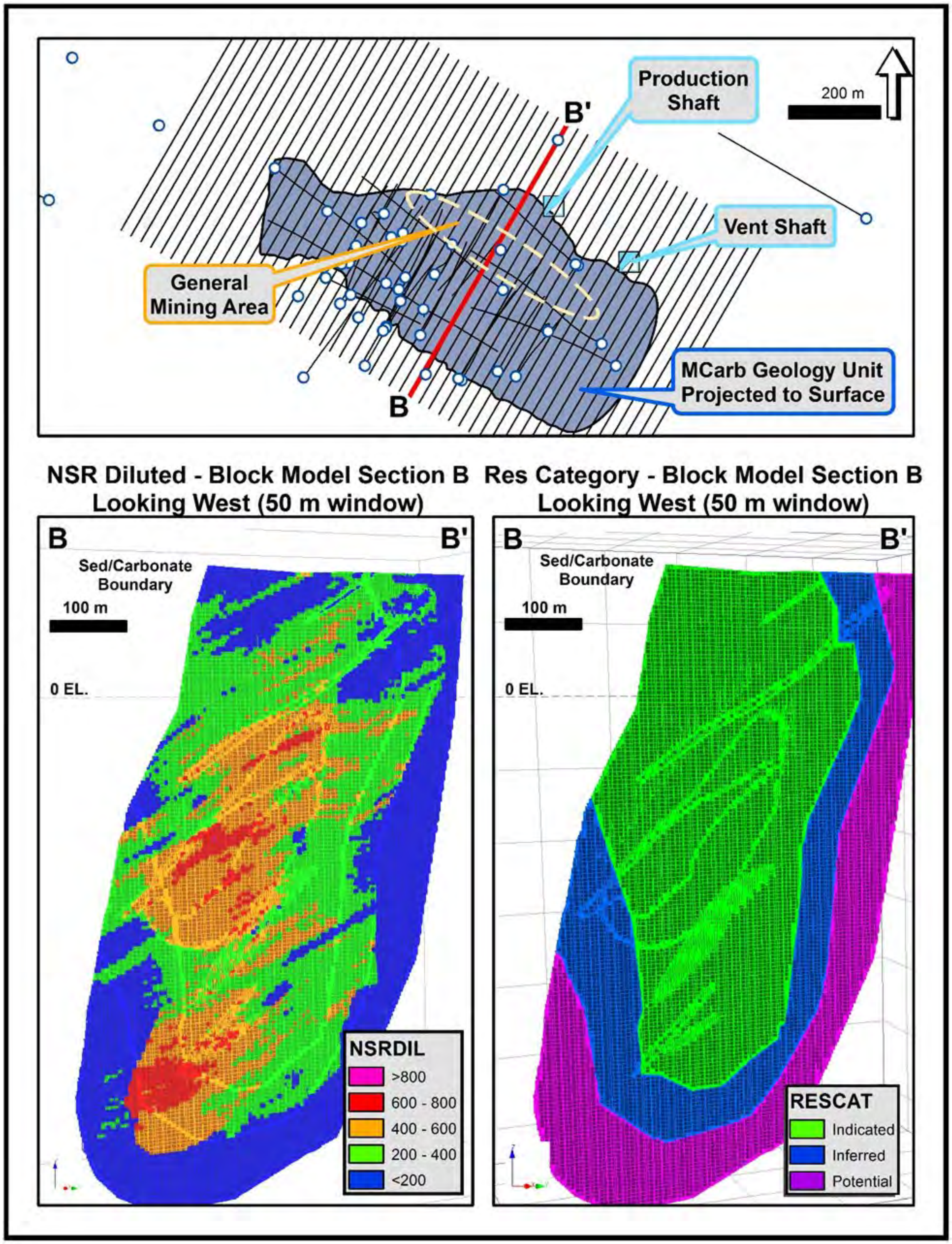
14.8 Mineral Resource Classification

The Elk Creek Deposit 2019 Mineral Resource Estimate was classified in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). Mineral resource classifications were assigned to broad regions of the block model based on Qualified Person's confidence and judgement related to geological understanding, continuity of mineralization in conjunction with data quality, spatial continuity based on variography, estimation pass, data density, and block model representativeness (see Figure 14-29, Figure 14-30 and Figure 14-31). Drill hole spacing for Mineral Resources in the Indicated and Inferred categories were approximately 50 m - 75 m and 75 m - 150 m respectively, where geology and grade continuity were reasonably understood and represented in the model. A wireframe was created past one boundary of the property for re-classifying any internal blocks as non-mineable.



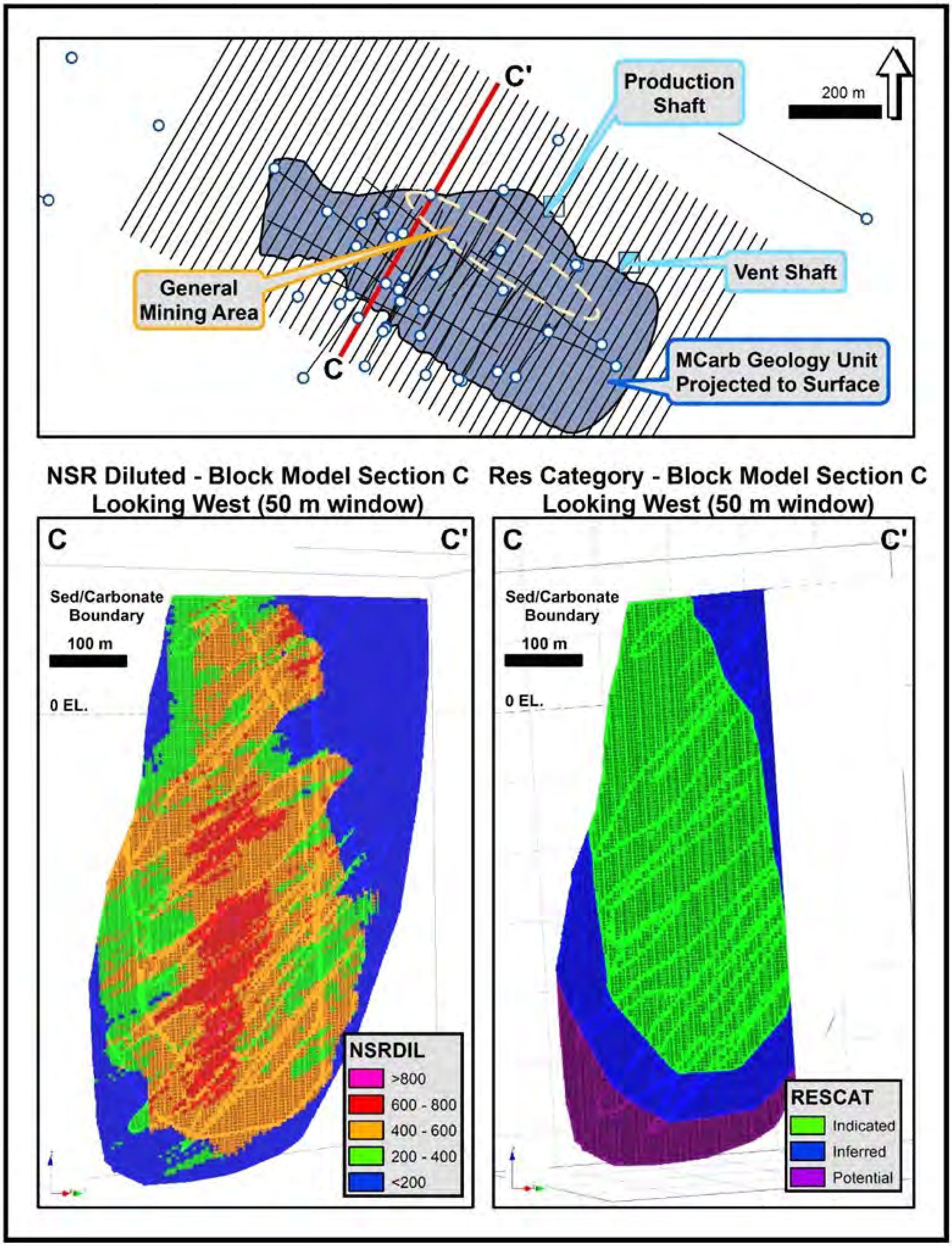
Source: Nordmin, 2019

Figure 14-29: Block Model Resource Categorization, Cross-Section A



Source: Nordmin, 2019

Figure 14-30: Block Model Resource Categorization, Cross-Section B



Source: Nordmin, 2019

Figure 14-31: Block Model Resource Categorization, Cross-Section C

The potential resource classification (RESCAT) wireframes are "exploration targets" that have potential quantity and grade that is conceptual. Further, there has been insufficient exploration to define a Mineral Resource, and it is uncertain if further exploration will result in the exploration target being delineated as a Mineral Resource. The disclosed potential quantity and grade has been determined based on internal analysis conducted by the Qualified Persons.

14.9 Reasonable Prospects of Eventual Economic Extraction

To fulfill the requirement to meet "reasonable prospects for eventual economic extraction," Nordmin estimated a potential underground mining cut-off grade using assumptions from previous technical studies and on known operating costs for UG mines operating in the region. Major portions of the project are amenable for underground extraction with a processing method to recover FeNb (as a saleable product of Nb), TiO_2 and Sc_2O_3 products.

The breakeven NSR cut-off grade (CoG) was estimated by Nordmin using input costs for mining, processing and general and administrative as outlined in Section 14.10.

14.10 Cut-Off Grade

The cut-off grade used for the 2019 Mineral Resource Estimate is an NSR of US$ 180 /tonne based on NioCorp's estimated break-even OPEX mining cost of US$ 180 per tonne and is based upon the assumptions outlined in Table 14-14.

Table 14-14: Economic Assumptions Used to Define Mineral Resource Cut-Off Value

Parameter	Value	Unit
Mining Cost	20.00	US$/t mined
Processing	125.00	US$/t mined
General and Administrative	5.00	US$/t mined
Total Cost	**180.00**	**US$/t mined**
Nb_2O_5 to Niobium conversion	69.60	%
Niobium Process Recovery	82.36	%
Niobium Price	39.60	US$/kg
TiO_2 Process Recovery	40.31	%
TiO_2 Price	0.88	US$/kg
Sc Process Recovery	93.14	%
Sc to Sc_2O_3 conversion	153.40	%
Sc Price	3,675.00	US$/kg
Calculated CoG NSR in-situ	180.00	US$/t
Calculated CoG NSR diluted 6%	180.00	US$/t

The NSR has been calculated as follows:

$$\text{NSR (US\$)} = \frac{\text{Revenue per block } Nb_2O_5 + \text{Revenue per block } TiO_2 + \text{Revenue per block Sc}}{\text{Tonnes per block}}$$

$$\text{Diluted NSR (US\$)} = \frac{\text{Revenue per block } Nb_2O_5 \text{ (diluted)} + \text{Revenue per block } TiO_2 \text{ (diluted)} + \text{Revenue per block Sc (diluted)}}{\text{Diluted tonnes per block}}$$

14.11 Mineral Resource Tabulation

The 2019 Mineral Resource Estimate for the Elk Creek Deposit is reported in accordance with NI 43-101 and has been estimated in conformity with current CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines.

This estimate includes the estimated Mineral Resources for Nb_2O_5, TiO_2, and Sc. The deposit Mineral Resources are not Mineral Reserves and do not necessarily demonstrate economic viability. There is no certainty that all or any part of this Mineral Resource will be converted into a Mineral Reserve. Inferred Mineral Resources are too speculative geologically to have economic considerations applied to them to enable them to be categorized as Mineral Reserves.

The base-case Mineral Resource Estimate is reported at an NSR cut-off of US$ 180 US$/tonne while other cut-offs are provided in order to demonstrate tonnage and grade sensitivities. The Mineral Resource Estimate is reported from within a diluted US$ 180 US$/tonne NSR to account for mineral continuity and potential mineability which excludes isolated blocks with little potential for mining.

Table 14-15 states the Indicated and Inferred Mineral Resources; Table 14-16 and Table 14-17 summarizes the sensitivity of the resource estimate to other potential mining cut-offs and Table 14-18, and Table 14-19 summarizes the changes from the 2017 Mineral Resource Estimate. The effective date of the Mineral Resource Estimate is February 19, 2019.

Table 14-15: Elk Creek Deposit 2019 Mineral Resource Estimate

Classification	Cut-off NSR (DIL) (US$/t)	Tonnage (t)	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (t)	TiO_2 Grade (%)	Contained TiO_2 (t)	Sc Grade (ppm)	Contained Sc (t)
Indicated	180	183,185,498	0.54	981,092	2.15	3,940,419	57.65	10,562
Inferred	180	103,992,535	0.48	498,864	1.81	1,886,181	47.38	4,928

Source: Nordmin, 2019. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.

- Mineral Resources are reported inclusive of the Mineral Reserve. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate and have been used to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Nordmin does not consider them to be material.
- The reporting standard adopted for the reporting of the MRE uses the terminology, definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves (May 10, 2014) as required by NI 43-101.
- CIM definition standards for Mineral Resources and Mineral Reserves (May 2014) defines a Mineral Resource as:
  - "(A) concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".
- Historical samples have been validated via re-assay programs, and all drilling completed by NioCorp has been subjected to QA/QC. All composites have been capped and then composited where appropriate, and estimates completed used ordinary kriging. The concession is wholly owned by, and exploration is operated by NioCorp Developments Ltd.

- The project is amenable to underground longhole open stoping mining methods. Using results from metallurgical test work, suitable underground mining and processing costs, and forecast product pricing Nordmin has reported the Mineral Resource at an NSR cut-off of US$ 180/tonne.
- Economic Assumptions Used to Define Mineral Resource Cut-Off Value:

$$\text{Diluted NSR (US\$)} = \frac{\text{Revenue per block } Nb_2O_5 \text{ (diluted)} + \text{Revenue per block } TiO_2 \text{ (diluted)} + \text{Revenue per block Sc (diluted)}}{\text{Diluted tonnes per block}}$$

Parameter	Value	Unit
Mining Cost	50.00	US$/t mined
Processing	125.00	US$/t mined
General and Administrative	5.00	US$/t mined
Total Cost	**180.00**	**US$/t mined**
Nb_2O_5 to Niobium conversion	69.60	%
Niobium Process Recovery	82.36	%
Niobium Price	39.60	US$/kg
TiO_2 Process Recovery	40.31	%
TiO_2 Price	0.88	US$/kg
Sc Process Recovery	93.14	%
Sc to Sc_2O_3 conversion	153.40	%
Sc Price	3,675.00	US$/kg
Calculated CoG NSR diluted 6 %	180.00	US$/t

- Price assumptions for FeNb, Sc_2O_3, and TiO_2 are based upon independent market analyses for each product.
- Price and cost assumptions are based on the pricing of products at the "mine-gate," with no additional down-stream costs required. The assumed products are a ferroniobium product (in metal form, approximately 65% Nb and 35% Fe), a titanium dioxide product in powder form, and scandium trioxide in powder form.
- The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate CoG, considering extraction scenarios and processing recoveries. Based on this requirement, Nordmin considers that major portions of the project are amenable for underground extraction with a processing method to recover FeNb (as the saleable product of Nb_2O_5), TiO_2, and Sc_2O_3 products.
- The result of positive indications from the company's metallurgical testing and development program, titanium (TiO_2) and scandium (Sc) were added to the Mineral Resource Statement in February 2015. Both metals can be recovered with simple additions to the existing process flowsheet and will possibly provide additional revenue streams that may complement the planned production of ferroniobium.
- Nordmin has provided reasonable estimates of the expected costs based on the knowledge of the style of mining (underground) and potential processing methods.
- Nordmin completed a site inspection of the deposit by Glen Kuntz, BSc, P.Geo., Consulting Specialist - Geology/Mining, an appropriate "independent qualified person" as this term is defined in NI 43-101.

14.12 Mineral Resource Sensitivity

The resources are tabulated using various NSR cut-off grades that demonstrate the robust nature of the resources for each of the NSR cut-off grades for both in-situ and diluted grade/tonnage (see Table 14-16 and Table 14-17).

Table 14-16: In-Situ Grade/Tonnage by NSR Cut-Off

Classification	NSR Cut-off	Tonnage	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (t)	TiO_2 Grade (%)	Contained TiO_2 (t)	Sc Grade (ppm)	Contained Sc (t)
Indicated	0	203,983,106	0.54	1,101,509	2.15	4,385,637	56.46	11,517
	100	199,056,651	0.54	1,074,906	2.19	4,359,341	57.77	11,500
	180	185,547,825	0.57	1,057,623	2.28	4,230,490	60.84	11,289
	200	181,158,985	0.57	1,032,606	2.3	4,166,657	61.75	11,187
	300	154,831,966	0.61	944,475	2.42	3,746,934	66.75	10,335
	400	118,785,620	0.67	795,864	2.53	3,005,276	72.67	8,632
	500	72,537,632	0.76	551,286	2.71	1,965,770	79.64	5,777
	600	30,364,982	0.91	276,321	3.01	913,986	87.38	2,653

	700	8,697,715	1.04	90,456	3.35	291,373	97.42	847
	800	1,720,152	1.12	19,266	3.54	60,893	109.79	189
	900	148,806	1.21	1,801	3.84	5,714	121.48	18
Inferred	0	154,955,659	0.45	697,300	1.65	2,556,768	36.21	5,611
	100	133,603,630	0.48	641,297	1.78	2,378,145	41.79	5,583
	180	107,252,537	0.51	546,988	1.91	2,048,523	49.43	5,301
	200	101,383,875	0.51	517,058	1.94	1,966,847	51.18	5,189
	300	73,389,799	0.56	410,983	2.08	1,526,508	59.34	4,355
	400	45,853,910	0.63	288,880	2.21	1,013,371	67.68	3,103
	500	20,844,827	0.73	152,167	2.44	508,614	76.04	1,585
	600	5,442,409	0.87	47,349	2.77	150,755	85.87	467
	700	1,079,948	1.02	11,015	3.09	33,370	97.11	105
	800	162,504	1.17	1,901	3.52	5,720	104.77	17
	900	348	1.35	5	3.24	11	114.86	0

Source: Nordmin, 2019. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.

Table 14-17: Diluted Grade/Tonnage, by NSR Cut-Off

Classification	NSR Cut-off	Tonnage	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (t)	TiO_2 Grade (%)	Contained TiO_2 (t)	Sc Grade (PPM)	Contained Sc (t)
Indicated	0	203,983,104	0.51	1,040,314	2.02	4,120,459	53.08	10,827
	100	198,287,692	0.51	1,011,267	2.06	4,084,726	54.48	10,803
	180	183,185,498	0.54	989,202	2.15	3,938,488	57.65	10,561
	200	178,253,124	0.54	962,567	2.18	3,885,918	58.6	10,446
	300	149,300,722	0.58	865,944	2.29	3,418,987	63.66	9,504
	400	107,995,291	0.65	701,969	2.41	2,602,687	69.83	7,541
	500	58,898,903	0.75	441,742	2.62	1,543,151	76.88	4,528
	600	19,782,363	0.91	180,020	2.96	585,558	85.25	1,686
	700	4,714,614	1.02	48,089	3.23	152,282	96.14	453
	800	636,150	1.09	6,934	3.44	21,884	108.79	69
	900	16,783	1.15	193	3.66	614	122.35	2
Inferred	0	154,955,656	0.42	650,814	1.55	2,401,813	34.04	5,275
	100	131,547,190	0.45	591,962	1.68	2,209,993	39.84	5,241
	180	103,992,535	0.48	499,164	1.81	1,882,265	47.38	4,927
	200	98,029,886	0.49	480,346	1.84	1,803,750	49.05	4,808
	300	68,182,922	0.54	368,188	1.97	1,343,204	57.23	3,902
	400	39,215,057	0.61	239,212	2.12	831,359	65.51	2,569
	500	15,068,144	0.73	109,997	2.39	360,129	73.82	1,112
	600	2,974,371	0.87	25,877	2.69	80,011	84.93	253
	700	529,654	1.02	5,402	3.08	16,313	94.55	50
	800	31,605	1.16	367	3.35	1,059	102.39	3
	900	0	n/a	0	n/a	0	n/a	0

Source: Nordmin, 2019. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.

14.13 Comparison with the Previous Estimate

Changes between the 2017 and 2019 Mineral Resource Estimates are primarily due to a reinterpretation of geological and structural controls and physical characteristics between the different mineral phases (see Table 14-18 and Table 14-19). Nordmin elected to create hard boundaries to separate the high grade mineralization from the low grade mineralization for each zone within each domain. This approach has the advantage of being able to interpret the mineralization in context with the deposit geology and associated geochemistry using explicit modelling, rather than the 2017 model which used implicit modelling. The explicit modelling separately grouped the low-high grade assays within each wireframe which constrained the grade estimate within each wireframe and prevented samples from one high grade wireframe from influencing grade within another high grade wireframe. Additionally, the measured specific gravity values demonstrated that as the specific gravity increases with increasing Nb_2O_5, TiO_2, and Fe_2O_3 grades, therefore the actual specific gravity values were adjusted to reflect these changing grade distributions.

Overall, comparing the cut-off grade of US$ 180 NSR/tonne resulted in an increase in tonnage and contained metal and a slight decrease in Nb_2O_5, TiO_2 and Sc grades for the Indicated category while the Inferred category has had a decrease in tonnage and contained metal.

In Nordmin's opinion, the approach outlined in Section 14 is best suited for this deposit and has minimized resource estimation risks associated with the deposit.

Table 14-18: NSR Comparison SRK 2017 In-Situ Versus Nordmin Diluted

Classification	Cut-off	Tonnage	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (x1000 tonnes)	TiO_2 Grade (%)	Contained TiO_2 (x1000 tonnes)	Sc Grade (PPM)	Contained Sc (x1000 tonnes)
NSR=100								
2017 Mineral Resource Estimate, In-Situ								
Indicated	100	92,645,546	0.65	602,196	2.58	2,390,255	69.3	6,420
Inferred	100	149,298,438	0.47	701,703	2.18	3,254,706	58.3	8,704
2018 Mineral Resource Estimate, Diluted								
Indicated	100	198,287,692	0.51	1,011,267	2.06	4,084,726	54.5	10,803
Inferred	100	131,547,190	0.45	591,962	1.68	2,209,993	39.8	5,241
Change, 2017 to 2018								
Indicated Change		+105,642,146	-0.14	+409,071	-0.52	+1,694,471	-14.82	+4,382
Percentage Change		+114.0%	-21.5%	+67.9%	-20.2%	+70.9%	-21.4%	+68.3%
Inferred Change		-17,751,248	-0.02	-109,740	-0.50	-1,044,713	-18.46	-3,463
Percentage Change		-11.9%	-4.3%	-15.6%	-22.9%	-32.1%	-31.7%	-39.8%
NSR=180								
2017 Mineral Resource Estimate, In-Situ*								
Indicated	180	90,900,000	0.66	598,800	2.59	2,353,900	70.0	6,300
Inferred	180	133,400,000	0.48	643,300	2.24	2,983,100	59.0	7,800
2018 Mineral Resource Estimate, Diluted								
Indicated	180	183,185,498	0.54	989,202	2.15	3,938,488	57.7	10,561
Inferred	180	103,992,535	0.48	499,164	1.81	1,882,265	47.4	4,927
Change, 2017 to 2018								
Indicated Change		+92,285,498	-0.12	+390,402	-0.44	+1,584,588	-12.35	+4,261
Percentage Change		+101.5%	-18.2%	+65.2%	-17.0%	+67.3%	-17.6%	+67.6%
Inferred Change		-29,407,465	0.00	-144,136	-0.43	-1,100,835	-11.62	-2,873
Percentage Change		-22.0%	0.0%	-22.4%	-19.2%	-36.9%	-19.7%	-36.8%
NSR=200								
2017 Mineral Resource Estimate, In-Situ								
Indicated	200	91,115,250	0.66	601,361	2.60	2,368,997	70.0	6,378
Inferred	200	144,169,295	0.48	692,013	2.21	3,186,141	59.6	8,592
2018 Mineral Resource Estimate, Diluted								
Indicated	200	178,253,124	0.54	962,567	2.18	3,885,918	58.6	10,446
Inferred	200	98,029,886	0.49	480,346	1.84	1,803,750	49.1	4,808
Change, 2017 to 2018								
Indicated Change		+87,137,874	-0.12	+361,206	-0.42	+1,516,922	-11.40	+4,068
Percentage Change		+95.6%	-18.2%	+60.1%	-16.2%	+64.0%	-16.3%	+63.8%
Inferred Change		-46,139,409	+0.01	-211,666	-0.37	-1,382,392	-10.55	-3,784
Percentage Change		-32.0%	+2.1%	-30.6%	-16.7%	-43.4%	-17.7%	-44.0%
NSR=300								
2017 Mineral Resource Estimate, In-Situ								
Indicated	300	85,999,510	0.69	591,693	2.65	2,276,162	71.7	6,170
Inferred	300	122,638,624	0.52	637,060	2.32	2,849,474	63.2	7,748
2018 Mineral Resource Estimate, Diluted								
Indicated	300	149,300,722	0.58	865,944	2.29	3,418,987	63.7	9,504
Inferred	300	68,182,922	0.54	368,188	1.97	1,343,204	57.2	3,902
Change, 2017 to 2018								
Indicated Change		+63,301,212	-0.11	+274,251	-0.36	+1,142,825	-8.08	+3,335
Percentage Change		+73.6%	-15.7%	+46.4%	-13.5%	+50.2%	-11.3%	+54.1%
Inferred Change		-54,455,702	+0.02	-268,872	-0.35	-1,506,271	-5.95	-3,846
Percentage Change		-44.4%	+4.0%	-42.2%	-15.2%	-52.9%	-9.4%	-49.6%
NSR=400								
2017 Mineral Resource Estimate, In-Situ								
Indicated	400	73,415,023	0.74	544,271	2.76	2,025,174	75.0	5,503
Inferred	400	69,852,339	0.65	455,600	2.63	1,836,373	70.2	4,903
2018 Mineral Resource Estimate, Diluted								
Indicated	400	107,995,291	0.65	701,969	2.41	2,602,687	69.8	7,541
Inferred	400	39,215,057	0.61	239,212	2.12	831,359	65.5	2,569
Change, 2017 to 2018								
Indicated Change		+34,580,268	-0.09	+157,698	-0.35	+577,513	-5.12	+2,039
Percentage Change		+47.1%	-12.3%	+29.0%	-12.6%	+28.5%	-6.8%	+37.0%
Inferred Change		-30,637,282	-0.04	-216,388	-0.51	-1,005,013	-4.69	-2,334
Percentage Change		-43.9%	-6.5%	-47.5%	-19.4%	-54.7%	-6.7%	-47.6%

Classification	Cut-off	Tonnage	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (x1000 tonnes)	TiO_2 Grade (%)	Contained TiO_2 (x1000 tonnes)	Sc Grade (PPM)	Contained Sc (x1000 tonnes)
NSR=500								
2017 Mineral Resource Estimate, In-Situ								
Indicated	500	51,795,343	0.81	420,136	2.90	1,500,307	80.0	4,142
Inferred	500	32,431,494	0.77	250,955	2.86	928,492	78.5	2,547
2018 Mineral Resource Estimate, Diluted								
Indicated	500	58,898,903	0.75	441,742	2.62	1,543,151	76.9	4,528
Inferred	500	15,068,144	0.73	109,997	2.39	360,129	73.8	1,112
Change, 2017 to 2018								
Indicated Change		+7,103,560	-0.06	+21,606	-0.28	+42,845	-3.08	+387
Percentage Change		+13.7%	-7.5%	+5.1%	-9.5%	+2.9%	-3.9%	+9.3%
Inferred Change		-17,363,350	-0.04	-140,958	-0.47	-568,364	-4.72	-1,435
Percentage Change		-53.5%	-5.7%	-56.2%	-16.5%	-61.2%	-6.0%	-56.3%
NSR=600								
2017 Mineral Resource Estimate, In-Situ								
Indicated	600	19,956,473	0.88	174,770	3.00	598,694	89.3	1,782
Inferred	600	9,516,767	0.86	81,838	2.99	284,551	88.4	841
2018 Mineral Resource Estimate, Diluted								
Indicated	600	19,782,363	0.91	180,020	2.96	585,558	85.3	1,686
Inferred	600	2,974,371	0.87	25,877	2.69	80,011	84.9	253
Change, 2017 to 2018								
Indicated Change		-174,110	+0.03	+5,249	-0.04	-13,136	-4.02	-95
Percentage Change		-0.9%	+3.9%	+3.0%	-1.3%	-2.2%	-4.5%	-5.3%
Inferred Change		-6,542,396	+0.01	-55,961	-0.30	-204,541	-3.45	-588
Percentage Change		-68.7%	+1.2%	-68.4%	-10.0%	-71.9%	-3.9%	-70.0%
NSR=700								
2017 Mineral Resource Estimate, In-Situ								
Indicated	700	2,734,381	0.96	26,250	3.12	85,313	102.6	281
Inferred	700	831,460	0.82	6,818	2.56	21,285	108.7	90
2018 Mineral Resource Estimate, Diluted								
Indicated	700	4,714,614	1.02	48,089	3.23	152,282	96.1	453
Inferred	700	529,654	1.02	5,402	3.08	16,313	94.6	50
Change, 2017 to 2018								
Indicated Change		+1,980,233	+0.06	+21,839	+0.11	+66,969	-6.50	+173
Percentage Change		+72.4%	+6.3%	+83.2%	+3.5%	+78.5%	-6.3%	+61.5%
Inferred Change		-301,806	+0.20	-1,416	+0.52	-4,972	-14.15	-40
Percentage Change		-36.3%	+24.4%	-20.8%	+20.3%	-23.4%	-13.0%	-44.6%
NSR=800								
2017 Mineral Resource Estimate, In-Situ								
Indicated	800	160,413	1.10	1,765	3.40	5,454	117.0	19
Inferred	800	433	1.14	5	3.66	16	109.1	0
2018 Mineral Resource Estimate, Diluted								
Indicated	800	636,150	1.09	6,934	3.44	21,884	108.8	69
Inferred	800	31,605	1.16	367	3.35	1,059	102.4	3
Change, 2017 to 2018								
Indicated Change		+475,737	-0.01	+5,169	+0.04	+16,430	-8.17	+50
Percentage Change		+296.6%	-0.9%	+293.0%	+1.2%	+301.2%	-7.0%	+268.9%
Inferred Change		+31,172	+0.02	+362	-0.31	+1,043	-6.72	3
Percentage Change		+7199.1%	+1.8%	+7327.1%	-8.5%	+6580.8%	-6.2%	+6749.5%
NSR=900								
2017 Mineral Resource Estimate, In-Situ								
Indicated	900	11,632	1.47	171	4.13	480	131.7	2
Inferred	900	0	n/a	0	n/a	0	n/a	0
2018 Mineral Resource Estimate, Diluted								
Indicated	900	16,783	1.15	193	3.66	614	122.4	2
Inferred	900	0	n/a	0	n/a	0	n/a	0
Change, 2017 to 2018								
Indicated Change		+5,151	-0.32	+22	-0.47	+134	-9.32	+1
Percentage Change		+44.3%	-21.8%	+12.9%	-11.4%	+27.9%	-7.1%	+34.1%
Inferred Change		0	n/a	0	n/a	0	n/a	0
Percentage Change		n/a	n/a	n/a	n/a	n/a	n/a	n/a

Source: Nordmin, 2019. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.

Table 14-19: NSR Comparison SRK 2017 In-Situ Versus Nordmin In-Situ

Classification	Cut-off	Tonnage	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (x1000 tonnes)	TiO_2 Grade (%)	Contained TiO_2 (x1000 tonnes)	Sc Grade (PPM)	Contained Sc (x1000 tonnes)
NSR=100								
2017 Mineral Resource Estimate, In-Situ								
Indicated	100	92,645,546	0.65	602,196	2.58	2,390,255	69.3	6,420
Inferred	100	149,298,438	0.47	701,703	2.18	3,254,706	58.3	8,704
2018 Mineral Resource Estimate, In-Situ								
Indicated	100	199,056,651	0.54	1,074,906	2.19	4,359,341	57.8	11,500
Inferred	100	133,603,630	0.48	641,297	1.78	2,378,145	41.8	5,583
Change, 2017 to 2018								
Indicated Change		+106,411,105	-0.11	+472,710	-0.39	+1,969,086	-11.53	+5,079
Percentage Change		+114.9%	-16.9%	+78.5%	-15.1%	+82.4%	-16.6%	+79.1%
Inferred Change		-15,694,808	+0.01	-60,405	-0.40	-876,561	-16.51	-3,121
Percentage Change		-10.5%	+2.1%	-8.6%	-18.3%	-26.9%	-28.3%	-35.9%
NSR=180								
2017 Mineral Resource Estimate, In-Situ*								
Indicated	180	90,900,000	0.66	598,800	2.59	2,353,900	70.0	6,300
Inferred	180	133,400,000	0.48	643,300	2.24	2,983,100	59.0	7,800
2018 Mineral Resource Estimate, In-Situ								
Indicated	180	185,547,825	0.57	1,057,623	2.28	4,230,490	60.8	11,289
Inferred	180	107,252,537	0.51	546,988	1.91	2,048,523	49.4	5,301
Change, 2017 to 2018								
Indicated Change		+94,647,825	-0.09	+458,823	-0.31	+1,876,590	-9.16	+4,989
Percentage Change		+104.1%	-13.6%	+76.6%	-12.0%	+79.7%	-13.1%	+79.2%
Inferred Change		-26,147,463	+0.03	-96,312	-0.33	-934,577	-9.57	-2,499
Percentage Change		-19.6%	+6.3%	-15.0%	-14.7%	-31.3%	-16.2%	-32.0%
NSR=200								
2017 Mineral Resource Estimate, In-Situ								
Indicated	200	91,115,250	0.66	601,361	2.60	2,368,997	70.0	6,378
Inferred	200	144,169,295	0.48	692,013	2.21	3,186,141	59.6	8,592
2018 Mineral Resource Estimate, In-Situ								
Indicated	200	181,158,985	0.57	1,032,606	2.30	4,166,657	61.8	11,187
Inferred	200	101,383,875	0.51	517,058	1.94	1,966,847	51.2	5,189
Change, 2017 to 2018								
Indicated Change		+90,043,735	-0.09	+431,246	-0.30	+1,797,660	-8.25	+4,808
Percentage Change		+98.8%	-13.6%	+71.7%	-11.5%	+75.9%	-11.8%	+75.4%
Inferred Change		-42,785,420	+0.03	-174,955	-0.27	-1,219,294	-8.42	-3,404
Percentage Change		-29.7%	+6.3%	-25.3%	-12.2%	-38.3%	-14.1%	-39.6%
NSR=300								
2017 Mineral Resource Estimate, In-Situ								
Indicated	300	85,999,510	0.69	591,693	2.65	2,276,162	71.7	6,170
Inferred	300	122,638,624	0.52	637,060	2.32	2,849,474	63.2	7,748
2018 Mineral Resource Estimate, In-Situ								
Indicated	300	154,831,966	0.61	944,475	2.42	3,746,934	66.8	10,335
Inferred	300	73,389,799	0.56	410,983	2.08	1,526,508	59.3	4,355
Change, 2017 to 2018								
Indicated Change		+68,832,456	-0.08	+352,782	-0.23	+1,470,772	-4.99	+4,165
Percentage Change		+80.0%	-11.3%	+59.6%	-8.6%	+64.6%	-7.0%	+67.5%
Inferred Change		-49,248,825	+0.04	-226,077	-0.24	-1,322,966	-3.84	-3,393
Percentage Change		-40.2%	+7.8%	-35.5%	-10.5%	-46.4%	-6.1%	-43.8%
NSR=400								
2017 Mineral Resource Estimate, In-Situ								
Indicated	400	73,415,023	0.74	544,271	2.76	2,025,174	75.0	5,503
Inferred	400	69,852,339	0.65	455,600	2.63	1,836,373	70.2	4,903
2018 Mineral Resource Estimate, In-Situ								
Indicated	400	118,785,620	0.67	795,864	2.53	3,005,276	72.7	8,632
Inferred	400	45,853,910	0.63	288,880	2.21	1,013,371	67.7	3,103
Change, 2017 to 2018								
Indicated Change		+45,370,597	-0.07	+251,593	-0.23	+980,102	-2.28	+3,130
Percentage Change		+61.8%	-9.6%	+46.2%	-8.3%	+48.4%	-3.0%	+56.9%
Inferred Change		-23,998,429	-0.02	-166,720	-0.42	-823,001	-2.52	-1,800
Percentage Change		-34.4%	-3.4%	-36.6%	-15.9%	-44.8%	-3.6%	-36.7%

Classification	Cut-off	Tonnage	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (x1000 tonnes)	TiO_2 Grade (%)	Contained TiO_2 (x1000 tonnes)	Sc Grade (PPM)	Contained Sc (x1000 tonnes)
NSR=500								
2017 Mineral Resource Estimate, In-Situ								
Indicated	500	51,795,343	0.81	420,136	2.90	1,500,307	80.0	4,142
Inferred	500	32,431,494	0.77	250,955	2.86	928,492	78.5	2,547
2018 Mineral Resource Estimate, In-Situ								
Indicated	500	72,537,632	0.76	551,286	2.71	1,965,770	79.6	5,777
Inferred	500	20,844,827	0.73	152,167	2.44	508,614	76.0	1,585
Change, 2017 to 2018								
Indicated Change		+20,742,289	-0.05	+131,150	-0.19	+465,463	-0.32	+1,635
Percentage Change		+40.0%	-6.3%	+31.2%	-6.4%	+31.0%	-0.4%	+39.5%
Inferred Change		-11,586,667	-0.04	-98,788	-0.42	-419,878	-2.50	-962
Percentage Change		-35.7%	-5.7%	-39.4%	-14.8%	-45.2%	-3.2%	-37.8%
NSR=600								
2017 Mineral Resource Estimate, In-Situ								
Indicated	600	19,956,473	0.88	174,770	3.00	598,694	89.3	1,782
Inferred	600	9,516,767	0.86	81,838	2.99	284,551	88.4	841
2018 Mineral Resource Estimate, In-Situ								
Indicated	600	30,364,982	0.91	276,321	3.01	913,986	87.4	2,653
Inferred	600	5,442,409	0.87	47,349	2.77	150,755	85.9	467
Change, 2017 to 2018								
Indicated Change		+10,408,509	+0.03	+101,551	+0.01	+315,292	-1.89	+872
Percentage Change		+52.2%	+3.9%	+58.1%	+0.3%	+52.7%	-2.1%	+48.9%
Inferred Change		-4,074,358	+0.01	-34,490	-0.22	-133,797	-2.51	-374
Percentage Change		-42.8%	+1.2%	-42.1%	-7.4%	-47.0%	-2.8%	-44.4%
NSR=700								
2017 Mineral Resource Estimate, In-Situ								
Indicated	700	2,734,381	0.96	26,250	3.12	85,313	102.6	281
Inferred	700	831,460	0.82	6,818	2.56	21,285	108.7	90
2018 Mineral Resource Estimate, In-Situ								
Indicated	700	8,697,715	1.04	90,456	3.35	291,373	97.4	847
Inferred	700	1,079,948	1.02	11,015	3.09	33,370	97.1	105
Change, 2017 to 2018								
Indicated Change		+5,963,334	+0.08	+64,206	+0.23	+206,061	-5.22	+567
Percentage Change		+218.1%	+8.3%	+244.6%	+7.4%	+241.5%	-5.1%	+201.9%
Inferred Change		+248,488	+0.20	+4,197	+0.53	+12,085	-11.59	14
Percentage Change		+29.9%	+24.4%	+61.6%	+20.7%	+56.8%	-10.7%	+16.0%
NSR=800								
2017 Mineral Resource Estimate, In-Situ								
Indicated	800	160,413	1.10	1,765	3.40	5,454	117.0	19
Inferred	800	433	1.14	5	3.66	16	109.1	0
2018 Mineral Resource Estimate, In-Situ								
Indicated	800	1,720,152	1.12	19,266	3.54	60,893	109.8	189
Inferred	800	162,504	1.17	1,901	3.52	5,720	104.8	17
Change, 2017 to 2018								
Indicated Change		+1,559,739	+0.02	+17,501	+0.14	+55,439	-7.17	+170
Percentage Change		+972.3%	+1.8%	+991.8%	+4.1%	+1016.5%	-6.1%	+906.6%
Inferred Change		+162,071	+0.03	+1,896	-0.14	+5,704	-4.34	17
Percentage Change		+37429.8%	+2.6%	+38417.4%	-3.8%	+35994.2%	-4.0%	+35937.0%
NSR=900								
2017 Mineral Resource Estimate, In-Situ								
Indicated	900	11,632	1.47	171	4.13	480	131.7	2
Inferred	900	0	n/a	0	n/a	0	n/a	0
2018 Mineral Resource Estimate, In-Situ								
Indicated	900	148,806	1.21	1,801	3.84	5,714	121.5	18
Inferred	900	348	1.35	5	3.24	11	114.9	0
Change, 2017 to 2018								
Indicated Change		+137,174	-0.26	+1,630	-0.29	+5,234	-10.19	+17
Percentage Change		+1179.3%	-17.7%	+953.0%	-7.0%	+1089.5%	-7.7%	+1080.3%
Inferred Change		+348	n/a	+5	n/a	+11	n/a	0
Percentage Change		n/a	n/a	n/a	n/a	n/a	n/a	n/a

Source: Nordmin, 2019. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.

14.14 Relevant Factors

Factors that may affect the 2019 resource estimate include: product price assumptions; changes in local interpretations of mineralization geometry and continuity of mineralization zones; changes to geotechnical, hydrogeological, and metallurgical recovery assumptions; input factors used to assess reasonable prospects for eventual economic extraction; assumptions as to social, permitting and environmental conditions; and additional infill or step out drilling or results obtained from the planned 2019 drill program.

Nordmin is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate that is not discussed in this Technical Report. As per NI 43-101, Mineral Resources, which are not Mineral Reserves, do not have to demonstrate economic viability.

15. MINERAL RESERVE ESTIMATE

15.1 Introduction

The Project is currently in the exploration phase and has not been developed. Based on geotechnical information and mineralization geometry, an underground longhole stoping method (LHS) has been determined to be suitable for the deposit. Paste backfill will be used to allow for high recovery of material.

The stopes dimensions are 15 m wide, and stope length varies based on Nb2O5 mineralization grade to a maximum of 25 m per panel with a level spacing of 40 m. The variation on stope length allowed for optimization of the Nb2O5 grade with a minimal increase to operating costs. The level spacing of 40 m was beneficial to operating and sustaining capital costs. Each block is mined with a bottom-up sequence. A partial sill pillar level is designed to be left between these two mining fronts/blocks. The extraction of ore from the partial sill pillar level is expected to be 62.5% using production up-holes through 25 m of the 40 m thick sill pillar and is accounted for within the reserves. This methodology will allow partial mining of ore on the sill pillar level, while at the same time allowing the development of the lower mining block and establishing an early start to the mining of the upper mining block. The backfill was designed to have an adequate strength to allow for mining adjacent to filled stopes, thus eliminating the need for rib pillars.

There will be two shafts, which will minimize the amount of development through water-bearing horizons located in the first 200 m from the surface. Both shafts are excavated at the same time using conventional shaft sinking methods in conjunction with a freezing process through the first 200 m from the surface.

The production shaft will facilitate main access and mechanical egress, material hoisting, fresh air intake, and material logistics. The production shaft is excavated to a lower elevation than in the previous 2017 SRK feasibility studies. This allows earlier access to higher grade ore in the central portion of the mine and to also access higher grade ore in the lower mining block with a more efficient material handling system.

The ventilation shaft will serve as the mine exhaust system, initial hoisting system of lateral development, as well as a second means of mechanical egress. In addition, the sinking of the ventilation shaft, which is not as deep as the production shaft, allows for an earlier start to key lateral development via the ventilation shaft.

Mined ore is to be transported from the stopes to the main production shaft hoisting system by underground LHDs, trucks, ore passes, crusher and conveyor circuit.

Access and infrastructure development for the underground was designed to support the mining method and sized based on mining equipment and production rate requirements. Surface infrastructure and tailings were designed to match the underground production rate requirements.

15.2 Conversion Assumptions, Parameters and Methods

Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described within this sub-section, to potential mining block shapes created during the mine design process. No Measured Resources are estimated and, as a result, no Proven Reserves are stated.

The undiluted tonnes and grade of each potential mining block are based on the resource block model estimated by Nordmin as described in Section 14 of this report. All Mineral Reserve tonnages are expressed as "dry" tonnes (i.e., no moisture) and are based on the density values stored in the block model.

15.2.1 Dilution

Mining dilution of approximately 6% was applied to all stopes and development, based on 3% for the primary stopes, 9% for the secondary stopes, and 5% for ore development. The mining dilution was added to the designed tonnage to account for unplanned sources of dilution such as backfill and host rock around the periphery of the ore mass. Mining dilution of host rock from around the periphery of the ore mass has been applied with zero grade as a conservative assumption even though some sources of this type of dilution carry grade. The primary stopes will have ore, host rock and backfill as material that will slough into them while being extracted. The ore portion of the sloughed material is not included in the 3% dilution factor, as this ore is accounted for in the adjacent stopes. Secondary stopes have more sources of material with no grade and less ore from adjacent stopes; therefore, a higher dilution factor of 9% has been applied to them.

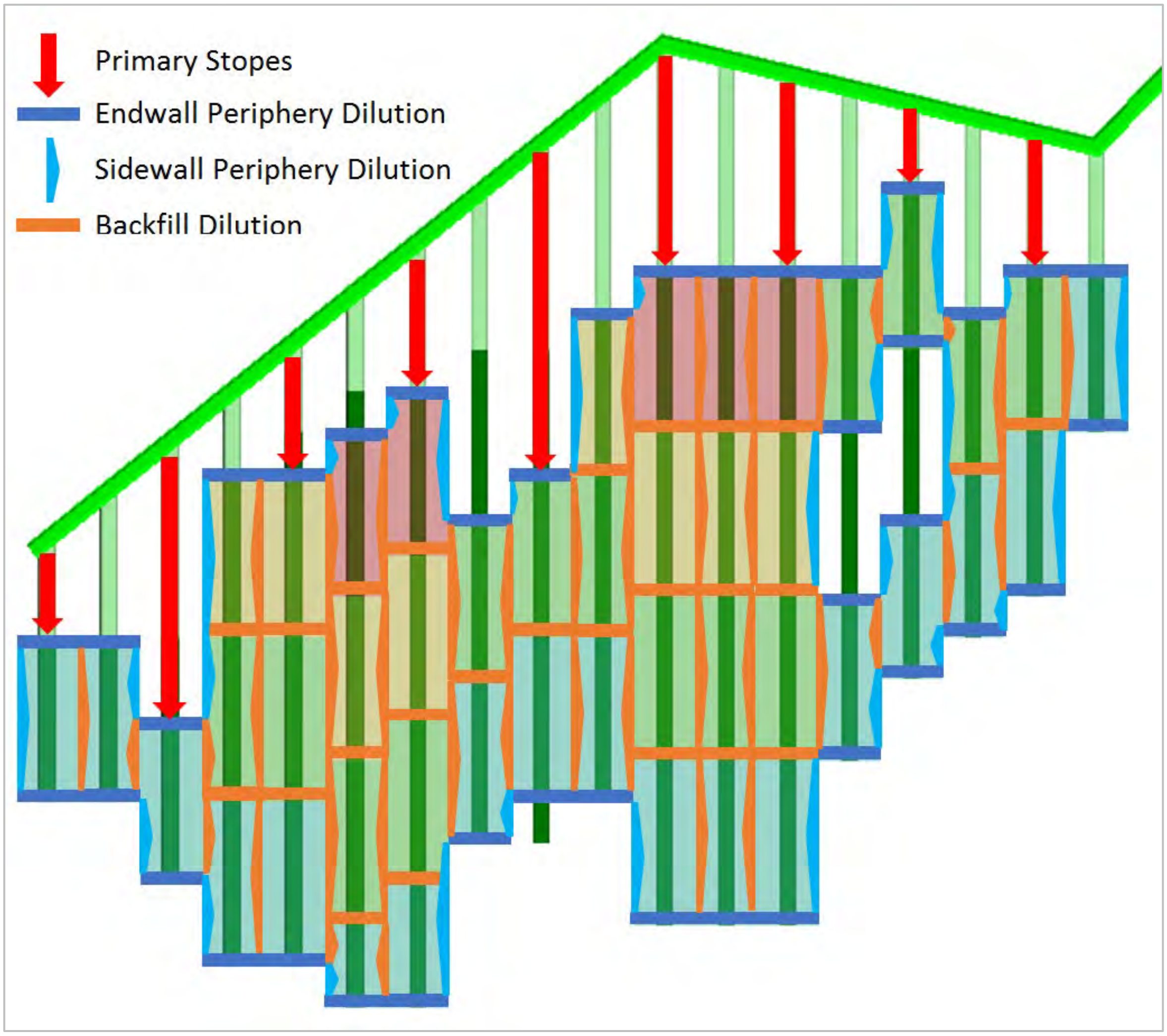
As stated in Section 16.2, the thickness of external dilution is estimated as equivalent linear overbreak/slough (ELOS), for moderately weathered carbonatite, and for fresh to slightly weathered carbonatite. Sidewall and back dilution are not expected to be a problem because in the primary stopes dilution (from adjacent secondary stopes) will be at grade, and dilution from secondary stopes is managed by controlling backfill strength.

As shown in Figure 15-1, sources of mining dilution for primary stopes include:

- Backfill material on the floor/sill with no grade.
- Backfill material from the hangingwall end with no grade if the stope is adjacent to a previously mined stope.
- Low grade periphery rock dilution in the hangingwall or footwall if the stope is not adjacent to other stopes.

As shown in Figure 15-1, sources of mining dilution for secondary stopes include:

- Backfill material on the floor/sill with no grade.
- Backfill material from the hangingwall end with no grade if the stope is adjacent to a previously mined stope.
- Low grade periphery rock dilution in the hangingwall or footwall if the stope is not adjacent to other stopes.
- For most situations, backfill material on both sidewalls with no grade.



Source: Nordmin, 2019

Figure 15-1: Sources of Mining Dilution for Typical Stope Layout (Not to Scale)

Table 15-1 shows the sources of mining dilution by stope type (primary and secondary) for typical stope geometry.

Table 15-1: Sources of Mining Dilution for Typical Geometry by Stope Type

Footwall	P2	P1	P0	Hanging Wall
	S2	S1	S0	
	P2	P1	P0	
	S2	S1	S0	
	P2	P1	P0	
	S2	S1	S0	

	Primary Stopes			Secondary Stopes		
	P0	**P1**	**P2**	**S0**	**S1**	**S2**
Hanging Wall Dilution - Rock	Yes	No	No	Yes	No	No
Footwall Dilution - Rock	No	No	Yes	No	No	Yes
Hanging Wall Dilution - Backfill	No	Yes	Yes	No	Yes	Yes
Footwall Dilution - Backfill	No	No	No	No	No	No
Sidewalls - Rock (Ore)	No	No	No	No	No	No
Sidewalls - Backfill	No	No	No	Yes	Yes	Yes
Floor/Sill Dilution - Backfill	Yes	Yes	Yes	Yes	Yes	Yes

Source: Nordmin, 2019

15.2.2 Recovery

A stope recovery factor of 95% was used. The following items were used to calculate this factor:

- Material loss into backfill (floor) or 0.4 m.
- Material loss to side and end walls (under blast) of 0.2 m.
- Material loss to mucking along sides and in blind corners.
- Additional loss factor due to rockfalls, unanticipated regional stress loads, and other geotechnical reasons.

A development recovery factor of 95% was used for all horizontal development.

15.2.3 Cut-Off Grade Calculation

Net Smelter Return (NSR) is a commonly accepted method of evaluating a mineral deposit where revenue is generated from multiple elements. NSR is defined as the proceeds from the sale of mineral products after deducting off-site processing and distribution costs. NSR is typically expressed on a dollar per tonne basis. For this Project, the NSR calculation takes into account revenue for three products, FeNb, TiO_2, and Sc_2O_3. A factor of 0.696 was used to convert Nb_2O_5 in the block model to Nb contained in the FeNb product. Similarly, a factor of 1.534 (1/0.652) was used to convert Sc to Sc_2O_3.

Recoveries used are based on metallurgical test work discussed in Section 13. The NSR was evaluated for each block in the 3D geologic resource block model. Table 15-2 shows NSR parameters and an example NSR calculation for an individual block.

Table 15-2: Example of an NSR Block Calculation

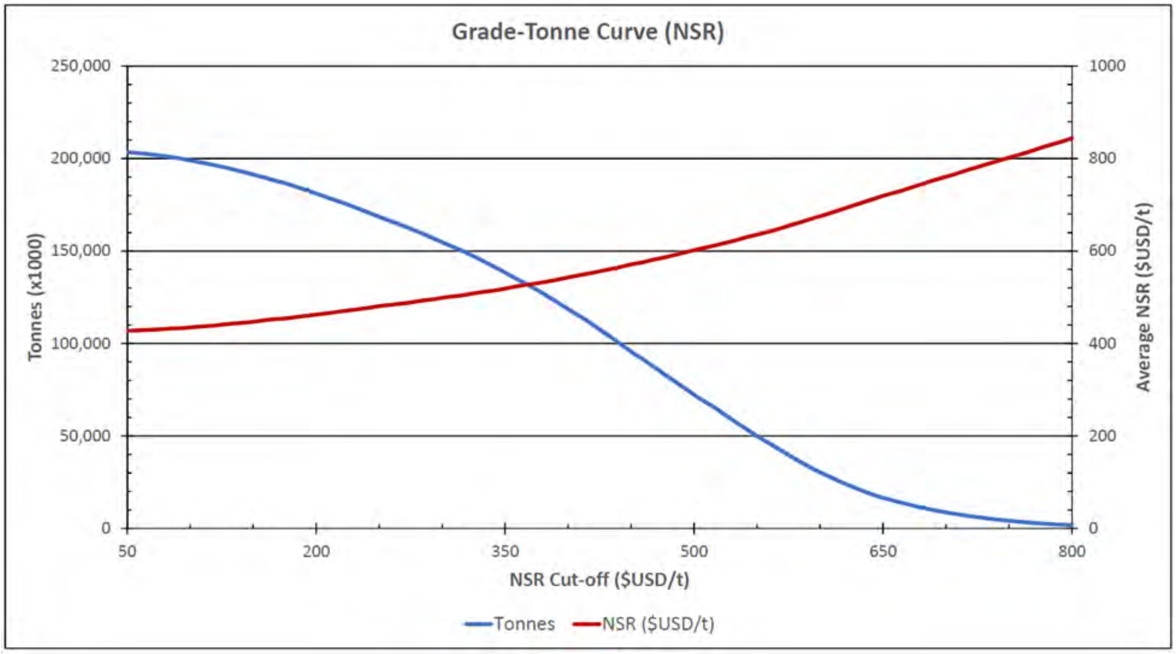
Input Parameters	Total	Nb_2O_5	TiO_2	Sc (1)
Example Block Model Mass	100 t			
Example Block Model Grades		0.70%	2.50%	60 ppm
Metallurgical Recoveries (2)		82.36%	40.31%	93.14%
Amount Payable		100.0%	100%	100.0%
Conversions from input grade to product		69.6%	100.0%	153.4%
Refining Charges		0	0	0
Price		US$ 39.60/kg	US$ 0.88/kg	US$ 3,675/kg
Calculate Contained Metal				
Nb_2O_5		700 kg		
TiO_2			2,500 kg	
Sc				6 kg
Calculate Saleable Metal (conversion to product, discounted by recovery)				
Nb		401 kg		
FeNb		617 kg		
TiO_2			1,008 kg	
Sc (as Sc_2O_3)				8.57 kg
Calculate Block Dollar Value for Each Metal				
FeNb		US$ 15,890		
TiO_2			US$ 887	
Sc				US$ 31,504
Total Block Value	US$ 48,281			
Block Value per tonne	US$ 482.81/t			

Source: Nordmin, 2019

(1) Stored as PPM in the block model. Sc % = Sc ppm/10,000.

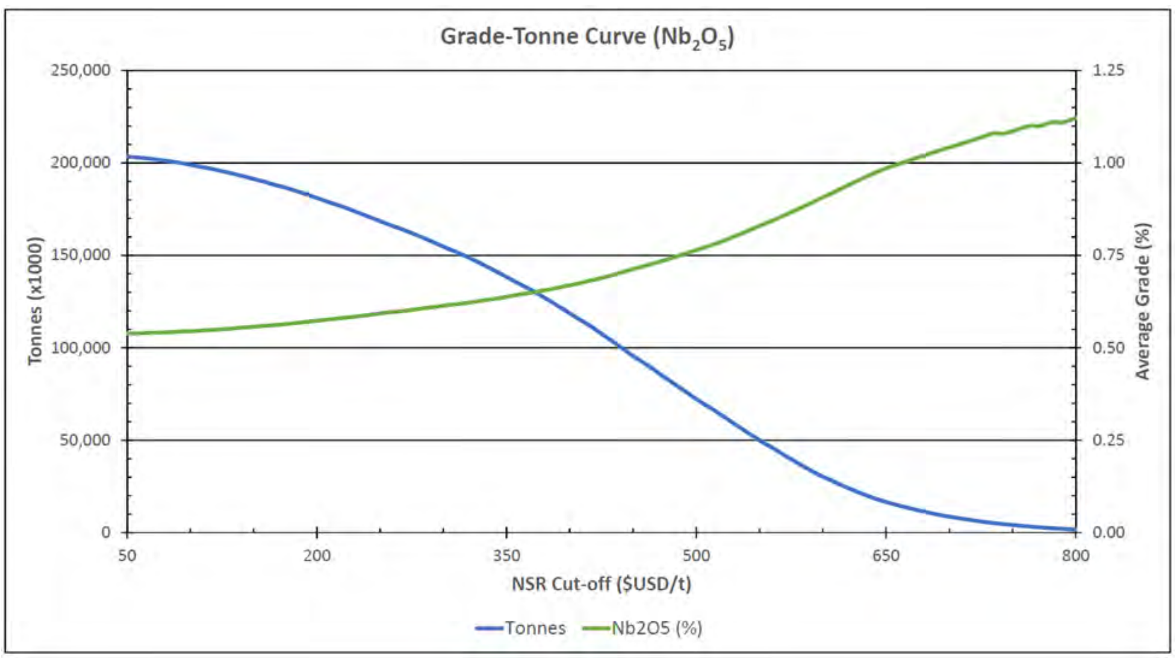
(2) Overall metallurgical recovery, including all losses

Figure 15-2 through Figure 15-5 provide a grade-tonne curve for the deposit using various NSR cut-off grades, (CoG). It includes only Measured and Indicated material and shows average grades for each grade variable. All Inferred material is treated as having a zero grade value in this Reserve.



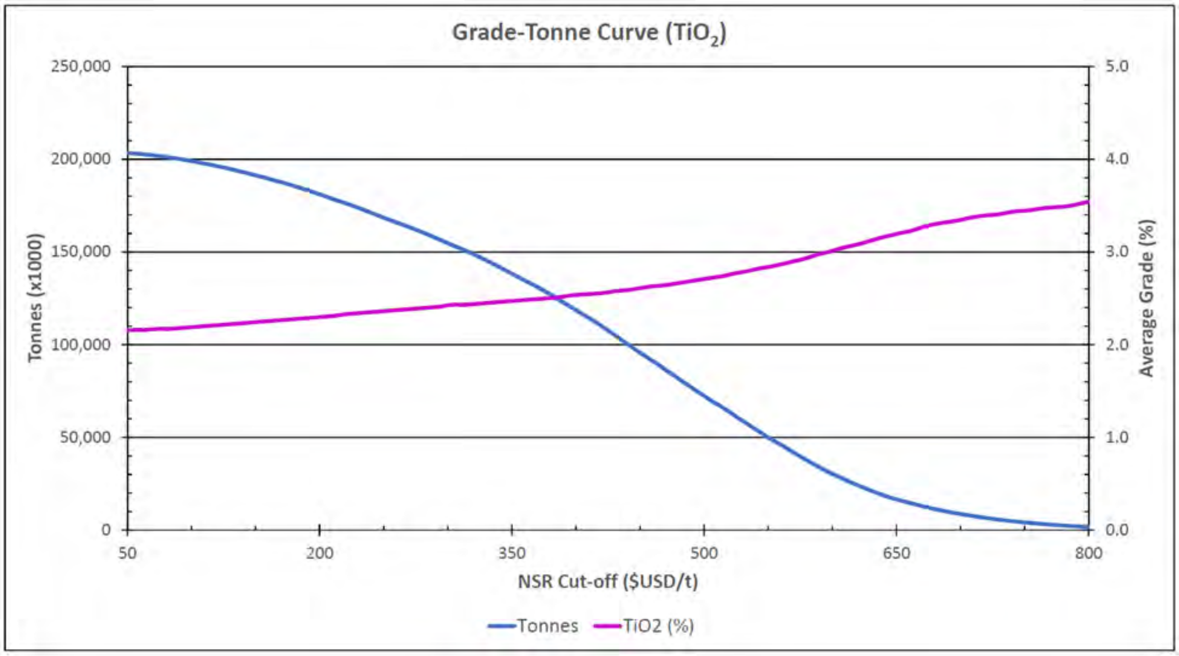
Source: Nordmin, 2019

Figure 15-2: NioCorp Grade/Tonne Curves Based on NSR Cut-Off (NSR)



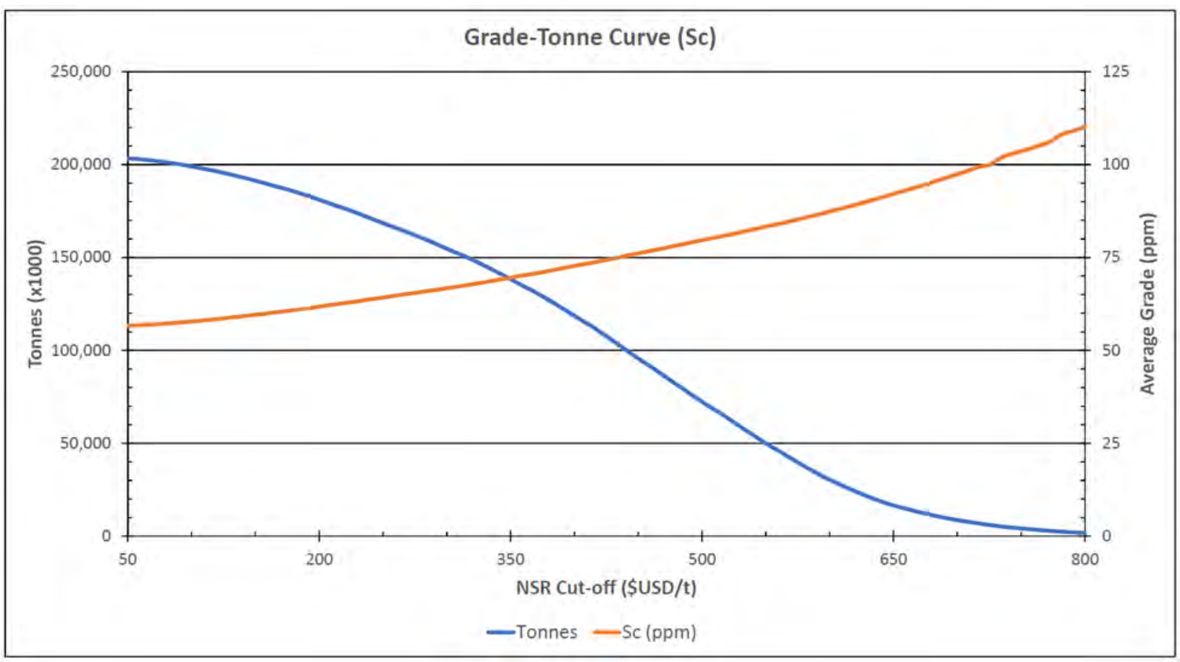
Source: Nordmin, 2019

Figure 15-3: NioCorp Grade/Tonne Curves Based on NSR Cut-Off (Nb_2O_5)



Source: Nordmin, 2019

Figure 15-4: NioCorp Grade/Tonne Curves Based on NSR Cut-Off (TiO_2)



Source: Nordmin, 2019

Figure 15-5: NioCorp Grade/Tonne Curves Based on NSR Cut-Off (Sc)

To establish the initial boundary of the mine design and to assure inclusion of all potential Mineral Reserves, a minimum CoG of US$ 180/t was used based on the estimated costs shown in Table 15-3.

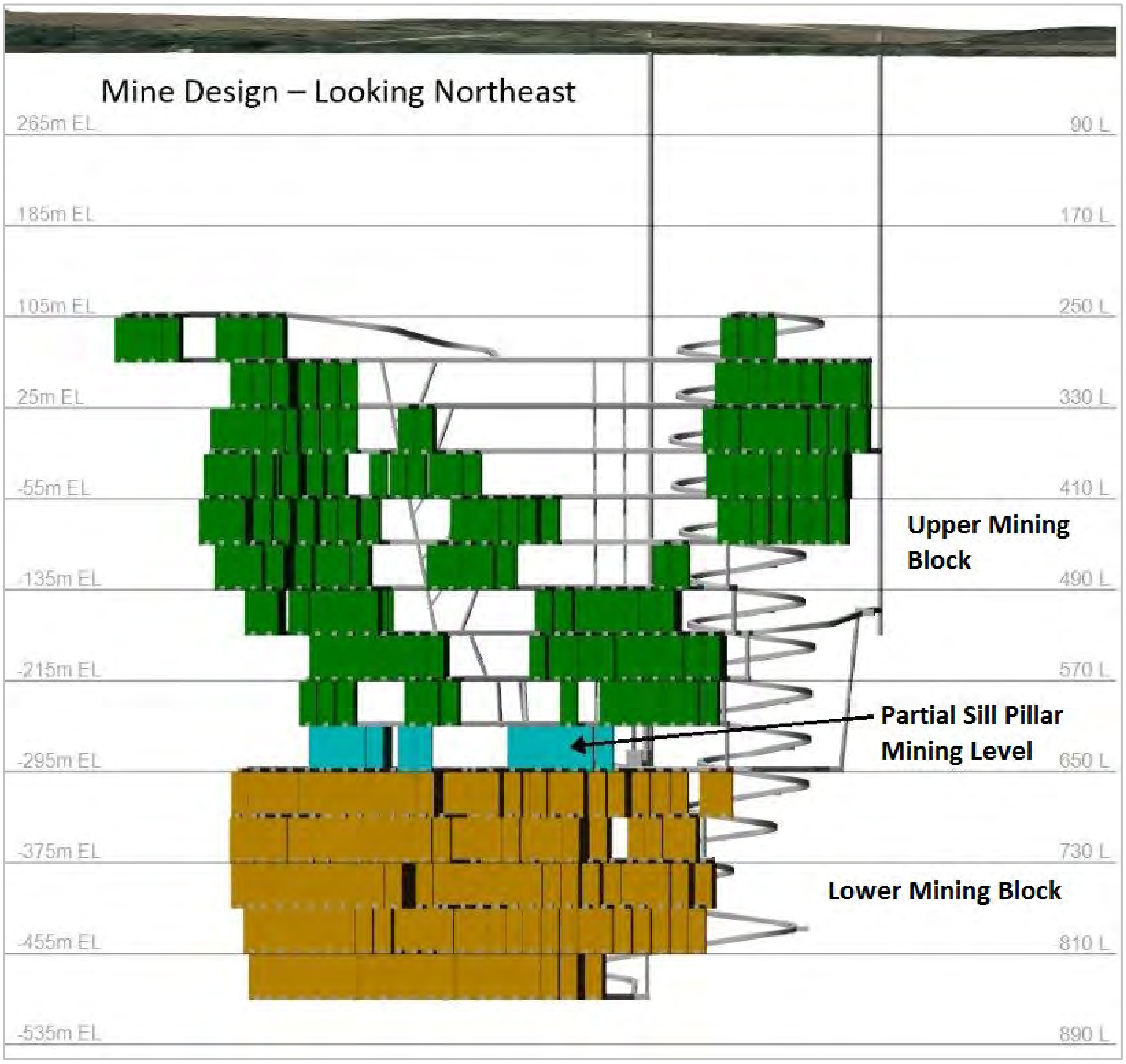
Table 15-3: Operating Costs Used for Mine Design NSR Cut-off

Item	Estimated Costs (US$/t)
Mining (1)	50.00
Processing	125.00
G&A	5.00
Total (2)	**US$ 180.00**

Source: Nordmin, 2019
(1) Includes backfill
(2) Values used here differ from the economic model generated from the final overall site design. Nordmin is satisfied that the values used were applicable to establishing the correct and optimum mining design.

15.2.4 Mine Design

Potential mining areas were identified using stope optimization within Datamine Minable Shape Optimizer software. The stope optimizer output was reviewed on a level by level basis, and a 3D mine design was generated. The estimated cut-off NSR value (CoNSR) of US$ 180/t from the SRK study was used as a starting point for this analysis. Generally, stopes would be selected based on the minimum CoG or CoNSR. As the CoNSR value is much lower than the resulting average stope NSR value, the Co NSR was not the decisive factor in the stope optimization process. Rather than using a minimum CoG or CoNSR, the mine design targeted higher annual ferroniobium production during the first five years of ore delivery, which resulted in an averaged annual production rate of 7,351 tonnes per year over this period. The steady-state average annual ferroniobium production was 7,220 tonnes annually. This strategy results in a LOM NSR average value of US$ 538.63/t. Note that two mining blocks are included in the mining design. There is additional mineralization potential below the lower mining block. The identified mining blocks provide an approximate 36-year LOM; therefore, the additional design below the lower mining block was not completed at this time. The design includes stopes, development accesses, and necessary infrastructure. Figure 15-6 shows the completed mine design.



Source: Nordmin, 2019

Figure 15-6: Completed Mine Design

15.3 Reserves

The 2019 Mineral Reserves were classified using the 2014 CIM Definition Standards. Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described earlier in this section, to potential mining block shapes created during the mine design process.

The underground mine design process resulted in a mine plan with a Mineral Reserve Estimate of 36.3 Mt (diluted) with an average grade of 0.81% Nb_2O_5, 2.86% TiO_2, and 65.7 ppm Sc. This estimate is based on a mine design using elevated CoGs and applying the US$ 180/t NSR CoG to capture all potential Mineral Reserves within the design. These numbers include a 95% mining ore recovery to the designed wireframes in addition to applying approximately 6% unplanned dilution as described in Section 15.2.1.

Table 15-4 summarizes the underground reserves as of February 19, 2019.

Table 15-4: Underground Mineral Reserves Estimate for Elk Creek, Effective Date February 19, 2019

Classification	Tonnage (x1000 t)	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (t)	Payable Nb (t)	TiO_2 Grade (%)	Contained TiO_2 (t)	Payable TiO_2 (t)	Sc Grade (ppm)	Contained Sc (t)	Payable Sc_2O_3 (t)
Proven	-	-	-	-	-	-	-	-	-	-
Probable	36,313	0.81	293,321	168,861	2.86	1,039,050	418,841	65.7	2,387	3,410
Total	36,313	0.81	293,321	168,861	2.86	1,039,050	418,841	65.7	2,387	3,410

Source: Nordmin, 2019. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.

- The Mineral Reserve is based on the mine design, mine plan, and cash-flow model utilizing an average cut-off grade of 0.788% Nb_2O_5 with an NSR of US$ 500/mt.
- Nordmin considers that the Mineral Reserve is amenable for underground extraction with a processing method to recover FeNb (as the saleable product of Nb_2O_5), TiO_2, and Sc_2O_3 products.
- The economic assumptions used to define Mineral Reserve cut-off grade are as follows:
    - Annual life of mine (LOM) production rate of ~7,220 tonnes of FeNb/annum,
    - Initial elevated five-year production rate ~ 7,351 tonnes of FeNb/annum
    - Mining dilution of ~6% was applied to all stopes and development, based on 3% for the primary stopes, 9% for the secondary stopes, and 5% for ore development.
    - Mining recoveries of 95% were applied.
    - Price assumptions for FeNb, Sc_2O_3, and TiO_2 are based upon independent market analyses for each product.
    - Price and cost assumptions are based on the pricing of products at the "mine-gate," with no additional down-stream costs required. The assumed products are a ferroniobium product (metallic alloy shots consisting of 65%Nb and 35% Fe), a titanium dioxide product in powder form, and scandium trioxide in powder form.

Parameter	Value	Unit
Mining Cost	43.55	US$/t mined
Processing	108.16	US$/t mined
Water Management and Infrastructure	13.71	US$/t mined
Tailings Management	1.35	US$/t mined
Other Infrastructure	6.96	US$/t mined
General and Administrative	8.65	US$/t mined
Royalties/Annual Bond Premium	7.53	US$/t mined
Total Cost	189.91	US$/t mined
Nb_2O_5 to Niobium conversion	69.60	%
Niobium Process Recovery	82.36	%
Niobium Price	39.60	US$/kg
TiO_2 Process Recovery	40.31	%
TiO_2 Price	0.88	US$/kg
Sc Process Recovery	93.14	%
Sc to Sc_2O_3 conversion	153.40	%
Sc Price	3,675.00	US$/kg

- The Mineral Reserve has an average LOM NSR of US$ 538.63 /tonne.
- Nordmin has provided detailed estimates of the expected costs based on the knowledge of the style of mining (underground) and potential processing methods (by 3rd party Qualified Persons).
- Mineral reserve effective date February 19, 2019. The financial model was run post-February 2019, which reflects a total cost per tonne of US$ 196.41 versus US$ 189.91 (February 19, 2019, Mineral Reserve Details Table above). Nordmin does not consider this a material change.
- Price variances for commodities are based on updated independent market studies versus earlier projected pricing. The updated independent market studies do not have a negative effect on the reserve.
- Nordmin completed a site inspection of the deposit by Jean- Francois St-Onge, P.Eng., Associate Consulting Specialist – Mining, an appropriate "independent qualified person" as this term is defined in NI 43-101.

15.4 Relevant Factors

Nordmin knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors which could materially affect the underground Mineral Reserve Estimate.

16. MINING METHODS

16.1 Geology Overview

The mine planning work is based on the resource geology and block model, described in Section 14 of this Technical Report. In addition to the mineralization, various other elements were estimated into the model for metallurgical purposes.

16.2 Geotechnical Design Parameters

From May 21, 2014, to July 30, 2015, SRK completed a geotechnical investigation program on site for the Project. The program was designed to characterize subsurface geotechnical conditions to assist in the development of a feasibility study design capable of meeting the requirements for basic engineering design.

The geotechnical database used for analyses includes data from two geotechnical borehole databases: the 2014 program and a previous 2011 program. A geotechnical model was created using the characterization information in the databases to estimate rock mass quality, rock strength, and major discontinuities in the carbonatite hanging wall and footwall and the Pennsylvanian Formation. The 3D geotechnical model was built using the Vulcan software, wherein representative volumes of rock mass quality domains were created.

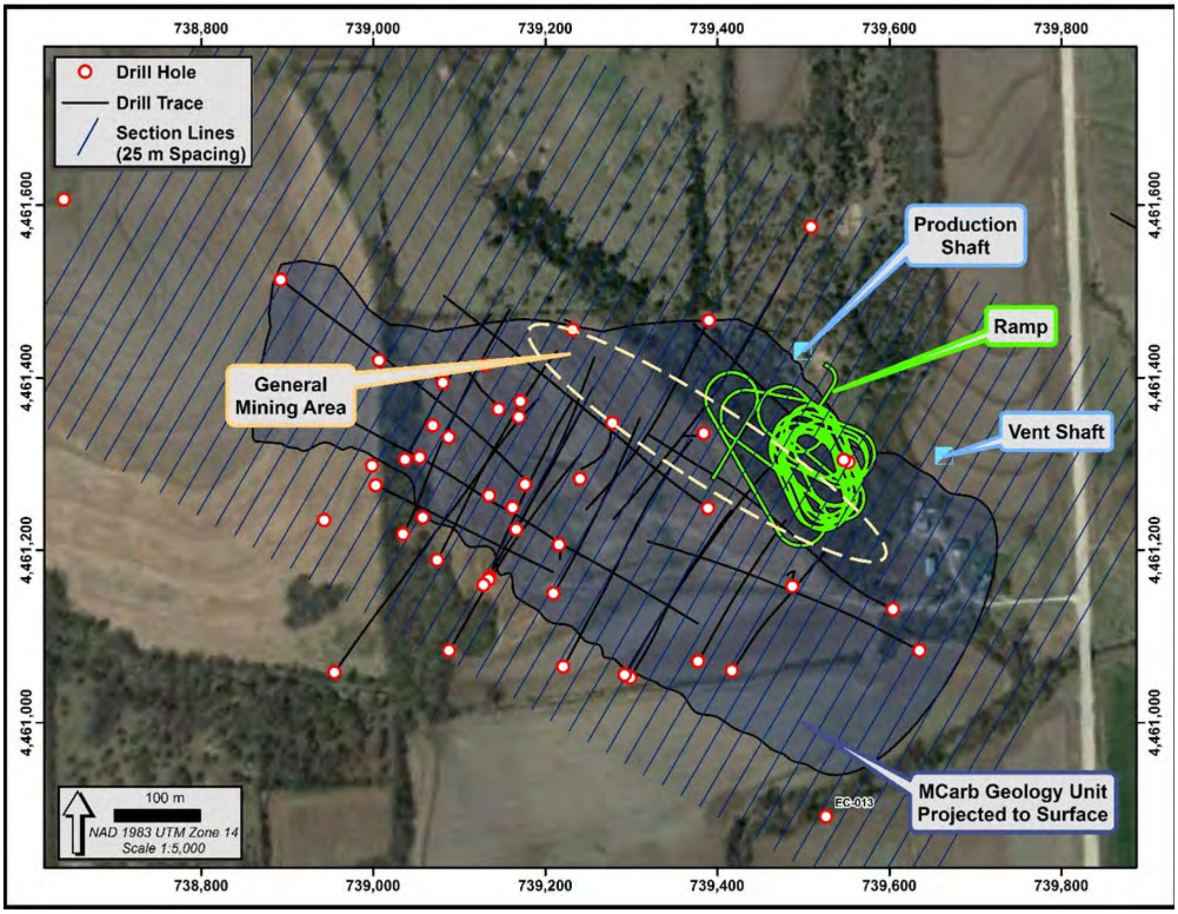
The following is a summary of the geotechnical parameters used to assess the mine design.

Data Collection

The geotechnical field investigation consisted of 23 drill holes, totalling 20,379 m. The program was designed to examine rock mass fabric and structural features in and around the mineralized zone at different depths and orientations. The drilling was conducted in three phases with incremental data collection designed to fill knowledge gaps on geotechnical conditions. Drill holes were drilled at varying orientations into the hanging wall, footwall, and mineralized rock to capture data on rock mass discontinuity variations.

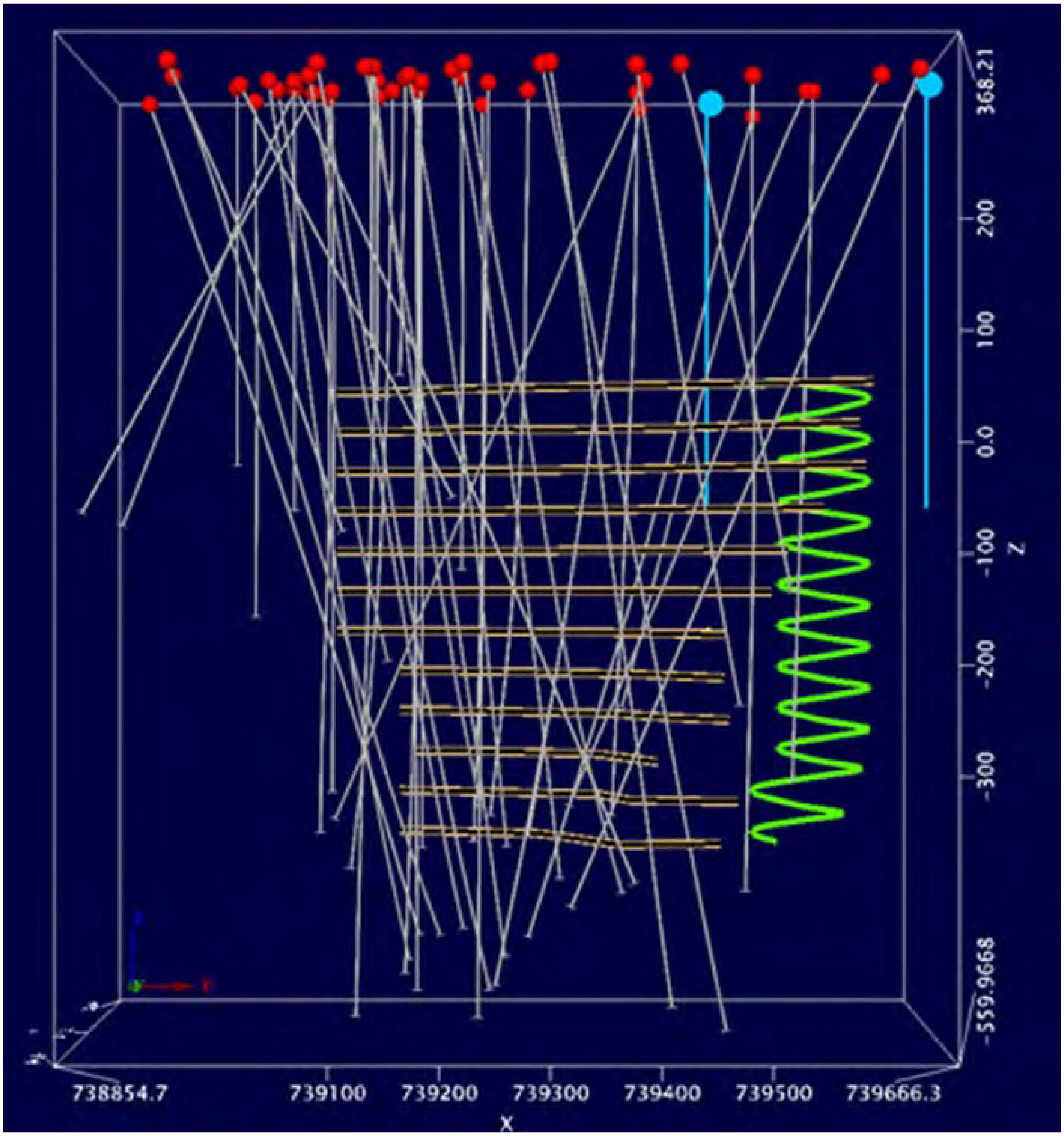
The drill holes are shown in Figure 16-1 in plan view. Figure 16-2 shows a vertical view looking towards the north with the mining levels and the planned access ramp. Figure 16-3 shows the same vertical view looking towards the north with wireframe shapes of the high grade niobium (Nb_2O_5 > 1%). The drill hole location data are summarized in Table 16-1. The field investigation included drilling of the core, geophysical borehole logging of structural features, geotechnical core logging, core sample collection for laboratory strength testing, and in situ stress measurements.

Structural features (discontinuities) encountered during this field investigation consisted of joints, lithological contacts, veins, dikes, foliation, faults, shear zones and fractures. Orientation data was collected using acoustic televiewer (ATV) down-hole equipment. Typical alteration on discontinuity surfaces was recorded in the logs and included mineralized coatings or infillings with occasional hematite staining. Faulting was interpreted using evidence of slickensided surfaces and the presence of gouge infilling and orientations verified using the ATV orientation data.



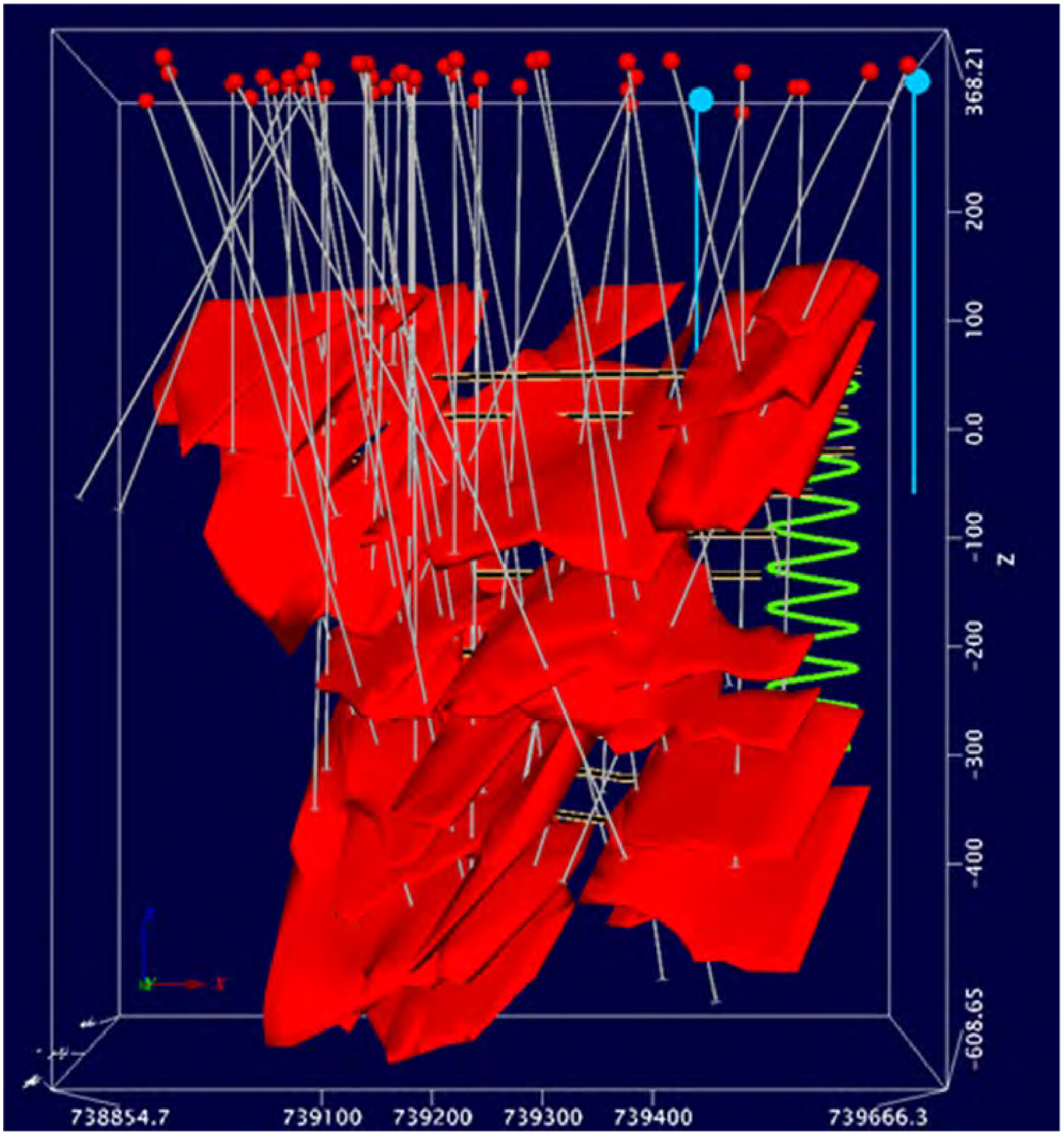
Source: Nordmin, 2019

Figure 16-1: Plan View Location of 2014-2015 Geotechnical Drill Holes



Source: Nordmin, 2019

Figure 16-2: Vertical View of the Location of 2014-2015 Geotechnical Drill Holes Looking Towards the North



Source: Nordmin, 2019

Figure 16-3: Vertical View of the Location of 2014-2015 Geotechnical Drill Holes Looking Towards the North with High Grade Niobium Wireframe (>1% Nb_2O_5)

Table 16-1: Drill Hole Orientation and Data Collection Methods

<table>
<tr><th>Year</th><th>Hole ID</th><th>Easting (m)</th><th>Northing (m)</th><th>Elev. (m)</th><th>Az (°)</th><th>Dip (°)</th><th>Length</th><th colspan="2">Geotechnical Data</th></tr>
<tr><td rowspan="3">2011</td><td>NEC11-001</td><td>739297.0</td><td>4461224.0</td><td>343.4</td><td>28.1</td><td>-72</td><td>900.4</td><td rowspan="3"></td><td rowspan="26">Rock Mass Charact-erization</td></tr>
<tr><td>NEC11-002</td><td>738950.0</td><td>4461083.5</td><td>343.4</td><td>33.5</td><td>-61</td><td>479.7</td></tr>
<tr><td>NEC11-003</td><td>739417.0</td><td>4461059.6</td><td>340.8</td><td>33.5</td><td>-61</td><td>508.7</td></tr>
<tr><td rowspan="19">2014</td><td>NEC14-006</td><td>739166.2</td><td>4461224.0</td><td>352.0</td><td>29.9</td><td>-71</td><td>772.7</td><td rowspan="19">Structure Orientation Data (ATV televiewer)</td></tr>
<tr><td>NEC14-007</td><td>739088.2</td><td>4461083.5</td><td>344.8</td><td>29.4</td><td>-71</td><td>907.4</td></tr>
<tr><td>NEC14-008</td><td>739128.1</td><td>4461159.4</td><td>351.2</td><td>30.8</td><td>-70</td><td>886.1</td></tr>
<tr><td>NEC14-009</td><td>739390.2</td><td>4461466.2</td><td>349.3</td><td>208.7</td><td>-70</td><td>751.3</td></tr>
<tr><td>NEC14-009a</td><td>739390.2</td><td>4461466.2</td><td>349.3</td><td>208.7</td><td>-70</td><td>897.0</td></tr>
<tr><td>NEC14-010</td><td>739209.5</td><td>4461149.8</td><td>347.8</td><td>30.0</td><td>-73</td><td>796.1</td></tr>
<tr><td>NEC14-011</td><td>738892.5</td><td>4461513.6</td><td>359.7</td><td>125.8</td><td>-65</td><td>900.4</td></tr>
<tr><td>NEC14-012</td><td>739635.1</td><td>4461083.4</td><td>339.9</td><td>299.8</td><td>-68</td><td>843.2</td></tr>
<tr><td>NEC14-013</td><td>739169.3</td><td>4461354.3</td><td>355.2</td><td>-</td><td>-90</td><td>880.3</td></tr>
<tr><td>NEC14-014</td><td>739034.8</td><td>4461218.6</td><td>346.3</td><td>28.6</td><td>-78</td><td>901.0</td></tr>
<tr><td>NEC14-015</td><td>739221.0</td><td>4461064.7</td><td>342.4</td><td>29.1</td><td>-72</td><td>827.8</td></tr>
<tr><td>NEC14-016</td><td>739509.1</td><td>4461574.7</td><td>354.7</td><td>210.5</td><td>-60</td><td>913.8</td></tr>
<tr><td>NEC14-020</td><td>739037.1</td><td>4461305.0</td><td>348.4</td><td>28.2</td><td>-71</td><td>587.7</td></tr>
<tr><td>NEC14-021</td><td>739074.3</td><td>4461188.0</td><td>347.1</td><td>29.51</td><td>-69</td><td>865.0</td></tr>
<tr><td>NEC14-022</td><td>739292.2</td><td>4461055.0</td><td>340.3</td><td>31.3</td><td>-68</td><td>950.4</td></tr>
<tr><td>NEC14-023</td><td>739377.6</td><td>4461071.0</td><td>341.5</td><td>30.2</td><td>-71</td><td>615.1</td></tr>
<tr><td>NEC14-MET-01</td><td>739240.4</td><td>4461282.7</td><td>352.8</td><td>-</td><td>-90</td><td>894.7</td></tr>
<tr><td>NEC14-MET-02</td><td>739171.1</td><td>4461372.4</td><td>355.8</td><td>-</td><td>-90</td><td>865.0</td></tr>
<tr><td>NEC14-MET-03</td><td>739129.9</td><td>4461414.5</td><td>355.4</td><td>-</td><td>-90</td><td>913.3</td></tr>
<tr><td rowspan="4">2015</td><td>NEC15-002</td><td>739046.5</td><td>4460708.9</td><td>344.4</td><td>303.8</td><td>-88</td><td>850.4</td><td rowspan="2"></td></tr>
<tr><td>NEC15-003</td><td>740346.2</td><td>4460854.1</td><td>341.6</td><td>306.5</td><td>-90</td><td>850.5</td></tr>
<tr><td>NEC15-004</td><td>739472.2</td><td>4461507.0</td><td>354.6</td><td>-</td><td>-90</td><td>413.6</td><td rowspan="2">ATV</td></tr>
<tr><td>NEC15-005</td><td>739514.8</td><td>4461418.5</td><td>351.2</td><td>-</td><td>-90</td><td>407.5</td></tr>
</table>

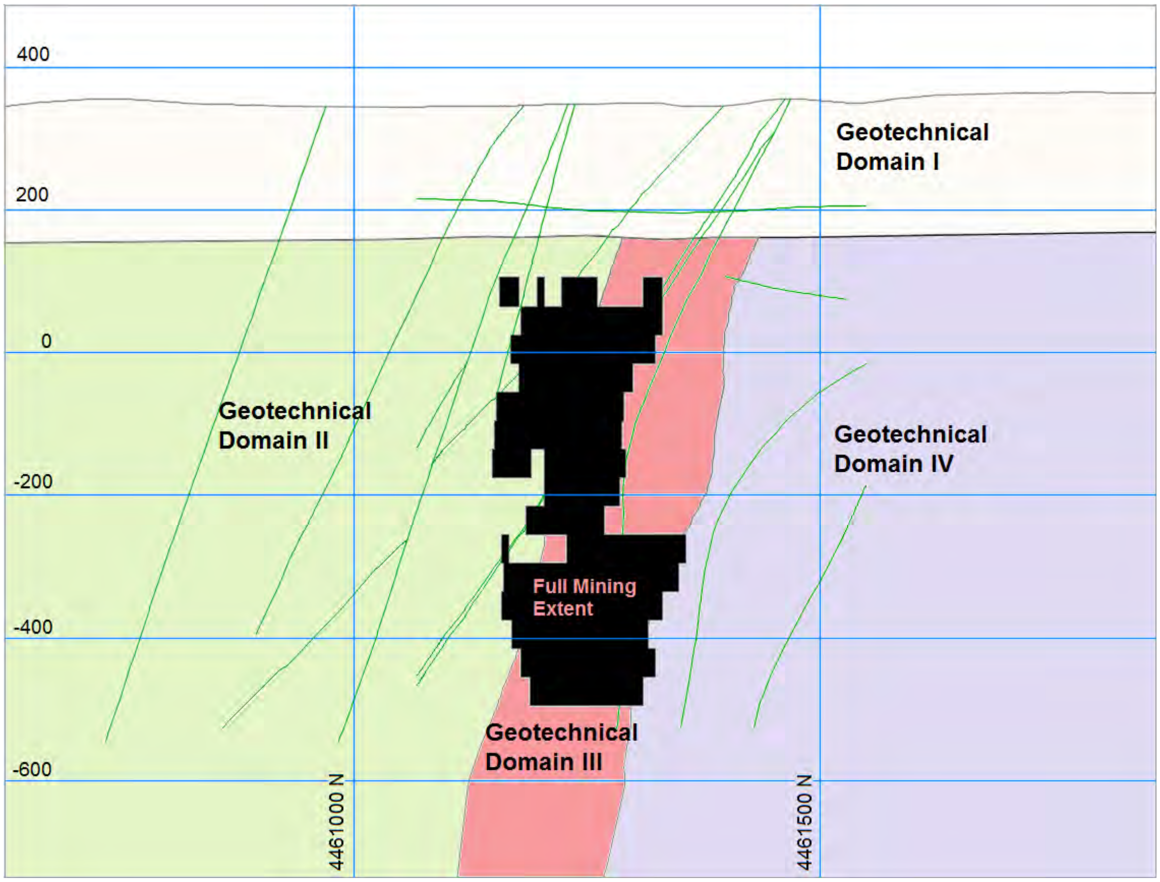
Source: SRK, 2017

Geotechnical Domains

Four spatial geotechnical domains were identified based on lithology, weathering, structural conditions and rock mass strength similarities. These geotechnical domains include:

- Pennsylvania Formation in the upper 200 m.
- Hanging wall material to the southwest of the orebody.
- Mineralized carbonatite orebody.
- Footwall material to the northeast of the orebody.

These domains, shown in Figure 16-4, were delimited based on intact rock properties and in situ rock mass quality from characterization logging. Characterization was based on Rock Mass Rating (RMR76) (Bieniawski, 1976) and the Q-system (Barton et al., 1974). These values were then used with empirical design methods to assess the basic inputs for underground mine design.



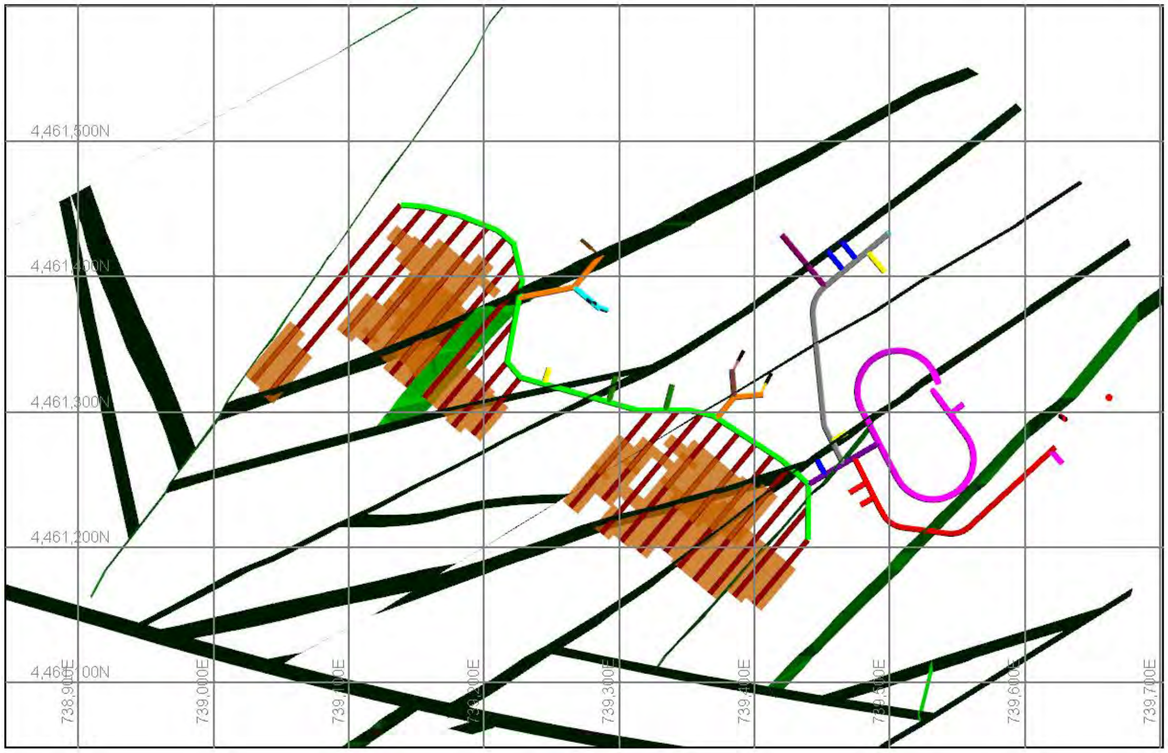
Source: Nordmin, 2019

Figure 16-4: Geotechnical Model, Vertical Cross Section (N40°E Section)

Structural Faulting

Data on the regional structural geology and the borehole ATV logging data was used to identify 21 major structures in the local mine-scale geology.

Figure 16-5 shows a plan view section through the Vulcan™ model with the position of the stopes and footwall accesses relative to the geologic structures on the +60 m elevation level in Block 1 mining stopes.



Source: Nordmin, 2019

Figure 16-5: Plan View of Geologic Structures (Green) on +60 m Elevation Level in Block 1

Rock Mass Quality

The laboratory testing program included 71 Unconfined Compressive Strength (UCS) tests of intact samples, 17 Triaxial Compressive Strength (TCS) tests of intact samples, and 38 Direct Shear Strength (DSS) tests of jointed rock samples. Triaxial tests were conducted at different confinement levels (80 tests) to account for anticipated variations in stress conditions. A set of 29 static and dynamic elastic constant measurements were collected to characterize the elastic properties of the rock. A total of 12 Brazilian Tensile Strength (BTS) tests were conducted to characterize the tensile strength of the rock mass. The UCS results were used to calibrate the spatial variation of UCS values interpreted from 2,180 Point Load Tests (PLT) conducted in the field during the core logging program. The laboratory tests were sufficient to develop shear strength parameters (intact and discontinuity) and provide estimates of the statics and dynamics elastic constants.

The majority of the Rock Quality Designation (RQD) values indicate fair to good rock quality (RQD = 80 to 100) throughout the drill holes. Regions with lower RQD (RQD = 10 to 60) were generally associated with weathered or altered rock zones and/or minor geological intrusions. Fracture frequencies ranged from 0.1 to 9 fractures per metre, with an average of 3 fractures per metre.

The core was generally fresh to slightly weathered with weathering limited to the surfaces of the discontinuities of slight rock mass alteration. Field index strength tests indicate that the core was strong on average (R4). The core tended to break along pre-existing planes of weakness such as veins, foliation and healed structural features.

Laboratory UCS tests results indicate that the carbonatite and lamprophyre strength ranges from strong to very strong (UCS = 50 to 250 MPa), whereas the mafic dikes and mudstone and limestone of the Pennsylvania Formation are moderately strong to strong (UCS = 25 to 100 MPa). Table 16-2 shows a summary of the rock mass properties by domain.

Table 16-2: Summary of Rock Mass Characterization by Domain

Geotechnical Domain	Weathering (%)	Density (t/m^3)	IRS (MPa)	RQD (%)	Fracture Frequency (FF/m)	RMR76/ GSI	Q'
I Pennsylvania (13%)	Limestone 48%	2.57	21 – 75	95 - 100	0 – 0.3	73 – 94	17 – 88
	Mudstone 43%		14 – 50	96 – 100	0 – 0.7	68 – 92	14 – 88
II Hanging Wall (22%)	Fresh 49%	2.84	43 – 90	96 – 100	0.5 – 2	59 – 77	6 – 25.1
	Moderated Weathered 41%		28 – 59	81 – 100	1 – 4	51 – 67	4 – 12
	Highly Weathered 10%		8 – 44	17 – 90	2 – 28	33 – 55	0.3 – 5
III Mineralized Carbonatite (50%)	Fresh 72%	3.02	33 – 100	94 – 100	0.3 – 3	57 – 79	6 – 27
	Moderated Weathered 22%		26 – 63	80 – 100	1 – 4	51 – 69	4 – 14
	Highly Weathered 6%		16 – 50	64 – 85	2 – 12	40 -58	1 – 9
IV Footwall (15%)	Fresh 69%	2.84	45 – 113	91 - 100	0.3 – 2	61 - 82	6 – 28
	Moderately Weathered 24%		27 – 59	70 - 90	1 – 5	50 – 70	3 – 14
	Highly Weathered 7%		4 – 28	15 – 85	4 – 26	32 – 48	0.4 – 6

Source: SRK, 2017

The ATV data was used to establish the structural domains and for input to the structural model. The oriented core investigation included a total of 16,790 m of ATV scans of which 9,494 m were sufficient for discontinuity interpretation (i.e. 57% of televiewer success). Structural sets were identified for each domain based on orientation clusters and discontinuity type. Table 16-3 is a summary of the ATV data obtained in each drill hole.

Table 16-3: Discontinuity Orientation Data for 2014 Geotechnical Investigation

Drill Hole ID	Drill Hole Length (m)	Total Drill Hole Televiewed (m)	Success of Televiewer (%)	Total Discontinuities Logged
NEC14-006	772.67	499	65%	671
NEC14-007	907.39	723.6	80%	1,132
NEC14-008	886.05	758	86%	908
NEC14-009	751.33	287	38%	754
NEC14-009a	897.03			
NEC14-010	796.14	564	71%	1,157
NEC14-011	900.38	807	90%	1,748
NEC14-012	843.23	751	89%	1,305
NEC14-013	880.26			
NEC14-014	900.99	716	79%	2,129
NEC14-015	827.84	706	85%	1,691
NEC14-016	913.79	758.4	83%	2,048
NEC14-020	587.7			
NEC14-021	865			
NEC14-022	949.7	897	94%	3,824
NEC14-023	615.1			
NEC14-MET-01	894.74	560.2	63%	1,222
NEC14-MET-02	865.02	820	95%	1,280
NEC14-MET-03	913.33			
NEC15-004	413.6	383	93%	1,241
NEC15-005	407.5	264	65%	995

Source: SRK, 2017

The RMR_{76} values ranged from 50 to 70 in the hanging wall rock, with Barton Q' values ranging from 4 to 20 with the majority of the rock mass being of fair to good quality. In the footwall, RMR_{76} values ranged from 50 to 80, with Q' values ranging from 4 to 30 with the majority of the rock mass being of fair to good quality. The mineralized rock had the greatest RMR_{76} variations, where values ranged from 60 to 80, and Q' values ranged from 5 to 25, but overall the rocks are fair to good quality.

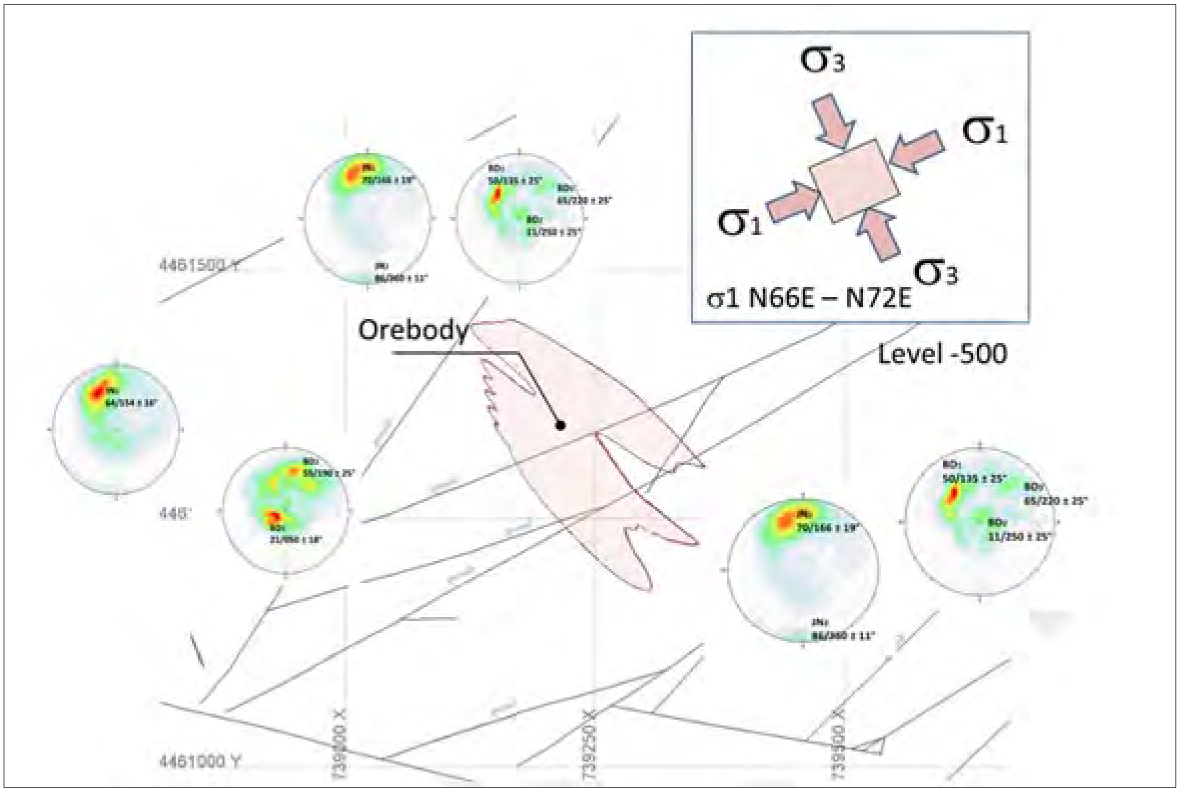
Pre-Mining Stress Regime

Since no information on in situ stresses was available in the region and this underground mine would be the first in the district, a stress measurement program was undertaken.

In September 2014, Agapito Associates, Inc. (AAI, 2014) performed downhole in situ horizontal stress testing at the site using the Sigra IST tool. The purpose of the testing program was to estimate the in situ horizontal stress field in the two basic rock types present at the site: the un-mineralized Pennsylvania rock (above 200 m bgs depth) and the mineralized carbonatite ore zone (below 200 m bgs). A total of thirteen tests were conducted; eight of which were successful. The results of the study were as follows:

- There is an apparent increase of stress with the depth of about 36 kPa/m for the major horizontal stress (σ_H) and 21 kPa/m for the minor horizontal stress(σ_h);

- The major horizontal stress is about 20% greater than the vertical stress (σ_H / σ_V =1.2), and the minor horizontal stress is 71% of the vertical stress (σ_h / σ_V =0.7);
- The average orientation of the major stress is N 66° E. However, a calculation using ATV borehole breakout data provides a better estimate; and
- Figure 16-6 shows a summary of the major and minor stress orientation relative to the fracture set orientations and fault structure orientations.



Source: Nordmin, 2019

Figure 16-6: Orientation of Stress Measurements Relative to Faults and Fracture Orientations

<u>Seismicity</u>

A high-level assessment of the local seismic earthquake potential from the International Building Code suggests a local Peak Ground Acceleration (PGA) of 0.02 g for a 50-year return earthquake event. The source of the peak ground acceleration is the 2002 USGS Interactive National Seismic Hazard Map (Frankel et al., 2002). It shows the Maximum Design Earthquake (MDE) with an expected 1% probability of having an earthquake of magnitude greater than 5.0 in 100 years.

No additional studies of seismicity were conducted since the region is not particularly known for large earthquake activity.

<u>Mine Layout Parameters</u>

SRK evaluated different mining methods. Given constraints on limiting the extent of surface disturbance, long-hole open stoping with backfilling was selected as the optimal mining method for the orebody. To maximize ore extraction, SRK has selected a primary/secondary stope extraction sequence, whereby primary stopes are mined first on the first pass and backfilled prior to mining secondary stopes.

The orebody shape is longest in the northwest-to-southeast direction, which is nearly perpendicular to the major principal stress (σ_1). Considering the principal stress orientations, the major geologic structures, and the local discontinuity orientations, SRK has chosen the stopes orientation to be N 60°E creating the most favourable ground conditions during mining. In this way, the major principal stress is redistributed around the smallest dimension of the primary stopes.

To minimize long-term, mining-induced damage to access drifts, the setback distances used in the design of the mine is 25 m for haulage drives and 75 m for the main ramps. These values represent the minimum distance between the drifts and induced stresses from stope mining. These setback distances were verified from the results of 3D numerical modelling of the mining sequence (SRK, 2017).

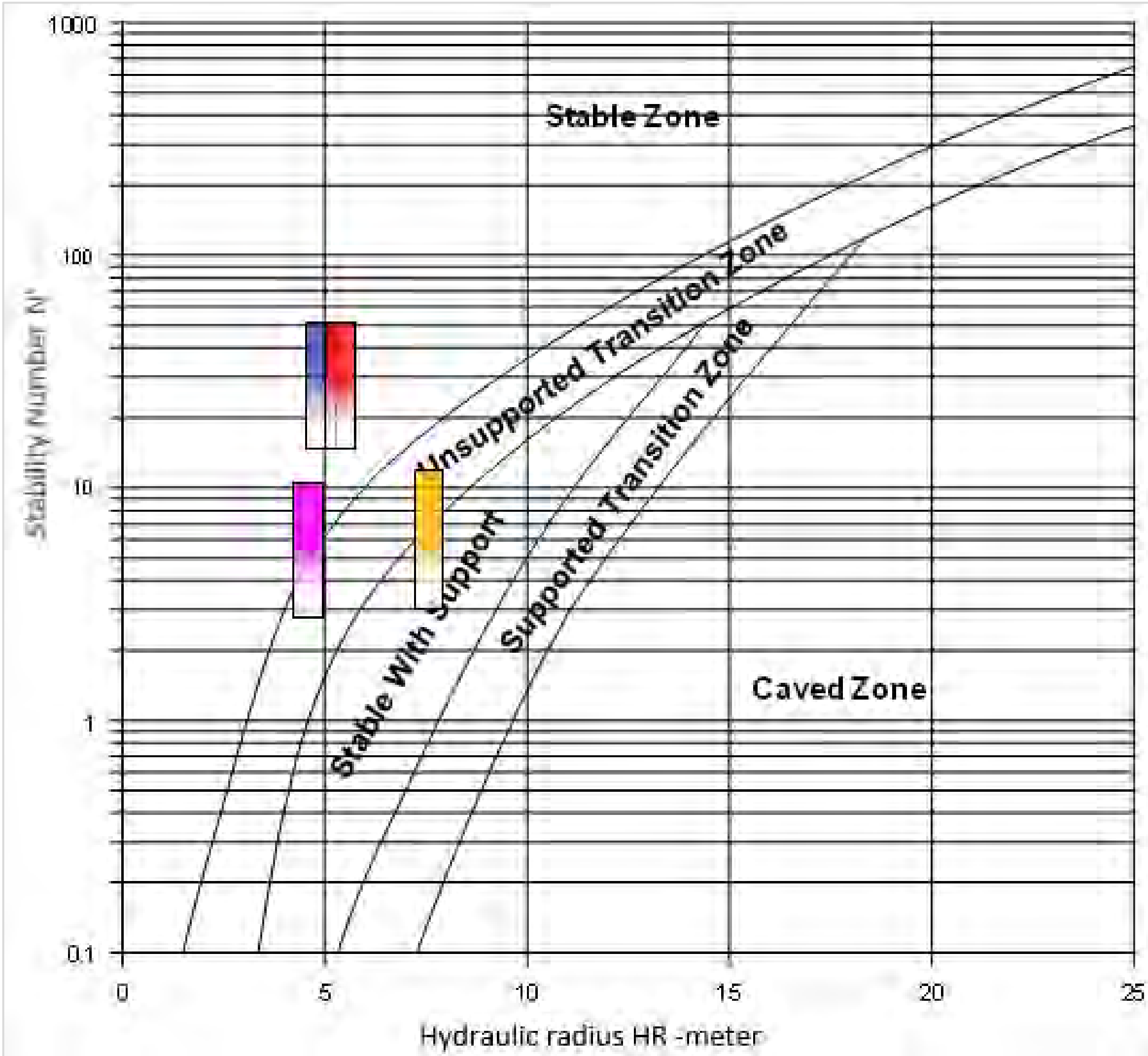
Stope Dimensions

A stope stability assessment was completed using the stability graph method (Potvin and Milne, 1992 and Nickson, 1992) with open stope dimensions of:

- Width: 15 m
- Height: 40 m
- Length: 40 m in fresh rock mass (70% of time), 25 m in moderately weathered rock mass (30% of time)

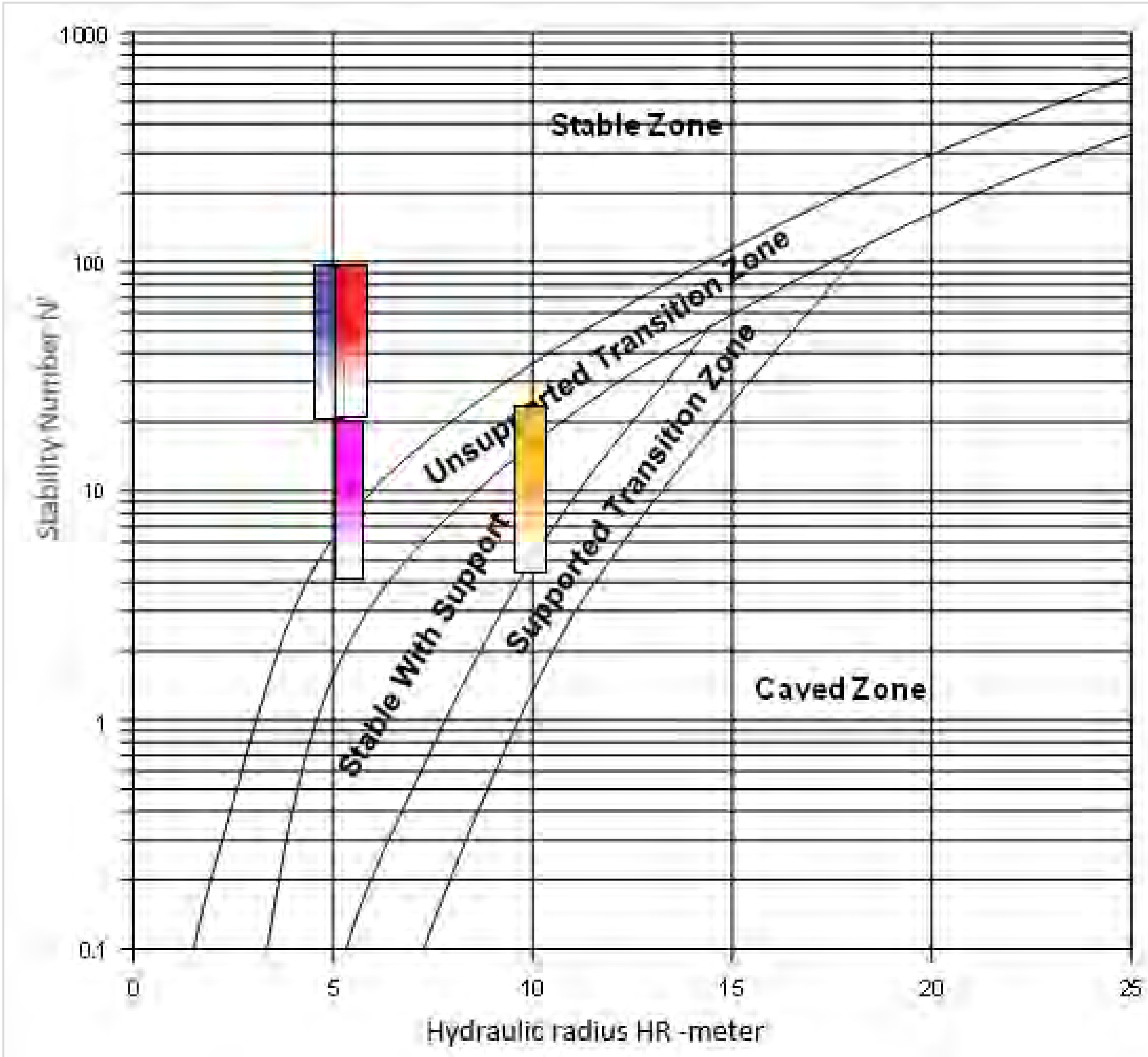
The method compares the hydraulic radius (area divided by perimeter) of a stope face to a stability index number. The stability index number accounts for the rock mass quality (primarily Q values) with adjustments for local fracture orientations, potential block failure mode into the stope, and induced mining stresses. The results from the stability assessment indicated that the initial dimensions for stopes are stable. Figure 16-7 and Figure 16-8 show the stability chart of each stope face under slightly weathered and moderately weathered carbonatite conditions, respectively. The range of stability numbers shown for each stope face consider changes in depth and variations in rock mass quality within each geotechnical domain.

The difference between these stope dimension from the 2017 feasibility study is that the stope height is taller by 10 m, but the stope lengths are shorter by 10 m. the result is a very similar hydraulic radius for the critical sidewall stability. The narrow stope widths, oriented along the major principal stress, are the same as in the 2017 feasibility study, so the taller stopes have minimal stability impact for the hanging wall and footwall.



Source: SRK, adopted from Potvin, 2001

Figure 16-7: Empirical Stope Design Chart for Moderately Weathered Rock Mass (Red=Footwall Face, Blue=Hanging Wall Face, Magenta=Back, Yellow=Sidewall Faces)



Source: SRK, adopted from Potvin, 2001

Figure 16-8: Empirical Stope Design Chart for Fresh and Slightly Weathered Rock Mass (Red=Footwall Face, Blue=Hanging Wall Face, Magenta=Back, Yellow=Sidewall Faces)

A second stability assessment was completed by using larger stope dimensions (40 m length) assuming 15% dilution along the walls. Results of this analysis indicate that these diluted stopes should be stable but may require some support depending on rock mass quality. The actual in situ rock mass conditions need to be incorporated in the final stope designs.

Stope stability was simulated using a 3D numerical model for the 2017 feasibility study stoping sequence and dimensions (SRK, 2017). Although the design stope dimensions and mining sequence have been changes since these analyses were conducted, the results are considered generally applicable because the hydraulic radius and net extraction ratios are quite similar. The modelling results confirm that stopes and access drifts are predicted to remain stable during active mining. SRK recommends that the numerical analyses should be re-assessed as the project design stage is advanced.

Backfill Strength

An initial backfill strength assessment was completed for the primary/secondary stopes. This assessment identified the minimum strength required for the primary panels to be self-supporting when they are exposed on the east and west walls. Two methods were used to determine the minimum backfill strengths: 2D analytic vertical stope method and numerical modelling using FLAC3D stope-scale model (SKR, 2017), although the current stope sizes are slightly different.

The analytical methods included Li and Aubertin (2014) and Belem and Benzaazoua (2000) for single stope face of backfill exposed during secondary stope mining (exposing two backfill faces simultaneously is not planned). Using a factor of safety of 2.0, a 14-day UCS strength of 1.0 MPa was considered the minimum strength for the primary stope panels. The numerical modelling results confirmed these values (SRK, 2017). The model was also used to verify that secondary stopes would remain stable in that no “sit-down” failure mechanism would occur against two adjacent backfilled stopes during the progressive mining of the secondary stope.

Dilution

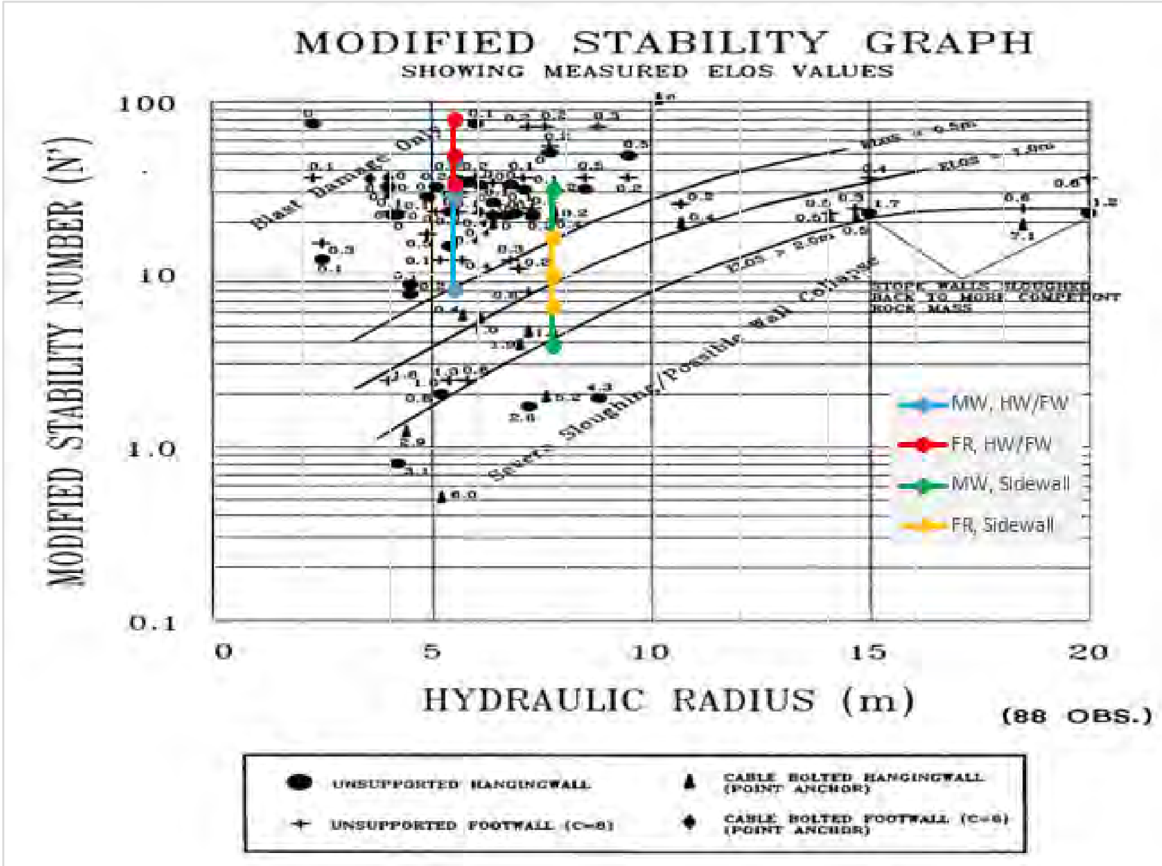
Dilution was estimated using the method developed by Clark and Pakalnis (1997) based on an empirical model calibrated to case histories for dilution into open stopes. The method predicts the quantity of unstable wall rock for a given rock mass quality from a given stope size. The parameters plotted on the dilution chart are the stability number, N’, and hydraulic radius.

The thickness of external dilution is estimated as an ELOS. Figure 16-9 shows the range of predicted equivalent linear overbreak/slough (ELOS) for moderately weathered carbonatite, and for fresh to slightly weathered carbonatite. Sidewall and back dilution are not expected to be a problem because in the primary stopes dilution (from secondary stopes) will be at grade, and dilution from secondary stopes is managed by controlling backfill strength. Given the low fracture frequencies (about 3 m / fracture) dilution is anticipated to be variable.

For stope sidewalls, the dilution depends on stope length and depth below ground and is summarized as follows.

- 25 m long stopes in moderately weathered rock mass dilution from HW & FW = 0.25 m (blue)
- 25 m long stopes in moderately weathered rock mass dilution from Sidewalls = 0.25 m in shallow stopes increasing to about 2 m in deepest stopes (green)
- 40 m long stopes in fresh (unweathered) rock mass dilution from HW & FW = 0.10 m (red)
- 40 m long stopes in fresh (unweathered) rock mass dilution from HW & FW = 1 m in shallow stopes increasing to about 2 m in deepest stopes (yellow)

For design purposes, an average of 0.25 m of ELOS dilution has been assumed for endwalls 0.75 m for sidewalls. In backfilled sidewalls, an average of 0.2 m has been assumed.



Source: SRK, 2017, adapted from Clark & Pakalnis, 1997

Figure 16-9: Empirical ELOS Estimate – Moderately Weathered and Fresh Rock Mass

Ground Support

Ground support requirements have been estimated using empirical support charts developed by Barton (1974). The method relates the rock mass quality (Q) to the equivalent dimension of the excavation (De). De is the ratio of the excavation width (D) to the excavation support ratio (ESR) index. The ESR value accounts for a degree of safety required depending on the use of the excavation. Values range from ESR=1.6 for long-term critical access drifts to 2.5 for short-term temporary accesses into stopes. The recommended ground support consists of three general classes of support levels based on rock mass quality, drift usage and drift dimensions. The three-class system includes:

- Support Type 1 – spot bolting for Q>4 (~69% of drifts);
- Support Type 2 – systematic bolting and 4 to 10 cm of fibre reinforced shotcrete for Q<4 and Q>1 (~24% of drifts); and
- Support Type 3 – systematic bolting, steel mesh, and 4 to 10 cm of plain shotcrete for Q<1 (~7% of drifts).

Table 16-4 summarizes the drift dimensions used to estimate ground support requirements. The support specifications are summarized in Table 16-5.

Table 16-4: Barton Parameters for Different Excavations

Excavation	Type of Excavation	Opening Dimensions W x H (m)	ESR	D	De
Main Ramps	Long Term (6-10 years)	5.5 x 5.5	1.6	5.5	3.4
Footwall Accesses	Medium Term (1 year)	5.0 x 5.0	2.0	5.0	2.5
Stope Accesses	Short Term (1-2 months)	4.5 x 4.5	2.5	4.5	1.8

Source: SRK, 2017

Table 16-5: Preliminary Support According to Barton Method

Geotechnical Zone	Q	Excavation	Support Categories	Bolt Length	Bolt Spacing	Other Support
Footwall, highly weathered (6%)	0.4 to 6.2 (Very Poor)	Main Ramp	3-Bolts, mesh and shotcrete	2.5 m	1.2 m	Fully grouted rebar, mesh, 5 cm shotcrete
		FW Access	2-Systematic bolting	2.5 m	1.2 m	Split sets and mesh
		Stope Access	2-Systematic bolting	2.5 m	1.6 m	Split sets and mesh
Footwall, moderately weathered (24%)	3.2 to 13.8 (Poor-Fair)	Main Ramp	2-Systematic bolting	2.5 m	1.2 m	Fully grouted rebar, mesh
		FW Access	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Split sets
		Stope Access	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Split sets
Footwall, slightly weathered (70%)	5.9 to 28.1 (Fair-Good)	Main Ramp	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Grouted rebar
		FW Access	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Split sets
		Stope Access	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Split sets

(%) Amount of Expected Ground
Source: SRK, 2017

Smaller drifts in the competent ground may use short length bolts (i.e., 1.8 m long versus 2.5 m) in the good quality ground, depending on ground conditions. Also, in short-term accesses in the good quality ground, it may be possible to substitute swellex or split set bolts, depending on ground conditions. The need for cable bolts and/or shotcrete in stope brows will be dependent on ground conditions and the degree of fracturing in the brow area. The final decision on these substitutions will be up to the on-site geotechnical ground support engineer.

Crown Pillar Stability

Crown pillar stability assessment was conducted using the empirical Scaled Span Method (Carter, 1992) and a limit equilibrium analysis in the CPillar software. The results of the empirical analysis indicate that if the rock mass quality in the crown has a Q value greater than 37, then all stopes could be mined in Level 1 with a Factor of Safety (FOS) of more than 2.0. No additional analyses are required at this time because the stopes are planned to be tightly backfilled and the FOS will be much higher.

The results of the 3D numerical model (SRK, 2017) confirm that the crown pillar will remain stable during and after mining. The total predicted surface displacement is anticipated to be less than about 1 cm at the end of mining.

16.3 Hydrogeology Design Parameters

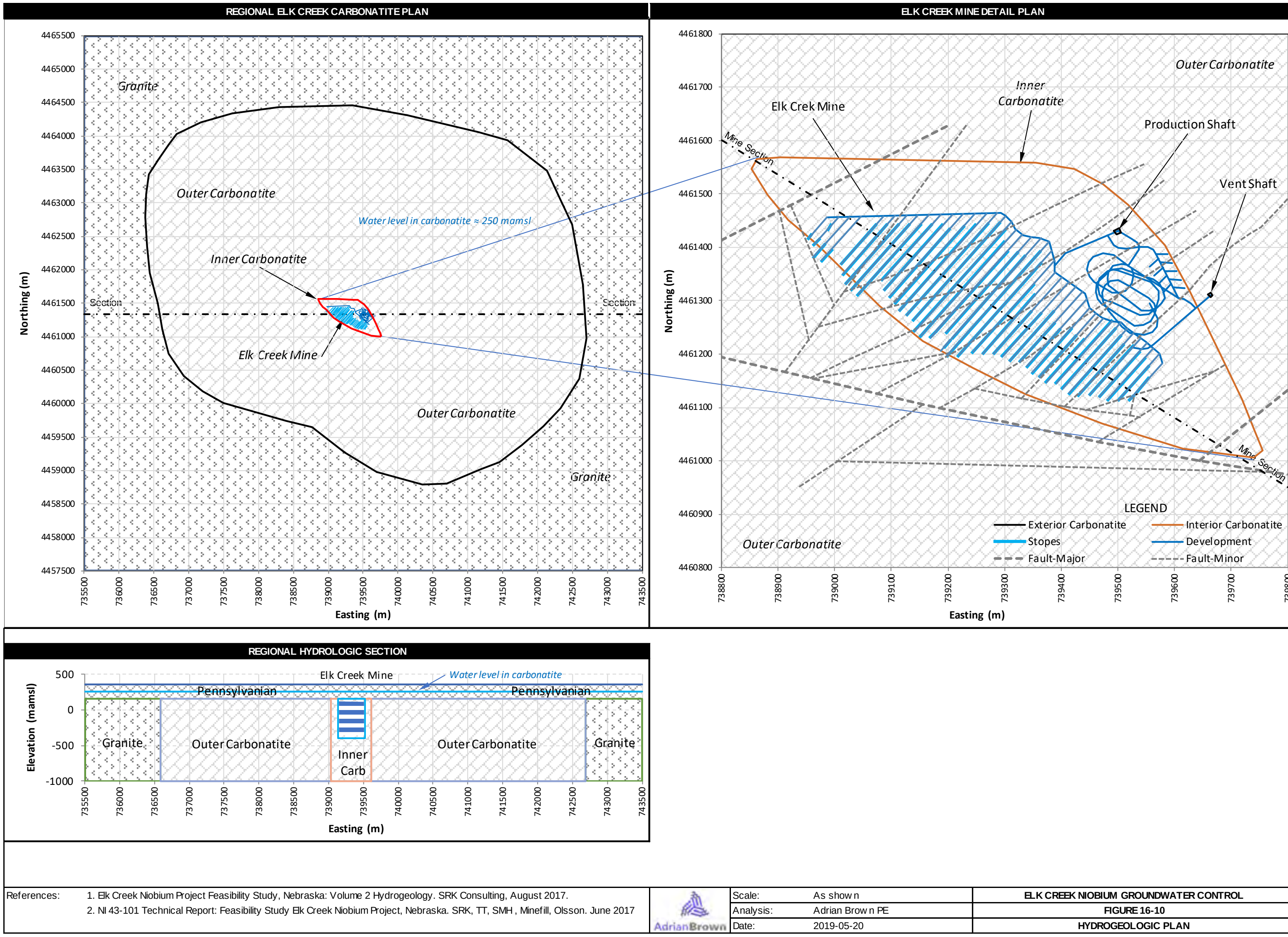
The hydrogeology of the deposit was characterized based on three phases of work:

1. Phase I: The first phase of hydrogeological characterization was conducted during phases 1 and 2 of the core drilling program and consisted of packer testing, installation of piezometers, and measurement of water levels. Specifically, the program included:
    - 42 downhole packer-isolated injection and airlift tests in drill holes.
    - Installation of six 2“ PVC standpipe piezometers isolated in the carbonatite and open to large intervals of the deposit.
    - Installation of two nominal 2“ PVC standpipe piezometers isolated in the 180 m thick Pennsylvanian aquitard above the carbonatite.
    - Frequent measurement of water levels in open drill holes and piezometers over a period of six months.
2. Phase 2: Following the second phase of resource-related core drilling, a 10-day airlift pumping test was completed using a deep, open, vertical PQ drill hole as a pumping well. Water levels from the surrounding piezometers were recorded over the duration of the test and for several weeks following the test.
3. Phase 3: The third phase of hydrogeological characterization involved installation of two multi-level, distally-located piezometers and a deep 6” diameter injection well completed to depths of 850 m, followed by the performance of a nominal 30-day injection test. The piezometers were completed within the carbonatite at distances of 0.6 km and 1.2 km from the center of the injection well, which was located at the center of the orebody. The injection test was chosen as a test method over a standard pumping test due to the salinity of the groundwater and the expense of handling the discharge water. During the injection test, surface water from Todd Creek was injected at rates of between 22 L/s and 30 L/s (350 to 480 gpm) over a period of 33 days, including downtime. Response to the injection test was monitored over the duration of the test and for more than eight weeks following the test.

16.3.1 Conceptual Geohydrology

Geology

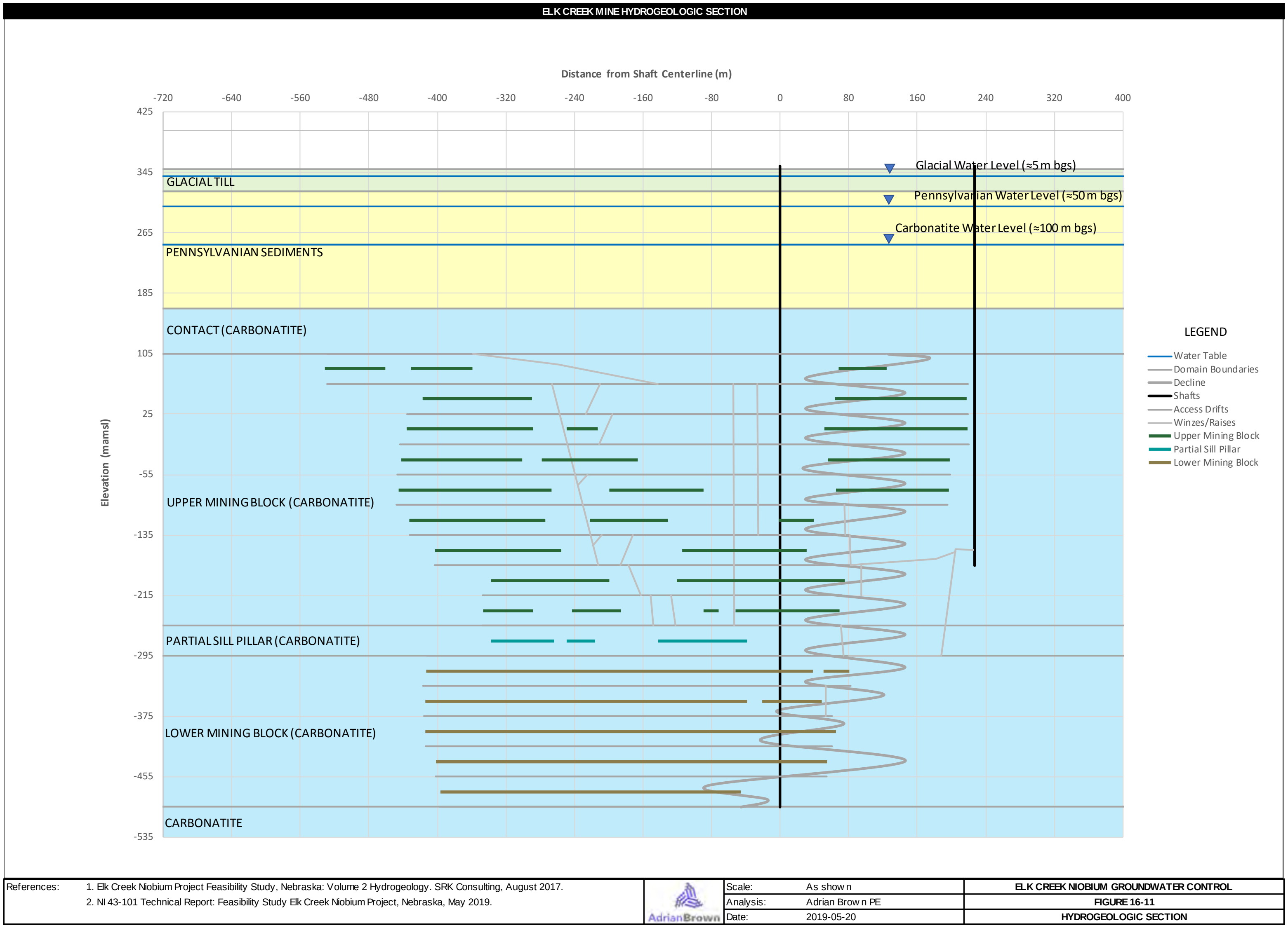
The Elk Creek Deposit is hosted in the Elk Creek Carbonatite, a volcanic carbonatite plug located in south-east Nebraska. The carbonatite plug is 6 km to 8 km in diameter and contains the orebody at its approximate center. Surrounding the plug is low permeability Precambrian-aged granite bedrock (see Figure 16-10).



Source: Adrian Brown Consultants, 2019

Figure 16-10: Hydrogeologic Plan

The local geology generally consists of a 30 m thick layer of low permeability Pleistocene-aged glacial till overlying a 180 m thick low-permeability Pennsylvanian-aged shale and limestone, which rests on top of a moderate-permeability Phanerozoic-aged carbonatite volcanic plug extending to a depth in excess of 1,000 m (see Figure 16-11).



Source: Adrian Brown Consultants, 2019

Figure 16-11: Hydrogeologic Section

Glacial Till

Pleistocene-aged glacial till covers the surface of the site, to a depth of approximately 30 m. It is variably permeable, with lenticular glacial outwash features providing potable water to shallow wells that service local agricultural activities. Water levels in these wells are typically within 10 m of the ground surface.

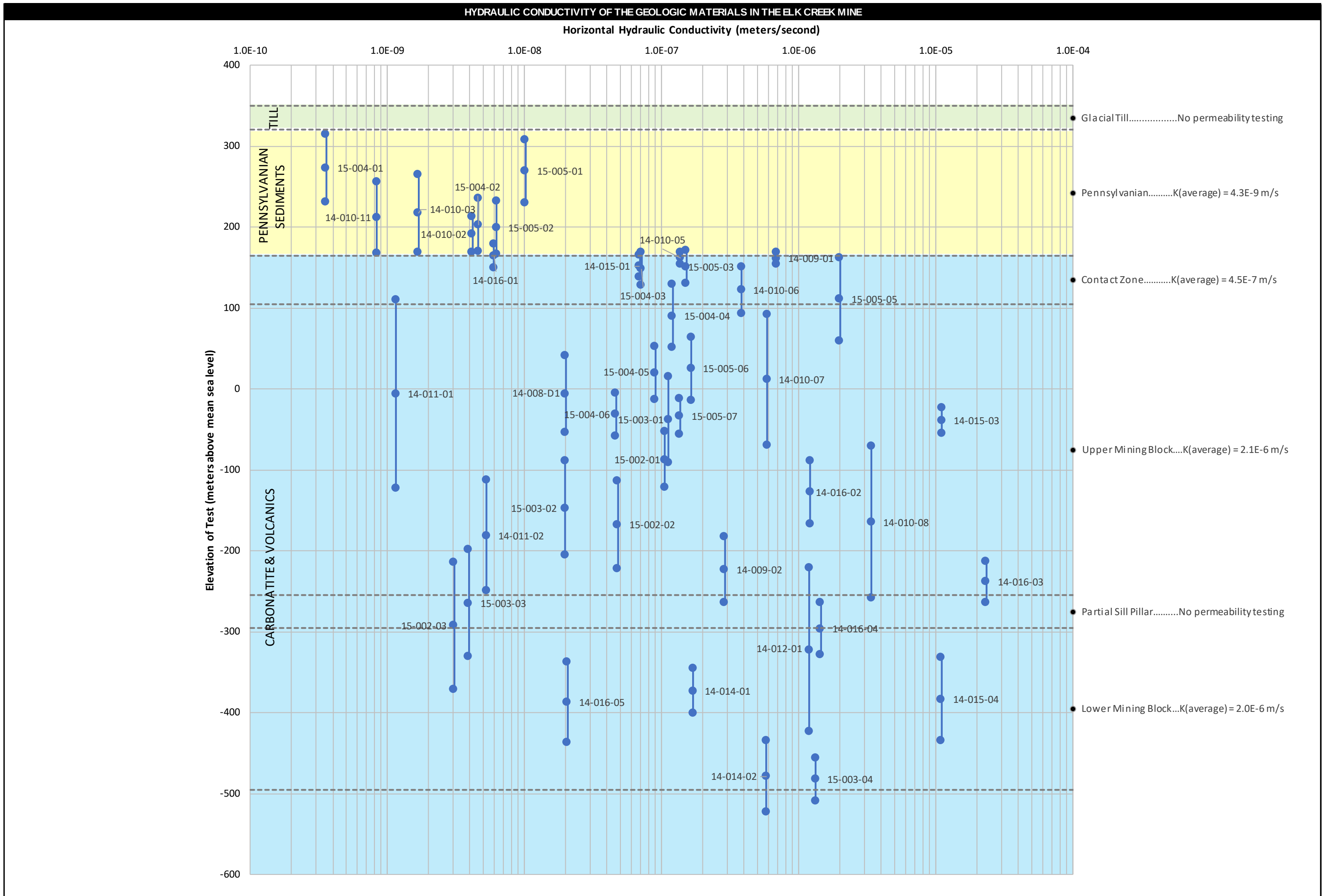
Pennsylvanian Sediments

The Pennsylvanian sediments are made up of interbedded shale and limestone strata. The ensemble displays horizontal hydraulic conductivity on the order of 10^{-9} m per second in localized testing from single wells, with the vertical hydraulic conductivity of the horizontally layered strata likely 100 times lower. This unit hydraulically isolates the glacial till groundwater system from the deeper carbonatite and effectively functions as an aquiclude for vertical water infiltration to the carbonatite below.

Water levels in wells completed in this unit are typically 50 m below ground surface, indicating a vertical downward head gradient from the glacial till above to the carbonatite below. However, due to the very low vertical permeability of the sediments, there is essentially no vertical downward groundwater flow through them.

Phanerozoic Carbonatite

The Phanerozoic carbonatite unit is made up of carbonatite (volcanic calcium-magnesium-iron carbonate) with siliceous lamprophyre dikes and sills interspersed throughout. The intact carbonatite and lamprophyre rocks are essentially impermeable, and the rock mass is generally lightly fractured, resulting in generally low hydraulic conductivity. However, the plug is intersected by fractured zones and dissolution structures that are interpreted to be related to faulting that likely occurred at the time of emplacement of the carbonatite. As a result, the total orebody material is of moderate hydraulic conductivity, on the order of 10^{-6} m per second (based on the multiple well pumping and injection tests), ranging locally from 10^{-9} to 10^{-5} m per second (based on the results of single hole testing, shown in Figure 16-12).



References:	1. SRK 2017: *Elk Creek Niobium Project Feasibility Study, Nebraska: Volume 2 Hydrogeology*. SRK Consulting, August 2017.		Scale:	As shown	ELK CREEK NIOBIUM GROUNDWATER CONTROL
	2. NI 43-101 Technical Report: *Feasibility Study Elk Creek Niobium Project, Nebraska*. SRK, TT, SMH, Minefill. June 2017.	AdrianBrown	Analysis:	Adrian Brown PE	FIGURE 16-12
	3. Brown A, 2019: *Control of Inflow to the Elk Creek Mine by Grouting*; Adrian Brown Consultants, Inc., dated May 2019.		Date:	2019-05-20	HYDRAULIC CONDUCTIVITY OF THE ELK CREEK OREBODY

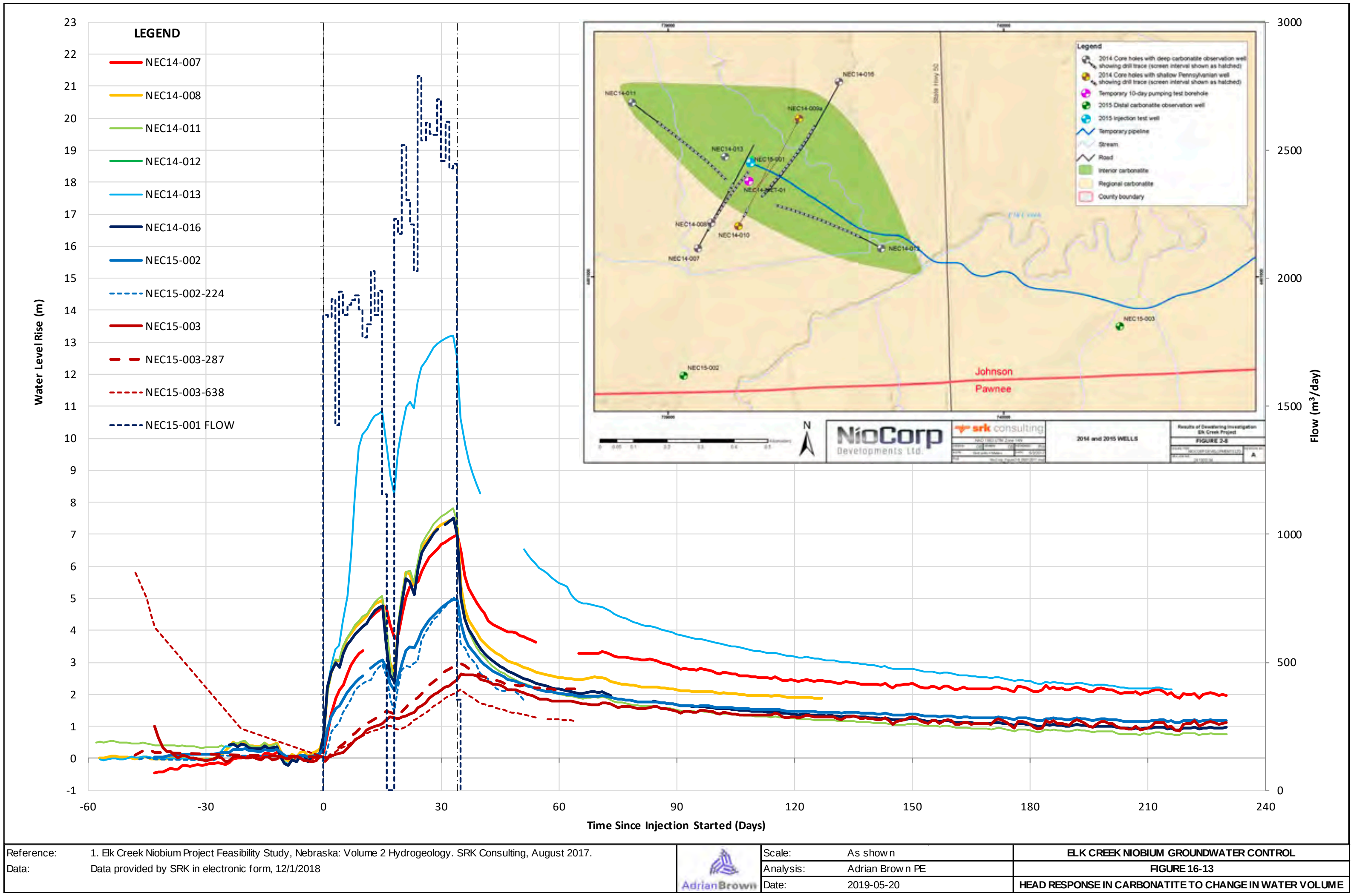
Source: Adrian Brown Consultants, 2019

Figure 16-12: Permeability Testing of the Elk Creek Rock Mass

The nature of the fracturing and dissolution of the carbonatite changes with depth in the orebody, which also causes variability of permeability with depth, as follows:

1. Contact Zone: The uppermost 90 m of the carbonatite show heavy fracturing and occasional dissolution cavities, indicating that it has been subject to long-term leaching by fresh groundwater from above, likely occurring prior to the emplacement of the overlying Pennsylvanian sediments. The ubiquitous presence of this zone in all drill holes penetrating the carbonatite and the appearance of this rock in core suggests it forms a thin dissolution caprock on the entire carbonatite plug, similar to caprocks seen at the top of salt domes. The permeability of this zone averages 4×10^{-7} m per second but is highly variable.
2. Mining Zone: The orebody from approximately 90 m to approximately 600 m below the top of the carbonatite is made up of largely unfractured carbonatite and other volcanics, intersected with rubble zones, probably caused by faulting, with some associated evidence of dissolution on joints and fractures. These broken rock zones impart an elevated hydraulic conductivity to some locations in the orebody, which collectively cause this portion of the rock mass to have an average hydraulic conductivity in the order of 2×10^{-6} m per second. The local hydraulic conductivity is also highly variable, with values up to 2.5×10^{-5} m per second.

Within all zones, the permeability of the carbonatite appears to be locally quasi-isotropic and relatively homogenous, based on the uniform drawdown cone observed at the end of the large-scale injection test (see Figure 16-13).



Source: Adrian Brown Consultants, 2019

Figure 16-13: Response to Injection in Carbonatite - End of Test

Water in the carbonatite is saline. When sampled from wells unaffected by injection of fresh water by drilling or testing, the carbonatite groundwater has a total dissolved solids concentration of approximately 19,000 mg/L, and a sodium chloride concentration of approximately 16,500 mg/L, comprising 89% of the total dissolved solids concentration (the balance is mostly calcium, magnesium, and carbonate). The high salinity of the water in the carbonatite cannot have originated from the overlying materials, which are essentially devoid of sodium chloride. Accordingly, the sodium chloride must either have come from dilution of connate seawater, or dissolution of evaporates located in the carbonatite rock mass. Given the volcanic origin of the carbonatite, seawater is considered to be the more likely genesis. This indicates that the water in the carbonatite is a captured volume of seawater and has been diluted over geologic time by slow infiltration of fresh water through the overlying Pennsylvanian strata, with the resulting dense water very slowly discharging from the carbonatite at great depth through the surrounding Precambrian granite.

Water levels in wells and piezometers completed in the carbonatite unit are currently approximately 100 m below ground surface. This water level is approximately equal to the water level in the Missouri River, 52 km to the east, which demonstrates that the water level in the carbonatite is not controlled by regional drainage to the river. The reasons that the water level is depressed below the level expected to exist in an open groundwater flow system may include the following:

1. Salinity. The water in the carbonatite is saline, approximately half the sodium chloride concentration of the ocean, and is approximately 0.1% denser than pure water. This column of water in the carbonatite balances the water pressure in the granite outside, which it is reasonable to expect is in general non-saline. As a result, at any point above the point where saline carbonatite water seeps into the surrounding granite, the static (saline) water level in the carbonatite will be less than the static water level in the granite. In the case of the Elk Creek Carbonatite, the water level in the carbonatite would be expected for this reason alone to be approximately 5 m to 10 m lower than the water level in a pure water system, such as the overlying till.
2. Isolation. If the carbonatite is an isolated system, its current water level will not bear any set relation to the water levels in adjacent materials. The injection test performed as part of this project injected a total of 68,588 m^3 of (fresh) water into the carbonatite, which produced a long term – apparently permanent – increase in the water level in the entire 40 km^2 carbonatite plug of approximately 1 m (see Figure 16-13). The storativity displayed by the rock mass is 0.003, which is typical for a thick confined fractured rock aquifer. This permanent head-change behaviour is indicative of an effectively isolated system, in which the water level that is currently observed is the result of all the inputs and outputs of water to the carbonatite over geologic time.
3. Isostatic Uplift. The carbonatite lies within the area of late Winsconinan glaciation and was most recently covered by between 2 km and 4 km of the glacial ice sheet between 10,000 and 25,000 years ago. This ice sheet acted as a load on the carbonatite system, compressing it with an added total stress of approximately 20 to 40 Mega-Pascal (MPa). When the glacial sheet ice melted, this load was removed quite rapidly in geological terms, causing a large reduction in total stress in the carbonatite. As the carbonatite is isolated, this reduction in total stress resulted in a corresponding reduction in the porewater pressure in the carbonatite, reducing the water head in a hypothetical piezometer in the carbonatite by up to between 2,000 m and

4,000 m. The porewater pressure in the carbonatite and the water level along with it has been slowly recovering since, by drawing water from the surrounding Precambrian granite and overlying Pennsylvanian sediments into the carbonatite. It appears that when the current investigation program began, the water level had recovered to within 70 m of the equilibrium level, which is represented by the current water level in the glacial till.

4. Investigation Extraction. The investigation of the carbonatite has injected water into and withdrawn water from the carbonatite. As was seen in the injection test, this can permanently influence the water level in the carbonatite, as it is an effectively closed system. In the case of the current program, the drilling has generally used coring technology, which involves the injection of water to keep the bit cool and to remove cuttings. Lost circulation has been a persistent issue, so the drilling has if anything caused a net injection of water into, and an increase in water level in, the carbonatite. The only deliberate withdrawal from the carbonatite occurred during the 10-day extraction test, which extracted 1,900 m^3 of water. Based on the injection test result, this would have reduced the water level in the carbonatite by a net 0.03 m (3 cm).

In conclusion, the carbonatite is an isolated, fractured volcanic plug, surrounded and covered by essentially impermeable materials. It has a moderate hydraulic conductivity at the top of the plug, decreasing with depth due to reduced fracturing and dissolution. The head conditions and the large-scale testing of this material show that it is hydraulically isolated, and the water pressure is lithostatically controlled, with effectively no exit for water from the carbonatite to any other material.

<u>Precambrian Granite</u>

Precambrian granite surrounds the carbonatite and acts as a containment structure for the water within it. The granite has a hydraulic conductivity of less than 10^{-4} m per day based on the results of modelling the long-term injection test. The granite provides very limited opportunity for groundwater flow laterally to or from the carbonatite, consistent with the above conclusion about the genesis of the brine contained within the carbonatite rock mass, the observed static water level in the carbonatite, and the permanent water level changes induced by injection testing at the site.

16.3.2 Mine Inflow Control

<u>Concept</u>

Mine inflow control will be achieved in the Elk Creek Mine by limiting groundwater inflow to the mine to a maximum flow of 66 L/s (1,000 US gpm). This will be achieved by freezing and grouting the shafts, grouting the rock around development drifts and orebody stopes, and backfilling worked-out stopes with cemented backfill. Mine inflow will be pumped to the surface for treatment, with the filtrate used or discharged.

<u>Shaft Inflow Control</u>

Groundwater inflow to the shaft and the associated breakout will be controlled by freezing of the rock in advance of shaft sinking. A sealed concrete pressure liner will be emplaced in the shaft progressively during sinking to control inflow after completion of the shaft sinking and cessation of freezing.

Breakout stations will be grouted from within the shaft liner to control inflow during development and may be lined or dentally grouted to complete flow control once accessed.

Inflow to the shaft and breakout stations is expected to be nominal, with a peak estimated at 10 L/s (150 US gpm). If the flow exceeds this value, additional freezing or dental grouting will be employed.

Development Inflow Control

Inflow to all permanent development drifts, transport haulage ways, ramps, and other non-production underground facilities will be prevented by grouting significant inflow conduits (generally faulted zones within the orebody that contain rubble and dissolution pathways). The method of grouting will be developed during the initial excavation of the mine, and is expected to be generally as follows:

1. Prior to development of the underground facility, cover holes will be drilled from the access point at a spacing of approximately 5 m (15 ft), and grouted with superfine cement grout to a pressure equal to 150% of the static water pressure originally computed for that location, to create a low-permeability envelope around the facility advance.
2. Following excavation of the facility or drift, all locations where there is visible inflow to the mine in excess of 1 L/s per 100 m of facility length will be sealed by dental grouting, using shear-activated grout injected at a pressure up to 150% of the observed water pressure in the area to be grouted.

Stope Inflow Control

Inflow to all mined stopes will be controlled during mining by grouting any major inflow conduits (generally faulted zones within the orebody that contain rubble and dissolution pathways). After mining is complete, inflow will be controlled by sealing the stopes using cemented paste backfill. The method of mine inflow control to be used for stopes will be developed during the mining process, and is expected to be generally as follows:

1. Prior to the development of a stope, a drill hole will be advanced along the stope centerline to the distal end of the stope.
2. In the event that the free-flow from the advance drill hole exceeds 3 L/s (50 gpm), the advance hole will be pressure grouted with ultrafine neat cement grout at a pressure up to 200% of the calculated pre-mining static water pressure at that location to control inflow during the mining of the ore in the stope.
3. After mining is completed, the stope will be completely backfilled with cemented paste, creating a low permeability inclusion in the stope to reduce flow from the stope to less than 0.5 L/s (10 gpm). If the outflow from the stope to the mine workings exceeds 0.5 L/s, a concrete bulkhead will be constructed in the access drift to the stope with a pressure rating equal to 150% of the calculated pre-mining static water pressure at that location. Any remaining leakage past the bulkhead will be controlled by dental grouting of the rock surrounding the bulkhead.

Impact of Mine Inflow Control on Groundwater Pressure in the Carbonatite

The groundwater control system proposed for the Elk Creek Mine will result in the extraction of up to 66 L/s from the isolated carbonatite volcanic plug. This will have the effect of reducing the water head pressure in the carbonatite plug at a rate of up to 30 m per year, which will dewater orebody down to the base of the proposed mine by the end of mining.

This dewatering has an impact on the optimal order of mine development, favouring early development in low permeability deep portions of the orebody, and later mining of higher-permeability portions of the orebody when the water pressure in them has been reduced by the prior mine inflow. The current mine plan takes advantage of this opportunity, at the same time as targeting the higher-grade ore, which is generally in the deeper portion of the mine, for early extraction.

16.4 Mine Design

16.4.1 Selection of Mining Method

The mining method selected for this ore body was based on economic parameters and geotechnical information, ensuring it was suitable for the mineralization geometry. Due to its depth and requirement for selectivity in mill feed grades, the underground longhole stoping method (LHS) was selected. Given the bulky geometry of the deposit, a block caving or sub-level caving method could have possibly been economically viable. However, the limited selectivity of such methods would not allow for optimizing the higher value of this deposit given their production constraints. To maximize the recovery of the high grade zones, a longhole stoping method utilizing paste backfill was used.

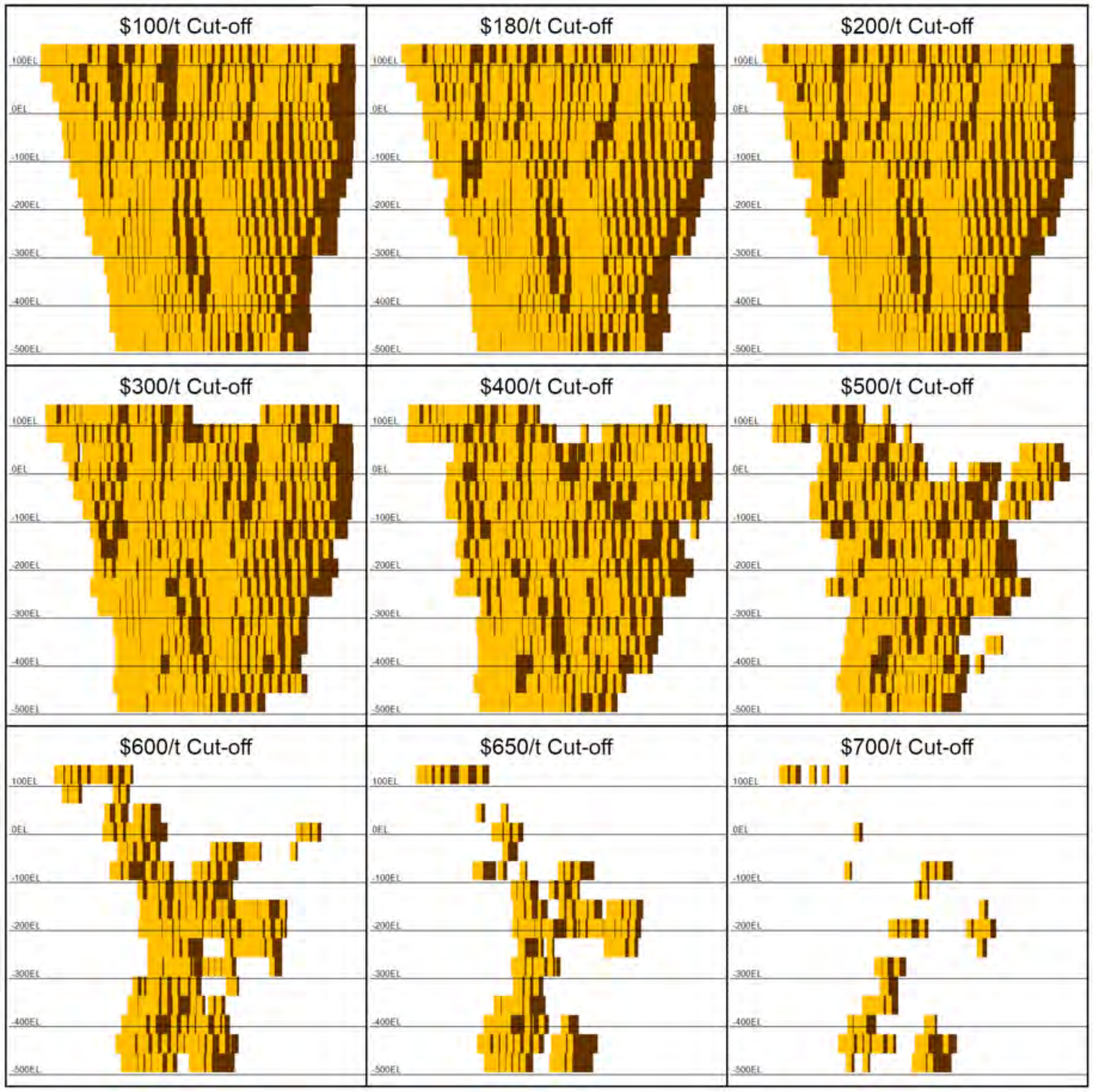
The stopes dimensions are 15 m wide, and stope length varies based on Nb_2O_5 mineralization grade to a maximum of 25 m per panel with a level spacing of 40 m. The variation on stope length allowed for optimization of the Nb_2O_5grade with a minimal increase to operating costs. The level spacing of 40 m was beneficial to operating and sustaining capital costs. Each block is mined with a bottom-up sequence. A partial sill pillar level is designed to be left between these two mining fronts/blocks. The extraction of ore from the partial sill pillar level is expected to be 62.5% using production up-holes through 25 m of the 40 m thick sill pillar and is accounted for within the reserves. This methodology will allow partial mining of ore on the sill pillar level, while at the same time allowing the development of the lower mining block and establishing an early start to the mining of the upper mining block. Using this approach minimizes the impact on initial capital investment. The backfill was designed to have an adequate strength to allow for mining adjacent to filled stopes, thus eliminating the need for rib pillars. The backfill will have an adequate strength to allow for mining adjacent to filled stopes, thus eliminating the need for rib pillars.

The mine design process involved using a minable shape optimization software to determine potentially mineable areas based on an estimated minimum cut-off NSR (CoNSR) value, Nb_2O_5 grades and mining dimensions parameters. As the CoNSR value is much lower than the resulting average stope NSR value, the CoNSR was not the decisive factor in the stope optimization process. Mining dilution of approximately 6% was applied to all stopes and development, based on 3% for the primary stopes, 9% for the secondary stopes, and 5% for ore development. The mining dilution was added to the designed tonnage to account for unplanned sources of dilution such as backfill and host rock around the periphery of the ore mass. An ore recovery factor of 95% was applied to account for unrecoverable ore left within the stopes.

The mine design and schedule were based on a milling constraint (2,764 tpd), provided by NioCorp, to produce approximately 7,000 t/y of ferroniobium and a LOM of over 36 years. Optimization work indicated that the grade of Nb_2O_5, (0.81%) at a unit NSR over US$ 500/t could sustain and produce a consistent ferroniobium production over the LOM. The mill production rate was established at 2,764 t/d from which an annualized production averaging approximately 7,220 t of ferroniobium per year is derived. Scandium trioxide and titanium dioxide accompany the ferroniobium production in the mine plan. Nordmin favoured a higher NSR value in its approach to maximize the LOM NPV for production scheduling while at the same time maintaining the annual ferroniobium requirement.

16.4.2 Stope Optimization

As mentioned in Section 16.4.1, the minable shape optimization software provided by Datamine was used to determine potentially mineable areas based on NSR, Nb_2O_5 grades and mining dimensions parameters. The estimated cut-off NSR value (CoNSR) of US$ 180/t from the SRK study was used as a starting point for this analysis. Generally, stopes would be selected based on the minimum CoG or CoNSR. As the CoNSR value is much lower than the resulting average stope NSR value, the CoNSR was not the decisive factor in the stope optimization process. Rather than using a minimum CoG or CoNSR, the mine design targeted higher annual ferroniobium production during the first five years of ore delivery, which resulted in an averaged annual production rate of 7,351 tonnes per year over this period. The steady-state average annual ferroniobium production was 7,220 tonnes annually. This strategy results in a LOM NSR average value of US$ 538.63/t. Figure 16-14 and Table 16-6 show the stopes optimized for varying CoNSR scenarios. An average dilution of approximately 6% was added to the designed tonnage which accounts for unplanned sources of dilution such as backfill and the host rock around the periphery of the ore mass while a recovery factor was applied to account for unrecoverable material which left within the stopes.



Source: Nordmin, 2019

Figure 16-14: Undiluted Stope Optimization Results for Varying Cut-Off Grades

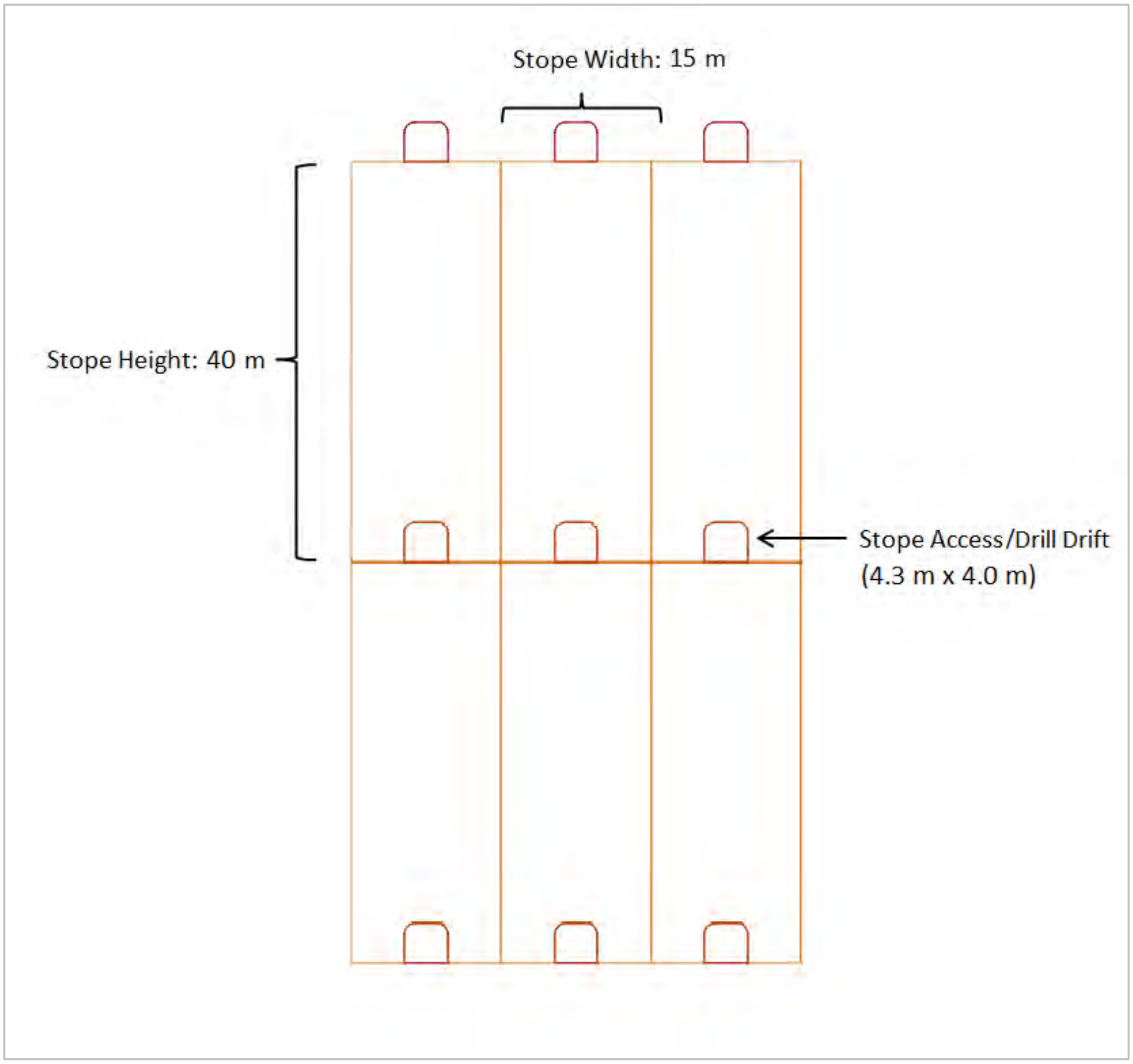
Table 16-6: Undiluted Stope Optimization Results for Varying Cut-off Grades

NSR Cut-off (US$/t)	Tonnes (t)	Nb_2O_5 (%)	Sc (ppm)	TiO_2 (%)	NSR (US$/t)
100	201,552,457	0.519	55.0	2.09	413.7
180	185,082,731	0.545	58.3	2.20	437.8
200	179,820,623	0.553	59.4	2.24	445.0
300	150,707,597	0.594	64.6	2.39	482.7
400	113,434,034	0.645	70.6	2.51	525.8
500	66,791,705	0.718	77.4	2.67	578.8
600	21,756,196	0.843	85.5	2.97	650.5
650	9,758,092	0.919	89.5	3.19	689.8
700	3,281,664	0.972	95.3	3.38	732.8

Source: Nordmin, 2019

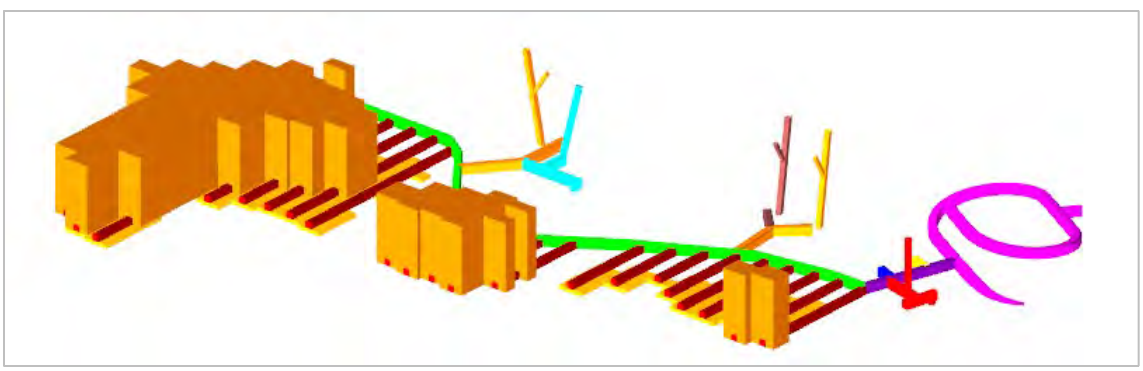
16.4.3 Stope Design

Figure 16-15 shows typical stopes cross section. The stope width is a constant 15 m, and vertical height is 40 m from floor to floor. The length of the stopes is on average 19 m ranging from 10 m to a maximum panel length of 25 m. Figure 16-16 shows a typical level arrangement of the stopes, x-cut, footwall drive, ramp and other infrastructures servicing a level. The mine plan stope orientation is perpendicular to the general strike of the deposit, which is 20° off the measured principal stress. Nordmin does not feel that this offset will have a significant impact on stope stability. The actual mine plan stope lengths have a maximum length of 25 m in both fresh and moderately weathered rock, which is a conservative design in relation to the stability assessment described in Section 16.2, Geotechnical Design Parameters.



Source: Nordmin, 2019

Figure 16-15: Stopes and Cross-Cut Accesses (Cross Section View)



Source: Nordmin, 2019

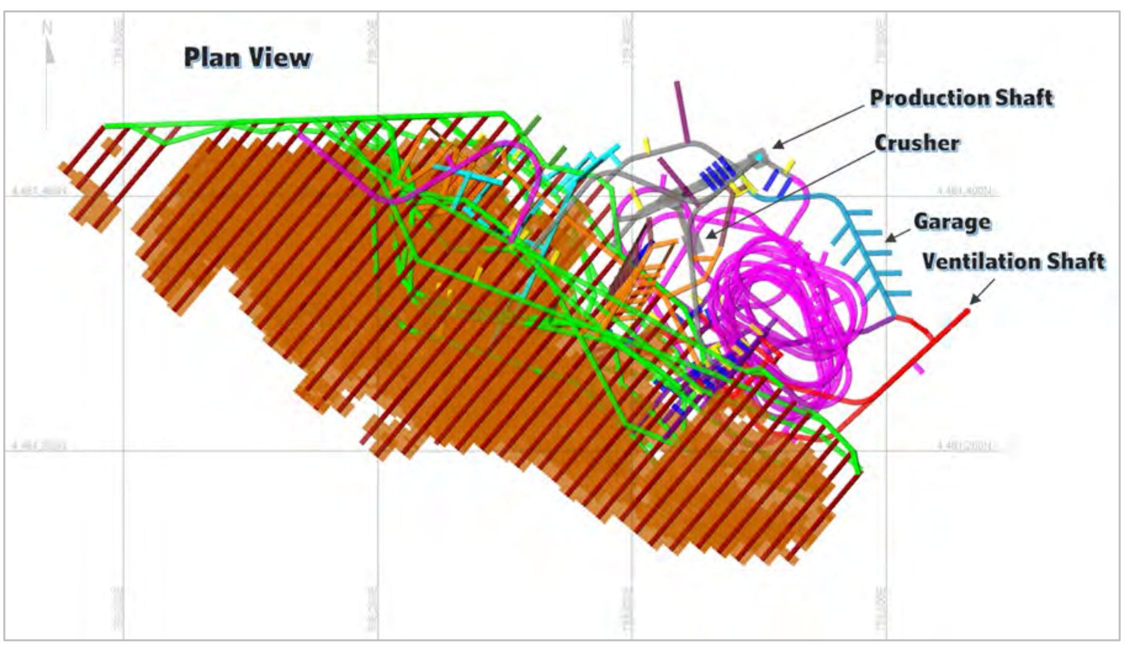
Figure 16-16: Level Layout with Stopes and Footwall Accesses (Rotated View Looking North)

The stoping sequence is based on primary/secondary stopes progressing level by level, where on any given level primary stopes must be separated by a secondary stope. The primary stope is backfilled with cemented paste fill to ensure safe recovery of the adjacent stopes. As the thickness of the ore body is generally more than the maximum length of a single stope, the primary and secondary recovery must also make use of the cemented paste fill to mine stopes from one end (hanging wall) of the ore body across to the other end (footwall).

16.4.4 Development Design

The stopes are accessed through a footwall drive with about 25 m offset from the stopes. The cross-cuts (x-cuts) are driven in the center of the stopes from the footwall drives, as shown in Figure 16-17. These drives are connected by the ramp system, ventilation raises and on some levels they are connected to the production shaft. Most of the mine infrastructure is located in waste, but some areas can be in lower grade material as it gets closer to the ore body.

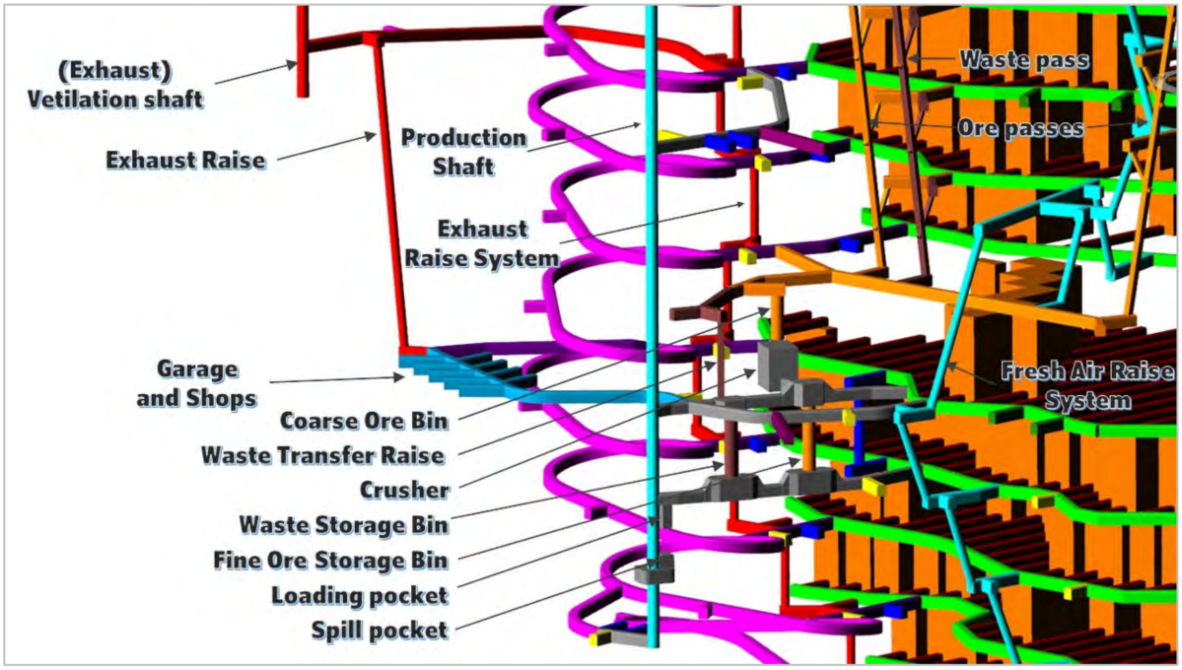
There is a significant increase in the overall depth and vertical extent of the mined ore zone from the previous SRK 2017 feasibility study. The designed vertical extent is 600 m with a bottom elevation of -495 m, versus the previous vertical extent of 450 m with a bottom elevation of -375 m. To efficiently develop the increase in depth and vertical extent, the production shaft and exhaust system (ventilation shaft), were excavated to lower depths. The ventilation shaft is designed to a 530 m depth versus the previous ventilation raise depth of 386 m. The production shaft is designed to a 755 m depth versus the previous depth of 440 m. The deeper production shaft and related crushing and conveying system is complemented with an ore pass and waste pass system that results in an overall material handling system that has suitable ore storage above and below the crusher station, fewer haulage trucks, and fewer ventilation requirements. The decrease in ventilation requirements in turn allowed for a 6.0 m diameter production shaft versus the 7.5 m diameter production shaft in the previous SRK 2017 feasibility study.



Source: Nordmin, 2019

Figure 16-17: Completed Mine Design (Plan View)

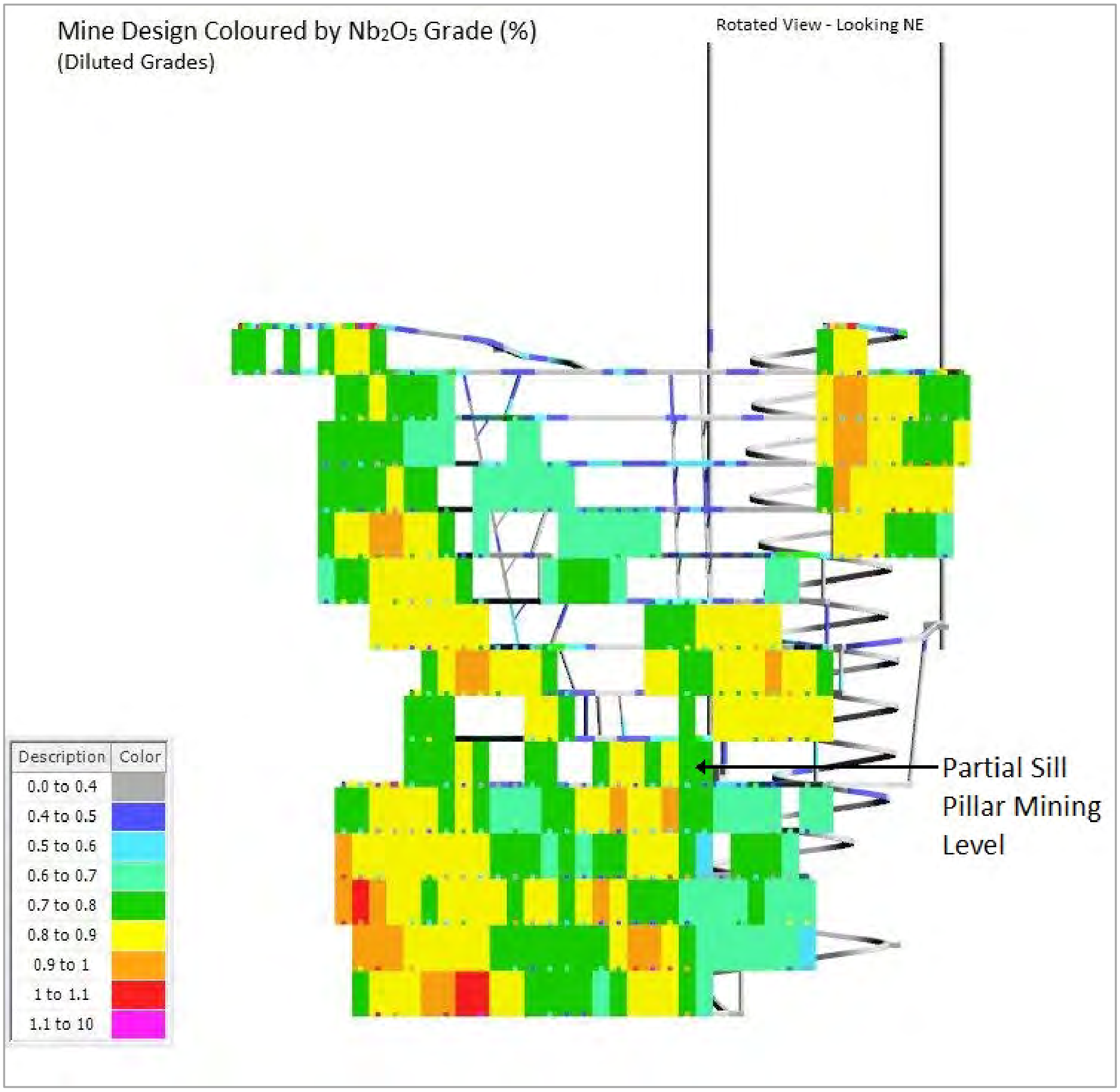
Figure 16-18 shows the completed mine design main infrastructure area. The shafts, internal raises, and underground infrastructure included in the design are discussed in other subsections. The two mining horizons are generally mined simultaneously. Altogether, they provide an estimated LOM of 36 years.



Source: Nordmin, 2019

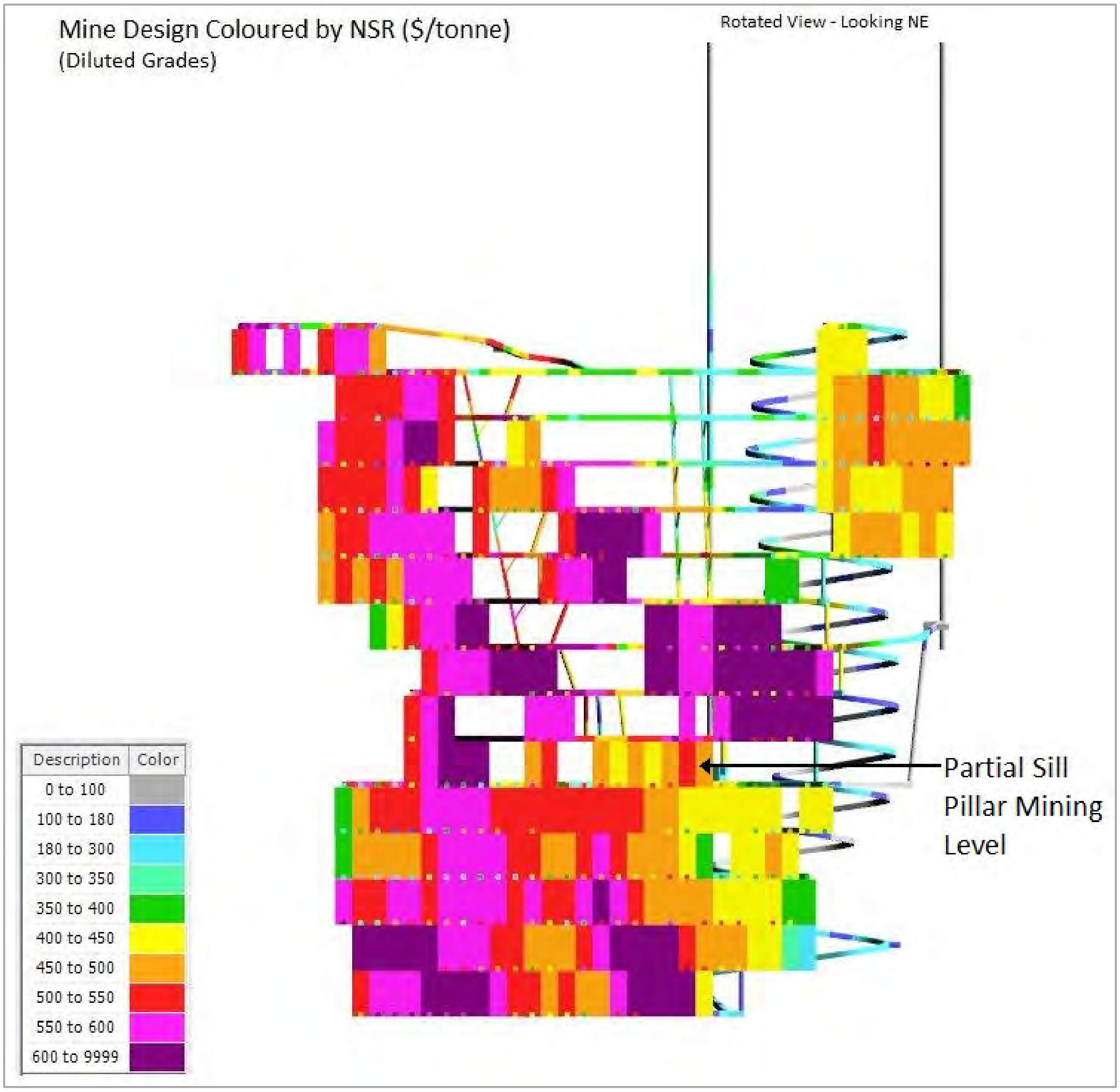
Figure 16-18: Completed Mine Design - Main Infrastructure (Looking South)

Figure 16-19 and Figure 16-20 show the mine design coloured by Nb_2O_5 grade and NSR, respectively.



Source: Nordmin, 2019

Figure 16-19: Mine Design Coloured by Nb_2o_5 Grade.



Source: Nordmin, 2019

Figure 16-20: Mine Design Coloured by NSR

Table 16-7 summarizes the mine design by activity type.

Table 16-7: Mine Design Summary - by Activity Type

General Summary	LOM Statistics
Ore Tonnes (t)	36,312,808
FeNb (t)	259,786
Nb_2O_5 Grade (%)	0.808
Sc Grade (ppm)	65.7
TiO_2 Grade (%)	2.86
Development Ore Tonnes (t)	926,932
Stope Production Tonnes (t)	35,385,875
Waste Tonnes (t)	2,944,407
Total Tonnes Moved (t)	39,257,214
Lateral Development:	
Main Ramp - 5.0 x 5.3 (m)	4,777
Ramp Access to Level - 4.5 x 5.3 (m)	1,187
Shaft Access to Level (m)	834
Footwall Access - 4.5 x 5.3 (m)	7,787
Fresh Air Raise Access - 4.5 x 5.3 (m)	720
Return Air Raise Access - 4.5 x 5.3 (m)	1,075
Ore Pass Access - 4.5 x 5.3 (m)	1,225
Waste Pass Access - 4.5 x 5.3 (m)	265
Stope Access Drift - 4.3 x 4.0 (m)	43,000
Other Lateral Development - Shop, Crusher, Sumps, etc. (m)	1,630
Total Lateral Development (m)	62,500
Vertical Development:	
Production Shaft - 6.0 m Finished Diameter (m)	755
Ventilation Shaft - 6.0 m Finished Diameter (m)	530
Fresh Air Raise - 3.3 m Diameter (m)	684
Return Air Raise - 3.3 m Diameter (m)	702
Ore Pass - 3.0 x 3.0 (m)	648
Ore Pass Fingers - 2.0 x 2.0 (m)	273
Waste Pass - 3.0 x 3.0 (m)	359
Waste Pass Fingers - 2.0 x 2.0 (m)	147
Other Vertical Development - Bins, Conical Sump (m)	115
Total Vertical Development (m)	4,212

Source: Nordmin, 2019

16.4.5 Mine Access

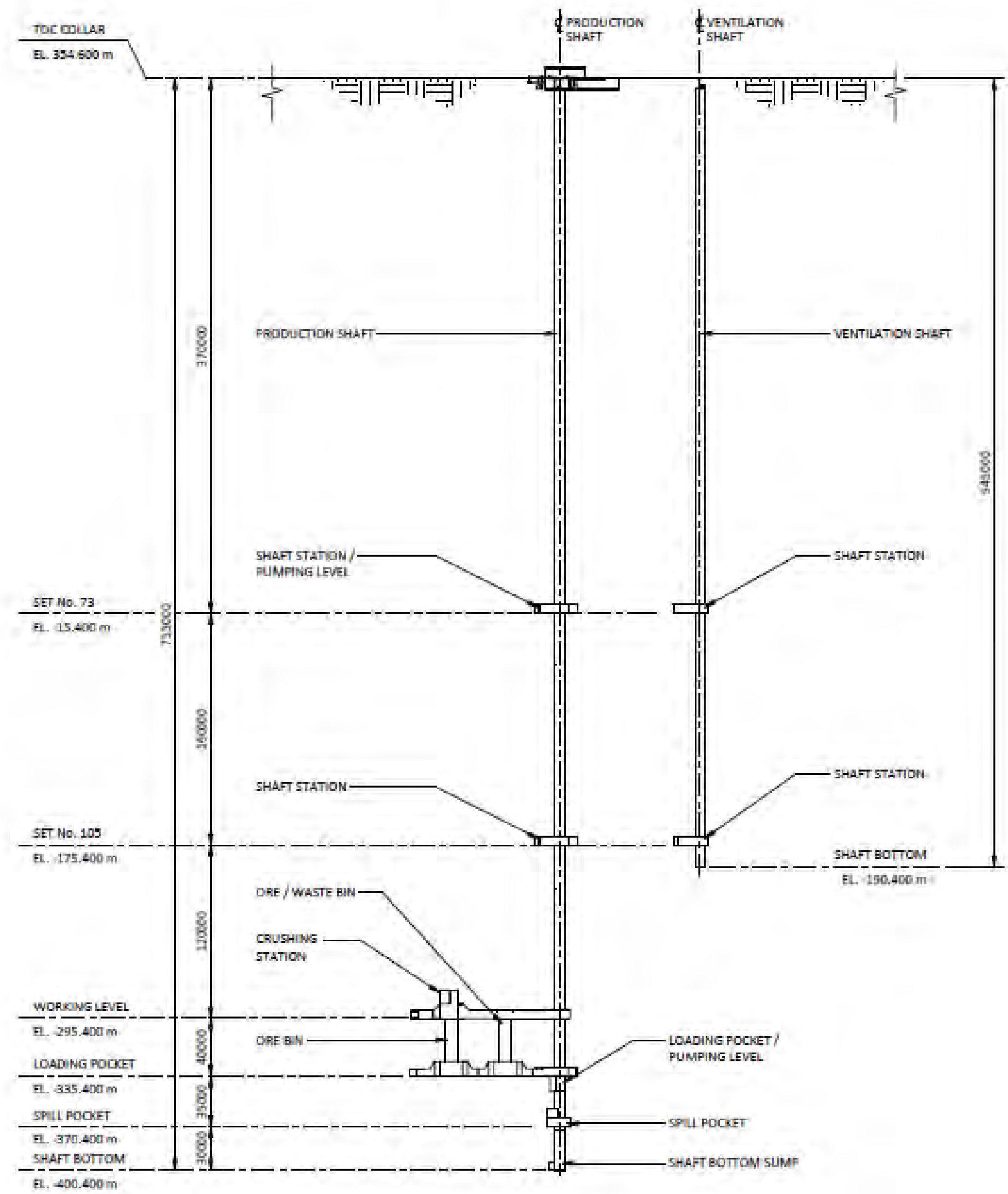
The underground mine will be accessed from the surface at a collar elevation of 354.6 m (1,163.4 ft) ASL, via twin 6.0 m (19’-8”) diameter concrete lined shafts, named the “production shaft” and the “ventilation shaft” (see Figure 16-21). Coordinates for the shafts are N4461430.850 m, E739499.590 m for the production shaft and N4461310.000, E739663.000 m for the ventilation shaft. The shafts are excavated by means of conventional shaft sinking and will be combined with freezing down to the potential water-bearing contact between the Pennsylvanian sediments and carbonatite unit, (reference Figure 7-5). This method, unlike a raisebore method of excavation, allows control of potential water inflows.

The production shaft will facilitate the movement of larger mining equipment, workforce, services, material hoisting, and act as the supply route for the mine ventilation system. The production shaft is excavated to a lower elevation than in the previous 2017 SRK feasibility studies. This allows earlier access to higher grade ore in the central portion of the mine and to also access higher grade ore in the lower mining block with a more efficient material handling system.

The ventilation shaft will be dedicated to moving workforce and smaller material, hoisting for initial lateral development, as well as act as an exhaust route for the mine ventilation system. A second temporary hoist, hoist room, and headframe is installed for the ventilation shaft sinking process and will be utilized to hoist waste from initial lateral mine development prior to the completion and installation of the permanent hoisting arrangement in the production shaft.

Main access to the lower working levels will be gained by means of the production shaft which records a shaft bottom elevation of -400.4 m (-1313.65 ft), with stations at the -15.4 m (-50.5 ft), -175.4 m (-575.5 ft), -295.4 m (-969 ft), and -335.4 m (-1,100.5 ft), and access to the spill pocket at -370.4 m (-1,215 ft).

Stations and underground development on the -15.4 m (-50.5 ft), -175.4 m (-575.5 ft) levels, allow for easy access between the production shaft and the ventilation shaft.



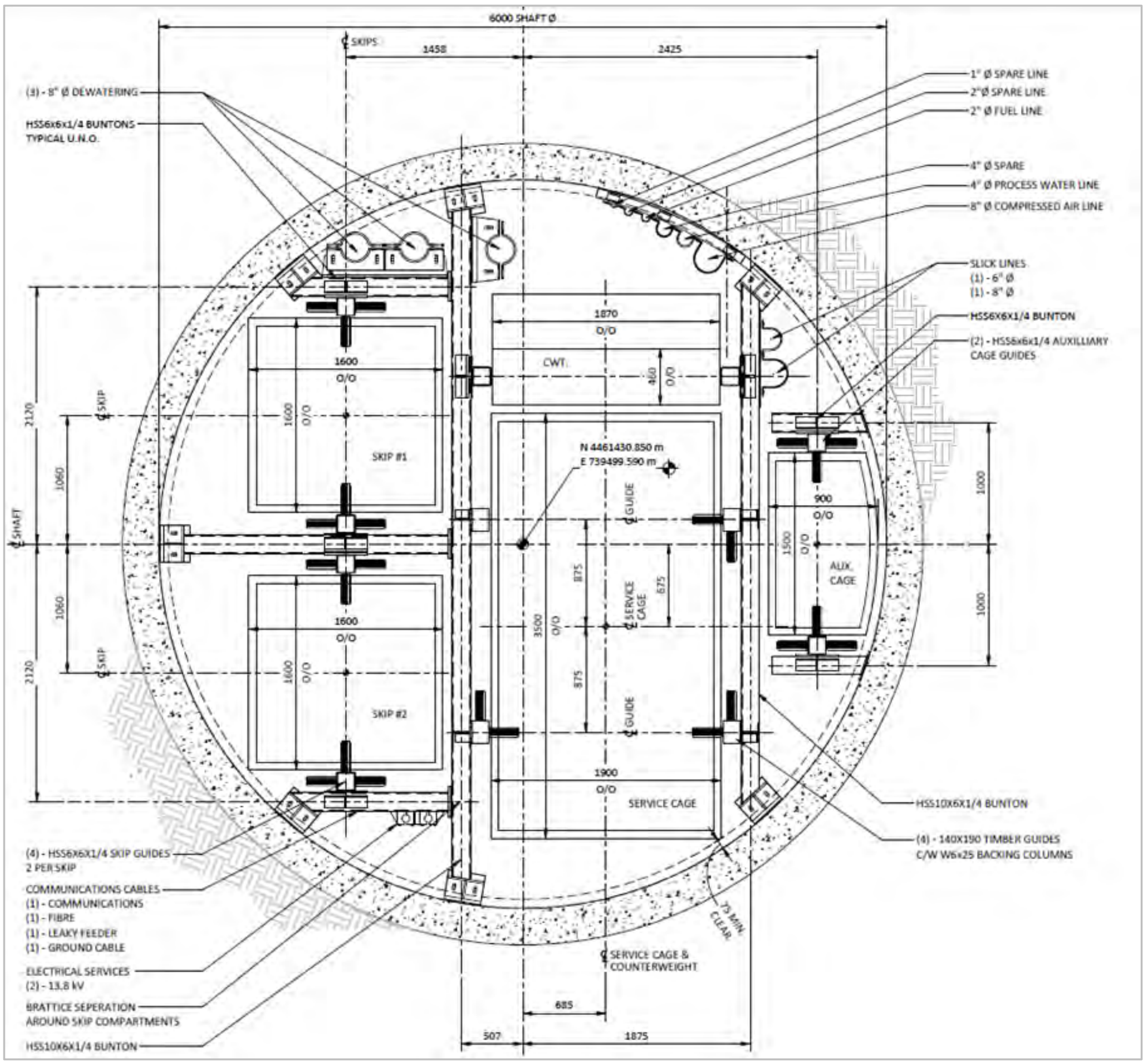
Source: Nordmin, 2019

Figure 16-21: Underground Mine Access Via Twin Concrete Lined Shafts

16.4.5.1 Shaft Layouts

Access to the underground mine is via either the 6.0 m diameter concrete lined production shaft or the 6.0 m diameter concrete lined ventilation shaft. Atop the production shaft lies a 71.5 m (235 ft) tall headframe, with three sheave decks for five rope sheaves. The production shaft will host the two production skips, the main service cage and counterweight, and auxiliary cage as well as house all services to the underground including:

- 8.0” diameter dewatering lines, a quantity of three;
- 8.0” compressed air line;
- 8.0” and 6.0” slick lines;
- 4.0” process water line;
- 2.0” fuel line;
- 4.0”, 2.0” and 1.0” spare cables;
- 13.8 kV power lines; and
- communication, fibre, leaky feeder and ground cables (see Figure 16-22).



Source: Nordmin, 2019

Figure 16-22: Production Shaft Layout

Services within the production shaft are located for ease of access for required inspections from the skips, service cage, and the auxiliary cage. The production shaft will be partitioned into two main areas with the use of buntons and brattice. The twin skips running on steel guides will be separated from the rest of the shaft by brattice panels throughout the length of the shaft. The other section of the shaft will house the auxiliary cage, the service cage and the service cage counterweight. A 5 m shaft set spacing is used in both the ventilation shaft and the production shaft.

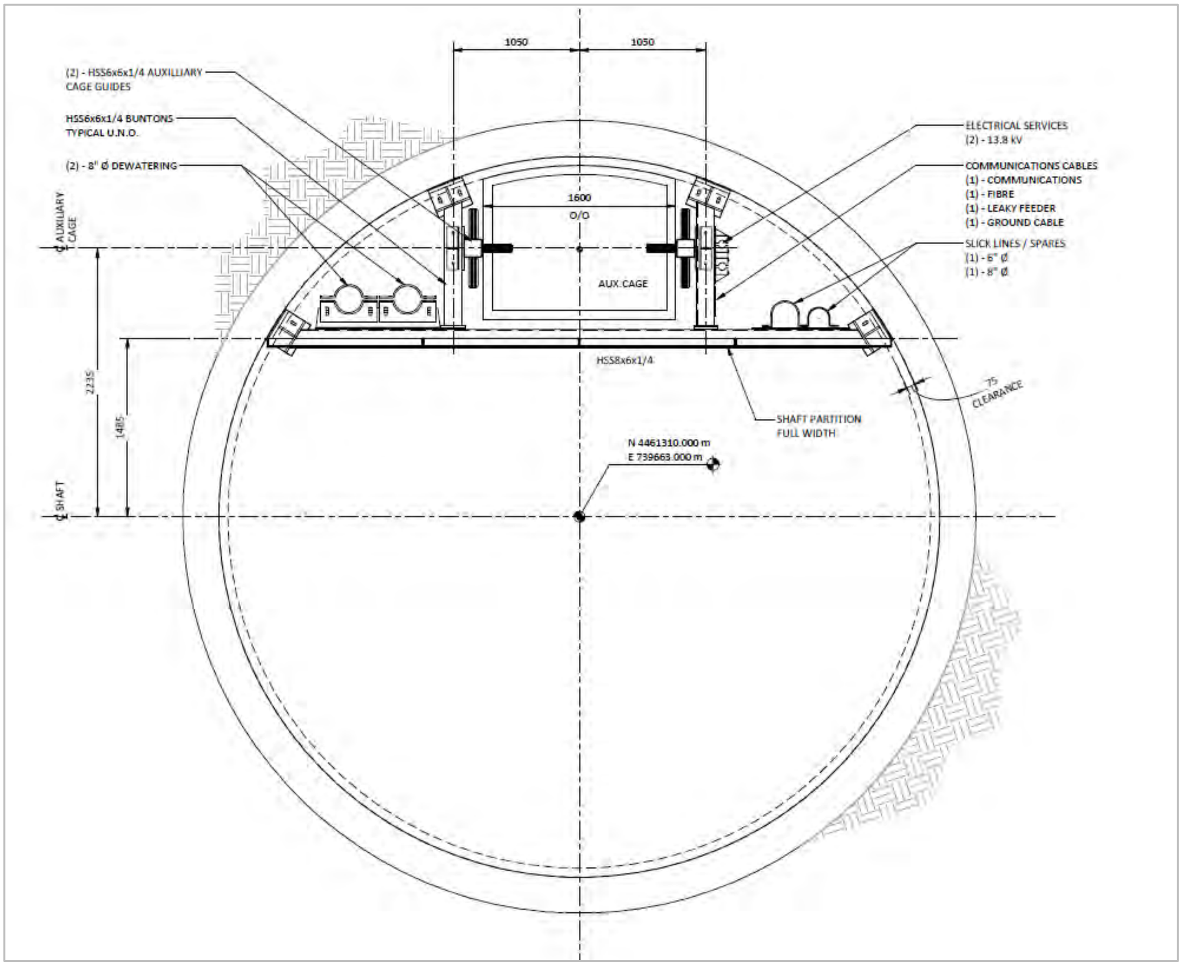
Access to the shaft will be from within the headframe. Additional safety gates and hydraulically actuated collar doors will reside atop the shaft at collar elevation.

The 6.0 m diameter concrete lined ventilation shaft will be used to host a secondary auxiliary cage, identical to the production shaft auxiliary cage and minimal services, including:

- 13.8 kV power cables;
- communication, fibre, leaky feeder and ground cables; and
- 8.0" and 6.0" slick lines.

Similar to the production shaft headframe, the ventilation shaft headframe will house a single sheave deck, a set of collar doors and safety gates.

The ventilation shaft is partitioned into two main areas, one dedicated for the auxiliary cage, and one dedicated for the ventilation system. Both sections will be segregated by the use of brattice panels from the lower depths of the shaft to the top of the ventilation sweep for the exhaust system. This will guarantee an unobstructed ventilation pathway for the mine air exhaust system (see Figure 16-23).



Source: Nordmin, 2019

Figure 16-23: Ventilation Shaft Layout

16.5 Production Schedule

The production schedule is based on the mine design and reserves discussed in previous sections.

16.5.1 Productivity

Productivities were developed from first principles. Input from mining contractors, blasting suppliers and equipment vendors, were used for the key parameters. The rates developed from first principles were adjusted based on benchmarking and the experience and judgment of Nordmin's team.

The productivity rates used for mine scheduling are shown in Table 16-8, followed by a description of the general and activity-specific parameters upon which the productivity rates are based.

Table 16-8: Productivity Rates

Activity	Type	Dimensions	Rate
Lateral Development	Priority Face	See Table 16-9	5.0 m/d
	Non-Priority Face		3.0 m/d
	Shaft Station	Varies	2.0 m/d
Vertical Development	Production Shaft	6.0 m diameter	2.3 m/d
	Ventilation Shaft	6.0 m diameter	2.3 m/d
	Fresh Air Raise	3.3 m diameter	3.6 m/d
	Return Air Raise	3.3 m diameter	3.6 m/d
	Ore Pass	3 m x 3 m	3.6 m/d
	Ore Pass Fingers	2 m x 2 m	3.6 m/d
	Waste Pass	3 m x 3 m	3.6 m/d
	Waste Pass Fingers	2 m x 2 m	3.6 m/d
	Bins and Conical Sump	Varies	1.3 m/d
Individual Stoping	Slot Development	-	10.5 d
	Drilling	-	960 m/d
	Stope Production	-	930 t/d
	Backfill Preparation	-	10.0 d
	Backfilling	-	1200 m^3/d
	Backfill Curing	-	28.0 d

Source: Nordmin, 2019

Typical dimensions by heading types are presented in Table 16-9. These will all be developed by contractors in accordance with the productivity rates and levelled in the schedule by crew assignments. The main development headings were slightly decreased in dimension size from the previous SRK 2017 feasibility study. The reasoning for the smaller dimensions was to decrease development costs.

Table 16-9: Dimensions by Heading Types

Heading Types	Width (m)	Height	Area (m^2)
Ramp	5.0	5.3	27
Elect Sub	4.5	5.3	24
FAR Access	4.5	5.3	24
FW Drift	4.5	5.3	24
Ore/Waste Access	4.5	5.3	24
X-Cuts	4.3	4.0	17
Remucks	4.5	5.0	23
Sump	4.5	5.3	24
Refuge Station	4.5	5.3	24

Source: Nordmin, 2019

General Parameters

Table 16-10 provides the general schedule parameters applicable to all underground mining activities.

Table 16-10: Workforce Schedule Parameters for Underground Mining

Schedule Parameters	Units	Value
Annual Mining Days	days/year	365
Mining Days per Week	days/week	7
Shifts per Day	shifts/day	2
Scheduled Shift Length	hrs/shift	12
Scheduled Deductions:		
- Travel Time Between Underground and Surface	hrs/shift	1.00
- Workplace Examinations and Equipment Pre-shift Inspections	hrs/shift	0.25
- Lunch	hrs/shift	0.50
- Breaks	hrs/shift	0.50
Total Scheduled Deductions	hrs/shift	2.25
Operating Time (Scheduled Shift Length Less Scheduled Deductions)	hrs/shift	9.75
Effective Time (Operating Time Reduced to a 50 Minute Hour, i.e., Multiplied by 83.3%)	hrs/shift	8.125

Source: Nordmin, 2019

Table 16-11 provides the ground support requirements.

Table 16-11: Ground Support Requirements

Geotechnical Zone	Q*	Excavation	Support Categories	Bolt Length	Bolt Spacing	Other Support
Footwall, highly weathered (6%)	0.4 to 6.2 (Very Poor)	Main Ramp	3-Bolts, mesh and shotcrete	2.5 m	1.2 m	Fully grouted rebar, mesh, 5 cm shotcrete
		FW Access	2-Systematic bolting	2.5 m	1.2 m	Split sets and mesh
		Stope Access	2-Systematic bolting	2.5 m	1.6 m	Split sets and mesh
Footwall, moderately weathered (24%)	3.2 to 13.8 (Poor-Fair)	Main Ramp	2-Systematic bolting	2.5 m	1.2 m	Fully grouted rebar, mesh
		FW Access	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Split sets
		Stope Access	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Split sets
Footwall, slightly weathered (70%)	5.9 to 28.1 (Fair-Good)	Main Ramp	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Grouted rebar
		FW Access	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Split sets
		Stope Access	1-Spot bolting (15/10 m)	2.5 m	1.6 m	Split sets

*(%) Amount of Expected Ground
Source: SRK, 2017

The mine plan developed by Nordmin has conservatively allowed for grouted rebar in the back, (roof) of all excavations. Split sets were acceptable in walls of excavations, but not in the back.

16.5.2 Shaft Sinking – Production Shaft and Ventilation Shaft

Shaft sinking operations at both shafts will be carried out simultaneously. This allows the initial lateral development to begin from the bottom of the ventilation shaft while the production shaft continues to be excavated to a lower elevation to facilitate extraction of higher niobium grade stopes located at the lower levels.

The 6.0 m (inside diameter) Production Shaft is excavated to a depth of 755 m. The shaft is excavated using conventional shaft sinking methods in conjunction with a freezing process through the first 200 m from the surface to ensure ground and water control. Upon completion of the first 200 m section, the shaft sinking continues, but freezing is no longer required to reach the bottom elevation. The rate of excavation averages 2.30 m/d; this rate was developed in collaboration with contractors for the material expected to be encountered. The average rate includes sinking, lining, furnishing and adjustments in rate due to rock types and shaft depth. The material is removed by a temporary shaft sinking hoist, hoist room, and headframe system and placed in the lined temporary stockpile location adjacent to the shaft.

The Ventilation Shaft is excavated with the same diameter and method as the production shaft, but only to a depth of 530 m. Conventional shaft sinking is combined with freezing down to the potential water-bearing contact between the Pennsylvanian sediments and carbonatite unit (reference Figure 7-5). This method, unlike a raisebore method of excavation, allows control of potential water inflows. A second temporary hoist, hoist room, and headframe is installed for the sinking process and will be utilized to hoist waste from lateral mine development prior to the completion and installation of the permanent hoisting arrangement in the production shaft.

16.5.3 Development and Production Schedule

The production and development schedules were completed using the Deswik scheduling software. The production schedule is based on the rate assumptions shown in Table 16-12.

A delay of 28 days was used before driving on paste fill or mining adjacent to a paste filled stope. These delays account for curing time as well as multiple pours.

The mining operation schedule is based on 365 days/year, 7 days/week, with two 12 hour shifts each day. A production rate of 2,764 t/d was targeted with a ramp-up to full production as quickly as possible. The schedule timeframe is monthly for the pre-production period, two years for production, then quarterly for three years, and annually for the remainder of the LOM.

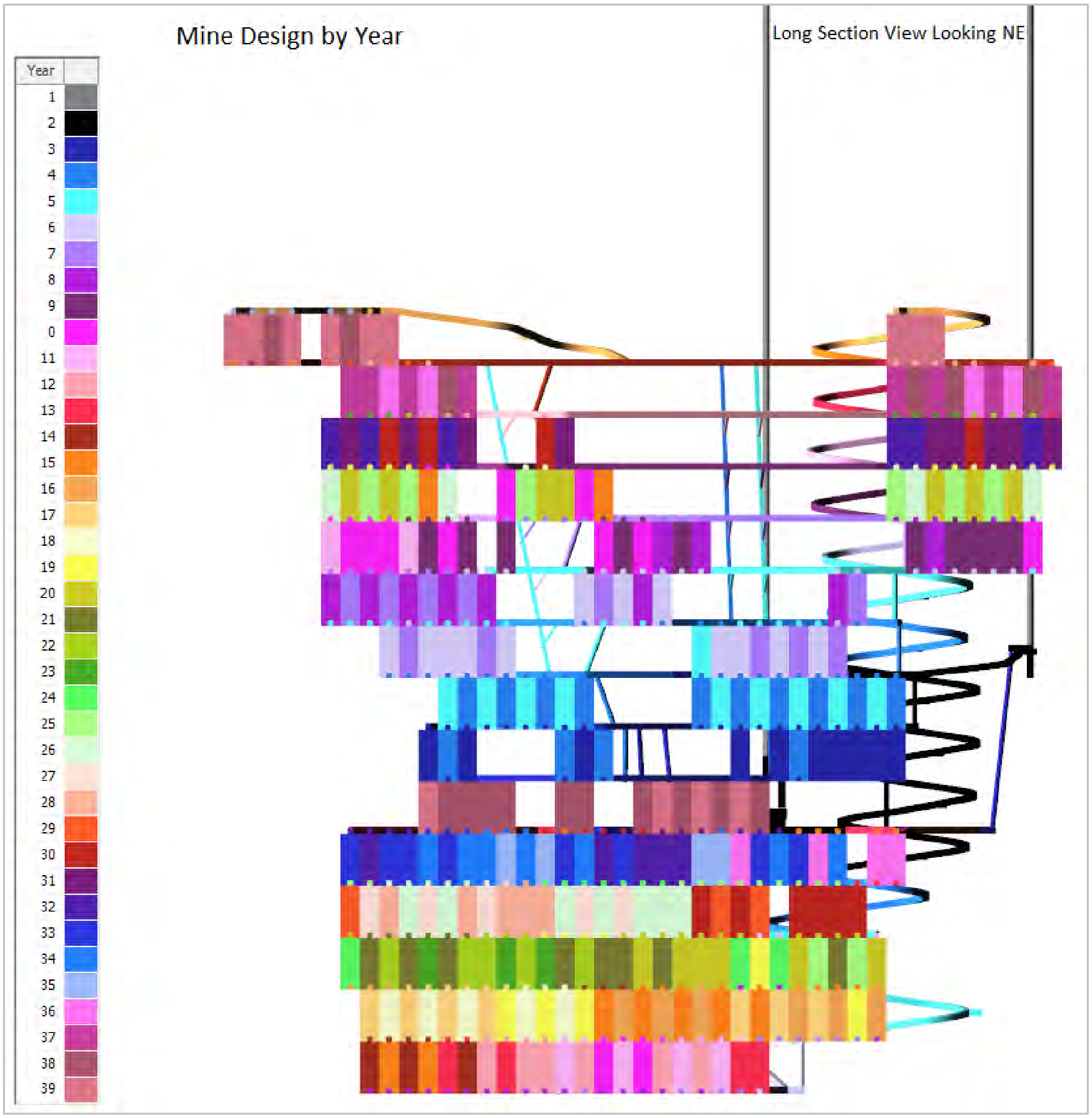
Production shaft and ventilation shaft sinking preparation begins eight months after the commencement of detailed engineering with the actual sinking beginning five months later and subsequent lateral mine development beginning nine months later. Production stoping begins sixteen months after the start of lateral development, with a production ramp-up period through the next six months, after which the mine and plant are operating at full capacity.

Table 16-12 shows the annual mine production schedule, and Figure 16-24 shows the mine production schedule coloured by year.

Table 16-12: Mine Production Schedule

Year	Ore Tonnes (t)	Nb_2O_5 (%)	Sc (ppm)	TiO_2 (%)	Waste Tonnes (t)	Backfill Volume (m^3)
Year 0	-	-	-	-	-	-
Year 1	-	-	-	-	80,213	-
Year 2	12,014	0.588	64.8	2.20	303,899	-
Year 3	478,065	0.818	78.1	2.99	222,341	96,053
Year 4	990,692	0.855	77.6	3.10	251,524	327,771
Year 5	970,287	0.878	75.1	3.08	277,679	299,534
Year 6	1,011,500	0.842	71.5	2.97	242,475	331,190
Year 7	991,845	0.816	67.3	2.95	212,947	301,494
Year 8	982,017	0.824	68.5	2.93	175,284	318,544
Year 9	1,013,848	0.797	63.1	2.96	187,332	324,575
Year 10	1,034,227	0.753	65.5	2.82	72,866	326,597
Year 11	1,000,792	0.816	69.8	2.87	101,196	333,041
Year 12	1,018,813	0.798	71.5	2.99	111,533	320,314
Year 13	1,008,922	0.803	69.3	2.95	116,389	341,954
Year 14	925,636	0.872	67.0	3.06	113,769	318,853
Year 15	1,047,234	0.784	68.2	2.61	94,574	304,254
Year 16	980,466	0.826	73.4	2.82	80,622	341,682
Year 17	972,774	0.843	75.1	3.03	10,365	314,195
Year 18	916,797	0.876	74.3	3.14	17,154	308,961
Year 19	1,000,501	0.809	69.6	2.93	45,348	292,997
Year 20	1,046,208	0.729	60.3	2.63	18,951	331,114
Year 21	1,017,067	0.804	62.3	2.75	6,420	340,583
Year 22	958,165	0.853	67.0	2.86	15,089	313,450
Year 23	1,023,413	0.798	65.7	2.76	12,118	327,255
Year 24	1,021,338	0.797	68.7	2.90	47,026	322,492
Year 25	1,057,957	0.766	54.7	2.79	19,518	340,791
Year 26	1,044,664	0.771	53.3	2.60	15,114	356,097
Year 27	1,008,196	0.807	60.8	2.78	10,662	351,698
Year 28	1,005,226	0.808	62.4	2.74	8,017	303,401
Year 29	994,018	0.829	62.0	2.97	4,308	329,079
Year 30	1,020,916	0.796	59.2	2.84	20,033	346,102
Year 31	1,018,164	0.794	59.3	2.80	12,843	337,648
Year 32	1,033,847	0.759	61.9	2.61	12,064	322,923
Year 33	989,450	0.841	61.4	2.98	8,668	337,242
Year 34	1,000,738	0.812	61.5	2.98	13,282	328,691
Year 35	1,030,705	0.797	64.2	2.81	2,783	336,457
Year 36	1,046,559	0.788	62.5	2.77	-	333,297
Year 37	1,022,677	0.786	60.4	2.75	-	359,523
Year 38	1,046,217	0.785	66.5	2.76	-	406,353
Year 39	570,853	0.791	61.1	2.72	-	266,603
Totals	**36,312,808**	**0.808**	**65.7**	**2.86**	**2,944,407**	**11,892,808**

Source: Nordmin, 2019



Source: Nordmin, 2019

Figure 16-24: Mine Production Schedule - Coloured By Year

16.6 Mining Operations

16.6.1 Production

The ore feed to the plant primarily comes from the stope production as the development contributes to less than 3% of the total ore. Stopes are mined using the longhole open stoping method. Individual stope blocks are designed to be 15 m wide, up to 25 m long oriented roughly parallel to the main stress. Levels are spaced 40 m apart, and each stope block has top and bottom access called the crosscut (x-cut: 4.3 m x 4 m flat back drifts).

Stopes are drilled downward from the top access using 114 mm (4.5 in) diameter holes. Initial opening is done using stope slots drilled with a slot reamer machine and blast holes. A level by level bottom up, primary/secondary extraction sequence is followed. Primary stopes are backfilled with high strength cemented paste backfill. Secondary stopes are backfilled with high strength cemented paste backfill when more than one panel needs to be mined adjacent to one another. When development waste rock is not available for backfilling secondary stopes, low strength paste backfill is utilized as needed.

All blasting is performed with bulk emulsion. The slot is expanded first followed by one or two mass blasts to complete the stope.

Ore is mucked from the lower x-cut access using a 6.2 m^3 (14 t) LHD with remote control capability. The ore is transported by the LHD to either an ore pass directly or to a remuck bay to maximize the efficiency of the stope mucking operations as a function of the haulage distance. When needed, a second LHD and a fleet of 40-tonne haul trucks are used to transport ore from the remuck bays to the grizzly feeding the underground material handling system. Multiple remuck bays are used on each level to avoid interference between the LHD and the haul trucks.

16.6.2 Development

Lateral development includes interlevel ramps, level accesses, stope accesses, and short connecting drifts for ventilation. The designed development headings were slightly decreased in dimension size from the previous SRK 2017 feasibility study. The reasoning for the smaller dimensions was to decrease development costs. The interlevel ramp system is 5 m wide by 5.3 m high at a maximum 15% gradient. Level accesses is 4.5 m wide by 5.3 m high and is mined higher at the remuck bays to allow the haul trucks to be loaded by the LHD. Stope access drifts 4.3 m wide by 4 m high. Stope accesses are oriented perpendicular to the strike of the orebody.

The lateral development is sized for the operation of the mining equipment fleet selected for the operation. The development profiles include allowances for ventilation ducting and services.

Raiseboring is used to establish ventilation connections between level access drifts.

<u>Truck Haulage</u>

The mine plan assumes that 6.2 m^3 (14 t) LHDs load the 40-tonne haul trucks from remuck bays that are strategically located throughout the development workings. Ore and waste haulage distances and cycle times were calculated using a haulage model built in Microsoft Excel® and are based on estimated underground truck speeds, as shown in Table 16-13. The outputs from the haulage profile module are a one-way haulage distance and an average truck cycle time (round trip). The truck haulage demand/usage was decreased from the previous SRK 2017 feasibility study by installing an ore and waste pass system in the overall material handling system.

Table 16-13:Truck Hauling Speeds

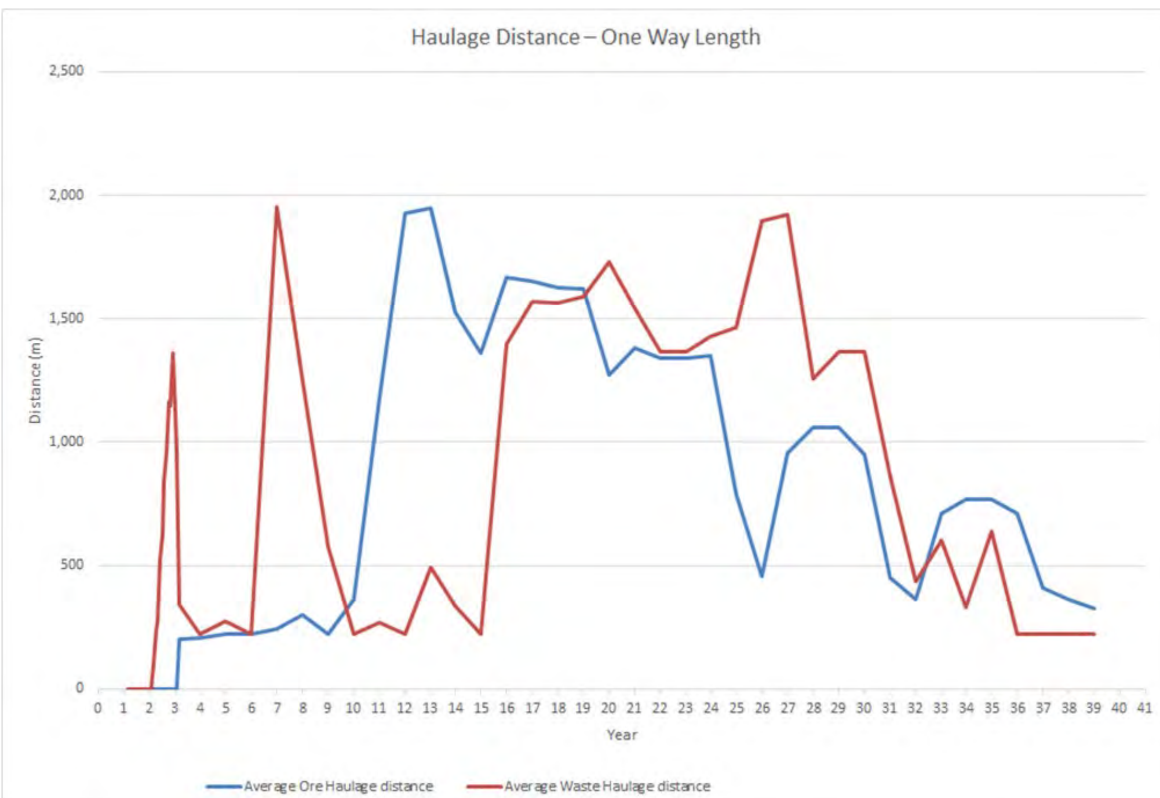
	Road Grade (%)	Speed (km/h)
Loaded	0%	8.0
	-15%	6.0
Empty	0%	8.0
	15%	10.0

Source: Nordmin, 2019

The ore haulage distances were evaluated from the mine design. Based on this evaluation, ore haulage routes were measured according to the distance from the truck loading area to the truck dump, per level. Microsoft Excel® was then used to generate a one-way ore haulage distance and an average cycle time (round trip) using the speed parameters shown in Table 16-13.

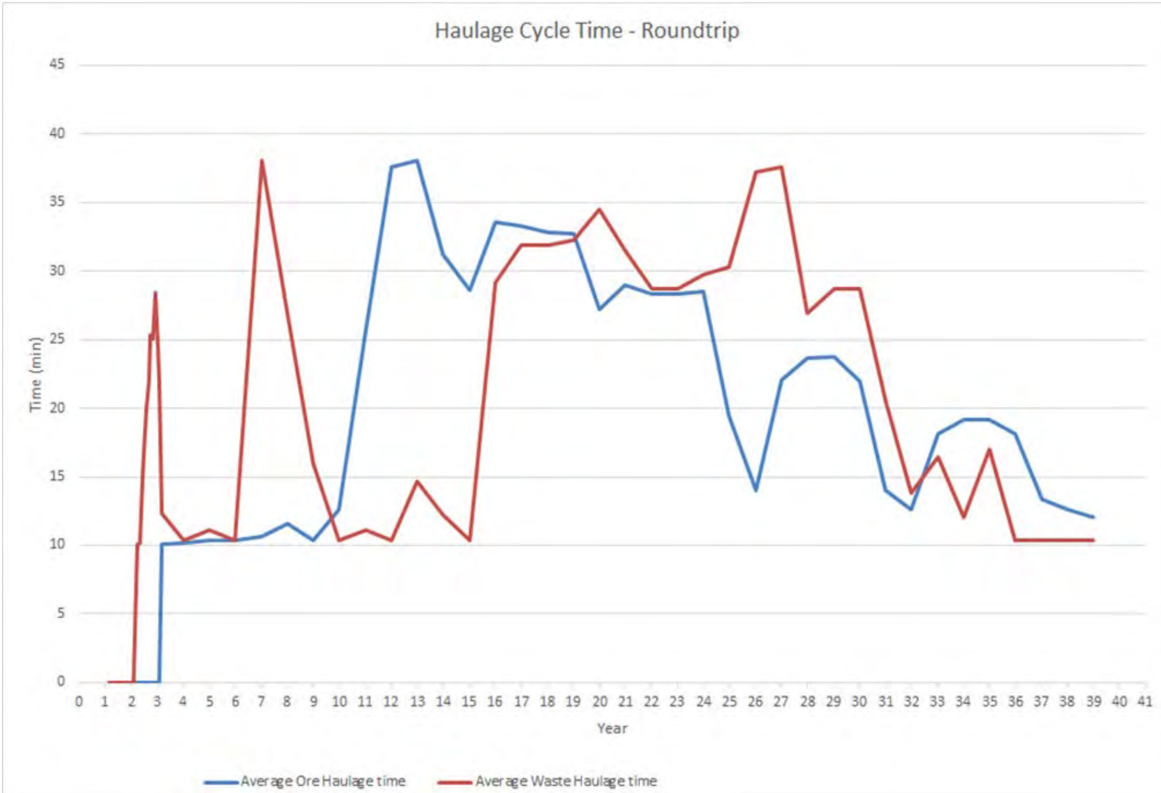
The availability of mined-out secondary stopes was evaluated to determine the quantity of low strength paste backfill placed underground during each time period. Based on this evaluation, waste haulage routes were created to approximate the location of development waste mining and waste rock dumping for each time period. A Microsoft Excel® haulage profile was then used to generate a one-way waste haulage distance and an average cycle time (round trip) using the speed parameters shown in Table 16-13.

The average one-way ore haulage distances are approximately 240 m early in the LOM through the first eight years and increase to approximately 1,275 m from years 9-16 and 945 m for the remainder of the LOM; the LOM average is 940 m. Waste haulage distances vary considerably depending on the time period. At the peak, five haul trucks are required to transport the ore and waste. Figure 16-25 and Figure 16-26 show the haulage distance and cycle time by monthly and yearly time periods. Nordmin notes the cycle times reflected in this summary are indicative as there is a fixed component including the loading time, dumping time, positioning time, and additional delays that are included in the productivity and equipment quantity determinations and not included in the information summarized in these figures.



Source: Nordmin, 2019

Figure 16-25: Haulage Distance — One-Way Length



Source: Nordmin, 2019

Figure 16-26: Haulage Cycle Time – Roundtrip

During the pre-production period, before mining of stopes and the commissioning of the plant, waste and mineralized material is hoisted to the surface and stored separately in a designated lined storage facility. During the pre-production period, the mine produces approximately 467,000 tonnes of waste and 42,000 tonnes of mineralized material. The mineralized material is fed into the processing plant during commissioning.

16.6.3 Backfilling

The mine production sequence includes the use of cemented paste backfill to fill the voids left by the stopes to maintain the mine structural integrity. The mine utilizes a high strength backfill paste that has a 5% cement content in the primary stopes. For secondary stopes, lower strength paste with 2% cement is used to supplement development waste rock, whenever development waste rock is not available to backfill stopes.

Section 18.14 discusses the surface plant and system to move the pastefill underground to the stopes. A backfill operations crew installs barricades in the lower access drift to the stopes, extends the pipe delivery system from the production shaft via the upper access drift into the stopes, and monitors the backfill as the stope fills. Once the stope is filled the backfill is allowed to cure (28-days) to design strength of over 1 MPa before blasting on the adjoining stope.

Table 16-14 provides the LOM backfill breakdown by volume and type.

Table 16-14: Backfill Volume Summary - By Type

Backfill Type	Volume (m^3)
High Strength Backfill (5% Cement Pastefill)	5,137,125
Low Strength Backfill (Waste Rock, Cemented Rock Fill, or 2% Cement Pastefill)	6,755,683
Total Backfill	**11,892,808**

Source: Nordmin, 2019

16.6.4 Ground Support

The lateral development assumptions include systematic bolting with screen for all areas. Some specific excavations include shotcrete either because they have been identified as weaker ground conditions or for long term stability requirements. As there are faults and discrete structures that may impact stability locally, the unit cost of the various type of development has been increased to take into account that a portion of the development will require shotcrete locally. Table 16-15 shows the estimated amount of the total development that is projected to require extra support, whether it is shotcrete or cable bolting. The extra support accounts for intersections with wider span or geological structures that would require special attention. It is assumed that all the stope brow will need extra support, which attributes for 25% of the x-cuts requiring extra support.

Table 16-15: Extra Support Assumptions by Heading Type

Heading Type	% Extra Support
Ramp	15%
Electric Substation	100%
FAR Access	20%
FW Drift	15%
Ore/Waste Pass Access	100%
X-Cuts	25%
Remucks	10%
Sump	100%
Refuge Station	100%

Source: Nordmin, 2019

16.6.5 Grade Control and Reconciliation

The objective of an underground grade control program, as part of a routine mining sequence, is to maximize the value of ore mined and fed to the surface plant. The grade control (or ore control) process involves the predictive delineation of the tonnes and grade of ore that will be recovered by mining. The predictions have several common characteristics across all mineralization and mining types, for instance, from small, low production rate, metalliferous underground mines to large world-class open pits.

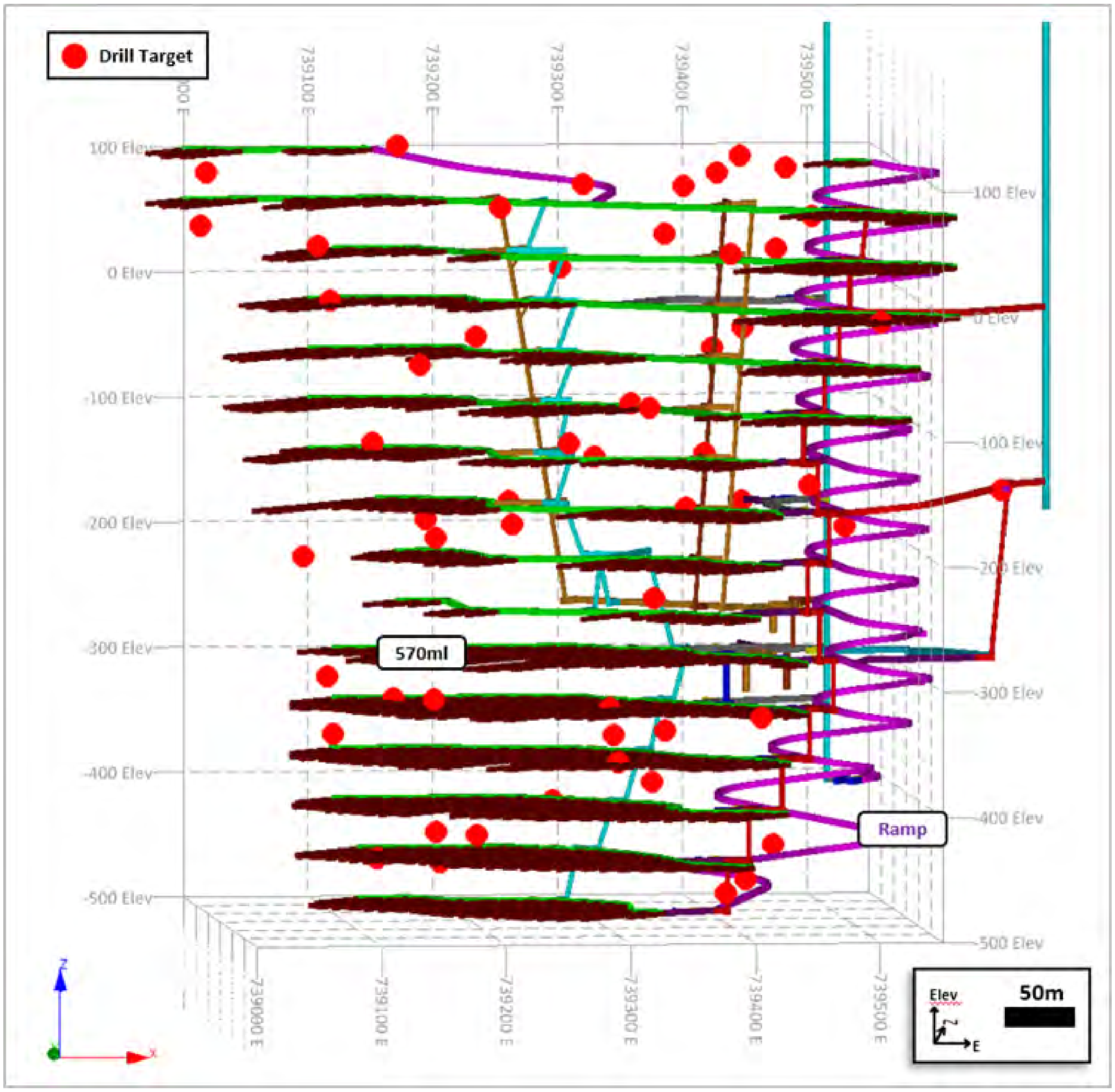
Accordingly, a dedicated grade control sampling practice must ensure the following:

- The program aims to deliver the most economic tonnes to the mill via an accurate definition of "ore" and waste;
- The program aims to identify variations in the dip, strike and width, impact on a local scale from faulting effects, and grade continuity/type. Variations in geometry at the edge of the mineralization require a geological understanding to ensure optimum grade, minimal dilution and maximum mining recovery;
- Safe practices are followed during the sampling process;
- Sampling remains as unbiased as possible;
- Representative (i.e. correct in terms of Gy's sampling theory ("Wikipedia Contributors," 2019); and
- Timely (so that the results can usefully define the ore blocks).

A successful program in an underground environment is completed through detailed geological mapping and grade sampling ahead of the mining. The mine geologist is to perform daily mapping and define the ore/waste contact for the mining team to progress. The mapping is incorporated into a digital format to improve the geological model further and enable the development of short-term estimation. The grade control strategy is related to the mining method and orebody type. For underground operations sampling methods include chip, channel and panel samples, grab/muck pile samples, and drill based samples.

NioCorp will establish a daily grade control strategy as outlined above before commencing mining. The strategy will involve an infill drilling and underground chip sampling program. The goal of the infill drilling and underground chip sampling program is to monitor and provide close spaced sampling that is required to define the boundaries of mineable ore blocks. The amount of sampling is constrained by practical limitations and cost considerations and will be adjusted over the LOM based upon the needs of the mining operation.

The infill drilling will occur from established drilling stations on each level, with underground diamond drilling using an NQ core diameter drilled across the width of the known mineralization. Drill logging is required to collect geological and structural measures in conjunction with assaying. The current protocols and procedures developed by NioCorp for exploration work require further development to support a daily production environment. NioCorp will drill multiple holes in a fan pattern from each station to gain information for levels above and below as required. The current geometry of the orebody supports completion of the infill drilling in advance of the mining to enable the design of the ore blocks to be based on true grade control sampling. Accordingly, Figure 16-27 outlines many of the critical areas that require further infill definition drilling.



Source: Nordmin, 2019

Figure 16-27: The Critical Areas Requiring Further Infill Definition Drilling

Additionally, NioCorp will rely on underground chip sampling to provide infill sampling for grade control purposes. Underground chip sampling continues to depend primarily on manual methods of extraction, i.e. collecting rock chips using a hammer and chisel. The proposed mining method will involve the development of cross-cuts at regular intervals across the width of the mineralization at the top and bottom of a stope before mining. Samples will be taken across the full width of the exposed mineralization with sufficient volume to ensure accurate assay. The sample weight will be the equivalent of at minimum half NQ core for the sampling interval. The samples will be logged geologically marking the width of the mineralization and any hanging wall or footwall mineralization.

Additionally, blasted material is available for grab sampling to test grades, which will be input into a production database and be used to confirm head grades and used for reconciliation purposes.

The close spaced sampling collected from drilling, chip sampling and grab sampling are utilized within the mine reconciliation process, where mine reconciliation is completed progressively by accumulating predictions for treated ore and comparing them to the production and mill results.

NioCorp will establish a consistent reconciliation framework, monitored on a daily/weekly/monthly and yearly process. The essential steps of the framework include the following:

- Establish an audit trail for all data.
- Agree to report results routinely in a consistent format and ensure that there are cross-functional reconciliation meetings in place to discuss results and develop action plans.
- Tabulate the data.
- Report variations based on consistent volumes (bench by bench, stope by stope) or periods (monthly, quarterly, annually).
- Graph the variations (or factors) for each parameter to determine trends.
- Analyze the differences and annotate the graphs to explain the differences.
- Alter the input parameters systematically to reduce future reconciliation differences.

A consistent framework of establishing reconciliation has been established by Harry Parker (2012) that used the following definitions:

$$F1 = \frac{\text{short range model depletions}}{\text{long range model depletions}} \quad \text{i.e. } F1 = \frac{\text{GRADE CONTROL (PREDICTION)}}{\text{ORE RESERVE (PREDICTION)}}$$

and

$$F2 = \frac{\text{received at mill}}{\text{delivered to mill}} \quad \text{i.e. } F2 = \frac{\text{MILL (PRODUCTION)}}{\text{GRADE CONTROL (PREDICTION)}}$$

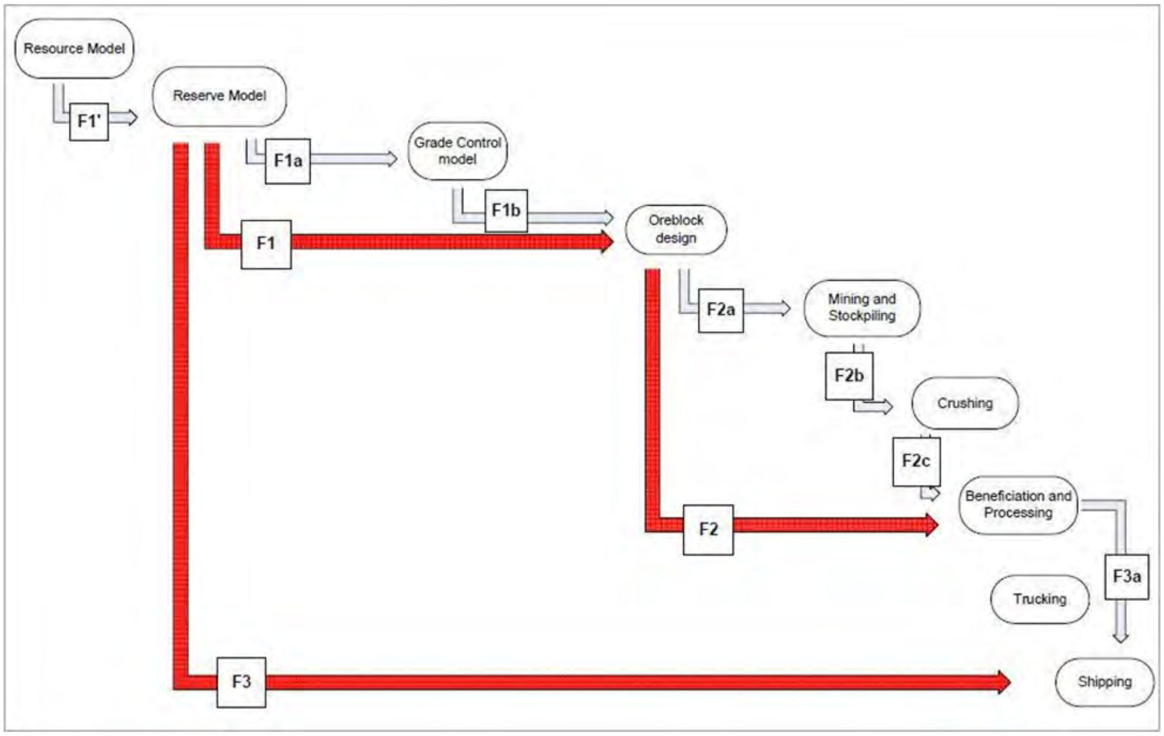
and

$$F3 = \frac{\text{received at mill}}{\text{long range model depletions}} \quad \text{i.e. } F3 = \frac{\text{MILL (PRODUCTION)}}{\text{ORE RESERVE (PREDICTION)}}$$

then it is now evident that $F3 = F1 * F2$

Source: Shaw, W.J, et al., 2013

By ensuring that reconciliation calculations are all done as factors (for tonnes, grade and metal), and each stage of the chain is used as a numerator when compared to the previous component in the chain, all of the various components of a mine reconciliation scheme can be rationalized and compared (see Figure 16-28).



Source: Shaw, W.J, et al., 2013

Figure 16-28: Elaboration of the Reconciliation Process Defining Additional Steps

Nordmin recommends that NioCorp establish a series of protocols covering all grade control tasks, reconciliation from mapping to sampling, and integration with the database. Review the ongoing quality assurance/quality control monitoring to ensure protocols and staff are updated as required.

16.7 Ventilation

The ventilation design was completed based on the mine design and production schedule described in previous sections. The backbone of the design includes the main production shaft and the ventilation/exhaust shaft, which both have an inside diameter of 6.0 m. Both shafts have two parallel surface fans, which are connected to a plenum arrangement that in turn is connected to the shaft collar area. Each of the four fans has a power rating of 224 kWh.

The production shaft serves as the mine fresh air intake, and the ventilation shaft provides the mine exhaust. The volume of air travelling through the shafts is 283 m^3/s. When surface temperatures are lower than 4° C, heaters in front of the two intake surface fans are activated.

16.8 Airflow Requirements

The airflow requirements were based on the engine power for the mobile equipment to be operated, within the entire mine, and within any given segment of the mine. The mucking and haulage fleet will be diesel powered. The drill carriers and utility vehicles will be electric powered with batteries. A factor of 0.063 m^3/s per kW of engine power was used to ensure the dilution and dissipation of diesel engine emissions. Minimum airflow requirements for the mine were estimated based on the equipment list and estimated utilization. The airflow required is 283 m^3/s for the mine at maximum productivity. This mine-wide airflow is directed, via regulators and auxiliary ventilation fans, to the areas where it is required for specific equipment, and where it is needed to maintain a minimal amount of airflow in the absence of diesel equipment.

The use of ore and waste passes has resulted in fewer haulage trucks than used in the SRK 2017 feasibility study, which in turn lowered the ventilation volume requirement. An opportunity exists to make further reductions with the introduction of battery-powered electric mucking and haulage equipment as the mine progresses beyond the initial development phase.

Air velocity limitations vary according to airway type. The minimum recommended air velocity for an area with workers in a drift is 0.3 m/s, to provide perceptible movement. In areas such as return airways and shafts where personnel are not expected to work, higher velocities are acceptable. Table 16-16 contains relevant selections from a Table on page 9-13 of Subsurface Ventilation Engineering by Malcolm J. McPherson (1993). It provides the threshold airflow velocities for various airway types. Maximum velocities were considered for shaft and development drift profiles.

Table 16-16: Maximum Airflow Velocities (m/s)

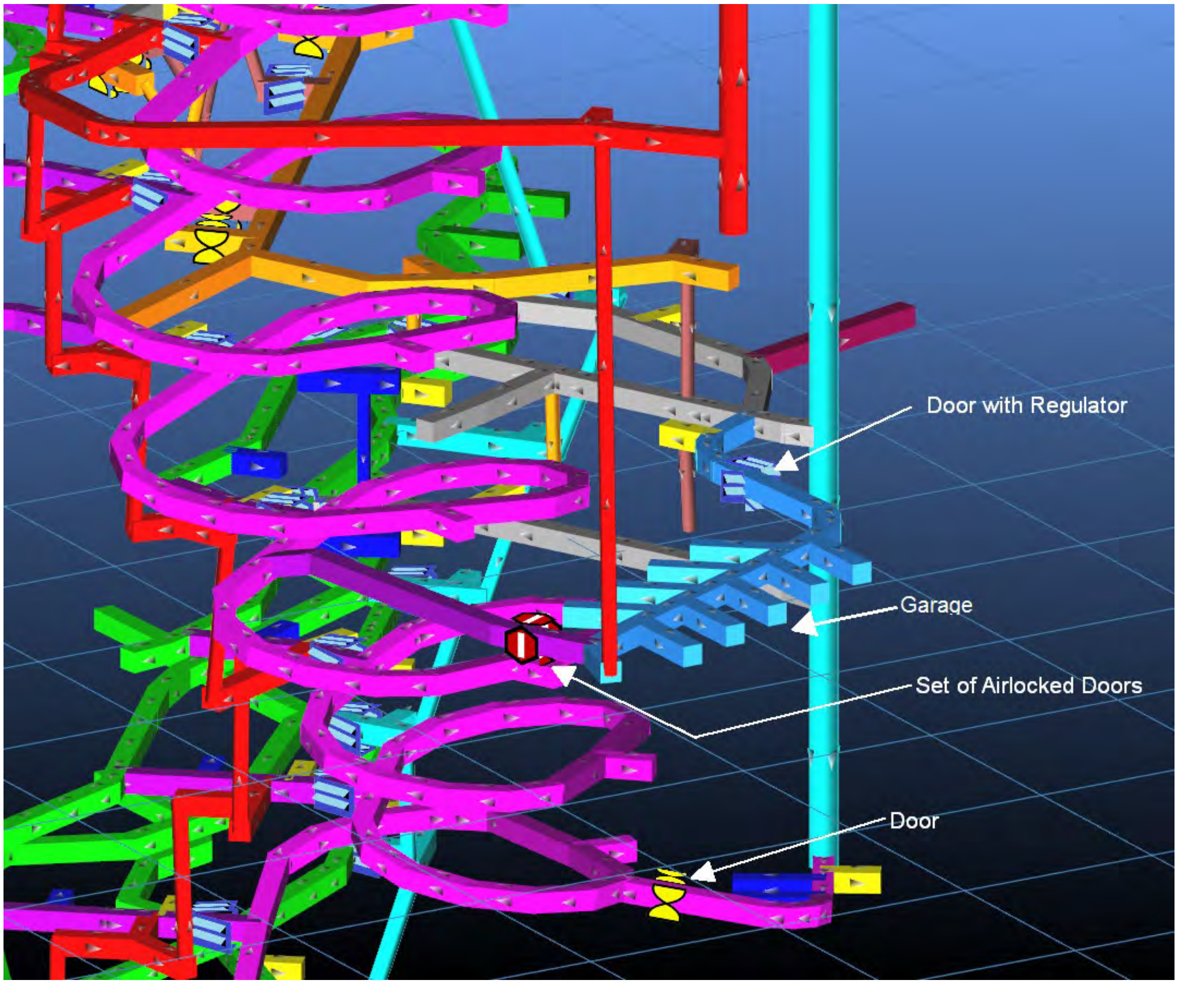
Area	Velocity (m/s)
Working Faces	4
Conveyor Drifts	5
Main Haulage Routes	6
Shafts with Hoisting	10

Source: Nordmin, 2019

16.8.1 Ventilation Controls

<u>Fixed Facilities Controls</u>

Figure 16-29 demonstrates the suggested controls for the garage and shaft bottom area. Air from the garage flows directly to a return air raise that ventilates directly into the ventilation/exhaust shaft. This way, in the event of a fire within the garage area, smoke and fumes cannot contaminate the mine. Fire doors are included in all fuel bays and the garage to prevent the spread of fumes in the event of a fire with remote controlled doors. Interlocked equipment doors with sliding regulators are used to regulate the quantity of air delivered to various levels of the mine.



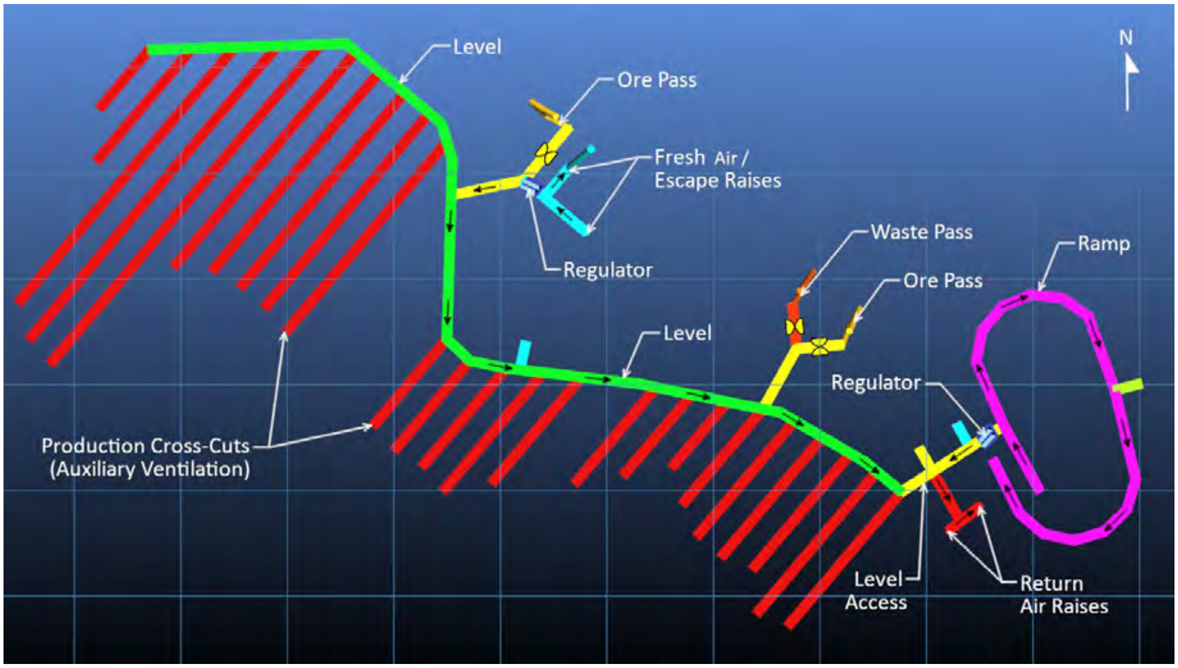
Source: Nordmin, 2019

Figure 16-29: Garage Area Ventilation Controls (Looking Northwest)

Level Ventilation Controls

Air enters the mining levels from the intake raise system and is regulated onto the levels depending on the air requirement at that stage of production. Portions of the air travelling within the flow-through route on a level are picked up and used in production headings via auxiliary ventilation fans and ventilation ducting. Once used in a heading, the air then re-enters back to the flow-through route to be carried to the exhaust raise system and into the ventilation/exhaust shaft. Enough air is pulled from the intake raise system a allow a small amount of the air to flow out to the ramp. This system allows mining on more than one level, without contaminating downstream work areas with upstream diesel exhaust and dust (see Figure 16-30).

The construction of temporary bulkheads is required when stopes are open between levels, to maintain control of the ventilation system and prevent short-circuiting of air. These temporary bulkheads can be a simple curtain type bulkhead made from brattice material.



Source: Nordmin, 2019

Figure 16-30: Level Ventilation Controls (Plan View)

When mining of the level is complete, the regulators are sealed off with door access bulkheads, which will allow re-entry for inspection if required. These bulkheads will prevent short-circuiting of air into these mined out levels.

16.8.2 Ventilation Model

Ventsim Design was used to generate the ventilation model using pre-set resistance factors programmed into the software. Ventilation models were generated for various phases of the mining sequence.

Staged Modelling Fan Results - Main Fans

Table 16-17 provides the main surface fan duties. This information was used to size fans and determines power requirements to ventilate the mine. The installed powers were calculated with an assumed fan efficiency of 70% and the required air power. The assumed air density at the fans is 1.18 kg/m^3.

Table 16-17: Duties of Main Surface Fans

Description	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)(1)	Total Motor Power (kW)(1)
2 Intake Fans (in parallel, per fan)	1.62	283	156	223	446
2 Exhaust Fans (in parallel, per fan)	1.62	283	156	223	446

Source: Nordmin, 2019
(1) Assume 70% fan efficiency

The operating pressures supplied should be considered applied pressures; that is, the pressures reported do not account for losses associated with fan housings, ducts, plenums, or diffusers.

Staged Modelling Fan Results - Surface Heating Fans

The surface intake fans are low-pressure, high-volume fan(s), as required for the air heaters. These fans produce just enough pressure to overcome the losses from the heater, fan, and plenum so that there is a slight positive pressure within the production headframe. The total airflow through the intake fans is slightly greater than through the underground workings because additional air is used to slightly upcast air in the main production shaft in order establish a positive pressure in the production headframe. Table 16-18 provides the total airflow throughout the mine workings.

Table 16-18: Total Mine Airflow

Mining Area	Airflow (m^3/s)
Upper Mining Area	142
Lower Mining Area	142
Total	**283**

Source: Nordmin, 2019

16.8.3 Auxiliary Ventilation

In areas that are not in the path of flow-through ventilation, including the production area to the west of the fresh air raises as well as the stope access crosscuts, auxiliary ventilation is used. The auxiliary ventilation consists of a fan and ducting attached to the fan, that is run to the mining face, draw point or drill drift. Fans and ducting are selected to deliver enough air to provide 0.063 m^3/s for every 1 hp of the maximum equipment that will operate at the same time in the area.

16.8.4 Recommended Ventilation Infrastructure

Sensors

Several different types of remote sensors are recommended for installation at the mine. These sensors can help predict wear on the fans, alarm in the event of a fire, low temperature or harmful gasses and can tie into the ventilation modelling software. Bundled air quality and quantity sensors are recommended for each fan installation, intake shaft, fixed facilities and each working level. These include fan monitoring, air quality, air quantity, and psychrometric sensors.

Regulators

Drop board regulators consist of large wooden or steel boards which are slotted from the ground up. More boards in place result in a smaller air opening and consequent generation of higher airway resistance. Regulators can also form the frame for a bulkhead. Drop board type of regulators are recommended to be placed at the intake and exhaust of each level.

Slide regulators are typically a piece of steel on a slider that can be adjusted manually to increase/decrease the area of an opening in a door or a regulator.

Louvred regulators are manufactured items that consist of steel slats that rotate on a horizontal axis within a frame and are controlled electronically. These regulators can be controlled from the surface and can form part of a ventilation on demand (VOD) system.

Bulkheads

Temporary bulkheads for stopes with an open brow are constructed from flexible plastic PVC brattice material. They do not experience high pressures and serve to prevent loss of ventilating air from stopes while in an LHD mucking phase.

Permanent bulkheads such as for a level no longer in use consist of a shotcrete wall with a steel personnel door, to allow worker access.

Equipment Doors

Pneumatically operated, steel equipment doors allow passage of vehicles and materials, and control the volume of airflow to or from an area. Dual air locking doors in series are used where the pressure is highest. Personnel doors are installed beside the main equipment doors to safeguard workers against being struck by a closing door.

Air Heaters

Natural gas air heaters are used at the surface intake shaft. Low-pressure high-quantity fans are used with the air heaters. These fans produce just enough pressure to overcome the losses from the heater, fan, and plenum so that there is a slight positive pressure in the production headframe.

16.8.5 Ventilation Power Consumption

Based on the fan operating points, the total motor power of the main fans is estimated to be 895 kW. The monthly power consumption averages to 470,838 kWh/month based on applied load and utilization factors.

16.8.6 Air Heating

Winter ice buildup can cause airways to be restricted and lead to hazardous conditions. An intake air heating system, as described below, is recommended that will mitigate these conditions. Based on the average temperatures shown in Table 16-19, air heating is required for seven months out of the year. A 7.5 MW heater provides 70,689,000 MJ per year, based on average temperatures and an airflow of 283 m^3/s. Instrumentation for this heater includes a thermostat in mixed air an appropriate distance downstream of the heater, as well as a carbon monoxide monitor with an alarm and automatic fuel cut-off to the heater.

Table 16-19: Surface Temperatures Near the Elk Creek Mine

	Jan	Feb	Mar	Apr	Oct	Nov	Dec
AVG C	-10.6	-8.3	-2.2	3.9	3.9	-2.2	-8.9

Source: Nordmin, 2019

16.9 Mine Infrastructure & Services

16.9.1 Material Handling System

The underground material handling system is designed for both waste and ore, to provide surge and storage capacity underground, to size ore, and to be an efficient, automated system from underground mining areas to the mineral processing plant on the surface via the production shaft and surface conveyors.

During underground operations, mined waste and ore will be dumped through a typical 300 mm x 300 mm (12” x 12”) scalping grizzly complete with rock breaker and will be stored in either a waste pass or either of two ore passes. The three passes will serve the majority of mining levels, (-215 El. and above) and will lead to either a waste or an ore storage bin. The ore then passes over an ore handling apron feeder which will feed the ore sizing and storage circuit. The waste from the storage bin feeds onto the loading pocket conveyor.

The apron feeder will supply ore from the ore pass to a grizzly feeder, which will allow undersized material to be removed from the crushing circuit prior to the crusher. The oversized material will continue to the single C series jaw crusher, which will size the ore to the mineral processing plant required 115 mm (4½”). All ore will then be passed through conveyor systems and transfer cars to either the dual use waste or ore storage bin or the single-use ore storage bin. Both underground storage bins are designed to hold up to 2,500 tonnes (2,750 tons), representing approximately two day's production through the processing plant.

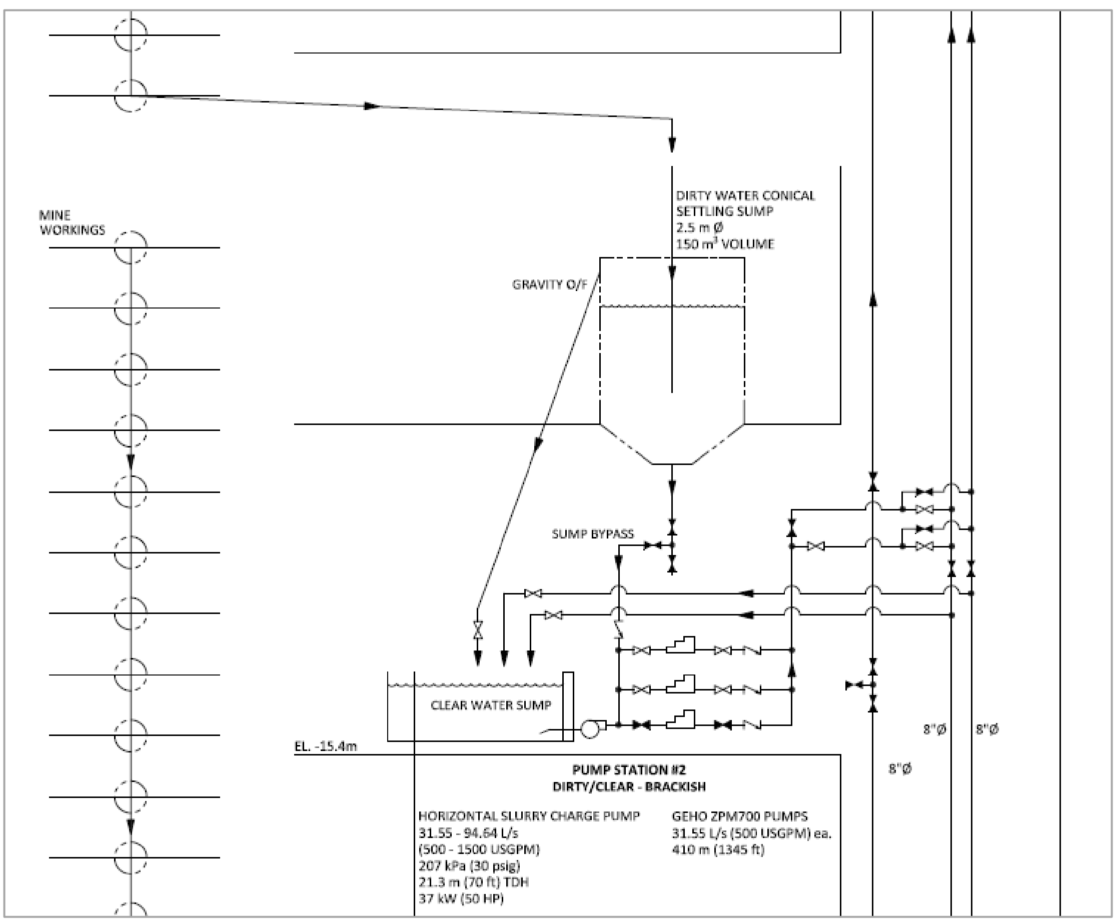
Ore or waste from the loading pocket conveyor is forwarded via conveyor and transfer car to the twin 11 tonnes (12 tons) weighted flasks at the periphery of the production shaft. The flasks are weighed, and the feed and discharge system is controlled via a PLC/PC interface. Control monitoring around the facility will use CCTV cameras located at strategic points, with load sensors, bearing and motor monitors, etc. Rock breakers will have the option to be remotely operated by the hoist operators via joystick and camera, or by an operator located underground at the rock breaker. The entire system will be monitored by the hoisting personnel.

Once the flasks are adequately filled, ore or waste will be forwarded to the skips within the shaft. Loaded skips will be hoisted to the headframe on the surface and dumped into a chute which will direct the material to either a 1,000 tonne (1,100 tons) waste storage bin or to a 2,000 tonne (2,200 tons) ore storage bin via a transfer car. The waste will be stored in the bin and further loaded into haul trucks to be brought to the dedicated waste storage area. The stored ore will pass onto a conveyor system, complete with manual loading area, and will report to the mineral processing plant.

16.9.2 Mine Dewatering System

Mine dewatering at the Project is designed to accommodate groundwater inflows from the shaft and mine workings, along with inflows from drill and other underground operating equipment. Total inflows have been estimated to be approximately 32 L/s (500 US gpm), which includes service water and can be accommodated with one main dewatering station pump running. To accommodate for uncertainty in the water model, the design capacity of the underground dewatering system can accommodate an elevated flow rate of 63 L/s (1,000 US gpm) with two station pumps running, and a maximum flow rate up to 95 L/s (1,500 US gpm) with all three station pumps in operation.

The system design incorporates two main pumping stations, one on the -15.4 m (-50.5 ft) level and one on the -335.4 m (-1,100 ft) level, that work in series to lift mine water from the lowest depths of the mine to the surface via the production shaft. Each station is comprised of a single 150 m^3 (5,297 ft^3) conical settling sump (borehole) allowing clear water to pass along to three positive displacement pumps (GEHO ZPM800) which lift the water from the station to the next available level through 8” pipes within the shaft. Additionally, during upset conditions and when required, the PD pumps will allow operations to lift dirty water from the settling sumps up through the shaft piping to surface. Figure 16-31 demonstrates a typical pumping system.



Source: Nordmin, 2019

Figure 16-31: Typical Pumping System (-15.4m and -335.4 m)

The cone bottom settling sumps will also integrate a flush water port to be utilized to assist draining of settled solids to a desired location on the pumping level. The shaft will house three 8” diameter carbon steel schedule 80 pipe columns. Water from the underground workings will be forwarded to the water treatment center on the surface, where it will be purified to supply the hydromet and pyromet and underground with process water and offices, mine dry and other buildings with potable water.

16.9.3 Compressed Air System

Compressed air will be supplied to the production surface mining structures (headframe and hoist house) via twin compressors located within the production shaft hoist house. Each compressor will be rated for 0.94 m^3/s (2,000 scfm) at the intake with a discharge pressure of up to 862 kPa (125 psig), which will ensure that the compressed air delivered underground remains above 690 kPa (100 psig). An ASME Section VIII air receiver (complete with water purge valve and safety relief valve) located within the hoist house will also ensure that the compressors are not required to cycle on and off excessively. Surface branch connections will include the hoist house and headframe where a requirement for air driven hand tools exits. No requirement for instrument air, requiring a drying system exists within the production hoist house.

An 8” diameter carbon steel pipe will deliver the compressed air from the hoist house to the headframe via an underground services trench. This pipeline will then proceed to the production shaft, where it will form part of the services hung from the shaft steel, down to the working levels, including crusher station, conveyance level, loading pocket and spill pocket. Each level will have an individual line branched from the shaft line, that will service the level. No additional underground storage will be required, as all main lines serving the levels will be large enough to accommodate the usage and will supply additional storage.

The requirement for compressed air within the backfill plant and the ventilation shaft and headframe will be filled with small portable compressors housed within the structures.

16.9.4 Underground Water Supply

Process

Industrial or process water supply for drilling and dust control will be supplied from the water treatment center via a 4” nominal carbon steel line to the production headframe and down the production shaft. Individual branch connections will report to each of the levels that require process water, including those that require water for the dust suppression system. As the static pressure increases, the deeper the line extends (to the lower shaft stations), pressure reducing stations will be utilized.

Pressure reducing stations are strategically located down the shaft, at the crushing station and loading pocket levels. Typically, a 4” diameter Ford Figure #12 Pressure Reducing Valve (PRV) with either flanged or grooved ends, will be employed at each station. Ford Figure #12 PRVs is used to reduce supply pressures from a maximum of 2758 kPa (400 psig) to the desired discharge service pressure. The PRV stations will be located as close as possible to the shaft.

16.9.5 Underground Fuel Storage and Distribution

Fuel from the surface storage facility will be delivered to the underground storage system via a 2” diameter fuel transfer pipeline within the production shaft. The fuel line will run from the surface, down to the underground shops level where the line will be routed to a storage area at a fuel bay for fueling vehicles. The fuel pipe feeds fuel to either of two 3,785 L (1,000 US gallons) storage and distribution systems, located within a cut out on the south of fuel bay, via a motorized three-way valve.

Each storage and distribution station will be a bladder type, with up to 150% containment, complete with the following safety functions: 4 hr rated UL approved roll-up door, thermally activated fuel shut off valve to the dispensing system, anti-syphon valve, and a dry chemical automatic fire suppression system with detection and actuation. Each station will be individually alarmed, by means of a PLC with level alarms, and a level switch.

Additionally, fusible link fire doors are also included in the underground layout, these twin fire doors, upon actuation, will isolate the fueling area from the main shops.

16.9.6 Workshop, Maintenance Bays, and Warehouse

The maintenance area consists of nine large bays, of approximately 16 m long by 7 m wide to accommodate vehicular traffic. One wash bay is included in the workshop layout. A drainage trench with covering grating runs the length of the bay to carry water to a nearby oil capture sump. Grading of the area will reduce the possibility of oil contamination. Three maintenance bays are equipped with an overhead crane to facilitate the maintenance work on vehicles.

Warehouse and tool cribs are included within the maintenance area.

Airlock doors separate the maintenance area from the rest of the mine. An office is located at the end of a drift located in the maintenance area.

16.9.7 Explosives Storage

The mine design includes underground powder and primer magazines. The mine explosives are stored off-site at a vendor location and deliveries are on as needed basis with the underground magazines providing the capacity required for production needs. The explosives pricing includes the contractor storage and supply totes, as per the manufacturer's recommendation, all of which are included in the capital estimate.

16.9.8 Refuge Stations

Two mobile refuge chambers have been included within the underground mine design. Each refuge chamber will be sufficiently equipped to house 12 or more persons, depending on location and unit size, for up to 36 hours. The stations are self-sufficient in that they include seating, a chemical toilet, emergency food and water, back-up power, lighting, and communications via external antenna and 12V power supply. The breathable air system that is incorporated within the refuge chambers includes a standard compressed air line tie in, oxygen cylinders connection, as well as an oxygen candle. Each chamber can be located at the most strategic location as dictated by the mining operation and underground workings. The chambers are easily transported by forklifts or LHD units.

In addition to the two mobile refuge stations, there will be a permanent refuge station located on the 530 Level and 65O Level. Both permanent refuge stations will be equipped in a similar fashion to the two mobile refuge stations, but with a capacity of 30 persons per station.

16.9.9 Hoist House Substation Surface Electrical Distribution

Electrical power will be supplied to the hoist house substation via a 44 kV overhead line from the main substation. Power will be stepped down to 13.8 kV by two 20/25 MVA Delta/Wye resistance grounded transformers, supplying a main-tie-main primary distribution switchgear.

The E-House in the hoist house substation will distribute power at 13.8 kV to the production hoist and service hoist drive transformers, one underground feeder (3C 4/0 AWG), the hoist house 4160 V power distribution (via two 13.8 kV-4160 V step down transformers) and an overhead line. The overhead line will supply power to the surface infrastructure (offices, administration, dry) and ventilation shaft power distribution switchgear. The second underground circuit is fed from this ventilation shaft switchgear unit.

Two backup diesel generator units are included in the power system for use during a utility outage. The first backup generator is connected to the hoist house switchgear and includes a regenerative load bank. The second backup generator is connected to the ventilation shaft switchgear, at 13.8 kV, and includes a regenerative load bank.

Critical loads will include both auxiliary hoists, auxiliary hoist MCCs, downcast fan #1, and the underground mine feeder #1. The underground feeder #1 will provide power to critical underground ventilation during a utility outage.

Additional loads on the 4160 V hoist house switchgear include downcast fan #2, compressors and 480 V services.

The 480 V unit substation will supply power to all the 480 V motor control centers which will supply power to all other electrical loads in the hoist house, headframe/collar house, hoist house substation and downcast fan heater building. The hoist house MCC will service 480 V loads, ventilation, lighting and low voltage services for the hoist house. The production hoist MCC will supply all loads to all ancillary equipment for the operation of the production hoist. The headframe/collar house MCC will be in the collar house and will service all 480 V loads in the headframe/collar house, including ventilation, lighting loads, and skip handling equipment.

16.9.10 Underground Electrical Distribution

Two 13.8 kV shaft feeders will supply power underground. Underground feeder #1 will originate in the hoist house substation, traverse through the hoist house, to the headframe. The cable will then be hung vertically in the production shaft with the cable turning into selected levels to distribute power. Underground feeder #2 will originate at the ventilation shaft switchgear and will be hung vertically in the ventilation shaft.

At each supplied level, there will be a dual load break switch in order to select which feeder is used to supply power to the level. A 15 kV junction box will be used to provide a junction point to supply the necessary equipment and provide a point for expansion at each level.

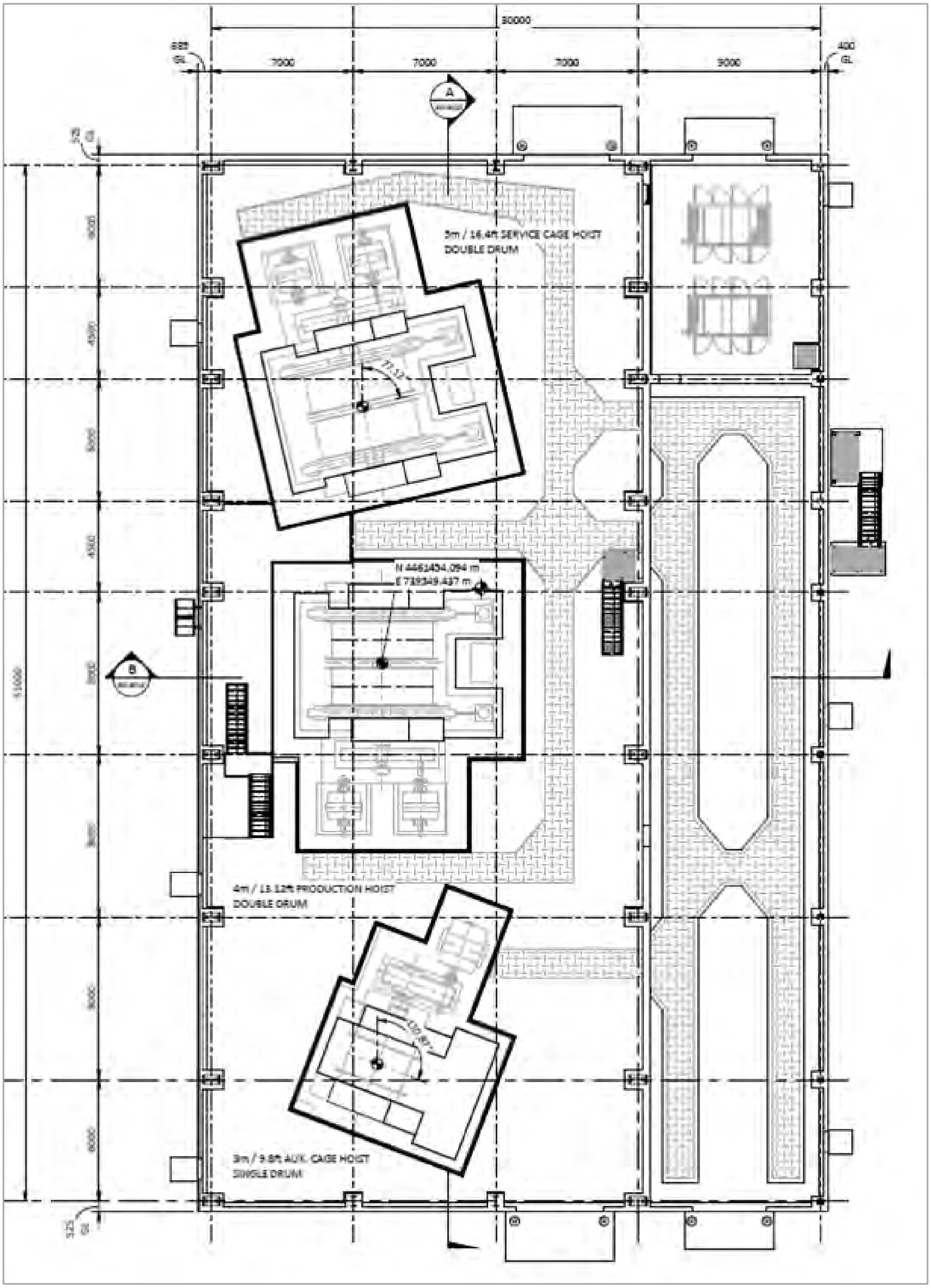
16.9.11 Overhead Pole Line Electrical Distribution

A 13.8 kV overhead pole line requires construction to distribute power to temporary loads during shaft freezing and sinking. The overhead line will originate at a temporary generator farm, which will be used to supply power during the initial stages of construction. The line will remain in service after temporary power has been removed, and will be permanently connected to the 13.8 kV main substation. This overhead distribution will supply power to the ventilation shaft switchgear unit, and the office/administration/dry structure.

16.9.12 Hoisting Plants

The hoisting plants are designed to serve as both an efficient means of hoisting ore and waste to the surface and of lowering and lifting labour forces, materials, and equipment between the surface and the underground working levels of the mine.

The production shaft hoisting plant (see Figure 16-32) is comprised of three hoists: the service cage double drum (DD) single-clutched hoist in a balanced condition, the skipping DD single-clutched hoist in a balanced condition and the auxiliary cage single drum (SD) hoist. The service cage hoist is the largest of the three at 5 m in diameter and supports both a 3500 mm x 1900 mm (138” x 75”) double deck main service cage with a service capacity of 20,000 kg (22,045 lb) and a counterweight. The skipping DD hoist is the next largest hoist at 4 m in diameter and supports twin 11,000 kg (12,125 lb) balanced payload bottom dump skip conveyances. The smallest of the three production hoists is the 3 m diameter SD auxiliary cage hoist which supports a 1500 mm x 900 mm (59” x 35”) double deck cage with a total payload capacity of 5,000 kg (11,000 lb).

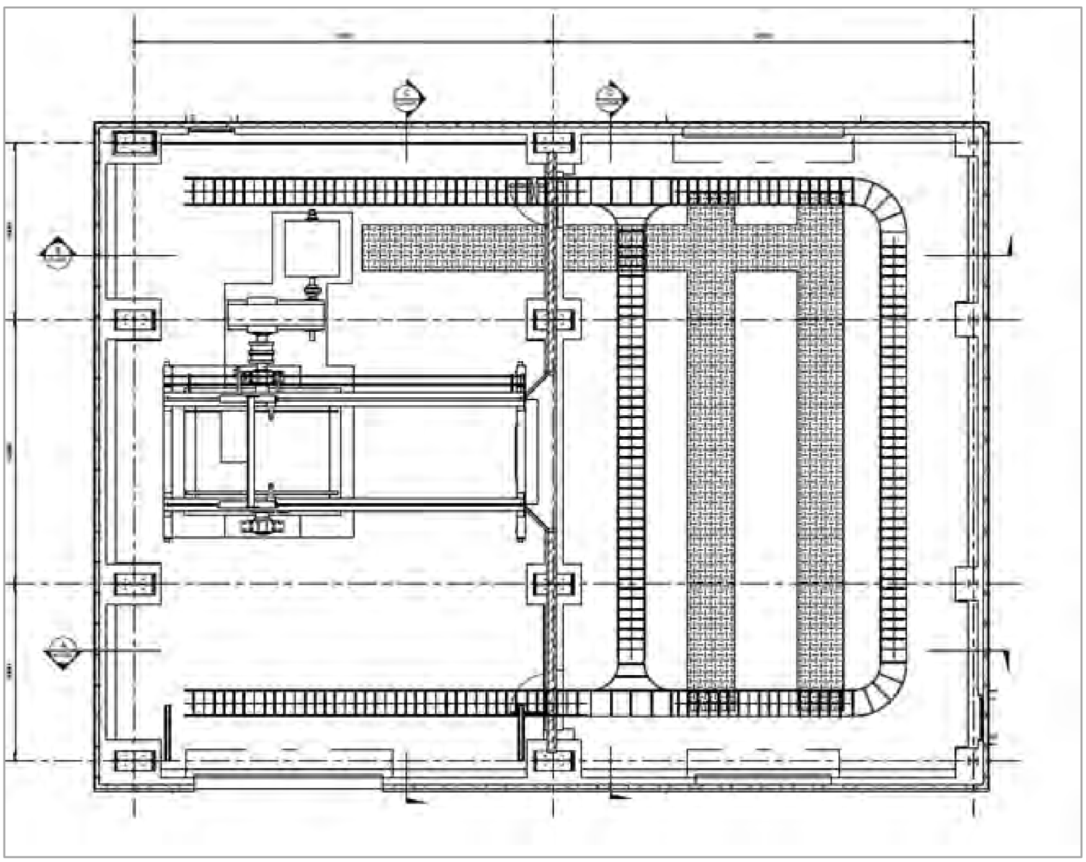


Source: Nordmin, 2019

Figure 16-32: Production Shaft Hoisting Plant

All three hoists will be powered through the main mining substation, which is dedicated to mine infrastructure both on surface and underground. Additionally, a 2 MW diesel powered generator set will supply back up power to the auxiliary hoist, ventilation system and one air compressor feeding compressed air to the underground workings.

The ventilation shaft hoisting plant (see Figure 16-33) is comprised of one 3 m diameter SD auxiliary cage hoist which supports a 1500 mm x 900 mm (59” x 35”) double deck cage with a total payload capacity of 5,000 kg (11,000 lb). The auxiliary hoist and cage are duplicates of those installed within the production hoisting plant and shaft, thus ensuring a common platform for both systems. As within the production hoisting plant, the auxiliary cage hoist in the ventilation shaft is powered both by the main surface infrastructure and a secondary back up diesel generator. Thus, ensuring not only a secondary means of mechanical egress from underground but twinned mechanical egresses from either the production shaft or the ventilation shaft.



Source: Nordmin, 2019

Figure 16-33: Ventilation Shaft Hoisting Plant

16.9.13 Dust Suppression System

A multi-zoned underground dust suppression system will aid in reducing the amount of air born dust created by the ore and waste handling systems. Each ore and waste transfer point within the underground material handling system will have a set of air and water nozzles fed from both the compressed air and process water systems. The compressed air and process water are piped to a regulating station cabinet where the fluids are cleaned (filter and strainer) and pressure regulated.

From the cabinet, the individual cleaned, and regulated process water and compressed air are fed to a set of strategically placed nozzles which combine the streams and produce a light mist which is enough to drive air born dust down. Regulating stations will be located at all material handling open draw areas including truck dump sites, apron feeder area, vibratory feeders, conveyor systems, transfer car chutes, the crushing station, the transfer area downstream of the crusher, bin conveyor feed area, bin discharge areas, and the flask infeed area.

16.9.14 Communications System

The mine will be equipped with a leaky feeder system that will allow internet, phone, and radio communications underground. The mine will have standard underground call phones with intercom. A control system will allow remote operation of the rock breaker and CCTV system to monitor dump points, crusher, and key material handling locations.

16.9.15 Safety and Health

Mine Safety and Health Administration (MSHA) safety standards are incorporated in the mine design and include dual secondary means of mechanical egress, backup power for both auxiliary hoists, partial ventilation system and one air compressor which feeds compressed air to the underground. Twelve person mobile refuge chambers are included and will be in active working areas over the LOM. In addition, there is a cut-out on both the 530 Level and 650 Level to facilitate the installation of two permanent 30-person refuge chambers.

The mine will have a communications system that has both mine phones and wireless communication through a leaky feeder system. A mine rescue team will be required to support the mine’s underground operation. The mine safety program will integrate with local providers in case of any mine emergency. Additionally, a stench gas emergency warning system will be installed in the mine's intake ventilation system. This system can be activated to warn underground employees of a fire situation or other emergency whereupon emergency procedures will be followed. The shop areas and underground fueling station will be equipped with automatic closure doors that will operate in case of fire.

16.9.16 Workforce

Workforce levels are estimated based on the production schedule and equipment needs. The productivities used reflect a mix of local and skilled labour with an experienced management team.

The estimate is based on the utilization of a contractor for mining development and operations with an ownership senior management team to oversee mining activities. The rotating contractor crews will be using an operating schedule consisting of 12 hours per shift, two shifts per day, and seven days per week. A four crew arrangement supports the 12-hour shift with two crews onsite at any given time (per rotation). The ownership, senior management and technical team are planned to work five 8 hour days per week.

Table 16-20 shows the maximum required workforce. There are 96 people on a two-week rotation and 24 ownership senior management and technical team on a weekly basis. The workforce increases over time to a maximum of 216 in year five. There will be a maximum of 120 people onsite at any given time.

Table 16-20: Typical Mining Labour

Management / Technical Support		**Total Qty.***
Mining Manager		1
Mine Superintendent		1
Maintenance Superintendent		2
Chief Engineer		1
Geotechnical Engineer		1
Long Term Mine Planner		1
Short Term Mine Planner		1
Project Engineer (ventilation, water, construction)		2
Chief Geologist		1
Resource Geologist		1
Grade Control Geologist		2
Administrative / Mine Clerks		1
Chief Surveyor		1
Mine Surveyor		3
Material Handling / Shaft Shift Supervisor		2
Mine Services Shift Supervisor - Construction		1
Maintenance Shift Supervisor - Fixed Equipment		1
Electrical General Foreman		1
Total Management / Technical Support		**24**
Rotating Crews	**Per Rotation Qty.***	**Total Qty.***
Shaft Services	2	4
Hoistperson	2	4
Deckman	2	4
Skip Tender / Crusher	2	4
Safety Technician / Trainer	2	4
Development / Production Shift Supervisor	2	4
Vertical Development Crew	2	4
Blasting/Powder Crew	4	8
Blasting/Powder Crew Helper	4	8
Jumbo Operator	4	8
Longhole Drill Operator	3	6
LHD Operator	7	14
Haul Truck Operator	8	16
Bolter Operator	8	16
Cable Bolter Operator	2	4
Nipper	4	8
Shift Supervisor - Logistics	1	2
Utility / Construction Crew	4	8
Grouting Lead	1	2
Grader Operator	1	2
Conveyor Attendant	2	4
Diamond Driller	4	8
Maintenance Supervisor - Mobile Fleet	2	4
Mine Electrician	7	14
Heavy Equipment Mechanic	12	24
Welder	2	4
Instrumentation Technician	2	4
Total Rotating Crews	**96**	**192**
Grand Total		**216**

Source: Nordmin, 2019

*This value represents peak contractor and ownership workforce.

16.9.17 Equipment

The underground equipment used, shown in Table 16-21, is typical for a sublevel stoping mining method with the number of pieces of equipment calculated from the production rates and typical availabilities for equipment in underground mines.

The estimate uses typical availabilities and utilization rates for mining equipment used for this mining method. Each shift of 12 hours is reduced by 2.25 hours to represent shift change, breaks, lunch, fuel/grease/inspect time and travel to and from work areas. This provides an equivalent working day of 19.5 hours or 9.75 hours per shift. The resulting reductions result in 5,931 productive hours per year of mining time. It should be noted that the layout of this mine and mining on multiple levels requires the addition of equipment to reduce equipment move time. This reduces the overall utilization of the equipment fleet.

Table 16-21 summarizes the mine equipment totals for peak production.

Table 16-22 summarizes the major fixed equipment for the mine.

Table 16-21: Mine Mobile Equipment

Type of Equipment	Quantity*
Drill Jumbo	3
Haul Truck (40 t)	5
LHD (6.2 m^3)	4
Longhole Drill	2
Cable Bolter	1
Bolter	4
Grader	1
Personnel Carrier	2
Pick-up Trucks	6
Utility Vehicle	3
Boom Truck	2
Scissor Lift	2
Shotcrete Sprayer	2
Anfo / Emulsion Loader	3
Development Emulsion Loader	1
Production Emulsion Loader	1
Portable Grout Unit	2
Blockholer	1
Exploration Drill	2

Source: Nordmin, 2019

* This value represents peak production mine mobile equipment fleet.

Table 16-22: Mine Fixed Equipment

Type of Equipment	Quantity*
Production Shaft Skip Hoist	1
Production Shaft Service Cage Hoist	1
Production Shaft Auxiliary Cage Hoist	1
Ventilation Shaft Auxiliary Cage Hoist	1
Mine Ventilation Supply Fans (224 kW)	2
Mine Ventilation Exhaust Fans (224 kW)	2
Auxiliary Ventilation Fans (112 kW)	6
Auxiliary Ventilation Fans (56 kW)	4
Natural Gas Mine Air Heaters	1
Surface Apron Feeders	2
Surface Conveyors	3
Service Cage	1
Auxiliary Cages	2
Skips	2
Rock Breakers	4
Grizzly Feeder	1
Apron Feeder	1
Jaw Crusher	1
Crusher Discharge Conveyor	1
Ore Bin Conveyor	1
Loading Pocket Conveyor	1
Vibratory Belt Feeder	2
Belt Magnets	3
Chain Gates	5
Crusher O/H Crane	1
Electric Battery Stations	3
Underground Shop O/H Cranes	3
Main Dewatering Pumps	6
Primer Pumps	2
Submersible Pumps	10
Main Air Compressors	2

Source: Nordmin, 2019
* This value represents peak production mine fixed equipment.

17. RECOVERY METHODS

17.1 Process Plant Design Criteria

17.1.1 Surface Crushing, Ore Storage & Mineral Processing Plant

The primary driver of the comminution circuit design is the dry processing of ore, which will be used to avoid an expensive drying operation prior to acid leaching.

The process design relies upon two things; receiving a primary crusher product with a characteristic particle size of (P_{80}) 115 mm at the comminution circuit feed bin and producing feed material for the downstream hydrometallurgical processing at a characteristic particle size of (P_{80}) 1.1 mm.

The primary crusher product will be fed to the secondary cone crusher system, operating in closed circuit with a double deck screen. The screen undersize from the cone crusher system will be fed to an HPGR unit, operating in closed circuit with another double deck screen. The HPGR screen undersize is the comminution product that will report to the hydrometallurgical process. The process design criteria are provided in Table 17-1.

17.1.2 Hydrometallurgical Plant

The hydrometallurgical process design criteria have been established based on bench and pilot scale test work, conducted by SGS, Hazen and KPM, as well as similar projects, and standard industry practices. The key items are listed in Table 17-2.

17.1.3 Pyrometallurgical Plant

The purpose of the pyrometallurgical (pyromet) plant is to reduce the niobium pentoxide in the hydromet feed by converting it into a saleable ferroniobium metal. The pyromet also plays an important role in the purification of the FeNb by removing excess Ti in the slag portion of the smelting. Since niobium is commonly alloyed with various high grade steels to increase their mechanical properties significantly, producing ferroniobium metal is an attractive and suitable option to be created for use in the steel industry.

The pyrometallurgical process design criteria were established based on thermodynamic calculations, inspired by test results completed by KPM and supported by the literature available on the aluminothermic reduction as well as on the niobium pyrometallurgy. Table 17-3 presents the pyromet design criteria.

The aluminothermic reduction has been selected as the technology to convert the hydrometallurgical Nb_2O_5 precipitate into a FeNb metal. Aluminum shots and iron oxide pellets will be introduced on a continuous basis along with the fluxing agents to initiate and complete the exothermic chemical reduction of the Nb_2O_5. This reduction is performed in a single electrical arc furnace with a continuous feed of precipitate, additives and fluxes to produce a saleable FeNb metal alloy.

Table 17-1: Design Process Criteria

Description	Value	Unit
Throughput and Operational Time		
Non-operational Time	0	h/a
Planned Down Time	252	h/a
Unplanned Down Time	1,276	h/a
Available Time	7,232	h/a
Availability	85	%
Annual Design Throughput	1,008,129	t/a
Process Plant Throughput	125	t/h
Ore Characteristics		
Average Specific Gravity	2.96	-
Moisture in Ore	5	%
Bulk Density	1.8	t/m^3
Angle of Repose	37	degrees
Angle of Reclaim	60	degrees
Test Work Parameters		
JK Drop Weight Test		
A x b - Maximum	58.4	-
A x b - Minimum	44.3	-
SMC *Test*		
A x b - Maximum	56.4	-
A x b - Minimum	34.9	-
$M_{,a}$ - Design	19.7	kWh/t
Mih - Design	15.0	kWh/t
Crushability and Grindability Tests		
Cwi	12.0	kWh/t
Rw, - Design	17.9	kWh/t
Bw, - Design	15.4	kWh/t
A, - Design	0.112	g
Crushing Circuit		
Feed Rate to Secondary Crusher	139	t/h
Primary Crusher Product Size (Pao)	115	mm
Primary Crusher Product Size (Ploo)	203	mm
Crushed Ore Bin Reclaim Feeder Type	Vibrating Feeder	
Design Feeder Capacity (Total)	160	t/h
Number of Feeders	3	-
Secondary Crusher Screen		
Screen Type	Double Deck Vibratory	
Number of Screens	1	-
Fresh Feed Throughput	139	t/h
Secondary Crusher Recycle Throughput	171	t/h
Total Screen Feed	311	t/h
Number of Decks	2	-
Top Deck Opening Size	50	mm
Bottom Deck Opening Size	25	mm
Product Sze (Pao)	22.4	mm
Screen Size - Area	18	m^2
Secondary Crusher		
Crusher Type	Cone	
Average Throughput	171	t/h
Number of Units	1	-
Feed Size - Maximum (Firm)	203	mm
Feed Size (Fao)	115	mm

Close Side Setting	25	mm
Product Size (Pao)	26	mm
Selected Crusher Size	HP300 or Equivalent	-
Crusher Motor Size	200	kW
HPGR Circuit		
Crusher Type	HPGR	
Feed Size (Fao)	22.4	mm
Fresh Feed Throughput	139	t/h
Total Throughput	198	t/h
Number of Units	1	
Specific Energy Consumption	4.18	kWh/t
Selected Size	POLYCOM 14/08 - 02 or Equivalent	
Installed Power	1,000	kW
Product Size (Pao)	1.1	mm
HPGR Screen		
Screen Type	Double Deck Vibratory	
Number of Screens	1	-
Screen Throughput	198	t/h
Screen Recycle Throughput (to HPGR)	59	t/h
Top Deck Opening Size	6	mm
Bottom Deck Opening Size	3	mm
Product Size (Pao)	1.10	mm
Screen Size - Area	18	m^2
Fine Ore Bin		
Fine Ore Bin - Storage Time	48.0	h
Crushed Ore Bin - Live Capacity	6,000	t
Fine Ore Bin Reclaim Feeder		-
Feeder Type	Vibrating Feeder	
Design Feeder Capacity (Total)	144	t/h
Number of Feeders	3	-

Source: Tetra Tech, 2017

Table 17-2: Hydrometallurgical Processing Design Criteria

Description	Value	Unit	Source
HCl Leach Unit			
General feed characteristics			
Temperature	AMB		HMB
Percent Solids	95%		
Mass flow rate	2764	dmt/d	
WPL Feed Composition			
Nb_2O_5	0.81	%w/w	Mine Plan
TiO_2	2.86	%w/w	Mine Plan
Sc_2O_3	100.75	ppm	Mine Plan
Al_20_3	2.24	%w/w	PEA II - PDC
BaO	4.41	%w/w	Mine Plan
CaO	16.98	%w/w	Mine Plan
FeO	7.36	%w/w	Mine Plan / Test Work Data
Fe_2O_3	9.60	%w/w	Mine Plan / Test Work Data
K_2O	1.60	%w/w	PEA II - PDC
MgO	8.66	%w/w	Mine Plan

MnO	0.62	%w/w	PEA II - PDC
Na_2O	0.24	%w/w	PEA II - PDC
P_2O_5	0.83	%w/w	PEA II - PDC
SiO_2	9.68	%w/w	Mass Balance
SrO	0.27	%w/w	PEA II - PDC
ZrO_2	270.00	ppm	Test Work Data
CO_3	34.00	%w/w	v01 Mine Plan
ThO_2	472.0	ppm	Mine Plan
UO_3	58.5	ppm	Mine Plan
REE_2O_3	0.363	%w/w	Engineering Design
Leach Conditions			
Temperature End	40	°C	Test Work/Eng. Design
Residence time	3.3	h	Test Work/Eng. Design
Acid Bake Unit			
PUG Mill			
Residence Time	1.0	H	Test Work/Eng. Design
Acid Addition Rate	925	kg/mt	
Temperature	220	°C	Test Work/Eng. Design
Solids Fraction at Discharge	81%	wt%	Mass Balance
Hollow Flight			
Residence Time	1.5	H	Test Work/Eng. Design
Temperature	300	°C	Test Work/Eng. Design
Water Leach Conditions			
Temperature	35	°C	Test Work/Eng. Design
Water addition rate	3	kg/kg WPL(s)	Test Work
Solids Fraction at Discharge	14.5%	wt%	Mass Balance
Iron Reduction Tank			
Temperature	Variable	°C	Test Work
Residence Time	1	H	Test Work
Iron Addition Rate	0.375	Stoich Fe+Ti	Test Work
Nb Precipitation Conditions			
Temperature	90-100	°C	Test Work/Eng. Design
Residence Time	4	H	Test Work
Solids Fraction at Discharge	1.2%	wt%	Mass Balance
NbP Calcination Conditions			
Temperature	950	°C	Test Work/Eng. Design
Nb Caustic Leach Conditions			
Temperature	105	°C	Test Work
Caustic Addition	1.00	kg/kg	Test Work
Caustic Solution Strength	35.00	wt%	Test Work
Dilution Ratio (Water to NaOH Solution)	5.23		Test Work
Solids Fraction at Discharge	12.6%	wt%	Mass Balance
TiP Neutralization			
Temperature	Ambient	°C	Test Work/Eng. Design
Target Acidity	15.00	gpl	Test Work
Ti Precipitation			
Temperature	100	°C	Test Work/Eng. Design

Residence Time	2	H	Test Work
TiP Calcination Conditions			
Temperature	950	°C	Test Work/Eng. Design
Sc Precipitation Iron Reduction			
Temperature	75	°C	Test Work
Residence Time	0.25	H	Test Work
85% Phosphoric Acid Addition	4.5	kg/m^3	Mass Balance
Iron Powder Addition	2.9	kg/m^3	Mass Balance
Sc Precipitation			
Temperature	75	°C	Test Work
Target pH	3.25		Test Work
Reagent Used	15.30	kg/m^3	Mass Balance
Sc Releach			
Temperature	85	°C	Test Work/Eng. Design
Target pH	0.37		Test Work
Hydrochloric Acid Strength	20%	wt%	Test Work
Sc Solvent Extraction Circuit			
Solvent Conditioning Acid Preparation Tank			
HCI Addition Ratio	0.185	$m^3(a) / m^3(o)$	Test Work
HCI Concentration	36%	wt%	Engineering Design
Scandium Extraction Mixer-Settlers			
Mixer Retention Time	3.4	min	Test Work
Overall O:A Ratio	1 : 8		Mass Balance
Internal O:A Ratio	1 : 1		Engineering Design / Test Work
Settler Retention Time	6.8		Test Work
Solvent Wash Mixer-Settler			
Hydrochloric Acid Wash Solution Strength	36%	wt%	Engineering Design / Test Work
Mixer Retention Time	19	min	Test Work
Overall O:A Ratio	1 : 6		
Internal O:A Ratio	1 : 1		
Settler retention time	68	min	
Scandium Scrub Mixer-Settlers			
Mixer retention time	16	min	Test Work
Overall O:A Ratio	1 : 4		Mass Balance
Internal O:A Ratio	1 : 1		Engineering Design / Test Work
Settler retention time	43	min	Test Work
Scandium Stripping			
Retention time	50	min	
Scandium Hydroxide Leach			
Sulphuric Acid Concentration	96	wt%	
Target Residual Sulphuric Acid Concentration	50	gpl	
Scandium Oxalate Precipitation			
Temperature	75	°C	Test Work
Residence Time	1	H	
Sulphate Conversion Conditions			

Temperature	1050	°C	Test Work
Calcium Loop Calciner Sulphate Conversion Conditions			
Temperature	1050	°C	Test Work
Tailings Neutralization			
Temperature	Ambient	°C	Test Work/Eng. Design
Residence Time	1	H	
Tailings Calcination Conditions			
Temperature	1050	°C	Test Work/Eng. Design

Source: Tetra Tech, 2017

Table 17-3: Pyrometallurgical Processing Design Criteria

Section	Description	Value	Units
Nb_2O_5 Precipitate Pelletized	Nb_2O_5 Precipitate Feed Rate (Dry Basis)	2.94	t/h
		64.7	t/d
	Moisture Content (After Pelletizing)	<1	%
	Nb Precipitate Pellets d_{80}	8	mm
Niobium Pentoxide Precipitate Composition	Nb_2O_5	30.5	%w/w
	TiO_2	63.5	%w/w
	P_2O_5	0.4	%w/w
	Al_2O_3	0.4	%w/w
Nb Precipitate Pellets	Pellets feed rate	2.94	t/h
	Number of bins	1	#
	Storage time	4.6	days
	Capacity	324	t
Aluminum (Al) pellets	Aluminum (Al) feed rate	0.52	t/h
	Number of bins	1	#
	Storage time	13	days
	Capacity	162	t
Hematite (Fe_2O_3) Pellets	Hematite (Fe_2O_3) feed rate	0.48	t/h
	Number of bins	1	#
	Storage time	13	days
	Capacity	150	t
Sodium Dioxide (Na_2O)	Feed rate	0.03	t/h
	Super sacks rack (1 Tm or 2Tm)	1	#
Limestone ($CaCO_3$)	Limestone feed rate	0.174	t/h
	Super sacks rack (1 Tm or 2Tm)	1	#
FeNb Furnace – Aluminothermic Reduction	Total Feed to FeNb Furnace	4.18	t/h
	Operating Temperature	1900	°C
FeNb Furnace Power	Electric Arc Furnace	754	kWh
	Power Consumption Per Tonne Precipitate Pellets	182	kWh/t
	Furnace Thermal Efficiency	60.0	%
	Furnace Design Power	1000	kW
	Nb Recovery	96	%
Furnace Cooling system	Water Flow Rate	64.7	m^3/h

	Cooling Tower	1	#
FeNb Furnace - FeNb Alloy Composition	Nb	63.3	%w/w
	Fe	33.2	%w/w
	Ti	0.9	%w/w
	P	0.3	%w/w
	Al	1.4	%w/w
FeNb Alloy Tapping	FeNb Alloy Flowrate	0.917	t/h
	FeNb Alloy Tapping Schedule	2	taps/12 hour shift
		4	taps/day
	FeNb Alloy Tapping Time	10.0	min/tap
	FeNb Alloy Tapping Flowrate	5.5	t/tap
		22.0	t/d
	FeNb density	8.2	t/m3
Furnace Slag Rate	Slag Average Production Rate	2.96	t/h
Furnace Slag Composition	Nb_2O_5	1.0	%w/w
	Fe_2O_3	1.1	%w/w
	FeNb (particles)	0.3	%w/w
	Fe_2O_3	0.8	%w/w
	TiO_2	62.3	%w/w
	Al_2O_3	30.9	%w/w
	CaO	3.2	%w/w
	Na_2O	1.5	%w/w
	P_2O_5	0.02	%w/w
	Slag density	4.0	t/m^3
	Slag Flowrate	2.96	t/h
	Slag Tapping Schedule	18	taps/12 hour shift
		36	taps/day
	Slag Tapping Time	15.0	min/tap
	Slag Tapping Flowrate	3.94	t/tap
		71.04	t/day
FeNb Furnace Off-gas Handling	Dusts all recycled to the furnace: Dust loss	0	%
	Generation of CO_2 (use of limestone)	0.07	t/day
FeNb Pelletizing system	Cooling water	15.1	m^3/h

Source: Metallurgy Concept Solutions, 2019

17.1.4 Acid Plant

The sulphuric acid plant will primarily be used to regenerate off gas coming from the calciner (SO_2). The design basis is described below.

Calciner-Off Gas

Hot gas from the calciner contains SO_2 and H_2SO_4 which is removed in a gas cleaning system consisting of venturi scrubbers in series. The gas is also cooled to remove water from the gas. The gas conditions at the inlet to the drying tower (battery limits) are:

- Temperature 50°C
- Pressure - 4 in. WC
- Gas Composition SO_2 14.51 vol%
- (Wet Basis) O_2 0.66 vol%

N_2 63.12 vol%

H_2O 10.69 vol%

CO_2 11.02 vol%

Overall Plant Conversion

Overall SO_2 to SO_3 conversion 99.7%

Autothermal Limit

The minimum SO_2 concentration that can be handled by the acid plant is 5 vol%.

Turndown

The plant will be capable of operating at 50% of the design capacity.

Product Acid

The product acid produced by the plant will meet the following criteria at the acid plant battery limits:

Temperature	40°C maximum
Concentration	96 wt% H_2SO_4 +/-0.5 wt%
Fe Content	50 ppm maximum

Cooling Water

Cooling Water Supply	30°C
Cooling Water Return	40°C

Instrument Air (oil-free quality)

Pressure	800 kPa
Dewpoint	-40°C
Site Barometric Pressure	97.65 kPa

17.2 Flowsheets and Process Description

17.2.1 Surface Crushing, Ore Storage & Mineral Processing Plant

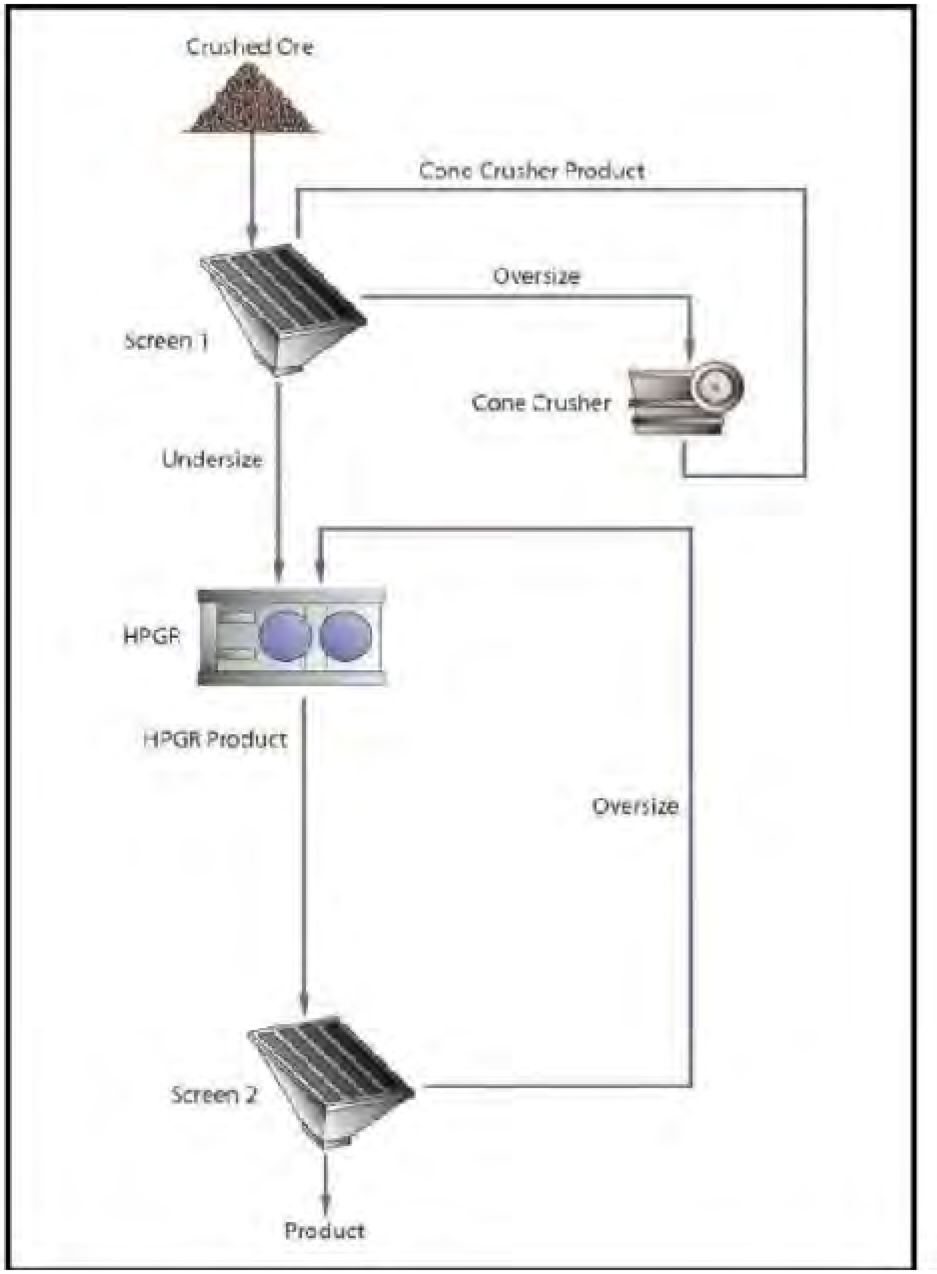
The ROM ore will be crushed underground in a primary crusher, and the crushed product with a top size of 203 mm and characteristic size (Pao) of 115 mm, will be delivered to the crushed ore bin located at the surface. The ore from the bin will be reclaimed by three vibrating feeders with a total capacity of 136 t/h and passed on to the secondary crusher circuit via the secondary crusher screen feed conveyor.

At the secondary crushing stage, the ore will be sized on a dry, double deck screen with a top deck aperture size of 50 mm and bottom deck aperture size of 25 mm. The screen oversize from both decks will report to the secondary crushing stage. The screen undersize will be conveyed to the HPGR circuit.

The screen oversize fractions will be crushed in a single secondary cone crusher operating with a closed side setting of 25 mm. The secondary crushed product will be sized by the same double deck screen with the primary crusher discharge ore.

The screen undersize, at an approximate characteristic particle size (Pao) of 22 mm, will be further crushed in the HPGR circuit. The HPGR circuit will consist of a single HPGR crusher, with a separate double-deck vibrating screen with top and bottom deck aperture sizes of 6 mm and 3 mm, respectively. The recirculating load of the HPGR circuit is expected to be in the range of 30 to 40% of the circuit new feed.

The HPGR screen undersize will be the final comminution product and is expected to have a characteristic particle size (Pao) of 1.1 mm. The ore will be stored in a fine ore bin, then reclaimed by a vibrating feeder with a design capacity of 132 t/h, and then passed on to the acid leach circuit via the acid leach feed conveyor for further processing. The HPGR conceptual block flow diagram can be seen in Figure 17-1.



Source: Tetra Tech, 2017

Figure 17-1: HPGR Conceptual Block Flow Diagram

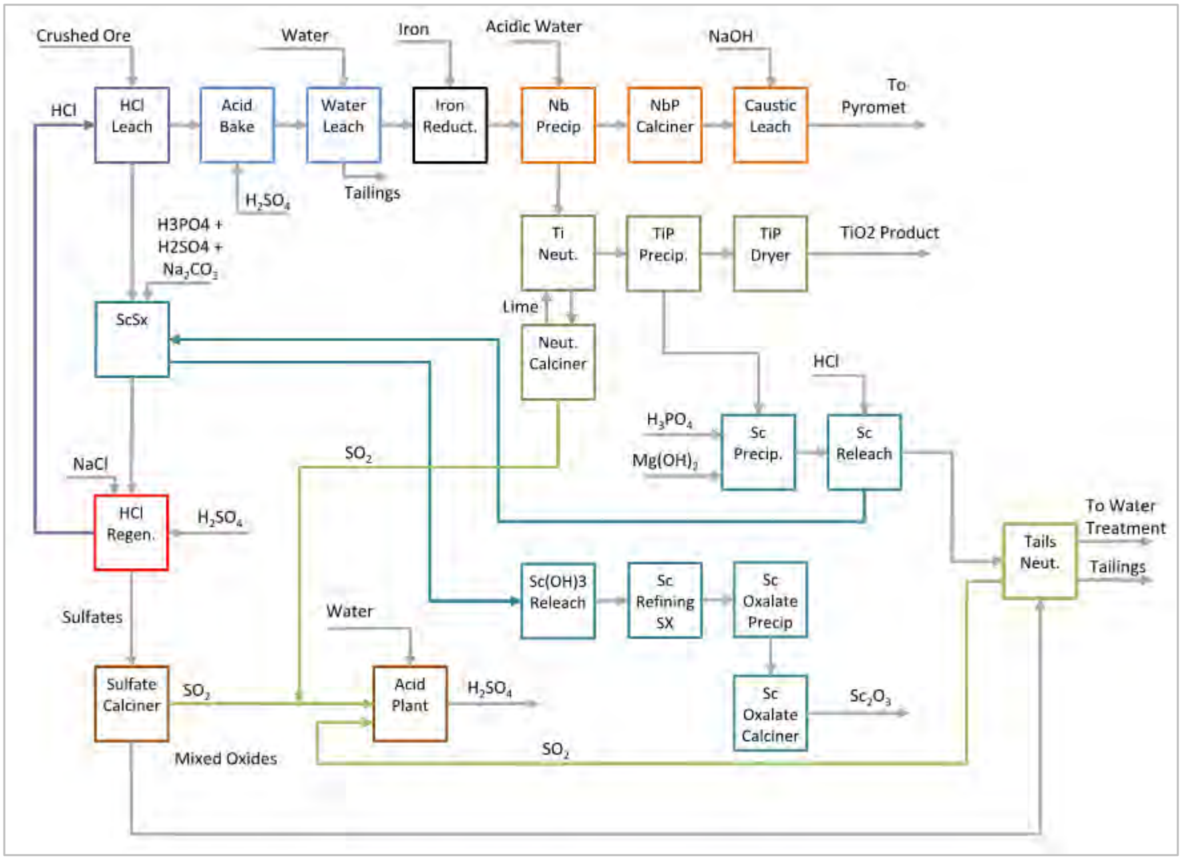
17.2.2 Hydrometallurgical Plant

The role of the hydrometallurgical (Hydromet) plant is to separate the three pay elements Nb, Ti, Sc, from the crushed ore while utilizing processes to minimize the operating cost of the plant. This requires a large amount of acid, both Hydrochloric (HCl) and Sulphuric (H_2SO_4). The Hydromet plant includes acid recovery processes to lower the operating expense of the process by requiring a small amount of fresh acid and sulphur to be brought onsite. The HCl and H_2SO_4 recoveries are 99% and 85% respectively. The other operating cost reduction comes from utilizing impurities in the ore separated out in the process as reagents in the process, which minimizes the need for fresh reagents brought onsite. The added benefit to utilizing impurities as reagents reduces the amount of tailings from the process that needs to go to the Tails Storage Facility (TSF), reducing the overall size of the storage area.

The hydrometallurgical process is divided into fifteen units:

1. Hydrochloric Acid Leach (605)
2. Sulphuric Acid Bake (610)
3. Water Leach (615)
4. Iron Reduction (620)
5. Niobium Precipitation and Phosphorus Removal (625)
6. Scandium Precipitation (628)
7. Sulphate Calcining and Mixed Oxides Handling (630)
8. Titanium Precipitation (635)
9. Scandium Solvent Extraction (640)
10. Scandium Refining (645)
11. Product Handling and Packaging (650)
12. Sulphuric Acid Plant (655)
13. Hydrochloric Acid Regeneration (660)
14. Tailings Neutralization (665)
15. Tailings Filtration (670)

The majority of the unit processes selected for the hydrometallurgical flowsheet have been extensively reported on in literature and are predominately proven and existing processes. The plant consists of multiple buildings that will house 15 separate physical and chemical processes required to separate the niobium, scandium and titanium that are contained in the ore and to regenerate and recover reagents for reuse. More details can be found in Figure 17-2 and Section 17.3.2.



Source: SMH, 2017

Figure 17-2: Simplified Sheet

Hydrochloric Acid Leach (605)

The Hydrochloric Acid Leach unit is designed to leach the majority of the impurities and the scandium present in the feed material to reduce the size of subsequent process equipment. The feed coming from the Mineral Processing Plant at a rate of 2,764 t/d (115 t/h) is fed to the HCl Leach Feed Bin. The crushed ore is then distributed using screw conveyors to two parallel trains each consisting of a primary HCl leach tank followed by two secondary HCl leach tanks. Hydrochloric acid from the Hydrochloric Acid Regeneration unit is combined with the crushed ore in each train and reacts at a controlled temperature. The discharge slurries (9 wt% solids) from each train of hydrochloric leach tanks are combined and fed to a dewatering and washing circuit consisting successively of a thickener and four parallel filter presses. The solids are washed in a series of counter current washing stages to ensure removal of the residual chloride ions that may be present in the cake moisture. The filtrate and wash liquors are combined along with the thickener overflow and sent to the PLS Aging section ahead of the Scandium Solvent Extraction unit and the Hydrochloric Acid Regeneration unit. The filter cake is sent to the Acid Bake and Water Leach unit.

The PLS Aging tank receives the combined thickener overflow, leach filtrate and wash liquor along with the secondary scandium re-leach liquor. Titanium and other minor elements contained in the PLS are oxidized with the use of an oxidizing agent, precipitated and are further separated by a clarifier. The clarified PLS is sent to the Scandium Solvent Extraction unit for scandium recovery while the solids are sent to the Titanium Precipitation unit.

Sulphuric Acid Bake (610)

The Acid Bake unit is used to convert all of the unleached metal content into sulphate compounds. The Hydrochloric Acid Leach cake is combined with pre-heated sulphuric acid and mixed in a pair of pug mills before being fed into a hollow flight screw and maintained at reaction temperature. The hollow flight provides the necessary reaction time at elevated temperature to convert the metal compounds to sulphate compounds. Off-gas from the Acid Bake is sent to a condensing column where the sulphuric acid is condensed and sent to the Hydrochloric Acid Regeneration unit. The Acid Bake discharge is continuously fed through a discharge lump breaker to the Water Leach unit.

Water Leach (615)

The Water Leach unit is used to solubilize all soluble sulphates while separating non-soluble impurities. The circuit is composed of a series of three cascading agitated tanks discharging to centrifuges. The feed is delivered to the cascading agitated tanks where it is combined with leach water. The discharge from the last cascading tank is pumped to dewatering centrifuges. The centrate, which contains the soluble sulphates, is sent to the Iron Reduction unit while the cake is transported via screw conveyor to a three-stage counter-current washing process. The Water Leach residue from the wash centrifuge is transferred by conveyor to the Paste Back Fill Plant.

Iron Reduction (620)

The Iron Reduction unit is used to reduce iron (III) sulphate ($Fe_2(SO_4)_3$) present in the solution to iron (II) sulphate ($FeSO_4$). The titanium (IV) oxysulphate $TiOSO_4$ is believed to also be reduced to titanium (III) oxysulphate ($Ti_2O(SO_4)_2$). Addition of iron solids to the solution at room temperature reduces iron and titanium compounds. In this unit, the acidic Water Leach discharge is received into the Iron Reduction column where it is contacted with iron. From this reduction column, the liquid is gravity fed to the agitated reduction tank where the reduction is completed with the addition of more iron. The discharge of the Iron reduction tank is sent to the Niobium Precipitation unit.

Niobium Precipitation and Phosphorus Removal (625)

The Niobium Precipitation unit uses water dilution (RO water) to selectively hydrolyze niobium sulphate and precipitate it as niobium oxyhydroxide. The Iron Reduction discharge is diluted with hot water, acidified with sulphuric acid, and cascaded through a series of agitated tanks. The dilution water to feed volume ratio is 0.6:1, while the sulphuric acid addition is adjusted to provide the required precipitant acid concentration. The precipitation reaction temperature is maintained at or near boiling by steam jacketed agitated tanks. The discharge of the Niobium Precipitation tanks is pumped to a clarifier. The overflow liquid is directed through a polishing filter before being forwarded to the Titanium Precipitation circuit. The solids slurry from the clarifier is pumped to two centrifuges for further dewatering. The centrate is sent to the Titanium Precipitation unit while the cake is sent to the direct fired Niobium Calciner where water is driven off, and niobium oxyhydroxide is oxidized to niobium pentoxide (Nb_2O_5) at 950°C while converting any sulphate trapped in the precipitate to an oxide thus liberating SO_2 in the off-gas. The calcination also converts a portion of the Phosphorus content to a leachable form. The calcined material is cooled in a rotary cooler and then pneumatically transported and fed via a rotary feeder into a series of cascading agitated caustic leach tanks. A Sodium hydroxide solution is added to the calcined niobium concentrate to leach Phosphorus to an acceptable residual concentration. The caustic leach discharge is pumped to another series of tube presses for dewatering and washing with RO water.

The filtrate and wash liquor are combined and sent to the Tailings Neutralization unit, while the cake is sent to a pelletizer before being fed into a sintering kiln. The discharge from the sintering kiln is then conveyed to the Pyrometallurgical plant for further processing.

Titanium Precipitation (635)

The Titanium Precipitation is achieved through hydrolysis of the titanium oxyhydroxide using heat at a reduced free acid content. The titanium rich solution from the Niobium Precipitation along with the scrubbing liquor from the Scandium Solvent Extraction are partially neutralized using fresh and recycled CaO from the Calcium Loop. The gypsum precipitate containing scandium is filtered on a vacuum belt filter and sent to the rotary kiln where it is calcined back to oxides before being recycled back to the neutralizing tanks. A portion is purged from the loop to maintain impurity levels and sent back to the Hydrochloric Acid Leach unit where the scandium and any trapped titanium is recovered. The off-gas from the Calcium Loop containing the SO_2 is combined with the off-gases from sulphate calciners, is cleaned and sent to the Acid Plant for sulphur recovery.

The titanium rich filtrate from the vacuum belt filter is heated with steam directly injected in a series of agitated tanks where titanium hydrolyzes and precipitates as titanium oxyhydroxide. The slurry is then pumped to a clarifier. The thickened slurry is then fed to tube presses for additional dewatering and washing with RO water. The overflow from the clarifier along with the filtrate and wash liquor from the tube presses are combined and sent to the Scandium Precipitation circuit. The solids from the tube presses are calcined to drive off any remaining sulphur and water to convert the titanium to TiO_2. The titanium dioxide is then sent to the packaging area where it will be loaded into super sacks and/or plastic-lined steel drums according to the client's specifications.

Scandium Precipitation (628)

The Titanium Precipitation filtrate is fed to the Scandium Precipitation unit where it is first mixed with Phosphoric acid (H_3PO_4). Iron is also added. Magnesium carbonate is used to adjust the pH to ensure the precipitation of the scandium. The slurry is pumped through a clarifier to a filter press where the liquids are separated and recycled to the clarifier. The clarifier overflow is sent to the Tailings neutralization circuit. The cake is conveyed to the Scandium Re-Leach tank where hydrochloric acid is added to re-leach the scandium. The resulting PLS is sent back to the PLS aging tank in the HCl leach circuit before being treated in the Scandium Extraction circuit.

Sulphate Calcining and Mixed Oxides Handling (630)

The sulphate calcining unit recovers sulphur from the different cakes formed throughout the process. The feed comes from two sources: the tailings neutralization filter cake and the HCl Regeneration filter cake. The cake from the Tailings Neutralization unit is processed similarly but separately from the remaining sulphate cake to maintain the desired neutralizing potential of the resulting mixed oxides.

The initial stage (Primary Kiln) of the calcination of the Acid Regeneration cake operates at a lower temperature and recovers the free H_2SO_4 acid, which is not associated with any other elements. All of the water content in the sulphate cake is also driven off in the off-gas stream. This gas stream is condensed in a nearby three-stage condensing and scrubbing circuit. The first stage condenses as much as 98% of the H_2SO_4 and recovers it in a solution containing more than 96% sulphuric acid. The second stage completes the condensation of the H_2SO_4 as 80% sulphuric acid. The last stage finalizes the scrubbing to minimize the H_2SO_4 release. The combined 96% sulphuric acid produced in the condensing circuit is sent back to the HCl Regeneration unit. The last stage of the two-stage calcining process of the Acid Regeneration cake reaches elevated temperatures to decompose all of the solid sulphates present in the kiln. Sulphur is added to the sulphate cake at this stage to compensate for sulphur losses and is burned to SO_2 during the decomposition of the sulphates. This decomposition releases SO_2 and H_2O. The sulphur dioxide gas from this kiln is combined with that of the Tails Neutralization sulphate cake calciner and scrubbed in a three-stage scrubbing circuit. The scrubbing circuit is identical to the first stage scrubbing circuit and condenses H_2SO_4 and water from the SO_2 gas before sending it to the Acid Plant for conversion back to H_2SO_4. The mixed oxides produced are pneumatically conveyed to the Tailings Neutralization as a neutralizing reagent.

The calcination of the Tailings Neutralization sulphate cake only has one stage as it contains no free H_2SO_4 acid that is not associated with any other elements. The calcination also releases SO_2 and H_2O. The sulphur dioxide gas from this kiln is combined and scrubbed as described above. The mixed oxides are sent to Tailings Neutralization via screw conveyor. Calcium loop calcined oxides are pneumatically conveyed back to Ti Neutralization and to the HCl Leach. The mixed oxides produced are conveyed to the Paste Backfill unit via belt conveyor. A portion, adjusted to minimize sulphur losses to the wastewater, is recycled back to the Tailings Neutralization for neutralization.

Scandium Solvent Extraction (640)

The Scandium Solvent Extraction unit is a four-stage D_2EHPA solvent extraction circuit followed by a wash stage, a three-stage scrubbing circuit and two-stage Stripping Circuits used to selectively recover scandium from the leach solution. The extracting organic solution is prepared in an agitated tank. The Barren organic from the stripping section is conditioned with HCl to remove any Hydroxides and to convert it from the Na+ form to its H+ form before being recycled to the Extraction section with fresh organic (small amount as required from time to time to adjust volume). The barren organic flows through the extraction mixer-settlers in series. The aqueous feed from PLS Aging is fed countercurrent to the barren organic through the extraction mixer-settlers. The scandium loads to the organic along with titanium and other elements in small amounts. The raffinate is sent to the HCl Regeneration unit while the loaded organic moves on to a single stage wash with an HCl solution to finalize the separation of aqueous impurities from the organic. The titanium and other impurities that loaded on the organic are then scrubbed in a scrubbing section with a solution containing H_2O_2 and H_2SO_4. The Sc rich, loaded organic is then sent to the Stripping Circuit. The scrubbed organic solution is combined with NaOH and agitated in a series of cascading tanks which strips the scandium of the organic and precipitates it. Separation is accomplished by a 3-phase settler where the organic, aqueous and solids are separated. The aqueous phase containing excess NaOH is moved to a holding tank and is recycled back to be re-used as strip solution. The barren organic is sent to a coalescer and is recycled to the conditioning step upstream the Extraction. The settler underflow is further refined in the Scandium Refining unit.

Scandium Refining (645)

Scandium Refining involves removing Zirconium and other impurities such as titanium and niobium from the scandium produced. The process is operated in batches. The first step is to re-dissolve the scandium rich solids in H_2SO_4 and dilute this solution down with RO water. The scandium rich solution is then contacted with a mix of two organics and diluent in a single mixer-settler tank. The Zr, Ti and Nb are loaded onto the organic, which is decanted and moved to a stripping stage while the decanted Sc rich raffinate is sent to the oxalate crystallization step. There Zr, Ti and Nb are stripped from the loaded organic with H_2SO_4 in a single mixer-settler tank. The stripped organic solution then moves on to the organic regeneration circuit to be recovered for re-use in another cycle, and the stripping solution is sent to Tailings Neutralization. The Sc rich raffinate is mixed with oxalic acid to form scandium oxalate crystals that are filtered and washed on a belt filter. The filtrate and wash liquor are sent to Tailings Neutralization while the cake is calcined to convert the solids to Sc_2O_3. The calciner discharge is transferred by screw conveyor to a screen where the final product is weighed and bagged for sale.

Sulphuric Acid Plant (655)

Gas from the calciner is delivered to the acid plant drying tower cleaned and cooled to a temperature of 50°C. The gas is deficient in oxygen, so ambient air is added to the clean gas leaving the gas cleaning system to provide sufficient oxygen for the conversion of SO_2 to SO_3 in the acid plant contact section. A minimum O_2:SO_2 ratio of 1:1 is required for efficient reaction.

The diluted gas still contains water which must be removed before the gas enters the contact section. Drying of the gas is done in the drying tower by counter current contact with concentrated sulphuric acid in the packed section. Concentrated sulphuric acid readily absorbs water from the gas. The dry gas leaves the top of the packed section and passes through a mesh pad mist eliminator before leaving the tower.

The dry gas enters the main acid plant blower, which compresses the gas for delivery through the acid plant contact section. The cold gas leaving the blower must first be heated to the catalyst ignition temperature. This is done in a series of gas-to-gas heat exchangers which transfer heat from the hot gas leaving the catalyst beds.

The contact process consists of multiple catalyst bed conversion stages with interstage gas-cooling heat exchangers, followed by two absorption stages. The conversion stages convert SO_2 to SO_3, while the absorption stages capture the SO_3 to produce concentrated sulphuric acid.

Primary conversion is obtained in the first three catalyst beds with the cooling of the process gas between each bed. The gas leaving the third catalyst bed is cooled prior to entering the intermediate absorber tower. In the absorber tower the SO_3 formed up to this stage is absorbed by counter current contact with concentrated sulphuric acid in the packed section. The gas leaving the absorber tower passes through a set of high-efficiency mist eliminators before being reheated to the Bed 4 inlet temperature.

The gas undergoes the final stage of SO_2 to SO_3 conversion in Bed 4. The removal of SO_3 in the

intermediate absorption tower results in a higher overall conversion rate in the final bed. The gas leaving the final catalyst bed is cooled before entering the final absorber tower. In the final absorber tower, the SO_3 formed in Bed 4 is absorbed by counter current contact with concentrated sulphuric acid in the packed section. The gas leaving the absorber tower passes through a set of high-efficiency mist eliminators before being discharged to the atmosphere through the acid plant stack.

The acid plant is designed for an overall conversion rate of 99.7% and a product acid concentration of 96%.

Hydrochloric Acid (HCl) Regeneration (660)

In this area, chlorides are recovered in the form of Hydrochloric Acid (HCl) for reuse in the Hydrometallurgy Process. The scandium raffinate stream contains significant concentrations of dissolved metal chloride compounds in water. Metal chlorides react with sulphuric acid to produce metal sulphates and HCl gas. The HCl gas is vented to absorber columns to recover the HCl in solution. Sulphate compounds are precipitated as solids in a sulphuric acid solution and recovered by filtration.

The process contains a sulphuric acid recirculation loop. Sulphuric acid is added to the Precipitators and collected in Filtrate Tanks after the solids are filtered. The sulphuric acid from these tanks is heated and returned to the HCl Regen Reactors

Water interferes with precipitation of sulphate solids and must be removed along with HCl. Two water removal steps are provided with an initial feed vacuum flash and a second post-reaction vacuum flash unit.

Tailings Neutralization (665)

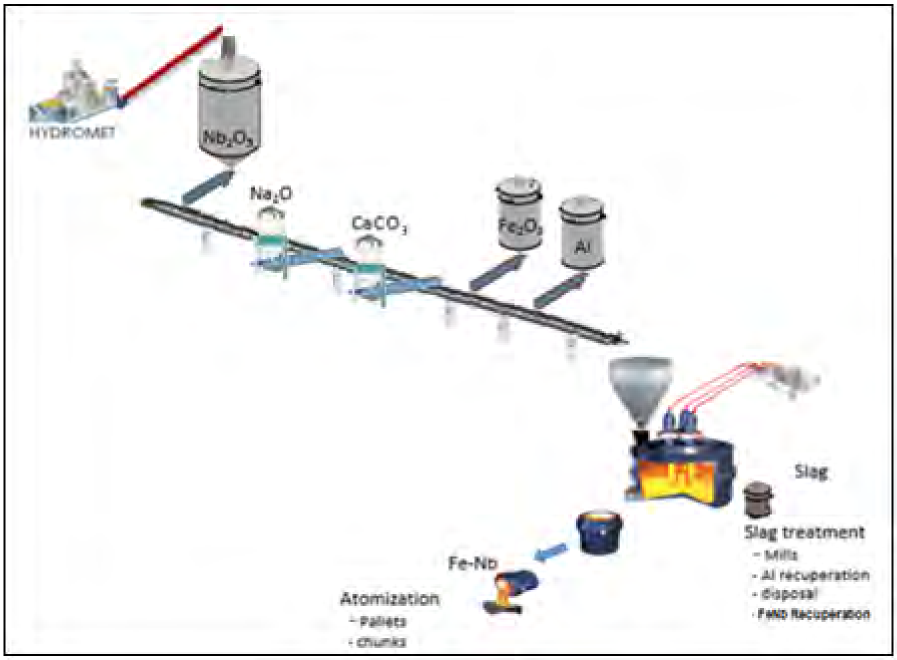
The Tailings Neutralization unit is fed by the centrate and filtrate from the Titanium Precipitation unit as well as other acidic tailings streams. The combined feed is reacted with recycled mixed oxides in a series of agitated tanks in order to raise the pH to around 9.5. The discharge slurry is pumped to Tailings Handling.

Tailings Handling (670)

The discharge slurry from Tailings Neutralization is successively dewatered in a thickener and filter presses. The filtrate is returned to the thickener, the filter cake is sent to the sulphate calciner, and the thickener overflow is sent to water treatment.

17.2.3 Pyrometallurgical Plant

The high-level pyromet plant flowsheet is presented in Figure 17-3. The selected process is based on the aluminothermic reduction of niobium pentoxide (Nb_2O_5) present in the hydromet precipitate. The dry Nb_2O_5 precipitate pellets are fed by conveyor to the Furnace Feed Preparation Area (FPA), and stored in a closed bin, giving a total of five days storage time. Aluminum grains and hematite Fe_2O_3 pellets to supply iron units are also stored in feed bins with a six-day storage capacity for each component. The additives and reductants complete for the aluminothermic reaction recipe, where super sacks will be used to handle these products. The three usage bins are loaded by conveyor while an overhead crane will be used to replace the empty super sacks.



Source: Tetra Tech, 2017

Figure 17-3: Pyrometallurgical Processing Simplified Flowsheet

Six flow control bin/sacks discharge for aluminum, hematite, limestone and sodium oxide provide measured feed to the FeNb Furnace. According to the need, FeNb metal off-spec material can also be recycled along with the Nb_2O_5 feedstock from the hydromet. All the conveyors under the feed material storage units are on load-cells, as part of the furnace feed preparation mass measurement system which is automatically controlled via PLC.

The furnace feed preparation is performed as a continuous process with specified mass measurement of the Nb_2O_5 precipitate pellets with the required aluminum, hematite, and fluxes to satisfy a "recipe" to produce on-spec FeNb alloy (ratio Fe/Nb = 0.35/0.65). Each ingredient is fed onto the furnace feed conveyor at a pre-determined rate to provide a continuous charge to the furnace.

This allows tight control on continuous feed of the mixed charge into the furnace, to maintain furnace levels of slag and metal alloy. Furnace feeding will be stopped briefly for the tapping of both molten slag and FeNb alloy, according to levels of slag and metal in the furnace.

The tapping of slag and FeNb metal is scheduled over two 12 hour shifts:

1. Slag: 18 taps x per 12-hour shift, 15-minute tapping duration. 1.81 t per tap.
2. FeNb metal: 2 taps x per 12-hour shift, 10-minute tapping duration. 5.5 t per tap.

A tapping drill and clay gun unit is used to open each slag and metal tap-hole and plug each tap-hole with clay after the tap is complete. A molten heel or pool of metal is left remaining in the furnace, with some slag layer covering the metal. This is carried out according to measured furnace levels with the slag and metal masses, providing ongoing control and continuous operation of the furnace. The FeNb furnace will be operated at a temperature in the range of 1850 to 1900°C.

Electrical energy is supplied to the furnace to initiate and maintain heat input into the furnace to complete the reduction of Nb_2O_5 and Fe_2O_3. Aluminum is the primary reductant and on oxidation to Al_2O_3 forms a large part of the slag system with TiO_2, fluxed with limestone ($CaCO_3$) and sodium oxide (Na_2O).

The Al_2O_3-TiO_2-CaO type slag produced in the furnace is tapped into steel molds multiple times (18 times/shift) each shift where it can cool before being moved to a storage bunker area. The steel molds are sectioned so that the slag is easily removed from the mold and is in small, easily crushed pieces. The molds are reused. The slag is moved from the loadout bunker with a front-end loader (FEL) to the vibrating feeder that feeds the Slag Jaw Crusher. The crushed slag is then treated by gravity separation to recuperate the FeNb particles stuck in the slag. The remaining slag is transferred to the tailing's impoundment.

The FeNb alloy metal is tapped via a short launder into the FeNb pelletizing pan where the molten droplets will solidify in a cold water basin to form particles ranging in size from approximately 6 mm to 15 mm. The cooled FeNb pellets will then be removed from the basin by a pocket conveyor and transferred to a rotary dryer where the moisture will be driven off prior to screening and packaging. Undersize FeNb pellets are collected and sent to the off-spec feed bin to be reintroduced into the EAF.

Dust from the FeNb Furnace Feed Preparation Area are captured via ducting through a dry cyclone — bag-house system. All dust from this area are returned to the FPA and placed in a separate bin. As required, according to the furnace charge mix recipe, these fines are bled back into the furnace charge for smelting.

The FeNb furnace off-gas, since it contains a fraction of SO2, will be sent to the sulfuric acid for recovery. The slag and metal tapping fumes, and casting fumes above each mold are captured and ducted to the furnace off-gas baghouse. Baghouse dust is recycled to the EAF or the Pelletizer.

Both the Feed Preparation and dust collectors cleaned air exhausts are ducted respectively to their own exhaust stack. Each air exhaust duct may be monitored by sampling to meet environmental regulations.

17.2.4 Acid Plant

Gas from the calciner is delivered to the acid plant drying tower cleaned and cooled to a temperature of 50° C. The gas is deficient in oxygen, so ambient air is added to the clean gas leaving the gas cleaning system to provide sufficient oxygen for the conversion of SO_2 to SO_3 in the acid plant contact section. The diluted gas still contains water which must be removed before the gas enters the contact section. Drying of the gas is done in the drying tower by counter current contact with concentrated sulphuric acid in the packed section. Concentrated sulphuric acid readily absorbs water from the gas. The dry gas leaves the top of the packed section and passes through a mesh pad mist eliminator before leaving the tower.

The contact process consists of multiple catalyst bed conversion stages with interstage gas-cooling heat exchangers, followed by two absorption stages. The conversion stages convert SO_2 to SO_3, while the absorption stages capture the SO_3 to produce concentrated sulphuric acid.

The acid plant is designed for an overall conversion rate of 99.7% and a product acid concentration of 96%.

17.3 Mass Balances

17.3.1 Surface Crushing, Ore Storage & Mineral Processing Plant

The mass balance for the comminution circuit can be seen in Table 17-4.

Table 17-4: Comminution Circuit Mass Balance

Description	Primary Crusher Feed	Secondary Crusher Screen Feed	Secondary Screen Fines	Secondary Screen Coarse	Secondary Crusher Product	HPGR Product	HPGR Screen Coarse	HPGR Screen Fines	To Shuttle Conveyor	HCl Leach Feed
Solids (t/h)	139.30	310.64	139.30	171.34	171.34	197.81	58.51	139.30	139.30	125.00
Liquid (t/h)	7.17	15.96	7.17	8.80	8.80	10.25	3.08	7.17	7.17	6.35
Gas (t/h)	-	-	-	-	-	-	-	-	-	-
Density (t/m^3)	2.70	2.70	2.70	2.70	2.70	2.70	2.70	2.70	2.70	2.70
Volume (m^3/h)	54.23	120.91	54.23	66.68	66.68	77.07	22.84	54.23	54.23	48.58
Nb (t/h)	0.78	1.74	0.78	0.96	0.96	1.11	0.33	0.78	0.78	0.70
Sc (t/h)	0.01	0.02	0.01	0.01	0.01	0.01	0.00	0.01	0.01	0.01
Ti (t/h)	2.33	5.19	2.33	2.86	2.86	3.30	0.98	2.33	2.33	2.09

Source: Tetra Tech, 2017

17.3.2 Hydrometallurgical Plant

Based on the design criteria and the flowsheet, a mass balance for the hydrometallurgical processing plant has been developed. The mass balance was prepared for an average feed of 2,764 t/d or 115 t/h operating 332 days per year at 0.8% Nb_2O_5. The mass balance for the plant was calculated to provide tonnages and flow rates to different sections and equipment in the plant. The mass balance was designed, and a model of the hydrometallurgical plant was done using the flowsheet integrator METSIM. A summarized mass balance can be found in Figure 17-4.

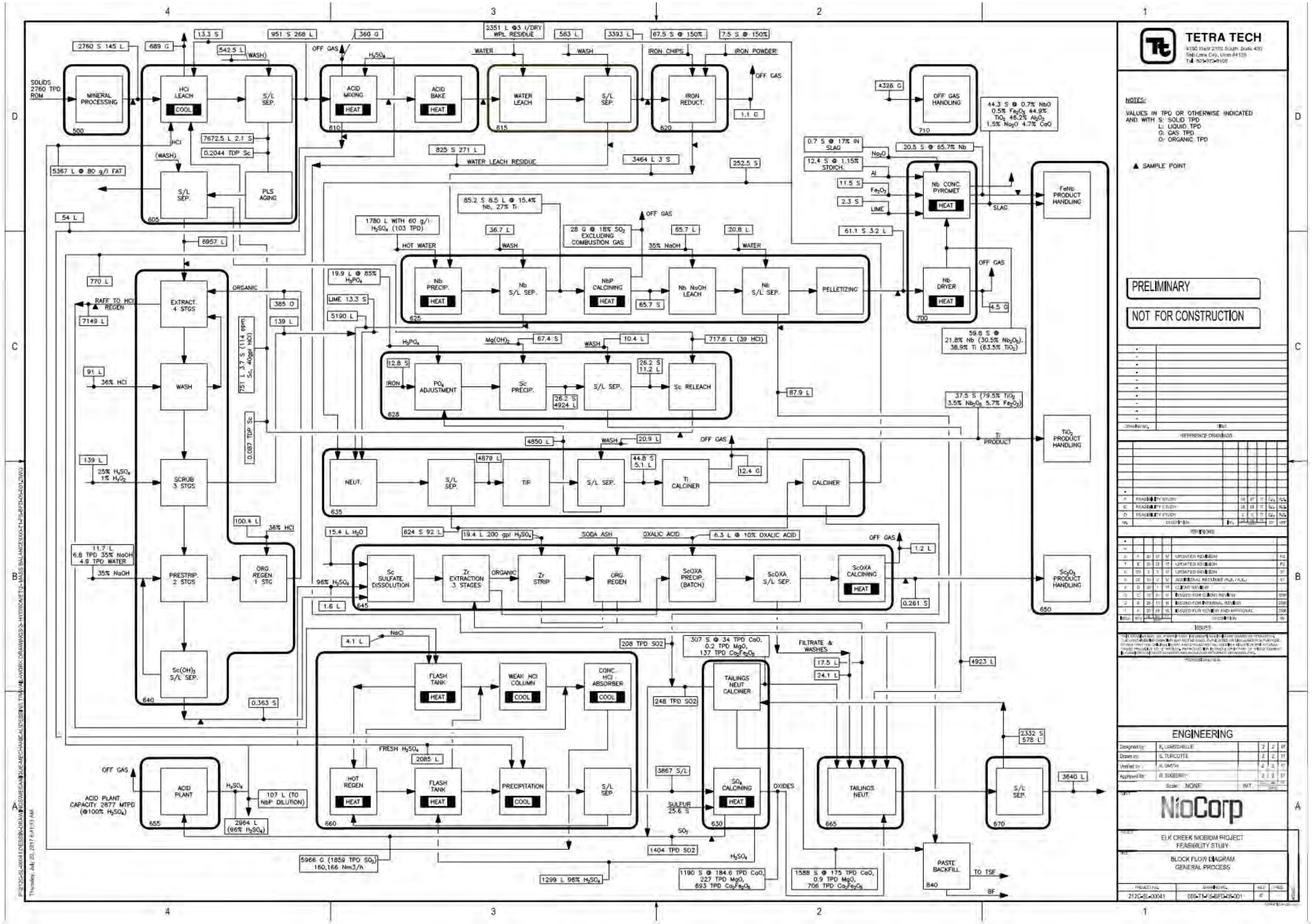


Figure 17-4: Block Flow Diagram and Summary Mass Balance

17.3.3 Pyrometallurgical Plant

Based on the design criteria and the pyrometallurgical flowsheet, a mass balance model with energy requirements was developed for the pyromet plant. The mass balance was prepared for an average feed rate of 64.7 t/d (dry basis) or 2.94 t/h at 30.7% Nb_2O_5 with a 91.7% overall plant availability. The mass balance for the pyrometallurgical plant was calculated to provide tonnages and flow rates to different sections and equipment in the plant. A partition coefficient method was used to define the split of elements between furnace slag and FeNb metal alloy. These partition coefficients were estimated based on KPM test work, slag and alloy chemistry (thermodynamic & literature, supported by data from other FeNb alloy industry operations).

The major element partition coefficients defining the mass balances are shown in Table 17-5.

Table 17-5: FeNb Furnace Partition Coefficients

Element	To Metal	To Slag
	%	%
Nb	96.0	4.0
Fe	95.0	5.0
Ti	1.4	98.6
P	42.6	57.4
Na	-	100.0
Si	56.1	43.9
Al	4.1	95.9
Mg	-	100.0
Ca	-	100.0
K	-	100.0

Source: Metallurgy Concept Solutions, 2019

From Table 17-5, the Nb recovery in the pyrometallurgical process plant is targeted at 96%, given the assumptions made in the design criteria. For this study, it is assumed that all dust and metal fines with Nb units in fumes are collected and recycled to the FeNb furnace. This includes Nb bearing dust and fume from the Feed Preparation Area, FeNb Furnace Off-gas, Tapping & Casting, and FeNb Crushing and Screening areas.

The FeNb metal production was calculated at 19.4 t/d, corresponding to 7,096 t/yr average with an availability of 91.7% and considering a grade of 0.35Fe - 0.65Nb ratio. However, the pyromet has been designed to process an extra capacity of 25% precipitate in order to support peak production rate and/or to recover shutdown time periods that can occur during the year.

The slag rate output is estimated at 61.6 t/d, corresponding to 22.4 Mt/y (same availability), with an estimated Nb_2O_5 grade of 1% not transformed.

17.3.4 Acid Plant

The acid plant mass balance is shown in Table 17-6.

Table 17-6: Acid Plant Mass Balance

Annual Acid Requirement			Mt/y	1,038,425
Operating Days			Days	365
Daily Production				
	as 100%	H_2SO_4	Mt/d	2,845
	as 96%	H_2SO_4	Mt/d	2,964
		Inlet Gas		
		%	Nm^3/h	
SO_2		14.51	21,933.84	
O_2		0.66	997.68	
N_2		63.12	95,414.46	
H_2O		10.69	16,159.39	
CO_2		11.02	16,658.23	
		100	151,163.60	
SO_2 Conversion		%	99.7	
Acid Production		Mt/d	2,964	
Operating Days			365	
Annual Production		Mt/a	1,038,425	

Source: Tetra Tech, 2017

17.4 Process Equipment

17.4.1 Surface Crushing, Ore Storage & Mineral Processing Plant

The primary equipment list (see Table 17-7) and the ancillary equipment list (see Table 17-8) for the comminution area were prepared based on the process design criteria. The installed power of the major equipment determined during the process design is shown in Table 17-7.

Although the ancillary equipment list for the comminution area is shown in this report for completeness, the associated installed power is not determined as part of the process design process. The installed motor power of ancillary equipment is reported as provided by the Qualified Person for materials handling design.

Table 17-7: Primary Equipment List

Comminution Circuit Primary Equipment	No. of Units	Unit Installed Power (kW)	Total Installed Power (kW)
Double Deck Vibrating Screen	1	5.5	5.5
Secondary Crusher (Metso HP300 or equivalent)	1	200	200
High Pressure Grinding Rolls (Polycom 14/08 - 02 or equivalent)	1	1,000	1,000
HPGR Product Double Deck Screen	1	37.5	37.5

Source: Tetra Tech, 2017

Table 17-8: Ancillary Equipment List

Comminution Circuit Ancillary Equipment	No. of Units	Unit Installed Power (kW)	Total Installed Power (kW)
Crushed Ore Bin Vibrating Feeder	3	3.75	11.25
Secondary Crusher Screen Feed Conveyor	1	75	75
Secondary Crusher Recycle Conveyor	1	15	15
HPGR Feed Conveyor	1	11.5	11.5
HPGR Screen Feed Conveyor	1	11.5	11.5
HPGR Recycle Conveyor	1	22	22
Fine Ore Bin Feed Conveyor	1	30	30
Fine Ore Bin	1	Not Applicable	Not Applicable
Fine Ore Bin Vibrating Feeder	3	3.75	11.25
Fine Ore Conveyor	1	30	30
Conveyor Scales	1	Not Determined	Not Determined
Tramp Magnet	1	3.75	3.75

Source: Tetra Tech, 2017

17.4.2 Hydrometallurgical Plant

The equipment list of the Hydrometallurgical Plant was developed based on the design criteria and using the mass balance provided by the METSIM model. Table 17-9 provides a summarized list of equipment for reference in this report. A more detailed list and sizing were used in the capital cost estimate.

Table 17-9: Summarized List of Equipment

Equipment	Equipment Name	Qty
600-BOL-001	BOILER	1
600-OCR-001	BRIDGE CRANE 20 TON CAPACITY	2
605-BIN-001	HCI LEACH FEED BIN	1
605-BIN-002	DISCHARGE BIN	1
605-CVO-001	ACID MIXER BIN FEED CONVEYOR	1
605-CVO-002	HCI LEACH RESIDUE FEED CONVEYOR	4
605-FPR-001	HCI LEACH RESIDUE FILTER PRESS	4
605-HTX-001	HEAT EXCHANGER	1
605-SCC-001	HCI LEACH FEED SCREW CONVEYOR	2
605-SLP-001	HCI LEACH RESIDUE THICKNER PUMP	4
605-SLP-003	HCI LEACH RESIDUE PUMP	3
605-SLP-005	CLARIFIER PUMP	2
605-SOP-003	INTERMEDIATE WASH PUMP	1
605-SOP-005	FLOCC MIX TANK PUMP	1
605-SOP-006	INTERMEDIATE WASH PUMP	1
605-SOP-103	INTERMEDIATE WASH PUMP	1
605-SOP-105	FLOCC MIX TANK PUMP	1
605-SOP-106	INTERMEDIATE WASH PUMP	1
605-TAK-001	HCI LEACH TANK	6
605-TAK-005	HCI LEACH RESIDUE SURGE TANK	1
605-TAK-006	INTERMEDIATE WASH TANK	2
605-TAK-009	HCI RESIDUE FEED TANK	1
605-TAK-010	PLS AGING TANK	1
605-TAK-011	TANK CLARIFIER	1
605-THK-001	HCI LEACH RESIDUE THICKNER	1
610-BIN-001	PUG MILL FEED BIN	1
610-BOL-001	PUG MILL HEATER	1
610-CON-001	ACID CONDENSING COLUMN	1
610-HTX-002	ACID PLATES HEAT EXCHANGER	1
610-PUG-001	PUG MILL	3
610-SCC-001	PUG MILL FEED SCREW	2
610-SCR-001	ACID CONDENSING SCRUBBER	1
610-SLP-001	PUG MILL ACID FEED PUMP	2
610-SOH-001	ACID FEED TANK HEATER	1
610-SOP-002	SPENT SCRUBBER SOLUTION PUMP	2
610-SOP-005	ACID CONDENSING COLUMN DISCHARGE PUMP	2
610-TAK-001	ACID BAKE ACID FEED TANK	1
615-CEN-001	WL CENTRIFUGE	2
615-CEN-003	WL RESIDUE WASHING CENTRIFUGE	3
615-PBO-001	WL CENTRATE PUMPBOX	1
615-SCC-001	WL CENTRIFUGE SCREW CONVEYOR	2
615-SCC-003	WL RESIDUE SCREW CONVEYOR	3
615-SLP-001	WL RESIDUE PUMP	2
615-SLP-003	WL RESIDUE WASHING PUMP	6
615-SOP-001	WL CENTRATE PUMP	2
615-TAK-001	WATER LEACH TANK	2
615-TAK-003	WATER LEACH TANK DECANT FEED TANK	1
615-TAK-004	WL RESIDUE WASHING REPULP TANK	3
620-BIN-001	Fe POWDER BIN	1
620-PAC-001	IRON REDUCTION COLUMN	1
620-SLP-003	IRON REDUCTION SOLUTION TRANSFER PUMP	2
620-TAK-003	IRON REDUCTION TANK	1
625-BIN-001	NbP ROTARY COOLER DISCHARGE BIN	1

625-BIN-003	NbP PELLET STORAGE	1
625-BIN-004	NbP ROTARY COOLER FEED BIN	1
625-CAL-001	NbP CALCINER	1
625-CEN-001	Nb PRECIPITATION CENTRIFUGE	2
625-CVO-001	NbP TUBE PRESS DISCHARGE CONVEYOR	2
625-DIL-001	DILUTION SKID	1
625-HPP-001	TUBE PRESS HYDRAULIC POWER PACK	2
625-HPP-002	TUBE PRESS HIGH PRESSURE PUMP SYSTEM	2
625-HVS-001	TUBE PRESS HYDRAULIC VACUUM SYSTEM	2
625-ILF-001	NbP CLARIFIER 0/F AUTOMATIC BACKWASH FILTER	1
625-KLN-001	NbP SINTERING KILN	1
625-LPP-001	TUBE PRESS LOW PRESSURE PUMP SYSTEM	2
625-PBO-001	NbP CLARIFIER 0/F PUMPBOX	1
625-PBO-003	TUBE PRESS FILTRATE PUMPBOX	1
625-PEL-001	NbP PEL MIXER	1
625-RTC-001	NbP ROTARY COOLER	1
625-SLP-001	NbP CLARIFIER FEED PUMP	2
625-SLP-002	NbP CLARIFIER U/F PUMP	2
625-SLP-003	NbP CLARIFIER 0/F PUMP	2
625-SLP-005	NbP TUBE PRESS FEED PUMP	2
625-SLP-006	NbP CAUSTIC LEACH TUBE PRESS FEED PUMP	2
625-SLP-110	NbP CENTRIFUGE FEED PUMP	1
625-SOP-001	TiP NEUTRALISATION FEED PUMP	2
625-SOP-004	NbP CAUSTIC LEACH TUBE PRESS FILTRATE PUMP	2
625-TAK-001	NbP TANK	4
625-TAK-005	NbP CLARIFIER FEED TANK	1
625-TAK-007	NbP TUBE PRESS FEED TANK	1
625-TAK-008	NbP CAUSTIC LEACH TANK	2
625-TAK-010	NbP CAUSTIC LEACH FEED TANK	1
625-TAK-011	NbP CLARIFIER	1
625-TAK-012	TiP CENTRIFUGE FEED TANK	1
625-TAK-013	NbP TANK	2
625-TAK-015	NbP SOLUTION BUFFER TANK	1
625-TUF-001	NbP TUBE PRESS	3
625-TUF-004	NbP CAUSTIC LEACH TUBE PRESS	3
628-BIN-001	NgCO3 Bin	1
628-CVO-001	Sc FILTER PRESS COLLECTION CONVEYOR	1
628-FPR-001	Sc FILTER PRESS	1
628-SLP-001	Sc RE-LEACH PUMP	1
628-SLP-002	PO4 ADJUSTMENT PUMP	1
628-SLP-003	Sc PRECIP PUMP	2
628-SLP-005	Sc PRECIP CLARIFIER U/ PUMP	2
628-SLP-006	FILTER FEED TANK PUMP	2
628-SLP-101	Sc RE-LEACH PUMP	2
628-SLP-102	PO4 ADJUSTMENT PUMP	2
628-TAK-003	Sc PRECIP TANK	1
628-TAK-004	Sc PRECIP SLURRY TANK	1
628-TAK-005	TANK	1
628-TAK-006	TANK	1
628-TAK-007	Sc PRECIP CLARIFIER	1
628-TAK-008	FILTER FEED TANK	1
630-BIN-001	DISCHARGE BIN	1
630-BIN-002	DISCHARGE BIN	1
630-BIN-003	DISCHARGE BIN	1
630-BIN-005	DISCHARGE BIN	1

630-CON-002	ACID CONDENSING COLUMN	3
630-CON-004	VENTURI SCRUBBER	1
630-CYC-001	CYCLONE	3
630-CYC-004	TAILS NEUT KILN CYCLONE	1
630-HTX-001	ACID PLATES HEAT EXCHANGER	4
630-ILF-001	VENTURI SCRUBBER FILTER	1
630-KLN-002	SECONDARY KILN	3
630-KLN-007	TAILS NEUT KILN	1
630-SCC-001	SCREW CONVEYOR DRYER	3
630-SCC-005	DISCHARGE BIN SCREW CONVEYOR	1
630-SCC-006	MIXED OXIDE SCREW CONVEYOR	3
630-SCR-001	ACID CONDENSING SCRUBBER	2
630-SOP-005	ACID CONDENSING COLUMN DISCHARGE PUMP	2
630-SOP-008	ACID CONDENSING SCRUBBER DISCHARGE PUMP	4
630-SOP-009	VENTURI SCRUBBER DISCHARGE PUMP	2
630-SOP-010	ACID CONDENSING COLUMN DISCHARGE PUMP	4
635-BIN-003	KILN DISCHARGE STORAGE BIN	1
635-BIN-004	WPL FEED STORAGE BIN	1
635-BIN-005	LIME POWDER FEED BIN	1
635-BIN-001	PRODUCT STORAGE BIN	1
635-BIN-002	TiP NEUT FEED BIN	1
635-CVO-001	TiP TUBE PRESS DISCHARGE CONVEYOR	1
635-CVO-002	TiP NEUT PRESS DISCHARGE CONVEYOR	1
635-CYC-001	CYCLONE	1
635-DRM-001	OVERSIZE DRUM	1
635-DRY-001	TiP DRYER	1
635-DUC-001	DUST COLLECTOR	1
635-FPR-001	TiP NEUTRALIZATION FILTER PRESS	1
635-HPP-001	TUBE PRESS HYDRAULIC POWER PACK	1
635-HPP-002	TUBE PRESS HIGH PRESSURE PUMP SYSTEM	1
635-HVS-001	TUBE PRESS HYDRAULIC VACUUM SYSTEM	1
635-ILF-001	TiP CLARIFIER 0/F AUTOMATIC BACKWASH FILTER	1
635-KLN-001	TiP NEUT KILN	1
635-LPP-001	TUBE PRESS LOW PRESSURE PUMP SYSTEM	1
635-PBO-001	TiP CLARIFIER 0/F PUMPBOX	1
635-PNC-001	PNEUMATIC CONVEYING SYSTEM	2
635-SCC-002	KILN DISCHARGE SCREW CONVEYOR	1
635-SCC-001	TiP ROTARY COOLER DISCHARGE SCREW CONVEYOR	1
635-SCN-001	SWECO SCREEN 80 MESH	1
635-SLP-001	TiP CLARIFIER FEED PUMP	2
635-SLP-002	TiP CLARIFIER U/F PUMP	2
635-SLP-005	TiP CLARIFIER 0/F PUMP	2
635-SLP-007	TiP TUBE PRESS FEED PUMP	2
635-SLP-008	TiP NEUT TANK PUMP	2
635-SOP-001	Sc PHOSPHATE PRECIP FEED PUMP	2
635-SUP-001	TiP SUMP PUMP	2
635-TAK-001	TiP NEUTRALIZATION TANK	2
635-TAK-002	TiP TANK	2
635-TAK-007	TiP CLARIFIER FEED TANK	1
635-TAK-010	TiP TUBE PRESS FEED TANK	1
635-TAK-011	TiP CLARIFIER	1
635-TAK-013	TiP SOLUTION BUFFER TANK	1
635-TUF-001	TiP TUBE PRESS	3
640-CLS-001	STRIPPED ORGANIC COALESCER	1
640-SEX-001	EXT MIXER SETTLER	4

640-SEX-005	ORG WASH MIXER SETTLER	1
640-SEX-006	SCRUB MIXER SETTLER	3
640-SLP-001	AQUEOUS PUMP	2
640-SLP-002	NaOH PRECIPITATION PUMP	2
640-SLP-003	3-PHASE SETTLER UNDERFLOW PUMP	2
640-SLP-004	SCANDIUM PRECIPITATION PUMP	2
640-SLP-007	COALESCER TANK PUMP	2
640-SOP-001	BARREN ORGANIC PUMP	2
640-SOP-002	ACID MIX TANK PUMP	2
640-SOP-003	AQ TANK PUMP	3
640-SOP-007	Sc SX FEED PUMP	2
640-SOP-009	AQUEOUS PUMP	2
640-SOP-010	STRIPPED ORGANIC PUMP	2
640-SOP-012	ORG TANK PUMP	2
640-SOP-014	COALESCER PUMP	2
640-SOP-112	ORG TANK PUMP	2
640-TAK-001	BARREN ORGANIC HOLDING TANK	1
640-TAK-002	EXT ACID MIX TANK	1
640-TAK-003	AQ TANK	1
640-TAK-007	Sc SX FEED TANK	1
640-TAK-008	ORG TANK	1
640-TAK-009	COALESCER TANK	1
640-TAK-010	SCANDIUM PRECIPITATION TANK	2
640-TAK-012	STRIPPED ORGANIC HOLDING TANK	1
640-TAK-013	NaOH PRECIPITATION TANK	1
640-TPS-001	3-PHASE SETTLER	1
645-BEF-001	Sc OXALATE BELT FILTER	1
645-BIN-001	PRODUCT STORAGE BIN	1
645-CAL-001	Sc CALCINER	1
645-CVO-001	Sc OXALATE TRANSFER CONVEYOR	1
645-DRM-001	OVERSIZE DRUM	1
645-PSC-001	PLATFORM SCALE	1
645-SCN-001	SWECO SCREEN 80 MESH	1
645-SLP-002	Sc PRECIPITATION FEED PUMP	2
645-SLP-003	Sc PRECIPITATION TRANSFER PUMP	2
645-SLP-004	Sc PRECIPITATION WASH PUMP	2
645-SLP-005	Sc FILTER FEED PUMP	2
645-SLP-006	FILTRATE PUMP	2
645-SLP-008	Sc DISSOLUTION LIQUOR PUMP	1
645-SLP-009	ZR STRIPPING FEED PUMP	1
645-SLP-010	ORGANIC REGEN FEED PUMP	1
645-SLP-011	ORGANIC REGEN DISCHARGE PUMP	1
645-SLP-012	ORGANIC BARREN PUMP	1
645-SLP-013	Sc LIQUOR PUMP	1
645-SLP-014	STRIP LIQUOR PUMP	1
645-SLP-015	Na2CO3 MIXING TANK PUMP	1
645-TAK-002	Sc PRECIPITATION FEED TANK	1
645-TAK-003	Sc OXALATE PRECIPITATION TANK	1
645-TAK-004	Sc OXALATE FILTER FEED TANK	1
645-TAK-009	Sc DISSOLUTION LIQUOR TANK	1
645-TAK-010	ZR LOADING FRP TANK	1
645-TAK-011	ZR STRIPPING FRP TANK	1
645-TAK-012	ORGANIC REGEN TANK	1
645-TAK-013	ORGANIC/AQU SEPERATION TANK	1
645-TAK-014	Na2CO3 MIXING TANK	1

660-TK-90 A-C	HCI REGEN FEED TANKS	3
660-H-90	SALT HOPPER	1
660-TK-100 A-B	FEED WATER FLASH TANK	2
660-AG-100 A-B	FEED WATER FLASH TANK AGITATORS	2
660-R-110 A-C	HCI REGEN REACTORS	3
660-AG-110 A-C	HCI REGEN REACTORS AGITATORS	3
660-TK-115 A-C	VACUUM FLASH TANKS	3
660-AG-115 A-C	VACUUM FLASH TANKS AGITATORS	3
660-TK-120 A-C	PRECIPITATORS	3
660-AG-120 A-C	PRECIPITATORS AGITATORS	3
660-TK-125	FILTRATE TANK	1
660-TK-105	WEAK HCI DISTILLATE DRUM	1
660-TK-135	CONCENTRATED HCI RECEIVER	1
660-C-105 A-B	WEAK HCI ABSORBER COLUMN	2
660-C-105 A-B	WEAK HCI ABSORBER INTERNALS	2
660-C-130 A-B	CONCENTRATED HCI ABSORBER COLUMN	2
660-C-130 A-B	WEAK HCI ABSORBER INTERNALS	2
660-HX-100 A-B	FEED FLASH HEAT EXCHANGERS	2
660-HX-120 A-C	PRECIPITATOR COOLERS	3
660-HX-125 A-D	SULPHURIC ACID RECYCLE HEATERS	4
660-HX-105 A-R	WEAK ACID CONDENSERS	18
660-HX-110 A-B	HCI REGEN CONDENSERS	2
660-HX-130 A-H	CONCENTRATED ACID CONDENSERS	8
660-P-90 A-B	HCI REGEN FEED PUMPS	2
660-P-100 A-B	FEED FLASH PUMPS	4
660-P-110 A-C	HCI REGEN REACTOR PUMPS	6
660-P-115 A-C	VACUUM FLASH PUMPS	6
660-P-120 A-C	PRECIPITATOR PUMPS	6
660-P-125 A-B	FILTRATE PUMP	2
660-P-106 A-C	WEAK HCI ABSORBER COLUMN PUMPS	3
660-P-105 A-B	WEAK HCI ABSORBER COLUMN REFLUX PUMPS	2
660-P-130 A-C	CONCENTRATED HCI ABSORBER PUMPS	3
660-P-135 A-B	CONCENTRATED HCI RECEIVER PUMPS	2
660-F-125 A-B	SOLIDS FILTER	2
660-VP-105 Al, A2	VACUUM PUMP	2
660-ED-90	SALT EDUCTOR	1
660-CVO-1	CONVEYOR #1	1
660-CV0-2	CONVEYOR #2	1
660-CV0-3	CONVEYOR #3	1
660-CV0-4	CONVEYOR #4	1
665-BIN-001	TAILS NEUTRALIZATION FEED BIN	1
665-BIN-002	TAILS NEUTRALIZATION FEED BIN	1
665-ROV-001	ROTARY FEEDER	1
665-ROV-002	ROTARY FEEDER	1
665-SLP-002	TAILS NEUTRALIZATION PUMP	2
665-TAK-002	TAILS NEUTRALIZATION FEED TANK	1
665-TAK-003	TAILS NEUTRALIZATION TANK	2
670-CVO-003	TAILS FILTER PRESS DISCHARGE CONVEYOR	2
670-FPR-001	TAILS FILTER PRESS	2
670-SLP-001	TAILS THICKENER 0/F PUMP	2
670-SLP-002	TAILS THICKENER U/F PUMP	2
670-SLP-003	TAILS FILTER PRESS FEED PUMP	2
670-TAK-001	TAILS THICKENER 0/F TANK	1
670-TAK-002	TAILS FILTER PRESS FEED TANK	1
670-THK-001	TAILS THICKENER	1

Source: Tetra Tech, 2017

17.4.3 Pyrometallurgical Plant

Based on the design criteria and mass balances, major process equipment and some minor equipment has been sized. These pieces of equipment were used to determine the capital and operating costs of the pyrometallurgical plant.

An allowance was made for some minor equipment and facilities where required. The major equipment items are listed in Table 17-10.

Table 17-10: Pyrometallurgical Processing Major Equipment List

Equipment Name	Qty	Description/Size/Model
FeNb Furnace Feed Preparation		
Nb_2O_5 Pellets Bins (5 days)	1	4.88 m dia. x 9.76 m height
Aluminum Shot Feed Bin (6 days)	1	4.88 m dia. x 9.76 m height
Fe_2O_3 Pellet Feed Bin (6 days)	1	4.27 m dia. x 5.48 m height
Nb_2O_5 Pellet Operations Bin	1	1.00 m dia. x 1.68 m height
Aluminum Shot Operations Bin	1	1.00 m dia. x 1.68 m height
FeNb Off Spec Operations Bin	1	1.37 m dia. x 2.28 m height
Fe_2O_3 Pellets Operations Bin	1	0.76 m dia. x 0.91 m height
FeNb Furnace		
FeNb Furnace	1	Electric Arc Furnace, 1000kW
FeNb Pelletizing Basin	1	6.00 m^3
Rotary Dryer	1	0.61 m dia. x 4.60 m len., 741 kBTU/hr
Slag Jaw Crusher	1	
Screening system	1	
Cooling Tower	1	
Dust Collection	1	

Source: Tetra Tech, 2017

17.4.4 Acid Plant

Based on the design criteria and mass balances, major process equipment has been sized and listed in Table 17-11.

Table 17-11: Acid Plant Equipment List

Equipment	Qty	Description	kW
Contact Section			
Blower	2	2 operating, single-stage centrifugal, electric motor, lube oil system	5500
Cold Heat Exchanger	1	Shell and tube, CS	
Hot Heat Exchanger	1	Shell and tube, SS	
Converter	1	4 bed, 304 SS	
Intermediate Reheat Exchanger	1	Shell and tube, SS	
Cold Reheat Exchanger		Shell and tube, CS	
Strong Acid			
Drying Tower	1	Packed tower, CS shell, acid resistant brick lining, membrane, ceramic packing, mesh pad mist eliminator, acid distributor	
Drying Acid Pumps	2	1 op/1 stdby, vertical submerged centrifugal	240
Drying Acid Cooler	1	Shell and tube, anodically protected	
Drying Acid Pump Tank	1	Horizontal, CS shell, acid resistant brick lining	
Intermediate Absorber Tower	1	Packed tower, CS shell, acid resistant brick lining, membrane, ceramic packing, high-efficiency mist eliminator, acid distributor	
Final Absorber Tower	1	Packed tower, CS shell, acid resistant brick lining, membrane, ceramic packing, high-efficiency mist eliminator, acid distributor	
Absorber Acid Pumps	3	2 op/1 stdby, vertical submerged centrifugal	240
Absorber Acid Cooler	1	Shell and tube, anodically protected	
Absorber Acid Pump Tank	1	Horizontal, CS shell, acid resistant brick lining	
Product Pumps	2	1 op/1 stdby, vertical submerged centrifugal	25
Strong Acid Area Sump Pump		Vertical centrifugal, UHMWPE	4

Source: Tetra Tech, 2017

17.5 Power Requirements

17.5.1 Surface Crushing, Ore Storage & Mineral Processing Plant

The estimated power requirements for the mineral processing are shown above in Table 17-7 and Table 17-8.

The Mineral Processing total installed power is 4000 kVA. After applying the power factor and a 90% utilization rate, the installed operating power requirement is 2,700 kVA, which gives a total annual electrical energy consumption 23,622 MVAh/y.

The power requirement is estimated from vendor data for major power users such as the crusher, HPGR, vibrating screen and conveyor motors. Building lighting and other smaller users are also included in the total.

17.5.2 Hydrometallurgical Plant

The total installed power for the Hydrometallurgical process plant is 25,546 kVA. After applying the power factor and a 92% utilization rate, the installed operating power requirement is 17,244 kVA, which gives a total annual electrical energy consumption 151.053 MVAh/y. A summary unit breakdown is shown in Table 17-12.

Table 17-12: Hydromet Power Requirements

Processing Unit	Units	Value
Sulphuric Acid Plant	kVA	10,133
HCl Regeneration	kVA	3,074
Hydromet Total (all other processes)	kVA	12,339
Total	kVA	25,546

Source: Tetra Tech, 2017

17.5.3 Pyrometallurgical Plant

For the Pyrometallurgical process plant, the total installed power is 5,200 kVA (including the furnace). After applying the power factor and a 90% utilization rate, the installed operating power requirement is 3,500 kVA, which gives a total annual electrical energy consumption 30,724 MVAh/y.

The power requirement was estimated based on scoping test work and from calculations from previous FeNb test work (XPS, KPM, and Hazen). Furnace equipment / technology vendors also confirmed the estimated power requirement for the FeNb Furnace, as summarized in Table 17-13.

Table 17-13: FeNb Furnace Power Requirements

Furnace Power Parameter	Units	Value
Electrical Power per ton Furnace Feed	kWh/t	334
Furnace Efficiency	%	60
Total Peak Power Input	kW	950
Furnace Design Power	kW	1,000

Source: Tetra Tech, 2017

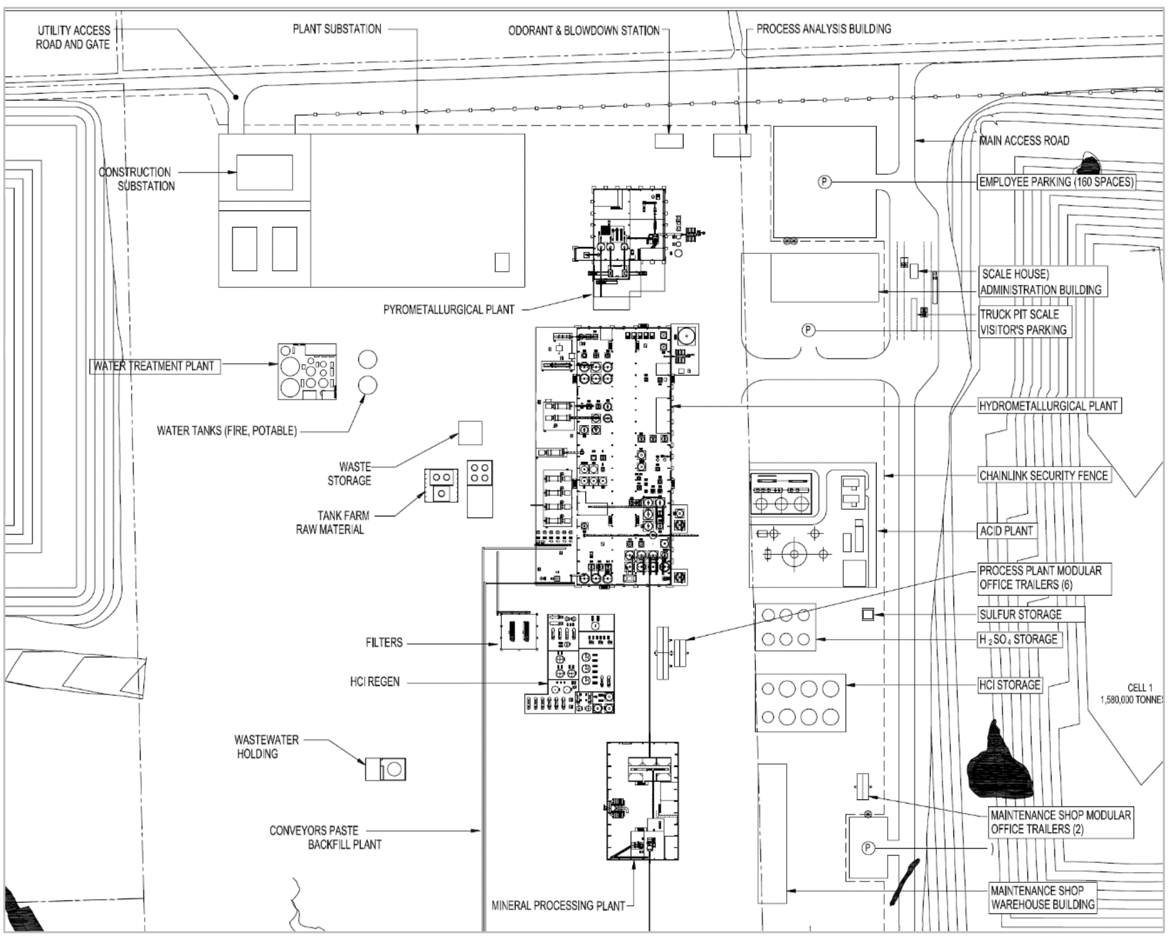
17.5.4 Acid Plant

The estimated power requirements for the acid plant are shown above in Table 17-11.

17.6 Plant Layout

17.6.1 General

The site process facilities include the Mineral Processing Plant, Hydrometallurgical Plant and the Pyrometallurgical Plant (Figure 17-5). The Hydrometallurgical Plant further includes both the HCl Regeneration Plant and the Acid Plant to support its operation. These facilities, as well as other support and Infrastructure facilities, are located west of State Hwy 50, and south of County Road 721.



Source: Nordmin, 2019

Figure 17-5: Process Plant Layout

17.6.2 Mineral Processing Plant, Surface Crushing and Ore Storage

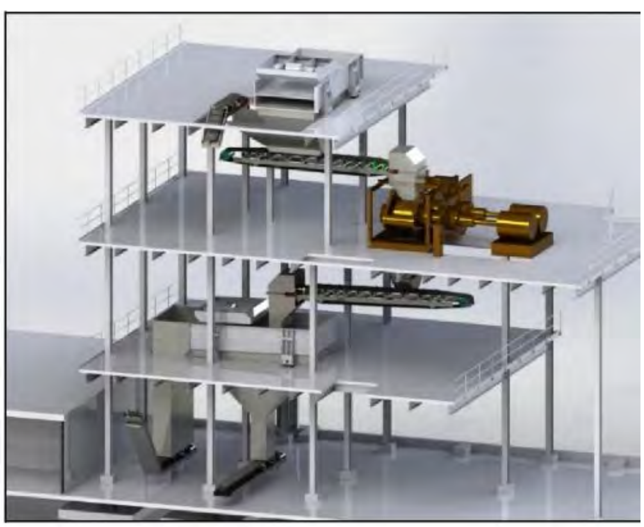
Figure 17-6 through Figure 17-11 present illustrations of the mineral processing plant, surface crushing and ore storage.

The Mineral Processing building will house all of its equipment within a single large building. This building will be an engineered steel structure with dimensions approximately 61.8 m x 46.7 m (203 ft x 153 ft) with a 31.7 m (104 ft) eave height. The equipment has been placed to allow for ease of material movement and maintenance access.

Source: Tetra Tech, 2017

Figure 17-6: Mineral Processing Building Southeast View

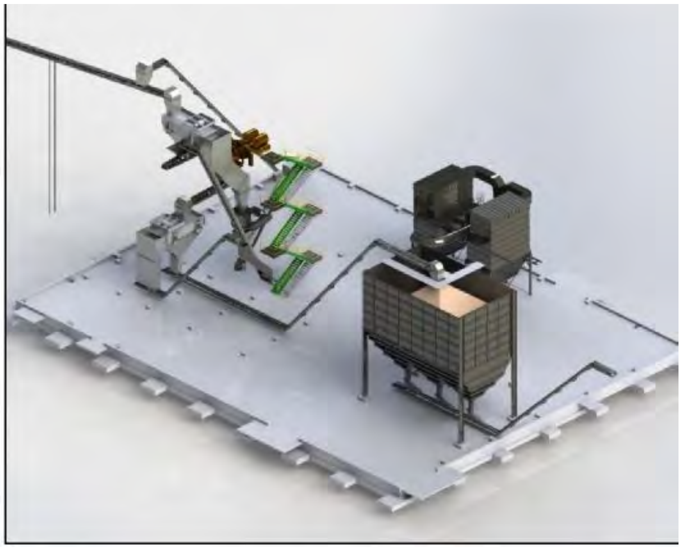
The crushing and grinding areas are located in the south half of the building and utilizes a multilevel structural deck to support the heavy equipment and conveyors, to provide operational and maintenance access to each piece of equipment and to utilize gravity to enhance flow characteristics.



Source: Tetra Tech, 2017

Figure 17-7: Crushing/Grinding Deck

The fine ore bin is located in the north half of the building and also utilizes decking and stairs to allow access to the equipment. The fine ore bin discharge provides the feed product for the initial Acid Leach Process in the Hydromet utilizing belt conveyor to move product between buildings.



Source: Tetra Tech, 2017

Figure 17-8: Mineral Processing Building Northwest View Without the Building Steel

The two-story personnel space contains offices, restrooms, Control Room and a break room and is located in the Northwest quadrant of the building. The electrical room has been located close to the crushing and grinding equipment in the southeast corner to minimize the length of electrical cable runs.

Source: Tetra Tech, 2017

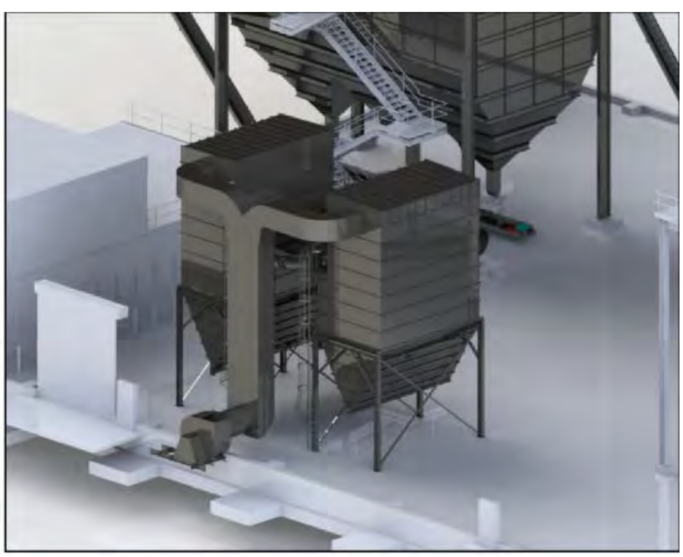
Figure 17-9: Office/Control Room Location

A 10-ton bridge crane that spans the whole building has been provided to assist with maintenance and operation functions. It was sized to accommodate the large crushing and grinding components that will require periodic replacement.

Source: Tetra Tech, 2017

Figure 17-10: Mineral Processing Building Plan View

Dust from the dry material handling process will be collected and routed to the dust collection baghouse, which is centrally located inside on the west end of the building. The dust collection fan and discharge are located outside of the building.



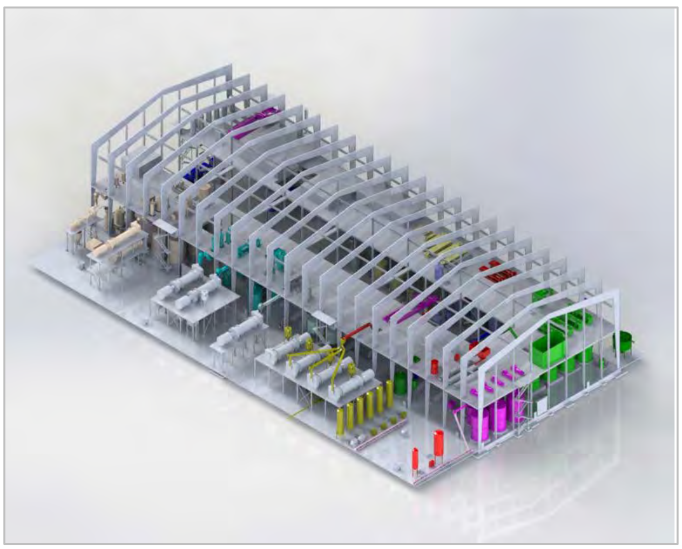
Source: Tetra Tech, 2017

Figure 17-11: Dust Collection System

17.6.3 Hydrometallurgical Plant

Figure 17-12 through Figure 17-24 present illustrations of the hydrometallurgical plan.

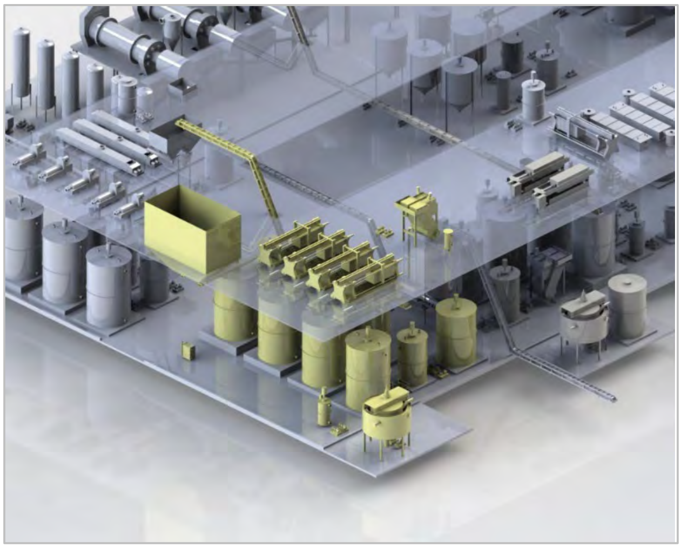
The Hydromet Plant building is a very large multi-level engineered steel structure with dimensions approximately 167.64 m x 60.96 m (550 ft x 200 ft) with a 30.5 m (100 ft) eave height. The building will house the equipment on two levels for the 15 individual processes required to separate the three recoverable elements. The equipment has been placed to allow for ease of material movement and maintenance access. Some of the equipment, such as the calcinators and kilns, will be located outside on elevated steel support structures adjacent to the building. The electrical room is centrally located on the west side of the building. Personnel areas such as offices, break rooms, maintenance rooms and the Control Room are located near the electrical room. Longitudinally the building is split into three long bays to allow two separate 20-ton bridge cranes to service the east and west sides of the building. The center bay is open for vehicle and maintenance access.



Source: Tetra Tech, 2017

Figure 17-12: Hydromet Plant

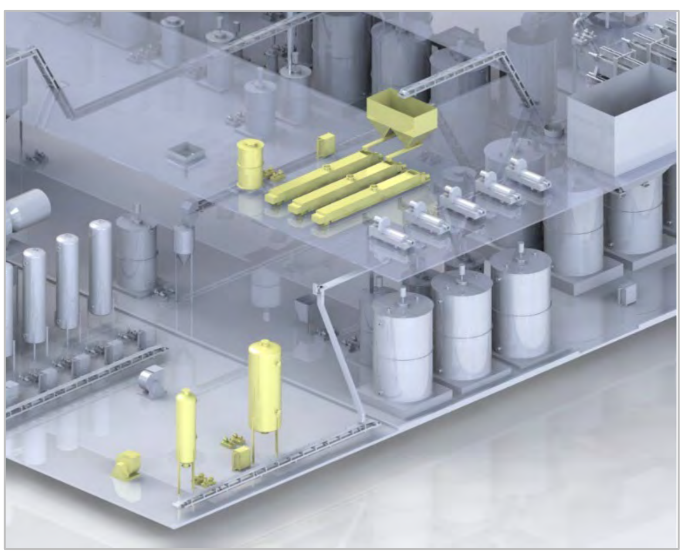
Hydrochloric Acid (HCl) Leach (605) - The Hydrochloric Acid Leach unit is designed to leach the majority of the impurities and the scandium present in the feed material to reduce the size of subsequent process equipment. It contains agitated tanks, thickeners and filter presses.



Source: Tetra Tech, 2017

Figure 17-13: HCl Leach Equipment

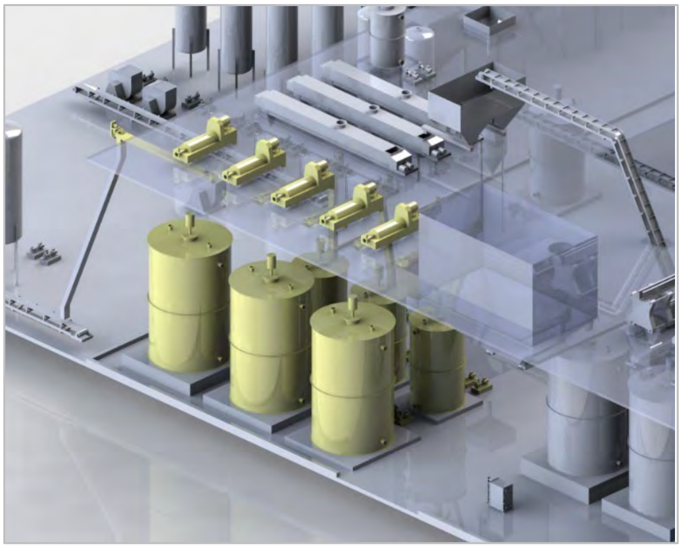
Sulphuric Acid Bake (610) - The Acid Bake unit is used to convert all of the unleached metal content into sulphate compounds. It contains a set of three pug mills that heat the paste to remove excess water and sulphuric acid. The discharge from this unit becomes the feed for the Water Leach process.



Source: Tetra Tech, 2017

Figure 17-14: Sulphuric Acid Bake Equipment

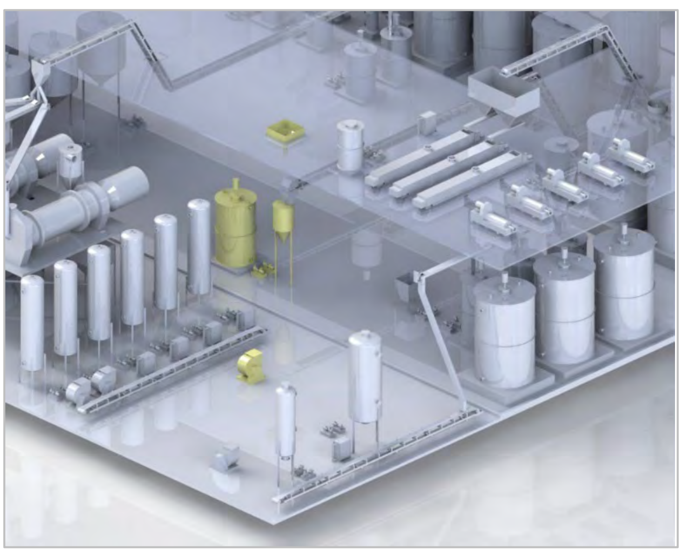
Water Leach (615) - The water leach unit is used to solubilize all soluble sulphates while separating non-soluble impurities. The water leach circuit is composed of a series of agitated tanks discharging to centrifuges. The Water Leach Residue is transferred by conveyor to the Paste Back Fill Plant.



Source: Tetra Tech, 2017

Figure 17-15: Water Leach Equipment

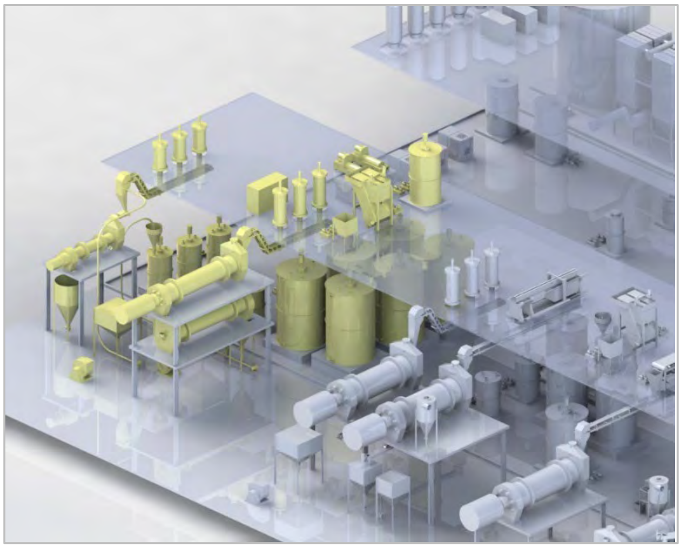
Iron Reduction (620) - The Iron Reduction unit is used to reduce iron (III) sulphate ($Fe_2(SO_4)_3$) present in the solution to iron (II) sulphate ($FeSO_4$) and also reduces titanium by adding iron solids (iron briquettes and iron powder) to the solution at room temperature.



Source: Tetra Tech, 2017

Figure 17-16: Iron Reduction Equipment

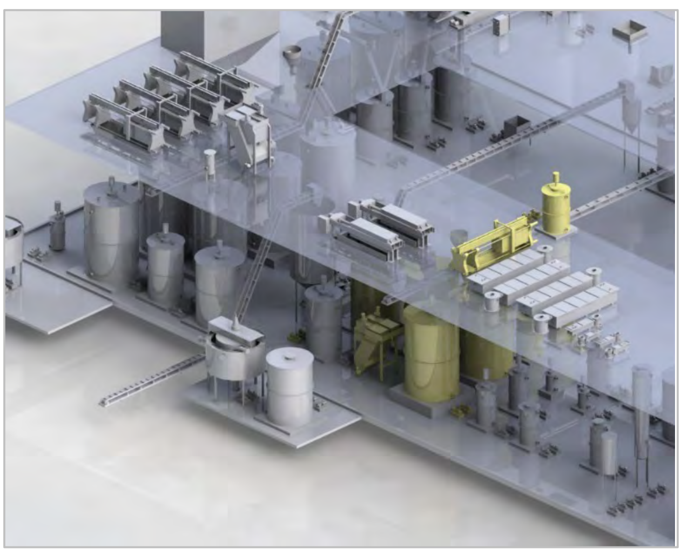
Niobium Precipitation and Phosphorus Removal (625) — This unit uses water dilution to selectively hydrolyze niobium sulphate and precipitate it as niobium oxyhydroxide. The Iron Reduction discharge is diluted with hot water, acidified with sulphuric acid, and cascaded through a series of five agitated tanks. The niobium oxyhydroxide is subsequently leached with sodium hydroxide to remove phosphorus before being dried and advanced to the pyromet process.



Source: Tetra Tech, 2017

Figure 17-17: Niobium Precipitation Equipment

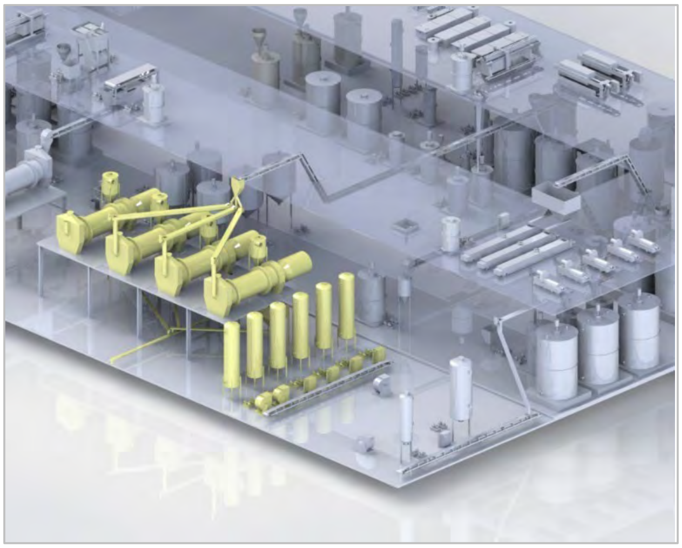
Scandium Precipitation (628) - The feed for the Scandium Precipitation circuit comes from the titanium precipitation filtrate. This material is mixed with phosphoric acid and iron powder in an agitated tank in order to prepare the precipitation of the scandium as a phosphate compound.



Source: Tetra Tech, 2017

Figure 17-18: Scandium Precipitation Equipment

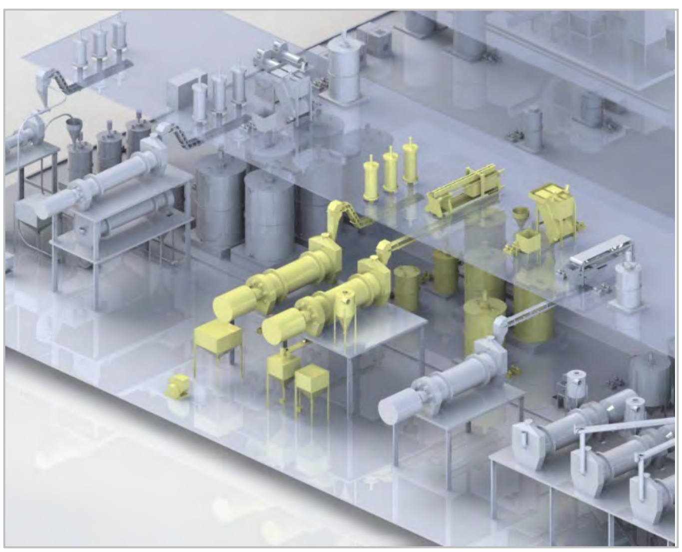
Sulphate Calcining and Mixed Oxides Handling (630) — This three-stage calcination process operates at elevated temperatures to recover the majority of free sulphuric acid, which is not associated with any other elements. All water content in the ferrous sulphate cake will follow the sulphuric acid gas stream.



Source: Tetra Tech, 2017

Figure 17-19: Sulphate Calcining Equipment

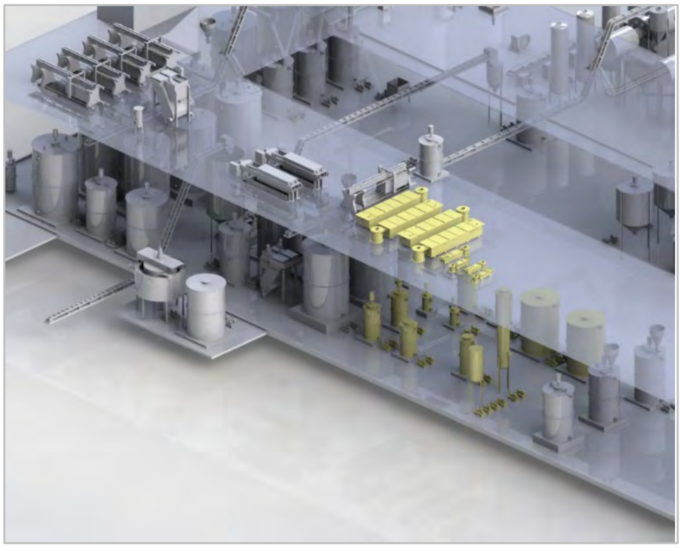
Titanium Precipitation (635) - The Titanium Precipitation is achieved through hydrolysis of the titanium oxyhydroxide using heat at a reduced free acid content. The unit contains tube presses, filter presses, calcination kilns and agitated tanks.



Source: Tetra Tech, 2017

Figure 17-20: Titanium Precipitation Equipment

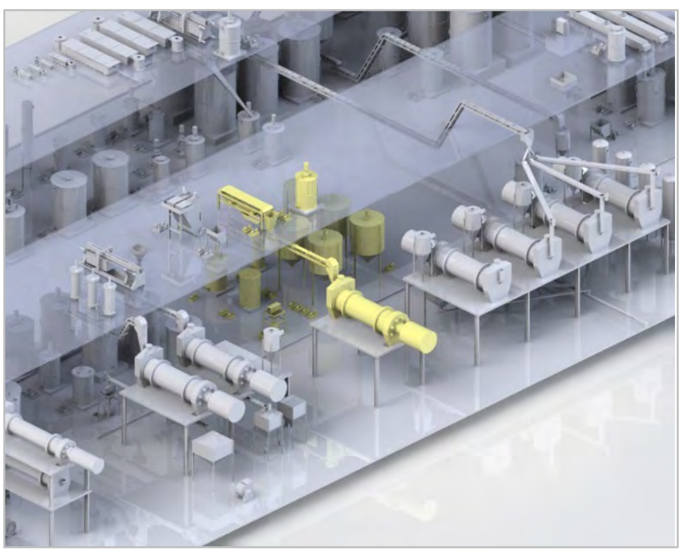
Scandium Solvent Extraction (640) - The scandium solvent extraction unit is a four-stage extraction circuit followed by a wash stage, a three-stage scrubbing circuit and two-stage Stripping Circuits used to selectively recover scandium from the leach solution.



Source: Tetra Tech, 2017

Figure 17-21: Scandium Solvent Extraction Equipment

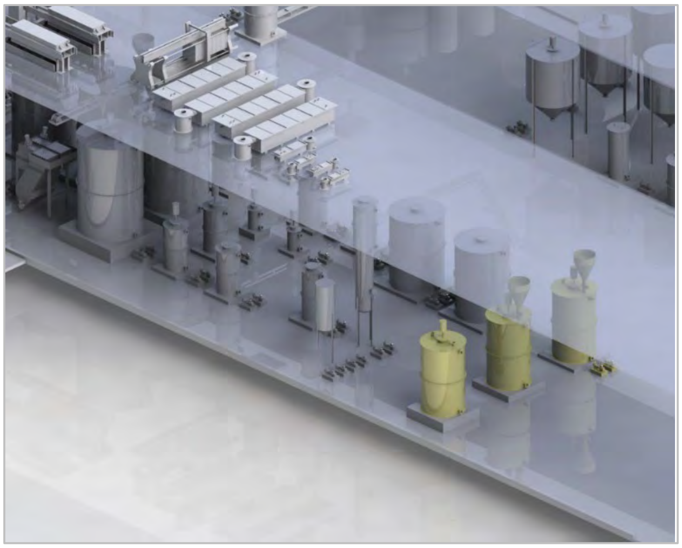
Scandium Refining (645) - The Scandium refining unit consists of a batch solvent extraction circuit followed by the oxalate scandium crystallization. The scandium cake coming from the Scandium Extraction unit is mixed with sulphuric acid in order to dissolve the scandium into solution.



Source: Tetra Tech, 2017

Figure 17-22: Scandium Refining Equipment

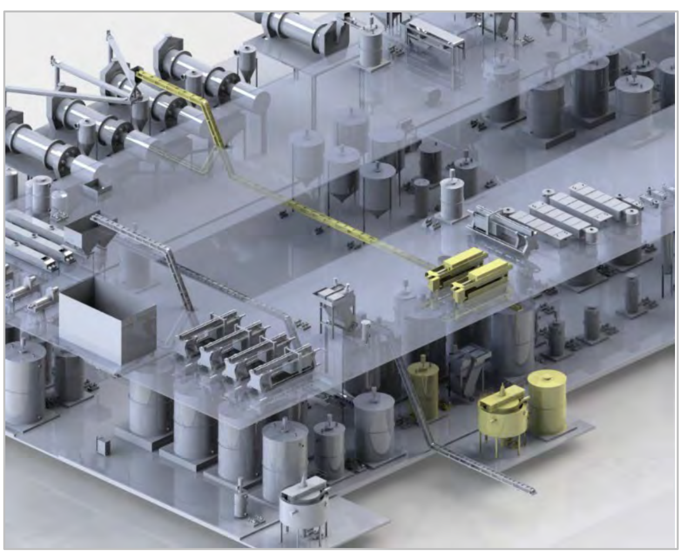
Tailings Neutralization (665) - The tailings neutralization unit is fed by multiple acidic tailings streams. The tailings neutralization feed is combined with limestone and the caustic leach tailings in a series of agitated tanks to raise the pH to around 9.5. The discharge slurry is pumped to tailings filtration.



Source: Tetra Tech, 2017

Figure 17-23: Tailings Neutralization Equipment

Tailings Filtration (670) - The discharge from tailings neutralization is successively dewatered with a thickener and belt filters. The filtrate is returned to the thickener, the filter cake is sent to the sulphate calciner, and the thickener overflow is recycled as process water or sent for water treatment.



Source: Tetra Tech, 2017

Figure 17-24: Tailings Filtration Equipment

17.6.4 Pyrometallurgical Plant

Figure 17-25 through Figure 17-31 depict the pyromet plant.

The pyromet building will house most of its equipment within a single building. This building will be an engineered steel structure with dimensions approximately 45.7 m x 45.7 m (150 ft x 150 ft) with a 22.9 m (75 ft) eave height. The open floor layout will allow for ease of material movement and maintenance of equipment.

Source: Tetra Tech, 2017

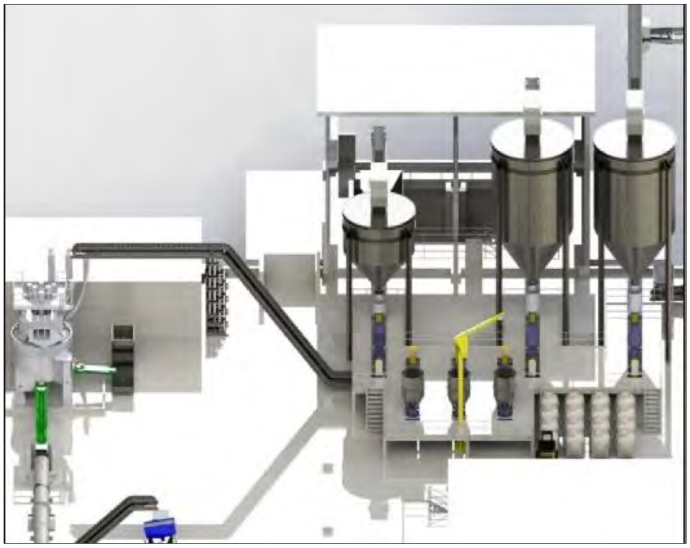
Figure 17-25: Pyromet Building Southeast View

The bulk furnace feedstock storage and prep areas are located in the southwest quadrant of the building and utilize inclined sidewall conveyors to elevate the feed product into the storage bins.

Source: Tetra Tech, 2017

Figure 17-26: Bulk Feed and Storage

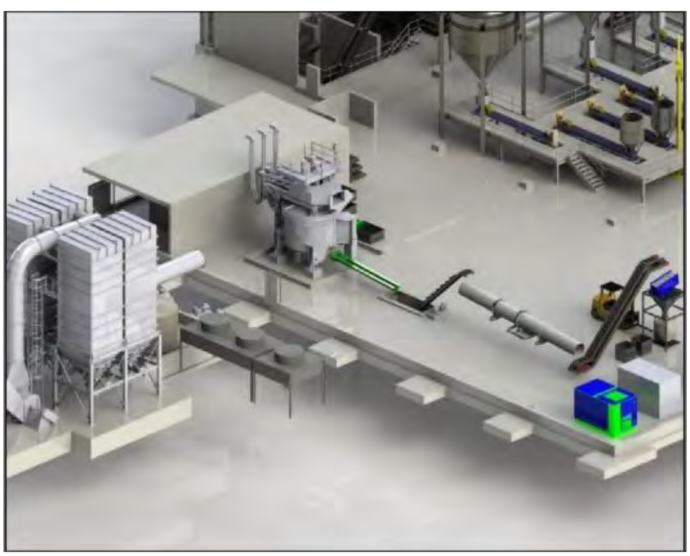
The furnace feed preparation is performed as a continuous process with specified mass measurement of the niobium oxide and other reagents and fluxes. Each ingredient is fed onto the furnace feed conveyor at a pre-determined rate to provide a continuous charge to the furnace.



Source: Tetra Tech, 2017

Figure 17-27: FeNb Furnace Feed System

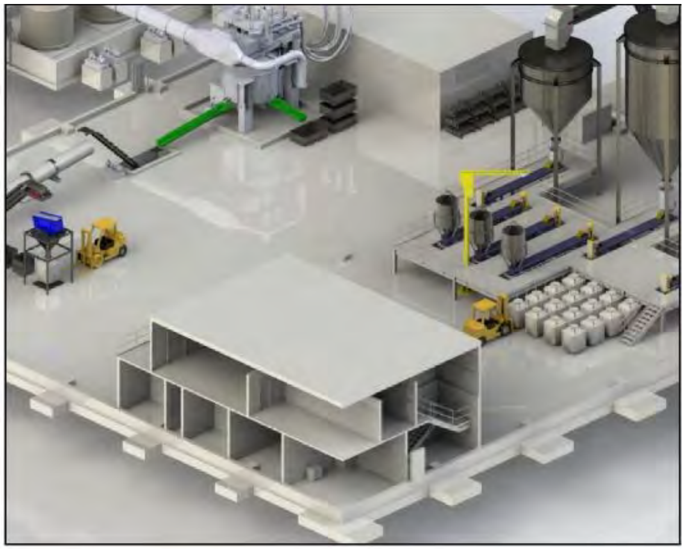
The FeNb furnace, dryer, pelletization basin and product packaging are located on the east side of the building. The electrical room has been located close to the furnace to minimize the length of the high voltage water cooled cables for the furnace.



Source: Tetra Tech, 2017

Figure 17-28: FeNb Furnace, Pelletization Basin, Dryer and Packaging Equipment

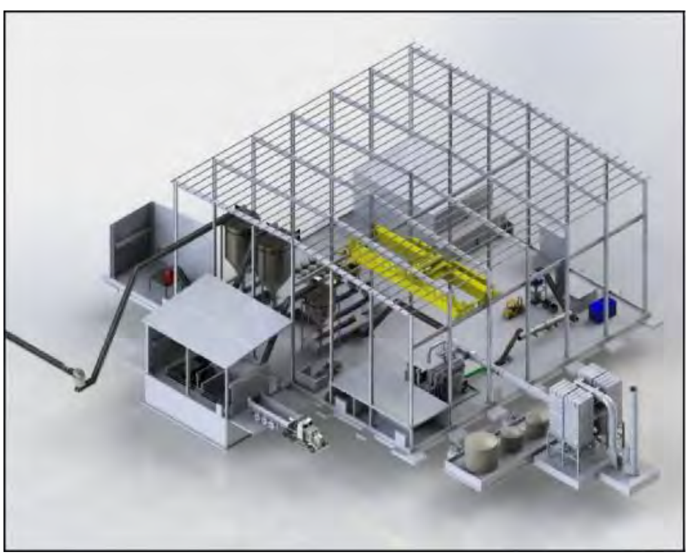
The personnel spaces such as offices, control room, restrooms and break room are located in the northwest quadrant of the building and utilize a two-level design to minimize the space requirements and to allow better communication.



Source: Tetra Tech, 2017

Figure 17-29: Office and Control Room

Several small building extensions are included to provide protection for reagent delivery equipment and the slag crusher equipment. A bridge crane shown in yellow below has been provided over the furnace equipment on the east side of the building to assist with maintenance and operation functions.



Source: Tetra Tech, 2017

Figure 17-30: Pyromet Building Northwest View

The dust collection equipment which includes the baghouse, a fan and a stack is located outside of the building as are the cooling towers, pumps and water storage tanks containment area. All equipment is located close to its functional use point to minimize piping, ducting and energy consumption. As for the furnaces off-gas system, ducting will be arranged in order to treat the stream into the sulfuric acid plant.

Source: Tetra Tech, 2017

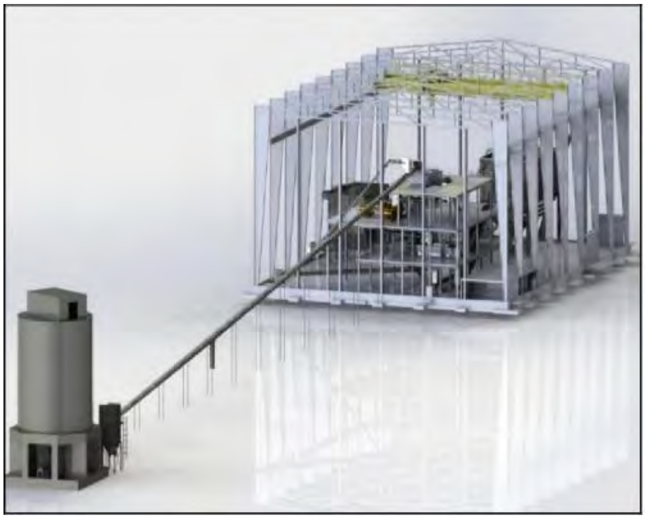
Figure 17-31: Dust Collection and Cooling Systems

17.7 Supporting Facilities Layout

Figure 17-32 through Figure 17-37 present illustrations of the supporting facilities.

17.7.1 Material Handling from Ore Storage Bin

Crushed ore is pulled from the Crushed Ore Storage Bin using three vibrating feeders controlled by knife gate valves and is conveyed to the Mineral Processing Building with a standard belt conveyor. Strategically placed magnets will provide tramp metal protection. Feed chutes are instrumented with Plugged Chute detection. The conveyor is instrumented with drift, zero speed and E-Stop switches. Dust control is installed at key dust generation points and is collected in a nearby baghouse.



Source: Tetra Tech, 2017

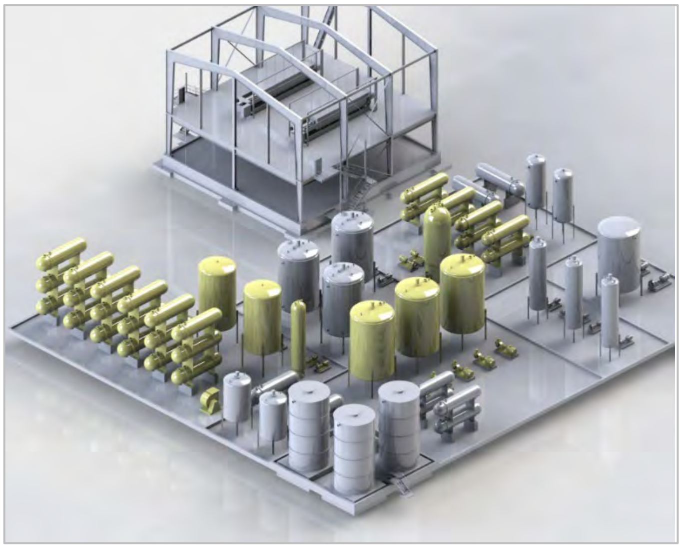
Figure 17-32: Coarse Ore Feed Conveyor

17.7.2 HCl Regeneration Plant

The Hydrochloric Acid Regeneration plant is composed of a large open air tank farm type equipment area with containment capability situated next to a single large building. This building will be an engineered steel structure with dimensions approximately 22.86 m x 22.86 m (75 ft x 75 ft) with a 16.76 m (55 ft) roof height.

The open air tank farm area will contain the reactors, agitated tanks, heat exchangers and absorber columns needed for the regeneration of hydrochloric acid. All of these vessels are located within the system containment area.

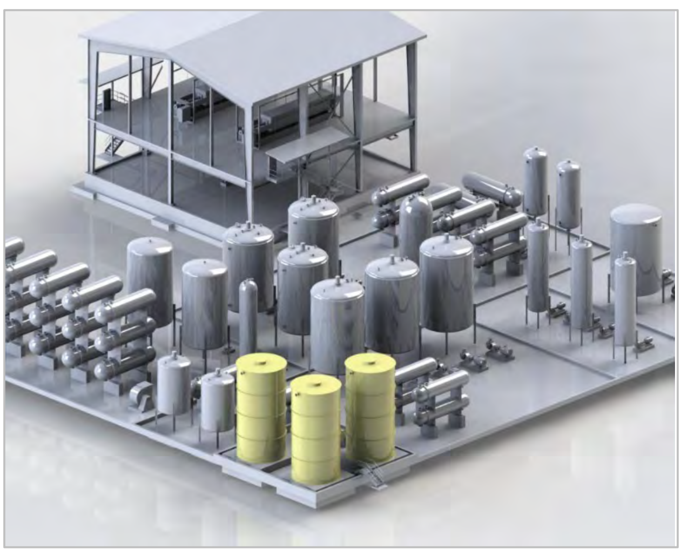
The electrical equipment supporting this plant has been selected for outside placement and will be located outside near the plant. No bridge crane has been provided as all of the equipment is accessible to a mobile crane.



Source: Tetra Tech, 2017

Figure 17-33: HCl Regeneration Plant Layout

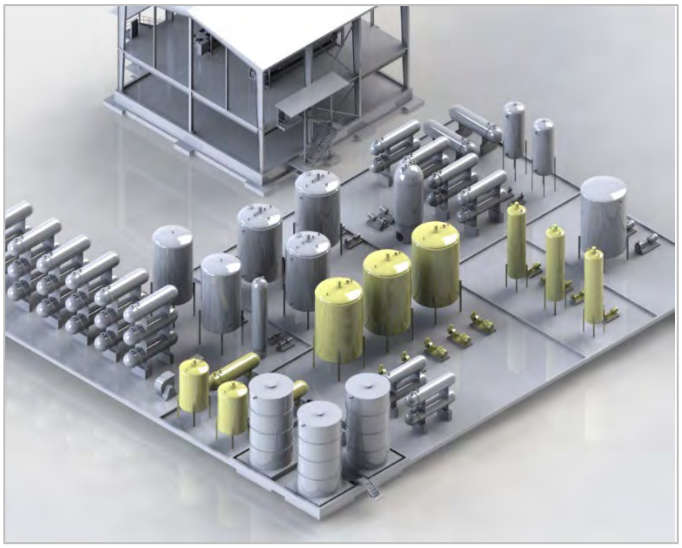
The feed tanks receive and hold the scandium raffinate containing metal chloride solutions prior to transfer to the HCI Regeneration process. These tanks are each located within their own containment area.



Source: Tetra Tech, 2017

Figure 17-34: Feed Tanks

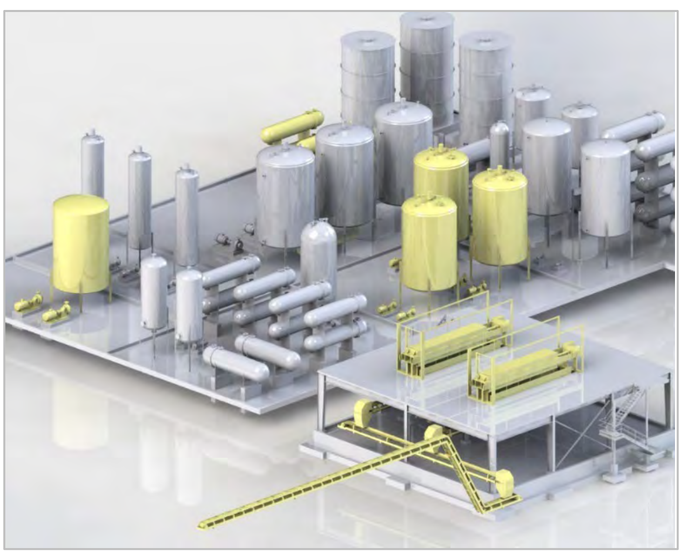
The regeneration process includes a series of heated reactor vessels and flash tanks to remove the moisture followed by cooled precipitation vessels. The precipitate from the reactors is fed to the two filter presses located in the nearby building.



Source: Tetra Tech, 2017

Figure 17-35: Reactor System

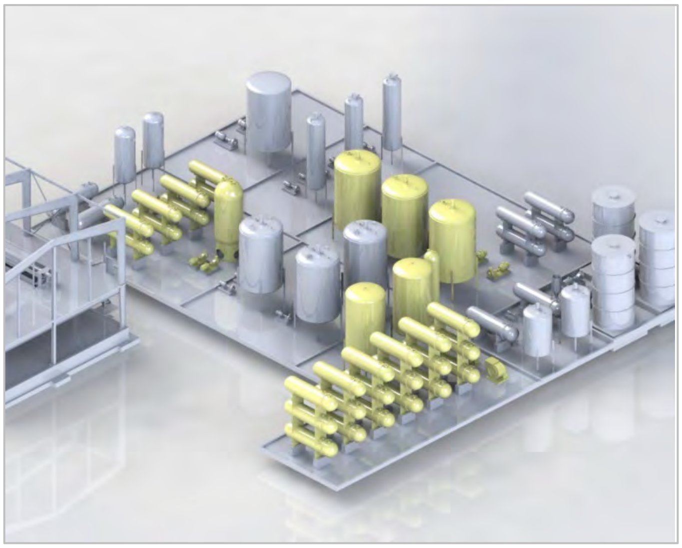
The two filter presses are located on the upper-level deck to provide room for gravity feed of the filter cake material to the conveyor located just under the upper-level deck. The filter cake becomes feedstock for the Mixed Oxide Calciners while the filtrate liquid moves on to the absorber columns.



Source: Tetra Tech, 2017

Figure 17-36: Filter Presses and Precipitator

The absorber system contains a series of vessels, tanks and pumps where the Hydrochloric Acid is concentrated into two separate streams that include a weak 20% HCI solution and a stronger 36% HCI solution. Both of these products are then returned to the Hydromet Plant for re-use.



Source: Tetra Tech, 2017

Figure 17-37: Absorber System

17.7.3 Acid Plant

The Acid Plant is composed of a large open equipment area with dimensions approximately 82 m x 72.3 m (269 ft x 237 ft). Within this general area is a containment area with dimensions approximately 40 m x 26.8 m (131 ft x 88 ft) that contains the absorber and drying towers, acid coolers, acid dryers and pumps. The equipment has been placed to allow for ease of material movement and maintenance access. The remaining open area will contain the heat exchangers, superheater, converter and blowers used for the regeneration of sulphuric acid. The electrical equipment supporting this plant has been selected for outside placement near the plant.

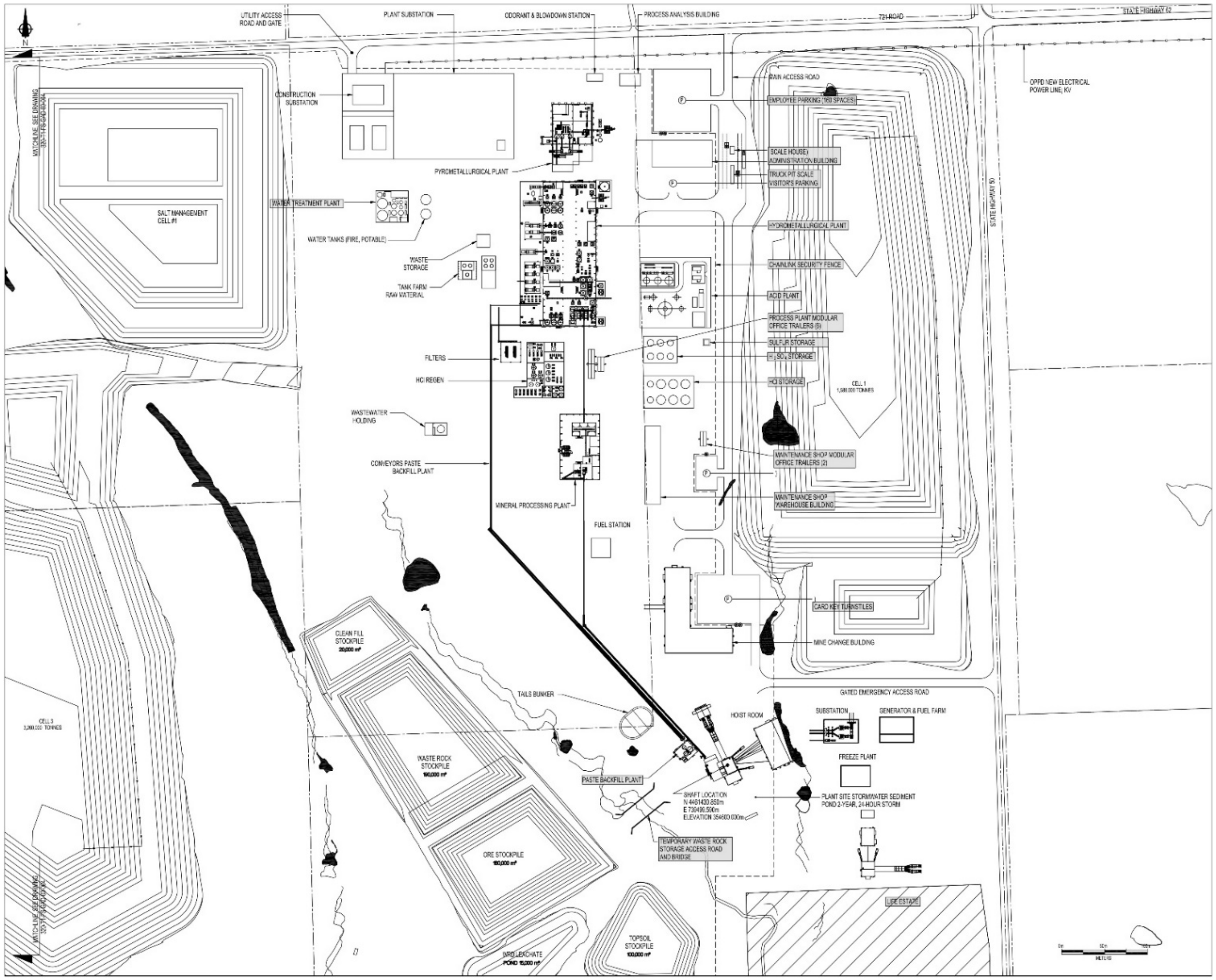
18. PROJECT INFRASTRUCTURE

18.1 General Information Site Layout

There are several local communities near the Project site including Elk Creek, Tecumseh and Lewiston that are intended to provide local housing for the Project construction, and operating staff. There are several other communities within driving distance, and the large cities of Lincoln and Omaha are also within reasonable driving distance of the site. Both cities have substantial regional airports.

Presently, the site has no existing infrastructure except for access via the Nebraska state highway 50 and County Road 721. The Project site will be accessed from County Road 721 through a guarded gatehouse into the Project property.

The site comprises an area of over 218.53 hectares. Figure 18-1 shows the layout.



Source: Nordmin, 2019

Figure 18-1: Elk Creek Project Site Plan Layout

18.2 Electrical Power

18.2.1 Electrical Power Line & Substation

The local power utility (Omaha Public Power District) will provide power to the site. This will require approximately 29 km (18 miles) of new transmission line be installed by the utility to provide power to the Project site main sub-station to meet the required power demand. The local power utility will also design and install the main substation that will be owned and maintained by the utility. This infrastructure will be paid back through rate charges on electrical usage.

18.2.2 Electrical Power Distribution - Plant and Facilities

The main substation will feed the site distribution substation with 44 kV. A 44 kV pole line will be constructed on the Project site to supply main power throughout the site and to the mine sub yard. In addition, this substation will include two 20/25 MVA transformers to provide 13.8 kV for distribution through the above ground facilities with approximately 1,100 m (3,610 ft) of power cables in vaults, and approximately 1,600 m (5,250 ft) of overhead lines.

18.2.3 Electrical Power Distribution - Underground

The underground electrical distribution will be fed from both the production and vent shafts, at 13.8 kV. Duplex fused disconnect switches will be present at several levels to allow power to be selected from either 13.8 kV feeder, providing redundancy. Power for utilization is accomplished through portable mine power centers, located at each production level. The duplex fused switches are not on every level but are distributed to adjacent levels through medium voltage junction boxes and boreholes.

18.2.4 Emergency Power Generation

Independent emergency power generation at the hoist house and vent shaft switchgear will be provided for back-up generation for surface infrastructure. Ventilation and hoisting are all powered from the surface, and thus, no emergency power is fed to the underground electrical distribution. Emergency power generation for the hoisting and ventilation systems will be supplied with from two diesel-powered generators, one at the hoist house and one at the vent shaft.

18.3 Control & Communications

18.3.1 Process Control System

Distributed processing will be implemented throughout the processing facilities. The Allen Bradley Control Logix 5000 PLC/PAC family of processors or equivalent will be used. Several networks will be utilized to maintain security, throughput and functionality.

18.3.2 Site Communications

Site communications are handled via phone service, radio communications and email communications.

18.3.3 Access and Security System

The entire site will be enclosed with a barbed wire fence. Site access will be permitted through a manned security gate for vehicles, or through employee turnstiles operated electronically by card key. A security network will be installed, allowing for control of gate access and security camera control.

18.4 Natural Gas

18.4.1 Natural Gas Pipeline to Site

Natural gas, to be used throughout the Elk Creek during the construction and operation phases of the project, will be brought to the site via pipeline from the local utility company. NioCorp has a natural gas transportation contract with Tallgrass Energy, which operates the Rockies Express (REX) pipeline. Tallgrass will construct a 45 km (28 mile) gas pipeline lateral from the main REX pipeline system in Kansas to the project site. The lateral will be sized to provide a minimum of 27.5 dekatherms of gas per day. Natural gas will be distributed to all on-site facilities utilizing buried height density polyethylene (HDPE) natural gas distribution pipe. Natural gas piping above ground and located inside of facilities will consist predominately of carbon steel pipe. Maximum on-site pipeline distribution pressure will be 100 psig. Natural gas will be used for facility heating, water heating, and for natural gas-fired process equipment.

18.4.2 Natural Gas Distribution on Site

Natural gas will be distributed to all on-site facilities utilizing HDPE natural gas distribution pipe. Natural gas piping located inside of facilities will consist predominately of carbon steel pipe. Maximum on-site pipeline distribution pressure will be 100 psig. Natural gas will be used for facility heating, water heating, and for natural gas-fired process equipment.

18.5 Plant Water

18.5.1 Water Treatment Plant

Water used for all on-site for all process needs and activities will be supplied from mine dewatering activities, local groundwater wells and from a local water utility (Tecumseh Board of Public Works). Mine water will be pumped to the Water Treatment Plant (WTP) that will produce approximately 2,908 gpm of treated water. Approximately 2,154 gpm of water will be produced from the Reverse Osmosis and Evaporation/Crystallization units, and 754 gpm of water will be produced from the Cooling Tower Makeup (CTMU) system.

The Water Treatment System is designed to reduce the hardness, metals, and dissolved solids of the process wastewater, cooling tower blowdown, well/utility and mine water streams. The system consists of precipitation softening, clarification, pH adjustment, multimedia filtration (MMF), and reverse osmosis (RO). Concentrated brine from the RO system will be sent to a thermal evaporator and crystallizer to produce a salt cake for disposal with the distillate being returned and combined with RO permeate for reuse.

The Process Water Treatment System includes the following major equipment units:

1. Process Water Influent Equalization Tank
2. Softening Reactor
3. Clarifiers
4. pH Adjustment Reactor
5. Multimedia Filters
6. Reverse Osmosis Units
7. Sludge Holding Tank
8. Filter Presses (shared with CTMU system)
9. Evaporator/Crystallizer System
10. Crystallizer Solids Dewatering System
11. Chemical Feed Systems

The following Table 18-1 was used as the design basis.

Table 18-1: Design Requirements

Parameter	Quantity (gpm)	Notes
Plant Source Water	3,375	From mine dewatering and water wells
Pyromet Feed Make-up (2 points)	5	From RO Units
	5	From RO Units
Hydromet Feed	200	From RO Units (Additional Hydromet Feed Water from the Acid Plant (450 gpm) and Potable Water Wells (1575 gpm) are untreated)
Acid Plant	40	From RO Units
Hydromet Cooling Tower	1,415	From RO Units & CTMU System
Pyromet Cooling Tower	1,158	From RO Units & CTMU System
Hydromet Return Water	750	Constituents: Na, Cl, Ca, SO^4 and Fe

Following is a summary description of the proposed Water Treatment Plant.

18.5.1.1 Flow Equalization

Process wastewater and underground mine water from NioCorp will be pumped into an equalization tank. Cooling Tower Blow Down (CTBD) will also be added to this tank since it will contain elevated total dissolved solids and hardness. The tank will also receive intermittent return flows from MMF backwash and the sludge dewatering system. The equalization tank will allow for storage during a shutdown and to sustain consistent flow to the system. The combined process wastewater and mine water will be pumped from the equalization to the softening reactor at a controlled rate. In the case of a system shutdown, it was assumed there would be enough storage capacity to accommodate reduced or no flow of mine water to the treatment system.

18.5.1.2 Softening and Clarification

The combined streams will enter a Turbomix® softening reactor where chemicals will be added for precipitation softening. The advantage of the Turbomix design is that it promotes precipitation/crystallization of the dissolved particles to maximize their size and density. This results in faster settling rates, improved sludge handling characteristics, and improved sludge thickening and dewatering rates. To enhance the crystallization reaction kinetics and to maximize the density of the settled sludge, a portion of the precipitated sludge collected in the downstream clarification process will be recycled back to the Turbomix draft tube. Hydrated lime and soda ash will be fed to the Turbomix based on the flow rate, hardness, and alkalinity of the incoming water. A coagulant also will be added.

The Turbomix reactor will overflow to two flocculating clarifiers to provide redundancy to allow one unit to be taken down for short durations for maintenance. The polymer will be added to the clarifier center well to promote flocculant growth and improve the settling characteristics of the precipitated solids. A rotating rake assembly including two long rake arms will move the settled solids to a center sludge discharge sump. The clarifier rake drive will be equipped with a high torque alarm and an automatic rake lift to raise the rotating rake mechanism should a torque overload condition occur.

The settled sludge will be withdrawn from the bottom of the clarifiers continuously by underflow pumps. The settled softening sludge is expected to have a solids concentration of close to 10%.

The clarifier effluent will be collected in a launder and will exit the clarifier through a drop box and be conveyed to the pH adjustment reactor tank ahead of the multimedia filters. The pH will be reduced to near neutral. This will allow any residual aluminum to precipitate for subsequent removal in the Multimedia Filter (MMF). An oxidant will also be added to this tank for ammonia removal. Water will be pumped from this tank to the MMF to further reduce the suspended solids prior to RO.

18.5.1.3 Multimedia Filtration

The effluent from the pH Adjustment Reactor (MMF Feed tank) is pumped to the MMF System. The goal of the filtration system is to reduce the inlet suspended solids concentration prior to RO. The vessels contain three separate layers of filtration media and a gravel support bed. The gravel supports the top three active filter layers consisting of anthracite, sand and fine garnet. This layered media profile provides a high sediment holding capacity as compared to conventional dual media/sand filters. The larger incoming particles are trapped on the upper layer of the media allowing the smaller particles to continue through the bed where they are trapped in the lower layers, producing a high-quality effluent. A filter aid will be added to the inlet of the MMF to enhance solids-liquid separation process and achieve deep bed filtration versus conventional surface filtration.

During operation, the softened water enters the multimedia filter vessel under pressure at the top and is distributed uniformly over the top layer of the media bed. After passing through the media bed, the filtered service water exits the vessel through the under-drain assembly at the bottom. As the water flows through the media bed, the suspended solids and turbidity present in the feed water will be removed. The filter media bed slowly exhausts from top to bottom. When the turbidity and/or the differential pressure from the media bed approaches a predetermined set point, the media bed is exhausted and is subjected to cleaning/backwash cycle.

18.5.1.4 Reverse Osmosis (RO) System

Filtered water from the MMF is collected in the RO Feed Tank and will be pressurized through a single pass RO system for removal of total dissolved solids. A small portion of the filtered water will be utilized for Multimedia Filter backwash purposes.

The RO process separates dissolved contaminants from the feed water by passing through a semipermeable thin film composite membrane. These membranes remove 95 ~ 99% of the dissolved solids present in the feed water and essentially perform a complete removal of all particulate matter.

During operation, the filtered water from the RO feed tank is pumped to the cartridge filter vessels. The water pressure forces the feed water through the filter elements while leaving any residual impurities behind on the filter element surface. The cartridge slowly exhausts, and when they are clogged with impurities, the pressure drop across the cartridge filter system exceeds the desired limit, and the dirty filter elements are taken out of service for replacement. An antiscalant will be added at the RO cartridge filter inlet to prevent any potential scaling issues across the downstream RO system.

The filtered water from the cartridge filter is then pressurized using the RO booster pump and is fed to the first stage membranes in the RO system. The concentrate from the RO system is routed to the RO Reject Tank prior to being discharged. The concentrate will be sent to the evaporation/crystallization process for further concentration. Permeate stream from the system is collected in the RO Product tank where it blends with the distillate from the evaporator and crystallizer and is pumped to the Hydromet process, cooling tower and other water users.

Over a period of time, the RO membrane elements will be subjected to potential fouling by suspended material or sparingly soluble material that may be present in the feed water. Upon an increase of the feed pressure or decline of permeate quantity/quality, the RO system will be taken offline, and the membranes will be cleaned.

18.5.1.5 Cooling Tower Makeup System (CTMU)

The CTMU Treatment System is designed to reduce iron and manganese in the groundwater supply for cooling tower makeup. Limited data was available on the groundwater; data from a nearby farmer's groundwater well shows that manganese is present at 0.4 mg/L. Cooling tower suppliers typically recommend that manganese be reduced to <0.05 mg/L to prevent deposition and fouling on the cooling tower fill and cooling loop systems. Based on the final water balance, there will be excess RO permeate available to blend with the groundwater (40:60 blend). Based on the projected blended quality, it is expected that the cooling towers can be operated at up to seven cycles of concentration.

The CTMU system will consist of a separate second treatment system to reduce iron and manganese using a filter media for this process. The filter backwash from the CTMU system will be combined with the Process Water Treatment system.

The CTMU Treatment System includes the following major equipment units and redundancy:

1. Manganese Removal Media Filters
2. Sodium Hypochlorite Feed System

The untreated well water will flow through Manganese Removal filters. It is assumed that the well water pressure is adequate for feeding the filters without re-pumping. Normally all filters are online, except when one filter requires backwashing, where the remaining filters handle the design flow. Sodium hypochlorite is injected in-line prior to the filters. The filters operate like the MMF units described previously. Upon high differential pressure, each filter is taken off-line for backwashing. Backwash water will be pumped from the downstream CTMU tank to the filters. Dirty backwash water will be conveyed to the PW sump and sent through the sludge handling system.

The treated water will be collected in a CTMU Tank and pumped to the cooling towers. Excess RO permeate/distillate from the PW treatment system will also be used as CTMU when available. Chemical storage systems will be shared between the CTMU and PW treatment systems.

18.5.1.6 Sludge Handling

Sludge from the PW will be collected in a sludge storage tank.

Intermittently the sludge from the storage tank will be pumped to the filter presses for dewatering. Pumps are provided to feed the filter presses. Filter press filtrate will flow by gravity to the building sump and then pumped to the Process Water Equalization Tank using sump pumps. The building sump will also receive filter backwash from the MMFs.

18.5.1.7 Evaporation and Crystallization System

<u>Evaporator Brine Flow</u>

The RO concentrate will be processed through an Evaporator/Crystallizer system to produce a salt cake for disposal. The RO concentrate contains a certain amount of alkalinity. In order to prevent calcium carbonate fouling of the Evaporator heat exchanger, it is important to eliminate all the carbonate alkalinity in the feed stream. This is accomplished in a three-stage process: feed acidification with sulphuric acid, feed preheating and feed deaeration/decarbonation. Feed acidification (via metered sulphuric acid addition) is performed within the Evaporator Feed Tank. The sulphuric acid converts the carbonate and bicarbonate ions to CO_2. The CO_2 is subsequently stripped out of the feed stream in the Feed Deaerator following heat recovery in the Feed Preheater. Brine from the Evaporator Feed Tank is pumped to the Feed Preheater where the temperature is increased by exchanging heat with the Evaporator and Crystallizer condensate. The feed then enters the Feed Deaerator where vapour and non-condensable gasses (NCGs) vented from the shell side of the Evaporator, heats the feed and allows for the release of CO_2 to the atmosphere. The feed then enters the Evaporator.

The purpose of the Evaporator is to remove the majority of the water in the most energy and cost efficient manner prior to the crystallization system. The feed flow enters the vapour body and is pumped up through the center of the heater via Evaporator Recirculation Pump. The recirculating brine stream is introduced into a vertical heat exchanger tube bundle utilizing Veolia's double distributor plate design. The brine falls down the inside of the heater tubes where it is heated by vapours condensing on the outside of the tubes, causing the brine to boil. The concentrated brine gathers in the vapour body below the heater, where it is recirculated again.

Antifoam can be added to the Evaporator on an as needed basis to ensure that no liquid is carried over through the mist eliminators. Caustic is added to the Evaporator to maintain the pH between 8.0 and 8.5 to ensure the system will not be susceptible to corrosion.

The concentrated brine leaves the Evaporator via a purge line off the discharge of the Evaporator Recirculation Pump and is pumped to the Crystallizer Feed Tank for further concentration.

Low-pressure steam is created by the auxiliary boiler. This steam is utilized for start-up purposes and as supplemental heat for the system when required.

18.5.1.8 Crystallizer Brine Flow

The concentrated brine from the Evaporator is pumped to the Crystallizer Feed Tank. Caustic is again added to the system at the Crystallizer Feed Tank. Caustic is needed at this point to make up for metal hydroxides that precipitate as the brine is concentrated. The target pH in the Crystallizer is 8.0-8.5. The Crystallizer is a forced circulation unit meaning the recirculation pump circulates the concentrated brine through the Crystallizer Heater, where heat is transferred through the tubes. The hydrostatic head from the level in the Crystallizer Vapor Body suppresses boiling in the tubes. This prevents scaling that may occur if dry spots form on the heater tubes (which can be the case if boiling occurs in the tubes).

Brine entering the Crystallizer Vapor Body from the heater flash boils and releases heat in the form of water vapour. The concentrated brine collects in the vapour body and is re-circulated through the heater again. As the evaporation process continues, the concentration of the brine contained in the vapour body increases. As the concentration increases, the solution becomes supersaturated, and salts precipitate from solution resulting in a brine slurry.

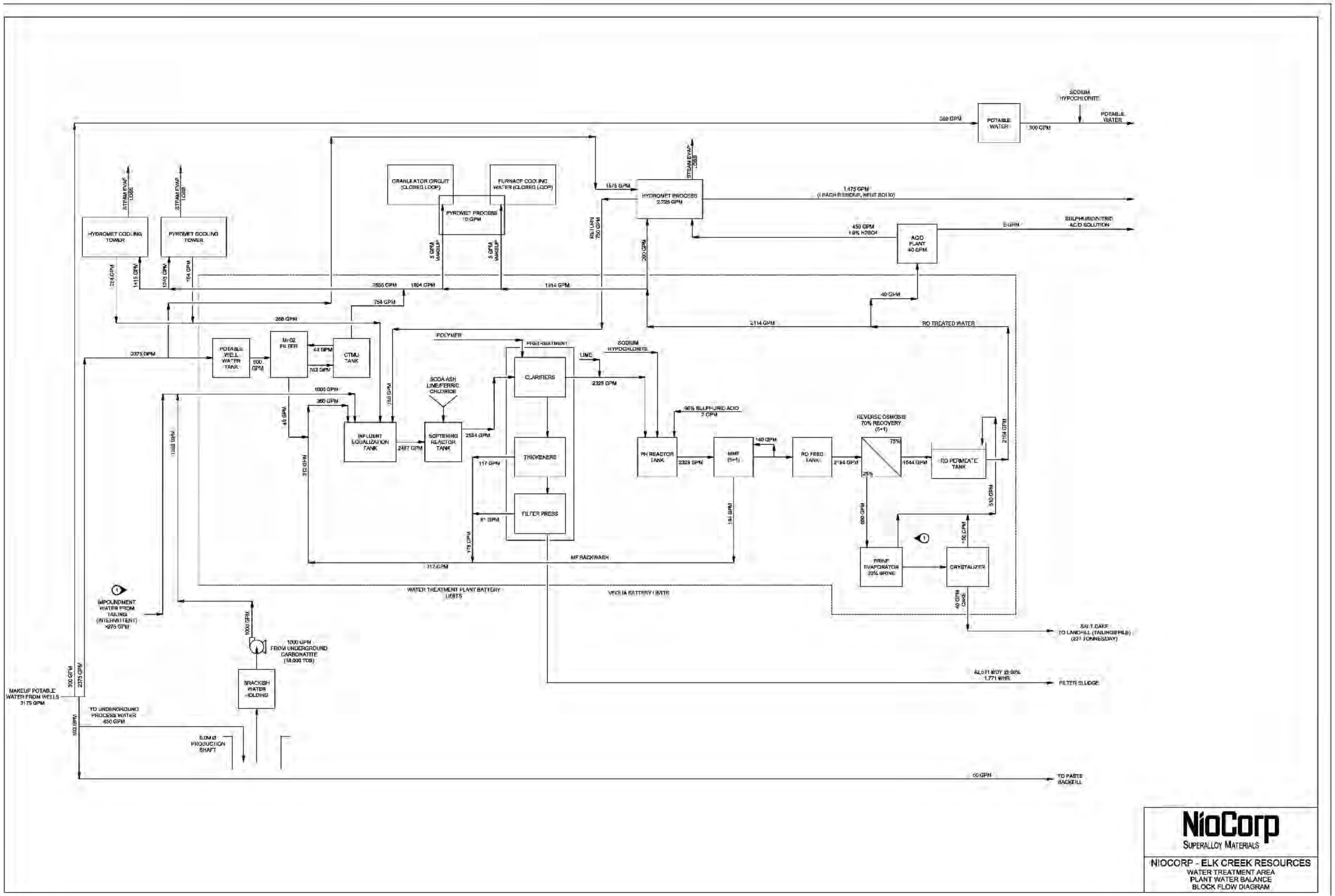
Antifoam can be added to the Crystallizer on an as-needed basis to ensure that no liquid is carried over through the mist eliminators into the Crystallizer First Stage Fan during upset conditions.

Slurry from the Crystallizer is removed from the vapour body and is pumped through a recirculation loop to the Crystallizer Centrifuges by the Slurry Pump. The feed flow to each centrifuge is controlled to maintain the proper slurry density, ~25 wt% suspended solids, in the recirculating brine. The slurry is pumped from the vapour body, and a slipstream is diverted to each centrifuge for dewatering while the remaining portion recirculates back to the Crystallizer. This recirculating slurry highway is utilized to maintain a relatively high fluid velocity to avoid any solids settling and plugging in the piping.

The centrifuges process the Crystallizer product slurry. The resultant wet-cake is discharged for on-site disposal. The centrate is sent to the Centrate Tank and returned to the Crystallizer.

Make-up steam can be added as necessary but is normally only needed during start-up.

Figure 18-2 is the block flow diagram of the proposed Process Water Treatment Plant



Source: NioCorp, 2019

Figure 18-2: Process Water Treatment Plant Block Flow Diagram

18.5.2 Process Water

Process water will be produced at the Water Treatment Plant. Plant Process water will be required in the Hydromet, Acid Plant, HCl Regeneration Plant, Paste Backfill plant and the Pyromet. Additional treated water will be required for both the Mine, as well as for site potable and fire water systems.

The vast majority of process water will be required in the Hydromet. The Paste Backfill plant will utilize RO permeate for backfill, as will the mine for underground operations. The remaining plants identified will require small quantities of make-up water primarily for cooling and chilling purposes.

Mineral Processing Plant

The water requirements for the Mineral Processing plant are minimal. Plant water will be available for use during the cleanup.

Hydrometallurgical Plant

The water requirement for the Hydrometallurgical Plant is to provide dilution, make up and wash water to various sections of the plant. Table 18-2 provides a summary of the water requirement.

Table 18-2: Summary of Hydrometallurgical Process Water Requirement

HCl Leach	542.5 t/d
Water Leach	2,914 t/d
Nb Precipitation	1,713.7 t/d
Nb Caustic Leach	63.5 t/d
Sc Precipitation and Refining	24.0 t/d
Ti Precipitation	20.9 t/d
Total	5,272.9 t/d
gpm	967

Source: Tetra Tech, 2017

Pyrometallurgical Plant

The water requirements (Table 18-3) for the Pyrometallurgical Plant provide make up water to supply the FeNb Furnace cooling systems and the FeNb Furnace Pelletizer basin for cooling and pelletizing the FeNb product.

Table 18-3: Pyrometallurgical Water Requirements

Water Item	Units	Value
Furnace Cooling Water Flowrate	m^3/h	64.7
Cooling Water Flowrate Addition for Pelletizing	m^3/h	15.1
Water Volume Required	m^3/tap	68.5
Steam Produced	%	20
Steam Flowrate	m^3/mi	0.05
Make-up water Required	m^3/mi	0.05

Source: Tetra Tech, 2017

18.5.3 Fire Water

The firewater system will be comprised of two 225,000 gal insulated fire water tanks and two independent fire water pumps capable of delivering 2,000 gpm for a minimum period of four hours. The primary pump will be electrically driven while the backup pump will be diesel powered. A fire water distribution system will be installed throughout the site. Dry and wet sprinkler systems, hydrants, hose reels and fire extinguishers will be utilized per the design.

All infrastructure facilities on the surface, except for the gate house, will include fire suppression systems. Process building fire suppression systems will include wet sprinklers in all office spaces and control rooms. Dry sprinkler systems will be utilized in the hydrometallurgical buildings within specified high hazard areas. The remaining open process/factory areas of these two process facilities, as well as the open areas of the mineral processing building, will utilize fire hose protection from outside hydrants, as well as interior located fire hose reels.

18.5.4 Potable Water

Potable water will be supplied from three possible available sources at an operational flow rate of 300 gpm to dedicated potable water tankage. These possible sources with their expected flow rates include; a supply line furnished by the Tecumseh Board of Public Works (2,000 gpm), a well and supply line from the Landowner 1 property (500 gpm), and two (2) wells and a supply line from the Landowner 2 property (1,500 gpm). Potable water will be distributed to all site facilities via a dedicated pumping system at 50 psig pressure. The nominal flow rate will be 100 gpm for the entire facility, with a peak flow rate of 750 gpm during shower usage.

18.6 Auxiliary Buildings and Facilities

The designs undertaken for both the processing mill (including Hydromet and Pyromet) and for the mining systems (surface and underground) allow for the two distinct areas to be partially autonomous at the feasibility stage.

18.6.1 Mining Infrastructure

Mine Change House (Mine Dry)

The Mine Change House will be house within the multi-use complex, located to the north of the production headframe. The Mine Change House is designed to accommodate the use by both male and females. The facility will include locker rooms and shower facilities for personnel working in the mining areas.

Administration

The Mining Administration will also be housed within the multi-use facility. Mining support staff and management will be housed within the Administration section of the building.

Warehouse

A Warehouse dedicated to mining use is included in the multi-use facility. All mining related items will be received and stored within the Warehouse.

Maintenance Shop

A Maintenance Shop for mining vehicles is included within the multi-use facility. The maintenance shop is a single story and will have the capability to house wash bays, storage areas, and maintenance areas.

Figure 18-3 shows the multi-use facility.

Source: Nordmin, 2019

Figure 18-3: NioCorp Multi-Use Facility

18.6.2 Processing Mill

Administration & Service Building

The Administration Building will consist of modular units or a long term leased building that will house offices for support staff and site management.

Maintenance Shop

The Maintenance Shop and the Warehouse will both be located in a single-story, steel-framed structure, 91.7 m x 18.6 m (301 ft x 61 ft) located centrally within the site. The Maintenance shop will include a wash bay and a Maintenance Shop area.

The Warehouse section will be 42.7 m x 18.6 m (140 ft x 61 ft). The Warehouse will be the primary hub for receiving all parts and materials for the mine and processing facilities and shipping of all products.

Process Plant and Maintenance Modular Offices

Additional office space for processing area and maintenance personnel will be provided in modular units located in the vicinity of the Hydrometallurgical Building.

Assay Laboratory

The Process Analysis Laboratory will be housed in a single story, steel-framed building located near the main processing facilities with dimensions of 24.7 m x 14.9 m (81 ft x 49 ft).

Gate House

A Gate House, a portable lease building of 9.1m x 4.9 m (30 ft x 16 ft), will be located at the main site access point. The Gate House will host the security personnel controlling access into the site.

Site Drainage

Stormwater will be collected on-site by a stormwater collection system that will consist of a combination of buried HDPE pipe and surface swales and ditches. Surface water from disturbed areas will be collected in a stormwater retention basin prior to its release into the local stream.

Stormwater that is collected from areas of potential contamination from hazardous material from process areas will be collected separately from other surface water sources and analyzed prior to discharge to the stormwater collection system. The fuel island, as well as the retention pond pipe inlet, will include oily water separators to ensure any petroleum that is in the surface water is not discharged to local waters and is collected for off-site disposal.

Sanitary Wastewater System

Sanitary Wastewater will be transported from the on-site holding tanks to the municipal wastewater treatment plant located in Tecumseh.

Wastewater from all site facilities will be collected in the on-site sanitary wastewater system through an underground PVC SR35 sewer piping network combining manholes and sewage lift stations. The system is designed for a peak flow rate of 750 gpm during peak shower usage and 27,300 gpd daily nominal volume.

First Aid Facilities

The Administration building, as well as the Mine Change building, will each have a first aid station for treatable on-site injuries. There will be an on-site emergency mine rescue vehicle and a rescue trailer. A helicopter pad will be located within the site property for the evacuation of personnel.

Laydown Area / Cold Storage

During normal operation of the mine and processing facilities, there will be minimal need for laydown areas or additional, covered or enclosed storage. The spacing between buildings has been chosen to provide adequate clearances for construction, and space for staging equipment and replacement parts for maintenance and plant turnarounds.

Reagent Storage

Reagents will be used in the Hydromet, Pyromet, Acid Plant, HCI Regeneration Plant and the Paste Backfill Plant. Liquid reagents will be stored in the reagent or raw material tank farm located west of the Hydromet Plant. The tank farm will include truck unloading stations and transfer pumps to transfer reagent to their required process. All tanks of specific reagents will be isolated from other reagents and located within their own diked containment areas.

Process facilities will also store reagent tanks, bunkers, bins and silos. Additional plant reagents will be stored in the process buildings.

Products Storage, Packaging, Shipping

Packaging of the three main products will take place at the outlet of the respective final processing equipment, as a continuation of the process flow.

The anticipated production rates of ferroniobium and titanium dioxide will warrant multiple shipments per week of each product if shipped via over-the-road vehicles. The anticipated production rate of scandium will yield much smaller volumes. Frequency of shipments of scandium will be less frequent justifying the transfer of the containers to a designated storage area inside the Maintenance / Warehouse Building until shipment.

The types and locations of loading and unloading facilities will be specific to the material or products being received and shipped.

Waste Storage

This storage area will include a concrete diked containment area for the storage of wastes, including any hazardous wastes generated at the facility prior to offsite disposal.

Truck Scale

A truck scale will be located near the primary site access.

Fuel Storage - Surface Fuel Station

Fueling facility for surface vehicles will be provided.

18.7 Roads

18.7.1 Main Access Road to Site

The primary access to the site will be from County Road 721. Access into the site will be controlled by security personnel. The site access road will be leading to the main access points to the mine, the administration building and the primary traffic destinations on the site.

18.7.2 Secondary Site Access Roads

A second, emergency access to the site will be connecting to Nebraska State Route 50. The entrance to the emergency access road will be secured with a locked gate.

18.7.3 Secondary Site Roads (to tailings, etc.)

Secondary roads on site include haul roads connecting the plant site to TSF cells and light vehicle access roads connecting infrastructure throughout the site. Haul traffic is expected to include 40-tonne haul trucks delivering tailings and water treatment system residual salt to the active TSF and salt cells and support equipment for the haul fleet. Light vehicles include light-duty pickups and service vehicles supporting infrastructure.

Haul Roads

Haul roads are required to provide access between the plant site and TSF and salt cells. A haul road will provide access to Plant Site TSF Cells 2 and 3 and Salt Management Cell 1 (SMC-1), as well as the Area 7 TSF and salt management Cell 2 (SMC-2). The Area 7 haul road will require a connection between Highway 50 and the TSF area. Improvements to the public roadway may be required based on the haul fleet size delivering tailings and salt to Area 7. Highway-compliant tractor-trailer trucks may be used instead of off-road trucks.

Haul roads are designed to allow trucks to pass safely. The haul trucks assumed for this study are 40 ton articulated off-road trucks. The width of a truck is approximately 4 m wide. To provide safe passage for two-way traffic, the suggested width of the travel way (driving surface) is 3.5 times the width of the truck or 14 m. Haul roads are shown with a width of 20 m, allowing for up to 6 m for safety berms. The road widths and berm placement will be determined during the final engineering design. Much of the roadway may not require a safety berm due to height, but a running width of 20 m is used for estimating and preliminary design purposes. Haul traffic speed is generally slow (30 to 40 km/h); the design shown does not incorporate engineering controls for a specific design speed, but rather are shown for estimating and preliminary design purposes only.

Light Vehicle Access Roads

Light vehicle access roads are located throughout the site. They provide access to dewatering well pads and infrastructure such as ponds, embankment crest and toe fills.

Expected traffic on light duty roads includes light-duty pickup trucks, maintenance equipment, and the occasional haul truck. Light vehicle roads assume occasional use, single-lane traffic with areas to safely pull out of the traffic lane should vehicles meet. A typical light-duty vehicle is approximately 3 m wide. Road widths are designed at 6 m in width.

Speeds are expected to be slow (20 to 30 km/h); the design shown does not incorporate engineering controls for a specific design speed, but rather are shown for estimating and preliminary design purposes only.

Construction

Geotechnical information for soils underlying road alignments is not available at this time. The construction of the roadways assumes similar construction practices as defined for the TSF embankment construction, including removal of 1 m (+/-) of topsoil, replacement with suitable compacted sub-grade fill, and the provision of structural support for traffic with a durable gravel surface. Geotextile fabric will be installed at the base of the gravel layer to provide stability. A minimum of 0.5 m of compacted gravel is assumed for the driving surface.

All roadways will be designed to promote drainage off of the driving surface. This requires that the roadways be elevated slightly above the surrounding ground elevations and crowned, and/or a drainage ditch be provided as needed in areas of elevation transition from cut to fill. In areas where berms are required, notches in the berms should be provided at regular intervals to allow stormwater to discharge off of the roadways. In areas where safety berms are not required, shoulder slopes should not exceed 3:1 (horizontal to vertical), and 4:1 is preferred to reduce the chance of a vehicle rollover should they divert from the roadway.

18.8 Temporary Waste Rock Stockpile

The temporary waste rock stockpile will be used during the sinking of the shaft for storage of topsoil, waste rock and limited quantities of ore. The feasibility design incorporates the following parameters and details:

i. The facility has been divided into three cells (Figure 18-4) to enable waste materials to be stockpiled separately.

ii. A minimum of 1 m of subbase soils will be removed prior to construction of the TSF and stockpiled at the location shown in Figure 18-5.

iii. Based on the current geochemical analysis of the waste rock and ore, the temporary waste rock stockpile will be geomembrane-lined. The liner system for the facility is shown in Figure 18-9 and will incorporate:

- A minimum of 0.6 m of glacial till, amended if necessary, with bentonite, and compacted in layers to result in hydraulic conductivity of less than or equal to 1×10^{-7} cm/s; and
- An 80-mil high-density polyethylene (HDPE) geosynthetic liner placed over the low permeability basin and inside embankment sideslopes.

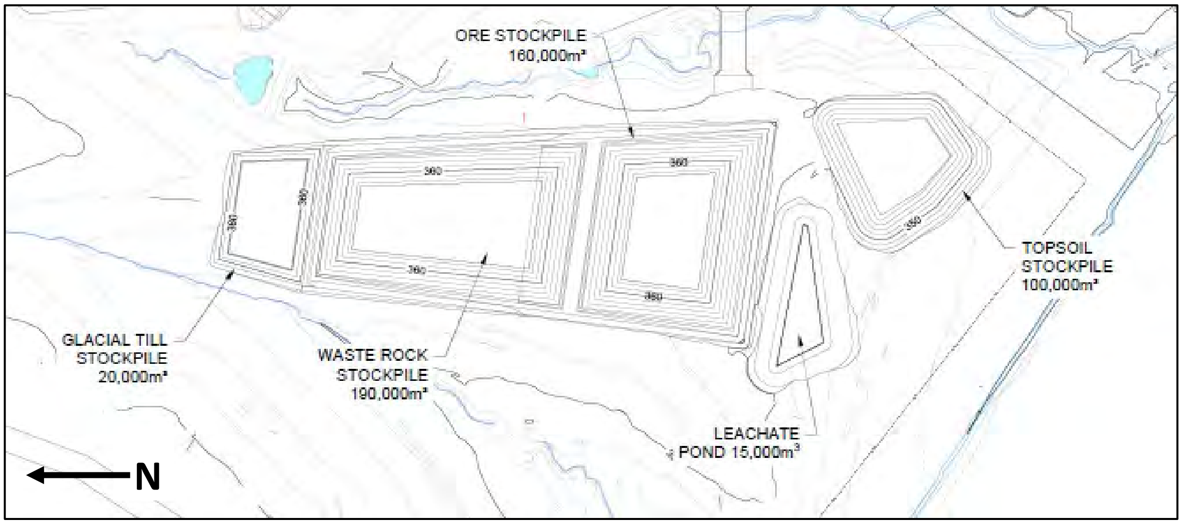
iv. Runoff from the stockpile will gravity drain into a water management pond located to the south of the facility (Figure 18-4). The water management pond liner system will incorporate:

- A minimum of 0.6 m of glacial till, amended if necessary with bentonite, and compacted in layers to result in hydraulic conductivity of less than or equal to 1×10^{-7} cm/s;
- A 60-mil HDPE secondary liner comprised of either an Agru DrainLiner® or geonet and smooth liner;
- An 80-mil high-density polyethylene (HDPE) primary liner;

The DrainLiner or geonet/geomembrane secondary liner will route intercepted flows into a leak collection and recovery system (LCRS) that facilitates pumping of collected seepage water back into the water management pond via a submersible pump and riser pipeline arrangement. As shown in Figure 18-10, the riser pipeline will be contained in a "port" pipeline installed between the two liners. The LCRS sumps are gravel-filled containment areas between the primary and secondary liners, with a horizontal perforated pipe section within the gravel for pumping.

v. Once the plant is operational, the ore will be removed and processed. Waste rock will be used as overliner during TSF construction, and any remaining material will be placed in Plant Site TSF Cell 1 for final disposal.

vi. It is currently anticipated that all waste material will be removed from the temporary waste rock stockpile by Year 2.



Source: SRK, 2019

Figure 18-4: Temporary Waste Rock Stockpile Layout

18.9 Water Management

18.9.1 Surface Water Management for TSF and Temporary Stockpile Areas

The Project is located primarily in the Elk Creek Watershed, near its confluence with Todd Creek. Todd Creek is a tributary of North Fork Big Nemaha River which becomes Big Nemaha River approximately 48 km (30 miles) downstream and joins the Missouri River approximately 72 km (45 miles) downstream.

The TSF and associated ponds will all be located outside and above the limits of the FEMA approximate Zone A flood zone (Figure 18-5).

Hydrologic and hydraulic analyses were performed to evaluate stormwater control requirements for the Elk Creek TSF and supporting facilities. In general, the TSF facilities are located in the uppermost reaches of small catchments in the Elk Creek watershed, and therefore only local diversion of small upstream flows (run-on) around facilities is required.

Stormwater control designs include spillways on the TSF cell and water management pond embankments, and channels on the embankment crests for management of storm runoff from the closed and re-graded surfaces. External stormwater controls include triangular channels (v-ditches) and sediment traps located at the toe of embankments for sediment and erosion control, and culverts to pass flows in drainages through access road crossings (Figure 18-5).

All TSF cell and pond spillways are configured as a 0.5 m deep by 3 m wide trapezoidal channel (notch) with 10:1 sideslopes, oriented perpendicular to the embankment crest and can pass the PMF storm event; a channel (down-chute) lined with riprap (or HydroTurf) will convey flows down embankments and into stilling basins.

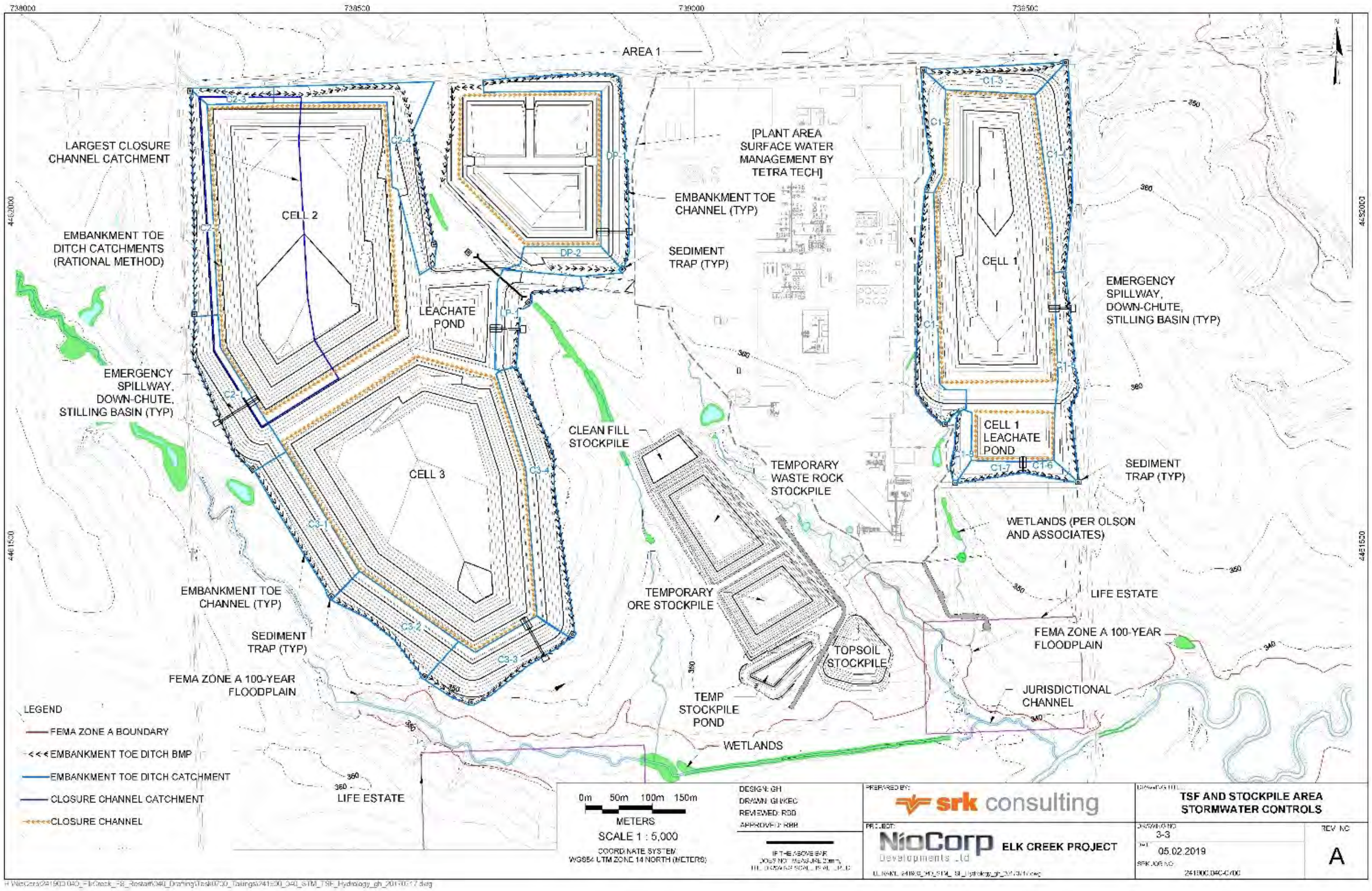


Figure 18-5: TSF and Stockpile Area Storm Water Control

All other stormwater controls are design to withstand the peak flow rate from the 100-year, 24-hour storm event.

Peak flow rates for spillway, culvert and closure channel sizing were determined utilizing HEC-HMS version 4.2 (HEC-HMS), released by the US Army Corps of Engineers, Hydrologic Engineering Center (USACE 2016). The Rational Method was used to evaluate peak flow rates and channel (v-ditch) sizing for the embankment toe channels. Hydraulic calculations were performed in Bentley FlowMaster version 8i (Bentley 2009), and HY-8 version 7.5 (HY-8) released by the Federal Highway Administration (FHWA 2016).

Rainfall data used in the hydrologic analyses was obtained from the National Oceanic and Atmospheric Administration (NOAA), Precipitation Frequency Data Server (PFDS) which for the State of Nebraska utilizes NOAA Atlas 14, Volume 8 Version 2 (NOAA 2013) (Table 18-4). The probable maximum precipitation (PMP) storm depth of 21” (533 mm) was obtained from the document Nebraska Statewide Probable Maximum Precipitation Study (Tomlinson et al., 2008), recommended for use in Nebraska by Nebraska Department of Natural Resources.

Table 18-4: 24-Hour Precipitation Depth-Frequency Data for Elk Creek Project

24-Hour Storm Event		
Frequency	**Depth (mm)**	**Depth (inches)**
2-Year	81	3.2
10-Year	122	4.8
25-Year	150	5.9
50-Year	173	6.8
100-Year	198	7.8
PMP (1)	533	21
Point Precipitation Depths		
Latitude: 40.2667°		
Longitude: -96.1833°		
Elevation: 1126 ft (est. from Google Earth)		

Source: NOAA Atlas 14 Precipitation Frequency Server

(1) PMP estimate from Nebraska Statewide Probable Maximum Precipitation Study (Tomlinson et al., 2008) by Applied Weather Associates (recommended for use by Nebraska Department of Natural Resources).

18.10 Tailings Surface Logistics

The tailings will be transported by conveyor from the Hydromet building to a temporary staging area, which will have concrete containment and will be sheltered by a sprung structure comprised of tensioned-fabric over a steel frame structure. Slag will be transported from the Pyromet building by truck or skid loader to the same temporary staging area. The material will be transported by truck to the active TSF cell or utilized for paste backfill by the surface paste backfill plant, once the slag has been properly characterized for use in the backfill application.

18.11 Tailing Storage and Associated Facilities

Preliminary investigations performed by SRK included a comparison of potential TSF sites for both slurry and filtered (dewatered) tailings disposal options. This comparison considered potential engineering, strategic, permitting and closure issues, including:

- Engineering: Containment area, required reclaim for water balance on tailings impoundment, relative embankment heights, distance to plant, pumping head for slurry (plant to impoundment) and reclaim water (impoundment to plant), upstream stormwater management, major road crossings, potential residential relocations, and potential road relocations.
- Strategic: Proximity to major roadways, churches and cemeteries, visual embankment heights, and property ownership.
- Permitting: Major drainage crossings and major road encroachment.
- Closure: Closure cover areas and volumes, seepage potential, and mass stability.

Of eight potential sites, Area 7 and Area 1 ranked first and second for both slurried and filtered tailings, respectively. This evaluation included the development and implementation of a preliminary foundation characterization plan for both Area 1 and Area 7 and development of preliminary water balance spreadsheets for both slurried and filtered tailings options for both sites.

Following the development of the 2015 PEA, the decision was made to only generate dry tailings, by calcining and filtration processes, and a more detailed foundation characterization investigation was performed for Area 7. Revised planning indicated that a significant portion of the filtered tailings would be used for underground backfill operations, limiting the total tailings tonnage to be disposed of in the TSF cells to around 1,070 dry t/d for a life of 36 years (from the original plan for 4,930 t/d for 30 years).

This significant decrease in deposition rate, as well as the finding that the calcined tailings material will be a dry "clinker" with a sandy gravel or gravelly sand gradation (i.e., well drained), led to NioCorp's decision to evaluate the plant site (refer to TSF Cells 1, 2 and 3 in Figure 18-6) as feasible tailings storage and stormwater management locations for the first 19 years of operations, with the following significant advantages:

- No access roads or conveyors crossing Elk Creek.
- Shorter distance from Plant Area for tailings transport and reclaim water management.
- Reduction in stormwater management.
- Consolidation of disturbance into a much smaller area (without Area 7).

The plant area was therefore considered the best option for the first 19 years of management and storage of dry tailings (in three, State-approved “solid waste” disposal facilities or cells), and management of precipitation contacting the tailings via runoff and infiltration in separate double-lined leachate collection ponds. Once the Plant Site TSF cells are full, a new facility will be constructed at Area 7.

The feasibility design incorporates the following parameters and details:

i. Topography: Feasibility design has been performed using 1 m contoured topography.

ii. Feasibility Design: Feasibility design of the TSF Cells is based on dam safety regulations, solid waste regulations (including tailings placement/compaction/interim covering), leachate water management regulations, and radioactive licensing regulations, all discussed in Section 20. The design is intended to demonstrate compliance with Nebraska industrial solid waste regulations for design, operation and closure and is based on a meeting held between NioCorp, SRK and the Nebraska Department of Environmental Quality Solid (NDEQ).

iii. Embankment Cross-Section: All TSF and LCP embankment sections will incorporate a 20 m crest width and 3 (horizontal) to 1 (vertical) sideslopes as shown in Figure 18-7 and Figure 18-9. The vegetation of the embankment crests and downstream sideslopes will be provided for erosion protection, immediately after construction completion of each embankment.

iv. Leachate Collection Ponds (LCPs): Each of the three LCPs will be utilized for management of precipitation runoff and drainage from the tailings' solids. The ponds will be lined with two layers of geomembrane liner (primary and secondary), sandwiching a permeable spacer that allows evacuation of all leakage through the primary liner to be collected in a lined sump area, or leakage collection and recovery system, and pumped back into each leachate collection pond, thereby providing a means of long-term leakage control (refer to Figure 18-10).

v. Tailings Production Rate: The tailings production rate is an average of 2,460 dry t/d consisting of 1) 825 dry t/d of water leach residue tailings; 2) 1,588 dry t/d of calcined excess oxide tailings, and 3) 46 t/d of slag. Of this, an average of 1,390 dry t/d will be placed into the mine backfill (Section 18.14), and 1,070 dry t/d will be placed in the TSF. Testing performed on the excess, and insoluble oxides indicate that a loose (non-compacted) dry density of 1.6 t/m^3 will be achieved without compaction, and that placement and spreading of the dry tailings will increase the density to between 1.7 and 1.8 t/m^3.

vi. Growth Media Salvage: A minimum of 1 m of subbase soils will be removed prior to construction of each TSF cell and stockpiled at the locations shown in Figure 18-9, as follows:

- For Plant Site TSF Cell 1, the topsoil will be stockpiled in the Plant Site TSF Cell 3 footprint;
- For Plant Site TSF Cell 2 and Cell 3, topsoil will be stockpiled in the footprint of the temporary waste rock stockpile (which will have ended its design life and been removed to Plant Site TSF Cell 1 by the time that these facilities are constructed; and
- For Area 7 Cell 1 topsoil will be stockpiled in the specified location in Area 7.

vii. TSF Area, Storage and Time Characteristics: The TSF system includes four TSF cells, Plant Site Cell 1, Cell 2 and Cell 3 and Area 7 Cell 1. Table 18-5 provides a summary of TSF cell footprint areas, storage characteristics and time periods.

Table 18-5: TSF Area, Storage and Time Characteristics

Cell No.	Approximate Footprint Area (Ha)	Storage Capacity (@ dry density of 1.7 t/ m^3) (Mt)	Time Period (years after commissioning)
Plant Area TSF			
1	8.8	1.5	3
2	14.1	3.1	4-10
3	13.9	3.1	10-18
Area 7 TSF			
1	26.6	6.7	19-36

Source: SRK, 2019

viii. Leachate Collection Pond (LCP) Storage Characteristics: The LCP system includes three ponds; one for Plant Site Cell 1 (LCP-1), one for Plant Site Cells 2 and 3 (LCP-2) and one for Area 7 Cell 1 (Area 7-LCP). Table 18-6 provides a summary of LCP footprint areas and leachate and stormwater storage characteristics.

Table 18-6: LCP Area, Storage and Time Characteristics

LCP	Approximate Footprint Area (Ha)	Total Storage Capacity ([1]) (n^s)
LCP-1	0.9	29,200
LCP-2	1.5	61,600
Area 7 LCP	4.3	240,000

Source: SRK, 2019
(1) For operating and 100-year stormwater conditions.

ix. Structural Embankment Foundation Preparation: Foundation preparation for all TSF and LCP embankments will incorporate removal of a minimum of 0.5 m of native soils, and re-compaction in layers to form a non-settling structure as shown in Figure 18-9 and Figure 18-10.

x. Embankment Compaction: All TSF and LCP embankments will be constructed using soil borrowed from within the respective TSF basins and compacted in layers to form a non-settling structure, as shown in Figure 18-9 and Figure 18-10.

xi. Embankment Raises: All tailings embankments will be constructed to completion before each cell is commissioned as shown in Table 18-7, thereby eliminating the need for raising extension of the TSF liner and embankment drainage elements at any stage. This assists in preventing the facility from being affected by potential liquefaction of the tailings' solids under seismic loads.

xii. TSF Liner and Above-Liner Drainage System: The components of the facility liner/drainage systems are described below and shown in Figure 18-7 through Figure 18-10.

a. The TSF basins and inside embankment sideslopes will incorporate:

i. A minimum of 0.6 m of glacial till, amended if necessary, with bentonite, and compacted in layers to result in hydraulic conductivity of less than or equal to 1×10^{-7} cm/s;

ii. An 80-mil high-density polyethylene (HDPE) geosynthetic liner placed over the prepared basin subgrade and inside embankment sideslopes; and

iii. For Plant Site TSF Cell 1, an above-liner centralized drain (i.e., from north to south) directing drain flows into an above-liner, double-lined, leak-detected sump that facilitates pumping into LCP 1.

iv. For Plant Site TSF Cells 2 and 3 and Area 7 TSF Cell 1, above-liner embankment toe drains (along entire inside perimeter), as well as centralized drains that gravitate drain flows into above-liner, double-lined, leak-detected sumps that facilitate pumping into LCP 2 and Area 7 LCP, respectively.

v. Typical drain sections are provided in Figure 18-8 for the centralized drain (Plant Site Cells 1, 2 and 3), and the perimeter inside toe drains (Plant Site Cells 2 and 3 and Area 7 Cell 1).

b. Lining for LCP Basins and Inside Sideslopes: LCP basins and inside embankment sideslopes, as well as the leak collection and recovery system (LCRS) sumps within the TSF cells, will incorporate:

i. A minimum of 0.6 m of glacial till, amended if necessary, with bentonite, and compacted in layers to result in hydraulic conductivity of less than or equal to 1×10^{-7} cm/s;

ii. A 60-mil HDPE secondary liner incorporating an Agru DrainLiner® system or geonet layer plus smooth geomembrane;

iii. An 80-mil HDPE primary liner;

The DrainLiner or geonet layers will gravitate into an LCRS sump that facilitates pumping of collected seepage water back into the LCPs via a submersible pump and riser pipeline arrangement. As shown in Figure 18-13, the riser pipeline will be contained in a "port" pipeline installed between the two liners. The LCRS sumps are gravel-filled collection areas between the primary and secondary liners, with a horizontal section of perforated pipe within the gravel for pumping.

xiii. Tailings Solids Transportation and Deposition: A cost trade-off study was performed to compare trucking to conveying costs and trucking as selected as the preferred option. Tailings solids will be trucked from the process plant directly to each planned deposition location at the TSF Cells, dumped, spread and compacted using a bulldozer, and graded to slope to facilitate control of surface water. Tailings will be placed in sections in the cells starting at the high point in the base grading and working toward the sumps. Each cell will be closed in phases every 3 to 4 years, once the full depth of tailings has been achieved in each section, as described in Section 20.5.

a. For Plant Site TSF Cell 1 and Area 7 TSF Cell 1, tailings placement will be performed from south to north.

b. For Plant Site TSF Cell 2 and 3, tailings placement will be performed from north to south.

xiv. TSF Stage-Area-Capacity Data: Stage-capacity data is provided in Table 18-7 and summarizes TSF elevation, area, cumulative volume, capacity in tonnes, time in years and rate-of-rise in metres per year.

xv. Surface Water Management: Surface water management comprises both precipitation-induced contact water and non-contact water.

a. Surface water contacting the tailings will be managed via dedicated pump arrangements for all three cells that comprise slotted HDPE riser pipes located above the liner system at the impoundment low topography on the embankment inside slopes. Submersible pumps will be used to pump collected water into the LCPs. The submersible pumps will be maintained above the current tailings elevation at all times. The locations of the riser pipes are shown in Figure 18-9. Any infiltrating surface water will be collected in the TSF above-liner drainage system.

b. The average rainfall is shown in Table 18-8. Based on the average monthly precipitation, pump back from the TSF underdrainage system has been estimated:

- For Plant Site TSF Cell 1 at an average of 36 gpm varying between 64 gpm and 11 gpm,
- For Plant Site TSF Cell 2 at an average of 58 gpm varying between 104 gpm and 18 gpm, and
- For Plant Site TSF Cell 3 at an average of 57 gpm varying between 102 gpm and 18 gpm.
- For Area 7 TSF Cell 1 at an average of 105 gpm varying between 180 gpm and 35 gpm.

c. Storm-related precipitation depths are provided for 25-year and 100-year, 24-hour duration storms in Table 18-4. Based on 100-year precipitation depth, the pump back requirements for the 100-year condition is estimated to require 115 gpm from Plant Site Cell 1, 190 gpm from both Plant Site Cells 2 and 3, and 373 gpm from Area 7 Cell 1.

d. Non-contact surface water will be managed via channels, spillways, and culverts as described in Section 18.9.1 and shown in Figure 18-6. Spillways are sized to pass the PMF storm event, and all other stormwater controls are designed to accommodate 100-year, 24-hour storm event precipitation.

Table 18-7: Tailings Storage Facility Stage-Area-Capacity Data

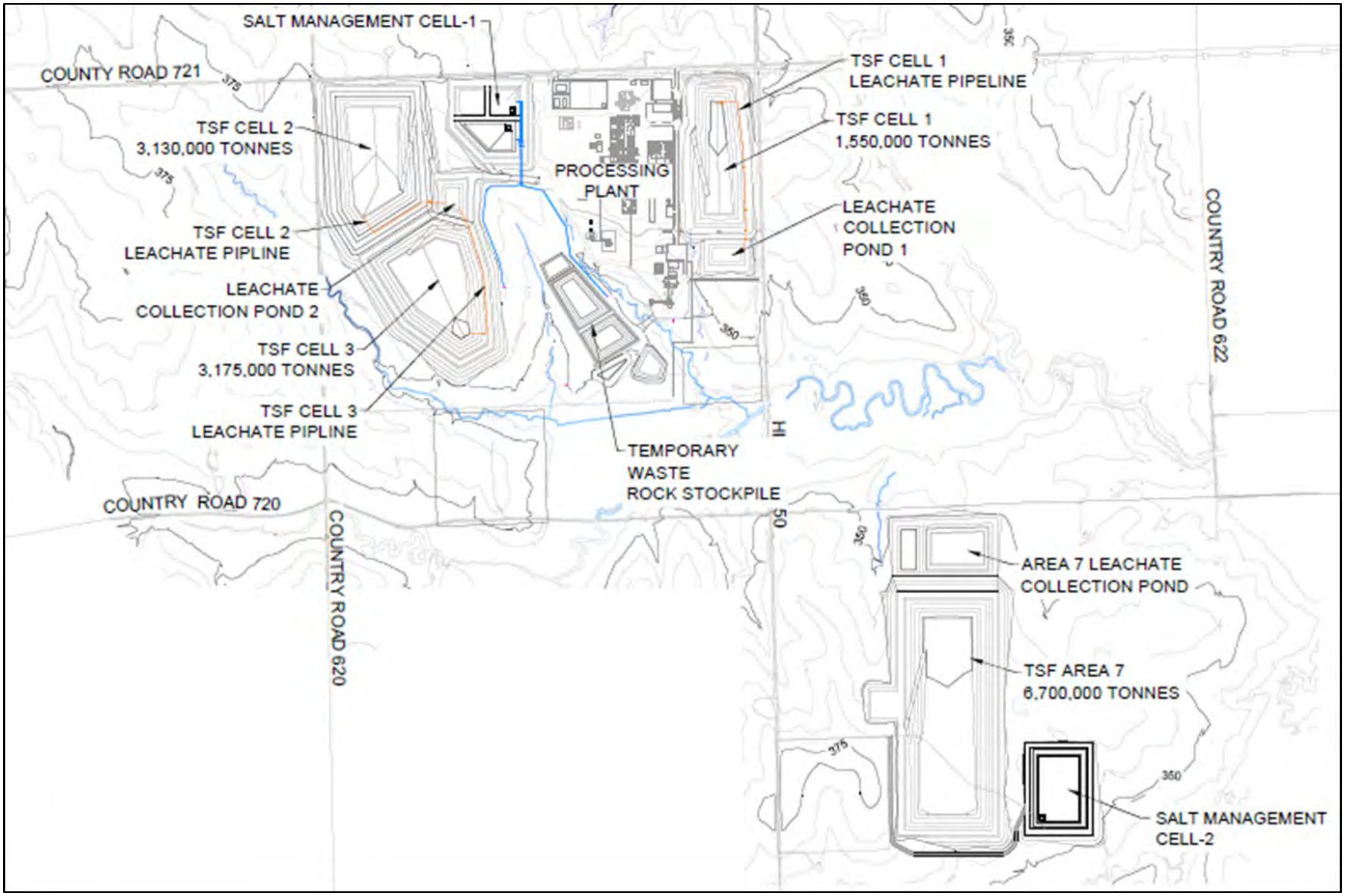
Cells	TSF Elevation (masl)	Area (m^2)	Cumulative Volume (m^3)	Capacity ([1]) (t)	Years	Rate of Rise (m/y)
Plant Site Cell 1	349	3,343	0	0	0.0	0.0
	350	9,570	6,456	10,330	0.0	36.6
	352	26,589	42,022	67,235	0.2	10.7
	354	36,889	107,528	172,045	0.5	6.7
	356	43,353	187,734	300,375	0.8	5.7
	358	50,109	281,160	449,856	1.2	4.9
	360	57,163	388,395	621,431	1.6	4.3
	362	64,516	510,036	816,058	2.2	3.8
	364	72,168	646,682	1,034,692	2.7	3.4
	366	80,119	798,932	1,278,291	3.4	3.0
	368	88,369	967,383	1,547,813		
Plant Site Cell 2	360	25,179	21,391	34,225	3.8	14.0
	362	63,468	109,856	175,770	4.2	4.4
	364	74,603	249,913	399,861	4.8	3.2
	366	81,976	406,456	650,329	5.5	2.9
	368	89,630	578,027	924,842	6.2	2.7
	370	97,567	765,189	1,224,302	7.0	2.5
	372	105,786	968,507	1,549,611	7.8	2.3
	374	114,286	1,188,544	1,901,670	8.8	2.1
	376	123,069	1,425,864	2,281,382	9.8	2.0
	378	132,133	1,681,031	2,689,649	10.8	1.8
	380	141,480	1,954,609	3,127,374		
Plant Site Cell 3	351	14,534	8,924	14,279	11.5	26.5
	352	32,842	32,613	52,180	11.6	10.0
	354	68,836	139,323	222,917	12.0	3.8
	356	76,009	284,360	454,976	12.6	3.2
	358	83,004	443,339	709,343	13.3	2.9
	360	90,251	616,564	986,502	14.0	2.7
	362	97,765	804,545	1,287,272	14.8	2.5
	364	105,554	1,007,830	1,612,527	15.7	2.3
	366	113,609	1,226,960	1,963,136	16.6	2.1
	368	121,919	1,462,457	2,339,931	17.6	2.0
	370	130,486	1,714,827	2,743,724	18.7	1.8
	372	139,324	1,984,603	3,175,365		
Area 7 Cell 1	348	3,936	0	0	0.0	0.0
	350	39,195	43,131	73,323	19.4	11.0
	352	78,499	160,825	273,402	19.9	4.0
	354	125,046	364,370	619,428	20.8	2.3
	356	155,733	645,148	1,096,752	22.0	1.7
	358	167,568	968,449	1,646,363	23.3	1.5
	360	179,701	1,315,718	2,236,721	24.8	1.4
	362	192,133	1,687,552	2,868,839	26.4	1.3
	364	204,863	2,084,549	3,543,733	28.1	1.2
	366	217,892	2,507,304	4,262,416	29.9	1.1
	368	231,219	2,956,414	5,025,904	31.8	1.1
	370	244,844	3,432,477	5,835,211	33.8	1.0
	372	258,768	3,936,089	6,691,351	35.9	0.9
	373	265,842	4,198,394	7,137,270	37.0	0.9

Source: SRK, 2019. (1) Tonnes of storage is based on an assumed dry density of 1.7 t/m^3.

Table 18-8: Mean Monthly Average Precipitation

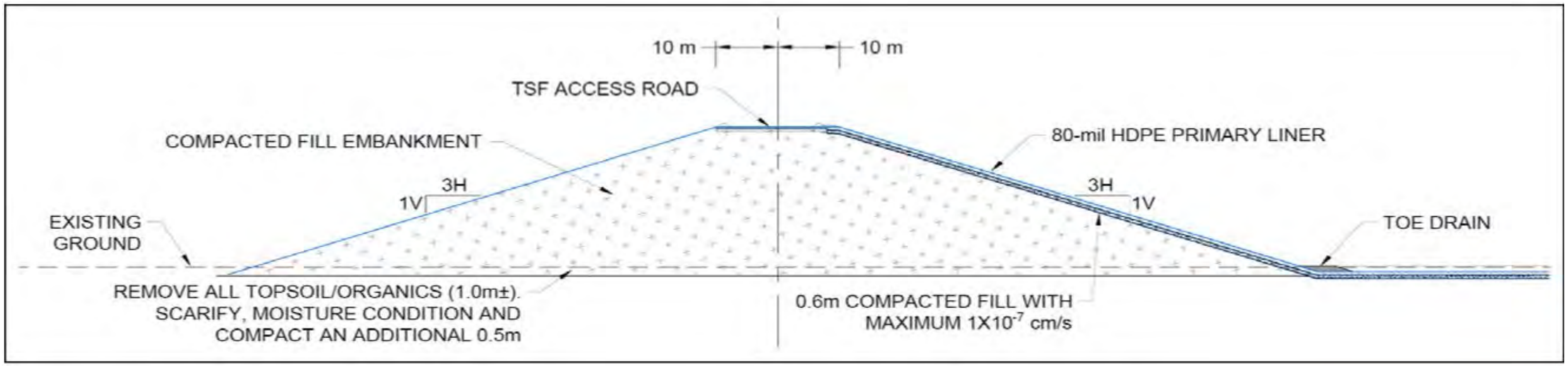
Station	Mean Monthly Precipitation	Mean Monthly Pan Evaporation	Mean Monthly Lake Evaporation	Annual Potential Evapotranspiration (PET)
	Tecumseh Station' (mm)	Sabetha Lake Station[2] (mm)	Sabetha Lake Station[2] (mm)	Rainwater Basin Station[3] (mm)
Jan	21		-	30
Feb	28	-	-	32
Mar	49	-	-	66
Apr	72	131	98	84
May	111	167	126	98
Jun	117	186	139	98
Jul	99	210	158	102
Aug	97	190	142	87
Sep	89	138	103	86
Oct	58	103	77	81
Nov	39	57	43	58
Dec	26	-	-	29
Annual	805	1182	887	851
Seven-Year Wet-Cycle	**6,662**			
Seven-Year Dry-Cycle Total	**4,318**			

1. Tecumseh station data (WRCC, DRI) is considered the most representative based on elevation and proximity to the Project site.
2. Data from Southwest Climate and Environmental Information Collaborative (WRCC, DRI); Sabetha Lake station data is considered the most representative based on elevation and proximity to the Project site.
3. RAWS Network (DRI), ASCE Standardized Reference ET Calculations
4. 5-year average from 2009 through 2013
5. Based on Lake Evaporation as 75% of Pan Evaporation



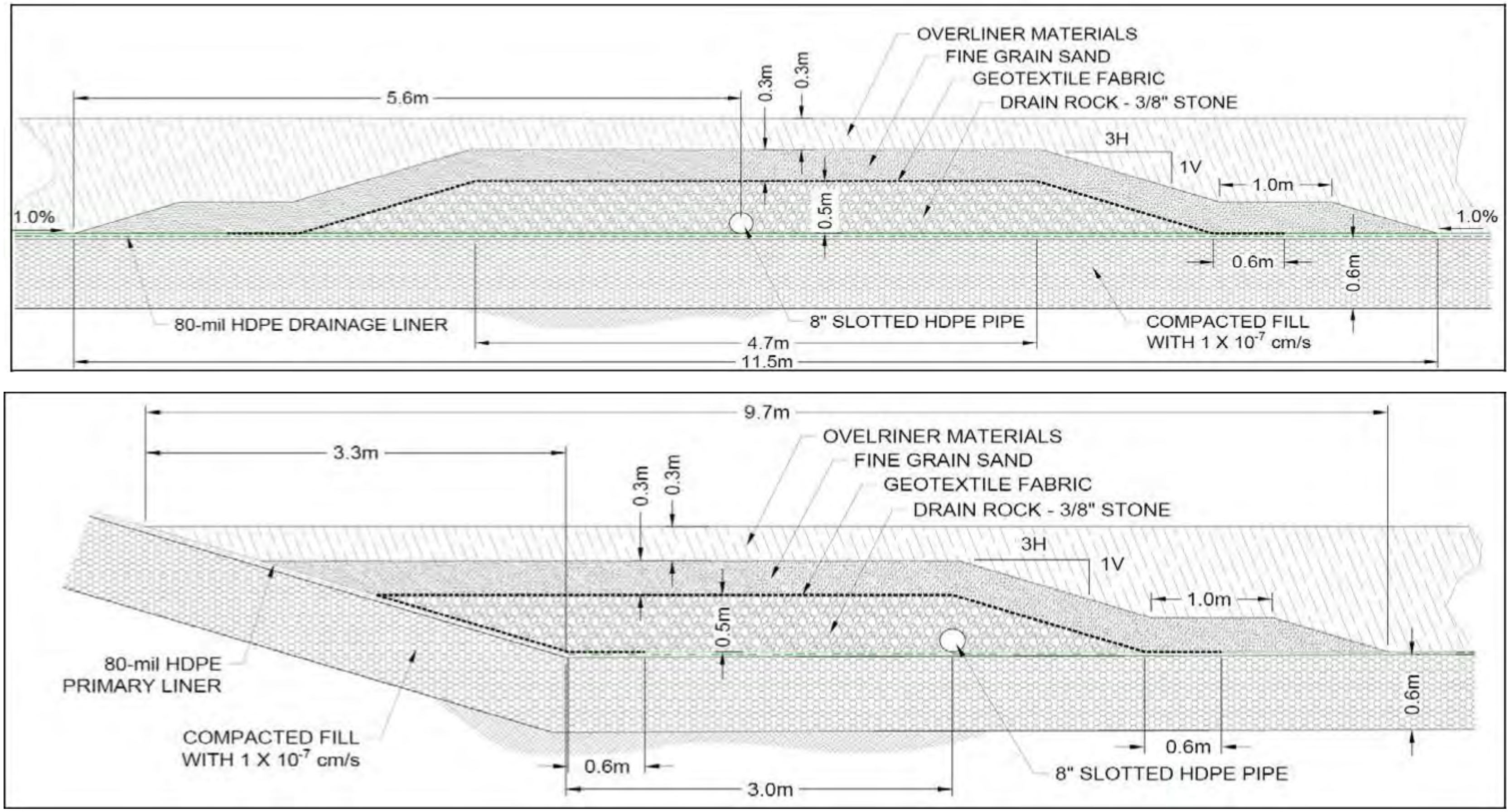
Source: SRK, 2019

Figure 18-6: Tailings Storage Facility Layout Showing Plant Site Cells 1, 2 and 3 and Area 7 Cell 1



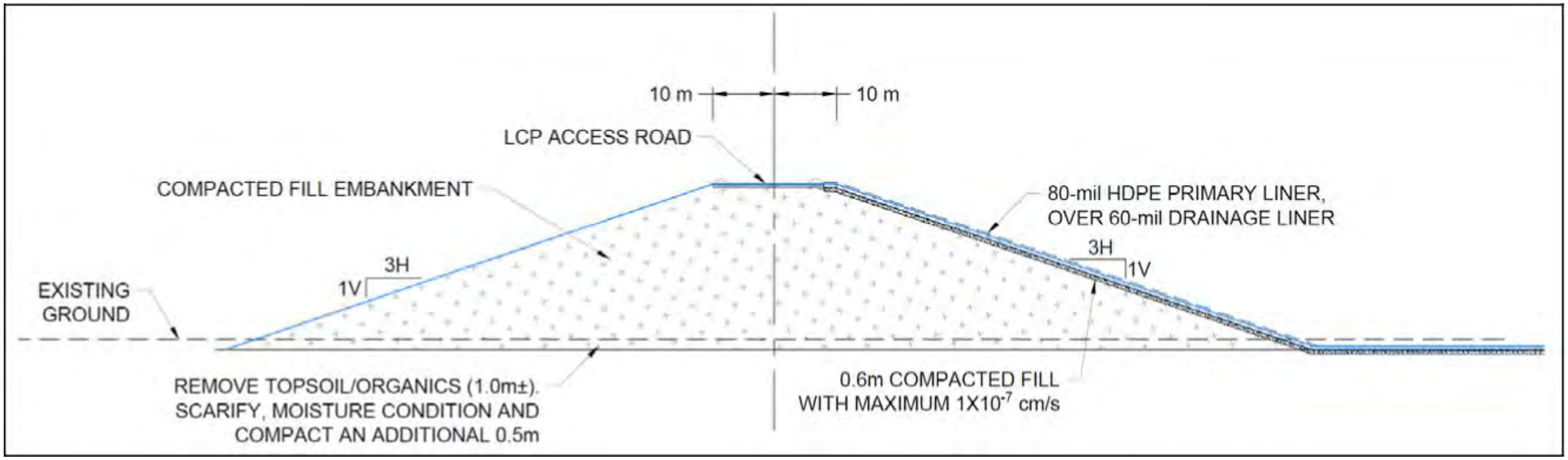
Source: SRK, 2019

Figure 18-7: Tailings and Waste Rock Storage Area Embankment Cross-Section



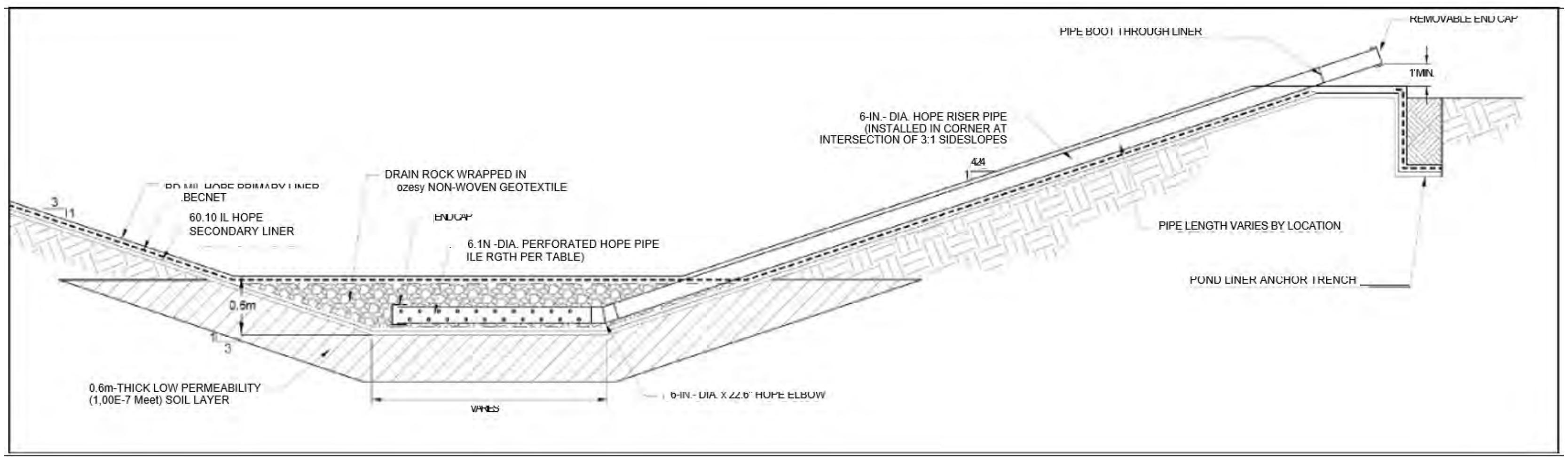
Source: SRK, 2019

Figure 18-8: Tailings Storage Facility Central and Toe Drain Details



Source: SRK, 2019

Figure 18-9: Leachate Collection Pond Embankment Cross-Section



Source: SRK, 2019

Figure 18-10: Leachate Collection Pond LCRS System

18.12 Salt Management Cells

The crystalline salt produced as a waste product by heating and evaporating brine from the Reverse Osmosis (RO) water treatment plant will be transported by conveyor to the temporary salt staging area within the aforementioned sprung structure over concrete containment. The salt will then be transported by truck to the dedicated salt management cells.

The feasibility design incorporates the following parameters and details:

A. Feasibility design has been performed using 1 m contoured topography.

B. The salt management cell embankment sections will incorporate a 20 m crest width and 3:1 sideslopes, as shown in Figure 18-8 and Figure 18-10. Revegetation of the embankment crests and downstream sideslopes will be provided for erosion protection immediately after the construction of each embankment.

C. A minimum of 1 m of subbase soils will be removed prior to construction of the SMCs and stockpiled for future use as growth media for site closure.

D. Foundation preparation for the embankments will incorporate removal of a minimum of 0.5 m of native soils, and re-compaction in layers. The embankments will then be constructed using soil borrowed from within the active SMC footprint, and compacted in layers to form a non-settling structure.

E. SMC embankments will be constructed to their ultimate configuration before each cell is commissioned, as opposed to construction in phases.

- The SMCs will incorporate, as described from the sub-base vertically upwards, the following:
- A minimum of 0.6 m of glacial till, amended if necessary with bentonite, and compacted in layers to result in hydraulic conductivity of less than or equal to 1×10^{-7}cm/s;
- A 60-mil HDPE secondary liner incorporating an Agru DrainLiner® system or geonet plus smooth geomembrane; and
- An 80-mil high-density polyethylene (HDPE) primary liner.

The DrainLiner or geonet layers will route collected leakage into an LCRS sump that facilitates pumping of collected water back into the SMCs or LCPs via a submersible pump and riser pipeline arrangement as shown in Figure 18-10.

F. The salt production rate is anticipated to be 45,250 m^3 per year with a total of 1.63 million m^3 required for the life of the mine. The SMC system includes two cells, SMC-1 and SMC-2, adjacent to TSF Cell 2 and Area 7 TSF Cell 1, respectively. Table 18-9 provides a summary of SMC footprint areas and storage capacities.

Table 18-9: SMC Footprint Areas and Storage Capacities

LCP	Approximate Footprint Area (Ha)	Total Storage Capacity (m^3)	Time Period (year after commissioning)
SMC-1	9.7	700,000	15.5
SMC-2	10.2	930,000	36

Source: SRK, 2019

G. Stormwater will be managed within each Salt Management Cell by spray evaporating within the open portion of each cell until the useable area is too small or the volume of stormwater collected is too great. The base of each cell will be graded to an internal sand drain, and sump from which captured runoff will be pumped to the internal evaporator system or the nearest tailings leachate collection pond. Salt will be placed in each cell (via trucks), starting upstream and progressing downstream to the internal sump location. Temporary covers will be employed to minimize the exposed salt to water. As portions of each cell reach maximum design height, they will be progressively closed using geomembranes and soil cover to minimize exposed salt and contact water again. When the salt placement sequence requires filling over the internal sump, a riser housing consisting of an HDPE pipe laid along the cell sideslope will be installed to provide protection to the pump and discharge lines.

18.13 Paste Backfill Plant and Underground Distribution

18.13.1 Surface Plant

The backfill system at the Elk Creek Mine is designed to be a multiproduct system, which during construction will produce concrete and grout for construction and water control. During mine production the plant will add paste backfill to its products. The paste backfill will be made from leach residue and oxide material produced by the Hydromet plant.

Source: Nordmin, 2019

Figure 18-11: NioCorp Concrete and Paste Backfill Plant

18.13.2 Backfill Testwork

The backfill paste formulation and characterization tests were performed at SGS Canada in Lakefield, Ontario between March and July 2017. The mix designs were formulated to achieve 1 MPa at 28-days cure time, meeting the strength requirements for the mine operation. Several mixes were tested using a 5% binder, including 100% cement, 100% fly ash, and mixtures of the two. As well, the ratios between oxide and leach residue were tested at 75/25 and 60/40 mixes.

The resulting Phase 2 tests demonstrated that the formulations met or exceeded the 1 MPa at 28-day mining requirement using a 5% binder. The mixes tested are presented in Table 18-10.

Table 18-10: Paste Backfill Formulations for Phase 2 Testwork.

Mix #	EO/LR	Excess Oxide	Leach Residue	Type GU Cement	Fly Ash Binder	Note
1	75/25	71%	24%	5%	0%	Control
2	60/40	57%	38%	5%	0%	Design Production Rate
3	75/25	71%	24%	2.50%	2.50%	50/50 GU/FA
4	75/25	71%	24%	3.75%	1.25%	75/25 GU/FA

GU Is "General Usage" Or Standard Portland Cement.

The subsequent results at 7 days cure time were positive. All of the mixes resulted in high strength fill (see Table 18-11).

Table 18-11: Results of Phase 2 UCS Testing of paste backfill samples after 7 days

Mix #	Cube 1 (MPa)	Cube 2 (MPa)	Cube 3 (MPa)	Average (Mpa)
1	5.9	6.0	6.3	6.1
2	8.3	8.0	8.6	8.3
3	4.9	4.9	5.0	4.9
4	5.6	6.2	6.4	6.1

The results of the test work proved that a) the use of the oxides with the leach residue was suitable for backfill use and b) the resulting product would have sufficient strength. Based on these results, further work can be done to optimize the recipe, as the total binder requirement to achieve 1 MPa at 28 days is clearly lower than 5%.

A calcination pilot program carried out at Hazen Lab in Golden, Colorado suggested that the formation of Srebrodolskite ($Ca_2Fe_2O_5$) significantly reduced the CaO content in the oxide solids. This has allowed the backfill paste system to utilize the oxide from the Hydromet plant as a source of solids without having a self-heating issue due to the exothermic reaction from hydrating CaO.

18.13.3 Paste Plant Process

The paste plant receives 28.1 t/h of the leach residue filter cake and up to 70 t/h of the oxide solids for making paste backfill. The leach residue filter cake is expected to have 20% to 25% moisture, and the oxide solids are completely dry (due to the high-temperature calcination process). If required, the oxide solids will be crushed to minus 2 mm, with a suitable particle size distribution, before they are used in the process. The crushing process will be a dry process in order to avoid the introduction of additional water into the paste mix. The filter cake from the Hydromet plant will be fed to a conveyor belt with a live bottom feeder, while the dry oxide solids will be stored in a silo before they are fed to the system. The system has a nominal production of 37.8 m^3/h or 81.9 wet t/h. However, the design production rate is 62.4 m^3/h or 137.8 wet t/h. This design rate allows the paste plant to catch up to the supply from the Hydromet (at a throughput of 65% over the nominal rate) and keep pace with the required paste fill for the underground mining effort. The system is expected to run continuously once the stopes become available for backfill, except for planned maintenance.

Concrete production from the plant will be supplied by raw materials from three silos (fly ash and cement) as well as an aggregate bunker. Fly ash is used to reduce the cost and to improve the performance of the cement. Typically, 15% to 30% of the cement can be replaced with the fly ash. Dependent upon the scheduled completion of the plant, the Elk Creek Mine will also be able to supply all concrete for foundations, shaft liners and other installations throughout the site. Prior to the completion of the plant, the mine will use concrete supplied by a local third-party producer.

Concrete produced either on-site in the plant or off-site from a third party producer will be fed into the shaft slicklines via truck and hopper located within either of the two headframes.

18.13.4 Underground Distribution of Paste Backfill

The paste is pumped underground from the backfill plant with positive displacement pumps (one in operation and one spare), through the production headframe, into the shaft via either of two carbon steel schedule 80 pipes anchored to buntons, discharging on the appropriate level. A short section of surface pipe will be required to broach the gap between the paste backfill plant and the production headframe.

It is imperative that the paste backfill lines remain clean between each use. In order to ensure cleanliness, water is flushed through the system from the backfill plant through to the underground. Additionally, cleaning pigs are to be used to remove any materials from the inside walls of the slicklines.

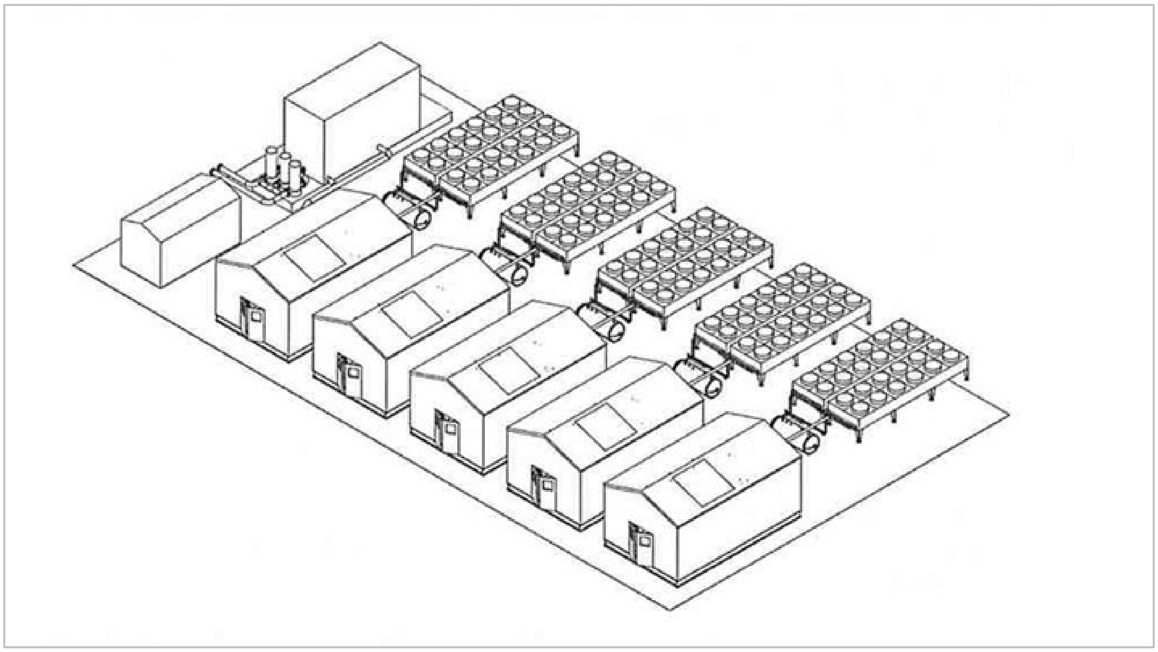
As detailed within Section 16.6.3, barricades are to be installed in the lower access drift to the stopes, development level pipe extensions are added to the shaft slicklines from the production shaft via the upper access drift into the stopes, backfill paste flows and fills the stope. Once the stope is filled the backfill is allowed to cure (28 days) to the design strength of over 1 MPa before blasting on the adjoining stope. This ensures the maximum loading on the barricade is kept under 200 kPa.

18.14 Freeze Plant

Key to the revised plan to develop the shafts for the mine access will be the installation of a freeze plant that will provide super-cooled brine to be utilized for freezing the ground from the surface through the limestone to the carbonatite interface. The use of this technology allows the project to complete these excavations without the need for an extensive pumping system.

The freeze plant will require a 4 MW cooling facility that will prepare and recirculate supercooled brine through a number of deep boreholes surrounding the two shafts. The boreholes, which will be 200 mm (8 ") in diameter, will utilize insert pipes of a smaller diameter to push the brine down to the carbonatite and allowing it to recirculate to the surface and back to the freeze plant.

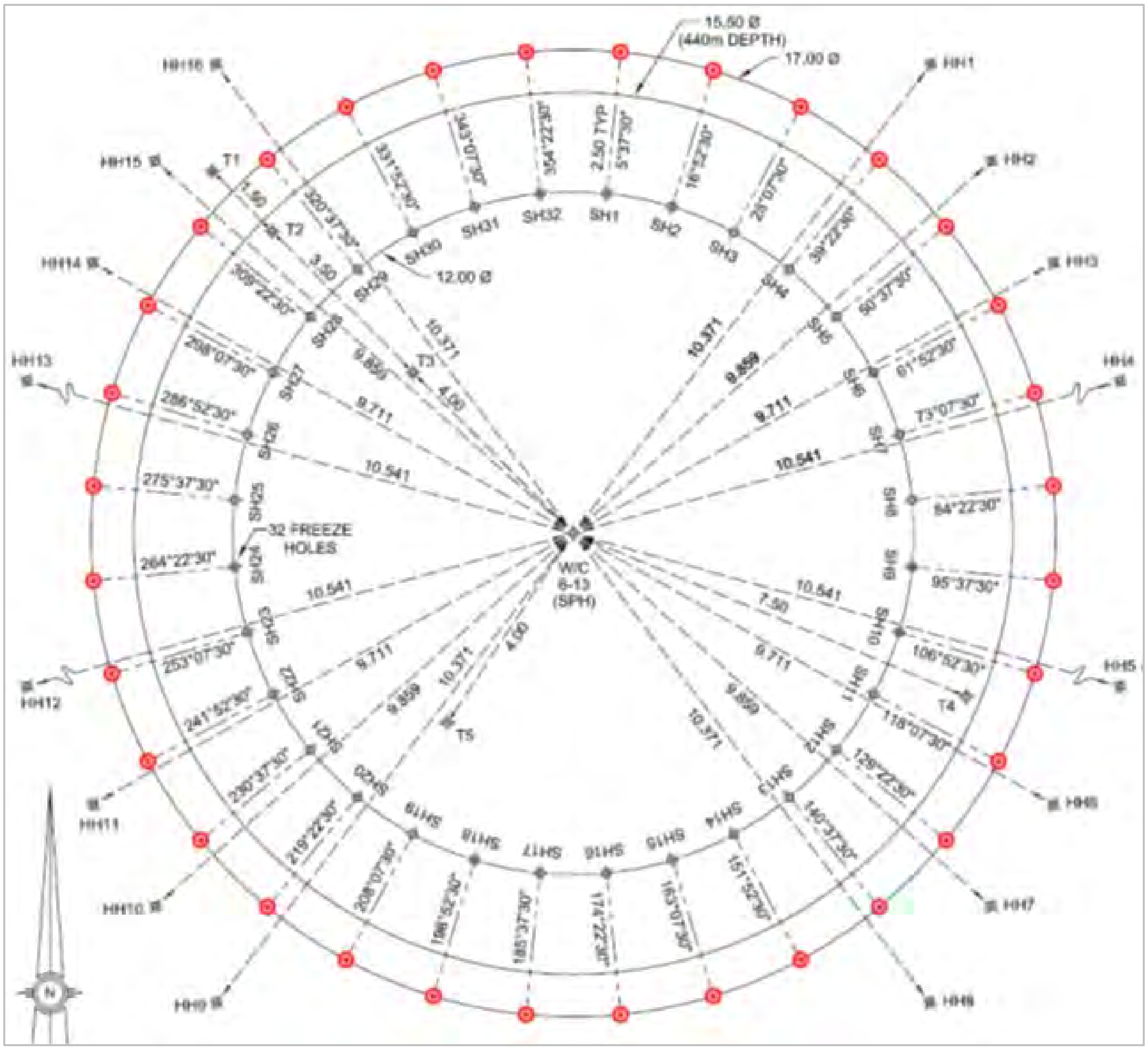
The plant itself will consist of compressor houses and cooling coil sets in gangs according to the final required capacity. A typical arrangement is shown in Figure 18-12.



Source: Nordmin, 2019

Figure 18-12: Typical Freeze Plant Configuration
(with gangs of compressors and cooling coils in series to make up the total capacity of the plant)

The boreholes around each shaft are arranged radially around the planned perimeter of the excavation, as shown in Figure 18-13. The actual working diameter of the freeze hole perimeter and the number of holes is determined by geotechnical design.



Source: Nordmin, 2019

Figure 18-13: Typical Layout of a Freezewall Borehole System

The red perimeter holes are used for freezing the shaft envelope, which is shown as the inner ring.

The stabilization of the shaft envelopes down to the carbonatite is critical to the progress of the project. To this end, the freeze will start three to six months prior to commencement of shaft sinking and will be left in place until one month after the shaft liner is socketed and sealed into the carbonatite.

19. MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Market studies for niobium, titanium dioxide and scandium trioxide are an important part of the proposed Elk Creek Mine. These products, especially niobium and scandium trioxide (scandium), are thinly traded without an established publicly available price discovery mechanism.

19.1.1 Niobium Market Overview

Niobium is a versatile element that adds value to a range of applications. Niobium improves material properties, which often leads to increased efficiency, safety, performance and transform's the properties of advanced steels, cast aluminum, glass, batteries and electronics. Ferroniobium in steelmaking consumes approximately 90% of the available world supply of niobium. The remainder goes into a wide range of smaller volume but higher value applications, such as high-performance alloys (which includes superalloys), carbides, superconductors, electronic components and functional ceramics.

Commercial trade of niobium occurs in several forms, the most common of which is ferroniobium. Ferroniobium is sold most commonly as steel grade (65% Nb content) as well as a higher purity technical grade.

19.1.1.1 Niobium Supply

The niobium market is generally described as an oligopoly with three major producers dominating supply. These three producers are Companhia Brasileira de Metalurgia e Mineração (CBMM), Magris Resources and China Molybdenum Co. Ltd (CMOC). However, in practical terms, the market operates as a monopoly with a single company (CBMM) setting the price and the other operations acting as price takers. In addition, CBMM performs its own research and development activities to evaluate additional/increasing usage of niobium, which provides a significant benefit for other market participants. Over many decades, CBMM has become a very reliable producer and has significantly reduced supply disruptions and in return has increased supply to accommodate overall demand growth. In terms of ferroniobium production, Table 19-1 provides the reported annual production capacity from the three largest mine operations along with the project's estimates.

Table 19-1: Comparison of Project Versus Selected Niobium Producers

Mine/Project	Owner	Country	Reserves (Est.)	Annual Ferroniobium Production (Est.)
Araxa (OP)	CBMM	Brazil	829 Mt @ 2.5% Nb_2O_5 (weathered) 936 Mt @ 1.57% Nb_2O_5 (fresh)[1]	110 kt/y[2]
Niobec (UG)	Magris Resources	Canada	Proven 19.9 Mt @ 0.51% Nb_2O_5 Probable 54.5 Mt @ 0.51% Nb_2O_5[3]	9.2 kt/y[3]

Elk Creek (UG)	NioCorp	USA	Probable 36 Mt @ 0.81% Nb_2O_5	7.2 kt/y
Catalao (OP)	CMOC	Brazil	Area I 37.4 Mt at 0.97% Nb_2O_5 Area II 217.7 Mt at 0.34% Nb_2O_5[4]	13.8 kt/y[5]

Source: NioCorp, 2019

[1]CBMM, 2017, Sustainability Report. CBMM does not report reserves, only resources

[2]Roskill, 2018

[3]Roskill, 2017

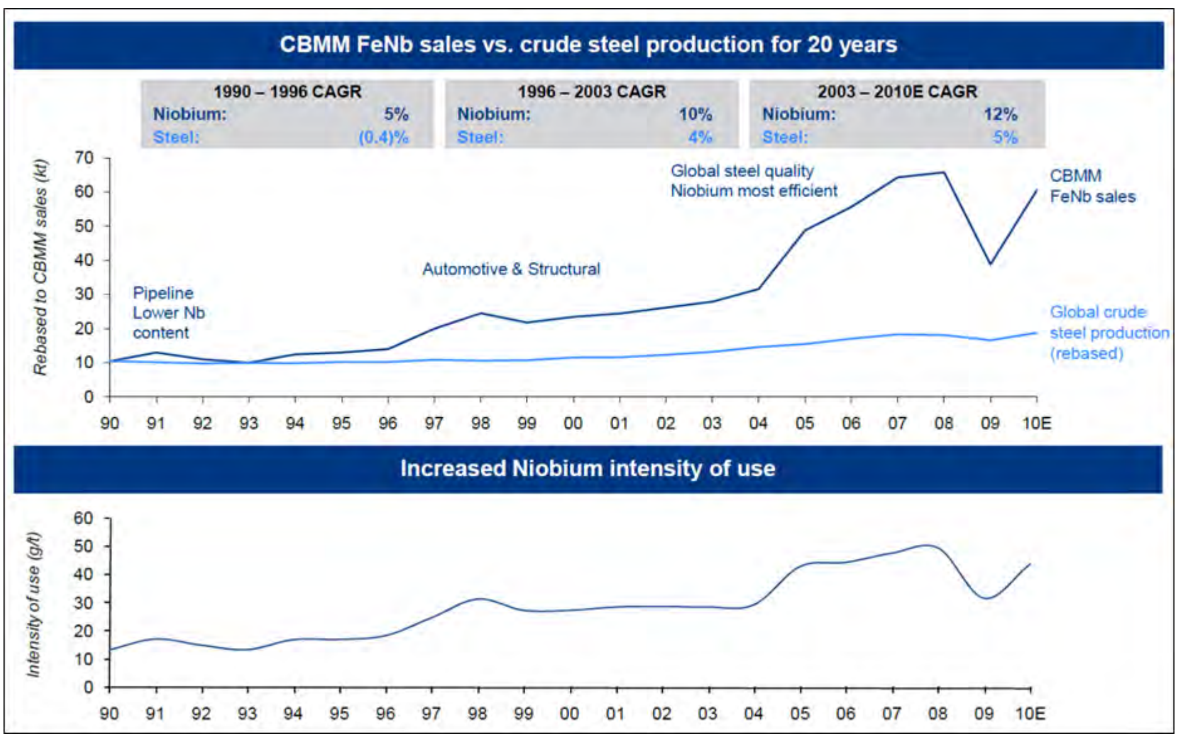
[4]CMOC Annual Report, 2017

[5]Roskill, 2017

Niobium is not traded in public markets. Transactions generally occur directly between mine operators and downstream consumers. Trading firms also play a smaller role in the market as intermediaries. There are several quoted prices for various ferroniobium and niobium oxide products that are established based on these transactions with traders.

19.1.1.2 Niobium Demand

Based on market information provided by CBMM, Niobium demand showed an active profile of growth over almost 20 years from the early 1990s to late 2000s, greatly exceeding growth rates in steel demand (see Figure 19-1).

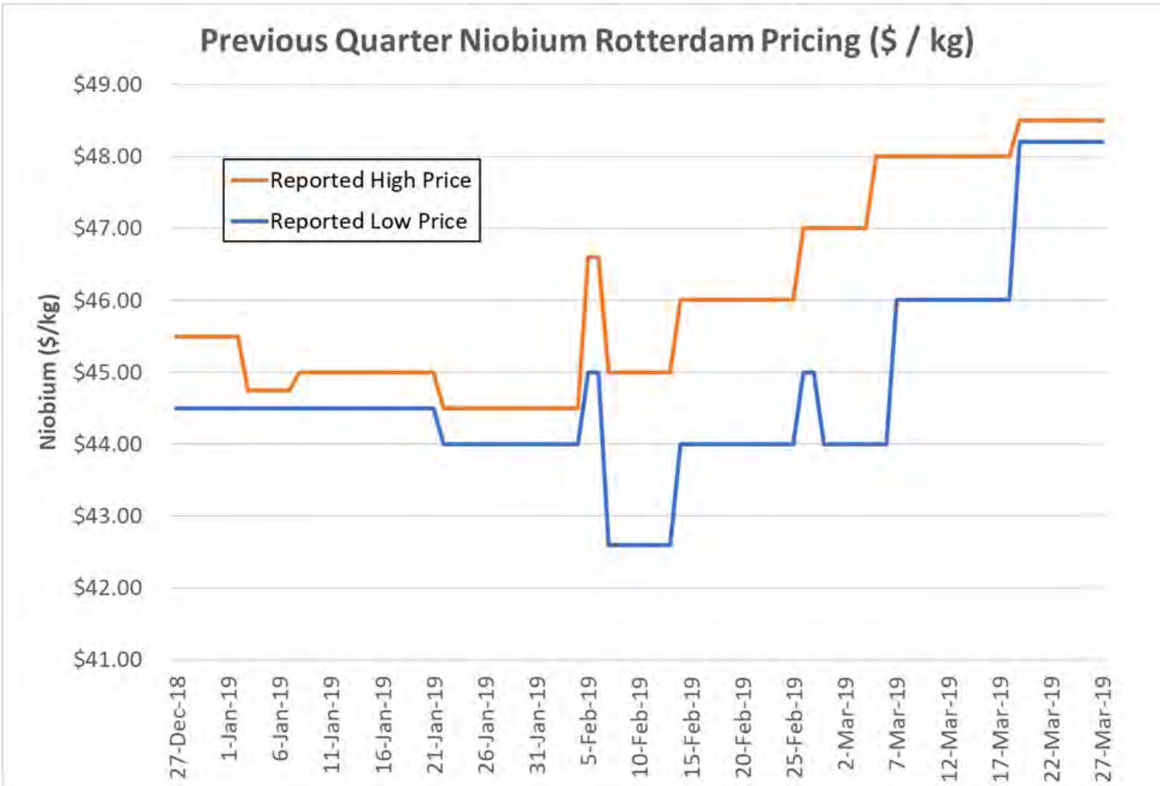


Source: CBMM, 2013

Figure 19-1: CMBB Niobium Sales Versus Steel Demand and Niobium Intensity of Use

Pricing

Figure 19-2 demonstrates recent price trends for 65% ferroniobium (pricing basis anticipated for NioCorp).



Source: Metal Pages, 2019

Figure 19-2: Ferroniobium (65% - EU) Price Trends Previous Quarter

Future prices of niobium are highly dependent upon the intention of CBMM. CBMM could flood the market with low-cost production, dropping the price and driving out its competitors, however to date, CBMM has shown a tolerance for other producers and has dropped its production levels to maintain a stable price.

Roskill's Global Industry, Markets and Outlook 2018 (Roskill, 2018) has indicated that Niobium prices are historically very stable. They moved little in the period up to about 2006, when a producer-driven doubling in the pricing began and have remained stable at the higher benchmark. Ferroniobium prices, in particular, are an inelastic demand, with the 2009 slump in demand from the global steel industry having only minimal impact on pricing. The outlook for prices is one of a gentle but steady increase; spikes are unlikely. The economic analysis in this report used the recommended real 2019 US dollar base price of US$ 47/kg Nb as the forward-looking price for steel grade (65%) ferroniobium.

19.1.2 Titanium Dioxide Market Overview

In 2017, the global titanium dioxide market size was US$ 15.03 billion and was expected to grow steadily, owing to its growing demand in end-use industries such as plastics, coatings, paper, cosmetics and others. Furthermore, technological innovations in manufacturing processes, which have resulted in higher and good quality yield, positively impact the overall titanium market (see Figure 19-3, Adroit Market Research, March 2019).

Source: Adroit Market Research, 2019

Figure 19-3: Global Titanium Dioxide Market Value and Volume 2014-2025

TiO_2 is used extensively as paint pigment with some minor, though increasing, demand from the aerospace industry as an alloy in next-generation aircraft. Average domestic US consumption in 2018 was 920,000 t of which 270,000 t was imported. With the Project producing approximately 12,000 t TiO_2 per year during LOM, it is assumed that this annual production volume can easily be absorbed into the domestic market.

19.1.2.1 Titanium Dioxide Demand

The competitive landscape of global titanium dioxide market is highly fragmented with a large number of global and regional players including Henan Billions Chemicals Co., The Chemours Company, Huntsman International LLC, NL Industries, Inc., Tronox Limited and others. These prominent players have always looked forward to implementing essential strategies through partnerships, agreements, collaborations and business expansions.

Nordmin completed the high-level titanium dioxide (TiO_2) market research. Formal market studies were not completed at this time as TiO_2 represents only 2% of the overall revenue in the economic analysis. All market information for titanium and titanium dioxide is derived from USGS Commodity Market Summaries (Bedinger, 2019) and Adroit Market Research (Johnson, 2019).

The economic analysis assumes a constant long-term price of US$ 0.99/kg based on rutile concentrate FOB Australia benchmark with no discounts (see Table 19-2). Pricing has shown a significant rebound in the recent period. Although the market is well-established and mature, the key risk to maintaining this price is the domestic US and global economic growth.

Table 19-2: Titanium Mineral Concentrates Pricing History (Rutile Concentrate FOB Australia)

	2014	2015	2016	2017	2018	2019
95% TiO_2 Price (US$ /kg)	0.95	0.84	0.74	0.74	0.99	0.99

Source: USGS MCS 2019

19.1.3 Scandium Trioxide Market Overview

Scandium, as the lightest rare earth element in the periodic table, has critical utilization in areas such as the aerospace industry, solid fuel cells, electronics industry and is also used in metallurgical applications (Altinsel et al., 2018). Scandium is the 50th most abundant element with a crustal abundance of 20 - 30 ppm. However, scandium does not have any identified single deposit type due to its natural occurrence being as a dispersed state. Due to the scarcity of high grade scandium deposits, and high processing costs, scandium production rate is relatively limited. Scandium is generally produced as a co-product of primary metal processes, wastes and reprocessed tailings (Altinsel et al., 2018).

NioCorp engaged OnG Commodities LLC (OnG) to produce a preliminary market assessment for scandium, published in July 2015 (OnG, 2015). This study formed the basis for the economic analysis undertaken in the 2015 PEA. For this report, the market assessment was updated by OnG in March 2019 (OnG, 2019) to reflect the current and future market for Scandium.

The studies examine current scandium production trends (~20 t/y) from existing and emerging producers plus an outlook for supply to 2030. The outlook then reviews the current and emerging applications for scandium including fuel cells, aerospace, industrial and other uses plus and an outlook for demand to 2030.

This study outlines that even though the current global market for scandium is approximately 15 t/y in the form of Sc_2O_3, this relatively small amount of production is due to the market demand being relatively muted given the diminutive size of the global market along with a lack of stable supply. The scandium supply is highly reliant on China as a co-product or by-product of rare earth mining along with increasing supply from the Russian Federations. Accordingly, the distribution of supply, as much as the amount of scandium available, should be seen as an impediment to scandium demand growth. Consequently, this lack of supply has been an inhibitor of demand growth and the lack of demand has depressed supply growth.

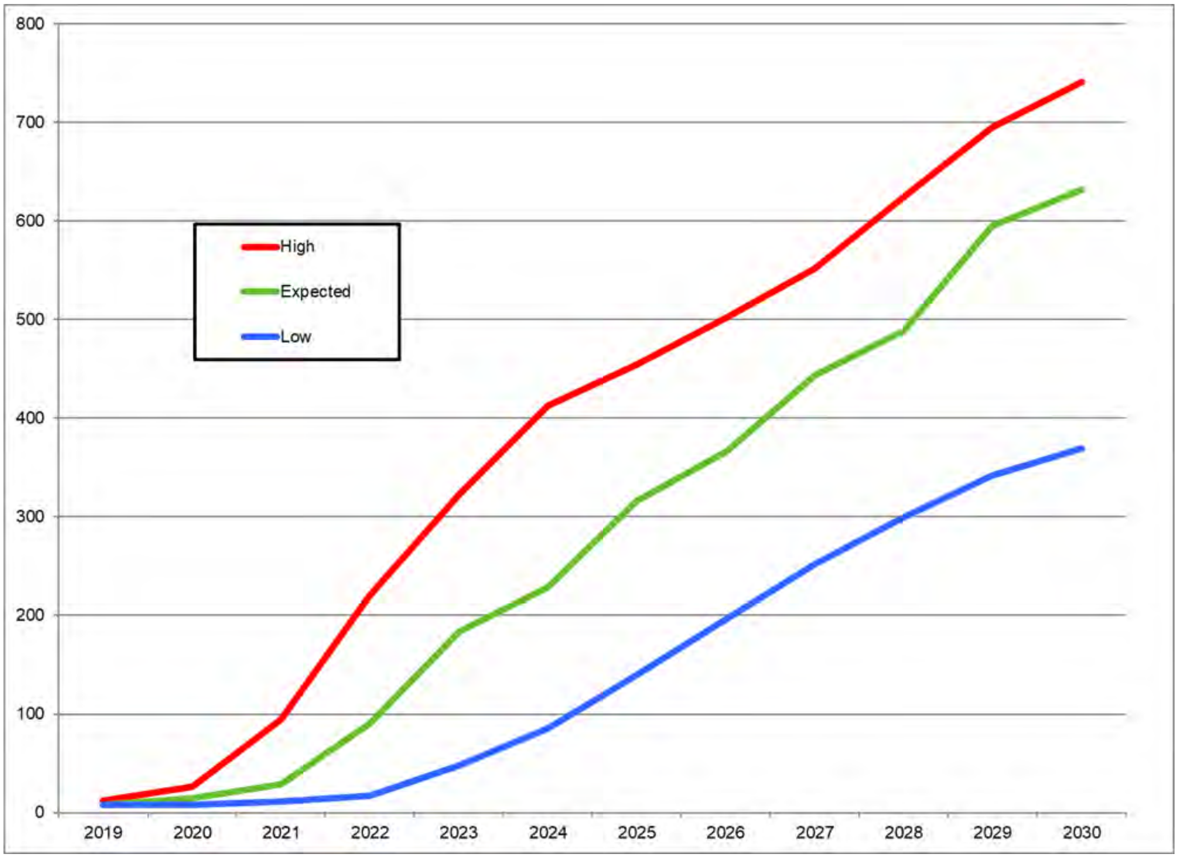
It is reasonable, to take the view that until 2010, scandium while promising in principle, was little more than an academic curiosity; due to the unwillingness of any large potential user to commit to developing supply. The situation changed exclusively by the actions of Bloom Energy. Bloom has contracts with numerous existing and emerging scandium suppliers and is constrained first by the availability of scandium and only second by the price of scandium.

19.1.4 Key Aspects of OnG Commodities Report

<u>Scandium Trioxide Market Supply</u>

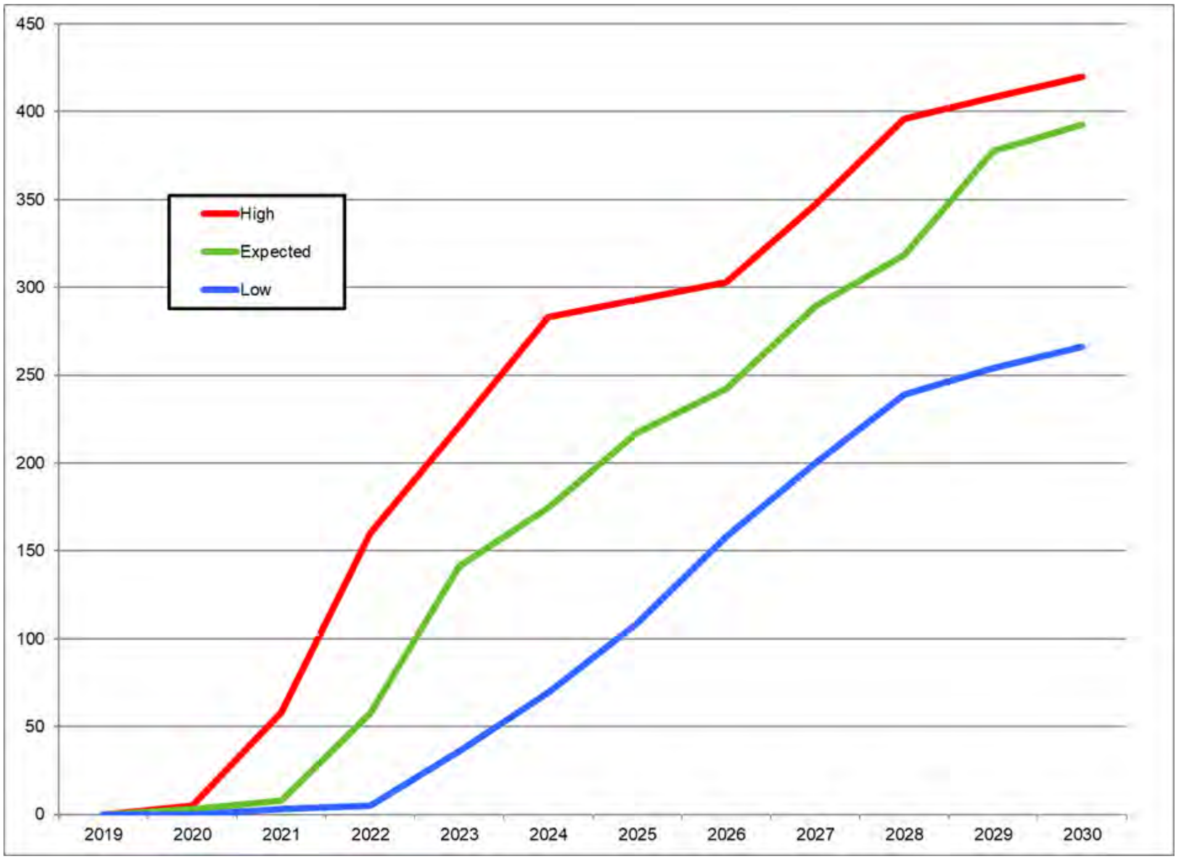
Currently, approximately 75% to 80% of all scandium production originates in China, as a by-product or co-product of rare-earth production. Drawdowns of former USSR stockpiles and by-product recovery from uranium in situ leaching operations represent the balance of the current world supply. The Sumitomo Taganito scandium plant in the Philippines was expected to begin operations in 2018, producing 7.5 t/y of scandium oxide according to the Sumitomo Metal Mining 2018 3-year business plan, however, no announcement has been delivered, and the plant may still be in commissioning and qualification.

OnG develops a detailed analysis of various production sources expected to come online in the next few years. These include resources in Australia, the USA, Turkey, Canada, and India, in addition to the expansion of existing resources within China and Russia. Thorough analysis details the relative challenges entrants may face monetizing these resources given the new technologies being developed. OnG develops two primary forecast ranges differentiated by the inclusion of Russia and China (see Figure 19-4 and Figure 19-5).



Source: OnG, 2019

Figure 19-4: High, Expected, and Low Case Forecasts for Scandium Oxide Potential Supply 2019 – 2030, Tonnes per Year

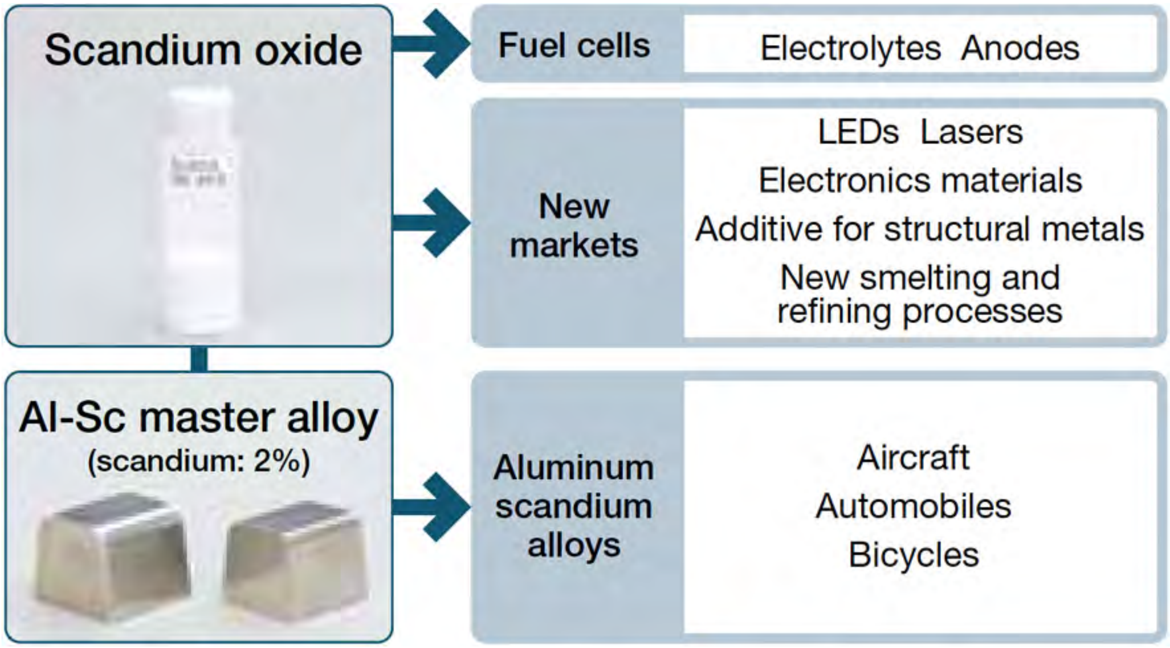


Source: OnG 2019

Figure 19-5: High, Expected, and Low Case Forecasts for Scandium Oxide Potential Supply 2019 – 2030, Tonnes per Year, Excluding Russia and China

Scandium Trioxide Market Demand

OnG speculates that scandium has two primary main applications today (1) As an alloying agent in aluminum alloys (with aerospace the largest candidate market) and (2) in solid oxide fuel cells (SOFC). The SOFC market is the largest single consumer of scandium and is almost entirely constituted by Bloom Energy of the US. Also, fuel efficiency standards driven by an increasing focus on carbon emissions in the EU is anticipated to lead to a dramatic increase in scandium usage within the transportation sector (see Figure 19-6).



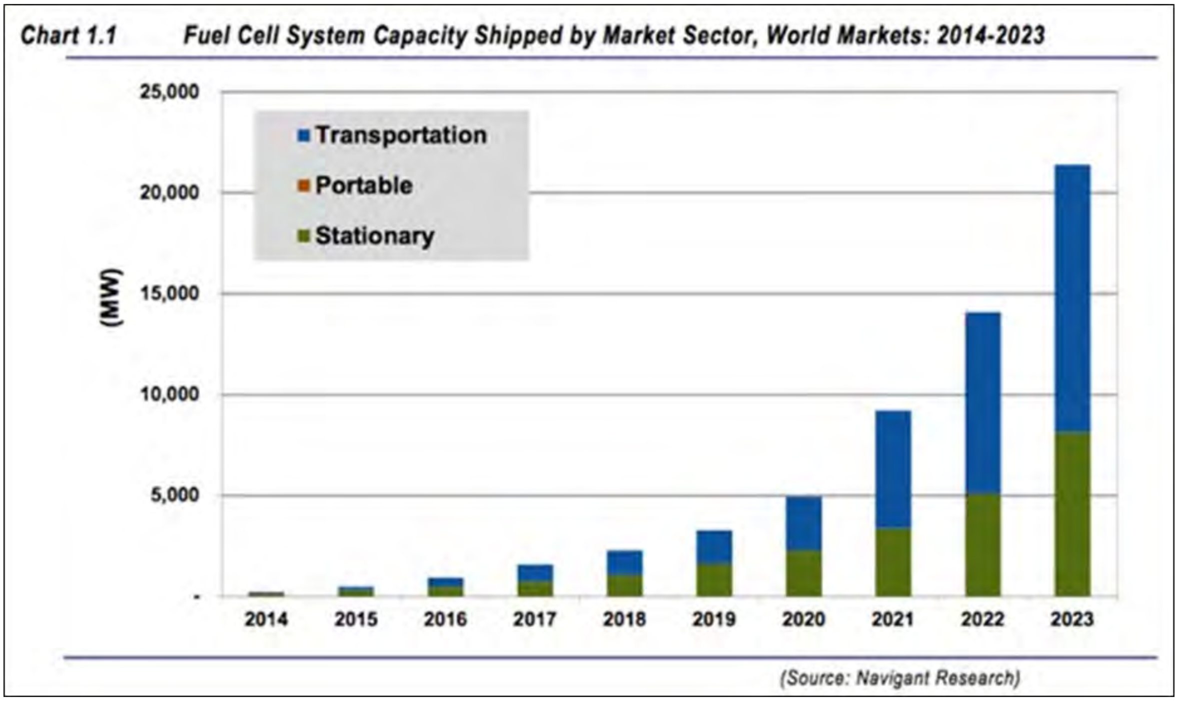
Source: Sumitomo Metal Mining Co., 2019

Figure 19-6: Current/Potential Scandium Market

Solid Oxide Fuel Cells

OnG provides an overview of scandium use in the SOFC market and the technology that necessitates its use. Scandium is an essential component of Bloom Energy's SOFCs and delivers high reliability and the ability to operate the fuel cell at a much lower temperature than competing SOFC technologies reliant on yttrium doped zirconia. The lower operating temperature simplifies construction and allows for less costly materials of construction. Using published data, SOFC manufacturing requires an estimated 150 kg of scandium oxide per MW of power. There are no true substitutes that deliver an equivalent level of performance.

Navigant Research has forecast sales of fuel cells to 2023. They project annual shipments exceeding 20,000 MW by 2023, of which about 8,000 MW is expected to be stationary power (the market segment where SOFCs are used). Growth rates exceed 40% per year across the period. Navigant's forecast shown in Figure 19-7.



Source: Navigant Research, 2016

Figure 19-7: Fuel System Capacity Shipped by Market Sector: Navigant Research

Assuming that SOFC use sustains growth at this rate, and at the same time the intensity of use of scandium oxide in SOFCs is reduced at a rate equal to any price reduction for the fuel cells SOFC demand for scandium oxide will grow at 40% per year and will increase to around 180 t/y in 2025. Note that Bloom Energy completed an IPO in 2018 and its growth rate and forecast orders are consistent with the outlook above.

Aircraft Aluminum Alloys

OnG provides significant background on the Aluminum Scandium (AlSc) market opportunity. The aerospace industry was an early adopter of the alloy and has accumulated years of experience with the materials. The lack of a reliable supply has been the primary barrier to broad commercial market adoption.

Very little scandium is necessary for AlSc alloys, and less than 0.5% scandium is sufficient and loading as low as 0.1% can be adequate (although Airbus' patented alloys can contain up to 1.3% scandium). A typical single-aisle jetliner, such as an Airbus A320 or a Boeing 737, has a dry weight of 45 t to 50 t, which is mostly (80% by weight) aircraft aluminum. According to Airbus, scandium alloys can reduce this weight by an estimated 15% to 20%, or by 6 t to 10 t.

Assuming the AlSc alloy is 1% scandium, each aircraft would require approximately 600 kg of scandium oxide, approximately US$ 2.1 million at current market prices, while the lifetime value of fuel savings would total US$ 20 to 30 million.

The weight reductions come to some degree from the ability to use less aluminum alloy when it is alloyed with scandium. The majority of weight savings accrue from the ability to weld the airframe. Welding eliminates thousands of rivets currently needed to fasten an aluminum aircraft together.

Welding also has the potential to save time and cost in aircraft assembly, offering further benefits to a switch to AlSc alloys.

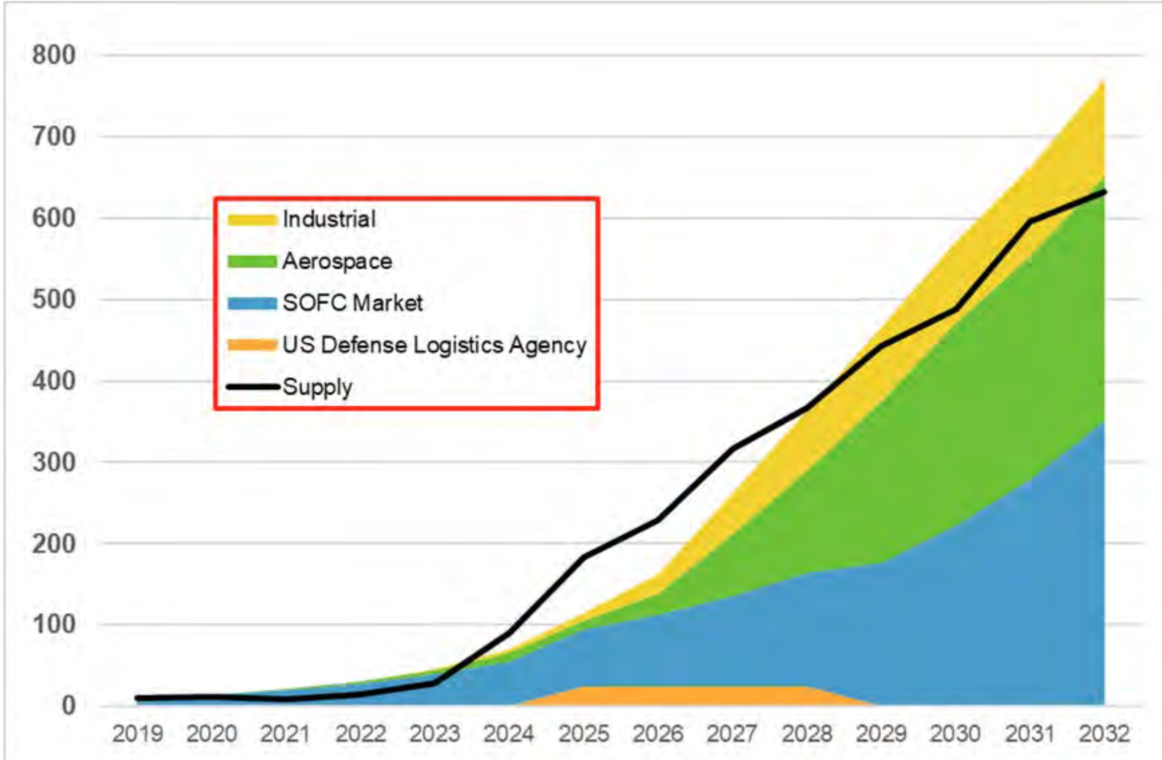
OnG notes a key question for scandium demand growth will be driven by the pace of adoption by aerospace firms. Specifically referencing the A320, current production rates would necessitate at least 100 t/y of scandium oxide if only key components transitioned to AlSc alloys with 1% Sc content. That grows to nearly 250 t/y if all aluminum components were transitioned, and the buy-to-fly impact on required input scandium would increase both these quantities substantially (aluminum buy-to-fly ratios in civilian aerospace vary by component but can commonly reach 5:1 - as reported for Constellium's Airware alloys (deployed in the Airbus A350) for example).

Widespread adoption will, therefore, take time. Under realistic supply-side scenarios, the early to mid-2020s is the earliest period when large-scale deployments of AlSc could be expected in passenger aircraft. This will be because of supply chain issues primarily because AlSc alloys are well characterized and understood for aerospace applications. OnG draws a link to the development of the Airbus A380 which required new LiAl alloys for wing main spars, new ingot casting techniques, and new manufacturing and assembly. The entire process from inception to launch required seven years. Scandium could potentially be adopted faster if the supply side is well established because the foundational alloy development and understanding has already been completed. Further, global capacity for lithium aluminum alloys is approaching 50,000 t/y, which if replicated for scandium would represent 250 to 500 t/y of scandium usage depending on the level of scandium doping.

<u>Other Markets</u>

OnG provides additional context on the broader adoption of AlSc alloys in the transportation sector, defence sector, and as a replacement for titanium. The potential also exists for growth in smaller existing markets, such as sporting equipment, stadium lighting, handguns, specialty alloys and lasers. These potentials are disregarded for price forecasting in the OnG analysis.

Figure 19-8 provides a summary chart of the aggregate demand by the differing sectors.



Source: OnG, 2019

Figure 19-8: Supply-Demand Forecast for Scandium Oxide to 2028, Tonnes, Base Case

Each of the independent market segments is expected to drive significant demand growth over the next ten years. As supply begins to align with demand in 2025, a deficit again appears in 2028.

Market Pricing

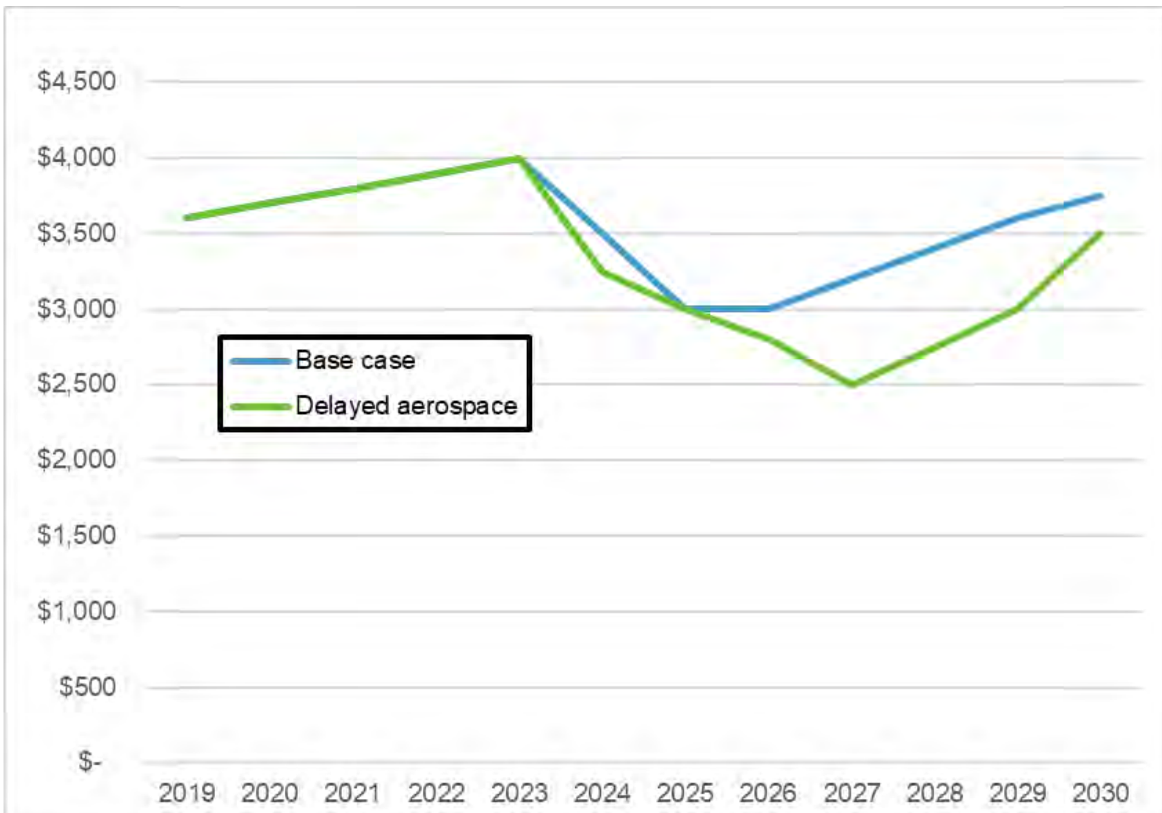
OnG provides a forecast of market pricing and the context of current scandium pricing with the following statements:

- Price trends are more reliable than the actual quoted numbers. The general increase in scandium oxide pricing reported by the USGS since 2010, and the narrowing of the spread between low purity and high purity scandium oxide pricing, does reflect an increase in consumption (by Bloom Energy of California) as well as a willingness to purify lower grade scandium oxide through secondary reprocessing. OnG goes on to note that Bloom has, to date, been willing to purchase all the scandium available to it, has entertained long term supply agreements with many (if not all) of the existing and emerging scandium suppliers, and has managed to grow at rates exceeding 40% per year despite scandium oxide prices in the range of US$ 3,500 to 4,000/kg.
- Too much supply would inevitably depress prices in the long run. A substantial increase in supply, from a more diverse set of countries and, is underwritten by well-capitalized mining operations could increase the size of the scandium market and support prices at today's levels.

OnG presents two scenarios for market pricing driven by aggregate aerospace adoption of the increased supply of scandium. Under the "base case" assumptions, scandium oxide prices will likely rise slowly from their current level of US$ 3,500/kg to US$ 4,000/kg by 2022 as demand begins to outstrip supply. With new Western operations beginning from 2023 – 2027, there is likely to be a period of moderate oversupply, causing a softening of prices to US$ 3,000/kg. This oversupply period is expected to support substantial growth in aerospace demand. By 2027, as demand begins to outstrip supply prices will likely rise.

This scenario is considered probable even if the market returns to undersupply in 2027 since suppliers will have entered into contracts as they commission plants and because market tightness will take time to manifest. From 2028 the market should recover strongly to a level of US$ 3,750/kg.

However, if aerospace demand is slow to materialize, prices may fall through 2027 to a level of around US$ 2,500/kg, before turning around in 2028 as delayed aerospace growth begins to tighten the supply of scandium oxide. Prices are unlikely to fall below this due to the relatively short periods of supply excess. Further, Bloom and industrial users are likely to make efforts to accelerate growth (see Figure 19-9).



Source: OnG, 2019

Figure 19-9: Scandium Oxide Pricing Outlook, US$/kg, 2019 – 2030

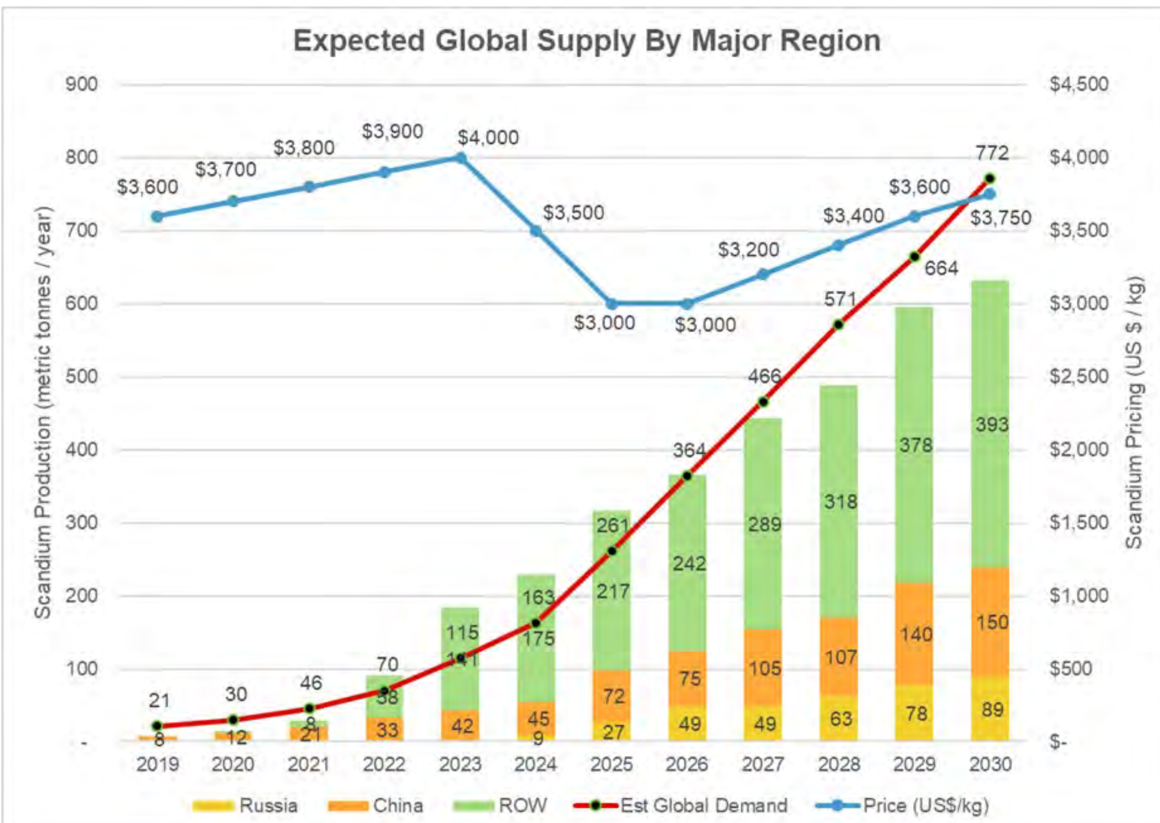
<u>Summary</u>

Based on these inputs, the following summaries of OnG pricing forecasts and global demand volumes by year to 2030 based on estimated production costs and supply-demand balances. These forecasts, plus the Project's estimated annual scandium production volumes, are shown in Table 19-3 and Figure 19-10.

Table 19-3: Scandium Supply, Demand and Price Forecast Summary

Description	Price (US$/kg)	Est. Global Supply (kg)	Est. Global Demand (kg)	Project Annual Production (kg)	% of Est. Global Supply	% of Est. Global Demand
2019	3,600	8,000	21,168	-	-	-
2020	3,700	15,000	30,236	-	-	-
2021	3,800	29,000	45,530	-	-	-
2022	3,900	90,750	70,342	47,750	53%	68%
2023	4,000	183,810	114,731	112,110	61%	98%
2024	3,500	228,950	162,861	108,500	47%	67%
2025	3,000	316,000	260,705	103,400	33%	40%
2026	3,000	366,330	364,488	95,330	26%	26%
2027	3,200	443,120	465,755	96,120	22%	21%
2028	3,400	488,380	571,452	91,380	19%	16%
2029	3,600	595,430	664,029	96,730	16%	15%
2030+	3,750	631,670	771,577	99,770	16%	13%

Source: OnG, 2019



Source: OnG, 2019

Figure 19-10: Global Scandium Supply/Demand and Price Projections Summary

From an overall market standpoint, demand for scandium oxide is straining supply, and there are few other truly near-term opportunities to increase supply. So, for emerging larger scale producers such as NioCorp, a few extra tonnes of supply out of Russia will create the potential for further market growth as they develop their supply, just as the Sumitomo project will be beneficial for all.

Currently, most of the scandium production is from China, which does not have transparency in reserves and cost reporting. If production from other parts of the world and Russia begins to take off as projected, it is not clear whether Chinese or even Russian production will increase and keep new entrants from entering the market. Conversely, if the entrance of a few new producers to the market stimulates demand, and the new entrants and existing producers cannot meet that demand, pricing could conceivably increase over the estimates provided in the OnG market report.

19.2 Contracts and Status

At the time of this report, NioCorp had entered into three offtake agreements covering ferroniobium and scandium trioxide production from the Project.

Each ferroniobium agreement has a ten-year term which, when combined, means 75% of the projected production is contracted at a 3.75% discount to the quoted Metal Pages [1] price (unless a premium can be achieved by the offtake customers, which is uncertain).

The scandium trioxide offtake agreement is structured similarly. The agreement has a ten-year term and a minimum of 12 t/y. At that rate, approximately 10 - 15% of the projected annual production is contracted. Further, the customer may elect to take more material in any given year above the prescribed minimum quantity.

No offtake agreements have been executed at the time of the report for the titanium dioxide product from the Project. It is assumed this product and all other material not covered by an offtake agreement will be sold on a spot price, ex-mine gate basis.

Supply contracts for electric power with the Omaha Public Power District (OPPD), natural gas transportation from Tallgrass Energy and natural gas supply from Tenaska have been executed at the time of this report.

[1] As of May 6, 2014, Metal-Pages Ltd. operates as a subsidiary of Argus Media Limited. https://www.argusmedia.com/metals/argus-metal-prices

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

20.1.1 Soils

According to the Natural Resources Conservation Service (NRCS), soils in the vicinity of the Project are primarily comprised of clay, silty clay, silt loam, and clay loam within an ecological site that is typified as “Rangeland.” For all soil types, the depth to any soil restrictive layer is more than 200 cm below ground surface (bgs), and the infiltration is generally “slow” to ”very slow.” Soils in the area are generally eroded and range in slopes from 2% to 30%, with the majority of the area having slopes of between 6% and 11%. (NRCS, 2015)

20.1.2 Climate/Meteorology/Air Quality

A dedicated meteorological station was installed at Elk Creek in July 2014. Parameter measurements included in the overall instrument package include:

- Wind Speed
- Wind Direction
- Temperature
- Temperature Difference
- Dew Point Temperature
- Precipitation
- Pressure
- Solar Radiation

The meteorological data thus far collected includes continuous monitoring that has been audited by a third party and can subsequently be used in air quality modelling and permitting. A continuation of data review will be conducted on a monthly basis, as specified in the Meteorological Monitoring Plan, and a final review will be necessary prior to submittal for inclusion in dispersion modelling.

In September 2016, NioCorp met with the Nebraska Department of Environmental Quality (NDEQ) regarding the on-site air monitoring program and the air quality permit application process. It was decided that the ambient monitoring program needed to include $PM_{2.5}$ data collection, in light of the attention that this parameter has been given recently by the U.S. EPA. Air quality monitoring was conducted from March 6 to August 20, 2017: the $PM_{2.5}$ monitoring was initiated at the Elk Creek site in February 2017; a PM_{10}, monitor was added in March 2017, along with co-located $PM_{2.5}$, and monitoring for four gasses including CO, NO_x, SO_x and ozone (O_3).

20.1.3 Cultural and Archeological Resources

There were at least 15 Native American tribes that have inhabited the Great Plains region now incorporated in the State of Nebraska, including the Kansa and Otoe tribes of southeastern Nebraska. Of these original inhabitants, there are four federally recognized Indian tribes that remain in Nebraska today, including:

- Omaha Tribe of Nebraska;
- Winnebago Tribe of Nebraska;
- Iowa Tribe of Kansas and Nebraska;
- Ponca Tribe of Nebraska; and
- Santee Sioux Tribe of Nebraska.

Reservations associated with these tribes are located in the northeastern part of the state, over 200 km to the north of Elk Creek.

The Otoe Tribe once lived south of the Platte River in the region of the proposed mine, but in 1881, sold all of their lands in Nebraska to the federal government and moved to Indian Territory (now Oklahoma). No direct tribal consultation appears to be necessary at this time.

In January 2017, Cultural Resources Consulting of Hickman, Nebraska (CRC) conducted archeological resources investigations within the proposed area of potential effect, including the proposed mine Project area and the waterline corridor to the Missouri River. The investigation was intended to determine if there are known archeological sites recorded, or currently unknown, but potentially significant cultural resources that may be impacted within the defined area of potential effect. As currently designed, no significant archeological resources will be impacted by construction of the Elk Creek Mine and processing area, the evaporation pond and tailings impoundment area, or installation of the waterline. It is recommended that no further Historic Preservation compliance actions are warranted, and the Project be allowed to proceed as currently planned. (CRC, 2017) During construction, the Project will still be subject to the provisions of the *Nebraska Unmarked Human Skeletal Remains and Burial Goods Protection Act*.

The waterline to the Missouri was eliminated from the scope of the project after the archeological resource's investigations were completed.

20.1.4 Vegetation

Cultivated cropland (principally corn, soy, and alfalfa) makes up the majority of the surface area within the Project boundary. Native and non-agricultural vegetation exist primarily in the form of hedgerows and windbreaks along field margins, and in riparian areas along surface water drainages. According to ecosite descriptions from the NRCS (2015), plant communities within the vicinity of Project consist of annual and perennial weedy forbs and less desirable grasses from abandoned farmland, as well as big bluestem (*Andropogon gerardii*), smooth brome (*Bromus inermis*), tall fescue (*Schedonorus arundinaceus*), switchgrass (*Panicum virgatum*), Indiangrass (*Sorghastrum nutans*), sideoats grama (*Bouteloua curtipendula*), little bluestem (*Schizachyrium scoparium*), Scribner's rosette grass (*Dichanthelium oligosanthes var. scribnerianum*), porcupinegrass (*Hesperostipa spartea*), sedge (*Carex*), leadplant (*Amorpha canescens*), eastern redcedar (*Quercus macrocarpa*), honey locust (*Gleditsia triacanthos*), and smooth sumac (*Rhus glabra*).

20.1.5 Wildlife

According to Schneider et al. (2011), the Project is located in Nebraska's Tallgrass Prairie Ecoregion which is home to more than 300 species of resident and migratory birds and 55 mammal species, most of which can also be found in central and western Nebraska. The small mammal fauna of the Tallgrass Prairie Ecoregion consists of species such as the plains pocket gopher (*Geomys bursarius*), prairie vole (*Microtus ochrogaster*), thirteen-lined ground squirrel (*Spermophilus tridecemlineatus*), and Franklin's ground squirrel (*Spermophilus franklinii*). White-tailed deer (*Odocoileus virginianus*) are the common big game species in the region. The most abundant large predator of the region is the coyote (*Canis latrans*), but other predators such as the red fox (*Vulpes vulpes*) and American badger (*Taxidea taxus*) can be found in the Tallgrass Prairie Ecoregion as well. The bobcat (*Lynx rufus*), least weasel (*Mustela nivalis*), and American mink (*Neovison vison*) can be found in wooded areas, wetlands and along river valleys (Schneider et al. 2011).

20.1.6 Threatened, Endangered, and Special Status Species

The Project and surrounding areas lie in the Southeast Prairies Biologically Unique Landscape within the Tallgrass Prairie Ecoregion of Nebraska (Schneider et al., 2011). No species that are listed as Threatened or Endangered under the federal Endangered Species Act, or the Nebraska Non-game and Endangered Species Conservation Act, have been identified as inhabitants of the Southeast Prairies Biologically Unique Landscape. According to Schneider et al. (2011) special status species which have been identified as "Tier I at-risk species" by the state of Nebraska, as well as those species that may be headed for state or federal listing, that may occur in the vicinity of the Project include the following:

- Birds:
  - Greater Prairie-Chicken (Tympanuchus cupido);
  - Henslow's Sparrow (Ammodramus henslowii);
  - Loggerhead Shrike (Lanius ludovicianus); and
  - Wood Thrush (Hylocichla mustelina).
- Reptiles:
  - Massasauga (Sistrurus catenatus); and
  - Timber Rattlesnake (Crotalus horridus).
- Insects:
  - Iowa Skipper (Atrytone arogos iowa);
  - Regal Fritillary (Speyeria idaliaI);
  - Married Underwing (Catocala nuptialis); and
  - Whitney Underwing (Catocala whitneyi).
- Mollusks:
  - Pimpleback (Quadrula pustulosa);
  - Pistolgrip (Tritogonia verrucosa); and
  - Plain Pocketbook (Lampsilis cardium).
- Mammals:
  - Plains Harvest Mouse (*Reithrodontomys montanus griseus*).

The Pistolgrip is known to only occur in one other biologically unique landscape in Nebraska, while the Massasauga and Plain Pocketbook are known to occur in only two other biologically unique landscapes in Nebraska. No nesting Piping Plovers (*Charadrius melodus*), Interior Least Terns (*Sternula antillarum athalassos*), migrant Whooping Cranes (*Grus Americana*), or nesting Bald

Eagles (*Haliaeetus leucocephalus*) are known to occur in the vicinity of the Project area (Brown, 2014). The Massasauga's primary habitat is wet meadows while the Timber Rattlesnake generally inhabits rocky outcropping and adjacent habitats. If any construction is to be conducted in the range of either of the snake species or any Tier I species, a proper impact analysis is required to be executed. This will be accomplished during the federal review process, if necessary.

The development of the mine may also need to consider potential impacts to several sensitive species, including bats that might be affected by ground-clearing activities during construction at the project site. These risks are considered negligible.

20.1.7 Land Use

Since the settlement of Johnson County, farming for livestock, crops, and pasture has been the most important land use enterprise. Over the years, crop production has shifted from orchards, oats, barley, and rye to corn, soy, wheat, alfalfa, and grain sorghum. Livestock in the county generally consists of hogs, cattle, and milk cows (USDA SCS, 1984). About 4,046.86 hectares (10,000 acres) in Johnson County is irrigated cropland, while about 16,996.78 hectares (42,000 acres) is used for pasture. About 12,949.94 hectares (32,000 acres) of Johnson County is used for rangeland, which includes both native prairie that was never broken from sod and areas that were cultivated and then reseeded. Based on known soil types, land use in the vicinity of the Project is best suited for rangeland and native hay, introduced or domestic grasses for pasture and, if irrigated, corn, sorghum, and soybeans (USDA SCS, 1984).

20.1.8 Hydrogeology (Groundwater)

A hydrogeological characterization of the deposit was conducted during the core drilling program. The program included:

- 42 downhole packer-isolated injection and airlift testing in core holes.
- Installation of six, 2" PVC standpipe piezometers isolated in the carbonatite and open to large intervals of the deposit.
- Installation of two, nominal 2" PVC standpipe piezometers isolated in the 180 m (590 ft) thick Pennsylvanian aquitard above the carbonatite.
- Frequent measurement of water levels in open core holes and piezometers over six months.

The hydrogeological characteristics of the resource area were significant enough that a 10-day pumping test was conducted in the fall of 2014. During this initial test, an open borehole was pumped at 35 gpm, and the response was observed in nearby piezometers. These data were used to establish the prospective mine water inflow prediction that appears in the 2015 PEA level documents. However, the hydrogeological issues associated with these initial findings were considered to be significant enough for a second test, conducted in May and June of 2015. For this second test, a large diameter injection well was installed in the approximate center of the deposit, and two additional distant monitoring piezometers were established. Water was injected at a rate of 22 to 30 L/s (350 to 480 gpm) over a nominal 30-day period, and the response was measured by a series of instrumented piezometers. Analysis and interpretation of the data from these testing programs have been completed, and a preliminary conceptual model developed.

In 2017, NioCorp and Nordmin engaged Adrian Brown, a local hydrogeological consultant, to re-analyze the data set generated during the previous investigations. They concluded that ground freezing for the shafts is feasible, as is spot grouting in the mine. This is anticipated to reduce the mine water inflow to around 1,000 gpm. Water treatment can now be effectively handled with Reverse Osmosis (RO) treatment. Treated water may be used in the process circuit (or discharged, as necessary – though not anticipated) and the brines from the RO will be evaporated/crystallized to form a solid salt; this salt, in turn, will be disposed of in the engineered and lined Salt Management Cells.

While water samples collected from these deep holes, NEC 14-014 and Met-1, and the follow-up investigation by Nordmin, indicate very similar quality, overall, water sampling results are variable across the site. This includes total dissolved solids which can range in concentrations of over 18,000 ppm, with the major contributors being sodium and chloride. Both of the wells noted above also exceed EPA primary Maximum Contaminant Levels (MCLs) with respect to the following:

- Arsenic;
- Gross alpha; and
- Ra-226 + Ra-228.

Water from both of these wells also exceeds secondary MCLs with respect to chloride, fluoride, iron, manganese, sulphate, as well as total dissolved solids. NEC 14-014 also exceeds the secondary MCL for aluminum. There were no detectable pesticides or herbicides. Although the deep groundwater is not currently a drinking water source, concentrations were compared to drinking water standards as a reference to possible regulatory and management implications of groundwater disposal from future mine dewatering. Given the variability of water quality across the site, additional testing may be necessary to appropriately characterize the deep aquifer.

The deep groundwater chemistry data indicate a low-oxygen, chemically reducing groundwater system that is out of chemical equilibrium with surface conditions. Supporting evidence of this conclusion includes:

- Nitrogen species are mostly dominated by ammonia rather than nitrate or nitrite.
- Iron is elevated at neutral pH, a condition which is unlikely to occur in an oxygenated, natural system.
- Groundwater brought to the surface at some boreholes is initially black, changes to orange over a time period ranging from hours to days, then eventually turns clear while forming an orange precipitate. This is characteristic of water initially containing reduced ferrous iron that eventually oxidizes to ferric iron.

Further investigation is needed to determine the origin of the elevated concentrations in the groundwater, as well as refinement of the overall pumping requirements for the underground mining operation. Because of difficulties in handling these waters once they have been pumped to the surface, the additional testing remains a recommendation and must wait until surface management structures (ponds) and permitting have been completed.

20.1.9 Hydrology (Surface Water)

Surface water samples have been collected as part of baseline sampling on a quarterly basis since early 2014. Surface water sampling locations were selected to establish a baseline monitoring perimeter both upstream and downstream from all proposed facilities in the Project area. All samples were analyzed by Midwest Laboratory in Omaha for a comprehensive suite of metals and other inorganic analytes plus a panel of pesticides and herbicides. The preliminary results of the baseline program are as follows:

- Surface water in and around the Project area exhibits minor water quality impairment, as indicated by concentrations outside the limits of several secondary drinking water standards and several aquatic life criteria (i.e., aluminum, iron, and manganese).
- Average stream TDS concentrations fluctuate appreciably; however, this variability is most likely the result of post-harvest runoff containing excess sediments.
- Stream pH is consistently circum-neutral, ranging from about 6.6 to 8.2 standard units.
- Gross alpha, beta, Ra-226 and Ra-228 have been detected in several surface water samples, but at concentrations below their respective EPA MCL.

20.1.10 Wetlands/Riparian Zones

Project Site

Olsson Associates was retained in 2015 to conduct a wetland delineation and potential jurisdictional waters assessment in Sections 3, 28, 29, 32, 33, Township 3; 4 North, Range 11 East, Pawnee and Johnson counties, Nebraska. The purpose behind this investigation was to identify wetland and drainage features within the proposed Project boundary that were likely to be classified as jurisdictional waters of the U.S., and therefore be subject to permitting requirements by the USACE.

The study area, at the time of the site visit, consisted of existing agricultural fields, pastures, farmsteads and unnamed tributaries to Todd and Elk creeks. The majority of unnamed tributaries consisted of riparian areas and ponds that drained to Todd and Elk creeks. Many of the wooded areas that were not situated along drainages were located along fence lines as windbreaks. Most of the study area had been impacted by grazing livestock.

Wetlands were identified in agricultural fields, pastures, roadside ditches and abutting stream channels. During 2015 Olsson identified a total of 45 wetlands encompassing a total area of approximately 4.3 hectares (10.64 acres). Nine unnamed streams were also found during the field investigation for a total length of approximately 4.18 km (13,726 ft). An approved jurisdictional determination was obtained from the USACE on September 6, 2016. This approved jurisdictional determination confirmed jurisdiction of 11 stream reaches and associated wetlands as waters of the U.S. within the Project area.

20.1.11 Geochemistry

A geochemical characterization program for the mineralized material, waste rock, and tailings has been initiated by SRK for the Project. Preliminary results are provided in the following sections.

Niobium Mineralized Material

Preliminary results suggest that the mineralized material has the potential to leach various constituents due to exposure to meteoric precipitation. Laboratory leach tests of a composite sample of this material from drill hole NEC11-001 indicate that, at a minimum, fluoride and nitrate are likely to be mobilized during surface stockpiling. Note: fluoride is the only analyte that exceeds the EPA MCL in the leach testing. Nitrate and several metals are detectable, but not at concentrations exceeding their respective MCL for drinking water).

Contained within the mineralized material are naturally occurring uranium and thorium. Based on existing drilling data, the average thorium and uranium content in the Mineral Resource is 0.034% and 0.0045%, respectively (0.0395% in total). Leach testing of potential waste materials has not produced concentrations of these radionuclides above regulatory limits. However, the concentrations in the rock are relatively elevated (approximate relative concentrations):

- Uranium = 33 ppm;
- Thorium = 303 ppm;
- Gross alpha = 200 pCi/g;
- Gross beta = 160 pCi/g;
- Radium 226 = 56 pCi/g; and
- Radium 228 = 18 pCi/g.

The current assay database for the Elk Creek Project contains 6,288 samples for which uranium and thorium were analyzed and detected. Of this dataset, 1,122 samples (~18%) had a combined uranium+thorium concentration of greater than (">") 500 ppm. The mean and median concentration of uranium+thorium was 336 ppm and 273 ppm, respectively. The mineralized material suitable for mill feed will require proper management during the periods it is exposed on the surface, prior to processing.

Waste Rock and Overburden

There are two basic types of waste rock associated with the Deposit. These include:

- Pennsylvanian limestones and mudstones – The upper 30 m (100 ft) of lithology consists of unconsolidated glacial till, underlain by a 170 to 180 m (560 to 590 ft) of low-permeability, Pennsylvanian-aged mudstone and limestone, otherwise known as the "Pennsylvanian strata" (PENN). The PENN is reportedly continuous across the state of Nebraska, and locally it behaves as a very effective aquitard. This material is neutralizing due to its high carbonate content. In terms of metal leaching characteristics, Meteoric Water Mobility Procedure (MWMP) testing suggests that the PENN has the potential to leach antimony and selenium at concentrations above general surface water standards. Additionally, the PENN exhibits a propensity to leach gross alpha and radium above regulatory limits. This lithology is the primary source for construction aggregate in Nebraska.
- Non-ore grade carbonatite – Preliminary assessment of the host rock identified visual sulphide content of up to 1% based on observations by core loggers. Laboratory analyses confirmed the sulphide content at around 1.34%. This sulphide consists mainly of pyrite, chalcopyrite, bornite, galena, sphalerite, and possibly pyrrhotite. However, even with detectable sulphide content, the carbonatite waste rock is still net neutralizing given the high carbonate content.
- Of the 94 rock samples collected over a 255 m (837 ft) vertical length of the waste rock and mineralized zone, eight samples (8.5%) registered a reading of >25 µRads/hour. These levels are not considered to be hazardous but may be used as a diagnostic tool to identify elevated concentrations of uranium and thorium.

Temporary surface disposal of waste rock will be predicated on minimizing meteoric infiltration and leaching of this material. NioCorp has conservatively elected to line the waste rock and low grade mineralized material stockpiles, and actively manage any runoff derived from these materials until such time as that, and residual ore and low grade mineralized materials can be processed, and the surface waste rock transferred to the TSF for final disposal.

Tailings

Representative quantities of post-process tailings from the metallurgical testing program have been limited. Geochemical testing and characterization (including radiological testing) of the tailings was completed in Q3 of 2017 when the testing of the beneficiation process was finalized, and the need for, and usability of, tailings as underground backfill was evaluated. Characterization of the various tailings materials has included both the TCLP and the SPLP, which are designed to determine the mobility of both organic and inorganic analytes present in the liquid, solid, and multiphasic wastes, and assist in the proper classification of waste materials. The most recent tailings material testing showed negligible mobility of regulated constituents (indicating a non-hazardous classification), although the pH of the TCLP/SPLP extracts remained high. While the calcined tailings are likely to produce heat when exposed to atmospheric moisture and precipitation (i.e., exothermic hydration), this reaction is not "violent" as defined under 40 CFR § 261.23(2) *Characteristic of reactivity* [for hazardous wastes] (adopted by the State of Nebraska under Title 128 - Nebraska Hazardous Waste Regulations). Given the limited quantities of ore available for this testing, further characterization of these materials is recommended in order to establish representativeness with the mineral deposit as a whole.

20.1.12 Known Environmental Issues

There are currently no known environmental issues that are likely to materially impact NioCorp's ability to extract the Mineral Resources or Mineral Reserves at the Elk Creek Project. However, there are several key permitting challenges and uncertainties associated with the dewatering and ground freezing program that may affect the Project financing and overall schedule. Risks are summarized in Section 24.2.

20.2 Waste Management and Disposal

20.2.1 Overburden and Waste Rock

Overburden developed during mine construction will be excavated, crushed and used as a construction material. The limited quantities of waste rock will be temporarily stored on the surface prior to final disposal within the lined tailings impoundment. The majority of the waste rock generated by the mining operations will remain underground and be placed in secondary backfill stopes. Because of the potential presence of low levels of Naturally Occurring Radioactive Materials (NORMs) in some of the waste rock brought to the surface, NioCorp will take the conservative approach of placing this material on a lined containment facility from which any surface water runoff or seepage can be controlled and managed. It is not anticipated that any of these materials will remain on the surface post closure.

20.2.2 Tailings

NioCorp has chosen to design the solids portion of the TSF to include.61 m (2 ft) of compacted soil liner with a permeability of 1×10^{-7} cm/s or less, overlain by an 80-mil HDPE liner, overlain by an overliner drain system. The water retaining portion of the facility will be lined with a double lined system consisting of a 60-mil HDPE secondary liner and 80-mil HDPE primary liner with an active leak detection system between. This conservative approach will likely ensure adequate protection of local groundwater resources. Additional details regarding the TSF are provided in Section 18.11. Closure of the TSF is discussed in Section 20.5.3.

20.2.3 Project Waste Disposal

Solid Waste

The solid waste generated by the Project, as defined by 40 CFR § 261.2, will be collected and transported to the Douglas County/Pheasant Point Landfill, located near Elk City in northwest Douglas County, 140 km (87 miles) from the Project site. Under current management practices, the Pheasant Point Landfill has 92 years of projected remaining life (NDEQ, 2012).

Reject brines from the proposed RO water treatment plant are currently anticipated to be evaporated (crystallized) and the “solid” residue disposed of in the engineered and lined Salt Management Cells. Alternatively, these RO brines may be piped away from the mine site and re-injected into the deep underground aquifer, though this option still requires considerable evaluation before being considered viable.

Hazardous Waste

Any hazardous waste generated by the Project will be transported by licensed operators to the Clean Harbors Environmental Services facility in Deer Trail, Colorado, 756 km (470 miles) away, in accordance with hazardous waste manifest and pre-transport requirements.

20.2.4 Site Monitoring

Surface water and groundwater monitoring will continue throughout the LOM, as initiated during the baseline study program. Additional monitoring locations may be added during the regulatory review process. This will include, but not necessarily be limited to groundwater monitoring downgradient of the tailings storage facility, mine water collection pond, and the discharge to the Missouri River under the state NPDES program.

Geotechnical monitoring of the TSF facility will also occur on a regular basis as per state regulatory requirements.

Ambient air quality monitoring will likely continue and may include emissions control monitoring once operations commence. This will be conducted in accordance with all applicable state regulatory requirements.

The presence of NORMs in the mineralized ore and several of the process waste streams will necessitate the need for comprehensive site-wide monitoring. At a minimum, the Broad Scope License will require the development and implementation of a formal Radiation Safety program for the facility, including environmental and personnel monitoring programs, which are discussed further in the following sections.

20.2.5 Water Management

Operational Water Management

For the first several years of construction, the advancement of the shaft and underground workings will require limited dewatering, anticipated to be through lower-level sumping and pumping for surface collection and disposal. Initially, water will be stored in the lined Salt Management Cell #1 during construction or will be trucked off-site for treatment at a local publicly owned treatment works. Excess water in the Salt Management Cells will be spray evaporated within the footprint of the Cell, to avoid the reintroduction of soluble salts into the water treatment system. Temporary on-site storage or off-site shipment and disposal of the crystallizer solid waste may be necessary until construction of the Salt Management Cells is completed.

Once full operations commence, NioCorp anticipates a shortfall of approximately 3,700 gpm of operational and processing water, as the underground mine dewatering is only expected to produce 1,000 gpm. To make up this shortfall, NioCorp proposes the following sources for additional water:

1. Tecumseh Board of Public Works water supply line (~2,000 gpm) – Tecumseh Board of Public Works, which maintains the infrastructure and supplies residential and commercial users in the City of Tecumseh, might run a line to the project site to supply all of the necessary shortfalls.
2. Local Landowner Well #1 (~500 gpm) – A new well on a local landowner's property has the potential to supply up to 500 gpm of the project's needs. Because there will be a transfer of water from one property to another, a Groundwater Transfer Permit will need to be issued by the Nemaha Natural Resources District pursuant to Chapter 11 of the *Management Area Rules and Regulations for Groundwater Quantity Management Areas*.
3. Local Landowner Well #2 – NioCorp has the option to connect to an existing well as well as install a new well to supply an additional 1,500 gpm.

NioCorp is pursuing approval of all three sources as insurance that there are no disruptions in the water supply during operations. None of the permitting for these alternative water sources is considered particularly onerous or time-consuming

Once tailings begin being deposited in the TSF, internal contact water (from residual moisture in the tailings and precipitation falling within the impoundment footprint) will need to be actively managed. This water will be collected and treated using lime softening to precipitate hydroxide and carbonate solid forms for many of the inorganic constituents. The treated water will be filtered to remove the solids (which will be returned to the TSF for disposal), and the clean water will be pumped to the process plant RO system for further treatment. The clean water from the process plant RO unit will be used in the process plant, and the reject concentrate will be crystallized and deposited back into the Salt Management Cells.

<u>Post-Closure Water Management</u>

Upon cessation of mining, the limited subsurface dewatering operations will be halted, and the workings will be allowed to flood. Until such time that the TSF closure cover can be constructed, and any residual water or seepage eliminated, the TSF contact water will require active management. Whether the singular TSF brine stream from the RO plant can continue to be crystallized and deposited in the Salt Management Cells or if another disposal method needs to be considered (i.e., disposal in the deep mine workings of off-site disposal facility), will be evaluated during the final years of operation.

20.2.6 Chemical and Reagents Handling

Process reagents and chemicals, including but not limited to: sulphur (molten), sodium hydroxide (NaOH), magnesium hydroxide (MgOH), lime (CaO), aluminum, iron (scrap and powder), boiler feed chemicals, water treatment plant chemicals, cooling tower chemicals, and solvent extraction circuit chemicals, will be trucked to the site and stored in specially designed and constructed containers located within concrete and concrete-bermed areas. For liquid chemicals and reagents, these bermed areas will be designed to contain at least 110% of the capacity of the largest storage tank or tanks in series within the berm. Solid chemicals and reagents will be stored in flow bins or silos specifically designed for these materials. Reagents will be stored in a manner that inhibits any inter-mixing and subsequent reactions.

Fuel (i.e., gasoline, diesel fuel, and propane), antifreeze, petroleum oils, and solvents will be delivered to the mine in tanker trucks, totes and barrels for transfer to authorized storage tanks. Storage tanks or tanks in series will be enclosed by berms sized to contain at least 110% of the capacity of the largest tank in the event of a spill or tank rupture. NioCorp will develop a comprehensive *Spill Prevention Control and Countermeasures Plan (SPCC)* to be implemented in the event of a spill or release of petroleum products.

Explosive materials transported to the site will include blasting agents and initiation devices. Blasting agents are comprised primarily of ammonium nitrate and fuel oil. The ammonium nitrate and fuel oil will be stored in appropriate storage bins separate from the explosives magazine. Blasting initiation devices will be stored in prefabricated magazines in conformance with U.S. Bureau of Alcohol, Tobacco and Firearms (BATF), MSHA, and applicable state and local regulations.

20.3 Project Permitting Requirements

Engagement of local, state and federal regulators has commenced. Initiation of the balance of permitting for the Project is dependent upon the completion of the mine plan and surface facilities being developed as part of this technical document. Typically, larger mining operations such as this have the benefit of a pre-feasibility stage of analysis and development from which permitting is generally initiated. With the completion and publication of this feasibility study, the balance of permitting for the Project can commence.

The Project has considered, and will likely be held to permitting requirements that are determined to be necessary by Johnson and Pawnee counties, the State of Nebraska, and the U.S. Environmental Protection Agency and USACE national policies, such as the National Environmental Policy Act (42 U.S.C. 4321) and the Clean Water Act (33 U.S.C. 1251 *et seq*.). The list of potentially applicable permits and authorizations for the Project are presented in Table 20-1.

Table 20-1: Project Permits

Permit/Approval	Issuing Authority	Permit Purpose	Status
Federal Permits Approvals and Registrations			
Explosives Permit	U.S. Bureau of Alcohol, Tobacco and Firearms (BATF)	Storage and use of explosives	Mine Safety and Health Administration (MSHA) and the Department of Homeland Security (DHS) will also regulate explosives at a mining operation.
EPA Hazardous Waste ID No.	U.S. Environmental Protection Agency (EPA)	Registration as a Conditionally Exempt Small Quantity Generator (CESQG) or a Small Quantity Generator (SQG) of waste	NioCorp laboratory facilities are likely to generate small quantities of hazardous waste.
Spill Prevention, Control, and Countermeasure (SPCC) Plan	U.S. Environmental Protection Agency (EPA)	Regulation of facilities having an aggregate aboveground oil storage capacity greater than 1,320 gallons or a completely buried storage capacity greater than 42,000 gallons with a nexus to jurisdictional waters	**REQUIRED.** Adjacent jurisdictional drainages.
Notification of Commencement of Operations	Mine Safety and Health Administration (MSHA)	Mine safety inspections, safety training plan, mine registration	**REQUIRED.** All mining operations in Nebraska.
Obstruction Evaluation / Airport Airspace Analysis (OE/AAA)	Federal Aviation Administration (FAA)	Notification of the Administrator of the FAA for any construction or alteration exceeding 200 ft above ground level.	**REQUIRED:** If and project components exceed 200 feet in height.
Federal Communications Commission Permit	Federal Communications Commission (FCC)	Frequency registrations for radio/microwave communication facilities	**REQUIRED**. If NioCorp intends to use business radios to transmit on their own frequency.
State Permits, Authorizations and Registrations			
Permit to Appropriate Water	State of Nebraska Department of Natural Resources (DNR)	Regulates the use and storage of surface and ground waters	**REQUIRED** to appropriate water.
Explosives Permit	Nebraska State Patrol	Regulates the use, storage, or manufacture of explosive materials.	**REQUIRED.** Also regulated by BATF, MSHA, and DHS.
Permit to Discharge under the National Pollutant Discharge Elimination System (NPDES)	State of Nebraska Department of Environmental Quality (DEQ)	Multiple permits are applicable to the discharge of industrial wastewater and stormwater.	**REQUIRED.** The project will require an industrial stormwater discharge permit. The project will not discharge wastewater.
Mineral Exploration Permit	State of Nebraska DEQ	Regulates the exploration for minerals by boring, drilling, driving, or digging.	**REQUIRED.** Already obtained for the exploration drilling program.
Air Construction Permit	State of Nebraska DEQ (under Federal PSD Program)	Regulates emissions during construction activities to protect ambient air quality.	**REQUIRED**. Under Nebraska Administrative Code (NAC) Title 129.
Air Operating Permit	State of Nebraska DEQ (under Federal PSD Program)	Regulates emissions during operation to protect ambient air quality. Will be based on a Feasibility Study mine plan.	**REQUIRED**. Class I (Title V) federal major source PSD operating permit will likely be required as per NAC 129.
Water Well Installation Declaratory Ruling Request	Nebraska Department of Health and Human Services, Division of Public Health	Water well installation requirements; well must be registered with the Department of Natural Resources.	**REQUIRED.** Already obtained for the hydrogeological portion of the exploration drilling program.
Authorization for Class V Well Underground Injection	State of Nebraska DEQ	All activities conducted pursuant to Title 122 - Rules and Regulations for Underground Injection and Mineral Production Wells.	**REQUIRED.** Already obtained for the hydrogeological portion of the exploration drilling program. Will also be required for future disposal of tailings and/or crystalized RO brine gels in underground workings.
Septic Systems – Permit for Onsite Wastewater Treatment System Construction/Operations	State of Nebraska DEQ	Protects surface water and groundwater as well as public health and welfare through the use of standardized design requirements.	**REQUIRED**. Needed if the septic system does not meet the "Authorization by Rule" requirements due to the quantity or quality of the wastewater, as per NAC 124.
Boiler Inspection Certificate	Nebraska Department of Labor	Protects public safety through an inspection and approval process of boilers.	**REQUIRED**. For installation of the boiler(s) is installed in any of the facility buildings.
Section 401 Water Quality Certification	State of Nebraska DEQ	The program evaluates applications for federal permits and licenses that involve discharge to waters of the state and determine whether the proposed activity complies with NAC Title 117- Nebraska Surface Water Quality Standards. Isolated wetlands are included in NAC Title 117.	**NOT REQUIRED**. Only required as part of Section 404 authorization. Not currently anticipated.
Development Permit	State of Nebraska DEQ/Johnson County Floodplain Administrator	The program regulates building requirements for any structures that are constructed on a floodplain.	**REQUIRED.** Will be needed if NioCorp constructs any building on a designated floodplain.

Permit/Approval	Issuing Authority	Permit Purpose	Status
Fire and Life Safety Permit	Nebraska State Fire Marshall	Review of non-structural features of fire and life safety.	**REQUIRED**. Project proponent to submit operating and building plans. State Fire Marshall will then determine required inspections as per NFPA 101.
State Business License	Nebraska Secretary of State	License to operate in the state of Nebraska.	**REQUIRED**. All business entities in Nebraska.
Retail Sales Permit or Exemption Certificate	Nebraska State Tax Commissioner	Permit to buy wholesale or sell retail.	**MAY BE REQUIRED**. Will be required if NioCorp is direct selling niobium product.
Solid Waste Management Permit	State of Nebraska DEQ	Regulates the construction and operation of solid waste management facilities.	**REQUIRED**. Will be needed if NioCorp intends to create an on-site solid waste management facility. This may include the TSF.
Drinking Water Construction Permit	Nebraska Department of Health and Safety	The Drinking Water Construction Permit regulates the design and construction of a public water system.	**MAY BE REQUIRED**. All drinking water systems that serve more than 25 individuals and are considered to be "non-transient and non-community" are required to obtain a Drinking Water Construction Permit. This will include the use of RO permeate produced at the plant site.
Drinking Water Permit to Operate	Nebraska Department of Health and Safety	Defines testing and water quality criteria for public drinking water systems.	**MAY BE REQUIRED**. All drinking water systems that serve more than 25 individuals and are considered to be "non-transient and non-community" are required to obtain a Drinking Water Permit to Operate.
Radioactive Materials Program and Licensing	Nebraska Department of Health and Human Safety	Regulates and inspects users of radioactive materials.	**REQUIRED.** If the plant uses sealed sources for process measurements or if naturally occurring, radioactive materials are possessed as a result of beneficiation activities.
Hazardous Waste Management	State of Nebraska DEQ	Management and recycling of hazardous wastes.	**REQUIRED**. As per Title 128 of the *Nebraska Hazardous Waste Regulations* NioCorp must notify the NDEQ of hazardous wastes generated or transported from the facility.
Dam Safety Approval	State of Nebraska DNR	Regulates the design and construction of any dam (i.e., any artificial barrier with the ability to impound water or liquid-borne materials).	**REQUIRED.** Will be required for TSF (dam) and may be required for the Mine Water Pond depending on the final design capacity.
Water Storage Permit	State of Nebraska DNR	Regulates any water impoundment that has a normal operating water volume of at least 15 AF below the spillway.	**MAY BE REQUIRED.** May be required for the Mine Water Pond, if it will impound greater than 15 AF below the spillway.
Local Permits for Johnson and Pawnee Counties			
Water Well Permit	Nemaha Natural Resources District	Regulates installation of groundwater wells	**REQUIRED**. This permit will be required to install a new water supply well.
Water Well Transfer Permit	Nemaha Natural Resources District	Regulates transfer of groundwater off overlying land	**REQUIRED**. This permit will be required to transfer water from wells located on a separate property to be used for water supply.
Building and Construction Permits	Johnson County Zoning Administrator	Ensure compliance with local building standards/requirements.	**REQUIRED**. This permit will most likely be included with the Permitted Use Zoning Permit
County Road Use and Maintenance Permit/Agreement	Johnson County Zoning Administrator	Use and maintenance of county roads.	**MAY BE REQUIRED**. Will be needed if NioCorp intends to maintain any of the area county roads.
County Road Use and Maintenance Permit/Agreement	Pawnee County Commission	Use and maintenance of county roads.	**MAY BE REQUIRED**. Will be needed if NioCorp intends to maintain any of the area county roads.
Permitted Use Zoning Permit	Johnson County Zoning Administrator	Regulates and authorizes permitted uses.	**REQUIRED**. Issuance of this permit will require completion on an application form, and at least one meeting with the county zoning regulators and at least one public comment meeting.
Special Use Permit	Pawnee County Assessor	Regulates and authorizes permitted uses	**REQUIRED**. TSF land currently zoned for agriculture. Zoning regulations allow for mineral extraction.

Source: SRK, 2017

The following is a brief discussion of some of the more material permits which are likely to be required for the project (Note: with respect to the Underground Injection Control (UIC) permit, the discussion is included only as an alternative to the planned treatment and disposal of excess water).

20.3.1 Nebraska Underground Injection Control (UIC)

In the event that crystallization of the RO water treatment brines becomes impractical, NioCorp may alternatively opt to reinject the reject waters back underground. This activity will, necessarily, require a permit. The UIC Program of the NDEQ Water Division issues and reviews permits, conducts inspections and performs compliance reviews for wells used to inject fluids into the subsurface. The program must ensure that injection activities are in compliance with state and federal regulations, and that groundwater is protected from potential contamination. Injection wells are classified by activity, with most activity concentrating on Class I, II, III, and V wells. Class II wells are associated with oil and gas production and are regulated by the Nebraska Oil and Gas Conservation Commission. NDEQ has authority over and manages, Class I, III and V wells. A water treatment system brine re-injection well is likely to be a Class V well.

The EPA delegates the UIC program to the NDEQ and provides authority for the program through the Safe Drinking Water Act. The Natural Resource Districts across the state have also developed sets of rules and regulations (NDNR) regarding permitting requirements and the installation of wells based on specific Groundwater Management Plans, and the NDNR requires that all wells installed in the state must be registered. Additionally, the NDNR is charged with issuing permits for industrial use of groundwater.

20.3.2 DHHS Radioactive Materials Program and Licensing

The Elk Creek Mineral Resource, and thus the residual post-processing tailings, will contain trace amounts of uranium and thorium, which are Naturally Occurring Radioactive Materials (NORM). At issue will be the ultimate classification of the tailings because of these constituents, and the occurrence of these constituents in the processing circuit. Preliminary discussions with the State of Nebraska have indicated that either a Specific or Broad Scope Radioactive Materials License, issued under 180 NAC 3-013 by the Nebraska (DHHS), will likely be necessary, as confirmed with the DHHS on December 6, 2018.

As defined by the Nebraska Radiation Control Act, radioactive material means any material, whether solid, liquid, or gas, which emits ionizing radiation spontaneously. Radioactive material includes but is not limited to, accelerator-produced material, by-product material, naturally occurring material, source material, and special nuclear material. The classification of radioactive material appears to be irrespective of any concentration – it merely has to emit ionizing radiation. The material for processing, waste rock, and tailings are likely to be seen as naturally occurring material, and therefore, classified as a radioactive material.

The DHHS retains the right to require registration or licensing of [any] radioactive material in order to maintain compatibility and equivalency with the standards and regulatory programs of the federal government or to protect the occupational and public health and safety and the environment [NRS 71-3507(2)]. At the same time, the DHHS can exempt certain sources of radiation or kinds of uses or users from licensing or registration requirements when the department finds that the exemption will not constitute a significant risk to occupational and public health and safety and the environment [NRS 71-3507(4)].

At a minimum, the Broad Scope License will require the development and implementation of a formal Radiation Safety program for the facility, including environmental and personnel monitoring programs, appropriate warning signage be displayed around the site, and a final permanent closure cover for the TSF be engineered and constructed. DHHS oversight and the Broad Scope License will necessarily cover all points of potential worker exposure, including but not limited to underground mining, crushing, transportation and stockpiling, conveying, and processing, especially in areas were airborne dust containing uranium and thorium (as well as radon gas) can occur. Worker protection from ionizing radiation and radon will also be regulated by the U.S. Department of Labor, Mine Safety and Health Administration (MSHA) under 30 CFR PART 57 – Safety and Health Standards – Underground Metal and Nonmetal Mines, Subpart D – Air Quality, Radiation, Physical Agents, and Diesel Particulate Matter. Both programs will examine potential exposure limits, engineering and administrative control requirements, the use of appropriate Personal Protective Equipment (PPE), and monitoring/reporting programs to ensure worker protection.

In the likely event that the Elk Creek facility is regulated in this way, some land restrictions may be invoked at the time of mine closure. While these requirements appear to be directed at uranium mills and commercial radioactive waste disposal facilities, and not necessarily mine tailings for operations containing NORM, the law makes no clear distinction between the facility types; the State of Nebraska may apply them under either scenario, which might even include the possibility of deeding the land to the State of Nebraska following closure.

Irrespective of ultimate classification, the tailings (and their disposal facility) will require financial assurance for reclamation and closure. Again, these rules appear to be directed at uranium mill tailings and low-level radioactive waste facilities but are non-specific enough that they may be applied to other situations where NORMs are being actively managed. In addition to a direct reclamation financial assurance, it is probable that the state will require a funding mechanism (i.e., trust fund, escrow, etc.) for monitoring and maintenance of the facility in the longer term as part of a Broad Scope License.

<u>DHHS License Timing</u>

NioCorp estimates that a Broad Scope License for the Project will take approximately 16 months to acquire once the formal application has been submitted and will involve several months of discussions and negotiations related to engineering, design, monitoring, and terms and conditions. At this time, the federal Nuclear Regulatory Commission shall play a purely advisory role in these negotiations.

20.3.3 Nebraska Air Quality Permitting

The Nebraska air regulations are primarily based on regulations developed by the U.S. EPA to address the Clean Air Act requirements. Air quality permits are the primary tool used by the NDEQ to implement the Clean Air Act. For businesses that intend to operate unit sources that emit regulated pollutants that will exceed Nebraska air quality thresholds, a construction permit will be required.

There are two types of construction permits: state construction permits and federal construction permits, known as New Source Review or Prevention of Significant Deterioration (PSD) permits. The type of construction permit that is needed will depend on the quantity of air pollutants that potentially may be released from the new plant or expansion project.

Because the Project includes a primary sulphuric acid plant [a regulated facility under 40 CFR § 52.21(b) which anticipates emissions in excess of the regulatory thresholds], and since Nebraska is currently classified as in "attainment" of all ambient air quality standards, a federal PSD construction permit will be required. The entire permit process is expected to take at least 190 days, provided that there are no significant technical issues or problems in obtaining information, and the facility has submitted a complete application (including detailed air dispersion modelling). Typically, however, PSD permits require over one year in order to complete.

The PSD permitting process includes both public and EPA review and comment periods. Part of the EPA review of the application includes additional scoping through the issuance of a PSD Public Notice Package to other federal agencies and land managers, local officials, affected states and others, as necessary. This can lengthen the permit timeline. However, opportunities exist within the program to authorize certain early construction activities (typically limited to ground clearing and grading activities) prior to permit issuance. The nature and extent of these variances must be negotiated and applied for with the NDEQ.

The sulphuric acid plant is currently assumed to only exist for the purpose of supplying sulphuric acid to the super-alloy materials production process. Alternatively, the sulphuric acid plant may be built with enough capacity to provide the majority of sulphuric acid off-site as a saleable product.

In addition to the construction permit, the NDEQ also issues operating permits based on a source's level of emissions. There are two types of operating permits: major source (federal program) and minor source (state program). As before, the potential to emit associated with the sulphuric acid plant will necessitate the issuance of a major source permit for the operation. The federal major source program (a.k.a., Class I or Title V) regulates larger sources of air pollution. A Class I source has the potential-to-emit quantities greater than:

- 100 t/y of any criteria air pollutant, excluding lead;
- 10 t/y of any single hazardous air pollutant (HAP) or 25 t/y of a combination of HAPs; or
- 5 t/y of lead.

The operating permit incorporates all of a source's requirements into one permit, including all construction permit limitations and federal regulations. Operating permits usually require additional monitoring, stack testing, reporting, and recordkeeping. However, the application for the operating permit need only be submitted within 12 months after the emissions unit(s) begin operation, or within 12 months of becoming subject to the operating permit requirements, whichever is earlier.

Earthworks associated with digging holes, grading soil, stockpiling of topsoil, and land clearing where the new source will be located, which will not result in a change in actual emissions, and are not of a permanent nature, do not require a construction permit or prior approval of the NDEQ under Title 129, Chapter 17 (*Acceptable Pre-Construction Dirt Work* dated August 2016).

20.3.4 Nebraska Dam Permitting

The Department of Natural Resources (DNR) regulates the construction, operation, and maintenance of dams in Nebraska to protect life and property from dam failures. The DNR regulates all dams in the state that:

- Have a total height of 7.62 m (25 ft) or more and an impounding capacity at the top of the dam that is greater than 1.85 hectare-15 (15 acre-ft);

- Have an impounding capacity at the top of dam of 6.17 hectare-metres (50 acre-ft) or more and a total height that is greater than 1.8 m (6 ft); or
- Are located in a high hazard potential location.

As promulgated in Chapter 46, Article 16 - Safety of Dams and Reservoirs, approval of applications shall be issued within 90 days after receipt of the "completed" application plus any extensions of time required to resolve matters diligently pursued by the applicant. At the discretion of the DNR, one or more public hearings may be held on an application (46-1654). This will, of course, add additional time to the overall permitting process for the TSF and Mine Water Pond.

20.3.5 Greenhouse Gas Permitting

The NDEQ defines Greenhouse Gases (GHG) as chemical compounds that, when emitted into the atmosphere, have the potential to cause climate change. There are currently 73 GHG chemicals identified in 40 CFR § 98 Table A-1 to Subpart A, which include, but are not limited to CO_2, CH_4, N_2O, and Fluorinated GHGs (SF6, PFCs, HFCs). Recent rulemaking by the EPA incorporates changes impacting the regulation of GHGs and establishes emission thresholds for GHG emissions, while provides the State of Nebraska (among others) the authority to issue PSD permits governing GHGs.

Because not all GHGs remain in the atmosphere for the same amount of time or have the same potential effect in the atmosphere, a system of equivalents (using CO_2 as a baseline or CO_2^e) was developed to account for the variation between compounds. For New Sources, the PSD permitting threshold is 100,000 t/y CO_2^e (as of July 1, 2011). Preliminary calculations for the Project suggest that the operation will be above this threshold.

To date, the EPA has not implemented a minor source program for GHGs, and Nebraska has not chosen to implement a minor source program either. At this time, no fees will be collected, but all sources will be required to report GHG emissions.

20.3.6 Permitting Status

Initial permitting activities commenced in January 2015 with the submission of a Jurisdictional Delineation report to the USACE for the mine site. In addition, several high-level meetings with federal, state and local agencies have been held in order to introduce the Project to the local regulatory communities.

<u>NDEQ Construction Air Permit</u>

A pre-application meeting took place with the NDEQ on September 8, 2016. Subsequent to this meeting, the project team has begun analysis of processes that result in emissions regulated by the NDEQ and have completed initial air emissions calculations. These calculations continue to evolve as the process develops. At the current time, it appears the Project will require a PSD major construction and operating permit. The PSD process requires ambient air monitoring for a number of pollutants, and a request to trim that list to $PM_{2.5}$ was submitted to the NDEQ on December 6, 2016. The NDEQ denied NioCorp's request, and the Company has proceeded to conduct air monitoring for all of the monitoring parameters required under the PSD program.

NioCorp currently anticipates submitting the Air Construction Permit application by mid-year 2019 with an anticipated permit issuance in 2020.

<u>Temporary Limestone Processing</u>

The Project may require temporary limestone processing during the construction of the mine shaft. Third-party portable limestone processing equipment may be used on site to crush and handle limestone removed from the mine shaft, so long as that material meets construction specifications and does not leach potentially deleterious constituents (i.e., heavy metals or NORMs). The NDEQ has confirmed that third-party operators will be required to have an air quality permit to operate equipment on site.

20.3.7 Post-Performance and Reclamation Bonding

In addition to lacking hardrock mining regulations for reclamation and closure, there are also limited requirements for the provision of financial sureties with respect to hardrock mining operations in Nebraska. One possible exception may include the scenario in which the facility falls under a broad scope radiological license, which has financial assurance requirements for reclamation and closure ("decommissioning funding plan"). As noted before, however, these rules appear to be directed at uranium mill tailings and low-level radioactive waste facilities, but are vague enough that they may be applied to other situations where NORMs are being managed, though NioCorp has conservatively assumed that the licensure program and financial surety requirements will apply to the Project. These surety requirements extend to long-term site monitoring, maintenance, and care, and include the following mechanisms:

- Pre-payment (Trust Fund)
- Surety Bond
- Insurance
- Letters of Credit
- Corporate Guarantee (provided parent company passes the financial test)

In addition, financial assurances will also be required for the TSF, for which jurisdiction will fall under the NDEQ Title 132 - Integrated Solid Waste Management Regulations, and includes the requirement for a detailed, third-party closure cost estimate, proper disposal of all materials or wastes left at the site, and post-closure care for the solid waste disposal area in compliance with the post-closure plan. Allowable mechanisms for financial assurance under the solid waste regulations include:

- Trust Funds
- Surety Bonds Guaranteeing Payment or Performance
- Letters of Credit
- Insurance
- Corporate Financial Tests
- Local Government Financial Tests
- Corporate Guarantees
- Local Government Guarantee

At this time, the type and phased amount of financial surety for the Project has not yet been established, though the amount of bond will only reflect the liability on the ground at any given time (i.e., NioCorp is not likely to be required to bond for reclamation of all of the TSF cells when only one will be active and unreclaimed at any time). The specific requirements will be refined through meetings and negotiations with the two agencies and the submission of formal permit applications.

20.4 Community Relations and Social Responsibilities

Community relations and stakeholder engagement have been undertaken in parallel with field operations in Nebraska and have included town hall and individual meetings with local landowners. Some early communications have occurred between NioCorp and Johnson, Pawnee, Nemaha and Richardson County representatives (including the county commissioners) as well as the Southeast Nebraska Development District. Given the schedule proposed by NioCorp for the Project, all of the relevant regulatory agencies will need to be formally engaged as soon as possible using the designs presented herein as the basis for permitting. Any significant deviations from this design may have an impact on overall Project timing.

NioCorp is committed to ensuring that a proper Social License is garnered from the community and stakeholders. Thus far, support for the Project has been positive from those who have been engaged and notified of the pending Project. However, as with any major mining project, there remain vocal opponents and non-governmental organizations (NGOs) who will oppose the Project on principal alone. These groups are likely to include organizations such as Bold Nebraska, a citizen group focused on "*taking actions critical to protecting the Good Life*." NioCorp has already engaged with Bold Nebraska in early discussions about the Project on May 23, 2016, and has kept the group informed of major developments.

20.4.1 Safety and Health

Occupational Safety and Health at the Project will be strictly regulated by the U.S. Department of Labor, Mine Safety & Health Administration (MSHA), under Title 30 of the Code of Federal Regulations, Mineral Resources, Parts 1 through 199 (30 CFR Parts 1 through 199). This includes all of the training requirements specified in 30 CFR Parts 46 through 49. Given the radiological nature of the mineralized material, MSHA will likely institute radon exposure and monitoring requirements on all underground workers in accordance with 30 CFR § 57.5039 thru § 57.5047.

Because Nebraska has not enacted any workplace safety and health rules, the federal Occupational Safety and Health Act (OSH Act) governs workplace health and safety requirements in private (private businesses and non-profit organizations) sector workplaces. In addition, the Nebraska Occupational Safety and Health Surveillance Program (NOSHP), established in 2010 under the Nebraska Department of Health & Human Services, provides state-based occupational health surveillance, while the Nebraska Department of Labor (DOL) Office of Safety is charged with the protection of people and property through enforcement of the Nebraska Amusement Ride, Boiler Inspection, and Conveyance Safety Acts. With respect to the Project, DOL safety staff will inspect boilers and pressure vessels to ensure that they are properly installed and maintained.

20.5 Reclamation and Closure

Without specific hardrock mining regulations, there are limited obligatory requirements for reclamation and closure of mining properties in Nebraska. There are provisions, however, within the applicable regulatory framework which are likely to be applied to the Project during the permit and licensing processes, specifically those associated with the TSF. The following sections provide a summary of the key elements to the approaches proposed for closure and reclamation of the Project and form the basis for the closure cost estimate.

20.5.1 Surface Disturbance

The principal objective of the surface reclamation plan will be to return disturbed lands to productive post-mining land use. Soils, vegetation, wildlife and radiological baseline data will be used as guidelines for the design, completion, and evaluation of surface reclamation. Final surface reclamation will blend affected areas with adjacent undisturbed lands so as to re-establish original slope and topography and present a natural appearance. Surface reclamation efforts will strive to limit soil erosion by wind and water, sedimentation, and re-establish natural drainage patterns.

20.5.2 Buildings and Equipment

All surface structures and equipment will be evaluated for appropriate post-closure re-use or disposal. Buildings and equipment will be decommissioned, decontaminated (as necessary), dismantled, and either salvaged or disposed of in an appropriate on-site or off-site disposal facility.

All wells, including dewatering and production wells, monitoring wells, and any other wells within the Project Area used for the collection of hydrologic or water quality data or incidental monitoring purposes, will be properly abandoned in accordance with NDEQ and DNR requirements.

20.5.3 Tailings Disposal Facility

Since the definition of Solid Waste in Chapter 1 of Title 132 – *Integrated Solid Waste Management Regulations* includes material generated from mining operations, the Tailings Storage Facility (TSF) and the Salt Management Cells at the Project will likely be subject to all or part of the Title 132 regulations, including the closure requirements. The design of the TSF cells allows for concurrent reclamation in order to reduce the amount of precipitation contact water that will require active management. Once a cell of the TSF has reached design capacity, it will be closed. For purposes of closure cost estimating and potential future bonding requirements, this approach will assume that only one cell will be active at any given time for which reclamation (and bonding) may be required. In addition, the approach to TSF construction and material placement will allow the operator to concurrently close portions of each cell as they reach capacity.

The initial closure cover will consist of surface grading and placement of a geomembrane liner over the graded tailings. This liner requires an over-liner drainage system that discharges to the outer slope of the embankment of each TSF cell, and placement of adequate thickness of cover to allow for vegetation; though a root barrier may be necessary to prevent rooting into the tailings. With respect to post-closure requirements, operators of solid waste disposal areas shall provide for post-closure care for a period of at least 30 years. At this time, there is no anticipated post-closure solution/draindown management consideration for the TSF cells given the nature of the tailings materials and the conceptual closure approach. This approach to the closure of the TSF cells is considered conservative and was selected to demonstrate the feasibility and permit ability with respect to the NDEQ landfill regulations and on the advice of the agency. Given the current LOM expectation, additional technologies and/or approaches to equally effective closure options may likely be developed prior to actual reclamation of the site.

The Salt Management Cells will be closed in a manner similar to the TSF.

20.5.4 Closure Cost Estimate

Direct reclamation and closure costs for the Project, including estimates for post-closure monitoring and maintenance, have been estimated at approximately US$ 44.7 million. Including financial assurance premiums for the first five years of operations brings the total to US$ 50.2 million. This conservative approach and estimate consider the fact that 1) none of the facilities are constructed (i.e., final actual configurations are unknown), 2) costs for materials and services are difficult to predict 30 years in advance, and 3) no trade-off studies or final risk assessments have been performed on the closure approach (normally done later in the LOM).

20.6 International Standards and Guidelines

The United States is a Designated Country with respect to the Equator Principles. Designated Countries are those countries deemed to have robust environmental and social governance, legislation systems, and institutional capacity designed to protect their people and the natural environment (Equator Principles Association, 2011).

The current release of the Equator Principles (EP III), launched on June 4th, 2013, covers more projects and streamlines the process to focus on legal compliance in Designated Countries. The reworded Principle 3, states that: “the Assessment process should, in the first instance, address compliance with relevant host country laws, regulations and permits that pertain to environmental and social issues.” This is in an effort to streamline assessments, especially in Designated Countries whose laws already meet the requirements of environmental and/or social assessments (Principle 2), management systems and plans (Principle 4), stakeholder engagement (Principle 5) and, grievance mechanisms (Principle 6). In this case, an evaluation of compliance with host country laws is considered sufficient. Elsewhere (non-Designated Countries), compliance with the applicable International Financing Corporation (IFC) Performance Standards (updated in January 2012) and the World Bank Group Environmental, Health and Safety Guidelines are generally required. Other changes in EP III included:

- public disclosure of the Environmental and Social Impact Assessment for the Project, or at a minimum, it's summary statement;
- analysis of alternatives to address greenhouse gas reductions;
- in non-designated countries, consideration of the new IFC Performance Standards around labour standards, and occupational health and safety diligence requirements in relation to primary supply chain employees and contracted workers, and possibly human rights due diligence in limited “high-risk circumstances.” The new IFC Performance Standards also require Informed Consultation and Participation of affected peoples, or even, Free Prior and Informed Consent for certain projects. This latter is stronger than most of the legally required consultation frameworks around the world.

At this stage, the Project is in compliance with EP III.

21. CAPITAL AND OPERATING COSTS

Capital and operating cost estimates were prepared by Nordmin, SRK, Tetra Tech, Adrian Brown Consultants, SMH, Optimize Group, Zachry Engineering Corporation and Metallurgy Concept Solutions with contributions from NioCorp.

21.1 Capital Cost Estimate

21.1.1 Basis of Estimate

The estimate meets the classification standard for a Class 3 estimate as defined by AACE international and has an intended accuracy of ± 15%. The estimate is reported in Q1 2019 U.S. constant dollars.

The capital cost estimate reflects a detailed bottom-up approach that is based on key engineering deliverables that define the Project scope. This scope was described and quantified within material take-offs (MTO's) in a series of line items. Capital costs are divided among the areas of underground mining, processing, infrastructure, water management, tailings management, mining indirects and contingency. Sustaining capital costs are related to underground mining fixed equipment and development, process plant, infrastructure maintenance, tailings management, mine closure and contingency.

21.1.1.1 Mining, Process, and Infrastructure Capital Costs

The mining Capital Costs were developed, including a combination of vendor and contractor quotations, first principles buildup, allowances, and historical database costs. The estimates include labour, materials, fixed equipment purchase and operation cost, rental equipment, supplies, freight, and energy. The costs developed include direct and indirect costs and included separate contingencies on both. Fixed equipment purchase costs include freight, an allowance for transporting underground, initial training and commissioning.

21.1.1.2 Tailings and Tailings Water Management Capital Costs

The capital cost for tailings facility construction was based on contractor estimates for earthworks and liner installation. A local equipment supplier quoted equipment for loading, hauling and placement of the tailings. SRK developed some costs internally for items where no quotes were obtained. The SRK estimates were developed from recent and relevant costs on other projects or developed from first principles. Approximately 10% of the tailings facilities costs were from SRK estimates.

21.2 Capital Cost Summary

Table 21-1 shows the breakout in LOM initial and sustaining capital estimates, which total US$ 1,609 million. An overall 9.67% contingency factor has been applied to the initial capital estimate, while a smaller 6.25% contingency was applied to the sustaining capital estimate. The pre-production period is defined from April 2019 to the end of construction in June 2022 plus a six-month ramp-up period through the end of December 2022. Commercial production is then to be declared on January 1, 2023. The initial capital estimate of US$ 1,143 million will be partially offset by a Gross Pre-production Revenue Credit of US$ 265 million, (generated by pre-production product sales) which equates to a net cost of US$ 879 million.

Table 21-1: Capital Costs Summary (US$ 000's)

Description	Initial	Sustaining	Total
Capitalized Pre-production Expenses	82,531	-	82,531
Site Preparation and Infrastructure	40,569	15,007	55,576
Processing Plant	367,439	96,448	463,886
Water Management & Treatment	73,756	23,613	97,369
Mining Infrastructure	256,731	180,438	437,170
Tailings Management	21,423	78,855	100,277
Site Wide Indirects	7,368		7,368
Processing Indirects	96,028		96,028
Mining Indirects	39,766		39,766
Process Commissioning	13,350		13,350
Mining Commissioning	1,444		1,444
Owner's Costs	33,619		33,619
Mine Water Management Indirects	8,520		8,520
Closure and Reclamation		44,267	44,267
Subtotal	1,042,542	438,628	1,481,171
Contingency	100,797	27,429	128,227
Total Capital Costs	**1,143,340**	**466,058**	**1,609,397**
Pre-production Revenue Credit	(264,747)		(264,747)
Net Project Total	**878,593**	**466,058**	**1,344,651**

Source: Nordmin, 2019

21.2.1 Capitalized Pre-production Costs

Pre-production costs are defined as production operating expenses that are incurred in the pre-production period before the declaration of Commercial Production phase.

For this study, the NioCorp finance team directed the Project team to categorize these costs as capital for taxation purposes which, per US federal tax rules, 70% of the annual cost can be expensed in the year incurred with the remaining 30% of cost amortized over next five years.

Table 21-2 shows the breakout between the different operating production costs incurred from April 2019 with the start of mine development activities and throughout the nine-month commissioning and ramp-up period from March 1, 2022, and December 31, 2022.

Consequently, all production operating costs incurred after the planned declaration of Commercial Production on January 1, 2023, are 100% expensed in the year incurred and fully deductible for taxation purposes.

Table 21-2: Capitalized Pre-production Cost Summary

Description	US$ 000's
Mining	23,926
Processing	45,880
Other Infrastructure	2,351
Water Management	8,647
Tailings Management	1,727
Total	82,531

Source: Nordmin, 2019

21.2.2 Mining Capital Costs

Mining capital costs primarily comprise the following areas: shaft sinking, lateral mine development, and stationary/fixed mine infrastructure. Nordmin has assumed that a shaft sinking and mine development/production contractor would be operating at the site from the beginning of the project to the end of mine life. The mine contractor would be responsible for sinking both shafts concurrently, developing the underground drifts, including the internal ramp, footwall and hanging wall access drifts, other underground mine infrastructure, the ventilation system and full production activities. The contractor would also develop all internal vertical development (ventilation raises, ore and waste passes). The Mining Capital costs were divided between direct and indirect costs.

The direct mining capital cost contribution is summarized in Table 21-3.

Table 21-3: Initial Direct Mine Capital Cost Estimate

Category	US$ 000's
Surface Infrastructure	114,154
Shaft and Structure	53,817
Underground Development	52,911
Underground Other	31,902
Spares	3,947
Subtotal Category	**256,731**
Contingency	24,952
Total Category	**281,683**

Source: Nordmin, 2019

A further breakdown of each category is summarized as follows:

- The Surface Infrastructure includes:
    - Production shaft permanent hoist house
    - Production shaft permanent headframe/collarhouse/bins

- Ventilation shaft permanent hoist house
- Ventilation shaft permanent headframe/collarhouse
- Permanent surface mine ventilation systems
- Surface material handling
- Temporary generator farm
- Shaft sinking freeze plant
- Backfill plant
- Mine dry/offices/warehouse
- Mine electrical substations
- Surface services
- Surface site work

- The Shafts and Structures include:
  - Shaft geotechnical drilling
  - Shaft sinking setup at the production shaft and ventilation shaft
  - Temporary shaft sinking facilities (both shafts)
  - Shaft sinking for both the production shaft and ventilation shaft
- Underground Development includes vertical and lateral development during the pre-production phase of the mine.
- Underground Other includes underground mine ventilation, dewatering, material handling, garage/shops, and services.
- Spares includes all capital spare parts.

The contingency is based on a line item analysis by category and averages 9.87% for the mine infrastructure capital. No contingency is included on sustaining capital. Table 21-4 shows the contingency by category.

Table 21-4: Initial Direct Mine Capital Cost Contingency Estimate

Category	Percent
Surface Infrastructure	9.87%
Underground Development/Other	9.93%
Shafts and Structures	9.78%

Source: Nordmin, 2019

Indirect Cost

The indirect mining cost is summarized in Table 21-5. A contingency of 10.0% was applied to the indirect cost. The indirect costs include detailed engineering, testing programs, EPCM, per diem, temporary power generation and distribution and energy costs.

Table 21-5: Mining Indirect Cost

Category	US$ 000's
Contractor Indirects	16,047
Owner Indirects	18,386
Diamond Drill Program	5,333
Pre-production Opex	23,926
Commissioning	1,444
Subtotal Mining	**65,135**
Contingency	6,513
Total Mining Indirect	**71,649**

Source: Nordmin, 2019

21.2.3 Processing Plant Capital Costs

The surface processing plant capital summarized in Table 21-1 is further broken down in Table 21-6.

Table 21-6: Process Plant Costs Summary

Item	US$ 000's
Mineral Processing	24,871
Hydromet	243,700
Pyromet	22,341
Acid Plant	76,526
Total	**367,439**

Source: Tetra Tech, 2019

Each category includes the following:

- Building, including foundation, structural steel, roofing, envelope, louvres, doors, elevated floors, control room, offices & electrical rooms, overhead cranes, etc.
- Building services, including ventilation, heating, plumbing, natural gas distribution, compressed air, etc.
- Mechanical equipment
- Chutes
- Dust collection equipment, including ducting
- Process piping
- Utility piping
- Protective coating on equipment & piping where applicable
- Electrical work

- Instrumentation & control

21.2.3.1 Processing Indirects

The processing indirects capital summarized in Table 21-1 is further broken down in Table 21-7.

Table 21-7: Processing Indirect Costs Summary

Item	US$ 000's
Detailed Engineering, Procurement & Construction Management (EPCM)	62,592
Other Professional Services Temporary Services	6,471
Construction Management Facilities other than EPC	439
Worker's Lodging, Meals & Incidentals (Per Diem)	19,016
Early Operations & Construction Energy	269
Inventory and First Fills	6,233
Capital Spares	1,007
Total	**96,028**

Source: Tetra Tech, 2019

Each category includes:

- Detailed Engineering, Procurement & Construction Management (EPCM):
  - Detailed process engineering
  - EPCM contractor fee and expenses
  - EPC contractors' fees and expenses
- Other Professional Services:
  - Hydromet process testing program
  - Software programming
  - Supplementary Geotechnical study
  - Surveying and quantity control
  - Quality control of fabrications
- Construction Management Facilities other than EPC:
  - Rental and installation of modular trailer offices
  - Office consumables
- Worker's Lodging, Meals & Incidentals (Per Diem):
  - Construction workers: 90% of workers will come from outside the region and receive the Per Diem
  - Construction management and supervision personnel: all personnel will receive the Per Diem

- Early Operations & Construction Energy:
  - Maintenance of temporary power distribution system
  - Operation and maintenance of the Water Treatment Plant, including chemical products
- Inventory and First Fills:
  - Two weeks consumption of reagents
  - Diesel and Fuel Gas tanks filled
  - Wear and tear, consumables store items, one set of each
  - Maintenance supplies and repair parts estimated at 50% of annual cost
  - Lubricant estimated at 25% of annual cost
- Capital Spares: High-Pressure Grinding Rolls

21.2.3.2 Process Commissioning

Process commissioning totals US$ 13.35 million and include the following:

- Pre-commissioning includes pre-operational verifications by contractors, vendors, and specialists, beginning three months before commissioning.
- Commissioning:
  - Commissioning by Operations personnel
  - Assistance from vendors
  - Commissioning spares and consumables
  - Assistance from engineering
  - Assistance from contractors.

21.2.4 Tailings Water Management and Salt Management Cells

Basis

Capital cost for tailings and salt management facility construction was based on contractor estimates for earthworks and liner installation. A local equipment supplier quoted equipment for loading, hauling and placement of the tailings. SRK developed some cost internally for items where no quotes were obtained. The SRK estimates were developed from recent and relevant costs on other projects or developed from first principles. Approximately 10% of the tailing's facilities costs were from SRK estimates.

Initial Capital

Tailings Plant Site Cell 1 is located directly east of the process plant site. Salt Management Cell 1 is located west of the process plant site. Construction for this cell will occur during the summer and fall before the plant goes into production. Plant Site Cell 1 construction will consist of approximately 465,000 m^3 of cut to fill earthworks and 111,000 m^2 of geomembrane installation for the tailings facility and associated stormwater pond. The estimated costs, including earthworks, project management, piping and geosynthetic costs, and stormwater management, are summarized in Table 21-10.

Salt Management Cell 1 will be constructed early in the construction schedule in order to be ready to receive salt from Water Treatment operations once they commence and to support hydrogeological testing. This cell will consist of approximately 446,400 m^3 of cut to fill earthworks and 120,800 m^2 of geomembrane installation. A small salt holding facility will be constructed adjacent to the Water Treatment Plant to temporarily store salt in advance of hauling the salt to the Salt Management Cell. The estimated costs, including earthworks, project management, piping and geosynthetic costs, are summarized in Table 21-8.

Table 21-8: Pre-production Tailings and Salt Management Facility Construction Cost

Item	US$ 000
Salt Storage Facility	416
Pre-production Salt Haulage	313
Salt Management Cell	3,6,36
Subtotal	**4,365**
Contingency	435
Total	**4,800**

Source: SRK, 2019

In addition, the Project will haul and deposit both tailings and salt. Tailings and salt will be loaded from their respective storage areas with a front-end loader and hauled by articulating trucks to the tailings and salt management cells. Both the tailings and salt will be spread in thin lifts with a mid-size dozer and compacted with a soil compactor. The cost of this equipment, based on budgetary pricing from a local equipment supplier, is shown in Table 21-9.

Table 21-9: Tailings and Salt Placement Equipment Pre-Production Capital

Item	Unit Cost US$ 000	Number of Units	Total Cost US$ 000
Cat 980M Loader	558	2	1,116
Cat D6TXL dozer w/ ripper	418	1	418
Cat 815K compactor	582	1	582
Ledwell LW2000 gal water truck	108	1	108
Magnum MTL4060K Light Plant	10	1	10
Cat 730C2 Articulating Truck	453	2	907
Subtotal			**3,140**
Contingency			157
Total			**3,297**

Source: SRK, 2019

Capital Contingency

Capital contingency was assigned, with some exceptions, on the following assumptions

- 5% for equipment with quotes from supplier;
- 10% for earthwork and liner with contractor quotes; and
- 15% for items estimated from similar projects.

21.2.4.1 Temporary Waste Rock Storage Facility

The Temporary Waste Rock Storage Facility construction will be very similar to the construction of the tailings cell. Costs for similar activities from the tailings cell were applied to the Temporary Waste Rock Storage Facility. Contingencies were also estimated for this construction as they were for the tailings cell.

The storage facility construction will consist of approximately 16,600 m^3 of cut to fill earthworks and 69,900 m^2 of geomembrane installation. The estimated costs for this facility, broken out by major components, are shown in Table 21-10. Construction of the Temporary Waste Rock Storage Facility is scheduled for 2019/20.

Table 21-10: Pre-production Temporary Waste Rock Storage Facility Construction Cost

Item	US$ 000
Project Management	150
Access Road/Pipeline Corridor	161
Site Preparation	557
Earthworks	258
Geosynthetics	1,376
Overliner & Drains	567
Subtotal	**3,070**
Contingency	290
Total	**3,360**

Source: SRK, 2019

21.2.5 Water Management and Infrastructure

Water Management and Infrastructure include the costs to construct a Water Treatment Plant which will treat mine water, process wastewater, cooling water blowdown and fresh water to supply the facility with its operational water needs as well as produce a solid salt that will be impounded on site. Veolia provided the capital cost for the Water Treatment Plant on a Design-Build-Operate (DBO) basis, inclusive of commissioning, indirect and contingency costs.

Water Management and Infrastructure also include the costs for a series of hydrogeologic investigations and the costs for supplying additional water to the facility from two local landowners and the Tecumseh Board of Public Works. These costs are detailed in Table 21-11.

Table 21-11: Water Management and Infrastructure Cost

Item	US$ 000
Hydrogeology Investigation	6,050
Water Treatment Plant	64,730
Water Supply	2,976
Subtotal	**73,756**
Contingency	446
Total	**74,202**

Source: NioCorp, 2019

21.2.6 Site Preparation and Infrastructure Capital Costs

The site preparation and infrastructure capital summarized in Table 21-1 is further broken down in Table 21-12.

Table 21-12: Site Preparation and Infrastructure Costs Summary

Item	US$ 000's
Site Preparation	18,495
On-site Infrastructure	15,265
Auxiliary buildings	6,146
Surface Mobile Equipment Fleet	664
Total	**40,569**

Source: Tetra Tech, 2019

Summary of the general items of each category:

Site Preparation

- Site clearing and grubbing
- Topsoil removal and berm construction
- Site grading, pad preparation & access way
- Site roads and parking infrastructure
- Site fencing and access gates
- Construction silt fencing/control, stormwater sediment retention pond
- Architectural landscaping at the main entrance

On-site Infrastructure

- Electrical main substation
- Electrical main power distribution
- Natural gas distribution to site loads

- Surface fuel storage and delivery system
- Firewater distribution system
- Wastewater network
- Potable water distribution
- Water treatment plant feeder line from the water pipeline
- Stormwater drainage
- Tailings conveyors
- Tailings impoundment facility water recovery system
- Process control, telecommunications, IT, CCTV
- Truck scale at the main gate
- Site lighting on poles
- Waste storage

Auxiliary Buildings

- Gatehouse (leased estimate includes furniture and equipment only)
- Administration and service building (leased estimate includes furniture and equipment only)
- Process analysis laboratory
- Maintenance shop and warehouse building
- Processing plant modular office trailers (leased estimate includes furniture and equipment only)
- Maintenance shop modular office trailers (leased estimate includes furniture and equipment only)
- Mine change house

Surface Mobile Equipment Fleet

- Carry Deck Crane (5T)
- Weld Truck Ford (1T) 4WD
- Ambulance and fire services will be supplied from local municipalities
- Mine Rescue Vehicle
- Mine Rescue Trailer
- Snow Removal Plow blade for Dump Trucks
- Pick-up trucks and service cars will be leased

21.2.6.1 Site Wide Indirects

The site-wide indirects capital summarized in Table 21-1 is further broken down in Table 21-13.

Table 21-13: Site Wide Indirect Costs Summary

Item	US$ 000's	
Temporary Works	3,192	
Temporary Services	4,176	
Total	**7,368**	

Source: Tetra Tech, 2019

Each category includes the following:

Temporary Works

- Construction and silt fencing
- Environmental protection
- Main construction parking for processing plants
- Secondary construction parking for processing plants
- Mining area construction parking
- Contractors' trailer park,
- Laydown area
- Temporary access gate
- Temporary gravel roads
- Temporary electrical power & lighting
- Sanitary installations
- Communications
- Removal of all temporary facilities

Temporary Services

- Site security
- Snow removal
- Grading of parking, roads and lay down areas
- Dust abatement
- Solid Waste management during the pre-production period
- Janitorial services
- Potable water and first aid
- Medical and first aid

21.2.7 Owner's Costs

Table 21-14 shows the Owner's cost breakout totalling US$ 33.6 million. No contingency is applied to the Owner's Costs.

Table 21-14: Owner's Costs Summary

Item	US$ 000's
Permitting	655
Land Acquisition	11,675
Electrical Utility Company Cash Down Payment	726
Owner's Team during Project	4,250
Operations Readiness	12,880
Construction Umbrella Insurance	3,253
Other Costs	180
Total	**33,619**

Source: Tetra Tech, 2019

The general items that make up each category are summarized as follows:

- Permitting
- Land acquisition
- Electrical Utility Company prepayment for the construction of the main substation
- Owner's team during Project execution includes the cost of salary and expenses of the Owner's personnel dedicated to Project execution
- Operations readiness includes:
  - Specialized assistance for preparing and monitoring the Operations Readiness plan.
  - External assistance for the hiring of personnel.
  - Relocation of personnel.
  - Training program.
  - Preparation of the operations procedures.
  - Preparation of the operating and maintenance schedules, including programming and data input.
  - Procurement activities to fill stores.
  - ERP software, including system configuration, Block development, HMI graphics, Programming, System architecture drawings, Network drawings and training.
  - Other software, including purchase, licenses, and maintenance.
  - Construction umbrella insurance including General Liability, Employer's and Excess Liability, Worker's Comp, Builder's Risk, Contractor's Pollution Liability, Owner's Protective Professional and Professional Liability.
  - Other Costs includes the environmental monitoring program.

21.2.8 Closure and Reclamation

Closure Cost Basis

The closure cost estimate for the Project was developed using the Standardized Reclamation Cost Estimator (SRCE) (available at www.nvbond.org) and a user-defined cost data file (CDF). The inputs to the CDF were obtained from the following sources:

- Equipment costs have been obtained from Gana Trucking, a local Nebraska contractor. These include all-in operator rates, fuel consumption, consumables, and preventive maintenance.
- The operator rates are included in the equipment hire costs. The labour rates are input separately for non-operator rates only.
- Material costs have been obtained from current quotes, where available.

Plant and Mine Facilities

Facilities and equipment associated with the underground mine and processing plant will be reclaimed as follows:

- Plant site buildings will be decontaminated, the buildings will be demolished, and the debris hauled off-site.
- Stockpile underliners will be removed and hauled to the underground for disposal.
- Ponds no longer in use will have sediment and liners removed and hauled to the underground mine for disposal.
- Residual wastes (solid and/or hazardous), will be hauled to appropriate off-site disposal facilities.
- Groundwater wells will be no longer required at the end of operations and will be plugged and abandoned.
- Underground workings (production shaft and ventilation shaft) will be capped to prevent public access post-closure.
- On-site water pipelines, such as the dewatering pipes, will be removed.
- On-site powerlines within the Project boundary will be removed. The utility company will own the substation and would be responsible for its continued use or demolition, once site operations are complete. The natural gas metering station on site that would be owned by a utility would also be managed in this fashion.
- General disturbances will be covered with growth media if necessary and revegetated.

Tailings and Salt Storage Facilities

The Tailing and Salt Management Facilities consist of a series of separate impoundments (cells) for which the exposed tailings and salt surface will be reclaimed as follows:

- Subgrade preparation (i.e., tailings regrading, assumed to be part of the operational costs).
- Synthetic liner installation.
- Above-liner drainage layer construction (i.e., gravel drain layer).
- Above-liner growth media layer placement.

- Amendment of the growth media layer (scarification and nutrient addition) and placement of sod as an anti-erosion measure.

Reclamation will be carried out concurrently as each of the cells reaches its design capacity.

Post-Closure Monitoring

Monitoring is assumed to continue for 30 years after the end of operations and includes baseline and radiochemical profiles. Monitoring around the tailings and salt management cells will be conducted at three points. Long-term management costs will be limited to the maintenance of a fence around the cells, which is on private property, in perpetuity.

21.2.9 Sustaining Capital Costs

A contingency is included on sustaining capital only on tailings to address construction unknowns.

Mining

The sustaining capital is in the categories of lateral and vertical waste development, mine fixed equipment, and definition drilling/exploration drilling. The sustaining capital captures all costs related to supporting mining activities and applies a percentage of cost towards fixed equipment purchase prices over the life of mine. Table 21-15 presents the sustaining capital for mining.

Table 21-15: Sustaining Capital for Mining (US$ 000's)

	Lateral Waste Development	Vertical Waste Development	Fixed Equipment	Definition/ Exploration Drilling	Subtotal	Contingency	Total
Year 4*	10,471	3,962	1,179	6,667	**22,279**	2,228	**24,506**
Year 5*	13,454	4,906	1,155	6,667	**26,182**	2,618	**28,800**
Year 6*	9,642	1,315	1,204	4,667	**16,827**	1,683	**18,510**
Year 7*	5,664	930	1,180	2,667	**10,442**	1,044	**11,486**
Year 8*	1,766	623	1,169	2,667	**6,224**	622	**6,846**
Year 9*	9,609	930	1,206	2,667	**14,412**	1,441	**15,854**
Year 10 to 19*	20,665	1,539	11,788	13,333	**47,325**	4,733	**52,058**
Year 20 to 29*	0	0	12,110	10,000	**22,110**	2,211	**24,321**
Year 30 to 39*	0	0	11,638	3,000	**14,638**	1,464	**16,102**
Total	**71,271**	**14,205**	**42,629**	**52,333**	**180,438**	**18,044**	**198,482**

Source: Nordmin, 2019
* All years are from project start with Year 0 being the initial project year.

Process and Infrastructure

The sustaining capital for process plants, buildings and infrastructure was estimated using a ratio to the direct cost Capital Costs. This ratio has been set to zero from year one to year five and then ramped up to its maximum value between years six to 30 of the Project life. The ratio is decreasing year 29 and 30 and set at zero for years 31 to 36. Table 21-16 presents the sustaining capital for process and infrastructure.

Table 21-16: Sustaining Capital for Process and Infrastructure (US$)

Sustaining Capital	Applicable Capital Costs	Yr 0-5	Yr 6	Yr 7	Yr 8-28	Yr 29	Yr 30	Yr 30-36
Infrastructure	29,504,000	-	49,000	98,000	147,520	98,000	49,000	-
Buildings	53,476,000	-	178,000	357,000	534,760	357,000	178,000	-
Process Plants	292,274,000	-	1,461,000	2,923,000	4,384,110	2,923,000	1,461,000	-
Capital Costs per year		**-**	**1,688,000**	**3,378,000**	**5,066,390**	**3,378,000**	**1,688,000**	**-**

Source: Tetra Tech, 2019

Tailings, Salt and Tailings Water Management

A new tailings facility (Plant Site Cell 2) will be needed in 2024, and that facility will have to be expanded (Cell 3) in 2031. When Plant Site Cell 3 in nearly full, a separate facility will be constructed in “Area 7” in 2040. The Area 7 facility is designed to handle all the remaining tails for life of the Project. Based on preliminary designs, the construction cost of these facilities is shown in Table 21-17. A second Salt Management Facility will be constructed in 2038 to replace the initial Salt Management facility which will have reached capacity.

Table 21-17: Tailings and Salt Facility Sustaining Capital Cost

Item	Sustaining Capital US$ 000
Tailings Cell 2	14,860
Tailings Cell 3	13,455
Salt Management Cell 2	5,365
Subtotal	**62,455**
Contingency	5,987
Total	**68,441**

Source: SRK, 2019

Equipment to load, haul and place tailings will be replaced over the life of the Project. It was assumed that the mobile equipment would be replaced at 40,000 machine hours, with the articulating trucks and the soil compactor replaced at 30,000 machine hours. Light plants are replaced at 10,000 hours. Area 7 Tailings and Salt Management Facility #2 will require hauling tailings on Nebraska highway 50. When the Area 7 tailings facility goes into service, a contractor will be used to haul the tailings and salt using over the road trucks. The Project will load the contractor's trucks and be responsible for spreading and compacting the tailings and the salt. Table 21-18 shows the estimated LOM replacement cost of the tailings placement of mobile equipment.

Table 21-18: Tailings Placement Equipment LOM Replacement Capital

Item	Unit Cost US$ 000	Number of Units	Total Cost US$ 000
Cat 980M Loader	558	2	1,116
Cat D6TXL Dozer w/ Ripper	418	1	418
Cat 815K Compactor	582	1	582
Ledwell LW2000 Gal Water Truck	108	1	108
Magnum MTL4060K Light Plant	10	1	10
Cat 730C2 Articulating Truck	453	2	907
Subtotal			**3,140**
Contingency			157
Total			**3,297**

Source: SRK, 2019

21.2.10 Contingency

The contingency is based on a line item analysis by category and averages 10.33% for the capital. Table 21-19 shows the contingency by category for the initial capital.

Table 21-19: Initial Capital Contingency Summary

Capitalized Pre-production Expenses	Percent (%)	Total US$ 000's
Site Preparation and Infrastructure	10.6%	4,298
Processing Plant	13.9%	50,912
Mine Water Management & Infra	0.6%	446
Mining Infrastructure	9.7%	24,952
Tailings Management	9.2%	1,964
Site Wide Indirects	14.6%	1,075
Processing Indirects	8.8%	8,481
Mining Indirects	10.0%	6,369
Process Commissioning	16.2%	2,157
Mining Commissioning	10.0%	144
Owner's Costs	**0%**	**0**
Total Contingency	**10.33%**	**100,797**

Source: Nordmin, 2019

21.3 Operating Cost Estimate

21.3.1 Basis of Estimate

Operating cost estimates were developed to show monthly and annual costs for production. All unit costs are expressed as US$/tonne processed and are based on Q1 2019 US$.

21.3.1.1 Mining Operating Costs

The development and operation of the underground mine will be carried out by mining contractors. Mine operating costs were developed from first principles. Input from mining contractors, blasting suppliers and equipment vendors, was considered for the key parameters and contractor unit rates. The rates developed from first principles were adjusted based on benchmarking and the experience and judgment of Nordmin's team in collaboration with mining contractors. Vendor quotations for high use materials were obtained that included freight. Productivity information was developed based on first principles for the mining tasks. The required labour was developed based on mine plan requirements for equipment and material movements. The mine plan quantities also dictated the material quantities required and unit pricing based on vendor quotes were applied to determine material costs. Maintenance supplies and labour for fixed equipment, as well as management and technical personnel, were included in the mining cost. A unit cost for backfill was developed and included in the mine operating cost. The costs vary by year based on production requirements. Haulage distance was taken into consideration on haulage costs. A contingency was applied to mine operating costs.

21.3.1.2 Process Plants Operating Costs

The operation of the surface plants will be carried out by the mine owner. The annual process operating costs were determined by estimating the required quantities of workforce, natural gas, electrical power, reagents, and consumables required for one year and applying current unit cost rates to develop an annual operating cost for each area.

21.3.1.3 Tailings and Tailings Water Management Operating Costs

Basis of the tailings operating cost includes the following cost items supplied by the client:

- The hourly wage rate for the equipment operator and truck driver: US$ 23.29/hour
- Labor burden: 35.91%
- Dyed diesel fuel: US$ 2.09/gallon

The equipment operating cost was developed from Infomine Costmine (2016)

The tailings and salt operating cost for haulage from the plant site to Area 7 TSF and Salt Cell #2 are based on quotes provided by Gana Construction.

21.3.1.4 Site G&A Operating Costs

The annual Site G&A operating costs were determined by estimating the required quantities of workforce and using allowances for fixed costs such as consumables, supplies, etc., based on SRK's experience with analogous projects and discussions with NioCorp Project team members.

21.3.1.5 Owner's Costs Capital Costs

The Owner's Capital Costs were estimated mainly from allowances based on Tetra Tech's experience with analogous projects and discussions with NioCorp Project team members.

21.3.1.6 Water Supply Operating Costs

The operating cost is based on discussions between NioCorp and area landowners as well as a cost estimate provided by the Tecumseh Board of Public Works.

21.3.1.7 Closure and Reclamation

The closure cost estimate for the Project was developed using the Standardized Reclamation Cost Estimator (SRCE) (available at www.nvbond.org) and a user-defined cost data file (CDF). The inputs to the CDF were obtained from the following sources:

Equipment costs have been obtained from Gana Trucking. These include all-in operator rates, fuel consumption, consumables, and preventive maintenance.

Operator rates are included in the equipment hire costs. The labour rates are input separately for non-operator rates, only.

Material costs have been obtained from current quotes, where available.

21.3.2 Operating Cost Summary

Table 21-20 summarizes the operating costs estimate by area, which equals US$ 196.41/t ore. These unit rates are stated on a LOM basis where the costs are estimated from the beginning of construction to the end of mine life. LOM operating costs include the pre-production and first/last years of production.

Table 21-20: LOM Operating Cost Unit Rate Summary

Description	LOM US$/t ore
Mining Cost	43.04
Process Cost	106.70
Water Management Cost	16.78
Tailings Management Cost	1.99
Other Infrastructure	5.47
Site G&A Cost	8.29
Other Expenses	6.30
Subtotal	**188.56**
Royalties/Annual Bond Premium	7.84
Total LOM Operating Costs	**196.41**

Source: Nordmin, 2019

21.3.2.1 Mining Operating Costs

Mine operating costs for the LOM, (pre-production and steady state) are US$ 43.04 /t of ore produced. Table 21-21 summarizes the breakdown by mine production activities and shows the general services costs and labour that are allocated over all the tonnage produced. These costs include the cost to drill, blast, install ground support, shotcrete, grouting, load and haul, crush and handle materials to the surface, ventilation, pumping, general maintenance, technical services, backfill, and mine management. The operating cost varies by year, by mine location and production. The annual operating cost varies by year but averages approximately US$ 44 million per year over the LOM. The mining operating cost is based on a Q1 2019 cost basis.

Table 21-21: Steady State Mining Operating Unit Cost (after pre-production)

Description	Steady State (US$ 000's)	Cost per Tonne Ore
Production Drill, Blast, Backfill	454,430	12.80
Development	164,562	4.59
Trucking and Hauling	282,988	7.90
Power	177,681	4.96
Underground Services and G&A	334,618	9.34
Subtotal Operating Cost	**1,418,279**	**39.59**
Contingency	141,828	3.96
Total Operating Cost	**1,560,108**	**43.55**

Source: Nordmin, 2019

21.3.2.2 Process Plant Operating Costs

The annual LOM operating costs for the Process and Infrastructure portion of the plant is estimated at US$ 106.70/t of mineralized material processed. This estimate includes four primary areas of the surface plant; Mineral Processing, Hydrometallurgical Plant, Pyrometallurgical Plant and Infrastructure. The estimate for each of these four areas was developed by determining the required quantities of workforce, energy (natural gas, electrical power, and fuel) reagents, consumables and other general costs required for one year of operation and then applying current unit cost and feed rates to develop an annual operating cost for each area. These costs were then used to calculate other valuable metrics, such as dollars-per-ton-milled. Table 21-22 summarizes the costs for each area.

Table 21-22: ROM Processing Operating Cost Unit Rate Breakdown

Cost Items	Annual Cost (2,764 t/d) (US$/y)	Annual Cost Per Tonne Milled (2,764 t/d) (US$/y)
Mineral Processing	**4,416,103**	**4.38**
Workforce	1,480,925	1.47
Energy	1,398,197	1.39
Reagents	0	0.00
Consumables	1,445,243	1.43
Other Processing	91,738	0.09
Hydromet	**85,488,398**	**84.73**
Workforce	5,312,935	5.27
Energy	39,588,483	39.24
Reagents	34,566,368	34.26
Consumables	5,743,396	5.69
Other Processing	277,216	0.27
Pyromet	**17,745,296**	**17.59**
Workforce	1,856,808	0.99
Energy	1,838,105	1.82
Reagents	12,909,042	12.80
Consumables	1,002,734	0.99
Other Processing	138,608	0.14
Infrastructure	**1,805,954**	**1.79**
Water Management	**16,822,949**	**16.67**
Product Packaging	**875,015**	**0.87**
Other	**2,835,046**	**2.81**
Total Process Cost	**129,988,762**	**128.84**

Source: Tetra Tech & Veolia, 2019

The operating costs for this Project are based on processing 2,764 t of ore per day to produce an average of 7,220 t/y of ferroniobium. These operating costs are based on Q1-2019 pricing data.

21.3.2.3 Tailings, Salt and Tailings Water Management Operating Costs

Tailings Operating Costs

During tailings and salt disposal operations, tailings and salt will be loaded from a storage building near the backfill plant and water treatment plant respectively and hauled to the tailing storage or salt impoundment facilities in 30 ton articulating trucks. In order to maximize the density, both the tailings and salt will be dozed into thin lifts and compacted using a soil compactor.

Tailings storage at the backfill plant is limited, so costs for tailings placement were estimated by assuming that the work will be completed on a 10-hour shift, seven days per week. Two crews of operators, consisting of four persons per crew, will alternate on a four days on, four days off schedule. Each crew will consist of two equipment operators and two truck drivers. Half of the year, it was assumed that there would be a full-time water truck driver. One equipment operator will run the front end loader to load the trucks. The second operator will alternate between the dozer and soil compactor at the tailings cell. Alignment of the work schedules for tailings personnel with the balance of mine and plant operational schedules will be evaluated at the detailed design stage. This same crew would also transport the salt from the water treatment storage building to the salt impoundment facilities.

Truck productivity was calculated using Caterpillar Inc.'s Fleet Production and Cast Analysis (FPC) software to determine the haul times required the trucks to place the tailings. Parameters used in determining the haulage requirements to Plant Site Cell 1 are shown in Table 21-23. This analysis shows that two trucks can handle the amount of tailings produced with sufficient capacity in case of equipment downtime.

Table 21-23: Tailings Haulage Calculations

Description	Value	Unit of Measure
Average tailings stacking	1100	t/d
Tailings bulk density	1.6	t/m^3
Haul trucks	Cat 730C2	
Haul truck capacity	28.00	t
Haul truck capacity	17.50	m^3
Estimated load	13.50	m^3
Estimated load	21.60	tonne
Loads/day	51	loads
Trucks operating	2.00	each
Hours per shift	10.00	hr
Loads per truck-shift-hr	2.55	loads/hr
Plant Site Cell 1 haul - one way	1200	m
Potential cycle time (FPC)	9.4	min
Utilization	80%	
Potential 2 truck production	221	t/h

Source: SRK, 2019

Plant Site Cells 2 and 3 have longer haul distance, but the equipment fleet will be able to handle the additional haulage cycle time.

Equipment operating costs were obtained using Costmine, modified for fuel pricing. Operating cost includes the costs associated with major component rebuild. Adjusted equipment hourly costs are shown in Table 21-24.

Table 21-24: Tailings and Salt Mobile Equipment Hourly Operating Cost

Equipment Type	Model	Fuel US$/hr	Lube US$/hr	Tires (US$/hr)	Overhaul (US$/hr)	Maint (US$/hr)	Wear Items (US$/hr)	Total Cost US$/hr
Loader	Cat 980M Loader	19.98	6.64	17.56	7.75	14.40	0.66	66.99
Dozer	Cat D6TXL dozer w/ ripper	11.41	3.58	-	6.15	9.21	9.00	39.35
Compactor	Cat 815K compactor	13.15	4.47	0.34	19.68	16.11	-	53.75
30 Ton Articulating Truck	Cat 730C2 Articulating Truck	10.40	5.32	3.39	4.50	8.34	-	31.94
Skid Steer		4.56	0.78	0.32	0.88	1.66	0.14	8.33
Water Truck 2000 Gallon	Ledwell LW2000 gal water truck	10.68	3.48	2.55	1.10	2.65	-	20.46
Light Plant	Magnum MTL4060K Light Plant	0.70	0.19	0.02	0.27	0.50	-	1.69

Source: Infomine, 2016

Costs were calculated on a period basis. It was assumed that all tailings haulage operators would be paid based on working a full 10-hour shift every day of the period. Equipment utilization factors were assumed for each equipment type. Table 21-25 shows the utilization factors for equipment usage.

Table 21-25: Tailings and Salt Mobile Equipment Utilization

Equipment Type	Utilization
Loader	80%
Dozer	45%
Soil Compactor	40%
Water truck	60%
Articulating trucks	85%
Light plant	15%

Source: SRK, 2019

In Year 19 when tailings and salt disposal move to the Area 7 TSF, highway trucks will be required when hauling tailings and salt to Area 7 and Salt Cell #2 (instead of the articulating trucks used for disposal in the Plant Site TSFs). A quote was received from a contractor to tailings haulage only for US$ 2.55/t tailings. NioCorp will be responsible for loading the contractor's trucks and for spreading and compacting the tailings.

Estimated costs for tailings loading, haulage and placement are shown in Table 21-26.

Table 21-26: Cost for Tailings and Salt Placement

Haulage Type	Cost, US$ /t
Owner hauled tailings and salt	3.00
Contractor hauled tailings and salt	4.38

Source: SRK, 2019

21.3.2.4 Site G&A Operating Costs

SRK estimated LOM Site General and Administrative (Site G&A) operating costs for the Project from first principles. The results are presented in Table 21-27, which shows a LOM unit rate of US$ 8.29/t ore (versus a US$ 8.37/t ore on an ROM basis). On an annual basis, Site G&A costs average US$ 8.4 million, as shown in Table 21-28.

Table 21-27: LOM Site G&A Operating Costs

Description	US$ 000's	US$ /t
Site Management	43,092	1.19
Processing Overhead	73,832	2.03
Technical Services	62,560	1.72
Health & Safety	19,646	0.54
Human Resources	14,931	0.41
Supply Chain Management	21,778	0.60
Information Services	13,978	0.38
Finance	15,507	0.43
Community and Social Responsibility	3,210	0.09
Environmental and Permitting	22,668	0.62
Site Services	9,901	0.27
Total	**301,103**	**8.29**

Source: Nordmin, 2019

Table 21-28: Site G&A Annual Operating Costs

Description	US$ 000's
Site Management	
Salaries and Wages	268
Office Supplies	60
Rents/Premiums/Travel	880
Subtotal Site Management	1,208
Processing Overhead	
Salaries and Wages	2,070
Materials, Supplies, Consumables, Training	-
Subtotal Processing Overhead	2,070
Technical Services	
Salaries and Wages	1,754
Materials, Supplies, Consumables, Training	-
Subtotal Technical Services	1,754
Health and Safety	
Salaries and Wages	465
Materials, Supplies, Consumables, Training	86
Subtotal Health & Safety	551
Human Resources	
Salaries and Wages	259
Recruitment/Relocation	100
L&D Training Programs	60
Subtotal Human Resources	419
Supply Chain Management	
Salaries and Wages	561
Materials, Supplies, Consumables	10
Contract Services/Head Office Support	40
Subtotal SCM	611
Information Services	
Salaries and Wages	235
IT Equipment/Licenses	157
Subtotal Information Services	392
Finance	
Salaries and Wages	292
Contract Services/Head Office Support	143
Subtotal Finance	435
Community and Social Responsibility	
Salaries and Wages	-
Marketing Supplies	15
Community Funding	75
Subtotal CSR	90
Environmental and Permitting	
Salaries and Wages	536
Operating Costs	50
Contract Services	50
Subtotal E&P	636
Site Services	
Salaries and Wages	258
Site Facilities Maintenance	20
Subtotal Site Services	278
Grand Total	**8,442**
Total Salaries and Wages	6,696
Total Other Fixed Costs	1,747

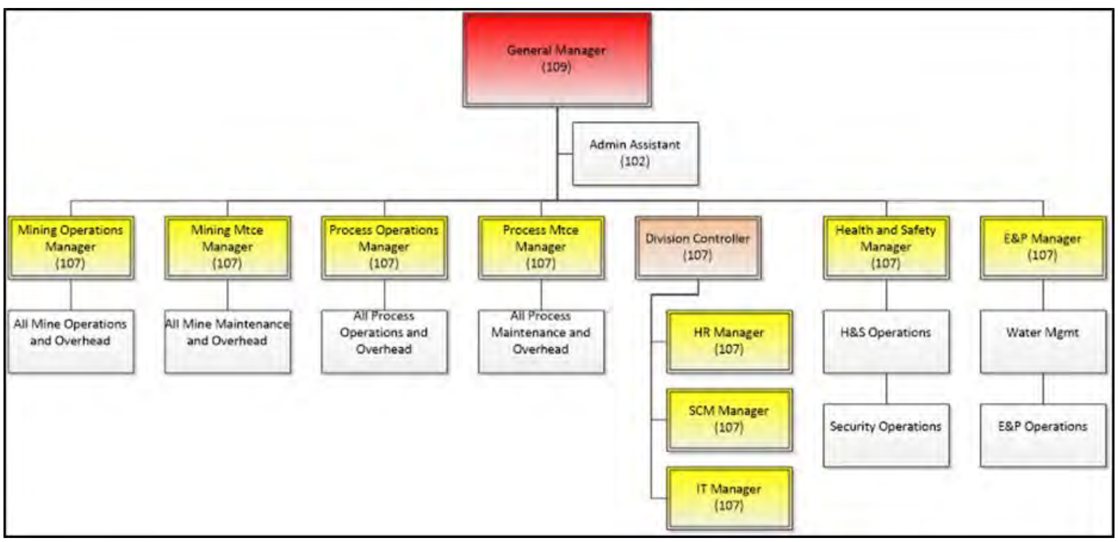
Source: Nordmin, 2019

The Site G&A labour costs, as shown in Table 21-28 of US$ 6.7 million per year are derived by the 80.5 staff headcount, as shown in Table 21-29. The overall organizational chart for the entire operation during LOM is shown in Figure 21-1. The Site G&A numbers do not include any mining or any direct processing operations or maintenance staff. However, both Processing Overhead and Technical Services that roll up in the Site G&A area are grouped within mining and processing areas in the organizational chart for simplicity.

Table 21-29: Average Annual G&A Headcount during Operations

Description	Headcount
Site Management	2.5
Processing Overhead	21
Technical Services	23
Health & Safety	6
Human Resources	3
Supply Chain Management	9
Information Services	2
Finance	3
Environmental and Permitting	6
Site Services	5
Total G&A Personal	**80.5**

Source: Nordmin, 2019



Source: Nordmin, 2019

Figure 21-1: Elk Creek Project LOM organizational chart.

In terms of fixed costs, allowances were developed for each category based on experience from similar US mining projects (see Table 21-30). The average annual Site G&A fixed cost (non-labour) estimate is US$ 1.7 million per year. The Project is somewhat unique that although it is located in a rural area in southeastern Nebraska, it is only one hour drive from the city of Omaha, an eight-minute drive to a 40-bed county hospital, and has a high capacity fiber optic trunk bypassing the property on the adjacent county road.

Table 21-30: Average Annual G&A Fixed Costs During Operations

Description	Whole US$
Site Management	
Materials, Supplies, Consumables	
Office Supplies	30,000
Postage, Courier and Light Freight	10,000
Copying and Printing	20,000
Subtotal Materials, Supplies, Consumables	60,000
Insurance	
Property, Business Interruption, Bldgs, Equip, Liability	750,000
Insurance	750,000
Government Surcharges/Fees(1)	
County	10,000
Other	5,000
Subtotal Property Taxes Government Surcharges/Fees	15,000
Travel/Professional	
Conferences and Meetings	25,000
Outside Accounts (Small vehicle repair, dining, catering)	20,000
Dues and Subscriptions	20,000
Business Travel & Accommodation	50,000
Subtotal Travel/Professional	115,000
Total Site Management Fixed Costs	**940,000**
Total Processing Overhead Fixed Costs	**-**
Total Technical Services Fixed Costs	**-**
Health and Safety	
Emergency Supplies	
First Aid Stations	10,000
Fire Extinguishers	50,000
Basket/Stretcher	1,250
Subtotal Emergency Supplies	61,250
Training	

Description	Whole US$
Staff Training Contractor	-
Training Supplies/ Classes	25,000
Subtotal Training	25,000
Total H&S Fixed Costs	**86,250**
Human Resources	
Recruitment/Relocation	
Recruitment Allowance	15,000
Recruitment Fees	15,000
Relocation and Assignment Cost	70,000
Subtotal Recruitment/Relocation	100,000
L&D Training Programs	
Staff Training Contractor	25,000
Training Supplies/ Classes	25,000
NG Corporate Head Office Support	10,000
Subtotal Human Resources	60,000
Total HR Fixed Costs	**160,000**
Supply Chain Management	
Subtotal Materials, Supplies, Consumables	10,000
Contract Services - Trucking	10,000
Corporate Head Office Support	30,000
Total SCM Fixed Costs	**50,000**
Information Services	
IT Equipment/Licenses	
IT Equipment	7,355
IT Software (On-site IT Support FTE)	90,000
Private Mobile Radio (PMR)	10,000
Carrier Services	50,000
Total IS Fixed Costs	**157,355**
Finance	
Contract Services/Head Office Support	
Contract Services - Legal	50,000
Contract Services - Consultants (tax, acctg, mgmt)	50,000
Contract Services - External Audits	13,000
NG Corporate Head Office Support	30,000

Description	Whole US$
Total Finance Fixed Costs	**143,000**
Community and Social Responsibility	
Subtotal Materials, Supplies, Consumables	15,000
Community Funding	
Charitable Contributions	50,000
Sponsorships	25,000
Subtotal Community Funding	75,000
Total CSR Fixed Costs	**90,000**
Environmental and Permitting	
Subtotal Materials, Supplies, Consumables	50,000
Contract Services - Annual ESR Studies	50,000
Total E&P Fixed Costs	**100,000**
Site Services	
Subtotal Materials, Supplies, Consumables	20,000
Total Site Services Fixed Costs	**20,000**
Grand Total	**1,746,605**

Source: Nordmin, 2019

(1) Does not include annual county property taxes which are included in operating expenses in the technical economic model as a direct cash cost.

22. ECONOMIC ANALYSIS

22.1 Cautionary Statement

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this Report include, but are not limited to, statements with respect to future niobium, scandium and titanium prices, the estimation of Mineral Resources and Mineral Reserves, the estimated mine production and niobium scandium and titanium recovered, the estimated capital and operating costs, and the estimated cash flows generated from the planned mine production. Actual results may be affected by:

- Unexpected variations in the quantity of ore, grade or recovery rates, or presence of deleterious elements that would affect the process plant or waste disposal
- Unexpected geotechnical and hydrogeological conditions from what was assumed in the mine designs, including water management during construction, mine operations, and post mine closure
- Differences in the timing and amount of estimated future niobium, scandium and titanium production, costs of future niobium, scandium and titanium production, sustaining capital requirements, future operating costs, requirements for additional capital, unexpected failure of plant, equipment or processes not operating as anticipated.
- Changes in government regulation of mining operations, environment, and taxes.
- Unexpected social risks, higher closure costs and unanticipated closure requirements, mineral title disputes or delays to obtaining surface access to the property.

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Report and may result in changes to the calendar timelines presented and the information and statements contained in this Report. No development approval has been forthcoming from the NioCorp Board and statutory permits, including environmental permits, are required to be granted prior to mine commencement.

22.2 Methodology Used

The Project has been evaluated using discounted cash flow analysis. Cash inflows consist of annual revenue projections. Cash outflows consist of initial capital expenditures, sustaining capital costs, operating costs, taxes, royalties, and commitments to other stakeholders. These are subtracted from revenues to arrive at the annual cash flow projections. Cash flows are taken to occur at the end of each period. To reflect the time value of money, annual cash flow projections are discounted back to the Project valuation date using the yearly discount rate. The discount rate appropriate to a specific project can depend on many factors, including the type of product, the cost of capital to the Project, and the level of Project risks (i.e. market risk, environmental risk, technical risk and political risk) in comparison to the expected return from the equity and money markets. The base case discount rate for the 2019 Feasibility Study is 8%. The discounted present values of the cash flows are summed to arrive at the Project's NPV. In addition to the NPV, the IRR and the payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. The payback period is calculated as the time required to achieve positive cumulative cash flow for the Project from the start of production.

22.3 Financial Model Parameters and Assumptions

The indicative economic results summarized in this section are based upon work performed by Nordmin and NioCorp in 2019. They have been prepared on both a periodic monthly/quarterly format and an annual format. The metrics reported in this volume are based on the annual cash flow model results. The metrics are on both a pre-tax and after-tax basis; a 100% equity basis with no Project financing inputs; and are in Q1 2019 U.S. constant dollars.

Key criteria used in the analysis are discussed in detail throughout this section. Principal Project assumptions used are shown summarized in Table 22-1.

Table 22-1: General Assumptions

Description	Value
Pre-Production Period	4 years
Process Plant Life	36 years
Mine Operating Days per Year	365
Mill Operating Days per Year	365
Discount Rate	EOP @ 8%
Commercial Production Year	2023

Source: Nordmin, 2019

All costs incurred prior to April 2019 are considered sunk with respect to this analysis.

The selected Project discount rate is 8% as directed by NioCorp, and the valuation uses standard end of-period discounting. A sensitivity analysis of the discount rate is discussed later in this section.

Foreign exchange impacts were deemed negligible as most, if not all costs and revenues are denominated in US dollars.

The major criteria adopted to define when the Project enters into Commercial Production include the following: (1) all major capital expenditures to bring the mine to nameplate capacity have been completed; (2) the process plant, and other facilities have been transferred to the control of the Operations team from the Commissioning team; (3) the plant has reached at least 80% of initial design capacity following an adequate ramp-up period; (4) product recoveries are at or near expected levels; (5) the mine has the ability to sustain ongoing production of ore at the required CoG; and (6) costs are under control or within expectations.

<u>Mineral Resource, Mineral Reserve and Mine Life</u>

The Mineral Resource discussed in Section 14 was converted to the Mineral Reserve outlined in Section 15. The estimated Mineral Reserve will support a 36-year production life, using the mine plan as provided in Section 16.

<u>Metallurgical Recoveries</u>

The basis for the process recoveries is included in Section 13, and the process design is outlined in Section 17.

Product Prices

The product price basis is discussed in Section 19.

Capital and Operating Costs

The capital and operating cost estimates are detailed in Section 21.

Closure Costs and Salvage Value

Reclamation costs were included with the capital cost estimate.

Financing

The economic analysis assumes 100% equity financing and is reported on a 100% project ownership basis.

Inflation

The economic analysis assumes constant prices with no inflationary adjustments.

22.3.1 Physicals

Mining

Table 22-2 is a summary of the estimated mine production over the 36-year LOM. Ore mined refers to Probable Mineral Reserves.

Table 22-2: Mining Physicals

Description	Value
Ore Mined (kt)	36,313
Ore Mining Rate (t/d)	2,764
Niobium Grade	0.81%
Scandium Grade (ppm)	65.71
TiO_2 Grade	2.86%
ContainedNb_2O_5 (kt)	293
Contained Sc (t)	2,387
Contained TiO_2 (kt)	1,039

Source: Nordmin, 2019

Processing

A summary of the estimated process plant production for the Project is contained in Table 22-3 for a 36-year operating life at an average capacity of 1.01 Mt/y. Table 22-4 shows more detail of process recovery rates for each product in the three plants. Ore processed refers to Probable Mineral Reserves.

Table 22-3: Processing Physicals

Description	Value
Total Ore Processed (kt)	36,313
Processing Rate (kt/y)	1,009
Average Recovery, Nb	82.4%
Average Recovery Sc	93.1%
Average Recovery TiO_2	40.3%
Recovered Nb_2O_5 (kt)	241
Recovered Sc (t)	2,223
Recovered TiO_2 (kt)	419

Source: NioCorp, 2019

Table 22-4: Processing Recovery Summary

Description	Nb	Ti	Sc
Mineral Processing Plant	100.0%	100.0%	100.0%
Hydrometallurgical Plant	85.8%	40.31%	93.1%
Pyrometallurgical Plant	96.0%		
Overall Recovery	**82.4%**	**40.3%**	**93.1%**

Source: Tetra Tech Memo, 5/15/2017

22.3.2 Revenue

Based on data discussed in Section 19, Table 22-5 and Table 22-6 show benchmark product pricing assumptions used in the economic analysis. The following criteria apply to the calculation of revenue:

- Niobium measured in the resource and reserve as Nb_2O_5 but is produced as commercial ferroniobium, which is a mixture typically containing 65% Nb and 35% Fe. Ferroniobium pricing is based solely on its Nb content.
- TiO_2 is measured as TiO_2 in the resource and reserve and is produced as that same compound.
- Scandium is measured as Sc in the resource and reserve and is produced and sold as the compound Sc_2O_3.

Table 22-5: Pricing Assumptions

Description	Tonnes Saleable Product	LOM Benchmark Price US$/t product
Payable Nb	168,861	46.55
Payable Sc_2O_3	3,410	See Table 22-6
Payable TiO_2	418,841	0.99

Source: NioCorp, 2019

Table 22-6: Scandium Trioxide Pricing Assumptions

Year	US$/kg
2019	3,600
2020	3,700
2021	3,800
2022	3,900
2023	4,000
2024	3,500
2025	3,000
2026	3,100
2027	3,200
2028	3,400
2029	3,600
2030+	3,750

Source: OnG, 2019

The following is a breakdown of netback pricing assumptions for each product:

<u>Niobium</u>

- Ferroniobium (65% Nb) product (FeNb product) with constant LOM Benchmark/Provisional Price of US$ 47/kg Nb.
- All settlement Nb prices have a 3.75% discount to the netback price of benchmark price minus Buyers Logistics Costs (BLC) except with customers buying on spot pricing.
- It is assumed that all FeNb product purchases have a 10 Net Days Outstanding (NDO) A/R term. At the time of this report, the Project had two committed offtake customers signed up for 10-year terms with all remaining annual FeNb production sold on a spot basis:
  - Buyer #1 – US-based metals trader with mill operations located in the southern half of the US:
    - 10-year commitment to purchase 25% of annual offtake production to a maximum of 1,875 t/y.
  - Buyer #2 – European-based manufacturer with global mill operations:
    - 10-year commitment to purchase 50% of annual offtake production to a maximum of 3,750 t/y.
  - Spot Buyer - It is assumed that all annual FeNb production not sold under an offtake agreement is sold at spot (or benchmark) pricing of constant US$ 47/kg Nb on an ex-mine gate basis with a 10-day NDO A/R term.

Based on these pricing assumptions, the average realized LOM Nb price is US$ 46.55/kg.

<u>Titanium Dioxide</u>

- No offtake agreements have been negotiated at the time of writing this report.
- It is assumed that all annual TiO_2 production is sold at spot (or benchmark) pricing of constant US$ 0.99/kg on an ex-mine gate basis with a 10-day NDO A/R term.

Scandium Trioxide

- Scandium Trioxide (Sc_2O_3) product with an average realized LOM price of US$ 3,675/kg.
- It is assumed that all Sc_2O_3 product purchases have a 10-day NDO A/R term.

At the time of this report, the Project has one committed offtake customer signed up for a 10-year term with all remaining Scandium Trioxide sold on a spot basis.

- 10-year commitment to purchase a minimum of 12 tonnes per year.
- Under the agreement, the buyer has exclusive rights to the aerospace and sporting goods sectors.

22.3.3 Operating Costs

Operating cost metrics in the technical, economic model are reported on a LOM basis meaning that all of these unit rates are stated on a LOM basis where the costs are estimated from the beginning of construction to the end of mine life. LOM operating costs include the pre-production and first/last years of production.

The total LOM operating cost unit rate of US$ 196.41/t processed is summarized in Table 22-7.

Table 22-7: Operating Cost Summary

Description	US$/t ore
Mining	43.04
Processing	106.70
G&A	8.29
Water Management	16.78
Tailings Management	1.99
Other Infrastructure	5.47
Other Expenses	6.30
Subtotal Operating Costs	**188.56**
Royalties/Bond Premium	7.84
Total All-in Operating Costs	**196.41**

Source: Nordmin, 2019

22.3.4 Capital Costs

Total LOM capital costs totalling US$ 1,565 million, not including US$ 44 million of final closure/reclamation costs are summarized in Table 22-8. Total initial capital costs of US$ 1,143 million, including a 10.3% contingency, are part of this total.

Table 22-8: Capital Cost Summary (US$ 000's)

Description	Initial	Sustaining	Total
Capitalized Pre-production Costs	83	-	83
Site Preparation and Infrastructure	41	15	56
Processing Plant	367	96	464
Water Management & Treatment	74	24	97
Mining Infrastructure	257	198	455
Tailings Management	21	79	100
Site Wide Indirects	7	-	7
Processing Indirects	96	-	96
Mining Indirects	40	-	40
Commissioning	15		15
Owner's Costs	34	-	34
Mine Water Management Indirects	9	-	9
Contingency	101	9	110
Total Capital Costs	**US$ 1,143**	**US$ 422**	**US$ 1,565**

Source: Nordmin 2019

Further detail of the initial capital estimate is presented in Table 22-9 which shows it is partially offset by a Gross Pre-production Revenue Credit of US$ 265 million generated from product sales made in the period from March 1, 2022, to December 31, 2022, which is before the start of commercial production on January 1, 2023.

Table 22-9: Initial Capital Cost Summary

Description	US$ Million	% of Total
Capitalized Pre-Production Costs	83	7%
Process Commissioning	15	1%
Subtotal Pre-production Costs	**97**	**9%**
Site Preparation and Infrastructure	41	4%
Processing Plant	367	32%
Mine Water Management	74	6%
Mining Infrastructure	257	22%
Tailings Management	21	2%
Subtotal Direct Costs	**US$ 760**	**66%**
Site Wide	7	1%
Processing	96	8%
Mining	40	3%
Owner's Costs	34	3%
Mine Water Management	19	1%
Subtotal Indirect Costs	**US$ 185**	**16%**
Project Total Before Contingency	**US$ 1,043**	**91%**
Contingency of 11.1 %	101	9%
Project Total Before PP Revenue Credit	**US$ 1,143**	**100%**
Gross Pre-production Revenue Credit*	(265)	
Project Total*	**US$ 879**	

Source: Nordmin, 2019
*Revenue from sales occurring during commissioning and ramp-up phases

An estimate of US$ 30 million of working capital is estimated for the last year before and the first year of commercial production. The assumptions used for this estimate are as follows:

- Accounts Receivable (A/R) – product /offtake agreement specific (see revenue section)
- Accounts Payable (A/P) – 30-day delay
- Consumable Inventory – 60-day supply

Annual adjustments to working capital levels are made in the technical economic model with all working capital recaptured by the end of LOM resulting in a LOM net free cash flow (FCF) impact of 0.00 US$.

22.4 Cashflow Forecasts and Annual Production Forecasts

Cashflow Forecasts are summarized on a LOM basis in this section.

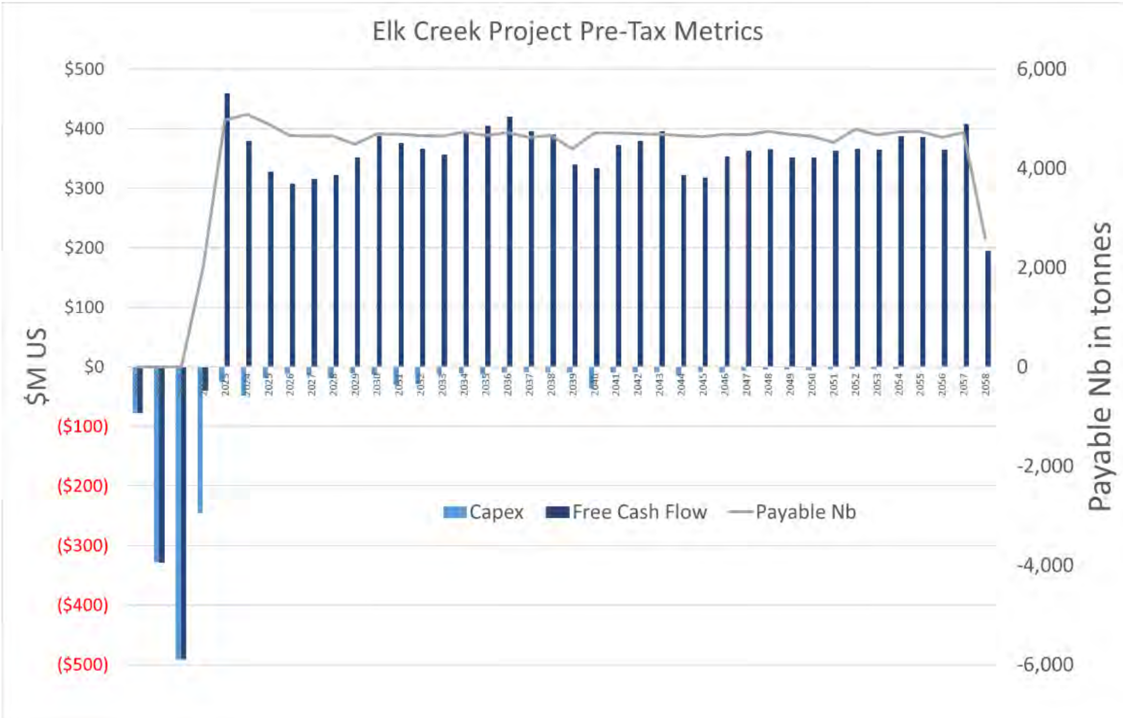
The technical, economic model metrics are prepared on an annual pre-tax and after-tax basis, the results of which are summarized in Table 22-10. Based on current assumptions and design listed in this report, the Project returns a pre-tax NPV 8% of US$ 2,564 million and an IRR of 27.3% along with an after-tax NPV 8% of US$ 2,098 million and IRR of 25.8%.

Figure 22-1 and Figure 22-2 presents annual pre-tax and after-tax free cash flow versus payable Nb production and shows that the Project is expected to generate a stable positive pre-tax free cash flow of US$ 375 to US$ 450 million per year and after-tax free cash flow of US$ 275 to US$ 350 million per year through the LOM.

Table 22-10: Indicative Economic Results

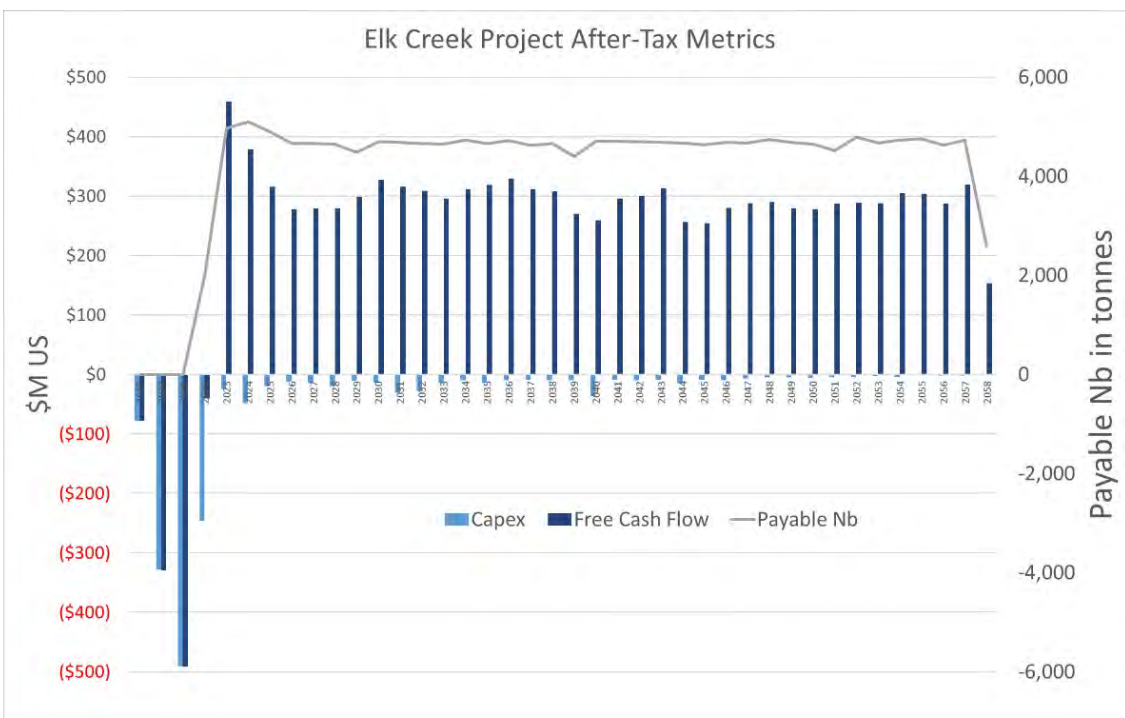
Description	Value
Realized Market Prices	
Nb	US$ 46.55
TiO_2	US$ 0.99
Sc_2O_3	US$ 3,676
Payable Metal	
Nb (t)	168,861
TiO_2 (t)	418,841
Sc_2O_3 (t)	3,410
Total Gross Revenue	**US$ 20,807,083**
Operating Costs	
Mining Cost	(1,562,803)
Process Cost	(3,874,533)
Site G&A Cost	(301,103)
Concentrate Freight Cost	(10,290)
Other Infrastructure Costs	(198,532)
Water Management Cost	(609,195)
Tailings Management Cost	(72,228)
Property Tax	(218,634)
Royalties	(279,224)
Annual Bond Premium	(5,500)
Total Operating Costs	**(US$ 6,717,578)**
Operating Margin (EBITDA)	**US$ 13,675,041**
Effective Tax Rate	17.51%
Income Tax	(2,319,660)
Total Taxes	**(US$ 2,319,660)**
Working Capital	0
Operating Cash Flow	**US$ 11,354,694**
Capital	
Initial Capital	(1,143,340)
Sustaining Capital	(421,791)
Reclamation/Salvage Capital	(44,267)
Total Capital	**(US$ 1,609,397)**
Metrics	
Pre-tax Free Cash Flow	US$ 12,064,956
Pre-tax NPV @ 8%	**US$ 2,564,433**
Pre-tax IRR	**27.3%**
Pre-tax Undiscounted PB from Start of CP (Years)	2.85
After-tax Free Cash Flow	US$ 9,745,296
After-tax NPV @ 8%	**US$ 2,098,167**
After-tax IRR	**25.8%**
After-tax Undiscounted PB from Start of CP (Years)	2.86

Source: Nordmin, 2019



Source: NioCorp, 2019

Figure 22-1: Annual Project Metrics Summary (Pre-Tax)



Source: NioCorp, 2019

Figure 22-2: Annual Project Metrics Summary (After-Tax)

22.5 Taxes, Royalties and Other Interests

Due to the Project's location in a rural area of Nebraska with little industrial activity, taxes and depreciation for the Project were modelled based upon input from NioCorp, as well as a review of various guidelines such as the Nebraska Advantage Act, Nebraska tax credit and Federal tax rates. As such, a detailed tax methodology was developed for the technical, economic model to model the impacts of various government tax incentives.

The assumptions used in the methodology are described in this section and assumptions are as follows:

- Taxes calculation is on an annual basis.
- Corporate Income Tax (CIT) rates are 21% for Federal and 7.81% for Nebraska.
- County property tax based on the end of year value of Project, net of capital improvements and depreciation taken, multiplied by 0.017292. A 10-year tax abatement has also been established based on the Company's successful application for tax benefits under the Nebraska Advantage Act.
- Net Operating Losses (NOL) is carried forward indefinitely and can be used up to 100% of annual positive taxable income per period.
- Federal Depletion allowance is calculated using the co-product percentage depletion method as it was determined that the cost depletion method would be too small compared to the former method. The percentage of depletion rates applied against Gross Income from Mining (subject to 50% of Net Income from Mining limit) are:
  - Nb - 22%
  - TiO_2 – 22%
  - Sc_2O_3 – 14%
- Tax Depreciation allowance is calculated each year by the following methods:
  - Mining Development/Capitalized Pre-production Costs: 70% of cost expensed in the year incurred and remaining 30% amortized over 5 years.
  - Mine Fleet Equipment: 7 year Modified Accelerated Cost Recovery System (MACRS) depreciation starting in the year when the cost is incurred.
  - Plant: 7-year MACRS depreciation starting in the first year of commercial production.
  - Infrastructure: 10-year MACRS depreciation starting in the first year of commercial production.
- Tax credits available to the Project include:
  - Nebraska Investment Tax Credit (ITC) is applied against NE state income tax payable from a beginning balance of US$ 144.1 million based on a formula incorporating development capital spent in area to date.

The calculated effective income tax rate for the Project is 17.51% (CIT Payable/Adjusted EBITDA).

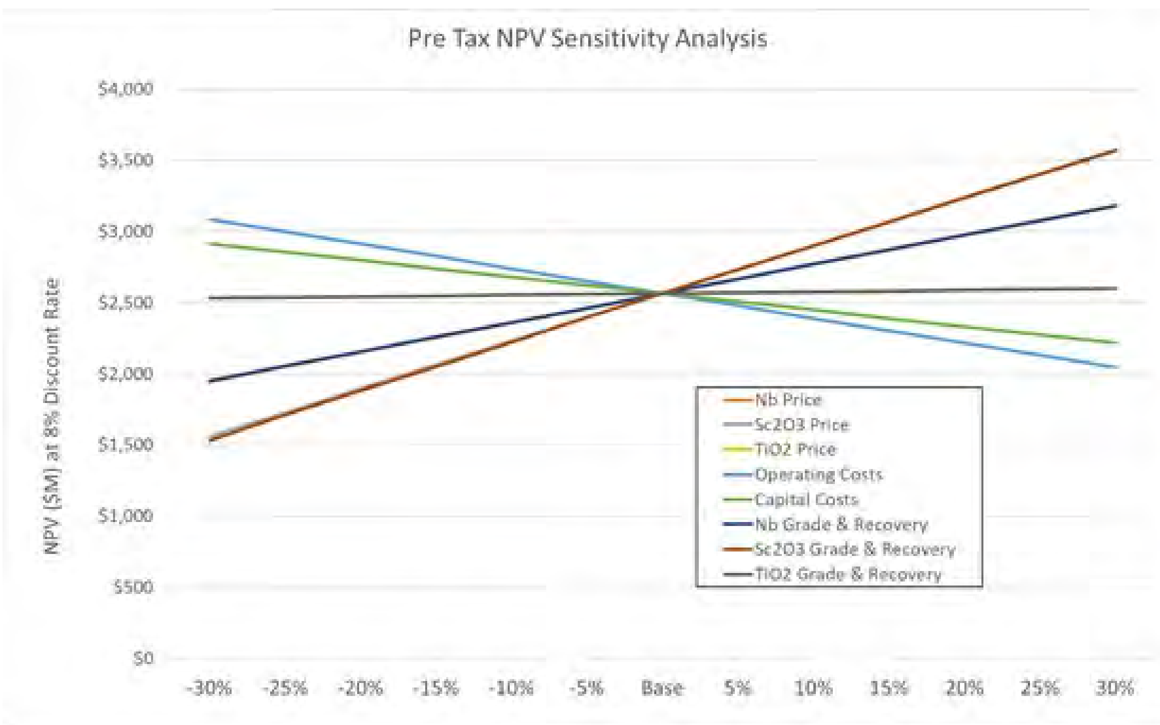
The Project is subject to a private third party NSR royalty of 2%. For the purposes of this economic analysis, this royalty is better defined as a "Net Proceeds" royalty as annual operating costs are deducted along with freight/insurance costs.

There is a US$ 5.50 million reclamation bond premium payable on the Project to be paid quarterly in a five-year period from 2022 through 2026, at which point the Project will be eligible based on its financial statements to provide other means of financial assurance to the State of Nebraska.

22.6 Sensitivity Analysis

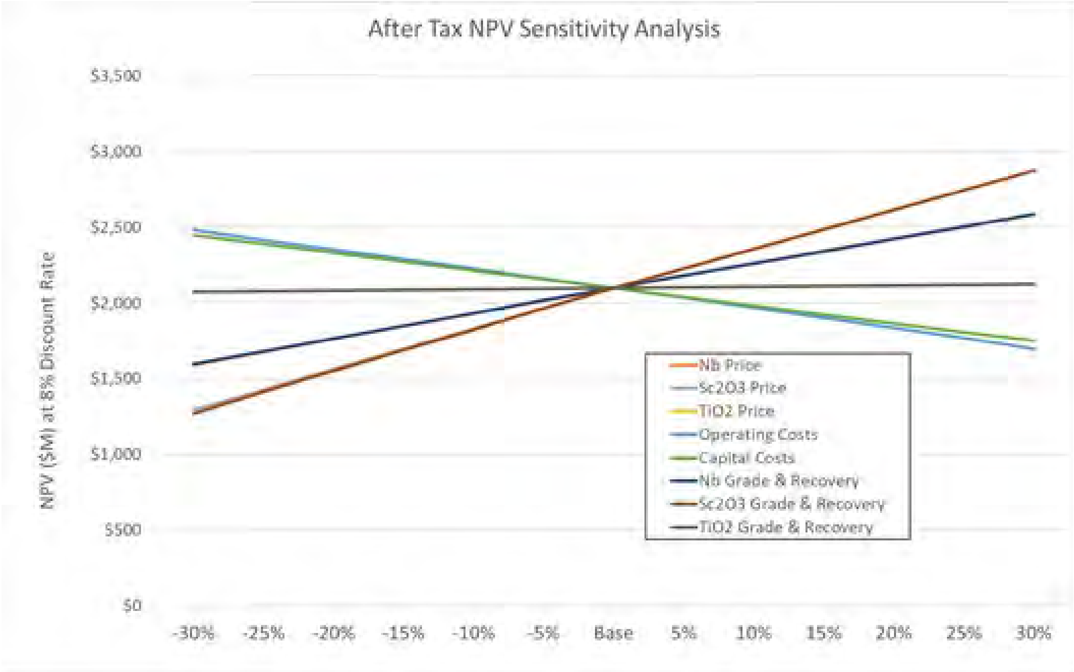
The cash flow model was tested for sensitivity to variances in milled tonnes, head grades (Nb, Sc, and Ti), process recoveries (Nb, Sc, Ti), metal prices, initial/sustaining capital expenditure and operating costs (mining, processing, water management, tailings management, site G&A and royalties).

Figure 22-3 and Figure 22-4 illustrate the results of pre/post tax basis with respect to four of the operational parameters and product prices along with recovery and head grades. The anticipated project cash flow is sensitive to the price of scandium and niobium compared to capital and operating costs, which were both quite similar.



Source: NioCorp, 2019

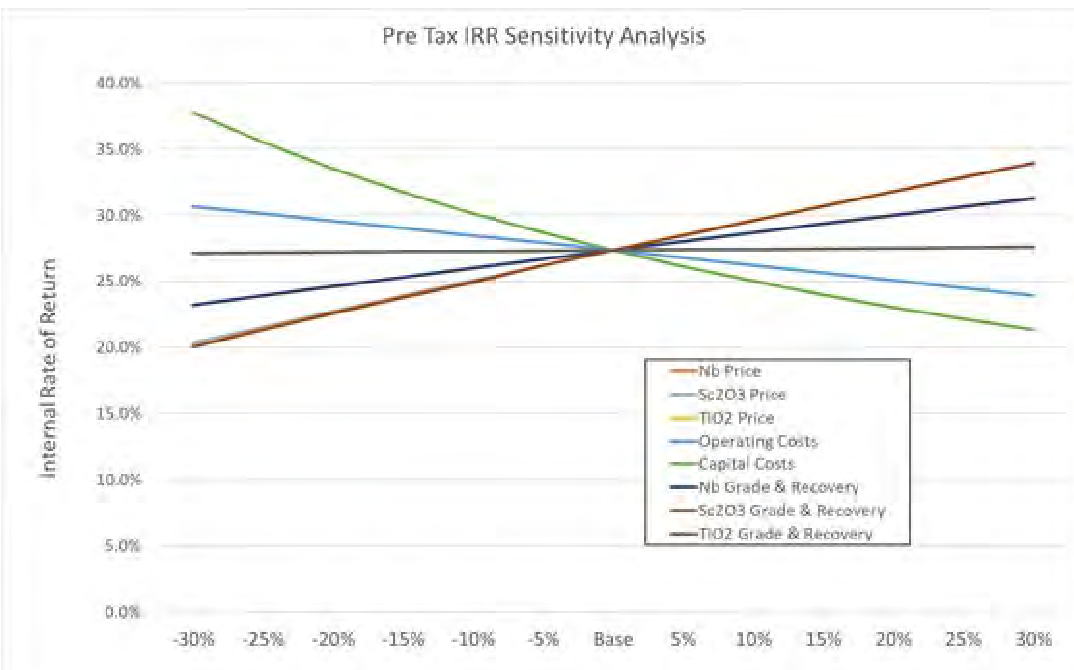
Figure 22-3: Pre-Tax NPV 8% Sensitivity Graph



Source: NioCorp, 2019

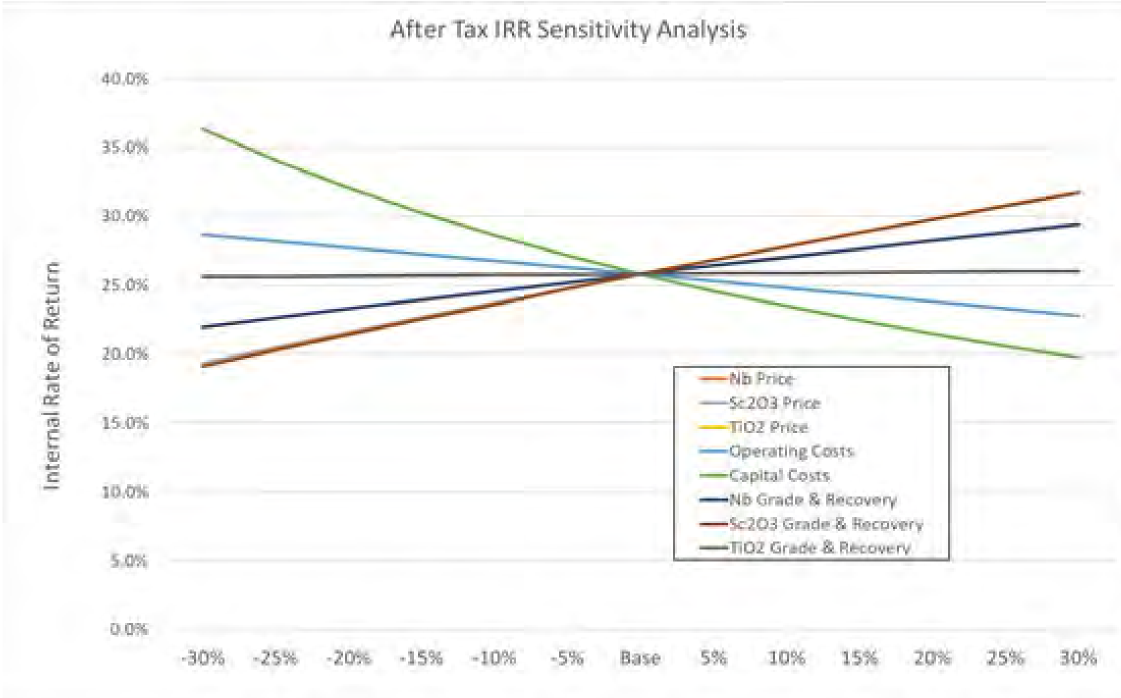
Figure 22-4: After-Tax NPV 8% Sensitivity Graph

Sensitivity graphs in Figure 22-5 and Figure 22-6 demonstrate the Project IRR is sensitive to changes in Sc_2O_3 and Nb prices on both a pre-tax and after-tax basis, but capital costs clearly have a greater effect than operating costs.



Source: NioCorp, 2019

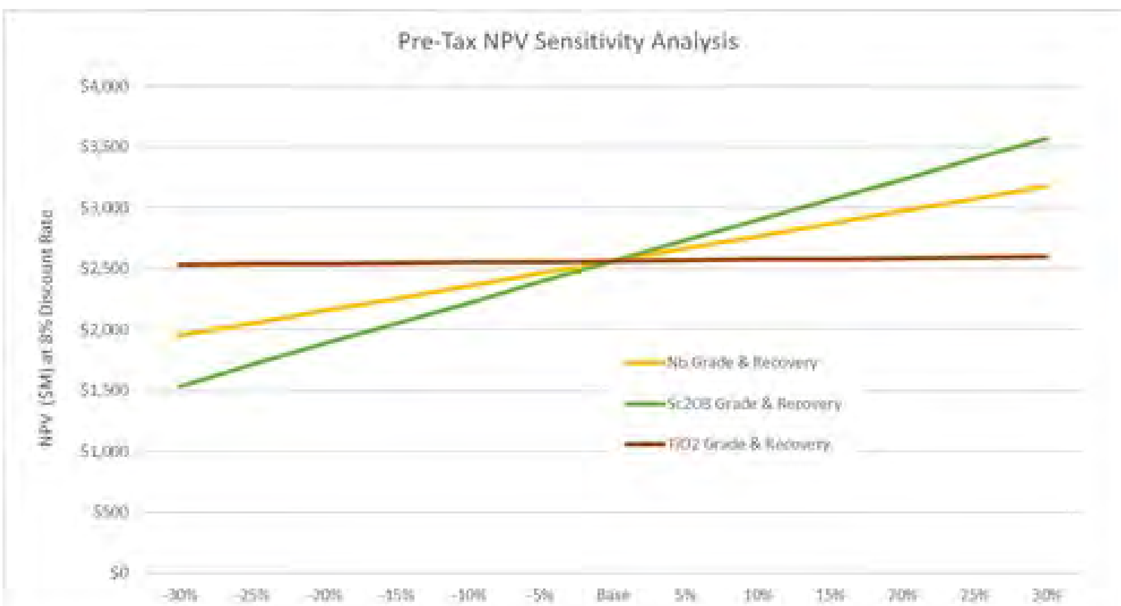
Figure 22-5: Pre-Tax IRR Sensitivity Graph



Source: NioCorp, 2019

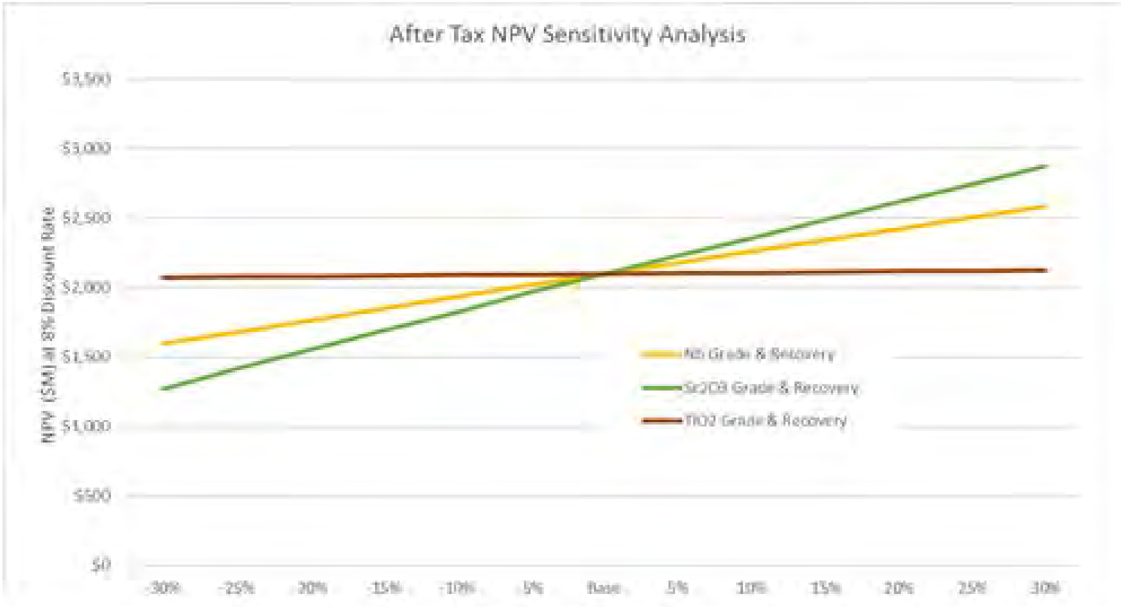
Figure 22-6: After-Tax IRR Sensitivity Graph

Figure 22-7 and Figure 22-8 illustrate the results of pre/post tax basis with respect to head grades and process recoveries of the three products. Not surprisingly, the impact of a head grade reduction is exactly equivalent to a process recovery reduction for each of the products.



Source: NioCorp, 2019

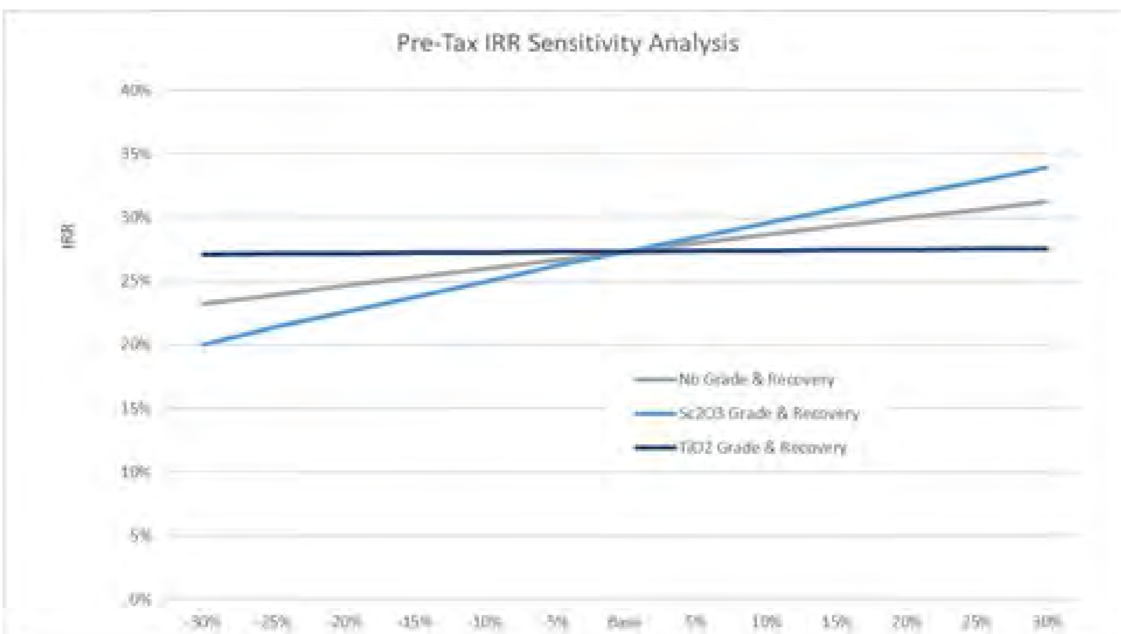
Figure 22-7: Pre-Tax NPV 8% Sensitivity Graph



Source: NioCorp, 2019

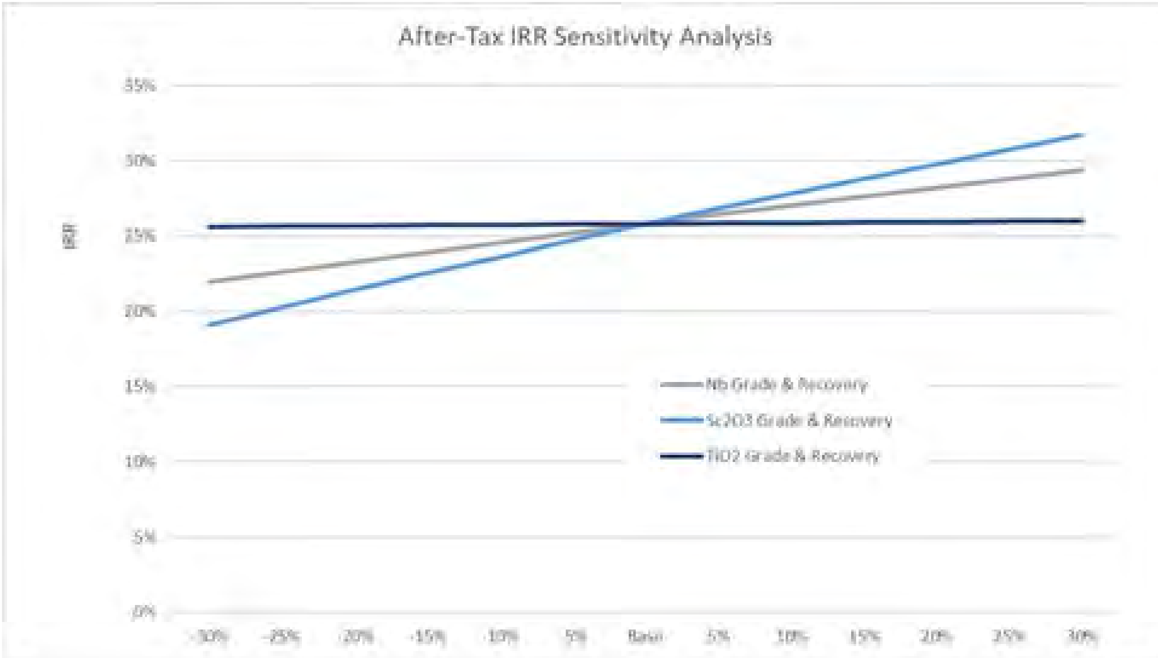
Figure 22-8: After-Tax NPV 8% Sensitivity Graph

Sensitivity graphs in Figure 22-9 and Figure 22-10 demonstrate the Project IRR is sensitive to changes in Sc_2O_3 and Nb head grade and recovery on both a pre-tax and after-tax basis, but with limited to no impact from TiO_2.



Source: NioCorp, 2019

Figure 22-9: Pre-Tax IRR Sensitivity Graph



Source: NioCorp, 2019

Figure 22-10: After-Tax IRR Sensitivity Graph

Given the relative sensitivity and impact of product pricing on project returns, Table 22-12 and Table 22-13 further summarize the financial results at different niobium and scandium price points.

For each table, the prices for the other products are held constant to their base case values. For example, when the price of niobium is raised to US$ 55.87/kg (120% of the base case value), scandium is held at an average of US$ 3,675/kg and titanium to US$ 0.99/kg.

Table 22-11 and Table 22-12 demonstrate that at a US$ 0/kg price for niobium, the project retains a US$ 706 million NPV (pre-tax) and a US$ 482 million NPV (after-tax). For scandium, the project's break-even pricing is US$ 1,028/kg (pre-tax) and US$ 1,130 (after-tax). These scandium prices represent, respectively, 28% and 31% of the base pricing.

Table 22-11: Niobium Price Sensitivity (Sc and Ti Prices Remain Constant)

Niobium Pricing (US$/kg)	% of Base Model	Pre-Tax NPV (US$ million)	Pre-Tax IRR	After-Tax NPV (US$ million)	After-Tax IRR
60.53	130%	3,182	31.3%	2,588	29.4%
55.87	120%	2,976	30.0%	2,425	28.2%
51.21	110%	2,770	28.7%	2,262	27.0%
46.55	100%	2,564	27.3%	2,098	25.8%
41.89	90%	2,359	26.0%	1,932	24.6%
37.23	80%	2,153	24.6%	1,763	23.3%
32.57	70%	1,947	23.2%	1,594	21.9%
27.91	60%	1,741	21.7%	1,424	20.6%
23.24	50%	1,535	20.3%	1,254	19.2%
11.59	25%	1,021	16.5%	824	15.6%
0.00	0%	506	12.4%	387	11.7%

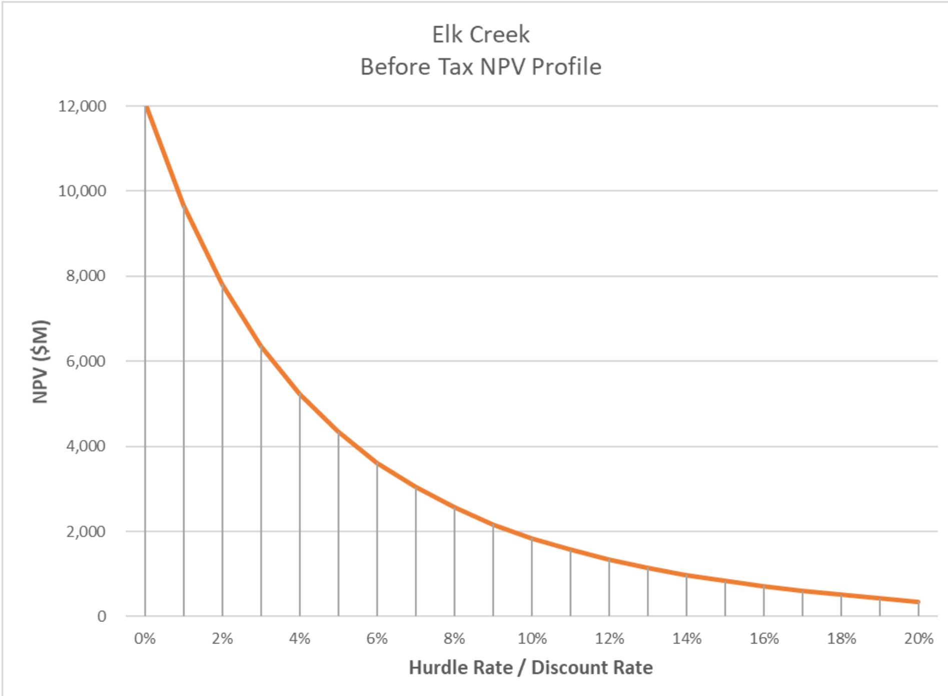
Source: Nordmin, 2019

Table 22-12: Scandium Price Sensitivity (Nb and Ti Prices Remain Constant)

Scandium Pricing (US$/kg)	% of Base Model	Pre-Tax NPV (US$ million)	Pre-Tax IRR	After-Tax NPV (US$ million)	After-Tax IRR
4,778	130%	3,568	33.9%	2,872	31.7%
4,411	120%	3,234	31.8%	2,615	29.8%
4,043	110%	2,899	29.6%	2,357	27.8%
3,676	100%	2,564	27.3%	2,098	25.8%
3,308	90%	2,230	25.1%	1,832	23.7%
2,940	80%	1,895	22.7%	1,562	21.6%
2,573	70%	1,560	20.3%	1,292	19.3%
2,205	60%	1,226	17.9%	1,017	17.0%
1,838	50%	891	15.3%	737	14.6%
1,470	40%	557	12.7%	455	12.1%
1,103	30%	222	9.9%	171	9.6%
893	24.29%	31	8.3%	0	8.0%
859	23.37%	0	8.0%	(28)	7.7%

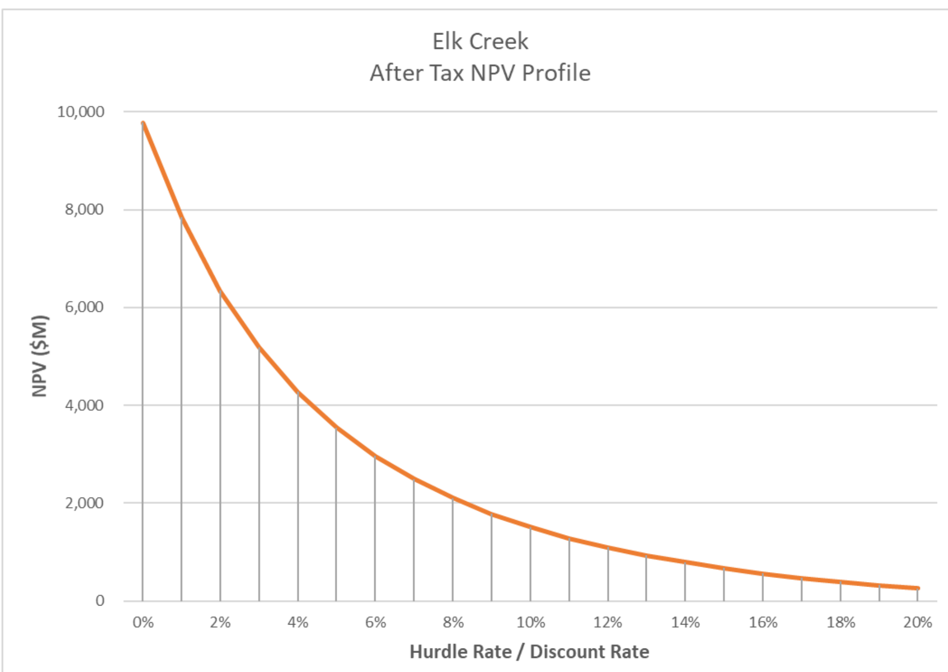
Source: Nordmin, 2019

Discount rate sensitivity is always important in a Project valuation, and with respect to this Project, there is a complex process plant flow sheet and market uncertainty to account for. NPV profile charts are presented in Figure 22-11 and Figure 22-12, which shows pre and after-tax NPV results for 100 basis point increments between 0% and 20%. It should be noted that with current assumptions, the Project breaks even at a ~20% hurdle rate on an after-tax basis.



Source: Nordmin, 2019

Figure 22-11: Before-Tax NPV Profile



Source: Nordmin, 2019

Figure 22-12: After-Tax NPV Profile

23. ADJACENT PROPERTIES

There are no significant properties adjacent to the Elk Creek Project.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation Plan

The key project objectives are to:

- Deliver the Elk Creek Mine Project on time and on budget.
- Ensure to meet environmental compliance.
- Ensure the safety of all Project stakeholders.
- Ensure compliance with all applicable laws and regulations, at the local, state and federal levels.
- Ensure positive economic impacts for SE Nebraska, including the use of local businesses wherever feasible, the employment of local residents and tax benefits for local governments.
- Maintain a high level of engagement and communication with all stakeholders.
- Ensure the Project meets design parameter objectives; throughput, quality, and operating budget objectives.

The Project Implementation Plan (PIP) execution is based on the use of two main EPCM contractors. One contractor with responsibilities for all mining related work and a second contractor responsible for all other site-wide related work. Certain portions of the site-wide work will be performed with EPC sub-contracts awarded to companies that specialize in process and technology related packages, such as the acid plant. The approach is reflected in the capital cost estimate for the Project.

24.1.1 Project Cost Objectives

Table 22-8 and Table 22-9 present the cost of the Project by the main category of work. The cost objective of the Project is to reach 100% of production capacity within the total cost of US$ 879 million (includes gross pre-production revenue credit). Numbers are rounded to the nearest thousand.

24.1.2 Project Schedule Objectives

The scheduling objective is to deliver a fully constructed and commissioned mining facility as per the following timeline.

The project timeline is based on achieving the First Metal milestone at 42 months after Authorization to Proceed, plus an additional four months of the ramp-up to 100% of production capacity for a total Project schedule lasting 46 months.

The schedule highlights are as follows:

- The total duration of the project is 46 months from Authorization to Proceed to the end of the ramp-up period.
- A four-month ramp-up period (included in the overall schedule) is allotted to increase the site throughput to 100% of nameplate rating.
- The Project timeline is linked to both the mining-related activities and the surface operations in both sequencing and duration. The construction of the main surface plant buildings and supporting infrastructure is not on the critical path.

- The critical path activities include the following:
    - Completion of the air permit.
    - Completion of drilling, sampling and final hydrogeological investigation.
    - Engineering and procurement for shaft sinking and mining components.
    - Construction of temporary power plant for shaft sinking and construction activities.
    - Construction of the temporary freeze plant for shaft sinking activities.
    - Establishing commercial natural gas and electricity service to the Project site.
    - Construction of the backfill plant.
    - Sinking both the production and ventilation shafts.
    - Underground mine infrastructure.
    - Completion of commissioning of the processing plant up to First Metal.
    - Ramp up of processing plant to full production capabilities.

Table 24-1 provides a summary of key activities leading up to First Ore (deemed "Advance of"), while all activities completed after First Ore are deemed "Post."

The Project Pre-production Schedule, provided in Appendix C, makes use of a monthly timescale, utilizing a declining monthly countdown (i.e. Authorization to Proceed is 38 months in advance of First Ore (-38)).

Table 24-1: Key Project Milestones

Activity	Completion Month (With Respect to First Ore)
Full Project Authorization	38 (Advance of)
DEQ Air Construction Permit	28 (Advance of)
Concrete Batch/Grouting/Backfill/Shotcrete Plant Complete	28 (Advance of)
Shaft Freezing to Limestone/Carbonatite Interface	26 (Advance of)
Commence Production Shaft Sinking	24 (Advance of)
Commence Ventilation Shaft Sinking	24 (Advance of)
Natural Gas Available	20 (Advance of)
Ventilation Shaft Sinking Complete	16 (Advance of)
Permanent Power Available	14 (Advance of)
Production Shaft Sinking Complete	10 (Advance of)
Water Treatment Plant Construction Completion	6 (Advance of)
Mineral Processing Commissioning Completion	1 (Advance of)
Underground Pre-Production Development Complete	0
Underground Major Infrastructure Complete	0

First Ore	0
Water Treatment Plant Commissioning Completion	1 (Post)
Hydromet Commissioning Completion	3 (Post)
Acid Plant Commissioning Completion	4 (Post)
HCl Regen Commissioning Completion	4 (Post)
Pyromet Commissioning Completion	4 (Post)
First Metal	4 (Post)
Acid Plant Commissioning Completion	4 (Post)
Full Mill Production Begins	8 (Post)

Source: Nordmin, 2019

24.1.3 Early Works

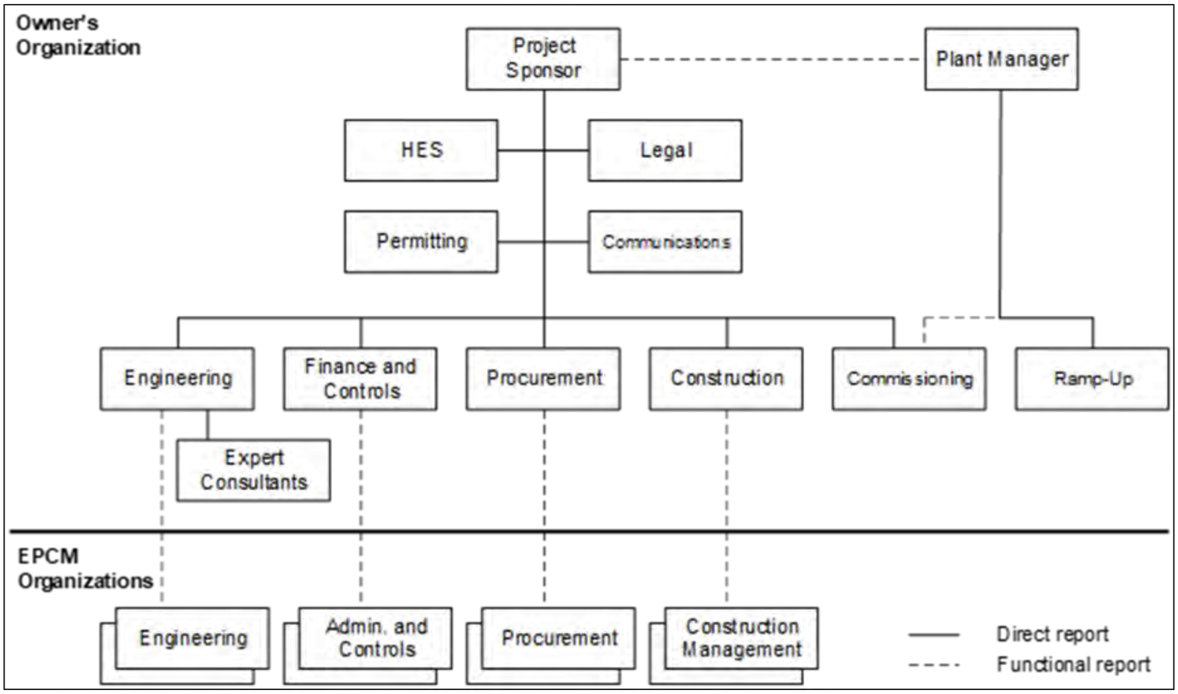
Project Execution requires key early work that includes the following:

- Finalize contracting approach and contract key contractors (EPCM).
- Optimization metallurgical testing to confirm details for plant detailed design.
- Permitting activities required for early works construction activities.
- Complete the drill program to finalize the hydrogeological and geotechnical reviews in order to properly locate both shafts, and determine the final approach to shaft sinking.
- Perform detailed engineering and procurement of long lead time items as available.
- Commence construction on the third-party natural gas pipeline and electric power supply by Omaha Public Power District.

24.1.4 Project Team

As previously stated, the primary execution of the Project will be performed with an EPCM approach utilizing one specialized EPCM contractor to manage and execute the construction of the mining related items and a second EPCM contractor to manage and execute the remainder of the Project construction. The two EPCM contractors will report to the NioCorp Project Sponsor.

The Owner's Project team organization will mirror the EPCM management structure. As presented in Figure 24-1, the Owner's team will have environmental, safety, and permitting staff, personnel for engineering, finance, controls, procurement oversight, construction management including scheduling and reporting personnel. The commissioning personnel will include key vendors and NioCorp operations personnel that will continue after the construction effort is completed, in order to operate the plant.



Source: Nordmin, 2019

Figure 24-1: Summary Level – Owner's Project Team

The NioCorp corporate team will remain in Denver, CO with a Project Team located both on the Elk Creek Mine site and in the company's offices in nearby Tecumseh, NE.

24.1.5 Project and Document Control

NioCorp will utilize a project controls system for monitoring, reporting, and controlling the Project schedule, the cost, and the scope of work (change management).

- The NioCorp Project Team will be responsible for establishing the project controls procedures and assuring its consistent application throughout the Project timeline.
- The Project team will also develop a control budget to aid in managing the overall effort and will develop an appropriate Project accounting system to meet the Project needs.
- The accounting system will be sued to baseline the Project cost and aid in forecasting cash flow needs. The system will aid in the creation of Earned Value Reporting (EVR) for the Project.
- The Project team will maintain the Project schedule with the use of scheduling software such as PrimaveraTM P6 or equivalent. The schedule will be updated on a regular basis to track Project progress, note any deviations.
- Change management will also be a function of the Project controls system and will be used to identify and track changes in the scope of work throughout the course of the Project.
- The Project controls system will provide Project KPIs (Key Performance Indicators) through dashboards, monthly reports, and management reports. KPIs will be determined by management in conjunction with the EPCM contractors to measure Project success.

24.1.6 Engineering

Following the completion of the Feasibility Study, design and engineering activities will be undertaken by engineering consultants. The design and engineering activities will be managed by the EPCM Contractor Engineering Managers and will be divided as follows:

- Mine water management
- Mining and mine infrastructure
- Tailings facility
- Paste backfill
- Processing plants
- Above ground infrastructure
- Water Treatment Plant
- HCl Regeneration Plant
- Acid Plant

24.1.7 Supply Chain and Procurement

The supply chain management responsibilities will reside with the EPCM Contractors. These duties include procurement, contracting, site material management, and development and management of work packages. The contracting strategy will include the use of a "pre-qualified bidders list" and contracts that are fixed price, lump sum or time and materials (T&M), as the work package dictates. As mentioned previously, certain packages will be turnkey EPC contracts. The EPCM contractors will perform procurement work consisting of:

- Development of the Long Lead Equipment list.
- Development of site-wide procurement needs and packages.
- Development of Equipment Procurement Packages.
- Procurement of goods and services as required.
- Administration of purchase orders.
- Expediting of deliveries.
- Quality Control of Fabrications.
- Logistics.

The key long lead time equipment currently identified are:

- Mine hoists, and conveyances.
- Mine substation transformer.

24.1.8 Construction Management

The EPCM contractors will perform construction management functions, including planning, organizing, and resolving issues involving the site contractors. Ensuring that contractors' work is performed according to the Project's safety, quality, schedule, and cost requirements. Additionally, the EPCM contractors are required to provide the facilities and services, including security, to support the sub-contractors. This practice will ensure that quality standards are maintained and will improve the use of shared resources and equipment. The primary functions include planning and coordination, contractor management, quality assurance, resolving design engineering issues, quantity measurement, and materials management.

24.1.9 Commissioning, Operational Readiness, and Early Operations

Commissioning

The Owner's team, in conjunction with the EPCM Contractors, will be responsible for commissioning activities. The team will develop a detailed commissioning plan during the course of the work that will address the following:

- Lists of Handover Packages & Commissioning Systems.
- Transition process.
- Commissioning Sequence.
- Alignment of Boundaries between Handover Packages and Construction Work Packages.
- Commissioning Schedule.
- Roles and Responsibilities.
- Scope of Work Alignment.
- HES Management for Commissioning.
- Handover Documentation.
- Vendor Management.
- Monitoring of Inspection and Testing performed by work Contractors.
- Commissioning Deficiencies.
- Acceptance process.
- Reporting.

The team will also partner with other key stakeholders (vendors, and suppliers) to complete the commissioning effort to hand over the Project to operating personnel for early operations and ramp up.

Operational Readiness and Ramp-up

Two Operations Readiness Plans will be prepared: the first Plan will be specific to the operation of the mine; the second Plan will be specific to the surface plant.

Training on equipment (both factory-based and on-site) will be provided by vendors. Request for quotations will require all vendors to supply Operation and Maintenance manuals, lists of spare parts for the first year of operation, list of commissioning spare parts, and training manuals. Vendors may be requested to perform on-site training based on the complexity of the equipment and/or its controls.

Ramp-up consists of bringing the plant production from First Metal, achieved by commissioning of the plant, up to 100% of commercial capacity. For the purpose of ramp-up, commercial capacity involves the production of Superalloy materials at 80% of the facility nameplate capacity, in salable quality.

NioCorp internal resources will execute the ramp-up under the responsibility of the Owner's Team.

Early Operations

The Project has a number of early operational tasks that are required at the onset of the Project. These activities include:

- Main Plant and mine electrical substations.
- Natural gas distribution.
- Concrete / Paste Backfill Plant.
- Temporary Freeze Plant.
- Temporary and permanent electrical distribution.
- Specific areas may be operated and maintained by internal Project staff or by third-party work contractors to be decided on a case by case basis. Early operations activities are included through the completion of the Project.

24.2 Opportunities and Risk Assessment

An opportunity and risk analysis was completed by Nordmin in conjunction with Optimize, SRK, Tetra Tech, Adrian Brown, Zachry, MCS, SMH and NioCorp at the project level, reviewing both opportunities and risks that were identified both during the previous 2017 feasibility study and the 2019 feasibility study by Nordmin.

Nordmin provided each QP with a semi-quantitative risk matrix where the likelihoods and consequences were assigned numbered levels that were multiplied to generate a numerical description of risk ratings. The values that were assigned to the likelihoods and consequences were not related to their actual magnitude, but to the numerical value that was derived for risk (see Figure 24-2). This approach provided for a standardized grouping and generation of indicated risk ratings. Each QP worked independently and reported back to Nordmin their findings which were then compiled and summarized below and provided in Appendix D, including possible mitigation strategies.

Likelihood x Consequence Matrix

		Consequence				
Likelihood		Negligible	Minor	Moderate	Major	Severe
	Score	*1*	*2*	*3*	*4*	*5*
Rare	*1*	1	2	3	4	5
Unlikely	*2*	2	4	6	8	10
Possible	*3*	3	6	9	12	15
Likely	*4*	4	8	12	16	20
Almost Certain	*5*	5	10	15	20	25

Low
Medium
High
Very High

Source: Nordmin, 2019

Figure 24-2: Likelihood and Consequence Matrix

24.2.1 Opportunities

Opportunities recognized during the analysis included:

Mine Operations

- Optimizing the mine plan based upon market conditions. At present, the production stopes are dictated by their niobium content. There are existing areas within the footwall zone that have high concentrations of scandium, but they have been dismissed as ore due to their lower content of niobium. If the scandium market demand remains intact and the processing plant can increase scandium throughput possibly through a separate circuit, then there would be additional ore within the existing vertical extent of the present mine design.
    - i.e. The current resource model has many resource blocks that have an NSR greater than $500/tonne that are currently not in the mine plan for they do not meet the niobium head grade requirements but do consist of high grade Scandium. As such, if market conditions change, there is an opportunity for the operation to adjust to meet the market needs.
- After completion of additional diamond drilling underground and development within the ore zone, there could be a reason to increase the width of the stopes from 15 m wide to 20 m wide, if geotechnical factors allow. This would decrease ore drive development by 25%, which is the predominant development activity.
- There could be an opportunity to replace the mining contractor after approximately three years of steady-state production. After this period of time, the full requirements to obtain sustainable production levels would be understood, and the owner could replace the contractor with their own workforce. The resulting operating cost should decrease; this would be partially offset with the purchase and sustainable capital for mobile equipment.

Ventilation

- The decrease in ventilation requirements. With present-day equipment manufacturing capabilities, it would be unreasonable to expect a mining contractor to equip themselves with an electric powered mucking and hauling fleet. It is reasonable to transition the diesel-powered haulage fleet to electric power possibly three years into full production. This change over to an electric powered fleet would have a substantial decrease in demand for ventilation underground, although part of the savings related to this would be offset with higher haulage costs to cover the more expensive equipment.

Mine Backfill

- Optimization of the backfill recipe. The work on the pastefill demonstrated that a 5% binder yielded very high backfill strength. There is potential for a considerable positive impact to the OPEX by optimizing the recipe and reducing the binder requirements.

Resource/Reserve Expansion Potential

- The current deposit is open in the hanging wall, foot wall and at depth and along strike. Further drilling during the infill definition drill programs can be used to determine if the ore body can be expanded.

Cost Estimating

- Use the Hydromet and Mineral Plant buildings for tanks fabricated on site.
- Consider surface-based stormwater drainage.

EPCM Phase

- Mineral Processing and Pyromet buildings: Stick built building vs prefab building.
- Hydromet building: Stick built first story and prefab second.
- Quality: Specialized contractor for installation of the liner in tailing and active dewatering pond.
- Environmental protection: Environmental barrier at ground level during construction.
- Update Paste Backfill Cement Content based on additional testing and evaluate lower cement content.

24.2.2 Risks

The risk analysis defined 58 risks and their associated potential mitigation strategies (see Appendix D).

- 25 risks were considered as a pre-response consequence of moderate, major or severe and a likelihood of likely or almost certain.
    - If the action plan is initiated, the post response consequence for these high-risk items reduces to 6 risks.
- 30 risks were considered as a pre-response consequence of minor or moderate and a likelihood of unlikely or possible.
    - If the action plan is initiated, the post response consequence for these moderate risk items reduces to 12 risks.

The major group of risks identified and have an action planned assigned in Appendix D are the following:

Mine Operational Risks

- Shaft Location - Drilling pilot holes for shaft locations to determine local geological, geotechnical and hydrological characteristics and conditions that would be encountered during shaft sinking.
- Resource/Reserve and Mine Design - Significant infill definition drilling is required during construction and operations phases to determine local geological, geotechnical and hydrological characteristics and conditions in conjunction.
- Grade Control - A daily grade control monitoring program is required to maximize the value of ore mined and fed to the surface plant. The grade control process involves the predictive delineation of the tonnes and grade of ore that will be recovered by the mining team. The program will involve incorporating the results from the infill drilling program in conjunction with an underground chip sampling program to define the boundaries of mineable ore blocks and determine the daily/weekly feed grades to the plant.

- UG Ground Support/Hydrogeology – an ongoing probe hole drill program/grout program needs to be established to support mining activities and not create significant production delays. The need to develop and deploy a high-pressure grout injection system is required to protect the mine from excess inflow to safeguard the project from injury, property damage and loss of life or equipment.

Hydrometallurgical Process Risks

A summary of the recommended test work is presented below to reduce further the risks associated with the Hydromet process design. It is expected that the work would proceed in parallel with detailed engineering for the project and would take an estimated 4 months to complete. Two commercial labs, one in the US and one in Canada, have been identified with the capabilities to complete this test work.

HCl Leach

- Optimize leaching of iron (Fe) to correlate with optimum niobium (Nb) precipitation and Fe/Nb ratios– aiming for the highest recovery of Nb while preventing titanium (Ti) co-precipitation.
- Validate the method used in the aging of the HCl Leach liquor prior to scandium (Sc) Solvent Extraction.

Acid Bake – Water Leach

- Perform vendor testing and optimization of Acid Bake operations and equipment.
- Validate process control and equipment capabilities - optimizing mixing time, temperature, acid to residue ratio.
- Optimize water to residue ratio in Water Leach.

Iron Reduction

- Verify reaction kinetics and the use of briquettes.

Nb Precipitation

- Optimize FeNb ratio.
- Optimize Precipitant (dilution water) acidity to maximize Nb precipitation and Ti selectivity.
- Optimize Final Free Acid (FAT) to maximize selectivity against Ti.

Ti Precipitation

- Further test work required to maximize the removal of uranium and thorium from the Titanium dioxide product to increase its value.

Sc Precipitation

- Optimize the H_3PO_4 addition.
- Optimize the Fe addition.
- Perform locked cycle tests on the Calcium loop.

Sc Refining

- Optimize and further evaluate Zr/Nb removal using mixed organics – stripping acid.
- Optimize conditions to minimize Sc losses.

Sc oxalate Precipitation

- Verify precipitation using solid oxalic acid – optimal amount for optimal recovery.
- Optimize acidity, temperature, and g/l with solid oxalic acid.
- Optimize the washing of Sc oxalate for calcining equipment integrity.

Acid Regeneration

- Optimize the filtration – evaluate equipment and filtration media.

Sulfate Calcining

- Optimize residence time.
- Vendor testing of different equipment and assembly.

General

- Equipment selection, material of construction and vendor guarantee testing.
- Consider a fully integrated pilot testing to be operated onsite during construction of a full-size plant to make final adjustments and equipment selection.
- Further perform process engineering during the detailed design phase.
- Perform process simulation of the yearly or monthly elemental feed composition using the METSIM model and the compositions from the mine plan.

Scandium Market Risks and Sales Plan

At the time of this report, NioCorp had entered into one offtake agreements covering scandium trioxide production from the Project.

The scandium trioxide offtake agreement is structured similarly to the Niobium contracts. The agreement has a ten-year term and a minimum of 12 t/y. At that rate, approximately 10 - 15% of the projected annual production is contracted. Further, the customer may elect to take more material in any given year above the prescribed minimum quantity.

NioCorp is also working with other potential customers at the time of writing and discussions with these potential customers are proceeding under the provisions of Non-Disclosure Agreements (NDAs). These potential customers can be separated into the following categories or final end products:

- Scandium/Aluminum alloys used in aerospace, automotive, and other applications to increase strength and allow for a reduction of weight. Interested customers are situated at various points in the supply chains for aerospace manufacturing and operation; specialty alloy manufacturing; and specialty minerals and metal brokers/distributors.
- Solid Oxide Fuel Cells. Scandium is used in the electrolyte of solid oxide fuel cells to increase the conductivity at lower temperatures, allowing for higher efficiency and longer life. Discussions with interested customers in this industry and its supply chains are continuing.

NioCorp has produced a small quantity of 99.9% pure Scandium Trioxide during lab pilot testing, which meets or exceeds the purity needed for virtually all mainstream commercial applications. This material has been sent and will continue to be sent to interested customers for their analysis.

The U.S. Congress is currently on track to approve an initiative contained in the FY2018 National Defense Authorization Act that is aimed at raising the profile of NioCorp's domestic Sc production with the U.S. Department of Defense (DoD) and with the many defense contractors who supply the DoD with technologies and platforms that could incorporate Sc. NioCorp is in talks with the DoD about how the Department can help to ensure the onset of U.S. production of Sc.

25. INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Technical Report.

25.2 Geology & Mineral Resource

Nordmin constructed mineralization models for the deposit based on all available drilling data. As summarized in Section 14, block modelling was completed in Datamine through explicit modelling of each domain (high grade Nb_2O_5 /TiO_2, high grade Sc, and low grade). Structural and mineralization trends were used in the interpretation and for selection of modelling parameters. A block model was built by estimating and combining block models for each domain, and the final block model has been fully validated with no material bias identified.

Nordmin has classified the Mineral Resource into Indicated and Inferred resource categories based on geological and grade continuity as well as drill hole spacing. The Mineral Resource Estimate has been defined based on NSR cut-off grade to reflect processing methodology and assumed revenue streams from Nb_2O_5, TiO_2, and Sc for the deposit. The updated Mineral Resource represents an increase in contained metal.

Additional material exists in the geological model, which has not been classified as Indicated or Inferred resource. The deposit remains open along strike in both directions and at depth, and there exists significant resource expansion potential based both on these factors as well as areas of the block model that require improved definition through diamond drilling.

25.3 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

Exploration completed to date has resulted in the delineation of the Elk Creek Deposit and a number of exploration targets.

Nordmin is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. In Nordmin's opinion, the drilling, core handling, logging, and sampling procedures meet or exceed industry standards and are adequate for the purpose of Mineral Resource Estimation.

Nordmin considers the QA/QC protocols in place for the Project to be acceptable and in line with standard industry practice. Based on the data validation and the results of the standard, blank, and duplicate analyses, Nordmin is of the opinion that the assay and bulk density databases are of sufficient quality for Mineral Resource Estimation at the Elk Creek Deposit.

No limitations were placed on Nordmin's data verification process. Nordmin considers the resource database reliable and appropriate to support a Mineral Resource Estimate.

25.4 Processing and Metallurgical Testing

<u>Mineral Processing</u>

The Feasibility-level comminution test work was completed in two stages at SGS. The primary stage was conducted on six composite samples and 13 variability samples and included the determination of standard comminution parameters (SGS 2016a). The second stage of comminution test work was conducted on a single composite sample, using a LABWAL HPGR semi-pilot scale test work program (SGS 2016b). The test work results indicate that the Project ore is categorized as soft to moderately hard in terms of ore hardness, and amenable to standard grinding as well as an HPGR operation.

Hydrometallurgical Plant

Pilot test programs showed that high recovery rates of the niobium, scandium and titanium could be achieved, and that recycling and regeneration of reagents was also possible; thus, minimizing fresh reagent input and waste generation. Recoveries of 85.8% Nb_2O_5 and 93.1% Sc_2O_3 have been demonstrated while achieving 40.3% recovery of TiO_2.

Further understanding of the Process was achieved with respect to the kinetics of each unit operation, which suggested that the design be adjusted from what was initially shown in the 2015 PEA. Among the changes, the following are of interest:

- The temperature of the HCl Leach was adjusted to control leaching of the iron. The Fe to Nb ratio in the Leach residue has an impact on the precipitation of Nb and the co-precipitation of titanium.
- Acid Bake total mixing and reaction time was reduced to 2.5 hours.
- Iron Reduction step was optimized based on actual reduction of Fe3+, which resulted in an improved iron consumption.
- Dilution ratio in the Niobium Precipitation was reduced from 5:1 to 0.6:1, thus reducing reagent consumption and equipment size. This, however, comes at the expense of a slight reduction in Nb recovery and an increase in Ti co-precipitation.

  Secondary scandium recovery from the barren sulphate solution was developed. Selective precipitation of the scandium over impurities was achieved. Scandium precipitated in this section is combined and recovered in the Sc Solvent Extraction.
- A scandium purification step was added that provided a 99.9% scandium product (as Sc_2O_3).
- HCl Acid Regeneration development proved that recovery of chlorides in excess 99% is achievable.
- Further test work and development provided the basis to greatly reduce the need for neutralizing reagents while increasing the recovery of sulphur; therefore, greatly reducing the need for sulphur import.
- Mixed sulphur oxide gas is treated and cleaned prior to being sent to the Acid Plant, therefore, reducing the size and cost of the Acid Plant.

Pyromet

Lab testing has confirmed most of the anticipated findings from the mathematical model that was developed by applying thermodynamic principles:

- The aluminothermic reduction of niobium oxide precipitate and iron oxide has been demonstrated. Ferroniobium particles have been formed, and the chemical proportion of iron and niobium is just what was expected.

- The change to produce a higher TiO_2 content product from the Hydro-Met did not change what was anticipated: Ti content in the FeNb alloy did not increase, and the reduction of Nb_2O_5 did not seem to be affected by this higher amount of titanium oxide.
- Different temperatures have been used in various tests which have provided good reference points on the slag behaviour. The Electric Arc Furnace (EAF) temperature during the operation is expected to correspond to a temperature between 1850°C and 1900°C.

25.5 Mining & Mineral Reserve

Geotechnical

A geotechnical field characterization program has been undertaken to assess the expected rock quality. This program included logging core, laboratory strength testing, in situ stress measurements and oriented core logging of jointing. The results of this program have provided adequate quantity and quality data for the feasibility-level design of the underground workings.

A geotechnical assessment of the orebody shape and ground conditions has determined that long-hole open stoping mining is an appropriate mining method. Stopes have been sized to maintain stability once mucked empty. A primary/secondary extraction sequence with tight backfilling allows optimization of ore recovery while maintaining ground stability. Primary stopes will be backfilled with cemented pastefill, while secondary stopes will be backfilled with either light-cement pastefill or uncemented waste rock from development.

The design has been laid out using empirical design methods based on similar case histories. The stability of the 2019 feasibility study mine design has been checked with 3D numerical stress-strain models of the working, which included consideration for mine-scale faulting. The modelling results confirm that stopes and access drifts are predicted to remain stable during active mining, including areas adjacent to pastefilled primary stopes. The revised stope dimensions have been reverified using empirical design methods. The current design has not been reverified using numerical analyses, but this reverification is recommended as the mine design is advanced to the final design.

Ground support requirements have been based on empirical ground support methods and have considered variable levels of required ground support.

The location of underground infrastructure (i.e., shafts, ventilation raises, shops, etc.) have been situated to minimize the adverse impact of encountering geologic structures (i.e., weaker faults and shear zones).

Mine Design

Longhole stoping is seen as the appropriate mining method for the deposit geometry. The large stope sizes minimize the mining cost. The increased dilution due to large stopes sizes is not particularly material to the mine plan as dilution has some grade.

An NSR approach was used focused on targeted amounts of Nb_2O_5 and takes into account revenue for three elements (Nb_2O_5, TiO_2, and Sc) and generates three separate products (TiO_2, FeNb, and Sc_2O_3). Stope optimization was completed to identify economic mining areas. The 3D mine design was completed on an elevated CoG, which achieved over three times the actual calculated cut-off. Two mining blocks were designed, giving a 36-year LOM, although additional material, classified as indicated, exists below the mine plan presented here.

The underground mine is accessed through a 6.0 m diameter production shaft system. A 6.0 m diameter ventilation/exhaust shaft serves as the mine exhaust, the second means of access, and second mechanical emergency egress. Both shafts are excavated using conventional shaft sinking methods in conjunction with freezing technology to an elevation 200 m below surface.

Tonnage and grades presented in the reserve include dilution and recovery and are benchmarked to other similar operations. Productivities were generated from first principles with inputs from mining contractors, blasting suppliers, and equipment vendors where appropriate. The productivities were also benchmarked to similar operations. Equipment used in this study is standard equipment used worldwide with only standard package/automation features.

A monthly/quarterly/yearly production schedule was generated using Deswik scheduling software. The steady-state mine production schedule of 2,764 t/d ore was based on the processing throughput. The mine design targeted higher annual ferroniobium production during the first five years of ore delivery, which resulted in an averaged annual production rate of 7,351 tonnes per year over this period. The steady-state average annual ferroniobium production was 7,220 tonnes annually.

25.6 Recovery Methods

Based upon the ore body samples retained, all bench testing performed, and process analyses completed to date, SMH, MCS and Zachry are confident that the current design will yield the FeNb, TiO_2 and Sc_2O_3 in the quantities expected. While this level of design is feasibility, it is expected that additional design effort during the detail phase will likely yield better results and further improve the efficiency and yields of the processes.

25.7 Infrastructure

Onsite and Offsite Infrastructure

Based upon the most current operating and process design information and expectations, the on-site and off-site infrastructure and services will meet each of the required needs of this entire facility.

Infrastructure buildings, office space, locker facilities and showers were sized and designed based upon current workforce projections for the site, as well as a tentative work schedule of 12-hr shifts for shift personnel, and standard 8-hr shifts for non-shift staff. A change in the number of shifts and/or shift durations may have an impact on the requirements of these facilities.

Likewise, both potable water and wastewater distribution systems were sized based upon the above shift criteria. Changes in the number of personnel, and/or changes in numbers of shifts and shift durations may have an impact on the potable and wastewater demands which must be addressed during the detail phase of this design.

Off-site infrastructure in the form of natural gas and electrical power services provided by others are readily available, and well within the current demand requirements of the facility. Potable water sources yielding approximately 4,000 gpm are available from the local municipality (City of Tecumseh), as well as from two private landowners. The public water source would require a service extension from the existing system, while the private sources would require pipelines from the respective owner's wells.

Changes to the process during the detail phase, in particular, changes to the Hydromet process could also have an impact on both potable and process water and an impact on the WWTP (Waste Water Treatment Plant). Additionally, changes to the process could have an impact on the quantities and type of reagents required, which could, in turn, change the size of storage tanks and facilities, as well as the types and materials of construction of these facilities (tanks, totes, bunkers, etc.).

Foundation designs for large loads and structures, as well as roadway designs, were based upon the most current geotechnical report and best engineering practices for the local site conditions.

The most current geotechnical report partially addressed the recommended designs for deep foundations or foundations for large loads; building columns, columns with bridge crane loads, large process equipment or structures. It will be important that the final geotechnical report address these types of loads and provide specific recommendations, but that the final geotechnical site evaluation includes test borings in the final locations of buildings, process equipment and major structures. Recommendations should further include expected settlements, as well as pavement designs with material and compaction recommendations.

25.7.1 Tailings

The tailings storage facilities (TSF) are designed for storage of dry tailings solids in lined facilities permitted under State of Nebraska Industrial Solid Waste regulations. Separate lined “leachate collection ponds” (LCPs) will be used for management of precipitation contacting the tailings solids. Based on the parameters and assumptions outlined in Section 18.12, the Plant Site and Area 7 TSFs have been designed with adequate containment and capacity to manage the planned filtered water leach residue, calcined excess oxide, and slag deposition for a 36-year LOM.

25.8 Environmental, Permitting and Social or Community Considerations

NioCorp has developed information and conducted a number of environmental studies related to baseline characterization for the Project, the most important of which are the studies related to hydrogeology and geochemistry. The production rate and geochemistry of dewatering water will dictate what is critical to the onsite water balance and any additional management (active or passive) that may be required.

The geochemistry and characterization/classification of the ore and waste materials (including the final process waste streams making up the bulk of the tailings mass and the crystallized RO water treatment salts), directly influences the management of these materials given the presence of naturally occurring radioactive materials (NORMs) (i.e., uranium and thorium) and the potential for limited reaction to contact with water. These materials currently classify as non-hazardous based on regulatory testing. Site-wide management of non-contact and contact stormwater will be essential to Project compliance.

Engagement of local, state and federal regulators has commenced. Initiation of the formal permitting program for the Project is dependent upon the completion of the mine plan and surface facilities being developed as part of this technical document, as well as additional characterization of the waste materials and potential worker exposures under the jurisdiction of the Nebraska Department of Health and Human Services (DHHS) and U.S. Department of Labor — Mine Safety and Health Administration (MSHA), both of whom will have primary oversight of worker safety and monitoring programs with respect to the presence of NORMs in the ore and waste rock.

Without specific hardrock mining regulations, there are limited obligatory requirements for reclamation and closure of mining properties in Nebraska. There are provisions, however, within the applicable regulatory framework that is likely to be applied to the Project during the permit and licensing processes, specifically those associated with the TSF and mineral processing facilities. This will include the provision of financial surety for proper closure and reclamation of the site. The currently estimate costs for closure and reclamation of the Project is US$ 45 million.

Overall, the Project appears to be sufficiently advanced to initiate the submission of formal permit applications which will govern the construction, operation, and closure of the mine. However, given the complexity of the mine design, process operations, accelerated schedule currently envisioned by NioCorp, and the inexperience of the state regulators with this type of mining, one must recognize that risks remain within the permitting process that could slow Project development, even with the overwhelming support that the Project appears to have from the communities and stakeholders.

25.9 Market Studies and Contracts

Market studies for niobium, titanium dioxide and scandium trioxide are an important part of the proposed Elk Creek Mine. These products, especially niobium and scandium trioxide (scandium), are thinly traded without an established publicly available price discovery mechanism.

Marketing studies and product price assumptions are based on research and forecasts for the following products:

- Ferroniobium (FeNb): Roskill's Global Industry, Markets and Outlook 2018 (Roskill, 2018)
- Scandium Trioxide (Sc_2O_3): OnG Commodities LLC (OnG, 2019) - specializes in the scandium alloys and scandium markets.
- Titanium Dioxide (TiO_2): USGS Commodity Market Summaries (Bedinger, 2019) and Adroit Market Research (Johnson, 2019).

NioCorp is considering selling ferroniobium, scandium trioxide and titanium dioxide products from the Project through all avenues, which include entering into long-term contracts with buyers.

At the time of this report, NioCorp had entered into three off-take agreements covering ferroniobium and scandium trioxide production from the Project.

No off-take agreements have been executed at the time of the report for the titanium dioxide product from the Project. It is assumed this product and all other material not covered by an off-take agreement will be sold on a spot price, ex-mine gate basis.

25.10 Capital and Operating Costs

The estimate meets the classification standard for a Class 3 estimate as defined by AACE international and has an intended accuracy of ± 15%. The estimate is reported in Q1 2019 U.S. constant dollars.

Total LOM capital costs, including initial, sustaining and reclamation costs, are US$ 1,565 million. The initial capital estimate of US$ 1,143 million can be partially offset by a Gross Pre-production Revenue Credit of US$ 265 million (generated by pre-production product sales) to net to a cost of US$ 879 million.

The operating cost estimates were developed to show annual costs for production. Unit costs are expressed as US$ 196.41/tonne processed. LOM operating costs are estimated to be 1,560 million.

The operating cost varies by year, by mine location and production. The annual operating cost varies by year but averages approximately US$ 44 million per year over the LOM. The mining operating cost is based on a Q1 2019 cost basis.

25.11 Economic Analysis

The 2019 Technical Report is based on an assumption of processing of 36,313 (kt) over a 36-year life of mine (LOM) to produce 168,861 tonnes of Nb in the form of ferroniobium, 3,410 tonnes of Sc_2O_3 and 418,841 tonnes of TiO_2.

On a pre-tax basis, the NPV (8% discount) is US$ 2,564 million, the IRR is 27.3%, and the assumed payback period is within 2.85 years.

On a post-tax basis, the NPV (8% discount) is US$ 2,098 million, the IRR is 25.8%, and the assumed payback period is within 2.86 years.

26. RECOMMENDATIONS

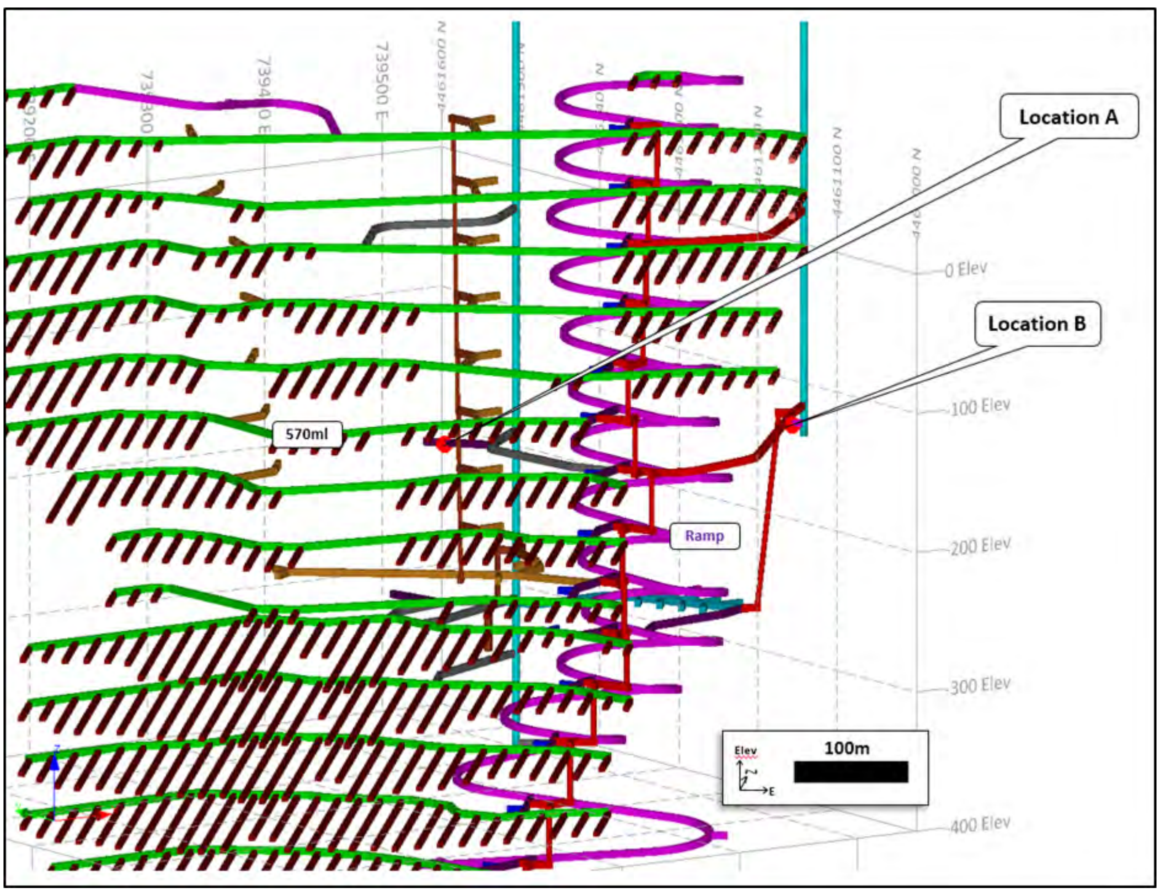
26.1 Recommended Work Programs

26.1.1 Geology and Resources

Nordmin recommends extensive definition drilling as development progresses. After development reaches an accessible area, definition drilling should immediately commence. Two locations have been identified as drill stations:

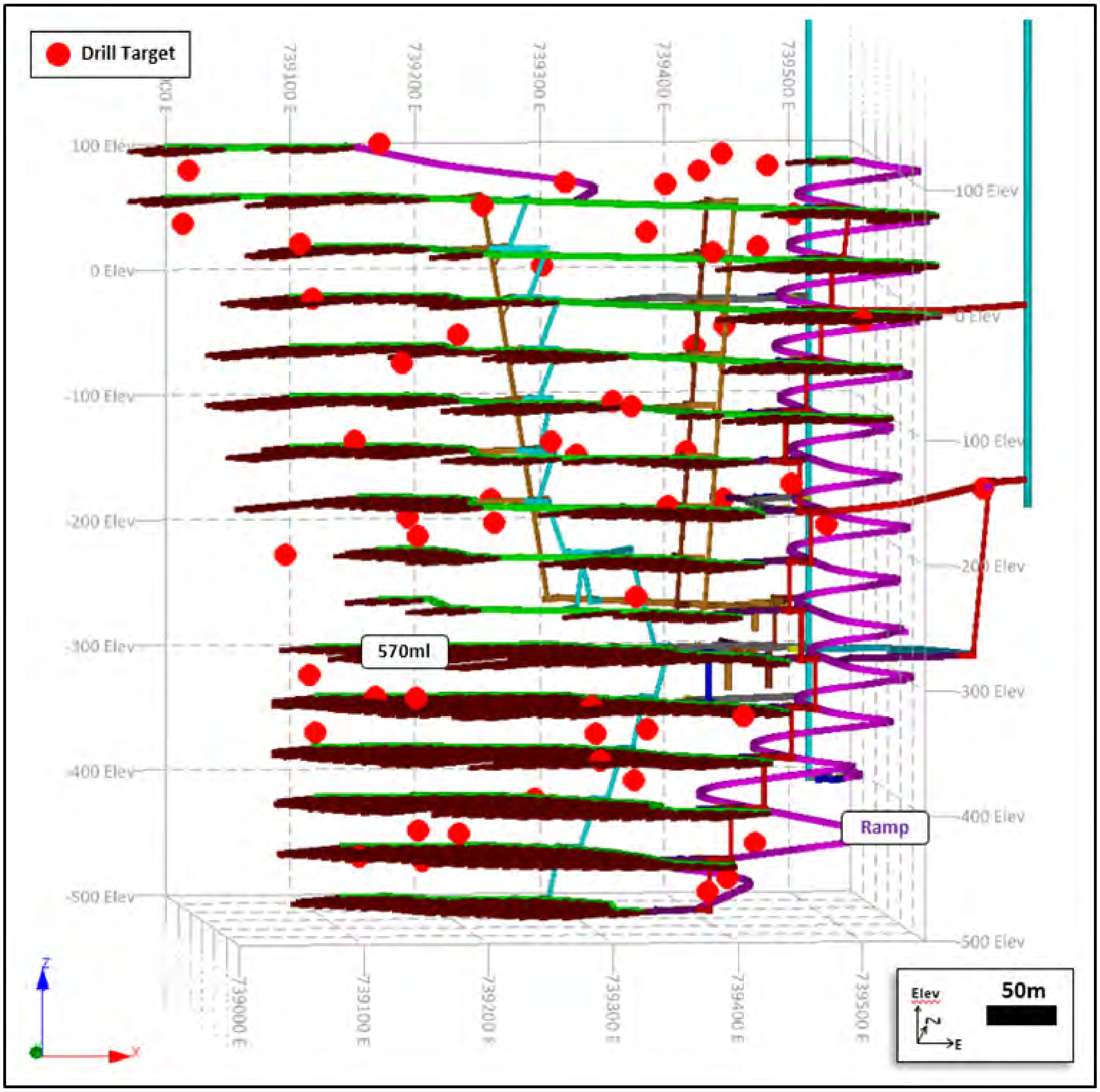
- Location A is the 570 m level shaft station drift (preferably the planned refuge station); and
- Location B is the planned remuck crosscut off the planned ventilation shaft located on the 570 m level access (see Figure 26-1).

Initially, drilling should be prioritized with the primary purpose of improving the level of confidence in grade, quantity and resource classification. Nordmin recommends that three drills be on-site, including two standard definition drills and one mobile unit. The recommended meterage for total planned drilling per year is 50,000 m for the first five years, with 30,000 m per year for subsequent years. Priority target areas have been defined for initial drilling (see Figure 26-2), comprised primarily of a lower number of composites used per block, as well as the block Nb_2O_5 grade.



Source: Nordmin, 2019

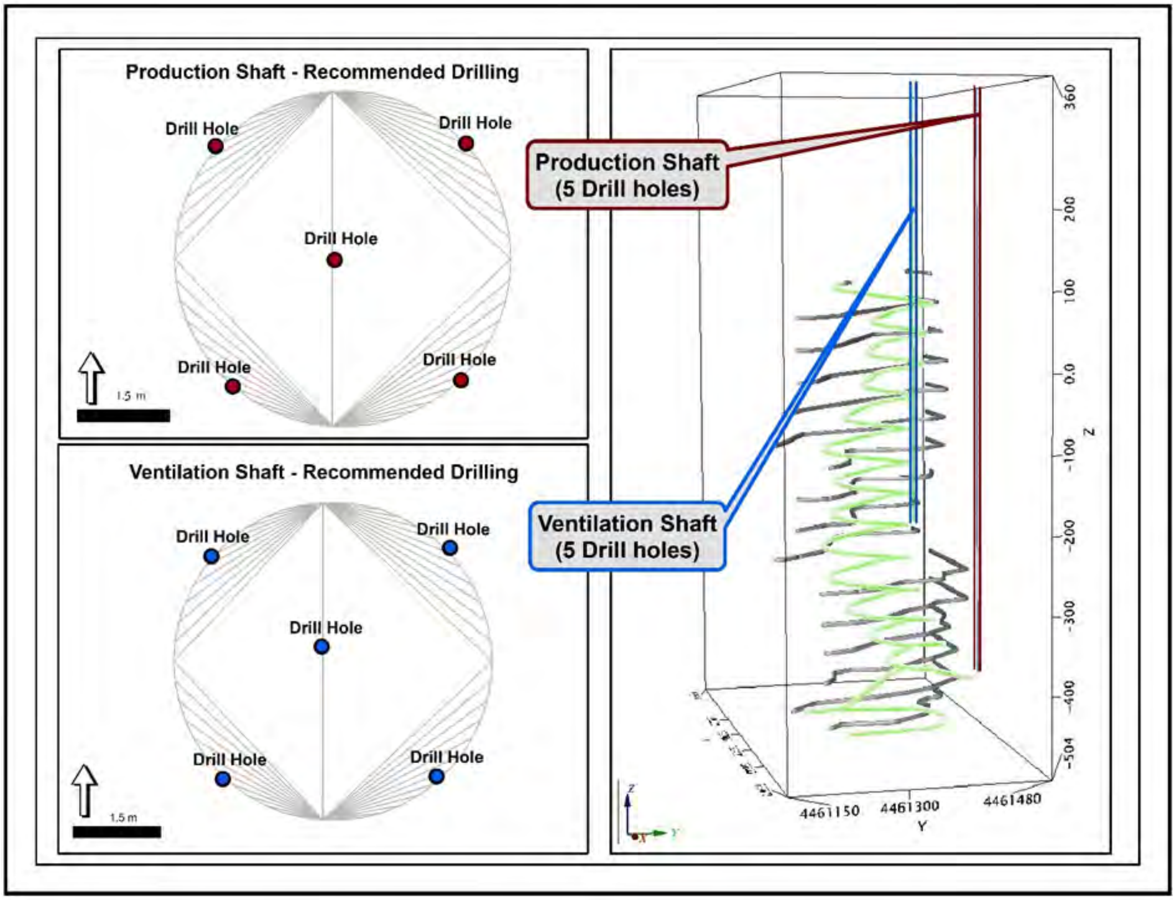
Figure 26-1: Drill Collar Locations



Source: Nordmin, 2019

Figure 26-2: Definition Drilling Targets

Nordmin recommends that for the geological and structural definition of the immediate area where the production shaft and ventilation shaft are to be developed, ten total drill holes be drilled. For each of the production and ventilation shafts, one hole is to be drilled in the planned centre, and a hole is to be drilled in each of four shaft edge corners to planned depth (see Figure 26-3). The production shaft holes would be 700 m each, and the ventilation shaft would be 525 m each, for a total of 4900 m. Each hole is to be navigationally controlled and will be logged for lithology and structural geology. Nordmin recommends the geotechnical parameters, hydrogeology, and piezometry be analyzed, and an acoustic televiewer used. Additionally, it is recommended each hole is plugged when complete and tested for water seepage.



Source: Nordmin, 2019

Figure 26-3: Recommended Definition Drilling for the Production and Ventilation Shafts

26.1.1.1 Quality Assurance/Quality Control

Nordmin recommends the following quality assurance/quality control procedures be created and followed:

- At least three certified reference material samples (CRMs) to be consistently included during sampling, comprised of the low, medium, and high values for the standardized assay.
- A clear protocol to manage CRM failures.
- Regular monitoring of the high/low CRM bias on an ongoing basis.
- A clear audit trail for re-assay.
- Perform the analysis on the 2011 assay program, which did not include selected re-assays.
- To track samples through the assay process, a work order is to be included in the assay summary sheet.
- Submit to SGS an additional, comprehensive set of samples with CRMs, explicitly focusing on mining grade ranges between 0.5 and 1.5% Nb_2O_5, to determine if a bias exists and if correction factors may be required.

- Local standards should be created for Nb_2O_5, TiO_2, and Sc using material from the Project site. This would eliminate the use of standards that are not appropriate for the deposit both from a grade and chemistry perspective.

26.1.2 Hydrometallurgical Plant

Adequate test work was conducted to support a feasibility-level design for the Hydromet Plant, and all sections of the process have been successfully tested at the pilot scale required for a feasibility study. However, optimization was not achieved in all areas, and certain areas will certainly benefit from further "post feasibility study" test work, preferably before detailed engineering activities begin. A number of factors have not been optimized in this study, and further testing will be preferable to achieve optimal results. Such optimization could also be achieved with the help of the process simulation of the yearly or monthly elemental feed composition using the METSIM model and the compositions from the mine plan.

A summary of the recommended test work is presented below.

- HCl Leach
  - Optimize leaching of Fe to correlate with optimum FeNb ratio and Nb Precipitation – aim to best recovery of Nb while preventing Ti co-precipitation.
  - Optimize the method used in the aging of the HCl Leach liquor prior to Sc Solvent Extraction.
- Acid Bake – Water Leach
  - Perform vendor testing and optimization of Acid Bake operations and equipment.
  - Optimize process control and equipment capabilities - optimizing mixing time, temperature, acid to residue ratio.
  - Optimize water to residue ratio in Water Leach.
- Iron Reduction
  - Verify reaction kinetics and the use of briquettes.
- Nb Precipitation
  - Optimize FeNb ratio.
  - Optimize Precipitant (dilution water) acidity to maximize Nb precipitation and Ti selectivity.
  - Optimize final free acid (FAT) to maximize selectivity against Ti.
- Ti Precipitation
  - Further test work required to maximize Th/U removal from the titanium dioxide product to increase its value.
- Sc Precipitation
  - Optimize the H_3PO_4 addition.
  - Optimize the Fe addition.
  - Perform locked cycle tests on the Calcium loop.

- Sc Refining
  - Optimize and further evaluate Zr/Nb removal using mixed organics – stripping acid.
  - Optimize conditions to minimize Sc losses.
- Sc oxalate Precipitation
  - Verify precipitation using solid Oxalic Acid – optimal amount for optimal recovery.
  - Optimize acidity, temperature, and g/l with solid Oxalic Acid.
  - Optimize the washing of Sc Oxalate for calcining equipment integrity.
- Acid Regeneration
  - Optimize the filtration – evaluate equipment and filtration media.
- Sulphate calcining
  - Optimize residence time.
  - Vendor testing of different equipment and assembly.
- General
  - Equipment selection, material of construction and vendor guarantee testing.
  - Further perform process engineering during the detailed design phase.
  - Perform process simulation of the yearly or monthly elemental feed composition using the METSIM model and the compositions from the mine plan.

26.1.3 Geotechnical

To advance to the final mine design, additional characterization data will be required to reduce geotechnical uncertainty. SRK recommends the following characterization and design activities:

- Drill holes at the final shaft and ventilation raise location to confirm ground conditions for the shaft ground support.
- An additional 4 to 6 geotechnical drill holes in the footwall infrastructure and planned stope mining areas to verify the range of expected ground conditions. This includes collecting:
  - RMR/Q data
  - Structural orientation data
  - Updating the structural model and geotechnical models
  - Updating mine design parameters
- Additional geotechnical drill holes to characterize ground conditions for the final alignment of the ramps and footwall drives. These holes should be drilled from underground after the shaft is constructed and the initial access drives are mined.
- The geotechnical model should be updated to reflect the additional characterization information from new drill holes.
- The numerical model of stope stability should be reanalyzed given the revised stoping sequence. This analysis would consider any new characterization information in the geotechnical model and recent adjustments to underground infrastructure and development.

- A Ground Control Management Plan (GCMP) should be developed for guiding the initial underground development activities. This plan should include plans for geotechnical monitoring and initial Triggered Action Response Plans (TARPs) specific to ground control, including a TARP for sudden groundwater inflows and grouting plans.

26.1.4 Mining and Reserves

No additional work or costs have been identified or recommended beyond the work outlined in the 2019 feasibility study.

26.1.5 Recovery Methods

Mineral Processing

Based on the metallurgical test work and the process design completed for the feasibility study, Zachry recommends further pilot testing of the HPGR option. The pilot HPGR test work program should be conducted prior to the detailed engineering stage in order to confirm and determine:

- The main operating variables of the HPGR
- Flake strength
- Wear rate:
  - Abrasion Index

The results from the pilot testing will be used to confirm the unit energy consumption, optimize the HPGR operating variables, and confirm the size of the HPGR unit for industrial operation.

The sampling requirement for the pilot testing is estimated to be approximately 1.5 t. The cost of pilot testing will be in the range of US$ 8,500 to US$ 30,000.

Hydrometallurgical Plant

Any additional work required is included in the detailed engineering scope of work and included in the feasibility cost.

Pyromet

Even though the testing has shown good results and is aligned in accordance with the mathematic model developed using thermodynamic calculations, a few minor issues remain to be addressed:

- Optimize the capacity of the Hydromet to increase the proportion of Nb_2O_5 in the precipitate. A target ratio of Nb_2O_5 / TiO_2 of 1 would be suitable.
- Perform large scale testing with an EAF to ensure good separation of slag/metal liquid and ensure the homogeneity of the ferroniobium alloy.
- Develop a flux that will enhance the fluidity of the slag at 1850 °C and 1900°C.
- Select a material for the refractory that will resist the aluminothermic conditions in the Electrical Furnace.

26.1.6 Infrastructure

General Infrastructure

Tetra Tech recommends additional geotechnical investigation and design recommendations based upon the detail design requirements addressed in Section 25.5; borings in the selected building and large equipment locations, high load and deep foundation recommendations, as well as pavement design recommendations based upon the type and frequency of vehicle traffic.

Any additional work required is included in the detailed engineering scope of work and included in the feasibility cost.

Tailings

The detailed design phase of the Plant Area and Area 7 TSFs will include characterization of any additional tailings materials, additional geotechnical characterization of Plant Area TSF foundation and borrow materials, and confirmation of feasibility-level containment, water balance and stability design.

Any additional work required is included in the detailed engineering scope of work and included in the feasibility cost.

Salt Management

The final salt product will be characterized for solubility, runoff chemistry, and geotechnical characteristics to aid in the detailed design of the proposed salt management cells.

Backfill

Testing of the backfill mixture is recommended during the detailed engineering phase of the project in order to help reduce the cement content, maximum strength and minimize the overall plant operating cost. By doing so in the early stages of the project, the design for the backfill plant can be modified, and recipes perfected prior to construction.

Additionally, investigating potential synergies between the surface mill and underground mining could potentially make use of concrete produced on site in the backfill plant rather than purchasing the mixture through a third-party supplier.

26.1.7 Environmental and Social

With respect to environmental, permitting and social/community issues for the Project, SRK provides the following recommendations to NioCorp:

- Remain engaged and transparent with Bold Nebraska and other stakeholders/non-governmental organizations throughout the permitting process and provide them with an opportunity to participate in any public meetings or town hall discussions. This tends to garner less opposition when it comes time for formal public comments on permit applications.
- Complete more detailed hydrogeological investigations of the orebody to more accurately and precisely define the quantity and long-term quality of dewatering water expectations, and assess the feasibility of RO water treatment brine reinjection.
- Continue characterization work on the mine waste rock, process tailings, and RO water treatment crystallized salt materials in order to define the extent and partitioning of radionuclides more precisely. Assess the potential effects of the exothermic reactions from the hydration of the calcined tailings materials on the overall TSF facility, worker safety, and surrounding environment, including the potential for rad-containing, fugitive dust generation.

26.1.8 Summary of Costs for Recommended Work

Costs for recommended work programs are summarized in Table 26-1.

Table 26-1: Summary of Costs for Recommended Work

Area	**Program**	**Cost Estimate (US$)**
Geology and Resource	No additional work or costs have been identified or recommended beyond the work outlined in the Feasibility Study.	
Processing & Metallurgical Testing	Costs of the testing program for process optimization and/or vendor equipment selection have been included in the feasibility cost estimate.	$1,500,000
Processing Plants mainly the Hydrometallurgical Plant	Costs of the program in Section 26.1.2 have been included in the Detailed Engineering Phase of Work included in the Feasibility Cost Estimate.	$1,500,000
Mining & Reserves	No additional work or costs have been identified or recommended beyond the work outlined in the Feasibility Study.	
Geotechnical	Costs of the program in Section 26.1.3 have been included in the Detailed Engineering Phase of Work included in the Feasibility.	-
Recovery- Processing Plant	Zachry recommends further testing of the HPGR option.	$8,500 to $30,000
Recovery- Hydrometallurgical Plant	Any additional work required is included in the detailed engineering scope of work and included in the feasibility cost as noted in 26.1.5.	-
Recovery- Pyrometallurgical Plant	Further testing is recommended during the next phase of work to optimize the system, as noted in Section 26.1.5.	-
General Infrastructure	Tetra Tech and Nordmin recommend additional investigation per Section 26.1.6.	-
Tailings	Costs of the program in Section 26.1.6 have been included in the early stages of Detailed Engineering Phase of Work included in the Feasibility.	$450,000
Environmental and Social	No additional work is identified other than that included in workplan provided for the detailed engineering phase of work and cost estimate included within the feasibility study capital estimate	-
Economic Analysis	No additional work is identified other than that included in workplan provided for the detailed engineering phase of work and cost estimate included within the feasibility study capital estimate	-
Total		**$3,458,500-$3,480,000**

Source: Nordmin, 2019

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28. GLOSSARY

The Mineral Resources and Mineral Reserves have been classified according to CIM (CIM, 2014). Accordingly, the resources have been classified as Measured, Indicated or Inferred, the reserves have been classified as Proven, and Probable based on the Measured and Indicated resources as defined below.

28.1 Mineral Resource

A **Mineral Resource** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

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.

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 the 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.

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 the 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.

28.2 Mineral Reserve

A **Mineral Reserve** is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at pre-feasibility or feasibility-level as appropriate that include the application of modifying factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a pre-feasibility study or feasibility study.

A **Probable Mineral Reserve** is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the modifying factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

A **Proven Mineral Reserve** is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the modifying factors.

28.3 Definition of Terms

Table 28-1 summarizes the general mining terms potentially used in this Technical Report.

Table 28-1: Definition of Terms

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	The initial process of reducing the ore particle size to render it more amenable for further processing.
Cut-Off Grade (CoG)	The grade of mineralized rock, which determines as to whether or not it is economical to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with ore.
Dip	The angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of an orebody or stope.
Gangue	Non-valuable components of the ore.
Grade	The measure of the concentration of gold within the mineralized rock.
Hanging wall	The overlying side of an orebody or slope.
Haulage	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimize the estimation error.
Level	A horizontal tunnel, the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
LRP	Long Range Plan.

Material Properties	Mine properties.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Assets	The Material Properties and Significant Exploration Properties.
Ongoing Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore reserve	See Mineral Reserve.
Pillar	Rock left behind to help support the excavations in an underground mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Sill	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high-temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or dolt phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope	The underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	The direction of the line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulphide	A sulphur-bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures, including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

28.4 Abbreviations

The following abbreviations may be used in this Technical Report.

Abbreviation	Unit or Term
A	ampere
AA	atomic absorption
$Ai_{rn}2$	amperes per square metre
ANFO	ammonium nitrate fuel oil
Au	gold
BATF	U.S. Bureau of Alcohol, Tobacco and Firearms
bgs	below ground surface
°C	degrees Celcius
CAA	Clean Air Act
CAPEX	capital expenditure
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
CoG	cut-off grade
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
cfm	cubic feet per minute
ConfC	confidence code
CRec	core recovery
CRC	Cultural Resources Consulting
CRM	certified reference material
CSS	closed-side setting
CSV	comma separated values
CTW	calculated true width
°	degree (degrees)
dia.	diameter
DOL	Department of Labor
DNR	Department of Natural Resources
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
EPA	U.S. Environmental Protection Agency
ft	foot (feet)

ft^2	square foot (feet)
ft^3	cubic foot (feet)
g	gram
g/cm^3	grams per cubic centimetre
gpd	gallons per day
g/t	grams per tonne
Ga	giga-annum (1 billion years)
gal	gallon
GHG	greenhouse gases
g/L	gram per litre
g-mol	gram-mole
gpm	gallons per minute
g/t	grams per tonne
>	greater than
ha	hectare (10,000 m^2)
HAP	hazardous air pollutant
HDPE	height density polyethylene
HG	high grade
High-Ti	high titanium basalt
hp	horsepower
HTW	horizontal true width
ICP	induced couple plasma
ID2	Inverse-Distance Squared
IFC	International Finance Corporation
ILS	intermediate leach solution
IRR	internal rate of return
kA	kiloamperes
kg	kilogram
kg/m^2	Kilogram per cubic metre
kg/m^3	Kilogram per square metre
km	kilometre
km^2	square kilometer
koz	thousand troy ounce
kt	thousand tonnes
kt/d	thousand tonnes per day
kt/y	thousand tonnes per year

kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
<	less than
L	litre
L/s	litres per second
L/s/m	litres per second per metre
LG	low grade
lb	pound
LHD	long-haul dump truck
LLDDP	linear low-density polyethylene plastic
LOI	loss on ignition
LOM	life of mine
m	metre
m^2	square metre
m^3	cubic metre
m^3/h	cubic metre per hour
masl	metres above sea level
Ma	mega-annum (1 million years)
MCL	maximum contaminant levels
MDA	Mine Development Associates
µm	micrometre per micron
µRads/hour	microradian/hour
mg/L	Milligrams per litre
M	million
MJ	megajoules
mm	millimetre
mm^2	square millimetre
mm^3	cubic millimetre
MME	mine & mill engineering
Moz	million troy ounces
MSHA	Mine Safety and Health Administration
Mt	million tonnes
Mtpa	Million tonnes per annum
MTW	measured true width

MW	million watts
MWMP	meteoric water mobility procedure
m.y.	million years
NDEQ	Nebraska Department of Environmental Quality
NORM	naturally occurring radioactive material
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
NN	Nearest Neighbour
NPDES	national pollutant discharge elimination system
NRCS	Natural Resources Conservation Service
OK	Ordinary Kriging
OP	open pit
OPEX	operating expense
opt	ounce per tonne
OSC	Ontario Securities Commission
oz	troy ounce
%	percent
%w/w	percent mass fraction for percent mass
PENN	Pennsylvanian-aged mudstone and limestone (Pennsylvanian strata)
pCi/g	picocuries per gram
PLC	programmable logic controller
PLS	pregnant leach solution
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
PSD	prevention of significant deterioration
QA/QC	quality assurance/quality control
RC	rotary circulation drilling
RO	reverse osmosis
ROM	run of mine
RPD	relative percentage difference
RQD	rock quality description
SEC	U.S. Securities & Exchange Commission
sec	second
SG	specific gravity
SOFC	solid oxide fuel cells

SPCC	spill prevention, control, and countermeasure
SPLP	synthetic precipitation leach procedure
SPT	standard penetration testing
st	short ton (2,000 pounds)
t	tonne (metric ton) (2,204.6 pounds)
TCLP	toxicity characteristic leaching procedure
t/m^3	tonnes per cubic metre
t/h	tonnes per hour
t/d	tonnes per day
t/y	tonnes per year
TSF	tailings storage facility
TSP	total suspended particulates
UG	underground
USACE	U.S. Army Corps of Engineers
UIC	underground injection control
USGS	United States Geological Survey
UTM	Universal Transverse Mercator
V	volts
VFD	variable frequency drive
W	watt
XRD	x-ray diffraction
y	year

Appendices

Appendix A: Certificates of Qualified Persons

CERTIFICATE OF QUALIFIED PERSON

I, Adrian Brown, P.E., P.Eng., MIEAust., of Denver, Colorado, USA do hereby certify:

1. I am the Principal Groundwater Engineer with Adrian Brown Consultants, Inc. with a business address at 130 West 4th Avenue, Denver, Colorado.
2. This certificate applies to the Technical Report entitled "NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the "Technical Report").
3. I am a graduate of Monash University, Victoria, Australia, with a Bachelor of Engineering with Honors (1967), a Master of Engineering Science (1970), and a Master of Administration (1974).
4. I am Licensed Professional Engineer in good standing in the following jurisdictions: State of Colorado (license 22215); State of Nebraska (license E-17078); Province of British Columbia (member 10,141); and a Member of the Institution of Engineers, Australia (number 34,903).
5. My relevant experience includes 52 years of experience in groundwater engineering, geotechnical engineering, and hydrogeochemistry in mining and industrial projects worldwide. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
6. My personal experience of the Elk Creek Property located in southeast Nebraska, USA, approximately 105 km (65 miles) southeast of Lincoln, Nebraska occurred between March 2018 and May 2019, and comprised review of aerial photography, drilling records, drill core photographs, drillhole geology records, groundwater test records, and geotechnical logging and test records. I have not visited the Elk Creek Project or inspected the undeveloped site.
7. I am responsible for groundwater control evaluation of Section 16.3, and the preparation of two supporting documents entitled "Elk Creek Niobium Mine Groundwater Control" (2018), and "Control of Inflow to the Elk Creek Mine by Grouting" (2019). In addition, I am responsible for input into Sections 1, 16, 20, 21, 24, 25, 26 and Appendix D as related to mine water control.
8. I am independent of NioCorp Developments Ltd., as defined by Section 1.5 of the Instrument.
9. I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
10. As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information relating to the Elk Creek Project that is required to be disclosed to make the Technical Report not misleading.
11. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Signed and dated this 29th day of May 2019, at Denver, Colorado, USA.

PROFESSIONAL CIVIL ENGINEER
ADRIAN
BROWN
E-17078
STATE OF NEBRASKA

Adrian Brown

Adrian Brown, P.E., P.Eng., MIEAust.
Principal Groundwater Engineer,
Adrian Brown Consultants, Inc.

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CERTIFICATE OF QUALIFIED PERSON

I, Chris Dougherty, P.Eng., of Thunder Bay, Ontario do hereby certify:

1. I am a Principal, Consulting Specialist and Civil Engineer with Nordmin Engineering Ltd. with a business address at 160 Logan Ave., Thunder Bay, Ontario.
2. This certificate applies to the Technical Report entitled "NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the "Technical Report").
3. I am a graduate of Lakehead University, 1994 with a Bachelor of Engineering in Civil Engineering, and of Cambrian College of Applied Arts and Technology with a Diploma in Civil Engineering Technology, 1991.
4. I am a member in good standing of the Association of Professional Engineers of Ontario and registered as a Professional Engineering, license number 90416876.
5. My relevant experience includes over 30 years of experience in the mining industry in the design and evaluation of mining infrastructure systems, including shafts, hoisting systems, backfill systems, materials handling and classification systems. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
6. I am responsible for Paste Backfill Plant and Underground Distribution Section 18.13, Freezing Plant Section 18.14, Taxes, Royalties and Other Interests Section 22.5 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
7. I am independent of NioCorp Developments Ltd., as defined by Section 1.5 of the Instrument.
8. I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information relating to the Elk Creek Project that is required to be disclosed to make the Technical Report not misleading.
10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Signed and dated this 29th day of May 2019, at Thunder Bay, Ontario.

"Original document signed and stamped by Chris Dougherty, P.Eng."

*Chris Dougherty*______________________________

Chris Dougherty, P.Eng.
Principal, Consulting Specialist and Civil Engineer
Nordmin Engineering Ltd.

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CERTIFICATE OF QUALIFIED PERSON

I, Glen Kuntz, P. Geo., of Thunder Bay, Ontario do hereby certify:

1. I am the Consulting Specialist – Geology/Mining with Nordmin Engineering Ltd. with a business address at 160 Logan Ave., Thunder Bay, Ontario.
2. This certificate applies to the Technical Report entitled “NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the “Technical Report”).
3. I am a graduate of the University of Manitoba, 1991 with a Bachelor of Science in Geology.
4. I am a member in good standing of the Association of Professional Geoscientist of Ontario and registered as a Professional Geoscientist, license number 0475.
5. My relevant experience includes 28 years of experience in exploration, operations and resource estimations. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).
6. My most recent personal inspection of the Property was November 11 - 12, 2018 inclusive. I visited the Elk Creek Project located in southeast Nebraska, USA, approximately 105 km (65 miles) southeast of Lincoln, Nebraska.
7. I am responsible for property, geology and resource Sections 4 through 12, 14, 16.1, 19, 23 and portions of Sections 1, 21, 22, 25 and 26 summarized within this Technical Report.
8. I am independent of NioCorp Developments Ltd., as defined by Section 1.5 of the Instrument.
9. I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
10. As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information relating to the Elk Creek Project that is required to be disclosed to make the Technical Report not misleading.
11. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Signed and dated this 29th day of May 2019, at Thunder Bay, Ontario.

“Original document signed and stamped by Glen Kuntz, P.Geo.”

*Glen Kuntz*______________________________

Glen Kuntz, P.Geo.
Consulting Specialist – Geology/Mining
Nordmin Engineering Ltd.

SRK Consulting (U.S.), Inc.
Suite 600
1125 Seventeenth Street
Denver, CO 80202

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Joshua D. Sames, P.E. Civil, B.Sc., do hereby certify that:

1. I am Senior Consultant at SRK Consulting (U.S.), Inc., 5250 Neil Road, Suite 300, Reno, NV, USA 89502.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the "Technical Report").
3. I graduated with a degree in Civil Engineering from University of Newcastle Australia in 2005. I am a registered Professional Engineer in the State of Nevada (PE No. 22346). I have worked as an engineer for a total of 13 years. My relevant experience includes site investigations, conceptual and detailed design of tailing storage facilities, construction supervision, management and operational assessments, mine reclamation permitting and closure design and permitting at mining properties in the western United States and South and Central America.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Elk Creek Project property, but have reviewed maps, photos and Google Earth street views with another qualified professional from SRK Consulting who has visited the project site.
6. I am responsible for the preparation of earthworks, tailings and salt management Sections 18.8, 18.10 through 18.12, 21.2.4, 21.1.4.1, 21.3.1.3, 21.3.2.3 of the Technical Report and portions of Sections 1, 22, 25 and 26 summarized within this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 29th day of May 2019, at Reno, Nevada, U.S.A.

"Original document signed and stamped by Joshua D. Sames"

Joshua D. Sames

Joshua D. Sames P.E
SRK Consulting (U.S.), Inc.

CERTIFICATE OF QUALIFIED PERSON

Sylvain Harton, P. ENG.
Address: 356 Luzern Rd, Midway, Utah, United States, 84049
Telephone: +1 (435) 776.6207
Email: sylvain.harton@MCS-engineer.net
Professional Engineer Ontario: #100128046

I, Sylvain Harton, P. Eng. do hereby certify that:

1. I am the President of Metallurgy Concept Solution, which is located at the same address than the one mentioned above.

   "NI 43-101 TECHNICAL REPORT FEASIBILITY STUDY, ELK CREEK SUPERALLOY MATERIALS PROJECT, NEBRASKA" of NORDMIN ENGINEERING LTD. effective date of April 16, 2019 and report date of May 28, 2019.

2. I graduated with a Bachelor in Engineering, Metallurgy Processing in 1993 at the ***École Polytechnique de Montréal*** and I have practiced the profession of metallurgy since my graduation. After being a research associate at the École Polytechnique de Montréal, I have been employed with Noranda from 1994 to 2004 as process and production Engineer in several operation departments. Afterwards, I joined engineering firms as a senior process engineer working in the field of the metallurgy extractive with, an emphasis in pyrometallurgy. I have a total of 27 years of experience in the metal transformation industry, including a background in processing and material engineering.

3. I graduated with a Bachelor in Engineering, Metallurgy Processing in 1993 at the ***École Polytechnique de Montréal*** and I have practiced the profession of metallurgy since my graduation. After being a research associate at the École Polytechnique de Montréal, I have been employed with Noranda from 1994 to 2004 as process and production Engineer in several operation departments. Afterwards, I joined engineering firms as a senior process engineer working in the field of the metallurgy extractive with, an emphasis in pyrometallurgy. I have a total of 27 years of experience in the metal transformation industry, including a background in processing and material engineering.

4. I am a Professional Engineer (P. Eng.) registered with the Association of Professional Engineer Ontario in Canada (#100128046).

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.5.

6. I am a co-author of the technical report entitled "**NI 43-101 TECHNICAL REPORT FEASIBILITY STUDY, ELK CREEK SUPERALLOY MATERIALS PROJECT, NEBRASKA**", for Nordmin Engineering LTD and I am responsible for the Pyromet process part discussed in Sections number 13, 17, 24, 25 and 26 in the study.

7. I am independent of issuer applying all of the tests done at KPM company related to the Nb_2O_5 aluminothermic testing realized in 2017.

8. I have had no prior involvement with the NioCorp project that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.

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

Effective Date: April 16, 2019

Signed Date: May 29, 2019

CERTIFICATE OF QUALIFIED PERSON

I, Eric Larochelle, BEng, do hereby certify that:

1. I am Owner, President of SMH Process Innovation, 2392 High Mountain Dr, Sandy, UT 84092.
2. This certificate applies to the Technical Report entitled “NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the “Technical Report”).
3. I graduated with a degree in Bachelor of Chemical Engineering from McGill University in 1990. I have worked as a Chemical Engineering for a total of 27 years since my graduation from university. My relevant experience includes numerous process development in the Magnesium, Titanium, Rare Earths industries.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the Elk Creek property on 24 October, 2014
6. I am responsible for mineral processing and metallurgical testing Sections 13.2, 17.1.2, 17.1.4, 17.2.2, 17.2.4, 17.3.2, 17.3.4, 17.4.2, 17.4.4, 17.5.2, 17.5.4, 17.6.3, 17.7.2 and 17.7.3, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is Revised NI 43-101 Technical Report, Feasibility Study, Elk Creek Niobium Project, Nebraska” with an Effective Date of June 30, 2017 .
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 29th day of May 2019, at Salt Lake City, Utah.

“Original document signed and stamped by Eric Larochelle, B.Eng.”

Eric Larochelle, BEng
SMH Director, Specialty Metals & Hydrometallurgy

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CERTIFICATE OF QUALIFIED PERSON

I, Gregory Menard, P.Eng., of Thunder Bay, Ontario do hereby certify:

1. I am a Project Manager and Senior Mechanical Engineer with Nordmin Engineering Ltd. with a business address at 160 Logan Ave., Thunder Bay, Ontario.
2. This certificate applies to the Technical Report entitled "NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the "Technical Report").
3. I am a graduate of Lakehead University, 1994 with a Bachelor of Engineering in Mechanical Engineering.
4. I am a member in good standing of the Association of Professional Engineers of Ontario and registered as a Professional Engineer, license number 90459355.
5. My relevant experience includes 23 years of experience in heavy industrial engineering, specifically in the areas of mining operations and development. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
6. I am responsible for Infrastructure Sections 16.4.5, 16.6.3, 16.8.2 through to 16.8.4, 16.8.7 through to 16.8.9, 16.8.12 through to 16.8.15, 18.1, 18.2, 18.3, 18.12, 24 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
7. I am independent of NioCorp Developments Ltd., as defined by Section 1.5 of the Instrument.
8. I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information relating to the Elk Creek Project that is required to be disclosed to make the Technical Report not misleading.
10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Signed and dated this 29th day of May 2019, at Thunder Bay, Ontario.

"Original document signed and stamped by Greg Menard, P.Eng."

*Gregory Menard*________________________________

Gregory Menard, P.Eng.
Project Manager and Senior Mechanical Engineer
Nordmin Engineering Ltd.

ZACHRY

CERTIFICATE OF QUALIFIED PERSON

I, Orest J. Romaniuk, P. Eng., of Maryland Heights, MO do hereby certify:

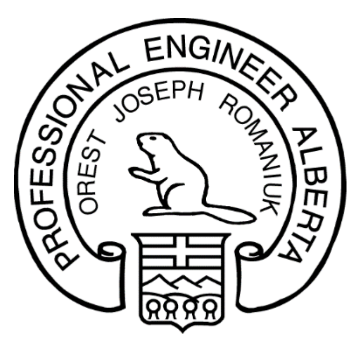
1. I am the Senior Engineer – Process Department with Zachry Engineering Corporation with a business address at 11885 Lackland Road, Suite 601, Maryland Heights, MO 63146.
2. This certificate applies to the technical report entitled “NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the “Technical Report”).
3. I am a graduate of the University of Alberta, 1981 with a Bachelor of Mineral Process Engineering.
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of Alberta (APEGA) and registered as a Professional Engineer, license number 32910.
5. My relevant experience includes 38 years of experience in plant operations, design engineering and process equipment testing and specification which including 20 years in mineral processing. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).
6. I have not visited the Elk Creek Superalloy Materials Project property located in southeast Nebraska, USA.
7. I am responsible for Sections 13, 17.1.1, 17.2.1, 17.3.1 and 17.4.1 and portions of Sections 1, 25, and 26 of the Technical Report.
8. I am independent of NioCorp Developments Ltd., as defined by Section 1.5 of the Instrument.
9. I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
10. As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information relating to the Elk Creek Superalloy Materials Project that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 29th day of May, 2019, at Maryland Heights, MO.

“Original document signed and stamped by Orest J. Romaniuk, P.Eng.”

Orest J. Romaniuk

Orest J. Romaniuk
Senior Engineer
Process Department



1121 North 102nd Court, Suite 301
Omaha, NE 68114
O: 402.548.4800 I F: 402.548.4999
www.zachrygroup.com

CERTIFICATE OF QUALIFIED PERSON

I, Jean-Francois, P. Eng., of Mississauga, Ontario do hereby certify:

1. I am the VP Mining for Optimize Group Inc. with a business address at 145 Wellington Street West, Suite 1001, Toronto, Ontario. I have worked as an associate Consulting Specialist Mine Engineer for Nordmin for the Feasibility Study described below.
2. This certificate applies to the Technical Report entitled "NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the "Technical Report").
3. I am a graduate of the Laval University of Quebec, 1993 with a Bachelor of Applied Science in Mining engineering.
4. I am a member in good standing of the Professional Engineers of Ontario and Ordre des Ingenieurs du Quebec, license number PEO 100215849 and OIQ 111717.
5. My relevant experience includes 26 years of experience in operations, mine technical engineering and consulting with Underground hard rock mines including reserve estimations. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
6. My most recent personal inspection of the Property was November 11 - 12, 2018 inclusive. I visited the Elk Creek Project located in southeast Nebraska, USA, approximately 105 km (65 miles) southeast of Lincoln, Nebraska.
7. I am responsible for mine design and reserve Sections 15 and portions of Sections 1, 16.4 to 16.8, 21, 22, 25 and 26 summarized within this Technical Report.
8. I am independent of NioCorp Developments Ltd., as defined by Section 1.5 of the Instrument.
9. I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
10. As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information relating to the Elk Creek Project that is required to be disclosed to make the Technical Report not misleading.
11. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Signed and dated this 29th day of May 2019, at Ouarzazate, Morocco.

"Original document signed and stamped by Jean-Francois St-Onge, P.Eng."

*Jean-Francois St-Onge*____________________

Jean-Francois St-Onge, P.Eng.
VP Mining
Optimize Group Inc.

SRK Consulting (U.S.), Inc.
Suite 600
1125 Seventeenth Street
Denver, CO 80202

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, John Tinucci, PhD, PE, ISRM, do hereby certify that:

1. I am a Principal Geotechnical Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report, Feasibility Study, Elk Creek Niobium Project, Nebraska” with an Effective Date of April 16, 2019 (the “Technical Report”).
3. I graduated with a degree in B.S. in Civil Engineering from Colorado State University, in 1980. In addition, I have obtained a M.S. in Geotechnical Engineering from University of California, Berkeley, in 1983 and I have obtained a Ph.D. in Geotechnical Engineering, Rock Mechanics from the University of California, Berkeley in 1985. I am member of the American Rock Mechanics Association, a member of the International Society of Rock Mechanics, a member of the ASCE GeoInstitute, and a Registered Member of the Society for Mining, Metallurgy & Exploration. I have worked as a Mining and Geotechnical Engineer for a total of 31 years since my graduation from university. My relevant experience includes 34 years of professional experience. I have 15 years managerial experience leading project teams, managing P&L operations for 120 staff, and directed own company of 8 staff for 8 years. I have technical experience in mine design, prefeasibility studies, feasibility studies, geomechanical assessments, rock mass characterization, project management, numerical analyses, underground mine stability, subsidence, tunneling, ground support, slope design and stabilization, excavation remediation, induced seismicity and dynamic ground motion. My industry commodities experience includes salt, potash, coal, platinum/palladium, iron, molybdenum, gold, silver, zinc, diamonds, and copper. My mine design experience includes open pit, room and pillar, (single and multi-level), conventional drill-and-blast and mechanized cutting, longwall, steep narrow vein, cut and fill, block caving, sublevel caving and cut and fill longhole stoping and paste backfilling.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I have not visited the Elk Creek property.
6. I am responsible for preparation of the geotechnical Section 16.3 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is in the preparation of the report titled, “Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska,” with an Amended Report Date of October 16, 2015.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th Day of May, 2019.

"Signed" "Sealed"

John Tinucci, PhD, PE, ISRM

SRK Consulting (U.S.), Inc.
5250 Neil Road, Suite 300
Reno, NV 89502

T: (775) 828-6800
F: (775) 828-6820
reno@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Mark Allan Willow, M.Sc., NV-CEM, SME-RM, do hereby certify that:

1. I am Principal Environmental Consultant and Practice Leader of SRK Consulting (U.S.), Inc., 5250 Neal Road, Ste, 300, Reno Nevada 89502-6568.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the "Technical Report").
3. I graduated with Bachelor's of Science degree in Fisheries and Wildlife Management from the University of Missouri – Columbia in 1987 and a Master's of Science degree in Environmental Science and Engineering from the Colorado School of Mines in 1995. I have worked as Biologist/Environmental Scientist for over 25 years since my first graduation from university. My relevant experience includes environmental due diligence/competent persons' evaluations of developmental phase and operational phase mines throughout the world, including small gold mining projects in Panama, Senegal, Peru, Ecuador, Philippines, Brazil, and Colombia; open pit and underground coal mines in Russia; several large copper and iron mines and processing facilities in Mexico and Brazil; bauxite operations in Jamaica; and a coal mine/coking operation in China. My Project Manager experience includes oversight of several site baseline characterization and mine closure projects. I draw upon my diverse background for knowledge and experience as a human health and ecological risk assessor with respect to potential environmental impacts associated with operating and closing mining properties, and have experienced in the development of Preliminary Remediation Goals and hazard/risk calculations for site remedial action plans according to U.S. EPA risk assessment guidance.

   I am a Registered Member (No. 4104492) of the Society for Mining, Metallurgy & Exploration Inc. (SME).

   I am a Certified Environmental Manager (CEM) in the State of Nevada (#1832) in accordance with Nevada Administrative Code NAC 459.970 through 459.9729. Before any person consults for a fee in matters concerning: the management of hazardous waste; the investigation of a release or potential release of a hazardous substance; the sampling of any media to determine the release of a hazardous substance; the response to a release or cleanup of a hazardous substance; or the remediation soil or water contaminated with a hazardous substance, they must be certified by the Nevada Division of Environmental Protection, Bureau of Corrective Action.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Elk Creek Project property on June 1, 2015 for two days.

6. I am the QP responsible for Environmental, Permitting and Social or Community Impact Section 20 and portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is in the preparation of the report titled, "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska," with an Amended Report Date of October 16, 2015.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 29th day of May 2019, at Reno, Nevada, U.S.A.

"Original document signed and stamped by Mark A. Willow"

*Mark A. Willow*________________________________

Mark A. Willow, M.Sc., NV-CEM, SME-RM
Principal Environmental Scientist
SRK Consulting (U.S.), Inc.

CERTIFICATE OF QUALIFIED PERSON

I, David R Winters, P.E., S.E., MBA, of Tetra Tech, Inc., Salt Lake City, UT, USA, do hereby certify:

1. I am the Consulting Specialist – Civil/Structural with Tetra Tech, Inc. with a business address at 4750 W 2100 S, Suite, 400, Salt Lake City, UT, USA.
2. This certificate applies to the Technical Report entitled "NI 43-101 Technical Report, Feasibility Study, Elk Creek Superalloy Materials Project, Nebraska with an Effective Date of April 16, 2019 (the "Technical Report").
3. I am a graduate of Lehigh University, Bethlehem, PA, 1978 with a Bachelor of Science in ESRM – Geology, Georgia Institute of Technology, Atlanta, GA, 1982, Bachelor of Science in Civil/Structural Engineering and University of Utah, Salt Lake City, UT, 2004, Masters Business Administration.
4. I am a member in good standing of the American Society of Civil Engineers, with Professional Engineering (or Structural Engineering) certificates in UT, PA, AZ, IN, WI, MN, HI. Licenses pending in NE, NV, ND, SD.
5. My relevant experience includes over 37 years in the mining and oil & gas industries in operations, design, facility management, process, project management, construction management, and project estimation. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
6. I am responsible for Civil, Structural and Infrastructure Sections 17.5.1, 17.6.1, 17.6.2, 17.7 (intro), 17.7.1, 18.4, 18.5.1 through 18.5.4, 18.6, 18.9 along with portions of Sections 1, 21, 25 and 26 summarized within this Technical Report.
7. I am independent of NioCorp Developments Ltd., as defined by Section 1.5 of the Instrument.
8. I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information relating to the Elk Creek Project that is required to be disclosed to make the Technical Report not misleading.
10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

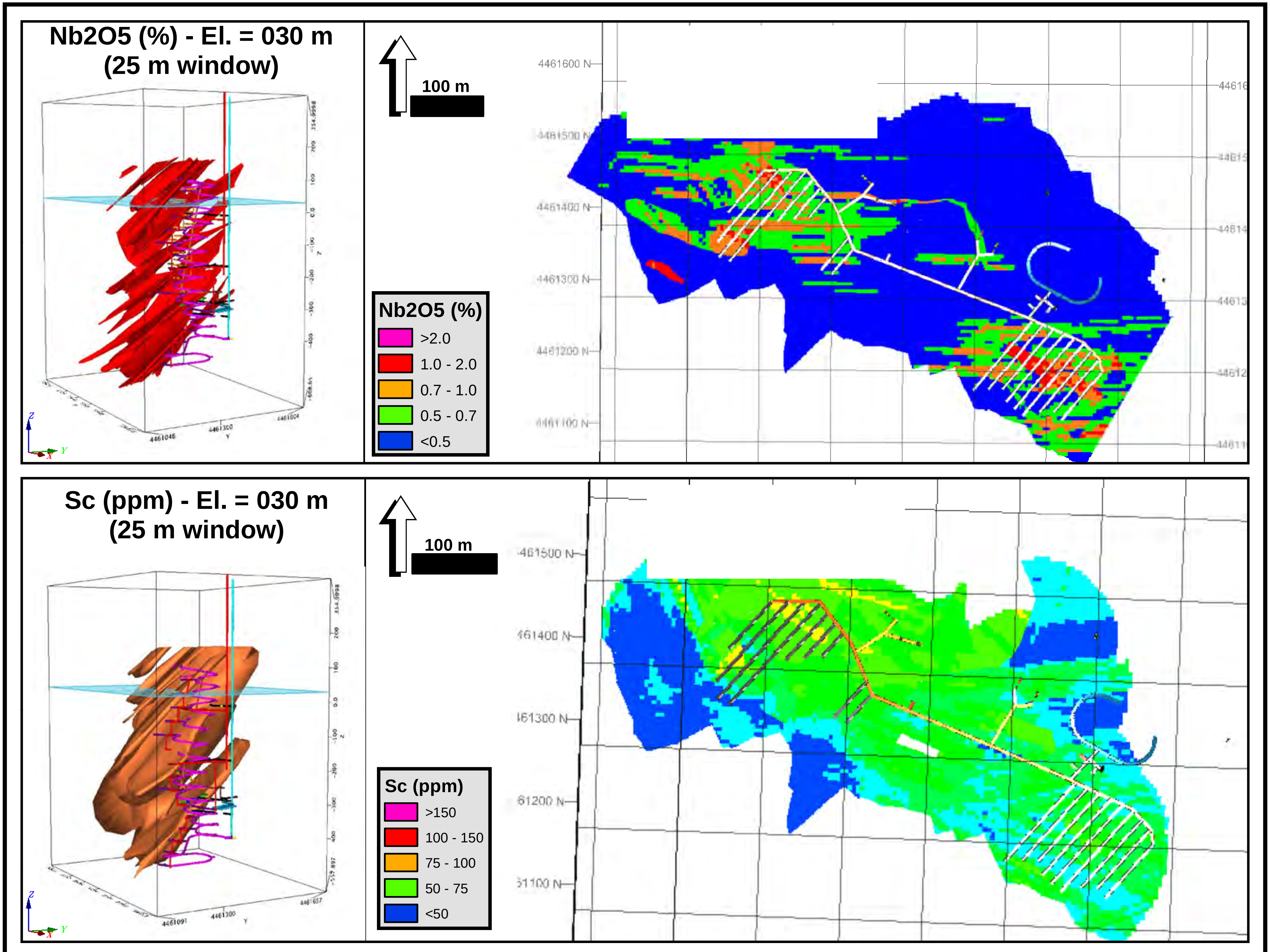
Signed and dated this 29th day of May 2019, at Salt Lake City, UT, USA.

"Original document signed and stamped by David R Winters, P.E., S.E., MBA."

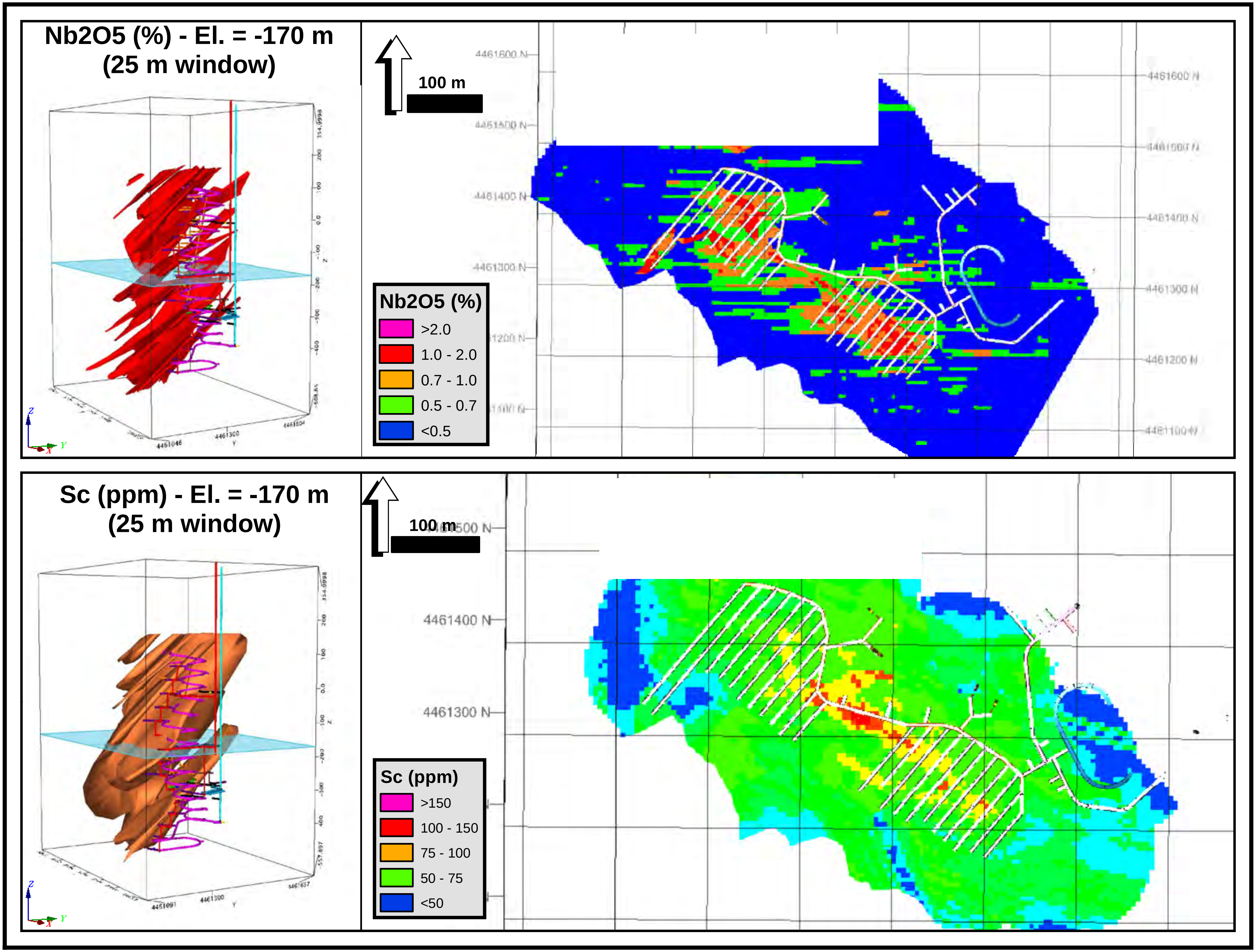
*David R Winters*______________________________

David R Winters, P.E., S.E., MBA
Consulting Specialist – Civil/Structural
Tetra Tech, Inc.

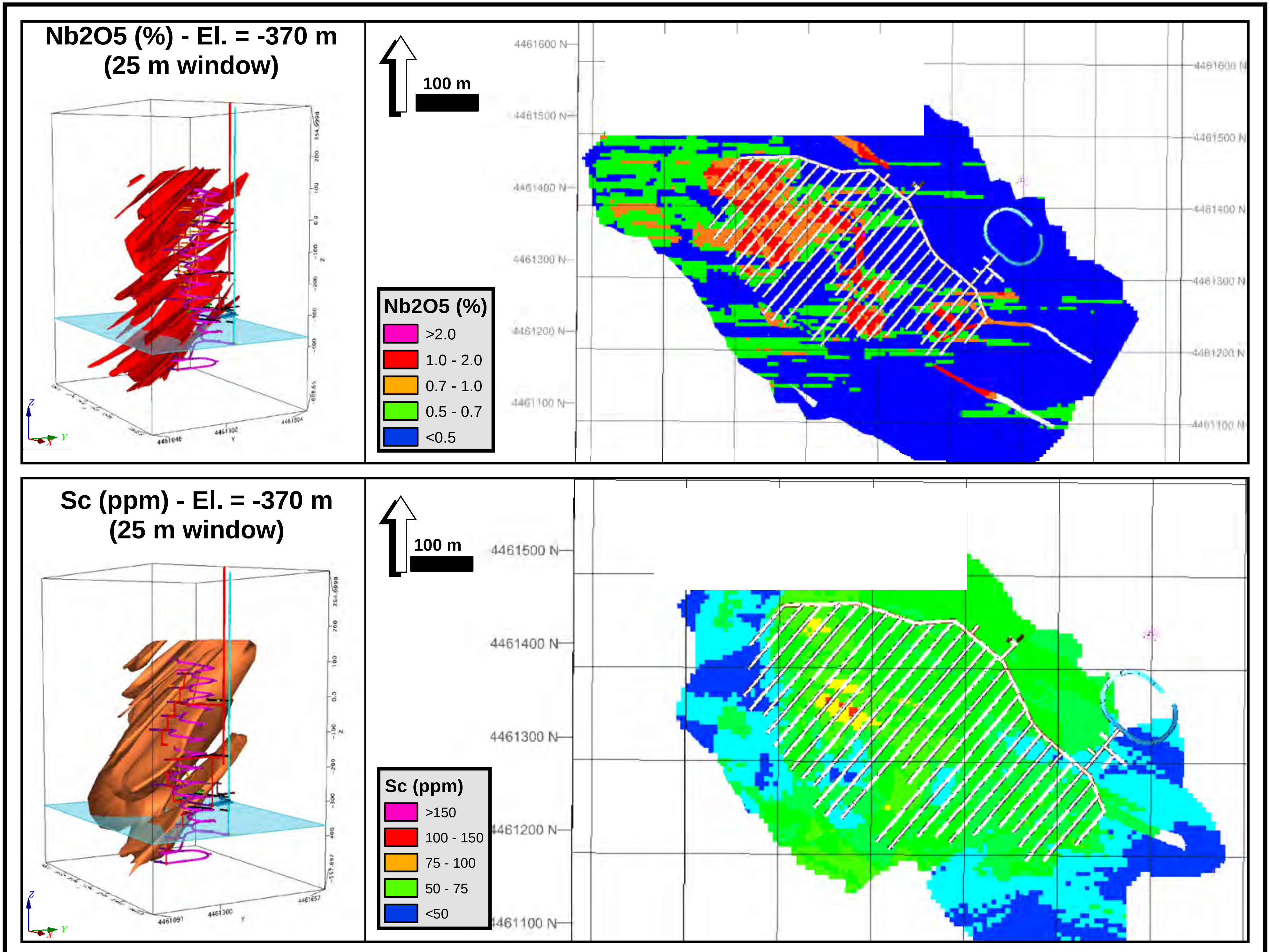
Appendix B: Plan View and Cross Sections

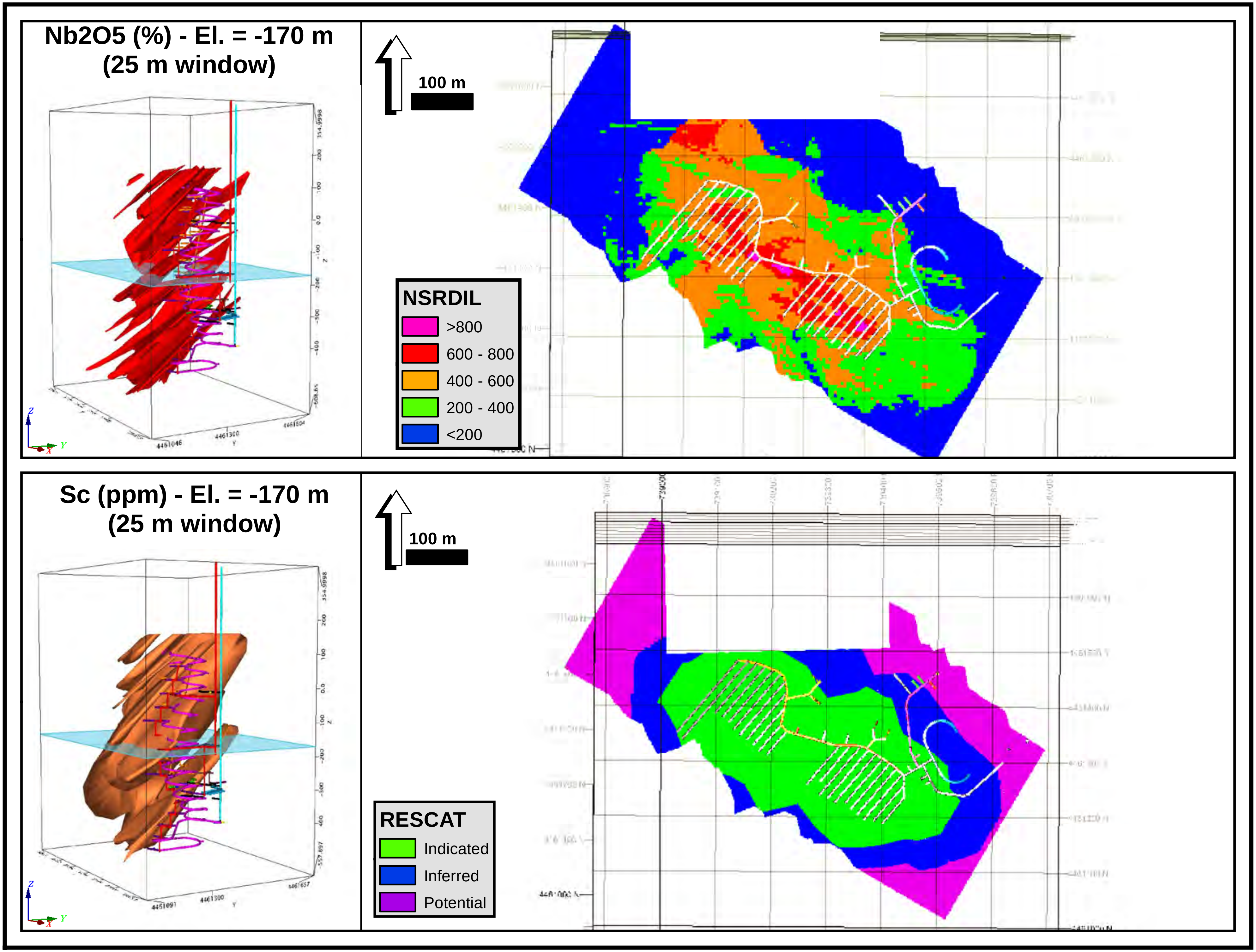


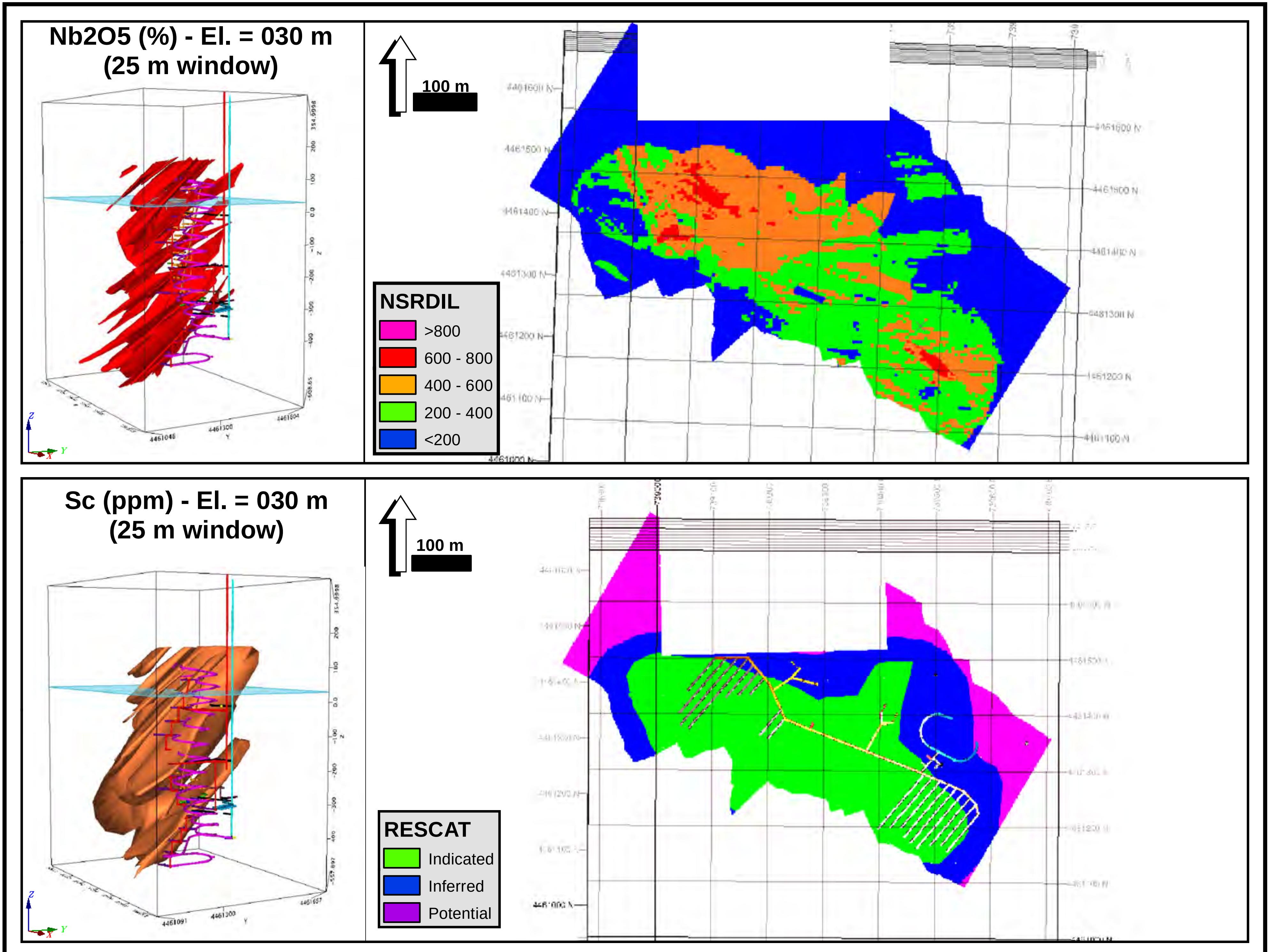
Nb2O5 (%) - El. = 030 m
(25 m window)
100 m
Nb2O5 (%)
>2.0
1.0 - 2.0
0.7 - 1.0
0.5 - 0.7
<0.5
4461600 N
4461500 N
4461400 N
4461300 N
4461200 N
4461100 N
Sc (ppm) - El. = 030 m
(25 m window)
100 m
Sc (ppm)
>150
100 - 150
75 - 100
50 - 75
<50



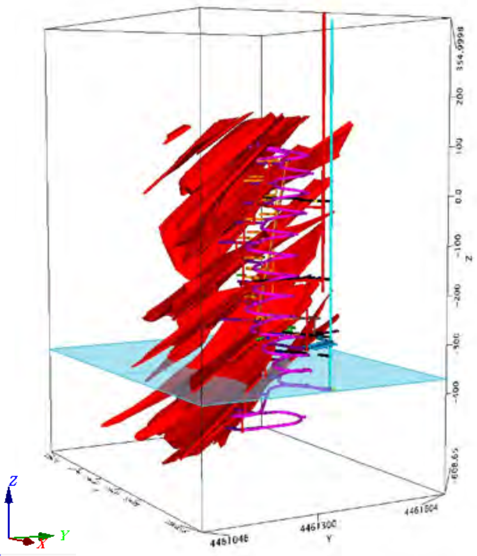
Nb2O5 (%) - El. = -170 m
(25 m window)
100 m
4461600 N
4461500 N
4461400 N
4461300 N
4461200 N
4461100 N
Nb2O5 (%)
>2.0
1.0 - 2.0
0.7 - 1.0
0.5 - 0.7
<0.5
Sc (ppm) - El. = -170 m
(25 m window)
100 m
4461500 N
4461400 N
4461300 N
Sc (ppm)
>150
100 - 150
75 - 100
50 - 75
<50

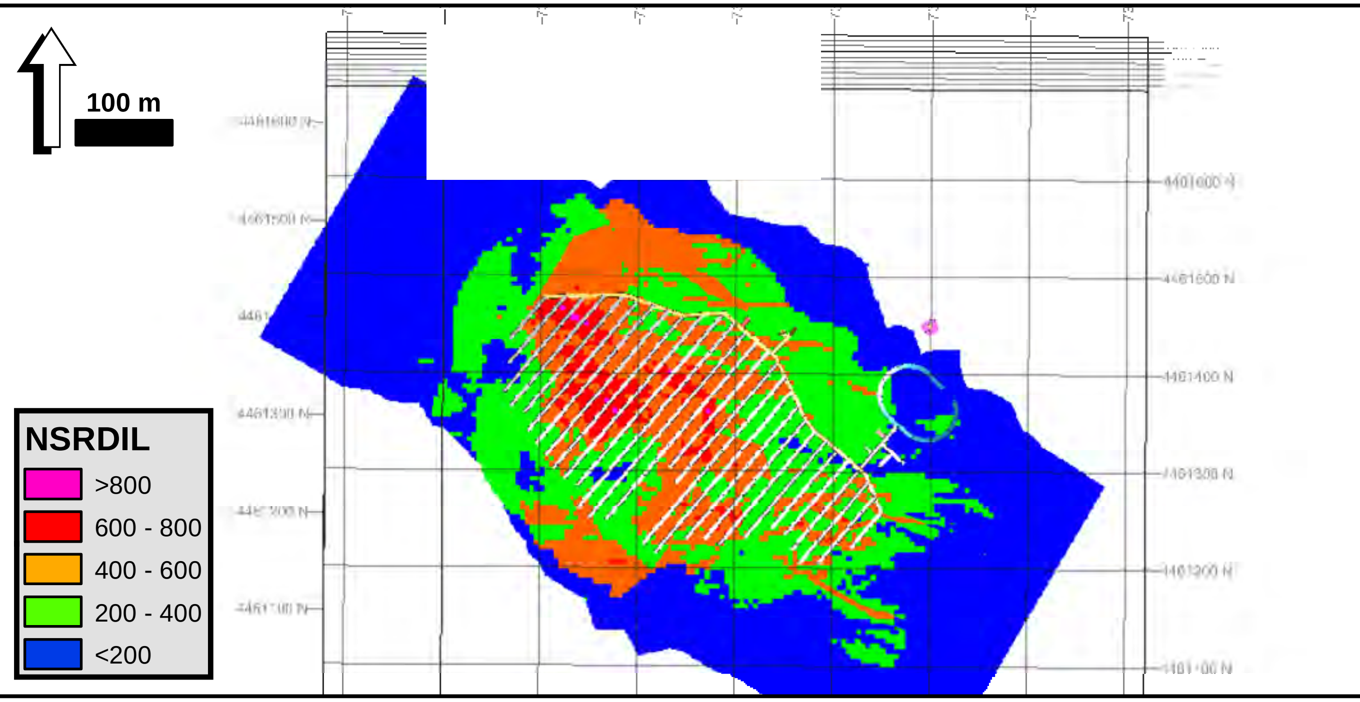

Nb2O5 (%) - El. = -370 m
(25 m window)
100 m
Nb2O5 (%)
>2.0
1.0 - 2.0
0.7 - 1.0
0.5 - 0.7
<0.5
4461600 N
4461500 N
4461400 N
4461300 N
4461200 N
4461100 N
Sc (ppm) - El. = -370 m
(25 m window)
100 m
Sc (ppm)
>150
100 - 150
75 - 100
50 - 75
<50
4461500 N
4461400 N
4461300 N
4461200 N
4461100 N


Nb2O5 (%) - El. = -170 m
(25 m window)
100 m
NSRDIL
>800
600 - 800
400 - 600
200 - 400
<200
Sc (ppm) - El. = -170 m
(25 m window)
100 m
RESCAT
Indicated
Inferred
Potential

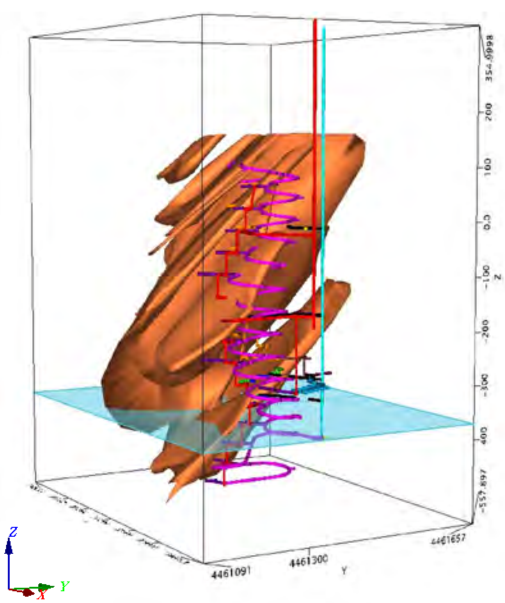

Nb2O5 (%) - El. = 030 m
(25 m window)
100 m
NSRDIL
>800
600 - 800
400 - 600
200 - 400
<200
Sc (ppm) - El. = 030 m
(25 m window)
100 m
RESCAT
Indicated
Inferred
Potential

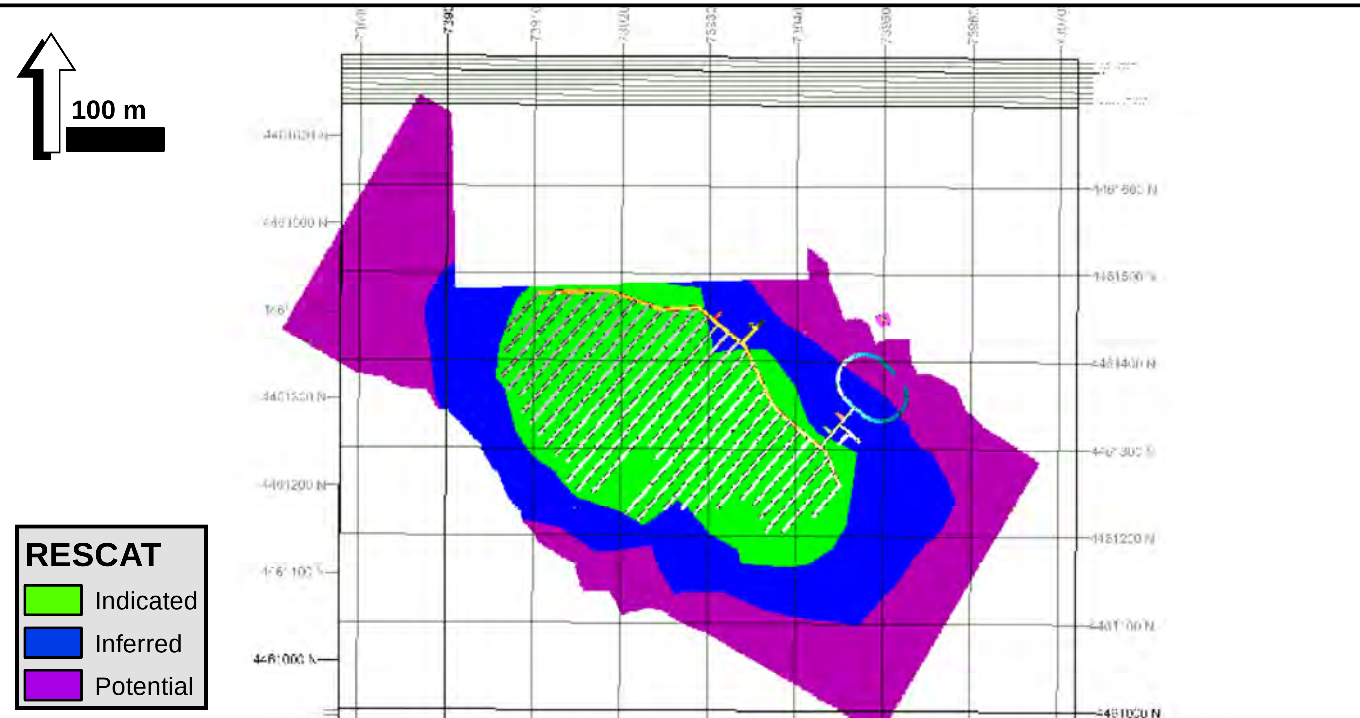
Nb2O5 (%) - El. = -370 m
(25 m window)

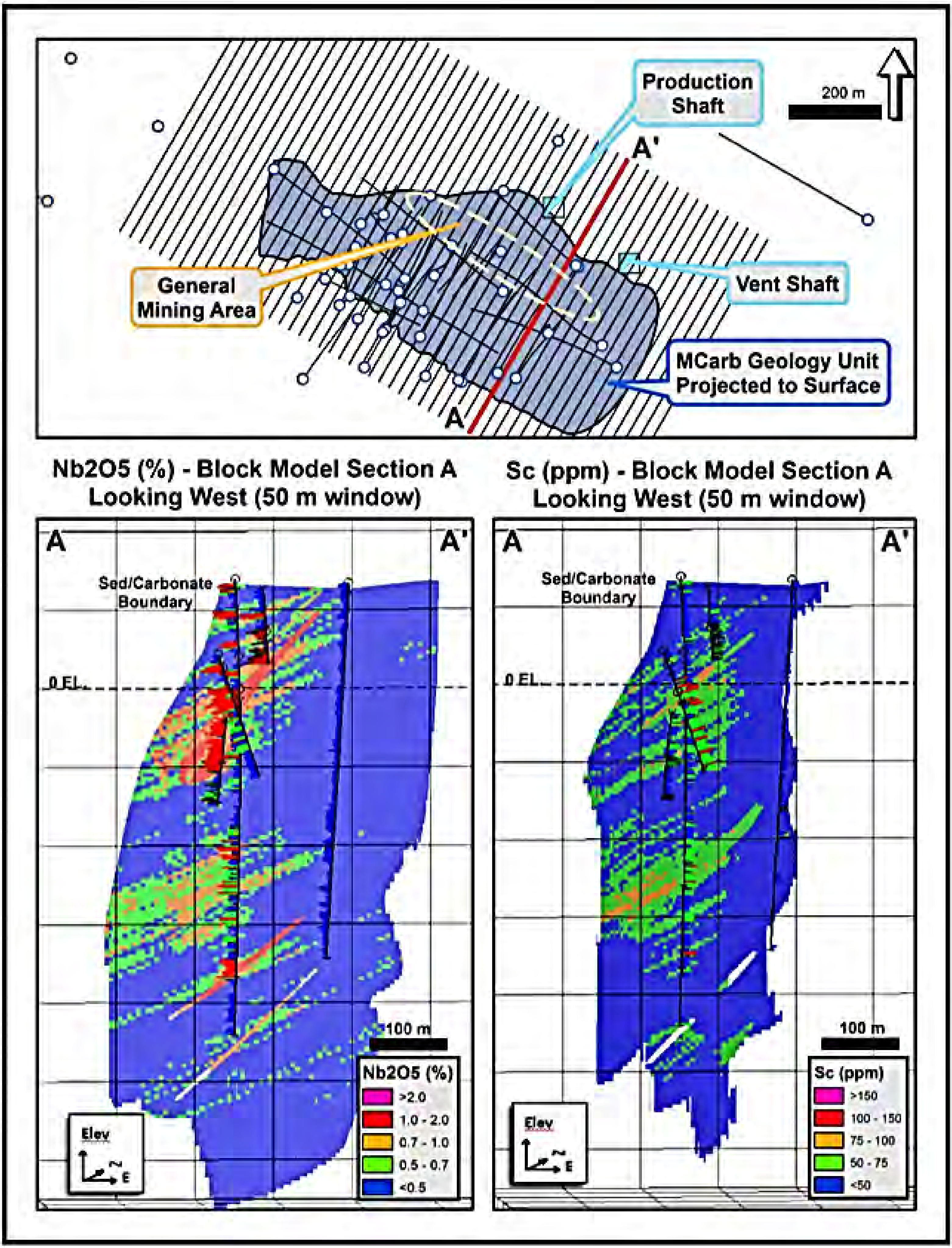


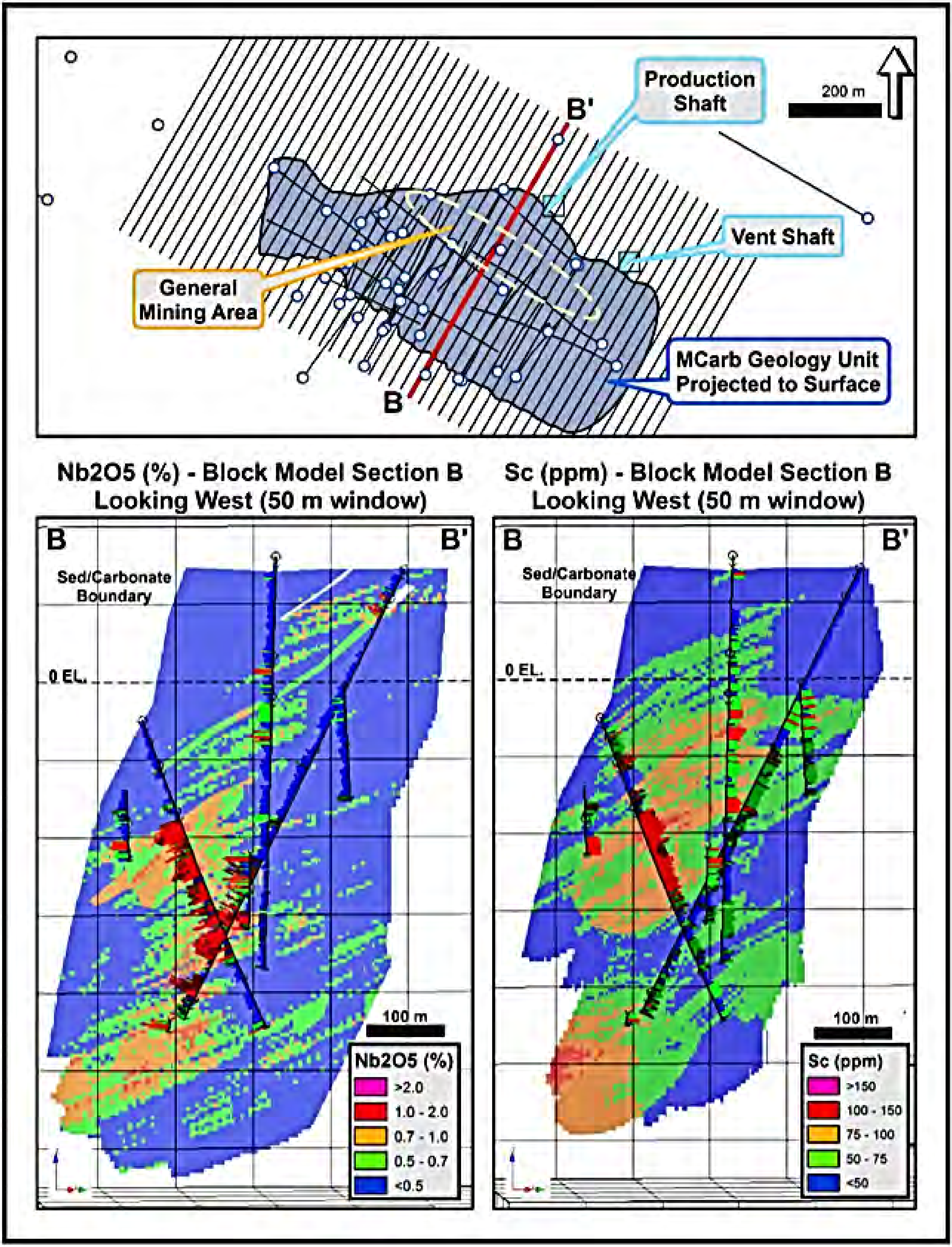


Sc (ppm) - El. = -370 m
(25 m window)

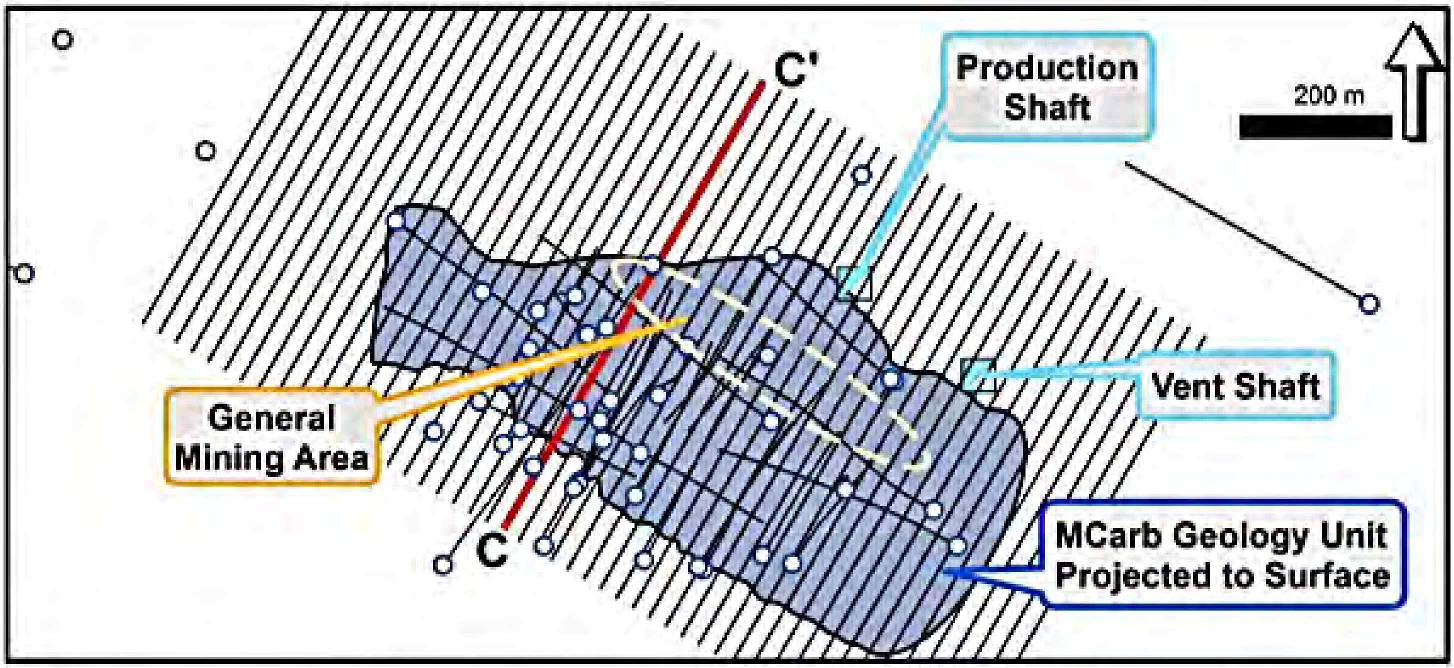




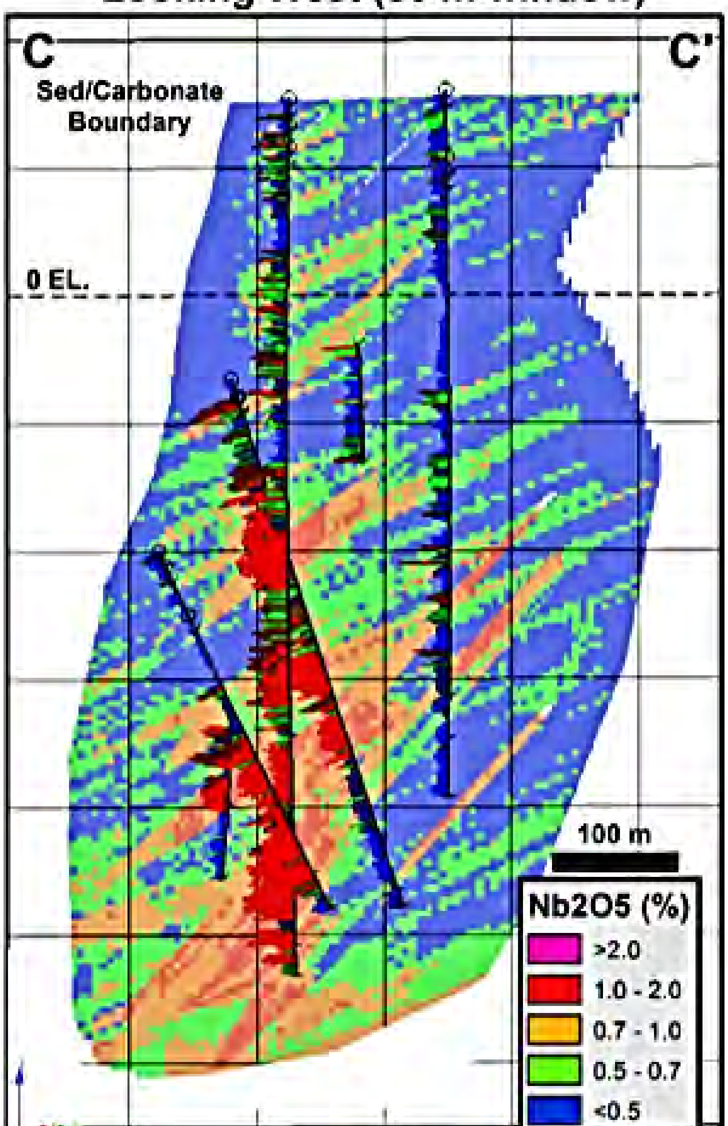

Production Shaft
200 m
A'
General Mining Area
Vent Shaft
MCarb Geology Unit Projected to Surface
A
Nb2O5 (%) - Block Model Section A Looking West (50 m window)
A
A'
Sed/Carbonate Boundary
0 EL.
100 m
Nb2O5 (%)
>2.0
1.0 - 2.0
0.7 - 1.0
0.5 - 0.7
<0.5
Elev
E
Sc (ppm) - Block Model Section A Looking West (50 m window)
A
A'
Sed/Carbonate Boundary
0 EL.
100 m
Sc (ppm)
>150
100 - 150
75 - 100
50 - 75
<50
Elev
E



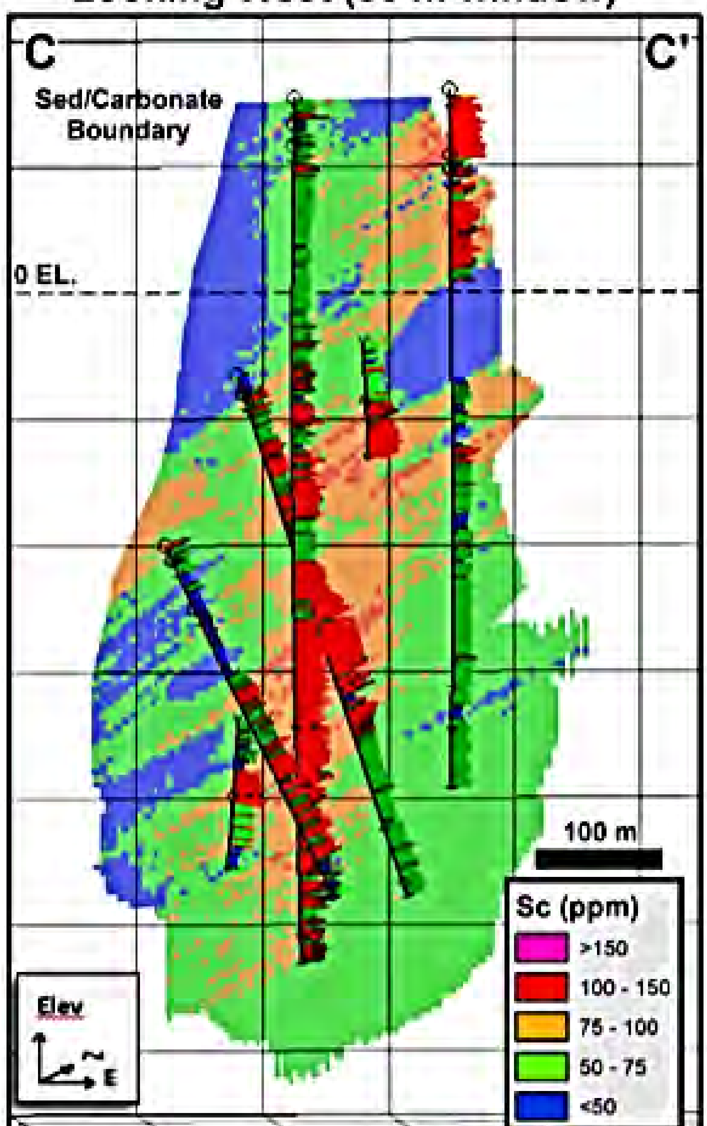
Production Shaft
200 m
B'
Vent Shaft
General Mining Area
MCarb Geology Unit Projected to Surface
B
Nb2O5 (%) - Block Model Section B Looking West (50 m window)
Sc (ppm) - Block Model Section B Looking West (50 m window)
B
B'
Sed/Carbonate Boundary
0 EL.
100 m
Nb2O5 (%)
>2.0
1.0 - 2.0
0.7 - 1.0
0.5 - 0.7
<0.5
B
B'
Sed/Carbonate Boundary
0 EL.
100 m
Sc (ppm)
>150
100 - 150
75 - 100
50 - 75
<50

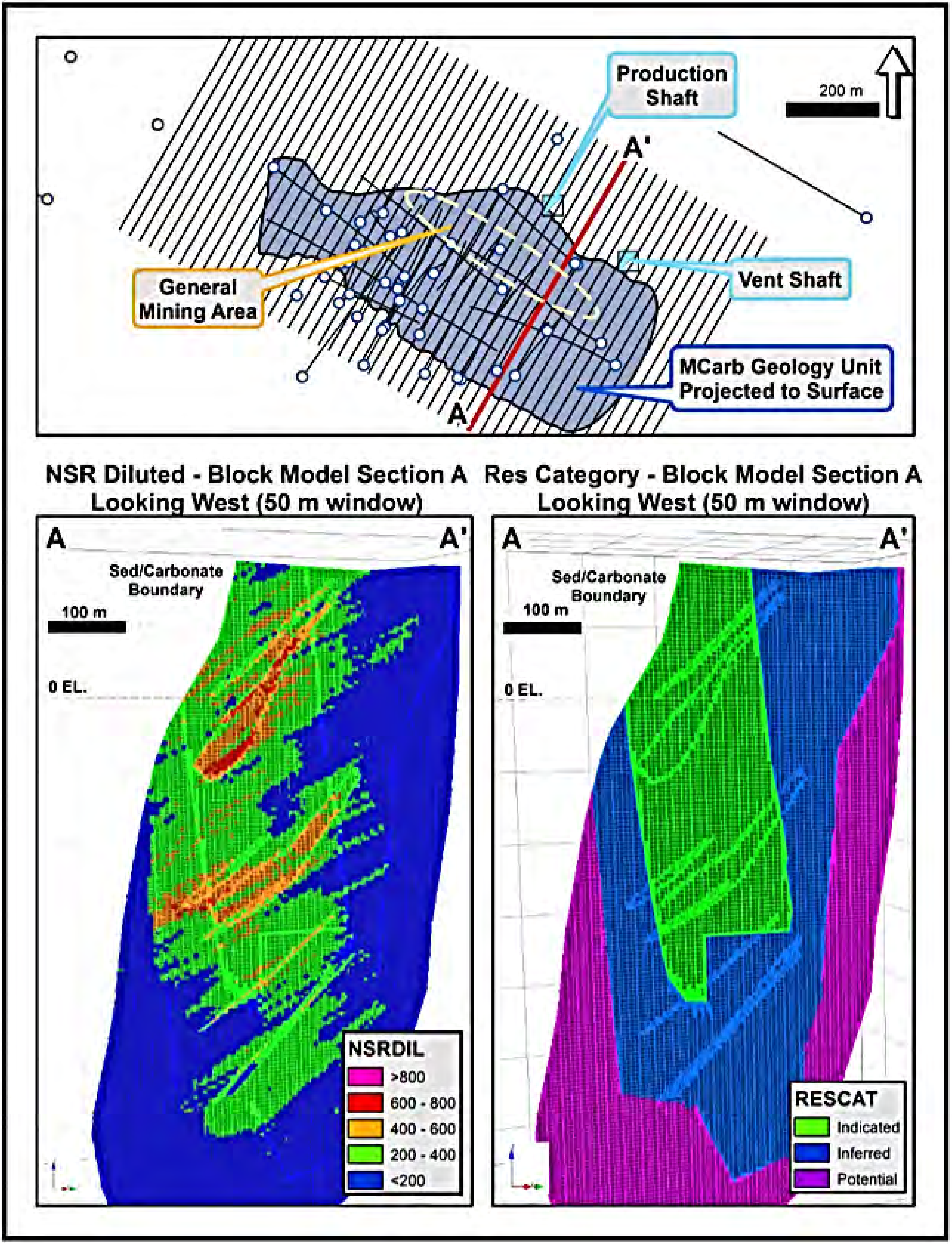


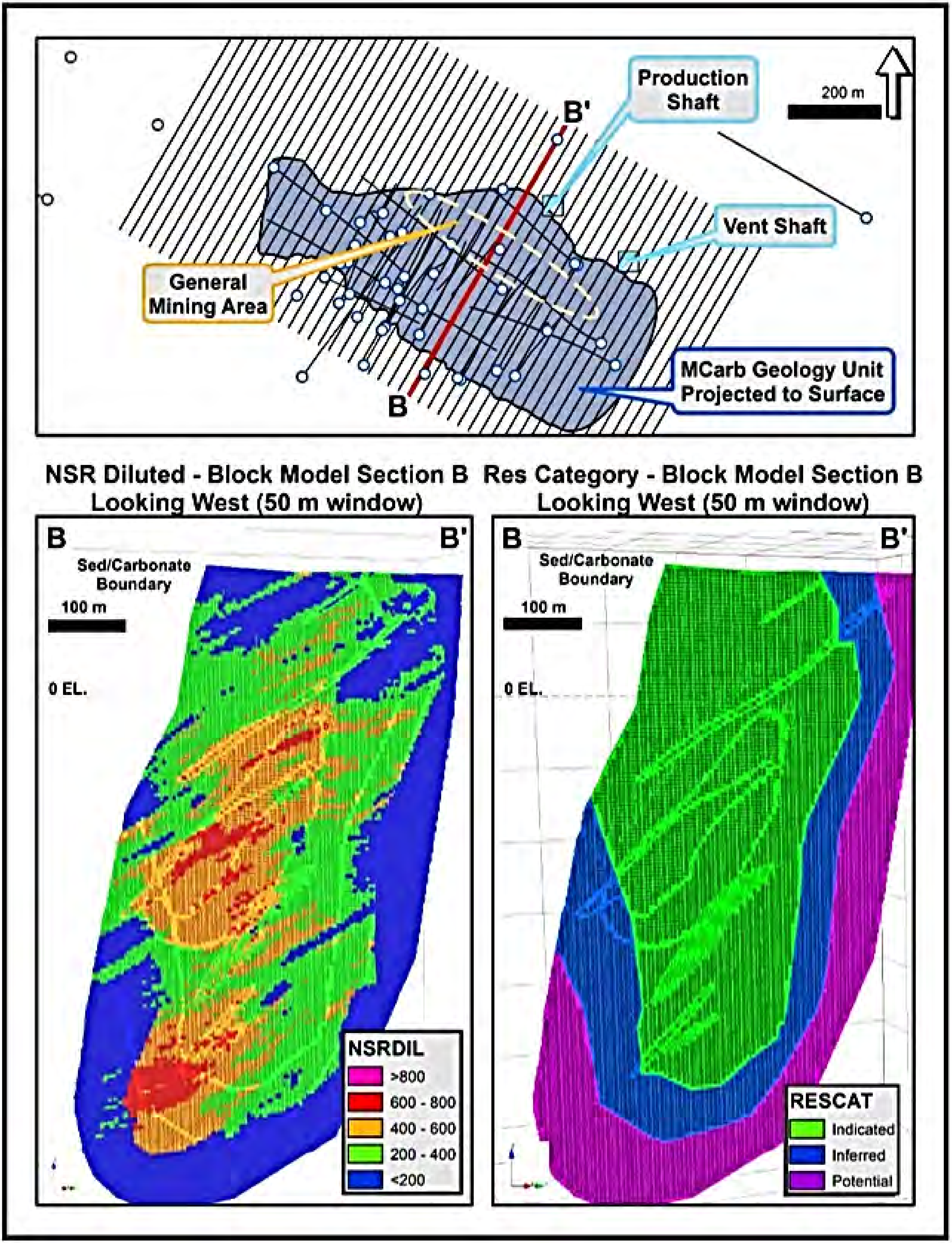
Nb2O5 (%) - Block Model Section C Looking West (50 m window)



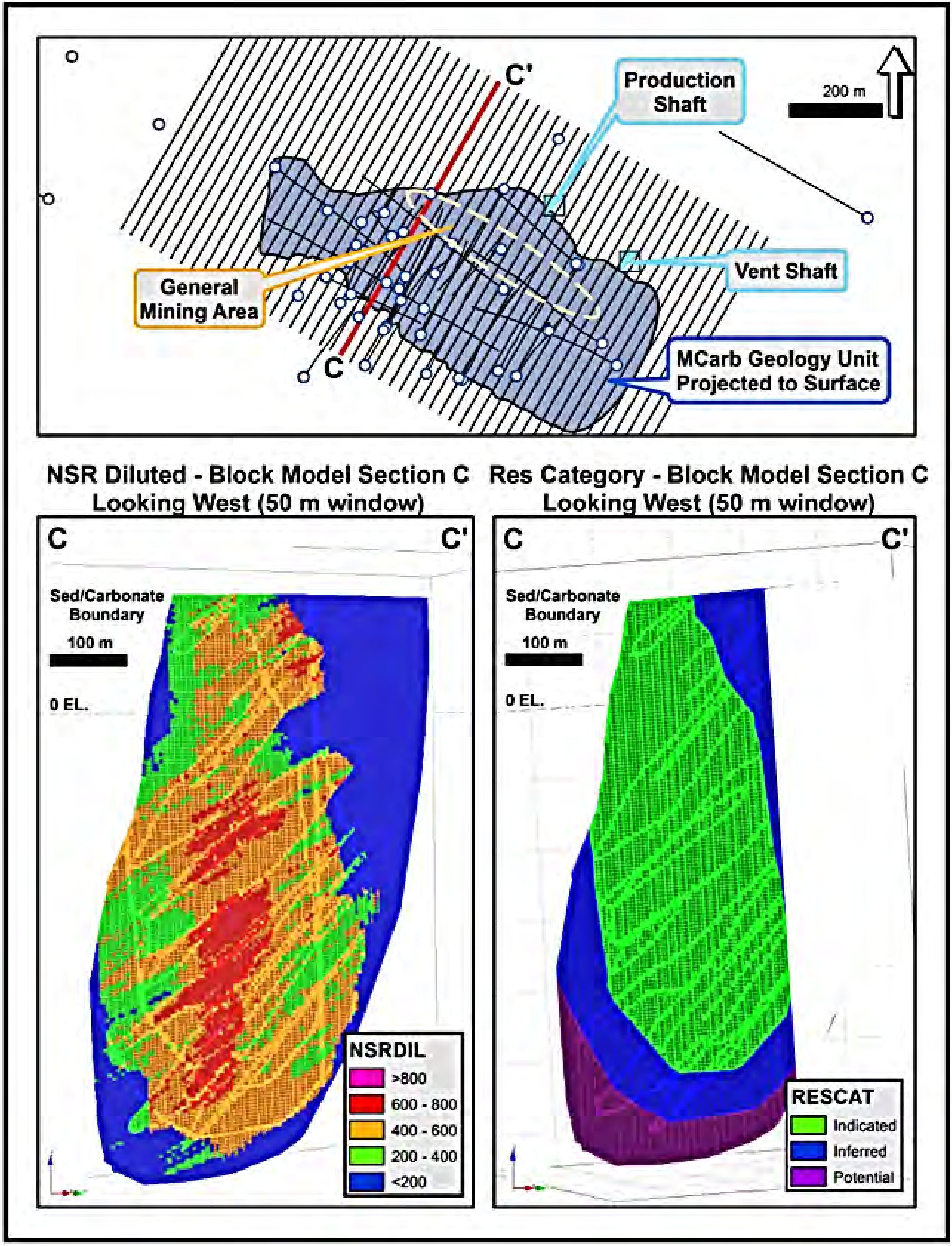
Sc (ppm) - Block Model Section C Looking West (50 m window)




Production Shaft
200 m
A'
General Mining Area
Vent Shaft
MCarb Geology Unit Projected to Surface
A
NSR Diluted - Block Model Section A Looking West (50 m window)
A
A'
Sed/Carbonate Boundary
100 m
0 EL.
NSRDIL
>800
600 - 800
400 - 600
200 - 400
<200
Res Category - Block Model Section A Looking West (50 m window)
A
A'
Sed/Carbonate Boundary
100 m
0 EL.
RESCAT
Indicated
Inferred
Potential

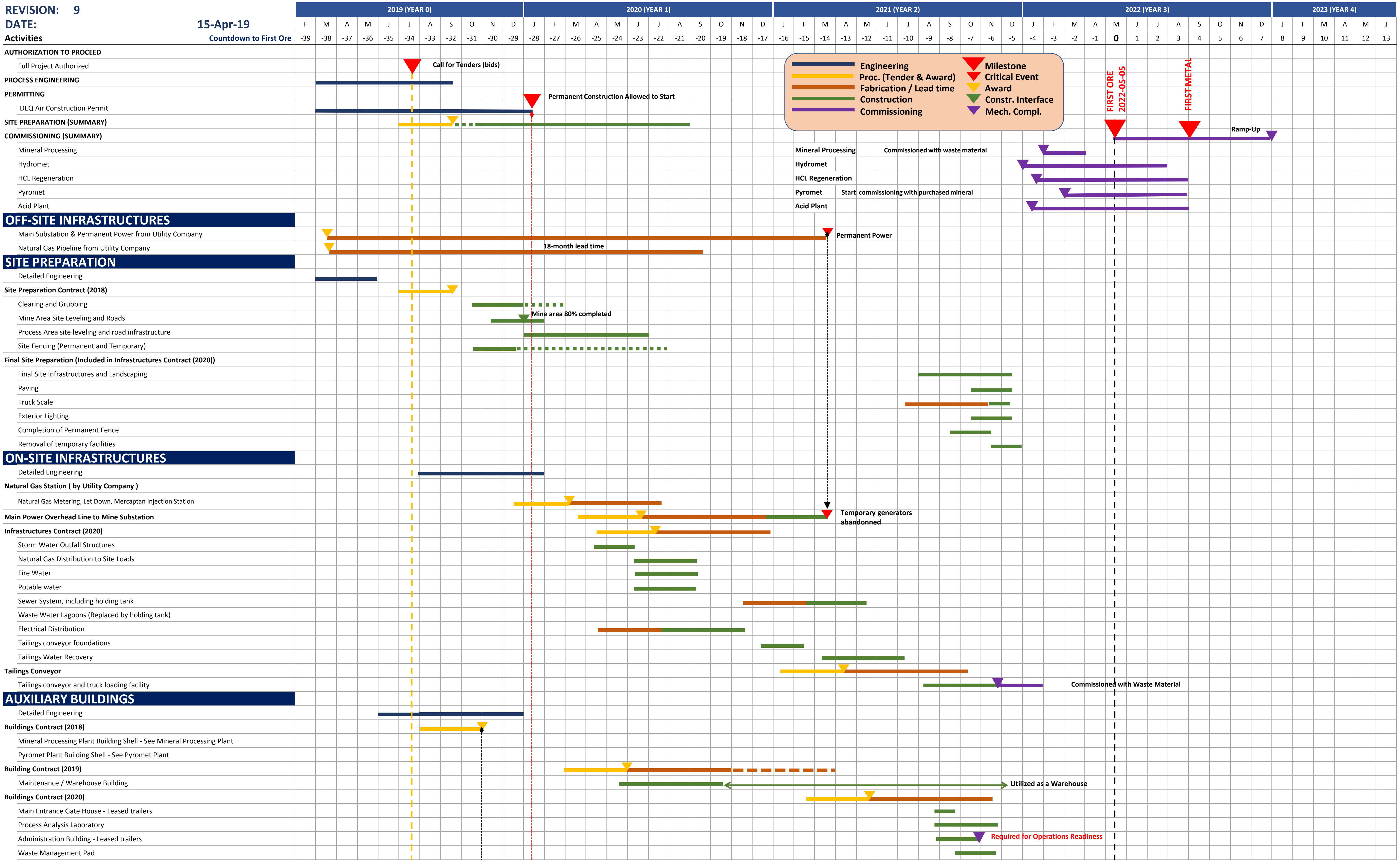


Production Shaft
200 m
B'
Vent Shaft
General Mining Area
MCarb Geology Unit Projected to Surface
B
NSR Diluted - Block Model Section B Looking West (50 m window)
B
B'
Sed/Carbonate Boundary
100 m
0 EL.
NSRDIL
>800
600 - 800
400 - 600
200 - 400
<200
Res Category - Block Model Section B Looking West (50 m window)
B
B'
Sed/Carbonate Boundary
100 m
0 EL.
RESCAT
Indicated
Inferred
Potential


C'
Production Shaft
200 m
General Mining Area
Vent Shaft
C
MCarb Geology Unit Projected to Surface
NSR Diluted - Block Model Section C Looking West (50 m window)
Res Category - Block Model Section C Looking West (50 m window)
C
C'
Sed/Carbonate Boundary
100 m
0 EL.
NSRDIL
>800
600 - 800
400 - 600
200 - 400
<200
C
C'
Sed/Carbonate Boundary
100 m
0 EL.
RESCAT
Indicated
Inferred
Potential

Appendix C: Pre-Production Schedule

NioCorp Elk Creek Niobium Project - Pre-Production Summary Schedule



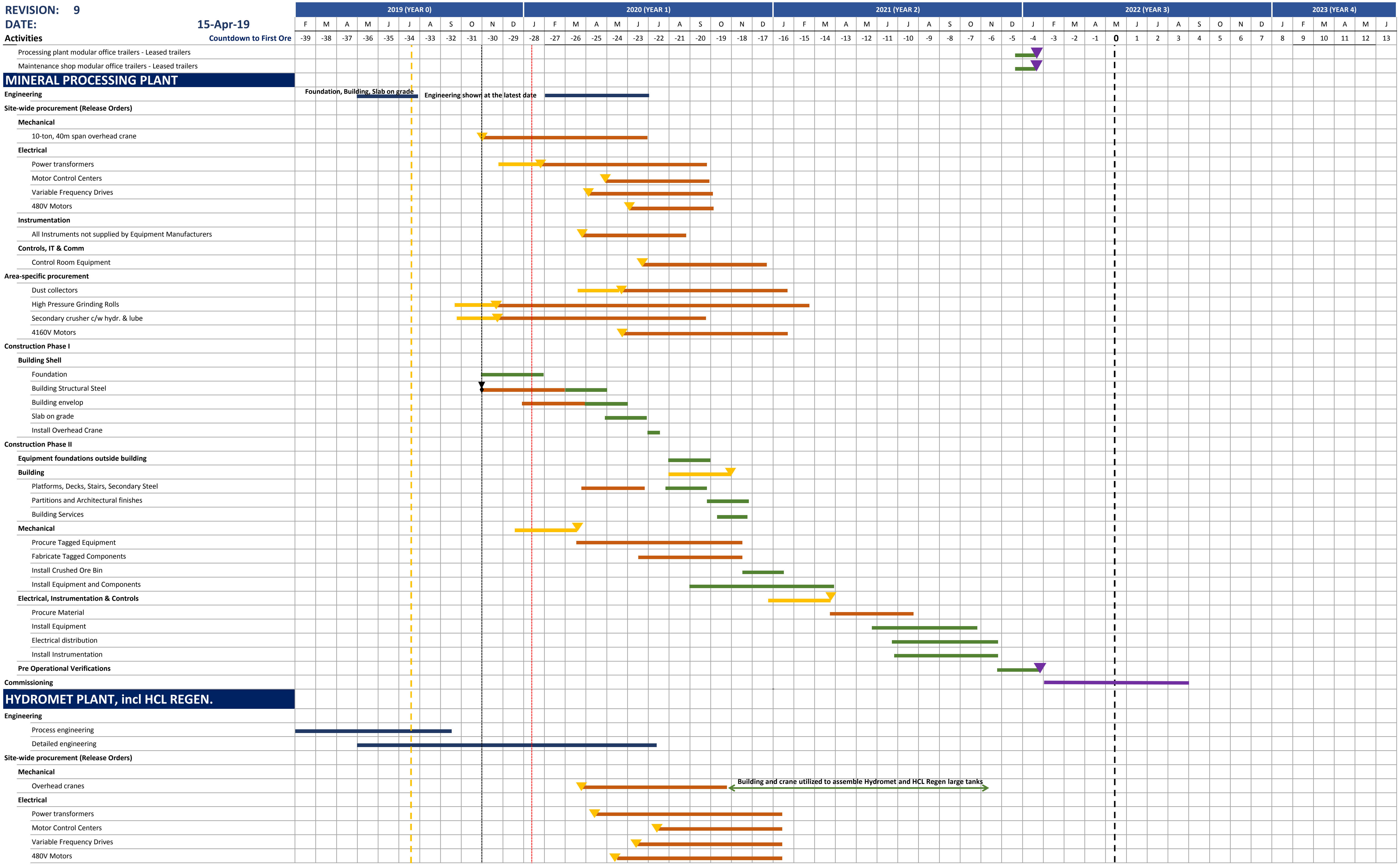
NioCorp Elk Creek Niobium Project - Pre-Production Summary Schedule

NioCorp Developments Ltd.

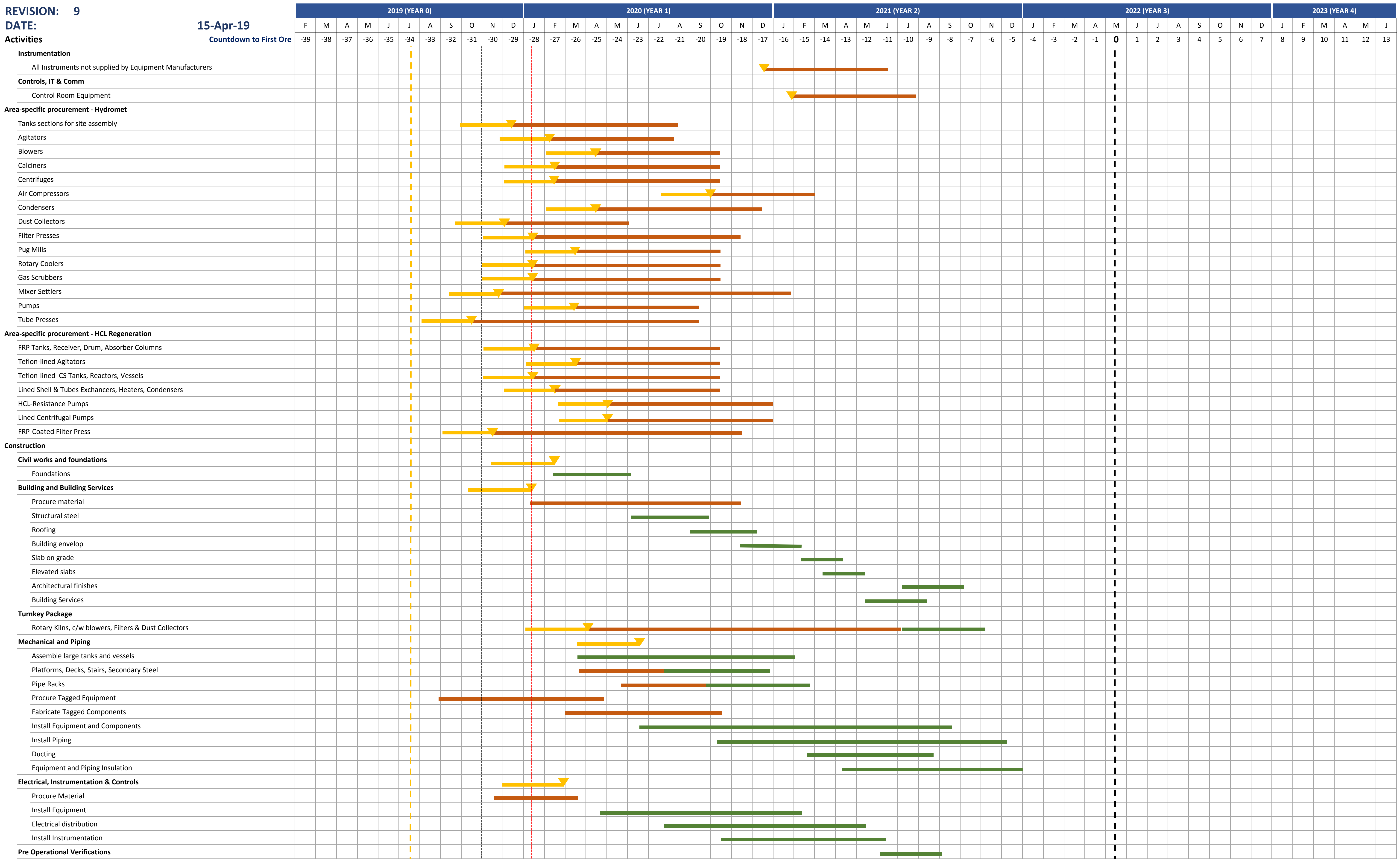
NORDMIN RESOURCE & INDUSTRIAL ENGINEERING — ZACHRY — TETRA TECH

REVISION: 9

DATE: 15-Apr-19



NioCorp Elk Creek Niobium Project - Pre-Production Summary Schedule



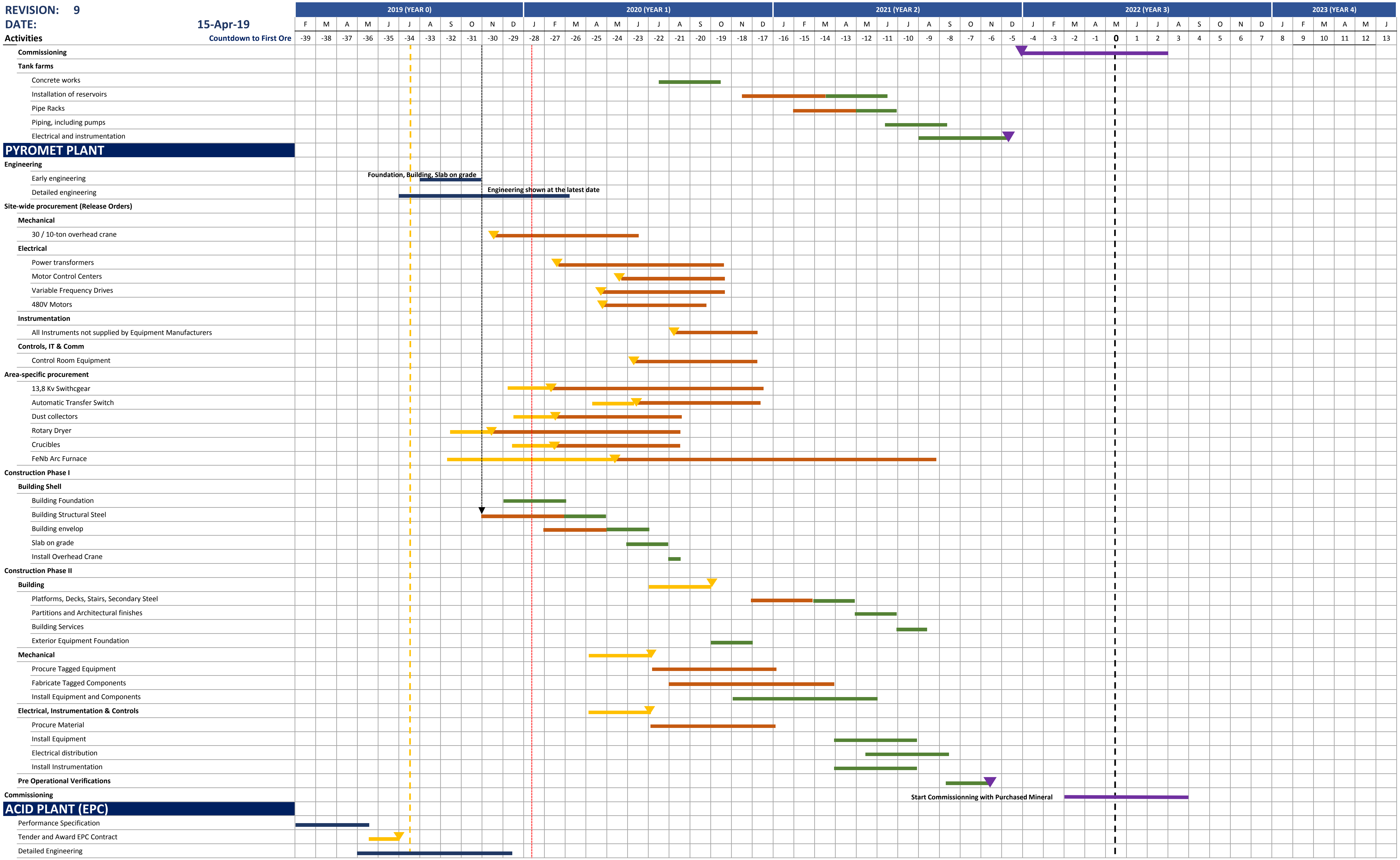
NioCorp Developments Ltd.

NioCorp Elk Creek Niobium Project - Pre-Production Summary Schedule

NORDMIN RESOURCE & INDUSTRIAL ENGINEERING | ZACHRY | TETRA TECH

REVISION: 9

DATE: 15-Apr-19



NioCorp Developments Ltd.

NioCorp Elk Creek Niobium Project - Pre-Production Summary Schedule

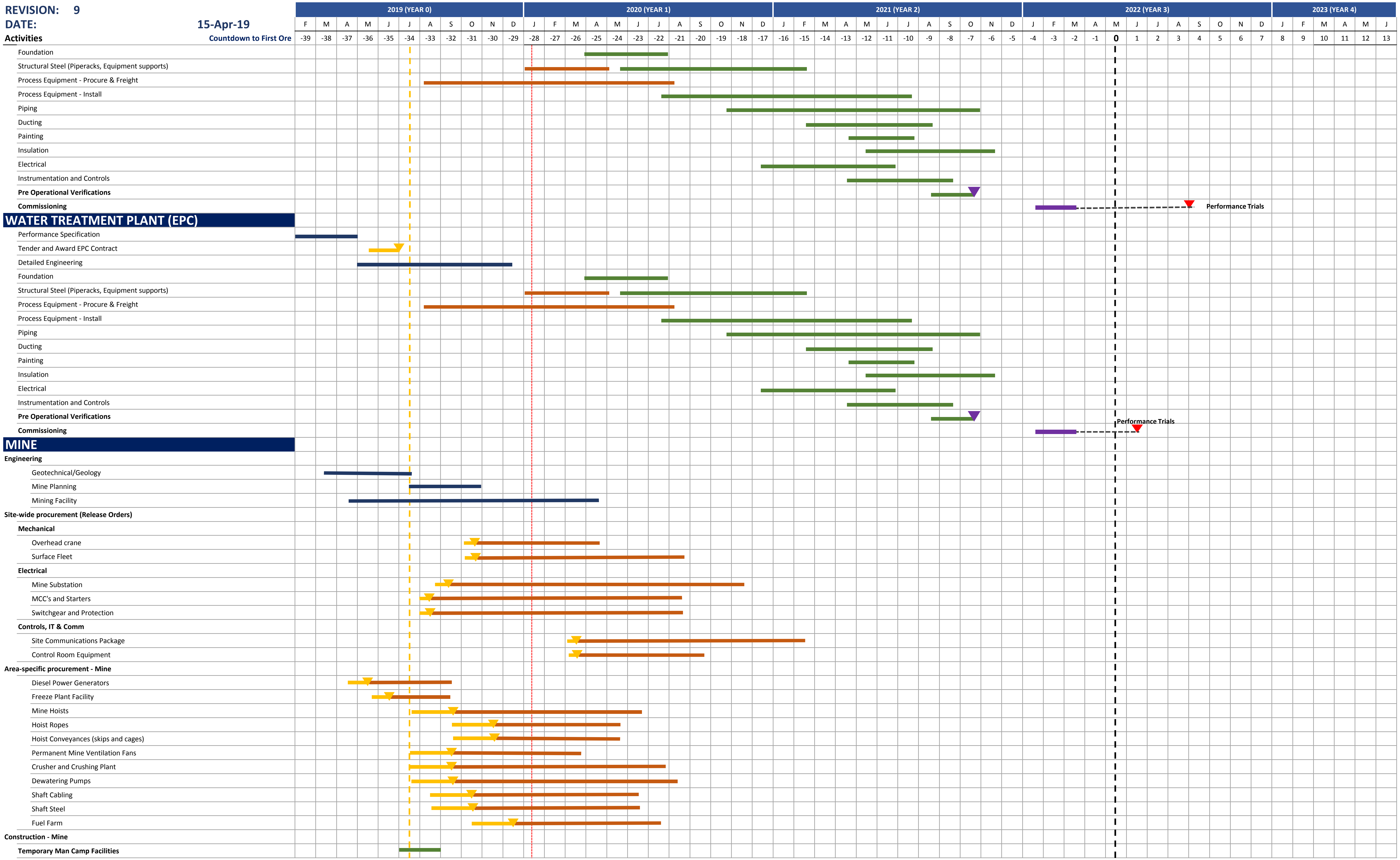
NORDMIN RESOURCE & INDUSTRIAL ENGINEERING

ZACHRY

TETRA TECH

REVISION: 9

DATE: 15-Apr-19



NioCorp Elk Creek Niobium Project - Pre-Production Summary Schedule

NioCorp Developments Ltd.

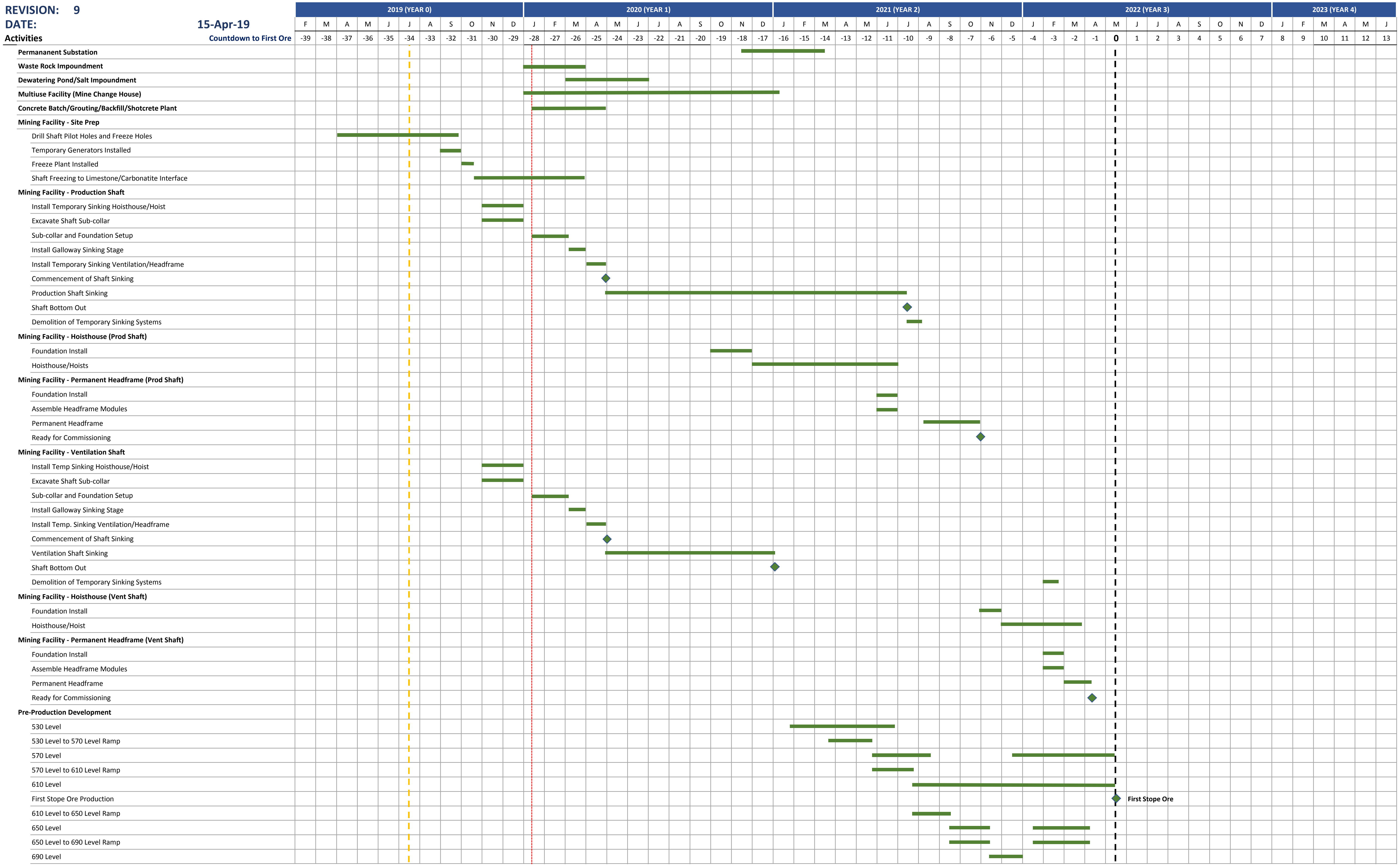
NORDMIN RESOURCE & INDUSTRIAL ENGINEERING

ZACHRY

TETRA TECH

REVISION: 9

DATE: 15-Apr-19



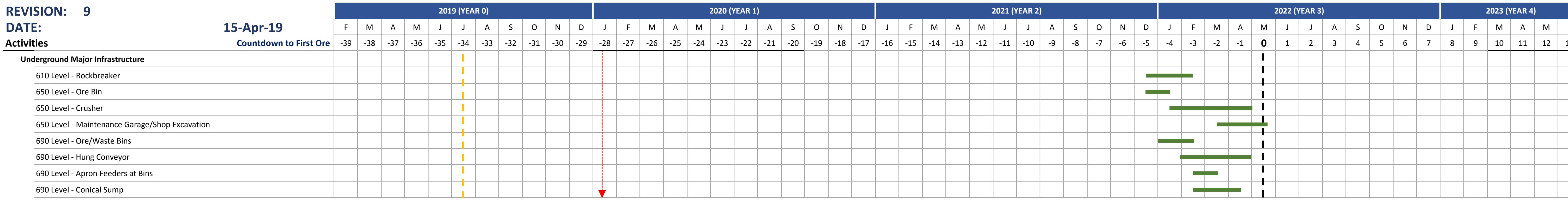
NioCorp Developments Ltd.

NioCorp Elk Creek Niobium Project - Pre-Production Summary Schedule

NORDMIN RESOURCE & INDUSTRIAL ENGINEERING | ZACHRY | TETRA TECH

REVISION: 9

DATE: 15-Apr-19



Appendix D: Feasibility Risk Register

RISK REGISTER

Project: Elk Creek Niobium Report	Project Manager: Greg Menard	Issue Date: 05/27/2019
Project Phase: Feasibility Report	Project Sponsor:	
	Project Risk Manager: Glen Kuntz	Revision: 002

Reference Numbering	Primary Source	Secondary Source	Risk Owner or Delegator	Headline Label	Risk or Opportunity	Root Cause and Risk Description	Risk Classification	Probable Consequence Description	Primary Consequence Category	Secondary Consequence Category	Pre-Response Consequence Level	Pre-Response Likelihood Level	Pre-Response Risk Rating	Strategies to Process Risk	Risk Response - Action Plan	Post-Response Consequence Level	Post-Response Likelihood Level	Post-Response Risk Rating
1	Geology	Mine Design	Geology	Resources & Reserves	Risk	The current mine plan is based upon indicated resources, with a total of 48 drill holes defining the entire resource. Drill hole spacing is approximately 50-75 m. Continuity of grade between diamond drill holes could be less than what is predicted in the model.	Threat	Grade throughput is significantly reduced	Production	Financial/ Costs	Severe	Almost Certain	25	Mitigate	It is recommend that parts of the deposit in the first 5-7 years of operation, be drilled to the frequency of measured minerla resources, to ensure that grade continuity supports the mine plan. Currently an ongoing 30,000-50,000 m definition drill program are required/planned and budgeted to support the first 5-7 years of production.	Moderate	Likely	12
2	Geology	Mine Design	Geology	Resources & Reserves	Risk	The complexity of the ore body could potentially lead to increased mining dilution. Grade control and proper mining execution will maintain minimal unplanned dilution, which would minimize potential impacts on grade, throughput, and operating costs.	Threat	Reducing Head Grade	Production	Financial/ Costs	Major	Likely	16	Mitigate	It is recommend/budgeted that a daily grade control program be established as outlined in recommendations.	Moderate	Likely	12
3	Geology	Mine Design	Mining	Engineering & Construction	Risk	The shafts could encounter unknown mineraation of varying grades that may impact further mine plans.	Threat	Development and production activity stopped	Production	Financial/ Costs	Minor	Almost Certain	10	Mitigate	Complete pilot hole drill program before shaft excavations begin.	Minor	Likely	8
4	Geology	Plant	Plant	Plant	Risk	Lack of detailed drilling to determine geochemical variations within the deposit.	Threat	Development and production activity stopped	Production	Financial/ Costs	Major	Likely	16	Mitigate	definition drilling program is required throughout LOM.	Moderate	Possible	9
5	Monitoring	Ground Support	Geotechnical	Safety, Health, Environment & Community	Risk	Lack of detailed drilling to determine poor ground conditions.	Threat	Worker injury or fatality	Health & Safety	Production	Major	Likely	16	Mitigate	definition drilling and daily mapping of geologicial, geotechnical parameters are required.	Major	Unlikely	8
6	Geology	Ground Support		Safety, Health, Environment & Community	Risk	Currently pilot holes have not been drilled for the proposed production and ventilation shafts to determine geological/geotechnical and hydrological characteristics.	Threat	Worker injury or fatality	Health & Safety	Production	Major	Likely	16	Mitigate	A plan, budget and drill plan has been designed to collect the required information.	Moderate	Possible	9
7	Hydrogeology	Mine Design	Mining	Engineering & Construction	Risk	Highly variable groundwater flow to stopes and development workings creates continual risk that a highly pressured, high permeabilty conduit may be encountered which delivers more water to the mine than can be handled by the installed equipment.	Threat	Flooding of one or more levels of the mine, generally starting at the deepest level	Production	Safety, Health, Environment & Community	Major	Possible	12	Mitigate	1. Maintain excess pumping capacity at lowest level of mine. 2. Segregate each level to provide short-term inflow response. 3. Maintain excess grouting capacity to plug local inflow zones. 4. Implement injection wells or bring in additional water treatment capacity.	Moderate	Possible	9
8	Hydrogeology	Mine Design	Mining	Safety, Health, Environment & Community	Risk	Mining in potentially karstic carbonatite formations located hundreds of meters below the piezometric surface of the deposit provides a continuing potential for very large inrushes of water when advancing. Controlling a large scale inrush in this mine setting will be extremely challenging.	Threat	Loss of life, loss of production, high cost plugging and recovery operations. Enterprise threatening.	Safety, Health, Environment & Community	Production	Severe	Possible	15	Explore	1. Probe drilling conducted in front of all mining advances. 2. Maintain personnel escapeways that remain open even under high flow conditions. 3. Grout all voids and producing fault and karst zones in advance of mining, to create structural and hydraulic integrity. 4. Operate flooding warning, integrated with mine-wide safety system, to allow safe exit of miners from rapidly-flooding work areas.	Major	Possible	12
9	Hydrogeology	Mine Design	Mining	Infrastructure	Risk	The mine water control system requires continuous large-scale grouting and long-term pumping of water inflow from the highly pressurized, high permeability carbonatite immediately outside the mine, which requires reliable sources of power, continuously servicable equipment, and trained personnel to operate them.	Threat	Uncontrollable inflow to the mine flooding critical infrastructure, potentially eliminating recovery options.	Production	Financial/ Costs	Major	Possible	12	Mitigate	1. Provide excess grouting equipment, due to the relatively low reliability of high pressure grouting equipment. 2. Provide onsite backup power supplies to ensure that any power outage can be bridged. 3. Have emergency backup plans, and drill mine personnel in their implementation.	Major	Possible	12
10	Ground Support	Monitoring	Geotechnical	Safety, Health, Environment & Community	Risk	Impurities in geochemistry impacting process recoveries.	Threat	Worker injury or fatality	Health & Safety	Production	Severe	Unlikely	10	Mitigate	Develop Ground Control Management Plan (GCMP) and conduct audits to ensure GCMP is being followed during construction and operations.	Major	Rare	4
11	Ground Support	Monitoring	Geotechnical	Safety, Health, Environment & Community	Risk	Intersecting unknown poor ground (mud, major fault gauge).	Threat	Worker injury or fatality	Health & Safety	Production	Severe	Unlikely	10	Mitigate	Develop Ground Control Management Plan (GCMP) and conduct audits to ensure GCMP is being followed during construction and operations.	Major	Rare	4
12	Hydrogeology	Lack of grouting	Geotechnical	Engineering & Construction	Risk	Failure to drill probe holes and properly grout ahead of development.	Threat	Development and production activity stopped	Production	Financial/ Costs	Moderate	Possible	9	Mitigate	Develop and implement Trigger Action Response Plan (TARP).	Minor	Unlikely	4
13	Monitoring	Ground Support	Geotechnical	Engineering & Construction	Risk	Lack of drilling to define Mineral Resources.	Threat	Worker injury/fatality or production activity stopped	Safety, Health, Environment & Community	Production	Severe	Unlikely	10	Mitigate	Monitor seismicity and identify burst prone areas (at depth) for which ground support can be increased.	Negligible	Unlikely	2
14	Stope dimensions	Ground Support	Geotechnical	Engineering & Construction	Risk	Rock mass strengths are lower than assumed for the stope designs or development ground support design. Intersecting unknown poor ground (mud, major fault gauge).	Opportunity/Threat	Stopes dilution or cave	Production	Financial/ Costs	Minor	Unlikely	4	Mitigate	Detailed mine plans get approval that includes geotechnical engineer.	Negligible	Rare	1
15	Pastefill design	Stope Dimensions	Geotechnical	Engineering & Construction	Risk	The variation in pastefill strength is insufficient to prevent sloughing into open adjacent stopes.	Opportunity/Threat	Stopes dilution or cave	Production	Financial/ Costs	Minor	Unlikely	4	Mitigate	Early pastefill performance needs monitoring and adjustments made to pastefill plant batching prior to major problems.	Negligible	Rare	1
16	Geotechnical Characterization	Mine Design	Geotechnical	Engineering & Construction	Risk	Faults encountered in development require additional ground support than anticipated.	Opportunity/Threat	Additional ground support required	Production	Financial/ Costs	Major	Unlikely	8	Mitigate	Develop Ground Control Management Plan (GCMP) and conduct audits to ensure GCMP is being followed during construction and operations.	Major	Rare	4
17	Geotechnical Characterization	Mine Design	Geotechnical	Engineering & Construction	Risk	Faults encountered in vertical shaft & raises require additional ground support than anticipated.	Opportunity/Threat	Additional ground support required	Production	Financial/ Costs	Moderate	Unlikely	6	Mitigate	Develop Ground Control Management Plan (GCMP) and conduct audits to ensure GCMP is being followed during construction and operations.	Moderate	Rare	3
18	Geotechnical Characterization	Mine Design	Geotechnical	Engineering & Construction	Risk	Faults encountered in infrastructure development areas require additional ground support than anticipated.	Opportunity/Threat	Additional ground support required	Production	Financial/ Costs	Moderate	Unlikely	6	Mitigate	Develop Ground Control Management Plan (GCMP) and conduct audits to ensure GCMP is being followed during construction and operations.	Moderate	Rare	3
19	Hydrogeology	Grouting	Mining	Engineering & Construction	Risk	Water leakage through gaps within the frozen zone and flowing into the shaft below the concrete liner, or inflows during during conventional sinking below the frozen elevation.	Threat	Delay and added cost to continuation of sinking	Financial/ Costs	Production	Moderate	Rare	3	Mitigate	Introduce a grout curtain as a second line of defense prior to continued sinking operations. Probe holes drilled in advance of sinking to search for water inflows prior to excavation operations.	Moderate	Rare	3
20	Geological Conditions	Poor execution of sinking operation	Mining	Engineering & Construction	Risk	Excavation takes longer than scheduled.	Threat	Delay to Production	Production	Financial/ Costs	Moderate	Rare	3	Mitigate	Initial shaft geotechnical holes to evaluate ground conditions and faults. Proper evaluation and selection of shaft sinking contractor.	Moderate	Rare	3
21	Procurement	Scheduling	Procurement	Supply	Risk	Delays to delivering of key equipment for development and production.	Threat	Delay to Production	Production	Financial/ Costs	Moderate	Unlikely	6	Mitigate	Complete detailed engineering in a timely fashion and establish an advanced procurement strategy.	Moderate	Rare	3
22	Tailings	Contact Water	Owner	Safety, Health, Environment & Community	Risk	Regulators require the TSF to be double-lined with LCRS.	Threat	Permit delay and construction delay with increased cost	Safety, Health, Environment & Community	Financial/ Costs	Major	Possible	12	Mitigate	Design TSF with double lined geomembrane and LCRS.	Moderate	Unlikely	6
23	Tailings	Contact Water	Owner	Safety, Health, Environment & Community	Risk	Breach in water containment or leak at tailings impoundment facility.	Threat	Potential shut down of Mill, loss of containment, saturation of foundation, loss of foundation strength	Safety, Health, Environment & Community	Financial/ Costs	Moderate	Possible	9	Mitigate	Monitoring; instrumentation and frequent human watch; Back-up pumps at the pond; Buried water line to the plant site. Spillways installed to prevent damage to embankment during overtopping episode. Double liner with above liner drainage in design.	Moderate	Unlikely	6
24	Tailings	Contact Water	Owner	Safety, Health, Environment & Community	Risk	Regulators requiring more frequent covering of the tailings than included in the operation plan.	Threat	Increased operation costs and loss of production	Production	Financial/ Costs	Moderate	Possible	9	Accept	Frequency of cover to be dictated by final tailings characterization results.	Minor	Unlikely	4
25	Tailings	Closure	Owner	Engineering & Construction	Risk	Conditions in place requiring installation of a root barrier to prevent rooting into the tailings.	Threat	Root uptake presents hazard to wildlife consuming vegetation	Financial/ Costs	Safety, Health, Environment & Community	Moderate	Possible	9	Prevent	Final closure cover includes geomembrane, preventing root penetration into tailings.	Moderate	Unlikely	6
26	Tailings	Foundation	Owner	Infrastructure	Risk	Foundation strength reduction due to loading and elevated pore pressure .	Threat	Excessive deformation of TSF, potential for tailings failure	Safety, Health, Environment & Community	Financial/ Costs	Major	Possible	12	Mitigate	Further characterization of foundation soils to confirm foundation design assumptions and strength properties. CQA during construction.	Minor	Unlikely	4
27	Tailings	Operations	Owner	Safety, Health, Environment & Community	Risk	Tailings emissions reqwuire abatement.	Threat	Emissions in excess of regulated standards	Safety, Health, Environment & Community	Financial/ Costs	Major	Possible	12	Explore	Additional characterization.	Moderate	Possible	9
28	Tailings	Operations	Owner	Engineering & Construction	Risk	Lower tailings density leading to higher volume occupation and either higher TSF embankments and/or additional area for TSF construction.	Threat	Excess pore pressure in the compacted tailings (static or seismic liquefaction), loss of storage capacity	Production	Financial/ Costs	Major	Likely	16	Mitigate	Additional characterization of geotechnical properties for water leach residue, calcined excess oxide, and slag tailings solids with additional laboratory testing including gradation, density, drainage/permeability, consolidation, and strength.	Moderate	Possible	9
29	Tailings	Operations	Owner	Engineering & Construction	Risk	Unsuitable construction material to build the TSF.	Threat	Delay construction of the embankment while suitable material is sourced from elsewhere	Production	Financial/ Costs	Major	Likely	16	Explore	Further characterization of borrow sources .	Minor	Unlikely	4
30	Tailings	Operations	Owner	Engineering & Construction	Risk	Construction delayed by rain, snow or shallow groundwater.	Threat	Delay construction and increased cost	Production	Financial/ Costs	Minor	Possible	6	Accept	Schedule construction outside of rainy season to avoid possible delays.	Negligible	Possible	3
31	Tailings	Operations	Owner	Safety, Health, Environment & Community	Risk	Fugitive dust generation off the placed tailings.	Threat	Impact to surrounding town and home owners with possible shut downs	Safety, Health, Environment & Community	Production	Major	Possible	12	Mitigate	Addition of a pug mill to tailings management circuit to increase grain size and density.	Moderate	Possible	9
32	Water Treatment	Operations	Owner	Engineering & Construction	Risk	Predicted salt production rate varies from predictions.	Threat	Increased size of salt management cells	Financial/ Costs	Production	Major	Likely	16	Improve	Provide factor of safety in design to accommodate more salt than predicted.	Major	Unlikely	8
33	Access	Construction	Owner	Engineering & Construction	Opportunity	Use non-potentially-acid-generating (non-PAG) waste rock from shaft sinking to produce gravel for road and parking infrastructure.	Opportunity	Decrease in cost and size of waste rock stockpile	Financial/ Costs	Financial/ Costs	Minor	Possible	6	Improve	Include this requirement in specification; ensure compliance during bid analysis.	Negligible	Unlikely	2
34	Tailings	Water Management	Owner	Engineering & Construction	Opportunity	Consider burying the Water Recovery Pipeline from the Tailings Impoundment Facility to the Water Treatment Plant.	Opportunity	Management of LCP	Financial/ Costs	Safety, Health, Environment & Community	Minor	Possible	6	Explore	Explore possible options.	Negligible	Unlikely	2

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35	Construction	Environment	Owner	Safety, Health, Environment & Community	Risk	Ensure environmental protection during construction.	Threat	Environmental incident	Financial/ Costs	Reputation	Minor	Possible	6	Mitigate	Make sure BMP's and surface water management structures are in place before and during construction.	Minor	Unlikely	4
36	Performance	Operations	Owner	Safety, Health, Environment & Community	Risk	Drier feed or diferent rock makeup could cause higher amounts of dust.	Threat	Environmental incident/ Lower Production Rate	Safety, Health, Environment & Community	Production	Moderate	Unlikely	6	Mitigate	Better dust control or possible addition of water.	Moderate	Rare	3
37	Performance	Operations	Owner	Plant	Risk	Exsessive recycle due to rock not breaking up as excepted.	Threat	Lower Production Rate	Production	Financial/ Costs	Moderate	Unlikely	6	Prevent	Make sure test work performed to ensure proper sizing of equipment.	Moderate	Rare	3
38	Performance	Operations	Owner	Plant	Risk	Final recipe at the paste plant requiring additional material (type and quantity) not anticipated.	Threat	Production delays	Production	Financial/ Costs	Moderate	Likely	12	Accept	During detailed design, conduct additional bench/pilot studies and vendor testing in specific areas to confirm metallurgical performance and increase confidence for startup. The MetSim simulation model will be completed and optimized to simulate and address any scale up issues.	Moderate	Likely	12
39	Performance	Operations	Owner	Plant	Risk	Hydromet scale-up from test lab: every step of the process was subject to a small scale pilot test.	Threat	Production delays	Production	Financial/ Costs	Moderate	Likely	12	Mitigate	Execute the recommended testing program during the detailed design process.	Moderate	Rare	3
40	Performance	Operations	Owner	Plant	Risk	Hydromet process design not being fully optimized.	Threat	Facility integration.	Production	Financial/ Costs	Major	Likely	16	Mitigate		Moderate	Rare	3
41	Operations	General	Owner	External	Risk	Product demand on a new scandium market.	Threat	Decrease cashflow.	Financial/ Costs	Financial/ Costs	Major	Likely	16	Explore	Current established market is approximately 20 t Sc_2O_3. NioCorp to continue efforts to add additional offtake agreements for scandium production to minimize market risk.	Major	Likely	16
42	Operations	General	Owner	External	Risk	Niobium demand not selling 100% of product.	Threat	Decrease cashflow.	Financial/ Costs	Financial/ Costs	Moderate	Possible	9	Explore	Continue to build long term contracts.	Moderate	Possible	9
43	Operations	General	Owner	External	Risk	Titanium impurities, cause delays in sales.	Threat	Decrease cashflow.	Financial/ Costs	Financial/ Costs	Moderate	Unlikely	6	Prevent	Continue to monitor chemistry, to determine if there are issues.	Moderate	Possible	9
44	Construction	General	Owner	Engineering & Construction	Risk	Failure to raise capital at the required time will delay construction and potentially excalate costs.	Threat	Delay/cancel project.	Financial/ Costs	Reputation	Severe	Possible	15	Improve	Finalise key lending contracts.	Moderate	Possible	9
45	Operations	General	Owner	Community	Risk	Insufficient housing and services offer in the local communities	Threat	Production delays.	Production	Financial/ Costs	Minor	Unlikely	4	Explore	Local market will respond to the demand. Schedule will allow for a reasonable mobilization.	Minor	Unlikely	4
46	Process	Technical	Plant	Plant	Risk	Testing in lab showed a thick and viscous slag .	Threat	Recipe additives/flux difficult to find - possible higher cost + longer starting operation window.	Production	Financial/ Costs	Minor	Possible	6	Improve	Push forward the investigation to decrease the level of TiO_2 at the Hydromet. In case of Hydromet unsuccess to reduce the TiO_2, ptimizing sequences might solve the issue.	Minor	Rare	2
47	Process	Technical	Technical	Human Resources	Risk	TiO_2 Feed level in Hydromet precipitate is significatively high . Slag tap number can bring some issue in terms of operation sequences.	Threat	Tasks might be challenging with high quantity slag removal.	Production	Production	Minor	Likely	8	Improve	Push forward the investigation to decrease the level of TiO_2 at the Hydromet. In case of Hydromet unsuccess to reduce the TiO_2, ptimizing sequences might solve the issue.	Minor	Rare	2
48	Process	Environment	Design	Safety, Health, Environment & Community	Risk	Presence of sulfates in the Hydromet feed can require acid treatment equipment or simply pipe connections to the sulfuric plant.	Opportunity/Threat	Non respect of the applicable environment rules & regulations.	Reputation	Safety, Health, Environment & Community	Major	Possible	12	Prevent	Consider the addition of a caustic scrubber to the pyromet emissions control design if the potential to emit SO_x from the pyromet operation is a matter of substance.	Major	Rare	4
49	Material	Monitoring	Design	Engineering & Construction	Risk	Slag viscosity might require to operate the EAF at high temperature, demanding high refractory resistance.	Threat	Refractory need to be replace more often than expected.	Production	Financial/ Costs	Negligible	Possible	3	Improve	Push forward the investigation to decrease the level of TiO_2 at the Hydromet. In case of Hydromet unsuccess to reduce the TiO_2, ptimizing sequences might solve the issue.	Negligible	Rare	1
50	Supply logistic	Procurement	Monitoring	Supply	Risk	There is only one aluminium shot supplier North America.	Opportunity/Threat	Inventory at zero stops production.	Organizational Effectiveness	Financial/ Costs	Moderate	Possible	9	Explore	Develop with other aluminum supplier to get a second or a third supplier or use cut Al wire.	Moderate	Rare	3
51	Supply logistic	Procurement	Monitoring	Supply	Risk	Fe_2O_3 suppliers are located in the North-East side of North America, distance/weather can be an issue.	Threat	Inventory at zero stops production.	Production	Financial/ Costs	Moderate	Unlikely	6	Transfer	Work with at least two or three suppliers.	Moderate	Rare	3
52	Procedures	HR management	Plant	Strategic Issues	Risk	Some FeNb will be lost in slag, in dust and as part of the equipment cleaning (tundish, launders, cooling equipment).	Opportunity/Threat	Losing FeNb at several steps/areas of the process.	Organizational Effectiveness	Financial/ Costs	Minor	Likely	8	Mitigate	Develop recycle techniques prior to and during commision and ramp up of the Pyromet Plant. Document these techniques and operting procedures and educate employees on the importance of minimizing ferroniobium losses and returning recovered ferroniobium to the Pyromet process.	Minor	Rare	2
53	Process/ Engineering	Management	Plant	Plant	Risk	Slag taps number can compromise operation stability.	Opportunity/Threat	Pyromet effeciency might not be optimum.	Production	Production	Minor	Unlikely	4	Explore	Push forward the investigation to decrease the level of TiO_2 at the Hydromet. In case of Hydromet unsuccess to reduce the TiO_2, ptimizing sequences might solve the issue.	Minor	Rare	2
54	Environmental	Permitting	Owner	Plant	Risk	The issuance of a Prevention of Significant Deterioration (PSD) air quality permit for the process operations (including sulphuric acid plant) will involve U.S. EPA review and public comment. While the inclusion of additional air monitoring and emission control devices generally mitigates the risk associated with permit acquisition, here too, the participation of a federal agency (and public review of the Project) has the potential to slow the permitting process.	Threat	Potential delay in project start up.	Financial/ Costs	Production	Moderate	Possible	9	Mitigate	Continue engagement with federal agency(s) and public reviews.	Minor	Unlikely	4
55	Environmental	Operations	Owner	Community	Risk	Once the full extent and scale of the project is officially publicized, organized opposition will likely increase. NioCorp has proactively engaged Bold Nebraska to mitigate some of this risk.	Threat	Potential project delay.	Financial/ Costs	Production	Moderate	Possible	9	Share	Continue engagement with NGO's, federal, state and public reviews.	Minor	Unlikely	4
56	Environmental	Permitting	Owner	Plant	Risk	Given the remaining uncertainty surrounding the various waste streams projected to report to the TSF, additional testing has the potential to result in one or more of these streams being classified as hazardous waste. While early testing of relatively small samples and laboratory-generated surrogate samples suggests that these materials will be non-hazardous from a toxicity perspective.	Threat	The potential reactivity of the calcined material to water may lead the NDEQ classification as hazardous, even though this material does not "react violently with water" as stipulated in the regulatory definition.	Financial/ Costs	Production	Moderate	Possible	9	Improve	Continue testing. Reassess. Further refinement required?			
57	Environmental	Permitting	Owner	Plant	Risk	Full and comprehensive characterization and disclosure of the partitioning of radionuclides in the various products and waste streams has not yet been completed. Uncertainty in this aspect of the project may lead the Nebraska DHHS to delay their permitting process and potentially place operating restrictions of the Project. This will include characterization of potential point of worker exposure, including areas of dust generation (i.e., crusher), and potential radon gas accumulation.	Threat	Nebraska DHHS delay of permitting process and potential placement of operating restrictions on the Project.	Financial/ Costs	Production	Moderate	Possible	9	Improve	Complete the full and comprehensive characterization and disclosure of the partitioning of radionuclides in the various products and waste streams. Reassess.			
58	Environmental	Health & Safety	Owner	Community	Risk/Opportunity	The reliance on shipments via trucks presents risk to the Project from public concern and opposition to the movement of large quantities of hazardous materials along the main thoroughfare (Hwy-50). The potential for human error and accidents along that corridor are necessarily greater than using rail.	Opportunity/Threat	The potential for human error and accidents.	Health & Safety	Financial/ Costs	Minor	Possible	6	Accept	Health and safety training, awarenes and adherance.	Minor	Possible	6