Amended NI 43-101 Technical Report Updated Preliminary Economic Assessment Elk Creek Niobium Project Nebraska

Effective Date: August 4, 2015
Original Report Date: September 4, 2015
Amended Report Date: October 16, 2015

Report Prepared for

NioCorp Developments Ltd.

Report Prepared by

srk consulting

ROCHE

SRK Project Number: 241900.030

Signed by Qualified Persons:

Martin Frank Pittuck, MSc, CEng, MIMMM (SRK Corporate Consultant, Mining Geology)
Benjamin Parsons, MSc, MAusIMM (CP) (SRK Principal Consultant, Resource Geology)
Vladimir I. Ugorets, PhD, MMSAQP (SRK Principal Consultant, Hydrogeologist)
Eric Larochelle, BEng (Roche Director, Specialty Metals & Hydrometallurgy)
Alain Dorval, BSc (Roche Manager, Mining and Mineral Processing)
Joanna Poeck, BEng Mining, SME-RM, MMSAQP (SRK Senior Consultant, Mining Engineer)
Jeff Osborn, BEng Mining, MMSAQP (SRK Principal Consultant, Mining Engineer)
John Tinucci, PhD, PE (SRK Principal Consultant, Geotechnical Engineer)
Clara Balasko, MSc, PE (SRK Senior Consultant, Civil Engineer)
Mark Willow, MSc, CEM, SME-RM (SRK Principal Environmental Scientist)
Valerie Obie, BS Mining, MA, SME-RM (SRK Principal Consultant, Mineral Economics)

Reviewed by:

Peter Clarke, BSc, PEng (SRK Principal Consultant, Mining Engineer)

Amended NI 43-101 Technical Report
Updated Preliminary Economic Assessment
Elk Creek Niobium Project
Nebraska

Report Prepared for

NioCorp Developments Ltd.
7000 South Yosemite Street, Suite 115
Centennial, CO 80112

Report Prepared by

SRK Consulting (U.S.), Inc.

Suite 600
1125 Seventeenth Street
Denver, CO 80202
e-mail: denver@srk.com
website: www.srk.com
Tel: +1.303.985.1333
Fax: +1.303.985.9947

And

Roche Ltd., Consulting Group

Centre d’affaires Henri-IV
1015 Wilfrid-Pelletier Ave.
Québec, QC G1W 0C4
Canada

SRK Project Number 241900.030

Effective Date: August 4, 2015
Original Report Date: September 4, 2015
Amended Report Date: October 16, 2015

Table of Contents

List of Tables

List of Figures

List of Appendices

1 Summary

This report was prepared as a Canadian National Instrument 43-101 (NI 43-101) Technical Report, Updated Preliminary Economic Assessment (PEA) for NioCorp Developments Ltd. (NioCorp or the Company) by SRK Consulting (U.S.), Inc. (SRK), and Roche Ltd, Consulting Group (Roche), (collectively referred to as the Consultants) on the Elk Creek Niobium Project (Elk Creek or the Project) located in southeast Nebraska. NioCorp was formerly known as Quantum Rare Earth Developments Corp. (Quantum), but changed its name to NioCorp effective March 3, 2014.

1.1 Property Description and Ownership

Elk Creek is an early stage exploration project located in southeast Nebraska, USA. It is located approximately 75 kilometers (km) southeast of Lincoln, Nebraska (the state capital), and 110 km south of Omaha, Nebraska. The mineralization is centered 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 work force 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 1,796 hectares (ha), of which the Company currently hold 15 active agreements (1,216 ha), with the remaining 7 option agreements currently undergoing re-negotiation. Option agreements are between NioCorp's subsidiary Elk Creek Resources Corp. (ECRC) and the individual land owners. 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 Agreements (ELAs), 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 land owners have title to the surface and subsurface rights, and the agreements are primarily with respect to 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 the exploration work. The options agreements that the company currently holds include all of the indicated and inferred resources described in this report.

1.2 Geology 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 (Sc) and is composed predominantly of dolomite, calcite and ankerite, with lesser chlorite, barite, phlogopite, pyrochlore, serpentine, fluorite, sulfides and quartz. Niobium is contained primarily within the mineral pyrochlore, and rare earth element (REE) mineralization is reported to occur as bästnasite, parisite, synchysite and monazite. Niobium has been the main element of interest for the current study, but recent developments since the November 2014 Technical Report within the metallurgical testwork indicates the potential to recover TiO_2 and Sc_2O_3 as part of the proposed process flowsheet. Work remains on-going to optimize and further test this at a pilot stage, but based on the work completed to date SRK considers these elements to have potential for economic extraction and therefore are discussed in the Technical Report, and included in the Mineral Resource Estimate.

1.3 Status of Exploration, Development and Operations

Drilling at the Project was conducted in three phases. The first was during the 1970's and 1980's 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.

Five holes for a total length 3,353.1 m, of additional drilling been drilled since the completion of the April 28, 2015 Mineral Resource Estimate. This drilling has been for the purpose of Hydrogeological and Geotechnical studies. No sampling has been completed of these holes to date and therefore they have not been considered for the Mineral Resource.

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.

SRK reviewed and validated the electronic database provided and concludes that the sampling methods, Quality Assurance/Quality Control (QA/QC), and database management practices employed by NioCorp are all at or above industry standards, and are suitable for use in resource estimation.

1.4 Mineral Processing and Metallurgical Testing

1.4.1 Mineral Processing and Metallurgical Testing

Comminution testwork performed at SGS Canada Inc. (SGS) indicated that the mineralized material is considered relatively hard giving a Bond Ball Mill Work Index of 14.5 kWh/t and not very abrasive giving an abrasion index of 0.066.

Developmental flotation testwork with mechanical flotation cells was performed by Hazen Research Inc. (Hazen) at SGS in Lakefield, Ontario in 2014 and early 2015. To achieve proper liberation with direct pyrochlore flotation, the flotation feed was ground to a P_{80} of 20 µm. Flotation column testwork was subsequently performed at Eriez and, with the use of wash water, provided superior results than that achieved using conventional flotation techniques conducted without froth washing. A final combined concentrate of 5.6% niobium pentoxide (Nb_2O_5) at a mass yield of 11.2%, with a Nb_2O_5 recovery of 72.6% was achieved at Eriez. COREM subsequently performed an intensive column flotation pilot plant testwork program during the first half of 2015. The cleaning flotation stage did not provide the desired metallurgical results in terms of mass pull versus recovery; more time, effort, and optimization would have been required. The difference between Eriez results and COREM results may be explained due to the fact that all the conditioning and flotation steps at COREM were done on a continuous basis and the mineralized material is very sensitive to reagent dosages. Hydrometallurgy testwork showed that direct leaching of the ground mineralized material (without flotation) significantly increased the recoveries associated with the process. As a result, the flotation testwork was put on hold in pursuit of whole ore direct leaching.

1.4.2 Hydrometallurgy and Metallurgical Testing

Metallurgical testwork were first conducted at SGS throughout 2014 and 2015 to properly design the required process units for the conversion of the flotation concentrate into a niobium product suitable for further treatment into ferroniobium. Testwork consisted of an exploratory bench and pilot scale hydrometallurgical test program aimed at defining an appropriate flowsheet using different reagents and technologies. Upon further consideration of the recoveries and in particular the scandium recovery being very low in the flotation, leach test work was conducted on coarse whole ore material. A leach using hydrochloric acid was introduced followed by the original sulfation. Coarse whole ore leach testwork showed that a high recovery of the scandium could be achieved without any added losses of titanium or niobium. A preliminary flowsheet was then established based on testwork performed in leaching, sulfation, purification and precipitation. Recoveries of 92% Nb_2O_5, 87.6% TiO_2 and 90% Sc_2O_3 have been demonstrated by the testwork performed to date.

1.4.3 Pyrometallurgy and Metallurgical Testing

Preliminary pyrometallurgical testwork has been carried out at XPS Consulting and Testwork Services in Sudbury, Ontario, Canada, with subsequent alumino-thermic reduction tests performed at Kingston Process Metallurgy (KPM) in Kingston, Ontario, Canada. Successful alumino-thermic reduction of Nb concentrates produced FeNb alloy metal. Four preliminary bench scale tests were performed; demonstrating the successful conversion of the niobium oxide in the niobium precipitate into a ferroniobium metal product. Though 85% Nb recovery was measured, a higher Nb recovery of over 95% to the FeNb alloy is to be expected based on literature and existing operations. Given the low P levels in the Nb concentrate feedstock, the % P in alloy is likely to be low at 0.1%, and would meet sales specifications.

1.5 Mineral Resource Estimate

Mineral Resources have been estimated in conformity with generally accepted Canadian Institute of Mining (CIM) Estimation of Mineral Resource and Mineral Reserves Best Practices guidelines, and are reported in accordance with the CIM Definition Standards – For Mineral Resources and Mineral Reserves, May 10, 2014.

The drillhole database used in the estimation is of high quality and has been independently verified by SRK. A three-dimensional geologic model was constructed using ARANZ Leapfrog® Mining Software (Leapfrog®). Modeling was based on logged geology in the drilling database, using a combination of geological controls and niobium grade shells. The grade estimation was confined to a hard boundary of three grade shell domains defined at 0.3%, 0.4% and 0.5% niobium pentoxide (Nb_2O_5%), with the estimation using only the composited samples from the same domain.

Developments in the metallurgical testwork indicated the potential to recover TiO_2 and Sc based on a revised flowsheet. Further, the Company completed a re-assay program of historical pulps, which were not included in the original 2011/2012 re-assay programs. The updated database has been provided to SRK who completed a review of the database and the QA/QC information for the additional elements to ensure their inclusion in the estimation process. External laboratory checks showed a bias between SGS and Actlabs laboratories, with Actlabs returning higher values for Nb_2O_5. The slight high bias confirms the slight over reporting noted in the routine submissions of standard reference materials (SRMs), which SRK estimates to be in the order of 4.0% to 4.4% (based on the SRMs). The bias in the external duplicates report a slight increase in the bias to 8.7%. SRK considers the level of bias to be within acceptable levels for use in the current Mineral Resource. SRK noted some gaps for TiO_2 and Sc still remain within the database. The gaps in the database are a result of the current re-assay program being limited to pulp material collected during the 2011 reanalysis program. Based on established relationships and statistical analysis, SRK is comfortable to use the revised database for the current Mineral Resource Estimation.

Mineral Resources utilized all the assay information from historical drilling and the NioCorp 2014 drilling program. Search distances were determined from directional variograms calculated using the capped and composited samples. A nested search ellipse estimation method consisting of three passes was used. The search ellipse has been rotated into the main dip and strike orientation of the deposit.

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 a nearest neighbor estimate completed as cross checks. Search distances were determined from directional variograms calculated using the capped and composited samples. A nested search ellipse estimation method consisting of three passes was used. The search ellipse has been rotated into the main dip and strike orientation of the deposit. Density has been assigned based on mean density per major geological unit from density determination values taken during the 2014 estimation program, using a combination of weight in air/weight in water, and volumetric analysis. Resources are reported as Nb_2O_5, TiO_2 and Sc (ppm).

Density has been estimated based on density determination values taken during the 2014 resource estimation program, using a combination of weight in air/weight in water, and volumetric analysis. Based on a statistical review of the density measurements and the assay results from the whole rock analysis, which including Fe_2O_3% and TiO_2%, a general trend of higher density with higher grade was identified and therefore an Ordinary Kriged estimate of density was chosen as the preferred option.

SRK has validated the Mineral Resource Estimates using a number of different validation techniques:

- Inspection of block grades in plan and section and comparison with drillhole grades;
- Comparative statistical study vs. composite data and alternative estimation methods; and
- Sectional interpretation of the mean block and sample grades (swath plots).

In the opinion of SRK, the Mineral Resource Estimate reported herein is a reasonable representation of the global Nb_2O_5 grades and the location of higher grade zones, which provide a reasonable underground mining target at the Project.

The Mineral Resources are classified under the categories of Indicated and Inferred according to CIM guidelines. Due to a lack of dense drilling (in the order of 35 m x 35 m), no Measured Mineral Resources have been assigned at this stage.

Classification of the resources reflect the relative confidence of the grade estimates. This classification is based on several factors including sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization, data verification to original sources, specific gravity determinations, accuracy of drill collar locations, accuracy of the topographic surface, quality of the assay data, and many other factors.

For the Indicated resource classification, a solid shape was constructed around the core of the deposit where most drillholes are spaced approximately 60 to 70 m apart, and blocks have been estimated with a minimum of two boreholes.

The Mineral Resources have been confined to estimated blocks within the Carbonatite. In order to determine the quantities of material offering “reasonable prospects for economic extraction” by an underground mining method, SRK has defined an underground mining cut-off grade (CoG) based on metal pricing, assumed costs and metallurgical recoveries. Costs and recoveries are based on bench mark studies completed for similar projects, and application of possible local variations. The blocks above the mining CoG form contiguous mining targets without isolated blocks that would be unlikely to warrant the cost of development. All material within the geological wireframes above a CoG of 0.3% Nb_2O_5 has been considered to have reasonable prospects of being mined via underground methods.

The Mineral Resource for the Project is summarized in Table 1.5.1, with a summary of the sensitivity of the tonnage and grade to CoG shown in Table 1.5.2.

Table 1.5.1: SRK Mineral Resource Statement for Elk Creek, 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 reasonably expects the Project to be amenable to a variety of Underground Mining methods. Using results from initial metallurgical testwork, 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, CEng, 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. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

Table 1.5.2: Grade Tonnage Showing Sensitivity of the Mineral Resource to CoG

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	0.60	59,700	0.82	489,200	2.94	1,750,000	74.2	4,400
	0.55	63,400	0.80	507,200	2.92	1,850,000	74.0	4,700
	0.50	65,200	0.79	515,000	2.91	1,900,000	73.9	4,800
	0.45	65,800	0.79	520,100	2.90	1,910,000	73.8	4,900
	0.40	68,100	0.78	531,000	2.87	1,950,000	73.7	5,000
	0.35	72,800	0.75	545,700	2.79	2,030,000	73.2	5,300
	0.30	80,500	0.71	571,600	2.68	2,160,000	72.0	5,800
Inferred	0.60	44,600	0.78	347,800	2.94	1,310,000	67.6	3,000
	0.55	50,700	0.76	385,100	2.92	1,480,000	67.3	3,400
	0.50	53,300	0.75	399,400	2.92	1,550,000	67.1	3,600
	0.45	54,300	0.74	401,600	2.91	1,580,000	66.9	3,600
	0.40	58,400	0.72	420,500	2.83	1,650,000	66.8	3,900
	0.35	67,500	0.67	452,400	2.69	1,810,000	66.0	4,500
	0.30	99,600	0.56	558,000	2.31	2,300,000	63.0	6,300

Source: SRK, 2015

1.6 Recovery Methods

The ferroniobium processing facility is designed with three distinct operation units: a mineral processing plant including a grinding circuit, designed to reduce the particle size prior to leaching; a hydrometallurgical plant (hydromet), which is designed to produce Nb_2O_5, scandium oxide (Sc_2O_3), and titanium oxide (TiO_2); and a pyrometallurgical plant (pyromet), designed to produce ferroniobium, an iron-niobium alloy.

The selected processing method for the mineral processing plant includes one crushing stage, one grinding stage, and one thickening stage. These three stages combined will deliver feed of the correct particle size and moisture content to the hydrometallurgy plant. Direct leaching of the ground mineralized material showed significantly increased recoveries, thus flotation is not present in the flowsheet.

The hydromet plant consists of eleven production units: Hydrochloric (HCl) Acid Leach, HCl Acid Scandium Extraction, Sulfuric Acid Bake and Water Leach, Iron Reduction and Crystallization, Sulfuric Acid Scandium Extraction, Niobium Precipitation and Phosphorus Removal, Titanium Precipitation, Sulfate Calcining, HCl Acid Regeneration, Tailings Neutralization, and a Sulfuric Acid Plant. Combined, these units will convert the ground mineralized material into three products: niobium oxide concentrate, scandium oxide, and titanium oxide.

The pyrometallurgical plant will take the niobium oxide and convert it via alumino-thermic reduction to ferroniobium. This reduction is performed in a single FeNb furnace, to produce a saleable FeNb metal alloy. The pyrometallurgical plant includes a niobium concentrate dryer, a furnace feed preparation area, a batch weighing and charging system, the FeNb furnace, a tapping and casting system, a slag granulation system, and a crusher and screen for the final FeNb product.

1.7 Underground Mining

Based on geomechanical information and mineralization geometry an underground longhole stoping (LHS) method with paste backfill is suitable for the deposit. The production rate is 2,700 t/d producing about 7,500 t ferroniobium product (FeNb) per annum, at average life-of-mine (LoM) grades of 0.80% Nb_2O_5, 2.84% TiO_2 and 73 ppm Sc. The deposit is divided into three blocks where the blocks are mined top down but the levels within a block are mined bottom up. Sill pillars are left in situ between blocks. Stopes are 15 m wide, 25 m tall, and vary in length based on the mineralization grade.

Mine design using Vulcan™ software was completed based on an estimated net smelter return (NSR) cut-off grade (CoG) of US$180/t. Stope optimization using elevated CoGs was used to determine mine plan resource areas and achieve the desired average grade for each level.

Table 1.7.1 summarizes the mine plan resource. This estimate is based on a mine design using elevated CoGs and applying the US$180/t NSR CoG to material within the design. These numbers include a 95% to 100% mining recovery based on type of opening (stope, development, etc.) to the designed wireframes in addition to a 0% to 5% unplanned waste dilution. An additional development allowance of 26% was applied to main ramps and 19% to level accesses to account for detail currently not in the design. A 7% additional allowance was applied to stopes where arched backs were not designed at the average grade of the stope. This percentage was determined based on percentage of stopes within the design where there is no stope above. Waste dilution for stopes was applied with grade, slightly lower than the cutoff grade, based on an analysis of the material around stopes in a representative area.

Table 1.7.1: Mine Plan Resource Classification (1)

Category	Tonnes (kt)	Nb_2O_5 (%)	TiO_2 (%)	Sc (ppm)
Measured	-	-	-	-
Indicated	31,086	0.80	2.84	73
Measured + Indicated	31,086	0.80	2.84	73
Inferred	-	-	-	-

Source: SRK, 2015
(1) Includes Measured and Indicated material reported using an NSR CoG of US$180/t.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

The design was then scheduled using iGantt software to generate a LoM production schedule summarized in Table 1.7.2.

Table 1.7.2: Annual Mining Schedule

Year	Mineralized Tonnes (kt)	Nb_2O_5 (%)	TiO_2 (%)	Sc (ppm)	Waste Tonnes (kt)	Backfill (m^3)
2016	-	-	-	-	66.7	-
2017	219.5	0.57	2.32	56.45	178.3	-
2018	986.9	0.76	2.85	62.50	71.1	239,379
2019	986.7	0.82	2.78	75.26	101.5	280,380
2020	984.6	0.82	2.74	69.01	128.0	348,168
2021	985.4	0.83	2.99	57.99	92.4	299,505
2022	986.6	0.79	2.84	67.35	97.0	304,887
2023	989.4	0.79	2.74	63.98	19.6	280,802
2024	986.2	0.81	2.85	71.42	7.9	269,808
2025	986.7	0.81	2.68	74.22	2.6	272,831
2026	986.1	0.79	2.84	79.40	3.8	273,712
2027	986.4	0.79	2.77	78.15	39.8	295,551
2028	985.4	0.79	2.77	80.57	40.4	315,736
2029	986.7	0.79	2.78	76.94	98.7	300,471
2030	986.6	0.79	2.77	76.32	110.1	323,672
2031	986.1	0.79	2.80	73.27	37.2	291,889
2032	989.1	0.79	2.83	75.10	9.1	267,658
2033	985.6	0.82	2.90	69.56	2.6	319,168
2034	985.6	0.87	2.92	71.11	-	375,794
2035	985.3	0.81	2.90	78.23	5.1	300,217
2036	985.4	0.81	2.92	67.33	1.1	307,560
2037	985.3	0.81	2.99	66.79	3.0	299,341
2038	985.8	0.80	2.91	78.88	4.2	305,000
2039	985.7	0.80	2.85	72.15	-	353,149
2040	985.6	0.82	2.92	64.35	1.8	312,336
2041	985.7	0.80	2.79	79.46	5.6	285,008
2042	985.7	0.82	2.85	73.72	-	374,333
2043	985.8	0.80	2.81	75.66	4.4	343,292
2044	989.9	0.79	2.78	80.79	2.5	271,278
2045	985.6	0.80	2.98	76.19	-	302,082
2046	985.7	0.81	2.86	78.45	-	441,975
2047	985.5	0.80	2.88	78.89	-	357,130
2048	985.5	0.84	3.00	81.11	-	354,014
2049	513.0	0.68	2.59	71.84	-	107,709
Total	**31,085.5**	**0.80**	**2.84**	**73.33**	**1,134.5**	**9,773,835**

Source: SRK, 2015

Mining operations within a stope include establishing top and bottom accesses, developing a slot raise at the far end of the stope (hangingwall side) and using a fan shaped drilling pattern to blast rings on retreat toward the level access. Drifting development such as main ramps and level accesses are sized as 5 m x 5 m openings with an arched back. Drifting top/bottom stope accesses are sized as 4.5 m x 4.5 m flat back openings. All drifting work is developed using two boom jumbos. Ramps are designed at a maximum gradient of 15% with a 40 m turning radius which is suitable for any underground truck. Stope and development material is mucked using 14 t LHD’s into 40 t underground trucks for haulage.

The mine is accessed through a shaft system. The surface facilities include a hoist house, headframe structure, hoisting system, and compressor room. The underground system includes a shaft, cages, skips, skip loader, and muck handling system at the bottom of the shaft. An emergency escape system is included in the exhaust (return air raise) air raise.

A paste backfill plant will be located on surface and the paste backfill product will be made of fly ash from a local (74 km away) coal power plant. Sand will be used as an aggregate source to regulate the strength gain characteristics of the paste. The paste will be fed by gravity via boreholes from surface.

The main ventilation during production will consist of main fans located underground near the dedicated 5.5 m exhaust raise. Fresh air will be taken down the shaft and be directed across the levels through the active workings and development faces and then exhaust through the exhaust vent raise. An auxiliary ventilation system moves air to the working faces and consists of fans and ducting. Air refrigeration is not necessary; however heating is required during some months of the year.

A surface well dewatering system will be used. After dewatering from the surface the mine is expected to produce approximately 18.9 L/sec of water and a mine dewatering system will be required. The dewatering system will be installed during development at the first sill level and will consist of a sump and pump system capable of pumping 63 L/sec.

The mine will operate on a 12 hour shift basis, 365 days per year, and the quantity of personnel and equipment are based on the production schedule.

1.8 Infrastructure

The Project will incorporate surface and underground infrastructure as well as tailings storage facilities. The off-site infrastructure includes a new 29 km 161 kV high voltage line from a delivery point on the existing regional power system, constructed by the local power utility. A 9 km natural gas pipeline connecting the site with a local utility's existing system will be constructed by the local gas utility. Telecommunications in the form of an optic line will be connected to a hub approximately 1.5 km from the site. A 7 km railway and rail unloading/loading/transfer facilities will support movement of chemicals and product to and from the site.

The on-site surface infrastructure will include an electrical substation and distribution system, on-site telecommunications, fuel storage and delivery system, process water system, water treatment, potable water system supplied by a nearby community, fire protection system, sewage system, natural gas distribution to site loads, access roads to the site with parking, fencing and security, laboratory, mine and process administrative and services buildings, warehouse, paste backfill plant, and maintenance shop. The mining related facilities will include a lined mine waste rock and mineralized material storage area, growth media storage area, surface water control facilities, and explosives storage area. The mine surface facilities include a headframe, hoist, and associated facilities. The underground will be serviced by a shaft and ventilation raise. The return air raise will have a fan system underground with an emergency hoist located at the surface.

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, service water, compressed air distribution, paste backfill distribution system, and ventilation system. The underground material handling system includes a grizzly, feeder, crusher, storage bin, conveyor, and skip loader system that loads skips in order to move the mined material to the surface facility.

Active mine dewatering will be utilized beginning before mine production and will include a series of surface wells that will dewater the mine and discharge into a lined collection pond. The water will then be pumped via a 50 km discharge pipeline to a diffuser at the discharge point at the Missouri River.

The two tailings storage facilities (TSF) will be constructed to contain the water leach residue, gypsum residue, neutralization residue, and iron oxide tailings, the first in Area 7, later in the mine life Area 1. Both TSFs will be constructed in phases. Area 7 will be constructed in three phases, storing approximately 26.2 Mt of tailings. Area 1 will be constructed in two phases, storing an additional 29.6 Mt to meet the LoM requirements. The tailings facilities 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.

1.9 Environmental Studies and Permitting

1.9.1 Required Permits

The Project will be subject to the permitting requirements of Johnson County, the State of Nebraska, and the U.S. Environmental Protection Agency and U.S. Army Corps of Engineers’ (USACE) national policies, such as the National Environmental Policy Act (42 U.S.C 4321, the Clean Air Act (42 U.S.C. 7401 *et seq*.), and the Clean Water Act (CWA) (33 U.S.C. 1251 *et seq*.). The list permits and authorizations required (or likely to be required) for the Project are presented in Section 20.1.

Project permitting commenced in January 2015 with the submission of a jurisdictional delineation report to the USACE. In addition, several high-level meetings with local, state and federal agencies have been held in order to introduce the Project to the regulatory community.

One of the most critical of the required permits and/or authorizations for Elk Creek will be the approvals to construct the mine, plant and tailings disposal facilities, as they cover considerable area, and cross various water features that fall under the jurisdiction of the state and federal governments. At the time of this report, all wetlands and waters/drainages in the Project study area have been assumed to be jurisdictional and subject to USACE regulation; however, no formal determination has yet been made by the agency. This conservative approach is being used by the various engineering groups to design the operations with as minimal potential impacts to federally jurisdictional features as possible. However, at this time, the delineation of the proposed rail and discharge pipeline corridors is still pending.

Section 404 of the federal CWA establishes a program to regulate the discharge of dredged or fill material into waters of the U.S. (WOUS), including wetlands and jurisdictional drainages/waterways. The USACE could require either a General Permit or an Individual Permit if the potential impacts to jurisdictional areas are deemed significant. Regardless, those potential impacts will need to be evaluated and disclosed through the National Environmental Policy Act (NEPA) process. The NEPA process generally involves one of two levels of analysis:

- Preparation of an Environmental Assessment (EA) and *Finding of No Significant Impact* (FONSI) where no significant impacts are expected or the potential impacts are unknown; or
- Preparation of an Environmental Impact Statement (EIS) where there is a potential for significant impacts.

It is important to remember that both EAs and EISs are public disclosure documents, not permit or approval documents. They are intended to disclose what, if any, environmental impacts may occur from the Project and guide the decisions of federal agencies. In the end, both NEPA processes are likely to result in the development of compensatory mitigation for the loss of wetlands and other jurisdictional features.

The time to review and evaluate the actual 404 Permit application is typically overshadowed by the NEPA review of the Project impacts. The time to complete an EA (generally accepted at approximately 12 months) is usually less than an EIS (3 to 5 years), as there are no statutory time frames and fewer bureaucratic procedures involved. Both include public scoping and public review processes. NioCorp's current understanding is that the simpler EA is likely the route to be taken by the USACE with respect to Elk Creek given the design emphasis that has been placed on limiting impacts to wetlands and riparian resources. NioCorp has initiated mitigation discussions with the USACE and commenced preparation of the formal permit application. However, inclusion of the dewatering water pipeline and discharge to the Missouri River, as well as several of the alternative discharge options are still under consideration and study, and are pending discussion with the USACE.

The other important permitting challenge will be dealing with the trace amounts of uranium and thorium that occur in the mineralized materials, and may ultimately be deposited in the tailings disposal facility. Preliminary discussions with the State of Nebraska have indicated that a Broad Scope Radioactive Materials License, issued by the Nebraska Department of Health and Human Services (DHHS), may likely be necessary for dealing with these Naturally Occurring Radioactive Materials (NORM). NioCorp estimates that a Broad Scope License for Elk Creek will take six to nine months to obtain, and will involve several months of discussions and negotiations related to engineering, design, monitoring, and terms and conditions.

Because the Project includes a primary sulfuric acid production plant [a regulated facility under 40 CFR §52.21(b)], and since Nebraska is classified as "attainment" of all ambient air quality standards , a federal Prevention of Significant Deterioration (PSD) air quality 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 modeling). Typically, however, PSD permits require over one year in order to complete. A federal operating permit will also be required; however, the application for the operating permit need only be submitted within 12 months after the emissions unit(s) begin operation.

Other important permit requirements prior to construction and operations include:

- National Pollutant Discharge Elimination System (NPDES) permit for the surface discharge of excess waters generated from mine dewatering and possibly mineral processing;
- Dam Safety permitting for the proposed tailings disposal facility and possible water storage reservoirs; and
- Greenhouse Gas (GHG) permitting if carbon dioxide thresholds are exceeded.

1.9.2 Engineering Design Criteria

The State of Nebraska does not have regulatory environmental protection requirements for the design and operation of hardrock mines, especially underground hardrock mines with chemical beneficiation circuits. As such, NioCorp has engaged in a conservative approach to minimize environmental risk and liability by adopting relevant Environmental Design Criteria (EDC). Without state or federal guidance in this matter, the EDCs for Elk Creek were fashioned after those from a jurisdiction dedicated to sustainable hardrock mining; the State of Nevada and the U.S. Bureau of Land Management. However, Nebraska does have regulations pertaining to the management of solid wastes, including mining wastes.

1.9.3 Environmental Studies

Preliminary information on various environmental resources has been collected from available literature as well as previous studies in the area of the Elk Creek Project. These include:

- Soils;
- Climate/Meteorology/Air Quality – a meteorological station has been erected at the site;
- Cultural and Archeological Resources;
- Vegetation;
- Wildlife;
- Threatened, Endangered, and Special Status Species;
- Land Use;
- Hydrogeology (groundwater);
- Hydrology (surface water);
- Wetlands/Riparian Zones; and
- Geochemistry.

1.9.4 Health and Safety

Occupational health and safety at the Project will be strictly regulated by the U.S. Department of Labor, Mine Safety & Health Administration (MSHA), including the possible implementation of radon exposure and monitoring requirements on all workers.

1.9.5 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, including, but not limited to requirements for the closure of the tailings disposal facility under the Nebraska *Integrated Solid Waste Management Regulations*.

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 would be under the scenario in which the facility falls under a broad scope radiological license, which will likely have financial assurance requirements for reclamation and closure. At this time, the type and amount of financial surety for the Project has not yet been established.

Current closure costs for the Project have been estimated at just over US$60 million, the bulk of which (US$40 million) is intended for reclamation and closure of the tailings disposal facility. These costs will be refined as part of the feasibility study, and may need to be adjusted based on specific regulatory agency requirements, particularly those associated with any radioactive material licensing of the plant and tailings facility.

1.9.6 Community Relations and Social Responsibilities

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, which includes local landowners, county representatives, and several state and federal regulatory agencies.

1.9.7 International Standards and Guidelines

Even though the United States is a ‘Designated Country’ with respect to the Equator Principles, NioCorp has committed to ensuring that Elk Creek is in compliance with international standards and guidelines, to the extent practicable, given the potential for international investment in the Project.

1.10 Preliminary Economic Assessment Results

Capital Cost Estimates

Table 1.10.1 contains a summary of capital costs for the underground development and operations of the Project. Capital costs include the design, procurement and construction of the underground mine and surface mine infrastructure, processing plants and auxiliary facilities, and infrastructure. At this level of study, and with the work performed to-date, the capital cost estimate is at an accuracy of +/- 25%.

Table 1.10.1: Capital Cost Summary

Description	Initial (US$000’s)	Sustaining (US$000’s)	LoM (US$000’s)
Mining	$177,269	$108,028	$285,298
Process	$391,220	$0	$391,220
Tailings and Infrastructure	$187,948	$228,658	$416,606
Owners Costs/Land Acquisition	$56,593	$0	$56,593
Closure Costs	$0	$71,309	$71,309
Contingency	$165,711	$0	$165,711
Total Capital	**$978,742**	**$407,995**	**$1,386,738**

Source: SRK, 2015

Operating Cost Estimates

The operating costs are based on processing 2,700 t of mineralized material per day to produce an average of 7,500 t/y of ferroniobium (rounded). The operating costs are based on Q1-2015 costs, and the estimate has been broken down into three main areas: mining costs (mine), processing costs (process), and general & administration (G&A).

The mine operating cost is estimated at US$53.00/t of the mineralized material milled and includes the manpower, energy, spares and maintenance supplies required for the underground development and production of the mineralized material as well as the paste backfill plant and underground distribution system, underground pumping systems, and ventilation.

The process operating cost is estimated at US$135.75/t of the mineralized material milled and consists of the manpower, energy, consumables, reagents, acid, spares and maintenance supplies required for the operation of the mineral processing, hydrometallurgical, acid plant and pyrometallurgical plants as well as the operating costs of the fresh water supply and treatment, surface dewatering wells and pumps, and tailings disposal.

The general & administration operating cost is estimated at US$8.11/t of the mineralized material milled. This includes all of the project's operating costs which are not related to the mining and processing plants. The G&A costs include the following subsections: administration manpower, and general costs for operations.

The overall LoM operating cost for the Project is estimated at US$6.1 billion, US$196.86/t mineralized material milled or US$39.28/kg of Nb (excluding benefit from the production of Sc_2O_3 and TiO_2). A summary of the operating costs for the Project is shown in Table 1.10.2 All costs presented are in US dollar per mineralized material milled or kg of Nb.

Table 1.10.2: Operating Cost Summary

Description	US$/t-Processed	US$/kg-Nb	LoM (US$000's)
Mine	$53.00	$10.58	$1,647,647
Process	$135.75	$27.09	$4,219,864
G&A	$8.11	$1.62	$252,000
Total	**$196.86**	**$39.28**	**$6,119,511**

Source: SRK, 2015

1.10.1 Indicative Economic Results

The technical economic model developed for the Project is on an after-tax basis and assumes 100% equity to provide a clear picture of the technical economic merits of the operation.

Table 1.10.1.1 outlines the model parameters used in the economic analysis for the base case scenario.

Table 1.10.1.1: Model Parameters

Description	Value	Units
Mine Life	32	years
Mineralized Material Processed	31,086	kt
Payable FeNb	239.7	kt
Payable TiO_2	766.7	kt
Payable Sc_2O_3	3.1	kt
FeNb Price (LoM avg)	$43.55	US$/kg
TiO_2 Price (LoM avg)	$2.10	US$/kg
Sc_2O_3 Price (LoM avg)	$3,883	US$/kg
Effective Tax Rate	23.9%	
Discount Rate	8%	

Source: SRK, 2015

The after-tax net present value (NPV) at an 8% discount rate over the estimated mine life is US$2.3 billion. The Project economic results are summarized and presented in Table 1.10.1.2.

Table 1.10.1.2: Economic Analysis (US$000's)

Description	Value	Units
Market Prices		
Niobium	**$43.55**	/kg
Titanium Oxide	**$2.10**	/kg
Scandium Oxide	**$3,883**	/kg
Estimate of Cash Flow (all values in $000's)		
Gross Revenue	**$18,925,111**	**$608.81**
Operating Costs		**US$/t-RoM**
Mining	($1,647,647)	$53.00
Processing	($4,219,864)	$135.75
G&A	($252,000)	$8.11
Product Freight	($97,800)	$3.15
Property/Severance taxes	$0	$0.00
By-product Credits (1)	1,610,089	($51.80)
Royalties	(286,358)	$9.21
Treatment Cost/Refining Cost	0	$0.00
Cash Closure/Reclamation	0	$0.00
Total Operating Costs	**($4,893,580)**	**$157.42**
Operating Margin (EBITDA)	**$14,031,532**	**$451.38**
Project Capital	($978,742)	$31.49
LoM Sustaining Capital	($336,686)	$10.83
Closure Costs	(71,309)	$2.29
Taxes	($3,033,191)	$97.58
After Tax Free Cash Flow	**$9,611,603**	**$309.20**
NPV @: 8%	**$2,301,735**	
Average Annual Niobium Production	**4,868,185**	**kg/y**
Average Annual Ferroniobium Production	**7,490**	**t/y**

(1) By-product credits of TiO_2
Source: SRK, 2015

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

1.10.2 Conclusions and Recommendations

Geology and Resources

SRK has no further recommendations for additional drilling needed to support the Mineral Resource for the impending feasibility-level study. SRK notes that the current understanding of the extents of the deposit, while sufficient for the current level of study, are still limited by the extents of the drilling, and that the deposit is locally open along strike and at depth. SRK is of the opinion that there may be an opportunity to further refine and improve the understanding of the mineralization, particularly near the top of the deposit where mining is scheduled to begin. This may be considered in planning for future exploration and mining, as a matter of course in the development of the project.

Recovery Methods

In order to improve process efficiency and minimize the potential risks of operating a full-scale plant, testing programs need to be carried out during the different phases of engineering studies. While some small-scale test methods provide adequate information for scoping or prefeasibility studies, it is suggested that pilot plant testing be conducted to provide sufficient information for the process development during the feasibility study at the ±15% accuracy levels. At the current stage of the study, a bench scale laboratory testing program including some mini-pilot testing has been conducted for understanding the mineralized material sample characteristics and its behavior under controlled conditions. During this program testing, sufficient data has been collected from the hydromet circuit to produce a PEA and has justified the need for a more detailed evaluation with a pilot plant program. Implementation of such testwork will provide additional key information to confirm bench test results and enable development of mass and energy balances, equipment selection and plant design. As process safety risk is an important factor, a pilot plant program will help to reduce possible risks associated with the construction and operation of the new full-scale process plant.

Mining and Reserves

No Mineral Reserves have been estimated for the Project. The available data indicate that underground operations using longhole stoping methods are viable for the Project. The mine maintains the target FeNb production for a 32 year period. An elevated NSR cut-off was used to minimize plant and capital requirements and to meet NioCorp forecasted market needs. Development of the shaft, initial ramp and accesses is imperative to achieving production in early years. Further optimization during a feasibility study would include overall optimization of mining system to minimize up front capital cost, accelerate initial development, optimize ventilation, further develop water handling systems as more data on water becomes available, and refine the paste backfill system.

Tailings Storage Facility

The Area 7 and Area 1 TSFs are capable of storing the tailings material generated for the life of mine. Further development of the design of the facility includes further characterization of the tailings material, construction material characterization including confirmation of the engineering properties of the materials, geochemical property review, and further developing a feasibility level design including appropriate geotechnical, water balance, and seismic characteristics.

Environmental Studies and Permitting

Initiation of formal permitting will commence upon completion of this PEA. While not necessarily complex, the timing required to complete permitting through the U.S. Army Corps of Engineers (404 Permit), the Nebraska Department of Environmental Quality (NPDES Discharge Permit), and the Nebraska Department of Health and Human Services (Radiological License), necessitates early engagement with all three agencies. Documentation of existing baseline environmental conditions at the site was initiated in 2014 and should continue throughout the permitting process. Geochemical programs for the characterization of the mineralized material (potential mineralized material), waste rock, and tailings (including radiological characterization) has been collected based on a preliminary PEA mine plan needs to be expanded for the feasibility study. Post-metallurgical geochemical testing of the tailings material is necessary to obtain solids and supernatant chemistry, and generate data needed to evaluate the closure alternatives for the underground workings and tailings impoundment, and the potential requirements for post-closure water management, if necessary. Additionally, development of project-specific environmental and social management plans based on the potential impacts identified during the permitting process will need to be initiated.

Capital, Operating Costs, and Economic Analysis

The Project as modeled provides a positive after-tax NPV of US$2.3 billion at an 8% discount rate with free cash flow of US$9.6 billion after taxes. The Project generates approximately 7,500 t/y of FeNb, 97 t/y Sc_2O_3 and a by-product of TiO_2 that offset substantial costs at current commodity price estimates. The upfront capital is US$978.7 million. The Project is net NPV positive through sensitivities of +/-25% on operating cost, capital cost, and recovery. Market pricing for FeNb and Sc_2O_3 was based on reputable market studies developed for the Project.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

2 Introduction

2.1 Terms of Reference and Purpose of the Report

This report was prepared as a Canadian National Instrument 43-101 (NI 43-101) Technical Report, Updated Preliminary Economic Assessment (PEA) for NioCorp Developments Ltd. (NioCorp or the Company) by SRK Consulting (U.S.), Inc. (SRK), and Roche Ltd, Consulting Group (Roche), (collectively referred to as the Consultants) on the Elk Creek Niobium Project (Elk Creek or the Project) located in southeast Nebraska. NioCorp was formerly known as Quantum Rare Earth Developments Corp. (Quantum) but changed its name to NioCorp effective March 3, 2014.

This report provides estimates of Mineral Resources within a PEA design mine plan, and a classification of resources prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards – For Mineral Resources and Mineral Reserves, May 10, 2014.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK'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 set forth in this report. This report is intended for use by NioCorp subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits NioCorp to file this 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 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.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

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, geotechnical, environmental, permitting, metallurgical testing, mineral processing, process design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this 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 normal professional consulting practice.

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 report, and are members in good standing of appropriate professional institutions. The QP's are responsible for specific sections as follows:

- ***Martin Frank Pittuck, MSc, CEng, MIMMM (SRK Corporate Consultant, Mining Geology)*** is the QP responsible for data verification and the mineral resource estimate Sections 12, and 14 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- ***Benjamin Parsons, MSc, MAusIMM (CP) (SRK Principal Consultant, Resource Geology)*** provided assistance in the preparation of the geological model and Mineral Resource Estimate under the guidance of Martin Pittuck. Mr. Parsons is the QP responsible for Sections 4 to 11 (except 5.4.1) and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- ***Vladimir Ugorets, PhD, MMSAQP (SRK Principal Consultant, Hydrogeologist)*** is the QP responsible for hydrogeology Sections 16.3, dewatering portion of 18.3, 20.3.8 and portions of Sections 1 and 26 summarized therefrom, of this Technical Report.
- ***Eric Larochelle, BEng (Roche Director, Specialty Metals & Hydrometallurgy)*** is the QP responsible for metallurgical testing and recovery methods Sections 13 (except 13.1) and 17 (except 17.1), and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- ***Alain Dorval, BSc (Roche Manager, Mining and Mineral Processing)*** is the QP responsible for mineral processing plant and infrastructure Sections 13.1, 17.1, 18 (except for 18.1.2, 18.2 and 18.3) and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- ***Joanna Poeck, BEng Mining, SME-RM, MMSAQP (SRK Senior Consultant, Mining Engineer)*** is the QP responsible for mining and reserves Sections 15, 16 (except 16.2, 16.3, 16.7.3. 16.7.4, 16.7.6, 16.8.2 through 16.8.6) and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- ***Jeff Osborn, BEng Mining, MMSAQP (SRK Principal Consultant, Mining Engineer)*** is the QP responsible for mining and infrastructure Sections 2, 3, 16.7.3. 16.7.4, 16.7.6, 16.8.2 through 16.8.6), 18.1.2, 23, 24, 27, 28 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- ***John Tinucci, PhD, PE (SRK Principal Consultant, Geotechnical Engineer)*** is the QP responsible for geotechnical Section 16.2 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- ***Clara Balasko, MSc, PE (SRK Senior Consultant, Civil Engineer)*** is the QP responsible for TSF Sections 5.4.1, 18.2, pipeline portion of 18.3, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- ***Mark Willow, MSc, CEM, SME-RM (SRK Principal Environmental Scientist)*** is the QP responsible for environmental studies, permitting and social or community impact Section 20 (except 20.3.8) and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.

- ***Valerie Obie, BS Mining, MA, SME-RM (SRK Principal Consultant, Mineral Economics)*** is the QP responsible for market studies, capital and operating costs and economic analysis Sections 19, 21 and 22 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.

2.3 Details of Inspection

Martin Pittuck (QP) visited the Project property between June 17 to 19, 2014. This included a cursory inspection of the deposit area, the exploration camp and sample preparation prior to dispatch. SRK has not visited the laboratory during the site inspection as all samples are shipped to Canada for analysis.

Vladimir Ugorets (QP) visited the Project property between September 8 to 10, 2014. This included an examination of core, supervising of installation of monitoring wells, supervision of hydrogeological testing, conducting control water level measurements and slug tests, observation of surface-water bodies, and reconnaissance of the Project area for assessment of boundaries of groundwater flow model (developed for the predictions of dewatering requirements for proposed underground mine).

Alain Dorval and Eric LaRochelle visited the property on October 21, 2014. This included an orientation to the company's land holdings and the resource, as well as an evaluation of locations for surface facilities and project infrastructure.

Mark Willow visited the property between June 1 to 3, 2015. This included an orientation to the company's land holdings and the resource, surface and groundwater conditions and meetings with key state agencies.

Clara Balaska visited the property on numerous occasions in November 2014 and June 2015 to supervise geotechnical field work associated with surface facilities and infrastructure.

SRK was given full access to relevant data requested, and conducted discussions with junior and senior project geologists regarding exploration procedures and interpretations.

Table 2.3.1 presents a site visit summary.

Table 2.3.1: Site Visit Participants

Personnel	Company	Expertise	Date(s) of Visit
Martin Pittuck	SRK Consulting	Overall QP for Mineral Resource Estimate	June 17 to 19, 2014
Cody Bramwell	SRK Consulting	Field Geologist/ Geotechnical	Site rotations May 2014 – December 2014
Dave MacDonnell	SRK Consulting (Associate)	Field Geologist/ Geotechnical	Site rotations May 2014 – December 2014
Shawn White	SRK Consulting (Associate)	Field Geologist/ Geotechnical	Site rotation July 20 – 31, 2015
Geoffrey Baldwin	SRK Consulting	Hydrogeology	Site rotations July 6, 2014 – November 5, 2014 February 20, 2015 – July 31, 2015
Paul Williams	SRK Consulting	Hydrogeology	June 6 to 15, 2014 June 26 to July 9, 2014 August 16 to 20, 2014 September 8 to 10, 2014 October 30 to November 9, 2014 March 27 to April 1, 2015 April 29 to May 2, 2015 May 18 to 22, 2015 June 8 to 24, 2015 July 13 to 14, 2015
Vladimir Ugorets	SRK Consulting	QP Hydrogeology	September 8 to 10, 2014
Goktug Evin	SRK Consulting	Hydrogeology	September 8 to 10, 2014
Mike Brewer	SRK Consulting	Hydrogeology	September 3 to 14, 2014 (approx..)
Clara Balasko	SRK Consulting	Tailings Geotechnical	November 10 to 15, 2014 June 22 to 24, 2015
Nikoliya Boyanich	SRK Consulting	Tailings Geotechnical	November 11 to 25, 2014
Jeevan Neupane	SRK Consulting	Tailings Geotechnical	June 22 to July 7, 2015

Source: SRK, 2015

2.4 Sources of Information

The sources of information include data and reports supplied by NioCorp personnel as well as documents cited throughout the report and referenced in Section 27.

2.5 Effective Date

The effective date of the report is August 4, 2015.

2.6 Units of Measure

The metric system has been used throughout this report. Tonnes are metric of 1,000 kg, or 2,204.6 lb. All currency is in United States dollars (US$) unless otherwise stated.

3 Reliance on Other Experts

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

SRK was reliant upon information and data provided by NioCorp including historic data inherited from previous owners. NioCorp have utilized the services of Dahrouge Geological Consulting Ltd. (Dahrouge) for the capture and databasing of the historical data, plus on-site geological management for the 2011 and 2014 exploration programs. SRK has been provided with adequate copies in digital format of the historical logs and provided full access to the Dahrouge dataroom. SRK has, where possible, verified data provided independently, and completed a site visit to review physical evidence for the Project.

SRK has relied upon information supplied by NioCorp (Mr. Scott Honan) during this current study. Land titles and mineral rights for the Project have not been independently reviewed in detail by SRK and SRK did not seek an independent legal opinion of these items.

SRK has relied upon market studies provided by NioCorp. The confidential marketing studies for ferroniobium were developed by Roskill (June 2015), a leader in international metals and minerals research for market reports. The confidential scandium study entitled "Scandium: A Market Assessment" was developed by OnG Commodities LLC (July 2015) and authored by Dr. Andrew Matheson. Dr. Matheson has extensive experience in specialty metals and consulted to global firms in the metals, materials and energy industries. The companies that provided the reports or information have specific knowledge of the specific commodity market and pricing. Pricing for titanium dioxide was estimated based on a lower quality product and in-house knowledge of the titanium market. Potential upgrade to the quality of the titanium dioxide produced to a pigment grade product could have a positive impact on market price. The scandium pricing includes price and market assumptions concerning an expanded demand in the scandium market. Based on the marketing study a slower uptake in demand in the scandium market will likely result in a lower initial price. SRK presented an alternative scenario to provide an indication of the sensitivity for the Project to scandium pricing. SRK has, where possible, verified data provided independently through review of available public documents.

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.

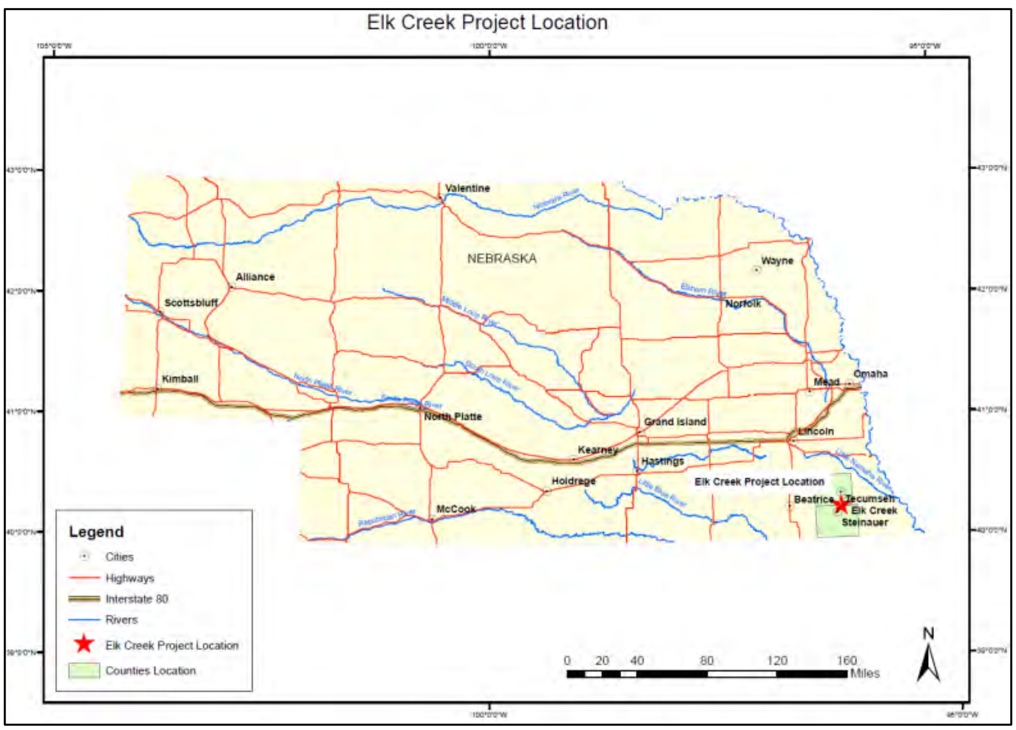
These items have not been independently reviewed by SRK and SRK did not seek an independent legal opinion of these items.

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 4.1.1 below and is located as follows:

- Within 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 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; and
- Approximately 5 km southeast of the Nemaha River a tributary of the Missouri River.



Source: SRK, 2014

Figure 4.1.1: Project Location Map

4.2 Property Description

The Project is a niobium-bearing carbonatite deposit located in Johnson County, southeast Nebraska. In addition to niobium, other elements of economic significance include titanium and scandium.

4.3 Mineral Titles

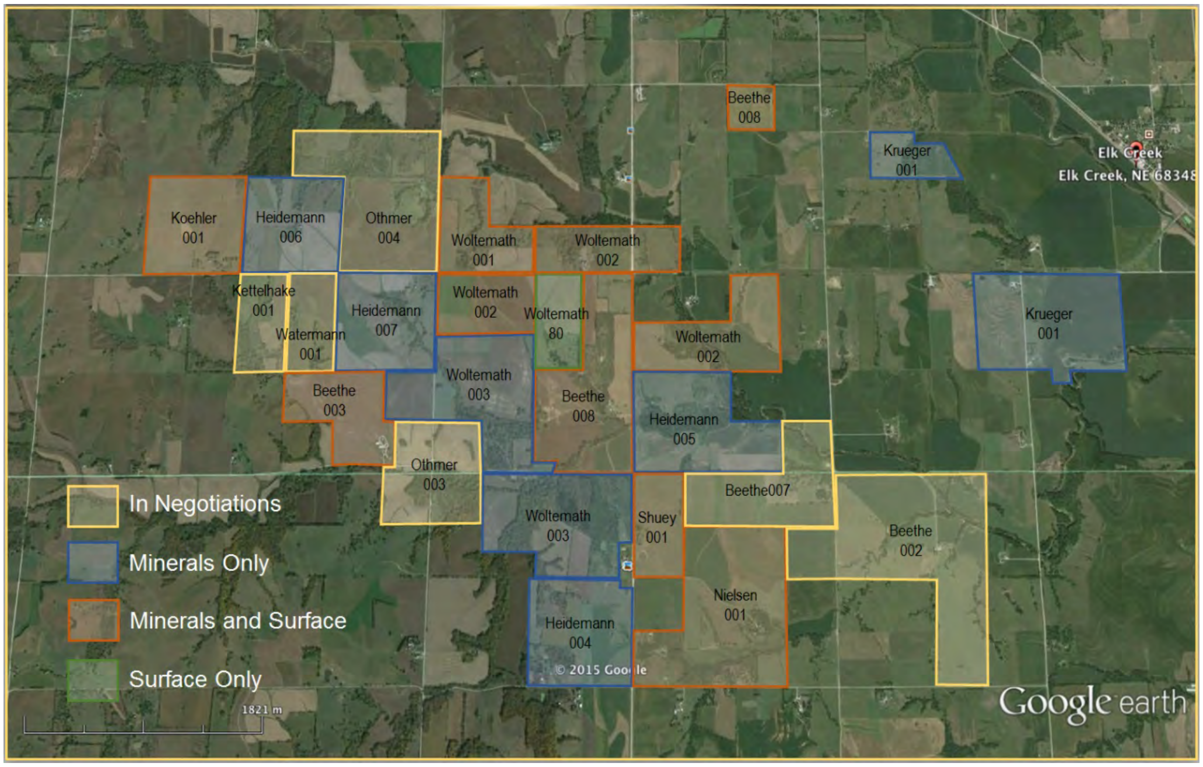
At the time of the previous PEA the Property consisted of 65 option agreements covering approximately 3,834 ha, of which the Company currently held 33 active agreements (1,879 ha).

As of the effective date of this Technical Report, the Company has reviewed the required option agreements to advance the Project and has chosen not to enter into an extension agreement on 44 agreements. These agreements covered mainly the larger area targeted in previous drilling campaigns. The Company current holds 15 of 21 targeted Land Option agreements and are in active negotiations with the remaining land holders. The current optioned land package covers an area of 1,215.96 ha, with the targeted package at 1,796.05 ha.

Option agreements are between NioCorp's subsidiary ECRC and the individual land owners. ECRC is a Nebraska based and wholly owned subsidiary of NioCorp. Land ownership is shown in Figure 4.3.1 and listed in Table 4.3.1. The two agreements covering the Mineral Resources are currently held by the Company and have been extended for a five year period. These two agreements are shown in bold on Table 4.3.1.

SRK has not researched property title or mineral rights for the Project and expresses no opinion as to the legal ownership status of the Project. As part of the option agreements the Company has where required secured surface rights to be able to conduct exploration work, as required to develop the Project.

The Mineral Resource is located in two option areas identified with bold text in Table 4.3.1; these are located on Section 33; Township 4N; Range 11; on the Tecumseh Quadrangle mapsheet.



Yellow polygons highlight option agreements are not held by ECRC, with infilled blue polygons indicating option agreements for minerals only. For the 80 acre parcel north of Beethe008, NioCorp has an option to purchase the surface rights and negotiations to secure the mineral rights are underway.
Source: Niocorp, 2015

Figure 4.3.1: Land Tenure Map

Table 4.3.1: Active Lease Agreements Covering the Project

Option Agreement Name	Code	Hectares	Acres	Original Agreement Sign Date	Original Agreement Expiry	New / Extended Agreement Expiry
Beethe, Elda E	**Beethe008**	**107.82**	**266.43**	**April 30, 2010**	**April 30, 2015**	**April 30, 2020**
Beethe, Harlan D. and Lisa M	Beethe003	48.69	120.32	April 15, 2010	April 15, 2015	June 24, 2020
Heidemann, Lavon L. and Robin Y	Heidemann003	48.56	120.00	March 17, 2010	March 17, 2015	March 17, 2020
Heidemann, Lavon L. and Robin Y	Heidemann004	62.96	155.58	March 15, 2010	March 15, 2015	March 15, 2020
Heidemann, Lavon L. and Robin Y	Heidemann005	79.55	196.57	March 16, 2010	March 16, 2015	March 16, 2020
Heidemann, Leland L. and Lola L	Heidemann006	64.75	160.00	March 26, 2010	March 26, 2015	March 26, 2020
Heidemann, Leslie L	Heideman007	64.75	160.00	March 25, 2010	March 25, 2015	March 25, 2020
Koehler, Robert and Ellen	Koehler001	64.75	160.00	June 4, 2010	June 4, 2015	June 12, 2020
Krueger, Gregory A and Joyce R	Krueger001	123.41	304.95	December 18, 2009	December 18, 2014	December 18, 2019
Nielsen, Rolande O. and Tami R	Nielsen001	112.81	278.75	March 31, 2010	March 31, 2015	June 25, 2020
Woltemath Roger - 80 acres surface	Woltemath80S	32.37	80.00	Not applicable	Not applicable	December 4, 2019
Woltemath, Eileen M	Woltemath001	48.47	119.77	December 4, 2009	December 4, 2014	January 21, 2020
Woltemath, Roger L. and Nancy A	Woltemath002	152.49	376.81	December 4, 2009	December 4, 2014	December 4, 2019
Woltemath, Victor L. and Juanita E	**Woltemath003**	**172.20**	**425.52**	**March 25, 2010**	**March 25, 2015**	**March 25, 2020**
Shuey, Dr. Keith	Shuey001	32.37	80.00	December 2, 2009	December 2, 2014	May 28, 2020
Othmer, Mark and Tom	Othmer003	75.89	187.52	February 10, 2010	February 10, 2015	Under Negotiation
Othmer, Mark and Tom	Othmer004	113.31	280.00	February 10, 2010	February 10, 2015	Under Negotiation
Watermann, Leona	Watermann001	145.69	360.00	May 6, 2010	May 6, 2015	Under Negotiation
Woltemath Family - 80 acres minerals	Woltemath80M	32.37	80.00	Not applicable	Not applicable	Under Negotiation
Beethe, Verlyn	Beethe007	66.27	163.75	April 14, 2010	April 14, 2015	Under Negotiation
Beethe, Glenn W	Beethe002	146.56	362.16	April 15, 2010	April 15, 2015	Under Negotiation
Kettelhake, Harold	Kettelhake001	32.37	80.00	June 9, 2010	June 9, 2015	Under Negotiation

Source: NioCorp, 2015

SRK notes that at the time of writing a number of the agreements included within Table 4.3.1 have expired but these do not directly influence the current Mineral Resource. NioCorp is currently negotiating with owners to obtain mineral and surface rights as appropriate. At the time of writing this report negotiations for the mineral rights are ongoing (personal communication, August, 2015). Surface rights only have been obtained for the “Woltemath_80S” option, located immediately north of the current Mineral Resource in an agreement dated December 4, 2014 with the surface owner, and are included in Table 4.3.1 as agreement Woltemath_80S.

The status of these agreements remains a current focus of the Company. SRK discussed the renewal process with NioCorp and understands that the Company is targeted all agreements covering potential Mineral Resource and any potential infrastructure. In areas outside of this the Company has made the decision to allow the Option agreements to expire.

The current Mineral Resource is wholly contained within parcels Woltemath_003 and Beethe_008, and extension agreements covering both of these properties have been secured. Negotiations for additional lands to support various configurations of the surface operations are underway, and the affected landowners are currently considering NioCorp’s offers to either extend the original agreements or enter into new agreements for these lands.

Discussion with NioCorp and review of the previous NI 43-101 completed in 2012, describe the option agreements and acquisition of the property by Quantum. Below is an excerpt from “Resource Estimate and Technical Report for the Elk Creek Nb-REE Project, Nebraska, USA”, completed by Tetra Tech Waldrop (Tetra Tech) for Quantum and dated April 23, 2012.

"The Property was acquired through [65] agreements between ECRC and individual land owners that are in the form of five-year pre-paid Exploration Lease Agreements (ELA), with an Option to Purchase (OTP) the mineral rights at the end of the lease (or for clarification at any point during the term). *The individual land owners have title to the surface and subsurface rights, and the agreements are primarily with respect to only the mineral interest of each property.*

The property boundaries are set out in a written description of each individual lease agreement. This property description is based on the Public Land Survey System (PLSS), descriptions of lots, and written descriptions of surface features (rivers, fences, roads, etc.).

The acquisition of the Elk Creek Property by Quantum involves the purchase of all of the issued and outstanding common shares of 859404 BC Ltd., ("859404") a private British Columbia company (Quantum News Release, Dec. 2010). 859404 holds 100% of the issued and outstanding shares of ECRC, the Nebraskan corporation that has secured individual agreements to acquire the mineral rights to the Elk Creek carbonatite. The property was held under a similar type of option agreement by Molycorp in the 1970's and 1980's.

In consideration for the common shares of 859404, Quantum will pay a total of US$500,000 and issue one common share of the Company for each common share of 859404 issued and outstanding. Of the total, US$200,000 has been paid by Quantum on signing of the agreement with 859404 and the balance of cash and shares is payable upon acceptance by the TSX Exchange."

It is SRK's opinion that ECRC's ability to securing the long term rights to land above and surrounding the Project will be key to completing a feasibility study.

4.3.1 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.

Some of the agreements 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 interest and in some cases the surface rights at any time during the term.

4.4 Royalties, Agreements and Encumbrances

The leases covering the Project are 100% owned by NioCorp and, with the exception of a 2% NSR royalty attached with some of the OTPs, have no other outstanding royalties, agreements or encumbrances.

4.5 Environmental Liabilities and Permitting

4.5.1 Environmental Liabilities

Existing environmental liabilities at the Project site are related to the exploration and hydrogeological and geotechnical investigation activities that have been undertaken to date. The Project consists of undeveloped (in terms of mining) farm land with no previous mineral development, mining or milling history. There are no existing liabilities associated with the utility rights-of-way or the highway (State Highway 50) in the Project area, which come under the responsibility of the company. A number of the option agreements, described above, provide for the establishment of an escrow account, where funds are deposited against the need to reclaim exploration areas once drilling is complete. At the time of writing, all reclamation work from the 2014 drilling programs has been completed, and all escrow monies have been released back to NioCorp. Five holes have been drilled in 2015, and escrow remains in place for three holes as the reclamation activity is ongoing at the time of writing.

Baseline environmental studies have been initiated and are discussed in Section 20 of the current report.

4.5.2 Required Permits and Status

The exploration work conducted to date on the Project has been completed under an Exploration Permit NE0211001 issued by the State of Nebraska, Department of Environmental Quality, which provided 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.

Subsequently, the proposed Project will be held to permitting requirements that are determined to be necessary by Johnson County, the State of Nebraska, and possibly the U.S. Army Corps of Engineers and Environmental Protection Agency, as is further detailed in Section 20 of this report.

4.6 Other Significant Factors and Risks

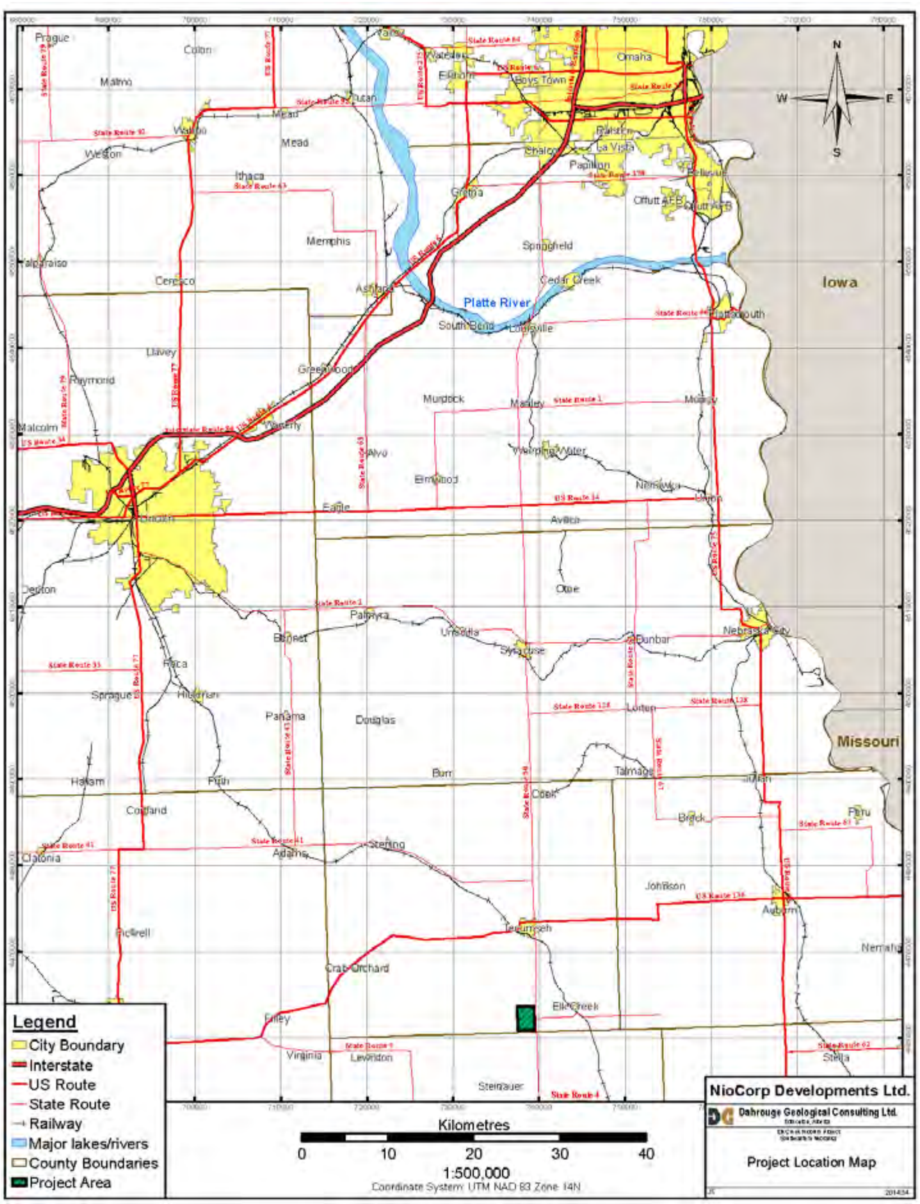
SRK notes that a potential risk for the development of the Project relates to the issue of renewing the current option agreements. SRK understands that the Company is currently in the process of renegotiating key options and agrees with the priority focusing on those directly above the mineralization and the surrounding leases, which may be required for surface infrastructure should the Project advance to more detailed levels of study. During this process, SRK highlights that the deposit remains open to the north of the current mineralization and there is potential to expand the deposit in that direction. No exploration has been completed into this area but it is known that main strike of the mineralization enters into the southwest corner of this parcel of land.

With the exception of the points raised above, 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 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 (Figure 5.1.1).



Source: Dahrouge, 2014

Figure 5.1.1: 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.

Geologists can be sourced from local universities. An experienced mining related workforce can be found in 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 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 (Table 5.2.1). Nebraska is located within an area known for tornados which runs 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 5.2.1: 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 (discussed in Section 4.3). It is expected that with appropriate studies and negotiations with land owners that land access and provision of land for infrastructure development will be achievable. There is sufficient 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 Infrastructure Availability and Sources

Elk Creek is the nearest town to the Project, with a population of approximately 100 people. Tecumseh, with roughly 1,700 inhabitants, is the nearest town of any size to the Project site and is situated approximately 11 km north of the Property. Tecumseh is well-suited as a staging base for future exploration work at the Project with available accommodations, fuel, and supplies. Contractors, bulk supplies, and skilled labor (engineering, surveying) may be sourced locally or from the cities of Lincoln or Omaha. Mining activities currently taking place in the area are limited to limestone and aggregate operations, to support the local cement manufacturing and construction industries. The Company has links to the University of Nebraska Lincoln which operates a geology department.

The Project is situated in a rural agricultural area that is covered by a well-developed network of paved highways and secondary gravel roads.

There are three electrical power generating stations within a 50 km radius of the Project that include the Beatrice and Sheldon coal generating stations, and the Cooper nuclear power generating station.

The nearest railway heads are found in both Tecumseh and Elk Creek. The Burlington Northern Santa Fe (BNSF) railway runs parallel to the Nemaha River connecting Kansas City to Omaha and Lincoln.

The nearest major airports are located in Lincoln and Omaha, Nebraska, and Kansas City, Kansas.

Water sources are available near the Property from local rivers and from groundwater wells for drilling requirements.

5.4.1 Potential Tailings Storage Areas

A TSF area required to support milling will need to be defined within the current mineral leases held by NioCorp. Further detail on the size and nature of this facility are discussed in Section 18.2 of the current report.

5.4.2 Potential Waste Rock Disposal Areas

A temporary disposal facility for waste rock (i.e., mined rock that does not contain economic concentrations of niobium, titanium or scandium) may be required to support mining and has been defined within the current land position held by NioCorp. Further detail is provided in Section 18.1 of this report.

5.4.3 Potential Processing Plant Sites

The Company holds sufficient surface rights to locate processing facilities at or near access to mineralization. The current options agreements contain sufficient land for the processing facilities.

5.5 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 meters above sea level (masl). Bedrock outcrop exposure is nonexistent in the Project area.

The majority 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 brief summary of the history of the Project and SRK has relied upon information provided in the 2012 NI 43-101 Technical Report produced by Tetra Tech for Quantum, entitled “Resource Estimate and Technical Report for the Elk Creek Nb-REE Project, Nebraska, USA”, effective date April 23, 2012.

6.1 Ownership History

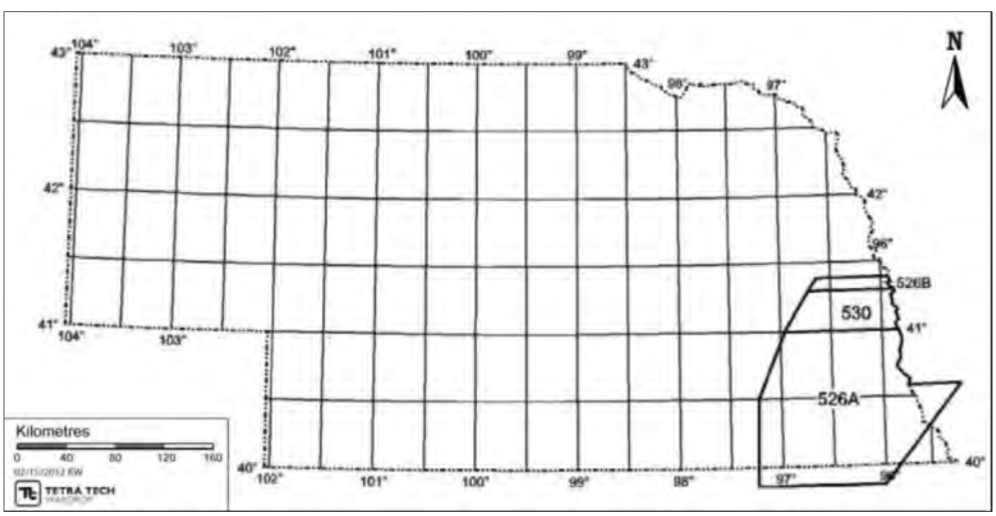
Initial regional geological work was completed by the USGS. The details of the initial ownership of the complete Project area are not clear, but it is reported that land packages were initially controlled by Cominco American Inc. (Cominco American) and Molycorp during the early 1970’s.

The majority of exploration over the Project area was completed by Molycorp prior to 1984. Between 1984 and 2010, at an unknown date, the title of the Project was held by Elk Creek Resources Corp. (ECRC). On May 4, 2010 Quantum announced the acquisition of ECRC and acquired 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

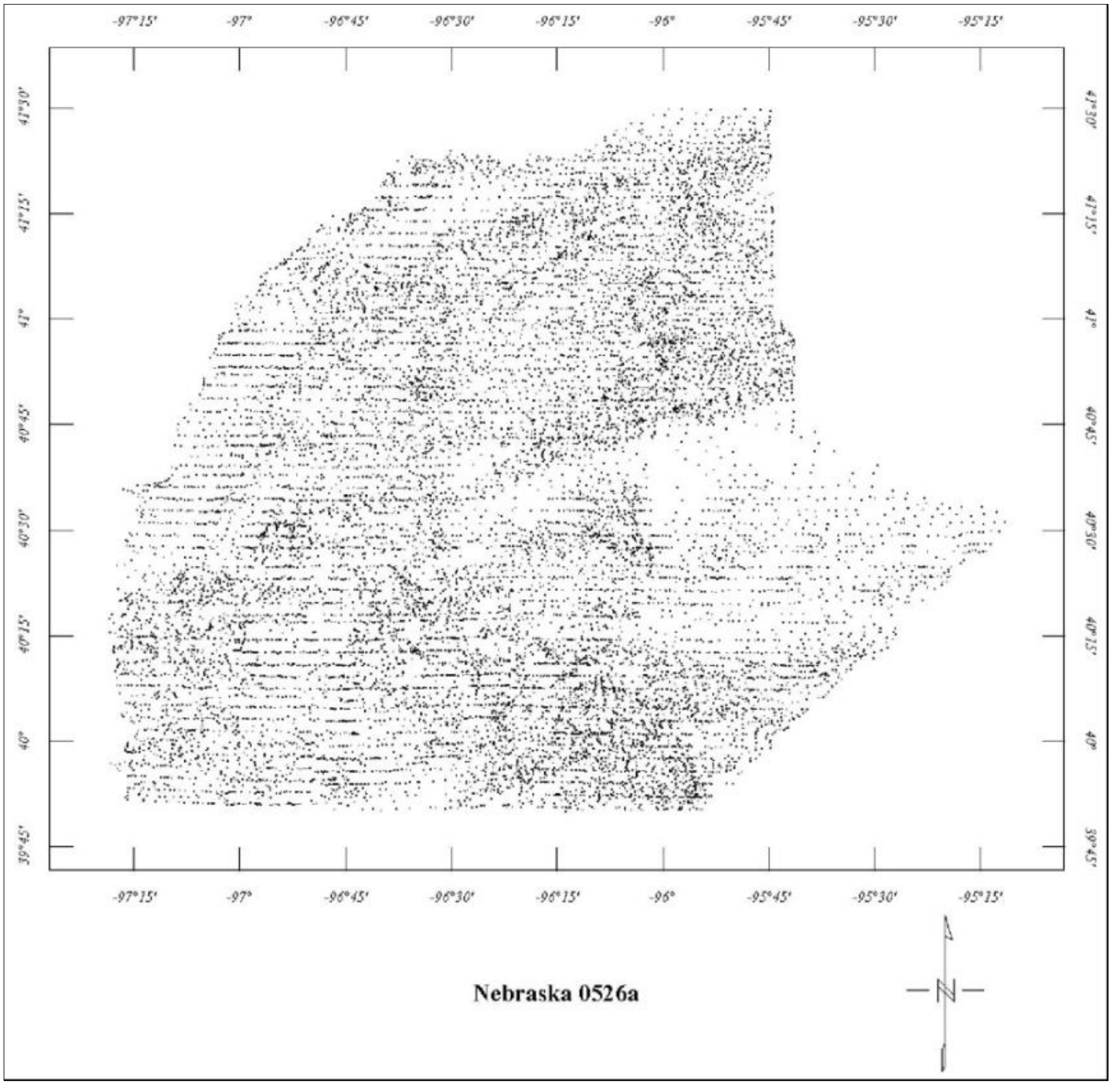
Between November 1963 and January 1964, the USGS flew three airborne magnetic surveys over southeast Nebraska. A total of 6,590 line km were flown (836, 209, and 5,544 line miles respectively) along east-west direction at a flight line spacing of 2 miles and at altitude of 305 m (1,000 ft) above ground (USGS website: OFR 99-0557). Figure 6.2.1.1 shows the area covered by the airborne survey.



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

Figure 6.2.1.1: 1964 USGS Aeromagnetic Survey Area Showing Surveys 526A, 526B, and 530 Respectively

The wide spacing of the flight lines illustrates only regional features and does not locate local anomalies (e.g., 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 6.2.1.2.



Source: Tetra Tech, 2012

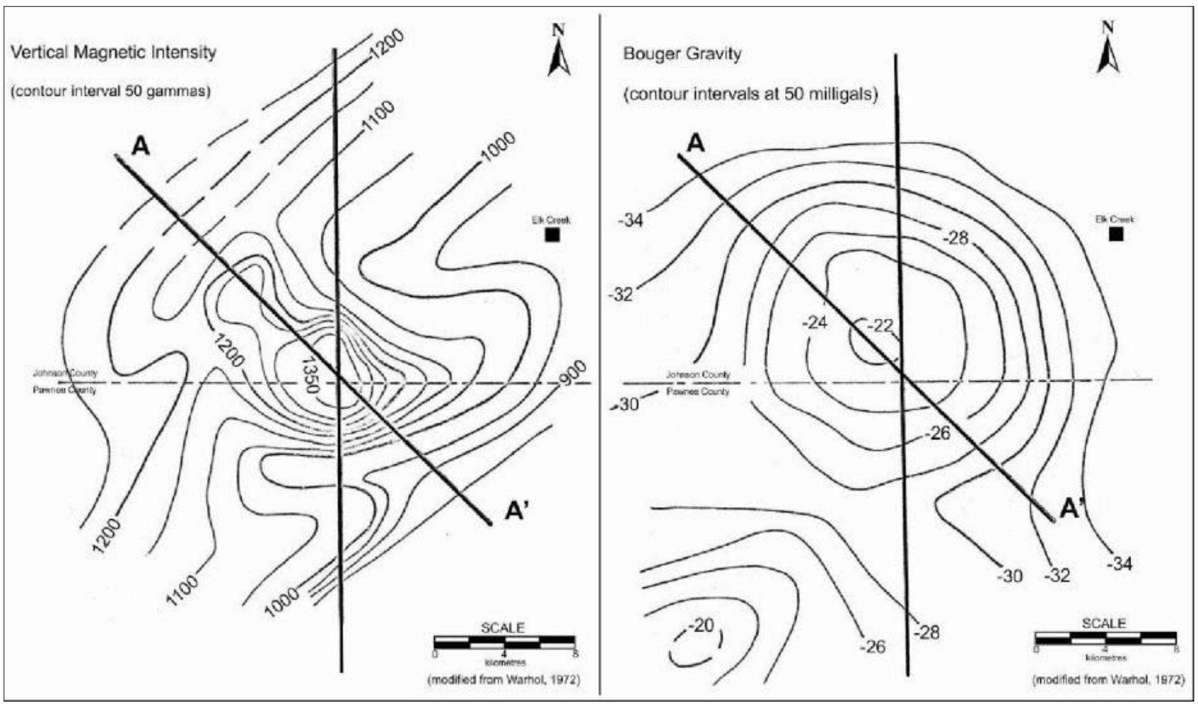
Figure 6.2.1.2: 1964 USGS Aeromagnetic Results (Merged 526A, 526B, and 530 Surveys)

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 time period the UNL geology department (operating independently), was mapping the magnetic expression of the Nemaha Arch and the Humboldt Fault.

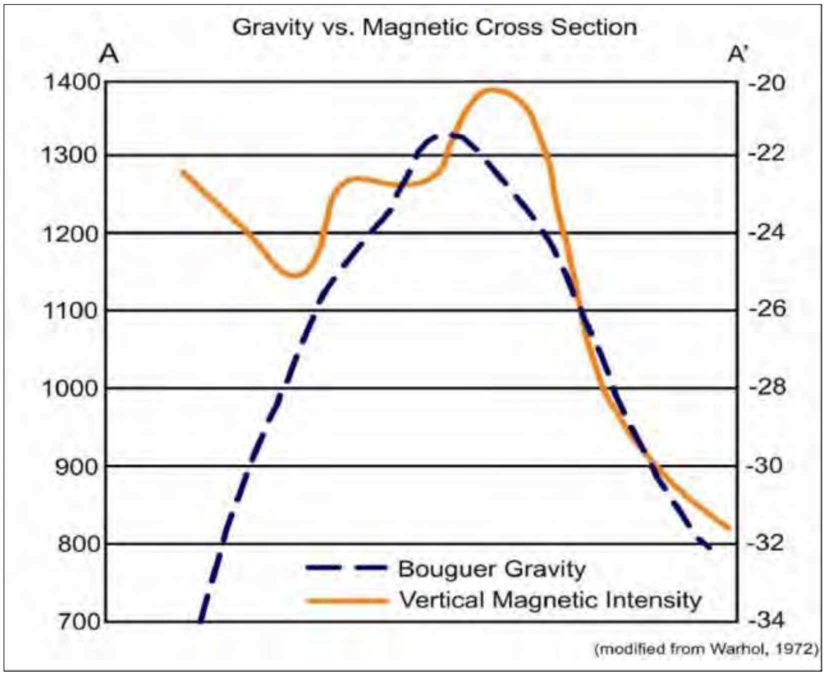
A comparison of the two geophysical survey results showed a positive anomaly that was coincident with a positive gravity anomaly over the area now defined as the Elk Creek gravity anomaly (Anzman, 1976). The geophysical gravity survey outlined a near-circular anomaly, along with a concurrent magnetic 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 1,700 m (5,500 ft; Burfeind et al. 1971). Figure 6.2.2.1 and Figure 6.2.2.2 illustrate the results of the two surveys.

In 1971, the Nebraska Geological Survey (NGS) commissioned a test drillhole 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 6.2.2.1: Comparison of the 1970 Magnetic and Gravity Geophysical Surveys



Source: Tetra Tech, 2012

Figure 6.2.2.2: 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 drillholes (CA-1 to CA-5) within the Elk Creek gravity anomaly. Details of the Cominco American drillholes and exploration activities within the property were not available. The information on drilling activities stated here was taken from the Molycorp database. SRK 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 were 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 drillholes: EC-1 to EC-4, targeting the Elk Creek anomaly and two other holes outside the Elk Creek anomaly area (Anzman, 1976). Drillholes 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 drillhole 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 centered on the Project within a radius of roughly 2 km. By 1984, Molycorp had completed 57 drillholes within the Elk Creek gravity anomaly area, which included 25 drillholes 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 by 610 m (roughly 2,000 by 2,000 ft.) with some closer spaced drillholes in selected areas.

By 1986, a total of 106 drillholes 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 were 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 reanalysis of historical samples.

The following excerpt was taken from McCallum and Cathro (Technical Report on the Elk Creek Property, 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 ¼ core duplicate). As discussed in Section 15, Molycorp Inc. utilized some standards to establish the calibration curve of the x-ray refraction (XRF) instruments. It is unclear if Molycorp also utilized inserted standard reference samples to test the analytical accuracy of their own laboratory or external commercial laboratories.

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% (with the exception of 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.

Subsequent to the 1983 "Niobium Analytical Standardization" report, Molycorp had 100 ft composite intervals of the majority of the drillholes (EC-1 to EC-105) sent to Metric Labs of Ste-Marthe-Sur-Le-Lac, Quebec for check analysis of niobium."

6.3 Historic 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 an NI 43-101 standard. The estimate is based on assay analysis conducted by

Molycorp at its own 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 its NI 43-101 Technical Report (April 2012) that the memo is the only evidence of an 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 an 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 drillhole spacing, drillhole location and sample data population.

The Mineral Resource Estimate for the deposit, at 0.4 Nb_2O_5% 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 6.3.2.1 and Table 6.3.2.2 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 6.3.2.1: Tetra Tech 2012 Indicated Mineral Resource Grade Tonnage Sensitivity for the Project

Cut-off Nb_2O_5 (%)	Density g/cm^3	Tonnes (000's t)	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

Source: Tetra Tech, 2012

Table 6.3.2.2: Tetra Tech 2012 Inferred Mineral Resource Grade Tonnage Sensitivity for the Project

Cut-off Nb_2O_5 (%)	Density g/cm^3	Tonnes (000's t)	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

6.3.3 SRK Estimate (September 2014)

In September 2014, SRK produced an NI 43-101 Technical Report for the Project based on the historical drillhole 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 drillhole spacing, drillhole location and sample data population.

The Mineral Resource Estimate for the deposit, at 0.3 Nb_2O_5% CoG, is 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$5%.

Table 6.3.3.1 presents the Indicated and Inferred Resource estimates for the Project, and Table 6.3.3.2 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 such as geotechnical and hydrogeological to improve the investigation and development of the Project.

Table 6.3.3.1: SRK Historical Mineral Resource Statement for the Project, Effective Date 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

Source: SRK, 2014

- 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.
- 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.
- SRK assumes the Project is amenable to a variety of Underground Mining methods. In the absence of definitive pricing for Nb and established rates of metallurgical recovery, SRK has 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.
- SRK Completed a site inspection to the deposit by Mr. Martin Pittuck, MSc., CEng, MIMMM, an appropriate "independent qualified person" as this term is defined in NI 43-101.

Table 6.3.3.2: 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, 2014

In February 2015, an initial estimate of the Nb_2O_5 Mineral Resource was completed by SRK Consulting on the Elk Creek deposit (Table 6.3.3.3). The estimate was based on the certified assays for Nb_2O_5 only, with the estimate subsequently updated on receipt of the TiO2 and Sc_ppm assays to produce the current Mineral Resource dated April 28, 2015.

Table 6.3.3.3: SRK Historical Mineral Resource Statement - Effective Date February 6, 2015

Classification	Cut-off (Nb_2O_5%)	Tonnage (000's Tonnes)	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

Source: SRK, 2014

- 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.
- 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.
- SRK assumes the Elk Creek deposit to be amenable to a variety of Underground Mining methods. Using results from initial metallurgical testwork, 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.
- 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.

6.4 Historic Production

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

7 Geological Setting and Mineralization

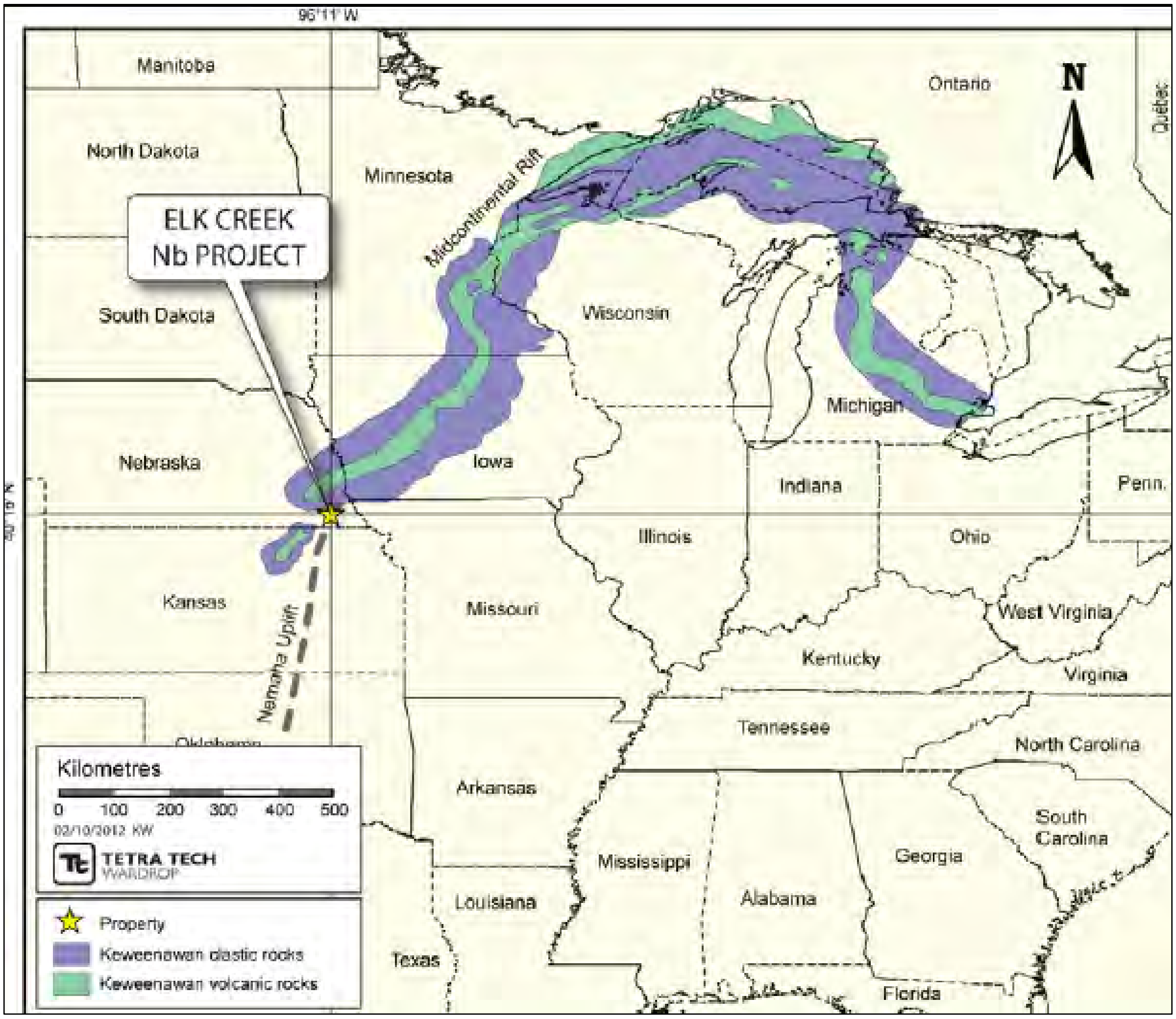
7.1 Regional Geology

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 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 7.1.1 illustrates the major rock types of the Midcontinental Rift system.

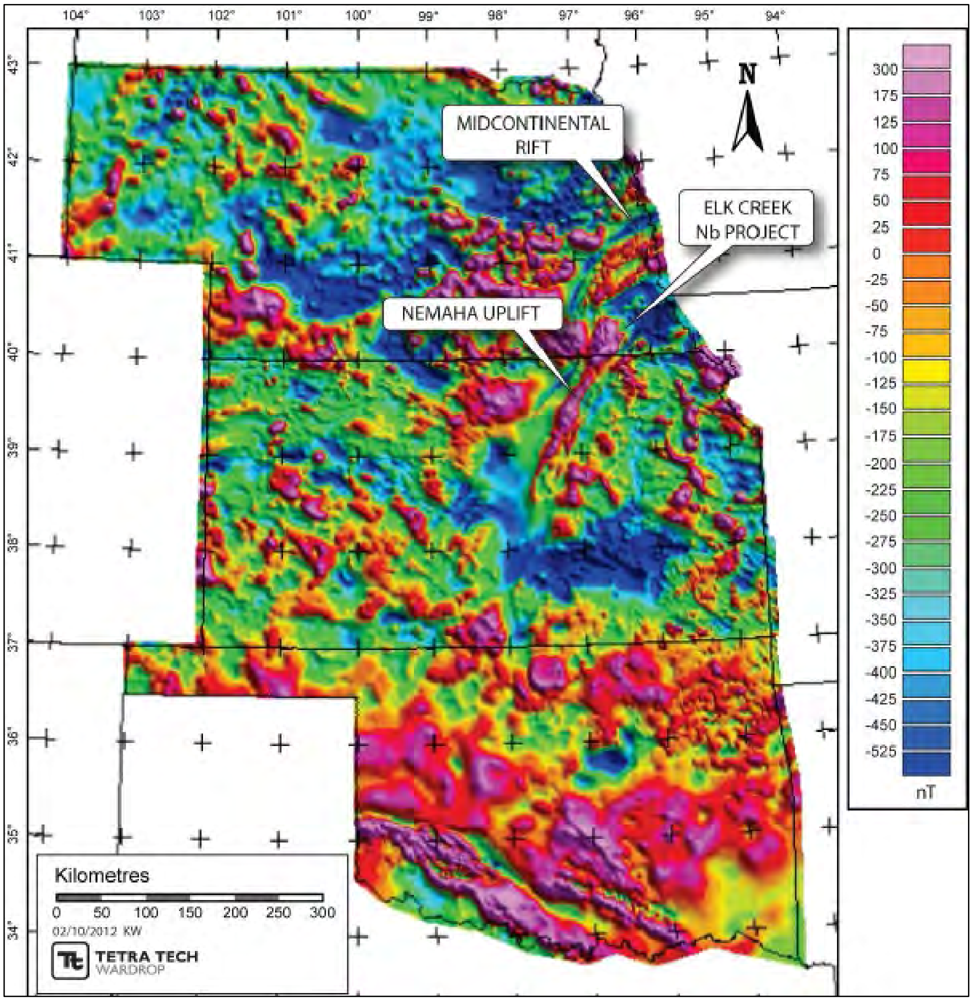
The Nemaha Uplift (300 Ma) extends southward as a narrow belt from southeastern Nebraska across Kansas along the midcontinent rift system (King, 1969) (Figure 7.1.1). Along the northern and eastern margins are complex fault zones and steeply dipping units. Regional north-northeast to northeast striking 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 7.1.2 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 7.1.1: Regional Geology Map



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

Figure 7.1.2: 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 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.

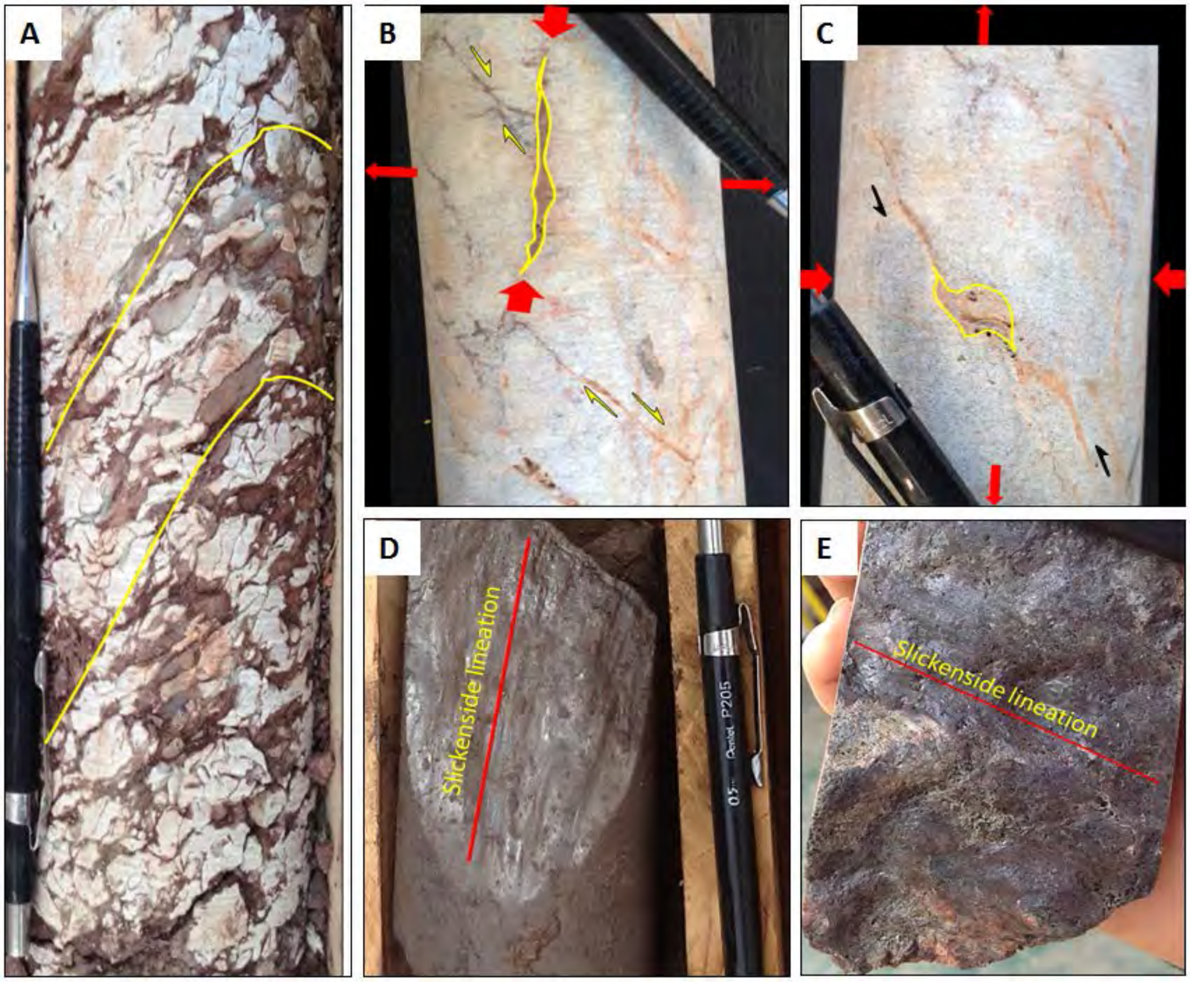
7.3 Elk Creek Carbonatite

The Elk Creek Carbonatite is an elliptical magmatic body with 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, 1997). It was first discovered by drilling in 1971 and tentatively identified as a carbonatite on the basis that it resembled rocks of the Fen District of Norway (Treves et al., 1972a and 1972b). The definitive confirmation of carbonatite was completed using Rare Earth Element (REE), P_20_5 and $^{87}Sr/^{86}Sr$ isotope analysis (Brookins et al., 1975). The Carbonatite has also been compared to the Iron Hill carbonatite stock in Gunnison County, Colorado on the basis of similar mineralogy (Xu, 1996).

The Carbonatite consists predominantly of dolomite, calcite and ankerite, with lesser chlorite, barite, phlogopite, pyrochlore, serpentine, fluorite, sulfides and quartz (Xu, 1996). It is, however, believed from stratigraphic reconstruction based on drill core observation in the area 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). However, 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 (Figures 7.3.1 and 7.3.2). Furthermore, both rock types appear to have been affected by at least one main brittle-ductile deformation event resulting in 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 (Figures 7.3.1 and 7.3.2). These faults could 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.

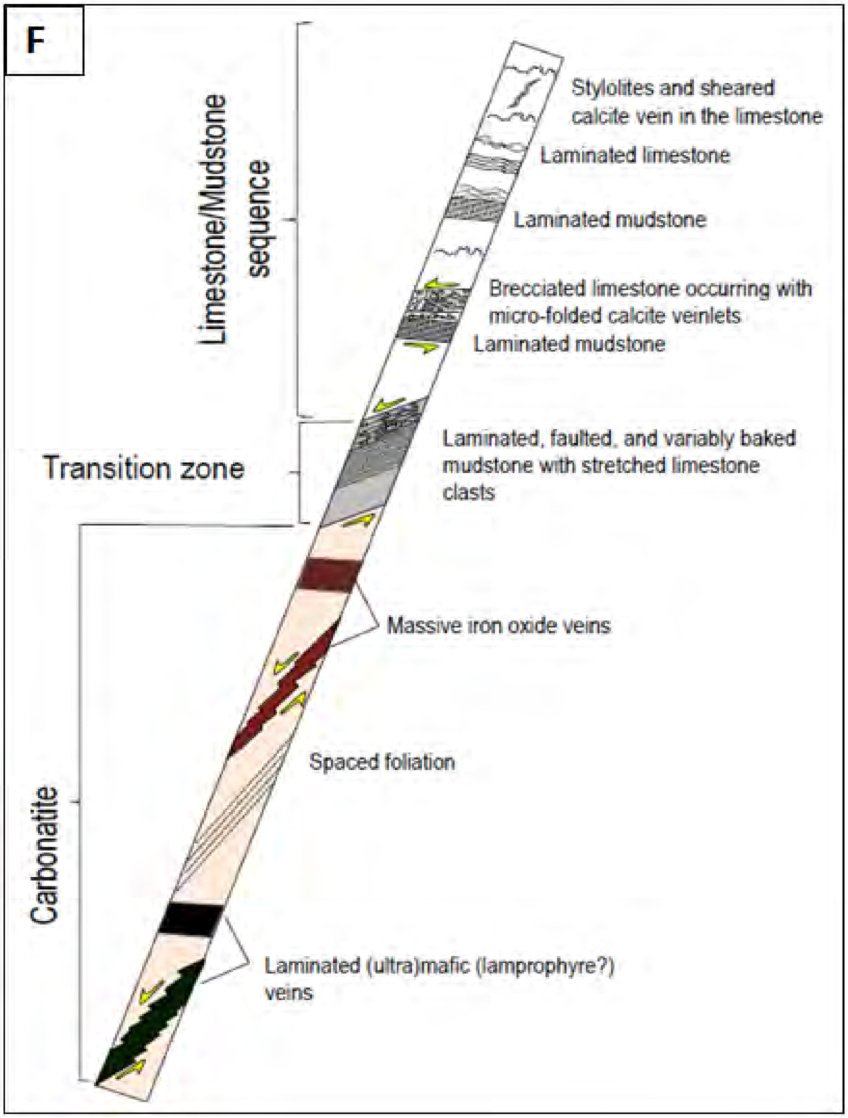
Microstructures presented in Figure 7.3.1 suggest the presence of extensional and strike-slip to oblique faults in the area as follows: (A) Spaced foliation and breccia in the contact zone between the Carbonatite and the Pennsylvanian sequence; Subvertical (B) and subhorizontal (C) tension veins and 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, 2014

Figure 7.3.1: Core Photographs Showing Microstructures

Figure 7.3.2 presents microstructures along a composite subvertical drill core suggesting that the Carbonatite is intrusive within the Pennsylvanian rock sequence.



Source: SRK, 2014
Illustration not to scale

Figure 7.3.2: Schematic of Drillhole Showing Typical Transition from Pennsylvanian Sediments to Carbonatite Units

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. However, 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. Whilst these radiometric dates provide a generalized time range for the Carbonatite intrusion, additional age dating is needed to establish a more precise date.

7.4 Carbonatite Lithological Units

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 7.4.1 (youngest at the top) represent the data captured during the original data capture in 2011. The information was compiled from the drill logs and the corresponding geology reports for each drillhole

Table 7.4.1: 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 drillholes by Xu (1996) identified two main phases within the area, a carbonate phase and a silicate phase. The study was based on drillholes 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; and
- Magnetite-dolomite carbonatite.

<u>Brecciated Carbonatite</u>

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

- Altered basalt;
- Altered lamprophyre; and
- Altered syenite.

In the 2014 drilling, the Dahrouge geologists have 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; and
- Magnetite-dolomite Carbonatite Breccia – mdolCarbBc.

SRK considers 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 (Table 7.5.1).

Table 7.5.1: 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

7.6 Structural Geology

On the basis of data provided to carry out this structural study, SRK concludes 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 SRK customized structural core logging data, and by undertaking a preliminary interpretation of the provided geophysics images.

SRK has used this data to model the fault pattern in 3-D 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 meter 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 occurs within the Elk Creek Carbonatite. The current known extents of the Carbonatite unit are approximately 950 m along strike, 300 m wide, and 750 m in dip extent, below the unconformity. For the purposes of this report, niobium, titanium and scandium are considered the main elements of interests, within additional background on REE mineralization included and discussed below.

In the Molycorp database, nearly every drillhole contains a separate geological report summarizing rock types, assay results and associated petrographic descriptions identifying niobium and/or REE minerals. Niobium is reported to be hosted in pyrochlore and REE mineralization is reported to occur as bästnasite, parisite, synchysite and monazite. SRK highlighted during the 2014 NI 43-101 Technical Report that the level of detail shown in the geological reports has not been transferred to the electronic database in completeness, this has been improved in the revised database with Dahrouge geologist 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 Mineralization

The deposit contains significant concentrations of niobium. Based on the metallurgical testwork 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.

7.7.2 Additional Elements of Economic Interest

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 resampling of selected historical core/pulps between 2011 and 2014.

As the metallurgical testwork advanced (discussed in Section 13 of this report) 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 between 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, but then a scatter of grades (on average considered higher grades 60 to 80 ppm, within the mdolCarb units.

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:

- Bästnasite ([Ce,La,Y]CO3F);
- Parisite (Ca[Ce,La]2[CO3]3F2);
- Synchysite (Ca(Ce,La)[F|CO3]2); and
- Monazite ([Ce,La]PO4).

A review of historic documents for drillhole EC-93, and part of Quantum's re-sampling program due to the high grade REE mineralization as noted in the Molycorp drill logs includes an excerpt as follows:.

"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 3.0 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 micrometer size range, aggregates of lanthanide minerals range from 0.23 to 8.0 mm. in maximum dimension. Monazite and bästnasite 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 7.7.3.1.

Table 7.7.3.1: 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 Rare Earth Oxides
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	

8 Deposit Type

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, dikes, 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); and
- 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 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 is covered by approximately 190 to 200 m of Pennsylvanian sedimentary rocks. No surface exploration has been completed with the 2014 exploration program focusing on infill drilling of the existing Mineral Resource using diamond drilling methods. The following section provides a summary of the 2012 NI 43-101 Technical Report for the exploration work completed by the Company since acquiring the Project in 2010.

9.1 Quantum, 2010-2011

9.1.1 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 historic reports.

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

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

9.1.2 Quantum Re-sampling Program, 2010

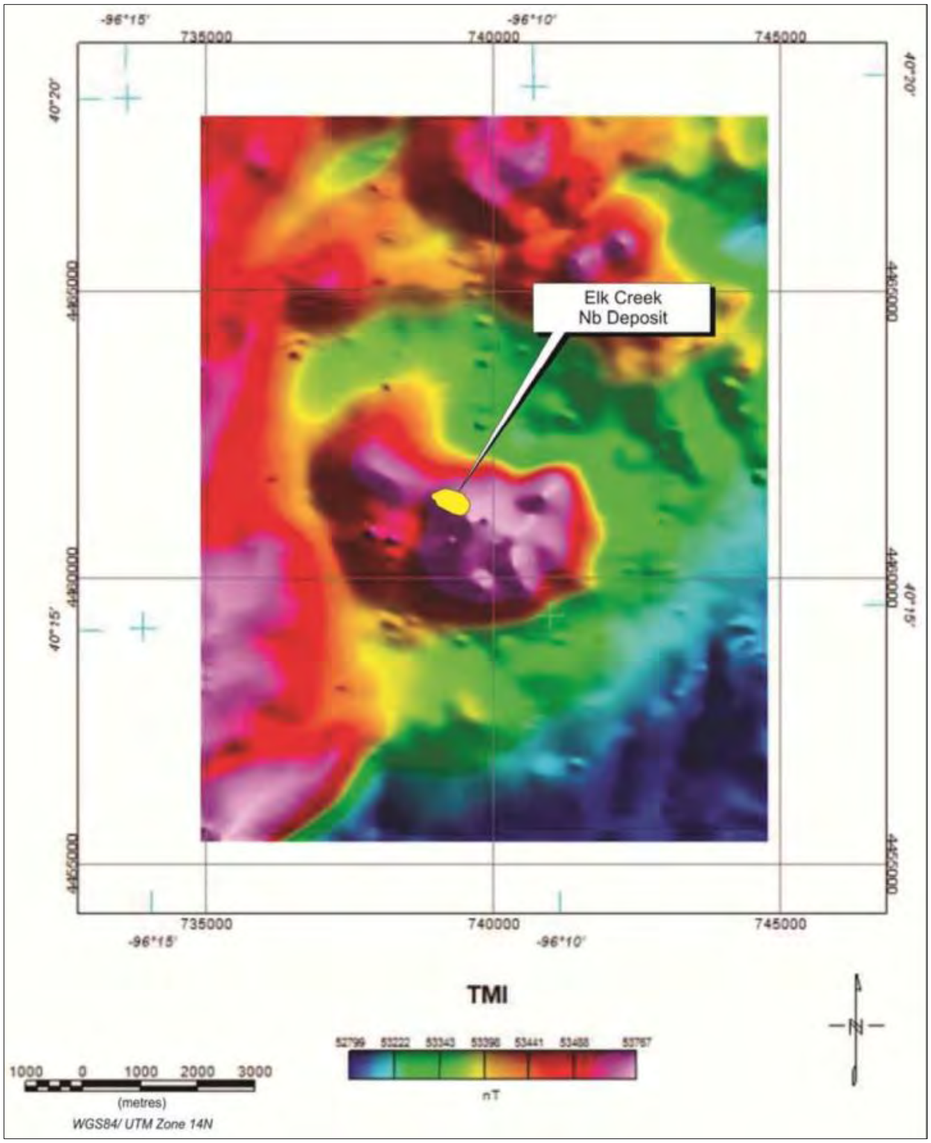
Commencing in November 2010, the Company contracted Dahrouge to undertake a re-sampling of the historic drill core pulps as part of a QA/QC program to ascertain the reliability of the historic drill core assay results and to obtain more detailed analysis of the REE content of the samples. The samples 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.

9.2 Quantum, 2011-2012

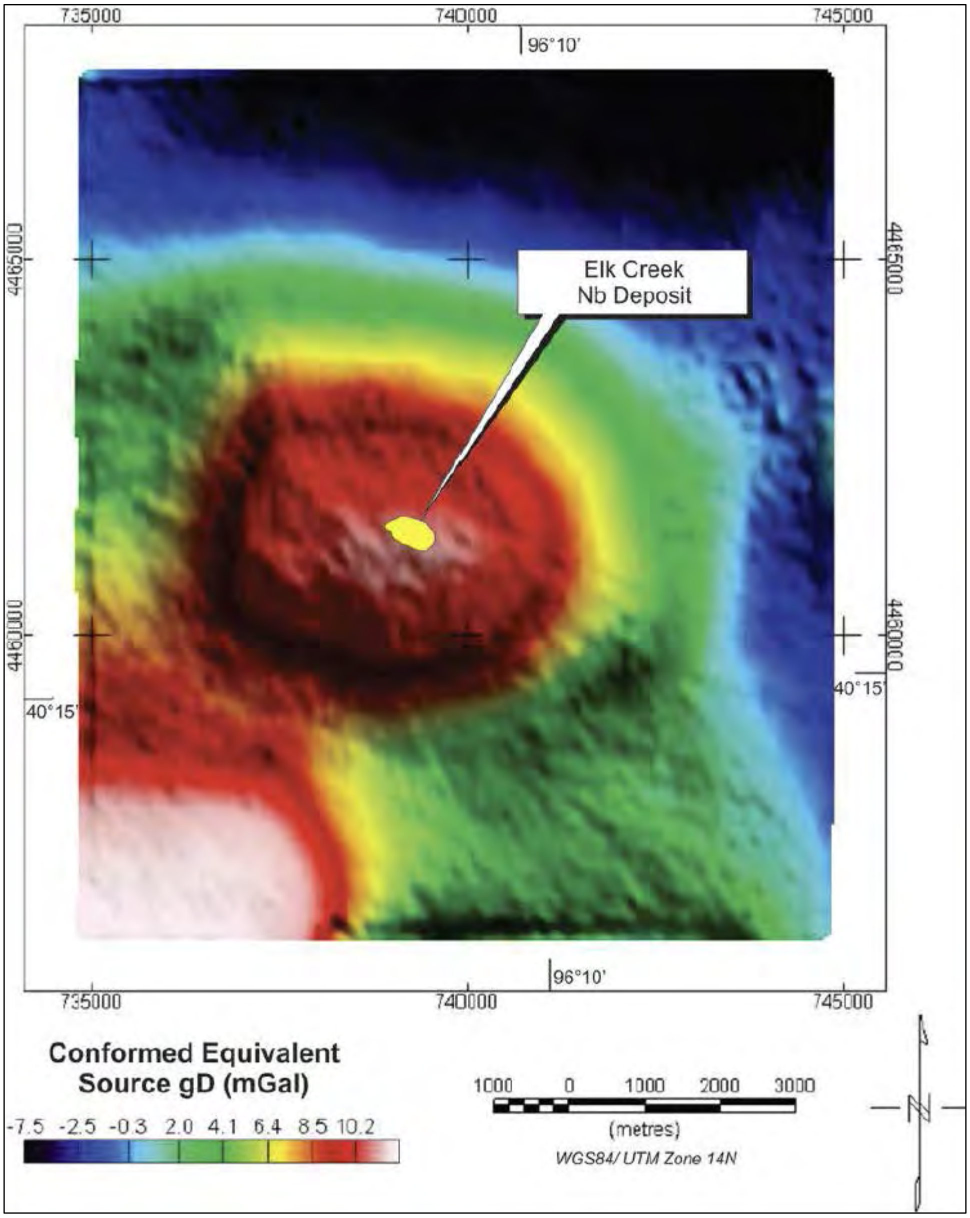
9.2.1 Airborne Gravity and Magnetic Geophysical Survey, 2011

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. The results of the gravity and magnetic geophysical surveys are shown in Figures 9.2.1.1 and 9.2.1.2 below.



Source: Tetra Tech, 2012

Figure 9.2.1.1: Airborne Total Magnetic Map



Source: Tetra Tech, 2012

Figure 9.2.1.2: Gravity Gradiometer Map

The survey area was centered on the Project and covered a total area of approximately 110 km^2 (approximately 10 by 11 km) around the deposit. A total of 1,176 line km were flown. Flight lines were oriented 000/180° azimuth on a nominal line spacing of 100 m. Five tie lines were flown, oriented at 090°/270° azimuth, spaced 2,750 m apart. All flight lines were flown at a nominal clearance of 100 m (Fugro, 2011).

9.3 Significant Results and Interpretation

It has been noted that the 2011-2012 geophysical surveys closely match the results of the CSD and UNL geophysical surveys in the early 1970's confirming the original gravity anomaly.

Subsequently, in October 2011, Colorado-based Condor Consulting Inc. (Condor) was retained by the Company to process and analyze the FALCON™ gravity and magnetic geophysical survey data. Condor noted coincident gravity and magnetic anomalies traversing 1,200 m to the east from the known Project. Several anomalies of higher relative density and magnetization have also been identified outside of the drilled prospect area (Condor, 2011).

No further geophysical studies targeting the Elk Creek Carbonatite have been completed as part of the current phase of exploration. SRK considers the exploration programs completed at the Elk Creek deposit to date to be appropriate for the style of mineralization.

10 Drilling

10.1 Type and Extent

Mineral Resource definition drilling at the Project was conducted in three phases. The first was during the 1970's and 1980's 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 (Figure 10.1.1). All drilling has been completed using a combination of Tricone, Reverse Circulation (RC) or Diamond Drillling (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 labor has been used by drilling contractors when preparing the drillhole 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 drillholes used RC drilling within the Pennsylvanian sediments, to increase the efficiency in drilling through the cover material, within areas of strong geological confidence.

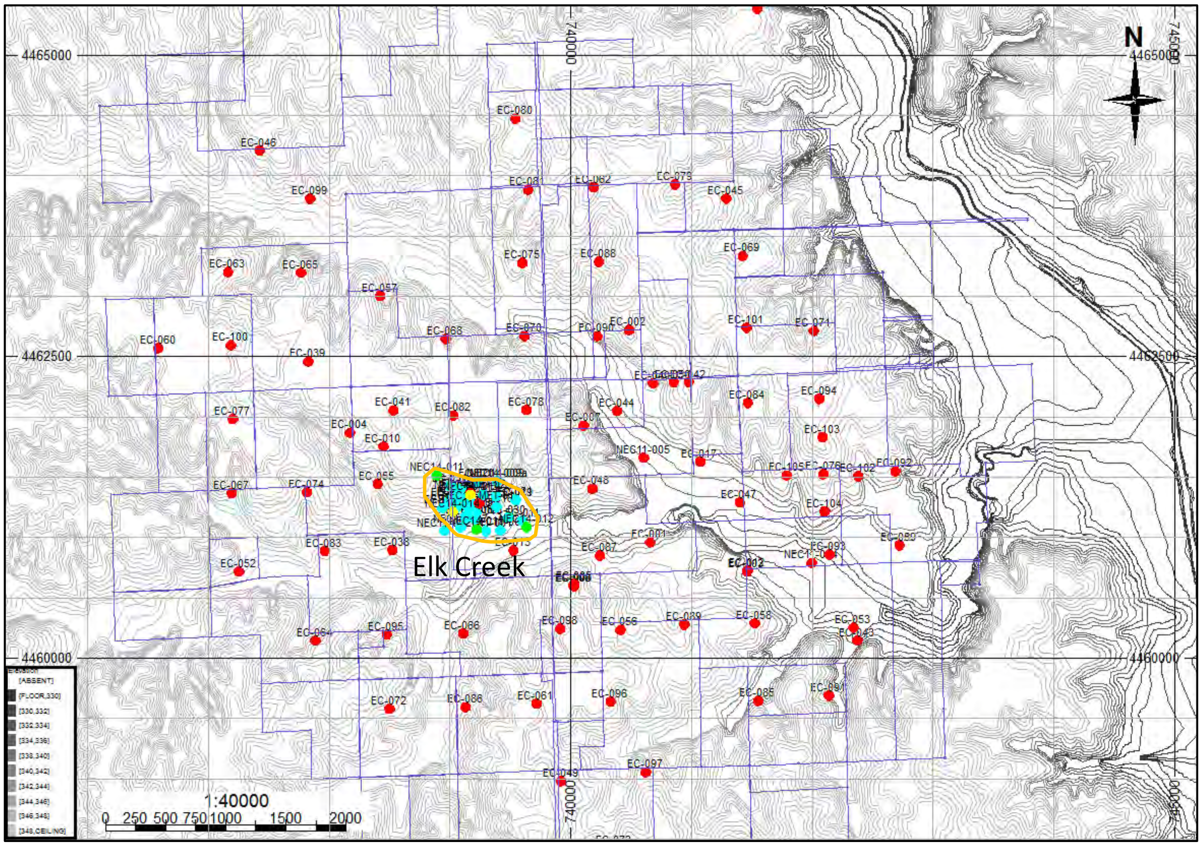
The following sections provide a brief summary of the resource drilling completed by Molycorp, Quantum and NioCorp (as shown in Table 10.1.1).

Table 10.1.1: 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: SRK, 2014

During 2015 five holes, for a total length 3,353.1 m, of additional drilling been drilled since the completion of the April 28, 2015 Mineral Resource Estimate. This drilling has been for the purpose of Hydrogeological and Geotechnical studies. The drilling has been completed by Idea Drilling and Envirotech Drilling LLC. No sampling of these holes have been completed to date and therefore they have not been considered in the Mineral Resource, and are excluded from Table 10.1.1 above. Inclusive of these holes the total drilling on the Project is 134 holes for 68,334.1 m.



Source: SRK, 2015

Figure 10.1.1: Drillhole Location Map of All Drilling vs. the Topographic Contour

Not all of the drillholes within the Project were used in the Mineral Resource Estimation that accompanies this report, as many do not intersect the Nb_2O_5 anomaly and are located at a significant distance away from the deposit. Of the 129 total drillholes within the Project, 48 drillholes are within the Elk Creek deposit area. Table 10.1.2 summarizes the drillholes used for the Mineral Resource Estimation.

Table 10.1.2: 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: SRK, 2015

10.2 Molycorp, 1973-1986

Between 1973 and 1986, Molycorp completed a regional scale drill program over approximately 7 by 7 km gravity anomaly that includes the Elk Creek deposit. The total program consisted of 106 drillholes 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 by 610 m (2,000 by 2,000 ft) with some closely spaced holes in selected areas within the gravity anomaly. A more detailed description of this program may be found in Section 6.2.4 in this report.

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

The Molycorp drillhole locations centered over the Elk Creek deposit are presented in Figure 10.1.1 (shown in blue).

10.3 Quantum, 2011

In April 2011, Quantum conducted a preliminary drill program (three holes) on the Elk Creek deposit and two REE exploration targets (two holes), which 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 in order 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 drillholes, totaling 3,420 m of NQ size diameter core. Inclusive of this total, three drillholes, totaling 2,318 m were drilled into the known Elk Creek deposit. The summary of the 2011 drill program is listed in Table 10.3.1.

Table 10.3.1: Summary of 2011 Drill Program

Drillhole	UTM Easting	UTM Northing	Elevation (m)	Depth (m)	Bearing (º)	Dip (º)
NEC 11-001	739299	4461052	341.49	900.38	28.1	-72.0
NEC 11-002	738955	4461058	343.88	908.61	33.5	-61.0
NEC 11-003	739417	4461060	340.79	508.71	34.3	-55.9
Outside Elk Creek Deposit; REE Exploration Targets						
NEC 11-004	741997	4460790	333.65	465.73	80.7	-55.6
NEC 11-005	740604	4461660	337.48	636.42	95.7	-56.0
Total				**3,419.85**		

Source: Tetra Tech, 2012

Hole NEC11-001 targeted the eastern portion of the deposit below historic drillhole EC-11 and between vertical holes EC-27 and EC-30. Hole NEC11-002 was drilled into the northwestern portion of the deposit. Hole NEC11-003 was drilled into the southeastern portion of the deposit. Drillholes NEC11-004 and 005 drilled into regional REE targets and are not subject to this report and have been excluded from the Mineral Resource Estimate.

The Quantum 2011 drillhole locations centered over the Elk Creek deposit are presented in Figure 10.1.1 (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 drillholes confirmed the high-grade potential of the niobium mineralization, as indicated by previous drilling completed by Molycorp.

10.4 NioCorp 2014 Program

No new drilling for the purpose of Mineral Resource definition has been completed since February 20, 2015 Mineral Resource Estimate.

NioCorp commenced drilling on the Project using a three phased program with the aim of increasing the confidence in the 2012 Mineral Resource Estimate from Inferred to Indicated. The three phased program was originally based on 14 drillholes for approximately 12,150 m (announced in a press release on April 29 2014), but was subsequently expanded during the program to 18 drillholes for approximately 15,482 m. Three of the 18 drillholes were drilled for the purpose of metallurgical characterization and process development studies. Two of these drillholes, NEC14-MET-01 and NEC14-MET-02 were not assayed, while NEC14-MET-03 was quarter cored with one quarter being assayed and the remainder used for metallurgical testwork. The drilling has been orientated to intersect the geological model from the southwest and northeast (perpendicular to the strike), with the exception of NEC14-011 and NEC14-012, which were oriented southeast and northwest, respectively.

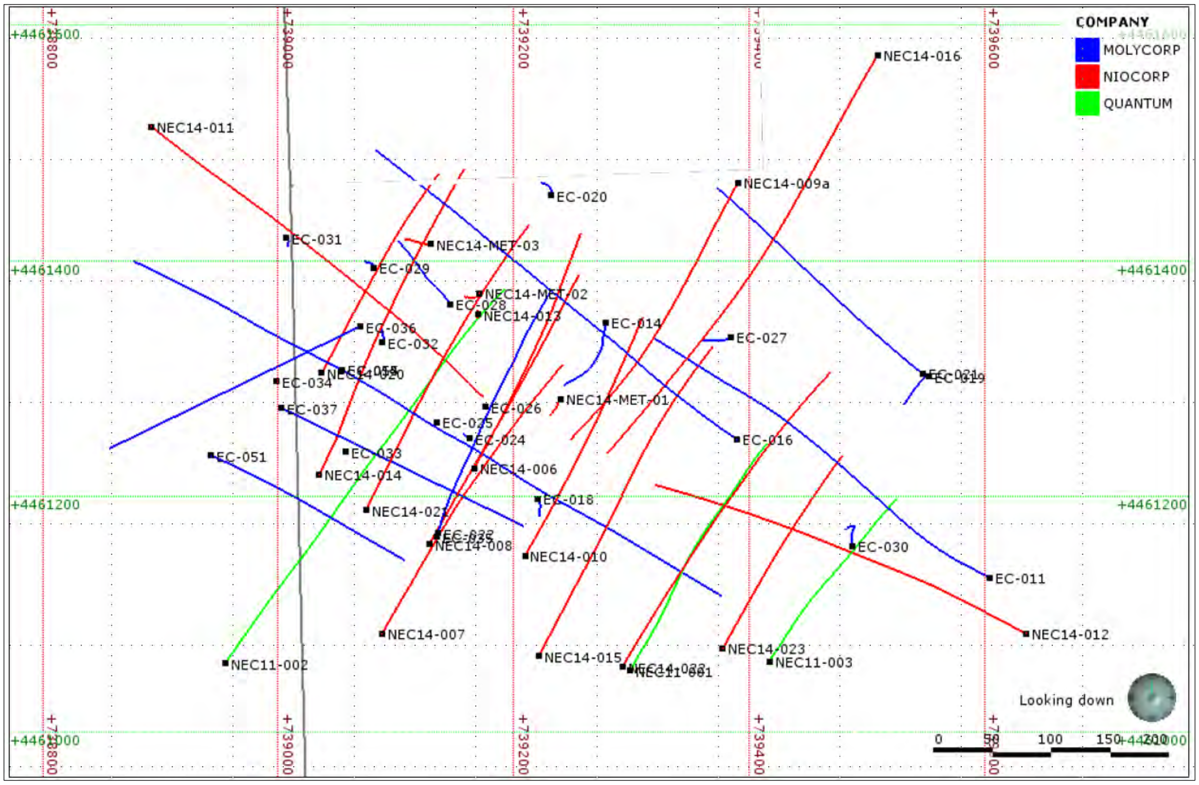
The NioCorp 2014 drillhole locations (shown in Table 10.4.1) are presented in Figure 10.4.1 (shown in red). The grey lines in Figure 10.4.1 are the mineral lease boundaries.

Table 10.4.1: Summary of NioCorp 2014 Phase 1 Drill Program

Drillhole	UTM Easting	UTM Northing	Elevation (m)	Depth (m)	Bearing	Dip	Comments
NEC14-006	739166.2	4461224.0	352.0	772.7	30	-70	
NEC14-007	739088.2	4461083.5	344.8	907.4	30	-70	
NEC14-008	739128.1	4461159.4	351.2	886.1	30	-70	
NEC14-009	739390.2	4461466.2	349.3	751.3	210	-70	
NEC14-009a	739390.2	4461466.2	349.3	897.0	210	-70	Wedge from 485.51 m
NEC14-010	739209.5	4461149.8	347.8	796.1	30	-70	
NEC14-011	738892.5	4461513.6	359.7	900.4	125	-65	
NEC14-012	739635.1	4461083.4	339.9	843.2	300	-65	
NEC14-013	739169.3	4461354.3	355.2	880.3	360	-90	
NEC14-014	739034.8	4461218.6	346.1	901.0	30	-75	
NEC14-015	739221.1	4461064.7	342.4	827.8	30	-70	
NEC14-016	739509.1	4461574.7	354.7	913.8	210	-60	
NEC14-020	739037.1	4461305.0	348.4	587.7	30	-70	
NEC14-021	739074.3	4461188.5	347.1	865.0	30	-70	
NEC14-022	739292.2	4461055.3	340.3	950.4	30	-69	
NEC14-023	739377.6	4461071.0	341.5	615.1	30	-70	
NEC14-MET-01	739240.4	4461282.7	352.8	894.7	360	-90	
NEC14-MET-02	739171.1	4461372.4	355.8	865.0	360	-90	
NEC14-MET-03	739129.9	4461414.5	355.4	913.3	360	-90	
Subtotal				**15,968.3***			

Source: SRK, 2015

* Does not equal total drilled meters due to NEC14-009a beginning at a depth of 485.51 m, total meters for 2014 drilling program is 18 holes for 15,482.8 m.



Source: SRK, 2015

Figure 10.4.1: Elk Creek Drillhole 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 SRK. 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 drillhole 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) was 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 drillholes to depths ranging from 18 to 37 m. The Pennsylvanian limestones and mudstones overlying the target carbonatite were drilled PQ-sized core and HQ-sized core for drillholes NEC14-020 to NEC14-023. The Carbonatite was drilled with HQ-sized core, with the exception of the three metallurgical holes (NEC14-MET-01, NEC14-MET-02 and NEC-14MET-03), which were drilled completely using PQ-sized core. Core size reduction took place just beneath the Pennsylvanian-carbonatite contact at depths ranging from 206 to 238 m. The core drilling rigs operated 24 hours/day and 7 days/week, with typical progress of 40 m/day.

During the drilling operation, the core is retrieved from the core barrel and laid sequentially into wooden core boxes by the drilling contractor. Interval blocks are then placed at all run breaks. Once the box is full, the ends and top of the box are labeled with drillhole identification and the sequential box number. Upon completing a box, it is stacked on a pallet or on a truck bed at the drill rig. At the end of each drilling shift, the boxes of core are transported by the drilling contractor in a pickup truck to the NioCorp field office. At this point, the core is in the custody of Dahrouge Geological Consulting Ltd. (Dahrouge). Eight of the 2014 drillholes had piezometers installed in them after drilling was complete. For these drillholes, 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 name plate attached to the casing. Surface completion for the drillholes that did not have piezometers installed consisted of a steel marker post and attached name plate. All name plates include drillhole number, total depth and orientation. Abandonment of the drillholes consisted of cementing from total depth to surface in the non-piezometer drillholes and from total depth to the bottom of the piezometers in the other drillholes with piezometer installations.

10.5.1 Collar Surveys

All hole collars were initially surveyed prior to drilling using a handheld GPS. On completion of the hole an external contractor ESP INC. (Engineering/Surveying/Planning), based in Lincoln, Nebraska, has been used to provide a detailed survey of the collar location using a using a Sokkia GS2700 IS GPS, which has 10 mm horizontal and 20 mm vertical accuracy. Data has been provided to SRK in digital format in UTM (NADS83 Zone 14) grid coordinates.

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

10.5.2 Downhole Surveys

Initial collar surveys of dip and azimuth have been taken using compass measurements for all holes (RC and DD). Downhole surveying has been 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 drillholes 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 potential tool movement, that particular 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 drillhole 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 is 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 are synchronized electronically with a receiver at surface, and recordings are collected every 30 seconds, after the tool has had a chance to equilibrate. 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 are completed through the drill rods and location data points are collected every 6.1 m (~20 ft).

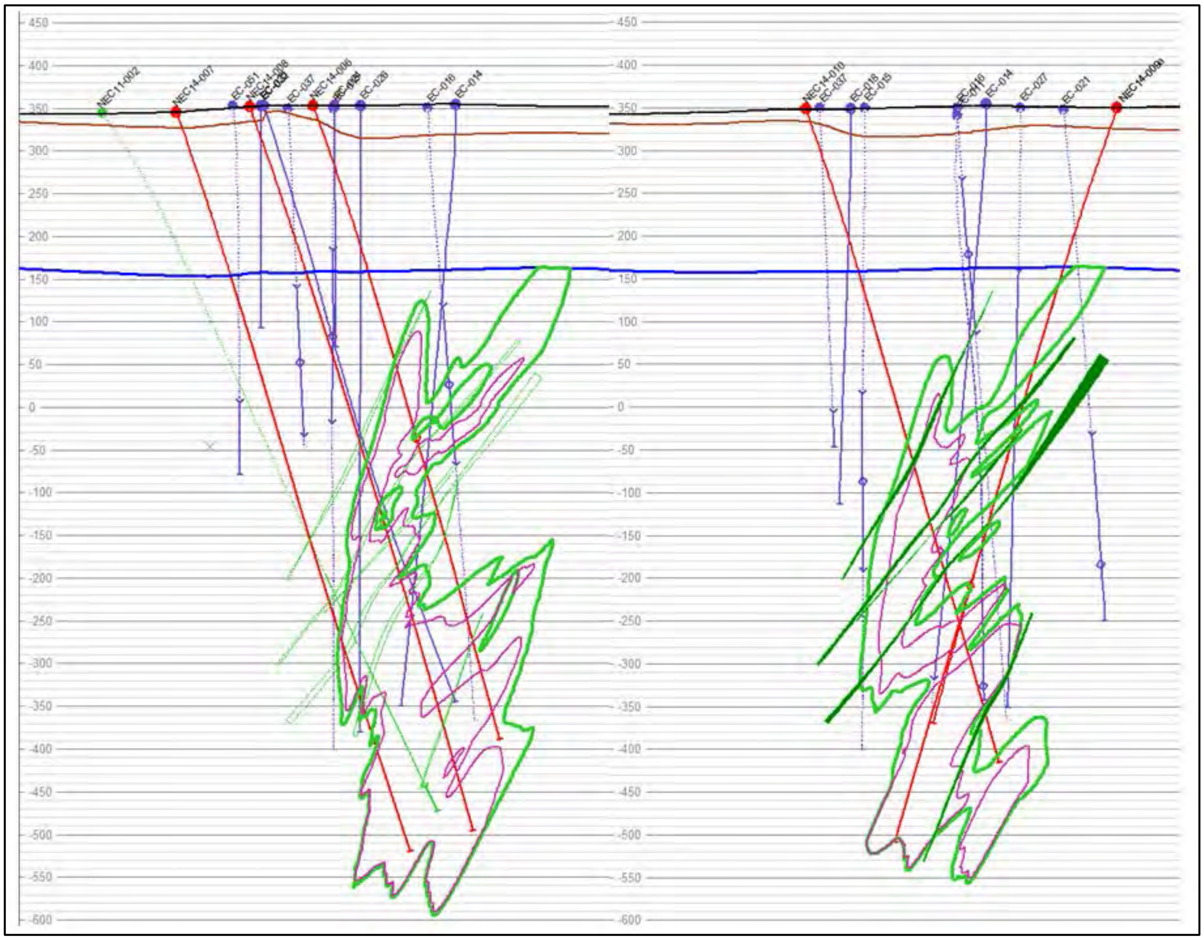
SRK considers the methods used for the downhole surveying during the 2011 and 2014 campaigns to be in line with industry best practice. Given the long hole lengths of over 700 m, the Company has used suitable techniques to provide a continuous (ranging from 3 to 6 m), measure of the drillhole trace from 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 hole location of the Molycorp drilling is considered lower due to their historic nature and the wider measurement spacing. Overall SRK consider the level of confidence in the downhole surveys to be sufficient for the declaration of an Indicated level of Mineral Resource.

10.6 Interpretation and Relevant Results

The drilling has been 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 historic drillholes within the deposit and Mineral Resource area were not conducted on a systematic grid and drill spacing varies from 25 to 225 m. The major drilling direction used by NioCorp has been towards the northeast (Figure 10.6.1). 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 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 (Figure 10.6.1)



Source: SRK, 2014

Figure 10.6.1: Typical Cross-sections looking Northwest showing NioCorp Holes Drilled to the Northeast and Southwest, Confirming the Width of the Deposit

SRK is of the opinion that the drilling operations were conducted by professionals using industry best practice that the core was handled, logged and sampled in an acceptable manner by professional geologists, and the results are suitable for support of a NI 43-101 compliant resource estimation.

11 Sample Preparation, Analysis 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 Molycorp, 1973-1986

Detailed descriptions of Molycorp's sample procedures, analyses and security have not been documented and reviewed by SRK. However, given the detailed nature of the historic drill logs and reports for the individual drillholes, and Molycorp's position as a leader in the rare earth industry at the time, it is considered likely that Molycorp applied the same standards to their sampling procedures.

A review of previous Technical Reports details by Ms. Beverly Beethe, a sampling technician for Molycorp, recalled the following procedures:

- The drill core was photographed;
- The drill core was split with a hydraulic core splitter;
- The core was crushed on-site, before sending samples to the lab (the crusher is no longer on site); and
- The core crusher was cleaned between samples by using limestone blank material.

Complete details of the sampling procedures were unclear as to whether the procedures had changed over the period of the drill programs. Photographs of the core were not included with Molycorp's available historic records.

Molycorp built two well-insulated, steel buildings, located on the property of Ms. Elda Beethe (Lease Agreement Beethe_008), within 100 m of the known deposit. The buildings were ceded to Ms. Beethe when Molycorp abandoned the Project. The following italicized text is excerpted from McCallum and Cathro 2010, which provides the best detail on the known and assumed methods used during Molycorp drilling:

"It is also uncertain what methods were used to crush, pulverize, blend and split any of the original 10' intervals and composite intervals. In order to confirm some of the analyses of Molycorp's internal laboratory, some of the 10' intervals were split and combined into either 50' or 100' composites and sent to commercial laboratories for independent assays. The procedure of creating the larger composites is unknown at this time."

Drill core samples collected were 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 some 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 hole EC-11.

The drill core, crushed (coarse reject), and pulverized material are currently being 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. The core was stored at two other storage facilities on UNL property, prior to its current location. Prior to the acquisition of the core by UNL in the early 2000's, the core was stored in the steel sheds on the property of Elda Beethe.

SRK completed a site visit to the Mead Core facility by Mr. Cody Bramwell on April 15, 2014. An inventory of the core was spot checked against a 2011 core inventory list originally compiled by Dahrouge and no discrepancies were found. The core investigation concentrated on 27 drillholes within the resource area. Table 11.1.1, is an inventory of core at the Mead facility filtered to include only the 26 drillholes within the resource area.

Table 11.1.1: Core Inventory of Drillholes within the Resource Area at the Mead Facility

Hole ID	Core Box Intervals		Depth		Missing Boxes
	Box # From	Box # To	From (m)	To (m)	
EC-11	7	41	207.6	310.3	1 - 6
EC-11A	1	180	233.2	769.6	
EC-14	13	188	43.9	707.1	1 - 12
EC-15	30	244	215.8	839.7	1 - 29
EC-16	7	218	214.6	817.5	1 - 6
EC-18	9	102	189.6	462.4	1 - 8
EC-19	9	178	194.2	664.2	1 - 8
EC-20	6	189	190.5	739.0	1 - 5
EC-21	6	156	210.3	644.3	1 - 5
EC-22	12	193	207.0	733.3	1 - 11
EC-24	11	39	191.7	281.9	1 - 10
EC-25	9	47	192.9	304.5	1 - 8
EC-26	14	191	199.3	733.0	1 - 13
EC-27 & 27A	13	186	202.4	702.0	1 - 12
EC-28	16	209	193.5	769.6	1 - 15
EC-29	14	182	196.9	726.0	1 - 13
EC-30	9	201	182.9	757.1	1 - 8
EC-31	16	117	203.3	512.4	1 - 15
EC-32	13	165	196.0	681.2	1 - 12
EC-33	14	83	199.0	405.4	1 - 13
EC-34	16	69	202.4	362.7	1 - 15
EC-35	6	27	192.0	260.0	1 - 5
EC-36	8	95	214.0	474.0	1 - 7
EC-37	14	87	239.9	457.5	1 - 13
EC-51	1	89	220.1	470.6	
EC-54	7	97	213.7	464.5	1 - 6

Source: SRK, 2014

An investigation of the drillholes within the resource area at the Mead facility led to the following conclusions:

- Core boxes were generally in good condition and labeled well;

- Not all of the historical core made it to the Mead facility with most drillholes missing between six and 26 of the first core boxes (Table 11.1.1);
- No drill core of the Pennsylvanian strata exists, hence no information on the strata was gathered;
- Drill core is typically NQ and some noted as being BQ;
- All drill core had been hydraulically split, removing the option of sampling for geotechnical purposes;
- Accurate geotechnical and hydrogeological parameters were difficult to estimate due to the core appearing to have been hydraulically split; and
- Identifying mineralization was difficult due to the fine grained nature of the rock and a lack of differences between mineralized and non-mineralized rock.

In addition to the drill core, there also exists an unknown inventory of sample pulps and rejects at the Mead facility.

11.2 Quantum Re-Sampling, 2010

The 2010/2011 re-sampling program utilized a total of 1,861 samples of pulverized material from the Molycorp drillholes that were prepared by the analytical division of Molycorp. Samples were derived from 1.52 m (5 ft) or 3.05 m (10 ft) intervals of split NQ or HQ diameter size core. The samples were selected based on the geological interpretation at the time and in areas of elevated Nb_2O_5 values. Not all samples have been selected continuously within each drillhole. SRK confirmed evidence of the resampling during the site inspection to the Mead Core facility.

A rigorous QA/QC protocol was used, and included the routine insertion of field duplicates, laboratory pulp duplicates, blanks and niobium certified reference standards. Samples were transported to the ALS Chemex (ALS) facility in Reno, Nevada, and prepared for analysis at the ALS testing facility in North Vancouver, B.C., using method XE-XRF10, whereby samples are prepared by pulverizing to 90% passing -70 µm, then decomposed utilizing a lithium borate flux, and analysis by XRF. A portion of niobium results were checked with Hazen of Golden, Colorado (Quantum news release February 22, 2011).

11.3 Quantum Drilling Program, 2011

For the 2011 sampling program, a rigorous quality assurance and quality control protocol was established. It involved the routine insertion of field duplicates, laboratory pulp duplicates, blanks, and certified reference standards. All samples were shipped to, and analyzed by Activation Laboratories (Actlabs) of Ancaster, ON. An eight-major oxides, rare earths, and trace element package was selected and samples were analyzed via fusion inductively coupled plasma (ICP) and inductively coupled plasma-mass spectrometry (ICP-MS) in addition to niobium by XRF, and fluorine by method 4F-F (news release, September 21, 2011).

11.4 NioCorp Drilling Program, 2014

Different drilling techniques, such as DDH, RC and tricone drilling, have been employed to drill through the overlying geological rock units (limestone & mudstone), but all carbonatite intervals have been diamond cored. All drilling contractors at Elk Creek utilized DDH utilized conventional wireline drilling techniques. Two drilling diameters have been used during the program with the upper portions of each cored hole drilled using PQ diameter (85 mm) for geotechnical testing and HQ diameter (63.5 mm) through the Carbonatite, with the exception of NEC14-MET-01, NEC14-MET-02 and NEC14-MET-03, which were drilled entirely with PQ. Geological and geotechnical logging is completed prior to mark-up and splitting of the core and completed by onsite geologists with the number of geologists used limited to ensure consistency in the logging codes used.

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

11.4.1 Core Recovery

Core recovery and RQD were generally good for most drill core. Core recovery has been recorded in the data base and is measured in the field at the drilling rig by the geologist. The borehole name is noted and the drilling interval, this is compared to the actual core recovered to back calculate the recovery. The recovery information is then loaded into the sample database.

SRK has reviewed the drill core recovery results and comments that while the recoveries per hole vary from a low as 2%, the typical minimum recovery is in the order of 47% to 100%, with the average recovery per hole ranging from 93% to 99%.

Drill core was digitally photographed under natural outdoor or fluorescent indoor lighting prior to core cutting. All digital photos are of high resolution and stored in a digital archive format. The geological logging included observations of color, lithology, texture, structure, mineralization, and alteration. All geological information is collected at sample interval scale and recorded in a digital logging program that has been custom formatted for carbonatite deposits. Detailed geological core logging of the Carbonatite intervals, alteration zones and its relationship to other intrusions allows sampling to be restricted by unique geological boundaries.

11.4.2 Sample Preparation for Analysis

Trained staff was involved at all stages of the sampling, sample packaging and sample transportation process. Day to day logging tasks were split between Dahrouge and SRK, whereby Dahrouge completed all geological and sampling related tasks, while SRK focused on geotechnical logging requirements. During the diamond drilling program (including the RC pre-collar drilling of RC/DD holes), staff members were based full time at the drill Project site to supervise the drilling and data collection including geological and geotechnical properties. Geological sampling was completed by geologists (Dahrouge), under the supervision of qualified professional geologists. Between four and six trained geologists acted as samplers.

Core sampling method and approach has been consistent through the 2011 and 2014 drill programs. Core was boxed on site and delivered each day to a core facility on the Project site where the core was logged and split. For the 2014 diamond drilling program, up to three coring drill rigs were monitored by two qualified professional geologists, one drill supervisor and an experienced geological team. Drill core was boxed and transported from each drill rig to the core processing facility (distances up to 800 m), at the end of each 12 hour shift. Core logging involved detailed geotechnical and geological information. All key geological features have been logged comprehensively. A Project database which contains the relevant rock codes and lithology descriptions has been created. A total of 22 detailed rock codes have been used during the logging, which is reduced to 10 codes under a simplified logging code defined as "MAJOR" in the database (Table 11.4.2.1). DDH core was sampled and assayed at predominantly 1 m intervals.

Table 11.4.2.1: Summary of Major Rock Codes Used by Dahrouge Geologist

Major	Description
Casing	Drillhole 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 drill core within each core box was marked up and then 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. Drillhole 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 site.

A summary of the sampling procedure used to collect core samples at the Project is as follows:

- The entire carbonatite intersection was sampled, including the geologically logged low-grade niobium carbonatite intervals of the footwall or hangingwall, for all holes with the exception of NEC14-020 to NEC14-023 where approximately 10 m of the hangingwall was sampled;
- Sample intervals, generally 1 m in length, were marked on the core and recorded in the geological database (Fusion Database);
- 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 prior to bagging and cutting equipment was regularly cleaned;
- Sampled intervals were placed in durable barcoded sample bags that were clearly labelled and contain back up sample tags within each bag;

- Sample bags containing original core sections and field inserted control samples were barcode scanned and secured in 5 gallon plastic shipping pails;
- Detailed shipping logs and preparation requests were sent in hard copy and digitally 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; and
- The unsampled half of the core is stored in labelled wooden core boxes at the Project site for reference or further sampling.

Core samples and the core library are securely stored at the Project facility work area. This material is stored inside locked metal buildings when the Project is not operating.

11.4.3 Security Measures

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

11.4.4 Sample Analysis

The 2011 and 2014 sawn core samples were shipped to Activation Laboratories Ltd. (Actlabs) 1336 Sandhill drive, Ancaster, Ontario Canada. Actlabs is the primary laboratory for sample preparation and for 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/IEC 17025:2005) accreditation from Standards Council of Canada and maintains current certification issued March 5, 2014 through February 27, 2018. Actlabs maintains ISO 17025 standards, which is obtained through experienced peer audits that ensure they conform to recognized analytical standards and that the accredited method validation verifies a number of analytical variables designed to ensures that data obtained from these methods are defensible. Actlabs maintain a custom Laboratory Information Management System (LIMS) system to provide the traceability necessary for today's stringent reporting requirements.

The 2014 sampling program employed SGS as an external check laboratory. SGS is an integrated geochemistry, mineralogy and metallurgy laboratory in Lakefield, ON, which has extensive experience with Nb_20_5 and REE analysis for both exploration and metallurgy projects. SGS Lakefield is ISO17025 accredited for the analysis methods used on this Project (GO_XRF76V & GE_ICP90A).

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;
- Collect as received sample weight (kg);
- Drying of the whole sample at 60°C for 12 hours, in a customized high air flow drying room;
- Collect 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 option of Jones Riffle split) coarse crushed sample and extract a 250 g sample;
- Pulverized 250 g sample using ESSA pulverizers with ring and puck bowls to 95% -200 mesh (75 μm), with quartz cleaner between each sample;
- Laboratory internal coarse-reject duplicates (1 in 50) and Pulp duplicates (1 in 30) are also routinely prepared; and
- Quality of the rejects and pulps are routinely monitored to ensure proper preparation procedures are performed.

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 and/or shipped to the external check laboratory, SGS (Lakefield), for analysis.

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. Whole Rock analysis and 43 Major Elements 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 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. Prior to analysis sample is prepped using a combined fusion with lithium metaborate and lithium tetraborate in induction furnace. Fluoride analysis was completed for 2014 drillholes, NEC14-006, NEC14-007, and NEC14-008.

All QC data are registered in the LIMS system and Assay results have been returned to NioCorp and the overseeing professional geologists in electronic format and loaded into the sample database with the batch number and date of assay recorded after review for QA/QC.

External pulp check samples were submitted to SGS (Lakefield) Labs, as a third party analytical result confirmation. Pulp samples and their control samples 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 prior to analysis using MISC80 package prior to analysis. 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 13 major Whole Rock oxides, following a borate fusion as defined under SGS package "GO XRF76V - ORE GRADE" (Table 11.4.4.1). Scandium analysis has been completed at SGS laboratory using GE_ICP90A package which has a detection limit of 5 ppm.

Table 11.4.4.1: 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: SRK, 2014

11.5 Quality Assurance/Quality Control Procedures

The Company has integrated a series of routine QA/QC procedures throughout the sampling and analytical analysis for both the 2011 and 2014 drilling programs, to ensure a high level of quality is maintained throughout the process. SRK has not reviewed any QA/QC data for the Molycorp drilling program, as no information has been detailed in the database. Definition of quality of the historical assays has been based on resampling/verification work completed by Dahrouge during 2010 – 2011. A total of 1,861 samples (approximately 44% of the original assays) were selected for reanalysis 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.

The following control measures were used to monitor both the precision and accuracy of sampling, sub-sampling, preparation and assaying. For the 2011 and 2014 sampling the QA/QC consisted of the insertion of duplicate samples taken from various stages of the process, insertion of known control samples (Standards Reference Material (SRM) and Blanks), plus an external check at a SGS laboratory. A summary of the type of samples, source and level of insertion is included in Table 11.5.1 and Table 11.5.2. Note percentages are reported as proportion of samples vs. the original submissions, unless otherwise noted.

Table 11.5.1: Summary of Designed Level of Insertion of Quality Control Submissions

Sample Type	Sample Sub-type	Type	Insertion Rate
Duplicates	Field quartered core	¼ HQ core	5.0%
	Coarse-Rejects	Reject split	3.0%
	Pulp	Pulp split	5.0%
Standard Reference Material SRM's	SX18-01 (Dilinger Hutte Lab)	Nb SRM	6.0%
	SX18-02 (Dilinger Hutte Lab)	Nb SRM	
	SX18-04 (Dilinger Hutte Lab)	Nb SRM	
	*SX18-05 (Dilinger Hutte Lab)	Nb SRM	
Blanks	Field Quartz Blanks	Optical Quartz	5.0%
External Lab Checks “Umpire Lab”	Pulp Splits		5.0%
	Nb & REE SRMs	Nb SRM	(5% of splits)
	Field Quartz Blanks	Optical Quartz	(5% of splits)

Source: Dahrouge, 2014

Table 11.5.2: Summary of Actual Submissions per Sample Type Within the 2014 Program

Sample Type	Type	Total Samples	Insertion Rate
Original Sections	1/2 HQ core, ¼ 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 (SRM's)	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 CRM’s	Pulp	49	10.6%*

Source: SRK, 2015
* Insertion rate is a percentage of total External Check Samples submitted

In addition to the QA/QC Program which accompanied the 2014 drilling program, there was also a QA/QC Program which accompanied the 2014 re-assay program which was undertaken to increase the database size for both titanium and scandium analysis. The 2014 re-assay program consisted of submitting 1,335 historic Molycorp pulps for scandium analysis. QA/QC for the re-assay program consisted of the insertion of pulp duplicate samples and SRMs. A summary of the type of samples, source and level of insertion for the re-assay program is included in Table 11.5.3.

SRK highlights that due to the timing of the relatively recent developments within the metallurgical database the routine submission of 2014 pulps did not include a scandium CRM. For the purpose of the current exercise SRK has relied heavily on the analysis of duplicate results to assign confidence, however SRK recommends the Company complete further verification using external checks and a suite of scandium SRM to increase the confidence further, provided such SRMs can be obtained.

SRK understands the Company has initiated these programs at the time of writing this report and data is expected to be available for review during the next Mineral Resource update.

Table 11.5.3: Summary of Actual Submissions per Sample Type within the 2014 Re-assay Program

Sample Type	Type	Total Samples	Insertion Rate
Original Sections	1/2 NQ core	1,335	NA
Pulp Duplicates	Pulp-split	7	0.5%
SRM's	Pulp	67	5.0%

Source: SRK, 2015

The following section provides details of the types of samples used at each section of the sampling process, followed by a discussion of the results. The QA/QC data was analyzed by the Project geologist on a routine basis prior to entering the data into the central database. Failures were reported directly back to the laboratory with systems (described in Section 11.5.1) in place for reanalysis (e.g. 10 samples before and after a failed standard).

SRK has been supplied with all the raw QA/QC data and has completed an independent check of the results.

11.5.1 Actions

The Company has a defined list of action points to review all QA/QC results. To review field quartz blanks a limit of 20 x ICP-MS detection limits and 2 x XRF detection limits, depending on the element being analyzed are applied. Results which report above this value are reported to the laboratory as having potential contamination.

The SRM has been sourced from Dillinger Hutte Laboratory (Germany). SRK has reviewed the certificates for each of the SRM's and notes that no standard deviation has been supplied and only a confidence interval of 95% is shown on the certificates (based on three laboratory round robin testwork). Due to a lack of information in the certificate the Company has elected to use a 5% error as a caution limit, and a 10% error as a failure. While this is not generally accepted as best practice, which would be based on 2x or 3x standard deviations for caution or failure, SRK agrees that the limits applied are reasonably tight which provide a reasonable level of control in assigning confidence to the assay results. SRK noted no significant difference in the potential pass/fail decisions using either the 10% or 3x 95% confidence limits from the certificate, and therefore considers the current limits to be acceptable.

In terms of the duplicate samples, no re-assay are requested based on the field duplicates, which are monitored for sample fluctuations and local variability. The reject and pulp duplicates are reviewed and re-assay s requested on values in excess of 20% difference using the equation:

$$\% \text{ Diff} = \text{ABS} [(X_1-X_2) / (X_1+X_2)] *100$$

When Duplicates or SRM's fail the 10 sequential samples on either side of the QC Sample are re-analyzed or re-analysis of the entire batch is requested, depending on the fail type, location, and sample range.

11.5.2 Field Sample Collection, Identification, Labeling, Insertion of Field Controls and Shipment

Sample tickets were assigned initially at the core shed using barcodes with duplicate tickets placed in the bag and on the outside of the bag, In addition to the routine samples a number of check samples (QC) were routinely inserted. Trained staff was involved at all stages of the sampling, sample packaging and sample transportation process.

Sample identification was confirmed using barcode labeling and visual sample type comparisons prior to sample shipment. Utilization of barcoded samples ensured both shipment forms and analytical labs used accurate information. Two types of QC samples were inserted at this stage of the process which includes the following:

- Field ¼-core duplicates – 1 in 20 (5%), inserted to test mineralization and sampling variability;
- Field quartz blanks – 1 in 20 (5%) Blanks were inserted within or immediately after samples collected from mineralized intervals, targeting zones of elevated visual mineralization, where possible; and
- SRM material – 1 in 20 (5%) is inserted in the field with the sample sequence.

11.5.3 Sample Preparation and Insertion of Pre-Selected and Quality Control Samples: Actlabs

Samples were dispatched to the laboratory via commercial transport. The laboratory received and weighed the samples. Receiving logs were monitored by Dahrouge, which were then checked against original sample lists to ensure accuracy.

The standard sample preparation at the laboratory targeting the criteria of 95% passing 200 mesh (-200 mesh). The high passing rate and the fine mesh are required to ensure the niobium minerals are sufficiently liberated for sub-sampling due to the fine size fractions known from metallurgical and petrographical studies.

To ensure quality throughout the sample preparation phase, NioCorp (via Dahrouge) utilized the insertion and splitting of pre-selected control and duplicate samples, based on the insertion rates shown in Table 11.5.1.

11.5.4 Results

Standards (SRMs)

The 2014 Program included 496 SRMs inserted in Actlabs batches as part of the routine sample submissions. The material was sourced from Dillinger Haute laboratory (Germany). A summary of the defined limits and results for Nb_2O_5 and TiO_2 are shown in Tables 11.5.4.1 and 11.5.4.2, respectively. The tables show the mean assay grades vs. the assigned, plus a summary of the number of samples returned outside of the warning and acceptable limits.

Table 11.5.4.1: Summary of Nb_2O_5 Results of SRM's Submitted to Actlabs

Standard ID	Assigned (%)	Count	Mean Assay (%)	Standard Deviation	Range	Minimum	Maximum	Difference From Assigned Grade	N outside 10%		N outside 5%	
SX18-01	0.695	169	0.712	0.023	0.176	0.593	0.769	2.4%	3	1.8%	36	21.3%
SX18-02	0.199	154	0.207	0.005	0.025	0.193	0.218	4.0%	0	0.0%	57	37.0%
SX18-04	1.32	8	1.016	0.019	0.055	0.988	1.043	4.4%	0	0.0%	3	37.5%
SX18-05	0.973	169	0.712	0.023	0.176	0.593	0.769	2.4%	3	1.8%	36	21.3%

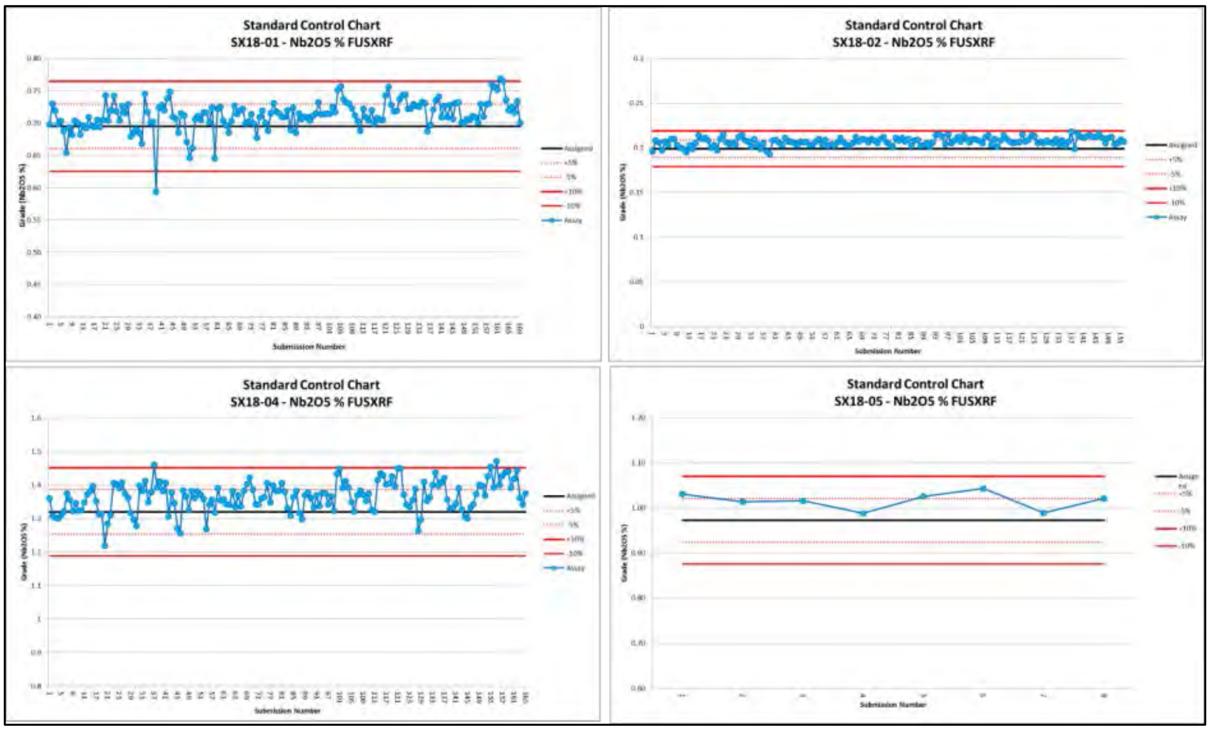
Source: SRK, 2015

Table 11.5.4.2: Summary of TiO_2 Results of SRM's Submitted to Actlabs

Standard ID	Assigned (%)	Count	Mean Assay (%)	Standard Deviation	Range	Minimum	Maximum	Difference From Assigned Grade	N outside 10%		N outside 5%	
SX18-01	0.266	169	0.254	0.013	0.141	0.234	0.375	-4.5%	10	5.9%	90	53.3%
SX18-02	0.237	154	0.231	0.008	0.062	0.201	0.263	-2.5%	5	3.2%	33	21.4%
SX18-04	0.287	165	0.265	0.011	0.056	0.235	0.291	-7.7%	34	20.6%	123	74.5%
SX18-05	0.295	8	0.284	0.006	0.018	0.271	0.289	-3.7%	0	0.0%	1	12.5%

Source: SRK, 2015

The results for Nb_2O_5 (Figure 11.5.4.1) from the SRM submissions have been within acceptable limits, with results generally reporting slightly above the assigned grades (between 2.4% and 4.4%). This can be seen in SX18-02 with the assay values typically reporting above the assigned value of the SRM. In general these range between the assigned value and the ±5% caution line. Statistically the results indicate a slight high bias across all grade ranges with the differences between the mean and assigned grades ranging from 2.4% to 4.4%. A total of six samples have reported outside the failure level of ±10%.

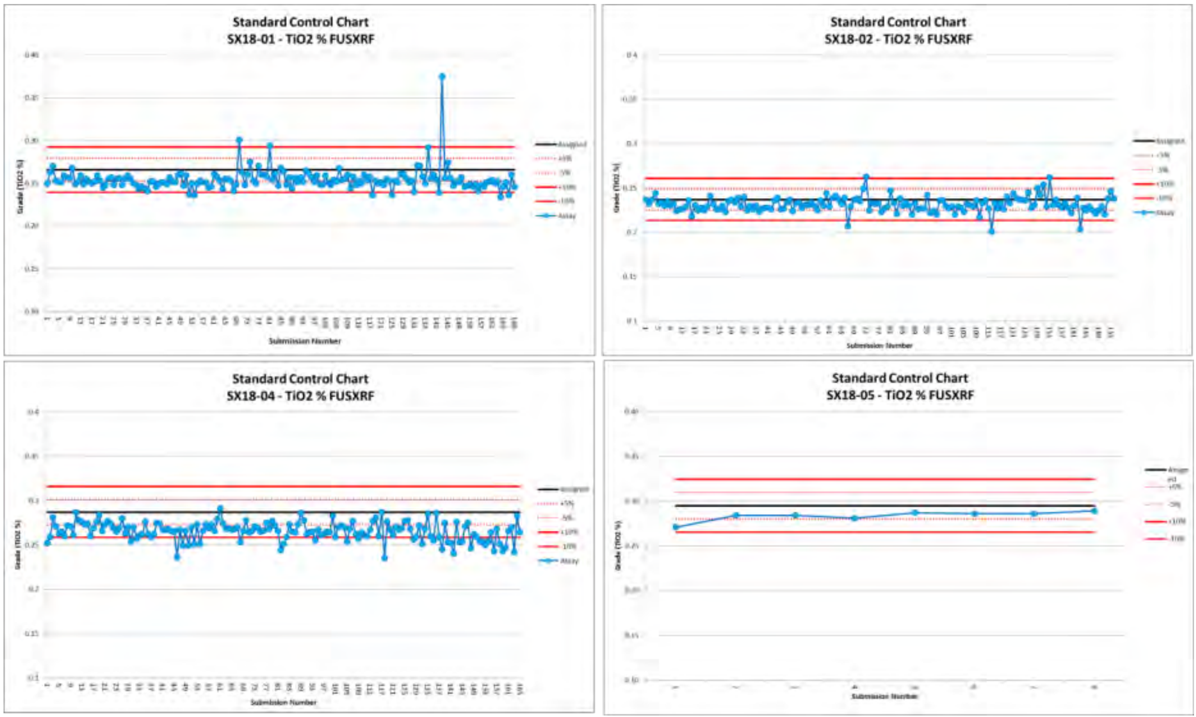


Source: SRK, 2015

Figure 11.5.4.1: Summary of SRM Control Charts for Nb_2O_5 Submitted to Actlabs (2014)

Analysis of the selected SRM used for Nb_2O_5 assay certificates also provides an expected value and associated range for TiO_2, for all four SRM's selected (Figure 11.5.4.2). The most significant issue to note is that the grade range of the TiO_2 in the SRM 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 geological wireframe. The results from the low grade analysis shows the analyses are typically below the assigned grade within 5% to 10%. The lowest performance is noted in SX-18-04 where a number of the assays report less than 10% low.

Given the low grade nature of the assays in the SRM's SRK has relied more heavily on the duplicate assays and external checks by SGS. SRK recommends the Company define a program where a small proportion of the assays across all grade ranges (1% to 2%) are sent for reanalysis with new SRMs that cover the full range of expected grades to add to the confidence in the assay database.



Source: SRK, 2015

Figure 11.5.4.2: Summary of CRM Control Charts for TiO_2 Submitted to Actlabs (2014)

Only 67 scandium SRM submissions have been analyzed to date (Figure 11.5.4.3). The routine submissions to Actlabs (which were analyzed for Sc) were completed prior to changes in the metallurgical flowsheet. With the revised focus on titanium and scandium the company conducted a re-assay program of 2011 sample pulps which had not previously been analyzed for TiO_2 or Sc. A scandium SRM was included within these batches. The results indicate good correlation between the laboratory scandium values and the expected grade with a difference in the mean of 0.36 ppm or 0.4%. A total of three samples have reported above the guideline line of 3 standard deviations during the study.

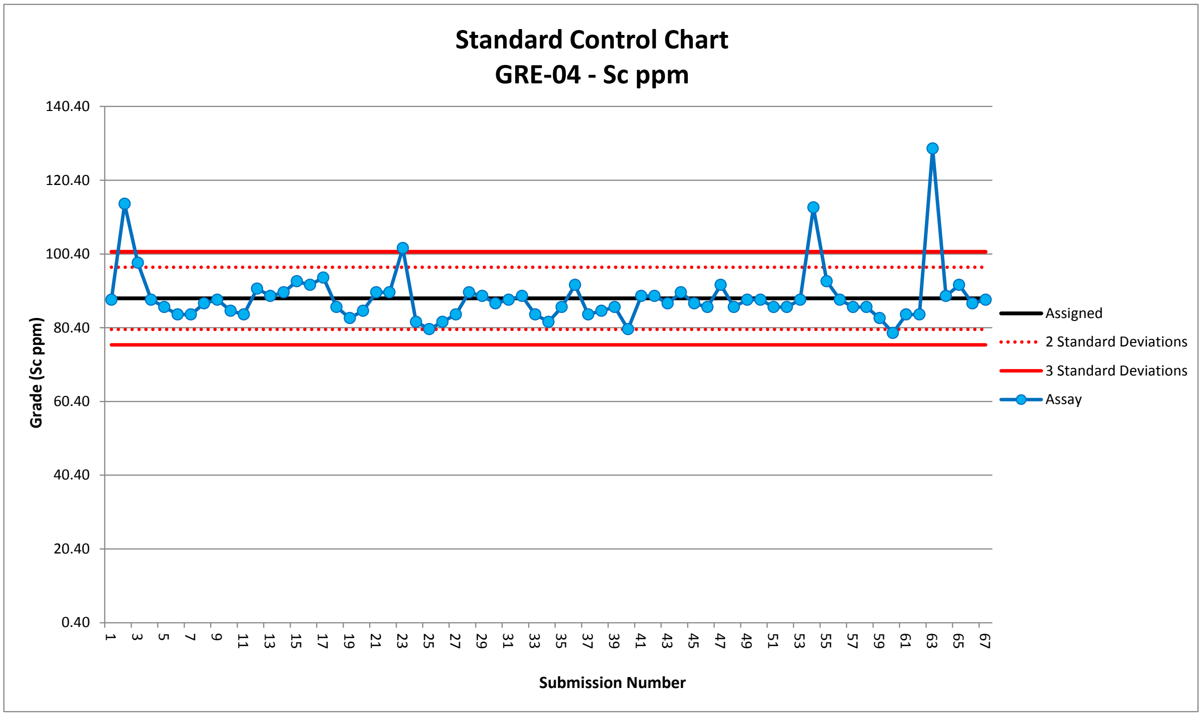


Figure 11.5.4.3: Summary of CRM Control Charts for Sc (ppm) Submitted to SGS (2014)

Overall SRK considers that the SRM's have performed within acceptable levels of error for the reporting of Mineral Resources. SRK has discussed the slight high bias in the Nb_2O_5 assays with the Project geologist who has raised the issue with the laboratory, as review of the laboratory internal SRM values indicated the assays are performing within the laboratory defined limits. The external CRM values however show the laboratory has over reported based on routine submitted SRM in the order of 2% to 4%.

Blanks

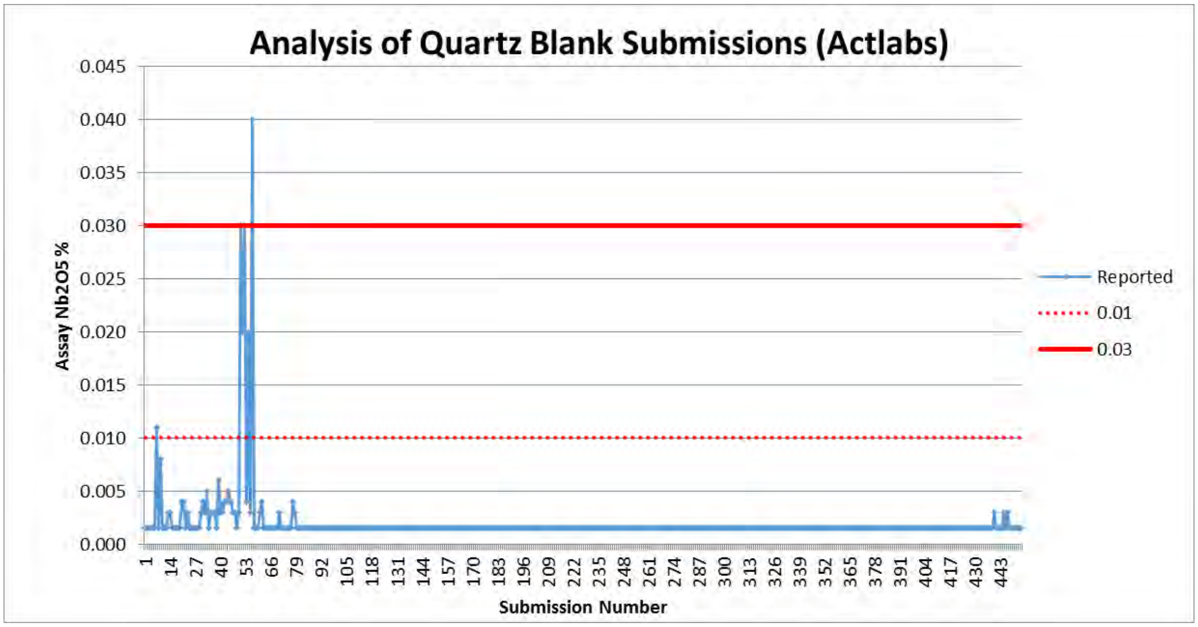
Coarse natural clear quartz blanks (sourced from an optical quality quartz quarry) were also included in order to:

- Pass through the same sample preparation system as the real samples and highlight any potential contamination; and
- Be indistinguishable from real samples and prevent these samples being treated in a different manner to real samples at the laboratory.

The following certified natural blanks were inserted within batches of samples sent to the laboratory. In total, 454 natural blanks (4.7% total submissions) were inserted at regular intervals within the sample suite which represents 4.7% of total sample submissions from the 2014 drilling program. The detection limits for Nb_2O_5 and TiO_2 are 0.003% and 0.001% respectively. SRK has assigned control limits (approximately 10x detection) at 0.01% and 0.03% for both Nb_2O_5 and TiO_2.

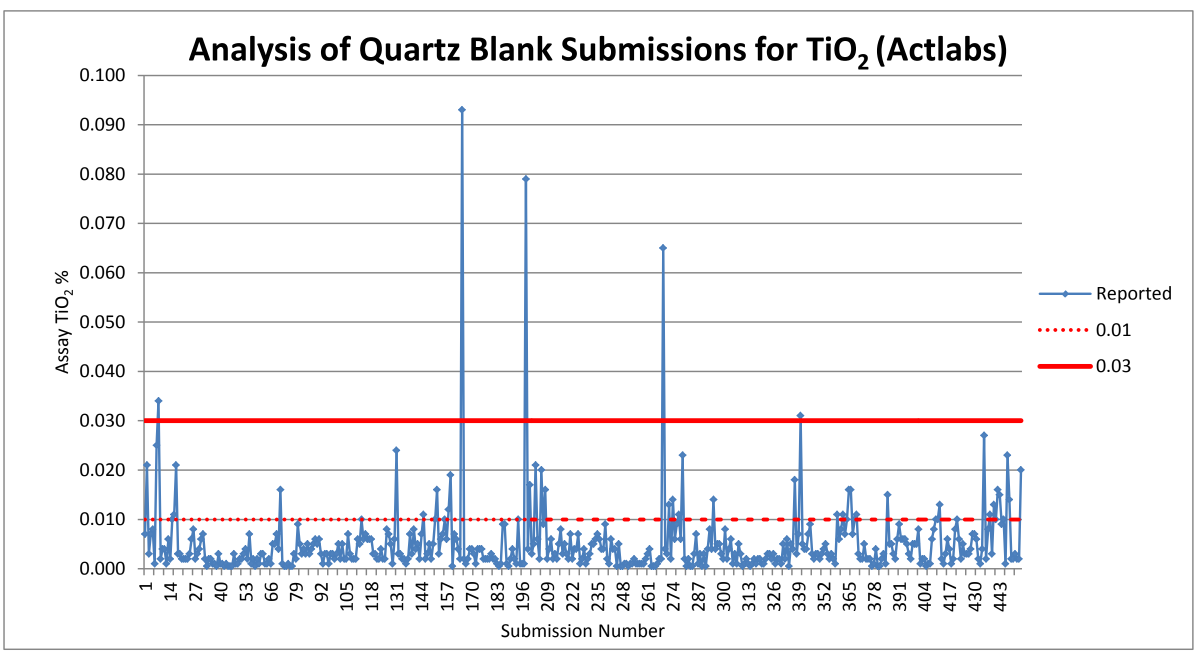
SRK notes that cluster assays (Figure 11.5.4.4), during the early stages of the drilling, displayed potential sample contaminations but SRK does not consider this to be material to the Mineral Resource Estimate.

SRK notes for the TiO_2 data (Figure 11.5.4.5) more variability is noted than within the Nb_2O_5 database, but overall the majority of the samples are less than 0.01% control line which is the equivalent of 10x the detection limit, above which potential contamination maybe identified. Overall SRK considers that the blank material has performed within acceptable levels of error and there is limited evidence of any major contamination issues at the laboratory.



Source: SRK, 2015

Figure 11.5.4.4: Summary of Blank Control Charts for Nb_2O_5 Submission to Actlabs (2014)



Source: SRK, 2015

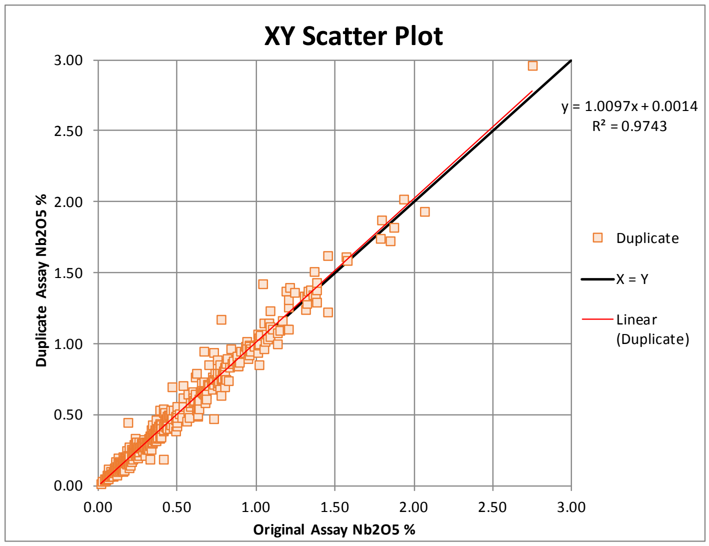
Figure 11.5.4.5: Summary of Blank Control Charts for Nb_2O_5 Submission to Actlabs (2014)

Duplicates

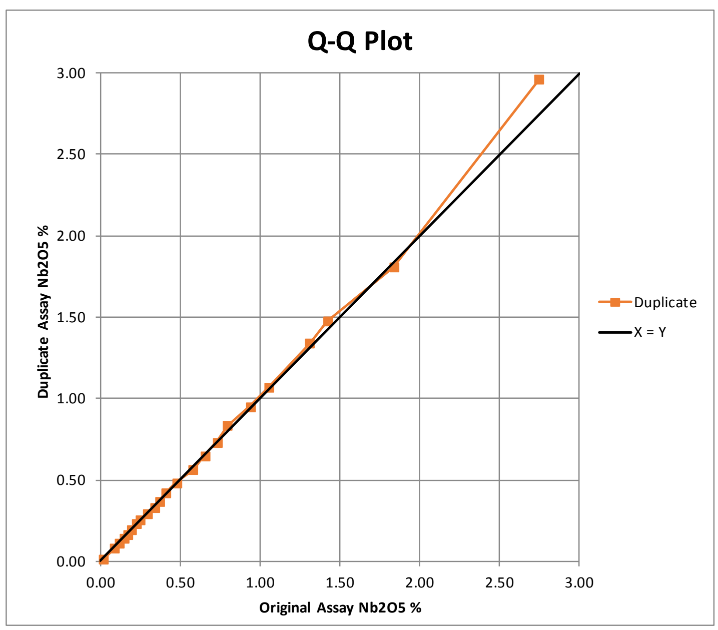
A total of 419 field duplicate samples, comprised of ¼ core, were resubmitted to Actlabs as part of the routine sample submission from DDH samples, which represent 4.3% of total sample submissions from the 2014 drilling program. The results are shown in Figure 11.5.4.6 and Figure 11.5.4.7, and indicate a reasonable comparison between the original and duplicate assays. SRK has also compared the base statistics for the two datasets and found the difference in the mean grades to be 1.3% for Nb_2O_5 and 1.0% for TiO_2, which indicates an acceptable level of precision at the laboratory.

ASSAY Value	*Original*	*Duplicate*
Mean	0.50	0.51
Standard Error	0.02	0.02
Median	0.37	0.37
Mode	0.21	0.10
Standard Deviation	0.40	0.41
Sample Variance	0.16	0.17
Kurtosis	2.97	3.39
Skewness	1.48	1.52
Range	2.74	2.95
Minimum	0.01	0.01
Maximum	2.75	2.96
Sum	210.51	213.15
Count	419	419
Confidence Level(95.	0.04	0.04

% Difference Mean **1.26%**



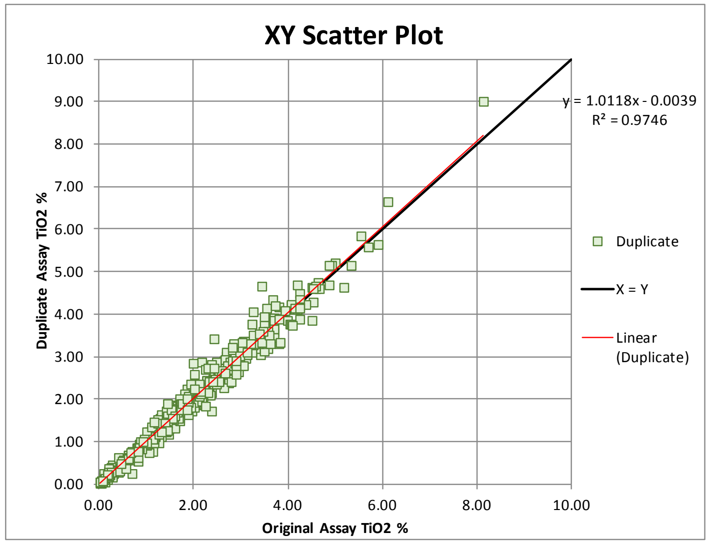
Percentile	Original	Duplicate
0.0%	0.01	0.01
5.0%	0.09	0.08
10.0%	0.12	0.11
15.0%	0.15	0.15
20.0%	0.17	0.17
25.0%	0.19	0.19
30.0%	0.22	0.23
35.0%	0.25	0.26
40.0%	0.30	0.29
45.0%	0.34	0.33
50.0%	0.37	0.37
55.0%	0.41	0.42
60.0%	0.48	0.48
65.0%	0.58	0.56
70.0%	0.65	0.65
75.0%	0.73	0.73
80.0%	0.80	0.83
85.0%	0.94	0.95
90.0%	1.06	1.07
95.0%	1.31	1.34
97.5%	1.42	1.47
99.0%	1.84	1.81
100.0%	2.75	2.96



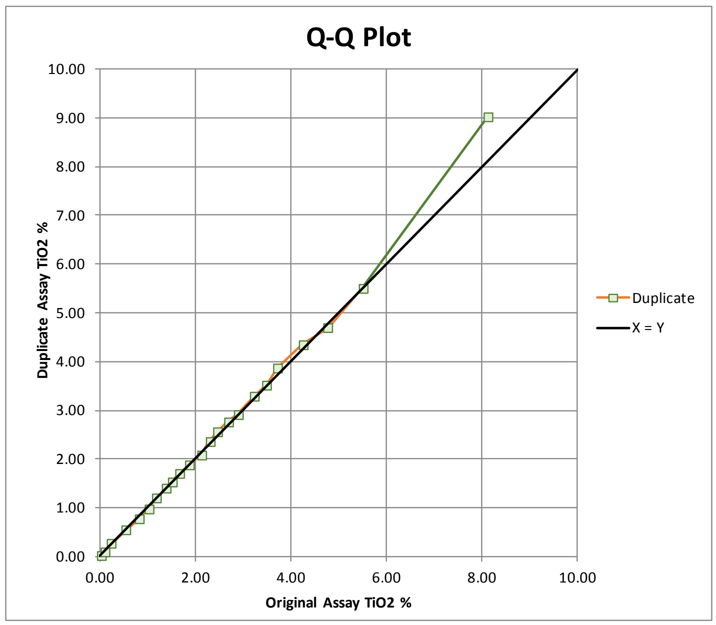
Source: SRK, 2015

Figure 11.5.4.6: XY Scatter and QQ Plot Showing Comparison of Original vs. Field Duplicate Analysis Nb_2O_5

ASSAY Value	*Original*	*Duplicate*
Mean	2.03	2.05
Standard Error	0.07	0.07
Median	1.87	1.88
Mode	1.64	1.65
Standard Deviation	1.34	1.37
Sample Variance	1.79	1.88
Kurtosis	0.41	0.94
Skewness	0.61	0.69
Range	8.10	8.99
Minimum	0.02	0.03
Maximum	8.12	9.02
Sum	849.55	857.91
Count	419.00	419.00
Confidence Level(95.	0.13	0.13
% Difference Mean		**0.98%**



Percentile	Original	Duplicate
0.0%	0.02	0.03
5.0%	0.11	0.11
10.0%	0.24	0.26
15.0%	0.55	0.55
20.0%	0.82	0.77
25.0%	1.02	0.98
30.0%	1.19	1.21
35.0%	1.40	1.40
40.0%	1.51	1.53
45.0%	1.67	1.70
50.0%	1.87	1.88
55.0%	2.13	2.09
60.0%	2.30	2.36
65.0%	2.47	2.56
70.0%	2.69	2.76
75.0%	2.90	2.91
80.0%	3.24	3.30
85.0%	3.50	3.52
90.0%	3.71	3.86
95.0%	4.25	4.35
97.5%	4.78	4.69
99.0%	5.50	5.51
100.0%	8.12	9.02

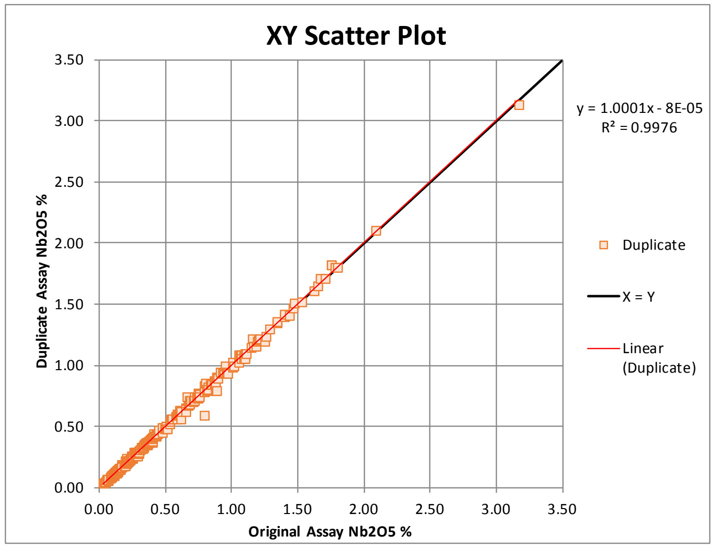


Source: SRK, 2015

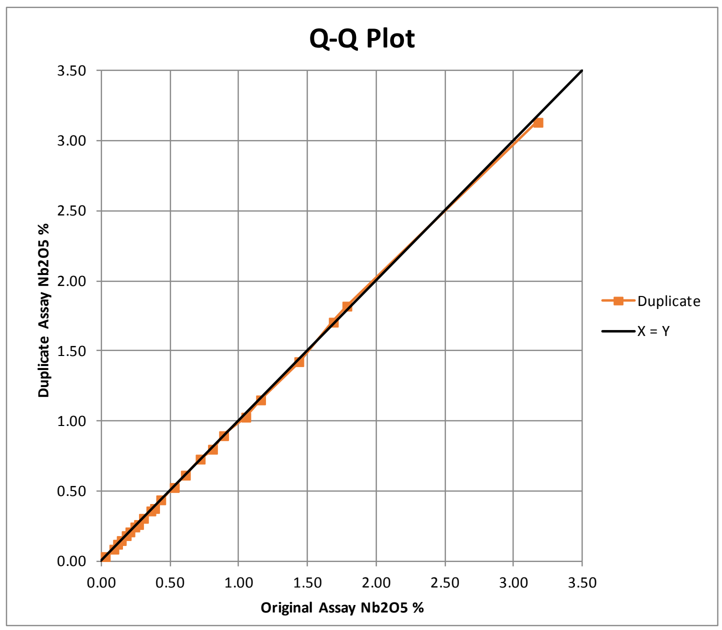
Figure 11.5.4.7: XY Scatter and QQ Plot Showing Comparison of Original vs. Field Duplicate Analysis TiO_2

260 reject duplicate samples, comprising a second riffled sample split taken after crushing, were submitted to Actlabs for reanalysis (blind) as part of the routine sample submission from DDH samples, which represent 2.7% of the total sample submissions from the 2014 drilling program. The results are shown in Figure 11.5.4.8 and Figure 11.5.4.9, and indicate a reasonable comparison between the original and duplicate assays. SRK has also compared the base statistics for the two datasets and found the difference in the mean grades to be 0.0% for Nb_2O_5 and 0.3% for TiO_2, which indicates an acceptable level of precision at the laboratory.

ASSAY Value	*Original*	*Duplicate*
Mean	0.55	0.55
Standard Error	0.03	0.03
Median	0.39	0.38
Mode	0.12	0.22
Standard Deviation	0.46	0.46
Sample Variance	0.21	0.21
Kurtosis	3.99	3.85
Skewness	1.59	1.59
Range	3.14	3.10
Minimum	0.03	0.03
Maximum	3.17	3.13
Sum	143.42	143.40
Count	260	260
Confidence Level(95.	0.06	0.06
% Difference Mean		**-0.01%**



Percentile	Original	Duplicate
0.0%	0.03	0.03
5.0%	0.09	0.09
10.0%	0.12	0.12
15.0%	0.15	0.15
20.0%	0.18	0.18
25.0%	0.21	0.21
30.0%	0.24	0.24
35.0%	0.27	0.27
40.0%	0.31	0.31
45.0%	0.36	0.36
50.0%	0.39	0.38
55.0%	0.44	0.44
60.0%	0.53	0.53
65.0%	0.62	0.62
70.0%	0.72	0.73
75.0%	0.81	0.80
80.0%	0.89	0.90
85.0%	1.05	1.03
90.0%	1.16	1.16
95.0%	1.44	1.42
97.5%	1.69	1.71
99.0%	1.79	1.82
100.0%	3.17	3.13

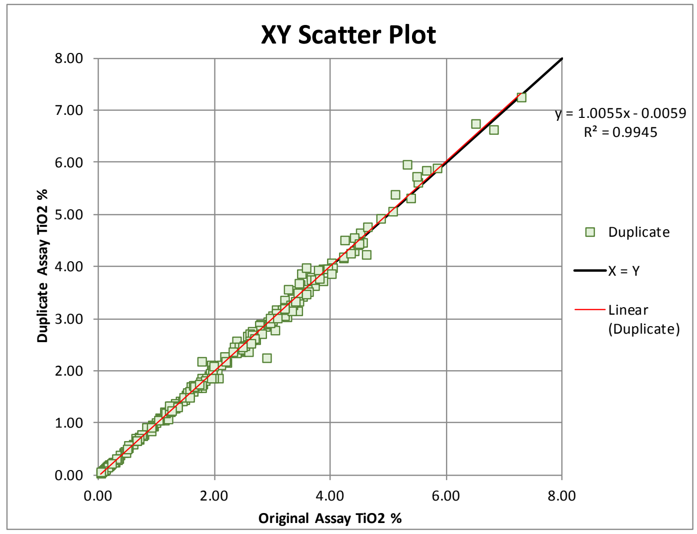


Source: SRK, 2015

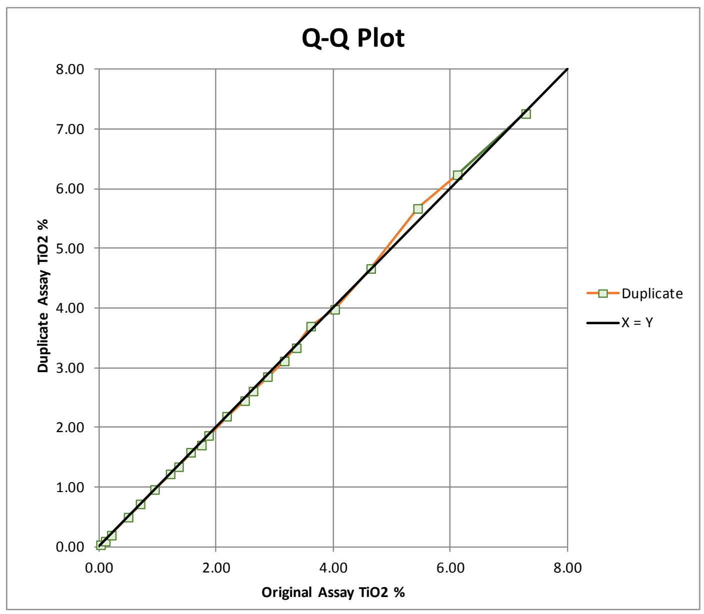
Figure 11.5.4.8: XY Scatter and QQ Plot Showing Comparison of Original vs. Reject Duplicate (Riffle Split) Analysis Nb_2O_5

ASSAY Value	*Original*	*Duplicate*
Mean	2.13	2.13
Standard Error	0.09	0.09
Median	1.87	1.86
Mode	0.94	1.44
Standard Deviation	1.47	1.49
Sample Variance	2.18	2.21
Kurtosis	0.18	0.27
Skewness	0.65	0.69
Range	7.25	7.23
Minimum	0.03	0.03
Maximum	7.28	7.26
Sum	552.69	554.18
Count	260.00	260.00
Confidence Level(95.	0.18	0.18

% Difference Mean **0.27%**



Percentile	Original	Duplicate
0.0%	0.03	0.03
5.0%	0.11	0.11
10.0%	0.20	0.20
15.0%	0.49	0.50
20.0%	0.71	0.72
25.0%	0.95	0.97
30.0%	1.21	1.22
35.0%	1.35	1.34
40.0%	1.57	1.58
45.0%	1.74	1.72
50.0%	1.87	1.86
55.0%	2.18	2.18
60.0%	2.48	2.45
65.0%	2.63	2.62
70.0%	2.88	2.85
75.0%	3.16	3.11
80.0%	3.36	3.35
85.0%	3.61	3.70
90.0%	4.02	3.98
95.0%	4.63	4.65
97.5%	5.44	5.67
99.0%	6.11	6.23
100.0%	7.28	7.26

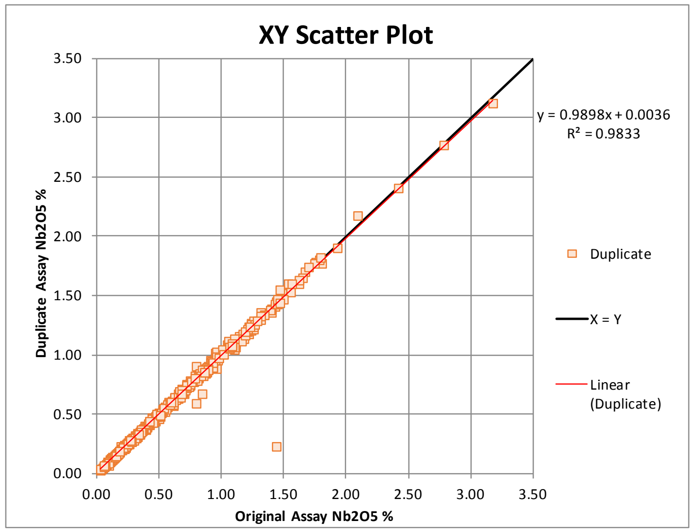


Source: SRK, 2015

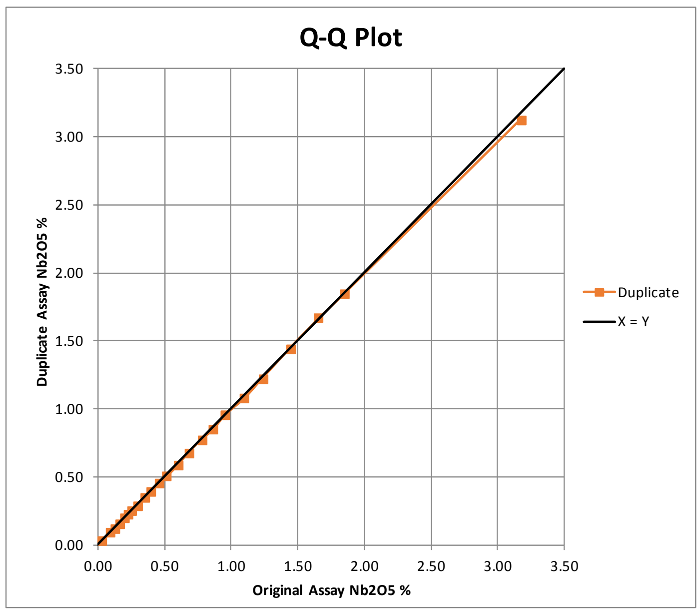
Figure 11.5.4.9: XY Scatter and QQ Plot Showing Comparison of Original vs. Reject Duplicate (Riffle Split) Analysis TiO_2

There were 468 pulp duplicate samples, comprising a second riffled sample split, taken after pulverization, were submitted as part of the routine sample submission from DDH samples, which represent 4.9% of total sample submissions from the 2014 drilling program. The results are shown in Figure 11.5.4.10 and Figure 11.5.4.11, and indicate a reasonable comparison between the original and duplicate assays. SRK 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 and 0.3% for TiO_2, which indicates an acceptable level of precision at the laboratory.

ASSAY Value	*Original*	*Duplicate*
Mean	0.59	0.59
Standard Error	0.02	0.02
Median	0.46	0.45
Mode	0.17	0.21
Standard Deviation	0.46	0.46
Sample Variance	0.22	0.22
Kurtosis	2.86	2.82
Skewness	1.38	1.39
Range	3.14	3.09
Minimum	0.03	0.03
Maximum	3.17	3.13
Sum	276.32	275.19
Count	468	468
Confidence Level(95.	0.04	0.04
% Difference Mean		**-0.41%**



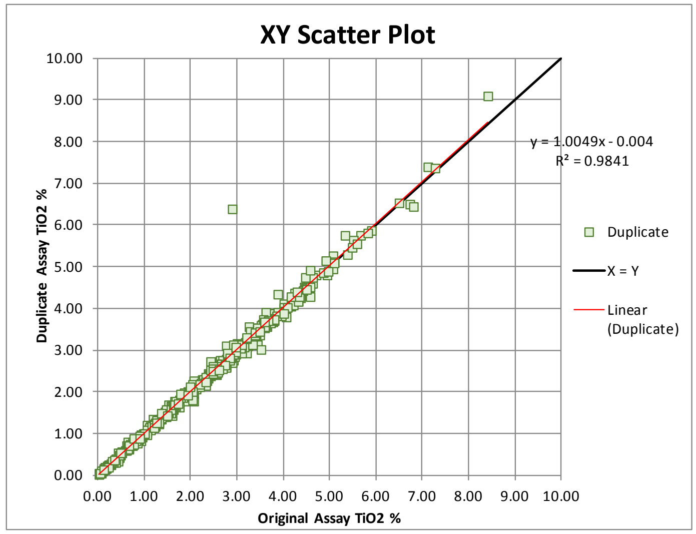
Percentile	Original	Duplicate
0.0%	0.03	0.03
5.0%	0.09	0.09
10.0%	0.13	0.12
15.0%	0.16	0.16
20.0%	0.20	0.20
25.0%	0.22	0.22
30.0%	0.26	0.26
35.0%	0.29	0.29
40.0%	0.35	0.35
45.0%	0.40	0.40
50.0%	0.46	0.45
55.0%	0.51	0.51
60.0%	0.60	0.59
65.0%	0.68	0.68
70.0%	0.78	0.77
75.0%	0.86	0.85
80.0%	0.95	0.96
85.0%	1.10	1.08
90.0%	1.24	1.22
95.0%	1.44	1.44
97.5%	1.65	1.67
99.0%	1.84	1.85
100.0%	3.17	3.13



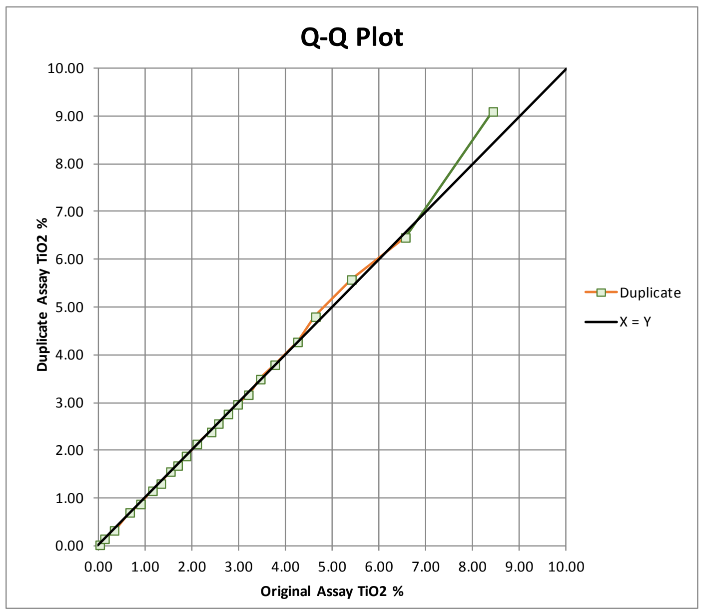
Source: SRK, 2015

Figure 11.5.4.10: XY Scatter and QQ Plot Showing Comparison of Original vs. Pulp Duplicate Analysis Nb_2O_5

ASSAY Value	*Original*	*Duplicate*
Mean	2.26	2.26
Standard Error	0.07	0.07
Median	2.10	2.13
Mode	0.94	0.53
Standard Deviation	1.47	1.49
Sample Variance	2.18	2.23
Kurtosis	0.46	0.76
Skewness	0.65	0.72
Range	8.39	9.06
Minimum	0.03	0.03
Maximum	8.42	9.09
Sum	1056.03	1059.36
Count	468.00	468.00
Confidence Level(95.	0.13	0.14
% Difference Mean		**0.32%**



Percentile	Original	Duplicate
0.0%	0.03	0.03
5.0%	0.14	0.15
10.0%	0.34	0.32
15.0%	0.66	0.70
20.0%	0.90	0.87
25.0%	1.14	1.16
30.0%	1.32	1.31
35.0%	1.54	1.55
40.0%	1.69	1.69
45.0%	1.88	1.87
50.0%	2.10	2.13
55.0%	2.40	2.40
60.0%	2.57	2.57
65.0%	2.77	2.76
70.0%	2.97	2.96
75.0%	3.21	3.15
80.0%	3.46	3.49
85.0%	3.77	3.80
90.0%	4.26	4.28
95.0%	4.64	4.81
97.5%	5.42	5.59
99.0%	6.58	6.46
100.0%	8.42	9.09



Source: SRK, 2015

Figure 11.5.4.11: XY Scatter and QQ Plot Showing Comparison of Original vs. Pulp Duplicate Analysis TiO_2

SRK has reviewed all the data available using XY Scatter Plots, QQ-Plots and ARD vs. percentage rank charts. Based on the review SRK concludes that no significant issues in terms of the precision exists from the Actlabs assays in the database, with all phases of the sample preparation displaying strong correlations between the original and duplicate assays. The results confirm the expected trend of greater precision within pulp duplicates vs. field duplicates, which are have more potential variability from the sample itself (geology), through the entire sampling process (laboratory precision).

11.5.5 Check Analysis SGS vs. 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 the equivalent of approximately 5% of the original submissions.

The SRM material submitted to SGS returned assays which were very close to the SRM values indicating slightly better accuracy than Actlabs for both Nb_2O_5 (Figure 11.5.5.1) and TiO_2 (Figure 11.5.5.2). The charts indicate that similar to Actlabs the SRM's return values at or above the assigned grades for Nb_2O_5 and values at or below the assigned grades for TiO_2. In SRK's opinion both laboratories provide sufficient accuracy for Indicated Mineral Resources.

A review of the XY Scatter plot (Figure 11.5.5.3) for Nb_2O_5 shows Actlabs reporting consistently higher across all grade ranges. SRK assumes this indicates some difference either in the method or equipment accuracy at one of the laboratories. A comparison of the mean Nb_2O_5 grades indicates an 8.7%high bias at Actlabs compared to SGS.

The bias is consistent with higher values reporting larger differences. SRK recommends the Company follow-up with both Laboratories to understand the fundamental difference in the sampling methods and identify the source of the bias. The results of the insertion of SRM material to the two laboratories indicate that SGS in general reports better accuracy than Actlabs, but the dataset is limited to 49 submissions vs. 492 submissions at SGS and Actlabs respectively. The results from SGS are assumed to present more accurate results SRK recommend that a more comprehensive set of samples are submitted to SGS with SRMs, specifically focusing on mid to high grades, so that a correction can be built into the Actlabs assays for future estimates. SRK does not suspect this will have a material impact on the overall Mineral Resource and the difference will be within acceptable level of errors of the current classification system. SRK understands that the Company has undertaken a supplemental assay program to address the bias as of the time of writing this report. The results of such analysis will be integrated into any future technical studies.

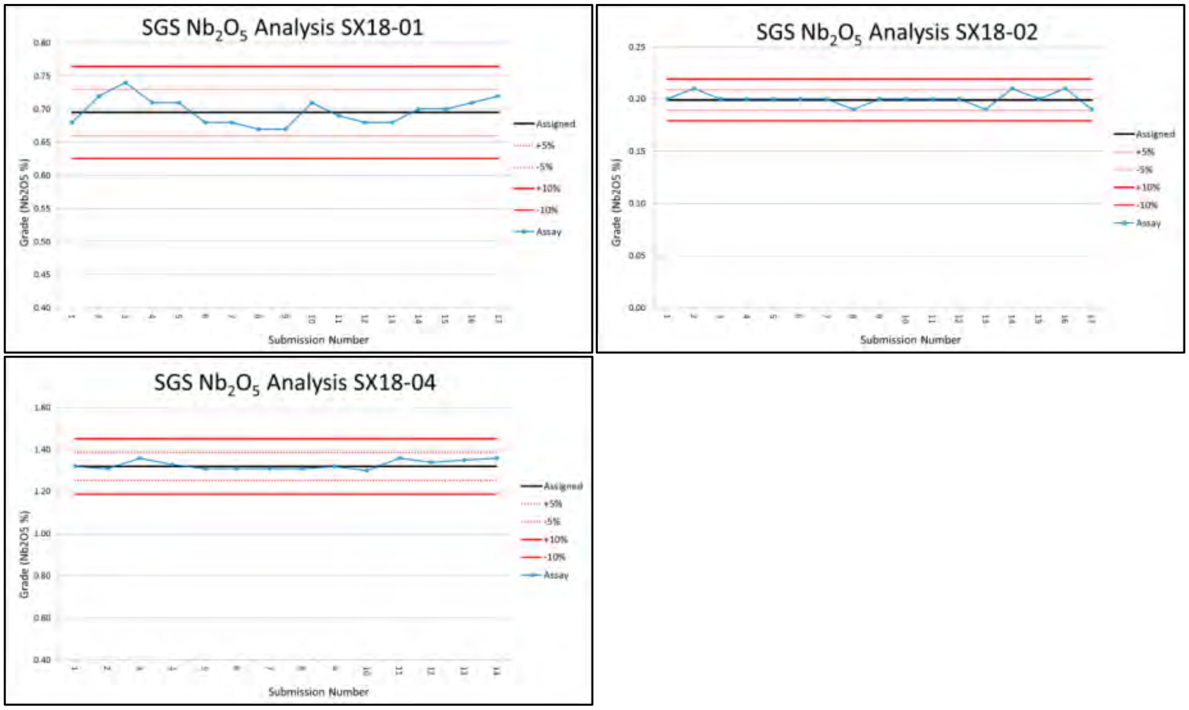
A review of the XY Scatter plot (Figure 11.5.5.4) for TiO_2 shows a slight low bias exists between the two datasets with Actlabs reporting consistently lower across all grade ranges. SRK assumes this indicates some difference in the either method or equipment accuracy at one of the laboratories. A comparison of the mean grades indicates a low bias towards the Actlabs results in the order of -4.6% on the mean grades (Actlabs reporting lower grades).

A review of the XY Scatter plot (Figure 11.5.5.5) for Sc shows good correlation between the two datasets with Actlabs reporting slightly lower grades across all grade ranges. A comparison of the mean grades indicates good correlation with Actlabs results in the order of -0.5% of the mean grades (Actlabs reporting lower grades).

In addition to the pulp duplicates a further 44 samples were submitted which were further duplicates of the external laboratory pulp submissions. SRK has reviewed this information and does not note any significant bias.

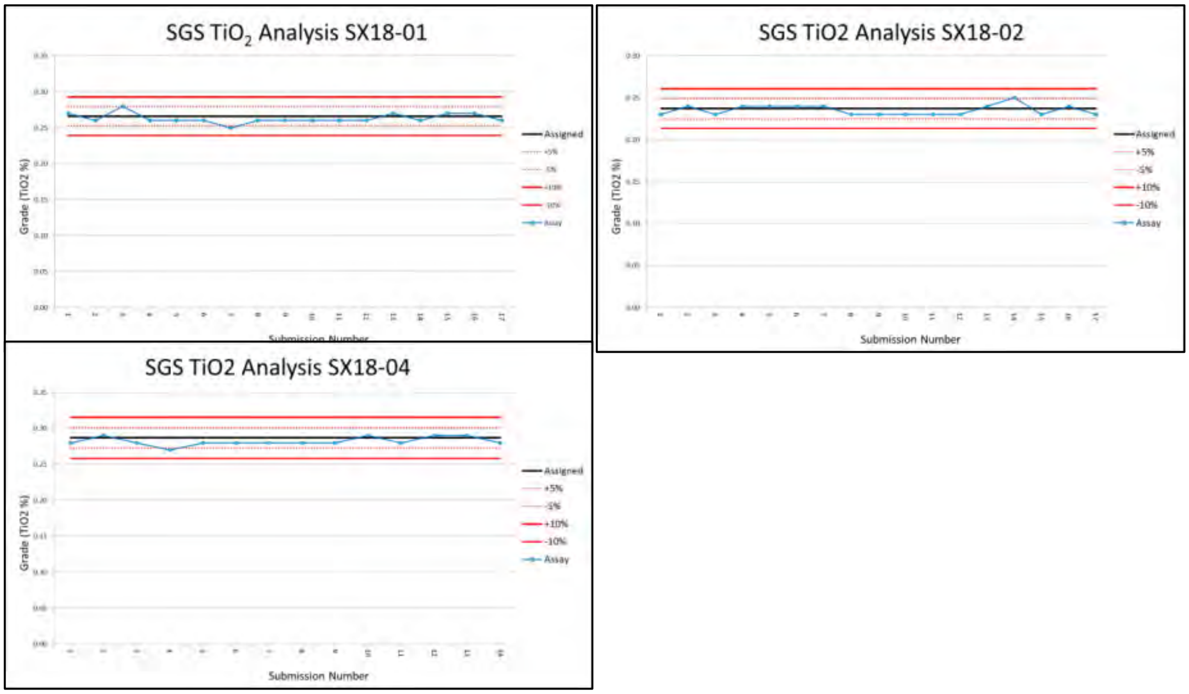
After considering the performance of the two laboratories for SRM material submitted, SRK concludes that a slight high bias exists in Nb_2O_5 assays for the 2014 database, with Actlabs returning higher assays than SGS. The bias has been reported to the laboratory and investigations into probable cause are ongoing. No conclusion on the source of the bias is available at the time of writing.

SRK concludes that while a bias exists it is currently within acceptable levels of error and therefore will not materially impact on the Mineral Resource. SRK has accepted the database as presented by Actlabs, and not made any adjustments to the assay information provided.



Source: SRK, 2015

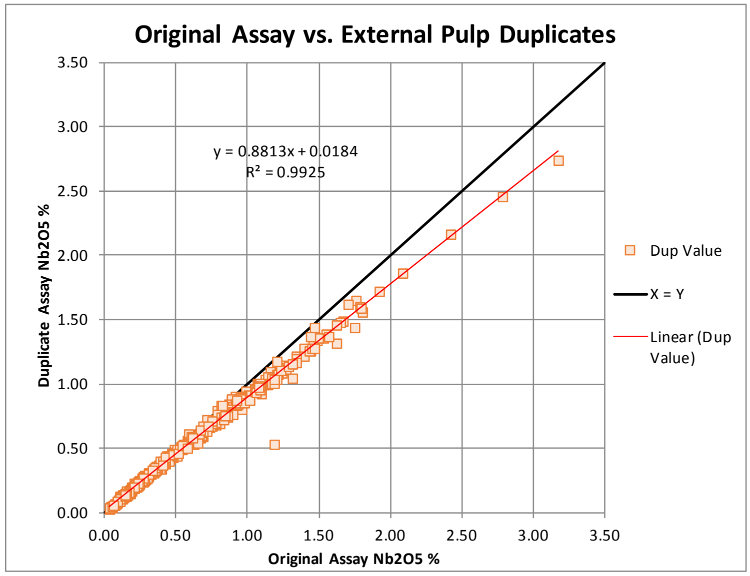
Figure 11.5.5.1: Summary of SRM Nb_2O_5 Assays Submitted to SGS During Check Analysis



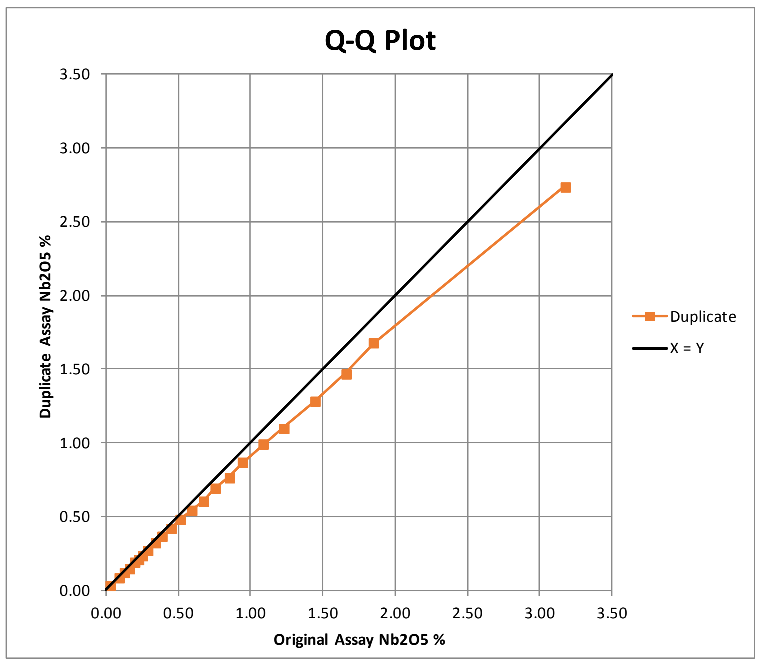
Source: SRK, 2015

Figure 11.5.5.2: Summary of SRM TiO_2 Assays Submitted to SGS during Check Analysis

ASSAY Value	*Original*	*Duplicate*
Mean	0.59	0.53
Standard Error	0.02	0.02
Median	0.45	0.43
Mode	0.17	0.20
Standard Deviation	0.46	0.41
Sample Variance	0.22	0.17
Kurtosis	2.99	2.68
Skewness	1.41	1.35
Range	3.14	2.71
Minimum	0.03	0.03
Maximum	3.17	2.74
Sum	270.55	246.94
Count	462	462
Confidence Level(95.	0.04	0.04
% Difference Mean		**-8.73%**



Percentile	Original	Duplicate
0.0%	0.03	0.03
5.0%	0.09	0.09
10.0%	0.12	0.12
15.0%	0.16	0.15
20.0%	0.20	0.19
25.0%	0.22	0.21
30.0%	0.25	0.24
35.0%	0.29	0.27
40.0%	0.34	0.32
45.0%	0.39	0.36
50.0%	0.45	0.43
55.0%	0.51	0.48
60.0%	0.59	0.54
65.0%	0.68	0.61
70.0%	0.75	0.69
75.0%	0.85	0.77
80.0%	0.95	0.87
85.0%	1.09	0.99
90.0%	1.23	1.10
95.0%	1.44	1.28
97.5%	1.66	1.47
99.0%	1.85	1.68
100.0%	3.17	2.74

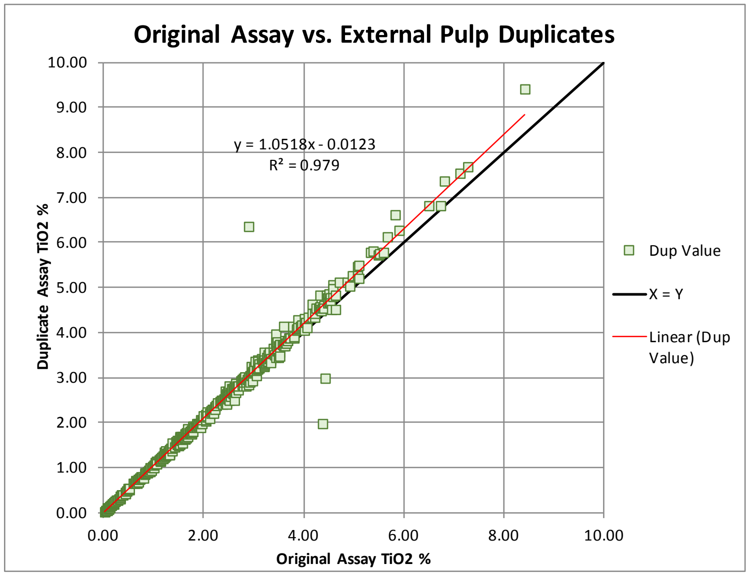


Source: SRK, 2014

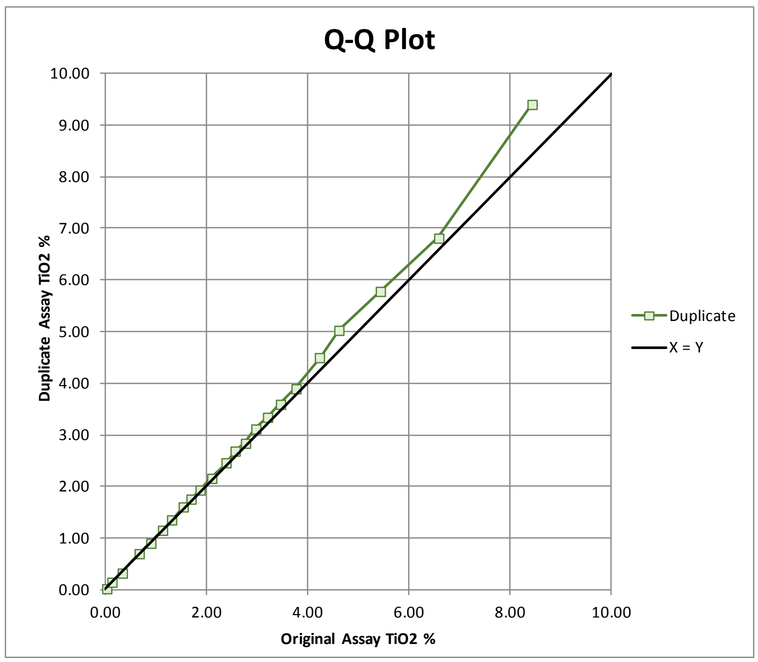
Figure 11.5.5.3: XY Scatter and QQ Plot Showing Comparison of Original vs. Umpire Laboratory Analysis Nb_2O_5

ASSAY Value	Original	Duplicate
Mean	2.25	2.35
Standard Error	0.07	0.07
Median	2.10	2.17
Mode	0.09	1.78
Standard Deviation	1.48	1.57
Sample Variance	2.18	2.47
Kurtosis	0.47	0.76
Skewness	0.66	0.74
Range	8.39	9.38
Minimum	0.03	0.02
Maximum	8.42	9.40
Sum	1039.25	1087.41
Count	462	462
Confidence Level(95.	0.14	0.14

% Difference Mean **4.63%**



Percentile	Original	Duplicate
0.0%	0.03	0.02
5.0%	0.14	0.15
10.0%	0.33	0.33
15.0%	0.66	0.69
20.0%	0.89	0.90
25.0%	1.13	1.16
30.0%	1.32	1.35
35.0%	1.53	1.59
40.0%	1.68	1.75
45.0%	1.87	1.93
50.0%	2.10	2.17
55.0%	2.39	2.46
60.0%	2.57	2.68
65.0%	2.77	2.84
70.0%	2.97	3.12
75.0%	3.21	3.35
80.0%	3.46	3.59
85.0%	3.77	3.90
90.0%	4.24	4.49
95.0%	4.63	5.03
97.5%	5.43	5.78
99.0%	6.59	6.81
100.0%	8.42	9.40

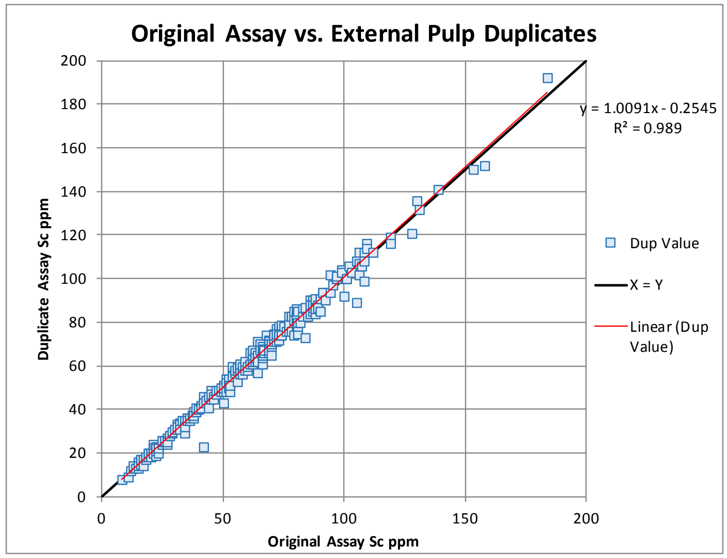


Source: SRK, 2015

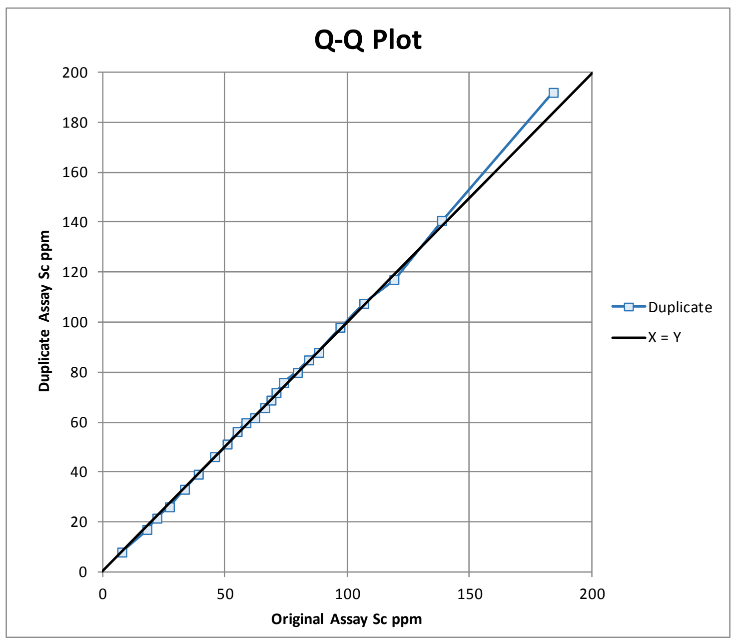
Figure 11.5.5.4: XY Scatter and QQ Plot Showing Comparison of Original vs. Umpire Laboratory Analysis TiO_2

ASSAY Value	*Original*	*Duplicate*
Mean	61.25	61.56
Standard Error	1.64	1.67
Median	62.00	62.00
Mode	62.00	20.00
Standard Deviation	28.76	29.18
Sample Variance	827.19	851.67
Kurtosis	0.79	0.84
Skewness	0.50	0.49
Range	176.00	184.00
Minimum	8.00	8.00
Maximum	184.00	192.00
Sum	18805.00	18898.00
Count	307	307
Confidence Level(95.	3.23	3.28

% Difference Mean **0.49%**



Percentile	Original	Duplicate
0.0%	8.00	8.00
5.0%	18.00	17.00
10.0%	22.00	21.60
15.0%	27.00	26.00
20.0%	33.20	33.00
25.0%	39.00	39.00
30.0%	45.80	46.00
35.0%	51.00	51.00
40.0%	55.00	56.40
45.0%	58.70	60.00
50.0%	62.00	62.00
55.0%	66.00	66.00
60.0%	69.00	69.00
65.0%	71.00	72.00
70.0%	74.00	76.00
75.0%	79.50	80.00
80.0%	84.00	84.80
85.0%	88.00	88.00
90.0%	97.00	97.80
95.0%	106.70	107.70
97.5%	119.00	117.05
99.0%	138.52	140.70
100.0%	184.00	192.00



Source: SRK, 2015

Figure 11.5.5.5: XY Scatter and QQ Plot Showing Comparison of Original vs. Umpire Laboratory Analysis Sc

11.6 Specific Gravity

NioCorp collected specific gravity (SG) measurements in 2011 and 2014 program, covering the spatial and temporal aspect of all drill campaigns and considering the various lithologies present. Two methodologies have been implemented, (1) water immersion specific gravity measurement and (2) volumetric dry density measurement. Initially only the water immersion measurements were taken but during the site inspection by SRK it was recommended that a volumetric wet and dry density measurements should also be taken, due to the porous or vuggy nature of some of the core causing possible errors in the water immersion method. The two methods used are described below:

Water immersion method determines the specific gravity by the following formula:

SG = (weight in air) / (weight in air – weight in water)

A 10 to 20 cm piece of whole, dry, HQ core was weighed dry on an Ohaus Scout Pro scale and the weight recorded. The weight in water is determined by attaching the core by a long nylon fishing line to the Ohaus balance, lowering the core piece into a large plastic tub located immediately below the scale and filled with purified water. The weight of the core while immersed is then recorded, and applied to the formula for determining the SG. Porous core samples of altered carbonatite cannot be accurately measured using this method and are better represented by using the dry density measurement.

Dry Volumetric method determines the Density by the following formula:

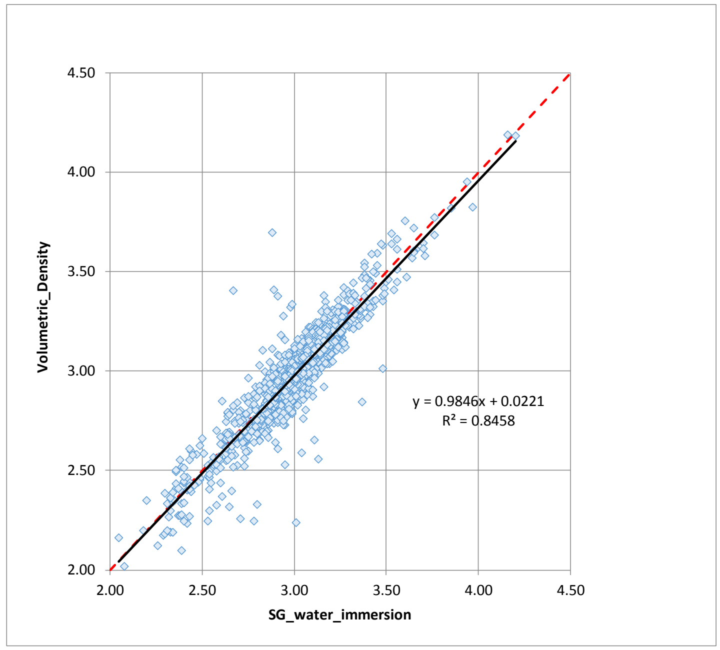
$$SG= [(\text{weight in air})]/ [(\pi)\ (\text{core length})\ (\text{core radius})^2]$$

A 10 to 15 cm piece of whole, HQ or PQ core, were dried in a convection oven for 60 minutes at 200°F. If the core still has moisture, it was left in the oven for a longer period of time. The exact length of the core was measured with a caliper and recorded. The sample was then weighed dry in air by suspending the core by a long nylon fishing line from an Ohaus Scout Pro balance and the weight of the core recorded. It is assumed the radius remains constant for each size of drill core: 31.75 mm for HQ and 41.50 mm for PQ. These measurements are applied to the formula for determining the SG. Calibration weights were occasionally used to verify the accuracy of the balance. The table used to complete the measurements is made of wood construction and tested for level by the technician.

A total of 1,225 samples have been analyzed using both method and a comparison between the two methods (Figure 11.6.1) shows that the water immersion method returns higher density values. A statistical analysis of the mean grades of the two populations where both methods have been recorded show a difference of approximately 1%. The correlation shown on the XY scatter indicates a strong correlation for the majority of cases, but for some samples there are significant differences with the volumetric density returning higher grades. This may be a result of voids, porous material.

SRK does not consider the difference to have a material impact.

	SG_water_immersion	*Volumetric_Density*
Mean	2.98	2.96
Standard Error	0.01	0.01
Median	2.98	2.96
Mode	2.97	2.69
Standard Deviation	0.24	0.26
Sample Variance	0.06	0.07
Kurtosis	2.36	1.83
Skewness	-0.04	-0.02
Range	2.15	2.17
Minimum	2.05	2.02
Maximum	4.20	4.19
Sum	3651.48	3622.14
Count	1225	1225



Source: SRK: 2015

Figure 11.6.1: Comparison of Density Measurements Using Volumetrics vs. Water Immersion Methods

11.7 Opinion on Adequacy

SRK comments that the decision for re-assay s of the SRMs/standards is based on a percentage and not the typical 2 x standard deviation, or 3 x standard deviation which is generally accepted as industry best practice. SRK has reviewed the original certificates for the SRMs submitted as part of the 2014 program and notes that no standard deviation is shown on the certificate. The limits have been requested from the supplier by Dahrouge but not supplied. The current method of using a ±5% and ±10% limit, while not ideal provides a reasonable level of confidence in the control samples, and the Company has addressed this issue by including an additional certified reference sample GRE-04, which provides certified standard deviations which form the basis of control lines.

SRK is of the opinion that these measures are consistent with or in excess of current industry best practices for projects at this scale of exploration.

12 Data Verification

The geological database has been provided to SRK by Dahrouge who have been involved with the Project since the Company acquired the Project, under its former name of Quantum. In addition to the digital database SRK has been provided access to historical copies of the data captured in scanned format for the drilling logs.

During the period of ownership by the Company a number of validation exercises have been completed on the database to provide a high level of confidence in the data available for the geological modelling and associated Mineral Resource Estimate.

The following Section provides a summary of the previous verification exercises completed by Dahrouge and Tetra Tech (as part of the previous NI 43-101 Technical Report), plus additional verification work completed by SRK as part of the current study.

12.1 Tetra Tech Data Verification, 2012

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

- The database consisted of 29 drillholes, totaling 18,159.15 m. Twenty-seven of the twenty-nine drillholes, totaling 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; however, these holes appear on the southwestern limit of the deposit and were only used in the interpretation of the deposit, not in the resource estimate;
- Quantum's 2010 to 2011 re-sampling data compared to historic values was less than 1% different in all cases except for one where the tolerance was less than 2%;
- REE assay values were not included in the 2012 Mineral Resource Estimate and therefore were excluded from the verification analysis;
- The results of the verification study found the following: During Quantum's 2011 drilling program, seven historic Molycorp drill collars were uncovered to survey the collar locations. The entries in the survey reports were recorded in imperial units (feet) and a factor of 0.3048 was used to convert the values into a metric system (meters). The database entries were verified against these converted values from the survey reports, and no inconsistencies were found;
- Completed verification of the digital database against the assay certificates for the three holes drilled within the Project (NEC11-001, NEC11-002, and NEC11-003), accounting for 1,195 of the total 6,078 samples, or 19.66% of the entire sample dataset., and no errors were observed;
- It was noted that when assaying yielded results below detection limit, half the detection limit (i.e. less than 0.003 Nb_2O_5%) was entered in the database for samples from 2011 drillholes, whereas a value of 0 was entered for all such occurrences in holes drilled prior to 2011. SRK has utilized a value of half the detection limit for all such occurrences;

- 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, however, entered in the database;
- Sixteen of the historic Molycorp drillholes were attributed with a negative azimuth value in the survey file. All drillholes 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 where logged in the field but were not transcribed into the database;
- All errors were correct by Tetra Tech prior to importing into technical software;
- The header, lithology, survey, and assays tables from the database were imported into Gemcom GEMS™ software, which has a routine that checks for duplicate intervals, overlapping intervals, and intervals beyond the length of the hole. No errors were identified; and
- Completion of a site inspection on February 8 to 9, 2012, which included the inspection of Quantum's 2011 drillhole locations and of the core logging, sampling and storage facility.

Independent check samples were collected during the site visit by Tetra Tech. Four ¼ 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 for niobium was conducted using XRF analysis;
- The Tetra Tech check sample analysis correlates well with Quantum's assay results for the same sample intervals in three of the four cases; and
- Tetra Tech concluded that the analytical results for Nb_2O_5% have been confirmed and that they are adequate for purposes of the 2012 Technical Report.

12.2 SRK Validation

SRK geologists under the guidance of Cody Bramwell and David MacDonnell were on site on a rotational basis during the 2014 drilling program conducted by NioCorp.

12.2.1 Site Inspection

In accordance with NI 43-101 guidelines, Martin Pittuck (Qualified Person) visited the Project between June 17 and 19, 2014. The main purpose of the site visit was to:

- Ascertain the geological and geographical setting of the Project;
- Witness the extent of the exploration work completed to date;
- Inspect the drilling rig(s);
- Review the sample preparation methodology;
- Inspect core logging and sample storage facilities;
- Discuss geological interpretation and inspect drill core; and
- Assess logistical aspects and other practicalities relating to the exploration property.

SRK was able to verify the quality of geological and sampling information and develop an interpretation of niobium grade distributions appropriate to use in the Mineral Resource model. A basic review of the electronic database against a number of drillhole intersections was also completed.

In addition to the site inspection by Martin Pittuck SRK has had a continual involvement reviewing data, interpretation and modelling outcomes.

12.2.2 Procedures

To verify the database SRK has looked at all aspects of the data collection. SRK checked the coordinates of all drillholes via handheld GPS for NioCorp 2014 drillholes. SRK notes that the drillholes are well-located and have been surveyed by an external company using high precision equipment.

Survey Information

During the review of the historical database a number of potential transcription errors between the historical locations and the captured information have been identified. This in part has been attributed to the collars co-ordinates being captured from detailed historical maps during the original data capture, which may potentially have had a different datum. Where possible historic collars (24 drillholes) were re-surveyed, but given the agricultural nature of the land were not always located at surface. Where this occurred, holes were re-located using metal detectors and dug out using a backhoe, and re-surveyed. The results showed a consistent shift between the historical collars used in the 2014 estimates. The difference in the UTM coordinates is consistently 4.7 m in the X coordinate and approximately 7.85 m in the Y coordinate (as shown in Table 12.2.2.1).

Table 12.2.2.1: Summary of Difference between DGPS vs. Digitized Collar Locations

BHID	Method	XCOLLAR_ DIFF-GPS	YCOLLAR_ DIFF-GPS	XCOLLAR_ MAP	YCOLLAR_ MAP	X_ Difference (m)	Y_ Difference (m)
EC-11	DIFF-GPS	739,604.1	4,461,131.2	739,599.4	4,461,139.1	-4.7	7.9
EC-14	DIFF-GPS	739,278.0	4,461,347.5	739,273.3	4,461,355.3	-4.7	7.8
EC-15	DIFF-GPS	739,054.2	4,461,307.6	739,049.5	4,461,315.4	-4.7	7.9
EC-16	DIFF-GPS	739,389.4	4,461,248.5	739,385.0	4,461,256.3	-4.4	7.8
EC-19	DIFF-GPS	739,552.7	4,461,301.9	739,548.0	4,461,309.8	-4.7	7.9
EC-20	DIFF-GPS	739,231.9	4,461,455.5	739,227.3	4,461,463.2	-4.6	7.7
EC-21	DIFF-GPS	739,547.0	4,461,304.5	739,542.3	4,461,312.4	-4.7	7.9
EC-22	DIFF-GPS	739,135.4	4,461,168.6	739,130.7	4,461,176.4	-4.7	7.9
EC-24	DIFF-GPS	739,162.1	4,461,249.3	739,157.4	4,461,257.2	-4.7	7.9
EC-25	DIFF-GPS	739,134.8	4,461,263.2	739,130.1	4,461,271.1	-4.7	7.9
EC-26	DIFF-GPS	739,176.3	4,461,276.0	739,171.7	4,461,283.9	-4.7	7.9
EC-27	DIFF-GPS	739,384.2	4,461,335.4	739,379.5	4,461,343.2	-4.7	7.9
EC-28	DIFF-GPS	739,145.6	4,461,363.4	739,141.0	4,461,370.9	-4.6	7.5
EC-29	DIFF-GPS	739,080.9	4,461,394.1	739,076.1	4,461,402.0	-4.8	7.9
EC-30	DIFF-GPS	739,487.3	4,461,158.0	739,482.2	4,461,166.5	-5.1	8.5
EC-31	DIFF-GPS	739,006.8	4,461,419.7	739,002.1	4,461,427.6	-4.7	7.9
EC-32	DIFF-GPS	739,087.8	4,461,330.8	739,083.1	4,461,338.6	-4.7	7.9
EC-33	DIFF-GPS	739,057.6	4,461,237.7	739,052.9	4,461,245.6	-4.7	7.9
EC-34	DIFF-GPS	738,998.8	4,461,297.8	738,994.1	4,461,305.6	-4.7	7.9
EC-35	DIFF-GPS	739,134.2	4,461,165.7	739,129.5	4,461,173.6	-4.7	7.9
EC-36	DIFF-GPS	739,069.2	4,461,344.2	739,064.5	4,461,352.1	-4.7	7.9
EC-37	DIFF-GPS	739,003.1	4,461,274.9	738,998.4	4,461,282.8	-4.7	7.9
EC-51	DIFF-GPS	738,942.9	4,461,234.7	738,938.2	4,461,242.6	-4.7	7.9
EC-54	DIFF-GPS	739,053.9	4,461,307.6	739,049.2	4,461,315.4	-4.7	7.9

Source: Dahrouge, 2014

Based on the investigation SRK has adjusted the collar locations accordingly to account for the higher confidence in the differential global positioning satellite (DGPS) measurements.

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 where by 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 latest database export provided to SRK 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 estimates this has been completed for approximately 10% of the assays within the 0.3% grade shell limit, and decreasing to <4% within the 0.4% grade shell. Based on a study of the mean grade using the original vs. the adjusted values the influence on the mean grade is negligible (<0.5%). As no defined reason for the adjustment has been noted, SRK has used the original data where no re-assays during the 2011/2012 verification program has been completed. SRK does not anticipate the use of the factored or unfactored historical assays will have a material impact on the current Mineral Resource Estimate. To ensure best practice and sufficient QA/QC is completed on the database SRK would recommend re-assaying the 10% of samples from the historical holes which lie within the 0.3% grade shell, where available in pulp or core is in sufficient quality to obtain a sample using the current QA/QC protocol. SRK understands that the Company has initiated this re-assay program at the time of writing.

Database Information

The database used for the resource estimate was constructed by Dahrouge and is stored in CAE Mining Fusion Database, and is considered to be of good quality. The use of a commercial database is considered industry best practice with the following key advantages:

- The system facilitates fast and accurate data collection and can be configured (via pick-lists) to meet all specific data schemes and logging standards relevant to each site;
- Drillhole related data can be recorded directly at the worksite on a touch-screen tablet or a notebook computer;
- Data is stored locally and synchronized to a single central database for the Project via a network connection. Transfer to and from the Central Database provides an audit trail for any edits made to the database;
- The QA/QC system allows users to achieve immediate data validation as information is captured. Only valid field values and labels are accepted, ensuring consistent logging standards are applied across multiple staff or sites;
- Importing laboratory results can be done directly to avoid potential transcription errors. The import function can proactively detect problems with analytical results; and

- Export routines can be created to provide the required data for use in technical software in a consistent format.

SRK has been supplied with exports from the database covering, collar, survey, lithology, assay, alteration and key structural indicators. SRK has used importing routines within Datamine Mining Software (Datamine) and Leapfrog®. During the importing routine the following errors have been noted:

- Assay values in the Molycorp database where Nb_2O_5% values are set to zero. These are assumed by SRK to represent values below detection limits and so SRK reset the values to half of the respective half detection limit;
- A search for sample overlaps or significant gaps in the interval tables, duplicate samples, errors in the length field, anomalous assay and survey results has been completed. No material issues were noted in the final sample database;
- The original signed electronic copies of the laboratory certificates were also spot verified for selected holes in the final electronic assay database and no errors were found; and
- Within the multi-element database a number of cases exist in the Molycorp assays have yet to be re-assayed for TiO_2 or Sc, as they were not included in the 2010 verification program. SRK has assumed that this is due to the original samples not being located for re-submission. SRK has ignored all cases where this occurs and inserted a default "NNS" for use in Leapfrog® during the geological modelling process.

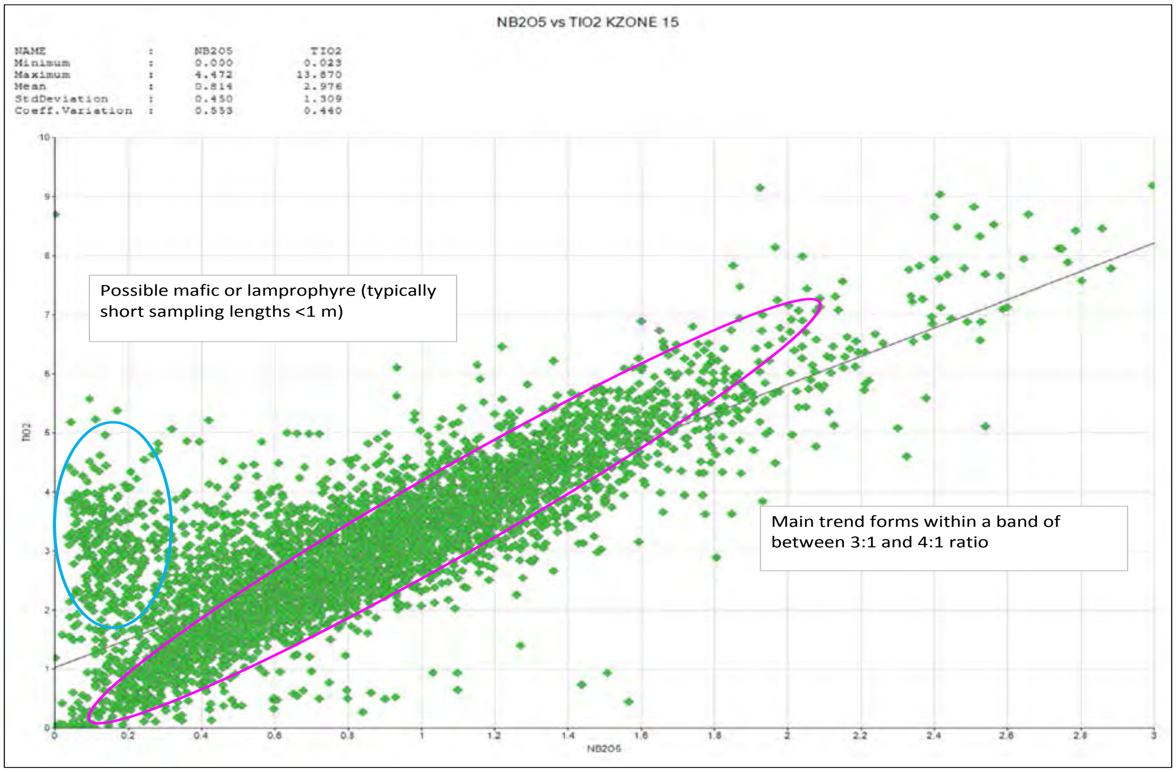
Absent TiO_2 and Sc assays

In total 6.0% and 7.1% of the Nb_2O_5 assays within the mineralized wireframes contain absent values for TiO_2 and Sc respectively. The average Nb_2O_5 grade for the absent values is approximately 0.3% Nb_2O_5. These samples were not included within the 2010 re-assay program, and therefore no pulp material was available for inclusion in the 2014 assay program. SRK has investigated alternative methods of how the absent values should be treated within the database.

Within the geological wireframes where multi-element data was absent, SRK has completed a regression analysis for absent TiO_2 and Sc values in the database. The sample regression was established by plotting XY Scatter charts of each element vs. the Nb_2O_5. SRK notes a very strong positive correlation between Nb_2O_5 and TiO_2 although a portion of the population where the TiO_2 grades are elevated shows a lesser corresponding increase in the Nb_2O_5 (Figure 12.2.2.1). In SRK's opinion, these may relate to more Lamprophyric material which tends to have lower Nb_2O_5 grades. Based on the analysis SRK elected to use an equation to derive missing values for TiO_2:

$$IF\ (TiO_2 == absent())\ TiO_2 = Nb_2O_5 \times 3.5$$

In addition to the assigned values SRK has also flagged the database with an indicator for quality where true assays equal 1 and assigned values equal 0. This allows SRK to review the quality of the original sampling data during the classification process.



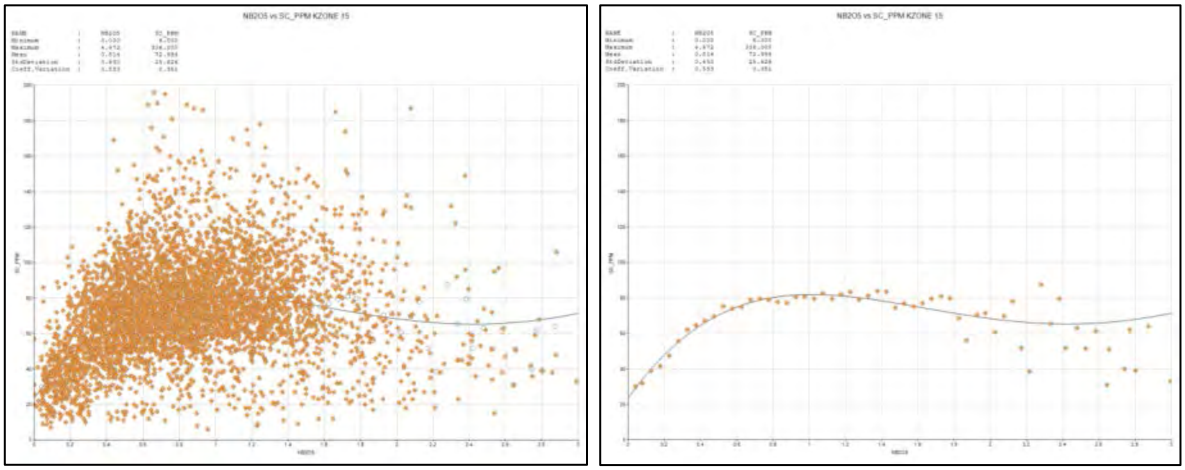
Source: SRK, 2015

Figure 12.2.2.1: XY Scatter Showing Relationship Between TiO_2 and Nb_2O_5

In contrast there is no linear trend which can be established between the Sc and Nb_2O_5 grades (left Figure 12.2.2.2). To establish a relationship SRK has created an average for the Sc values based on 0.05% interval bands of Nb_2O_5. The resultant chart shown on the right in Figure 12.2.2.2 shows a more defined model which has been used to assign values with missing Sc assays. To model the trend SRK has used linear trends at lower grades and then capped the Sc values above a given threshold. An example of the criteria used for estimation domain (KZONE) 15 is shown below:

- *IF(KZONE==15 AND SC_PPM==absent() AND Nb_2O_5<0.55)*
    - *SC_PPM=(Nb_2O_5*110)+20 END*
- *IF(KZONE==15 AND SC_PPM==absent() AND Nb_2O_5>=0.55)*
    - *SC_PPM=80 END*

The equations have been optimized per estimation domain with validation being completed by testing actual raw values to ensure the results remain within the plotted population. Additional checks have been completed to confirm the Nb_2O_5 population for the absent Sc values remains consistent with the sample population within Nb_2O_5 and Sc values, via histograms.



Source: SRK, 2015

Figure 12.2.2.2: Analysis of Sc vs. Nb_2O_5 grades within KZONE 15

To test the potential impact on the resultant Mineral Resource Estimate SRK completed three estimates as follows:

- Missing values reset to the detection limit and assumed to be waste;
- Missing values ignored and hence estimates rely only on neighboring values; and
- Missing values assigned based on Nb_2O_5 assays using regression formulae given above.

SRK compared the results using visual analysis via key cross-sections (10 in total per element), plus production of a global grade tonnage curve (not classified) for each scenario. The results were tabulated and compared to assess the level of risk in using each scenario.

Table 12.2.2.2: Summary of Analysis for Selection of Treatment for Absent TiO_2 and Sc Assays

Cut-Off	Tonnes	Detection Limit Assigned			Ignored Absent Values			Regression Analysis		
Nb_2O_5	(000's t)	Nb_2O_5	TiO_2	SC_PPM	Nb_2O_5	TiO_2	SC_PPM	Nb_2O_5	TiO_2	SC_PPM
0.00	1,271,000	0.10	0.38	10.4	0.10	0.43	12.1	0.10	0.41	12.2
0.30	180,000	0.63	2.37	59.5	0.63	2.55	66.8	0.63	2.48	67.0
0.40	126,000	0.75	2.83	67.6	0.75	2.86	70.3	0.75	2.85	70.5
0.50	118,000	0.77	2.91	68.5	0.77	2.92	70.6	0.77	2.91	70.8
0.70	75,000	0.86	3.01	69.0	0.86	3.02	71.2	0.86	3.02	71.4

Source: SRK, 2015

SRK preference has been to use the regression methodology in the current estimate as it is known from reviewing the 2014 assays that relationships exist between both the TiO_2 and Sc values with the Nb_2O_5 mineralization. Therefore by ignoring assigning a value of half detection limit would result in an underestimate, and ignoring the samples could potentially overstate the grade.

In comparison for the regression analysis the increase in the mean grade from 2.37% to 2.48% for TiO_2, and 59.5 ppm to 67 ppm for Sc at a cut-off of 0.3% Nb_2O_5. SRK considers these differences to be within acceptable levels of error, with the error reducing further at higher cut-offs. The reduction in the differences at higher cut-offs is due to samples having already been sent for either primary or reanalysis to obtain TiO_2 and Sc values.

SRK would recommend that the historical database which has not been submitted for reanalysis previously, be sent for TiO_2 and Sc assays to confirm the numbers used in the current estimate. SRK understands that the Company have put a program in place to locate and re-sample the historical pulps, which should be completed prior to any future updated Mineral Resources. SRK does not consider that this will make a material impact, but having absent assays within the geological wireframe is not considered industry best practice. Work programs will be required to increase the level of confidence for assay database further with the focus on two main areas:

- Assaying values which have not currently been assayed for TiO_2 and Sc which fall within or in close proximity to the current geological/mineralization wireframes; and
- Conducting a QA/QC program which includes submission of a low, medium and high grade TiO2 and Sc SRM (if one can be purchased), along with the submission of a range of grades from returned pulps to the primary laboratory. The aim of this exercise would be to confirm the accuracy of the laboratory as the precision is well established from the duplicate program.

12.2.3 Limitations

SRK 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.

The 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.

12.3 Opinion on Data Adequacy

SRK is of the opinion that appropriate scientific methods and best professional judgment were utilized in the collection and interpretation of the data used in this report. However, users of this report are cautioned that the evaluation methods employed herein are subject to inherent uncertainties.

In summary, SRK has accepted the sample database as provided by the Company and concludes that the data is sufficiently reliable to support Mineral Resource Estimation. SRK recommends that the issues raised previously between the umpire laboratory checks should be further reviewed. SRK would consider the work programs laid out above, in conjunction with further infill drilling will be required to gain confidence in the database to possibly delineate a Measured Mineral Resource.

13 Mineral Processing and Metallurgical Testing

Metallurgical testwork has been carried out in order to properly design processing facilities at a PEA level. In 2014 and 2015, comminution, magnetic and gravity separation, and flotation testwork was performed at multiple laboratories including Hazen Research Inc. (Hazen) in 2014, SGS Canada Inc. (SGS) in 2014 and early 2015, and at Eriez Flotation Division (Eriez) in 2014 and early 2015. During the first half of 2015, column flotation pilot plant testing was carried out at COREM.

The results of the flotation pilot testwork obtained at COREM demonstrated acceptable metallurgical recoveries but higher than targeted mass pull and thus, direct leaching of the ground mineralized material without a flotation circuit was selected as the most favorable process for treating the Elk Creek mineralized material.

The metallurgical, hydrometallurgical and pyrometallurgical testwork performed is described in Sections 13.1, 13.2, and 13.3, respectively.

13.1 Processing Plant

13.1.1 Metallurgical Testwork Summary

Samples

Metallurgical composite samples were selected throughout the 2014 core drilling program, to provide "representative" feed to the testwork program. As the project progressed and additional drilling was completed on the defined resource, the composite selection was extended vertically and laterally to encompass the proposed mine plan. Composite samples for metallurgical testing were designed to target the primary niobium-enriched rock unit, magnetite-dolomite carbonatite, using collected geological details, multi-element geochemistry, and the strongly correlated Nb_2O_5 analytical results. All composite samples, except "Nb Comp" and COMP-1 to COMP-5 identified below in Table 13.1.1.1, were continuously selected across targeted drillhole intervals, with the inclusion of internal dilution (low grade or waste rock) zones equal to or less than 3 m continuous interval length. Composite intervals selected for Nb Comp did not use a continuous interval selection and excluded the minor zones of internal dilution, potentially reducing the representativeness of the deposit. COMP-1 to COMP-5 were each selected by different rock type and depth within the deposit. The general composite details are summarized in Table 13.1.1.1 with their selection criteria defined in Table 13.1.1.2.

Table 13.1.1.1: General Composite Sample Details

Lab Comp-ID	Test Lab	Material Comp-ID	Testwork	Date Shipped	Material Source	Source Drillholes	Material Type	Quantity (kg)
Nb Comp	SGS	SGS-2014-03-24_QTR-COMP	Bench Test	2014-03-24	2011 Drill Core	NEC11-002 & NEC11-003	1/4-HQ Core	318
Nb 2nd Drill Core Comp	SGS	SGS-2014-07-15_QTR-COMP	Bench Test	2014-07-15	2014 Drill Core	NEC14-006	1/4-HQ Core	250
Mini-Pilot Comp	SGS	SGS-2014-08-20_REJ-COMP	Mini-Pilot Plant	2014-08-20	2014 Drill Core	NEC14-006 & NEC14-008	Coarse-Crush Assay Rejects	1895
18805 (SAN#)	Eriez	Eriez-2014-11-05_REJ-Comp	Column Testing	2014-11-05	2014 Drill Core	NEC14-013 & NEC14-015	Coarse-Crush Assay Rejects	340
Pilot Plant	Corem	Corem_2015-02-11_PilotPlant-COMPS (COMP-1 to COMP-5)	Column Testing	2015-02-11	2014 Drill Core	NEC-MET-01, NEC-MET-02, & NEC-MET-03	Full PQ Core	25,000

Source: Dahrouge, 2015

Table 13.1.1.2: Composite Sample Selection Criteria

Lab Comp-ID	Source Drillholes (Depth)	Targeted Area	Selection Criteria
Nb Comp	NEC11-002 (713 to 868 m) NEC11-003 (359 to 412 m)	Southern & central Resource area- multiple drillhole locations along deposit trend	• Targeted rock unit: magnetite-dolomite carbonatite • Selected from limited material availability (2011 drill core only) • Selection based off preliminary deposit understanding and Nb_2O_5 resource classifications • Composited sample intervals were alternated with material selected for Hazen in an attempt to provide representative material for two test labs • Alternating interval selection may have reduced mineralogical and textural representativeness
Nb 2nd Drill Core Comp	NEC14-006 (309 to 436 m)	Central Resource area – single drillhole location	• Targeted rock unit: magnetite-dolomite carbonatite • Selected to extend test area to central deposit area, north along trend of previous compositing: "Nb Comp" • Selection combines textural variations into a single composite • Extracted continuous quartered core material across the geologically defined unit
Mini-Pilot Comp	NEC14-006 (309 to 436 m) NEC14-008 (439 to 886 m)	Central Resource area – multiple drillhole locations across deposit trend	• Targeted rock unit: magnetite-dolomite carbonatite • Selected to represent material provided for Nb 2nd Drill Core Comp and external Hazen testwork • Selection combines textural variations into a single composite • Continuous intervals of coarse-crushed (coarse-reject) material representative of Nb 2nd Drill Core Comp was extracted for compositing
18805 (SAN#)	NEC14-013 (695 to 860 m) NEC14-015 (652 to 731 m)	North-central & central Resource Area- multiple drillhole locations along deposit trend	• Targeted rock unit: magnetite-dolomite carbonatite • Selected to extend test area to include material extracted north along trend of previous compositing, "Nb Comp" and "Nb 2nd Drill Core Comp" • Selection combines textural variations into single composite • Continuous intervals of coarse-crushed (coarse-reject) material was extracted for compositing
COMP-1	NEC14-MET-01, NEC14-MET-03	Combined low Grade Material	• Dolomite Carbonatite, background Nb grade
COMP-2	NEC14-MET-01, NEC14-MET-02, NEC14-MET-03	Material below 650 m depth	• Mixture of massive and porphyroclastic (1a/1b) magnetite-dolomite Carbonatite below 650 m
COMP-3	NEC14-MET-01, NEC14-MET-02, NEC14-MET-03	Material above 650 m depth	• Massive magnetite-dolomite Carbonatite (1a) above 650 m
COMP-4	NEC14-MET-01, NEC14-MET-02, NEC14-MET-03	Material above 650 m depth	• Porphyroclastic magnetite-dolomite Carbonatite (1b) above 650 m
COMP-5	NEC14-MET-01	Diluted Brecciated Zone	• Magnetite-dolomite Carbonatite Breccia (1c) with variable amounts intermixed Lamprophyre and/or dolomite Carbonatite. Commonly magnetite-dolomite Carbonatite Clasts within a Nb-diluted groundmass

Source: Dahrouge, 2015

Selected composites were extracted and shipped to SGS, Eriez, and COREM, directly from the Project site for all quartered HQ-core material, or from the certified analytical laboratory for all coarse-crush (coarse-reject) material. Shipments were completed using bonded trucking companies and recorded as received in good order at their final destination. SGS received three separate composite shipments, including Nb Comp (SGS-2014-03-24_QTR-COMP), Nb 2nd Drill Core Comp (SGS-2014-07-15_QTR-COMP), and Mini-Pilot Comp (SGS-2014-08-20_REJ-COMP). Eriez received a single composite shipment, SAN# 18805 (Eriez-2014-11-05_REJ-Comp). COREM received one shipment that included five composite samples, COMP-1 to COMP-5. Composite preparation and homogenization was completed independently at SGS and Eriez preparation facilities.

Mineralogy[1]

The two composite samples used for metallurgical testing at SGS, "*Nb Comp"* and "*2nd Nb Drill Core Comp"*, were submitted for quantitative analysis using QEMSCAN once received on-site at the SGS facility in Lakefield, Ontario ("Lakefield"). The following is a summary of observations:

- Both samples were stage-ground to K_{80} of 106 µm. The analysis was conducted on three size fractions (+106 µm, -106/+25 µm, and -25 µm).
- The Nb Comp and the 2nd Nb Drill Core Head consist of similar mineralogy although minor differences are recorded:
    - Carbonates 51% and 62%, respectively, Fe oxides (10% and 5%), quartz/feldspars (7% and 11%), biotite (3% and 4%), Fe-(Ti)-oxides (2%), barite (15% to 10%), sulfides (2% to 1%), and trace amounts of other minerals. Note that both magnetite and hematite were identified with XRD analysis.
- The D_{50} (50% passing value from the cumulative grain size distribution) shows that the niobium minerals are <20 µm in size.
- Pyrochlore carries most of the Nb (78% to 82%), followed by Nb-rutile (11% to 12%), and ilmenorutile (10% to 6%).
- Free and liberated pyrochlore accounts for 23% and 37% in the Nb Comp and 2nd Nb drill core head, respectively. Pyrochlore liberation is significant at 48% and 57% in the -25 µm fraction of the Nb Comp and 2nd Nb drill core head, respectively. This indicates the need for fine grinding to liberate the niobium minerals if a physical separation is used in the process flowsheet.
- Exposure of niobium minerals greater than 30% exposure is 39% and 48% in the Nb Comp and 2nd Nb drill core head.

1 *This section is taken from the report "Process Development Metallurgical Testing on Samples from the Elk Creek Deposit, Project 14379-002" prepared by SGS Canada Inc., dated April 10, 2015. Standardizations have been made to match the format of this report.*

Comminution

Comminution testwork performed at SGS Canada Inc. (SGS) indicated that the mineralized material is considered relatively hard giving a Bond Ball Mill Work Index of 14.5 kWh/t and not very abrasive giving an abrasion index of 0.066.

Flotation Testwork

SGS

During the second half of 2014 and early 2015, SGS carried out a flotation testwork campaign using drill cores samples. Over 125 flotation tests were performed during their developmental testwork program, most of which consisted of direct pyrochlore flotation using 2 to 4 kg of feed ground at 100% passing (P_{100}) of between 20 and 104 µm. Their program consisted of collector screening and the evaluation of various reagents, reagent dosages, grind sizes, and operation parameters. Their best tests produced flotation concentrates with over 2.3% Nb_2O_5, recoveries over 70% Nb_2O_5, and with weight recoveries of between 15% and 20%.

In October 2014, SGS ran a mini-pilot mechanical flotation plant in order to generate a large quantity of feed for hydrometallurgical testwork. Prior to flotation, the feed was ground using two ball mills followed by Low Intensity Magnetic Separators (LIMS) in series. The flotation circuit consisted of five rougher stages and four cleaner stages. In all, the mini pilot-plant processed 1,100 kg of feed over 30 hours, and generated a concentrate with 3.33% Nb_2O_5, 55.7% Nb_2O_5 recovery, in 10.2% mass pull.

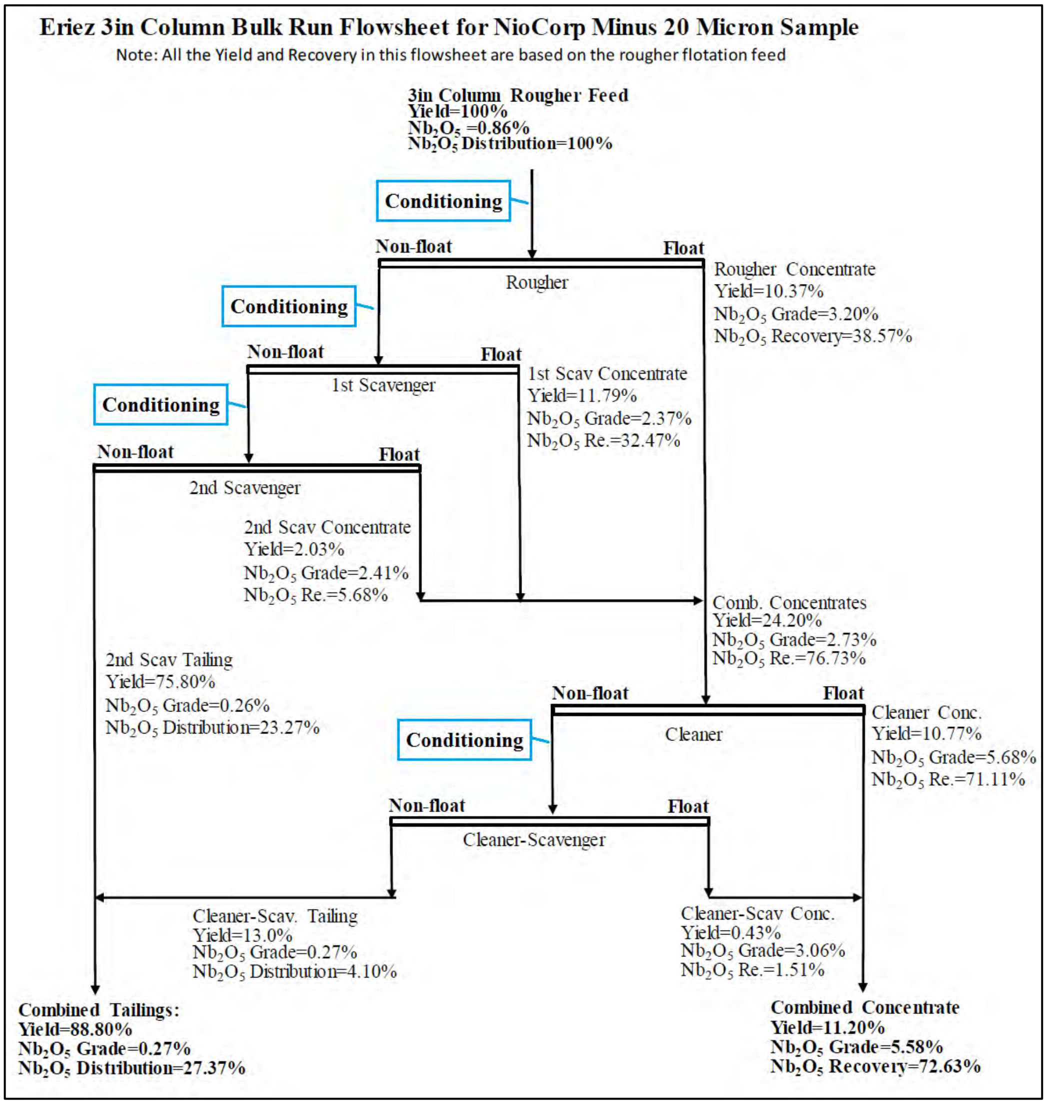
Due to the liberation requirement of the mineralized material, the best tests results were achieved with flotation feed ground to 100% passing 37 µm (equivalent to 80% passing 20 µm). Mechanical Flotation of fine materials usually causes a lot of entrainment, thus the option of using flotation columns without desliming was considered.

In most tests performed during the testwork campaign, the titanium recovery followed the same trend as the niobium recovery, and scandium recovery was approximately equal to the mass pull.

Eriez

In December 2014 and January 2015, a series of laboratory flotation tests were completed by Eriez. Promising results were achieved on a feed that was 100% passing 37 µm and 80% passing 20 µm. Several column flotation tests were performed including: ten rougher column tests, one column bulk rougher run under the optimized flotation conditions, a single 1st scavenger bulk run test on the rougher tail, a single 2nd scavenger bulk run test on the 1st scavenger tail, six cleaner tests on the combined concentrates of the rougher, 1st and 2nd scavenger, and four scavenger column tests on the cleaner tails.

All the test results showed that column flotation, with the use of wash water, provided superior results to those achieved using conventional flotation techniques conducted without froth washing. Under the optimized flotation conditions a rougher-scavenger-scavenger arrangement complete with a cleaner and cleaner scavenger step achieved a final combined concentrate of 5.6% Nb_2O_5 at a mass yield of 11.2%, and an Nb_2O_5 recovery of 72.6%. Final combined concentrate showed a TiO_2 grade of 21.4% with 77.6% recovery (feed grade of 3.1%). Scandium was found to follow the mass pull of the flotation, yielding approximately 11% recovery. The conceptual flowsheet with average bulk run results can be seen in Figure 13.1.1.1.



Source: Eriez, 2015

Figure 13.1.1.1: Column Flotation Flowsheet Developed at Eriez Using 3 Inch Column

COREM

During the first half of 2015, COREM performed pilot plant testwork to validate the flowsheet, developed by Eriez. Column flotation tests were performed on five representative composites. The flotation feed was ground to 100% passing (P_{100}) of 38 microns using a ball mill.

Continuous conditioning and continuous flotation required an intensive program to replicate Eriez results. Several pilot scale column flotations tests were performed, including various adjustments to reagents, air rates, feed rates, froth depth, and varying the number of columns. After an intensive campaign, COREM was able to replicate the results of Eriez's rougher and scavenger flotation steps. Unfortunately the cleaning flotation stages did not provide the desired metallurgical results in terms of mass pull versus recovery. A great deal more effort and time would have been required to achieve comparable results to the batch flotation steps achieved by Eriez.

Considering the poor mass pull performance achieved during column flotation pilot plant tests and given the prospect of dramatically increased niobium, titanium, and scandium recoveries by performing a whole ore leach on a much coarser ground material, it was decided not to pursue the flotation mineral processing beneficiation route. Therefore, mineral processing for the Elk Creek project has been limited to material handling and comminution prior to hydrometallurgical processing.

13.1.2 Process Selection

The first step in mineral processing consists of crushing the ROM to meet the required particle size for the grinding circuit feed. Grinding will be performed in a single stage by a SAG mill to achieve a P_{80} of 1100 µm, the target feed size for the hydromet plant. In order to increase the solids density of the feed to the hydromet plant, the grinding circuit discharge will be fed to a thickener. The thickener overflow will be recycled to the SAG mill feed.

13.1.3 Future Metallurgical Testwork

Further comminution testwork and grinding simulations are currently ongoing at SGS as a requirement for the feasibility study.

Comminution testwork has been performed on representative samples and included standard Bond indices, Bond Low-Energy Impact tests, JK Drop-weight tests, and SMC tests with abrasion indices. The tests were conducted on thirteen individual samples and six composites covering the potential variability of the grinding characteristics of the deposit

The grinding simulations to be finalized during the feasibility study will provide sizing data for the grinding mill(s), as well as the material balance projection (flow rate, water rate, % solids, particle size distribution, etc.), which can be used to confirm the sizing of other equipment such as pumps, water supply equipment, screens, crusher and cyclones. The predicted grinding circuit performance (power draw, specific power consumption (kWh/t) and operating work index) will be presented in each simulation report.

Samples will be collected for settling tests, paste backfill testwork and environmental characterization. Settling and rheology tests will have to be performed..

13.2 Hydrometallurgical Plant

13.2.1 Metallurgical Testwork Summary

Introduction

Metallurgical testwork were first conducted at SGS Canada Inc. (SGS) throughout 2014 and 2015 to properly design the required process units for the conversion of mineralized material into a niobium product suitable for further treatment into ferroniobium, as well as scandium and titanium products. Testwork consisted of an exploratory bench and pilot scale hydrometallurgical test program aimed at defining an appropriate flowsheet using different reagents and technologies. Upon further consideration of the recoveries and in particular the scandium recovery being very low in the flotation, leach test work was conducted on coarse whole ore material. A leach using an hydrochloric acid was introduced followed by the original sulfation. Coarse whole ore leach testwork showed that a high recovery of the scandium could be achieved without any added losses of titanium or niobium. A process flowsheet was then established based on testwork performed in leaching, purification, sulfation, and precipitation.

Samples

Samples were received at SGS Lakefield from the 2014 core drilling program and were used as feed material to test the feasibility of processing the whole ore within the hydrometallurgical process. A total of 800 kg of feed samples were processed by SGS Lakefield. For the PEA, 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 used as feed to the hydrometallurgical program. A summary of the combined feed material used in the testwork is given in in Table 13.2.1.1.

Table 13.2.1.1: Combined Whole Ore Feed Assay

Whole Ore Feed Assay (%)	
Si	4.78
Al	1.15
Fe	13.5
Mg	5.34
Ca	12.6
Na	0.31
K	1.21
Ti	1.97
P	0.33
Mn	0.51
Cr	0.01
V	0.03
Ba	4.16
Y (g/t)	181
Sc (g/t)	83
S	1.45
Nb	0.59
Th (g/t)	506
U (g/t)	52

Source: Roche / SGS, 2015

All samples were prepared by SGS Lakefield – the feed sample was crushed to specific particle size parameters. Each representative sample was sampled and analyzed to confirm the expected feed grade.

Leaching

Pre-Leach

There were a total of 13 hydrochloric acid pre-leach tests performed on the individual variability samples at the bench scale level. Using different hydrochloric acid concentrations and residence times, the leachability of the gangue material in the mineralized material was confirmed. The results supported compositing into one sample as there were little difference in HCl pre-leach results. An average weight reduction of 66% was achieved in the testwork. A summary of the design conditions and elemental extraction is found in Table 13.2.1.2.

Table 13.2.1.2: Pre-Leaching – Summary of Design Conditions and Elemental Extractions

Temperature	40	°C
Residence Time	4	H
Si	0	%
Al	26	%
Fe	64	%
Mg	95	%
Ca	98	%
Na	16	%
K	18	%
Ti	0	%
P	89	%
Mn	98	%
Ba	0	%
Sc	69	%
Sr	93	%
Nb	0	%

Source: Roche, 2015

Acid Regeneration

Synthetic solution and real pregnant leach solutions from the pre-leach testing were used in a series of acid regeneration tests, aimed at demonstrating the concept of hydrochloric acid regeneration and validating the theoretical mass balance calculations. Both the synthetic and real solution produced results in line with the theoretical calculations. Over 80 % of the consumed hydrochloric acid can be regenerated using sulfuric acid.

Acid Bake and Water Leach

The residues from the pre-leach testing were used in a series of acid bake tests, directed to extracting the niobium, titanium and remaining scandium after sulfation using sulfuric acid at high temperature in a kiln. Five acid bake tests were performed to confirm that the hydrochloric acid pre-leach residue would react similarly to the earlier sulfuric acid pre-leach residues. Twenty-four acid bake tests and seven strong acid agitated bake tests had previously been performed on sulfuric acid pre-leach residue to evaluate various acid doses, bake times, bake temperature and variation in feed materials. It was determined that the hydrochloric acid pre-leach residue reacted in a similar manner to the previous sulfuric acid pre-leach residues.

The resulting acid bake residues were contacted with water in a series of water leach tests, aimed at solubilizing the sulfated niobium, titanium and scandium. Five water leach tests were performed to confirm that the hydrochloric acid pre-leach residue, while being significantly (66%) reduced in mass, would react similarly to the previous sulfuric acid pre-leach residues. Previously, 24 water leach tests used the sulfuric acid pre-leached acid bake residues while seven more used the strong acid agitated bake slurries. These earlier tests looked into a selection of water doses, leach times, and temperature. A mini-pilot test was also operated on the sulfuric acid pre-leached residue produced in the sulfuric acid pre-leached acid bake pilot plant. A summary of the optimized conditions and elemental extraction for both the sulfuric acid and hydrochloric acid leach residues is shown in Table 13.2.1.3.

Table 13.2.1.3: Acid Bake and Water Leach – Extraction Results

Description	Sulfuric Acid Pre-leach Residue	Hydrochloric Acid Pre-leach Residue	Unit
AB Temperature	300	300	°C
AB Residence Time	4	4	H
AB Acid Ratio	1.5	1.5	t/t
WL Temperature	90	95	°C
WL Residence Time	2	3	H
WL Water Ratio	1.0	1.0	L/kg
Si	0	0	%
Al	23	34	%
Fe	99	100	%
Mg	97	100	%
Ca	95	100	%
Na	89	90	%
K	6	20	%
Ti	90	98	%
P	98	100	%
Mn	93	80	%
Ba	1	1	%
Sc	83	100	%
Nb	97	98	%

Source: Roche, 2015

<u>Reduction and Niobium Precipitation</u>

The water leach liquors were processed in a series of reduction tests using iron, aluminum, and sulfur dioxide followed by niobium precipitation tests, aimed at producing a niobium precipitate with sufficient purity to be further treated into a ferroniobium product. Preliminary bench scale work showed that the titanium and iron content of the niobium concentrate resulting from the precipitation of the water leach liquor were too high to produce a concentrate suitable for ferroniobium production. A reduction step was then introduced and tests produced a niobium concentrate with much higher niobium content suitable for further processing into ferroniobium product. Fifty-nine niobium precipitation tests evaluated a selection of precipitation methods, water quantities, reaction times and temperature. A mini pilot test was also operated on the water leach liquor produced in the acid bake and water leach pilot plant. A summary of the optimized conditions and elemental concentration is provided in Table 13.2.1.4.

Table 13.2.1.4: Reduction and Niobium Precipitation – Niobium Precipitation Results

Temperature	100	°C
Residence Time	4	H
Si	0	%
Al	4	%
Fe	0	%
Mg	0	%
Ca	1	%
Na	0	%
K	1	%
Ti	4	%
P	35	%
Mn	0	%
Cr	0	%
V	0	%
Ba	0	%
Sc	0	%
S	0	%
Nb	95	%

Source: Roche, 2015

Caustic Leach – Phosphate Removal

The niobium precipitates were used in a series of caustic leach tests, aimed at developing a suitable process for reducing the phosphate concentration in the final niobium precipitate. Eleven caustic leach tests looked into a selection of NaOH solutions at various concentrations, temperatures and contact times. A summary of the test conditions and key results is presented in Table 13.2.1.5.

Table 13.2.1.5: Phosphate Removal - Summary Results

Temperature	50	°C
Residence Time	2	H
Si	92	%
Al	99	%
Fe	2	%
Mg	2	%
Ca	3	%
K	91	%
Ti	2	%
P	100	%
Mn	2	%
Cr	24	%
V	83	%
Ba	1	%
S	100	%
Nb	4	%

Source: Roche, 2015

Titanium Precipitation

The resulting filtrate liquors from the niobium precipitation reactions were used in a series of titanium precipitation (TiP) tests. Although few in numbers due to the small amount of liquor available, the titanium precipitation tests assessed the production of a titanium precipitate with sufficient purity to be further processed into a pigment grade TiO_2 product. Preliminary bench scale work showed that the process is feasible and produces a crude TiO_2 precipitate. Four titanium precipitation tests evaluated oxidizing agent doses, reaction times, and temperature. A summary of the optimized conditions and elemental recovery to precipitate is found in Table 13.2.1.6.

Table 13.2.1.6: Titanium Precipitation –Titanium Dioxide Precipitation Results

Temperature	100	°C
Residence Time	2	H
Si	0	%
Al	5.1	%
Fe	1.1	%
Mg	0	%
Ca	0	%
Na	0	%
K	0	%
Ti	98	%
P	0	%
Mn	0	%
Ba	0	%
Sc	0	%
Nb	97	%

Source: Roche, 2015

Scandium Extraction

The pre-leach liquors were treated in a series of scandium extraction (ScSx) tests, aimed at developing a suitable process for extracting scandium from pre-leach liquors and titanium precipitation filtrate. Using liquors from pre-leach tests, organics were contacted with fresh pre-leach liquor. The iron concentrations in the resulting aqueous phase were the same as that of the feed liquor suggesting that co-extraction of iron is minimal.

The loaded organics from the extraction tests were stripped with different strip solutions. These solutions ranged from acidic to basic with varying concentrations. Based on these tests, it appears that scandium is better stripped by alkaline reagents. Testwork shows that 58% of the scandium loaded can be stripped with a 150 g/L sodium carbonate solution in a single stage. This suggests that recoveries greater than 90% can be achieved with a small number of stages. Conversely, the thorium in the organic system is preferentially stripped under acidic conditions, leaving the majority of the scandium in the organic. This will allow for an acid scrubbing step removing thorium prior to stripping scandium from the loaded organic. Testwork shows that 50.5% of the thorium loaded can be stripped with a 2M HCl solution in a single stage while only stripping 1.3% of the scandium. This suggests that recoveries greater than 90% can be achieved with a small number of stages. A summary of the test conditions and key results is presented in Table 13.2.1.7.

Table 13.2.1.7: Scandium Extraction - Summary Results

Sc Loaded	92	%
Fe Loaded	0	%
Th Loaded	35	%

Source: Roche, 2015

13.2.2 Process Selection

The numerous tests performed have provided the basis for the selected process. The first step consists of a hydrochloric acid (HCl) pre-leach. The solid residue is sulfated using concentrated

H_2SO_4 in a calciner at atmospheric pressure and a temperature of 300°C while the filtrate is processed in a solvent extraction circuit where scandium is loaded onto an organic phase. Stripping of the scandium is selectively achieved using an acidic stripping step removing thorium first followed by a stripping step using a sodium carbonate solution solvent extraction stage to recover scandium. The spent liquor is reacted with concentrated sulfuric acid to regenerate the hydrochloric acid.

The resulting residue is then leached with water at a rate of 1 liter per kilogram of solids in a series of agitated tanks where the niobium is solubilized along with titanium, iron, and remaining scandium. The pregnant liquor is reacted with elemental iron to reduce all iron(III) present in the solution to iron(II) and a portion of the Ti(IV) to Ti(III). The solution is then cooled in an evaporative crystallizer to precipitate pure iron(II) sulfate. The resulting pregnant liquor is processed in a solvent extraction circuit where scandium is loaded onto an organic phase. Stripping of the scandium is selectively achieved using an acidic stripping step removing thorium first, followed by a stripping step using a sodium carbonate solution solvent extraction unit to recover scandium.

Niobium is then selectively precipitated by diluting the pregnant liquor into boiling water. This final step provides a niobium concentrate that is leached with NaOH to remove impurities. This NaOH leach provides a high quality niobium concentrate that is suitable for pyrometallurgical treatment into a ferroniobium product.

The titanium in the filtered liquor is precipitated by oxidization using heat and sparged air in a series of agitated tanks. The filtered liquor is neutralized and the solids are sent to tailings.

The sulfates recovered in the acid regeneration step and the iron sulfate recovered in the iron(II) sulfate precipitation step are calcined in order to recover the sulfur as a gas. The gas in then sent to an acid plant that regenerates the H_2SO_4 for recycle in the hydrochloric acid and acid bake steps.

13.2.3 Future Metallurgical Testwork

Additional pilot-scale testing of the flowsheet is currently planned at SGS to further define and test all aspects of the process. Bench scale tests and mini pilots will be run to provide the final basis for the pilot testing. The tests will be conducted on whole ore samples resulting from representative samples. The pilot plant will validate the robustness of the hydrometallurgical process with regards to the variability of the mineralized material. From the pilot plant, samples will be collected for settling and filtration tests, paste backfill testwork, and environmental characterization. Settling and filtration tests will be performed by equipment suppliers to confirm equipment sizing.

13.3 Pyrometallurgical Plant

13.3.1 Introduction

Key drivers in pyrometallurgical testwork on the niobium concentrates from the Hydrometallurgical plant are the contents of Nb_2O_5, TiO_2, and phosphorous. Significant hydrometallurgical upgrades to the composition of the Nb concentrate precipitate have been achieved, particularly with the addition of a caustic leach (NaOH) step to remove phosphorous. The P_2O_5 levels have been reduced to 0.2% and lower in the hydromet concentrate. Nb_2O_5 grades have been significantly increased up to 90%, with lower TiO_2 levels.

Given the above niobium concentrate from the caustic leach, the pyrometallurgical plant comprises only alumino-thermic reduction to produce a relatively clean FeNb alloy containing less than 0.1% P.

13.3.2 Metallurgical Testwork Summary

Preliminary pyrometallurgical testwork has been carried out at XPS Consulting and Testwork Services (XPS) in Sudbury, Ontario, Canada.[2] Four preliminary bench scale tests were performed; demonstrating the successful conversion of the niobium oxide in the niobium precipitate into niobium metal. These tests were performed on lower grade Nb_2O_5 feedstocks and the resulting alloy Nb grades were low.

Subsequently, further preliminary calcination and alumino-thermic reduction tests were completed at Kingston Process Metallurgy (KPM) in Kingston, Ontario, Canada[3]. Summary points taken from the work at KPM are:

- Successful alumino-thermic reduction of Nb concentrates to produce FeNb alloy was achieved, albeit from small sample masses. FeNb alloy metal was produced.
- Niobium recovery of 85% was measured, but higher Nb recovery of over 95% to the FeNb alloy is to be expected and is feasible based on literature review and existing FeNb operations.
- The alumino-thermic reduction smelting temperature is at 1,650°C, and is consistent with FeNb industrial operations.
- The slag system was determined for smelting the Nb Concentrate, as a CaO-Al_2O_3-TiO_2 system, with some minor fluxing with fluorspar (CaF_2). Operation without SiO_2 additions helps to produce a low Si FeNb product.

---

[2] XPS Consulting and Testwork Services (XPS) Memorandum to NioCorp, 24 April 2015
Calcination and Aluminothermic Reduction of NioCorp FeNb Concentrate
[3] Kingston Process Metallurgy (KPM)
Memoranda to NioCorp:
12 June 2015, Niobium smelting test plan – update.
12 June 2015, Update on material characterization of NioCorp product.
23 July 2015, Results from smelting and reduction tests.

- Almost all of the phosphorous in the concentrate reports to the alloy, under such alumino-thermic reducing conditions. Given the low P levels in the Nb concentrate feedstock, the % P in alloy is likely to be low at 0.1%.
- Hematite powder (Fe_2O_3) was successfully added as the iron source for the alumino-thermic reduction smelting of FeNb. This could lead to the potential economic opportunity of using the Fe_2O_3 precipitate product from the hydrometallurgical circuit as the source of iron for the reaction.
- No carbon was added to the reduction smelt process, as Fe-Nb-carbides would form, and compromise the quality of the FeNb alloy for sale.

13.3.3 Process Selection

Alumino-thermic reduction has been selected to convert the hydrometallurgical Nb Concentrate into FeNb alloy. This reduction is performed in a single FeNb furnace, to produce a saleable FeNb metal alloy.

Given the fine particulate nature of the hydrometallurgical Nb Concentrate after the caustic leach the dried Nb concentrate is pelletized to feed the furnace.

From the KPM work, to form the slag system above, furnace additives of lime and fluorspar were selected. These are added together with aluminum powder, hematite and/or iron powder.

13.3.4 Future Metallurgical Testwork

Further pyrometallurgical testwork is envisaged at KPM following ongoing hydrometallurgical development work at SGS. The recommended pyrometallurgical testwork would comprise:

- Larger scale smelt testwork, on a larger bulk sample of Nb Concentrate, after the caustic leach. A bulk sample of 3,000 g would be a target mass of feedstock to supply multiple tests.
- The larger scale smelt tests on 250 g samples, would confirm Nb, P, Si and Ti deportments to the FeNb alloy and slag.
- Testwork would also be done in these reduction tests to finalize the slag chemistry, with a view to optimize temperature, slag fluidity, metal-slag separation and operating costs.
- Pelletizing testwork should also be carried out to ensure competent pellets to feed the furnace feed preparation area. Possible use of a binder should also be tested.
- Full characterization and density testwork on Nb Concentrate feeds, pellets, FeNb alloy, and slags should be completed.

14 Mineral Resource Estimate

14.1 Introduction

This section describes the Mineral Resource estimation methodology and summarizes the key assumptions considered by SRK. In the opinion of SRK, 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 Project at the current level of sampling. **No additional sampling/assays have been completed since the previous updated mineral resource estimation, and therefore no updated estimate has been completed as part of this PEA. The Mineral Resource remains effective of date of the previous PEA effective April 28, 2015.**

The Mineral Resources have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines and are reported in accordance with NI 43-101. 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 Mineral Reserve.

Martin Pittuck is the Qualified Person (QP) responsible for the resource estimation methodology and the resource statement. Mr. Pittuck has been assisted by Ben Parsons, MAusIMM (CP) a Principal Consultant at SRK Consulting (U.S.) Inc., who has constructed the geologic model and completed the grade estimation under the close supervision and review of Martin Pittuck. Mr. Parsons has 14 years in geological model and resource estimation and has completed the estimation using a combination of Leapfrog®, for geological modelling and CAE Mining Datamine software (Datamine) for grade estimation and reporting.

Due to time constraints and assay turnaround for the multi-element and scandium assays at the Laboratory, the Company requested SRK produce an initial Mineral Resource Estimate for Nb_2O_5 only, which has been reviewed and updated based on the addition of TiO_2 and Sc upon receipt of the assays.

The methodology used for preparation of the Mineral Resource Statement was as follows:

- Database verification;
- Construction of Nb_2O_5 mineralization wireframe models;
- Definition of resource domains;
- Preparing of data for geostatistical analysis and variography (capping and compositing);
- Block modelling and grade interpolation;
- Resource classification and validation;
- Assessment of “reasonable prospects for economic extraction” and selection of appropriate CoG;
- Preparation of a Mineral Resource Statement for Nb_2O_5;
- Database verification of the multi-element assay database;
- Verification/validation of the defined wireframes to the TiO_2 and Sc database;
- Block modelling and grade interpolation;
- Resource classification and validation; and

- Preparation of a Mineral Resource Statement using Nb_2O_5, as the primary economic assumption for determining the CoG.

The effective date of the Mineral Resource Statement is April 28, 2015.

14.2 Drillhole Database

The drillhole database was constructed by Dahrouge from Molycorp data and raw data captured by Dahrouge during the 2011 and 2014, drilling campaigns. The information has been exported from the Central Database and provided to SRK in .csv format. SRK determined the data to be of good quality. The database provided 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. SRK has assigned appropriate codes for missing samples and no recovery for use during the modeling procedures.

The complete database which covers the entire NioCorp concession area contains information from 129 diamond-core drillholes totaling approximately 64,981 m of drilling. There are no obvious gaps in the naming sequence of drillholes. The maximum drillhole depth is 950.4 m and the average is 501.7 m. Focusing on Elk Creek a total of 48 holes have been completed (inclusive of one wedged hole) for a total of 33.908.7 m of drilling.. A summary of the holes by drilling phase (Company) is shown in Table 14.2.1.

Table 14.2.1: Summary of Drilling Database over the Deposit by Phase

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		**34**	**649.1**	**33,908.7**

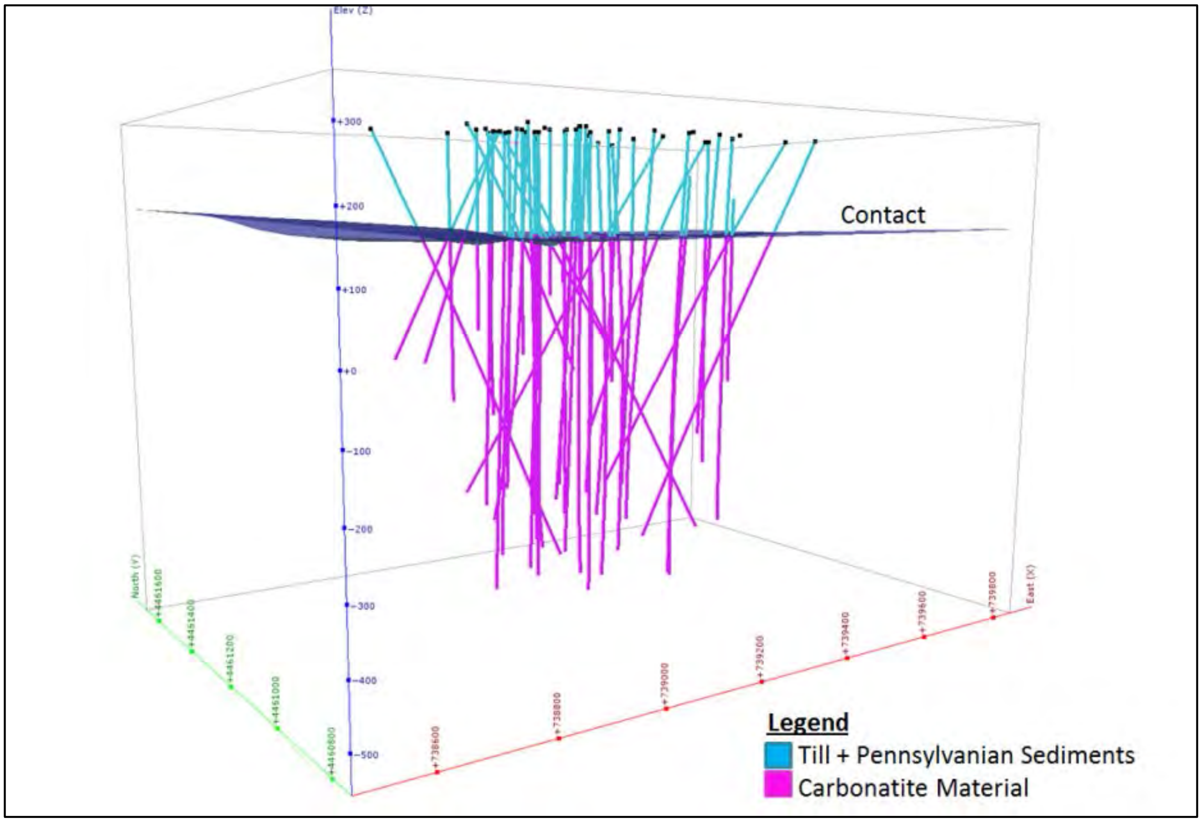
Source: SRK, 2015

14.3 Geologic Model

The drill log lithology data contains four major rock types based on the geologic observations of drill core, which based on the latest logging codes can be broken down into 19 sub-lithologies. The four main units considered during the analysis are:

- Overburden;
- Sediments;
- Carbonatite; and
- Mafic Units/Lamprophyre (low-grade domain).

The primary logging codes have been imported into Leapfrog® to create geological horizons for the base of overburden/till, plus the contact between the Pennsylvanian Sediments and the underlying Carbonatite (Figure 14.3.1). The models have been used by creating contact points within each drillhole at the contact between these major units.



Source: SRK, 2015

Figure 14.3.1: 3D View of Elk Creek Showing Modelled Base of Till and Unconformity between Pennsylvanian Sediments and the Elk Creek Carbonatite

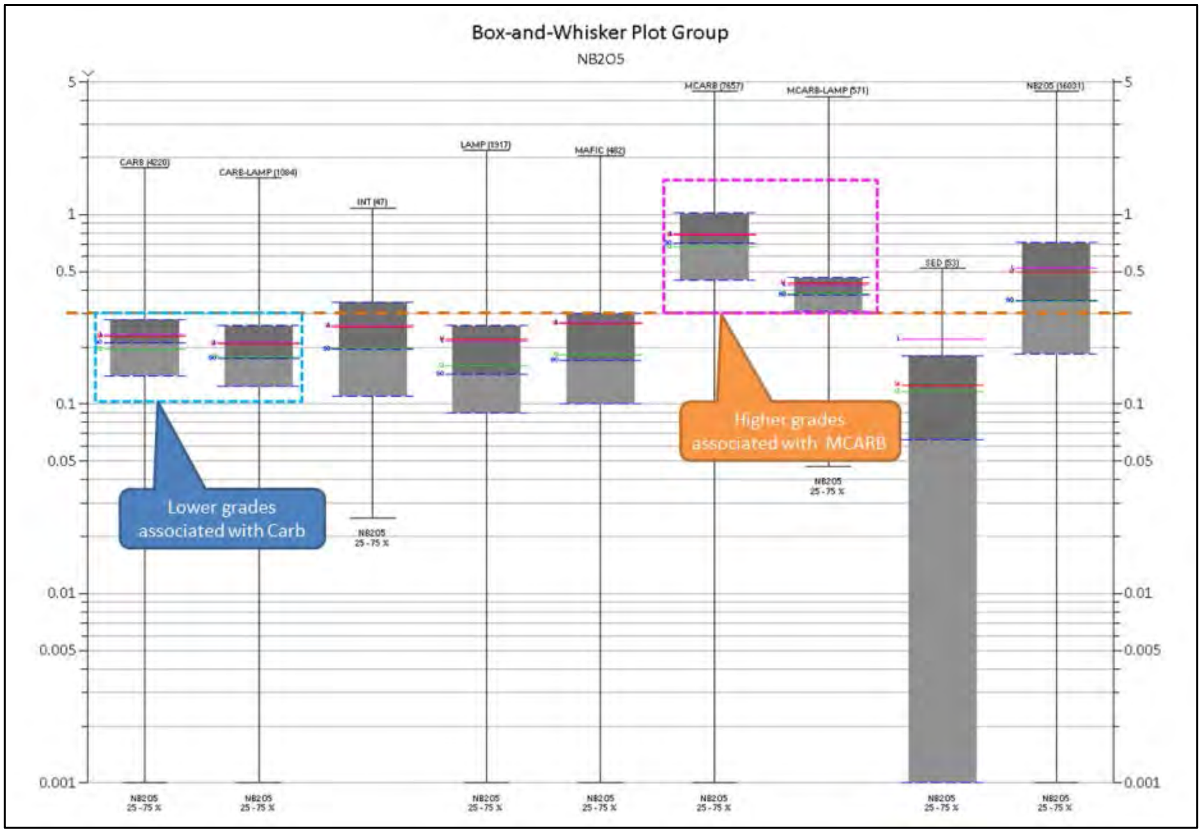
During the September 2014 geological model SRK did not consider the detail in the geological logs of the historical drilling to have sufficient detail to enable modelling of the geological units. Primarily this was due to a mixed population of higher grade magnetite-dolomite-carbonatite (mdolCarb), and lower grade dolomite-carbonatite (dolCarb), within material logged as dolCarb. As a result of this conclusion the decision was taken to review the historical information in addition to the collection of new drilling information, as discussed in Section 12.

Using the updated database SRK has completed a statistical analysis of the Nb_2O_5% grades per lithology using classical statistical methods. The database was then analyzed for relative abundance and Nb_2O_5 based on the main lithologies as shown in Figure 14.3.2. The box-whisker plot highlights the significant difference in the grade distributions between mCarb (pink square) and Carb (blue square). The MCARB and MCARB-LAMP account for 51% of the samples logged vs. 33% for the CARB and CARB-LAMP, which indicates these four codes cover 84% of the logged intervals. The other unit of significance in terms of logged intervals is the Lamporpyre units which accounts for 12%. The weighted average for all units is shown on the right of the chart.

The highest grades both in terms of values and mean grades, are found within the MCARB units with the next highest mean recorded in the MCARB-LAMP. The MCARB has grades in excess of 0.3% Nb_2O_5 for over 93% of the logged assays, while the MCARB-LAMP 79% of the logged values is greater than 0.3%. In comparison the CARB has 80% of the database below a nominal cut-off of

0.3% Nb_2O_5. The results confirm the importance of accurate geological logging and the improvement in geological domaining based on the relogging of the historical drilling.

In addition to the lower grades in the CARB units, SRK also noted lower grades within the MAFIC, INT and LAMP units. SRK has focused on trying to define these lower grade units via sectional analysis to domain these areas out of the geological model.



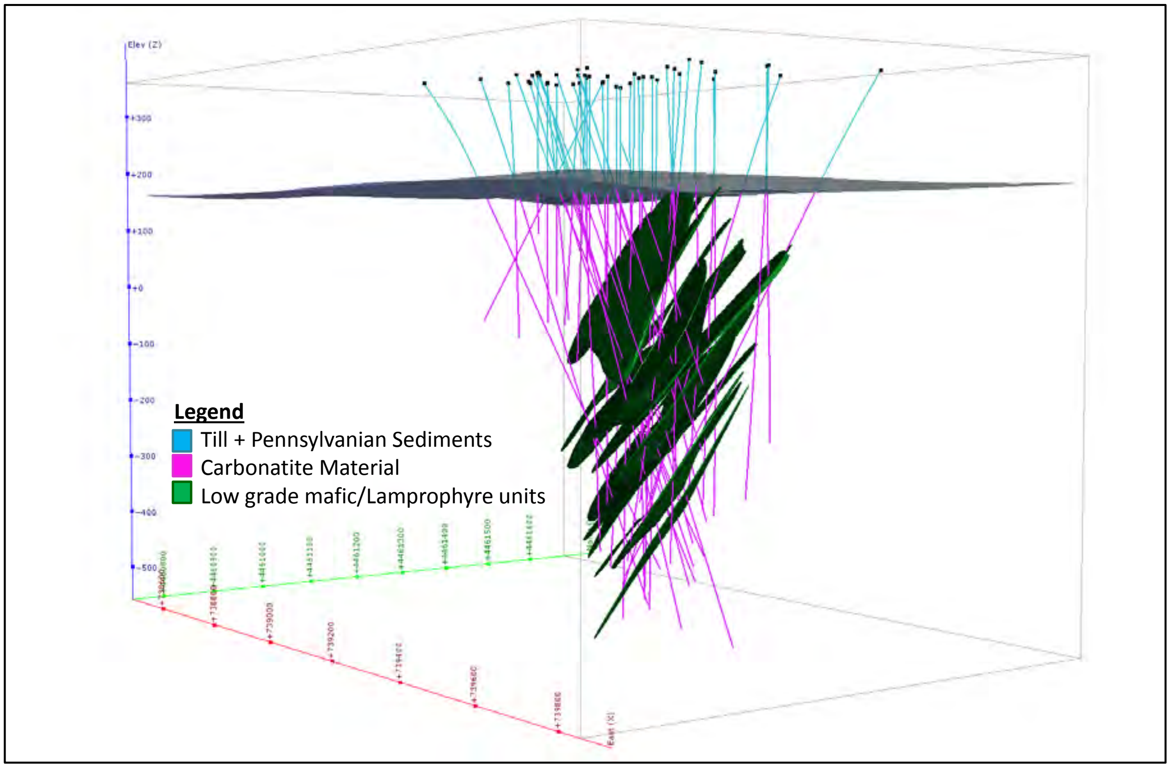
Source: SRK, 2015

Figure 14.3.2: Box Whisker Plot Showing Nb_2O_5 (%) Grades Split per Lithology

Historically within the historical database all early or late stage intrusives not defined as dolCarb have been assigned a mafic rock code. The NioCorp database shows a split between units logged as Mafic (considered to be late stage) which in general are low grade to barren, compared to Lamprohyre units which are shown to carry a mixture of low and high grades, which tend towards higher grades where units have been logged with a breccia texture. Given the nature of the rock-types and their similar properties, SRK consider there could remain a degree of mixing of data populations within these units. To improve the validation of the geological domaining for the low-grade a study of the multi-element database would increase the confidence required. SRK does not consider this to materially impact on the current geological model, but could provide additional confidence when looking to define Measured Mineral Resources.

SRK considers the presence of potentially late stage low/barren mafic units to be important. To be able to understand the distribution of Nb_2O_5 within the CARB and MCARB units SRK has first modelled the mafic units (Figure 14.3.3). To complete the model SRK has primarily used the lithology logging but has also used for guidance areas of low-grade to Nb_2O_5.and the overall trend of the mineralization. SRK notes that while the low-grade domain remains relatively easy to identify within cross-sections, the ability to link the structures between sections is difficult. In the September 2014 model SRK modelled a total of 6 units, but in comparison a total of 14 units have been modelled in the current update, with the strike length ranging from 150 to 650 m.

The resultant geological features have been imported into Datamine with the associated boreholes coded by the relevant geological units. The coding allows these units to be filtered from the geological modelling of the Carbonatite units.

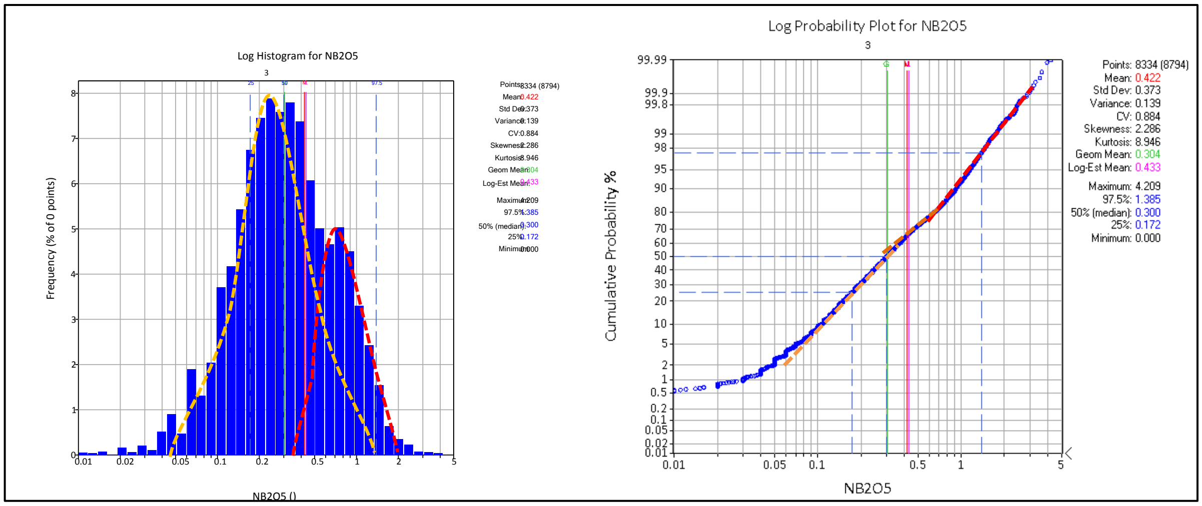


Source: SRK, 2015

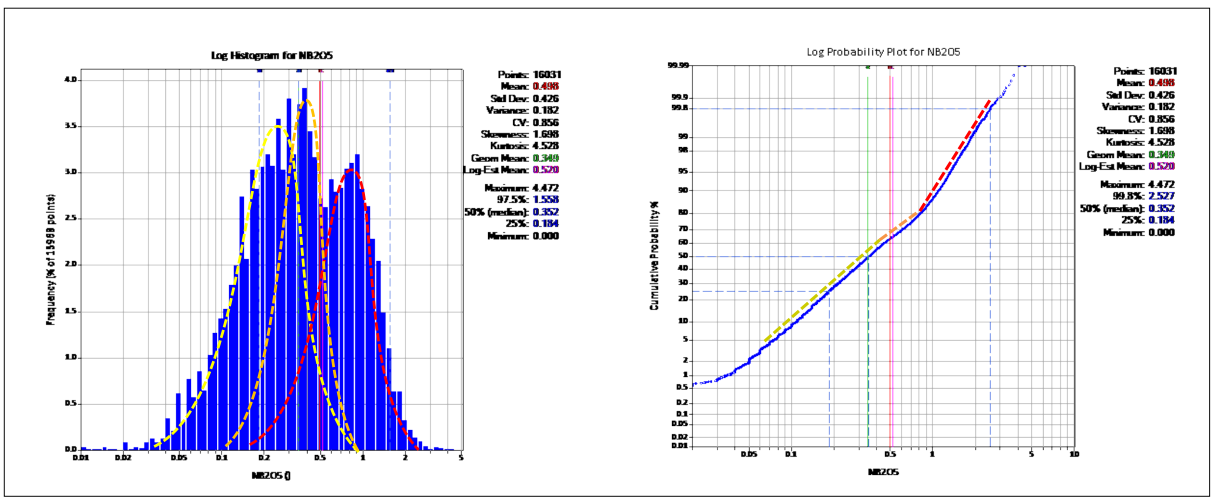
Figure 14.3.3: 3D view (looking northwest) of Elk Creek Showing Modelled Mafic Units Below the Carbonatite to Pennsylvanian Sediments Unconformity

Using log histograms and log-probability plots SRK has confirmed the box-whisker analysis that more than one sample population is present (Figure 14.3.4) at Elk Creek. The two main populations can be described as low-grade population ranging between 0.1% and 0.5% Nb_2O_5, and a higher-grade population in excess of 0.5% Nb_2O_5. The contact between the two populations is not clear on the charts and therefore SRK assumed some degree of transition between these two domains may exist. SRK has assumed that the lower grade population is defined by the CARB units, with the higher grades indicating the presence of MCARB.

The results indicated a slight change in the histogram compared to the September 2014 geological model, but overall supported the conclusions made at that time. The February 2015 geological model shows a more defined change in the trend for the higher grade domain, plus the transitional zone between the two main populations, can be seen by a third peak in histogram in the range of 0.3% to 0.4% Nb_2O_5.



September 2014 Model



February 2015 Model

Source: SRK, 2014/2015

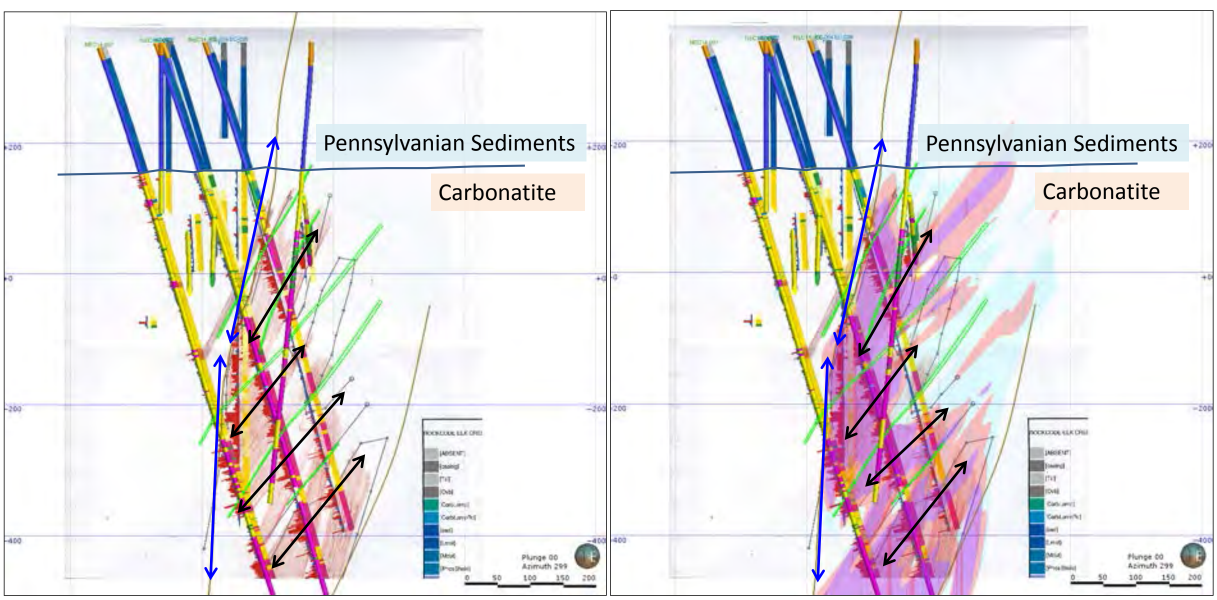
Figure 14.3.4: Statistical Analysis of Raw Nb_2O_5% Values within Elk Creek Carbonatite

Using Figure 14.3.4 and the assumption of a nominal lower grade cut-off of 0.3% Nb_2O_5, SRK has created grade shells at 0.3%, 0.4% and 0.5% Nb_2O_5. SRK found visually the best fits (in terms of correlation of grade and known higher-grade geological units), when using the 5 m composite data. At the shorter intervals, areas comprising of less than five continuous meters of low grade were producing isolates holes in the geological wireframes. SRK preferred the option to model larger more consistent wireframes, and are instead accounted for as internal dilution of lower-grade samples within the estimated blocks.

SRK tested multiple scenarios based on the raw and composite data to mimic the changes in niobium distributions between the CARB and MCARB beneath the unconformity, in addition to creating interpretations based solely on geological logging. Given the close relationship between the higher grades and the MCARB unit SRK has based the geological wireframe for the MCARB using an indicator methodology. To define the indicator model, values in the database are assigned a value of 0 or 1 based on a set criteria (such as 0.3% Nb_2O_5, 0.4% Nb_2O_5 and 0.5% Nb_2O_5). This criteria is then used as the mathematical basis for the definition of a grade shell within Leapfrog®. The aim in using an indicator over a traditional grade shell is it removes the influence of the grades (where higher grades may push further), and relies on the underlying relationship between mineralized (value equals 1) and non-mineralized (value equals 0) material, which in SRK view better mimics the geology at the Project. SRK initially used an indicator cut-off of 0.4% Nb_2O_5 using a range of thresholds between 0.25 and 0.5, with the resultant Leapfrog® grade shells validated against the geological logging, and a 0.35 threshold (isovalue), providing the best visual correlation.

To evaluate the preferred interpretation for the geological/grade shells boundaries, SRK has been provided with a series cross-sections and one long section by onsite geological staff (Dahrouge). SRK has also held technical meetings with the senior Project Geologist to assist in defining the key geological controls on the deposit. The interpretation remains consistent with the September 2014 model, which has been supported by confirmation drilling during Phase II and Phase III programs at Elk Creek.

To improve the continuity in the geological interpretation SRK has used two dominant trend surfaces. The southwest contact has been modelled using a strong sub-vertical trend (shown in blue), while the northeast of the deposit has followed moderate dipping trends (shown in black) parallel to the low-grade mafic units (shown in green) as shown in Figure 14.3.5.



Source: SRK, 2015
Boreholes show geology and assays (red histograms), used for validation.

Figure 14.3.5: Cross-section Showing Leapfrog® Model vs. Geological Interpretation

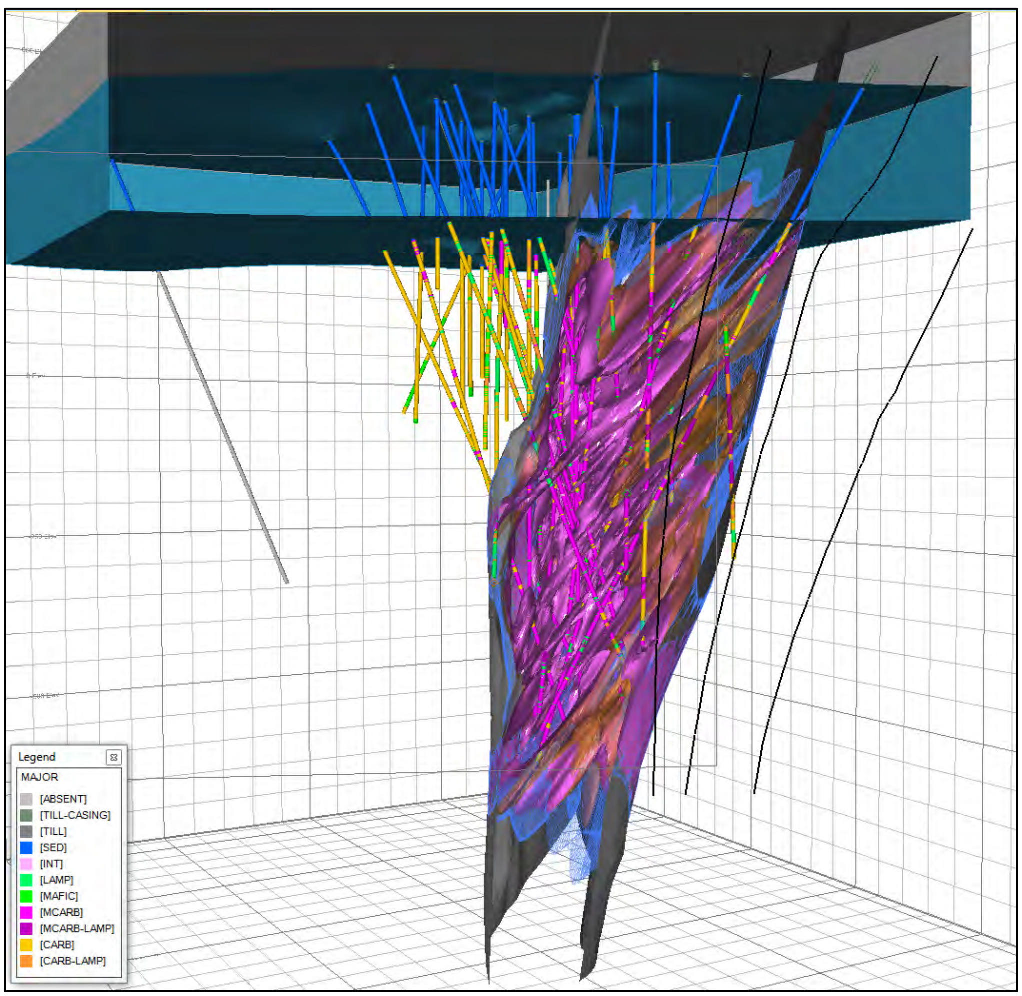
SRK has investigated the potential cause for the sharp contact on the southwestern edge of the deposit, to determine if the contact is structurally controlled. Based on a review of the drill core and

ATV surveys completed to date no fault has been established. SRK therefore assume that this forms a sharp igneous lithological contact.

During the geological modelling SRK noted that the 0.4% Nb_2O_5 defined domain closely correlates to the logged MCARB intervals, while the 0.3% Nb_2O_5 limit defines the edges of the mineralization, which is defined as CARB.

SRK noted a number of cases where the indicator model created significant volumes on the edge of the deposit in areas of limited drilling. SRK assumes these volumes to lack sufficient geological confidence for the definition of the Mineral Resource. SRK has limited the extent of the indicator wireframes in these cases to a corridor of mineralization bound by the steep southwest contact and a shallower northeast contact. The northeast contact has been based on geological and assay values at depth and projected to the unconformity (shown in brown on Figure 14.3.5).

The final wireframes (Figure 14.3.6) selected have been imported into Datamine and cropped accordingly to mimic the unconformity between the Carbonatite and the overlying Pennsylvanian sediments, to domain the drillhole information and for the generation of the geological block model.



Source: SRK, 2015

Figure 14.3.6: 3D Views Looking Northeast and Northwest of Selected Grade Shells Showing Pennsylvanian-Carbonatite Unconformity

Using the wireframes for the various CoGs the following domains have been defined for use in the Mineral Resource Estimate (Table 14.3.1).

Table 14.3.1: Summary of Geological Domains

KZONE	MAJOR	Description	Basis for Wireframe
1	TILL	Till	Geological contact between base of till and sediments
2	SEDT	Sediments	Geological contact between sediments and carbonatite
10	CARB	Carbonatite	Geological unit below the sediment contact
13	CARB	Low grade Carbonatite	Carbonatite material inside an Indicator wireframe of 0.3%
14	MCARB	Magnetite Carbonatite >0.4% Nb_2O_5	Carbonatite material inside an Indicator wireframe of 0.4% - validated against MCARB logging
15	MCARB	Magnetite Carbonatite >0.5% Nb_2O_5	Carbonatite material inside an Indicator wireframe of 0.5% - validated against MCARB logging
21	MAFIC/LAMP	Low-grade units	Defined from logging and low-grade samples, modelled in Leapfrog® as intrusive veins within the carbonatite

14.4 Assay Capping and Compositing

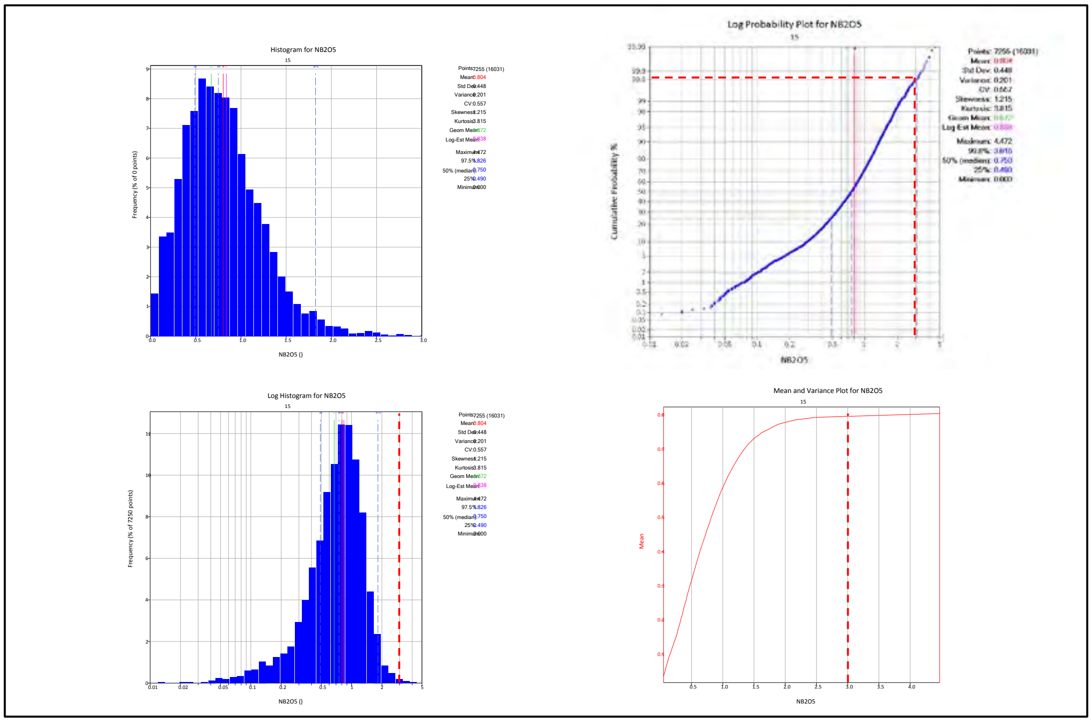
Prior to the undertaking of a statistical analysis, an outlier analysis has first been completed and samples need to be composited to equal lengths for constant sample volume, in order to honour sample support theories.

14.4.1 Outliers

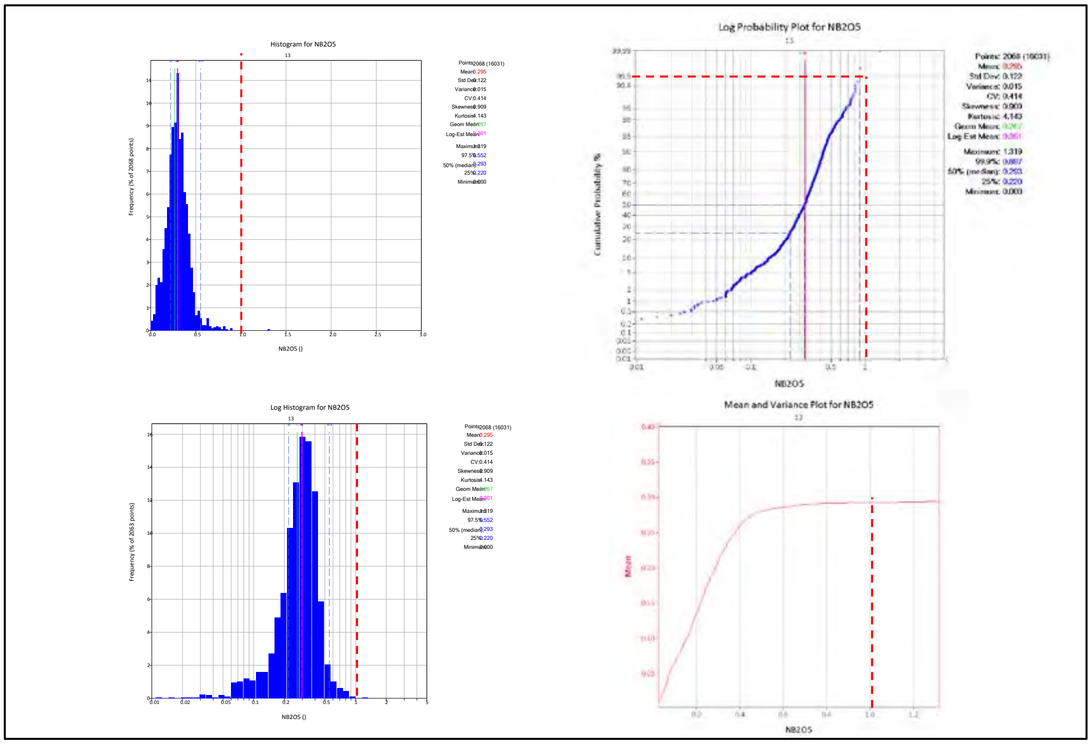
Outlier analysis has been completed for Nb_2O_5, TiO_2, Sc assays and density data per domain. The raw assay data was first plotted on histograms (Figure 14.4.1.1) and cumulative distribution plots (Figure 14.4.1.2) to understand its basic statistical distribution. High-grade capping was applied based on a combination of these plots, plus log histogram information. To create the plots the domained samples for all zones have been created in Datamine and imported into Snowden Supervisor v8.3 (Supervisor) for analysis.

The plots can be used to distinguish the grades at which additional samples have significant impacts on the local estimation and whose affect is considered extreme. Using this methodology top-cuts have been defined for each domain by reviewing the information from the different sample types.

The spatial occurrence of the capped values was visually verified to determine if they formed discrete zones which could potentially be modelled separately. Based on the analysis SRK has decided to apply a grade capping of 2.5% Nb_2O_5. For the mafic zones (mafic and Lamprophyre units) a cap of 1.0% Nb_2O_5 has been applied for the statistical analysis (Figure 14.4.1.1).



(a)

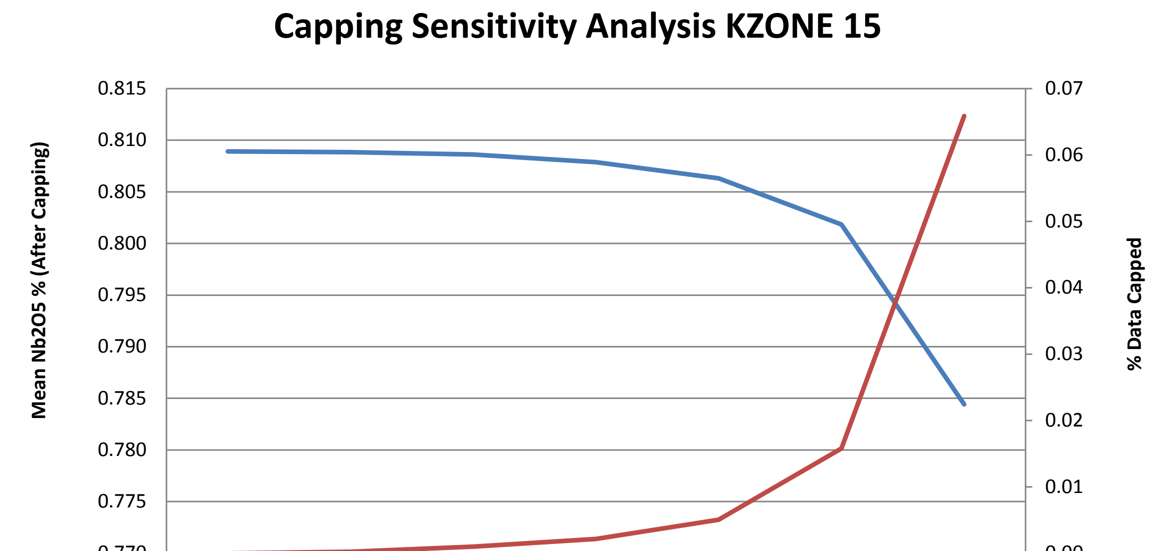


(b)

Source: SRK, 2014

Figure 14.4.1.1: SRK Capping Analysis, per Major Rock Type (a) MCARB, (b) CARB

The influence of the capping has been reviewed by SRK, to confirm the potential impact on the number of samples capped and the mean grades within each estimation domain (Table 14.4.1.1). Figure 14.4.1.2 provides an example of the study which reviews the number of samples capped within the 0.5% grade shell. The results show that approximately 0.62% of the database has been capped, with the mean grade reducing from 0.829% to 0.825% Nb_2O_5. SRK considers the capping to be appropriate for the style of mineralization.



	Raw Data	4.0% Cap	3.5% Cap	3.0% Cap	2.5% Cap	2.0% Cap	1.5% Cap
MEAN	0.81	0.81	0.81	0.81	0.81	0.80	0.78
% Capped	0.00	0.02%	0.10%	0.21%	0.51%	1.58%	6.59%

Source: SRK, 2015

Figure 14.4.1.2: Capping Sensitivity Analysis on Nb_2O_5% Grades within 0.5% Grade Shell

Table 14.4.1.1: Summary of the Capping Used per Domain and Element

KZONE	Major Rock	Rock Type	Cropped Density		
			Nb_2O_5	TiO_2	Sc
1	TILL	Till	n/a*	n/a*	n/a*
2	SED	Sediment	n/a*	n/a*	n/a*
10	CARB	Carbonatite (below cut-off)	1.0	4.0	80
13	CARB	Carbonatite (low grade)	1.0	4.0	95
14	MCARB	Magnetite Carbonatite (low grade)	1.5	4.5	110
15	MCARB	Magnetite Carbonatite (high grade)	3.0	6.0	150
21	MAFIC/LAMP	Mafic/Lamprophyre Units	1.0	3.5	65

* n/a due to no estimation of domain

14.4.2 Compositing

SRK has undertaken a sample composite analysis (Table 14.4.2.1) in order to determine the optimal sample composite length for grade interpolation. The analysis investigated both changes in composite length and minimum composite lengths for inclusion. Results are compared by reviewing the resultant mean grade against the length weighted raw sample mean grades, and the percentage of samples excluded applying the minimum composite length.

SRK has utilized a function in Datamine where all samples are maintained during the composite routine (MODE=1). MODE 1 forces all samples to be included in one of the composites by adjusting the composite length, while keeping it as close as possible to the 5 m interval selected by SRK. A review of the composite lengths per domain shows on average the mean length of the composite within the Carbonatite is in the order of 1.0 to 1.5 m, while the thinner mafic units average closer to 1.25 m. A comparison of the mean Nb_2O_5% grades shows the impact of the composite and capping routines results in slightly lower means at <0.4% in the Carbonatite. The reduction in mafic units reports larger differences of up to 15% but this is typically due to the low numbers in the samples populations. SRK assumes the differences in the mafics can be explained by differences in the logging of Molycorp drilling, and while this may have a degree of conservatism, the overall tonnage of the mafic units is low in comparison to the Carbonatite. SRK deemed the capping satisfactory, and no bias has been introduced during the capping and composite processes.

Table 14.4.2.1: Composite Length Analysis for Domain 15 (0.5 Nb_2O_5% grade shell)

Composite	% Min Length	N Samples	Minimum (Nb_2O_5%)	Maximum (Nb_2O_5%)	Mean (Nb_2O_5%)	Variance	Standard Deviation	CoV	% Difference from Mean
raw	all	8873	0	4.47	0.809	0.20	0.45	-	-
1	0.00	8337	0.0	4.47	0.803	0.17	0.41	-0.79%	0.51
1	0.25	8298	0.0	4.47	0.804	0.17	0.41	0.16%	0.51
1	0.50	8257	0.0	4.47	0.806	0.17	0.41	0.28%	0.51
1	0.75	8230	0.0	4.47	0.807	0.17	0.41	0.12%	0.51
1	1.00	8200	0.0	4.47	0.808	0.17	0.41	0.11%	0.51
2	0.00	4210	0.0	3.46	0.799	0.15	0.38	-1.08%	0.48
2	0.25	4171	0.0	3.46	0.802	0.15	0.38	0.38%	0.48
2	0.50	4149	0.0	3.46	0.804	0.15	0.38	0.20%	0.48
2	0.75	4092	0.0	3.46	0.809	0.15	0.38	0.69%	0.47
2	1.00	4057	0.0	3.46	0.812	0.15	0.38	0.27%	0.47
3	0.00	2848	0.0	3.50	0.795	0.13	0.37	-2.11%	0.46
3	0.25	2801	0.0	3.50	0.799	0.13	0.36	0.60%	0.46
3	0.50	2749	0.0	3.50	0.807	0.13	0.36	0.93%	0.45
3	0.75	2714	0.0	3.50	0.811	0.13	0.36	0.54%	0.45
3	1.00	2689	0.0	3.50	0.814	0.13	0.36	0.32%	0.44
4	0.00	2146	0.0	3.41	0.794	0.12	0.35	-2.42%	0.44
4	0.25	2118	0.0	3.41	0.798	0.12	0.34	0.50%	0.43
4	0.50	2069	0.0	3.41	0.805	0.12	0.34	0.90%	0.43
4	0.75	2032	0.0	3.41	0.811	0.12	0.34	0.67%	0.42
4	1.00	1989	0.0	3.41	0.817	0.12	0.34	0.79%	0.42
6	0.00	1736	0.0	2.71	0.793	0.11	0.33	-2.94%	0.42
6	0.25	1703	0.1	2.71	0.796	0.11	0.33	0.39%	0.42
6	**0.50**	1661	0.1	2.71	0.804	0.11	0.33	0.98%	0.41
6	0.75	1599	0.1	2.71	0.817	0.11	0.33	1.58%	0.40
6	1.00	1572	0.1	2.71	0.821	0.11	0.33	0.55%	0.40

Source: SRK, 2015

Table 14.4.2.2 shows a comparison of the mean grades within each zone based on the grade capping applied. Within the Carbonatite units the reduction in the mean is less than 0.5% for the Nb_2O_5 assays, while the difference in the means are more variable within TiO_2 and Sc database. Overall the reduction in the means are deemed acceptable by SRK and appropriate given the sampling distributions noted for each element.

Table 14.4.2.2: Comparison of Raw vs. Capped Composites Grades

	K Zone	Field	N Samples	Min.	Max.	Mean	Variance	Stand. Dev.	CoV	WGT Field	% Diff.
Raw Samples	2	Nb_2O_5	33	0.00	0.52	0.11	0.02	0.15	1.39	Length	
	10	Nb_2O_5	5609	0.00	2.32	0.20	0.02	0.15	0.77	Length	
	13	Nb_2O_5	2295	0.00	1.32	0.30	0.01	0.12	0.39	Length	
	14	Nb_2O_5	1939	0.00	1.18	0.36	0.03	0.16	0.45	Length	
	15	Nb_2O_5	8873	0.00	4.47	0.81	0.18	0.43	0.53	Length	
	21	Nb_2O_5	231	0.00	0.42	0.11	0.00	0.07	0.62	Length	
	2	SC_PPM	13	6.00	48.00	17.64	192.97	13.89	0.79	Length	
	10	SC_PPM	3885	4.00	196.00	32.52	371.10	19.26	0.59	Length	
	13	SC_PPM	2300	6.00	152.00	59.44	276.68	16.63	0.28	Length	
	14	SC_PPM	1940	8.00	156.00	62.82	459.68	21.44	0.34	Length	
	15	SC_PPM	8879	6.00	306.00	73.77	666.49	25.82	0.35	Length	
	21	SC_PPM	238	1.00	106.00	37.04	183.18	13.53	0.37	Length	
	2	TiO_2	36	0.07	1.82	0.58	0.22	0.47	0.82	Length	
	10	TiO_2	5630	0.01	6.80	0.94	0.86	0.93	0.98	Length	
	13	TiO_2	2296	0.02	5.22	1.38	0.51	0.72	0.52	Length	
	14	TiO_2	1940	0.02	7.33	1.80	0.71	0.84	0.47	Length	
	15	TiO_2	8878	0.02	13.87	2.98	1.62	1.27	0.43	Length	
	21	TiO_2	231	0.02	4.80	1.15	1.59	1.26	1.10	Length	
5 m Capped Composite	2	Nb_2O_5	28	0.00	0.48	0.12	0.02	0.15	1.22	Length	15.2%
	10	Nb_2O_5	1424	0.00	1.00	0.20	0.01	0.11	0.54	Length	-1.4%
	13	Nb_2O_5	556	0.00	0.66	0.30	0.01	0.07	0.25	Length	0.0%
	14	Nb_2O_5	469	0.00	0.74	0.36	0.01	0.11	0.32	Length	-0.2%
	15	Nb_2O_5	1664	0.00	2.60	0.80	0.11	0.33	0.40	Length	-0.1%
	21	Nb_2O_5	84	0.00	0.30	0.11	0.00	0.07	0.57	Length	0.0%
	2	SC_PPM	6	6.83	38.64	15.87	127.49	11.29	0.71	Length	-10.0%
	10	SC_PPM	672	10.00	80.00	31.93	240.94	15.52	0.49	Length	-1.8%
	13	SC_PPM	556	8.19	95.00	59.19	176.80	13.30	0.22	Length	-0.4%
	14	SC_PPM	469	9.50	110.00	62.66	356.25	18.87	0.30	Length	-0.2%
	15	SC_PPM	1664	10.85	150.00	73.48	455.10	21.33	0.29	Length	-0.4%
	21	SC_PPM	84	1.00	65.00	36.31	111.21	10.55	0.29	Length	-2.0%
	2	TiO_2	19	0.21	1.69	0.66	0.21	0.45	0.69	Length	14.1%
	10	TiO_2	1432	0.02	4.00	0.93	0.61	0.78	0.84	Length	-1.0%
	13	TiO_2	555	0.05	3.90	1.38	0.35	0.59	0.43	Length	-0.3%
	14	TiO_2	469	0.13	4.50	1.80	0.48	0.69	0.38	Length	-0.4%
	15	TiO_2	1664	0.05	5.97	2.96	0.90	0.95	0.32	Length	-0.7%
	21	TiO_2	84	0.02	3.50	1.09	1.24	1.12	1.03	Length	-5.6%

Source: SRK, 2015

14.5 Density

Dahrouge conducted density testing on the drill core to support the resource estimation. Approximately 2,045 samples were tested from the 2014 drilling program, completed using a combination of volumetric density (1,777 samples) determination and water immersion (1,493 samples) for confirmation. The density data was subdivided by the major lithologic groups used in the geologic model, and averages were calculated for each group.

The results are presented in Table 14.5.1. Density was assigned in the block model based on each block's lithology. Blocks outside of the resource estimation with unclassified lithology, were assigned a density of 2.82 g/cm^3, the average value for all the measurements taken.

Table 14.5.1: Density Determinations

Filters	SED	CARB	CARB-LAMP	MCARB	MCARB-LAMP	INT	LAMP	MAFIC
Samples	223	882	230	1940	113	12	382	24
Minimum	2.02	2.19	2.17	2.14	2.08	2.70	2.08	2.27
Maximum	2.85	3.96	3.44	4.19	3.30	3.77	4.19	3.41
Mean	2.49	2.89	2.85	3.04	2.91	2.90	2.85	2.95
Standard deviation	0.16	0.20	0.15	0.24	0.17	0.34	0.24	0.31
CV	0.06	0.07	0.05	0.08	0.06	0.12	0.09	0.11
10%	2.27	2.68	2.69	2.79	2.68	2.70	2.54	2.40
20%	2.36	2.79	2.77	2.87	2.82	2.73	2.69	2.65
30%	2.41	2.84	2.80	2.93	2.88	2.75	2.75	2.83
40%	2.44	2.86	2.83	2.98	2.92	2.81	2.82	2.98
50%	2.50	2.89	2.85	3.02	2.93	2.81	2.87	3.02
60%	2.55	2.92	2.89	3.07	2.96	2.83	2.92	3.02
70%	2.58	2.95	2.91	3.13	2.98	2.83	2.96	3.06
80%	2.62	2.99	2.95	3.19	3.02	2.83	3.00	3.19
90%	2.67	3.05	3.02	3.30	3.04	3.04	3.08	3.35
95%	2.72	3.18	3.05	3.45	3.08	3.40	3.16	3.35
99%	2.83	3.58	3.15	3.78	3.26	3.70	3.47	3.41

Source: SRK, 2015

During the February 6 Mineral Resource Estimate (Nb_2O_5 reported only) which formed the basis for the press release dated 09 February 2015, SRK assumed an average density based on the major geological units. The average density assigned is shown in Table 14.5.2. The breakdown of density has been based on the estimation domain (KZONE), with higher density values within the higher grade domains based on the relationship within higher magnetite content associated with the higher grades.

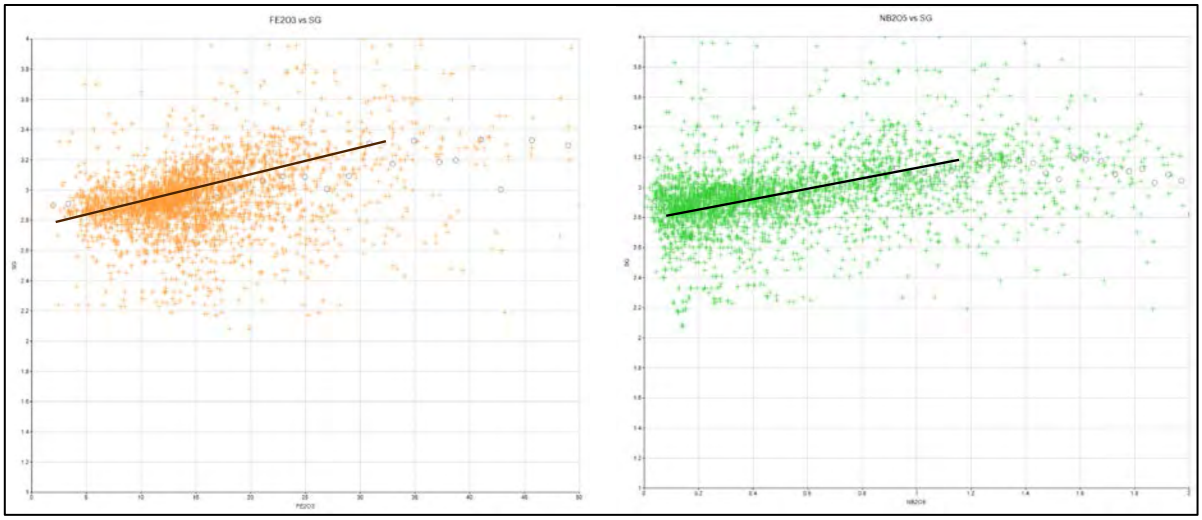
Table 14.5.2: Density used per Major Rock Type used in February 9, 2015 Mineral Resource Estimate

KZONE	Major Rock	Rock Type	Assigned Density
1	TILL	Till	2.00
2	SED	Sediment	2.48
10	CARB	Carbonatite (below cut-off)	2.82
13	CARB	Carbonatite (low grade)	2.85
14	MCARB	Magnetite Carbonatite (low grade)	2.90
15	MCARB	Magnetite Carbonatite (high grade)	3.05
21	MAFIC/LAMP	Mafic/Lamprophyre Units	2.86

Source: SRK, 2015

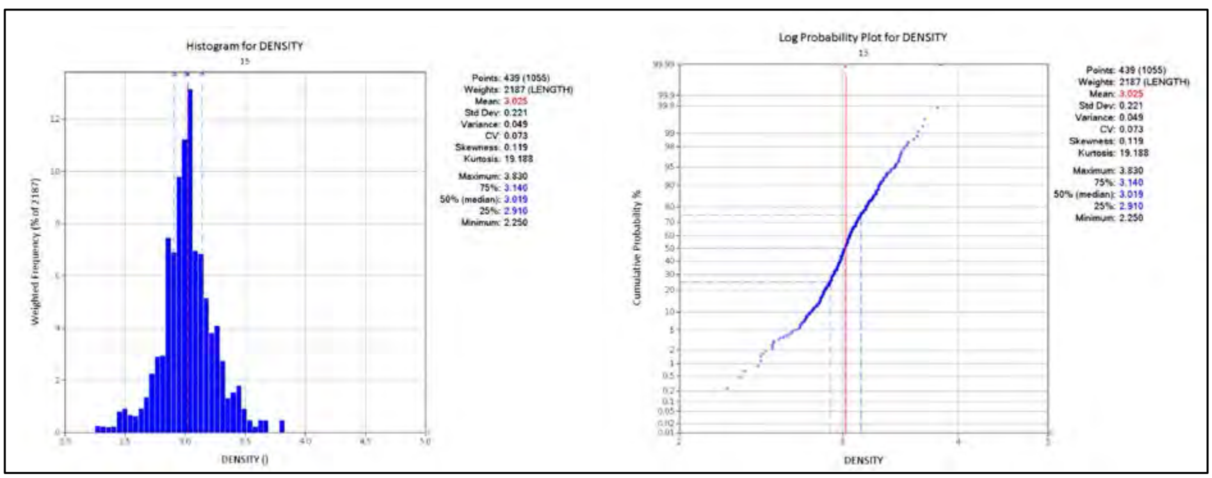
On receipt of the whole rock analysis database and prior to updating the Mineral Resource for the TiO_2% and Sc (ppm) estimates SRK conducted a review of the variability within the density values to determine/confirm if a relationship existed between the higher FeO_2% and the Nb_2O_5% (Figure 14.5.1). The study indicated that while a direct correlation is not established there is a trend for high density associated with high FeO_2% and Nb_2O_5%. Further review of the histograms for the density data per zone show variation in the density, and large enough sample populations for SRK to consider the estimation of density into the block model to be appropriate. SRK has assumed in terms of search orientations that the density values are associated with the same search orientations as the Nb_2O_5% distributions. To complete the analysis SRK has reviewed the histograms

(Figure 14.5.2) and applied capping to the density values per domain as appropriate. SRK has used the same methodology for reviewing outliers as discussed in Section 14.4.1 of this report.



Source: SRK, 2015

Figure 14.5.1: XY Scatter Plots of Density Values vs. Fe_2O_3 and Nb_2O_5



Source: SRK, 2015

Figure 14.5.2: Histogram and Log Probability Plot of Density Measurements within KZONE 15

Table 14.5.3: Summary of Capped Density Values per Domain

KZONE	Major Rock	Rock Type	Capped Density
1	TILL	Till	n/a (1)
2	SED	Sediment	n/a*
10	CARB	Carbonatite (below cut-off)	3.20
13	CARB	Carbonatite (low grade)	3.25
14	MCARB	Magnetite Carbonatite (low grade)	3.30
15	MCARB	Magnetite Carbonatite (high grade)	3.85
21	MAFIC/LAMP	Mafic/Lamprophyre Units	3.00

Source: SRK, 2015
(1) Used assigned density from previous study

In summary the change from the use of a single density per zone compared to the estimated density for the estimated domains (13, 14, 15, 21), reported a difference of 229,200,000 t vs. 228,200,000 t which is in the order of 0.5% (at a 0% Nb_2O_5 cut-off) which SRK does not consider to be a material change. SRK considers the use of estimated density to be more reasonable given the variable nature based on higher Nb_2O_5% and Fe_2O_3% (disclosed on February 23, 2015).

14.6 Variogram Analysis and Modeling

Variography is the study of the spatial variability of an attribute (in this case Nb_2O_5%, TiO_2%, Sc). Datamine and Supervisor have been utilized to test the geostatistical relationship for the deposit. Variogram analysis was performed on the capped and composited data filtered to include only the carbonatite domain. No stable semi-variograms have been achieved within the mafic units.

In completing the analysis the following has been considered:

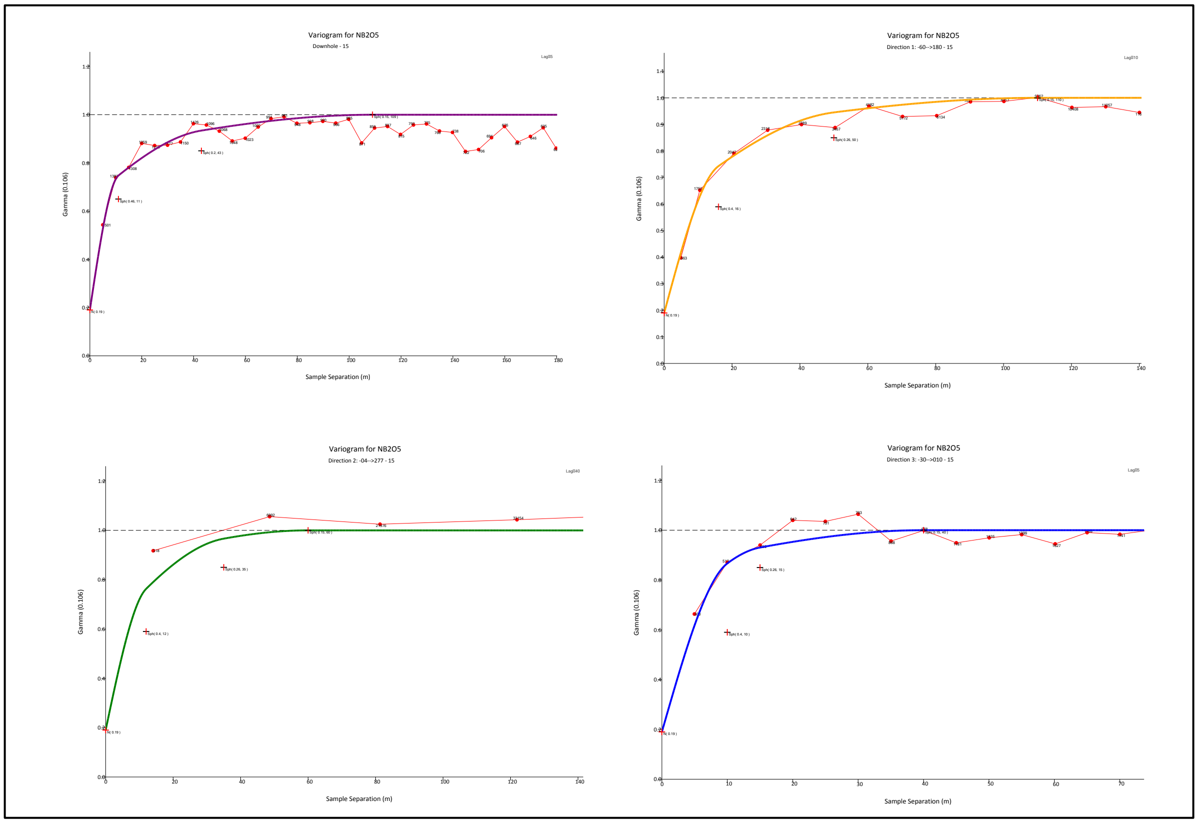
- Azimuth and dip of each zone was determined;
- The down-hole variogram was calculated and modelled to characterize the nugget effect;
- Experimental raw and pairwise relative semi-variograms, were calculated to determine directional variograms for the along strike, cross strike and down-dip directions;
- Directional variograms were modelled using the nugget and sill defined in the down-hole variography, and the ranges for the along strike, cross strike and down-dip directions; and
- All variances were re-scaled for each domain to match the total variance for that zone

A triple spherical structure was used to model the variograms for all three elements. A lag of 25 m was used with a variable separation based on the extents of the data. The semivariogram parameters are presented in Table 14.6.1. The experimental semi-variogram data is shown in Figure 14.6.1 fit with the model semi-variogram parameters listed in Table 14.6.1. The results indicate a reasonable nugget variance, but then a significant portion of the variability is within a short range (first sill) of between 7 to 20 m. SRK attributes the short scale variability to changes in the geological units between mdolcarb and dolcarb. Improving the geological model and hence geological domaining may improve the continuity noted within each of this units, which should be considered during the next Mineral Resource update.

Table 14.6.1: Semivariogram Model Results

Element	Sill	Variance	Variance %	Strike 120/0	Dip 30/-55	Across Strike 30/35
Nb_2O_5 (KZONE 13-15)	C0	0.19	0.19			
	C1	0.40	0.40	16	12	10
	C2	0.26	0.26	50	35	15
	C3	0.15	0.15	110	60	40
Nb_2O_5 KZONE 21)	C0	0.19	0.19			
	C1	0.40	0.40	12	12	12
	C2	0.26	0.26	50	50	50
	C3	0.15	0.15	105	105	105
TiO_2 (KZONE 13-15)	C0	0.24	0.24			
	C1	0.35	0.35	19	20	25
	C2	0.26	0.26	40	40	31
	C3	0.15	0.15	105	120	80
Sc ppm (KZONE 13-15)	C0	0.17	0.19			
	C1	0.26	0.40	13	12	18
	C2	0.12	0.26	61	36	53
	C3	0.45	0.15	180	84	75

Source: SRK, 2015



Source: SRK, 2015

Figure 14.6.1: Semi-Variogram Analysis for Domain 15 (0.5 Nb_2O_5% grade shell)

14.7 Block Model

The block model was constructed within the UTM grid (NADS83 Zone 14) coordinate limits listed in Table 14.7.1. A 5 m x 15 m x 5 m (x, y, z) block size was chosen as an appropriate dimension based on the current drillhole spacing and a potential underground smallest mining unit (SMU), compared to a drill spacing in the order of 60 m x 60 m within infill drilled sections. Sub-blocking has been allowed along the boundaries to a minimum of 0.5 m along strike, 2.5 m across strike and 1.0 m in the vertical direction, to maintain the geological interpretation. The block size has been based on the SMU, but it is SRK understanding that mine planning for the stopes will likely occur at a larger scale and not be based on individual blocks. The current block size will allow the mine planning to have the required level of flexibility when running the stope optimization. The topographic surface was created from the aerial survey of the topography and verified against the drill collars.

All modelling was conducted in Datamine for the Project grade estimation. The top of the carbonatite surface is located approximately 200 m below surface and is overlain by a sequence of Pennsylvanian sediments which have been modelled in Leapfrog®. All grade estimates are cropped to this contact.

SRK previously used a rotation to improve the geometric representation of the deposit. A rotated block model was created using a strike of 120º (set to -60º using Datamine convention). Based on work currently underway on the geotechnical aspects of the Project this rotation is noted to be oblique to some of the principal stresses, which have been supported by the fault model, and a specialized horizontal stress test completed as part of a geotechnical program. To improve the potential mine design and to reduce the potential for dilution SRK has rotated the block model to align with the key stress orientations.

To ensure no bias has been introduced in terms of dilution across the geological block model SRK has run the same model parameters using three different scenarios:

- Block model set-up based on key geological orientations (maximize grade continuity between blocks);
- Block model rotated 15º towards (half the required rotation) the principal stress orientation; and
- Block model rotated 15º towards (full rotation) into the principal stress orientation.

SRK noted that the difference in the global grades and tonnages between all three scenarios was negligible. Given the significant potential for improvement for the mine design (stope orientations), SRK elected to use the fully rotated prototype which aligns to the principal stress.

SRK has maintained the 5 m block size across strike as used in the geological model to ensure the vertical variation in the zones is modelled. A comparison of the block model dimensions used in 2014 and 2015 are shown in Table 14.7.1 and Table 14.7.2.

Table 14.7.1: Block Model Prototype used September 2014

Item	Origin	Rotation (Z Axis)	Block Dimension (m)	Number of Blocks	Minimum Sub-block
Easting	739,520		5	90	0.5
Northing	4,460,900	-60	15	55	2.5
Elevation	-600		5	200	1

Source: SRK, 2014

Table 14.7.2: Block Model Prototype used February 2015

Item	Origin	Rotation (Z Axis)	Block Dimension (m)	Number of Blocks	Minimum Sub-block
Easting	739,290		5	121	0.5
Northing	4,460,740	-30	15	70	2.5
Elevation	-650		5	220	1

Source: SRK, 2015

14.8 Estimation Methodology

The grade estimation has been completed using hard boundaries for the lithological (mafic) and mineralization (carbonatite grade shell) domains. Only the composites from the same domain have been used during estimation. This boundary corresponds to the geologic model presented in Section 14.3. The block model was first coded so that all blocks within this solid were flagged according to the relevant estimation domain (KZONE). The use of a soft boundary within the Carbonatite between the 0.4% and 0.5% limits has been tested during the September 2014 Mineral Resource Estimate, the findings of which indicated that the higher grades within the MCARB were smoothing into lower grade carbonatite material. The findings from the study showed in a previous iteration of the geological wireframe that removing the hard boundary increased the tonnage by 2%

and the grade by 5% at a cut-off of 0.3% (Nb_2O_5%). This increased at higher cut-offs to 32% more tonnage for a reduction of 7% in the grade. SRK concluded that the hard contact provided a better visual comparison to the raw sampling information. A review of the drillhole logs and core indicate a relatively sharp increase in the levels of magnetite and hence the definition of MCARB material. SRK considers this assumption to remain appropriate to the current geological model and estimation.

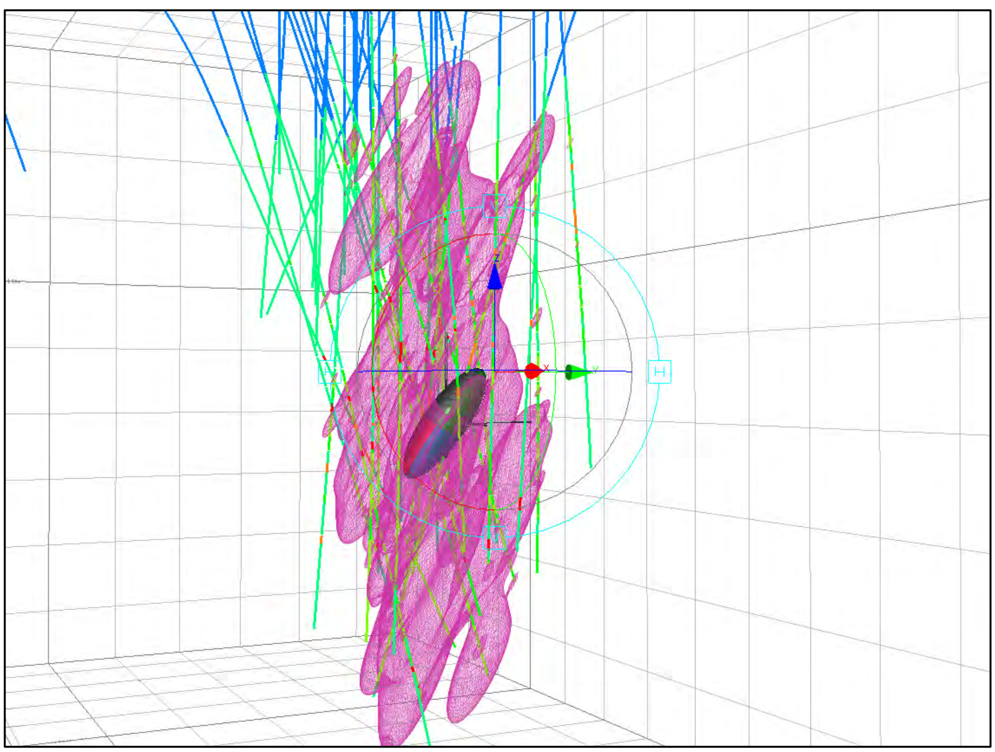
The Nb_2O_5% grade estimation utilized an OK algorithm supported by the 5 m sample composites for all units and elements, and density. A check estimate using Inverse distance weighting (IDW) to a power of 2 and nearest neighbor analysis has been completed for the Nb_2O_5 estimates for validation purposes. A nested search method consisting of three passes was used. The search ellipse has been rotated into the main dip and strike orientation of the deposit (Table 14.8.1).

Table 14.8.1: Ellipsoid Orientations

Domains	Rotation Angle 1	Axis of Rotation	Rotation Angle 1	Axis of Rotation
Nb_2O_5	30	Z-Axis	30	X-Axis
TiO_2	30	Z-Axis	30	X-Axis
Sc	15	Z-Axis	85	X-Axis
Density	30	Z-Axis	30	X-Axis

Source: SRK, 2015

Due to observed variations in the dip of the carbonatite and mafic units the search ranges have been rotated to best fit the semi-variogram orientation and the geological model (Figure 14.8.1). The search ranges are based on the results of the variography as well as the average drillhole spacing.



Source: SRK, 2014

Figure 14.8.1: Search Volume Orientation for Carbonatite Mineralization Shown vs. 0.5% Nb_2O_5 Grade Shell

In the first search passes for the Carbonatite, the estimation required a minimum of six and a maximum of 16 composites to assign grade to each block. A lower maximum number of 12 composites has been used in the mafics to account for the lower sample density and that commonly the mafics are represented by a single composite across the width of the wireframe. For the second pass, a minimum of three and a maximum of 12 composites were required to assign grade. In the third pass the minimum number of samples has been reduced to one sample and a maximum of 12 samples were required to assign grade. A maximum of three composites from a single drillhole were allowed for all passes, thus at least two drillholes were used for the first search pass. No blocks estimated in subsequent passes were allowed to overwrite the prior passes of estimation. No octant search restriction was applied.

The number of composites and drillholes used to estimate each block were stored during the estimation. Each pass of estimation was also recorded to show which blocks were estimated in which pass. The results show that an average of eight composites (using at least two holes) are used within the first two passes, which represents 73% of the number of blocks estimate. A detailed breakdown of the estimation parameters for the Carbonatite in each pass is shown in Table 14.8.2.

Table 14.8.2: Estimation Parameters and General Statistics for Carbonatite Estimate (0.3, 0.4, 0.5% Nb_2O_5 Grade Shells)

Parameter	KZONE 13 (0.3%)			KZONE 14 (0.4%)			KZONE 15 (0.5%)			KZONE 21 (MAFIC)		
	1	2	3	1	2	3	1	2	3	1	2	3
Major Axis (strike) (m)	75.00	150.00	300.00	75.00	150.00	300.00	75.00	150.00	300.00	50.00	100.00	250.00
Semi-Major Axis (dip) (m)	75.00	150.00	300.00	75.00	150.00	300.00	75.00	150.00	300.00	50.00	100.00	250.00
Minor Axis (across strike) (m)	20.00	40.00	80.00	20.00	40.00	80.00	20.00	40.00	80.00	10.00	20.00	50.00
Minimum Samples	6.00	3.00	1.00	6.00	3.00	1.00	6.00	3.00	1.00	3.00	4.00	1.00
Maximum Samples	16.00	12.00	12.00	16.00	12.00	12.00	16.00	12.00	12.00	12.00	24.00	20.00
Max per drillhole	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	2.00	2.00	2.00

Source: SRK, 2015

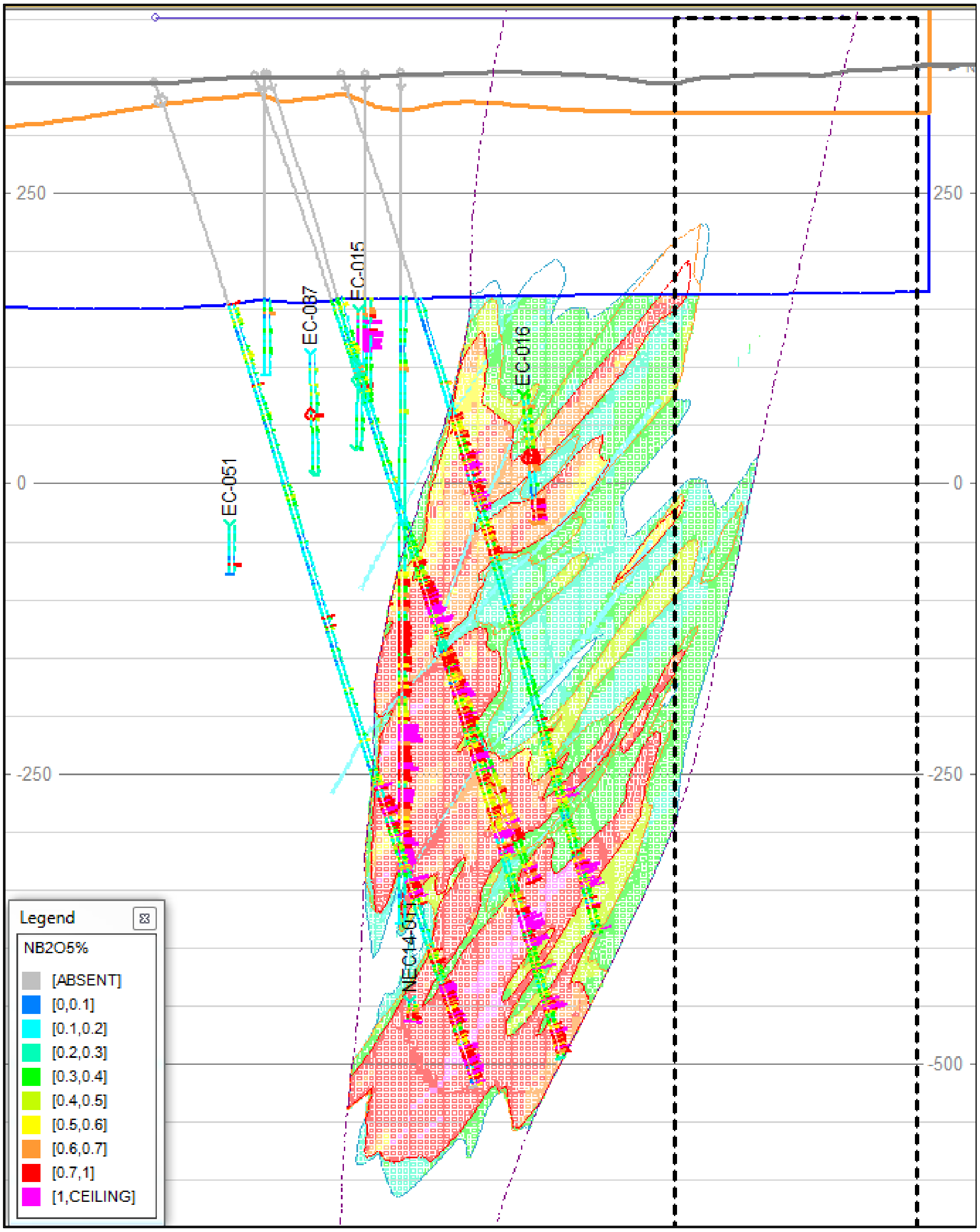
14.9 Model Validation

SRK has undertaken a thorough validation of the resultant interpolated model in order to confirm the estimation parameters, to check that the model represents the input data on both local and global scales and to check that the estimate is not biased. SRK has undertaken this using a using a number of different validation techniques.

- Inspection of block grades in plan and section and comparison with drillhole grades;
- Comparative Statistical study vs. composite data and alternative estimation methods; and
- Sectional interpretation of the mean block and sample grades (Swath Plots).

14.9.1 Visual Comparison

Visual validation provides a local validation of the interpolated block model on a local block scale, using visual assessments and validation plots of sample grades verses estimated block grades. A thorough visual inspection of cross-sections, long-sections and bench/level plans, comparing the sample grades with the block grades has been undertaken, which demonstrates good comparison between local block estimates and nearby samples, without excessive smoothing in the block model. Figure 14.9.1.1 shows an example cross-section of the visual validation checks and highlights the overall block grades corresponding with raw samples grades. Additional cross-sections showing the block estimates vs. the composite grades are shown in Appendix B.



Source: SRK, 2015

Figure 14.9.1.1: Cross-section looking northwest Showing Visual Validation of Boreholes to Grade Estimates

14.9.2 Comparative Statistics

SRK compared the composite grades to the estimated block grades within the wireframes for each domain. The composite grades are presented using the declustered weighting for comparison to the block statistics. Declustering was conducted using a cell-declustering algorithm, with default cell size of 20 m x 20 m x 20 m, improved correlation maybe achieved at different declustering grids. The comparison of the composite assays vs. the block estimates are shown in Table 14.9.2.1 for all three elements.

Table 14.9.2.1: Comparison of Block Estimates vs. Composite Samples (Carbonatite Domains)

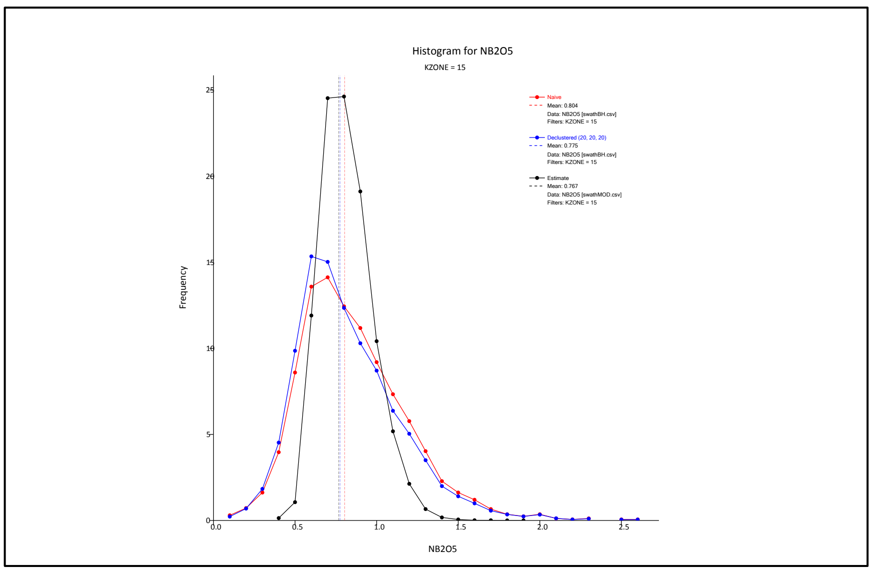
Element	KZONE	Statistic	Composite Sample Data	Declustered Sample Data	BlockData1 (Tonnage Weighted)	BlockData1 Vs Sample %Diff	BlockData1 Vs Declustered %Diff
Nb_2O_5	13	Mean	0.30	0.29	0.30	-0.49	1.12
		Std Dev	0.08	0.08	0.04		
		CV	0.26	0.27	0.13		
		Maximum	0.66	0.66	0.51		
		75%	0.35	0.35	0.32	-7.71	-7.44
		50%	0.31	0.30	0.30	-1.68	-0.36
		25%	0.26	0.25	0.27	7.06	9.40
	14	Mean	0.35	0.34	0.35	1.62	4.65
		Std Dev	0.12	0.13	0.06		
		CV	0.35	0.38	0.16		
		Maximum	0.74	0.74	0.58		
		75%	0.43	0.42	0.40	-7.67	-6.19
		50%	0.37	0.36	0.36	-2.97	-0.46
		25%	0.28	0.26	0.32	14.17	21.23
	15	Mean	0.80	0.78	0.77	-4.56	-1.09
		Std Dev	0.33	0.32	0.16		
		CV	0.41	0.41	0.20		
		Maximum	2.60	2.60	1.82		
		75%	0.99	0.95	0.86	-13.06	-9.55
		50%	0.75	0.72	0.75	-0.56	4.31
		25%	0.57	0.54	0.65	15.09	19.95
TiO_2	13	Mean	1.38	1.39	1.39	0.87	-0.06
		Std Dev	0.61	0.65	0.37		
		CV	0.44	0.47	0.27		
		Maximum	3.90	3.90	3.30		
		75%	1.63	1.65	1.63	-0.01	-0.86
		50%	1.25	1.26	1.38	10.53	9.57
		25%	1.00	1.00	1.15	15.03	15.12
	14	Mean	1.80	1.81	1.81	0.55	-0.24
		Std Dev	0.73	0.77	0.44		
		CV	0.41	0.42	0.25		
		Maximum	4.50	4.50	4.01		
		75%	2.19	2.23	2.08	-5.08	-6.83
		50%	1.63	1.64	1.76	7.80	7.19
		25%	1.35	1.34	1.53	13.27	14.39
	15	Mean	2.95	2.88	2.90	-1.91	0.70
		Std Dev	0.95	0.94	0.51		
		CV	0.32	0.33	0.18		
		Maximum	5.97	5.97	5.13		
		75%	3.56	3.44	3.21	-9.81	-6.75
		50%	2.87	2.81	2.85	-0.66	1.52
		25%	2.29	2.26	2.57	12.09	13.79
Sc	13	Mean	58.83	57.69	54.61	-7.17	-5.34
		Std Dev	13.99	15.12	11.50		
		CV	0.24	0.26	0.21		
		Maximum	95.00	95.00	88.66		
		75%	65.00	65.00	62.11	-4.45	-4.45
		50%	61.62	61.15	57.31	-6.99	-6.28
		25%	51.89	50.42	49.64	-4.34	-1.55
	14	Mean	61.49	60.04	61.13	-0.58	1.83
		Std Dev	19.74	20.63	13.29		
		CV	0.32	0.34	0.22		
		Maximum	110.00	110.00	107.92		
		75%	72.73	72.22	68.92	-5.23	-4.57
		50%	63.67	62.08	62.77	-1.41	1.12
		25%	50.05	46.92	52.84	5.58	12.62
	15	Mean	73.45	72.57	70.97	-3.37	-2.19
		Std Dev	21.37	21.51	15.23		
		CV	0.29	0.30	0.21		
		Maximum	150.00	150.00	133.82		
		75%	85.22	84.47	80.72	-5.28	-4.43
		50%	72.22	71.43	71.74	-0.66	0.44
		25%	60.29	59.23	61.10	1.34	3.17

Source: SRK, 2014

The results show acceptable levels of correlation between the mean blocks and declustered means for the Carbonatite domains. The 0.4 Nb_2O_5% and 0.5 $Nb_2O_{5\%}$ which combined for the majority of the metal above cut-off show difference in the mean grades typically reporting less than ± 2.5%, which SRK deems within acceptable levels.

The highest differences in the mean grades are noted within the 0.4 Nb_2O_5% grade shell, with the block model grades reporting approximately 4.7% higher than the composite mean, and the 0.3 Nb_2O_5% grade shell (Sc_ppm grades), which reported 5.3% lower than the composite means. The difference in the mean grades between the composite and the block estimates within the 0.4 Nb_2O_5% grade shell, is 0.02% to provide context. SRK still considered these levels of error to be within acceptable levels of error for the current level of confidence and drillhole spacing.

In addition to the statistical analysis comparative histograms (Figure 14.9.2.1) and distribution plots have been reviewed to assess the degree of smoothing. The result indicate the mean grade of the deposits are relatively close (as confirmed in the statistical analysis), with the block models typically smoothed towards the mean and a reduction in higher end of the distribution. The level of smoothing is a function of the current drill spacing and to increase the correlation between the datasets further drilling at a closer spacing would likely be required.



Source: SRK, 2015

Figure 14.9.2.1: Example of Comparative Histogram of Composites vs. Block Estimates

During the 2014 Mineral Resource update SRK noted in the mafic units the differences between the composite and block estimates were more significant than in the Carbonatite units. SRK attributed the differences to two main factors:

- The relatively small sample populations; and
- The data populations within the mafic units were more highly skewed and the influence of individual high grades on the overall statistical mean should be considered higher.

SRK comments that the mafic units represent a relatively small tonnage compared to the other units.

SRK has therefore remodelled these units based on the revised geological logging codes and infill drilling information. SRK considers the confidence in the mafic units geological interpretation remains lower than the Carbonatite, but the majority of the estimates fall below the economic cut-off. A review of the statistical comparison between the composite grades and the block estimates does show an improvement in the 2015 block estimates. SRK still considered these levels of error to be within acceptable levels of error.

Table 14.9.2.2: Comparison of Block Estimates vs. Composite Samples (Mafic/low grade Domain)

Element	KZONE	Statistic	Composite Sample Data	Declustered Sample Data	BlockData1 (Tonnage Weighted)	BlockData1 Vs Sample %Diff	BlockData1 Vs Declustered %Diff
KZONE 21	Nb_2O_5	Mean	0.12	0.13	0.12	2.22	-3.93
		Std Dev	0.07	0.07	0.03		
		CV	0.55	0.52	0.24		
		Maximum	0.30	0.30	0.23		
		75%	0.17	0.18	0.14		
		50%	0.12	0.14	0.13	8.79	-7.66
		25%	0.07	0.07	0.11	60.43	52.80
	TiO_2	Mean	1.18	1.23	1.27	6.87	2.89
		Std Dev	1.15	1.14	0.62		
		CV	0.97	0.93	0.49		
		Maximum	3.50	3.50	3.23		
		75%	2.11	2.11	1.73	-18.26	-18.26
		50%	0.69	0.70	1.23	78.39	76.09
		25%	0.32	0.34	0.76	140.28	120.54
	Sc	Mean	37.1	37.9	38.4	3.29	1.24
		Std Dev	11.9	12.5	6.7		
		CV	0.3	0.3	0.2		
		Maximum	65.0	65.0	63.8		
		75%	37.0	37.4	43.5	17.61	16.28
		50%	35.0	35.0	37.1	6.07	6.07
		25%	34.6	34.7	33.1	-4.44	-4.70

Source: SRK, 2015

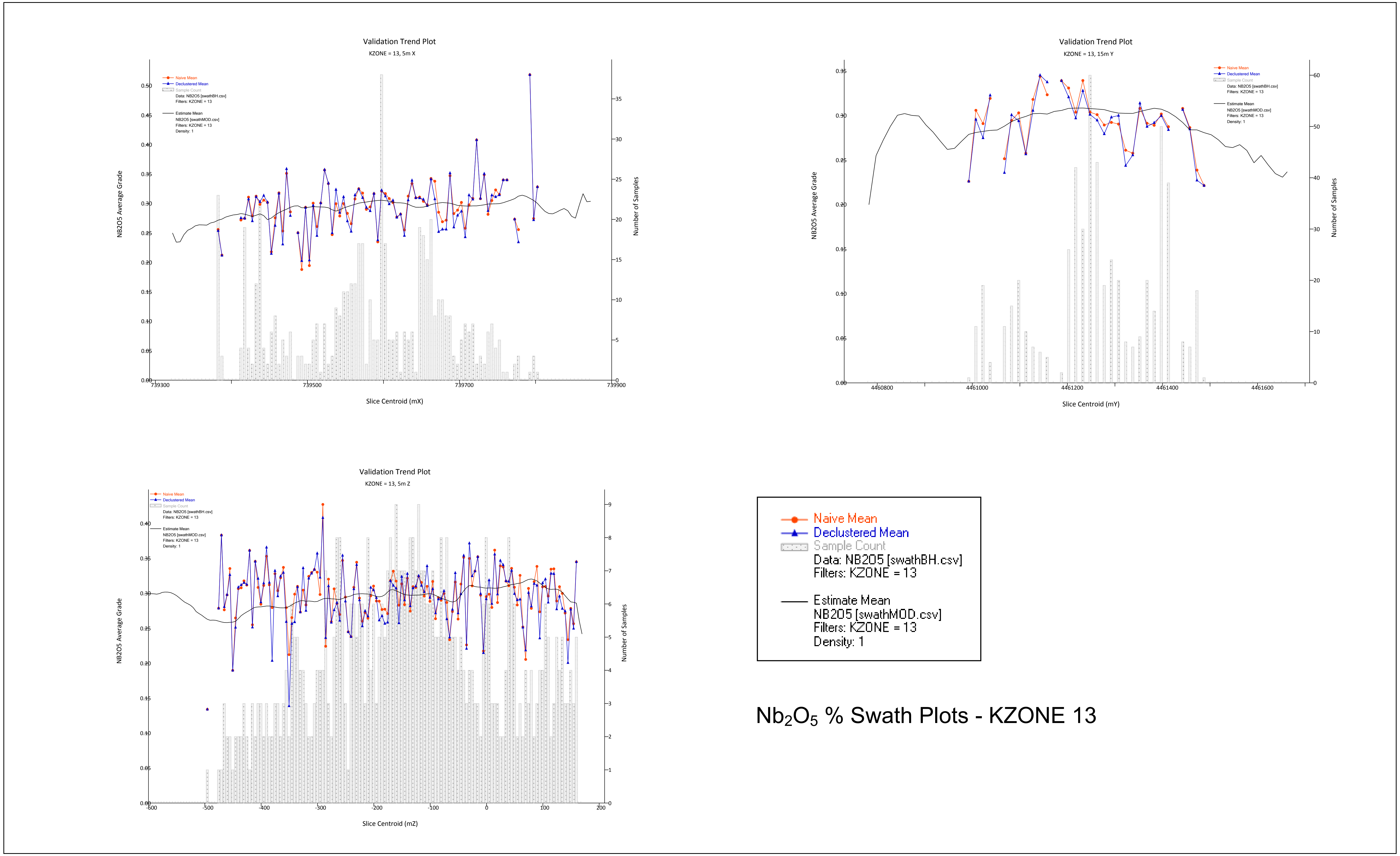
14.9.3 Swath Plots

Swath plots were generated, which show the mean grades in the block model as a function of their distribution along particular eastings, northings, and elevations.

The swaths compare the composite grades to the block grades, with the intention of ensuring that there are no significant deviations between the two which might mean that some bias exists in one part of the deposit. SRK calculated mean grades for composites and blocks within these swaths for all domains.

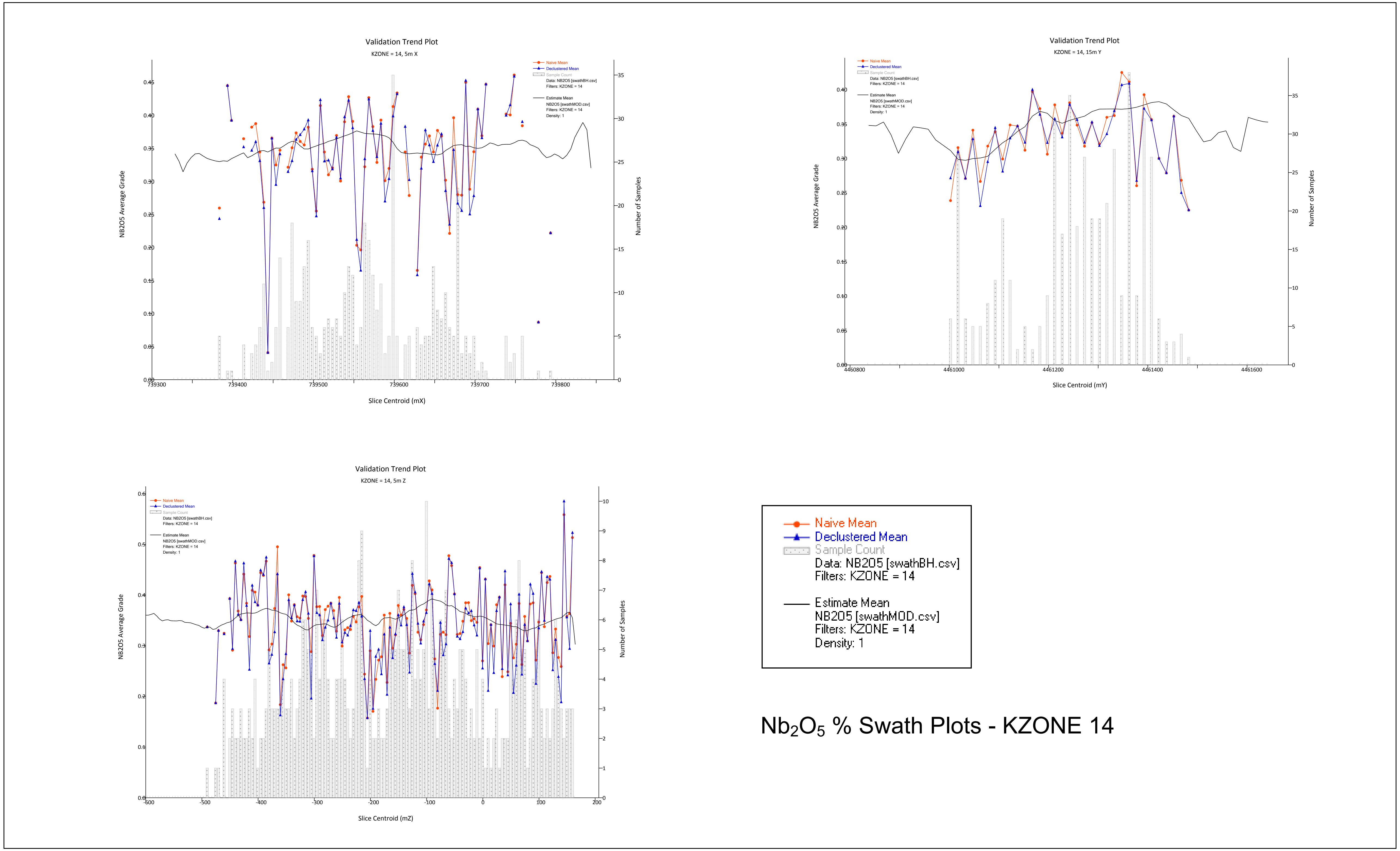
The resultant plots show a good correlation between the block model grades and the composite grades, with the block model showing a typically smoothed profile of the composite grades as expected. The plots for Nb_2O_5% generally confirm no indication of any significant bias introduced during the estimation, and generally display an adequate degree of smoothing. Based on the results of the analysis SRK have accepted the grades in the block model.

The swath analysis for the Carbonatite grade shells are shown in Figures 14.9.3.1 to 14.9.3.3.



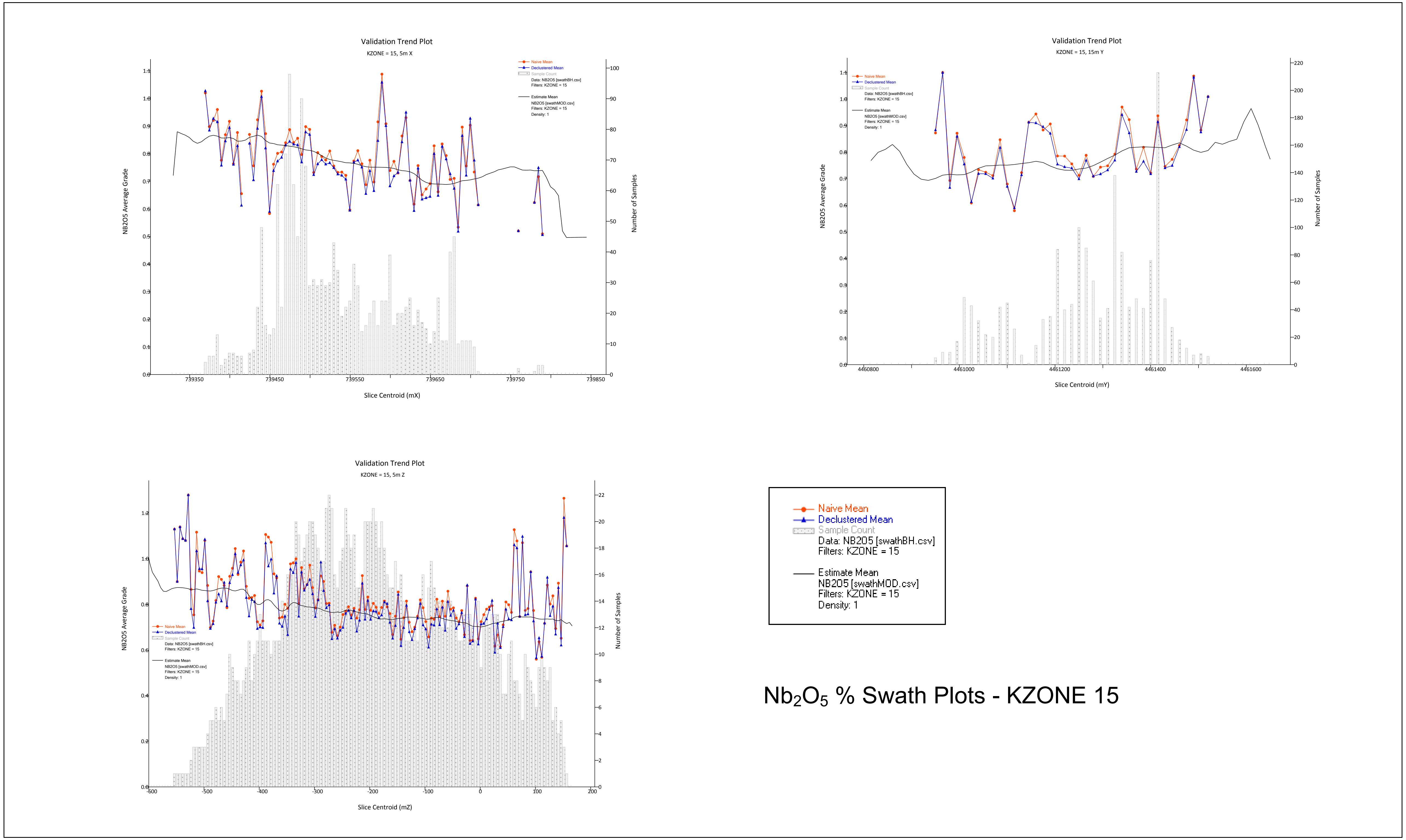
Source: SRK, 2015

Figure 14.9.3.1: Swath Plot for Nb_2O_5% Estimates within the 0.3% Grade Shell (KZONE=13)



Source: SRK, 2014

Figure 14.9.3.2: Swath Plot for Nb_2O_5% Estimates within the 0.4% Grade Shell (KZONE=14)



Source: SRK, 2014

Figure 14.9.3.3: Swath Plot for Nb_2O_5% Estimates within the 0.5% Grade Shell (KZONE=15)

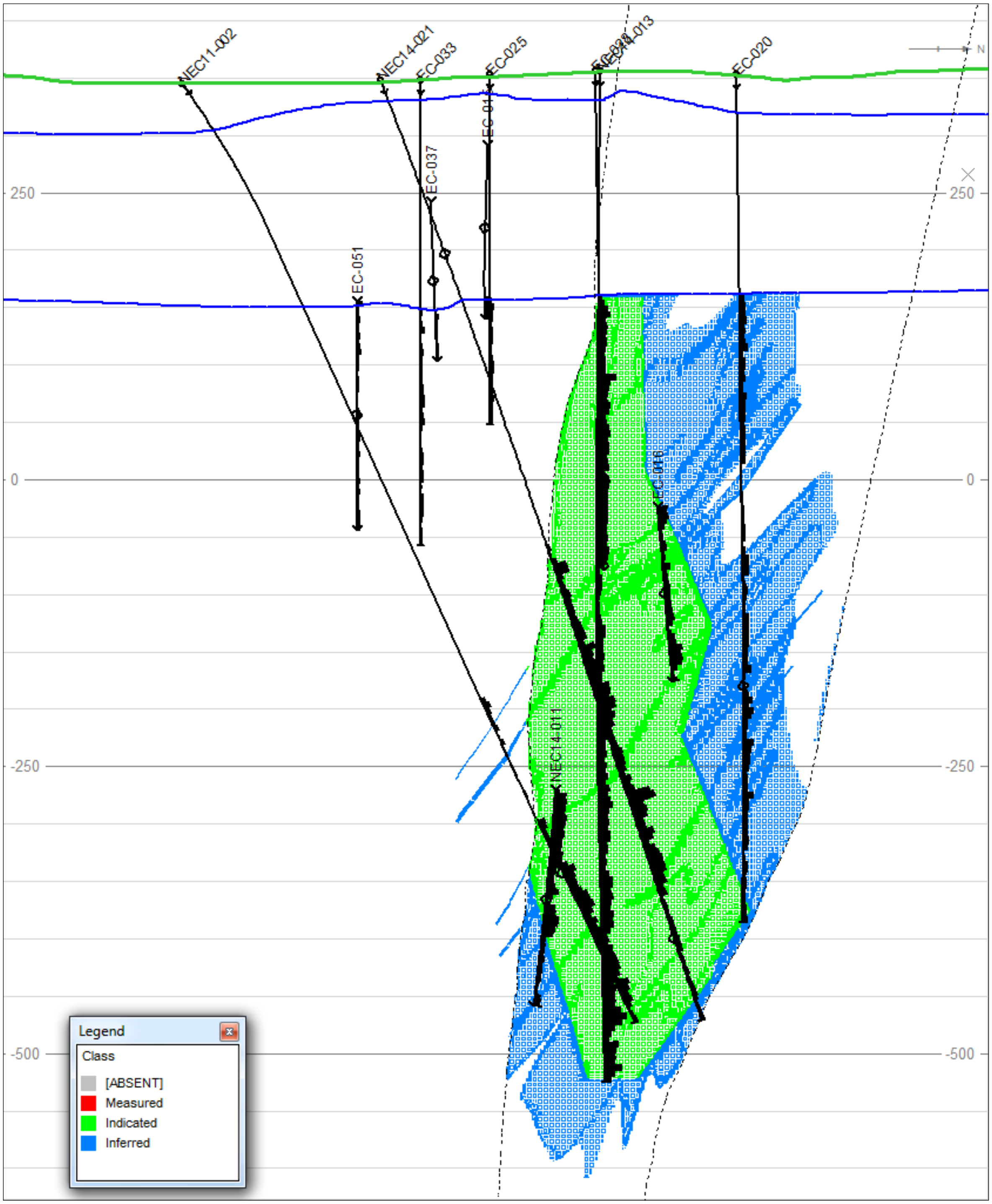
14.10 Resource Classification

The Mineral Resources are classified under the categories of Indicated and Inferred according to CIM guidelines. Due to a lack of dense (<35 m x 35 m) drilling and pending further analysis of the Actlabs vs. SGS accuracy issues no Measured Mineral Resource has been assigned at this stage for the Project.

SRK's classification mainly reflects the relative confidence of the geological model and the associated grade estimates. This classification is also based on sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization, data verification to original sources, specific gravity determinations, accuracy of drill collar locations, accuracy of topographic surface, quality of the assay data and many other factors, which influence the confidence of the mineral estimation. No single factor controls the resource classification rather each factor influences the result.

For the resource classification, a solid shape was constructed around the relatively well drilled core of the deposit resulting from the NioCorp Phase I to Phase III programs, where most drillholes are spaced approximately 60 to 70 m apart which allows typically three holes to be used for the first estimation search pass.

All blocks located within this area were classified as Indicated Mineral Resource (Figure 14.10.1). All blocks estimated outside of the perimeter of drillholes are classified as Inferred Mineral Resource, which typically extends 100 to 150 m beyond the drilling.



Source: SRK, 2015

Figure 14.10.1: Example of Classification

14.11 Mineral Resource Statement

The following section defines the updated Mineral Resource Statement for the Project. This statement includes the estimated Mineral Resources for Nb_2O_5, TiO_2 and Sc for the deposit and was disclosed on February 23, 2015, with an effective date of April 28, 2015. This represents the latest Mineral Resource for the Project.

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".

Portions of a deposit that do not have reasonable prospects for eventual economic extraction must not be included in a Mineral Resource.

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 taking into account extraction scenarios and processing recoveries. Based on this requirement, SRK considers that major portions of the Project are amenable for underground extraction with a processing method to recover Nb_2O_5, TiO_2 and Sc_2O_3 products.

The economic parameters were selected based on experience and benchmarking against similar projects (Table 14.11.1), and a 20% mark-up in the price assumptions to account for potential upside in market assumptions. Detailed technical studies have not been completed to date to confirm the assumed mining and processing costs, however SRK has provided reasonable estimates of the expected costs based on the knowledge of the style of mining (underground) and potential processing methods. The selected metal recovery is based on the initial metallurgical testwork completed during the Phase 1 program discussed in Section 13 of this current report.

Further work will be required to confirm these numbers via a detailed engineering study (prefeasibility or feasibility study). The reader is cautioned that the results are used solely for the purpose of testing the "reasonable prospects for economic extraction" by underground mining methods, and do not represent an attempt to estimate Mineral Reserves. There are no Mineral Reserves for the Project, and further work will be required to establish the costs to a higher level of confidence.

The estimated cost information presented here is used as a guide to assist in the preparation of a Mineral Resource Statement 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.

Table 14.11.1: Economic Assumptions Used to Define Mineral Resources

Parameter	Value	Unit
Mining Cost	26.00	US$/t mined
Processing	67.00	US$/t of feed
General and Administrative	1.50	US$/t of feed
Total Cost	94.50	US$/t of feed
Nb_2O_5 to Niobium conversion	69.9	percent
Niobium Process Recovery	60	percent
Niobium Price	50.00	US$/kg
TiO_2 Process Recovery	58.7	percent
TiO_2 Price	2.50	US$/kg
Sc Process Recovery	14.1	Percent
Sc Price	2,400	US$/kg
Calculated CoG Nb_2O_5	0.30	percent

Source: SRK, 2015

In order to determine the quantities of material offering "reasonable prospects for economic extraction" by an underground mining method, SRK has defined a suitable underground mining CoG based on assumed costs, pricing and metallurgical recoveries. The cost and recoveries used as the basis for the Mineral Resource have been based on the initial preliminary economic assessment completed in 2015 (disclosed in a press release, April 20, 2015). No update to the Mineral Resource statement has been made as part of the current technical report and therefore the assumptions shown in Table 14.11.1 remain valid. Increases in the recoveries and changes in the price assumptions shown in this study would result in a drop in the current selected cut-off grade.

The Mineral Resource has been 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% has been considered to have reasonable prospects of being mined via underground methods.

The 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 Statement in February 2015. Both of these metals can be recovered with simple additions to the existing process flowsheet, and would provide additional revenue streams that would complement the planned production of ferroniobium.

SRK defined a Mineral Resource on receipt of a validated database, to account for these additional revenue streams. No additional resource definition drilling has been completed since the press releases and therefore the drilling and sampling information presented in this technical report remain unchanged. 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 difference in the global tonnage between the estimated and assigned density has been accounted as < 1% change and is not considered material.

The Mineral Resource Statement in Table 14.11.2 has been determined using the economic parameters as defined in Table 14.11.1. The Mineral Resource was disclosed on February 23, 2015. This should be considered the latest estimate for the Project and be used in any future studies.

Table 14.11.2: SRK Mineral Resource Statement 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 assumes the Project is amenable to a variety of Underground Mining methods. Using results from initial metallurgical testwork, 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, CEng, 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. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

14.12 Mineral Resource Sensitivity

The grade tonnage distributions of the Measured and Indicated Mineral Resources at the Project are presented in Table 14.12.1 (based on the February 23, 2013 press release).

Table 14.12.1: Grade Tonnage Showing Sensitivity of the Project Mineral Resource to CoG, 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	0.60	59,700	0.82	489,200	2.94	1,750,000	74.2	4,400
	0.55	63,400	0.80	507,200	2.92	1,850,000	74.0	4,700
	0.50	65,200	0.79	515,000	2.91	1,900,000	73.9	4,800
	0.45	65,800	0.79	520,100	2.90	1,910,000	73.8	4,900
	0.40	68,100	0.78	531,000	2.87	1,950,000	73.7	5,000
	0.35	72,800	0.75	545,700	2.79	2,030,000	73.2	5,300
	0.30	80,500	0.71	571,600	2.68	2,160,000	72.0	5,800
Inferred	0.60	44,600	0.78	347,800	2.94	1,310,000	67.6	3,000
	0.55	50,700	0.76	385,100	2.92	1,480,000	67.3	3,400
	0.50	53,300	0.75	399,400	2.92	1,550,000	67.1	3,600
	0.45	54,300	0.74	401,600	2.91	1,580,000	66.9	3,600
	0.40	58,400	0.72	420,500	2.83	1,650,000	66.8	3,900
	0.35	67,500	0.67	452,400	2.69	1,810,000	66.0	4,500
	0.30	99,600	0.56	558,000	2.31	2,300,000	63.0	6,300

Source: SRK, 2015

14.13 Comparison with Previous Estimate

In comparison to the 2014 Mineral Resource Estimate for the Project, the updated estimate (February 20, 2015) represents a significant increase in the Indicated Mineral Resource when compared to the September 2014 estimate. The differences in the Mineral Resource can be attributed to the following points:

- Phase II and III infill drilling has decreased the drill spacing to the order of 60 to 70 m through the central portion of the deposit;
- Phase II and III infill drilling has targeted higher grade material at depth in the Mineral Resource; and
- Increase in the geological understanding of the controls on the niobium mineralization and grade domaining, based on the 2014 drilling program and relogging of historical holes.

To provide a like for like comparison of the Indicated Mineral Resources, SRK's 2014 block model reported using a CoG of 0.3 Nb_2O_5% has 22.6 Mt at a grade of 0.70% Nb_2O_5 which has increased to 80.5 Mt at a grade of 0.71% Nb_2O_5, within the Indicated category. This is an increase in the contained Nb_2O_5% from 177,000,000 kg to 571,600,000 kg, or 187% increase in the Indicated tonnage or 226% within the contained Indicated Nb_2O_5.

The Phase II and III infill drilling program initially only targeted the conversion of Inferred to Indicated within the current geological model, however as a direct result of the program additional Inferred material has been identified at depth and at the edges of the current Mineral Resource (limited to approximately 150 m along strike and 75 m down-dip.

The Inferred material when compared using a CoG of 0.3 Nb_2O_5% has reduced from 132.8 Mt at a grade of 0.55 Nb_2O_5% to 99.6 Mt at a grade of 0.56 Nb_2O_5%, which is a reduction from 733,700,000 kg to 557,800,000 kg (-24%) in contained Nb_2O_5 between the 2014 and 2015 models respectively.

Given the significant increase in the portion of Indicated material SRK considers the reduction in the Inferred to be reasonable as there is an increase in the global tonnage of approximately 20 Mt (approximately 10%). The deposit remains open at depth and along strike. A summary of the comparisons between CoG grades of 0.30 to 0.70 $Nb_2O_{5\%}$ is shown in Table 14.13.1.

Table 14.13.1: Comparison of 2012 to 2014 Tonnage and Grade per Category

		SRK September 2014 Estimates			SRK February 2015 Estimates							
	Cut-off	Tonnes (000's t)	Grade Nb_2O_5 %	Contained (000's kg)	Tonnes (000's t)	Grade Nb_2O_5 %	Contained (000's kg)	Grade TiO_2 %	Contained (000's kg)	Grade Sc (g/t)	Contained (000's kg)	Difference (% Nb_2O_5)
Indicated	0.70	10,800	0.84	91,300	45,200	0.87	391,800	3.01	1,361,200	73.9	3,300	**329.13%**
	0.65	13,500	0.81	109,500	53,300	0.84	446,800	2.97	1,581,800	74.1	3,900	**308.04%**
	0.60	15,800	0.78	123,700	59,700	0.82	486,600	2.94	1,751,100	74.2	4,400	**293.37%**
	0.55	17,400	0.76	133,100	63,400	0.80	508,200	2.92	1,850,400	74.0	4,700	**281.82%**
	0.50	19,100	0.74	142,000	65,200	0.79	517,700	2.91	1,897,300	73.9	4,800	**264.58%**
	0.45	20,700	0.72	149,300	65,800	0.79	520,800	2.90	1,912,100	73.8	4,900	**248.83%**
	0.40	22,600	0.70	157,600	68,100	0.78	530,100	2.87	1,950,600	73.6	5,000	**236.36%**
	0.35	25,300	0.66	167,800	72,800	0.75	547,600	2.79	2,029,500	73.2	5,300	**226.34%**
	0.30	28,200	0.63	177,000	80,500	0.71	571,600	2.68	2,159,400	72.0	5,800	**222.94%**
Inferred	0.70	34,400	0.85	291,100	29,800	0.00	251,600	3.02	900,800	67.7	2,000	**-13.57%**
	0.65	42,600	0.81	346,800	37,600	0.00	304,500	2.98	1,120,900	67.8	2,500	**-12.20%**
	0.60	51,900	0.78	404,900	44,600	0.78	348,100	2.94	1,313,200	67.6	3,000	**-14.03%**
	0.55	57,300	0.76	435,800	50,700	0.76	383,200	2.92	1,481,700	67.3	3,400	**-12.07%**
	0.50	63,700	0.74	469,600	53,300	0.75	396,800	2.92	1,554,700	67.1	3,600	**-15.50%**
	0.45	71,700	0.71	507,700	54,300	0.74	401,700	2.91	1,578,700	66.9	3,600	**-20.88%**
	0.40	87,200	0.66	573,300	58,400	0.72	419,000	2.83	1,654,700	66.8	3,900	**-26.91%**
	0.35	111,100	0.60	662,700	67,500	0.67	453,000	2.69	1,813,400	66.0	4,500	**-31.64%**
	0.30	132,800	0.55	733,700	99,600	0.56	557,800	2.31	2,304,500	63.0	6,300	**-23.97%**

Source: SRK, 2015

14.14 Relevant Factors

SRK is not aware of any environmental, permitting, legal, title, taxation marketing or other factors that could affect resources.

15 Mineral Reserve Estimate

No Mineral Reserves have been estimated for the Project.

16 Mining Methods

The Project is currently in the exploration phase and has not been developed. Mineralization is located approximately 200 to 1,000 m below the surface. Based on geomechanical information and mineralization geometry an underground longhole stoping method (LHS) is suitable for the deposit.

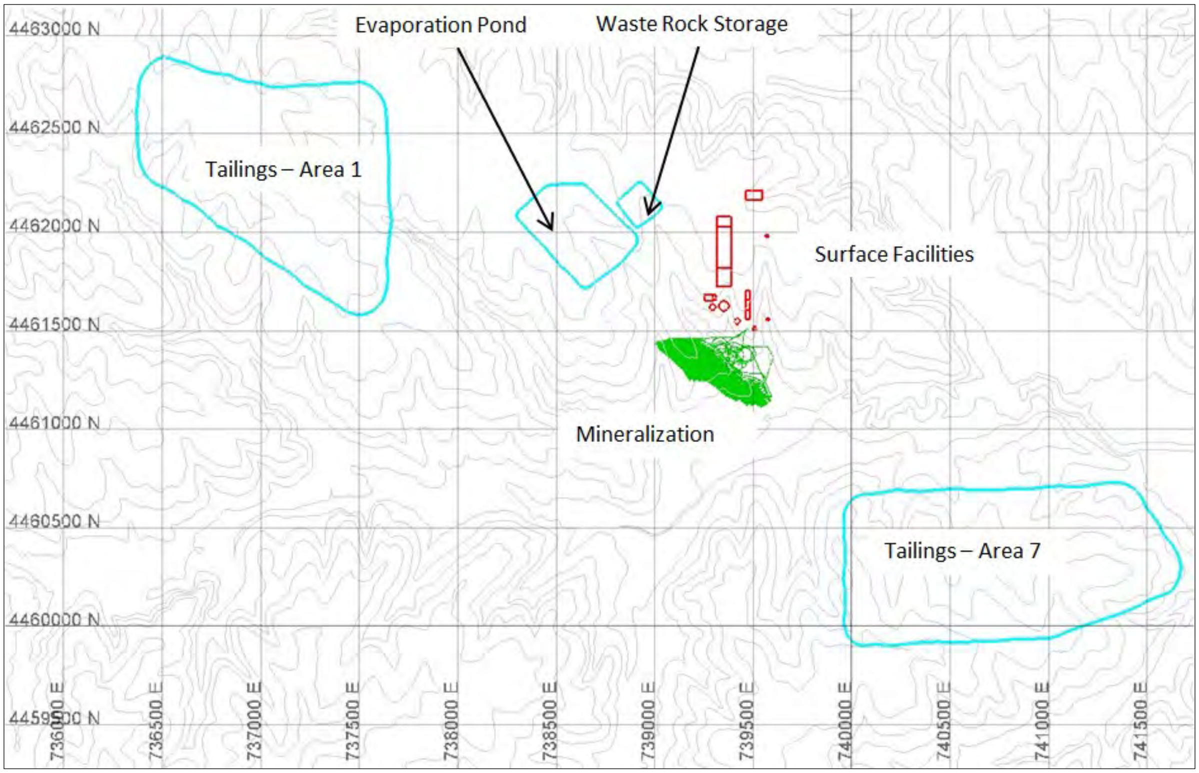
The stopes will be 15 m wide and stope length will vary based on mineralization grade. A spacing of 25 m between levels has been used. The deposit is mined in blocks where mining within a block occurs from bottom to top with the use of paste backfill. Sill pillars are left in situ between blocks. The backfill will have sufficient strength to allow for mining adjacent to filled stopes, thus eliminating the need for dip pillars.

The mine will be shaft access to minimize development through water bearing horizons. Mineralization will be transported from stopes to the shaft/hoist system by underground trucks. A single ventilation raise will serve as a dedicated exhaust raise and will be raisebored conventionally. Intake air will be down the shaft with fresh air entering a dedicated ventilation drift above the loading pocket. As levels are developed lower in the deposit short slot raises will be developed connecting levels for ventilation purposes.

The mine design process involved using stope optimization within Vulcan™ software to determine potentially mineable areas based on a CoG and minimum mining dimensions. Dilution and recovery were added to the designed tonnage to account for unplanned stope dilution and unrecoverable material within the stope.

NioCorp's current view on the marketability of FeNb is that approximately 7,500 t FeNb can be produced and sold per year. Based on metallurgical recoveries, estimated mine grades, and the 7,500 t FeNb target the process facility has been sized to 2,700 t/d. With this facility size an average grade of 0.80% Nb_2O_5 is targeted to produce the desired amount of product. A consistent average grade in the mine plan is achieved by varying the CoG for various levels of the mine.

Access and infrastructure development underground was designed to support the mining method and sized based on mining equipment and production rate requirements. Surface infrastructure and tailings were designed on lands which NioCorp has an option to purchase or which NioCorp is currently negotiating an option agreement with the landowner. The general layout of the mine and mill is shown in Figure 16.1.



Source: SRK, 2015

Figure 16.1: General Layout of Mine and Mill

16.1 Cut-off Grade Calculations

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. A factor of 0.699 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 testwork discussed in Section 13. The NSR was evaluated for each block in the 3-D geologic resource block model. Table 16.1.1 shows NSR parameters and an example NSR calculation for an individual block.

Table 16.1.1: Example NSR Block Calculation (1)

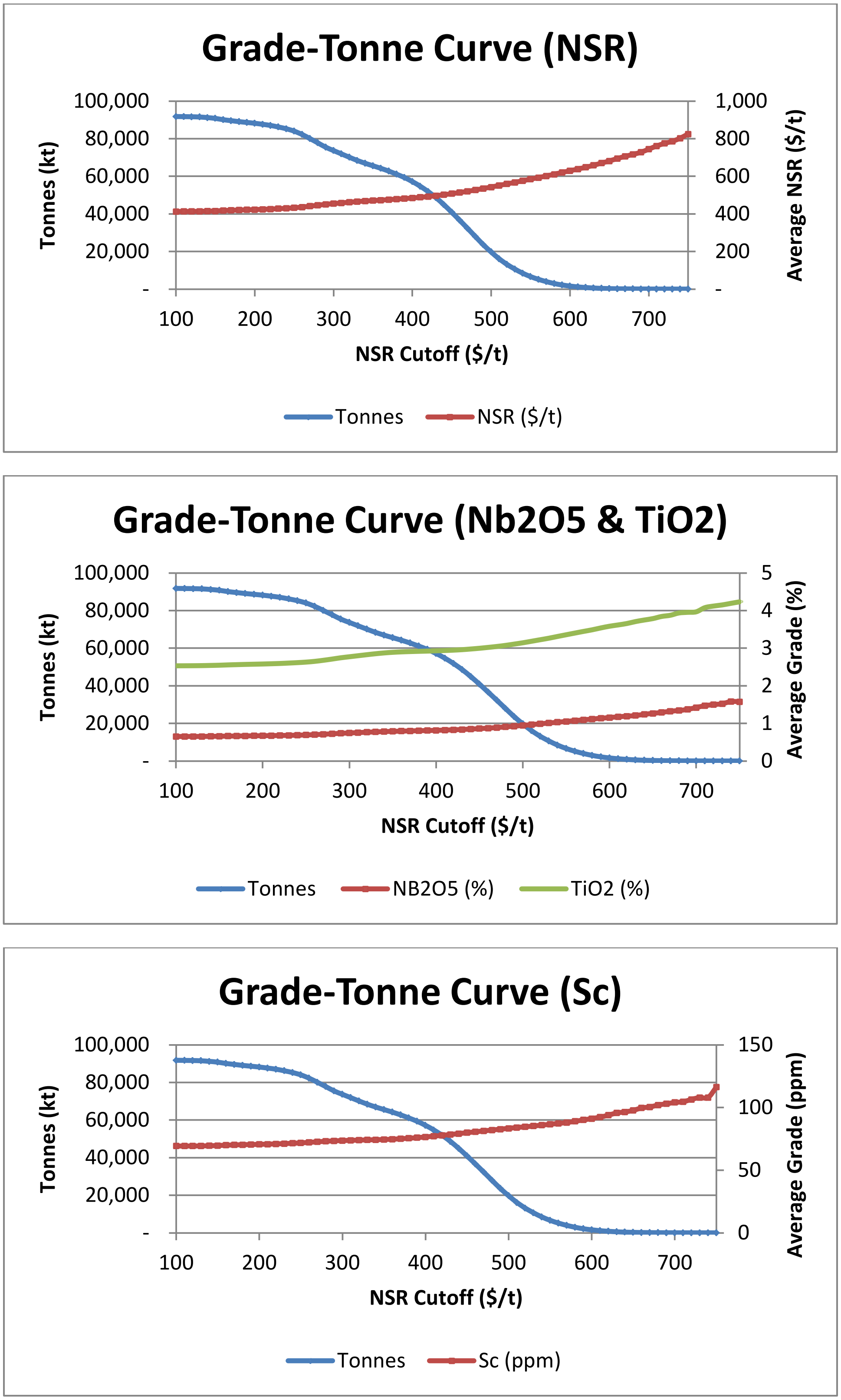
Input Parameters		Nb_2O_5	TiO_2	Sc (2)
From Block Model	100 t	0.70%	2.50%	60 ppm
Metallurgical Recoveries		89.2%	84.6%	90.0%
Payability		100.0%	99.0%	99.0%
Conversions from input grade to product		69.9%	100.0%	153.4%
Refining Charges		0	0	0
Price		US$44.00/kg	US$2.10/kg	US$2,000/kg
Calculate Contained Metal				
Nb_2O_5		700 kg		
TiO_2			2,500 kg	
Sc				6 kg
Calculate Saleable Metal – conversion to product, discount by recovery and payability				
FeNb (as Nb)		624.4 kg		
TiO_2			2,094 kg	
Sc (as Sc_2O_3)				8.20 kg
Calculate Block Dollar Value for Each Metal- subtracting refining charges				
FeNb		US$27,474		
TiO_2			US$4,397	
Sc				US$16,399
Total Block Value		US$48,270		
Block Value per tonne		US$482.70/t		

Source: SRK, 2015

(1) Values used here may differ from technical economic model.

(2) Stored as PPM in block model. Sc % = Sc ppm/10,000.

Figure 16.1.1 shows a grade-tonne curve for the deposit using various NSR CoGs. It includes only Measured and Indicated material and shows average grades for each grade variable. NioCorp has elected to not use Inferred material classification in this mine plan. All Inferred material is treated with zero grade.



Source: SRK, 2015

Figure 16.1.1: NioCorp Grade/Tonne Curves Based on NSR Cut-off

For mine design purposes a minimum CoG of US$180/t was used based on the estimated costs shown in Table 6.1.2.

Table 6.1.2: 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	**$180.00**

Source: SRK, 2015
(1) Includes backfill

16.2 Geotechnical

16.2.1 Geotechnical Characterization Program

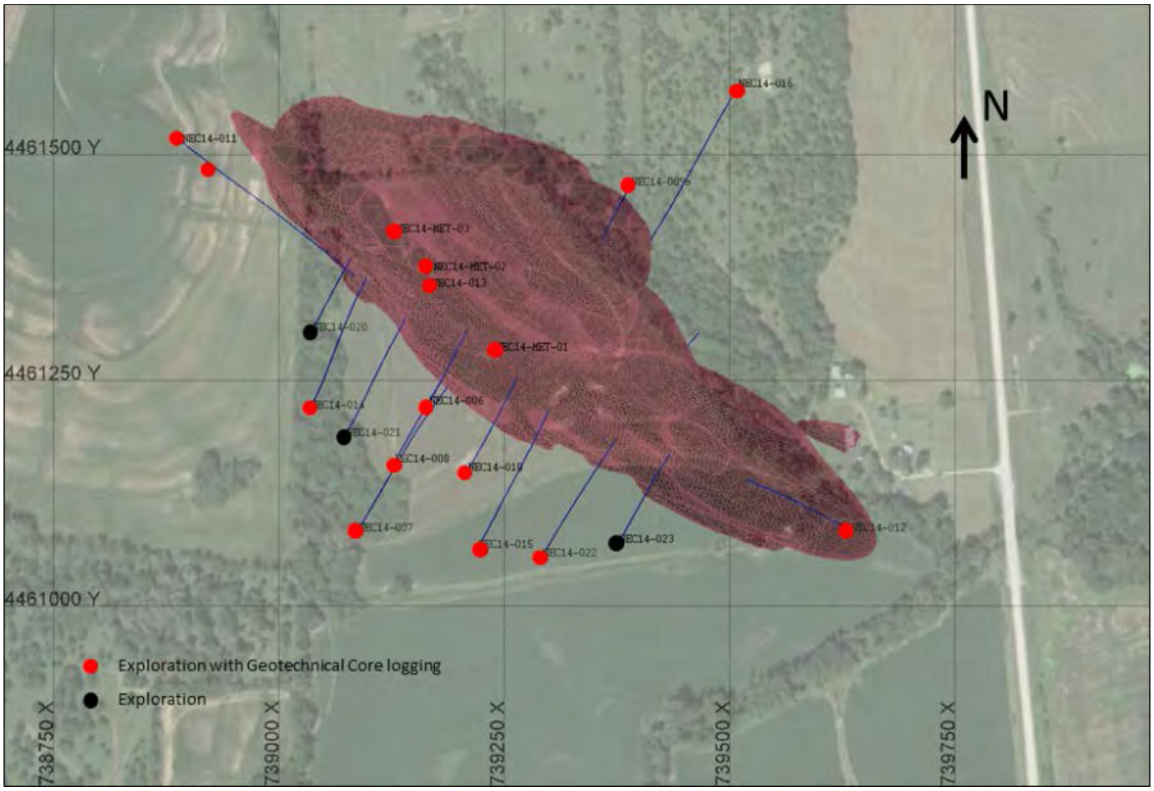
SRK has conducted a field geotechnical characterization program that included data collection, laboratory testing, and recommending geotechnical mine design parameters. The geotechnical characterization data collected for the TSF and mill infrastructure is documented in Section 18.

From May 21, 2014 to December 12, 2014 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-level design capable of meeting the requirements for Basic Engineering Design.

The geotechnical field investigation consisted of 18 drillholes (15,383 m) used for rock mass characterization, 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 within geotechnical conditions. Holes were drilled at varying orientations into the hangingwall, footwall, and mineralized rock. The field investigation included geophysical borehole logging of structural features, geotechnical core logging, core sample collection for laboratory strength testing, and in situ stress measurements. The location of the drillholes is shown on Figure 16.2.1.1.

Two recent drill holes were drilled in addition to the 18 characterization holes. These holes were drilled and cored along the centerline of the proposed production shaft location and ventilation hole location. The core was logged for geological and geotechnical characteristics, but analysis of the data has not been conducted to date. These holes were not considered as part of the PEA study.

The geotechnical investigation included a total of 12,986 m of acoustic televiewer scans where 9,345 m were successfully interpreted giving an overall televiewer interpretation of 72%. The rock testing program included 53 unconfined compression (UCS) tests, 17 triaxial compression (TCS) test, and 32 direct shear strength (DSS) tests of rock joint sampling. Tests were conducted at different confinement levels (80 tests). A set of 25 static and dynamic elastic moduli measurements, and 13 Brazilian tensile strength (BTS) tests were conducted. This information was used for calibration of the 1,992 point load tests conducted in the field and the 14,400 m of field estimated strength parameters estimated during the core logging. The laboratory tests were sufficient to develop discontinuity shear strength parameters and estimates of the static and dynamics elastic constants.



Shaft and Ventilation holes locations are not shown.
Source: SRK, 2015

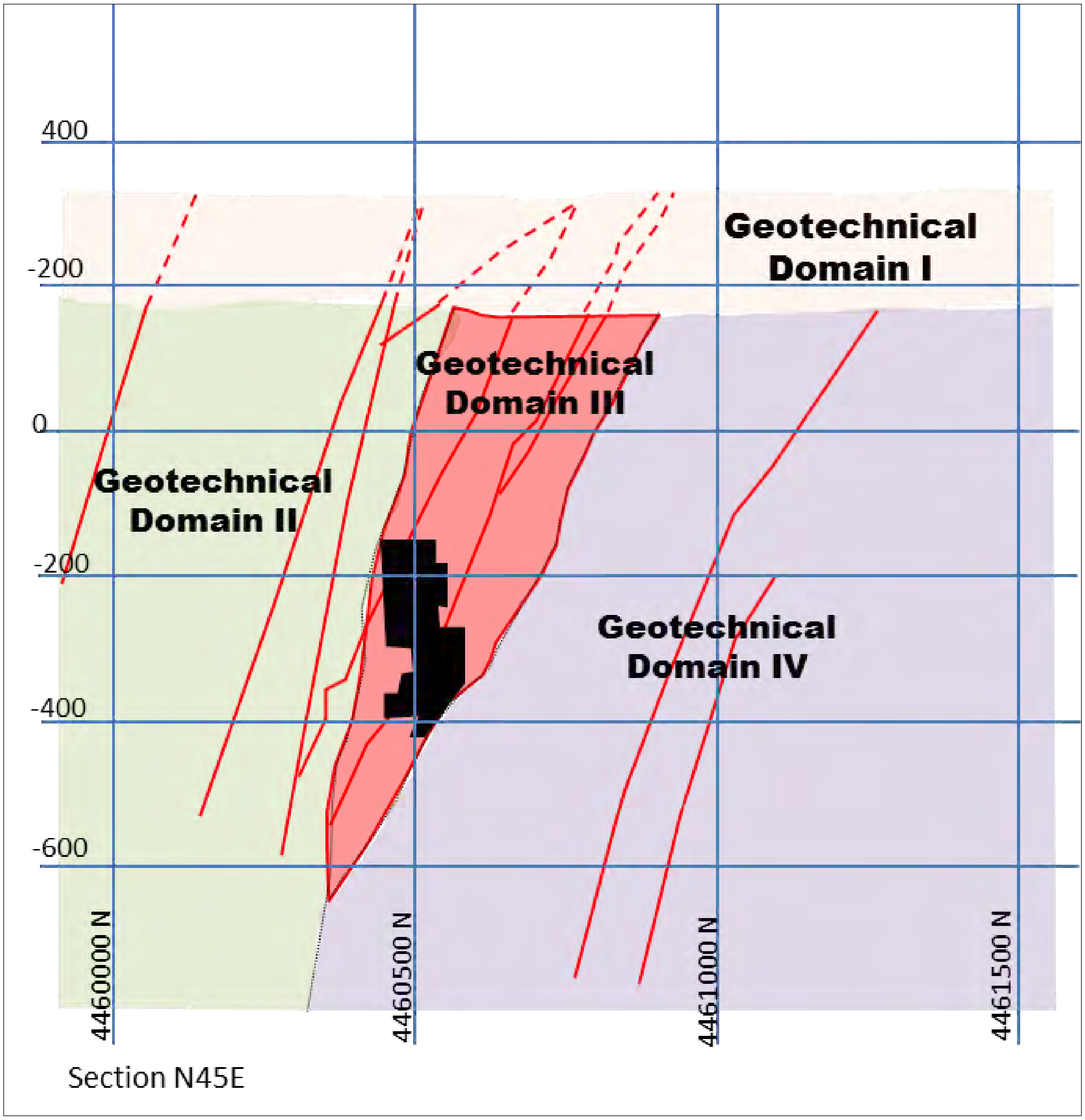
Figure 16.2.1.1: Location of 2014 Geotechnical Drillholes

16.2.2 Geotechnical Domains

Four 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;
- Hangingwall material to southwest of the mineralization;
- Mineralized carbonatite; and
- Footwall material to northeast of the mineralization.

The domains, shown on Figure 16.2.2.1, were delimited based on intact rock properties and in situ rock mass quality from characterization logging. Characterization was based on the Rock Mass Rating (RMR) (Bieniawski, 1976) and the Q-system (Barton, 1974). These value were then used with empirical design methods to assess the basic inputs for underground mine design.

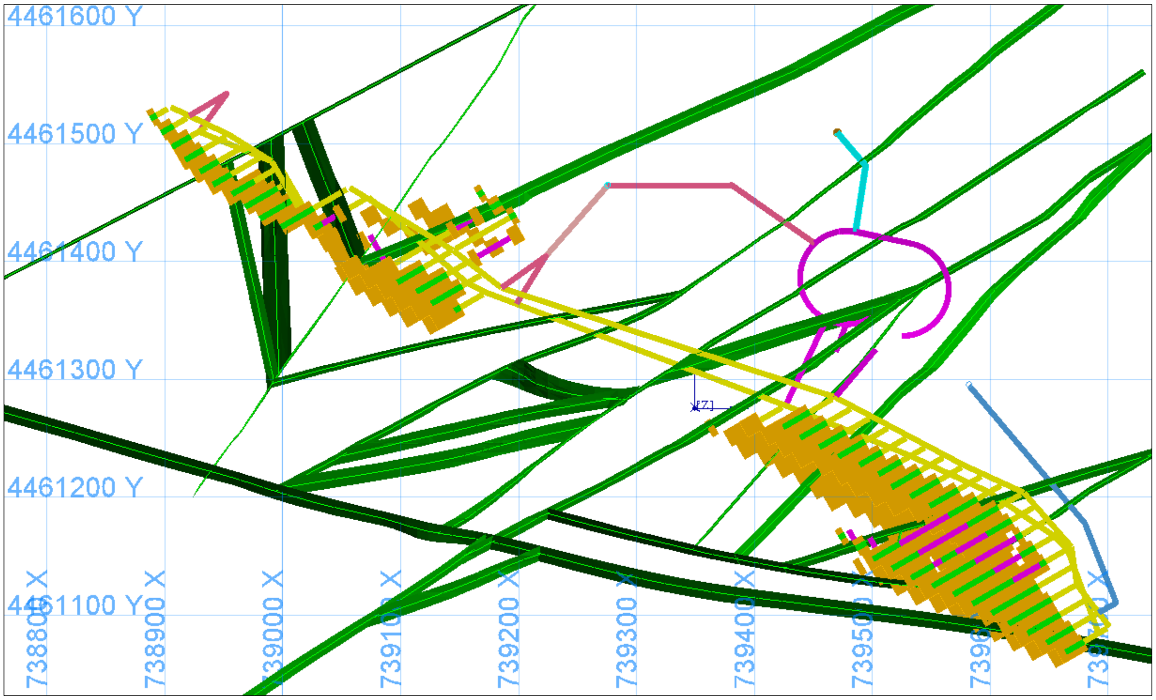


Source: SRK, 2015

Figure 16.2.2.1: Geotechnical Model, Vertical Cross Section (N45°E Section)

16.2.3 Structural Geology

The regional structural geology and the borehole logging data have been used to estimate the mine-scale structural geology. A total of 31 major structures have been identified. Figure 16.2.3.1 shows a plan view section with the position of stopes and footwall accesses relative to the geologic structures on the +60 m elevation level.



Source: SRK, 2015

Figure 16.2.3.1: Plan View of Geologic Structures (Green),+60 m Elevation

16.2.4 Rock Mass Properties

The laboratory testing and characterization data has been analyzed by domain and statistical ranges of values have been estimated. Table 16.2.4.1 shows a summary of the rock mass properties by domain.

In addition to the geotechnical core logging, acoustic televiewer data was used to establish the structural domains and for preparation of the major structural model. For each domain, structural sets were identified based on orientation and discontinuity type. Table 16.2.4.2 shows a summary of the discontinuity orientation by domain.

The Pennsylvania, domain I, is controlled by sub-horizontal structural sets, dipping between 0° and 18° mostly in the NE and SW direction, in combination with a sub vertical joint set, between 66° and 90° in the NW and SE direction.

The Hangingwall, domain II, joint set shows a joint set, dipping between 48° and 80° to SE, in combination with two bedding sets. The principal set dips between 3° and 39° in the NE direction and the secondary set dips between 30° and 60° in the SW direction.

The mineralized carbonatite, domain III, is characterized by one conjugate joint set (10°/166° and 86°/360°) and three defects logged as bedding sets. The principal bedding set is dipping between 25° and 75° to SE and the secondary set dips to the SW.

The footwall, domain IV, is controlled by two joint sets where the principal set is sub vertical and dipping SE (42°/138°±15°) and a secondary joint set dipping to the SW (66°/245°±10°). Both joint sets combine with two bedding sets dipping to the SE (30°/118°±19°) and SW (48°/234°±20°).

Based on joint spacing descriptions in the geotechnical logs and using the approach by Palmström (1995,1996), volumetric joint and block sizes have been estimated for each domain. The results, as follows, are variable depending on the fracture frequency and the degree of weathering.

- Domain I - approximately 11-12 joints/m^3;
- Domain II- approximately 8.5 and 15 joints/m^3;
- Domain III - approximately 9 to 13 joints/m^3; and
- Domain IV - approximately 15 to 20 joints/m^3.

This information is used to estimate ground support requirements and potential stope dilution.

Table 16.2.4.1: Summary of Rock Mass Characterization by Domain

Domain	Distribution (%)	Density (t/m^3)	UCS (MPa)	RQD (%)	Fracture Frequency (FF/m)	RMR_{76} /GSI	Q'
Domain I Pennsylvania	Limestone 48%	2.43-3.28 (2.86*)	62-84	66-90	2.9 – 3.9	54-74	77-103
	Mudstone 52%	2.43-3.28 (2.85*)	25-35	78-100	6.8 – 9.2	60-82	60-80
Domain II Hangingwall	Fresh 49%	2.68-3.62 (3.15*)	53-71	83-100	1.3-1.8	54-56	30-40
	Moderated Weathered 41%		36-48	75-100	3.6-4.8	43-57	27-37
	Highly weathered 10%	-	28-38	68-90	3.1-4.1	34-46	12-16
Domain III Mineralized Carbonatite Wall	Fresh 70%	2.82-3.10 (3.32*)	62-83	83-100	1.3-1.7	54-72	55-75
	Moderated Weathered 20%		39-53	81-100	2.6-2.2	46-62	37-50
	Highly weathered 10%	-	31-42	64-87	5.1-6.9	-	-
Domain IV Footwall	Fresh 46%	2.6-3.10 (3.05*)	84-112	84-100	0.5-0.7	59-79	65-85
	Moderated Weathered 51%		47-64	82-100	0.9-1.2	49-67	52-70
	Highly weathered 3%	-	-	-	-	-	-

*Average
Source: SRK, 2015

Table 16.2.4.2: Summary of Discontinuity Characterization by Domain

Domain	Type	Set	No of Defects	DIP (°)	DIPDIR (°)	VL (°)(68%)
Domain I Pennsylvania	Joint	JN 1A	30	83	150	10
		JN 1B	37	76	320	10
	Bedding	BD 1A	1067	6	061	10
		BD 1B	1291	10	249	8
Domain II Hangingwall	Joint	JN 1	660	64	154	16
		-		-	-	-
	Bedding	BD 1	211	21	50	18
		BD 2	193	55	190	25
Domain III Mineralized Carbonatite	Joint	JN 1	620	70	166	19
		JN 2	70	86	360	11
	Bedding	BD 1	110	50	135	25
		BD 2	60	11	250	25
		BD 3	32	65	220	25
Domain IV Foot Wall	Joint	JN 1	227	42	138	15
		JN 2	176	66	245	10
	Bedding	BD 1	26	30	118	19
		BD 2	19	48	234	20

Source: SRK, 2015

16.2.5 Rock Mass Quality

The geotechnical domains show most of the material to be fresh or moderately weathered with a RMR between 55 and 80. This translates to a Class II rock, considered good rock, and is described as blocky with fair/good joint surfaces, fresh to moderately weathered, planar rough and planar smooth in condition.

Other areas identified as moderately weathered have an RMR between 40 and 60. This translates to a Class III rock, considered fair rock, and is described as blocky with fair joint surfaces.

Core logs indicate that rock masses described as highly weathered are associated with major faults. These zones have an RMR of approximately 30 to 40. This translates to Class IV rock, considered poor rock, and is described as very blocky with fair and poor joint surfaces.

Table 16.2.4.1 includes the RMR_{76} values for each domain. The RMR_{76} values are the same as the Geologic Strength Index (GSI) used in Hoek-Brown strength criteria (Hoek and Brown, 2008).

16.2.6 Pre-Mining Stresses

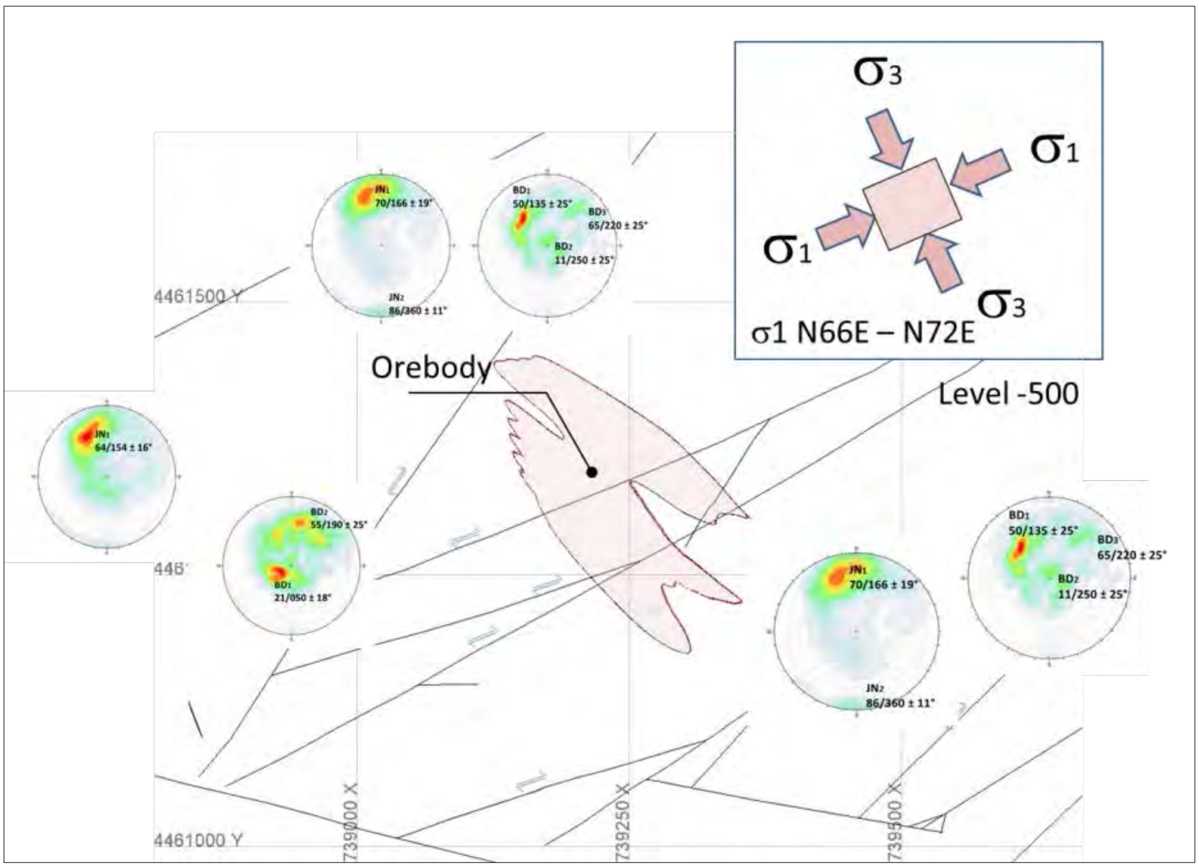
In September 2014, Agapito Associates, Inc. (AAI, 2014) performed downhole in situ stress testing at the site. The purpose of the work was to estimate the in situ horizontal stress field in the un-mineralized Pennsylvania rock (surface to 200 m depth) and the mineralized carbonatite zone (below 200 m). A total of thirteen tests were attempted, eight of which were successful. The results of the study concluded the following:

- There is an apparent increase of stress with depth of approximately 36 kPa/m for the major stress (σ_H) and 21 kPa/m for the minor stress(σ_h);
- The major stress (σ_H) is approximately 20% greater than the vertical stress (σ_V) and the minor stress (σ_h) is 71% of the vertical stress(σ_V);

- Both the major and minor stresses are approximately 66% lower than values predicted from the database of US-Canada non-coal sites;
- The average orientation of the major stress is N 66° E; however, a calculation using selected overcores provides an estimate of N 72° E; and
- The orientation of the principal stresses is well correlated with the orientation of the major fault structures, validating the major fault model.

SRK assumes that the major to minor stress ratio is 1.5 with the minor stress being the vertical stress.

Figure 16.2.6.1 shows a summary of the major and minor stress orientation relative to the fracture set and fault structure orientations.



Source: SRK, 2015

Figure 16.2.6.1: Principal Stress Orientation Relative to Major Geologic Structures and Discontinuities

16.2.7 Seismicity

A high-level assessment of the local seismic earthquake potential suggests that the local peak ground acceleration (PGA) of 0.02g for a 50 year return earthquake event.

These values are taken from the International Building Code, commonly used for mine applications. The source of the peak ground acceleration is the 2002 USGS, Interactive National Seismic Hazard

Map (Frankel et al, 2003). 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.

16.2.8 Underground Geotechnical Mine Design Parameters

Stope Dimensions

For the purposes of mine design the stopes have been oriented at an orientation of N60°E to create the most favorable ground conditions during mining. The orientation considers the major geologic structures, the local discontinuities and the principal stress orientations. The resource block model is also oriented along this same direction. Figure 16.2.8.1 shows the terminology used to describe the stope dimensions.

The sizing of stopes has been based on an empirical design method (Potvin, et. al. 1988). 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. Figure 16.2.8.2 shows a plot of the stability chart at three representative depths (400 m, 600 m, and 800 m) below ground and three representative rock qualities (fresh, moderately weathered and highly weathered). The chart includes points for the stope back, the hangingwall stope face, and the stope sidewall. The selected stope sizes used to compute the hydraulic radius are:

- Width 15 m;
- Height 25 m; and
- Length 100 m.

The figure indicates that these stope dimensions should remain stable when fully open and emptied of material. SRK notes that in weaker ground areas the stope lengths may need to be reduced depending on ground conditions. This should affect only a small percentage of stopes.

Sills and Mining Sequence

The longhole open stoping mining method is based on overhand mining so the deposit has been divided into three blocks where mining starts at the bottom of block 1 mining upwards for seven levels (i.e., top of the deposit). Mining then continues from the bottom of block 2 and mines the five levels upward leaving a sill between Blocks 1 and 2. Mining continues in this way for block 3. Based on tributary stresses, and preliminary numerical analyses, a 5 m high sill has been included in the design. This analysis assumes the mining rate is slowed for stopes immediately beneath the sill, stope sizes are reduced to account for increased stresses, and that cemented pastefill is used in all the stopes below the sill level. There may be an opportunity to increase stope lengths with additional stability analysis as the design is advanced.

The strategy for mining stopes is to use a primary/secondary stoping sequence that allows a gradual stress transfer on a given mining level. Optimal stoping sequence on a level would be from the center of the deposit out toward the abutments to push induced mining stresses away from active haulage/access areas.

Infrastructure Setback Distances

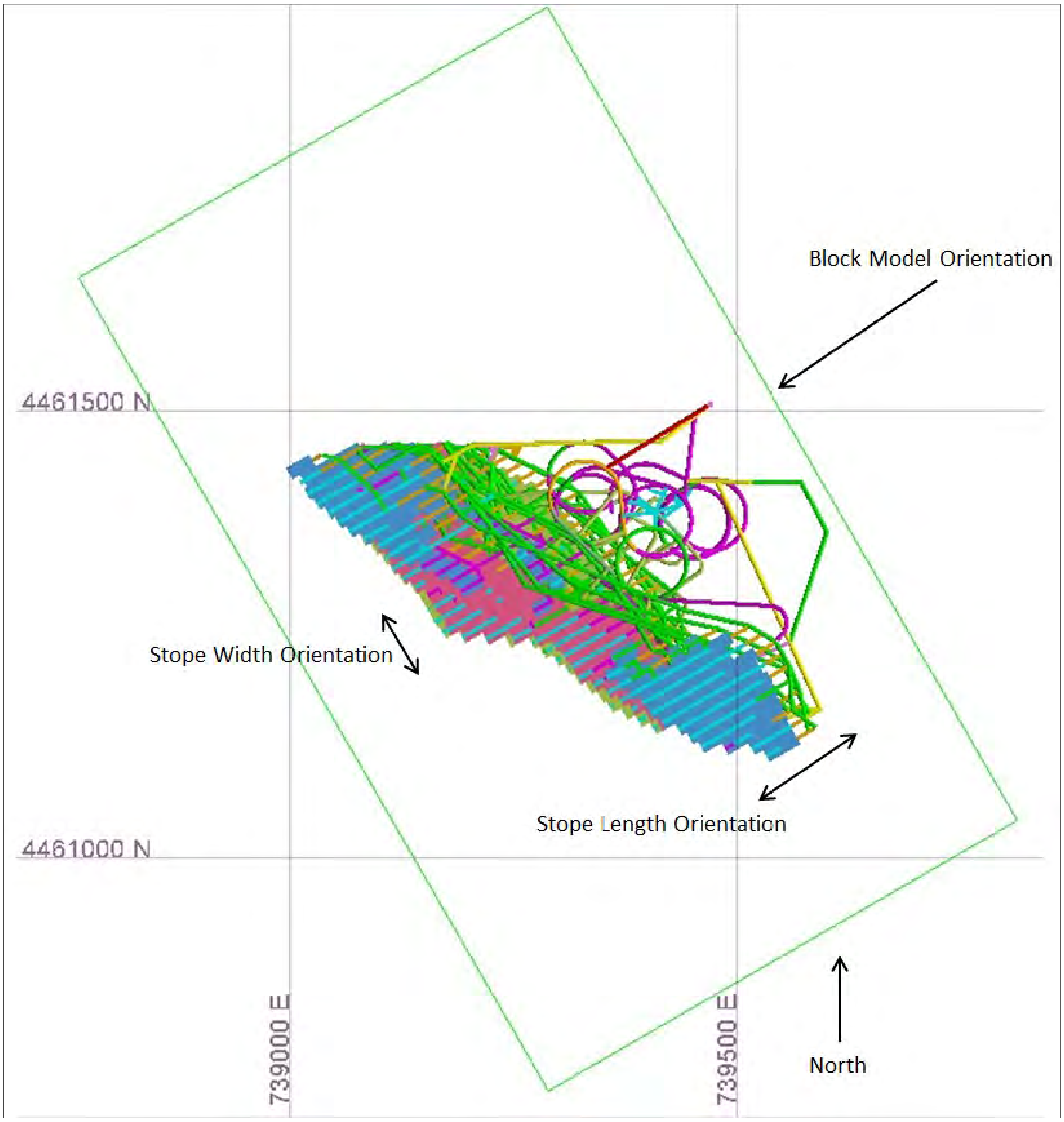
To minimize mining-induced damages to long-term drifts the setback distances used in the design are:

- Haulage setback: 25 m from stopes; and
- Main ramp setback: 75 m from stopes.

The setback distances are shown on Figure 16.2.8.4.

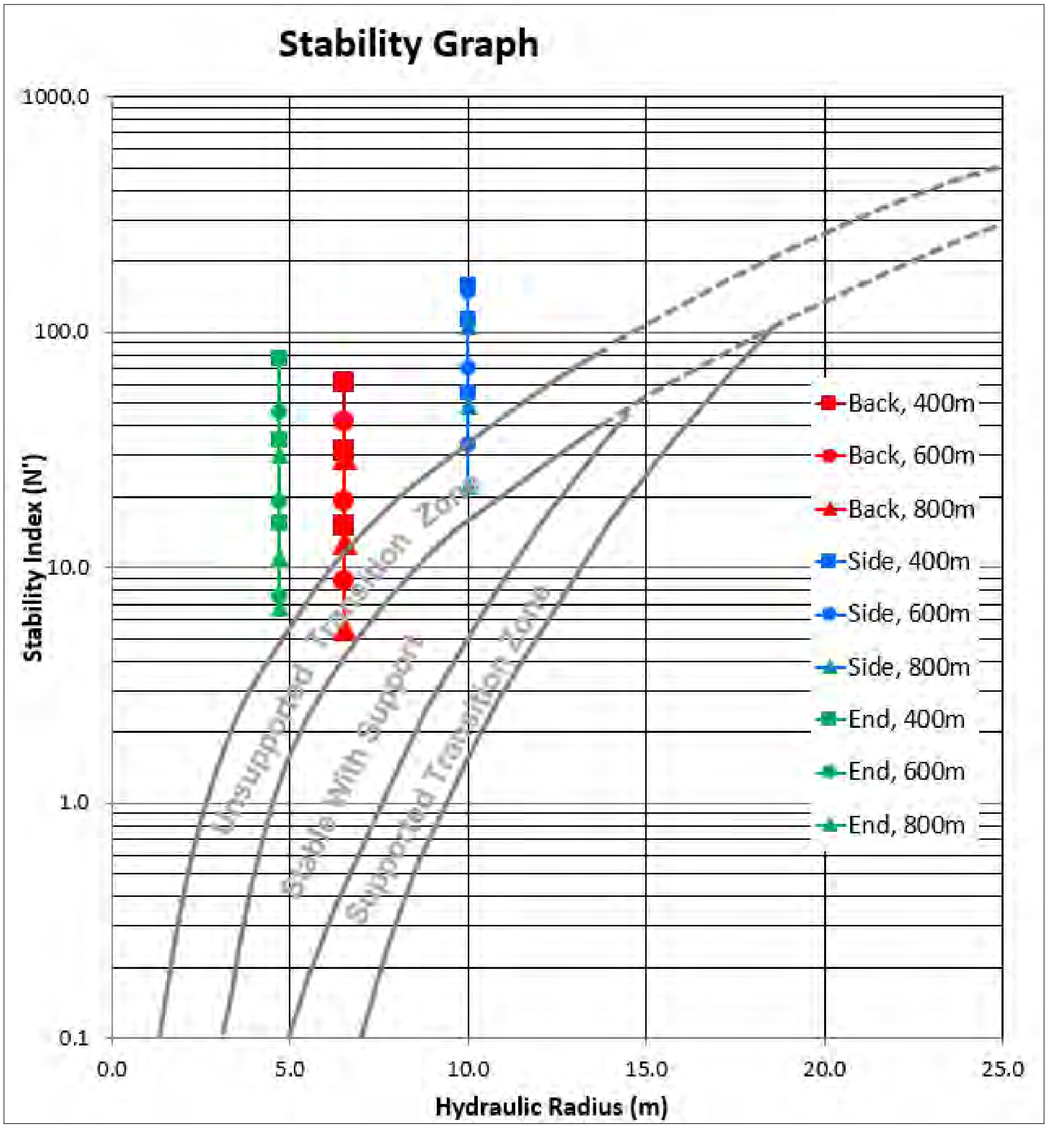
Backfill Requirements

The mining method requires that most of the stopes be backfilled. The backfill material will be a paste fill made of fly ash and sand. The primary/secondary extraction sequence requires that the primaries be backfilled with cemented pastefill having a minimum 14 day UCS strength of 1.0 MPa for single face fill exposure during mining of the adjacent secondary stope in high stress conditions. In lower stress conditions the minimum 14 day UCS strength can be 0.5 MPa. In secondary stopes where the backfill will never be exposed only sufficient binder is required to prevent liquefaction of the backfill during mining operations.



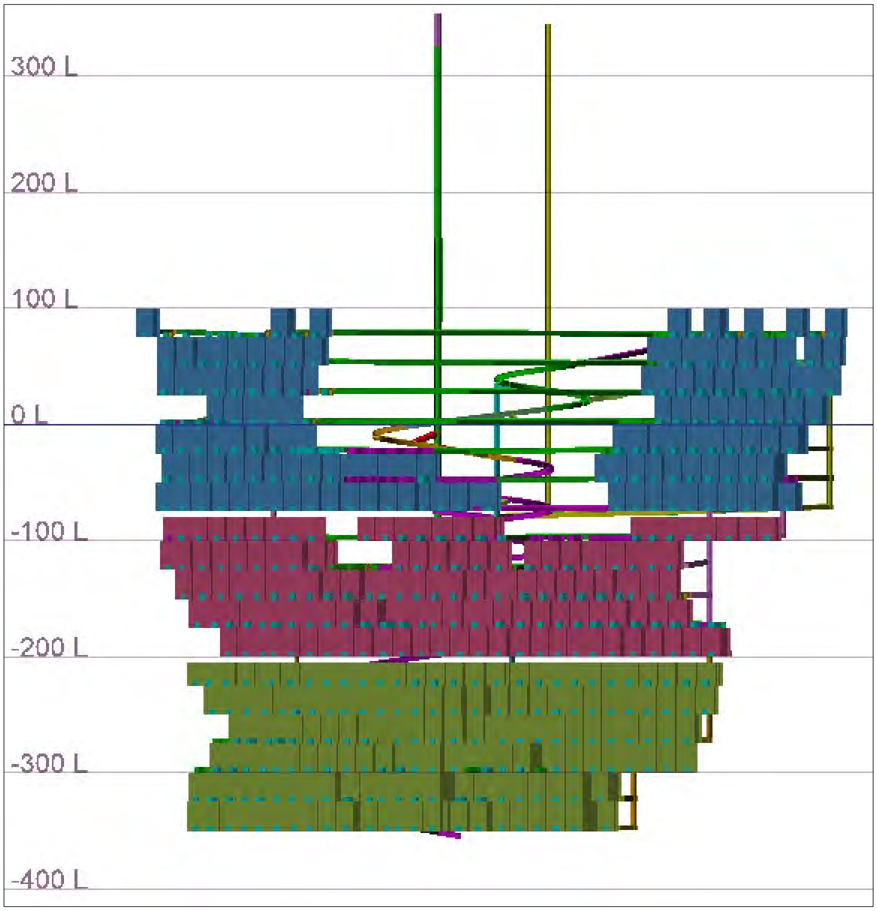
Source: SRK, 2015

Figure 16.2.8.1: Block Model and Stope Orientation



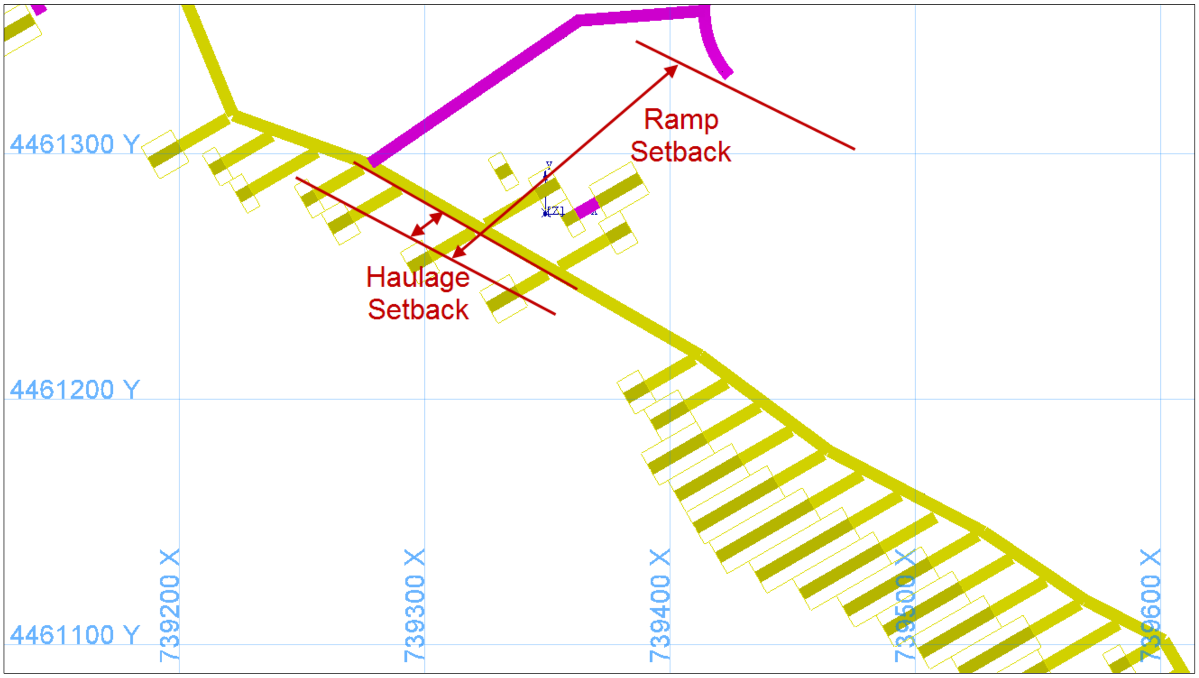
Source: SRK, 2015

Figure 16.2.8.2: Stability Chart of Rock Quality Stability Index Versus Hydraulic Radius at Three Representative Depths and Three Representative Rock Qualities.



Source: SRK, 2015

Figure 16.2.8.3: Location of Planned Sills Delimiting Mining Blocks



Source: SRK, 2015

Figure 16.2.8.4: Setback Distances for Haulages and Main Ramp

Rock Mechanics Description Around the Shaft

A total of five geotechnical units, as shown in Table 16.2.8.1, are expected to be crossed during shaft construction.

Table 16.2.8.1: Percent of Each Material Encountered Along Shaft

Geotechnical Unit		Total Material Intersected
Sediments		5%
Pennsylvania		40%
Foot Wall	Highly Weathered	0%
	Moderatly Weathered	15%
	Fresh	40%

Source: SRK, 2015

These geotechnical domains, were characterized using the Bieniawski (1976) rock mass rating and the Barton's Q-system (1974), to provide the basic inputs for mine designs. Tables 16.2.8.2 and 16.2.8.3 summarize the geotechnical parameters of the geotechnical units to be intersected during shaft construction.

Figure 16.2.8.5, shows a cross section of the shaft, indicating the depth and the geometry of each geotechnical domain that would be disturbed by the construction of this facility. This figure is based on the 2014 geotechnical drill characterization and does not consider data from the recent 2015 shaft and ventilation drill holes. The geotechnical domains should be updated to consider the 2015 data for the feasibility level design.

Table 16.2.8.2: Geotechnical Domain I Pennsylvania

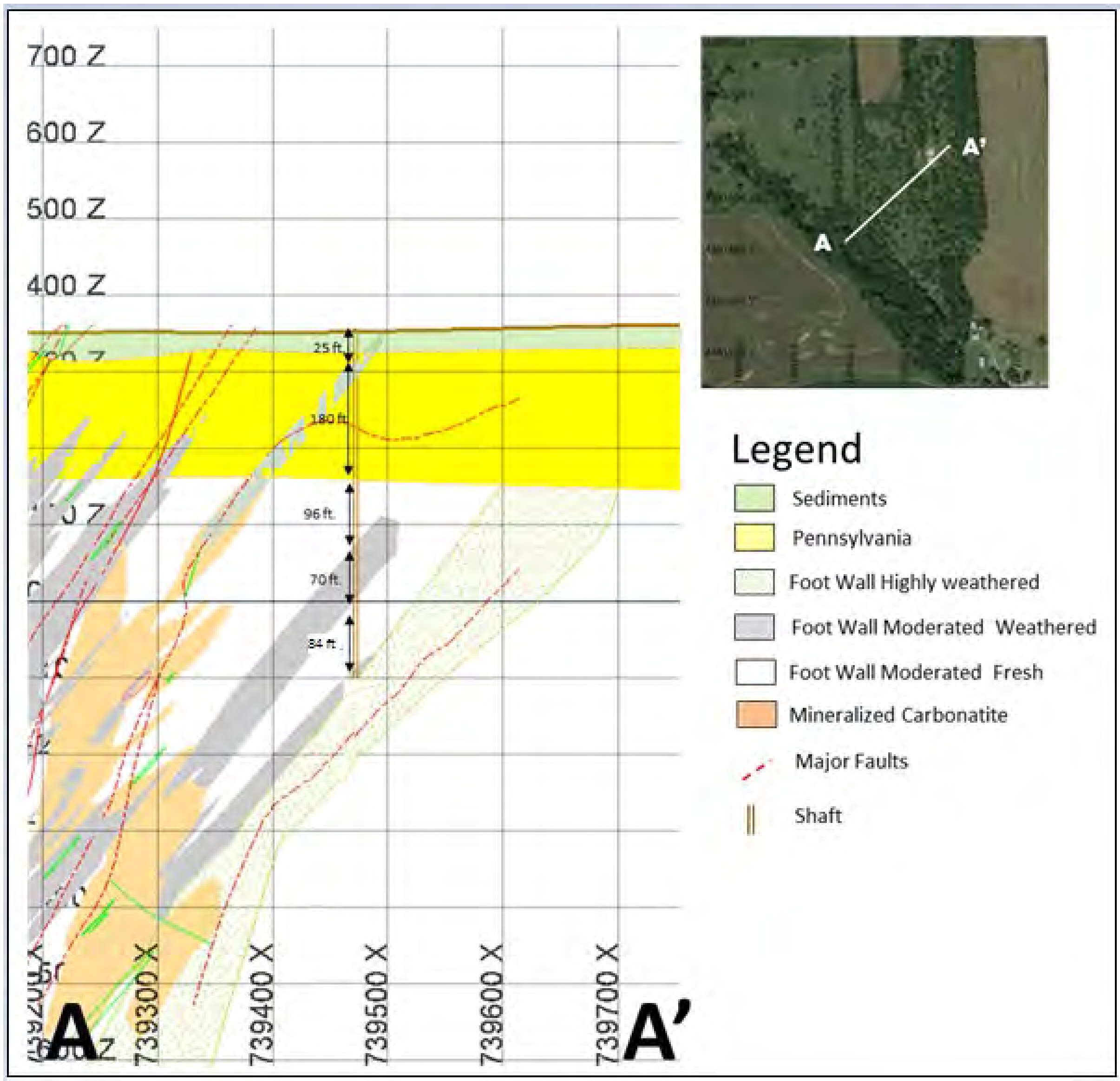
Domain	Unit	Geotechnical Parameter	Mean ± SD	Range True mean
Domain I Pennsylvania	Mudstone	Fracture Frequency (ff/m)	3.44 ± 12.46	2.92 – 3.95
		Spacing (m)	0.47 ± 0.68	0.40 – 0.54
		No. of sets	3	
		RQD	92± 18	78-100
		UCS (MPa)	29±16	25-35
		st (MPa)	-3	
		JRC	1.71± 3.6	1.46 - 1.97
		JCS	22	
		Jn	2.14 ± 2.75	1.82 – 2.45
		Jr	0.72 ± 0.82	0.61 – 0.83
		Ja	1.26 ± 1.21	1.07-1.44
		Jw	2	Slight
		Jv (joint/m^3)	11.82 ± 12.36	10.06 – 13.58
		Block size (m^3)	0.29 ± 0.12	0.25 – 0.34
		RMR_{76} = GSI	64.1± 17.2	54-74
		Q’	70± 55	60-80
	Limestone	Fracture Frequency (ff/m)	8.05 ± 8.77	6.85 – 9.25
		Spacing (m)	0.38 ± 0.58	0.33 – 0.44
		No. of sets	3	
		RQD	78± 27	66-90
		Field estimated strength (MPa)	64± 14	54 - 74
		sc (MPa)	55±33	62-84
		st (MPa)	-6	
		JRC	4.86 ± 6.88	4.14 – 5.59
		JCS	41	
		Jn	3.74 ± 2.57	3.18 – 4.29
		Jr	1.37 ± 1.29	1.16 – 1.57
		Ja	1.45 ± 0.73	1.23 – 1.66
		Jw	2	Slight
		Jv (joint/m^3)	11.08 ± 11.85	9.43 – 12.73
		Block size (m^3)	0.30 ± 0.10	0.26 – 0.35
		RMR_{76} = GSI	71.35± 12.36	60-82
		Q’	89.66± 50	77-103

Source: SRK, 2015

Table 16.2.8.3: Geotechnical Domain IV - Footwall

Domain	Unit	Geotechnical Parameter	Mean ± SD	Range True mean
Domain IV Foot Wall	Fresh	Fracture Frequency (ff/m)	0.60 ± 0.53	0.51 – 0.69
		Spacing (m)	1.08 ± 1.13	0.92 ± 1.25
		No. of sets	4	
		RQD	98± 14	84-100
		UCS (MPa)	98± 36	84-112
		σt (MPa)	-10	
		JRC	5.57 ± 6.56	4.74 – 6.40
		JCS	74	
		Jn	1.65 ± 1.02	1.41 – 1.90
		Jr	1.68 ± 0.92	1.43 – 1.92
		Ja	1.28 ± 0.90	1.09 – 1.47
		Jw	3	Moderate
		Jv (joint/m^3)	15.03 ± 14.01	12.79 – 17.27
		Block size (m^3)	0.30 ± 0.12	0.26 - 035
		RMR76 = GSI	69 ± 9.5	59-79
		Q'	76± 56	65-87
	Moderately Weathered	Fracture Frequency (ff/m)		
		Spacing (m)	1.03 ± 1.01	0.87 – 1.18
		No. of sets	4	
		RQD	96± 14	82-100
		UCS (MPa)	55± 17	47-64
		σt (MPa)		
		JRC	7.59 ± 4.81	6.46 – 8.72
		JCS	27.5	
		Jn	2.57 ± 1.63	2.19 – 2.95
		Jr	2.06 ± 0.64	1.75 – 2.36
		Ja	1.74 ± 0.71	1.48 – 1.99
		Jw	3	Moderate
		Jv (joint/m^3)	23.17 ± 14.4	19.72 – 26.62
		Block size (m^3)	0.23 ± 0.15	0.20 – 0.27
		RMR76 = GSI	58± 7	49-67
		Q'	61± 40	52-70
	Highly Weathered	Fracture Frequency (ff/m)	0.98 ± 0.00	0.83 – 1.13
		Spacing (m)	1.02 ± 0.00	0.87 – 1.17
		No. of sets	4	
		RQD	<25	<25
		UCS (MPa)	36 ±15	31-42
		σt (MPa)		
		JRC	2.50 ± 0.00	2.13 – 2.87
		JCS	(*)	(*)
		Jn	2.00 ± 1.41	1.70 – 2.30
		Jr	1.50 ± 0.00	1.28 – 1.72
		Ja	(*)	(*)
		Jw	3	Moderate
		Jv (joint/m^3)	14.86 ± 17.05	12.65 – 17.07
		Block size (m^3)	0.21 ± 0.13	0.18 – 0.24
		RMR76 = GSI	(*)	(*)
		Q'	(*)	(*)

Source: SRK, 2015



Source: SRK, 2015

Figure 16.2.8.5: Cross section Shaft Location and Geotechnical Units

16.3 Hydrogeology

The hydrogeology of the deposit is characterized based on three phases of work:

1. The first phase of hydrogeologic 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 coreholes;
    - Installation of six 2 inch PVC standpipe piezometers isolated in the carbonatite and open to large intervals of the deposit;

- Installation of two nominal 2 inch PVC standpipe piezometers isolated in the 180 m thick Pennsylvanian aquitard above the carbonatite; and
- Frequent measurement of water levels in open coreholes and piezometers over a period of six months.

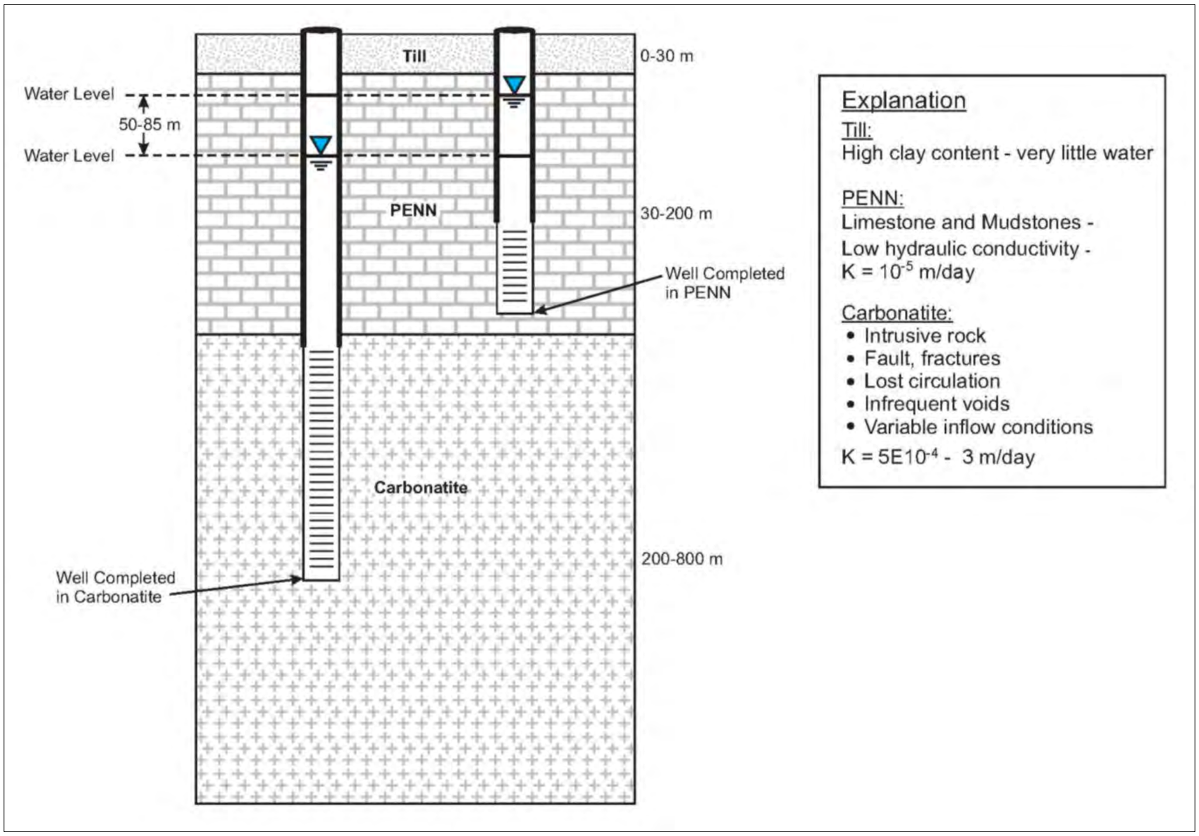
2. Following the second phase of resource-related core drilling, an 11 day airlift pumping test was completed using a deep, open, vertical PQ corehole as a pumping well. Water levels from the surrounding piezometers were recorded over the duration of the test and for several weeks following the test.
3. The third phase of hydrogeologic characterization involved installation of two multi-level, distally-located piezometers and a deep 6 inch diameter injection well completed to depths of 850 m, followed by completion 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 deposit. Water from Elk and Todd Creeks was injected at rates of between 22 and 30 L/sec (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 eight weeks following the test.

Conceptual Hydrogeology

Analysis and interpretation of the data from the testing program has been completed and the following preliminary conceptual model describes the site hydrogeology.

The upper 30 m of lithology is comprised of glacial till, underlain by a 170 to 180 m 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 beneath the glacial till and above the carbonatite in which the deposit is hosted. Testing in the PENN indicates the horizontal hydraulic conductivity of this unit is approximately 5×10^{-5} m/day. In addition, the PENN is highly stratified and is likely highly anisotropic, whereby the vertical hydraulic conductivity is several orders of magnitude lower than the horizontal. There is a vertically-downward hydraulic gradient across the PENN, resulting in very gradual leakage of water into the carbonatite. The carbonatite intrusive underlies the PENN to unknown depths. The rock within the mineralized zone is fractured and faulted, while the carbonatite beyond the immediate limits of the mineralized zone is likely to be less permeable. At a radial distance of about 4 km, granite county rock hosts the carbonatite. The hydrogeologic nature of the granite has not been well characterized, but results from the recent injection test indicate it is less permeable than the carbonatite and may act as a partial boundary to groundwater flow.

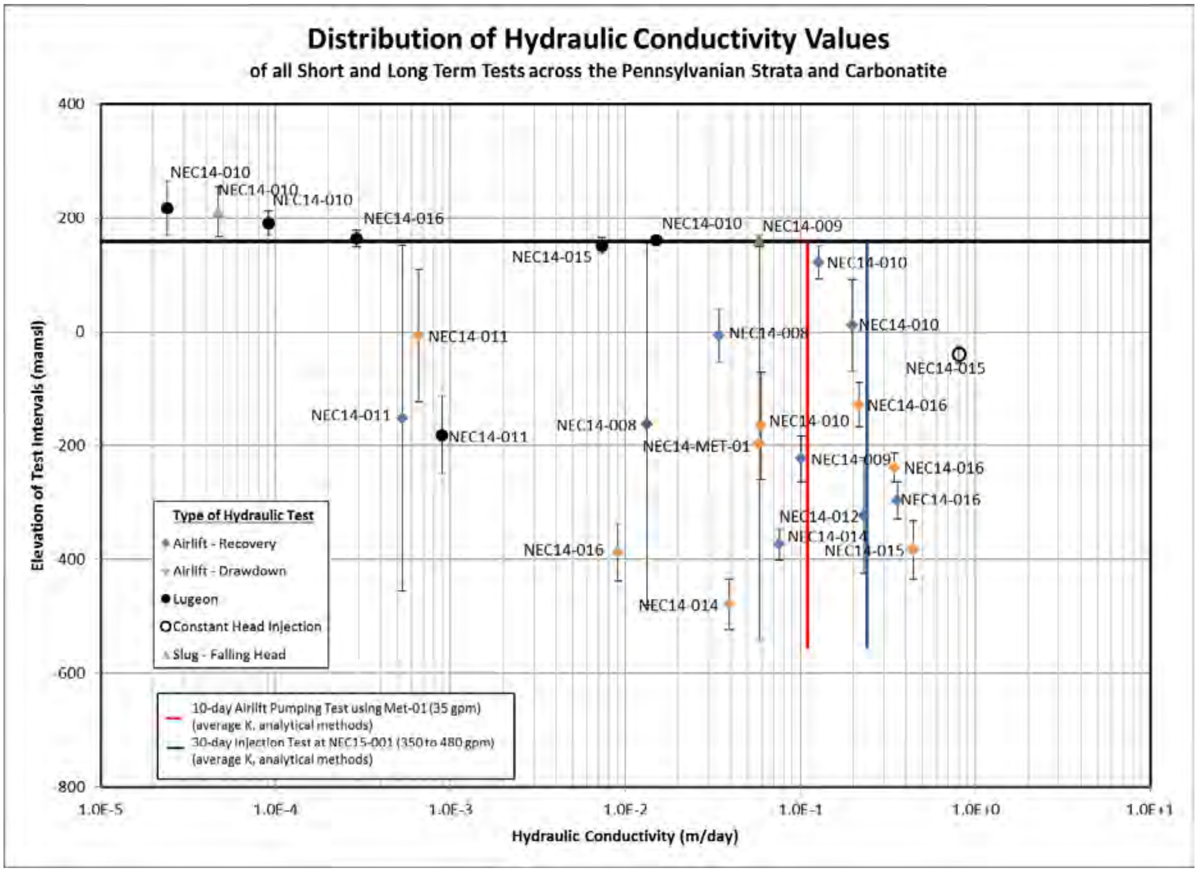
Water levels in wells completed in the carbonatite are consistently at about 100 m below ground surface, or approximately 251 m above mean sea level (mamsl); water levels in the wells completed in the PENN are 50 to 85 m higher than water levels in the carbonatite. This indicates that the groundwater system in the carbonatite is confined from above. Groundwater levels and pertinent hydrogeologic information are presented in Figure 16.3.1.



Source: SRK, 2015

Figure 16.3.1: Summary of Water Levels and Lithology

Testing in the carbonatite and subsequent analysis using an analytical method has indicated average hydraulic conductivities as high as 0.2 m/day to depths of 830 m below the surface. Sub-vertical faults have been mapped within the carbonatite proximal to the deposit, and these faults contribute to the elevated bulk hydraulic conductivity values generated from the testing program. A summary plot of hydraulic conductivity values generated during all three phases of the project is provided in Figure 16.3.2.



Source: SRK, 2015

Figure 16.3.2: Hydraulic Conductivity Estimates in Elk Creek Strata

The response observed in the surrounding piezometers over the course of the nominal 30 day injection test indicates that faults and fractures within the carbonatite are relatively well connected. The distribution of mounding seen during the injection test confirms that the stress propagates from the center of the deposit in a roughly radial geometry. Water from Elk Creek was injected into the 6 inch injection well completed in the carbonatite to a depth of approximately 830 m with shutter screens; the injection rate was 22 L/s (350 gpm) for the first two weeks of the test, at which time a severe storm caused a flood which shut the test down. After two days to allow the flood waters to recede, the test was restarted at approximately 30 L/s (475 gpm) for two more weeks. The rise in water levels within the carbonatite due to injection observed at the end of the injection test is presented in Figure 16.3.3, showing 4.85 m of response in NEC15-002 located 0.6 km to the southwest, and over 2 m of response in NEC15-003 located 1.2 km to the southeast. On a mine scale, the faulted and fractured carbonatite exhibits quasi-homogeneous and isotropic characteristics.

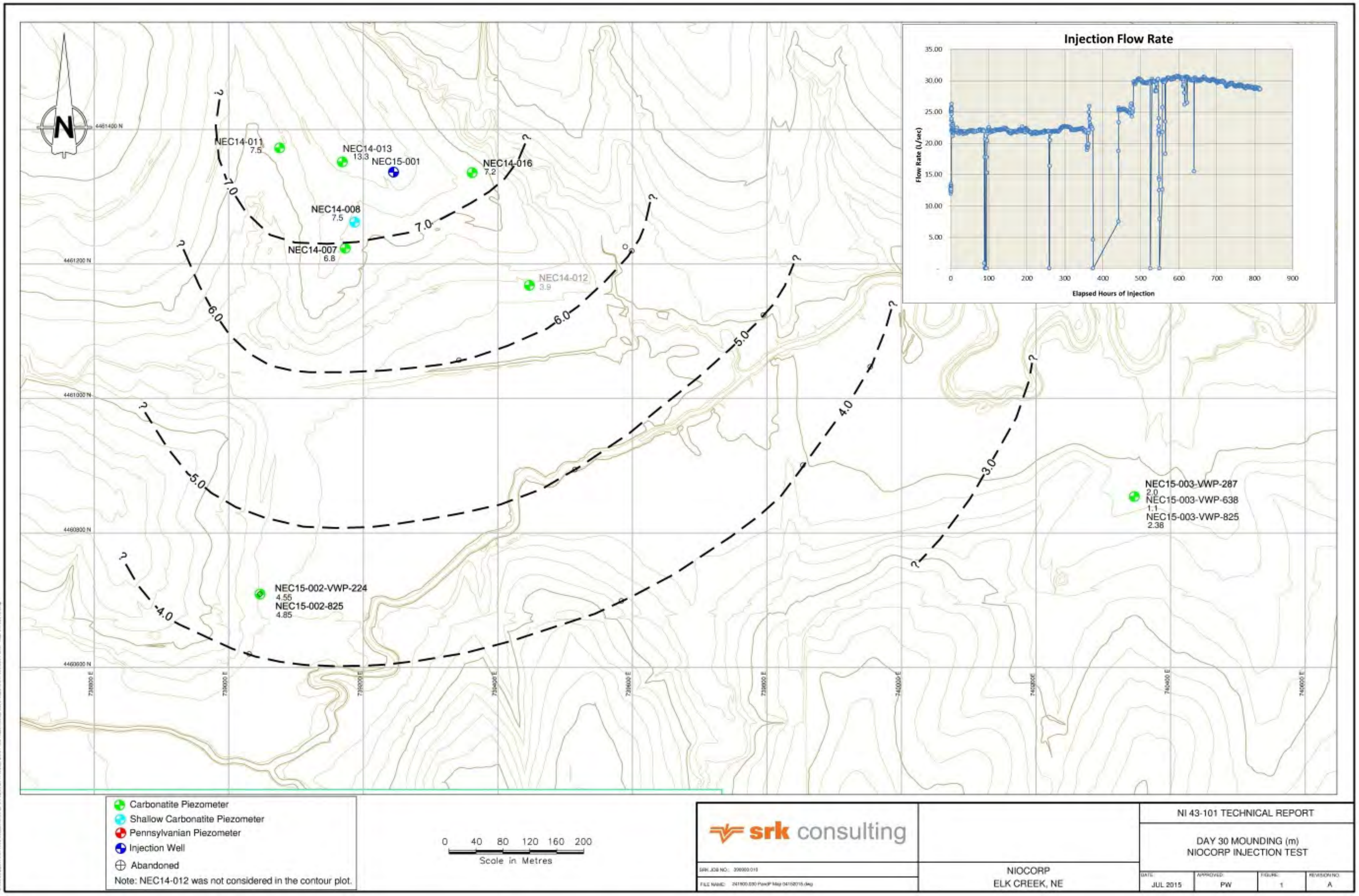


Figure 16.3.3: Rise in Water Levels Observed in Carbonatite at End of Injection Test

Groundwater temperatures from logging have been in the range of 35˚ to 51.7˚C (95˚ to 125˚F), and the water chemistry can be categorized as brackish. Chloride and sodium concentrations are approximately 10 and 8 g/L respectively, resulting in a TDS of 18 g/L.

Predicted Dewatering Requirements

Based on the data analysis and interpretation, SRK generated a prediction of dewatering requirements for the underground mine. A numerical groundwater flow model was built using MODFLOW-2000 finite-difference code to simulate mining progress, assuming the active mine blocks are sequentially dewatered as the mine progresses downward, using dewatering wells located along the edges of the mine. The model simulates the deep groundwater system below the confining PENN aquitard, which includes 860 m of carbonatite surrounded laterally by granite at a radial distance of 4 km. The modeled deep groundwater system includes six units as shown in Table 16.3.1. The model was calibrated to match the groundwater response observed during and after the nominal 30 day injection test. A summary of calibrated hydrogeological parameters are provided in Table 16.3.1.

Table 16.3.1: Hydrogeological Parameters Simulated by Numerical Groundwater Flow Model

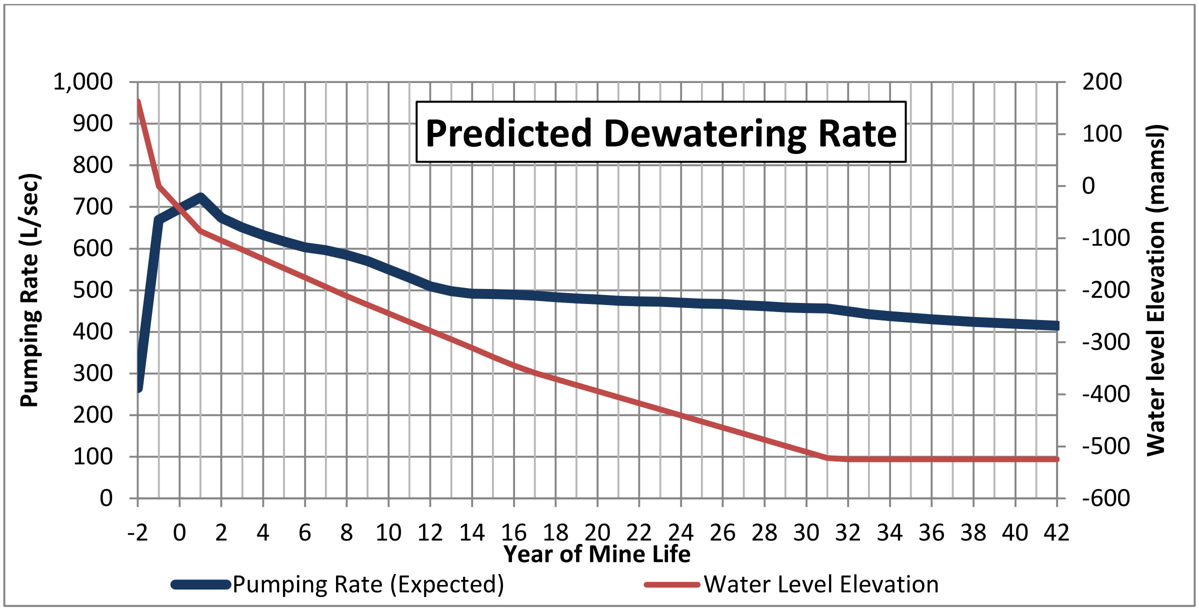
Hydrogeological Unit	Hydraulic Conductivity, K (m/day)		Specific Yield ( )	Specific Storage (1/m)
	K_h	K_v		
Mineralized Zone	2	2	0.005	2e-06
Faults within Mineralized Zone	3	3	0.005	2e-06
NE Faults outside of Mineralized Zone	0.5	0.5	0.005	2e-06
NW Regional Fault outside of Mineralized Zone	1	1	0.005	2e-06
Carbonatite outside of Mineralized Zone	0.15	0.15	0.005	2e-06
Granite	0.15/0.001	0.15/0.001	0.005	2e-06

Note: Unbounded/Bounded deep groundwater system.
Source: SRK, 2015

Compared to analytical method, calibration of the numerical model to injection test data indicates that mineralized zone and faults within carbonatite are more permeable (K varies from 0.5 m/d to 3 m/d) while carbonatite outside of mineralization zone has hydraulic conductivity about 0.15 m/day.

Two scenarios were considered regarding boundary conditions of the carbonatite within the deep groundwater system - unbounded and bounded by the granite. For example, in the case of bounded conditions, the hydraulic conductivity of the granite was assumed to be very low (almost impermeable) at K=0.001 m/day as shown in Table 16.3.1, while for an unbounded scenario a value of 0.15 m/d was simulated. Additionally, a range of storage parameters were considered in the dewatering projection, as shown in Table 16.3.1.

Expected pumping rates for dewatering activities are illustrated in Figure 16.3.4 and represent an average of the predicted flows for unbounded and bounded conditions as discussed above. It was assumed that installation of the dewatering well system (a number of wells drilled from surface) would begin two years in advance of production mining to allow dewatering of development workings such as the shaft, vent shaft and main access ramp. The maximum anticipated dewatering rate (~725 L/sec) is expected to occur one year into production mining, following three years of active dewatering. The expected dewatering rate is projected to decline to 500 L/sec in Year 13 and to 400 L/sec at end of mining.

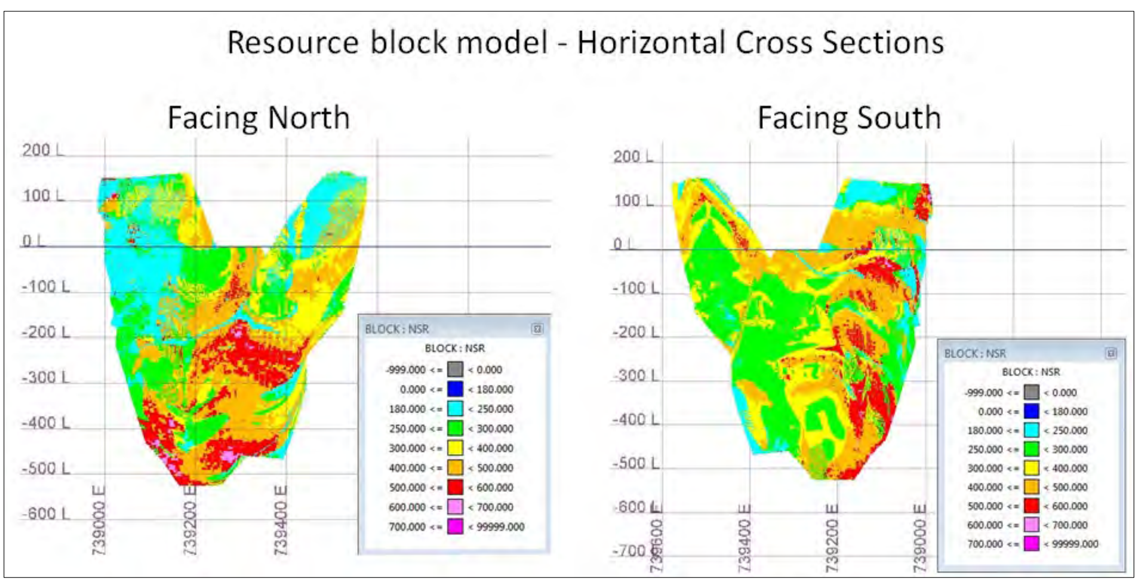


Source: SRK, 2015

Figure 16.3.4: Model-Predicted Dewatering Rates

16.4 Mine Design

Figure 16.4.1 shows cross sections of the resource block model blocks above a NSR CoG of US$180/t which have been classified as Measured and Indicated. There is a higher grade portion of mineralization at depth (approximately -400 to -600 m elevation) and the mineralization at those depths is approximately 300 m along strike. Higher in the deposit the mineralization is approximately 600 m in length along strike, and tends to have a lower grade central core area with higher grades at the edges of the deposit. This model formed the basis of the stope design.



Source: SRK, 2015

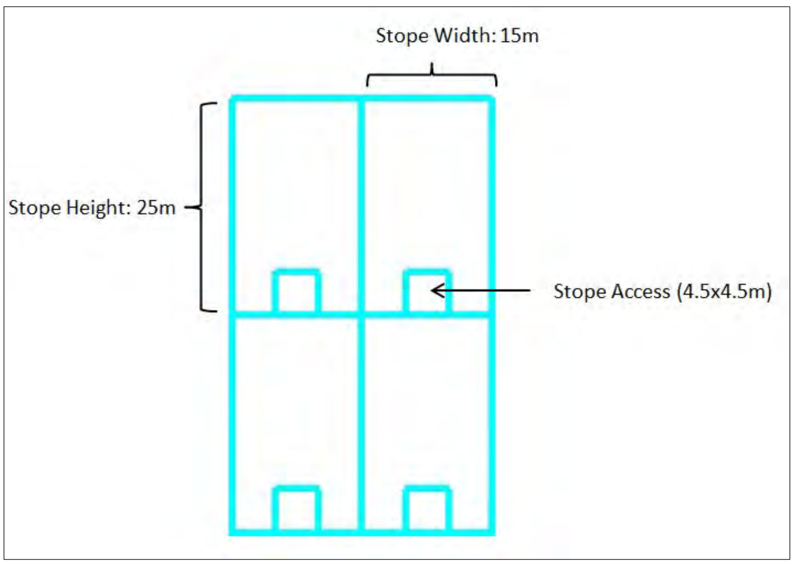
Figure 16.4.1: Cross Sections of Resource Model (NSR US$/t)

Stope optimization was completed in Vulcan™ using a minimum mining stope width of 15 m, a stope height of 25 m, variable length along strike, and an NSR CoG of US$180/t. The stopes and block model are oriented to match the direction of in situ stresses, which are elevated (~1.5H:1V). As discussed in Section 16.2, this stope orientation optimizes stress distribution and allows for mining of larger openings.

To produce 7,500 t FeNb annually with a process facility capacity of 2,700 t/d minimum average grade of 0.793% Nb_2O_5 must be mined. This is achieved by varying the mine design CoG by level to maintain the approximate average grade required. All CoGs used are above the marginal calculated CoG discussed in Section 16.1. Using elevated CoGs to maximize NPV or achieve other company goals has the effect of leaving lower grade resource material in situ.

16.4.1 Stope Design

Stope optimizer shapes were used as a basis for the design work. A typical level is made up of approximately 20 to 30 stopes. Stopes are 15 m wide and 25 m high with varying length. Each stope has a 4.5 m x 4.5 m access located at the bottom of the stope as shown in Figure 16.4.1.1. Top accesses are designed on most levels to give access to stopes on the next level and to allow for backfilling. For upper most stopes in a block or where there is no mining above, if the stope must be filled, it is assumed a hole can be drilled from adjacent development into the stope for backfilling purposes. The stopes are drilled top down and rings are blasted from the end of a stope toward the access. The blasted material is remotely mucked from the stope access.



Source: SRK, 2015

Figure 16.4.1.1: Stope Cross Section

A primary/secondary stoping sequence will be used, where on any given level, primary stopes must be separated by a secondary stope. Extraction of the secondary stope can only occur after the two immediately adjacent primary stopes have been mined, backfilled, and have had time to cure. Backfilling will be an integral part of the LHS mining cycle however with the quantity of stopes on a level and flexibility in sequencing mining should not be limited by backfilling operations. If a shorter level is encountered with fewer stopes then backfilling may be a limiting factor which could reduce mining rates on that level.

16.4.2 Development Design

The stope accesses are connected to a level access located in waste or low grade material. The level accesses are offset approximately 25 m from the end of stopes. Each stope access typically connects to the level access except in cases where stopes are small and long development is required to reach the stope. In those instances a connection from an adjacent stope is included in the design. This minimizes the amount of development, however it limits the sequencing order.

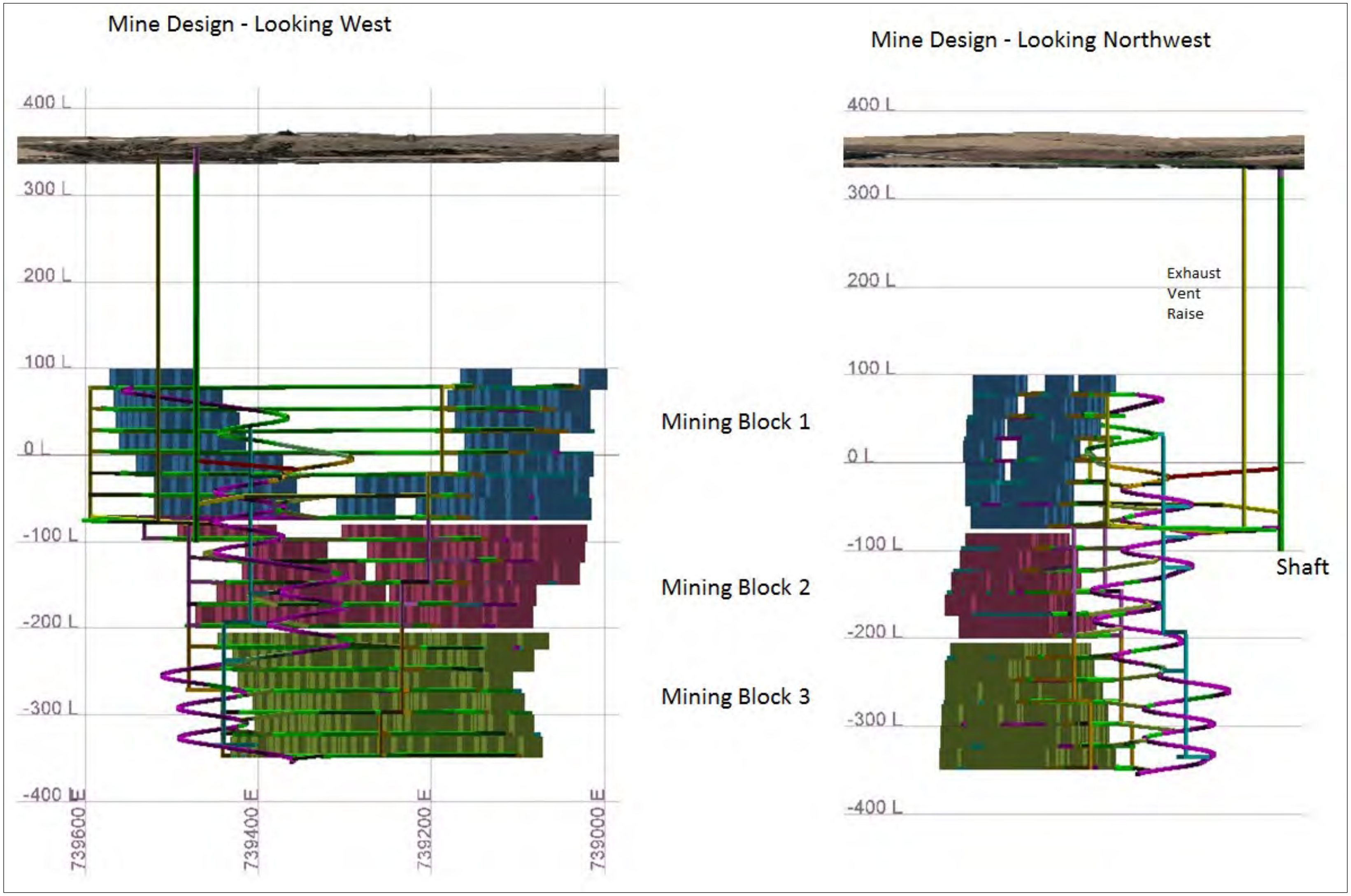
The level accesses connect to the main ramp which is offset at least 70 m from stoping into the footwall. On the southeast side of each level the level access connects to an exhaust air ventilation raise and on the northwest side connects to a fresh air raise. Figure 16.4.2.1 shows a typical level section.

Source: SRK, 2015

Figure 16.4.2.1: Typical Level Section

Shaft bottom development was not designed at this time, however a loading pocket and necessary accesses will be developed prior to production. Underground shops and associated development accesses are also assumed to be developed prior to production in the vicinity of the shaft. These infrastructure items should be located away from known faults.

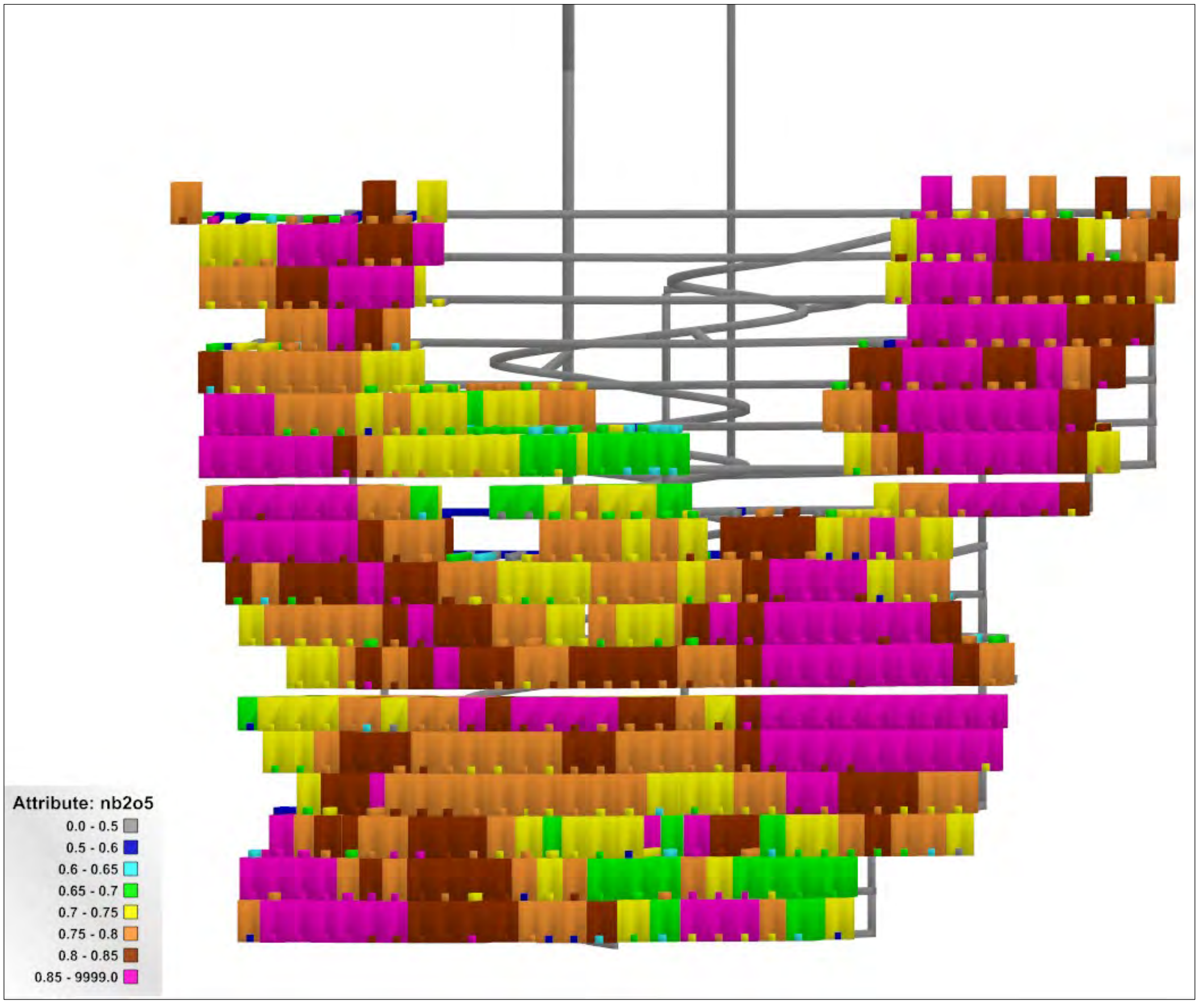
Figure 16.4.2.2 shows the completed mine design.



Source: SRK

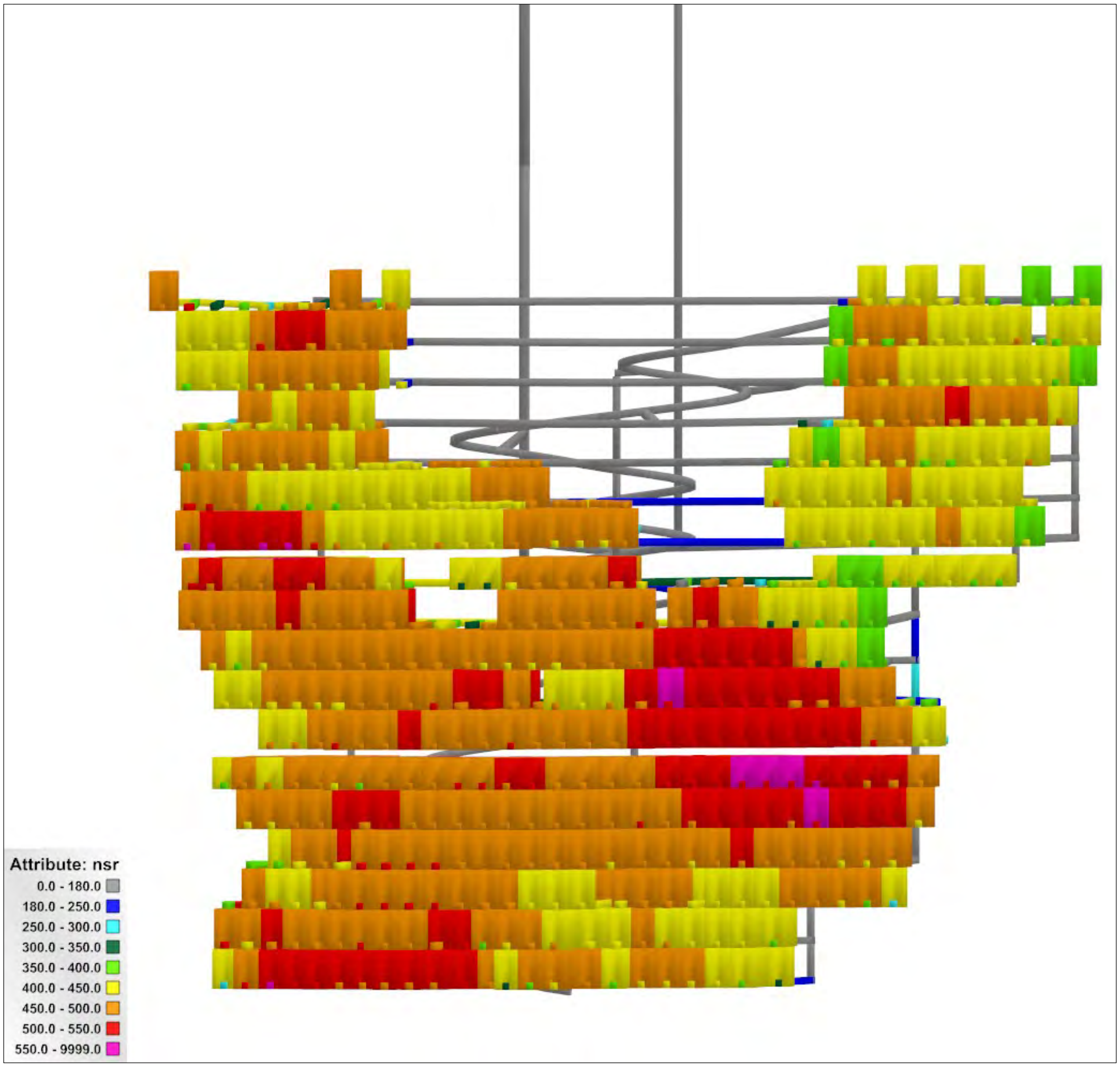
Figure 16.4.2.2: Completed Mine Design

Figures 16.4.2.3 and 16.4.2.4 show the mine design colored by Nb_2O_5 grade and NSR respectively.



Source: SRK, 2015

Figure 16.4.2.3: Mine Design Colored by Nb_2O_5 Grade



Source: SRK, 2015

Figure 16.4.2.4: Mine Design Colored by NSR

16.5 Mine Plan Resource

The underground mine design process results in mine plan resources of 31.1 Mt (diluted) with an average grade of 0.80% Nb_2O_5, 2.84% TiO_2, and 73 ppm Sc.

This estimate is based on a mine design using elevated CoGs and applying the US$180/t NSR CoG to material within the design. These numbers include a 95% to 100% mining recovery based on type of opening (stope, development, etc.) to the designed wireframes in addition to a 0% to 5% unplanned waste dilution. An additional development allowance of 26% was applied to main ramps and 19% to level accesses to account for detail currently not in the design. A 7% additional allowance was applied to stopes where arched backs were not designed at the average grade of the stope. This percentage was determined based on percentage of stopes within the design where there is no stope above. Waste dilution for stopes was applied with grade, slightly lower than the cutoff grade, based on an analysis of the material around stopes in a representative area.

Table 16.5.1 summarizes the mine plan resources.

Table 16.5.1: Mine Plan Resource Classification (1)

Description	Tonnes (kt)	Nb_2O_5 (%)	TiO_2 (%)	Sc (ppm)
Measured	-	-	-	-
Indicated	31,086	0.80	2.84	73
Measured + Indicated	31,086	0.80	2.84	73
Inferred	-	-	-	-

Source: SRK, 2015
(1) Includes Measured and Indicated material reported using an NSR CoG of US$180/t.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

16.6 Production Schedule

The production schedule is based on the rate assumptions shown in Table 16.6.1.

Table 16.6.1: Productivity Rates

Activity Type	Dimensions	Rate (1)
Main Ramps (single headings)	5 m x 5 m	5.1 m/d
Level Development (single headings)	5 m x 5 m	5.35 m/d
Drifting top/bottom stope accesses (multiple headings)	4.5 m x4.5 m	6.0 m/d
Stoping (2)	-	2,080 t/d
Shaft	7.5 m diameter	2.5 m/d
Raisebored Raise	5.5 m diameter	6 m/d
Slot Raises	5 m x 5 m	6 m/d
Backfilling	-	1,500 m^3/d

Source: SRK, 2015
(1) All rates are per face. Multiple areas/faces are mined together to generate the production schedule.
(2) Includes drilling, blasting, and mucking.

A delay of 9 days was used prior to driving on pastefill and a 28 day delay prior to mining adjacent to a pastefilled 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 2,700 t/d was targeted with ramp-up to full production as quickly as possible.

Shaft sinking activities begin in June of 2016 with mine development occurring in year 2017 and production commencing in 2018. Enough development is completed in 2017 to allow for production mining from stopes early in 2018 and minimizing the ramp up period. Table 16.6.2 presents the annual mining scheduled based on these assumptions. The annual schedule was completed using iGantt scheduling software. The iGantt scheduling work included backfill and its associated delays.

Table 16.6.2: Annual Mining Schedule

Year	Mineralized Tonnes (kt)	Nb_2O_5 (%)	TiO_2 (%)	Sc (ppm)	Waste Tonnes (kt)	Backfill (m^3)
2016	-	-	-	-	66.7	-
2017	219.5	0.57	2.32	56.45	178.3	-
2018	986.9	0.76	2.85	62.50	71.1	239,379
2019	986.7	0.82	2.78	75.26	101.5	280,380
2020	984.6	0.82	2.74	69.01	128.0	348,168
2021	985.4	0.83	2.99	57.99	92.4	299,505
2022	986.6	0.79	2.84	67.35	97.0	304,887
2023	989.4	0.79	2.74	63.98	19.6	280,802
2024	986.2	0.81	2.85	71.42	7.9	269,808
2025	986.7	0.81	2.68	74.22	2.6	272,831
2026	986.1	0.79	2.84	79.40	3.8	273,712
2027	986.4	0.79	2.77	78.15	39.8	295,551
2028	985.4	0.79	2.77	80.57	40.4	315,736
2029	986.7	0.79	2.78	76.94	98.7	300,471
2030	986.6	0.79	2.77	76.32	110.1	323,672
2031	986.1	0.79	2.80	73.27	37.2	291,889
2032	989.1	0.79	2.83	75.10	9.1	267,658
2033	985.6	0.82	2.90	69.56	2.6	319,168
2034	985.6	0.87	2.92	71.11	-	375,794
2035	985.3	0.81	2.90	78.23	5.1	300,217
2036	985.4	0.81	2.92	67.33	1.1	307,560
2037	985.3	0.81	2.99	66.79	3.0	299,341
2038	985.8	0.80	2.91	78.88	4.2	305,000
2039	985.7	0.80	2.85	72.15	-	353,149
2040	985.6	0.82	2.92	64.35	1.8	312,336
2041	985.7	0.80	2.79	79.46	5.6	285,008
2042	985.7	0.82	2.85	73.72	-	374,333
2043	985.8	0.80	2.81	75.66	4.4	343,292
2044	989.9	0.79	2.78	80.79	2.5	271,278
2045	985.6	0.80	2.98	76.19	-	302,082
2046	985.7	0.81	2.86	78.45	-	441,975
2047	985.5	0.80	2.88	78.89	-	357,130
2048	985.5	0.84	3.00	81.11	-	354,014
2049	513.0	0.68	2.59	71.84	-	107,709
Total	**31,085.5**	**0.80**	**2.84**	**73.33**	**1,134.5**	**9,773,835**

Source: SRK, 2015

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

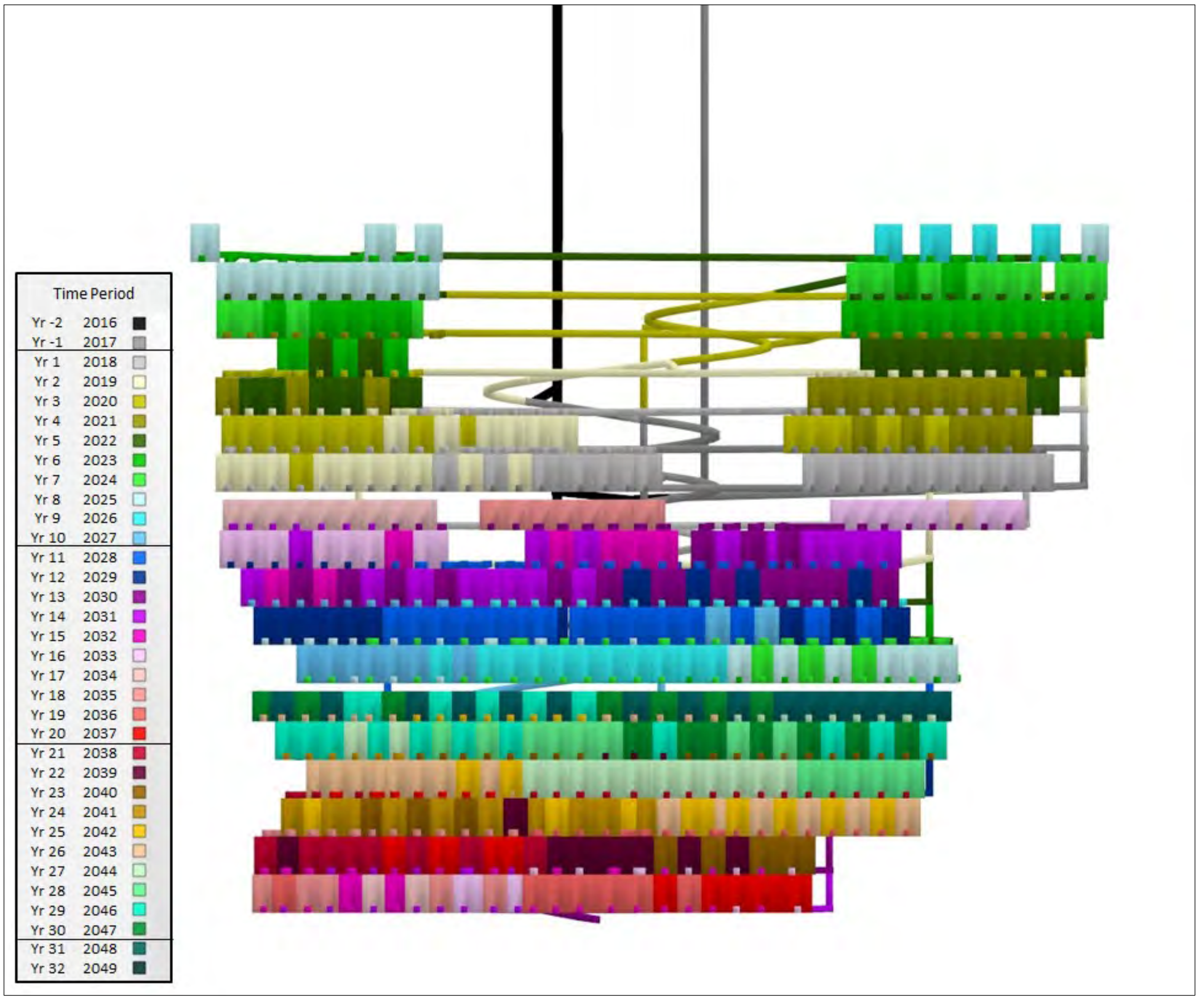
Table 16.6.3 summarizes the production schedule totals by development type.

Table 16.6.3: Production Schedule Totals by Activity Type

Develop/Production Type	Length (m)	Total Tonnes (kt)
Shaft	462	44.2
Raisebored Raise	406	25.7
Slot Raises	1,186	68.3
Main Ramp/Level Accesses	17,340	1,175.7
Level Development	47,835	2,892.9
Stoping	-	28,012.7
Total		**32,219.5**

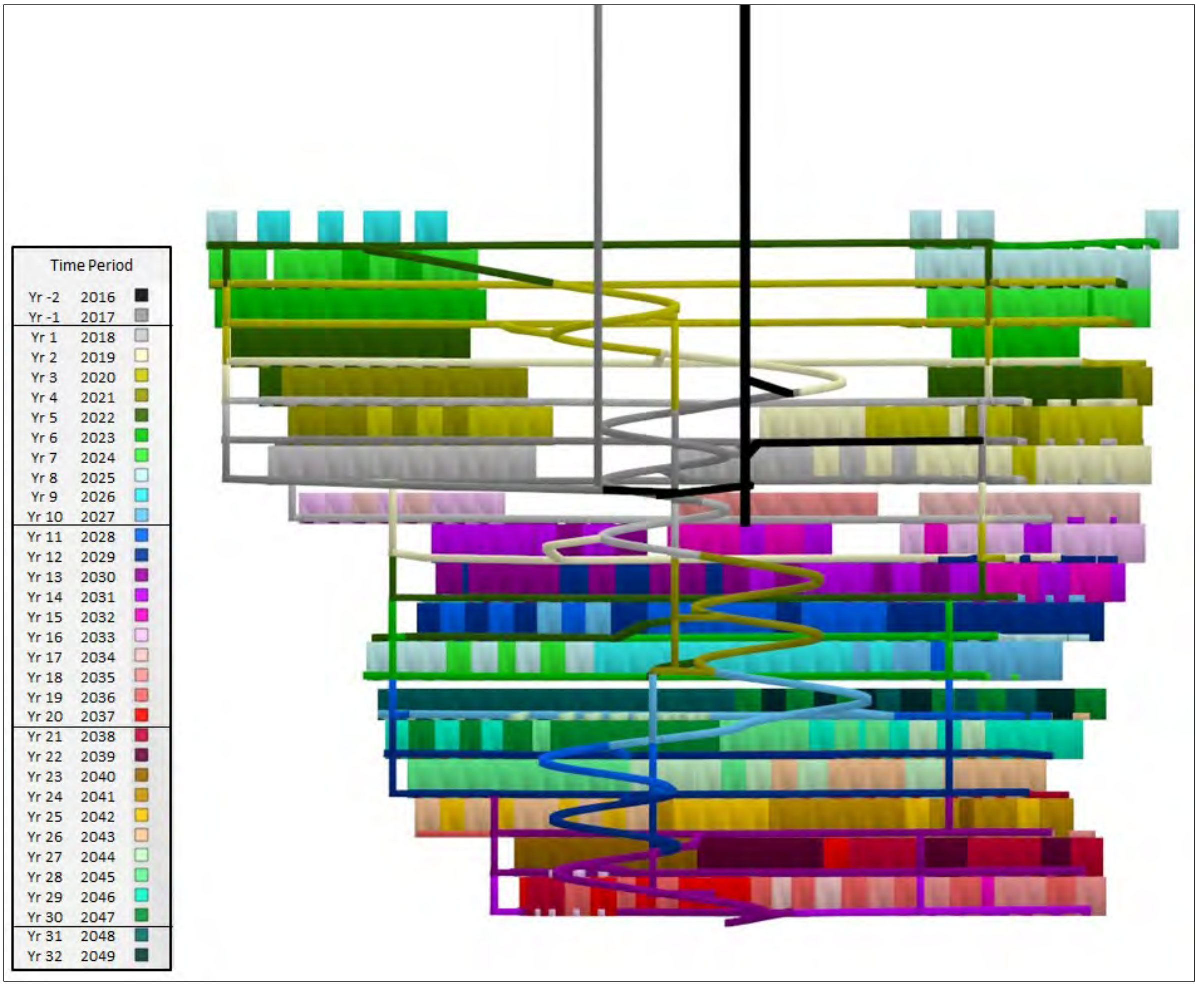
Source: SRK, 2015

Figures 16.6.1 and 16.6.2 shows the mine production schedule colored by year.



Source: SRK, 2015

Figure 16.6.1: Mine Production Schedule Colored by Year, Rotated View Looking Toward the Footwall (Northeast)



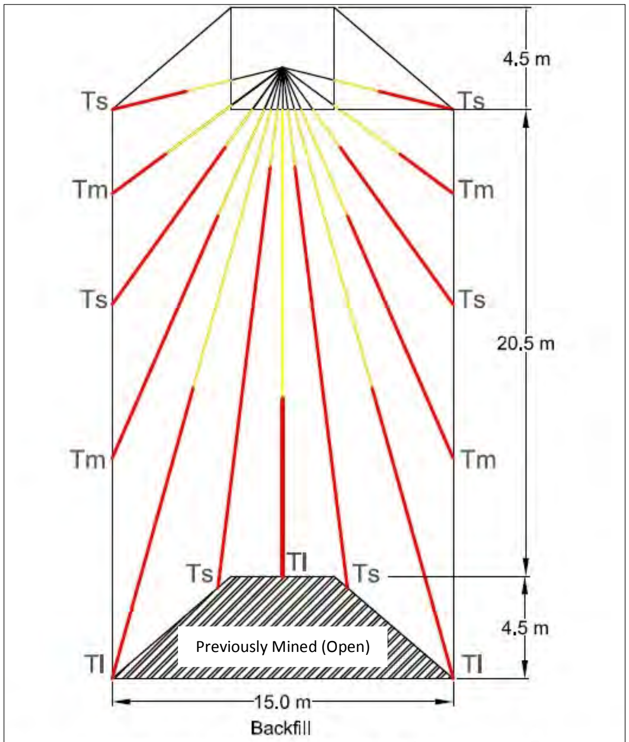
Source: SRK, 2015

Figure 16.6.2: Mine Production Schedule Colored by Year, Rotated View Looking Toward the Hangingwall (Southwest)

16.7 Mining Operations

16.7.1 Stoping

Stope lengths vary throughout the deposit ranging from 6 m to a maximum of 100 m giving a range of approximately 6,000 to 110,000 t per stope. After bottom and top accesses are established a slot raise will be developed at the far end of the stope (hangingwall side). Drilling will continue with the longhole drill using a fan shaped pattern as shown in Figure 16.7.1.1. Holes will be loaded with bulk emulsion and stope blasting will commence in the slot and subsequently rings will be blasted retreating toward the level access.



Hole #	Drilled Hole Length (m)	Collar Length (m)		Charge Length (m)	1/3 Charged	Steel Switchs
1	5.4	Ts	2.0	3.4	Pass	1
2	6.5	Tm	3.6	2.9	Pass	1
3	10.6	Ts	2.0	8.5	Pass	2
4	16.7	Tm	5.1	11.6	Pass	3
5	26.0	Tl	12.7	13.3	Pass	5
6	20.7	Ts	2.0	18.6	Pass	4
7	20.5	Tl	12.7	7.8	Pass	4
8	20.7	Ts	2.0	18.6	Pass	4
9	26.0	Tl	12.7	13.3	Pass	5
10	16.7	Tm	5.1	11.6	Pass	3
11	10.6	Ts	2.0	8.5	Pass	2
12	6.5	Tm	3.6	2.9	Pass	1
13	5.4	Ts	2.0	3.4	Pass	1

Source: SRK, 2015

Figure 16.7.1.1: Typical Stope Drilling Section

Remote mucking will be required for the majority of stope mucking so the load-haul-dump (LHD) operator can remain behind the brow of the stope. Stope material will be mucked primarily into a muck bay near the level access or the adjacent stope access. The material will then be loaded into trucks and hauled to the shaft for hoisting to surface. Once the stope is emptied a bulkhead will be placed in the 4.5 m x 4.5 m access and the stope void will be filled with paste backfill from the top access or via a drillhole at the top of the stope.

16.7.2 Development

Drifting development such as main ramps and level accesses are sized as 5 m x 5 m openings with an arched back. Drifting top/bottom stope accesses are sized as 4.5 m x 4.5 m flat back openings. These dimensions provide enough room for equipment, ventilation ducting, and utilities where necessary. Main ramps are typically a single heading environment. Level accesses are also typically single heading environments. Drifting top/bottom stope accesses are multiple heading environments.

All development will be mined using a double boom jumbo taking 4 m rounds. Blasted material will typically be mucked into a muck bay near the heading. The waste muck will subsequently be loaded into trucks and transported for disposal in the secondary stopes. As some development is in mineralized material grade control will be required to determine material destination on a round by round basis.

The ramp system is designed at a maximum gradient of 15%. A turning radius of 40 m was used which is suitable for any underground truck.

16.7.3 Mine Access

The underground mine will be accessed through a shaft system. The surface facilities include a hoist house, headframe structure, hoisting system, compressor room, MCC room, headframe structure with collar house, a mine electrical substation, and a development ventilation system that will be converted to a ventilation booster system during production. Shaft heating will also be provided.

The underground system includes the shaft, two cage over skips, an auxiliary cage, loading pocket and skip loader, and a muck handling system at the bottom of the shaft. An emergency escape system is included in the return air raise.

Surface Facilities

The hoist house is a single building with one large room that houses two hoisting plants and an annex that houses the electrical systems for the hoists, the control rooms, compressor room, mechanical cooling room, and other systems. The hoistroom will incorporate the use of an overhead crane to assist in installing the hoisting plants. The crane will have a capacity of 40 t with secondary hooks of 5 t capacity to assist with day-to-day maintenance on the hoists. The hoist foundations are reinforced concrete with some provision for a roping up winch in the hoistroom.

The hoist house contains a two-hoist system, a main production hoist and an auxiliary hoist. The production hoisting system is a 1,415 kW (1,900 HP), double drum system, 5 m in diameter, in a double-clutch configuration for maximum efficiency. The system allows for up to 5,000 t of muck hoisting within an eight hour shift, in addition to the use of the light duty service hoist. The second hoisting plant will utilize a 447 kW (600 HP), 2.44 m single drum hoisting plant that will serve as light duty service hoist and secondary means of egress during shaft construction. The auxiliary hoist will be connected to the emergency generator included in the plant electrical system, as it serves as an emergency egress from the shaft.

The headframe is a structural steel system founded on a mat foundation on the glacial till that extends to a depth of 30 m. The structural steel framing for the facility provides for four internal floors within the headframe, including:

- Production Sheave deck - The topmost floor in the headframe and houses the two 4.72 m diameter sheaves for the production hoisting facility.
- Service Sheave deck - The second floor from the top and houses the two 4.72 m diameter sheaves for the service/sinking hoist activities. It may also house sheaves for equipping stages (if needed) and the sinking sheaves for galloway winches.
- Dump floor - This provides maintenance access to the skip dumps. The floor also provides a dust barrier for the material being dumped and can house dust collection equipment.

- Collar floor - The main operating floor located at ground level. This provides access in and out of the shaft to the conveyances for service hoisting and to the hoist ropes and attachments for maintenance.
- A small bin will be utilized to capture and transfer mined material from the cages/skips to an apron feeder which will feed the conveyor system leading to the mineral processing facility.

The entry to the collar will be provided by a collar house attached to the headframe. The collarhouse provides a staging area for personnel and materials going down the shaft or returning to the surface and for maintenance. It is equipped with a 15 t overhead crane for material movement and conveyance removal/repair.

Shaft and Underground Access

The mine is accessed through a single 435 m deep, 7.5 m diameter shaft that supports both production and service hoisting operations. Shaft sinking is completed using conventional sinking methods. The shaft is compartmentalized with the primary skipping/mancage system on one side and the light duty system on the other. The compartments will be separated for the full length of the shaft with a brattice panel system of cladding or expanded metal mesh sheet in an angle frame.

The primary skipping system consists of a two skip/cage arrangement adjacent to each other on one side of the shaft. The cages have two decks and are 1.8 m x 1.8 m with the capability to handle 6 m long items internally. Utilizing this layout, each of the cages will have the ability to move approximately 39 persons per trip or 78 persons in one full cycle of the hoist (both conveyances unloading). Palletized consumables will be used for rapid handling. The cages will be a two-deck design with a removable upper deck that can be slung up or pulled out for handling long internal items. Four large pallets can be handled simultaneously, making movement of consumables relatively efficient. The skip will be located above the cage with a bottom dump mechanism. Each skip will have a 10.5 t payload.

The secondary light service hoisting system will have a capacity of approximately 5 t. The system will include a double-decked cage with a capacity of approximately ten people and will move smaller materials. In the event of a power loss connected emergency generator would come on automatically allowing the auxiliary hoist to transport personnel from the mine.

Near the shaft bottom the system includes a muck handling system with a grizzly, hydraulic breaker, jaw crusher, 3,000 t bin, conveyor belt, vibratory feeder, belt magnet, transfer car and two measuring pockets (flasks) in a typical loading pocket arrangement. The scalper, grizzly, hydraulic rock breaker, feeder and crusher will size the rock to a minus 15.25 cm (6 inch) size.

The system has an allowance for a control system that would integrate with the plant system.

One 5.5 m diameter ventilation raise to surface, approximately 410 m in length, is raise-bored by contractors prior to mine production. The raise will serve as an exhaust air raise and as emergency egress during production. The raise will contain an emergency hoist at the surface and bullet cage for emergency use.

16.7.4 Haulage

The mine will incorporate the use of four 14 t LHDs that muck material from stopes and development headings to either a muck bay or into 40 t underground trucks for haulage. Mineralized material is hauled to a grizzly/crusher that feeds the shaft loading bin. Early in the mine life average one-way haulage distances are approximately 540 m and 3 trucks are required. As the mine is developed deeper the haulage distance increases and additional trucks are required. Waste material from development will be moved from muck bays and hauled to secondary stopes as backfill in conjunction with the pastefill.

Table 16.7.4.1 shows the maximum one-way haul distance by mining block and the number of trucks required. Truck count includes waste haulage.

Table 16.7.4.1: Trucks and Haul Distance for Mineralized Material

Block	Max. One-way Haul Distance (m)	Number of Trucks
Block 1 (top)	845	3
Block 2	1,459	3
Block 3 (bottom)	2,614	4

Source: SRK, 2015

During the pre-production period, prior to mining of stopes and the commissioning of the plant, waste and mineralized material will be hoisted to surface. The material will be segregated during development and stored separately in a designated storage facility on the surface. The lined storage facility is designed with a capacity of 500,000 t with surface water controls as required. The mine will produce approximately 245,000 t of waste and 220,000 t of mineralized material in the two years prior to plant operation. The mineralized material will be fed into the processing plant during commissioning or as needed.

16.7.5 Backfilling

Paste Backfill Plant

A paste backfill plant will be located on surface and the paste backfill product will be made of fly ash from a local (74 km away) coal power plant. Sand will be used as an aggregate source to regulate the strength gain characteristics of the paste. The backfill mixture has a minimal amount of cement, as the fly ash is expected to gain some strength without the cement. Initial rheological testing has shown that fly ash used may begin to hydrate quickly and indicates the possible need for a retarder or water dilution with appropriate amounts of sand to allow for proper pipeline conveyance to the stopes. A 25% fly ash, 75% sand fill composition has been assumed. Primary stopes would be filled with 5% cement fill and secondary stopes would be filled with 2% cement.

The paste backfill plant has been designed to fill 1,500 m^3/day when operating. This is approximately 50% more than the void space generated from daily mining operations. Any waste rock mined underground and not hauled to surface will also be used as backfill material.

A simplified backfill process flow sheet is shown in Figure 16.7.5.1. Paste plant major mechanical equipment is shown in Table 16.7.5.1.

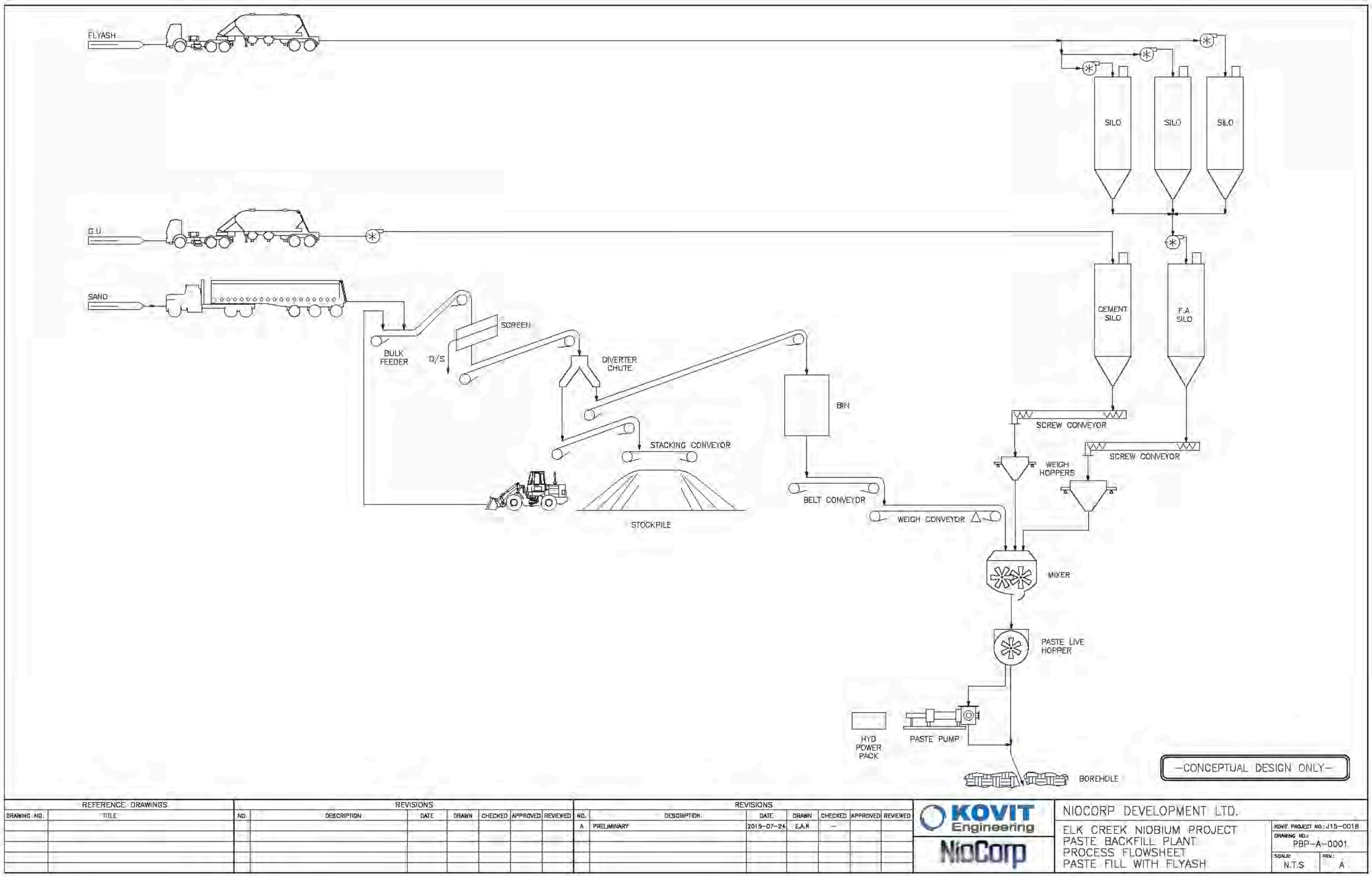


Figure 16.7.5.1: Simplified Backfill Process Flow Sheet

Table 16.7.5.1: Paste Plant Major Mechanical Equipment

Area	Description
Feeding Equipment	Live bottom receiving system
	Vibrating Screen
	Transfer conveyors (x3)
	Stacking Conveyor
Binder Equipment	Storage Silo – Fly ash & cement(x3)
	Screw conveyors (x2)
	Hoppers (x2)
Backfill Equipment	Sand Bin
	Conveyor (x2)
	Mixer w/ washing system
	Live bottom hopper
	Paste pump with hydraulic power pack
	Process Water Tank
	Process Water Pump

Source: Kovit

Underground Distribution

The paste location is near the east side of the deposit. A surface borehole is cased and twinned, where one hole acts as the main hole while the second is backup. A borehole orientation from surface of 70° was evaluated and underground holes between levels maintained a 60° to 70°dip. A 250 mm slump with a friction factor of 4 kPa/m can reach all areas of the underground mine using a gravity system only. Paste is then piped to the working areas through a network of steel and HDPE pipes.

Capital and operating costs for both the paste plant and the distribution system were provided by Kovit, and were reviewed and accepted by SRK.

16.7.6 Ground Support

The current knowledge of the geotechnical characteristics indicate that ground support will be required in the ramps and primary access drifts as well as the shaft and shop areas. The stope access drifts will require minimum ground support except at brows of the stopes. The ground support plan included use of swellex style bolts as a standard. The bolting will be supplemented with wire mesh, shotcrete, and additional support where required. The plan includes allowances for areas of full shotcrete, but it is not expected to be required in normal operations, just in areas encountering faulted or challenging ground conditions. A bolter will be utilized as normal practice and shotcrete equipment and transmixer are included in the estimate.

Table 16.7.6.1 shows the expected ground control systems for the various categories of rock conditions.

Table 16.7.6.1: Ground Control System and Rock Conditions

Rock Quality	Percentage	System
Good	70%	Spot bolting
Fair	20%	Systematic bolting
Poor	10%	Systematic bolting with mesh and Shotcrete
Faults	As encountered	Potential to Grout

Source: SRK, 2015

The systematic bolting is planned to use 2.5 m swellex bolts and a spacing of 1.2 m.

16.7.7 Grade Control

As the main ramp is developed, drill stations from the main ramp allow for fan drilling of the deposit prior to developing levels. This confirmatory drilling should be used to update the long term block model and provide confidence in expected tonnage and grades prior to level development.

Once a level is being developed, level and stope accesses will be sampled to determine material destination. This sampling can occur through use of a handheld XRF instrument which is able to sample the Nb grade. To provide more detail, samples can be taken from longhole cuttings and tested in the on-site lab.

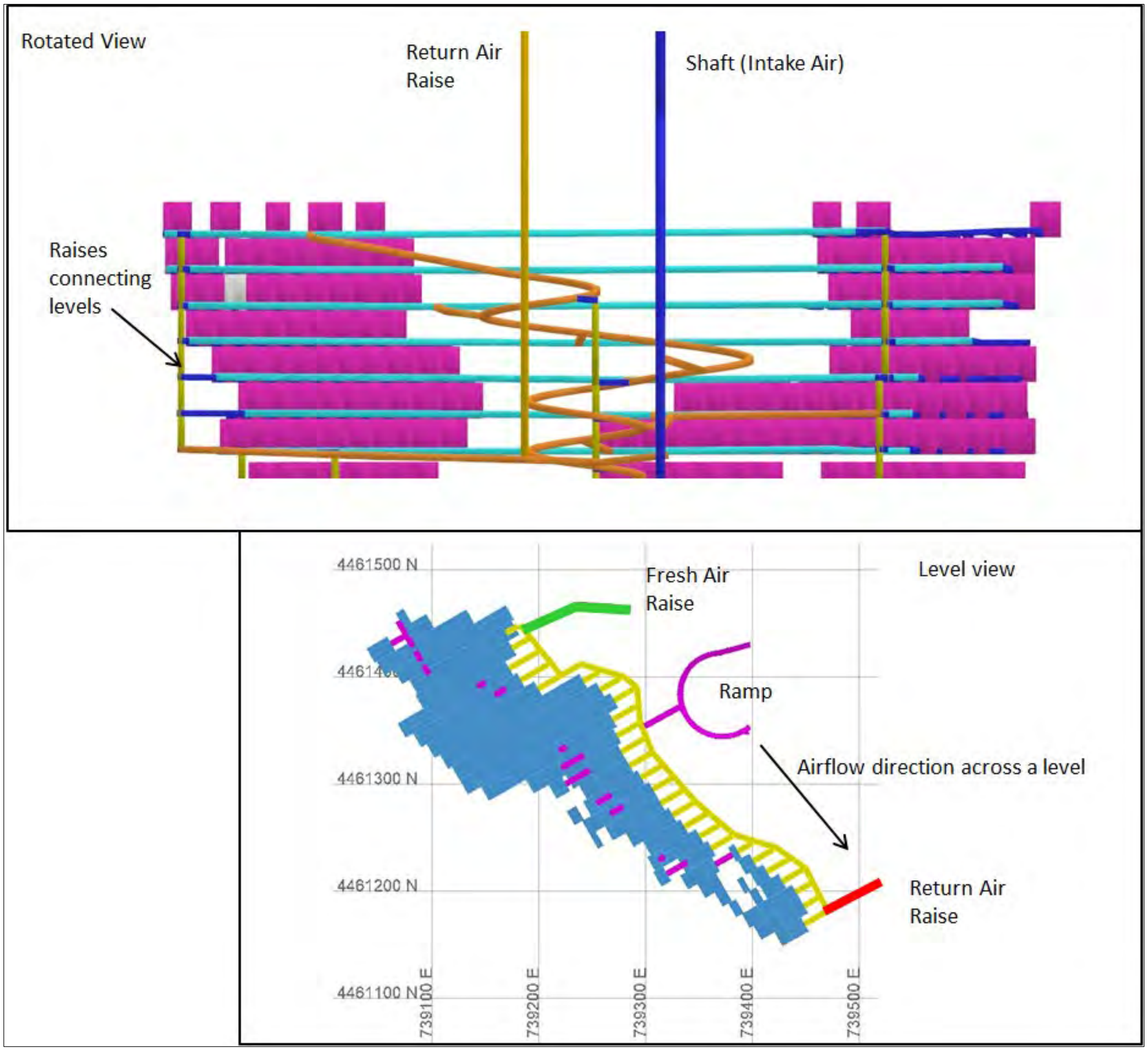
Once stope accesses are developed vertical holes will be drilled through the anticipated stopes and cuttings will be sampled to determine stope extents and estimated stope grades. Any samples tested in the lab should be used to update short term planning block models to better estimate tonnages and grades in the short term mine plan.

16.8 Mine Services

16.8.1 Ventilation

Primary Ventilation

The primary ventilation system during preproduction consists of fans installed on surface at the shaft. These surface fans become booster fans as the primary ventilation fans are installed underground near the return (exhaust) air raise (RAR). The fan system is designed to be adequate for the life of mine design with installation of ventilation doors and controls. Intake air is drawn down the hoisting shaft (and used for ramp ventilation) and across working levels as shown in Figure 16.8.1.1. Required airflow quantities are based on a ventilation rate of 0.08 m^3/kW of diesel equipment and assumed utilization factors for mobile equipment. A ventilation factor of 1.2 was applied to the diesel dilution airflow requirements for heat dissipation when the mining reaches the third block. A worst-case early mining scenario was modeled with three active stopes in series on one level (two actively mucking) and the exhaust temperature was kept under a design target of 26.5°C wet bulb.



Source: SRK, 2015

Figure 16.8.1.1: Typical Primary Ventilation System

Early in the mine life, the primary fan is expected to operate at 370 m³/s and 1.0 kPa, with the operating point eventually reaching 583 m³/s at 2.7 kPa during later years. The RAR is assumed to be built to finished internal diameter of 5.5 m. Slot raises used for ventilation were assumed to be 4.5 m × 4.5 m.

Total airflow required ranges from 350 m^3/s early in the mine life to 620 m^3/s late in the mine life.

Auxiliary Ventilation

An auxiliary ventilation system is required to provide ventilation from the level access to the stopes. A production heading is assumed to only have one 243 kW LHD requiring 19.4 m³/s of airflow. Models indicate a 65 kW fan should be used with a 1.2 m diameter duct for production headings.

Ramp and level development are assumed to have one 243 kW LHD and a 405 kW haul truck requiring 51.8 m³/s of airflow. These types of headings can be ventilated with a 302 kW fan and a 1.4 m diameter duct.

Mine Heating and Cooling

The need for mine air heating and/or cooling was assessed based on average temperatures for Elk Creek, NE as well as estimated virgin rock temperatures of approximately 35 °C and assumed rock thermal properties. Mine air refrigeration was determined to be unnecessary as the air could be cooled sufficiently with additional airflow. Approximately a 20% increase in airflow beyond the amount required for diesel exhaust dilution will be needed. The additional amount of airflow is included in the design of the system.

Heating is assumed to be required at some time during five months of the year. The system will heat the intake air to 4°C above freezing during the winter months

16.8.2 Pumping

The mine, after dewatering from the surface (discussed in 16.3), is expected to produce approximately 18.9 L/sec at maximum flows and thus an internal mine pumping system will be required. The system will be installed during development at the bottom of the shaft and will consist of two vertical agitated cone bottom tanks, sump, and two 260 kW positive displacement pumps per tank. This configuration will allow the system to be capable of pumping 63 L/sec up the shaft. Supplemental stage pumping skids will be located at the bottom of each block level and will consist of a 473 L/sec agitated holding tank and two dirty water pumps each capable of pumping 63 L/sec. One pump will operate and the other will be a spare. The skid system will be installed in series and report to the main pump station at the bottom of the shaft.

Face pumps will be employed at the development faces and will pump to the stage pumping skids that feed the main block sump.

16.8.3 Electrical Supply

All surface facilities will be supplied by power from the main project substation. The main surface substation will feed the mine substation located on the surface near the hoist house. The mine sub will feed the shaft electrical systems and the main underground infrastructure and equipment power down the shaft via a 13.8 kW feed line. The 13.8 kV power will feed throughout the mine to main load centers where the power will be stepped down to 480v for underground equipment use. Feeds will be provided at 110v and 220v for auxiliary use in the shops and for smaller loads such as fans, pumps, and auxiliary lighting.

A diesel backup generator at the surface will supply backup power for the emergency hoist systems and required ventilation systems to maintain minimum ventilation requirements in the case of emergency.

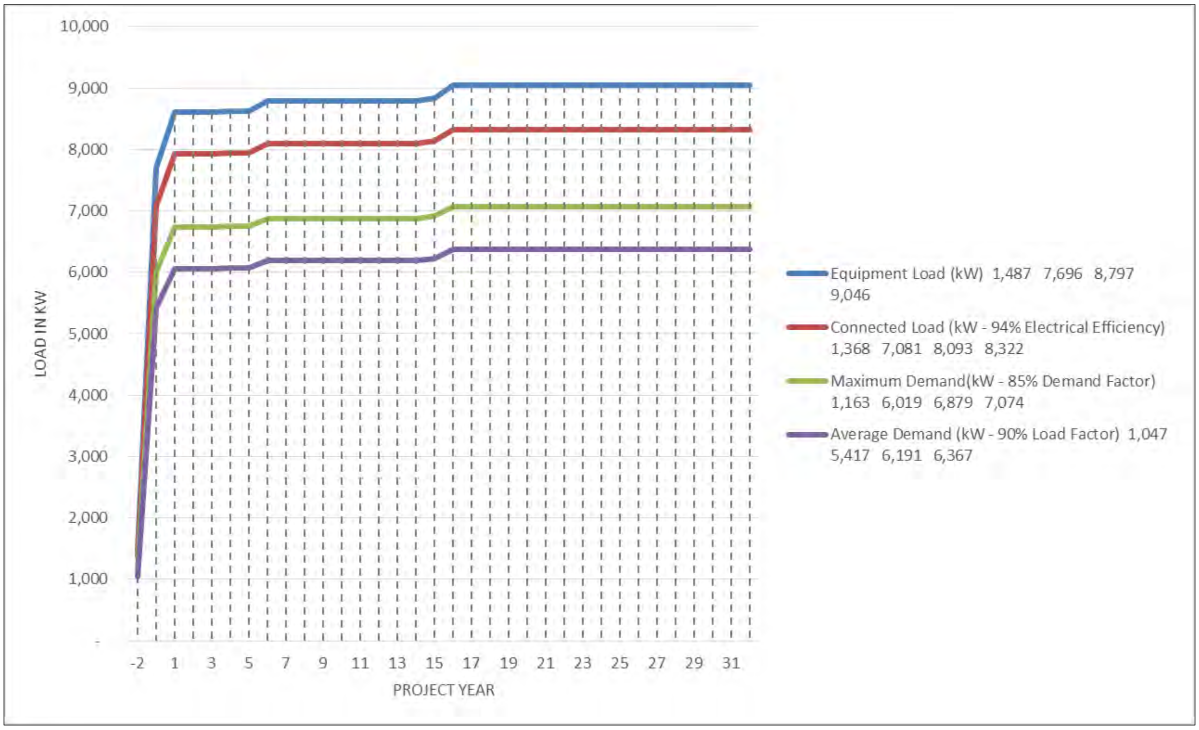
The connected electrical load is estimated to be 3.2 MW during construction and increasing to approximately 9.0 MW over the life of the mine. Table 16.8.3.1 summarizes the range of estimated loads.

Table 16.8.3.1: Typical Loads During Selected Time Periods (kW)

Load Category	Year -2	Year 1	Year 6	Year 28
Equipment Load (kW)	1,487	7,696	8,797	9,046
Connected Load (kW - 94% Electrical Efficiency)	1,368	7,081	8,093	8,322
Maximum Demand(kW - 85% Demand Factor)	1,163	6,019	6,879	7,074
Average Demand (kW - 90% Load Factor)	1,047	5,417	6,191	6,367

Source: SRK, 2015

The LoM electrical load is shown in Figure 16.8.3.1. The load changes correspond to increasing loads due to increased equipment and addition of ventilation and pumping due to depth.



Source: SRK, 2015

Figure 16.8.3.1: Mining Annual Electrical Load

The LoM equipment with electrical requirements are summarized in Table 16.8.3.2.

The main drivers of electrical consumption are the hoist, ventilation, mine pumps, and compressor. These systems account for 87% of the load.

Table 16.8.3.2: Life of Mine Equipment Electrical Requirement

Type of Equipment	Electrical Requirement per unit (kW)	Number of Units (Life of Mine)
Slot and Raise Drill	37	2
Jumbo (2 boom)	135	3
Bolter	70	2
Diamond Drill (Exploration)	40	2
Mobile 200 amp welders	6	2
Mobile 400 amp welder	12	2
Portable Water Pump- small	7	7
Portable Water Pump- medium	11	5
Main Sump Positive Displacement Pumps	261	2
Main Sump Feed Pumps	37	2
Block 1 Pumping Skid Pumps	37	2
Block 1 Sump Pump	43	1
Block 2 Pumping Skid Pumps	56	2
Block 2 Sump Pump	43	1
Block 3 Pumping Skid Pumps	75	2
Block 3 Sump Pump	43	1
Lighting Package Block 1	11	1
Lighting Package Block 2	11	1
Lighting Package Block 3	11	2
Hydrostroke Feeder	22	1
Grizzly Feeder	22	1
Jaw Crusher	112	1
Crusher Area Crane (40t)	34	1
HDPE Pipe Welder (8" and smaller)	5	3
Compressor (Electric)	448	3
Production Hoist	1,417	1
Auxiliary Hoist	448	1
Collar Door Hydraulic Power Pack	15	1
Headframe/Hoist House Lighting	11	1
Bins	6	1
Emergency Hoist	29	1
Supply Fan (Development/Booster)	597	2
Exhaust Fan (Production)	597	2
Development Fan	302	2
Auxiliary Fans (Electric)	35	13
Material handling system (skip pockets/bins/loading)	10	1
Rock Breaker plus grizzly (ore passes)	30	1
Dump Area Lighting Package	25	1
Shop Crane (UG Shop)	10	2
Warehouse	5	1
Shop	5	1
Offices	5	1
Diesel Storage (UG)	5	1
Refuge Chambers (12 person)	15	1

Source: SRK, 2015

16.8.4 Health and Safety

The mine design incorporates MSHA safety standards and includes an emergency hoist in the return air raise and a secondary light duty hoist in the main shaft. Both hoists are connected to backup power generation. Additionally three 12-person refuge chambers are included that will be located in active working areas over the LoM.

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 support the operation. The mine safety program will integrate with local providers in case of any mine emergency. 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.

16.8.5 Manpower

Manpower levels are estimated based on the production schedule and equipment needs. The productivities used reflect a mix of local and skilled labor with an experienced management team.

The estimate is based on owner mining using an operating schedule consisting of 12 hours per shift, two shifts per day, and seven days per week. The 12 hour shift is supported by a four crew rotation. The management and technical team are planned to work five 8 hour days per week.

Tables 16.8.5.1 to 16.8.5.3 shows the required workforce. The rotating crews will have a split of 46 people underground and 7 people on the surface. It is expected that the maximum personnel underground would be 59 per shift. The workforce will increase over time through the addition of staff to operate additional equipment.

Table 16.8.5.1: Typical Mining Labor by Shift

Day Shift (salaried)	Days
Mine Superintendent	1
Mine Planner	1
Maintenance Superintendent	1
Maintenance Planner	1
Maintenance Technician	1
Senior Mining Engineer	1
Geotechnical Engineer	1
Mine Planning Engineer	2
Surveyor	1
Mine Technologist	2
Geologist	1
Total	**13**

Rotating Shift (hourly)	Per Shift	Total
Mine Supervisor/Shift Boss	2	10
Safety / Mine Rescue / Training Supervisor	1	4
Hoistman	1	4
Toplander/Surface Laborer	2	8
Skip Tender	1	4
Surface Equipment Operator	0	0
Bolter Operator	2	8
Blasters	4	16
Ground Support, Hanging Services	3	12
Fuel/Lube/Boom/Grader/Telehandler	2	8
LHD & Truck Operator	5	20
Longhole and Jumbo Operator	4	16
Laborer	2	8
Diamond Driller	4	16
Backfill Crew - Bulkheads, Piping, Monitor	3	12
Paste Backfill Plant Operators-Surface	1	4
Mine Maintenance Supervisor/Lead Hand	1	4
Mechanic	4	16
Mechanic Helper	4	16
Electrician	3	12
Total	**49**	**198**

Grand Total	**62**	**211**

Source: SRK, 2015

A further breakdown of the staffing by function is included in Table 16.8.5.2.

Table 16.8.5.2: Typical Mine Labor by Function

Operations	Per Shift	Total
Mine Superintendent	1	1
Mine Planner	1	1
Mine Supervisor/Shift Boss	3	15
Total Operations Supervision	**5**	**17**
Hoistman	1	4
Toplander/Surface Laborer	2	8
Skip Tender	1	4
Surface Equipment Operator	0	0
Bolter Operator	2	8
Blasters	4	16
Ground Support, Hanging Services	3	12
Fuel/Lube/Boom/Grader/Telehandler	2	8
LHD & Truck Operator	5	20
Longhole and Jumbo Operator	4	16
Laborer	2	8
Backfill Crew - Bulkheads, Piping, Monitor	3	12
Paste Backfill Plant Operators-Surface	1	4
Total Operations Labor	**30**	**120**
Total Operations	**35**	**137**

Maintenance	Per Shift	Total
Maintenance Superintendent	1	1
Maintenance Planner	1	1
Maintenance Technician	1	1
Mine Maintenance Supervisor/Lead Hand	1	4
Total Maintenance Supervision	**4**	**7**
Mechanic	4	16
Mechanic Helper	4	16
Electrician	3	12
Total Maintenance Labor	**11**	**44**
Total Maintenance	**15**	**51**

Technical Services	Per Shift	Total
Safety / Mine Rescue / Training Supervisor	1	4
Senior Mining Engineer	1	1
Geotechnical Engineer	1	1
Mine Planning Engineer	2	2
Surveyor	1	1
Mine Technologist	2	2
Geologist	1	1
Diamond Driller	4	16
Total Technical Services	**13**	**28**

Grand Total Mining	**63**	**216**

Source: SRK, 2015

The workforce will vary from a low of 112 people in the first year of preproduction to a high of 219 starting in the 20th year of production. Table 16.8.5.3 shows the variation by year

Table 16.8.5.3: Mine Labor by Year

Year	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049
	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
Operations																																			
Mine Superintendent	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Planner	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Supervisor/Shift Boss	0	0	5	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	8	8	8	8	8
Total Operations Supervision	**0**	**0**	**7**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**10**	**10**	**10**	**10**	**10**
Hoistman	0	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Toplander/Surface Laborer	0	0	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Skip Tender	0	0	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Surface Equipment Operator	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Bolter Operator	0	0	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	4	4	4
Blasters	0	0	8	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	8	8	8	8	8
Ground Support, Hanging Services	0	0	8	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	4	4	4	4	4
Fuel/Lube/Boom/Grader/Telehandler	0	0	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
LHD & Truck Operator	0	0	12	20	20	20	20	20	28	28	28	28	28	28	32	32	32	32	32	32	32	32	36	36	36	36	36	36	36	36	36	28	28	28	28
Longhole and Jumbo Operator	0	0	12	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	8	8	8	8	8
Laborer	0	0	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	4	4
Backfill Crew - Bulkheads, Piping, Monitor	0	0	0	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Paste Backfill Plant Operators-Surface	0	0	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Total Operations Labor	**0**	**0**	**64**	**120**	**120**	**120**	**120**	**120**	**128**	**128**	**128**	**128**	**128**	**128**	**132**	**132**	**132**	**132**	**132**	**132**	**132**	**132**	**136**	**136**	**136**	**136**	**136**	**136**	**136**	**136**	**108**	**96**	**96**	**96**	**96**
Total Operations	**0**	**0**	**71**	**132**	**132**	**132**	**132**	**132**	**140**	**140**	**140**	**140**	**140**	**140**	**144**	**144**	**144**	**144**	**144**	**144**	**144**	**144**	**148**	**148**	**148**	**148**	**148**	**148**	**148**	**148**	**118**	**106**	**106**	**106**	**106**
Maintenance																																			
Maintenance Superintendent	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planner	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Technician	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Maintenance Supervisor/Lead Hand	0	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Total Maintenance Supervision	**0**	**0**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**
Mechanic	0	0	8	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16
Mechanic Helper	0	0	8	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	8	8	8	8
Electrician	0	0	8	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	8	8	8	8
Total Maintenance Labor	**0**	**0**	**24**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**44**	**32**	**32**	**32**	**32**
Total Maintenance	**0**	**0**	**31**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**39**	**39**	**39**	**39**
Technical Services																																			
Safety / Mine Rescue / Training Supervisor	0	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Senior Mining Engineer	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Geotechnical Engineer	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Planning Engineer	0	0	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1
Surveyor	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Technologist	0	0	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Geologist	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Diamond Driller	0	0	0	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	4	0	0
Total Technical Services	**0**	**0**	**10**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**20**	**15**	**15**	**15**	**11**	**11**
Grand Total Mining	**0**	**0**	**112**	**203**	**203**	**203**	**203**	**203**	**211**	**211**	**211**	**211**	**211**	**211**	**215**	**215**	**215**	**215**	**215**	**215**	**215**	**215**	**219**	**219**	**219**	**219**	**219**	**219**	**219**	**219**	**184**	**160**	**160**	**156**	**156**

Source: SRK, 2015

16.8.6 Equipment

The underground equipment used, shown in Table 16.8.6.1, is typical for a sublevel stoping mining method with the number of pieces of equipment calculated from the production rates and typical availabilities for underground mines.

The estimate uses an equipment availability of 85% and an operator efficiency factor (job factor) of 90%. Each shift of 12 hours is reduced by 1.5 hours to represent shift change, lunch, and travel to and from working areas. This provides an equivalent working day of 21 hours or 10.5 h hours per shift. An operational utilization of 85% is used for planning purposes. This nets approximately 5,000 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 tram time. This reduces the overall utilization of the equipment fleet. The surface fleet is available to handle the stockpiling and mill feed needs as well as supplement the ongoing work at the tailings facility.

The equipment totals by pre-production and production year are summarized in Table 16.8.6.2. The later years include additional trucks and LHD's due to increasing haul distance.

The mine will also have major fixed equipment that is summarized in Table 16.8.6.3.

Table 16.8.6.1: Mobile Equipment Life of Mine Summary

Type of Equipment	Suggested Equipment / Manufacturer	Size (m^3)	Size (t)	Diesel (kW)	Electric (kW)	LoM Total
LHD-3 m^3	Sandvik LH307 - 3 m^3	3	9	150		1
LHD-6.4 m^3 (14 T)	Sandvik LH514 - 6.4 m^3, some with ejector bucket	6.4	14	243		4
Haul Trucks – 40 T	Sandvik TH540 - 40T, backfill with ejector beds	20	40	405		6
Blind Bore Kit for Drill	Sandvik D30 for DU311					1
Downhole/ Slot and Raise Drill	Sandvik_DTH Drill_Orion_DU311-T_6200_Orion				37	2
Jumbo (2 boom)	Sandvik DD321-40 - 4.3 m (14 ft) feeds with RD520 Drill - R32 bit - 52mm (1.75 inch)			110	135	3
Bolter	Sandvik DS311-C – 3 m swellex capability			110	70	2
Scissor Lift	Getman - A64 Pipe Hanger/Fan Handler			130		1
Scissor Lift	Getman - A64 Scissor Lift			130		1
Shotcrete transmixer	Getman A64 HD R60			130		1
Shotcrete equipment	Getman - Shotcrete DMA			150		1
Emulsion loader	Getman - A64 Emulsion Charger			130		2
Road Grader	Getman - RDG1504C			110		2
Fuel / Lube Truck	Getman - Lube/Fuel Truck			130		1
Boom truck	Getman - Knuckle Crane Truck			130		1
Tractors	Kubota - RTV1140CPX -2 seat			19		20
Jackleg/Stoper	General Jackleg drill and leg					10
Diamond Drill (Exploration)	UG core drill, 914 m, 1.7 m feedlength, drill control panel, mounting fame power pack and pump				75	2

Source: SRK, 2015

Table 16.8.6.2: Mobile Equipment Totals by Year

Year	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Type of Equipment	**-3**	**-2**	**-1**	**0**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
LHD-3 m^3	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
LHD-6.4 m^3 (14 T)		-	2	3	3	3	3	3	4	4	4	4	4	4	4	4	4	4
Haul Trucks – 40 T		-	3	3	3	3	3	3	4	4	4	4	4	4	5	5	5	5
Blind Bore Kit for Drill		-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Downhole/ Slot and Raise Drill		-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Jumbo (2 boom)		-	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Bolter		-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Scissor Lift		-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Scissor Lift		-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shotcrete transmixer		-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shotcrete equipment		-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Emulsion loader		-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Road Grader		-	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2
Fuel / Lube Truck		-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Boom truck		-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tractors		2	12	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Jackleg/Stoper		6	8	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Diamond Drill (Exploration)		-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2

Source: SRK, 2015

Table 16.8.6.3: Major Fixed and Auxiliary Equipment Summary

Type of Equipment	Electrical Requirement per Unit (kW)	Number of Units (Life of Mine)
Portable Water Pump- small	7	7
Portable Water Pump- medium	11	5
Main Sump Positive Displacement Pumps	261	2
Main Sump Feed Pumps	37	2
Block 1 Pumping Skid Pumps	37	2
Block 1 Sump Pump	43	1
Block 2 Pumping Skid Pumps	56	2
Block 2 Sump Pump	43	1
Block 3 Pumping Skid Pumps	75	2
Block 3 Sump Pump	43	1
Hydrostroke Feeder	22	1
Grizzly Feeder	22	1
Jaw Crusher	112	1
Crusher Area Crane (40t)	34	1
Compressor (Electric)	448	3
Production Hoist	1,417	1
Auxiliary Hoist	448	1
Collar Door Hydraulic Power Pack	15	1
Bins	6	1
Emergency Hoist	29	1
Supply Fan (Development/Booster)	597	2
Exhaust Fan (Production)	597	2
Development Fan	302	2
Auxiliary Fans (Electric)	35	13
Material handling system (skip pockets/bins/loading)	10	1
Rock Breaker plus grizzly (ore passes)	30	1
Shop Crane (UG Shop)	10	2
Refuge Chambers (12 person)	15	1

Source: SRK, 2015

17 Recovery Methods

The ferroniobium processing facility is designed with three distinct operation units: a mineral processing plant including a grinding circuit, designed to reduce the particle size prior to leaching; a hydrometallurgical plant (hydromet), designed to extract niobium pentoxide (Nb_2O_5), scandium oxide (Sc_2O_3), and titanium oxide (TiO_2); and a pyrometallurgical plant (pyromet), designed to produce ferroniobium, an iron-niobium alloy.

17.1 Mineral Processing Plant / Grinding Circuit

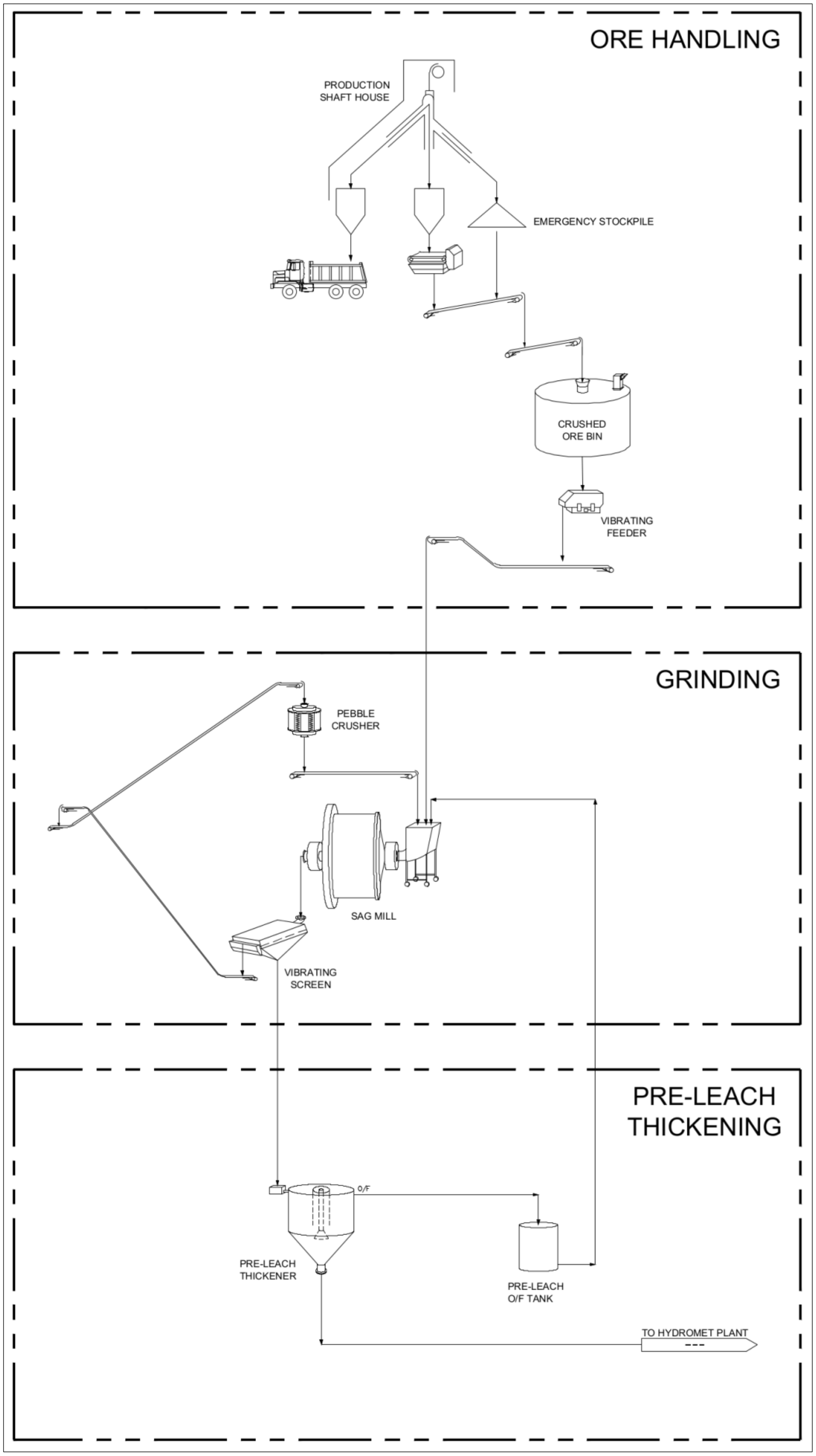
Following an intensive flotation testwork program, direct leaching of the ground mineralized material without a flotation pre-concentration circuit was selected as the most favorable process for treating the Elk Creek mineralized material due to a significant increase in recoveries associated with this process.

17.1.1 Flowsheet and Process Description

A preliminary flowsheet has been developed for the mineral processing plant and is shown in Figure 17.1.1.1. Run-of-mine mineralized material is crushed underground to a P80 of 115 mm with a jaw crusher. The crushed mineralized material is then conveyed at a rate of 337.5 t/h from a bin located at the mine shaft house to a mineralized material bin with a live capacity of 2,700 t.

Crushed mineralized material is reclaimed from the bin using vibrating feeders and is fed to the 6.71 m diameter x 2.44 m long (22 ft x 8 ft) semi-autogenous grinding (SAG) mill. The SAG mill is equipped with a 1,343 kW (1,800 hp) motor and a variable speed drive. The SAG mill operates in closed circuit with a classifying screen and an 89 kW (120 hp) pebble crusher. The SAG mill grinding circuit is designed at an average throughput of 122.3 t/h to produce a minus 2 mm product.

Classifying screen undersize flows by gravity to the SAG Mill discharge pumpbox where it will be pumped to the 6 m diameter pre-leach thickener. The thickener underflow with a P_{80} of approximately 1.1 mm is pumped to the hydrometallurgical plant while the thickener overflow will be recirculated to the SAG Mill.



Source: Roche, 2015

Figure 17.1.1.1: Grinding Circuit Simplified Flowsheet

17.1.2 Mineral Processing Plant Design Criteria

The mineral processing design criteria have been established based on the bench scale and pilot testwork results, conducted by SGS, the Roche in-house database from similar projects, and standard industry practices. The major design criteria are listed in Table 17.1.2.1.

Table 17.1.2.1: Mineral Processing Design Criteria

	Description	Value	Units
Feed	Mill availability	92.0	%
	Annual feed	985,500	t of mineralized material per year
	Daily throughput	2,700	t/d
	Mill hourly throughput	122	t/h
	Feed grade (Nb_2O_5)	0.80	%
	Feed grade (TiO_2)	2.84	%
	Feed grade (Sc)	73	ppm
Grinding	Ball Mill Work Index (BWI)	14.5	kW/h
	Rod Mill Work Index (BWI)	15.4	kW/h
	Abrasion index (Ai)	0.066	g
	SAG Mill Feed Size, F_{80}	115	mm
	SAG Mill product size, P_{80}	1100	microns
Thickening	Thickener sizing criteria	0.009	m²/t/d
	Thickener U/F density	75.9	%
	Thickener diameter	6	m
	Flocculant consumption	15	g/t

Source: Roche, 2015

17.1.3 Mass Balance and Equipment Selection

Based on the design criteria and the flowsheet, a mass balance for the mineral processing plant has been developed. The mass balance was prepared for an average feed rate of 2,700 t/d or 122.3 t/h with 92% plant availability at a feed grade of 0.80% Nb_2O_5, 2.84% TiO_2 and 73 ppm Sc.

Major process equipment as well as most minor equipment have been sized and selected based on the design criteria and mass balance.

An allowance was made for some minor equipment and facilities where required. The major equipment are listed in Table 17.1.3.1.

Table 17.1.3.1: Mineral Processing Major Equipment List

Equipment	Qty	Description / Size	Motor kW (each)
Crushed mineralized material bin	1	13.5 m dia. x 19 m height	0
Reclaim vibrating feeder	3	1,250 mm W x 2,000 mm L	5
SAG Mill	1	6.71 m dia. x 2.44 m long	1,343
SAG Mill vibrating screen	1	Double deck, 1.83 m W x 4.88 m L	37
Cone crusher (Pebble crusher)	1	HP120	89
Pre-leach thickener	1	6 m diameter	0
Pre-leach thickener overflow tank	1	2.7 m dia. x 3.3 m height	0

Source: Roche, 2015

17.1.4 Power Requirements

The power requirements for the major areas are listed in Table 17.1.4.1.

Table 17.1.4.1: Mineral Processing Power Demand by Area

Area	Installed (kW)
Crushed mineralized material handling	63
Crushed mineralized material storage	73
Grinding	1,825
Pre-leach thickening	71
Flocculant preparation	25
Total	**2,057**

Source: Roche, 2015

17.1.5 Grinding Circuit Plant Layout

From the crushed mineralized material bin located at the shaft house, crushed mineralized material will be transferred to a 2,700 t live capacity storage bin via a feeder and conveyors.

The bin is 13.5 m diameter x 19.0 m height. Under the storage bin, three reclaim vibrating feeders are installed to reclaim the crushed mineralized material and feed the SAG Mill feed conveyor.

The mineral processing equipment including the grinding circuit, thickening circuit, and flocculant preparation will be located within the hydrometallurgical plant. The dimensions of this area are 50 m length x 31.5 m width.

17.1.6 Mineral Processing Plant Water Management

Based on the design criteria and the mass balance above, a water balance for the mineral processing plant has been developed.

A water treatment plant will be built for the desalination of the underground mine. The treated water is required for the gland seals, reagent preparation and cooling water.

Process water will be supplied by recycled water coming from the hydrometallurgical plant tailings dewatering circuit, supplemented with make-up water from the underground mine dewatering system. The underground mine water will be needed to start the process and will act, once in operation, as make-up water.

The major water flows are listed below:

- 901 m^3/d of water contained in pre-leach thickener underflow from the mineral processing plant to the hydrometallurgical plant;
- 703 m^3/d of water will be recirculated from the pre-leach thickener overflow to the SAG Mill feed for pulp density control;
- 110 m^3/d of the treated water will be used as gland seal water;
- 22 m^3/d of the treated water will be used for flocculant solution preparation;
- 627 m^3/d of the process water will be used for dilution and SAG mill screen wash water.

17.2 Hydrometallurgical Plant

17.2.1 Flowsheet and Process Description

The preliminary hydrometallurgical processing plant flowsheet has been developed and is presented in Figure 17.2.1.1. The proposed process is divided into eleven units:

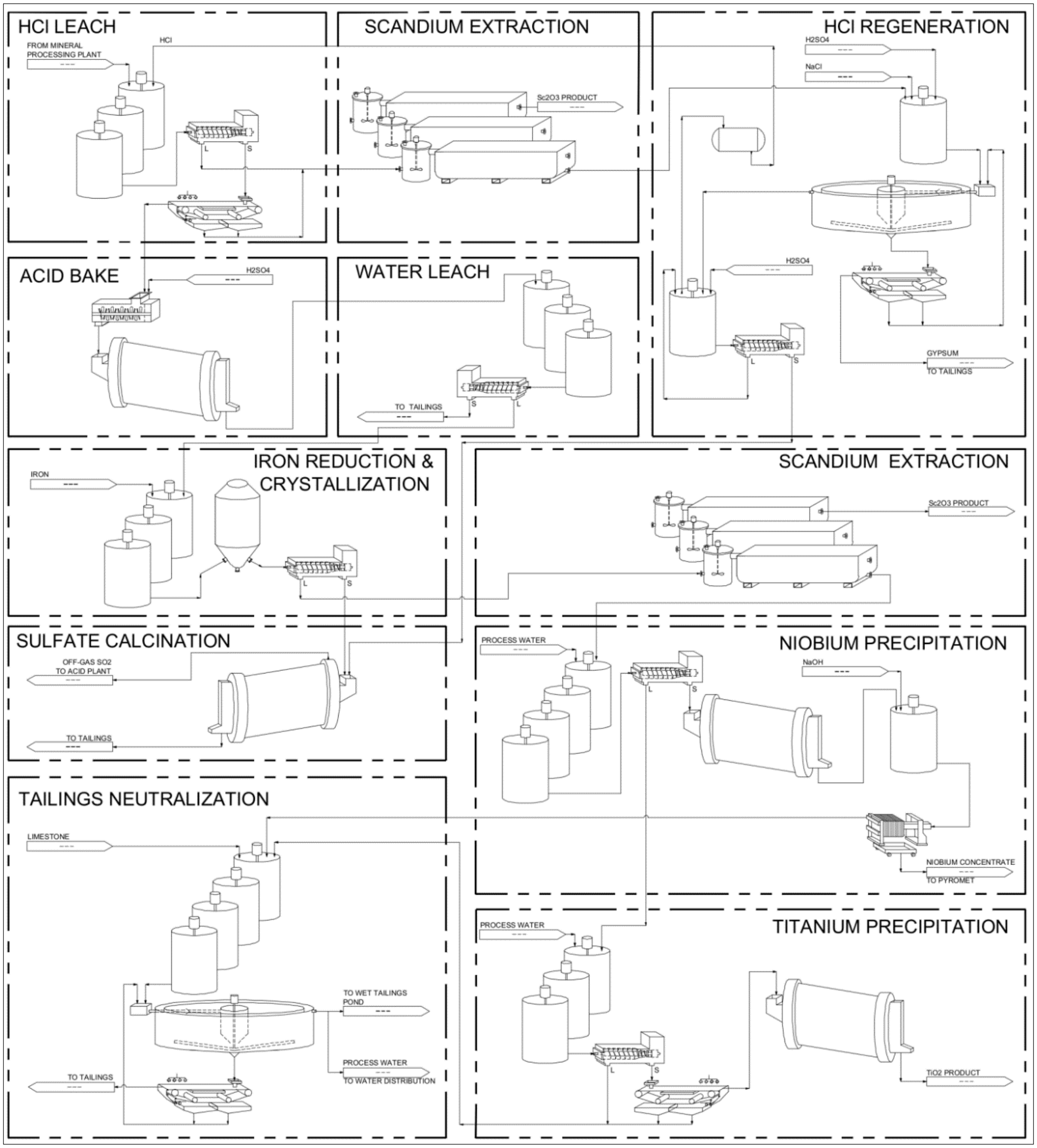
1. Hydrochloric Acid (HCl) Leach;
2. Hydrochloric Acid Scandium Extraction;
3. Sulfuric Acid Bake and Water Leach;
4. Iron Reduction and Crystallization;
5. Sulfuric Acid Scandium Extraction;
6. Niobium Precipitation and Phosphorus Removal;
7. Titanium Precipitation;
8. Sulfate Calcining;
9. Hydrochloric Acid (HCl) Regeneration;
10. Tailings Neutralization; and
11. Sulfuric Acid Plant (which is not shown on the flowsheet).

The unit processes selected for the hydrometallurgical flowsheet have been extensively reported on in literature and are predominately proven and existing processes.[4]

---

[4] An example of some of those references can be found below:

- Buxbaum, G, Industrial Inorganic Pigments (2008), p 52-53
- Grzmil, B. Y., Grela, D., & Kic, B., Hydrolysis of titanium sulphate compounds (2007)
- Gupta, C. K. & Mukherjee, T. K., Hydrometallurgy in Extraction Processes (1990), Volume 1, p 68-70
- Gupta, C. K. & Suri, A. K., Extractive Metallurgy of Niobium (1993), p 119
- Koerner, E. L., Smutz, M., & Wilhelm, H. A., Process of recovering niobium oxide from its mineralized materials (1941), US Patent 2259396A
- Koerner, E. L., Smutz, M., & Wilhelm, H. A., Niobium-tantalum separation (1963), US Patent 3107976A
- Young, R. S., Industrial Inorganic Analysis (1953), p 205



Source: Roche, 2015

Figure 17.2.1.1: Hydrometallurgical Processing Simplified Flowsheet

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 mineral processing plant product is pumped from the thickener underflow at a rate of 122 t/h (102 m³/h of slurry) and combined with 196 m³/h of hydrochloric acid from the hydrochloric acid regeneration unit and fed to the hydrochloric acid leach circuit. The hydrochloric acid leach circuit contains three parallel trains of three agitated tanks in series. The leaching reaction occurs at 40°C. The discharge of the hydrochloric leach tanks is dewatered successively with centrifuges and belt filters. The centrate and filtrate (222 m³/h) are sent to the hydrochloric acid scandium extraction unit ahead of the hydrochloric acid regeneration unit, while the belt filter cake (36.5 t/h) is sent to the acid bake and water leach unit.

The hydrochloric acid scandium extraction unit is a three-stage D2EHPA solvent extraction circuit followed by two stripping circuits used to selectively recover thorium and scandium from the leach solution. The hydrochloric acid leach solution is contacted with a D2EHPA organic solution in mixer-settler extraction units. The organic solution is first stripped of its thorium by a hydrochloric acid solution in mixer-settler stripping units, followed by stripping of its scandium by a sodium carbonate solution in mixer-settler stripping units. The scandium stripping stage also conditions the organic solution before it is recycled to the extraction step. The scandium-rich solution is evaporated to precipitate scandium hydroxide which is then dried to scandium trioxide. The thorium and scandium-free raffinate is sent to the hydrochloric acid regeneration unit, while the thorium strip solution is sent to tailings neutralization.

The hydrochloric acid regeneration unit uses sulfuric acid and sodium chloride to regenerate hydrochloric acid for the hydrochloric acid leach. The scandium-depleted hydrochloric acid leach centrate and filtrate (222 m³/h) are combined with sulfuric acid (21 m³/h) and sodium chloride (8 t/h) in the agitated low-temperature acid regeneration (LTAR) reactor tank where hydrochloric acid and gypsum are formed. The resulting slurry is sent to the LTAR thickener. The LTAR thickener underflow is continuously filtered using a belt filter to remove gypsum from the circuit which is transferred to the tailings (74 t/h). The LTAR thickener overflow is heated and combined with hot sulfuric acid in the high-temperature acid regeneration (HTAR) reactor tank at 130°C. Metal chlorides present in the solution react with sulfuric acid and form metal sulfates and hydrogen chloride which is vaporized and removed from the solution along with water. The vaporization of hydrogen chloride and water causes metal sulfates in the solution to precipitate. The hydrochloric acid gas generated from the HTAR reactor tank is condensed and returned to the hydrochloric acid leach unit (196 m³/h). The HTAR reactor tank discharge is transferred to a centrifuge. The cake, composed of metal sulfates, is sent to the ferrous sulfate calcining unit (106.8 t/h), while the centrate is sent to a flash tank. The liquid pumped from the bottom of the flash tank is returned to the HTAR reactor tank. The vapor from the top of the flash tank is condensed and sent to the tailings neutralization unit, while the non-condensables are sent to the plant scrubbing system.

The acid bake and water leach units are used to convert all remaining metals to sulfates (acid bake) and solubilize all soluble sulfates (water leach) while separating non-soluble impurities. The hydrochloric acid leach cake (36.5 t/h) is combined with 29 m³/h of pre-heated (150°C) fresh and recovered sulfuric acid and mixed at high percent solids in a pair of pug mills, before being fed into the acid bake calciner. The acid bake calciner provides heat to perform the acid bake reaction on the pug mill discharge. The acid bake discharge (25 t/h) is continuously fed via a screw conveyor into the water leach tanks, where it is combined with 64 m³/h of water. The water leach circuit is composed of a series of agitated tanks discharging to centrifuges. The centrate (79 m³/h), which contains the soluble sulfates, is sent to the iron reduction and crystallization unit while the cake (12 t/h) is discarded as tailings.

The iron reduction and crystallization unit is used to reduce iron (III) sulfate present in the solution to iron (II) sulfate and to remove a portion of it from the solution by cooling and crystallization. In the iron reduction and crystallization unit, the acidic water leach discharge (79 m³/h) is reacted with iron chips and/or shavings in a series of four agitated tanks. The discharge of the iron reduction tanks (81

m^3/h) is sent to the crystallizer circuit where is it cooled, causing iron (II) sulfate to precipitate out of the solution. The crystallizer discharges to centrifuges, where the centrate (66 m^3/h) is sent to the scandium extraction unit, while the cake (24 t/h) is sent to the ferrous sulfate calcining unit.

The sulfate solution scandium extraction unit is composed of a three-stage D2EHPA solvent extraction circuit followed by a two stripping circuits used to selectively recover thorium and scandium from the sulfate solution. The reduced iron water leach solution is contacted with a D2EHPA organic solution in mixer-settler extraction units. The organic solution is first stripped of its thorium by a hydrochloric acid solution in mixer-settler stripping units, followed by stripping of its scandium by a sodium carbonate solution in mixer-settler stripping units. The scandium stripping stage also conditions the organic solution before it is recycled to the extraction step. The scandium rich solution is evaporated to precipitate scandium hydroxide which is then dried to scandium trioxide. The thorium and scandium-free raffinate is sent to the niobium precipitation unit, while the thorium strip solution is sent to tailings neutralization.

The niobium precipitation unit uses water dilution to selectively hydrolise niobium and precipitate it as niobium oxide. The scandium-depleted crystallization discharge (66 m^3/h) is diluted with boiling water and flowed through a series of agitated tanks. The precipitation reaction temperature is maintained by direct steam injection in the agitated tanks. The discharge of the niobium precipitation tanks is dewatered using centrifuges. The centrate (243 m^3/h) is sent to the titanium precipitation unit while the cake (1.03 t/h) is sent to the niobium calciner. The calciner operates at 800°C and is used to drive off any remaining sulfur and water. The calcined material (1.01 t/h) is fed via a screw conveyor into the caustic leach tank, where it is combined with a sodium hydroxide solution. The caustic leach dissolves phosphorus from the calcined material, bringing the phosphorus concentration down to acceptable levels. The caustic leach discharge is sent to a filter press. The filtrate is sent to the tailings neutralization unit, while the filter cake (1.01 t/h) is sent to the pyrometallurgical plant for further processing.

The titanium precipitation unit triggers hydrolysis and precipitation as titanium oxyhydroxide using dilution and heat. The centrate from the niobium precipitation unit (243 m^3/h) is diluted with boiling water. Air may also be contacted if necessary through a series of agitated tanks. The reaction temperature is maintained by direct steam injection in the agitated tanks. The discharge of the titanium precipitation tanks is dewatered using centrifuges and belt filters. The filter cake on the belt filters is washed in a series of successive washing steps to ensure product purity. The centrate and filtrate (471 m^3/h) are sent to the tailings neutralization unit while the cake (4.1 t/h) is sent to a dryer/calciner. The material is dried and calcined using conditions which convert the titanium oxyhydroxide into a saleable titanium dioxide product (3.2 t/h).

The tailings neutralization unit is fed by the centrate and filtrate from the titanium precipitation unit as well as other acidic tailings streams. The tailings neutralization feed (517 m^3/h) is combined successively with limestone and the caustic leach tailings in a series of agitated tanks in order to raise the pH to around 7.0. The discharge is successively dewatered with a thickener and belt filters. The filtrate is returned to the thickener, the filter cake is sent to the tailings facility, and the thickener overflow is recycled as process water or sent for water treatment.

The ferrous sulfate calcining unit feeds the combined crystallization cake with the acid regeneration metal sulfate cake to a calciner to convert iron and magnesium sulfates to oxides while forming a mixture of gaseous sulfur dioxide and sulfur trioxide. These gases are further converted to sulfuric acid.

The ferrous sulfate from the crystallizer centrifuge cake (24 t/h) is combined with the metal sulfates from the HTAR centrifuge cake (107 t/h) and fed via a screw conveyor into the ferrous sulfate calciner. The calciner operates at a temperature of 1,000°C to decompose iron sulfate and magnesium sulfate into their respective oxides while producing sulfur dioxide and trioxide. The sulfur dioxide and trioxide gases are sent to the sulfuric acid plant for further treatment, while the metal oxides are sent to the tailings facility.

The ferrous sulfate calcination off-gas produced at a rate of 153,700 Nm^3/h containing 10.5% SO_2 gas is treated to reduce the sulfur dioxide emissions to the atmosphere. This process is achieved by oxidizing sulfur dioxide to sulfur trioxide, and converting it to sulfuric acid. The sulphur dioxide gas is cleaned through a series of electrostatic precipitation and scrubbing steps. The clean gas is then diluted with air and dried in a drying tower prior to entering the contact process. These steps allow the acid plant to produce sulfuric acid of an acceptable quality, to protect downstream equipment and for reuse in the hydrometallurgical plant. The circuit up to this point is suction fed, feeding into two single-stage centrifugal blowers which are used to compress the inlet gas to the contact process.

A sulfur burning plant operates in parallel with the ferrous sulfate calcination unit to allow for the production of 3,030 t/d of pure sulfuric acid (100% basis) required by the hydrometallurgical plant. Molten sulfur is delivered by train, unloaded and stored in storage tanks. Molten sulfur is pumped to the sulfur furnace where it is converted to sulfur dioxide. The furnace off-gas, at a temperature of about 950°C to 1,000°C is cooled through a waste heat boiler producing high pressure steam used in the hydro plant. The cooled gas is sent to the contact section for the conversion of sulfur dioxide to sulfur trioxide.

The contact process consists of multiple catalyst bed conversion stages with inter-stage gas-cooling heat exchangers followed by two absorption stages. The conversion stages convert sulfur dioxide to sulfur trioxide, while the absorption stages capture the sulfur trioxide to produce concentrated sulfuric acid. Primary conversion is obtained in the first three catalyst beds. The sulfur trioxide-rich gas formed is absorbed in the intermediate absorber tower with sulfuric acid. The first sulfur trioxide absorption results in a higher overall conversion rate allowing the remaining gas to convert in the final catalyst beds. The sulfur trioxide formed in the last conversion stage is absorbed in the final absorber.

A tail gas scrubbing system after the absorber towers reduces sulfur dioxide and other pollutants contained in the acid plant exhaust gas. A gas/liquid contact scrubber tower is designed to reduce remaining pollutant concentrations to within environmental regulations by reacting the gases with an alkaline solution.

The acid plant is designed for an overall conversion rate of 99.7% and a product acid concentration of 96%. The tail gas scrubbing system further reduces the SO_2 in the gas to achieve low emissions to the environment.

17.2.2 Hydrometallurgical Process Plant Design Criteria

The hydrometallurgical process design criteria have been established based on bench and pilot scale testwork, conducted by SGS and Hazen, as well as Roche's in-house database from similar projects, and standard industry practices. The key items are listed in Table 17.2.2.1.

Table 17.2.2.1: Hydrometallurgical Processing Design Criteria

Category	Description	Value	Units
Hydrochloric Acid (HCl) Leach	Feed rate (dry)	122.3	t/h
		2700	t/d
	Feed moisture content	25	%w/w
	Temperature	40	°C
	Residence Time	4	h
	Final filtrate acid concentration	100	g / L residual
	Solids moisture content after filtration	25	%w/w
	Recovery to leachate		%
	Nb	0	
	Ti	0	
	Fe	64	
	Sc	69	
	Recovery to residue		%
	Nb	100	
	Ti	100	
	Fe	36	
	Sc	31	
Hydrochloric Acid (HCl) Regeneration	Temperature	150	°C
	Residence Time	1	h
Sulfuric Acid Bake	Temperature Mixer	150	°C
	Temperature Bake	300	°C
	Residence Time	3	h
	Sulfuric Acid Ratio	1500	kg/t
	Sulfuric Acid Evaporation	25	% w/w
	Sulfuric Acid Recovery by Condensing	100	% w/w
	Phosphoric Acid Recovery to Solids	25	% w/w
	Recovery		%
	Nb	98	
	Ti	98	
	Fe	100	
	Sc	100	
Water Leach	Temperature	95	°C
	Residence Time	4	h
	Water addition ratio	1	L H_2O / kg feed
	Centrifuge residue % Solids	90	%w/w
	Recovery to solution		%
	Nb	100	
	Ti	100	
	Fe	100	
	Sc	100	

Category	Description	Value	Units
Iron Reduction	Temperature	95	°C
	Residence Time	2	h
	Iron ratio	0.88	mol Fe / mol ($Fe_2(SO_4)_3$ + $TiOSO_4$)
	Conversion Fe	 100	%
Niobium Precipitation	Temperature	100	°C
	Residence Time	4	h
	Dilution Ratio	2.5 : 1	
	Centrifuge Feed %Solids	0.4	%w/w
	Centrifuge Cake %Solids	50	%w/w
	Recovery to solution Nb Ti Fe	 95 4 0	%
Niobium Calcination	Temperature	800	°C
	Residence Time	3	h
Titanium Precipiation	Temperature	100	°C
	Residence Time	2	h
	Centrifuge Feed %Solids	0.5	%w/w
	Centrifuge Cake %Solids	80	%w/w
	Cake water wash cycle time	5	min
	Cake acid wash cycle time	5	min
	Cake water wash cycle time	5	min
	Recovery Nb Ti Fe	 97 98 1.1	%
Titanium Calcining	Drying Temperature	300	°C
	Calcining Temperature	1000	°C
	Residence Time	2	h
Tailings Neutralization	Residence Time	1	h
	Final pH	7	
Scandium Solvent Extraction circuits (chlorides and sulfates)	Temperature	40	°C
	Extraction O:A ratio Extraction Organic transfer ratio	1/3 1/40	
	Th Strip O:A ratio Th Strip Organic transfer ratio Th Strip Solution - HCl	1/1 1/1 12	 %w/w
	Sc Strip O:A ratio Sc Strip Organic transfer ratio Sc Strip Solution - Na_2CO_3	1/2 1/1 10	 %w/w
	Scandium Recovery	90	%
Scandium Precipitation	Temperature	Ambient	°C
	Residence Time	1.5	h
	Scandium Precipitate Filter Cake Moisture	25	%w/w
	Scandium Recovery	100	%
Ferrous Sulfate Calcining	First Stage Temperature	300	°C
	First Stage Residence Time	2	h
	Second Stage Temperature	1000	°C
	Second Stage Residence Time	2	h
Sulfuric Acid Plant	Gas % SO_2(g)	10.46	%w/w
	Gas flowrate (0°C, Atm pressure)	153706.7	Nm^3/h
	Overall Conversion SO_2 to SO_3	99.7	%
	Acid Production (100% H_2SO_4)	3030	tpd
	Required Acid Strength	96	%w/w
	Recovered Acid Temperature	Ambient	°C

Source: Roche, 2015

17.2.3 Mass Balance

Based on the design criteria and the flowsheet, a mass and energy balance for the hydrometallurgical processing plant has been developed. The mass balance was prepared for an average feed of 2,700 t/d or 122.3 t/h with 92% plant availability at 0.80% 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 using the flowsheet integrator METSIM.

17.2.4 Process Equipment

Based on the design criteria and mass balance major process equipment as well as some minor equipment has been sized. These pieces of equipment have been used to determine the capital and operating costs for the Project. An allowance was made for some minor equipment and facilities where it was required. The major equipment units are listed in Table 17.2.4.1.

Table 17.2.4.1: Hydrometallurgical Processing Major Equipment List

Equipment Name	Qty	Description / Size / Model
HCl Leach		
HCl Leach Tanks and Agitators	10	5.791 m dia. x 7.620 m height
HCl Leach Centrifuge	5	Bowl 0.470 m dia x 1.702 lg 4.72 m lg. x 1.27 m width x 1.57 m height
HCl Leach Belt Filter	3	4 m x 84m²
HCl Leach Scandium Extraction		
Sc Extraction Mixer-Settler	20	Settler: 12.1 m lg. x 2.7 m width x 1.8 m height Mixer: 1.524 m dia x 1.828 m height
Th Strip Mixer Settler	4	Settler: 3.6 m lg. x 0.9 m width x 0.7 m height Mixer: 0.914 m dia x 0.812 m height
Sc Strip Mixer Settler	2	Settler: 3.6 m lg. x 0.9 m width x 0.7 m height Mixer: 0.914 m dia x 0.812 m height
HCl Regeneration		
LTAR Tank and Agitator	1	6.706 m dia x 7.625 m height
LTAR Thickener	1	30m dia x 3 m height
LTAR Belt Filter	6	4m x 74 m²
LTAR Centrifuge	3	Bowl 0.470 m dia x 1.702 lg 4.72 m lg. x 1.27 m width x 1.57 m height
Sulfuric Rxn Tank and Agitator	1	6.706 m dia x 7.925 m height
HCl Condenser	1	3.048 m dia x 4.572 m length
Acid Bake		
Acid Bake Pug Mills	2	4.877 m width x 7.620 m length
Acid Bake Direct Heat Rotary Kiln	1	3.658 m dia. x 45.720 m length
Water Leach		
WL Tanks and Agitators	4	4.877 m dia x 6.401 m height
WL Centrifuges	3	Bowl 0.470 m dia x 1.702 lg 4.72 m lg. x 1.27 m width x 1.57 m height
Iron Reduction And Crystallization		
Iron Reduction Tanks and Agitators	4	3.658 m dia. x 5.486 m height
Crystallizer	2	Flow Rate 78 m³/h
FeSO4 Crystallizer Centrifuges	2	Bowl 0.470 m dia x 1.702 lg 4.72 m lg. x 1.27 m width x 1.57 m height
Scandium Extraction		
Sc Extraction Mixer Settler	5	Settler: 15.2 m lg. x 2.7 m width x 1.8 m height Mixer: 1.828 m dia x 1.828 m height
Th Strip Mixer Settler	4	Settler: 3.6 m lg. x 0.6 m width x 0.8 m height Mixer: 0.609 m dia x 0.711 m height
Sc Strip Mixer Settler	2	Settler: 3.6 m lg. x 0.6 m width x 0.8 m height Mixer: 0.609 m dia x 0.711 m height
Niobium Precipitation		
Niobium Precipitation Tanks and Agitators	5	6.096 m dia. x 7.620 m height
Niobium Centrifuges	5	Bowl 0.470 m dia x 1.702 lg 4.72 m lg. x 1.27 m width x 1.57 m height
Niobium Direct Heat Rotary Kiln	1	0.914 m dia. x 6.096 m length
Caustic Leach Tanks and Agitators	3	1.829 m dia. x 3.048 m height
Caustic Leach Filter Press	2	0.5 m x 0.5 x 24 plates
Titanium Precipitation		
Titanium Precipitation Tanks and Agitators	4	7.315 m dia. x 8.839 m height
Titanium Centrifuges	6	1.375 m width x 5.000 m length
Titanium Belt Filters	2	2.1m x 17.6 m²
Titanium Concentrate Direct Heat Rotary Kiln	1	1.219 m dia. x 15.240 m length
Ferrous Sulfate Calcining		
$FeSO_4$ Direct Heat Rotary Kiln	3	4.420 m dia. x 36.576 m length
Tailings Neutralization		
Tailings Neutralization Tanks and Agitators	4	7.315 m dia. x 8.839 m height
Tailing Dewatering Thickener	1	30 m dia. x 3 m height
Tailing Dewatering Belt Filters	3	4 m x 74 m²

Source: Roche, 2015

17.2.5 Power Requirements

The power requirements for the major areas are listed in Table 17.2.5.1.

Table 17.2.5.1: Hydrometallurgical Processing Power Demand by Area

Area	Installed (kW)
HCl Leach Unit & Scandium Extraction (1/2)	220
HCl Regeneration	300
Acid Bake & Ferrous Sulfate Calcining	2,400
Water Leach, Iron Reduction/Crystalization, & Sc Extraction (2/2)	240
Niobium Precipitation Unit & Titanium Precipitation Unit	680
Tailings Neutralization Unit	50
Reagent Services	100
Services (water, air, steam)	400
Sulfuric Acid Plant	10,814
Total	**15,204**

Source: Roche, 2015

17.2.6 Plant Layout

The hydrometallurgical processing building houses the mineral processing circuit (grinding, thickening and flocculant preparation) as well as ten (10) hydrometallurgical units: Hydrochloric Acid (HCl) Leach, Hydrochloric Acid Scandium Extraction, Sulfuric Acid Bake and Water Leach, Iron Reduction and Crystalization, Sulfuric Acid Scandium Extraction, Niobium Precipitation and Phosphorus Removal, Titanium Precipitation, Sulfate Calcining, Hydrochloric Acid (HCl) Regeneration, and Tailings Neutralisation in addition to an electrical room and mechanical and electrical shops for Hydromet plant maintenance. The dimensions of this area are 200 m x 120 m.

The Sulfuric Acid Plant will be in a separate building and the dimensions of this area are 100 m x 200 m.

17.2.7 Tailings Pumps and Piping

The hydrometallurgical tailings will go through the tailings neutralization unit, where they will be brought to a neutral pH of 7 using limestone and spent caustic leach sodium hydroxide prior to being thickened and filtered. The cake will be sent to the tailings facility, while the overflow produced will be sent to water treatment or recycled as process water.

17.2.8 Hydrometallurgical Plant Water Management

In order to assess water management in the hydrometallurgical plant, a water balance was developed with plant requirements based on 2,700 t/d throughput and 92% availability. A net total of 101 m^3/d of treated water will be required for washing of the filter cakes and reagent preparation. Treated water will be supplied from the mine dewatering operation and water treatment plant. The overflow of the tailings neutralization thickener will be pumped to the water treatment plant, where it will be treated and recycled as process water or discharged to the environment.

17.3 Pyrometallurgical Plant

17.3.1 Pyrometallurgical Process Plant Design Criteria

The preliminary pyrometallurgical process design criteria have been established based on bench scale testwork, conducted initially by XPS Consulting & Testwork Services (XPS) followed by testwork by Kingston Process Metallurgy (KPM), together with Roche's in-house database from similar projects, and standard industry practices. The key criteria have been listed below in Table 17.3.1.1.

Table 17.3.1.1: Pyrometallurgical Processing Design Criteria

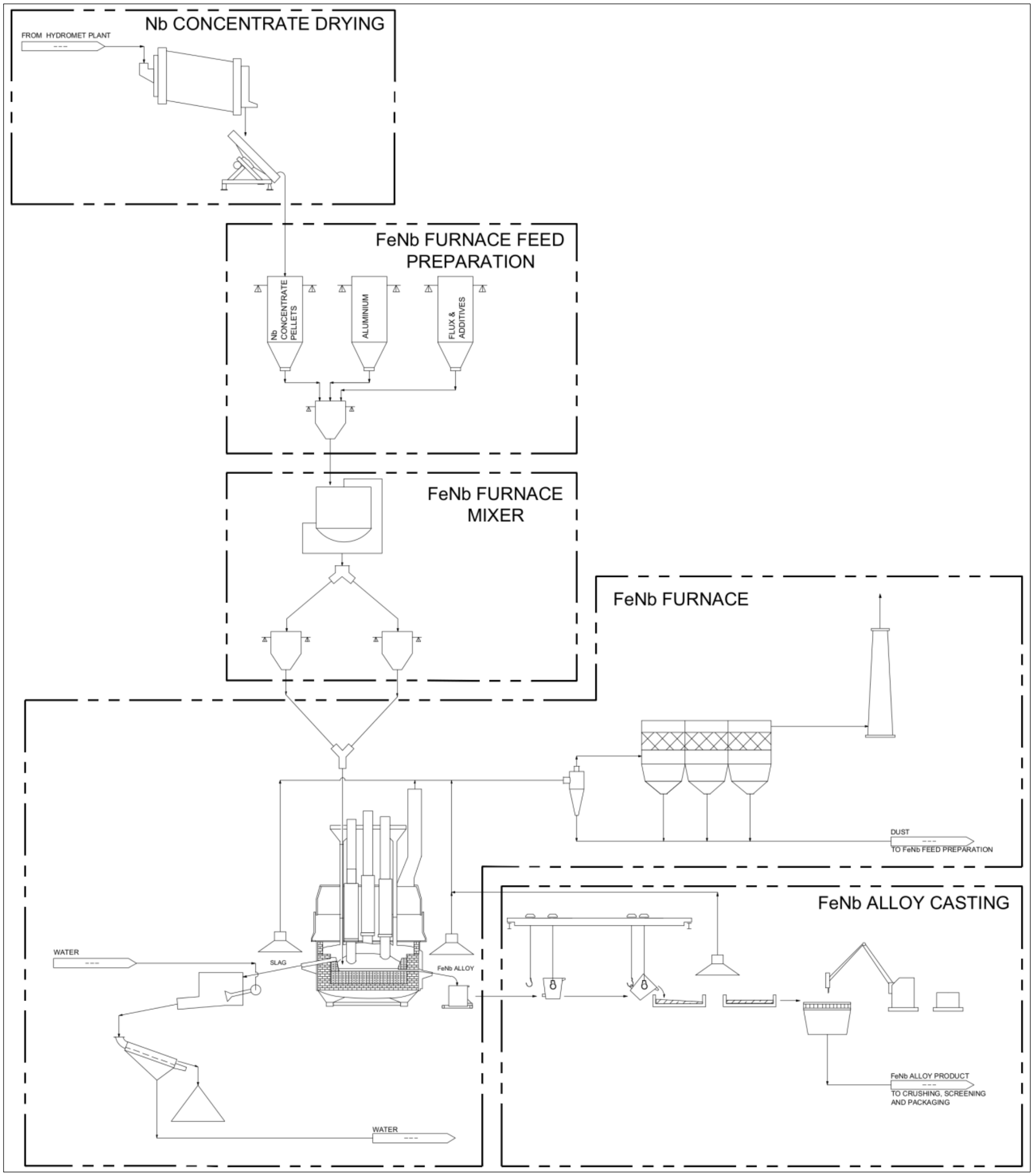
Section	Description	Value	Units
Nb Concentrate Drying & Pelletizing	Nb concentrate feed rate (dry basis)	1.01	t/h
		22.3	t/d
	Moisture content removed on drying	20	%
Niobium Concentrate Composition - Feed to Dryer (dry basis)	Nb_2O_5	90.3	%w/w
	TiO_2	7.00	%w/w
	P_2O_5	0.10	%w/w
	Al_2O_3	2.60	%w/w
Dryer	Temperature	300	^oC
	Dust Loss (all dusts recycled to furnace)	0.0	%
FeNb Furnace Feed Preparation	Nb Concentrate Pellets d_{80} (vol based)	6	mm
Nb Concentrate Pellets Feed Bin	Average Pellets feed rate	1.01	t/h
	Number of bins	2	#
	Storage time	7	days
	Capacity	156	t
Aluminum (Al) Powder Feed Bin	Aluminum (Al) feed rate	0.38	t/h
	Number of bins	1	#
	Storage time	14	days
	Capacity	118	t
Iron (Fe) Powder Feed Bin	Iron (Fe) feed rate	0.19	t/h
	Number of bins	1	#
	Storage time	14	days
	Live capacity	58.5	t
Hematite (Fe_2O_3) Powder Feed Bin	Hematite (Fe_2O_3) feed rate	0.18	t/h
	Number of bins	1	#
	Storage time	14	days
	Live capacity	55.6	t
Fluorspar (CaF_2) Powder Feed Bin	Fluorspar (CaF_2) feed rate	0.01	t/h
	Number of bins	1	#
	Storage time	14	days
	Live capacity	3.9	t
Lime (CaO) Feed Bin	Lime (CaO) feed rate	0.25	t/h
	Number of bins	1	#
	Storage time	14	days
	Live capacity	76.7	t
FeNb Furnace – Alumino-thermic reduction	Total Feed to FeNb Furnace	2.02	t/h
	Operating Temperature	1,650	^oC
FeNb Furnace Power	Smelt Electric Power	675	kW
	Power Consumption per tonne Nb Concentrate pellets	347	kWh/t
	Furnace Thermal Efficiency	60.0	%
	Total Peak Power Input	1,125	kW
	Furnace Design Power	1,500	kW
	Nb Recovery	97	%

Section	Description	Value	Units
FeNb Furnace - FeNb Alloy Composition	Nb	65.8	%w/w
	Nb Target Standard Grade	65.0	%w/w
	Fe	33.6	%w/w
	Ti	0.18	%w/w
	P	0.04	%w/w
	Al	0.43	%w/w
FeNb Alloy Tapping and Casting	FeNb Alloy Flowrate	0.93	t/h
	FeNb Alloy Tapping Schedule	3	taps/12 hour shift
		6	taps/day
	FeNb Alloy Tapping Time	10.0	min/tap
	FeNb Alloy Tapping Flowrate	3.42	t/tap
		20.5	t/day
	FeNb Alloy Density	7.7	t/m^3
Furnace Slag Rate	Slag average production rate	1.21	t/h
Furnace Slag Composition	Nb_2O_5	3.02	%w/w
	Fe_2O_3	0.37	%w/w
	TiO_2	5.61	%w/w
	Al_2O_3	61.36	%w/w
	CaO	20.54	%w/w
	P_2O_5	0.008	%w/w
Slag Granulation	Slag Flowrate	1.21	t/h
	Slag Tapping Schedule	3	taps/12 hour shift
		6	taps/day
	Slag Tapping Time	15.0	min/tap
	Slag Tapping Flowrate	4.44	t/tap
		26.7	t/day
	Water Flowrate Addition	2,400	m^3/hr
	Water Volume Requirement per tap	178	m^3/tap
		2,666	m^3/d
	Steam Produced	20.0	%
	Makeup Water Required	2.4	m^3/min
FeNb Furnace Off-gas Handling	Dusts all recycled to the furnace: Dust loss	0	%

Source: Roche, 2015

17.3.2 Flowsheet and Process Description

The preliminary pyrometallurgical processing plant flowsheet is presented in Figure 17.3.2.1. The proposed process is based on alumino-thermic reduction of niobium pentoxide (Nb_2O_5) in the Nb Concentrate which is a precipitate product supplied from the hydrometallurgical plant. Given the high grade of Nb in the Nb Concentrate (90% Nb_2O_5) with a very low phosphorous content, only a single reduction furnace is required, to produce ferro-niobium alloy (FeNb). Smelting energy is provided by the oxidation of aluminum, with additional electrical energy supplied to the submerged arc furnace (SAF), via a three electrode AC power input system.



Source: Roche, 2015

Figure 17.3.2.1: Pyrometallurgical Processing Simplified Flowsheet

Filtered Nb Concentrate is fed to a rotary dryer to drive off approximately 20% water in the concentrate. The rotary dryer is fired by a natural gas burner and operates at 300°C. The dried Nb concentrate is pelletized in a disc (pan) pelletizer unit, to produce pellets with a d_{80} of approximately 6 mm.

The dry Nb Concentrate pellets are fed by conveyor to the Furnace Feed Preparation Area (FPA), and stored in two closed bins, giving a total of 14 days storage time. Aluminum powder, the primary reductant, with hematite Fe_2O_3 powder to supply iron units and fluxes, lime (CaO) and fluorspar

(CaF_2) are each stored separately in bins. Seven flux and additive bins provide storage for aluminum, hematite, lime, fluorspar, furnace dusts and FeNb alloy fines recycle. These bins are loaded by crane, manually for each furnace additive, to maintain the required storage capacity. All bins are on load-cells, as part of the furnace feed preparation mass measurement system, automatically controlled via a PLC control system.

The furnace feed preparation is performed batch-wise with specified mass measurement of the Nb concentrate pellets with the required aluminum, hematite, and fluxes to satisfy a batch “recipe” for the production of on-spec FeNb alloy (65% Nb). Each batch is fed to a single mixer bin to feed to an Eirich Mixer for complete blending of the batch as a charge to the furnace.

Each charge is stored in one of two furnace charge bins, both on load cells. 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 would be stopped briefly for tapping of both molten slag and FeNb alloy, according to levels of slag and metal in the furnace.

The tapping of slag and FeNb alloy is scheduled over two 12 hour shifts:

1. Slag: 3 taps x per 12 hour shift, 15 minute tapping duration. 4.46 t per tap.
2. FeNb alloy: 3 taps x per 12 hour shift, 10 minute tapping duration. 3.42 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 is operated at 1,650°C.

Electrical energy is supplied to the furnace to initiate or 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 lime (CaO) and fluorspar (CaF_2).

On tapping, the slag is granulated with water, using a granulator system. High volume flows of water, via jets impact the slag continuously as the slag flows into a sloped launder system. The rapid cooling of the slag, forms slag granules at about 2 to 15 mm in diameter. Carried by the stream of water the slag granules pass over a screen to dewater and are transferred to a storage bunker area by conveyor belt. The slag is stored in a two concrete bunkers (one for loading, the other for transfer out, alternating). From the load out bunker the slag is removed with a front-end loader (FEL) for disposal in the tailing impoundment. Slag from either of the bunkers or from disposal areas, may be recycled back to the FeNb furnace to recover Nb units, when the slag Nb value is sufficient to cover the cost of such scavenger smelting.

The FeNb alloy metal is tapped via a short launder into a pre-heated ladle. After alloy metal tapping is complete the ladle is hoisted by overhead crane and moved to the casting bay area. The ladle is tipped using the crane’s second hoist, to pour the molten FeNb alloy (at about 1,600^{o}C), into cast iron refractory lined casting molds. The cast alloy in molds is allowed to cool for up to 24 hours, and thereafter the mold is transferred by crane to a casting grizzly, (square slots 350 mm x 350 mm), onto which the solidified FeNb alloy is discharged from the mold. A mechanical alloy breaker is used to break up the alloy through the grizzly into a tote box below. These totes are transferred by crane / vehicle to the FeNb Alloy Crushing and Screening plant, where the FeNb alloy is crushed and screened into specific size fractions, as required for sale to customers. FeNb alloy fines from the Crushing and Screening plant, if un-saleable are recycled back to the FeNb Furnace.

Dusts from the FeNb Furnace Feed Preparation Area, are captured via ducting through a dry cyclone – bag-house system. All dusts from this area are returned to the FeNb Furnace Feed Preparation Area 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, slag and alloy tapping fumes, and casting fumes above each mold are captured and ducted to the furnace off-gas dust collection cyclones and baghouse. These dusts are recycled to dust bin in the FeNb Furnace Feed Preparation Area.

Both the Feed Preparation and Furnace 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.

17.3.3 Mass Balance

Based on the design criteria and the pyrometallurgical flowsheet, a mass balance model with energy requirements was developed for the pyrometallurgical processing plant. The mass balance was prepared for an average feed rate of 22.3 t/d (dry basis) or 1.01 t/h at 90.3% Nb_2O_5 (63.1% Nb) with a 92% 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 assumed based on earlier KPM testwork, slag and alloy chemistry, and supported by other FeNb alloy industry operations.

The major element partition coefficients defining the mass balances are tabled below in Table 17.3.3.1. Other element distribution coefficients are provided in the Design Criteria back-up documents.

Table 17.3.3.1: FeNb Furnace Partition Coefficients

Element	% to Slag	% to Alloy Metal
Nb	3	97
Fe	1	99
Ti	96	4
Al	99	1
P	10	90

Source: Roche, 2015

From Table 17.3.3.1, the Nb recovery in the pyrometallurgical process plant is targeted at 97%, 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 dusts and fume from the Feed Preparation Area, FeNb Furnace off-gas, Tapping & Casting, and FeNb Crushing and Screening areas.

The FeNb alloy production was calculated as 20.5 t/d or 7,490 t/y, with a target standard alloy grade of 65.0%Nb. (Mass balance value at 65.8% Nb).

The FeNb Furnace slag output was estimated at 26.7 t/d or 9,730 t/y, with an estimated Nb_2O_5 grade of 3%. This slag may be recycled back to the FeNb Furnace, if economics permit.

17.3.4 Water Balance

The water requirements for the pyrometallurgical plant provide make up water to supply two systems:

1. The FeNb Furnace cooling systems for furnace sidewalls, tapping blocks and electrode clamps. This system is provided by the furnace package supplier, depending on the type and design of the furnace system. This water must be of a high purity, usually with a biocide and anti-scalant added. Treated potable water is added as make-up water, due to losses from the cooling tower.

2. The FeNb Furnace Slag granulation system for slag granulation, based on the slag production rates and tapping schedule, discussed above, the make-up water requirement is summarized below in Table 17.3.4.1.

Table 17.3.4.1: Pyrometallurgical Water Requirements

Water Item	Units	Value
Slag Water Flowrate Addition for Granulation	m^3/hr	2,400
Water Volume Required per Tap	m^3/tap	178
Steam Produced per tap	%	20
Steam Flowrate	m^3/min	2.4
Make-up water Required	m^3/min	2.4
Make-up water Required	m^3/d	213

Source: Roche, 2015

Note that for the above two water supply systems, there will be a first-fill water requirement to the plant water storage tanks. These water volumes have not been estimated. It is important to note that only dry baghouse units have been used in place of a gas scrubber – water quench system to clean the furnace off-gases and fumes. The potential presence of radio-active species (compounds with uranium and thorium, etc.) in the concentrates and solids which would contaminate the water, has dictated this decision.

17.3.5 Power Requirement

For the Pyrometallurgical process plant, the total installed power is 2,500 kW (including the furnace), and at a 92% utilization, the installed operating power requirement is 2,300 kW, which gives a total annual electrical energy consumption 20.15 MWh/y.

The power requirement was estimated based on scoping testwork and from calculations from previous FeNb testwork (XPS, KPM, and Hazen). Furnace equipment / technology vendors also confirmed the estimated power requirement for the FeNb Furnace, as summarized below in Table 17.3.5.1.

Table 17.3.5.1: FeNb Furnace Power Requirements

Furnace Power Parameter	Units	Value
Electrical Power per tonne Furnace Feed	kWh/t	347
Furnace Efficiency	%	60
Total Peak Power Input	kW	1,250
Furnace Design Power	kW	1,500

Source: Roche, 2015

17.3.6 Major Process Equipment

Based on the design criteria and mass balances, major process equipment as well as some minor equipment has been sized. These pieces of equipment have been 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 17.3.6.1.

Table 17.3.6.1: Pyrometallurgical Processing Major Equipment List

Equipment Name	Qty	Description/Size/Model
Nb Concentrate Drying		
Rotary Dryer	1	1 m dia. x 6.01 m length
Pan Pelletizer	1	6.01 m internal dia.
FeNb Furnace Feed Preparation		
Nb Concentrate Pellets Bins	2	4.75 m dia. x 9.50 m height
Aluminum Bin	1	3.25 m dia. x 6.50 m height
Fe Bin	1	3.25 m dia. x 6.50 m height
Fe_2O_3 Bin	1	3.00 m dia. x 6.00 m height
Fluorspar Bin	1	1.30 m dia. x 2.60 m height
Lime Bin	1	5.00 m dia. x 10.0 m height
FeNb Furnace Mixer		
FeNb Eirich mixer	1	Eirich model R19, 1800 kg capacity
FeNb Furnace		
FeNb Furnace	1	Submerged Arc Furnace, 1.5 MW
Granulator	1	Slag Capacity: 1.21 t/h
Dust Collector	1	High Temperature Baghouse at 600^{o}C

Source: Roche, 2015

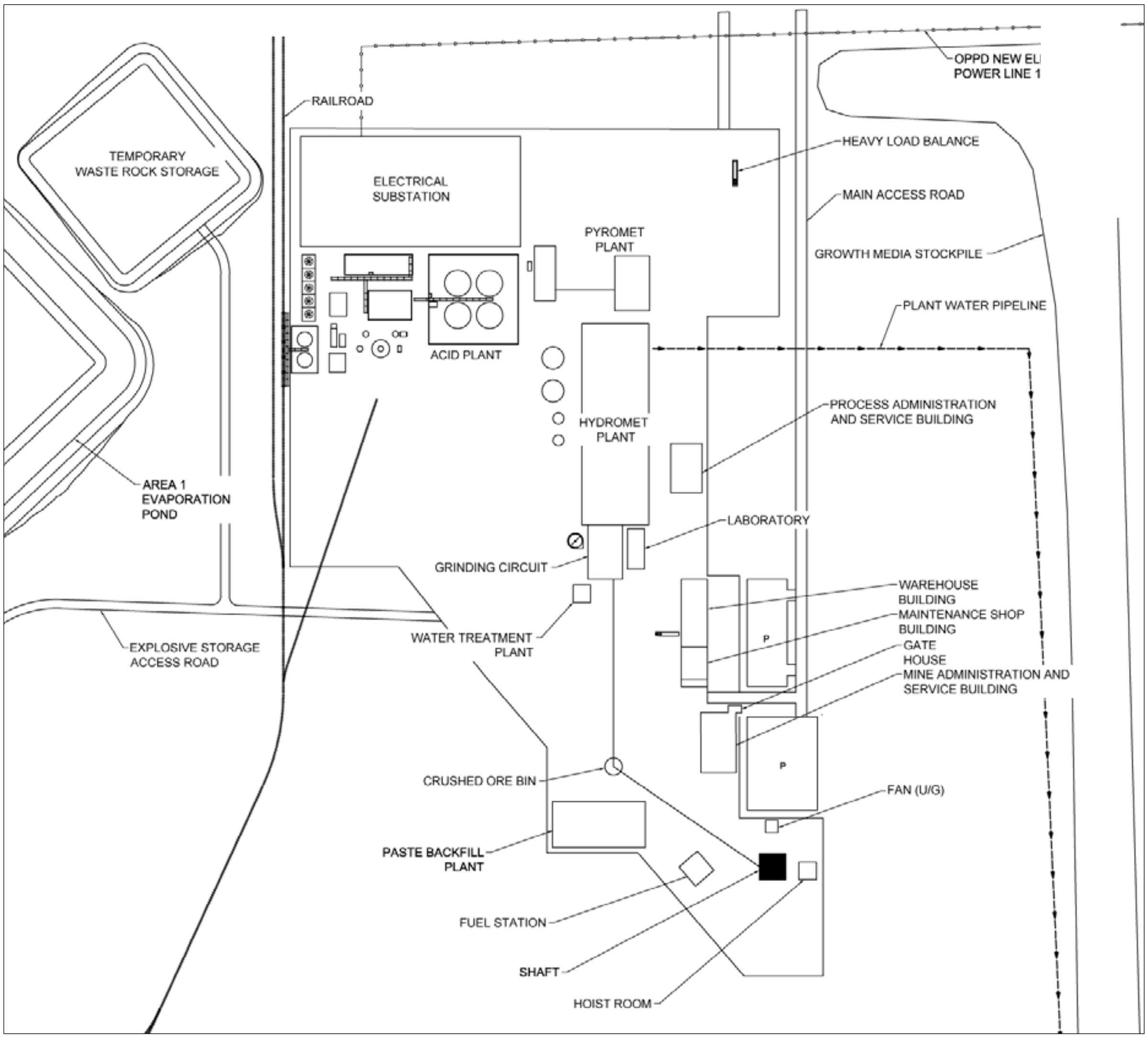
17.3.7 Pyrometallurgical Plant Layout

The pyrometallurgical processing plant is divided into two buildings (#1 and #2) for proper arrangement of the equipment and effective production operations.

The dimensions of the buildings #1 and #2 are approximately 50 m long x 19 m wide and 49 m long x 32 m wide, respectively.

18 Project Infrastructure

The project infrastructure includes a 161 kV electrical power line, a natural gas pipeline as well as other site infrastructure to support the mining and processing operations. It includes the site pad preparation, auxiliary buildings, utilities, access roads, railway with loading and unloading facilities, parking areas, etc. Figure 18.1 shows the site infrastructure layout.



Source: Roche, 2015

Figure 18.1: Site Infrastructure Layout

18.1 Infrastructure

18.1.1 Surface Infrastructure

Electrical Power Line & Main Substation

Electrical power for the project will be supplied by Omaha Public Power District (OPPD), which is the local electricity provider for South-east Nebraska. For the electricity requirements of the project, OPPD plans to build a new high voltage transmission line (161 kV) from their current substation to the mine site. The new transmission line is expected to be approximately 29 km long. OPPD will build a new electrical sub-station close to the process facilities. They will install one 33 MVA, 13,800 V distribution transformer to service the site; thus electricity will be delivered to NioCorp at medium voltage.

Site Power Distribution

NioCorp will be responsible for distributing electricity from OPPD's main substation at mine site to the various surface facilities on site. The site power distribution includes the above ground lines, underground cable trench, 15 kV cables, main 15 kV switchgear, secondary distribution switchgears, and emergency generator sets. It excludes the transformers, MCCs, etc., which are considered part of each building.

Telecommunications

Telecommunications for the project include all surface communications which includes the exterior backbone structure, telecommunication pathways, spaces and structured cables, IP network, digital land mobile radio communications, Wi-Fi communications, fixed voice communications, security, access control, video surveillance and enterprise system network.

General Plant Site Preparation and Parking

General plant site preparation has been planned and includes topsoil removal and storage, excavation, backfill material, drainage ditches and finishing surface to provide slopes and collect surface water. The site pad covers the following areas: process facilities, paste backfill facilities, hoist room, administration and services buildings, surface maintenance shop and warehouse, reagent storage and gate house. Sufficient space has been allocated to allow for a laydown area and parking lots for trucks and employees personal vehicles. Access to the mine site will be protected by a fence surrounding the area, automated gates for vehicles coming to and from the site, and pedestrian turnstiles for employees.

Access Road

An access road will be built to access the mine site from a local road. The road will lead to the main gate house and two parking lots: one for process employees and the other for the mine employees. The main lot will give access to the primary gatehouse and access to the process facilities and process administration building. A second entrance is also planned to give access to the hoist room and mine administration building. This second entrance will be operated remotely by the main gatehouse operator via video cameras and a remote access gate.

Auxiliary Buildings

The process buildings will be surrounded by auxiliary buildings which include the mine and process administration and services buildings, surface maintenance shop and warehouse, gate house, assay laboratory, and reagent storage.

Assay Laboratory

An assay laboratory is planned as a separate building and will include all the laboratory equipment and ventilation necessary to support the assay, wet, and environmental laboratories.

Surface Maintenance Shop and Warehouse

The surface maintenance shop will be equipped with proper equipment and tools to handle the maintenance of surface mobile equipment such as wheel loaders, dozer, pick-up trucks, boom truck, etc. The surface maintenance shop is combined with a warehouse which can be used to store spare parts, supplies, some reagents, etc.

Reagent Storage

A reagent storage building is planned to store reagents that will come in the form of super sacs, 45 gallons drums, etc.

Mine Administration and Service Building

The mine administration and service building is a two-story building and includes offices, cafeteria, conference rooms, dry, lockers, restrooms and showers. Also included are allowances for computers, software and licenses' fees for software.

Process Administration and Service Building

The process administration and service building is separate from the mine administration and service building to facilitate the displacement to and from the process facilities. It is a single story building and includes the same supplies as the mine administration building except for the lockers, men's and woman's dry changing rooms, showers, etc.

Gate House

A main gatehouse adjacent to the mine administration and service building is also planned to control access to and from the mine site. A second entrance is also planned to give access to the hoist room and mine workers. This second entrance will be operated remotely by the main gate house operator via video cameras and a remote access gate.

Process Water

Process water will be supplied by recycled water coming from the hydrometallurgical plant tailings dewatering circuit, supplemented with make-up water from the underground mine dewatering system. The underground mine water will be needed to start the process and will act, once in operation, as make-up water to avoid build-up of impurities in the hydrometallurgical circuit due to the recirculation of water. A pipeline will also be built to continuously bleed a fraction of the process water to the tailings evaporation ponds.

Water Treatment

A water treatment plant will be built for the desalination of the underground mine water (around 2,000 m^3/d) for use in the process. The treated water is required for gland seals, reagent preparation and cooling water.

Active Mine Dewatering and Pipeline System

This system is described in Section 18.3.

Potable Water

Potable water will come from the municipality water line located nearby. It will be used for showers, tap water, toilets, etc.

Site Fire Protection Loop

A site fire protection loop is planned around the buildings to distribute fire protection water to different buildings located within the site pad area. A fire water reservoir is also planned.

Sewage Treatment

Septic tanks are planned to accumulate and treat sewage.

Fuel Storage and Distribution

A surface fuel station including fuel tanks and pumps will be built to store a total of around 150,000 L of diesel fuel for surface equipment, mining equipment and emergency generator set(s). Piping to distribute diesel fuel to the underground has also been planned. A smaller gasoline tank with a pump is also planned for gasoline vehicles and/or smaller equipment.

Natural Gas Line

Northern Natural Gas Company and/or Black Hills Energy will build a natural gas pipeline to bring natural gas to a point near the Project location. NioCorp will be responsible for connecting its own receiving system to this pipeline and for the distribution of natural gas to the various surface facilities on the mine site.

Natural Gas Distribution

Natural gas will be used for heating buildings and mine ventilation air during the cold months and for process needs. Underground distribution is planned for each building. A provision has been made for natural gas heaters and ventilation systems on the surface, which supply the underground with heated air.

Surface Mobile Equipment

Surface mobile equipment was included to support the logistics required at the mine site. This equipment includes but is not limited to: pick-up trucks, dozers, wheel loaders, boom trucks, emergency environmental trailers, flatbed trucks, extendable boom forklifts, and skid steer loaders. These pieces of equipment are required to support the loading and unloading of goods, on-site material handling, maintenance, dry tailings handling and to ensure environmental emergency response is appropriate.

It is assumed that local emergency response is sufficient so that no ambulance or fire truck is required on site.

It is also assumed that most of the road maintenance will be given to local contractor and cost is included in the operating cost estimate.

Railway and Rail Loading/Unloading Facilities

In support of moving large quantities of various chemicals and minerals to and from Elk Creek, a railway system will need to be put in place.

In general terms, the railway requirements at the mine site will consist of:

- A rail unloading facility at the mine site for pressure unloading;
- A separate raised loading platform for loading from ground into box cars or gondolas;
- A gravity loading chute over railway tracks for bulk loading; and
- A number of yard tracks (3,000 to 5,000 ft in length) adjacent to the mine site facilities for the purpose of car storage – car placement and release.

From the mine facility railway site, the following railway installations will be required:

- The installation of approximately 7.2 km of track (115 lb rail) running from the mine site and connecting with the BNSF (Burlington Northern, Santa Fe Railroad) main line south of Elk Creek;
- An additional requirement for the installation of 3 to 4 transfer tracks (5,000 to 7,000 ft in length) adjacent to the BNSF main line will be required in order for BNSF to set-off, pick-up and exchange rail traffic with NioCorp;
- The two main line switches to be installed connecting NioCorp tracks with the BNSF main line will be:
  - Power operated switches controlled by the BNSF train dispatching center;
  - Equipped with snow-melters; and
  - Positive Train Control (PTC) compliant.

Molten sulfur unloading and handling for feeding the acid plant is included separately as part of the acid plant turn-key package.

Final Product Preparation

A provision for a building including automated packaging systems for the three products produced is also included as part of the auxiliary buildings. Drums will be used for ferroniobium packaging, super sacs for TiO_2, and jars for Sc_2O_3. The handling, packaging, and storage will be automated. It is assumed that the equipment for these three systems, as well as temporary storage, will be included under one building.

18.1.2 On-Site Infrastructure (Mining)

During the pre-production period of the Project, mine rock from the sinking of the shaft and the development of the underground workings, as well as some mineralized material encountered during development, will be stockpiled in a designated lined storage area. The material storage area will have appropriate surface water control structures included in the design. The mineralized material will be fed into the plant during plant operations. The waste material is planned to supplement the mine backfill system.

Growth Media Storage Area

During the construction process topsoil will be removed from the construction areas and stockpiled for use during the reclamation process. The material will be stored in a designated area and vegetated in a manner that minimizes erosion and with surface water controls in place.

Explosives Storage Area

The mine will utilize explosives during the mining process and a designated storage area will be located on site. The area will have controlled access and will have storage for both powder/gel and initiating caps. The facility will be operated by a qualified explosives manufacturer or contractor.

Underground Infrastructure

The underground facilities will include a mineralized material handling system including grizzly, feeders, crusher, and conveyance system, underground shop, warehouse, fuel storage and filling area, offices, explosives storage areas, electrical distribution system, water pumping and discharge system, service water, paste backfill distribution system, compressed air distribution, and ventilation system. The underground will be serviced by a shaft and a ventilation raise. The return air raise will have a fan system located underground near the vent hole as well as a bullet style emergency hoist system at the surface on the vent hole.

18.2 Tailings Storage Facility

Preliminary investigation performed by SRK included a comparison of potential TSF sites for both slurry and filtered (dry stack) tailings disposal options. The comparison considered potential engineering, strategic, permitting and closure issues:

- Engineering: Containment area, required reclaim for negative 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 impacts, 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; and
- Closure: Closure cover areas and volumes, seepage potential, and mass stability.

Of eight identified sites, Area 1 and Area 7 were considered appropriate for further evaluation. This evaluation included development and implementation of a preliminary foundation characterization plan for Area 1 and Area 7, as well as development of a preliminary water balance spreadsheet for both slurried and filtered tailings options for both sites.

In addition, further delineation of Waters of the US (WOUS) was conducted in Areas 7 and 1 as described in Section 20.1.1. For the PEA, this delineation was used to limit the footprints of the Area 7 and Area 1 TSFs.

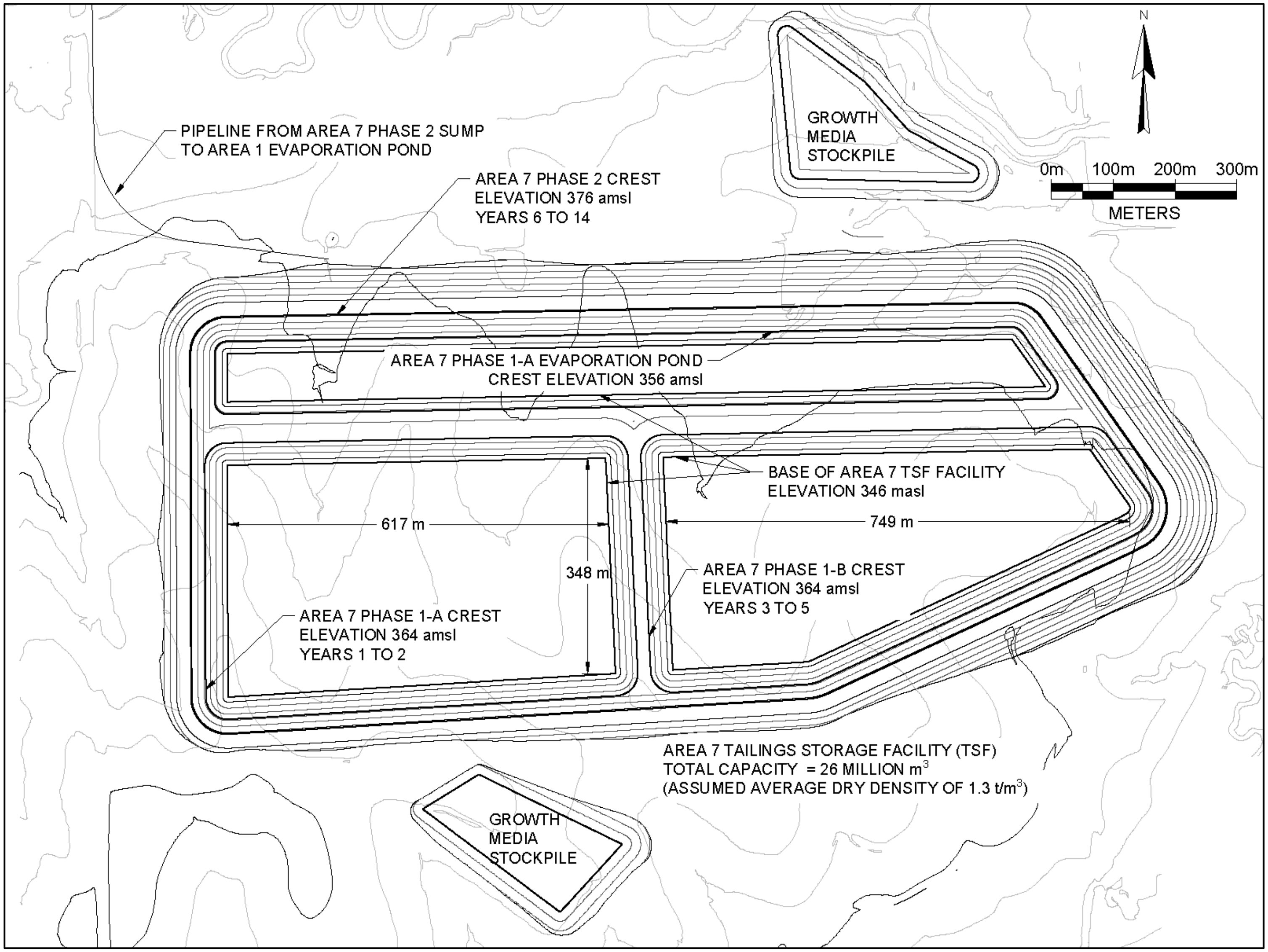
The facility incorporates the following assumed parameters and preliminary design details.

1. Design has been performed using 1 m contoured topography.
2. Liner and closure cover requirements for Area 7 and Area 1 TSFs are based on dam safety, solid waste, water storage and radioactive licensing regulations as shown in Table 320.1.1.

3. The tailings material will be predominantly dry (i.e., not in a slurry consistency) and incorporate two independent areas:
   a. An area for "dry stacking" and storage of tailings solids which will have composite liner consisting of a geo-synthetic liner overlying a compacted clay layer; and
   b. An evaporation pond which will be utilized for management of stormwater runoff and drainage from the tailings solids. This pond will be lined with two layers of geosynthetic liner (primary and secondary), sandwiching a permeable spacer that allows evacuation of all leakage through the primary liner to be collected in a sump and pumped back into the TSF, thereby providing a means of long-term leakage control as shown on Figure 18.2.4. This approach is known as a Leakage Collection and Recovery System (LCRS).
4. The tailings production rate is an average of 4,930 dry t/d consisting of water leach residue, gypsum residue, iron oxide (Fe_2O_3) and tailings neutralization residue. Water leach residue and the tailings neutralization residue will be filtered and are anticipated to be approximately 70:30 (solids: liquid) weight ratio. The gypsum residue and the iron oxide are products of calcined processes and are thus dry. As the materials will be dried or filtered prior to storage in the TSF, an average dry density of 1.3 t/m^3 is assumed for the mass tailings.
5. It is currently anticipated that the tailings materials will be transported via conveyor to the TSF. (Note: Trucking is an alternative option for cost trade-off in the feasibility study.) One meter of growth media will be removed prior to construction of the TSF and will be stockpiled at the locations shown on Figure 18.2.1 for use during closure cover construction.
6. The base of the Area 7 facility will be around 346 masl and incorporate the following phased construction schedule and details (Table 18.2.1 and Figures 18.2.1, 18.2.3, and 18.2.4):
   a. Area 7 Phase 1-A of the facility consisting of tailings solids storage (~ 36 ha), and an evaporation pond (~ 25 ha). The tailings solids storage area will have a crest elevation of 364 masl with a maximum embankment height of approximately 12 m. The evaporation pond will have a crest elevation of 356 masl with a maximum embankment height of 15 m. Area 7 Phase 1-A will be constructed for containment of approximately 5.3 Mt of tailings from Year 1 through Year 2.
   b. Area 7 Phase 1-B of the facility consists of an expansion of approximately 26 ha to the east of the Area 7 Phase 1-A tailings solids storage portion of the facility. No expansion of the Area 7 Phase 1-A evaporation pond is planned as it will be sufficient for containing runoff and drainage from Area 7 Phase 1-A and Phase 1-B. The Area 7 Phase 1-B tailings solids storage area will have a crest elevation of 364 masl with a maximum embankment height of approximately 25 m. Area 7 Phase 1-B will be constructed for containment of an additional 4.6 Mt of tailings from Year 3 through Year 5.
   c. Area 7 Phase 2 will consist of a downstream embankment raise of Area 7 Phase 1-A, Area 7 Phase 1-B and the Area 7 Phase 1-A evaporation pond for storage of tailings solids. A new evaporation pond covering approximately 20 ha will be constructed for Area 7 Phase 2 located in Area 1. Area 7 Phase 2 tailings storage will have a crest elevation of 376 masl with a maximum embankment height of approximately 39 m. Area 7 Phase 2 will be constructed for containment of an additional approximately 14 Mt of tailings from Year 6 through Year 14. All stormwater runoff and drainage will be pumped from a sump at the north toe of the Area 7 Phase 2 embankment to the Area 1 evaporation pond (Figure 18.2.1 and 18.2.2).
   d. The total capacity of the Area 7 facility is therefore 26.2 Mt.

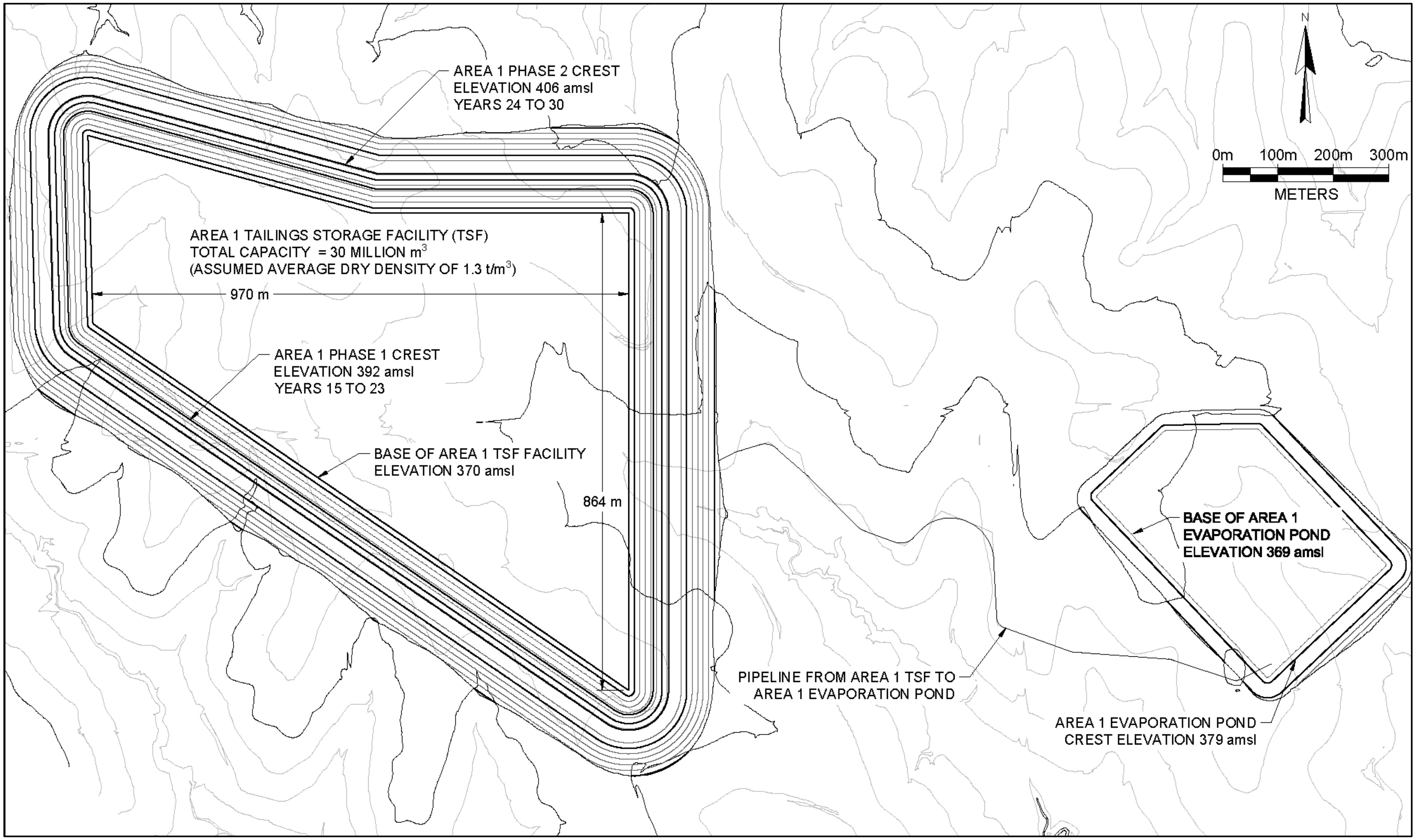
7. The base of the Area 1 facility will be around 370 masl and incorporate the following phased construction schedule and details (Table 18.2.2 and Figures 18.2.2, 18.2.3, and 18.2.4):
   a. The Area 1 evaporation pond utilized for water management of Area 7 Phase 2 will also be used for water management for Area 1 Phase 1 and Phase 2.
   b. Area 1 Phase 1 of the facility will cover approximately 87 ha for tailings solids storage. The tailings solids storage area will have a crest elevation of 392 masl with a maximum embankment height of approximately 27 m. Area 1 Phase 1 will be constructed for containment of approximately 16.1 Mt of tailings from Year 15 through Year 23.
   c. Area 1 Phase 2 will consist of a vertical expansion via downstream embankment raise of Area 1 Phase 1. It will have a crest elevation of 406 masl with a maximum embankment height of approximately 41 m. Area 1 Phase 2 will be constructed for containment of an additional approximately 13.5 Mt of tailings from Year 24 through Year 30.
   d. The total capacity of the Area 1 facility is therefore 29.6 Mt for a total storage (i.e., Area 1 plus Area 7) of 55.9 Mt.
8. Foundation preparation for all evaporation pond embankments, TSF starter embankments and embankment raises will incorporate removal of 1.5 m of native soils, re-compacted in layers to form a non-settling structure as shown on Figure 18.2.3 and 18.2.4.
9. All TSF and evaporation pond 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 on Figures 18.2.3 and 18.2.4.
10. All tailings embankment raises will incorporate the “downstream” method of tailings embankment construction, which requires progressive downstream relocation of the embankment crest with phased increases in embankment height as shown on Figure 18.2.3. This results in new raise construction being performed on original ground and not on top of tailings solids. This in turn allows independent extension of the TSF liner and embankment drainage elements, and prevents the facility from being affected by potential liquefaction of the tailings solids under seismic loads.
11. All TSF embankment sections will incorporate a 20 m crest width and 2 (horizontal) to 1 (vertical) side slopes as shown in Figure 18.2.3. Sod will be placed on the downstream faces of all Phase 1 embankments as an interim erosion control measure, and then removed prior to vertical Phase 2 expansions. The sod will then be placed on all Phase 2 downstream embankment slopes and will remain in place as a reclamation measure.
12. The components of the facility liner system as described below and shown in Figures 18.2.3 and 18.2.4:
   a. Foundation preparation to a depth of one meter incorporating TSF and evaporation pond base grading, and compaction of the sub-liner surface in layers to form a non-settling structure.
   b. The evaporation pond facilities will incorporate an 80-mil high-density polyethylene (HDPE) primary liner over a 60-mil HDPE secondary liner with a geonet drainage layer between the two liners. The geonet drainage layer will drain to either of two Area 7 evaporation pond sumps or of two Area 1 evaporation pond sumps). The LCRS sumps are geotextile-wrapped, gravel-filled facilities between the primary and secondary liners, with a riser pipe (perforated within the gravel) for leakage collection and removal. Within the sump areas, the soil beneath the secondary liner will be amended and compacted to create a low permeability soil layer beneath each sump area.

c. The TSF solids storage area base will incorporate an 80-mil high-density polyethylene (HDPE) primary liner over the compacted foundation, except for a portion of Area 7, Phase 2, which will be constructed using a pre-existing double-lined system in the Area 7 Phase 1-A evaporation pond.
d. The upstream face of the compacted embankment will incorporate a drained low-permeability compacted clay core (with bentonite amendment as necessary to achieve low permeability requirements), that is raised in a downstream configuration according to the phased construction. This will prevent potential ice damage to any liner on the upstream face of the embankment. To provide drainage on the upstream side of the low-permeability core, a continuous “perimeter drain” will be constructed within the embankment profile, which will include independent sumps to allow consistent pumping of collected water, thereby preventing leakage though the upstream face of the embankment. This drained, low-permeability core will tie into the above-liner drainage system described in Item 13.
e. A 500 mm thick above-liner, fine-sand drainage system, incorporating slotted pipes at 30 m centers, will be constructed above the lined base of the facility as shown on Figure 18.2.3. The above-liner drainage system will assist in reducing entrained moisture in the tailings via gravity drainage or pumped removal to the evaporation pond for each phase. This will minimize head on the TSF liner during its operational life, and provide a way to accelerate facility dewatering (primarily related to precipitation) and increase tailings consolidation during operations and closure.



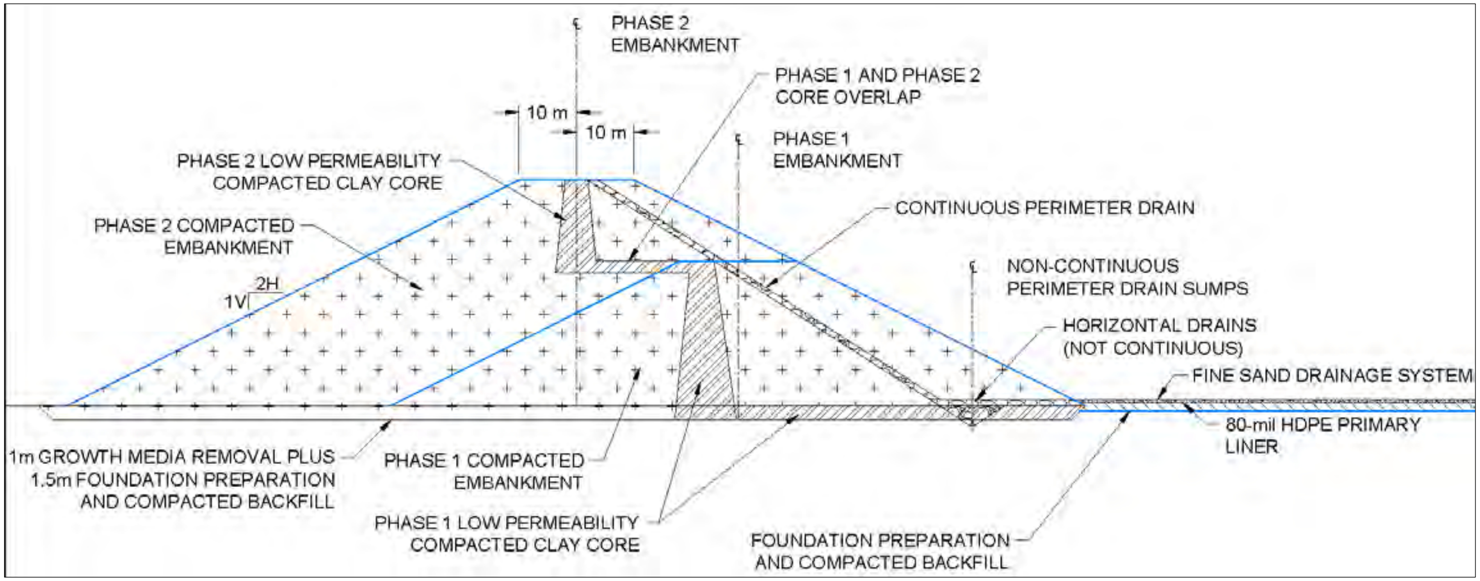
Source: SRK, 2015

Figure 18.2.1: PEA Concept Area 7 Tailings Storage Facility Layout



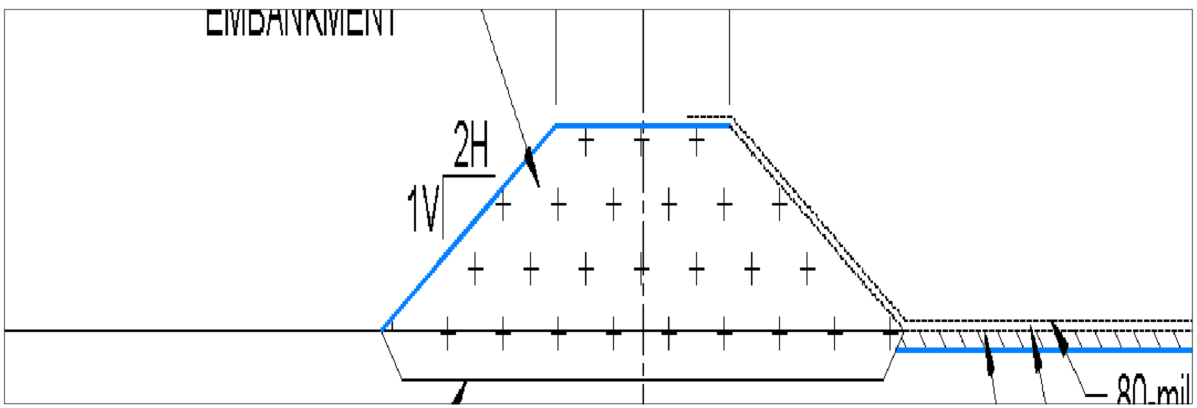
Source: SRK, 2015

Figure 18.2.2: PEA Concept Area 1 Tailings Storage Facility Layout



Source: SRK, 2015

Figure 18.2.3: PEA Concept Tailings Storage Area Embankment Cross-Section



Source: SRK, 2015

Figure 18.2.4: PEA Concept Evaporation Pond Embankment Cross-Section

Table 18.2.1: Area 7 Tailings Storage Facility Stage-Area-Capacity Data

Phase	TSF Elevation (mamsl)	Area (m^2)	Cumulative Volume (m^3)	Capacity (1) (t)	Years	Rate of Rise (m/y)
Phase 1-A	346	223,918				
	348	231,818	455,736	592,457	0.32	6.31
	350	239,819	927,373	1,205,585	0.65	6.10
	352	247,920	1,415,112	1,839,645	0.98	5.89
	354	256,122	1,919,153	2,494,899	1.34	5.70
	356	264,424	2,439,698	3,171,608	1.70	5.52
	358	272,826	2,976,948	3,870,033	2.07	5.35
	360	281,329	3,531,104	4,590,435	2.46	5.19
	362	289,933	4,102,366	5,333,076	2.85	5.03
	364	298,637	4,690,936	6,098,217	3.26	4.88
Phase 1-B	346	188,207				
	348	196,048	384,255	499,531	3.20	7.48
	350	203,989	784,291	1,019,579	3.56	7.19
	352	212,030	1,200,310	1,560,403	3.94	6.91
	354	220,172	1,632,513	2,122,266	4.33	6.65
	356	228,415	2,081,099	2,705,429	4.74	6.41
	358	236,757	2,546,271	3,310,153	5.16	6.18
	360	245,201	3,028,230	3,936,699	5.59	5.97
	362	253,745	3,527,176	4,585,328	6.04	5.76
	364	262,389	4,043,309	5,256,302	6.51	5.57
Phase 2	346	100,889				
	348	111,995	212,884	276,749	6.66	13.51
	350	123,201	448,079	582,502	6.82	12.22
	352	134,507	705,787	917,523	7.00	11.16
	354	145,914	986,208	1,282,070	7.20	10.25
	356	157,422	1,289,544	1,676,407	7.41	9.48
	358	184,168	1,631,133	2,120,473	7.64	8.42
	360	198,741	2,014,042	2,618,255	7.91	7.51
	362	212,794	2,425,577	3,153,250	8.20	6.99
	364	227,904	2,866,276	3,726,158	8.50	6.52
	366	835,852	3,930,032	5,109,041	9.24	2.70
	368	851,608	5,617,491	7,302,738	10.42	1.70
	370	867,464	7,336,562	9,537,531	11.61	1.67
	372	883,421	9,087,447	11,813,681	12.83	1.64
	374	899,478	10,870,346	14,131,450	14.07	1.61
	376	915,636	11,777,903	15,311,274	14.70	1.58

(1) Tonnes of storage are based on an assumed dry density of 1.3 t/m^3.
Source: SRK, 2015

Table 18.2.2: Area 1 Tailings Storage Facility Stage-Area-Capacity Data

Phase	TSF Elevation (mamsl)	Area (m^2)	Cumulative Volume (m^3)	Capacity [(1)] (t)	Years	Rate of Rise (m/y)
Phase 1	370	550,635				
	372	564,176	1,114,811	1,449,254	14.85	2.58
	374	577,818	2,256,805	2,933,846	15.64	2.52
	376	591,560	3,426,183	4,454,037	16.46	2.46
	378	605,403	4,623,145	6,010,089	17.29	2.40
	380	619,346	5,847,894	7,602,262	18.14	2.35
	382	633,389	7,100,629	9,230,818	19.01	2.30
	384	647,533	8,381,552	10,896,018	19.90	2.24
	386	661,778	9,690,864	12,598,123	20.81	2.20
	388	676,123	11,028,765	14,337,394	21.74	2.15
	390	690,568	12,395,456	16,114,093	22.69	2.10
	392	705,114	13,791,138	17,928,480	23.67	2.06
Phase 2	394	719,761	15,216,013	19,780,817	24.66	2.02
	396	734,507	16,670,281	21,671,365	25.67	1.98
	398	749,355	18,154,143	23,600,386	26.70	1.94
	400	764,302	19,667,800	25,568,139	27.75	1.90
	402	779,350	21,211,452	27,574,888	28.83	1.86
	404	794,499	22,785,301	29,620,892	29.92	1.83
	406	809,748	24,389,548	31,706,413	31.04	1.79

Source: SRK, 2015

Operational management of tailings material and supernatant water is addressed under Sections 17.1.7 and 17.2.7 and summarized below:

1. Tailings solids will be conveyed from the process plant to the planned deposition locations at the TSF. From these locations, the tailings will be placed by an equipment fleet consisting of 2 self-loading scrapers, a grader, a dozer and a water truck.
2. At Area 7, deposition will be managed from the southern embankments and the tailings will be graded to drain supernatant water towards a decant system on the northern embankment and thence to Area 7 Phase 1 evaporation pond or the Area 7 Phase 2 collection sump for pumping to Area 1 evaporation pond.
3. At Area 1, deposition will be managed from the western embankment and tailings will be graded to drain supernatant water towards a decent system on the eastern embankment and thence to the Area 1 evaporation pond.
4. Water collected in the above-liner drainage system will also be managed in the respective evaporation pond.
5. From the Area 7 and Area 1 evaporation ponds water will either pumping directly back to the plant (direct reclaim water), or to a water treatment facility (treated reclaim water).

The closure of the TSFs includes the following anticipated actions:

1. Removal of residual supernatant water from the Area 1 evaporation pond via pumping to the water treatment plant, followed by treatment and discharge to the stormwater management system, and thence to creeks.
2. On-going drainage or pumping of water from Area 7 and Area 1 above-liner drainage systems to remove entrained water and increase consolidation of the tailings mass.
3. Removal of all tailings deposition conveyors and water reclaim piping and pumps.

4. Placement of an engineered cover system on the TSF surface consisting of:
    a. An average of 500 mm of compacted underliner using borrow from Area 1;
    b. An 80-mil HDPE geomembrane liner;
    c. An overliner drain system including drainage piping, drain rock, and geotextile; and
    d. An average of 1 m growth media.
5. Scarification and vegetation of the final cover surface.

18.3 Active Mine Dewatering and Pipeline System

Active dewatering from the surface will be utilized in advance of development of the mine and will continue to be utilized throughout the mine life. The active dewatering system will consist of a network of approximately 12 perimeter wells, equally spaced around the deposit. The wells will be drilled to a depth of 50 to 75 m below the next active mine block. The wells will be a minimum of 12 to 18 inches in diameter and will be capable of pumping approximately 65 L/sec (1,000 gpm) each. The first round of wells must be installed and operational prior to “year zero” of mine production; therefore installation of those wells will begin two years prior to underground mining. A second round of 12 deeper replacement wells will be drilled to the bottom of the next planned mine block beginning in year 8 of mining to allow dewatering of the second mine block.

Each well will be equipped with a nominal 9.5 inch diameter, 820 kW (1,100 HP), submersible pump. Water from each well will be piped to a collection pond constructed as Phase 1 of Area 1 Evaporation Pond (Figure 18.2.2). The water will then be pumped via a mine dewatering discharge pipeline to the Missouri River. The total maximum anticipated input to the collection pond will be 725 L/sec, anticipated to occur during the first year of mine production (Year 1).

The collection pond will cover approximately 9 ha with a crest elevation of 379 masl and a base elevation of 369 masl. The maximum embankment height will be approximately 13 m. Assuming 2 m of dry freeboard, the pond has a capacity of approximately 520,000 m^3. The pond will provide 7 days of emergency storage at the maximum dewatering rate plus the 100 year, 24 hour storm.

The mine dewatering discharge pipeline has been designed to a maximum flow of 2,270 m^3/h. The discharge pipe would be 50 km long and would require a 24 inch steel pipe that would discharge through a diffuser at the Missouri River. It has been assumed that the pipeline would be buried to protect against freezing. A pump station would be located at the pond. All pumps could be installed at the pond pump station or at a booster station installed along the pipeline.

19 Market Studies and Contracts

SRK has reviewed public market analysis and commissioned studies performed for this Project and confirms the price assumptions for niobium, titanium dioxide and scandium oxide are reasonable and appropriate for use in the PEA.

19.1 Commodity

19.1.1 Niobium

Niobium is used in alloys including high strength low alloy steel and improves strength at lower temperatures. Alloys containing niobium are used in oil and gas pipelines, beams and girders for buildings and jet engines. Niobium also exhibits superconducting properties and is used in superconducting magnets. The USGS publication *Mineral Commodity Summaries 2015* states that niobium is consumed mostly in the form of ferroniobium by the steel industry and as niobium alloys and metal by the aerospace industry. Apparent US domestic consumption measured in contained niobium is estimate at 10,000 t in 2014 which was a 23% increase from 2013.

19.1.2 Titanium Dioxide

USGS publication *Mineral Commodity Summaries 2015*, lists the highest consumption of titanium dioxide in paints (62%), plastic (24%), paper (11%) and other uses (3%) which included catalysts, ceramics, coated fabrics and textiles, floor coverings, printing ink and roofing materials. Domestic consumption was estimated to have increased by 5% in 2014 in part due to the increase use of titanium dioxide containing products in the housing industry.

19.1.3 Scandium Oxide

USGS publication *Mineral Commodity Summaries 2015* lists principal uses for scandium in 2014 as solid oxide fuel cells (SOFCs) and aluminum-scandium alloys. Other uses include ceramics, electronics, lasers and lighting. Global supply and consumption of scandium was estimated to be between 10 and 15 t/y. Two characteristics of scandium oxide, which have the potential to increase the demand within the scandium market, include the oxide's ability to act as a highly efficient ion channel at lower temperatures in solid oxide fuel cells and the ability of the metal to serve as a highly effective grain refiner when alloyed with many metals such as aluminum.

Current technology employed in the manufacturing of solid oxide fuel cells uses materials, which operate at temperatures up to or exceed 1,000^{o}C. Competing technologies now in use show that the use of scandium oxides in the fuel cell have higher conductivity at reduced operating temperatures, which in turn can extend the life of fuel cells.

Benefit of scandium-aluminum alloys used in the aerospace industry have been known since the 1970's. Russian military aircraft used scandium-aluminum alloys for strength and reduction of weight. Currently, Airbus Group has advertised the potential uses of scandium-magnesium-aluminum alloys in aerospace, transportation, defense and leisure products due to its high specific strength, functionality and high corrosion resistance. The alloys also exhibit excellent weldability and good joint strength.

19.2 Market Price Projection

The payable products recovered through processing at the Elk Creek project site will be in the final salable form which includes ferroniobium, titanium dioxide and scandium oxide. Metallurgical testing to date indicates that all three products can be produced within acceptable market specifications.

Pricing was established using current data found in the public domain, confidential studies and in-house knowledge. Commodity prices used in the calculation of financial results are US$39 to US$44/kg Nb for ferroniobium containing 65% niobium, US$2.10/kg TiO_2 for titanium dioxide, and US$3,000 to US$4,000/kg Sc_2O_3 for scandium oxide.

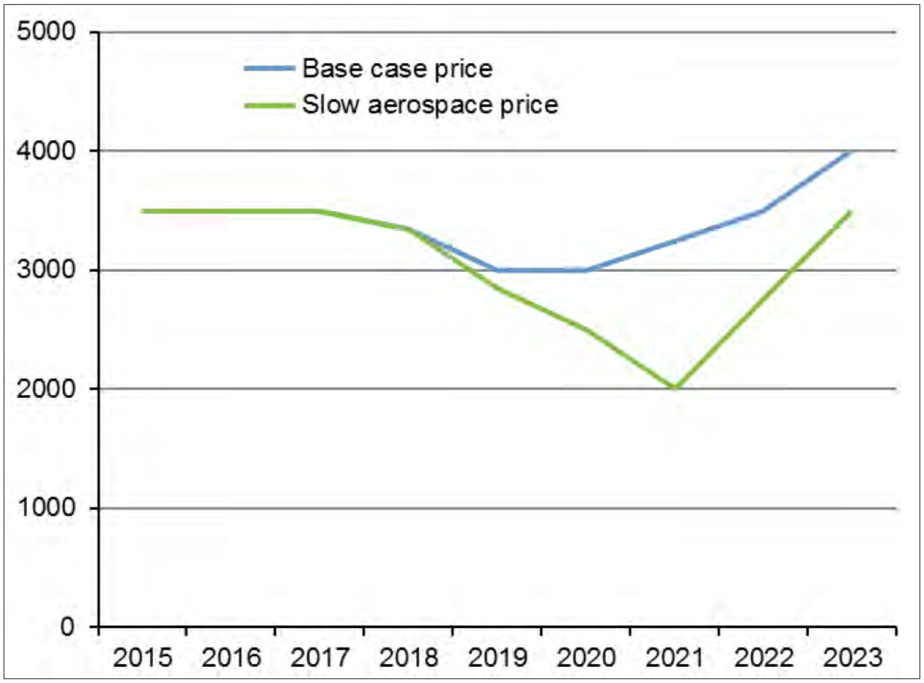
Pricing for ferroniobium was estimated using reputable market study data provided by Roskill which ranged from an average market price of US$39/kg Nb in 2015 to a 2020 forecast of US$44/kg Nb. The approach used in the PEA set the niobium price at US$39/kg for the first year of production, increased to US$44/kg in the fourth year and through LoM.

Pricing for titanium dioxide was estimated based on a lower quality product and in-house knowledge of the titanium market. Potential upgrade to the quality of the titanium dioxide produced to a pigment grade product could have a positive impact on market price. For the PEA, SRK used a conservative estimate of US$2.10/kg for the financial analysis.

Pricing for scandium oxide was estimated using a reputable market study entitled “Scandium: A Market Assessment” developed by OnG Commodities LLC during the preparation of the PEA,.and authored by Dr. Andrew Matheson. Dr. Matheson has extensive experience in specialty metals and consulted to global firms in the metals, materials and energy industries.

The base case pricing used for the economic analysis used a ranged price for scandium oxide of US$3,000/kg Sc_2O_3 to US$4,000/kg Sc_2O_3. It is estimated that the current price of US$3,500/kg Sc_2O_3 will decline slightly over the first three to four years of production to US$3,000/kg Sc_2O_3 as an increased reliable supply of scandium oxide becomes available. The market price projection is then estimated to increase to US$4,000/kg Sc_2O_3 and remain at that level for the LoM due to an anticipated increase in demand for the commodity in both the SOFC market and the aerospace industry. Understanding that these are forecasts in a currently undeveloped market, the sensitivity to the scandium oxide pricing was examined by a second price projection which reflected a slower ramp-up of consumption in the aerospace industry. This projection started with a price of US$3,500/kg Sc_2O_3 in the first year and ramping down to US$2,000/kg Sc_2O_3 by 2021 before ramping back to US$3,500/kg Sc_2O_3 in year 2023 and remaining at that level for the LoM.

Figure 19.2.1 shows the market pricing projection for both the base case and the slow aerospace growth scenarios. The chart reflects the initial drop in price as supply increases and the market becomes established, with a price recovery based on increased demand as industry consumption accelerates before it stabilizes.



Source: Ong Commodities LLC, 2015

Figure 19.2.1: Scandium Oxide Pricing Outlook, US$/kg

19.3 Contracts and Status

SRK confirms that NioCorp has entered into an offtake agreement with ThyssenKrupp Metallurgical Products GmbH whereby ThyssenKrupp Metallurgical Products will purchase approximately 3,750 t or roughly 50% of NioCorp's planned ferroniobium production from its Elk Creek deposit for an initial ten year term, with an option to extend beyond that time frame. The agreement presupposes the Company obtaining project financing, obtaining all necessary approvals and constructing a mine at Elk Creek.

For the purpose of the PEA, it is assumed that the remaining 50% of ferroniobium production during the first 10 years will be sold exWorks Elk Creek Plant and that all production sold after the first 10 years will be sold exWorks Elk Creek Plant.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Required Permits and Status

The proposed Project will be held to permitting requirements that are determined to be necessary by Johnson County, 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 likely permits and authorizations for the Project are presented in Table 20.1.1.

Table 20.1.1: Permits That May Be Required for the Project

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	**REQUIRED**. Mine Safety and Health Administration (MSHA) and 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	**REQUIRED**. NioCorp laboratory facilities 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.
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.
Clean Water Act Section 404 Permit	U.S. Army Corps of Engineers (USACE)	Permit for discharge of dredged or fill material into waters of the U.S. under Section 404 of the CWA	**REQUIRED**. Construction of mine, plant , pipeline and tailings disposal facilities will impact jurisdictional drainages and wetlands
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 applicable to the discharge of industrial wastewater and stormwater.	**REQUIRED.** Project likely to have excess water that will require some form of treatment and disposal.
Mineral Exploration Permit	State of Nebraska DEQ	Regulates the exploration for minerals by boring, drilling, driving, or digging.	**REQUIRED.** Already obtained for exploration drilling program.
Air Construction Permit	State of Nebraska DEQ	Regulates emissions during construction activities to protect ambient air quality.	**REQUIRED**. Under Nebraska Administrative Code (NAC) Title 129.

Permit/Approval	Issuing Authority	Permit Purpose	Status
Air Operating Permit	State of Nebraska DEQ	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 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 hydrogeological portion of 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 hydrogeological portion of exploration drilling program. May also be required for future disposal of water treatment sludge or RO brines.
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 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 boiler(s) is installed in any of the facility buildings.
Section 401 Water Quality Certification	State of Nebraska DEQ	Program evaluates applications for federal permits and licenses that involve a 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 17.	**REQUIRED**. Completed jointly with USACE during 404 permitting process.
Development Permit	State of Nebraska DEQ/Johnson County Floodplain Administrator	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.
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.	**MAY BE REQUIRED**. Will be needed if NioCorp intends to create an on-site solid waste management facility.
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. However, at this time, potable water for the Project is anticipated to come from Johnson or Pawnee county systems.

Permit/Approval	Issuing Authority	Permit Purpose	Status
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.	**MAY BE 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).	**LIKELY TO BE REQUIRED.** Will be required if the tailings facility (dam) a) has a total height of 25 ft or more and an impounding capacity at the top of dam that is greater than 15 acre-ft, or b) has an impounding capacity at the top of dam of 50 acre-ft or more and a total height that is greater than 6 ft, or c) is located in a high hazard potential location.
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.	**LIKLEY TO BE REQUIRED.** Will be required if NioCorp intends to construct any impoundment that impounds at least 15 AF below the spillway.
Local Permits for Johnson County			
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.
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.

Source: SRK

Project permitting commenced in January 2015 with the submission of a Jurisdictional Delineation report to the USACE. 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 community.

The following is a brief discussion of the more material permits which are likely to form the critical path in the permitting timeline.

20.1.1 USACE 404 Permit and NEPA

Clean Water Act Section 404 Permit

Perhaps the most critical of the required permits and/or authorizations for Elk Creek will be the approvals to construct the mine, plant and tailings disposal facilities, as they cover considerable area, and cross various water features that fall under the jurisdiction of the state and federal governments. The facilities for the operation have been designed to avoid jurisdictional features as much as possible. A wetlands delineation survey was conducted by Olsson Associates (October 2014) to define the nature and extent of potential jurisdictional features which could trigger the need for such permitting. Wetlands were identified in agricultural fields, pastures, roadside ditches and abutting stream channels. The total acreage of wetlands in the Project area is approximately 10.6 acres. Nine unnamed streams were also found during the site investigation for a total length of approximately 13,726 ft. At the time of this report, all wetlands and waters in the Project study area are assumed to be jurisdictional unless declared otherwise by the USACE. A site visit was conducted by the USACE on April 7, 2015 in order to inspect the site and assess the veracity of recommendations made by Olsson (2014) with respect to inclusion or exclusion of various features in the final determination. NioCorp and Olsson are currently working with the agency to make a final determination, which will be followed by the preparation and submission of a formal permit application.

Section 404 of the federal Clean Water Act (CWA) establishes a program to regulate the discharge of dredged or fill material into waters of the U.S. (WOUS), including wetlands and jurisdictional drainages/waterways. Activities in WOUS regulated under this program include fill for development, water resource projects (such as dams and levees), infrastructure development (such as highways and airports) and mining projects. Section 404 requires a permit before dredged or fill material may be discharged into WOUS.

Proposed activities are regulated through a permit review process. An individual permit is required for potentially significant impacts. Individual permits are reviewed by the USACE. However, for most discharges that will have only minimal adverse effects, a general permit may be suitable. General permits are issued on a nationwide, regional, or state basis for particular categories of activities. The general permit process eliminates individual review and allows certain activities to proceed with little or no delay, provided that the general or specific conditions for the general permit are met.

As part of the PEA, NioCorp has directed the engineering partners on the project to avoid both floodplains and any potentially jurisdictional water features during the design process. The mine and surface plant completely avoid these features, while the preferred tailings impoundment(s) impact just three small areas of potentially jurisdictional wetlands. This design approach is viewed very favorably by the USACE, as most agricultural entities tend to completely ignore USACE permit requirements.

National Environmental Policy Act Review

Within this permitting process, the need for environmental impact analysis is typically required. The National Environmental Policy Act (NEPA) requires federal agencies to consider the environmental effects of, and any alternatives to, their proposed actions. A USACE action that involves the placement of fill material into a WOUS and issuance of an Individual Permit (IP), the USACE must determine compliance with the CWA and NEPA prior to issuance of the permit.

The NEPA process generally involves one of two levels of analysis:

- Preparation of an Environmental Assessment (EA) and *Finding of No Significant Impact* (FONSI); or
- Preparation of an Environmental Impact Statement (EIS).

It is important to remember that both EAs and EISs are public disclosure documents, not permit or approval documents. They are intended to disclose what, if any, environmental impacts may occur from the Project and guide the decisions of federal agencies. The primary difference between the two types of documents is that an EA is prepared when no significant impacts are expected or the potential impacts are unknown, and an EIS acknowledges that there is a potential for significant impacts, and analyzes and discloses what those potential impacts are. Significance is determined based on a variety of factors such as compliance with state, federal, and local regulation regulations, USACE, EPA and USFWS policies and guidelines, and other site specific concerns such as removing water sites or critical habitat.

Compensatory Mitigation

Both NEPA processes are likely to result in the development of compensatory mitigation for the loss of wetlands and other jurisdictional features. Compensatory mitigation can occur through four methods: aquatic resource restoration, establishment, enhancement, or in certain circumstances, preservation. There are three mechanisms for achieving the four methods of compensatory mitigation (listed in order of preference as established by the regulations): mitigation banks, in-lieu fee programs, and permittee-responsible mitigation. The Omaha USACE is likely to require permittee-responsible mitigation at a ratio of 2:1. NioCorp will be required to perform the mitigation at the site of the permitted impacts, or at an off-site location within the same watershed.

USACE Permit Timing

NioCorp has initiated mitigation discussions with the USACE and commenced preparation of the formal permit application. The time to review and evaluate the actual 404 Permit application is typically overshadowed by the NEPA review of the Project impacts. The time to complete an EA (generally accepted at approximately 12 months) is usually less than an EIS (3 to 5 years), as there are no statutory time frames and fewer bureaucratic procedures involved. Both include public scoping and public review processes. NioCorp's current understanding is that the simpler EA is likely the route to be taken by the USACE with respect to Elk Creek given the limited impacts anticipated to wetlands and riparian resources. Inclusion of the dewatering water pipeline and discharge to the Missouri River is pending a delineation along the pipeline corridor before discussions with the USACE are initiated on this item. Several other dewatering water management options are also still under consideration and evaluation as part of the overall project feasibility study.

20.1.2 DHHS Radioactive Materials Program and Licensing

The Elk Creek 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. Preliminary discussions with the State of Nebraska have indicated that a Broad Scope Radioactive Materials License, issued under 180 NAC 3-013 by the Nebraska Department of Health and Human Services (DHHS), may likely be necessary.

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 tailings disposal facility be engineered and constructed. The final closure of the tailings will also require coordination with the Solid Waste Branch of the Nebraska Department of Environmental Quality (NDEQ).

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. As such, the State of Nebraska could apply them under either scenario, which could 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 could be applied to other situations where radioactive materials are being actively managed. In addition to a direct reclamation financial assurance, it is probable that the state will require a funding mechanism (e.g., trust fund, escrow, etc.) for monitoring and maintenance of the facility in the longer term as part of a Broad Scope License.

DHHS License Timing

NioCorp estimates that a Broad Scope License for Elk Creek will take 6 to 9 months to obtain, and will involve several months of discussions and negotiations related to engineering, design, monitoring, and terms and conditions.

20.1.3 Nebraska NPDES Permitting Program

The current Project water balance suggests that excess water from underground dewatering operations will need to be managed for discharge to the environment. The current preferred alternative will be to discharge dewatered groundwater via pipeline and diffuser directly to the Missouri River, approximately 56 km to the east of the project site. The Missouri River is believed to be the only regional water body of sufficient quantity (flow) and quality to accept the projected volume of water from the mine with minimal treatment, and still meet applicable discharge standards. Various rights-of-way and easements will also need to be negotiated with Johnson and Nemaha counties and private landowners. In addition, the location and design of the in-stream diffuser and impacts to any jurisdictional areas along the proposed pipeline corridor, will need to be addressed through the USACE.

In the State of Nebraska, all persons discharging or proposing to discharge pollutants from a point source into any waters of the state are required to apply for, and have a permit under the National Pollutant Discharge Elimination System (NPDES) to discharge, including all significant industrial users discharging to a publicly owned treatment works (POTW). The NDEQ is responsible for developing and issuing NPDES permits, and for insuring that permitted facilities comply with permit requirements.

Alternatively, NioCorp may still elect to treat the mine dewatering water to meet local discharge requirements and re-inject the treatment plant reject water back underground through an Underground Injection Control permit, administered through the Water Division of the NDEP. Additional studies for the viability of this option are pending. Other alternatives for water management at the site, which may or may not prove feasible in the long run, include:

- Direct discharge to surface waters;
- Direct deep well injection;
- RO treatment with surface discharge of permeate and crystallization of reject water;
- RO treatment with surface discharge of permeate and force freezing of the reject water;
- RO treatment with surface discharge of permeate and on-site storage of reject water; and
- RO treatment with surface discharge of permeate and deep well injection of reject water.

Most of these alternatives are not likely to be cost effective, but are still under consideration.

NPDES Permit Timing

Regulations require an individual, site-specific NPDES permit application be submitted to the NDEQ at least 180 days (six months) prior to the date of first discharge. This is predicated on an administratively and technically complete and accurate permit application. If changes are made or additional information is submitted or is required by the agency, the 180 day period may start over. NPDES permits are public noticed for 30 days before being issued. If comments are received and a hearing is required, the NDEQ would schedule a hearing and respond to any comments received at the hearing. This may require an additional 60 to 90 days.

20.1.4 Nebraska Air Quality Permitting

The Nebraska air regulations are primarily based on regulations developed by the U.S. Environmental Protection Agency (EPA) to address the Clean Air Act (CAA) requirements. Air quality permits are the primary tool used by the NDEQ to implement the CAA. 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 (NSR) 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 could be released from the new plant or expansion project.

Because the project includes a primary sulfuric 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 modeling). 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 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.

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 sulfuric 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 (PTE) 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.

20.1.5 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 25 ft or more and an impounding capacity at the top of dam that is greater than 15 acre-ft;
- Have an impounding capacity at the top of dam of 50 acre-ft or more and a total height that is greater than 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 both the proposed tailings disposal facility and treatment plant reject water storage pond.

20.1.6 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 Elk Creek 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.

20.2 Engineering Design Criteria

The State of Nebraska does not have regulatory environmental protection requirements for the design and operation of hardrock mines, especially underground hardrock mines with chemical beneficiation circuits. As such, NioCorp has engaged in a conservative approach to minimize environmental risk and liability by adopting relevant Environmental Design Criteria (EDC). Without state or federal guidance in this matter, the EDCs for Elk Creek were fashioned after those from a jurisdiction dedicated to sustainable hardrock mining; the State of Nevada and the U.S. Bureau of Land Management. However, Nebraska does have regulations pertaining to the management of solid wastes, including mining wastes.

While there are no specific regulations governing the construction and operation of hardrock tailings impoundments in the State of Nebraska, the definition of Solid Waste in Chapter 1 of Title 132 – Integrated Solid Waste Management Regulations includes material generated from mining operations and therefore the tailings facility at the Project may be subject to all or part of the Title 132 regulations. A final determination on the applicability of these regulations will be made once more information has been developed on the final tailings waste stream and chemical constituents. In the meantime, the PEA was intended to be developed under the assumption that these regulations will be applicable, and were addressed in the overall preliminary design of the facility.

Detailed design and development of the process facilities is ongoing, and, as such, environmental control equipment selection is somewhat premature. However, NioCorp remains committed to the integration and implementation of the most appropriate control technologies for both air emissions and water discharges in order to meet the pollution prevention requirements set by the regulatory agencies.

20.3 Environmental Study Results

20.3.1 Soils

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 slope from 2% to 30%, with the majority of the area having slopes between 6% to 11% (NRCS, 2015).

20.3.2 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; and
- Solar Radiation.

The data thus far collected are concluded to be a satisfactory start to the goal of a continuous one year record that can subsequently be used in PSD modeling. A continuation of data review will be conducted on a monthly basis as specified in Meteorological Monitoring Plan, and a final review will be necessary prior to submittal for inclusion in dispersion modeling.

20.3.3 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;
- 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 land in Nebraska to the federal government and moved to Indian Territory (now Oklahoma). An assessment of potential cultural and archeological resources in the area will need to be performed as part of the USACE NEPA analysis of the Project. However, no direct tribal consultation appears to be necessary at this time.

Since the passage of the National Historic Preservation Act of 1966, the governor of each state has been mandated to appoint a State Historic Preservation Officer to oversee preservation efforts. In Nebraska, the director of the Nebraska State Historical Society serves as State Historic Preservation Officer. The State Historic Preservation Office (SHPO) administers, among others, the National Register of Historic Places (NRHP), the Nebraska Historic Resources Survey & Inventory (NeHRSI), as well as the state Archeological Survey.

NioCorp will work with SHPO during the development of the Project to identify potentially significant cultural resources within the Project boundary and ensure that any such resources are properly managed ort mitigated.

20.3.4 Vegetation

Cultivated crop land (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*).

20.3.5 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 consist 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).

20.3.6 Threatened, Endangered, and Special Status Species

The Project and surrounding areas lie in the Southeast Prairies Biologically Unique Landscape (BUL) 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 BUL. 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 idalial*);
  - 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 BUL in Nebraska, while the Massasauga and Plain Pocketbook are known to occur in only two other BULs 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 would likely be accomplished during the USACE NEPA assessment.

20.3.7 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 10,000 acres in Johnson County is irrigated cropland, while about 42,000 acres is used for pasture. About 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).

20.3.8 Hydrogeology (Groundwater)

A hydrogeologic 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 inch PVC standpipe piezometers isolated in the carbonatite and open to large intervals of the deposit;
- Installation of two, nominal 2 inch PVC standpipe piezometers isolated in the 180 m thick Pennsylvanian aquitard above the carbonatite; and
- Frequent measurement of water levels in open core holes and piezometers over a period of six months.

The hydrogeologic 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 in the aquifer was observed in nearby piezometers. These data were used to establish the prospective mine water inflow that appears in the PEA. However, the hydrogeologic 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 piezometers were established. Water was injected at a rate of 22 to 30 L/sec (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 has been completed and a preliminary conceptual model developed. Details of this model are provided in Section 16.3 of this PEA.

The samples collected from NEC 14-014 and Met-1 indicate very similar water quality. Both wells have TDS over 18,000 parts per million, with the major contributors being sodium and chloride. Both wells exceed EPA primary MCLs with respect to the following:

- Arsenic;
- Gross alpha; and
- Ra-226 + Ra-228.

Water from both of these wells also exceeds secondary Maximum Contaminant Levels (MCLs) with respect to chloride, fluoride, iron, manganese, sulfate, and TDS. NEC 14-014 also exceeds the secondary MCL for aluminum. There were no detectable pesticides/herbicides in 4th quarter 2014 groundwater samples.

Although the 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.

The 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; and

- 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 overall pumping requirements for the underground mining operation (from surface wells and the underground mine).

20.3.9 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 program are as follows:

- Surface water quality can be classified as slightly impaired. Aluminum, iron, manganese, and TDS show recurring concentrations over the EPA secondary standards.
- Aluminum, iron, and manganese also exceed State aquatic life criteria at various locations on a recurring basis.
- Average stream TDS in the 2nd and 3rd quarters of 2014 was 560 and 430 mg/L respectively, jumping to 708 mg/L in the 4th quarter. This could be the result of post-harvest runoff containing more sediments.
- A single elevated concentration of arsenic (0.013 mg/L) in Elk Creek upstream sample ECKN-U was reported in the 2nd quarter of 2014. This has been the only exceedance of a primary EPA MCL.
- Stream pH is consistently 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 MCLs.

20.3.10 Wetlands/Riparian Zones

As noted above, Olsson Associates (2014) was retained 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 could 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. Olsson identified a total of 45 wetlands encompassing a total area of approximately 10.64 acres. Nine unnamed streams were also found during the field investigation for a total length of approximately 13,726 ft. Portions of these streams are likely to be classified as jurisdictional.

At the time of this report, all wetlands and waters in the Project study area are assumed to be jurisdictional. Olsson and NioCorp are currently working with the USACE in order to obtain a final determination.

20.3.11 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 potential to leach various constituents due to exposure to meteoric precipitation. Laboratory leach tests of a composite sample of this material from drillhole NEC11-01 indicate that, at a minimum, fluoride and nitrate could be mobilized during surface stockpiling.

Contained within the mineralized material are naturally occurring uranium and thorium. Leach testing has not produce concentrations of radionuclides above regulatory limits. However, the concentrations in the rock are relatively elevated:

- Gross alpha = 200 pCi/g;
- Gross beta = 160 pCi/g;
- Radium 226 = 56 pCi/g; and
- Radium 228 = 18 pCi/g.

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 niobium deposit. These include:

- **Pennsylvanian limestones and mudstones** – The upper 30 m of lithology consists of unconsolidated glacial till, underlain by a 170 to 180 m 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 strongly 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.
- **Non-ore grade carbonatite** – Preliminary assessment of the host rock identified visual sulfide content of up to one percent based on observations by core loggers. Laboratory analyses confirmed the sulfide content at around 1.34%. This sulfide consists mainly of pyrite, chalcopyrite, bornite, galena, sphalerite, and possibly pyrrhotite. However, even with

detectable sulfide content, the carbonatite waste rock is still net neutralizing given the high carbonate content.

Of the 94 rock samples collected over a 255 m 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 could be used as a diagnostic tool to identify elevated concentrations of uranium and thorium.

Additional drilling and testing has been proposed to better define the geochemical characteristics of the waste rock types before a proper management plan can be developed. Any surface disposal of waste rock will be predicated on minimizing meteoric infiltration and leaching of this material. Given the uncertainty surrounding the geochemistry of the Elk Creek Project waste rock and overburden (until additional testing is completed), 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 they can be processed, returned to the underground as backfill or properly closed in place.

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 is scheduled for late 2015 when the testing of the beneficiation process is complete and the need for and usability of tailings as underground backfill has been properly evaluated.

20.4 Health and Safety

Occupational health and safety 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 workers in accordance with 30 CFR § 57.5039 thru § 57.5047.

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.

20.5.1 Surface Disturbance

The principal objective of the surface reclamation plan will be to return disturbed lands to a 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 injection 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, tailings disposal facility at the Project will likely be subject to all or part of the Title 132 regulations, including the closure requirements. These requirements include:

- A final cover system shall be installed which shall be comprised of an erosion layer underlain by an infiltration layer as follows:
  - The infiltration layer shall be comprised of a minimum of 18 inches of earthen material that has a permeability less than or equal to the permeability of the bottom liner system or natural subsoil present, or a permeability no greater than 1×10^{-5} cm/sec, measured at the site, whichever is less; and
  - The erosion layer shall consist of a minimum of 18 inches of earthen material that is capable of sustaining adequate vegetative cover.
- Owners or operators of solid waste disposal areas shall prepare and submit a written closure plan that describes the steps necessary to close the solid waste disposal area in phases, or the entire area, whichever is applicable. This closure plan shall be part of the permit application. The closure plans shall include, but not be limited to, a description of the methods of closure which comply with the requirements of the law.

With respect to post-closure requirements, operators of solid waste disposal areas shall provide for post-closure care for a period of 30 years. Additional details regarding the closure of the tailings disposal facility are provided in Section 18.2 of this report.

20.5.4 Financial Surety Requirements

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 would be under the scenario in which the facility falls under a broad scope radiological license, which may have financial assurance requirements for reclamation and closure. 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 could be applied to other situations where radioactive materials are being managed. These surety requirements extend to long-term site monitoring, maintenance, and care.

In addition, financial assurances will also be required for the tailings disposal facility, which will fall under the NDEQ Title 132 - Integrated Solid Waste Management Regulations. Allowable mechanisms for financial assurance include trust funds, surety bonds, and letters of credit.

At this time, the type and amount of financial surety for Elk Creek has not yet been established.

20.5.5 Closure Cost Estimate

Closure costs for Elk Creek have been estimated at just over US$60 million, the bulk of which (US$40 million) is intended for reclamation and closure of the tailings disposal facility. Approximately US$15 million has been allocated for surface reclamation of the remaining facilities (i.e., building demolition, site regrading and revegetation, shaft closure, etc.), while the remaining US$5 million is set aside for post-closure monitoring and maintenance. These costs will be refined as part of the feasibility study, and may need to be adjusted based on specific regulatory agency requirements, particularly those associated with any radioactive material licensing of the plant and tailings facility.

20.6 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 County representatives (including the county commissioners) as well as the Southeast Nebraska Development District (SENDD). Given the accelerated 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, could, therefore, 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.

20.7 International Standards and Guidelines

Even though the United States is a Designated Country with respect to the Equator Principles, NioCorp has committed to ensuring that Elk Creek is in compliance with international standards and guidelines, to the extent practicable, given the potential for international investment in the Project. 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.

Potentially relevant international policies and/or guidelines for which the Project is likely to maintain compliance with include, but are not necessarily limited to:

- Equator Principles risk management framework for determining, assessing and managing environmental and social risk in projects;
- International Finance Corporation (Performance Standards) (IFC – PS) – social and environmental management planning;
- World Bank Guidelines (Operational Policies and Environmental Guidelines);
- Washington Convention of 1940 on Nature Protection and Wild Life Preservation in the Western Hemisphere;
- Vienna Convention for the Protection of the Ozone Layer;
- Montreal Protocol on Substances that Deplete the Ozone Layer;
- Basel Convention on the Control of Trans-boundary Movements of Hazardous Wastes and

their Disposal; and

- United Nations Climate Convention and the Kyoto Protocol.

Table 20.7.1 provides a brief assessment of the approach to compliance anticipated for Elk Creek with respect to the IFC Performance Standards, even though the U.S. is a Designated Country.

Table 20.7.1: IFC Performance Standard vs. Compliance Approach

IFC Performance Standard (PS)	Summary of Requirements	Project Compliance
PS1: Assessment and Management of Environmental and Social Risks and Impacts	Development of an ESMS appropriate to the nature and scale of the Project which includes a policy, identification of risks and impacts, management programs, organizational capacity and competency, emergency preparedness and response, stakeholder engagement, monitoring and review.	Project will be subject to environmental impact assessment and environmental management requirements at various stages of the state and federal permitting processes.
PS2: Labor and Working Conditions	Identification of risks, impacts and management requirements associated with working conditions and terms of employment, non-discrimination and equal opportunity, retrenchment, grievance procedures, child labor, forced labor, occupational health and safety, third party workers and the supply chain.	Project will be governed by Nebraska Department of Labor statutes and regulations, as well as state and federal OSHA and MSHA requirements
PS3: Resource Efficiency and Pollution Prevention	Promotes technically and financially feasible options to address resource efficiency (including greenhouse gas production and water consumption) and pollution prevention (with respect to wastes, hazardous materials management and pesticide use) across the Project life-cycle.	NPDES permitting, CWA 404 approvals, radioactive materials licensure, and air operating permits all ensure compliance with applicable environmental laws
PS4: Community Health, Safety and Security	Evaluation of risks and impacts to the health and safety of Project-affected communities over the Project life cycle. Issues to be considered include infrastructure and equipment design and safety, hazardous materials management, ecosystem services, community exposure to disease, emergency preparedness and response, and management of security personnel.	Nebraska DHHS will require extensive monitoring as part of any licensing of the facility; TSF safety regulated by DNR; environment incl. hazardous materials management will be overseen by NDEQ
PS5: Land Acquisition and Involuntary Resettlement	Applies to physical and or economic displacement resulting from Project acquisition of land rights or land use rights through expropriation, compulsory procedures, or negotiated settlements that if fail result in compulsory procedures. This PS also applies to Project situations requiring eviction of people occupying land without formal, traditional or recognizable usage rights and situations involving involuntary restrictions on land use or use of natural resources.	There will be no involuntary resettlement associated with the Project. Land acquisition being negotiated with individual land owners and tenants
PS6: Biodiversity Conservation and Sustainable Management of Living Natural Resources	Identification of risks and impacts on biodiversity and ecosystem services, especially focusing on habitat loss, degradation and fragmentation, invasive alien species, overexploitation, hydrological changes, nutrient loading and pollution. Guidance measures are dependent on type of habitat present (i.e. modified, natural or critical). Where a project is likely to adversely impact ecosystem service, a systematic review to identify priority ecosystem services is required.	Wetland and riparian resources impacted by the Project will be mitigated at ratio of 2:1 per USACE requirements; Region already fragments due to extensive agriculture; Aquatic habitats will be protected through NPDES discharge program
PS7: Indigenous Peoples	Avoidance of adverse impacts on indigenous peoples and active engagement with the affected communities. Free, prior and informed consent (FPIC) of affected communities of indigenous peoples is required for projects with potential impacts to lands and natural resources subject to traditional ownership or customary use, relocation of indigenous peoples from such lands, and impacts to critical cultural heritage.	There are no classifiable indigenous peoples in the region of the Project. The closest tribal reservations are over 200 km north of the site.
PS8: Cultural Heritage	Promotes protection of cultural heritage in Project design and execution including implementation of chance find procedures, consultation, and community access and mitigation hierarchy. Critical cultural heritage should not be removed, significantly altered or damaged.	NioCorp will work with the Nebraska State Historic Preservation Office (SHPO) to ensure that no cultural heritage is impacted by the Project

21 Capital and Operating Costs

21.1 Capital Cost Estimates

Table 21.1.1 contains a summary of capital costs for the underground development and operations of the Project. Capital costs contain the design, procurement and construction of the underground mine and surface mine infrastructure, processing plants and auxiliary facilities, and infrastructure. At this level of study, and with the work performed to-date, the capital cost estimate is at an accuracy of ±25%.

Table 21.1.1: Capital Cost Summary (US$000's)

Description	Initial	Sustaining	LoM
Mining	$177,269	$108,028	$285,298
Process	$391,220	$0	$391,220
Tailings and Infrastructure	$187,948	$228,658	$416,606
Owners Costs/Land Acquisition	$56,593	$0	$56,593
Closure Costs	$0	$71,309	$71,309
Contingency	$165,711	$0	$165,711
Total Capital	**$978,742**	**$407,995**	**$1,386,738**

Source: SRK, 2015
Sustaining capital for the processing plants has been included in operating costs.

Mining capital cost contains an estimate for the sinking of the shaft, underground development, underground mining equipment and infrastructure, underground pumping stations, backfill plant and distribution system and ventilation excavations and mechanicals. Initial capital requirements in the pre-production years is US$177.3 million and sustaining capital requirements total an additional US$108.0 million throughout the LoM (Table 21.1.2).

Table 21.1.2: Life of Mine Capital Cost for Mining ($000s)

Description	Initial	Sustaining	LoM
Shaft and Structure	$92,832	$0	$92,832
Initial Mine Capital	$12,660	$1,598	$14,258
Mine Equipment	$15,871	$60,161	$76,032
Ramp meters	$9,858	$19,455	$29,314
Short raise meters	$167	$1,730	$1,897
Vent raise (Long) meters	$7,174	$0	$7,174
Vent Connection meters	$8,480	$4,700	$13,180
Level Development	$3,735	$19,512	$23,247
Material Movement on Surface	$930	$0	$930
Underground Pumping	$4,433	$871	$5,304
UG Ventilation	$4,990	$0	$4,990
Backfill Plant	$16,140	$0	$16,140
Total Capital	**$177,269**	**$108,028**	**$285,298**

Source: SRK, 2015

Process and infrastructure costs contain an estimate for initial capital of US$579.2 million and LoM sustaining capital of US$228.7 million for an expansion of the TSF and replacement of surface dewatering wells (Table 21.1.3). Sustaining capital for the processing plants has been included in the estimated operating cost for the Project.

Table: 21.1.3: Life of Mine Process and Infrastructure Capital Costs

Description	Initial	Sustaining	LoM ($000s)
Early Design and Procurement	$1,500	$0	$1,500
Infrastructure	$62,335	$0	$62,335
Mineral Process Plant	$16,496	$0	$16,496
Hydrometallurgical Plant	$238,832	$0	$238,832
Acid Plant (Gas Cleaning + Contact Section)	$103,156	$0	$103,156
Pyrometallurgy	$32,736	$0	$32,736
Product Packaging	$2,122	$0	$2,122
Tailings and Water Management	$6,117	$0	$6,117
Tailings Storage Facility	$42,891	$159,358	$202,249
Mine Water Discharge Pipeline to river	$39,500	$0	$39,500
Surface Dewatering Wells and Pumps	$33,484	$69,300	$102,784
Other	$0	$0	$0
Total Capital	**$579,169**	**$228,658**	**$807,827**

Source: SRK, 2015

Capital costs above exclude contingency but include indirect costs which were estimated at 21% of direct costs.

Owner's costs include owners incurred costs prior to start of production which was estimated to be 3% of direct initial capital. Also included in Owner's cost is land acquisition, environmental closure of the mine, plant site and TSF, and post closure monitoring. Overall Project contingency is 20.4% of the total initial capital estimate. Table 21.1.4 contains the LoM summary of Owner's costs.

Table 21.1.4: Owner's Capital Cost and Estimated Capital Contingency

Description	Initial	Sustaining	LoM ($000s)
Owners Costs (3% of Initial Capex)	$22,693	$0	$22,693
Land Acquisition	$33,900	$0	$33,900
Exploration	$0	$0	$0
Environmental & Closure	$0	$15,000	$15,000
Tailings Closure	$0	$51,309	$51,309
Post Closure Monitoring	$0	$5,000	$5,000
Total Capital	**$56,593**	**$71,309**	**$127,902**

Source: SRK, 2015

21.1.1 Basis for Capital Cost Estimates

Underground mine development costs were estimated on a per meter basis of development. Table 21.1.1.1 contains the US$/m used for the various types of development. However, material movement on surface is based on US$/t. These costs were developed from SRK database information.

Table 21.1.1.1: LoM Underground Mine Development Costs

Description	Unit Rate (US$/m or t)	Quantity (m)	LoM (US$000's)
Ramp meters	4,500	6,514	29,314
Short raise meters	1,600	1,186	1,897
Vent raise (Long) meters	17,672	406	7,174
Vent Connection meters	3,200	4,119	13,180
Level Development	3,500	6,642	23,247
Material Movement on Surface	2.00	465	930
Total Capital			**$75,742**

Source: SRK, 2015

The equipment costs were from Sandvik 2014 cost information and recent quotations for similar equipment, as well as an SRK database and CostMine Mining Cost Service data.

Process equipment were sized and quantified based on this PEA process design criteria and mass and water balances. Process equipment costs were estimated with either historical data, budgetary quotations in some cases, or equipment cost databases. Minor equipment costs such as small bins or pumps were estimated based on historical data or with an allowance.

The process facilities equipment installation, freight, process piping and process electrical and instrumentation costs were estimated with factors of mechanical equipment cost. Factors did vary based on the application. For example, the pyrometallurgical plant process does not require significant process piping, therefore the cost for piping was adjusted accordingly.

The buildings' shells including earthworks, concrete, structural, architectural building related HVAC, electrical & instrumentation and piping, were estimated based on dimensions from conceptual drawings. Based on equipment sizing and quantities, a footprint with building height was established and used as a basis for the estimation.

As for the other infrastructure costs, most of the estimate was performed using quantities and unit costs for roads, site pad preparation, parking, etc. Auxiliary buildings were estimated based on conceptual drawings produced. Rail was estimated based on typical cost per kilometer of railway to build, cost estimate for connection with BNSF, a bridge, etc. Rail unloading was estimated based on the required equipment for unloading, loading and handling of products coming into and from the mine site. Finally, some smaller cost items such as potable water, sewage treatment system, and fuel station were estimated based on historical data from similar size mining projects.

21.2 Operating Cost Estimates

The operating costs are based on processing 2,700 t of mineralized material per day to produce an average of 7,500 t/y of ferroniobium (rounded). The operating costs are based on Q1-2015 costs, and the estimate has been broken down into three main areas: mining costs (mine), processing costs (process), and general & administration (G&A).

The mine operating cost is estimated at US$53.00/t of the mineralized material delivered to the processing operation and includes the manpower, energy, spares and maintenance supplies required for the underground development and production of the mineralized material, as well as underground paste backfill distribution, underground pumping systems, and ventilation.

The process operating cost is estimated at US$135.75/t of the mineralized material milled and consists of the manpower, energy, consumables, reagents, spares and maintenance supplies required for the operation of the mineral processing, hydrometallurgical, acid and pyrometallurgical plants, as well as the operating costs of the fresh water supply and treatment, surface dewatering wells, and tailings disposal.

The general & administration operating cost is estimated at US$8.11/t of the mineralized material milled. This includes all of the project's operating costs which are not related to the mining and processing plants. The G&A costs include the following subsections: administration manpower, and general costs for operations.

The overall LoM operating cost for the Project is estimated at US$6.1 billion, US$196.86/t mineralized material milled or US$39.28/kg of Nb (excluding Ti and Sc credits). A summary of the operating costs for the Project is shown in Table 21.2.1. All costs presented in this section are in US dollars per mineralized material milled, or kg of Nb, or US dollars. The details of the operating costs for each area are presented in Tables 21.2.1 to 21.2.4.

Table 21.2.1: Operating Cost Summary

Description	US$/t-Processed	US$/kg-Nb	LoM (US$000's)
Mine	$53.00	$10.58	$1,647,647
Process	$135.75	$27.09	$4,219,864
G&A	$8.11	$1.62	$252,000
Total	**$196.86**	**$39.28**	**$6,119,511**

Source: SRK, 2015

Table 21.2.2: Mine Operating Cost Summary (average over LoM)

Description	US$/y, US$/m or US$/t Mined	US$/t-Processed	LoM (US$000's)
Drift Development (4.5m x 4.5m)	3,200/m	4.50	139,890
Drift Development (5m x 5m)	3,500/m	0.47	14,643
Mineralized Material Production (MMP)	33.61/t-MMP	30.29	941,437
Material Handling at Surface	2.00/t-RoM	0.01	440
Underground pumping system	1,075/y	1.11	34,400
Paste Backfill Plant & UG Distribution	15.00/t-RoM	15.00	466,283
Ventilation	1,580/y	1.63	50,553
Total		**$53.00**	**$1,647,647**

Source: SRK, 2015

Table 21.2.3: Process and Infrastructure Operating Cost Summary

Description	US$/yr or US$/t Processed	US$/t Process	LoM (US$000s)
Mineral Processing			
Manpower	697	0.72	22,311
Energy (Mineralized Material Processing only)	0.97	0.97	30,153
Reagents	0.06	0.06	1,865
Consumables	1.92	1.92	59,684
Other Processing	0.09	0.09	2,798
Hydromet Plant			
Manpower	5,306	5.46	169,804
Energy (Hydrometallurgy only)	17.44	17.44	542,132
Reagents	25.72	25.72	799,520
Consumables	5.06	5.06	157,293
Other Processing and Product Packaging	0.51	0.51	15,854
Acid Plant			
Manpower	1,572	1.62	50,300
Energy (Acid Plant only)	4.97	4.97	154,495
Reagents	22.95	22.95	713,413
Consumables	7.60	7.60	236,250
Contingency (10%)	3.71	3.71	115,327
Pyrometallurgy Plant			
Manpower	1,524	1.57	48,775
Energy (Pyrometallurgy only)	1.18	1.18	36,681
Reagents	14.66	14.66	455,714
Consumables	0.63	0.63	19,584
Other Processing	0.09	0.09	2,798
Rail-Track, Loading & Unloading	**2.21**	**2.21**	**68,699**
Product Packaging	**1.30**	**1.30**	**40,411**
Water Management & Surface Operations	**2.54**	**2.54**	**78,957**
Tailings Opex (SRK increases in out years)	**2,796**	**2.93**	**91,040**
Mine Water discharge to river	**2,500**	**2.57**	**80,000**
Surface Dewatering Wells	**7,063**	**7.27**	**226,006**
Total		**$135.75**	**$4,219,864**

Source: Roche, 2015
Note: Based on total tonnes processed of 31,086 kt.

Table 21.2.4: G&A Operating Cost Summary

Description	US$/yr	US$/t Process	LoM (US$000s)
All-In G&A Costs	8,000	8.11	252,000
other	-		-
Total		**$8.11**	**$252,000**

Source: SRK, 2015

21.2.1 Basis for Operating Cost Estimates

The mining operating cost was developed as follows:

- The development costs are based on unit costs US$/m for specific mine development activities and are converted to a per tonne basis using the planned production rate of the mine.
- The stoping costs are developed from CostMine Mining Cost Service data with adjustments for haulage distance and additional equipment costs. The operating cost varies over time based on haul distance. Diesel fuel costs assumed in this study are US$0.806/L.

- Underground water pumping costs were based on maintenance parts, labor, and electricity costs for system.
- The underground backfill cost information was provided by Kovit based on their internal database information.
- The ventilation costs are based on estimated electricity use per year that is adjusted for mine depth and additional equipment as the mine deepens. The estimate is based on an electricity price of US$0.0575/kWh.

The G&A cost was estimated at approximately 6.8% of direct cost in a typical year and held constant for the LoM.

The process operating costs were determined by estimating the required quantities of consumables, reagents, manpower, natural gas and electrical power on a “dollars per tonne milled” basis.

The main consumables for the mineral processing plant are grinding mill liners and grinding media. The consumption rates were calculated based on the bond abrasion test results as well as consultation with mill suppliers for the liner consumption. The prices for the consumables were from recent quotes from suppliers for similar projects.

The reagent consumption rates are based on the laboratory test results used for the PEA with prices taken from suppliers’ quotes. The reagent consumption for acid plant, hydrometallurgical plant and pyrometallurgical plant accounts for around 50% of the total Process and Infrastructure operating cost.

The connected loads for the process equipment were taken from the electrical loads that were established based on the mechanical equipment lists for each plant. The cost of electricity was assumed to be US$0.0575/kWh.

The natural gas consumption rates are based on process equipment, with the hydromet kilns consuming the bulk of the natural gas. Cost for natural gas is $2.789/mmBtu taken from the current natural gas price for Nebraska as per U.S. Energy Information Administration.

Labor requirements were determined and estimated for various unit operations in the processing plants. The labor cost was then calculated using manpower rates taken from recent plant operation wages from Nebraska.

A factor of 4% of the mechanical equipment capital cost was considered for the spares and maintenance supplies. A factor of 2% of piping material capital cost and 8% of instruments capital cost were used for piping and instrumentation supplies costs.

Other processing costs for contractors for shut downs or other purpose were estimated for the different process facilities based on past experiences.

The rail operating cost, was based on Roche’s experience with similar rail maintenance cost estimates for both the Niocorp rail crew and outside contractors.

The paste backfill plant operation cost estimate is based on a rate 125 t/h, with US$0.060 kWh for electricity, US$15/t for flyash, US$150/t for cement as a binder, and US$40/man-hour for manpower costs.

22 Economic Analysis

22.1 Principal Assumptions and Input Parameters

The economic results summarized in this section are based upon work performed by SRK and Roche. The base case technical economic model developed for the Project is on an after-tax basis and assumes 100% equity to provide a clear picture of the technical economic merits of the operation.

Table 22.1.1 outlines the model parameters used in the economic analysis and includes the base case commodity pricing.

Table 22.1.1: Model Parameters

Description	Value	Units
Mine Life	32	years
Mineralized Material Processed	31,086	kt
Payable FeNb	239.7	kt
Payable TiO_2	766.7	kt
Payable Sc_2O_3	3.1	kt
FeNb Price (LoM avg)	$43.55	US$/kg
TiO_2 Price (LoM avg)	$2.10	US$/kg
Sc_2O_3 Price (LoM avg)	$3,883	US$/kg
Effective Tax Rate	23.9%	
Discount Rate	8%	

Source: SRK, 2015

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

22.2 Cashflow Forecasts and Annual Production Forecasts

The mine production summary includes 1,135 kt of waste material from the sinking of the shaft, access development of the ramp and drifts, and ventilation raises and connection drifts required for proper operation of the underground mine.

Mineralized material identified in the 32 year mine plant includes 31,086 kt of which 10%, or 3,073 kt, is from development and 90%, or 28,013 kt, is from production stopes (Table 22.2.1). The mine production rate of 2,700 t/d will carry the mine for an estimated 31 to 32 years. The first production of mineralized material associated with mine development will occur in late 2017. Full mine production is anticipated to start early in 2018 and the first full production year would by 2019.

Table 22.2.1: Mine Production Summary

Description	Value	Units
Mine Production		
Waste	1,135	kt
Mineralized Material	31,086	kt
Total Material	**32,220**	**kt**
Daily Mineralized Material Capacity	2,700	t/d
RoM Grade		
Nb_2O_5	0.80	%
Contained Metal		
Nb_2O_5	249.7	kt

Source: SRK, 2015

Process production is planned at 2,700 t/d matching the projected mine production rate, which will minimize the need for stockpiling production material. Estimated recoveries and product quantity of niobium, titanium and scandium are listed in Table 22.2.2.

Table 22.2.2: Process Production Summary

Description	Value	Units
RoM Mineralized Material Processed (incl. Stockpile)	31,086	kt
Daily Capacity	2,700	t/d
Metallurgical Recovery		
Nb_2O_5	89.2%	
TiO_2	87.6%	
Sc	90.0%	
Saleable Product		
Niobium	155.8	kt
Ferroniobium	239.7	kt
Titanium Oxide	766.7	kt
Scandium Oxide	3.1	kt

Source: SRK, 2015

Table 22.2.3 is a summary of the annual production schedule, produced metal content and the associated after-tax free cashflow and NPV at an 8% discount rate for the Project.

Table 22.2.3: Annual Production Summary

Year	Production			Produced Metal			Cashflow (After-Tax)	NPV@8% (After-Tax)
	Waste	Mineralized Mined	RoM to Process	FeNb-t	TiO_2-t	Sc-t		
2015	0	0	0	0	0	0	(12,170)	(12,170)
2016	67	0	0	0	0	0	(335,555)	(310,699)
2017	178	220	0	0	0	0	(631,017)	(540,996)
2018	71	987	987	7,227	24,350	84	236,498	187,740
2019	102	987	987	7,737	23,789	101	300,635	220,975
2020	128	985	985	7,711	23,363	93	301,998	205,535
2021	92	985	985	7,859	25,560	78	279,015	175,827
2022	97	987	987	7,499	24,301	91	272,468	158,982
2023	20	989	989	7,501	23,536	87	329,542	178,041
2024	8	986	986	7,657	24,366	96	365,428	182,805
2025	3	987	987	7,670	22,959	100	359,994	166,747
2026	4	986	986	7,514	24,253	107	380,346	163,124
2027	40	986	986	7,497	23,686	105	391,189	155,346
2028	40	985	985	7,499	23,628	108	378,927	139,331
2029	99	987	987	7,510	23,815	104	356,313	121,311
2030	110	987	987	7,499	23,725	103	350,841	110,600
2031	37	986	986	7,504	23,902	99	280,206	81,790
2032	9	989	989	7,499	24,310	102	326,584	88,266
2033	3	986	986	7,775	24,806	94	338,335	84,668
2034	0	986	986	8,191	24,959	96	316,191	73,265
2035	5	985	985	7,640	24,797	105	364,630	78,231
2036	1	985	985	7,698	24,970	91	325,200	64,603
2037	3	985	985	7,697	25,533	90	320,936	59,033
2038	4	986	986	7,578	24,904	106	363,012	61,826
2039	0	986	986	7,530	24,346	97	340,569	53,708
2040	2	986	986	7,784	24,966	87	278,333	40,642
2041	6	986	986	7,568	23,868	107	366,512	49,553
2042	0	986	986	7,738	24,379	99	351,544	44,009
2043	4	986	986	7,578	24,058	102	351,350	40,726
2044	3	990	990	7,505	23,865	109	369,476	39,655
2045	0	986	986	7,539	25,507	103	352,283	35,009
2046	0	986	986	7,615	24,474	106	359,747	33,102
2047	0	985	985	7,595	24,614	106	361,969	30,840
2048	0	985	985	7,907	25,597	109	378,390	29,851
2049	0	293	513	3,343	11,522	50	147,028	10,740
2050	0	0	0	0	0	0	(143)	(10)
2051	0	0	0	0	0	0	(1,000)	(63)
2052	0	0	0	0	0	0	(1,000)	(58)
2053	0	0	0	0	0	0	(1,000)	(54)
2054	0	0	0	0	0	0	(1,000)	(50)
2055	0	0	0	0	0	0	(1,000)	(46)
Total	**1,135**	**31,086**	**31,086**	**239,664**	**766,709**	**3,115**	**9,611,603**	**2,301,735**

Source: SRK, Roche 2015

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

The economic model assumes LoM average metal prices of US$43.55/kg for ferroniobium. The market price is set with a start point of US$39/kg Nb in 2015 and increases to US$44/kg Nb in 2020 through the remaining LoM. Market prices of US$2.10/kg for titanium oxide is used for revenue purposes. The base case market price for scandium oxide is set at US$3,500/kg for scandium oxide through 2017, declining to US$3,000/kg scandium oxide during 2019 and 2020, then increasing to US$4,000/kg scandium oxide by 2023 and remaining at that level for LoM.

The after-tax NPV at an 8% discount rate over the estimated mine life is US$2.302 billion with an IRR of 27.6%. The Project economic results are summarized and presented below in Table 22.2.4.

Table 22.2.4: Economic Analysis (US$000's)

Description	Value	Units
Market Prices		
Niobium	**$43.55**	/kg
Titanium Oxide	**$2.10**	/kg
Scandium Oxide	**$3,883**	/kg
Estimate of Cash Flow (all values in $000's)		
Gross Revenue	**$18,925,111**	**$608.81**
Operating Costs		**US$/t-RoM**
Mining	($1,647,647)	$53.00
Processing	($4,219,864)	$135.75
G&A	($252,000)	$8.11
Product Freight	($97,800)	$3.15
Property/Severance taxes	$0	$0.00
By-product Credits (1)	1,610,089	($51.80)
Royalties	(286,358)	$9.21
Treatment Cost/Refining Cost	0	$0.00
Cash Closure/Reclamation	0	$0.00
Total Operating Costs	**($4,893,580)**	**$157.42**
Operating Margin (EBITDA)	**$14,031,532**	**$451.38**
Project Capital	($978,742)	$31.49
LoM Sustaining Capital	($336,686)	$10.83
Closure Costs	(71,309)	$2.29
Taxes	($3,033,191)	$97.58
After Tax Free Cash Flow	**$9,611,603**	**$309.20**
NPV @: 8%	**$2,301,735**	
Average Annual Niobium Production	**4,868,185**	**kg/y**
Average Annual Ferroniobium Production	**7,490**	**t/y**

(1) By-product credits of TiO_2
Source: SRK, 2015

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

22.3 Taxes, Royalties and Other Interests

The State of Nebraska has a number of tax incentive programs that are available to businesses of all sizes and industries. These incentives provide reductions or elimination of property, payroll, income and/or sales tax liabilities.

The State of Nebraska Tax Incentives Programs can be grouped into the following categories related to the undeveloped niobium mining/extraction activities in Elk Creek, Nebraska and Johnson County:

- Nebraska Advantage Act;
- Nebraska Advantage Research and Development Act;
- Nebraska Customized Job Training Advantage; and
- Nebraska Advantage Rural Development Act.

Based on review of the various incentive programs, it is expected that the investment credit and the compensation credits could offset part of the corporate federal tax rate over the initial part of the Project. Most likely an offset of the property taxes will occur based on the Project being a contributor to the local economy in terms of job creation.

The tax rates assumed for the economic analysis has been set at an effective rate of 20% for the first 10 years of the Project followed by a 25% effective tax rate for the remaining LoM.

The Project also includes a standard 2% NSR on proceeds.

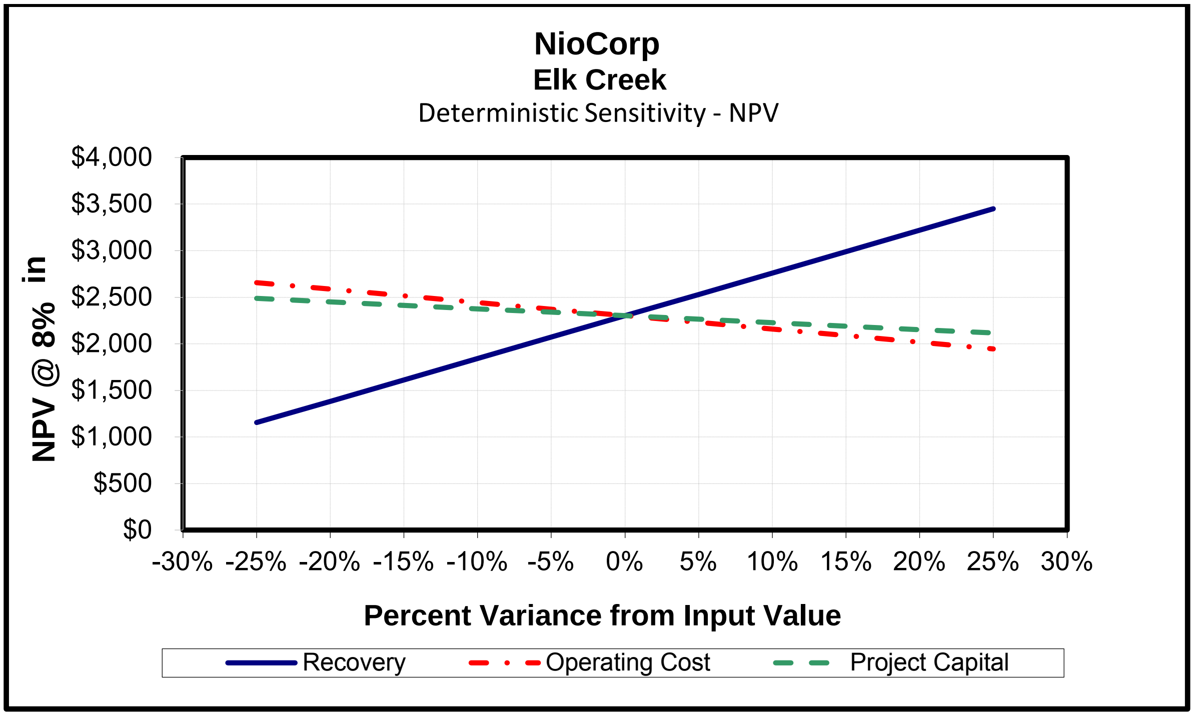
22.4 Sensitivity Analysis

Sensitivity analyses for key economic parameters are shown in Table 22.4.1 and Figure 22.4.1. This analysis suggests that Project economics are most sensitive to the metal recovery. The economics are less sensitive to operating cost followed closely by capital costs.

Table 22.4.1 Project Sensitivities NPV @ 8% After-Tax (US$ millions)

Item	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Recovery	1,154	1,383	1,613	1,842	2,072	2,302	2,531	2,761	2,991	3,220	3,450
Operating Cost	2,658	2,587	2,516	2,444	2,373	2,302	2,230	2,159	2,088	2,016	1,945
Project Capital	2,488	2,451	2,413	2,376	2,339	2,302	2,264	2,227	2,190	2,153	2,116

Source: SRK, 2015



Source: SRK, 2015

Figure 22.4.1 Project Sensitivities

22.5 Alternative Scandium Oxide Pricing

For the purpose of examining the sensitivity to the scandium oxide price projection, an alternate case was developed. The following economic analysis shows the results of the alternate case which assumes a slower ramp-up in consumption of scandium oxide in the aerospace industry. The economic model assumes LoM average metal prices of US$43.55/kg for ferroniobium. The market price is set with a start point of US$39/kg Nb in 2015 and increases to US$44/kg Nb in 2020 through the remaining LoM. Market prices of US$2.10/kg for titanium oxide are used for revenue purposes. The alternate case market price for scandium oxide is set at US$3,500/kg for scandium oxide for the first year, declining to US$2,000/kg scandium oxide by 2021 then increasing to US$3,500/kg scandium oxide by 2023 and remaining at that level for LoM.

The after-tax NPV at an 8% discount rate over the estimated mine life is US$1.959 billion with an IRR of 25.3%. The Project alternate economic results are summarized and presented below in Table 22.5.1.

Table 22.5.1: Alternate Economic Analysis (US$000's)

Description	Value	Units
Market Prices		
Niobium	**$43.55**	/kg
Titanium Oxide	**$2.10**	/kg
Scandium Oxide	**$3,416**	/kg
Estimate of Cash Flow (all values in $000's)		
Gross Revenue	**$17,381,497**	**$559.15**
Operating Costs		**US$/t-RoM**
Mining	($1,647,647)	$53.00
Processing	($4,219,864)	$135.75
G&A	($252,000)	$8.11
Product Freight	($97,800)	$3.15
Property/Severance taxes	$0	$0.00
By-product Credits	1,610,089	($51.80)
Royalties	(255,485)	$8.22
Treatment Cost/Refining Cost	0	$0.00
Cash Closure/Reclamation	0	$0.00
Total Operating Costs	**($4,862,707)**	**$156.43**
Operating Margin (EBITDA)	**$12,518,789**	**$402.72**
Project Capital	($978,742)	$31.49
LoM Sustaining Capital	($336,686)	$10.83
Closure Costs	(71,309)	$2.29
Taxes	($2,677,415)	$86.13
After Tax Free Cash Flow	**$8,454,637**	**$271.98**
NPV @: 8%	**$1,959,235**	
Average Annual Niobium Production	**4,868,185**	**kg/y**
Average Annual Ferroniobium Production	**7,490**	**t/y**

Source: SRK, 2015

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

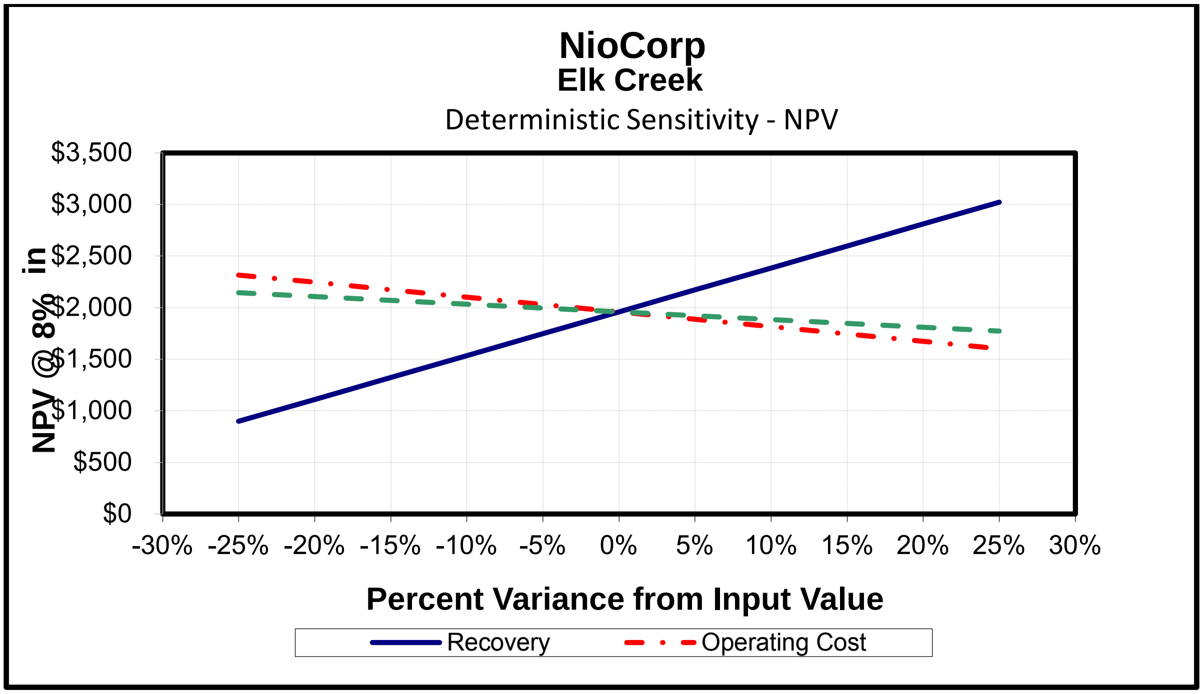
The PEA includes price and market assumptions concerning an expanded demand in the scandium market. There is no certainty that the PEA will be realized.

Sensitivity analyses for key economic parameters are shown in Table 22.5.1 and Figure 22.5.1. This analysis suggests that Project economics are most sensitive to the metal recovery. The economics are less sensitive to operating cost followed closely by capital costs. All results are positive within the +/- 25% range.

Table 22.5.1 Project Sensitivities NPV @ 8% After-Tax (US$ millions)

Item	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Recovery	897	1,109	1,322	1,534	1,747	1,959	2,172	2,384	2,597	2,809	3,022
Operating Cost	2,316	2,245	2,173	2,102	2,031	1,959	1,888	1,817	1,745	1,674	1,603
Project Capital	2,145	2,108	2,071	2,034	1,996	1,959	1,922	1,885	1,848	1,810	1,773

Source: SRK, 2015



Source: SRK, 2015

Figure 22.4.1 Alternate Project Sensitivities

22.6 Results

The project as modeled, in the base case, provides a positive after-tax NPV of US$2.30 billion at an 8% discount rate with free cash flow of US$9.61 billion after taxes. The Project generates approximately 7,500 t/y (rounded) of FeNb, 97 t/y Sc_2O_3 and a by-product of TiO_2 that offset substantial costs at current commodity price estimates. The upfront capital is US$978.7 million. The Project is net NPV positive through sensitivities of +/- 25% on operating cost, capital cost, and recovery.

23 Adjacent Properties

There are no significant properties adjacent to Elk Creek.

24 Other Relevant Data and Information

There is no additional information or explanation necessary to make the technical report understandable and not misleading.

25 Interpretation and Conclusions

25.1 Mineral Resource Estimate

SRK has constructed mineralization models for the deposit, based upon all of the available drilling information. Modelling has initially been completed in Leapfrog® by modelling the grade shells at 0.3, 0.4, 0.5 Nb_2O_5% intervals. The use of structural trends has been utilized to mimic the geological interpretation. The grade shells have been cross checked against the geological interpretation to select the optimum parameters.

SRK has undertaken a statistical study of the data, which demonstrates adequate splitting/domaining of the deposit. High grade statistical outliers have been controlled in the estimation through grade capping. SRK has undertaken a geostatistical study to investigate the niobium grade continuity which showed minor changes to the parameters used in 2014. The semi-variograms remain to have a relatively short first range of between 7 to 20 m, with the maximum range of influence of 80 to 110 m along strike and 60 m down-dip.

SRK has interpolated Nb_2O_5 grade data using OK into a block model of dimensions 5 m x 15 m x 5 m (based on an assumed mining unit) and using appropriate search and estimation parameters were then tested for sensitivity to the estimation process. The resultant block model has been fully validated and no material bias identified.

SRK has classified the Mineral Resource in the Indicated (51%) and Inferred (49%) Mineral Resource categories, mainly on the basis of the geological and grade continuity and the relatively wide drillhole spacing of up to 60 to 120 m on average. Additional Inferred material has been added to the geological model as a result of the Phase II and III drilling programs, at the end of the deposit and by increasing the model at depth, with the deeper vertical holes completed. The deposit remains open both along strike to the northwest and southeast and at depth. SRK notes that the highest grades are associated with mineralization at depth and this remains the best potential to increase the current Mineral Resource further.

The Mineral Resource Estimate for the deposit, at 0.3 Nb_2O_5% cut-off, is an Indicated Resource of 80.5 Mt at 0.71 Nb_2O_5%; and an Inferred Resource of 99.6 Mt at 0.56 Nb_2O_5%. The updated Mineral Resource represents a significant increase in the reported contained metal for the Indicated when compared to the 2014 estimate, while replacing a portion of the Inferred material which was upgraded in terms of confidence. The main reasons for the increases are:

- Phase II and III infill drilling has decreased the drill spacing to the order of 60 to 70 m through the central portion of the deposit;
- Phase II and III infill drilling has targeted higher grade material at depth in the Mineral Resource; and
- Increase in the geological understanding of the controls on the niobium mineralization and grade domaining, based on the 2014 drilling program and relogging of historical holes.

25.2 Mining and Mineral Reserve Estimate

No Mineral Reserves have been estimated for the Project. The available data indicate that underground operations using longhole stoping methods are viable for the Project. The mine maintains the target FeNb production for a 32 year period. An elevated NSR cut-off was used to minimize plant and capital requirements and to meet NioCorp forecasted market needs for FeNb. Development of the shaft, initial ramp and accesses is imperative to achieving production in early years.

25.3 Metallurgy and Processing

25.3.1 Mineral Processing

Several direct flotation and reverse flotation (carbonate flotation) experiments were conducted in mechanical flotation cells. The best result was obtained with a direct flotation. Testwork was also performed in flotation columns. Due to the entrainment of fine materials, column flotation with froth washing provided superior results to those achieved using conventional flotation techniques. A flotation column circuit was piloted, but the cleaner flotation stage did not provide the desired metallurgical results in terms of mass pull versus recovery. With additional time, effort and optimization, these results may have been improved. Furthermore, hydrometallurgy testwork showed that direct leaching of the ground mineralized material (without flotation) significantly increased the recoveries associated with the process. As a result, the flotation testwork was put on hold in pursuit of whole ore direct leaching.

25.3.2 Hydrometallurgy

Numerous tests have been performed both at the bench and mini pilot scale on the different steps of the hydrometallurgical process. These tests have led to the selection of a process that successfully produced a niobium product suitable for subsequent processing into a final ferroniobium product. In doing so, a titanium oxide product suitable for further treatment into a pigment grade product was also produced. Scandium was successfully loaded and selectively stripped from an organic in a solvent extraction stage. The best results were obtained using a chloride based process, followed by a sulfate based process involving an elevated temperature sulfation, followed by selective precipitation of niobium and titanium, and independent extraction of scandium from both the chloride and sulfate liquors.

Key results and interpretations are:

- Niobium recovery of 92%;
- Titanium recovery of 87.6%;
- Scandium recovery of 90%;
- The process development was challenging. Though the unit processes considered have been reported on in literature and are known process, the selection of the unit processes has been a challenge due to the required materials of construction, the recoveries of the different products and the quality required in the different products; and
- Solid liquid separation at the different steps of the process will be challenging. Centrifugation is a suitable, but expensive option.

25.3.3 Pyrometallurgy

Preliminary testing performed on the niobium oxide product from the hydrometallurgy testwork has demonstrated that ferroniobium can successfully be produced. An external source of heat will be required in order to meet the reduction reaction energy requirements. Niobium recovery is estimated at 97% for the pyromet process.

25.4 Tailings Storage Facility

Based on the parameters and assumptions outlined in Section 18.2, the Area 7 and Area 1 TSFs have been design with adequate capacity to manage planned water leach residue, gypsum residue, and iron oxide tailings deposition for a 30 year mine life.

25.5 Environmental and Social

A number of key permits and environmental management requirements have been identified for the Elk Creek Project, some of which need to be implemented as soon as practicable in order to maintain the proposed Project schedule.

- While not necessarily complex, the timing generally required to complete permitting through any federal regulatory agency requires that NioCorp engage key agencies (in this case the USACE and possibly the U.S. EPA) early on in Project development, and consider the siting and orientation of facilities carefully in order to minimized the risk of a protracted National Environmental Policy Act analysis of the Project. At this time, the design emphasis on limiting impacts to jurisdictional wetlands and waters of the U.S. should result in the use of an Environmental Assessment as opposed to an Environmental Impact Statement as the disclosure document for the USACE analysis of the Project. However, pursuing the option of including a 48 km pipeline and discharge of dewatering water to the Missouri River could trigger additional federal involvement and extend the scope of NEPA to EIS proportions.
- Perhaps one of the most critical approvals likely to be needed by the operation will be a radioactive materials license from the Nebraska Department of Health and Human Services, Office of Radiological Health. Because of their limited experience with hardrock mining in the State of Nebraska, much less mining that includes or NORM, the DHHS may require additional information and more time to approve the Project under a Broad Scope License. Early and frequent engagement is a necessity with respect to this regulatory agency.
- Documentation of existing baseline environmental conditions at the site was initiated in 2014 and should continue throughout the permitting process. Additional studies will need to be added once regulatory authorities have been given an opportunity to review the current mine plan presented in this PEA and assess their particular data needs for approval of the Project.
- Surface water monitoring should continue on a quarterly basis throughout the permitting process, and extend into construction and operations as part of the Environmental Management System. Flow monitoring of all surface water sampling locations should be added in order to calculate potential loading to the watershed(s). The NDEQ Water Quality Division will be engaged as soon as practicable in order to discuss the Project and potential data needs in order to initiate NPDES discharge permitting. This would include both local discharges as well as discharges to the Missouri River.

- A wetland delineation and potential jurisdictional waters assessment was conducted in late 2014 to identify wetland and drainage features within the proposed Project boundary that could be classified as jurisdictional waters of the U.S., and therefore be subject to the permitting requirements of the USACE. A total of 45 wetlands, encompassing an area of approximately 10.64 acres, were identified in agricultural fields, pastures, roadside ditches and abutting stream channels within the general project area and outside of floodplain areas identified by the State of Nebraska. Nine unnamed streams were also found during the field investigation for a total length of approximately 13,726 ft. At the time of this report, all wetlands and waters in the Project study area are assumed to be jurisdictional. Olsson Associates and NioCorp are currently working with the USACE in order to obtain a final determination. A delineation of potential jurisdictional features along the discharge water pipeline corridor to the Missouri River is still pending.
- 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 Section 20. Addition studies are ongoing.
- Closure costs for the Project have been estimated at just over US$60 million, the bulk of which (US$40 million) is intended for reclamation and closure of the tailings disposal facility. Approximately US$15 million has been allocated for surface reclamation of the remaining facilities (i.e., building demolition, site regrading and revegetation, shaft closure, etc.), while the remaining US$5 million is set aside for post-closure monitoring and maintenance. This estimate will be refined during development of the feasibility study.
- Community engagement has occurred in parallel with Nebraska field operations and has included public meetings, presentations to public agencies, communications with local and state politicians, and one-on-one meetings with area landowners.

25.6 Projected Economic Outcomes

The project as modeled provides a positive after-tax NPV of US$2.3 billion at an 8% discount rate with free cash flow of US$9.6 billion after taxes. The Project generates approximately 7,500 t/y of FeNb, a co-product of Sc_2O_3 and a by-product of TiO_2 that offset cost substantially at current commodity price estimates. The upfront capital is US$978.7 million. The Project is net NPV positive through sensitivities of +/- 25% on operating cost, capital cost, and recovery.

Variations in the metal prices have a significant impact on the financial results of the Project.

The Mineral Resource presented has been reported following CIM guidelines. The PEA is preliminary in nature, that it includes a level of engineering precision and assumptions which are currently considered too speculative to have the economic considerations applied to them that would enable Mineral Resources to be categorized as Mineral Reserves.

Inferred Mineral Resources are not included in the mine plan for this PEA. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

25.7 Foreseeable Impacts of Risks

Foreseeable risks and impacts include:

- Changes in product prices and market conditions could have a large impact on the project economics. SRK notes that the pricing for ferroniobium, the primary product that the operation would produce, has a more stable pricing history than other commonly mined commodities, such as gold and copper. SRK notes that the pricing of scandium is a significant portion of the Project revenue and achieving the revenue projected in this study is subject to market growth in scandium, which is a developing market with a risk of oversupply and/or undersupply disrupting pricing;
- Nebraska does not have clearly defined regulations with respect to permitting mines and the learning curve could potentially impact the total time to market for the project. SRK notes that this also could be considered an opportunity as the regulatory environment may be more streamlined;
- The construction and permitting timeline requires a carefully coordinated design, permitting, and scheduling effort that will need to occur concurrent to and immediately after completion of the next level of study to achieve the project startup dates. Any issues with this effort could impact the startup date; and
- The results of ongoing hydrogeology studies should clarify expected mine water quantities and quality and the resulting impact on the project mining and water treatment requirements.

26 Recommendations

26.1 Recommended Work Programs and Costs

26.1.1 Geology and Resource

SRK has no further recommendations for additional drilling needed to support the Mineral Resource for the impending feasibility study. SRK notes that the current understanding of the extents of the deposit, while sufficient for the current level of study, are still limited by the extent of the drilling, and that the deposit is locally open along strike and at depth. SRK previously highlighted to the Company a series of work programs to increase the confidence in the assay database for the historical drilling. SRK understands that the Company has programs in place to address all the outstanding issues, with the results to be reflected in the next updated Mineral Resource estimate. SRK is of the opinion that there may be an opportunity to further refine and improve the understanding of the mineralization, particularly near the top of the deposit where mining is scheduled to begin. This may be considered in planning for future exploration and mining, as a matter of course in the development of the project.

26.1.2 Mineral Reserve Estimate, Mining, and Geotechnical

SRK recommends that the following areas be reviewed during the next phase of work:

- Review drift sizes to optimize and minimize waste movement;
- Review and refine the development required during the pre-production period to minimize cost and maximize faces available for development;
- Detailed sequencing of shaft development and pre-production activities. To achieve production in the desired timeframe several activities may need to be completed in parallel;
- Review and confirm sill locations and optimize sill removal to maximize the recovery of the resource;
- Review the mine ventilation to optimize the system maximizing the use of all shafts. This should include reviewing the impact of utilizing more electric powered equipment to reduce ventilation needs. More specifically review use of electric LHD’s in the stopes and development mucking to muck bays.
- Review ventilation mine design to minimize the number of angles/turns in the system to reduce losses. Also evaluate sealing mined levels to eliminate losses while maintaining necessary accesses for ventilation system maintenance.
- Review shaft costs, productivities, and PEA assumptions to reduce cost, optimize sizing, and reduce upfront time to production. Consider impacts of knowledge of hydrology and geotechnical conditions in the mine shaft and key underground infrastructure areas. Provide geotechnical and hydrogeological analysis of the shaft and vent hole areas.
- Review the truck haulage system to see if there is a more cost effective method or options to reduce cost, increase productivity or reduce number of equipment and labor required for the haulage of mineralized material to the shaft.
- Refine the backfill quantities, material characteristics, and placement scheme to optimize the system and confirm PEA assumptions. This should include testwork to determine paste makeup and strength.

- Perform geotechnical analysis on sills to minimize loss and confirm required sill thickness and location.
- Optimization of the production schedule by delaying development to an as-needed basis.

26.1.3 Hydrogeology

The hydrogeologic characterization of the groundwater system, including short term testing during core drilling, nominal 10 day pumping test, and nominal 30 day injection test allowed definition of hydrogeological parameters for the carbonatite within the mineralization area. This testing confirmed that mineralization area is highly permeable, located within a confined groundwater system, and isolated from the shallow groundwater system by low permeable PENN strata. However, the lateral extent of highly permeable rock is currently unknown. Dewatering requirements were preliminarily estimated as an average of the total of pumping rates from dewatering wells predicted for an unbounded and bounded granite deep groundwater system.

SRK recommends to conduct additional hydrogeological testing of:

- The area of the proposed shaft to obtain a site specific hydrogeological data to better evaluate grouting procedure and cost of installation (completed in July 2015); and
- Carbonatite and granite outside of mineralization area to define lateral extent of zone high permeability within deep groundwater system (this work can be conducted at the detailed engineering phase).

Current early results of groundwater sampling efforts on the site indicate naturally occurring but locally elevated levels of salinity (TDS > 18,000 mg/L), plus arsenic and radiochemical parameters exceeding drinking water standards. These findings suggest that further groundwater sampling is warranted to characterize and delineate the water quality, and to determine design criteria for water treatment. Options for disposal or storage of water treatment residue also should be investigated. A monitoring program for selected existing piezometers should be established.

26.1.4 Processing Plan

In order to improve process efficiency and minimize the potential risks of operating a full-scale plant, testing programs need to be carried out during the different phases of engineering studies. While some small-scale test methods provide adequate information for scoping or prefeasibility studies, it is suggested that pilot plant testing be completed to provide sufficient information for the process development during the feasibility study at the ±15% accuracy levels. At the current stage of the study, a bench scale laboratory testing program including mini-pilot and pilot testing has been completed for understanding the mineralized material sample characteristics and its behavior under controlled conditions. During this program testing, sufficient data has been collected from the hydromet circuit to produce a preliminary economic assessment and has justified the need to complete hydromet pilot level studies. Implementation of such testwork will provide additional key information to confirm bench test results and enable development of mass and energy balances, equipment selection and plant design. As process safety risk is an important factor, a comprehensive pilot plant program will help to reduce possible risks associated with the construction and operation of the new full-scale process plant.

Mineral Processing

Comminution testwork should be finalized using representative samples and it should include a sufficient number of standard Bond indices, Bond Low-Energy Impact tests, JK Drop-weight tests, and SMC tests with abrasion indices. The tests should be conducted on several samples and composites covering the potential variability of the grinding characteristics of the deposit.

Grinding simulations should be performed to provide sizing data for the grinding mill(s), as well as the material balance projection (flow rate, water rate, % solids, particle size distribution, etc.), which can be used to confirm the sizing of other equipment such as pumps, water supply equipment, screens, crusher, cyclones, etc. The predicted plant performance (power draw, specific power consumption (kWh/t) and operating work index) will also have to be presented in each simulation report.

Samples should be collected for settling tests, paste backfill testwork, and environmental characterization.

Hydrometallurgy

An ongoing testwork program to further define and test all aspects of the preliminary process is required. Bench scale tests and mini pilots will be run to provide the final basis for additional pilot plant work. The tests will be conducted on whole ore samples resulting from representative samples. The additional pilot plant will validate the robustness of the hydrometallurgical process with regards to the variability of the mineralized material. From the pilot plant, samples will be collected for settling and filtration tests, paste backfill testwork, and environmental characterization. Settling and filtration tests will be performed by equipment suppliers to confirm equipment sizing.

Care should be taken in the subsequent testwork program to assess different materials of construction through the use of coupon testing or other means.

Pyrometallurgy

Further testwork should be performed on larger sample sizes to assess the energy requirement and to optimize the recovery. Induction furnaces as well as arc furnaces should be tested when considering the different types of reduction. Further testing of alumino-thermic reduction should be performed using a heel combined with an induction furnace. Testing of a reduction stage followed by oxidation of the impurities using oxygen should be performed to assess the benefit in product purity.

26.1.5 Tailings Storage Facility

The following work is recommended to facilitate further feasibility design of the Area 7 and Area 1 TSFs and associated evaporation areas:

- Confirm minimum design criteria applicable to the TSF via discussions with pertinent regulatory bodies, including closure and reclamation.
- Verify assumptions regarding WOUS for Area 7 and Area 1.
- Confirm design solid-to-liquid ratios anticipated from the water leach residue, gypsum residue, and iron oxide tailings.
- Confirm engineering properties of water leach residue, gypsum residue, and iron oxide tailings including gradation, settling potential, density, drainage/permeability, consolidation, and strength.

- Confirm geochemical properties of the tailings to verify the containment design (i.e. composite-lined tailings solids storage and double lined containment at evaporation ponds).
- Confirm containment requirements based on potential changes to the mine plan quantities that could impact the overall size and staging of the TSF.
- Confirm Area 7 and Area 1 TSF footprint foundation design assumptions for gradation, soil classification, moisture content, compaction, and strength.
- Determine applicable specifications for embankment compaction, base compaction, drain rock, filter sand, and low-permeability compacted clay core.
- Confirm assumptions related to mass stability including seismic stability.
- Refine assumptions for TSF water balance including 7 year dry and wet scenarios, and design storm criteria.

26.1.6 Environmental and Social

The following are general recommendations regarding the environmental and social management of the Project, and do not necessarily come with a specific cost for implementation (unless otherwise noted).

- A meteorological station has been set up at the Project site. NioCorp will continue to collect climatological and air quality data for use in air quality permitting, but should consider the installation of a second station in the location of the tailings impoundment, as well as installation of PM_{10} monitors to establish baseline fugitive dust conditions prior to the initiation of construction activities and deposition of tailings.
- Geochemical characterization of the mineralized material, waste rock and tailings (including radiological characterization) needs to be expanded. Post-metallurgical testing of the tailings material is necessary to obtain solids and supernatant chemistry, and generate data needed to evaluate the closure alternatives for the underground workings and tailings impoundment, and the potential requirements for post-closure water management, if necessary.
- Most of the major permits for the mine will require some form of public scoping and review, especially any federal permitting through the NEPA process. While stakeholder engagement has been undertaken by NioCorp, the breadth and scope of this commitment needs to be expanded in order to secure a Social License for the Project. This includes full disclosure of the findings of the PEA, as well as progress toward the feasibility study and proposed construction schedule.
- A detailed traffic study will need to be completed given the probability of using rail to transport materials to and from the site as well as semi-tractor trailer shipments. Both activities will impact state highway rights-of-way, including State Highway 50 and possibly Highway 62.
- As early as practicable, jurisdictional delineations along the proposed rail and discharge pipeline corridors should be completed and assessed for impacts, as well as the potential USACE expansion of involvement. In parallel, NioCorp needs to complete the feasibility assessment of water treatment and re-injection of reject water within the carbonatite formation as an alternative to direct discharge to the Missouri River.
- Post-closure management of the site needs to be considered during the planning and design phases in order to minimize potential long-term environmental and social impacts to the area; and

- NioCorp should initiate development of Project-specific environmental and social management plans based on the potential impacts identified during the permitting process.

26.1.7 Marketing and Economics

Market studies have been conducted on niobium and scandium oxide. A market study should be performed on titanium dioxide. The market studies provide summary commodity price projections, product valuations, market entry strategies, or product specification requirements. Transportation studies should also be conducted for the purpose of estimating transport and shipping costs for the commodities produced. The estimated cost of the studies is US$200,000.

26.1.8 Costs

In addition to the market studies the following work programs should be performed. Table 26.1.8.1 summarized the recommended work program costs.

Table 26.1.8.1: Summary of Costs for Recommended Work

Recommended Work	Cost Estimate (US$)
Feasibility study with hydrogeological, geochemistry, and geotechnical work programs.	6.0 million
Process feasibility study design and metallurgical testing program including backfill testing	2.4 million
Tailings geotechnical field testwork with drilling, logging, cone penetration testing, and in situ and borrow materials laboratory testing in Area 7	160,000
Marketing studies	200,000
Total	**$8.76 million**

Note: The majority of the work included in the table is ongoing at the time of writing.

27 References

ASDR, 2015. Roche, Elk Creek Niobium Project, Study Report and Scoping Level Paste Backfill Study, Annex A, Detailed Capital Cost and Operating Cost, June 3, 2015

Anzman, J.R., 1976. Interpretation of Gravity and Magnetic Data, Elk Creek Anomaly, Johnson and Pawnee Counties, Nebraska. Molycorp, Inc. June 29, 1976. 22 pages.

Berendsen, P. and Weis, T., 2001. New Kimberlite Discoveries in Kansas: Magnetic Expression and Structural Setting. Transactions of the Kansas Academy of Science. 104(3-4), pp. 223-236.

Brookins, D. G., Treves, S. B., and Bolivar, S. L., 1975. Elk Creek, Nebraska Carbonatite: Strontium geochemistry: Earth Planet. Sci. Lett., v. 28, p. 79–82.

Brown, 2014. RE: Endangered Species in Pawnee and Johnson Counties. Personal Communications from Mary Brown to Robert Joeckel on Mary 27, 2014.

Burchett, R.R. and Reed, E.C., 1967. Central Guidebook to the Geology of southeastern Nebraska. The University of Nebraska-Lincoln, Conservation and Survey Division, Lincoln, Nebraska Geological Survey.

Burchett, R.R. and Reed, E.C., 1967. Central Guidebook to the Geology of southeastern Nebraska. The University of Nebraska-Lincoln, Conservation and Survey Division, Lincoln, Nebraska Geological Survey.

Carlson, M.P. and Treves, S.B., 2005. The Elk Creek Carbonatite, Southeast Nebraska – An Overview. Natural Resources Research, Vol. 14, No. 1 March, 2005. pp. 39-45.

Carlson, M.P., 1992. Tectonic implications and influence of the Midcontinent. Rift System in Nebraska and adjoining areas. In: (Richard W. Ojakangas, Albert B. Dickas, John C. Green, Eds. Basement tectonics 10: proceedings of the Tenth International Conference on Basement Tectonics, Duluth, MN.

Condor, 2011. Assessment of the Elk Creek, Nebraska; FALCON Survey for Quantum Rare Earth Developments Corporation. Condor Consulting, Lakewood, Colorado, USA. October 16, 2011. 169 pages.

Cook W.B. and Sherer, R.L., 1986. Proposed Land Retention for 1986, Elk Creek, Nebraska. February 5, 1986.

Drenth, B.J., 2014. Geophysical expression of a buried niobium and rare earth element deposit: the Elk Creek carbonatite, Nebraska, USA. Interpretation 2, 169-179.

Erdosh, G., 1979. The Ontario Carbonatite Province and Its Phosphate Potential. Economic Geology, Vol. 74, pp. 331-338.

Farmer, G. L., R. M. Kettler, and P. L. Verplanck, 2013. Geochemical and isotopic constraints on the age and origin of the Elk Creek carbonatite complex, southeast Nebraska: Presented at Geological Society of America 65th Annual Meeting, 21-7.

Fugro, 2011, Elk Creek, Nebraska; FALCONTM Airborne Gravity Gradiometer Survey for Quantum Rare Earth Development Corp.; Processing Report. April 2011. 39 pages.

King, P.B., 1969. The Tectonics of Middle North America. Hafner Publishing Company.

McBee, W., 2003. Nemaha Strike-Slip Fault Zone. Search and Discovery Article #10055.

McCallum, N.G. and Cathro, M.S., 2010. Technical Report on the Elk Creek Property, South-East Nebraska; for Quantum Rare Earth Developments Corp. August 9, 2010. 173 pages.

Mitchell, R.H., 2005. Carbonatites and Carbonatites and Carbonatites. The Canadian Mineralogist; December 2005; v. 43; no. 6; p. 2049-2068.

NRCS 2015. United States Department of Agricultural Natural Resources Conservation Service Web Soil Survey. http://websoilsurvey.sc.egov.usda.gov/App/HomePage.htm. Accessed April 7, 2015.

Olsson Associates. 2014. Wetland Delineation Report. Prepared for Elk Creek Resources Corporation, NioCorp Development, Elk Creek Project, Johnson and Pawnee County, NE. OA Project No. 014-2723.

OnG Commodities LLC, 2015. CONFIDENTIAL: Scandium: A Market Assessment, prepared for Niocorp Ltd by OnG Commodities LLC, July 2015.

Palacas, J.G., Schmoker, J.W., Dawes, T.A., Pawlewicz, M.J., and Anderson, R.R.,1990, Petroleum source-rock assessment of Middle Proterozoic (Keweenawan) sedimentary rocks, Eischeid #1 well, Carroll County, Iowa, in Anderson, R.R., ed., The Amoco M.G. Eischeid #1 Deep Petroleum Test, Carroll County, Iowa, Preliminary Investigations: Iowa Department of Natural Resources, Geological Survey Bureau, Special Report Series No. 2, p. 119-134.

Richardson, D.G. and Birkett, T.C., 1996. Carbonatite associated Deposits; in Geology of Canadian Mineral Deposit Types, O.R. Ecstrand, W.D. Sinclair and R.I. Thorpe, Editors, Geological Survey of Canada, Geology of Canada Number 8, pp. 541-558.

Roskill, 2017. CONFIDENTIAL: Niobium: Market Outlook to 2017 - Twelfth Edition, 2013, Copyright © Roskill Information Services Ltd. ISBN 978 0 86214 592 7.

SGS 2015. Process Development Metallurgical Testing on Samples from the Elk Creek Deposit, Project 14379-002" prepared by SGS Canada Inc., dated April 10, 2015

SRK, 2014. NI 43-101 Technical Report on Resources, Elk Creek Niobium Project, Nebraska, Effective Date: September 9, 2014, Report Date: November 3, 2014, Prepared by SRK Consulting (U.S.), Inc. for NioCorp Development Ltd .

SRK, 2015. NI 43-101 Technical Report, Updated Mineral Resource Estimate, Elk Creek Niobium Project, Nebraska, Effective Date: February 20, 2015, Report Date: March 10, 2015, Prepared by SRK Consulting (U.S.), Inc. for NioCorp Development Ltd.

Schneider, R., K. Stoner, G. Steinauer, M. Panella, and M. Humpert (Eds.). 2011. The Nebraska Natural Legacy Project: State Wildlife Action Plan. 2nd ed. The Nebraska Game and Parks Commission, Lincoln, NE

Sisernos and Yernberg, Molycorp Internal Memo Niobium Analytical Standardization. June, 1983.

Tetra Tech, 2012. Report to: Quantum Rare Earth Developments Corp, Elk Creek Nb Project, Nebraska, US, Resource Estimate Update, Document No. 1291370100-REP-R0001-02, Effective Date: April 23, 2012, prepared for Quantum Rare Earth Developments Corp. by Tetra Tech Wardrop.

Treves, S.B., and Low, D.J., 1983. Precambrian Geology of Eastern and Central Nebraska. GSA Abstracts with Programs, north central Section, 15, No.4, pp.266-267.

United States Department of Agriculture (USDA) Soil Conservation Service (SCS), 1984. Soil Survey of Johnson County, Nebraska. National Cooperative Soil Survey. 1984.

Wayne, W.J. 1981. Kansan Proglacial Environment, east-central Nebraska. American Journal Science 281:375-389.

Woolley, A.R., 1989. The Spatial and Temporal Distribution of Carbonatites. In: Carbonatites, Genesis and Evolution (K. Bell, ed.). Unwin Hyman, London, pp 15-37.

Wyllie, P.J. and Lee, W-J., 1998. Model System Controls on Conditions for Formation of Magnesiocabonatite and Calciocarbonatite Magmas from the Mantle. Journal of Petrology, Volume 39, Number 11&12. Pp 1885-1893. 21 May 1998. 9 pages.

Xu, A., 1996, Mineralogy, petrology, geochemistry and origin of the Elk Creek carbonatite, Nebraska: Ph.D. thesis, University of Nebraska–Lincoln.

XPS Consulting and Testwork Services (XPS) Memorandum to NioCorp, 24 April 2015; Calcination and Aluminothermic Reduction of NioCorp FeNb Concentrate, Kingston Process Metallurgy (KPM)
Memoranda to NioCorp:
12 June 2015, Niobium smelting test plan – update.
12 June 2015, Update on material characterization of NioCorp product.
23 July 2015, Results from smelting and reduction tests.

U.S. Geological Survey, 2015, Mineral commodity summaries 2015: U.S. Geological Survey, pp 110-111; pp 140-141; pp 170-171.

28 Glossary

The Mineral Resources and Mineral Reserves have been classified according to the "CIM Definition Standards for Mineral Resources and Mineral Reserves" (May 10, 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 Resources

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 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 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 Reserves

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 prefeasibility or feasibility level as appropriate that include 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 mineralized material 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 prefeasibility 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

The following general mining terms may be used in this report.

Table 28.3.1: 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 mineralized material.
Crushing	Initial process of reducing 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 economic to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined.
Dip	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 the deposit or stope.
Gangue	Non-valuable components of the mineralized material.
Grade	The measure of concentration of gold within mineralized rock.
Hangingwall	The overlying side of the deposit or slope.
Haulage	A horizontal underground excavation which is used to transport mined material.
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 minimizes the estimation error.
Level	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
Material Properties	Mine properties.
Milling	A general term used to describe the process in which the mineralized material 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.
Pillar	Rock left behind to help support the excavations in an underground mine.
RoM	Run-of-Mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.

Term	Definition
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, mineralized material 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.
Stope	Underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur 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 report.

Table 28.4.1: Abbreviations

Abbreviation	Unit or Term
%	percent
°C	degrees Centigrade
μm	micron or microns
APS	Azimuth Pointing System
cfm	cubic feet per minute
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
CoG	cut-off grade
CWA	Clean Water Act
DDH	diamond drilling
DHHS	Nebraska Department of Health and Human Services
dia.	diameter
EA	Environmental Assessment
ECRC	Elk Creek Resources Corp.
EDC	Environmental Design Criteria
EIA	Environmental Impact Assessment
ELA	Exploration Lease Agreements
FeNb	ferroniobium
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
g	gram
G&A	general & administration
g/L	gram per liter
g/t	grams per tonne
Ga	billion years ago
gal	gallon
gD	gravity gradiometer
GHG	Greenhouse Gas
gpm	gallons per minute
GPS	global positioning system
h	hour

Abbreviation	Unit or Term
ha	hectares
hp	horsepower
ICP	induced couple plasma
IDW	inverse distance weighting
kg	kilograms
km	kilometer
km^2	square kilometer
koz	thousand troy ounce
kt	thousand tonnes
kt/d	thousand tonnes per day
kt/y	thousand tonnes per year
kV	kilivolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter
L/sec	liters per second
L/sec/m	liters per second per meter
lb	pound
LHD	load-haul-dump
LHS	longhole stoping
LoM	Life-of-Mine
m	meter
m.y.	million years
m^2	square meter
m^3	cubic meter
Ma	million years ago
mg/L	milligrams/liter
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
Moz	million troy ounces
MS	mass spectrometry
MSHA	U.S. Department of Labor, Mine Safety & Health Administration
Mt	million tonnes
MW	million watts
Nb_2O_5	niobium pentoxide
NEPA	National Environmental Policy Act
NI 43-101	Canadian National Instrument 43-101
NORM	Naturally Occurring Radioactive Materials
NPDES	National Pollutant Discharge Elimination System
NSR	net smelter return
OK	Ordinary Kriging
OTP	Option To Purchase
oz	troy ounce
PLSS	Public Land Survey System
ppb	parts per billion
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
RC	rotary circulation drilling
REE	rare earth element
REO	rare earth oxides
RoM	Run-of-Mine
RQD	Rock Quality Description
SAAB	strong acid agitated bake

Abbreviation	Unit or Term
Sc	scandium
Sc_2O_3	scandium oxide
sec	second
SG	specific gravity
SRM	standard reference material
t	dry metric tonne
t/d	tonnes per day
t/h	tonnes per hour
t/y	tonnes per year
TiO_2	titanium dioxide
TMI	total magnetic intensity
TSF	tailings storage facility
TSP	total suspended particulates
USACE	U.S. Army Corps of Engineers
USBM	U.S. Bureau of Mines
USEPA	U.S. Environmental Protection Agency
USGS	U.S. Geological Survey
W	watt
WOUS	waters of the U.S.
XRD	x-ray diffraction
XRF	x-ray refraction
y	year

Appendices

Appendix A: Certificates of Qualified Persons

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
City and County of Cardiff
CF10 2HH, Wales
United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.co.uk
Tel: + 44 (0) 2920 348 150
Fax: + 44 (0) 2920 348 199

CERTIFICATE OF QUALIFIED PERSON

I, Martin Frank Pittuck, MSc, CEng, MIMMM do hereby certify that:

1. I am Director and Corporate Consultant (Mining Geology) of SRK Consulting (UK) Ltd with an office at 5th Floor, Churchill House, Churchill Way, Cardiff CF10 2HH.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of August 4, 2015 (the "Technical Report").
3. I am a graduate with a Master of Science in Mineral Resources gained from Cardiff College, University of Wales in 1996 and I have practised my profession continuously since that time. Since graduating I have worked as a consultant at SRK on a wide range of mineral projects, specializing in rare metals and igneous deposits. I have undertaken many geological investigations, resource estimations, mine evaluation technical studies and due diligence reports. I am a member of the Institution of Materials Mining and Metallurgy (Membership Number 49186) and I am a Chartered Engineer;
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 June 17 to 19, 2014.
6. I am co-author of this report and the QP responsible for data verification and the mineral resource estimate Sections 12, and 14 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 was in the preparation of the reports titled "NI 43-101 Technical Report, Updated Mineral Resource Estimate, Elk Creek Niobium Project, Nebraska," with an Effective Date of February 20, 2015 and "NI 43-101 Technical Report, Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of April 28, 2015, both prepared for NioCorp Developments Ltd.
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 16th Day of October, 2015.

"Signed and Sealed"

____

Martin Frank Pittuck, MSc, CEng, MIMMM

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, Benjamin Parsons, MSc, MAusIMM (CP) do hereby certify that:

1. I am a Principal Consultant (Resource Geology) of SRK Consulting (U.S.), Inc., Suite 600, 1125 Seventeenth Street, Denver, CO 80202.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of August 4, 2015 (the "Technical Report").
3. I graduated with a degree in Exploration Geology from Cardiff University, UK in 1999. In addition, I have obtained a Masters degree (MSc) in Mineral Resources from Cardiff University, UK in 2000 and have worked as a geologist for a total of 15 years since my graduation from university. I am a member of the Australian Institution of Materials Mining and Metallurgy (Membership Number 222568) and I am a Chartered Professional.
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 co-author of this report providing assistance in the preparation of the geological model and Mineral Resource Estimate under the guidance of Martin Pittuck. I am the QP responsible for Sections 4 to 11 (except 5.4.1) 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 was in the preparation of the reports titled "NI 43-101 Technical Report, Updated Mineral Resource Estimate, Elk Creek Niobium Project, Nebraska," with an Effective Date of February 20, 2015 and "NI 43-101 Technical Report, Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of April 28, 2015, both prepared for NioCorp Developments Ltd.
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 16th Day of October, 2015.

"Signed and Sealed"

____
Benjamin Parsons, MSc, MAusIMM (CP)

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, Vladimir I. Ugorets, PhD, MMSAQP, do hereby certify that:

1. I am a Principal Hydrogeologist of SRK Consulting (U.S.), Inc., Suite 600, 1125 Seventeenth Street, Denver, CO 80202.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of August 4, 2015 (the "Technical Report").
3. I graduated with a degree as a Mining Engineer Hydrogeologist from the Moscow Geological Prospecting Institute (former USSR) in 1978. In addition, I obtained a Ph.D in Hydrogeology in 1984 from the Moscow Geological Prospecting Institute. I am a QP member (01416QP) of the Mining and Metallurgical Society of America (SME) with special expertise in Geology. I have worked as a Hydrogeologist for a total of 36 years since my graduation from university. My relevant experience includes planning field hydrogeological studies and analyzing their results for numerous mining projects, conducting numerical groundwater and solute transport modeling, and, optimizing wellfields for groundwater extraction.
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 September 8 to 10, 2014.
6. I the QP responsible for hydrogeology Sections 16.3, dewatering portion of 18.3, 20.3.8 and portions of Sections 1 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. I have not had prior involvement with the property that is the subject of the Technical Report.
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 contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th Day of October, 2015.

"Signed and Sealed"

____
Vladimir I. Ugorets, Ph.D., MMSAQP

CERTIFICATE OF QUALIFIED PERSON

Eric Larochelle, Eng.
Eric.larochelle@roche.ca

I, Eric Larochelle, B.Eng., Eng. of Sandy, UT, USA, do hereby certify:

1. I am currently employed as Director, Specialty Metals & Hydrometallurgy at Roche Engineering, Inc., Suite 502, 9815 Monroe, Sandy, UT, USA, 84070.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska," with an Effective Date of August 4, 2015 and Amended Report Date of October 16, 2015 (the "Technical Report").
3. I graduated from McGill University (Montréal, Qc, Canada) with a B. Eng in Chemical Engineering in 1989. I am a Senior Chemical Engineer, Member of the Ordre des Ingénieurs du Québec (#112819);

   I have worked as a chemical engineer in the mineral industry for 26 years. My technical expertise includes hydrometallurgical process evaluation, projects evaluation and plant design. I have been involved in several scoping studies and feasibility studies. I have participated in worldwide projects in specialty metals, rare earths, base metals and industrial minerals.
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 by 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 October 22, 2014 for one day.
6. I am responsible for metallurgical testing and recovery methods Sections 13 (except 13.1) and 17 (except 17.1), 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 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 contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th day of October 2015, Montreal, Quebec, Canada.

"Original document signed and sealed"
by Eric Larochelle, Eng.

Eric Larochelle, Eng.
Roche Engineering, Inc.

CERTIFICATE OF QUALIFIED PERSON

Alain Dorval, Eng.
alain.dorval@roche.ca

I, Alain Dorval, B.Sc., Eng. of Montreal, Quebec, do hereby certify:

1. I am Manager, Mining and Mineral Processing at Roche Ltd. Consulting Group with an office at 33 St-Jacques, 2nd floor, Montréal, Québec, Canada, H2Y1K9.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska," with an Effective Date of August 4, 2015 and Amended Report Date of October 16, 2015 (the "Technical Report").
3. I graduate from Laval University (Quebec City, Qc, Canada) with a B.Sc, in Mining Engineering in 1983. I am a member of good standing (#96127) of the Ordre des Ingénieurs du Québec (Order of Engineers of Quebec).

   My relevant experience includes over 32 years related to the mineral processing including industrial minerals, precious metals, base metals and iron ores. My experience includes mainly: mining operation, research process development for various minerals, consulting, and engineering.
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 by 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 October 22, 2014 for one day.
6. I am the QP responsible for mineral processing plant and infrastructure Sections 13.1, and 17.1, 18 (except for 18.1.2, 18.2 and 18.3) 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 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 contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th day of October 2015, Montreal, Quebec, Canada.

"Original document signed and sealed"
by Alain Dorval, Eng.

Alain Dorval, Eng.
Manager, Mining and Mineral Processing,
Roche Ltd, Consulting Group

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, Joanna Poeck, BEng Mining, SME-RM, MMSAQP, do hereby certify that:

1. I am a Senior Mining Engineer of SRK Consulting (U.S.), Inc., Suite 600, 1125 Seventeenth Street, Denver, CO 80202.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of August 4, 2015 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Colorado School of Mines in 2003. I am a Registered Member of the Society of Mining, Metallurgy & Exploration Geology. I am a QP member of the Mining & Metallurgical Society of America. I have worked as a Mining Engineer for a total of 11 years since my graduation from university. My relevant experience includes open pit and underground design, mine scheduling, pit optimization and truck productivity analysis.
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 the QP responsible for mining and reserves Sections 15, 16 (except 16.2, 16.3, 16.7.3. 16.7.4, 16.7.6, 16.8.2 through 16.8.6) 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 was in the preparation of the report titled, "NI 43-101 Technical Report, Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of April 28, 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 16th Day of October, 2015.

"Signed and Sealed"

Joanna Poeck, BEng Mining, SME-RM

U.S. Offices:		Mexico Office:		Canadian Offices:		Group Offices:
Anchorage	907.677.3520	Querétaro	52.442.218.1030	Saskatoon	306.955.4778	Africa
Denver	303.985.1333			Sudbury	705.682.3270	Asia
Elko	775.753.4151			Toronto	416.601.1445	Australia
Fort Collins	970.407.8302			Vancouver	604.681.4196	Europe
Reno	775.828.6800			Yellowknife	867.873.8670	North America
Tucson	520.544.3688					South America

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, Jeff Osborn, BEng Mining, MMSAQP do hereby certify that:

1. I am a Principal Consultant (Mining Engineer) of SRK Consulting (U.S.), Inc., Suite 600, 1125 Seventeenth Street, Denver, CO 80202.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of August 4, 2015 (the "Technical Report").
3. I graduated with a Bachelor of Science Mining Engineering degree from the Colorado School of Mines in 1986. I am a Qualified Professional (QP) Member of the Mining and Metallurgical Society of America. I have worked as a Mining Engineer for a total of 24 years since my graduation from university. My relevant experience includes responsibilities in operations, maintenance, engineering, management, and construction activities.
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 the QP responsible for mining and infrastructure Sections 2, 3, 16.7.3. 16.7.4, 16.7.6, 16.8.2 through 16.8.6), 18.1.2, 23, 24, 27, 28 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. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the report titled, "NI 43-101 Technical Report, Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of April 28, 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 16th Day of October, 2015.

"Signed and Sealed"

____

Jeff Osborn, BEng Mining, MMSAQP

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, do hereby certify that:

1. I am Principal Consultant, Geotechnical Engineer of SRK Consulting (U.S.), Inc., Suite 600, 1125 Seventeenth Street, Denver, CO 80202.
2. This certificate applies to the technical report titled “Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska” with an Effective Date of August 4, 2015 (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 the President 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 28 years since my graduation from university. My relevant experience includes 28 years of professional experience. I have 13 years managerial experience leading project teams, managing P&L operations for 25 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, tunneling, ground support, slope 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. I have geotechnical investigation and design experience in the oil and gas field including seismic design for offshore platforms, foundations, and gravel islands; civil design experience in tunneling, ground support; and underground nuclear waste disposal.
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 the QP responsible for geotechnical Section 16.2 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 was in the preparation of the report titled, “NI 43-101 Technical Report, Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska” with an Effective Date of April 28, 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 16th Day of October, 2015.

"Signed and Sealed"

John Tinucci, PhD, PE

SRK Consulting (U.S.), Inc.
5250 Neil Road, Suite 300
Reno, Nevada 89502

T: (775) 828-6800
F: (775) 828-6820

reno@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Clara Balasko, MSc, PE do hereby certify that:

1. I am Senior Consultant, Civil Engineer of SRK Consulting (U.S.), Inc., 5250 Neil Road, Suite 300, Reno, NV, USA, 89502.
2. This certificate applies to the technical report titled “Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska” with an Effective Date of August 4, 2015 (the “Technical Report”).
3. I graduated with a degree in Bachelors of Science in Geology from Texas A&M University in 2003. In addition, I have obtained a Master’s of Science in Geological Engineering from University of Nevada, Reno in 2003. I am a Professional Engineer in Civil Engineering of the Arizona Board of Technical Registration. I have worked as a Civil Engineer for a total of 12 years since my graduation from university. My relevant experience includes planning and conducting geotechnical investigations for tailings storage facility foundation and embankment design, design for construction, operation and closure of tailings storage facilities, calculating tailings storage facility water balances for operation and closure, and performing slope stability assessments.
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 November 10 to 15, 2014 and June 22 to 24, 2015.
6. I am the QP responsible for TSF Sections 5.4.1, 18.2, pipeline portion of 18.3, 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 was in the preparation of the report titled, “NI 43-101 Technical Report, Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska” with an Effective Date of April 28, 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 16th Day of October, 2015.

“Signed and Sealed”

____

Clara Balasko, MSc, PE

SRK Consulting (U.S.) Inc.
5250 Neil Road, Suite 300
Reno, Nevada 89502

T: (775) 828-6800
F: (775) 828-6820

reno@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Mark Willow, MSc, CEM, SME-RM do hereby certify that:

1. I am Principal Environmental Scientist of SRK Consulting (U.S.), Inc., 5250 Neil Road, Reno, Nevada 89511.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of August 4, 2015 (the "Technical Report").
3. I graduated with Bachelor's degree in Fisheries and Wildlife Management from the University of Missouri in 1987 and a Master's degree in Environmental Science and Engineering from the Colorado School of Mines in 1995. I have worked as Biologist/Environmental Scientist for a total of 22 years since my graduation from university. My relevant experience includes environmental due diligence/competent persons evaluations of developmental phase and operational phase mines through the world, including small gold mining projects in Panama, Senegal, Peru and Colombia; open pit and underground coal mines in Russia; several large copper mines and processing facilities in Mexico; and a mine/coking operation in China. My Project Manager experience includes several site characterization and mine closure projects. I work closely with the U.S. Forest Service and U.S. Bureau of Land Management on several permitting and mine closure projects to develop uniquely successful and cost effective closure alternatives for the abandoned mining operations. Finally, I draw upon this 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 under CERCLA activities according to current U.S. EPA risk assessment guidance. 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 have not visited the Elk Creek property.
6. I am the QP responsible for environmental studies, permitting and social or community impact Section 20 (except 20.3.8) 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 was in the preparation of the report titled, "NI 43-101 Technical Report, Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of April 28, 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 16th Day of October, 2015.

"Signed and Sealed"

Mark Willow, MSc, CEM, SME-RM

SRK Consulting (U.S.), Inc.
3275 West Ina Road, Suite 240
Tucson, AZ 85741

T: 520 544 3688
F: 520 544 9853

tucson@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Valerie Obie, BS Mining, MA, SME-RM, do hereby certify that:

1. I am Principal Mineral Economist of SRK Consulting (U.S.), Inc., 3275 West Ina Road, Suite 240 Tucson, AZ 85741.
2. This certificate applies to the technical report titled "Amended NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of August 4, 2015 (the "Technical Report").
3. I graduated with a degree in B.S. Mining Engineering from University of Arizona in1985. In addition, I have obtained a M.A. Organizational Management from the University of Phoenix in 1995. I am a Registered Member of the Society of Mining Engineers. I have worked as a Mining Engineer for a total of 30 years since my graduation from university. My relevant experience includes development of capital and production estimation and costing, depreciation, and development of cash flow and costing models. My experience also include open pit mine operations and planning for short to long term strategic and business plan since 1987 and have been involved in technical reports since 1995.
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 did not visit the Elk Creek property.
6. I am the QP responsible for market studies, capital and operating costs and economic analysis Sections 19, 21 and 22 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 was in the preparation of the report titled, "NI 43-101 Technical Report, Preliminary Economic Assessment, Elk Creek Niobium Project, Nebraska" with an Effective Date of April 28, 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 16th Day of October, 2015.

"Signed and Sealed"

____

Valerie Obie, BS Mining, MA, SME-RM