NI 43-101 Technical Report

Wicheeda Rare Earths Project PFS

Report date: **April 4, 2025**
Effective date: **February 28, 2025**

IMPORTANT NOTICE TO READER

This report was prepared by the qualified persons (QPs) listed in Table 2-1, as a National Instrument 43-101 Standards of Disclosure for Mineral Projects Technical Report. Each QP assumes responsibility for those sections or areas of this report that are referenced opposite their name in Table 2-1. None of the QPs, however, accepts any responsibility or liability for the sections or areas of this report that were prepared by other QPs. This report was prepared to allow Defense Metals Corporation (the “Owner”) to reach informed decisions respecting the development of the Wicheeda Project. Except for the purposes legislated under provincial securities law, any use of this report by any third party is at that party's sole risk, and none of the contributors shall have any liability to any third party for any such use for any reason whatsoever, including negligence. This report is intended to be read as a whole, and sections should not be read or relied upon out of context. This report contains estimates, projections and conclusions that are forward-looking information within the meaning of applicable securities laws. Forward-looking statements are based upon the responsible QP’s opinion at the time that they are made but in most cases involve significant risk and uncertainty. Although each of the responsible QPs has attempted to identify factors that could cause actual events or results to differ materially from those described in this report, there may be other factors that cause events or results to not be as anticipated, estimated or projected. None of the QPs undertake any obligation to update the forward-looking information. As permitted by Item 3 of Form 43-101F1, the QPs have, in the preparation of this report, relied upon certain reports, opinions and statements of certain experts. These reports, opinions and statements, the makers of each such report, opinion or statement and the extent of reliance are described in Section 3.0 of this report. Each of the QPs hereby disclaims liability for such reports, opinions and statement to the extent that they have been relied upon in the preparation of this report, as described in Section 3.0. None of the QPs undertake any obligation to update any information contained in this report, including, without limitation, any forward-looking information. Moreover, there can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. Accordingly, readers are cautioned not to place undue reliance on forward-looking statements. Unless required by securities laws, the authors undertake no obligation to update the forward-looking statements if circumstances or opinions should change.

Table of Contents

``

``

``

``

``

``

``

``

``

List of Tables

``

``

List of Figures

``

``

``

List of Attachments

``

1. Summary

1.1 Introduction

Defense Metals Corp (TSX-V:DEFN) retained Hatch Ltd and SRK Consulting (Canada) Inc. to conduct a preliminary feasibility study (PFS) on the Wicheeda rare earth element (REE) project in British Columbia (BC), Canada, and to present the outcomes in this National Instrument (NI) 43-101 independent technical report.

The PFS is based on an updated mineral resource which also is documented in this report.

Site visits for the purposes of personal inspections of the Wicheeda property have been undertaken by Mr. Douglas Reid, resource QP and Dr. Anoush Ebrahimi, mining QP, both with SRK (separate visits, Dr. Ebrahimi in October 2021 and Mr. Reid October 31 and November 1, 2024).

Note: Throughout this report, all currency is 2025, non-escalated United States dollars (USD) and all units are metric, unless otherwise specifically noted.

1.2 Property Description and Ownership

The Wicheeda Property is located in the Central Mining Division in central BC, approximately 80 km northeast of Prince George, BC and 50 km east of Bear Lake, BC.

The Wicheeda property is located at Wicheeda Lake, at the base of the Rocky Mountains, on the edge of the Central Interior Plateau, approximately 80 km northeast of Prince George and 50 km east of Bear Lake, BC. The property is comprised of 17 contiguous mineral claims, covering 11,800 ha within the Cariboo Mining Division. The claims are listed as 100%-owned by Defense Metals Corporation.

1.3 Geology and Mineralization

The Wicheeda deposit is a southeast-trending, north to northeast dipping composite layered syenite-carbonatite sill complex having dimensions of approximately 450 m north-south by 170-300 m east-west by 100-275 vertically . The carbonatite is intruded into syenite, mafic dikes, limestone and calcareous sedimentary wall rocks. The mineralization is interpreted as a moderately north-northeast dipping, shallowly north plunging, layered sill complex having low REE grade syenite at its base. It is overlain by transitional intermediate REE grade hybrid xenolithic carbonatite as well as relatively higher REE grade dolomite carbonatite rocks. The dolomite carbonatite zones form the main mineralization of the Wicheeda REE deposit, outcropping at surface.

1.4 Exploration Status

Defense Metals has not conducted any surface exploration on the Wicheeda project other than diamond drilling, obtaining high resolution LiDAR data and metallurgical testwork samples.

In 2008 and 2009, 19 holes totalling 2696 m were completed by Spectrum. These are discussed in Section 10.1.

``

During 2020 and 2021, all the 2008 and 2009 original drill core pulps were reanalyzed, utilizing a REE lithium metaborate fusion with an inductively coupled plasma mass spectrometry (ICP-MS) finish analytical method, to reduce the uncertainty regarding the historical incomplete X-ray fluorescence (XRF) analytical results.

Between 2019 and 2023, Defense Metals conducted diamond drilling exploration programs consisting of 66 diamond drill holes, totaling 14066 m further testing the extent of the mineralized carbonatite. All drill holes intersected variable lengths of significant REE mineralization.

The drill results have been used to support ongoing economic studies through the development of an updated geological model and MRE.

1.5 Metallurgical Testing

1.5.1 Flotation Processing

Between 2011 and 2024, metallurgical development programs were conducted using samples from the Wicheeda deposit to experimentally determine and optimize the beneficiation and hydrometallurgical flowsheets. Experimental testing was performed at both bench and pilot scales by SGS Canada Inc. at its Lakefield site. This work included flotation optimization test work for Defense Metals in 2019 and flotation pilot plant testing in 2020.

The work conducted for the Wicheeda project over the period 2011-2021 was presented in the SRK Report “Independent Preliminary Economic Assessment for the Wicheeda Rare Earth Element Project, British Columbia, Canada” Report Number 2CD031.000 January 2022 and is not repeated here.

From 2021 to 2024, SGS conducted comminution, metallurgical, and environmental test work using samples representing the main lithological domains of the Wicheeda deposit. In 2023, comminution test work included 18 samples: one master composite sample and 17 variability samples, such as the DC (Dolomite Carbonatite)-XEN1 (Xenolithic Carbonatite) composite. Tests included SMC and Bond ball mill grindability for all samples, with Bond rod mill grindability and Bond abrasion tests performed on the master composite sample only. Results indicated that Wicheeda ore is soft to medium soft and very amenable to semi-autogenous grinding (SAG) and ball mill grinding.

During this laboratory test campaign, a total of 86 flotation tests were completed, including 34 flowsheet development tests and 52 variability tests. Three flotation locked cycle tests (LCT) were conducted by SGS to identify the best circuit configuration for processing Wicheeda ore. LCT3, which used a larger charge size of 12 kg and had a feed grade of 2.83% total rare earth oxides (TREO), demonstrated better performance, with a second cleaner concentrate grade of 50.7% TREO at 85.4% recovery, making it the best circuit configuration for processing Wicheeda ore. It should be noted that rougher and rougher scavenger stages were conducted at 60°C and the cleaner stages at 75°C.

``

1.5.2 Hydrometallurgical Processing

Hydrometallurgical testing was conducted by SGS from 2021 to 2023 on concentrate produced in the concentrator pilot plant to develop a flowsheet incorporating acid baking, water leaching, impurity removal, solvent extraction, rare earth precipitation, and magnesium removal. This testing, which aimed to validate the various unit operations, investigate the effects of various operating parameters, and collect data required for process design, comprised various bench scale tests and two hydrometallurgical pilot plant campaigns. These campaigns operated for 107 h and 211 h, and processed 187 kg and 369 kg of concentrate, respectively.

Each of the main processing steps was tested, and the measured performance data was used as the basis for the process design criteria; for example:

1. Neodymium extractions of 91-94% were achieved at the optimal operating conditions,
2. Impurity (thorium, iron, phosphate) removals of 90% were achieved by Mg(OH)2 addition at approximately pH 2 and 100% Th and Fe removal was achieved at pH 6-6.5,
3. Ion exchange reduced uranium levels in the final product to less than 1 mg/kg,
4. Solvent extraction separation factors were generated,
5. ammonium bicarbonate achieved complete REE precipitation with minimal impurity entrainment,
6. 20% stoichiometric excess Ca(OH)2 addition achieved 100% magnesium precipitation from the recirculating liquor.

The experimental testing also allowed the identification of potential processing challenges to be assessed during future development work, such as materials handling challenges during acid mixing and acid baking. Furthermore, while all the main process steps were tested experimentally, some gaps in the pilot testing were identified, mainly the pilot-scale testing of process features added after the pilot campaigns were conducted. Additional pilot testing and equipment manufacturer tests are recommended to further validate and derisk the process and produce samples for potential off-take partners.

1.6 Mineral Resource Estimate

The Mineral Resource Estimate (MRE) was derived from the mineral resource block model presented in Section 14 of the report. The Mineral Resources stated below are constrained within an optimized pit shell to satisfy Reasonable Prospects of Eventual Economic Extraction (RPEEE) requirements. The Mineral Resources include 29.3 Mt of Measured + Indicated resource at an average grade of 2.27% TREO and 5.5 Mt of Inferred resource at an average grade of 1.40% TREO. No mining dilution has been incorporated into the Mineral Resources stated below. The Mineral Resources are stated inclusive of Mineral Reserves.

A summary of the surface mineable Mineral Resources by rock type and Resource classification is shown in Table 1-1.

``

Table 1-1: Summary of the Mineral Resources as of 28 February 2025

Mineral Resource	Rock Type	Ore	TREO	Pr6O11	Nd2O3	Tb4O7	Dy2O3
		kTonnes	%	ppm	ppm	ppm	ppm
Measured	Dolomite Carbonatite	5,350	2.99	1,161	3,158	12	35
	Xenolithic Carbonatite	300	1.64	662	1,950	11	36
	Syenite	50	1.40	560	1,631	11	40
	Limestone	10	1.96	837	2,310	13	41
	Total	5,720	2.90	1,128	3,079	12	35
Indicated	Dolomite Carbonatite	12,030	2.90	1,139	3,116	12	34
	Xenolithic Carbonatite	10,060	1.32	547	1,618	9	30
	Syenite	1,320	1.07	442	1,331	8	29
	Limestone	160	1.40	569	1,627	11	43
	Total	23,570	2.11	843	2,367	10	32
Measured + Indicated	Dolomite Carbonatite	17,380	2.93	1,145	3,129	12	34
	Xenolithic Carbonatite	10,360	1.33	550	1,628	9	30
	Syenite	1,370	1.08	447	1,343	8	29
	Limestone	170	1.44	588	1,675	11	43
	Total	29,290	2.27	899	2,506	11	33
Inferred	Dolomite Carbonatite	570	2.67	1,072	2,883	12	37
	Xenolithic Carbonatite	3,280	1.42	587	1,712	9	32
	Syenite	1,630	0.90	401	1,229	9	34
	Limestone	210	1.50	600	1,641	9	33
	Total	5,690	1.40	582	1,687	9	33

Source: SRK, 2025

Notes:

- CIM (2014) definitions were followed for Mineral Resources.
- The Qualified Person for the MRE is Doug Reid, P.Eng., EGBC (23347), an SRK employee.
- The effective date of the Mineral Resource is 28 February 2025
- Dollar values herein stated are United States Dollars (US$)
- Mineral Resources are calculated using the values derived from all REEs present in the deposit. Individual REO pricing provided by Adamas Intelligence was escalated by 15% and used for pit optimization. The key REO pricing is as follows:
  - NdPr Oxide 152.6 $/kg REO
  - Tb4O7 1567.3 $/kg REO
  - Dy2O3 508.8 $/kg REO
- Mineral Resources are defined within a pit shell derived from the optimization software, GEOVIA Whittle™
- Cut-off grade is based on the value factors generated in each block. The revenue and related costs vary based on the composition of different elements in each block. The value of a block is the revenue generated in that block minus the related processing and G&A operating costs.

``

- The base mining costs are assumed to be $4.50/t. The mining costs vary based by the bench and depth of the pit. The average mining costs for the life of mine is calculated to be $4.74/t mined.
- Processing costs consist of flotation concentrator plant cost at the mine site and a hydrometallurgical/solvent extraction plant that is off the mine property. The operating cost of the flotation concentrator plant is $27.60/t milled and the hydrometallurgical plant operating cost is $1,164.4/t of concentrate treated.
- General and administration costs of the mine site is $3.67/t for ore milled.
- Tailings management and storage cost is $6.55/t of ore.
- Off-site cost (transportation) is $87.76/t of precipitate products produced.
- Processing recoveries are calculated as follows:
    - Flotation recovery for TREO = -11.183*TREO^2 + 67.831*TREO - 20.42194.0%. For ore above 3% TREO, the flotation recovery is set to 82.4%. For grade less than 0.32% TREO, the flotation recovery is set to 0.0%.
    - Flotation recovery for each REE is calculated by multiplying the TREO recovery by that element's recovery factor. For example, the factors for Pr, Nd, Tb, and Dy are 0.995, 0.996, 0.734, and 0.636, respectively.
    - Similarly, hydrometallurgical recoveries are assigned for each REE, which for Pr, Nd, Tb, and Dy are 93.2%, 93.5%, 80.2%, and 73.4%, respectively.
- A 95% payability has been applied to the final hydrometallurgical product.
- Bulk density is assigned by lithology.
- No mining dilution has been applied.
- Mineral Resources are reported inclusive of those Mineral Resources converted to Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Figures are rounded to the appropriate level of precision for the reporting of mineral Resources. Due to rounding, some columns or rows may not sum as shown.
- The TREO grade encompasses 15 rare earth elements present in the deposit.
- The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

The QP, Douglas Reid, does not know of any legal, political, environmental, or other risks that could materially affect the potential development of the mineral Resources. Mr. Reid personally inspected the subject Wicheeda Project on October 31 and November 1, 2024.

1.7 Mining Methods

The Wicheeda deposit is to be developed as an open pit mining operation. The pit is to be developed in two phases, with Phase 1 targeting at-surface higher grade mineralization. Mining rates over the 15-year life-of-mine (excluding pre-production) vary from 5 to 10 Mtpa to maintain a 1.8 Mtpa mill feed rate to the flotation concentrator.

Ore is delivered to a pad near the pit where it is crushed and then conveyed to the flotation concentrator. Waste rock from the open pit is disposed of in a waste storage facility immediately south of the pit.

The mineral reserve estimate for the Wicheeda Rare Earth Element Project has been prepared as part of the 2025 Pre-feasibility Study (PFS) in accordance with the CIM Definition Standards adopted May 2014.

The mineral reserves were derived from the mineral resource block model presented in Section 14 of the report. The mineral reserves respective of the open pit are based on Measured and Indicated mineral resources that have been identified as being economically extractable and which incorporate mining losses and mining waste dilution. The mineral reserves include 26.3 Mt of mineable ore from one open pit at an average grade of 2.37% TREO. The mineral reserve includes variable mining dilution, and it is calculated after 1% ore loss.

A summary of the surface mineable mineral reserves by rock type and reserve classification is shown in Table 15-9.

Table 1-2: Summary of the Mineral Reserves as of February 28, 2025

Mineral Reserve	Rock Type	Ore	TREO	Pr6O11	Nd2O3	Tb4O7	Dy2O3
		kTonnes	%	ppm	ppm	ppm	ppm
Proven	**Dolomite Carbonatite**	5,300	2.96	1,152	3,134	12	35
	Xenolithic Carbonatite	260	1.71	690	2,031	11	37
	Syenite	40	1.44	574	1,658	11	39
	Limestone	10	2.01	858	2,359	12	40
	Total	**5,610**	**2.89**	**1,125**	**3,070**	**12**	**35**
Probable	**Dolomite Carbonatite**	12,020	2.86	1,120	3,067	12	34
	Xenolithic Carbonatite	7,810	1.38	569	1,674	9	29
	Syenite	760	1.20	482	1,427	8	26
	Limestone	140	1.38	558	1,585	10	38
	Total	**20,730**	**2.23**	**886**	**2,472**	**10**	**32**
Total	**Dolomite Carbonatite**	17,320	2.89	1,130	3,087	12	34
	Xenolithic Carbonatite	8,070	1.39	573	1,686	9	29
	Syenite	800	1.21	487	1,439	8	27
	Limestone	150	1.42	579	1,639	10	38
	Total	**26,340**	**2.37**	**937**	**2,600**	**11**	**32**

Source: SRK, 2025

Notes:

- The effective date of the Mineral Reserve is February 28, 2025.
- Mineral Reserves are calculated using the values derived from all REEs present in the deposit. Individual REO pricing provided by Adamas Intelligence was used for pit optimization. The key REO pricing is as follows:
  - NdPr Oxide 132.7 $/kg REO
  - Tb4O7 1362.8 $/kg REO
  - Dy2O3 442.5 $/kg REO
- Mineral Reserves are defined within the final pit design guided by pit shells derived from the optimization software, GEOVIA Whittle™

- Cut-off grade is based on the value factors generated in each block. The revenue and related costs vary based on the composition of different elements in each block. Value of a block is the revenue generated in that block minus the related processing and G&A operating costs.
- The base mining costs are assumed to be $5.00/t. The mining costs vary based by the bench and depth of the pit. The average mining costs for the life of mine is calculated to be $5.26/t mined.
- Processing costs consist of flotation concentrator cost at the mine site and a hydrometallurgical plant that is off the property. The operating cost of the flotation concentrator is $27.60/t milled and the hydrometallurgical plant operating cost is $1,164.4/t of concentrate treated.
- General and administration costs of the mine site is $3.67/t for ore milled.
- Tailings cost is $6.55/t of ore.
- Off-site cost (transportation) is $87.76/t of precipitate products produced.
- Processing recoveries are calculated as follows:
    - Flotation recovery for TREO = -11.183*TREO^2 + 67.831*TREO - 20.42194.0%. For ore above 3% TREO, the flotation recovery is set to 82.4%. For grade less than 0.32% TREO, the flotation recovery is set to 0.0%.
    - Flotation recovery for each REE is calculated by multiplying the TREO recovery by that element’s recovery factor. For example, the factors for Pr, Nd, Tb, and Dy are 0.995, 0.996, 0.734, and 0.636, respectively.
    - Similarly, hydrometallurgical recoveries are assigned for each REE, which for Pr, Nd, Tb, and Dy are 93.2%, 93.5%, 80.2%, and 73.4%, respectively.
- A 95% payability has been applied to the final hydrometallurgical product.
- Mining dilution varies based on the mining zone. The average mining dilution is calculated to be 2.9%, for the ore delivered to the mill. Tonnages reported as ore includes dilution.
- A 1% ore loss has been applied to the total reserve in each bench.
- Figures are rounded to the appropriate level of precision for the reporting of mineral reserves. Due to rounding, some columns or rows may not sum as shown.
- The overall strip ratio (the amount of waste mined for each tonne of ore) is 3.21 (W:O).
- The mineral reserve is stated as diluted dry metric tonnes.
- The mine plan underpinning the mineral reserves has been prepared by SRK Consulting (Canada) Inc.

The QP, Dr. Anoush Ebrahimi, does not know of any legal, political, environmental, or other risks that could materially affect the potential development of the mineral reserves. He personally inspected the subject project on October 26, 2021.

1.8 Recovery Methods

Material from the Wicheeda rare earth deposit is processed in a flotation concentrator to produce a rare earth flotation concentrate containing 50% TREO. The flotation concentrate is sent to a hydrometallurgical plant to produce a mixed RE carbonate precipitate with a REE distribution of 87 wt% NdPrO.

``

1.8.1 Flotation concentrator

The beneficiation plant is designed to process 1,800,000 tonnes of ore per year based on the production capacity developed for the proposed life of mine operation by SRK to produce a rare earth flotation concentrate containing 50% Total Rare Earth Oxide (TREO). The concentrator flowsheet will include primary crusher circuit, , SAG / Ball (SAB) mill grinding circuit, rougher and scavenger flotation cells and two stages of cleaner flotation at elevated temperature. The flotation concentrate will be thickened to about 70 wt.% solids and then filtered to about 8wt. % moisture. The filtered concentrate will then be loaded into half height containers and shipped by truck to the hydrometallurgy facility for further processing.

The final flotation tails will be thickened to about 60 wt.% solids and then filtered to about 9 wt.% moisture. Filter cake is transported using trucks to the filtered tailings storage facility for placement and compaction.

1.8.2 Hydrometallurgical Plant

The hydrometallurgical plant processes the concentrate to extract and purify the rare earth content, producing a mixed rare earth carbonate product suitable for further downstream separation.

This processing route employs sulphuric acid mixing/baking to convert the REE-bearing minerals into soluble sulphate species and water leaching to extract the rare earths from the concentrate. The baking off-gas, containing sulphuric acid and hydrofluoric acid fumes, is treated with lime slurry in a multi-step off-gas treatment system to prevent harmful gas emissions.

Multiple impurity removal precipitation steps, using magnesium hydroxide slurry, are employed to remove thorium, iron and phosphates. The leaching and neutralization solid residues are returned to the mine site to be combined with concentrator tailings, and the precipitation liquor is treated by ion exchange to remove uranium remaining in solution.

The purified leach liquor is subjected to a solvent extraction separation step to separate lanthanum and cerium from the remaining REEs, producing a REE stream that is enriched in high-value neodymium and praseodymium. The mixed REE stream is precipitated with ammonium bicarbonate to produce a mixed rare earth carbonate (MREC) product, which is dewatered, dried, then packaged. The remaining ammonium sulphate solution is treated with lime slurry to evaporate and recover ammonia gas. Ammonium bicarbonate for REE precipitation is regenerated via absorption from the recovered ammonia gas, make-up ammonia solution, and CO2 recovered from the REE precipitation off-gas and natural gas combustion exhaust.

The solvent extraction raffinate, containing primarily magnesium sulphate with lanthanum and cerium, is treated with lime slurry to precipitate the dissolved magnesium, lanthanum, and cerium to produce a hydrometallurgical waste precipitate and a gypsum-saturated liquor. The gypsum liquor is recirculated to the water leaching step of the process and used for lime and magnesium hydroxide slaking.

The excess liquor is directed to the hydrometallurgical effluent treatment plant for water recovery and waste disposal. The waste solids are directed to the hydrometallurgical waste storage facility for disposal.

``

The hydrometallurgical plant achieves 93% recovery of neodymium and praseodymium and produces a mixed rare earth carbonate product with 72 wt% TREO (dry basis), with a REE distribution of 87 wt% NdPrO.

1.9 Project Infrastructure

The Wicheeda Rare Earths project consists of two project sites: a mine site with a concentrator, and a separate hydrometallurgical plant site.

1.9.1 Mine Site Infrastructure

Mine site infrastructure includes all the required facilities for a remote site, including camp accommodation, first aid/emergency response, warehousing, maintenance facilities, explosives storage, etc.

1.9.2 Mine Site Tailings Management

Tailings generated in the flotation concentrator are transported via pipeline to a tailings filtration plant and dewatered to a filter cake before being hauled, placed and compacted in a lined Filtered Tailings Storage Facility (FTSF). The filtration plant is located adjacent to the FTSF to minimize haul distances, and both are located west of Wichcika Creek – the side opposite other project operations (pit, concentrator, etc.). The WL/PN residue and loaded uranium ion exchange resin produced at the hydrometallurgical plant are also stored in the mine site FTSF. Construction materials for drains, a starter toe embankment, water management pond and rock cladding on the slopes of the stack will be sourced from a local quarry and borrow areas.

1.9.3 Mine Site Water Management

The Wicheeda project includes a water management strategy that separates contact and non-contact water and provides sufficient storage for water treatment. Contact water will include runoff and groundwater inflows to the open pit, runoff and seepage from the waste rock storage area, and runoff collected in the FTSF water management pond. These inflows will be directed via channels or pumped to a central Contact Water Pond (CWP) located adjacent to the process plant. The PFS assumes all contact water will require water treatment. Non-contact water management will include diversion channels around the FTSF and waste rock storage area (WSA) to reduce the volume of contact water to be treated. The main parameters of concern for water treatment are molybdenum and uranium. The water treatment process for removal of molybdenum and uranium is ferric co-precipitation. The process works by adding a ferric sulphate ($Fe_2(SO4)_3$) to the CWP water in an agitated reactor tank.

1.9.4 Hydrometallurgical Site Infrastructure

The hydrometallurgical facility is a standalone industrial facility planned to be located in Bear Lake approximately 70 km North of Prince George just off Highway 97. Bear Lake has highway and rail access, access to the Chuckinka Forest Service Road (FSR), and is near both hydro-electrical power and natural gas supplies. This study assumes that an existing partially developed heavy industrial site will be available, but a specific site has not been selected. Obtaining access to a specific site will be investigated before and during the next phase of the project.

1.9.5 Hydrometallurgical Site Tailings Management

The WL/PN residue and loaded uranium ion exchange resin are dewatered via filtration at the hydrometallurgical plant and hauled by truck to the mine site FTSF before blending with the dewatered flotation tailings. Other wastes generated at the hydrometallurgical plant, predominantly gypsum materials, will be dewatered via filtration and stored in a lined facility adjacent to the plant. A physical location for the Hydrometallurgical Waste Storage Facility (HWSF) has not yet been identified but is assumed to be located adjacent to the hydrometallurgical plant at Bear Lake, approximately 50 km west of the mine site.

1.9.6 Hydrometallurgical Site Water Management

Runoff water as well as any potential seepage from the HWSF is collected by a full perimeter channel and directed to a lined water management pond. A diversion berm is also included on the downstream side of the channel to prevent any external non-contact water from entering the system. All contact water and filtrate is treated prior to reuse or discharge. Given topography and hydrological conditions are unknown, no other surface water management plans are currently proposed for the hydrometallurgical Site.

1.9.7 Off-Site Infrastructure

Off-site infrastructure includes:

Power, for both sites, supplied to the mine site via a new high-voltage overhead power line connecting to the BC Hydro 138 kV line (1L 365) west of the mine site, near Bear Lake.

The mine site access road utilizes the Chuchinka FSR for almost the entire distance. The FSR is upgraded as required and the bridge nearest the mine site repaired.

1.10 Environmental and Permitting

The Wicheeda project is in an ecologically rich area that supports diverse fish and wildlife populations. While the immediate project area lacks permanent settlements, it holds significant cultural and traditional value for the Indigenous peoples and supports the cultural, recreational, conservation and economical functions of the larger region.

Baseline studies for surface water quality, hydrology, groundwater, and meteorology and climate commenced in 2020 and were paused in Q4 2024 to allow for redesign of the monitoring network for the evolving mine infrastructure footprint. Meteorological data collection has been ongoing, and the water quality program will recommence in Q2, 2025. Fish and aquatic resources field studies began during the 2024 field season with additional work planned for 2025 and 2026. Planning for wildlife and wildlife habitat, soils and vegetation baseline studies is underway.

The Wicheeda project meets the current thresholds for both a federal and provincial environmental assessment and will need to complete an environmental assessment prior to receiving approvals under the provincial Mines Act and Environmental Management Act. The assessment process will be a coordinated effort between the provincial and federal governments with the goal of reducing duplication of review and effort. A renewed government commitment to streamline major project approvals and eliminate environmental assessment process duplication was announced in March 2025.

``

Defense Metals entered into a Co-Design agreement with McLeod Lake Indian Band (MLIB) that emphasizes a joint planning and decision-making process for the technical, social, engineering, environmental and regulatory aspects of the Wicheeda project.

The project is a recognized critical mineral project in British Columbia and will benefit from the BC Critical Mineral strategy. Through the strategy, the provincial government has committed to the provision of "concierge-like" service to Defense Metals supported by the provincial Critical Mineral Office. This support includes assistance with navigating regulatory processes and pursuing funding opportunities. This designation and the Co-Design agreement with MLIB are anticipated to accelerate the project’s progression and de-risk key decisions.

The project is within a region where the economic activity is heavily influenced by natural resources. Forestry, mining and mineral exploration, agriculture and hydroelectric power are active sectors, and the workforce is experienced in industrial operations and the surrounding region is well positioned to support the development of the project. Refined socio-economic baseline studies to support environmental assessment processes will commence in 2026.

1.11 Capital and Operating Costs

Capital costs for the project, including the mine, both processing plants, waste management facilities, and closure costs were estimated by combining unit rates for equipment, materials, labour, and subcontracts with unit quantities from various engineering deliverables. The unit costs were based a combination of vendor budget quotes, and reference costs from similar projects, factored costs and allowances. The total initial project capital cost is $1,441M, with the sustaining capital, closure costs, and post closure costs bringing the total to $2,007M. The capital costs are summarized in Table 1-3.

Table 1-3: Summary of capital costs

Area	Capital costs (US$M)				
	Initial	Sustaining	Closure	Post closure	Total
Mining	96.88	55.6	-	-	157.79
Concentrator	450.86	-	-	-	450.86
Hydrometallurgical	614.49	-	-	-	614.49
Mine Tailings	19.84	45.67	-	-	65.51
Hydrometallurgical Waste	10.66	14.04	-	-	24.70
Contact Water Pond	11.76	-	-	-	11.76
Mine Site Water Management	1.59	-	-	-	1.59
Mine Site Water Treatment	10.03	-	-	-	10.03
Hydrometallurgical Water Treatment	6.61				6.61
Closure	-	-	57.35	325.12	382.47
Contingency	217.85	15.30	7.17	40.64	280.96
Total	**1,441**	**136**	**65**	**366**	**2,007**

Source: Hatch, 2025

``

Note: Cost estimates do not consider cost escalation resulting from the imposition of new tariffs, counter-tariffs, import and/or export duties or other similar charges applicable to raw, semi-finished or finished materials and/or other products.

Cost Basis: Q1 2025; Exchange Rate: 1.40 CAD = 1.00 USD

The initial capital costs are spread over a three-year construction period. The earliest that construction is expected to start is Q1 2030 due to environmental permitting timelines. This would enable the concentrator and hydrometallurgical process plants to start up Q1 2033.

The operating costs for mining, the Concentrator plant, the hydrometallurgical plant, and the waste management were estimated using unit consumptions from various engineering deliverables, and recent unit prices for reagents, energy, consumables, and labour, along with cost factor for maintenance, and allowances based on similar projects. The costs are summarized in Table 1-4. Some costs were fixed over the project life while others are variable due to changes in the mine production. The total life of mine operating cost is estimated at $2,566M, which corresponds to an equivalent unit production cost of $38.42/kg$_{\text{NdPrO}}$.

Table 1-4: Operating Costs Summary

Operating Costs	LOM ($M)	LOM avg ($M/y)	LOM ($/kg NdPrO equivalent in MREC)
Mining	552	36.8	8.27
Concentrator	744	49.6	11.14
Hydrometallurgical Plant	999	66.6	14.95
Mine Site & Hydrometallurgical Plant G&A	135	9.0	2.03
Mine Site Tailings	94	6.3	1.41
Hydrometallurgical Waste	20	1.4	0.31
Contact Water Pond	2	0.2	0.04
Mine Site Water Treatment	11	0.7	0.16
Hydrometallurgical Plant Water Treatment	8	0.5	0.12
Total	**2,566**	**171**	**38.42**

Source: Hatch, 2025

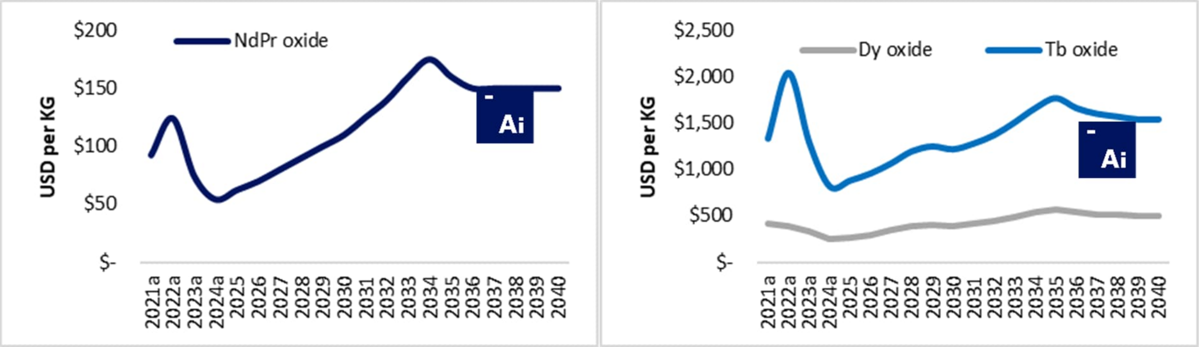
1.12 Economic Analysis

As per its latest “Rare Earth Pricing Quarterly Outlook” report (Q1 2025), Adamas Intelligence forecasted annual average prices for each rare earth oxide to 2040 under three scenarios. In the Base Case, Adamas expects the price of NdPr oxide to increase from US$55-60/kg last year to US$70-110/kg in the late-2020s. In a rational market, Adamas would expect these price increases to induce investment in new production capacity, however, owing to the long lead times to develop new rare earth supplies, Adamas sees potential for pervasive deficits to push prices above required inducement levels (estimated at US$100-150/kg) in the long term.

In Adamas’ view, the mixed rare earth carbonate (“MREC”) that Defense Metals plans to produce from Wicheeda would be marketable and desirable to existing and emerging rare earth separation facilities globally. Since the MREC is almost fully devoid of low value

``

lanthanum (“La”) and cerium (“Ce”), which typically comprise 50% to 70% of the rare earth contents in a standard MREC, a prospective processor of Wicheeda concentrate would not need to tie up capacity or expend costs to treat La and Ce making the Wicheeda concentrate a premium product. Inferring from Chinese processing costs, Adamas believes that from 2032 through 2040 Defense Metals could expect to receive a price for its MREC equal to 95% of the rare earth oxide value it contains (value based on China domestic prices, excluding VAT).



Source**:** Adamas Intelligence, 2025

* Forecasted prices are in Real 2025 US dollars and include 13% VAT; If selling into China, VAT should be deducted; if selling ex-China prices above should be taken at face value.

Figure 1-1: Forecasted magnet rare earth oxide prices to 2040 (Base Case)

The base case project economic model assumes long term prices of NdPr oxide $132.7/kg, Tb oxide $1,362.8/kg, and Dy oxide $442.5/kg (based on the Adamas prices less 13% VAT for sales outside China) which result in a basket value of $121.9/kg TREO after including the balance of REOs. The base case also assumes a 95% payability for the TREO in MREC which gives an average realized price of $115.8/kg TREO equivalent. At these prices the project achieves a positive NPV at an 8% real discount rate. A summary of key indicators is shown in Table 1-5.

The positive results of the economic analysis over a reasonably wide range of assumptions support the Wicheeda project proceeding to the next project development stage

Table 1-5: Key Indicators Summary

Financial Metrics	Units	Value
Pre-tax NPV @ 8%	US$M	1,746
After-tax NPV @ 8%	US$M	957
Pre-tax IRR	%	24.2%
After-tax IRR	%	18.6%
Pre-tax payback period from start of production	Years	3.2
After-tax payback period from start of production	Years	3.7
Initial capital expenditure	US$M	1,441
Average annual operating cost	US$M per annum	171.1
Average annual operating cost	US$/kg NdPrO equivalent in MREC	38.4
NdPrO price	US$/kg	132.70
MREC realized price	US$/kg TREO equivalent	115.8
Gross revenue (LOM)	US$M	9,062
EBITDA margin	%	71

Source: Hatch, 2025

The economic analysis is based on a discounted cash flow model in real terms. The model includes the 15-year mine production plan, operating costs, capital costs, and market assumptions discussed in this report, in addition to financial assumptions described in Section 22.

Returns are sensitive to input assumptions and should be viewed in the context of the sensitivity analysis provided Section 22 as well as the stated accuracies for items such as capital costs.

Product prices and payabilities could be materially different than modeled. China can influence the market through government intervention which can drastically affect prices and the Wicheeda product payability is preliminary and not yet supported by any customer agreements or detailed marketing studies. There is also uncertainty around the future structure and function of the markets for mixed rare earth carbonate given ongoing global trade disputes. There is no definitive timeline for project development and it is likely that the structure of the market will continue to evolve during the next project development phase.

1.13 Recommendations

Table 1-6 provides a summary of recommended investigations and their respective costs estimates to advance the Wicheeda project to the next stage feasibility study, as described in Section 26.

Table 1-6: Estimated Cost for Proposed Recommendations

Description	USD ($M)
Exploration drilling	1.5 - 2.0
Open pit rock mass geotechnical characterization and FS slope design and stability analysis	0.95 - 1.05
Feasibility study – resource and mining	2.0 - 2.5
Waste rock storage geotechnical studies	0.25 - 0.35
Waste material geochemistry studies	0.55
Tailings management studies	0.5 - 1.0
Water management / treatment studies	0.3 - 0.5
Concentrator and hydrometallurgical plant bench and pilot testing programs	5.0
Processing and infrastructure bridging studies	0.25
Feasibility study - processing and infrastructure	12.0
Subtotal	23.3 – 25.2
10% Contingency	2.3 - 2.5
TOTAL	25.6 – 27.7

Source: Hatch, 2025

The QPs are unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform the exploration work recommended for the Wicheeda project.

2. Introduction

This Preliminary Feasibility Study (PFS) report has been prepared by Hatch and SRK as an update to a previous PEA prepared by SRK in December 2021. The principal reasons for this PFS are to include the results of additional drilling, resource modelling, and metallurgical testwork; reporting a mineral reserve; updating the process plants design; updating the infrastructure design; and updating the costs, revenues, and economics for 2025.

The following independent consultants have contributed to this report:

- Hatch
- SRK.

Site visits are shown in Table 2-1.

A list of the qualified persons (QPs) responsible for each section of this report is provided in Table 2-1, and their QP certificates are appended to the back of this report.

All the qualified persons listed in Table 2-1 are independent of Defense Metals.

Table 2-1: Persons Who Prepared or Contributed to this Technical Report

Qualified Person	Employer	Date of Site Visit	Sections of Report
Doug Reid, P.Eng.	SRK	October 31 - November 1, 2024	1.3, 1.4, 1.6, 2.7, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.1, 25.2, 26.1
Anoush Ebrahimi, P.Eng.	SRK	October 9-10, 2021	1.7, 2.7, 15 (except for 15.2), 16, 25.3.2, 25.3.3, 26.2.2, 26.2.3
Bob McCarthy, P.Eng.	SRK	N/A	1.1, 1.2, 2, 4.1, 4.2, 5, 21.1.1, 21.2.1
Andy Thomas, P.Eng.	SRK	June 17-18, 2022	15.2, 25.3.1, 26.2.1
Ignacio Garcia, P.Eng.	SRK	N/A	1.9.2, 1.9.5, 18.4 (except 18.4.4, 18.4.6 and 18.4.11), 18.6.8, 21.1.4, 21.2.6, 25.4.2, 25.4.3, 26.3.2
Mauricio Herrera, P.Eng.	SRK	N/A	1.9.3, 1.9.6, 18.4.6, 18.5 (except 18.5.6), 21.1.5, 21.2.7, 25.4.4, 25.4.5, 25.5, 26.4
Soren Jensen, P.Eng.	SRK	N/A	1.9.3 (treatment), 18.5.6, 18.6.10, 21.1.5 (treatment), 21.1.6, 21.2.7 (treatment), 21.2.8, 25.4.4 (treatment), 25.4.4 (treatment) 25.5 (treatment), 26.4 (treatment)
Kirsty Ketchum, P.Geo.	SRK	N/A	18.3.2, 18.4.4, 18.6.9, 20.1.1, 20.2.2, 25.4.1, 26.3.1
Megan Miller, P.Eng.	SRK	N/A	18.3.3, 18.6.11, 21.3
Christina James, P.Eng.	SRK	N/A	1.10, 4.3, 4.4, 20 (except 20.1.1 and 20.2.2), 25.10, 26.8
Giuseppe (Joe) Paventi, P.Eng.	Hatch	N/A	1.5.1, 1.8.1, 11.5, 13.1, 13.2, 13.3, 17.1, 21.2.2, 25.6.1, 25.7.1, 26.5.1, 26.6.1
Jeff Adams, P.Eng.	Hatch	N/A	1.5.2, 1.8.2, 13.1, 13.2, 13.4, 17.2, 21.2.3, 25.6.2, 25.7.2, 26.5.2, 26.6.2
Gerald (Gerry) Schwab, P.Eng.	Hatch	N/A	1.1, 1.9.1, 1.9.4, 1.9.7, 1.11, 1.13, 2 (except 2.7), 18.1, 18.2, 18.4.11, 18.6 (except 18.6.9-12), 21.1.2, 21.1.3, 21.2.4, 21.2.5, 24, 25.8, 25.9, 26.7, 26.10
Stefan Hlouschko, P.Eng.	Hatch	N/A	1.12, 19, 22, 25.11, 26.9

Qualified Person	Employer	Date of Site Visit	Sections of Report

Source: SRK and Hatch, 2025

2.1 Issuer

The Wicheeda project is a REE mineral project, located in British Columbia, Canada. It is located approximately 80 km northeast of Prince George and 50 km east of Bear Lake. Defense Metals Corp. (Defense Metals), a publicly listed mining company trading under the ticker “DEFN” on the TSX Venture Exchange and headquartered in Vancouver, BC, is the proponent of the project.

2.2 Terms of Reference

In Q3 2023, Defense Metals commissioned Hatch Ltd (Hatch) and SRK Consulting (Canada) Inc. (SRK) to conduct a preliminary feasibility study (PFS) for the Wicheeda project. The services were rendered between then and April 2025 leading to the preparation of the PFS reported herein, which was disclosed publicly by Defense Metals in a news release on February 18, 2025.

This technical report documents an updated mineral resource statement for the Wicheeda project in addition to the results of the PFS. The report was prepared following the guidelines of the Canadian Securities Administrators’ National Instrument 43-101 and Form 43-101F1. The mineral resource statement reported herein was prepared in conformity with generally accepted CIM “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.”

This technical report summarizes the technical information available on the Wicheeda project and demonstrates that the Wicheeda project clearly has merit warranting additional exploration expenditures and technical development.

2.3 Work Program

The mineral resource for Wicheeda was updated in advance of the PFS. The mineral resource statement reported herein is a collaborative effort between APEX Geoscience Ltd. (APEX) and SRK personnel. SRK’s qualified person (QP), Mr. Reid, assumes responsibility for the mineral resources estimate.

In order to select a development strategy for the Wicheeda project, Hatch led an Optimization Study in Q3 2024. The subsequent PFS refined the mine plan, process design, infrastructure, and costing of the go-forward scenario. This work was completed in October 2024 to February 2025.

The technical report was assembled in Vancouver during the months of February and March 2025.

2.4 Sources of Information

This report is based on information collected during three site visits by SRK QPs (Section 2.7) and on additional information provided by Defense Metals throughout the course of the pre-feasibility study work. Other information was obtained from APEX, (geology, exploration, resource estimation), SGS (metallurgical testing), Adamas Intelligence (price forecasts), and the public domain. The QPs of SRK and Hatch have no reason to doubt the reliability of the information provided by Defense Metals.

2.5 Units of Measure

With respect to units of measure, unless otherwise stated, this Report uses:

- Abbreviated shorthand consistent with the International System of Units (International Bureau of Weights and Measures, 2006)
- ‘Bulk’ weight presented in metric tonnes (“tonnes” or “t”; 1,000 kg)
- Geographic coordinates are projected in the Universal Transverse Mercator (UTM) system relative to Zone 10 of the North American Datum 1983 (NAD 83)
- Currency in US dollars (US$), unless otherwise specified.

2.6 Acronyms

The acronyms and abbreviations used in this report are listed in Section 28.

2.7 Site Visit

In accordance with National Instrument 43-101 guidelines, Douglas Reid (mineral resources QP), Anoush Ebrahimi (mining QP), and Andy Thomas (pit geotechnical QP) visited the Wicheeda property separately on October 31 to November 1, 2024; October 26-27, 2021; and June 17-18, 2022, respectively. They were each accompanied by APEX employee, Mo Asmail.

The purpose of the site visits was to review the regional setting, terrain, drill hole core, geological quality control and quality assurance (QA/QC) protocols and verify collar locations. There was no active drilling during the 2024 site visit.

The SRK QPs were given full access to relevant data, and they conducted interviews with Defense Metals and APEX personnel to obtain information on the past exploration work, to understand procedures used to collect, record, store and analyze historical and current exploration data.

3. Reliance on Other Experts

This report is based on information provided by Defense Metals and other specialists throughout the course of the study. The qualified persons have taken reasonable measures to confirm information provided by others and have taken responsibility for the information.

The following specialists, who are not qualified persons for the purposes of this report, were relied upon for specific advice:

Title for the Wicheeda project was confirmed by independently reviewing the digital tenure records listed on the Province of British Columbia's "Mineral Titles Online" website (https://mtonline.gove.bc.ca). As of October 23, 2024, the seventeen mineral claims comprising the Property were shown to be active, in good standing and owned 100% by Defense Metals Corp. This information is relied upon in the ownership description in Section 4.1 of the Report.

The Hatch marketing QP has relied on a REE price forecast by Adamas Intelligence that was specific to the potential products at Wicheeda. This is discussed in Section 19.

Hatch was informed by Defense Metals that there are no known litigations potentially affecting the Wicheeda project.

John Goode / Mike Nees provided the solvent extraction process design parameters. The Hatch process engineer and the QP for recovery have reviewed the design used for Section 17 and judged that it is appropriate for the service.

The qualified persons responsible for these sections used their experience to determine if the information from the specialists was accurate.

4. Property Description and Location

4.1 Description and Location

The Wicheeda property is located at Wicheeda Lake, at the base of the Rocky Mountains, on the edge of the Central Interior Plateau, approximately 80 km northeast of Prince George and 50 km east of Bear Lake, BC (Figure 4.1). The property is situated within the 1:20,000 scale British Columbia Geological Survey (“BCGS”) map sheets 93J08, 93J050, 93J059, 93J060, 93I05 and is centred at approximately latitude 54° 31’ 48” N and longitude 122° 05’ 12” W. The mineral claims cover Wicheeda Lake and straddle a segment of Wichcika Creek. The principal area of interest, the Wicheeda Carbonatite, is centred between Wicheeda Lake and the Wichcika gravel pit.

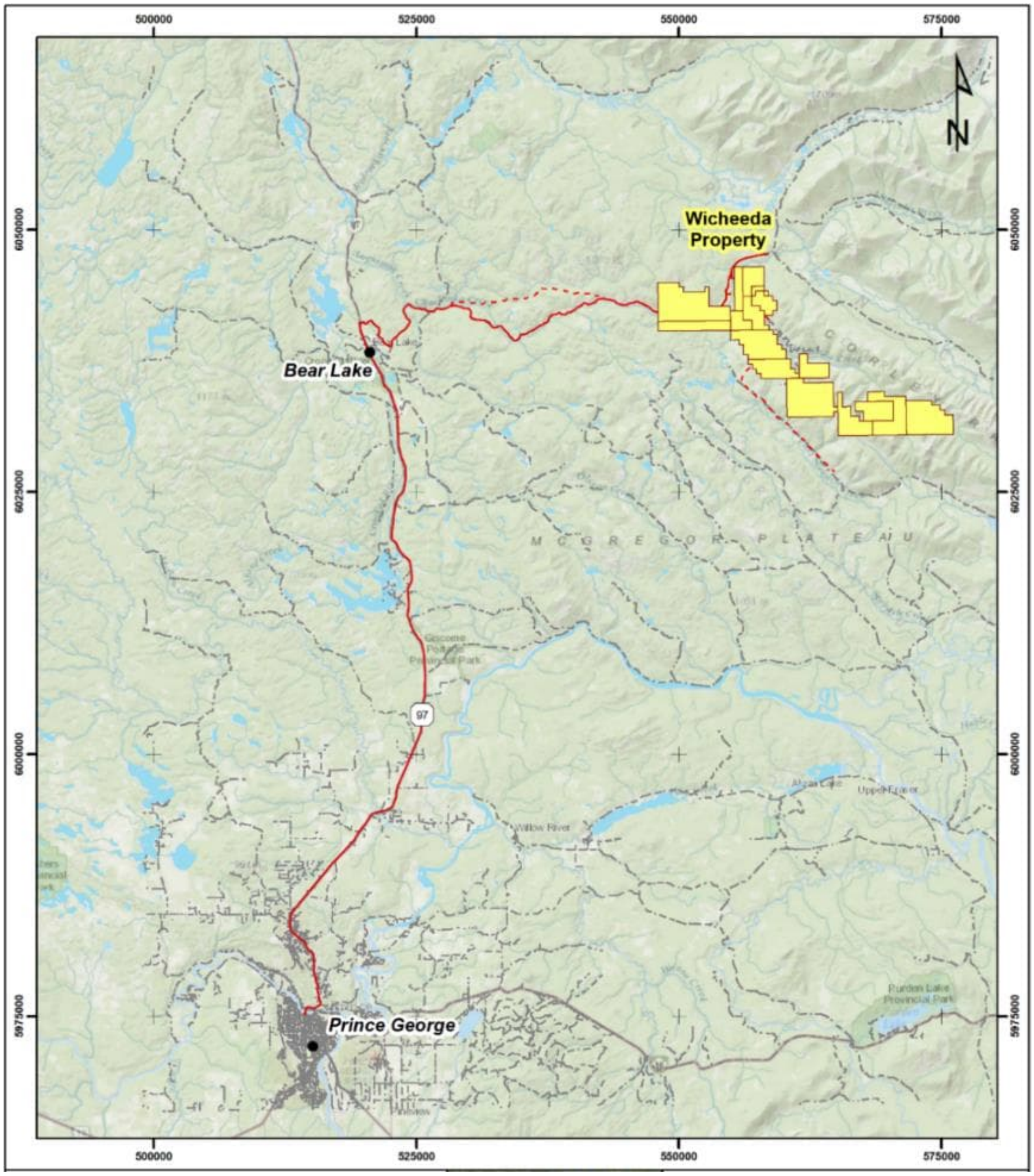
The property is comprised of 17 contiguous mineral claims, covering 11,800 ha within the Cariboo Mining Division (Table 4-1, Figure 4-2). The claims are registered on the Province of British Columbia's Mineral Titles Online (“MTO”) website and are listed as 100%-owned by Defense Metals Corporation. The individual claims and their respective anniversary dates are listed in Table 4-1.

No surface rights are held by Defense Metals. Should the project advance to the mining stage, Defense Metals will be required to obtain all necessary surface rights by way of filing an application for mining leases for the construction and operation of a mine on the project. Defense Metals is aware of a placer claim, held by others that will need to be resolved prior to advancing surface rights acquisition. Defense Metals is in discussions with the owners of the placer claim and an agreement is expected within 30 days of filing this PFS study.

Table 4-1: Wicheeda Property Mineral Claim Details

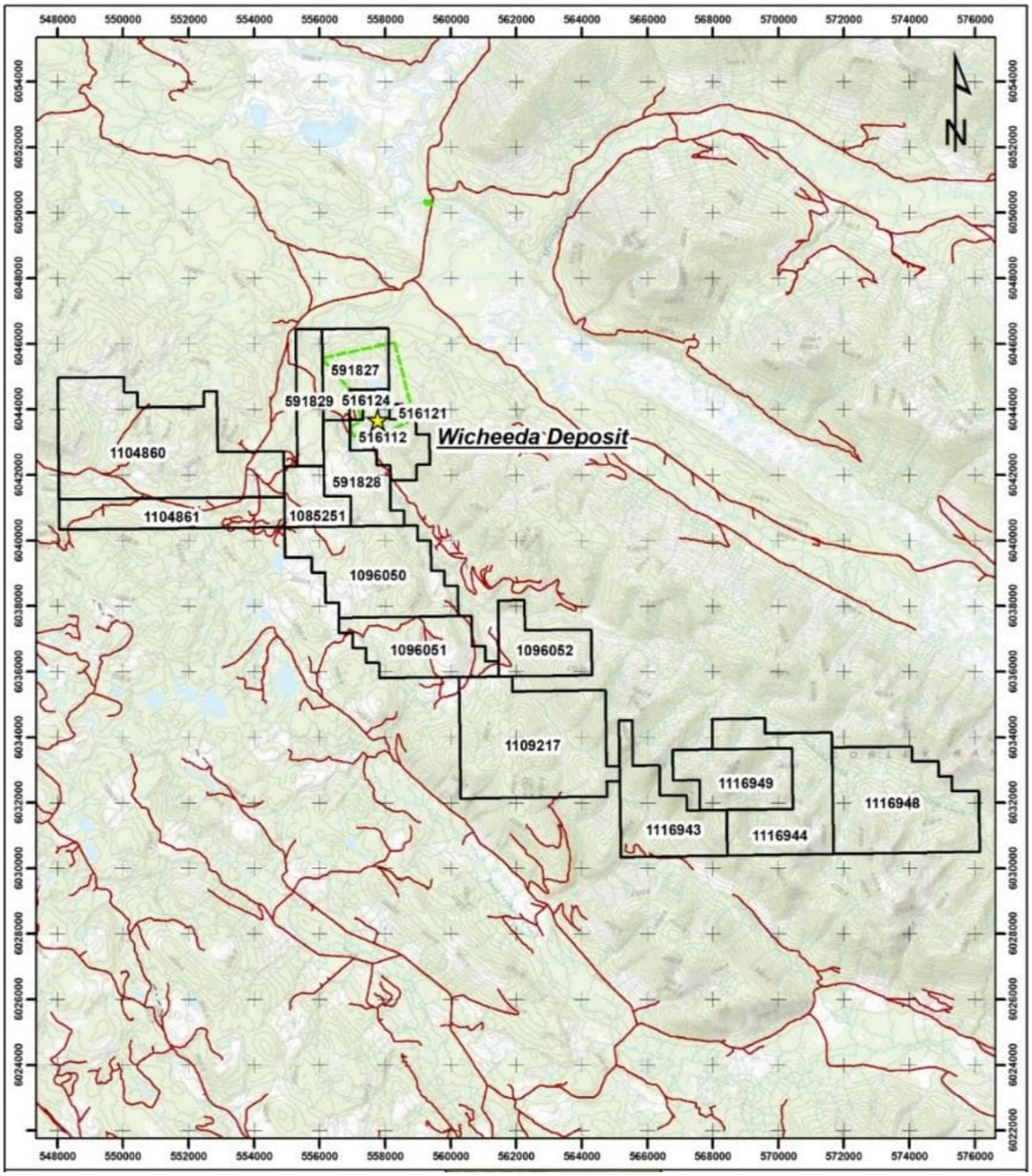
Tenure Number	Claim Name	Owner (%)	Area (ha)	Good to Date	Map Number
516112		Defense Metals Corp. (100%)	356.59	2034-09-30	093J060
516124	Wicheeda West	Defense Metals Corp. (100%)	75.05	2034-09-30	093J060
516121		Defense Metals Corp. (100%)	18.76	2034-09-30	093J060
591827	Wicheeda 6	Defense Metals Corp. (100%)	450.2	2034-09-23	093J060
591828	Wicheeda 7	Defense Metals Corp. (100%)	469.31	2034-09-23	093J060
591829	Wicheeda 8	Defense Metals Corp. (100%)	337.72	2034-09-23	093J060
1104860		Defense Metals Corp. (100%)	1858	2034-06-20	093J059 - 093J060
1104861		Defense Metals Corp. (100%)	638.38	2034-06-20	093J059 - 093J060
1096050		Defense Metals Corp. (100%)	1089.48	2034-06-06	093J050 - 093J060
1096051		Defense Metals Corp. (100%)	695.37	2034-06-06	093J050
1096052		Defense Metals Corp. (100%)	469.86	2034-06-06	093J050
1085251		Defense Metals Corp. (100%)	300.4	2034-05-05	093J060
1109217		Defense Metals Corp. (100%)	1542.11	2034-11-24	093J08
1116943		Defense Metals Corp. (100%)	715	2025-10-23	093I05
1116944		Defense Metals Corp. (100%)	921.87	2025-10-23	093I05
1116948		Defense Metals Corp. (100%)	1260.6	2025-10-23	093I05
1116949		Defense Metals Corp. (100%)	601.97	2025-10-23	093I05

Source: Apex, 2025



Source: APEX, 2025

Figure 4-1: Wicheeda Property Location Map



Source: APEX, 2025

Figure 4-2: Wicheeda Property Claim Map

4.2 Royalties and Agreements

The Property is 100%-owned and operated by Defense Metals Corp ("Defense"). The Property was subject to an Option Agreement dated November 22, 018 ("Option Agreement Effective Date"), where Spectrum Mining Corporation (former owner of the property) and its shareholders (collectively the "Vendors") granted Defense Metals an option to acquire ownership of the Wicheeda property.

On January 14, 2022, Defense Metals exercised its option and acquired 100% of the Wicheeda property through the acquisition of 100% of the issued and outstanding shares of Spectrum with the following common share issuances and payments:

a) Issued to the shareholders of Spectrum on a pro rata basis, such number of shares of Defense equal to 49% of the issued and outstanding common shares on a post-issuance basis (78,115,549 common shares issued).
b) Paid $100,000 in cash.
c) Issued 1,171,733 common shares pursuant to a finder's fee agreement with Mulgravian Ventures Corporation ("Mulgravian") entered into in connection with the November 22, 2018 option agreement.

On April 1, 2023, the Company amalgamated with Spectrum, all the issued shares of which were held by the Company, under section 273 of the Business Corporations Act (British Columbia) (the "Amalgamation"), with the continuing entity being Defense Metals. Accordingly, the shares of Spectrum were cancelled pursuant to the Amalgamation.

The Wicheeda property is subject to a 2.0% Net Smelter Return royalty with respect to the project, payable upon the commencement of commercial production. Defense Metals has the right to purchase one-half of the NSR Royalty from the Vendors, also on a basis pro rata to their current shareholdings in the Optionor (being 1.0% of Net Smelter Returns) for $1,000,000, leaving the Vendors with an aggregate of 1.0% NSR Royalty. For the purposes of this PFS, it is assumed that this option has been exercised.

4.3 Community and Local Relations

The project overlaps the Traditional lands of the McLeod Lake Indian Band ("MLIB"). A Treaty Impact Assessment on the project conducted by MILB confirms that the lands have supported MLIB in their exercise of constitutionally protected rights since time immemorial. The area is an important source of seasonally harvested foods, environmental livelihoods and remain cultural destinations for MLIB members. Defense Metals conducted several introductory information-sharing virtual meetings with the MLIB regarding its proposed exploration program from 2020 onward.

On September 7, 2022, Defense Metals announced it entered into a Mineral Exploration Agreement with the MLIB. The agreement addressed the immediate interests of the parties with respect to mineral exploration activities related to the project and put into place a framework for communication and cooperation going forward. In addition to providing MLIB with meaningful input into how site activities are to proceed, the agreement provided for economic opportunities for the community and established a roadmap for potential future commercial involvement.

On January 17, 2024, MLIB and Defense Metals Corp announced a strategic Equity Partnership and Co-Design Agreement. The agreement sets a new standard for collaborative project development among the parties and emphasizes a joint planning approach, empowering MLIB to play an integral part in the design and decision-making process in the technical, social, engineering, and environmental aspects of the Wicheeda project.

4.4 Environmental Liabilities, Permitting and Significant Factors

The Property exploration work to date has been conducted under a Multi-Year Permit, issued by the Ministry of Mining and Critical Minerals (MCM) to Spectrum on September 18, 2008, and amended on February 26, 2019 and November 16, 2023 (Permit MX-13-168). The permit was valid until December 31, 2024 and has expired.

Reclamation security funds totalling $24,300 were posted by Defense Metals to be held under Permit MX-13-168 by the British Columbia Minister of Finance. While this permit has lapsed, the reclamation funds are held until all the reclamation conditions of the permit are met in a manner satisfactory to the Chief Inspector of Mines.

A new Multi-Year, Area-Based permit for continued site investigations was submitted in May, 2024 and has been processed by MCM through government agency review and Indigenous consultation. A request for reclamation security of $157,000 CDN has been received. MCM will issue a decision on the permit upon submittal of a letter of credit for the reclamation security. The permit authorization will be for five years and will include drill sites, test pits sites, staging areas, new exploration trails, fuel storage, temporary bridges, water supply and camps. An occupant license to cut issued by the Ministry of Forests, Lands, Natural Resources Operations and Rural Development (FLNRORD) will be required for select tree removal to facilitate the site investigations.

There are no known environmental liabilities associated with the project as a result of any previous exploration. With respect to environmental and permitting risk and uncertainty the area surrounding Wicheeda Lake is known to have high cultural, recreational and ecological values, including native trout stocks in the lake. The lake and surrounding area is currently covered under Recreational Reserve REC6837 (Figure 4.2). As of October 23, 2023 the BC Ministry of Forests Land and Natural Resource (FLNR) indicated that given its current priorities and capacity there is no intent to establish a recreation site at Wicheeda Lake in the near future.

At present there are no restrictions on mineral exploration activities within REC6837. During 2009 five diamond drill holes were completed within REC6837. Recreation reserves are map notations or “an indication of interests” without any legal designation, as opposed to legally established recreational sites having specification of, or restriction against, permitted or specific uses. However, FLNR has requested that Defense Metals take all possible steps to minimize the impacts of exploration to the recreational ecological values associated with Wicheeda Lake.

The mine site area overlaps with endangered and threatened species including identified critical habitat for the threatened mountain caribou. The conservation status of caribou is an environmental issue and a matter of Indigenous rights. A mitigation strategy will need to be collaboratively developed, proposed and accepted by both Indigenous rightsholders and the government.

For the purpose of this document, carbon costs for the Output Based Pricing Systems were not calculated and will be in future studies.

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

5.1 Accessibility

The Wicheeda property is located at Wicheeda Lake, approximately 80 km northeast of Prince George and 50 km east of Bear Lake in the Cariboo region of British Columbia. Access to the Property from Prince George is facilitated via Provincial Highway 97 and two all-season gravel forest service roads. From Prince George, travel north on Highway 97 for approximately 80 km to the turn-off for the Chuchinka Forest Service Road (FSR), located just south of the community of Bear Lake. Travel east on the Chuchinka FSR to the Wichcika FSR; then south on the Wichcika FSR to the Wichcika gravel pit. The gravel pit was used as an equipment staging area and as a camp site during the 2019 - 2023 exploration programs. A hiking trail, approximately 1300 m in length, links the gravel pit and areas of diamond drilling.

5.2 Local Resources and Infrastructure

Bear Lake has access to the CN rail mainline, a natural gas pipeline, and a power transmission line. Dormant lumber industry sites such as a three-line sawmill, located immediately east of the Highway 97 near its junction with the Chuchinka FSR, could be utilized for Wicheeda project development, specifically for the location of the hydrometallurgical plant and access to electrical power, as could other nearby areas designated for heavy industrial use.

The city of Prince George, BC, known as the “Northern Capital of British Columbia”, is located 80 km southwest of the project.

More information on Bear Lake and Prince George are provided in Section 20.3.1

5.3 Site Topography, Elevation and Vegetation

The Wicheeda project is located on the eastern flank of the Rocky Mountains, on the edge of the Central Interior Plateau in east central British Columbia. The Property is characterized by subtle-moderate hills in the south/southeast and steep to cliffy topography in the east / southwest. Elevations range between 900 and 1520 m AMSL. Outcrops are sparse within the property, even in hilly areas.

The area is covered with stands of alder, birch, pine and spruce with variably thick undergrowth, or by logged areas. Forest plantations, buck brush and devil’s club are present in the property.

5.4 Climate

The climate of the project area is typical of northern continental areas, characterized by large seasonal temperature differences, with warm to hot (and often humid) summers and cold winters. Climate data indicate that temperatures vary from an average of 15.9°C in July, the warmest month, to -10.2°C in January, the coldest month, with an annual mean temperature of 4.1°C. The area averages 558 mm of precipitation annually.

6. History

6.1 Property History – Early Exploration

A regional aeromagnetic survey of the area, completed in 1961 by the Geophysics Division of the Geological Survey of Canada, identified a Magnetic high feature in the area of the Wicheeda Project. Prospecting of the area in 1976 and 1977 by Kol Lovang identified minor base metal showings which covered two mineral claims. No follow-up work was completed and the claims were allowed to lapse. However, later assaying of Lovang's samples by Teck Explorations Limited ("Teck") revealed anomalous niobium and Teck subsequently entered into a prospecting agreement with Lovang in early 1986 (Betmanis, 1987).

6.2 Property History – Teck Explorations Limited

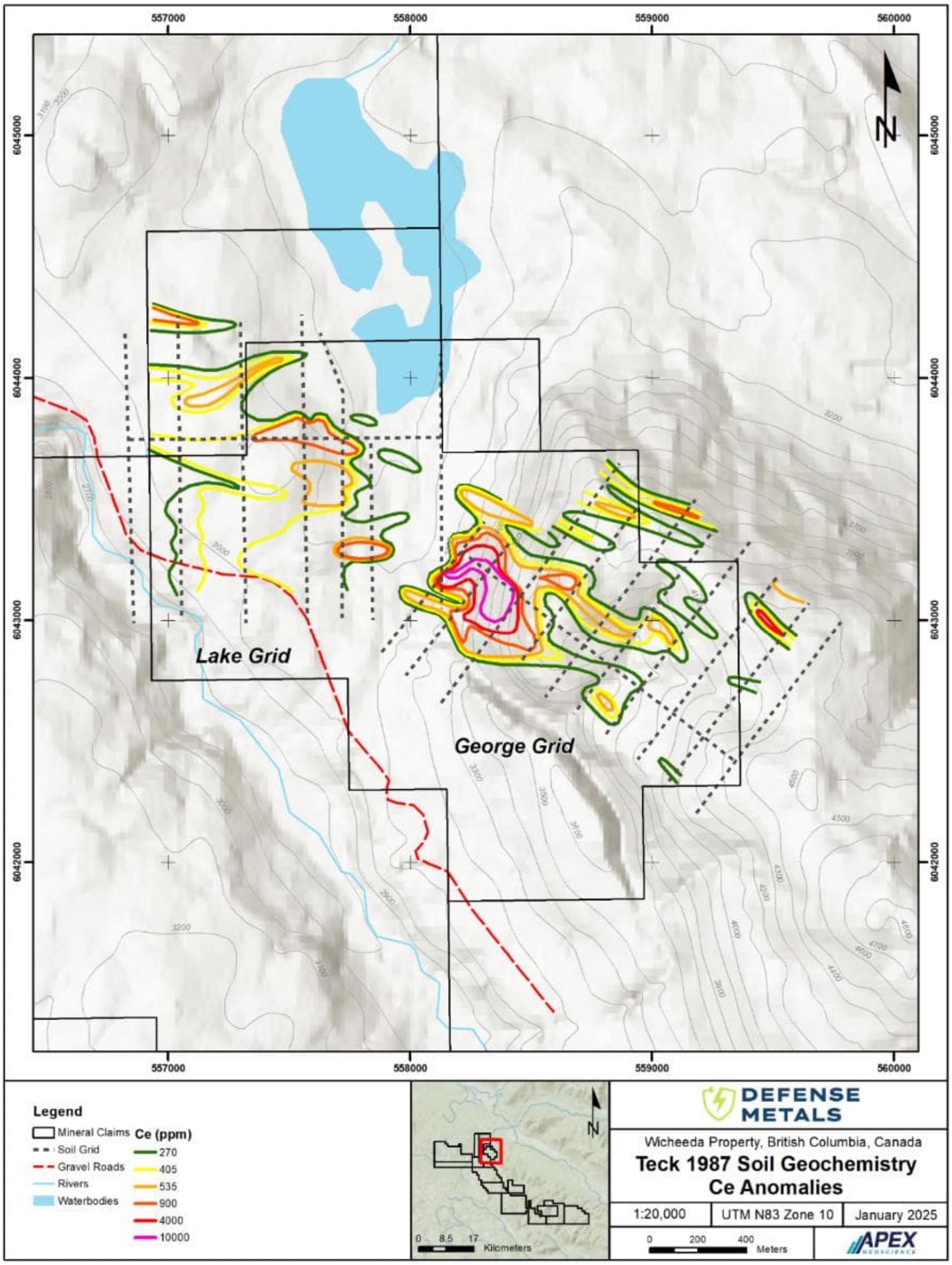
Teck staked its initial claims in April 1986 and proceeded with a helicopter-supported stream silt geochemical survey of the Wichcika Creek drainage. This work identified several anomalies, resulting in additional claims being staked (Betmanis, 1987). Within the claim group, Teck delineated 5 grids (Lake, George, D, F, and Prince) for reconnaissance work. Only the 'Lake' and 'George' grids are covered by the extent of the current claims (Figure 6-1)

Additional exploration completed in 1986 and 1987 consisted of soil and rock geochemical sampling, geological Mapping, trenching and ground Magnetic surveying (Betmanis, 1988 and 1987). Results from the soil geochemical surveys indicated a linear carbonatitic intrusion and a small syenite body hosted by limestone and calcareous fine-grained sedimentary rocks over a total strike length of 2.25 km contained within the 'Lake' and 'George' grids (Betmanis, 1987). Rock geochemical sampling and bedrock Mapping led to additional claims being staked during 1986 as the location of intrusive zones became better defined (Betmanis, 1987).

Pronounced cerium in soil geochemical anomalies partially cover both the 'Lake' and 'George' grid areas (Figure 6-1)

Locally, these anomalies coincide with barium and niobium highs and reflect the underlying intrusive rock. Intensely oxidized, coarse grained calcite carbonatite and fine-grained pyrochlore-bearing, pink calcite carbonatite was identified in trenches at the 'Lake' grid by Mader and Greenwood (1988). Ground Magnetometer surveys outlined modest Magnetic highs on both grids that are thought to be reflective of relatively narrow dykes that May or May not be genetically related to the intrusive carbonatites (Betmanis, 1987).

``



Source: APEX, 2025

Figure 6-1: Teck Exploration 1986-1987 Soil Geochemistry (Cerium)

``

Follow-up work outlined a deeply weathered carbonatite of unknown dimensions on the ‘Lake’ grid (Mader and Greenwood, 1988); and a semi-circular body of carbonatite, measuring about 250 m across, on the ‘George’ grid (Lovang and Meyer, 1978). A circular thorium (Th) radiometric anomaly, roughly 500 m in diameter, was found to coincide with the ‘George’ grid carbonatite and additional Th radiometric anomalies 100-200 m across followed a southeasterly trend (Lovang and Meyer, 1978; Mader and Greenwood, 1988). Soil geochemistry on the ‘George’ grid estimated the circular intrusive body at approximately 400 m in diameter. One or more narrow dyke-like carbonatite bodies were located south of ‘George’ grid (Lovang and Meyer, 1978; Minfile 093J 014) partially covering the southern portion of the Project area.

Lovang and Meyer (1987) found the carbonatites were generally anomalous in light rare earth elements (LREE) and niobium. A limited hand trenching program on the ‘George’ grid yielded encouraging values of lanthanum (La), neodymium (Nd) and cerium (Ce), modest values of niobium (Nb) and yttrium (Y), and anomalous values of samarium (Sm) and europium (Eu). Values for the principal LREE ranged from 202 to >1000 ppm La, from 104 to >1000 ppm Nd, and from 254 to >10000 ppm Ce over sample lengths of 2-10 m and an aggregate sample length of 87 m in three trenches spaced across the carbonatite body (Lovang and Meyer, 1987).

Subsequently, the claims were staked in March 2001 by Mr. Chris Graf after Teck allowed the claims to lapse. Mr. Graf, a principal of Spectrum, did not conduct any work of significance on the claims, and in September 2008, he transferred ownership of the claims to Spectrum.

6.3 Property History – Spectrum Mining Corporation

From late September to mid-October 2008, Spectrum completed four diamond drill holes (WI08-01 to WI08-04) with an aggregate length of 866 m within the ‘George’ grid area (Lane, 2009; Figure 6.2). The holes were drilled from a single helicopter-supported drill pad and included one vertical hole and three inclined holes drilled on different azimuths. Each drill hole was collared in intrusive carbonatite and confirmed the presence of a LREE-bearing dolomite carbonatite body of significance that outcrops on a west-facing slope 1 km south of Wicheeda Lake. Due to the limited amount of drilling, the overall geometry of the Wicheeda Carbonatite was not resolved; however, the 2008 campaign established an eastern structural footwall of the deposit. The western, northern, southern and depth components remained open (Lane, 2009).

The Wicheeda Carbonatite was found to contain significant concentrations of the LREEs cerium (Ce), lanthanum (La), and neodymium (Nd) as well as anomalous concentrations of Nb, Pr, Y, As, Ba, Mo, Mn, Pb, Sr, and Th (Lane, 2009). Significant weighted averages (XRF data) for selected drill hole are presented in Table 6-1.

``

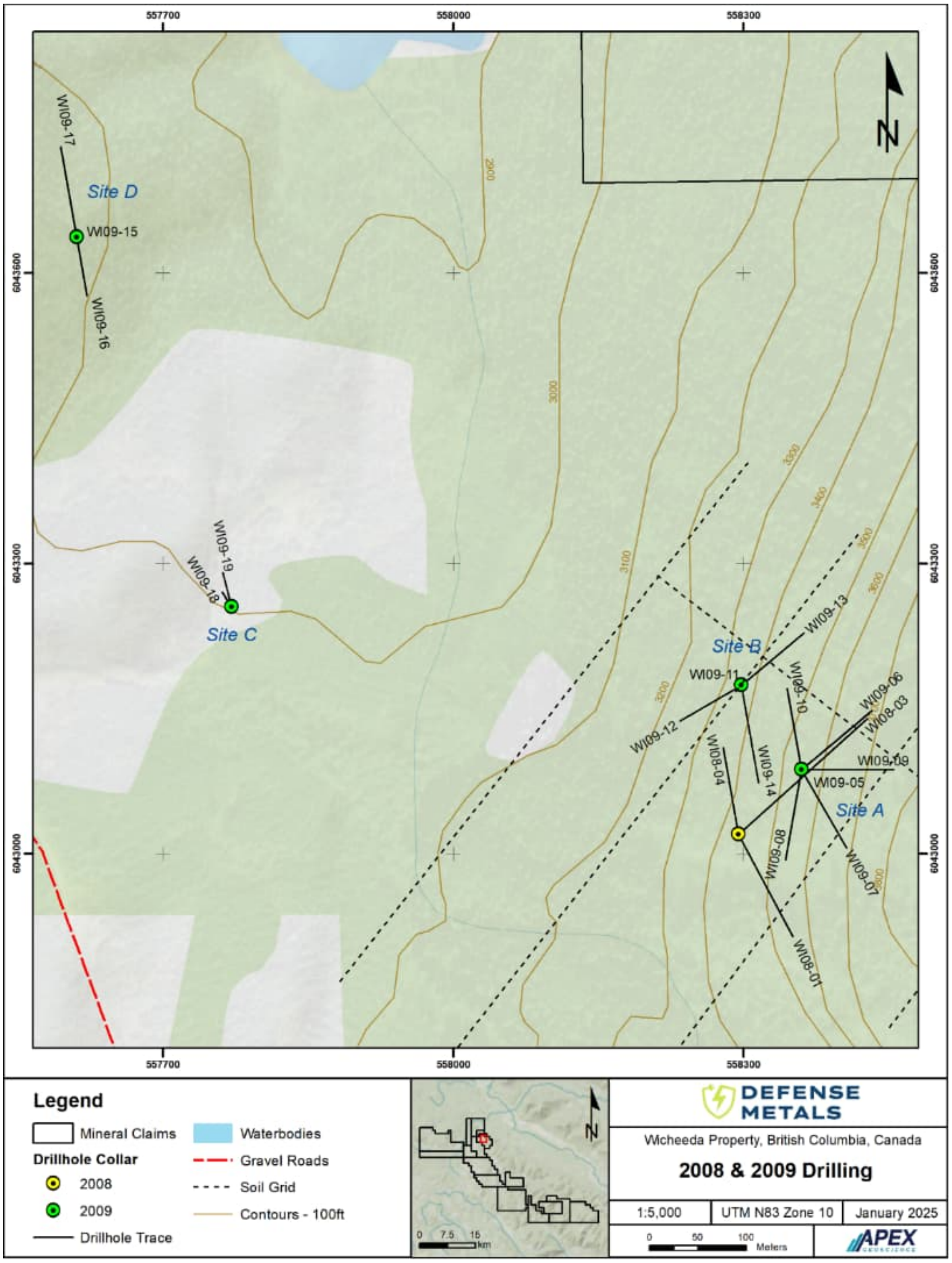
Table 6-1: 2008 and 2009 Wicheeda Carbonatite Significant Drill Hole Intercepts

Hole	From (m)	To (m)	Interval[1] (m)	CeO2 %	La2O %	Nd2O %	Pr6O11 %	Sm2O3 %	Dy2O %	Tb4O7 %	Eu2O3 %	Gd2O3 %	Ho2O3 %	TREE %	TREO[2] %
WI08-01	2.13	68.23	66.1	1.63	1.15	0.27	0.13	0.03	0.00	0.00	0.01	0.01	0.00	2.69	3.23
WI08-02	1.42	86.27	84.85	1.70	1.23	0.29	0.13	0.03	0.00	0.00	0.01	0.01	0.00	2.83	3.40
WI08-03	2.56	234	231.44	1.07	0.75	0.22	0.09	0.03	0.00	0.00	0.01	0.01	0.00	1.81	2.17
Including	2.56	75.55	72.99	1.69	1.23	0.26	0.12	0.03	0.00	0.00	0.01	0.01	0.00	2.80	3.36
WI08-04	1.57	121	119.43	1.63	1.17	0.31	0.13	0.03	0.00	0.00	0.01	0.01	0.00	2.75	3.30
WI09-05	1.52	56.39	54.87	1.18	0.86	0.28	0.10	0.03	0.00	0.00	0.01	0.01	0.00	2.06	2.47
Including	1.52	43.52	42	1.30	0.94	0.30	0.11	0.03	0.00	0.00	0.01	0.01	0.00	2.25	2.70
WI09-06	1.52	133.5	131.98	1.70	1.29	0.38	0.13	0.04	0.00	0.00	0.01	0.02	0.00	2.98	3.57
WI09-07	2.44	107.44	105	1.64	1.22	0.34	0.13	0.03	0.00	0.00	0.01	0.02	0.00	2.83	3.40
Including	2.44	74.44	72	1.96	1.47	0.40	0.16	0.04	0.01	0.00	0.01	0.02	0.00	3.38	4.05
WI09-08	1.83	97.83	96	1.49	1.08	0.31	0.11	0.03	0.00	0.00	0.01	0.01	0.00	2.54	3.04
Including	52.83	97.83	45	2.01	1.48	0.39	0.15	0.04	0.00	0.00	0.01	0.02	0.00	3.42	4.10
WI09-09	1.4	145.4	144	1.50	1.11	0.33	0.12	0.03	0.00	0.00	0.01	0.01	0.00	2.59	3.11
Including	1.4	67.4	66	1.76	1.31	0.38	0.14	0.04	0.00	0.00	0.01	0.02	0.00	3.05	3.65
WI09-10	2.44	148.13	145.16	1.33	0.98	0.28	0.10	0.03	0.00	0.00	0.01	0.01	0.00	2.28	2.74
Including	62.44	89.44	27	1.72	1.29	0.37	0.13	0.03	0.00	0.00	0.01	0.01	0.00	2.97	3.56
Including	128.44	148.13	19.69	1.92	1.41	0.38	0.15	0.03	0.00	0.00	0.01	0.01	0.00	3.26	3.91
WI09-11	3.2	57.2	54	1.45	1.07	0.31	0.11	0.03	0.00	0.00	0.01	0.01	0.00	2.49	2.99
WI09-12	6.7	39.7	33	0.97	0.67	0.21	0.08	0.02	0.00	0.00	0.00	0.01	0.00	1.65	1.98
WI09-13	1.83	147.52	145.69	1.22	0.86	0.29	0.10	0.03	0.00	0.00	0.01	0.01	0.00	2.10	2.52
Including	7.83	49.83	42	1.56	1.11	0.37	0.13	0.04	0.00	0.00	0.01	0.01	0.00	2.69	3.22
WI09-14	3	120	117	1.21	0.83	0.27	0.10	0.03	0.00	0.00	0.01	0.01	0.00	2.05	2.46
Including	3	39	36	2.15	1.53	0.42	0.16	0.04	0.00	0.00	0.01	0.01	0.00	3.60	4.32

[1] The true width of REE mineralization is estimated to be 70-100% of the drilled interval.

[2] TREO % sum of CeO2, La2O3, Nd2O3, Pr6O11, Sm2O3, Eu2O3, Gd2O3, Tb4O7, Dy2O3 and Ho2O3.

Source: APEX, 2023



Source: APEX, 2025

Figure 6-2: Spectrum Mining 2008-2009 Diamond Drilling

In 2009, Spectrum completed 15 additional drill holes (WI09-05 to WI09-20), totaling 1,824 m (Lane, 2010a; Table 6.1; Figure 6.2). Ten holes tested the Wicheeda Carbonatite from two different set-ups (sites A and B), two holes were drilled northwest of previous sites to intersect a small carbonatite dyke that outcrops on a trail leading to Wicheeda Lake (site C),

``

and three holes tested a REE soil anomaly located northwest of site C and southwest of Wicheeda Lake (site D). All ten holes drilled on the Wicheeda Carbonatite intersected significant intervals of REE-bearing dolomite ± calcite carbonatite from surface to variable depths (Table 6.1). The highest REE values correlated with dolomite carbonatite, dolomite carbonatite breccia and calcite carbonatite. High REE values also occurred in syenite breccia, later referred to as xenolithic dolomite carbonatite where dolomite carbonatite (matrix: clasts) formed > 20% and <70% of the rock Mass.

Drill site C was positioned on small knoll approximately 550 m south of the south end of Wicheeda Lake near an access trail that leads to the lake. Two holes targeted a narrow carbonatite dyke that crops out on the trail, one of which intersected 27.20 m of dolomite carbonatite similar to that observed at the Main Zone. Only core from drill hole WI09-19 was sampled; while the carbonatite dyke intersected was visually similar to that of the Main Zone, the former returned only weakly elevated concentrations of lanthanum with an individual high value of 523 ppm La over 3.0 m (Lane, 2010a).

Drill site D was located 315m southwest of the south end of Wicheeda Lake near an old Teck trench. Three holes were drilled from this site to evaluate a rare earth element soil geochemical anomaly. Each hole (WI09-15 to WI09-17) intersected calcite carbonatite and breccia that was consistently elevated in lanthanum. Hole WI09-15 averaged 345 ppm La over 96.00m; hole WI09-16 averaged 307ppm La over 91.71m, and hole WI09-17 averaged 307 ppm La over 146.30 m (Lane, 2010a).

During the same year, a bench scale heavy liquid – Magnetic separation was performed on a composite sample from the Wicheeda Lake carbonatite to separate minerals and produce a concentrate comparable to other well-known REE deposits around the world. The study achieved a high grade REE concentrate comparable with the Mianning bastnäsite-bearing carbonatite ore from Sichuan, China (Mariano, 2009).

In 2010, Spectrum contracted Hendex Exploration Limited of Prince George to conduct a GPS-controlled soil sampling survey over a 5.5 km2 area measuring approximately 2300 m north-south by 2200 m east-west (Graf, 2011; Figure 6.3). The survey covered the Wicheeda Carbonatite as well as other targets to the northwest that were drilled in 2009. A total of 977 soil samples were collected at stations spaced 50 m apart along east-west lines spaced 100 m apart. The survey data outlined three significant multi-element soil geochemical anomalies on the Project: the Wicheeda Carbonatite soil anomaly, the Southwest soil anomaly and the Northwest soil anomaly. Results from the Wicheeda Carbonatite anomaly indicated a strong correlation between cerium, lanthanum, yttrium, niobium, thorium, lead, Manganese, molybdenum, iron and phosphorous (Graf, 2011).

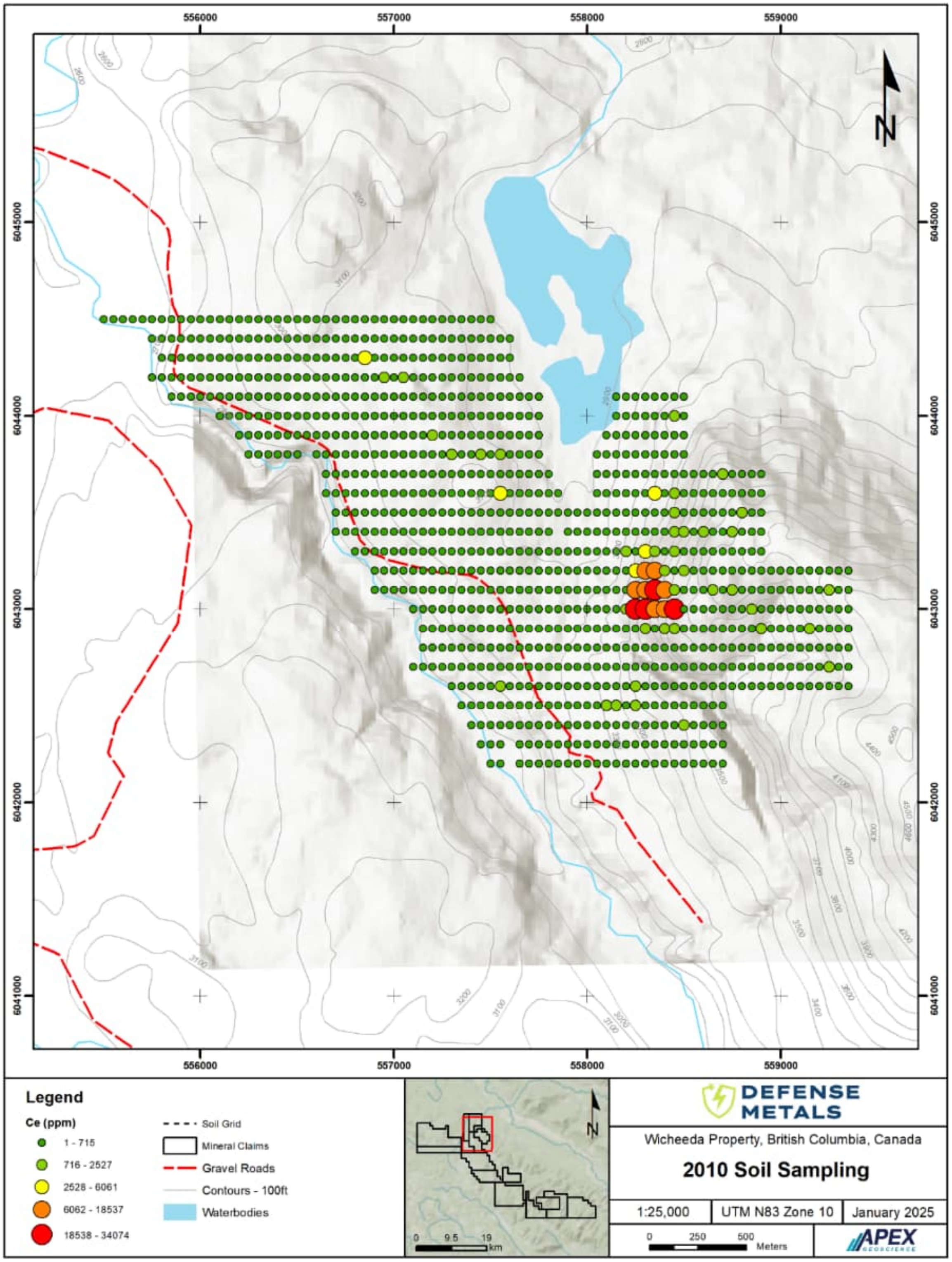
The Wicheeda Carbonatite is outlined by the approximately coincident contours of cerium (>400 ppm), lanthanum (>200 ppm), yttrium (>25 ppm), niobium (>100 ppm), thorium (>100 ppm), lead (>100 ppm), phosphorous (>2000 ppm), molybdenum (>10ppm), barium (>2000 ppm), Manganese (>2000 ppm) and iron (>50,000 ppm). The Wicheeda Carbonatite multi-element soil anomaly consists of a higher value north-south trending core area roughly 400 m long and 200 m wide east-west with a narrower northeast-trending (015° azimuth) extension that is 300 m long and 100 m wide (Figure 6.3). The entire multi-element anomaly extends over 700 m long by 100-200 m wide (Graf, 2011). The highest niobium and thorium

``

values are restricted to the core area of the soil anomaly and are significantly lower along the northeast extension.

A second coincident multi-element soil geochemical anomaly lies 300 m southwest of the Wicheeda Carbonatite anomaly on the same (195° azimuth) orientation as the northeast (15° azimuth) extension of the Main zone anomaly. This Southwest soil anomaly is outlined by the approximately coincident contours for cerium (400 ppm), lanthanum (200 ppm), yttrium (30 ppm), niobium (40 ppm), thorium (30 ppm), lead (50 ppm), phosphorous (1000 ppm), molybdenum (4 ppm), barium (1,000 ppm), Manganese (1,000 ppm) and iron (40,000 ppm) and extends in a northwest direction (Figure 6.3). The Southwest anomaly is 500 m long by 50-150 m wide (Graf, 2011). The location of the Southwest soil anomaly directly along strike with the northeast extension of the Wicheeda Carbonatite anomaly suggests that it May represent a southwest extension of the Wicheeda Carbonatite body.

A third coincident multi-element soil geochemical anomaly occurs roughly 400 m northwest of the Wicheeda Carbonatite soil anomaly. This Northwest soil anomaly is outlined by the approximately coincident contours for cerium (>200 ppm), lanthanum (>100 ppm), yttrium (>15 ppm), niobium (>100 ppm), thorium (>30 ppm), lead (>50 ppm), phosphorous (1500 ppm), molybdenum (>4 ppm), barium (>1,000 ppm), Manganese (>1,000 ppm) and iron (>40,000 ppm). The niobium values of the Northwest soil anomaly are as elevated as the niobium values in the Wicheeda Carbonatite soil anomaly; however, the cerium and lanthanum values are more subdued. This contrasts with the Wicheeda Carbonatite anomaly that has extremely elevated cerium and lanthanum values. The Northwest niobium anomaly (>100 ppm) contour is much larger than the Wicheeda Carbonatite niobium anomaly, measuring 600 m long north-south by 50-400 m wide east-west within a >40 ppm niobium anomaly that is 1100 m long north-south by 400 m to 700 m wide east-west. Additionally, the niobium values are consistently more elevated with a peak value of 901 ppm. There is a narrower multi-element soil anomaly along the west side of the Northwest soil anomaly that May represent a separate mineralized carbonatite dike or sill emanating from a larger carbonatite body that May underlie the larger soil anomaly. It is 300-500 m long by 50-100 m wide and has peak values of 1,893 ppm niobium, 1,512 ppm cerium and 915 ppm lanthanum (Graf, 2011).



Source: APEX, 2025

Figure 6-3: Spectrum Mining 2010 Soil Geochemistry (Ce ppm)

``

Prior to conducting the flotation test work, SGS carried out a high definition ("QEMSCAN"), electron microprobe and chemical analysis of the drill core composite samples. In June 2010, Spectrum submitted two composite samples from drill core, a Syenite Breccia (SB) and a Dolomite Carbonatite (DC), to the mineralogy department of SGS Canada Inc. ("SGS") at Lakefield, Ontario, to determine the overall mineral assemblage and textural characteristics for each sample, the liberation/association of the REE minerals and the grade recovery of REE (SGS, 2010). The mineralogical study of the samples identified dolomite, ankerite and K-feldspar as the dominant minerals and plagioclase, calcite, sericite /muscovite, biotite, other silicates and chlorites as minor minerals. The study identified parisite, bastnaesite, monazite and apatite as the Main REE minerals. Liberation analysis indicates monazite and cerium carbonate / oxides (Parisite + Bastnaesite) are well liberated at 67% to 74% and 79% to 86% in the SB Comp and DC Comp, respectively. The liberation of monazite (79%) and Ce-carbonate/oxides (86%) are higher in DC than the SB composite. The grade recovery of REE (Ce, La, Sm, Nd, Eu, Th, Y, Pr) for the SB Comp sample indicated REE grades between 56% and 48% for recoveries of 72% to 92%, respectively. The grade recovery of REE for the DC Comp sample indicated grades between 55% and 51% for recoveries of 85% to 96% respectively. Both the SB and DC fine fractions reported REE grades up to 97%.

Exploratory metallurgical test work has been carried out on Wicheeda mineralized dolomite-carbonatite by SGS in 2011 and 2012 using a representative composite sample created with core collected from seven separate drill holes at varying depths.

In 2011, Spectrum commissioned SGS to complete a test work program to investigate the direct flotation of rare earth oxide (REO) on samples from the Wicheeda Carbonatite (SGS, 2011). The tests concluded that the Wicheeda carbonatite orebody had a fairly homogeneous mineralization and as a result very little ore variability flotation tests would be anticipated in the future. The process test work produced a rare earth oxide concentrate suitable for further treatment in a hydrometallurgical process. The SGS test work successfully produced a 42% REE concentrate with recoveries shown in Table 6-2 .

Table 6-2: Locked Cycle Test Recoveries

REE	Recovery to Concentrate (%)
Ce	82.5
La	84.6
Nd	83.8

Source: SGS, 2011

In 2012 subsequent hydrometallurgical testing was conducted by SGS on a 2 kg composite sample of the Wicheeda flotation concentrates they had produced. The purpose was to develop a conceptual hydrometallurgical flow sheet consisting of pre-leaching, roasting, REE leaching and REE precipitation. The feed grade consisted of a flotation concentrate of 39.7% total rare earth oxides, which through pre-leaching was upgraded to 67% TREO (total rare earth oxide) Material and that in turn was further upgraded to 71% TREO by roasting the pre-leach residue. The hydrometallurgical tests were successful in producing a final purified precipitate of 69.7% TREO and removing 98% of the thorium from the concentrate (SGS, 2012). Additional details regarding mineral processing and metallurgical testing are discussed in Section 13.

6.4 Third-Party Regional Airborne Radiometric and Magnetic Surveys

In 2011, Bolero Resources Corporation conducted a helicopter-borne radiometric and Magnetic gradiometer survey over its vast Carbonatite Syndicate Property that encompasses the Project (Koffyberg and Gilmour, 2012, Figure 6.4). The survey was flown over a portion of the Project and outlined a potentially significant 500 m long by 200 m wide radiometric anomaly inside the southeastern most corner of the Wicheeda claims. There is incomplete soil sample coverage in this area, however, the existing soil sample data indicates that a multi-element geochemical anomaly May extend into this area and is potentially 400 m long.

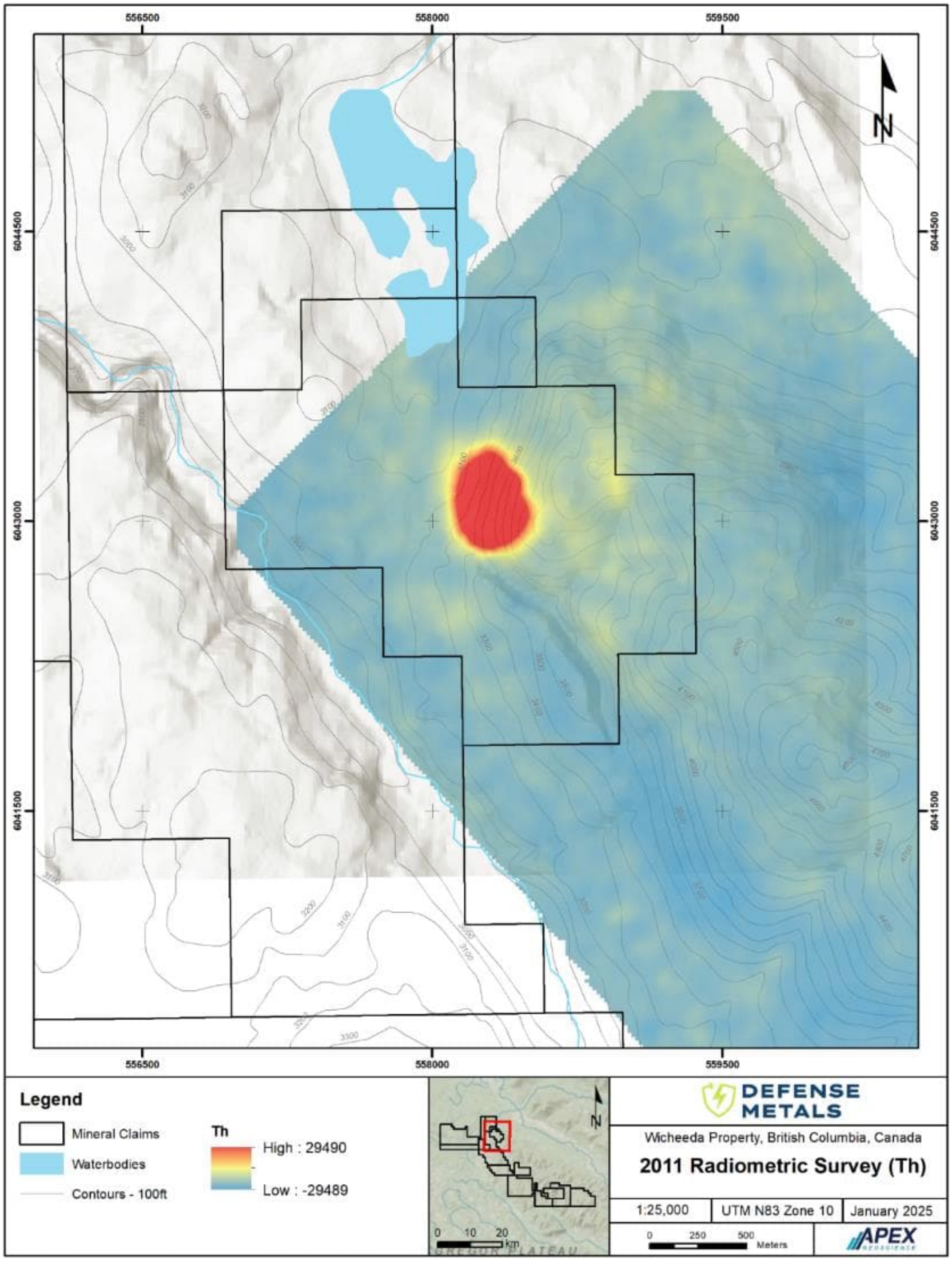
The strongest airborne Magnetic anomaly on the Project trends in a northwest direction and is 600 m long by 200 m wide. This anomaly May be the expression of a Magnetite-bearing syenites (Bird et al., 2019).

6.5 Property History – Academic Studies

Two academic studies were completed on the Wicheeda Carbonatite in 2014. One study focused on the nature and origin of the deposit; the principal results were:

- The carbonatite comprises a dolomitic core and a thin outer calcitic facies
- Bastnäsite-(Ce) and subordinate monazite-(Ce) are the Main REE minerals
- The REE mineralization was the product of Magmatic hydrothermal fluids which also fenitized the surrounding metasedimentary rocks (Trofanenko et al., 2014).

The other study evaluated the application of portable X-ray fluorescence (XRF) as an exploration tool for REE-enriched carbonatites. It concluded, based on the mineralogy of the Wicheeda carbonatite complex (detectable concentrations of Nb, Ta, La, Ce, Pr, Nd, and Y), that monazite, REE-fluorocarbonates and carbonates, and pyrochlore (± columbite) are prospective indicator minerals for Wicheeda carbonatite-type REE deposits (Mackay and Simandl, 2014).



Source: APEX, 2025

Figure 6-4: Bolero Resources 2011 Radiometric Survey Results (Thorium)

``

6.6 Property History – Defense Metals

In October 2018, Defense Metals collected a 30-tonne surface bulk sample at the Wicheeda Project for a multi-phase program of bench-scale metallurgical test work (SGS, 2019). The sample was submitted for metallurgical testing with SGS Canada. Select head assay results for the 30-tonne bulk sample include: 1.77% lanthanum-oxide, 2.34% cerium-oxide, 0.52% neodymium-oxide, and 0.18% praseodymium-oxide, for a total of 4.81% LREO (light rare-earth oxide). The metallurgical work also returned 10.1 times upgrade ratio, a low Mass yield concentrate at 8.2% and a recovery up to 85.7%.

During the summer of 2019, Defense Metals carried out a diamond drilling program. Thirteen diamond drill holes totalling 2,005 m delineated the higher-grade near surface dolomite-carbonatite unit and tested the Margins of the deposit. All drill holes intersected variable lengths of significant REE mineralization, Mainly in the carbonatite dolomite body and, to a lesser extent, in the lithologies enveloping the Wicheeda Carbonatite. The 2019 drilling campaign extended the Wicheeda deposit a 120 m along NNW-strike, 40 m to the southeast and 25 m to the southwest from previously defined resource model. More detail is presented in Section 10 of this report.

In 2020, Defense Metals completed a LiDaR survey over the eastern flank of the property. The increased resolution of the LiDaR allowed for more robust mine planning, particularly when considering the high relief within the project area.

Between 2020 and 2021, all the 2008 and 2009 original drill core pulps were reanalyzed, utilizing a REE lithium metaborate fusion with an ICP-MS finish analytical method, to reduce the uncertainty regarding the historical incomplete XRF analytical results. More detail is presented in Section 11 of this Report.

During 2020, Defense Metals initiated a flotation pilot plant based on the 2019 positive metallurgical test work. The work was carried out by SGS on the 30 tonne bulk sample and results indicated: an average REO recovery of 77.3% in a concentrate grading 51.6% TREO. The company also created a 3D geological model and produced a historic Mineral Resource Estimate (MRE) in 2020 (Raffle and Nicholls, 2020) which is discussed in Section 6.7.

During 2021, Defense Metals carried out a diamond drilling program. The program directive was to test the extent of the Wicheeda deposit where it is still open to the north and northwest, further delineate the relatively higher-grade near-surface dolomite unit, and to convert the inferred and/or indicated mineral resource into indicated and measured mineral resource. Twenty-nine NQ diameter diamond drill holes, totalling 5,366.3 m, were completed from five different drill pads, testing the southern, central and northern zones of the carbonatite. All 29 drill holes intersected significant intercepts of REE-mineralized dolomite carbonatite. Drilling at the Wicheeda Deposit delineated and expanded the carbonatite body to the north / northwest and Marginally around the deposit. More detail is presented in Section 10 of this report.

During 2022, Defense Metals carried out a diamond drilling program. The objectives of the program was to test the limit of the dolomite carbonatite-hosted REE mineralization, further delineate the relatively higher-grade, near-surface dolomite carbonatite unit, upgrade the mineral resource of the deposit and provide initial information on the geotechnical features of

``

the project. A total of 18 diamond drill holes were completed from nine separate drill pads totalling 5,510 m. Fifteen drill holes intersected significant intercepts of visible REE-mineralized dolomite-carbonatite at the Wicheeda Deposit and three exploration and geotechnical holes intersected barren to weakly altered sediments with trace REE mineralization. The 15 drill holes were drilled within the new block model to infill and better define the limits of the resource model, and to achieve an indicated and measured resource classification. Several drill holes were collared to the north and along strike to test the northern extension of the REE-mineralized carbonatite-dolomite. Drillholes in the central and southern portion of the deposit upgraded resource classification. More detail is presented in Section 10 of this report.

6.7 Mineral Resource Estimates & Preliminary Economic Assessment

A number of Mineral Resource estimates have been completed on the Project to date, the following section provides an overview of the estimates and methods used. A QP has not done sufficient work to classify the historical estimate as a current MRE or Mineral Reserve and the issuer is not treating the historical estimate as a current MRE.

In June 2019, an initial Mineral Resource Estimate (MRE) was prepared for Defense Metals by Moose Mountain Technical Services with an effective date of November 26, 2018 (Bird et al., 2019). The 2019 MRE was reported at a cut-off grade of 1.0% LREE (sum of Light Rare Earth Element Ce + La + Nd + Pr + Sm percentages) and comprised 11,260,000 tonnes at an average of 1.96% LREE inferred resource.

In June 2020, an updated MRE was prepared for Defense Metals by APEX (Raffle and Nicholls, 2020), with an effective date of June 27, 2020. The updated 2020 MRE included: 49% increase in the overall tonnage, 30% increase in the overall average grade, 730,000 tonnes increase of the inferred resource and conversion of 4,890,000 tonnes from previously inferred-defined resource to indicated resource. The mineral resource was reported at a cut-off grade of 1.5% LREE and comprised an average of 3.02% LREO indicated resource and an average of 2.52% LREE.

On November 24, 2021, as amended January 6, 2022, Defense Metals announced an updated MRE and Preliminary Economic Assessment (PEA) National Instrument (NI) 43-101 Technical Report for the Wicheeda REE Project. The NI 43-101 Technical Report, dated January 6, 2022, with an effective date of November 7, 2021, is titled “Preliminary Economic Assessment for the Wicheeda Rare Earth Element Project” and was published by SRK Consulting (Canada) Inc (McCarthy et al., 2022).

The MRE section of the 2022 “Preliminary Economic Assessment for the Wicheeda Rare Earth Element Project” report was completed by SRK Consulting and comprised a 5.0 million tonnes Indicated Mineral Resource, averaging 2.95% TREO (Total Rare Earth Oxides: CeO_2, La_2O_3, Nd_2O_3, Pr_6O_{11}, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_4O_7, Dy_2O_3 and Ho_2O_3) and a 29.5 million tonnes Inferred Mineral Resource, averaging 1.83% TREO, reported at a cut-off grade of 0.5% TREO within a conceptual Lerchs-Grossman (LG) pit shell. This MRE represented a 36% increase on a contained metal basis in comparison to the prior 2020 MRE (Raffle and Nicholls, 2020).

The 2022 Wicheeda REE Project Preliminary Economic Assessment technical report ("PEA") outlined a robust after-tax net present value (NPV@8%) of $517 million and an 18% IRR. This PEA contemplated an open pit mining operation with a 1.75:1 (waste: mill feed) strip ratio providing a 1.8 Mtpa ("million tonnes per year") mill throughput producing an average of 25,423 tonnes REO annually over a 16-year mine life. A Phase 1 initial pit strip ratio of 0.63:1 (waste: mill feed) would yield rapid access to higher grade surface mineralization in year 1 and payback of $440 million initial capital within 5 years (McCarthy et al., 2022).

In August 2023, APEX updated the Mineral Resource Estimate for the Wicheeda REE project. The 2023 MRE comprises a 6.4 million tonne Measured Mineral Resource, averaging 2.86% TREO CeO_2, La_2O_3, Nd_2O_3, Pr_6O_{11}, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_4O_7, Dy_2O_3 and Ho_2O_3); 27.8 million tonne Indicated Mineral Resource, averaging 1.84% TREO; and 11.1 million tonnes Inferred Mineral Resource, averaging 1.02% TREO, reported at a cutoff grade of 0.5% TREO within a conceptual Pseudoflow algorithm open pit shell provided in Table 14-17. The estimated metals include cerium (Ce), dysprosium (Dy), europium (Eu), gadolinium (Gd), holmium (Ho), lanthanum (La), neodymium (Nd), praseodymium (Pr), samarium (Sm), and terbium (Tb).

7. Geological Setting and Mineralization

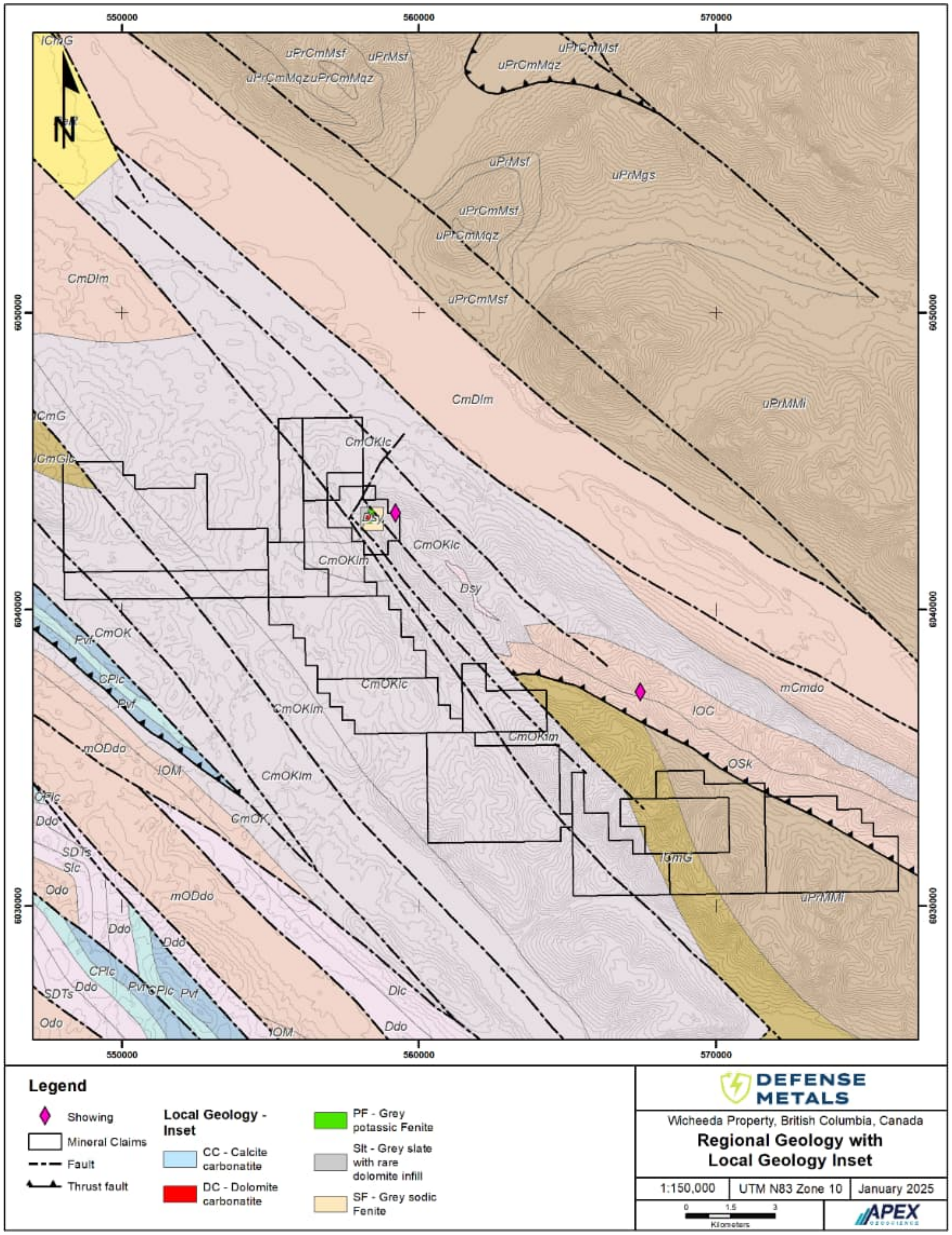
7.1 Regional Geology

The Wicheeda Carbonatite Complex is located in the Foreland Belt, a morphogeological belt of imbricated and folded miogeoclinal rocks that forms the eastern mountain ranges and foothills of the Canadian Cordillera (Gabrielse et al., 1991). In British Columbia, a small number of carbonatite-related complexes occur. These complexes are typically sub-circular to elongate in plan and commonly have well-developed metasomatic alteration haloes. Many of the intrusions that follow the trend of the Rocky Mountain Trench are Devonian to Mississippian in age. They were subjected to sub-greenschist facies metamorphism during the Columbian orogeny but behaved as inflexible and cohesive bodies during orogenesis and were rotated, tilted and/or transported eastwards in thrust panels (Pell, 1987). Well known carbonatite-alkaline complexes of the Foreland Belt include the Aley, Kechika, Ice River, Bearpaw and Rock Canyon (Pell, 1994).

The regional geology of the area was Mapped by Armstrong et al. (1969, McLeod Lake Map sheet) and Taylor and Stott (1979, Monkman Pass Map sheet). The regional bedrock underlying the Property and enclosing areas Mainly consists of limestone, Marble, siltstone, argillite and calcareous sedimentary rocks of the upper Cambrian to lower Ordovician Kechika Group. The Kechika Group sedimentary rocks are in fault contact with unassigned, Cambrian to Devonian carbonates, slates and siltstones to the east. To the west, the Kechika Group sedimentary rocks are in fault contact with Upper Proterozoic to Permian Gog Group quartzite rocks and Devonian to Permian unassigned felsic volcanic rocks. The Kechika Group lies on top of an erosional surface of uplifted Atan Group. Generally, the strata strike between 120 and 140° with steep dips to the southeast. The regional geology Map presented in Figure 7-1 is from a 1:250,000 scale digital compilation of the area (Digital Geology Map of British Columbia, BC MEMPR, Open file 2005-2).

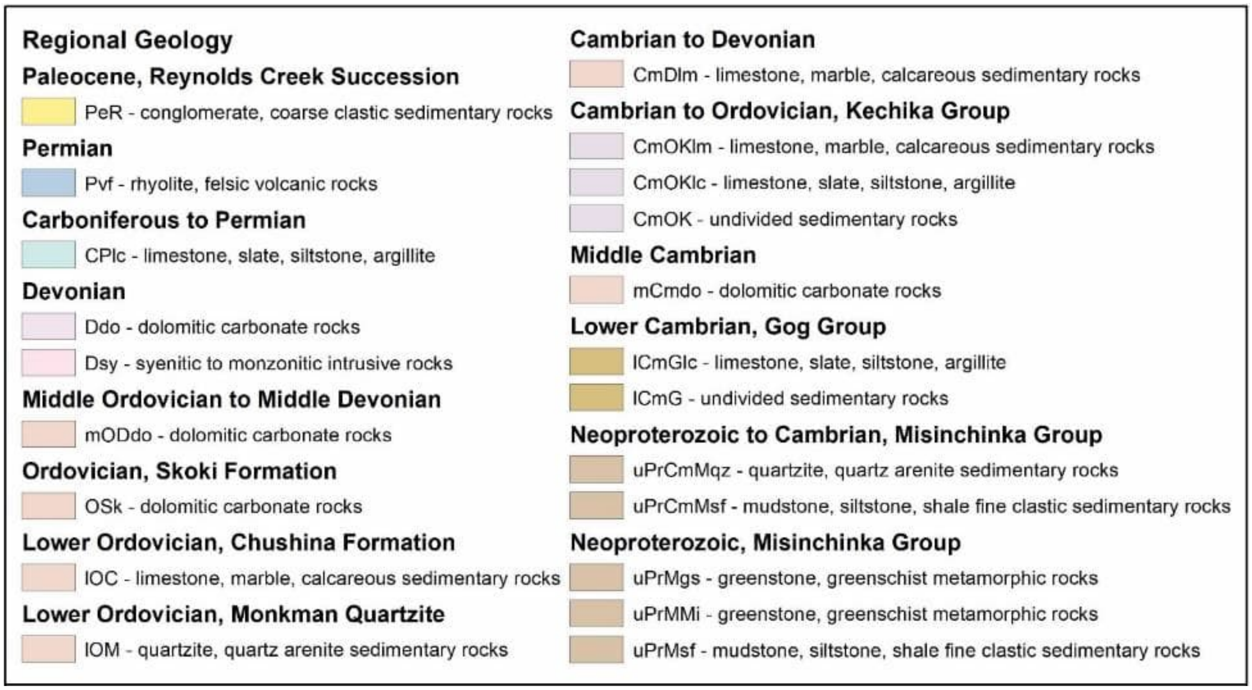
The complex is located within the McGregor Plateau between two dominant faults: the McLeod Lake fault to the west and the Rocky Mountain Trench to the east (Armstrong et al. 1969). The northwest-trending Rocky Mountain Trench parallel to the Parsnip River valley, a dominant structural and geographical feature, occurs east of the Property. Several other Major northwest trending faults occur in the area.

The age of carbonatite – alkaline complexes in British Colombia extend over 460 Ma. U-Pb and Th-Pb zircon dating defined three distinct ages of alkaline Magmatism; a Neoproterozoic (700-800 Ma), Late Cambrian (~500 Ma) and Upper Devonian to Lower Carboniferous (~340-360 Ma). The Neoproterozoic Magmatism corresponds to extensional settings during the initial break-up of the Rodina supercontinent while the other ages correspond to renewed extensional tectonics (Millonig et al., 2012).



Source: APEX, 2025

Figure 7-1: Regional Geology



Source: APEX, 2025

Figure 7-2: Regional Geology

7.2 Property Geology

Limited areas of the Wicheeda claim group have been covered by reconnaissance and/or grid-based bedrock Mapping. The REE-enriched carbonatites located on the Project are part of a narrow elongate, south-trending intrusive carbonatite-syenite complex cutting or occupying a structural panel within calcareous siltstones and limestones of the Cambrian to Ordovician Kechika Group. Some of the geological contacts observed in core are intrusive while others are almost certainly structural. The carbonatite complex extends southward from the south end of Wicheeda Lake for approximately 13 km.

Outcrop on a moderately steep, west-facing slope south of Wicheeda Lake, an area that coincides with part of the former ‘George’ grid, consists of a sequence of interbedded limestone, calcareous argillite and argillite with consistent northwest-trending attitudes and sub-vertical dips (Betmanis, 1987). A small intrusion cuts the sedimentary rocks in the southern part of the grid, just north of ‘A’ Creek. This feature was Mapped as syenite in 1986 by Betmanis (1987), although during a re-evaluation of the area (including trenching) the following year, it was concluded that the intrusion was a carbonatite (Lovang and Meyer, 1987).

Three types of narrow (0.5 m to 1.5 m), northwest-trending dykes were also observed in the gridded area, including: a K-feldspar phyric type with a fine-grained albite Matrix and abundant Fe-rich biotite; a blue sodalite-rich (as phenocrysts and Matrix) type, and; a feldspar and augite-phyric intermediate type with aphanitic groundmass that appears to be the youngest of the three varieties (Mader and Greenwood, 1988).

``

Outcrop in the area covered by the former ‘Lake’ grid is rare, but consists of strongly weathered, medium to coarse-grained calcite carbonatite, a band of fresh, fine-grained calcite carbonatite and related syenite were exposed in trenches (Mader and Greenwood, 1988).

7.3 Wicheeda Carbonatite and Mineralization

The Wicheeda Carbonatite is comprised Mainly of dolomite carbonatite (Figure 7.1), xenolithic dolomite carbonatite with varieties of Matrix to clast-supported fenite breccia where dolomite carbonatite occurs as the dominant Matrix component, and minor calcite carbonatite. This carbonatite body intrudes into syenite and minor Mafic dykes, limestone and calcareous sedimentary wall rocks. The upper part of the complex consists Mainly of dolomite carbonatite, brecciated dolomite carbonatite and lesser calcite carbonatite with minor fenitized limestone, Mafic dyke and syenite xenoliths whereas the lower part of the complex is weakly constrained by drilling and Mainly consists of xenolithic varieties of brecciated dolomite-carbonatite, fenitized limestone, syenite and country wall rocks.

The geometry of the Wicheeda carbonatite was originally interpreted to be sub-circular in plan (Lovang and Meyer, 1987; Mader and Greenwood, 1988). Subsequent modeling of the carbonatite body following diamond drilling showed a more oblong or lens-shaped with a long axis that is approximately north-south (Lane, 2009; 2010a), a subvertical dip and a plunge to the northwest. The Main carbonatite body was intersected over the extent of 215 m thick and is in fault contact with unaltered metasedimentary rocks of the Kechika Group on its western edge, and in intrusive contact with fenitized argillaceous limestones of the Kechika Group on is eastern Margin (Betmanis, 1987).

In their study of the Wicheeda Carbonatite on the Wicheeda Project, Trofanenko et al. (2016) proposed a preliminary model in which the carbonatite Magma exsolved a fluid which fenitized the host metasediments near the intrusion to potassic fenite and heated formational water distal to the intrusion, altering the metasedimentary rocks to sodic fenite. The REE were concentrated by Magmatic hydrothermal fluids, which partially dissolved the carbonatite, altered the dolomite, and lead to deposition of compositionally zoned dolomite and later bastnäsite-(Ce) and monazite-(Ce) in veins and vugs in response to cooling and an increase in pH.

REE mineralization at the Wicheeda carbonatite is zoned into high, moderate and low grade. High REE mineralization is directly related to dolomite-carbonatite and xenolithic dolomite carbonatite (defined as dolomite carbonatite contain >30% and less than 70 % xenolithic country rock dilution). Moderate REE mineralization is typically associated with mixed zones where xenolithic dolomite-carbonatite, fenitized limestone, syenite and Mafic dyke xenoliths exceed 30% and less than 70%. These mixed zones have the potential to add size to the deposit with more modest grades. Low REE mineralization is typically encountered in fresh and fenitized limestone, calcareous sedimentary rocks, syenite and fresh, weakly brecciated Mafic xenoliths.

Field observation of REE mineralization includes disseminated to clotty dark grey/blue – black columbite, disseminated, inclusion and fractured pyrochlore, rare fluorite and sphene / rutile and a combination of bastnäsite, monazite, and parasite-synchysite observed as aggregates and patches in veins and vugs. Vein-type mineralization was commonly noted in amorphous to coarse-grained dolomite-carbonate intersecting earlier fine-grained, dolomite carbonatite with disseminated fine-grained REE mineralization. Vein-type mineralization range in width from few centimeters to over a meter wide. On the other hand, vuggy and disseminated REE mineralization was noted in all lithologies, except unaltered limestone and calcareous sedimentary rocks, in variable percentages throughout the drill core.

8. Deposit Types

The principal deposit type of interest on the Wicheeda Property is a rare earth element enriched carbonatite deposit.

Carbonatites and carbonatite-associated deposits are mined worldwide for rare-earth elements (REEs) and Nb (e.g., Bayan Obo mine, Inner Mongolia; Kynicky et al., 2012, Araxa mine, Brazil; Biondi, 2005). Carbonatites can be economic targets for several other elements and mineral commodities including F, P, Al, Fe, Ti, Zr, C, Cu, Ni, Au, PGE, Ta, Sr, U, Th, phlogopite, vermiculite, olivine, lime, and barite (Marioano, 1989; Pell, 1996).

Carbonatites are defined by the International Union of Geological Sciences (IUGS) system of igneous rock classification as having more than 50% of primary carbonate minerals (such as calcite, dolomite, and ankerite) and less than 20% SiO2 (Le Maitre, 2002). Simandl and Paradis (2018) summarize the three main hypotheses regarding the origin of carbonatite melts:

- Immiscible separation of parental carbonated silicate magmas at crustal or mantle pressures
- Crystal fractionation of parental carbonated silicate magmas such as olivine melilitites or kamafugites
- Low-degree partial melting of carbonated mantle peridotite below 70 km depth.

Hypotheses involving a possible derivation of carbonatites from the earth’s crust (Lentz 1999; Ferrero et al. 2016) or from the Earth’s mantle with some crustal contribution (Cheng et al. 2017; Song et al 2017) have also been proposed. It is likely that not all carbonate forming melts are of the same origin.

Most carbonatites and alkaline-carbonatite complexes are emplaced in continental settings in Archean and Proterozoic rocks, or in Phanerozoic rocks underlain by a Precambiran basement. They form in extensional tectonic settings along major linear trends related to large-scale intra-plate fracture zones, in association with doming features or in relation to slab windows in subducting plates (Simandl and Paradis, 2018)

Birkett and Simandl (1999) provide the following concise description of carbonate associated deposits:

Carbonatites are small, pipe-like bodies, dikes, sills, and small plugs or irregular masses. The typical pipe-like bodies have subcircular or elliptical cross sections and are up to 3-4 km in diameter. Magmatic mineralization within pipe-like carbonatites is commonly found in crescent-shaped and steeply-dipping zones. Metasomatic mineralization occurs as irregular forms or veins. A fenitization halo (alkali metasomatized country rocks) commonly surrounds carbonatite intrusions; alteration mineralogy depends largely on the composition of the host rock. Typical minerals are sodic amphibolite, wollastonite, nepheline, mesoperthite, antiperthite, aegirine-augite, pale brown biotite, phlogopite and albite. Most fenites are zones of desilicification with the addition of Fe^{3+} and K is also common.

``

The REE minerals form pockets and fill fractures within ferrocarbonaitite bodies. Pyrochlore is disseminated, apatite can be disseminated or semi-massive; bastnaesite occurs as disseminated to patchy accumulations; fluorite forms as veins and masses, hematite is semi-massive disseminations, and chalcopyrite and bornite are found in veinlets. Magmatic ore mineralogy consists of one or more of: bastnaesite, pyrochlore, apatite, anatase, zircon, baddeleyite, magnetite, monazite, parasite and fersmite.

In the Canadian Cordillera, carbonatites were emplaced episodically, at ca. 810-700, 500, and 360-330 Ma, forming part of the British Columbia alkaline province, which defines a long (~1000 km), narrow (200 km) orogeny-parallel belt. The ca. 810-700 Ma and 500 Ma carbonatites were injected during protracted breakup of the supercontinent Rodinia and passive margin development on the western flank of Laurentia. In contrast to these the 360-330 Ma carbonatites were emplaced near the continental margin during subduction rather than in the cratonic interior during continent-building (Rukhlov et al., 2018). The carbonatites on the Wicheeda Project are believed to be part of this latter group.

In their study of the Wicheeda Main Zone, Trofanenko et al. (2014; 2016) proposed a preliminary model in which the carbonatite magma exsolved a fluid which fenitized the host metasediments near the intrusion to potassic fenite and heated formational water distal to the intrusion, altering the metasedimentary fluids, which partially dissolved the carbonatite, altered the dolomite, and lead to deposition of compositionally zoned dolomite and later bastnaesite-(Ce) and monazite-(Ce) in veins and vugs in response to cooling and an increase in pH.

‘

``

9. Exploration

A summary of historical exploration completed on the Project is provided in Section 6: History. Recent surface exploration completed by Defense Metals during 2023 includes: geological mapping, ground and airborne geophysical surveys, a LiDaR survey, diamond and geotechnical drilling, test pitting and updating the Wicheeda Deposit 3D geological model and MRE.

9.1 2023 Outcrop Geological Mapping and Test Pitting

During the summer of 2023, an outcrop mapping program was conducted within the Property. The objective of the program was to generate a lithological and structural map and to correlate surficial data with the downhole litho-structural data at the Wicheeda deposit.

The result of outcrop mapping and test pitting confirmed that outcrops in the lower elevation areas of the Property are scare. Outcrop mapping recorded multiple lithologies and inferred fold structures at the Property scale and improved the correlation of downhole and surficial lithologies at the deposit scale. Lithologies within the property were simplified into three general types: phyllite, limestone and mudstone interbedded with siltstone.

Phyllite rocks are characterized by variable deformation ranging from sub-mm laminae of moderately foliated to strongly foliated sericite. Phyllites are typically fissile, light brown to shiny beige (weathered) and light grey where fresh. Phyllite layers may contains mm-scale beds of siltstone-mudstone. Limestone layers are characterized by cm-thick, grey to pistachio green colour layers interbedded with minor siltstone and mudstone. Siltstone interbedded with mudstone was commonly mapped across the property and characterized by light grey, cm to dm-thick siltstone interbedded dark grey, mm scale mudstone.

The sedimentary packages follow a general northwest trend with moderate to subvertical dips. Along the northwestern claim boundary, anastomosing beds were mapped along an interpreted anticline with open folds. As defined by drilling and the 2023 mapping program, the Wicheeda carbonatite body stretches over 400 m along a northwest-southeast strike, 220 m east-west width and up to 250 m deep in the central down-dip portion of the body. Property-scale outcrop mapping is illustrated in Figure 9-1 and detailed surface outcrop and interpreted geology map Figure 9-2.

``

9.2 Ground Geophysics

During the July 2023, a ground magnetometer and radiometric survey was conducted over the Wicheeda deposit covering an area of approximately 800 m x 900 m over the main deposit. The ground survey comprised a total of 20 line-km along 50 m spaced, and locally 25 m infill, east-west oriented survey lines.

A NUVIA Dynamics PGIS-2 Gamma-ray spectrometer, equipped with a 0.347 Litre NaI detector and 512-channel resolution ADC was used. This backpack-mounted unit was operated in tandem with a ground magnetic survey utilizing a proton GEM-GSM-19T magnetometer. The GEM GSM-19T magnetometer with integrated GPS time synchronization, uses proton precession technology with absolute accuracy of ±0.20 nT and sensitivity of 0.15 nT at 1 Hz for efficient data collection. Base station magnetic data were recorded on internal solid-state memory and downloaded onto a field laptop using a serial cable and GEMLink 5.4 software. Profile plots of the base station readings were generated and reviewed at the end of each day.

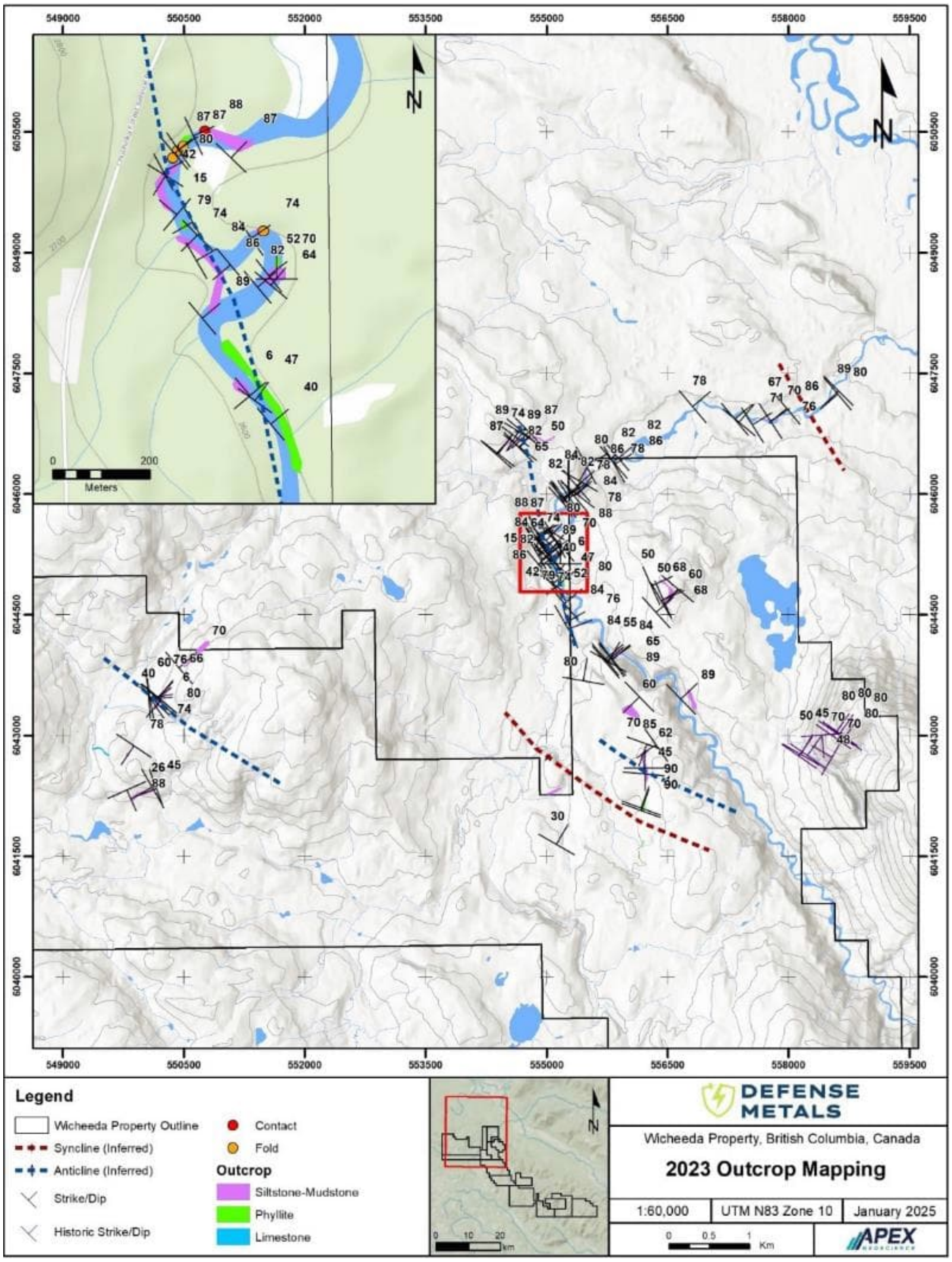
Data was collected at walking speed, even in challenging bush terrain, without compromising data integrity. Additionally, data was automatically synchronized with high-resolution integrated GPS, ensuring both time and location accuracy.

The spectrometer's self-stabilizing capabilities on natural radioactive elements such as K, U, and Th eliminated the need for frequent recalibration, assuring reliable and accurate gamma-ray measurements. Given that gamma rays are highly attenuated by overburden (approximately 90% attenuation at 20-30 cm overburden depth) ground radiometric surveys are only likely to detect outcropping or very near surface sources.

Results of this ground geophysical survey indicated anomalously higher magnetic values lie in the periphery of the dolomite carbonatite, in the contact zone to the syenite body Figure 9-3 illustrates the Residual Magnetic Intensity (RMI) Total Horizonal Gradient (THG). Additionally, two previously unknown linear radiometric anomalies were identified, each approximately 40 m in width and extending approximately 250 m northwest from the main body of the Wicheeda REE deposit (Figure 9-4).

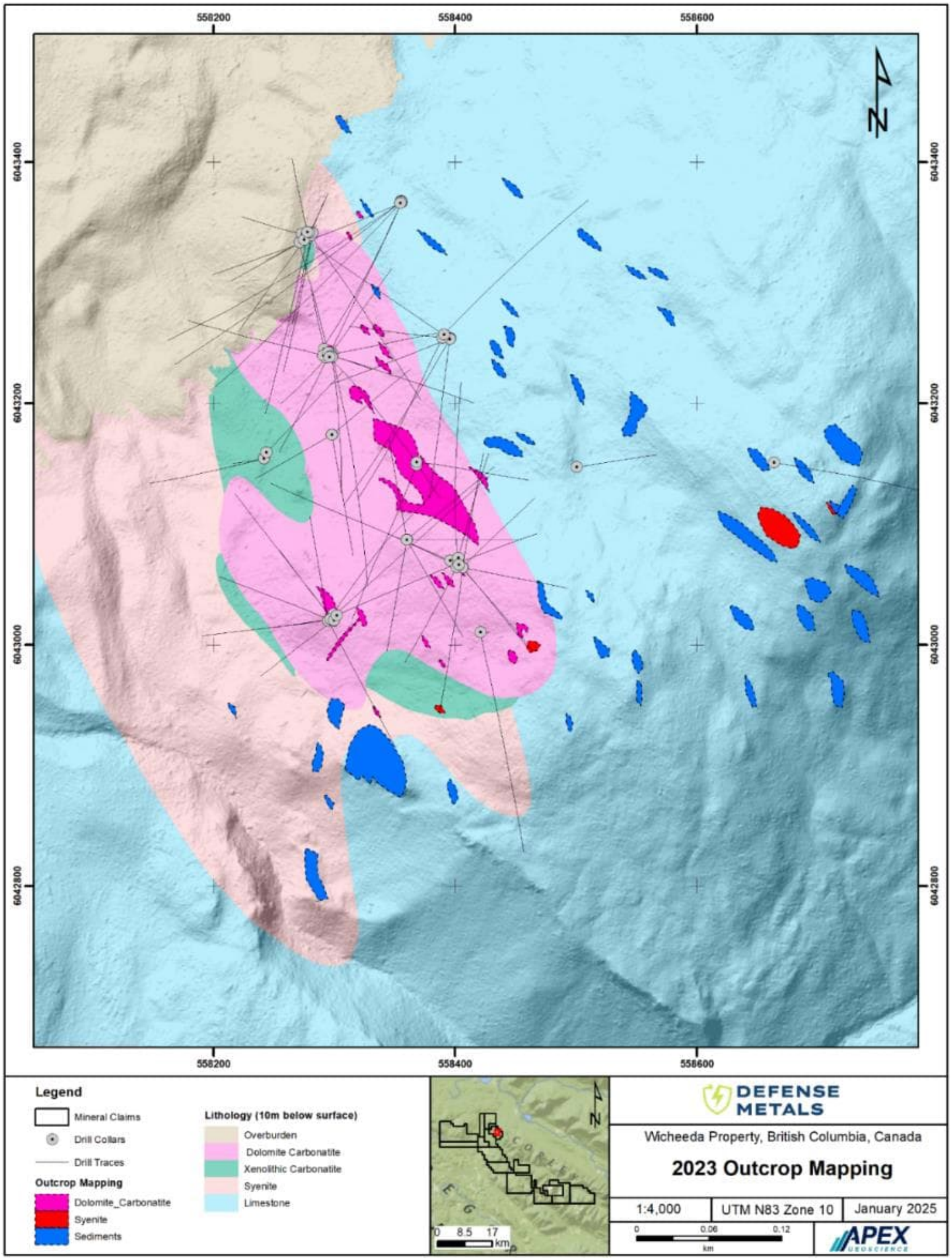
In the opinion of Mr. Reid the exploration programs conducted on the project, as outlined in Section 6 and in this Section 9 are appropriate for the style of mineralization identified. The current degree of geological knowledge and understanding of mineralization is considered adequate at this stage of exploration.

``



Source: APEX, 2025

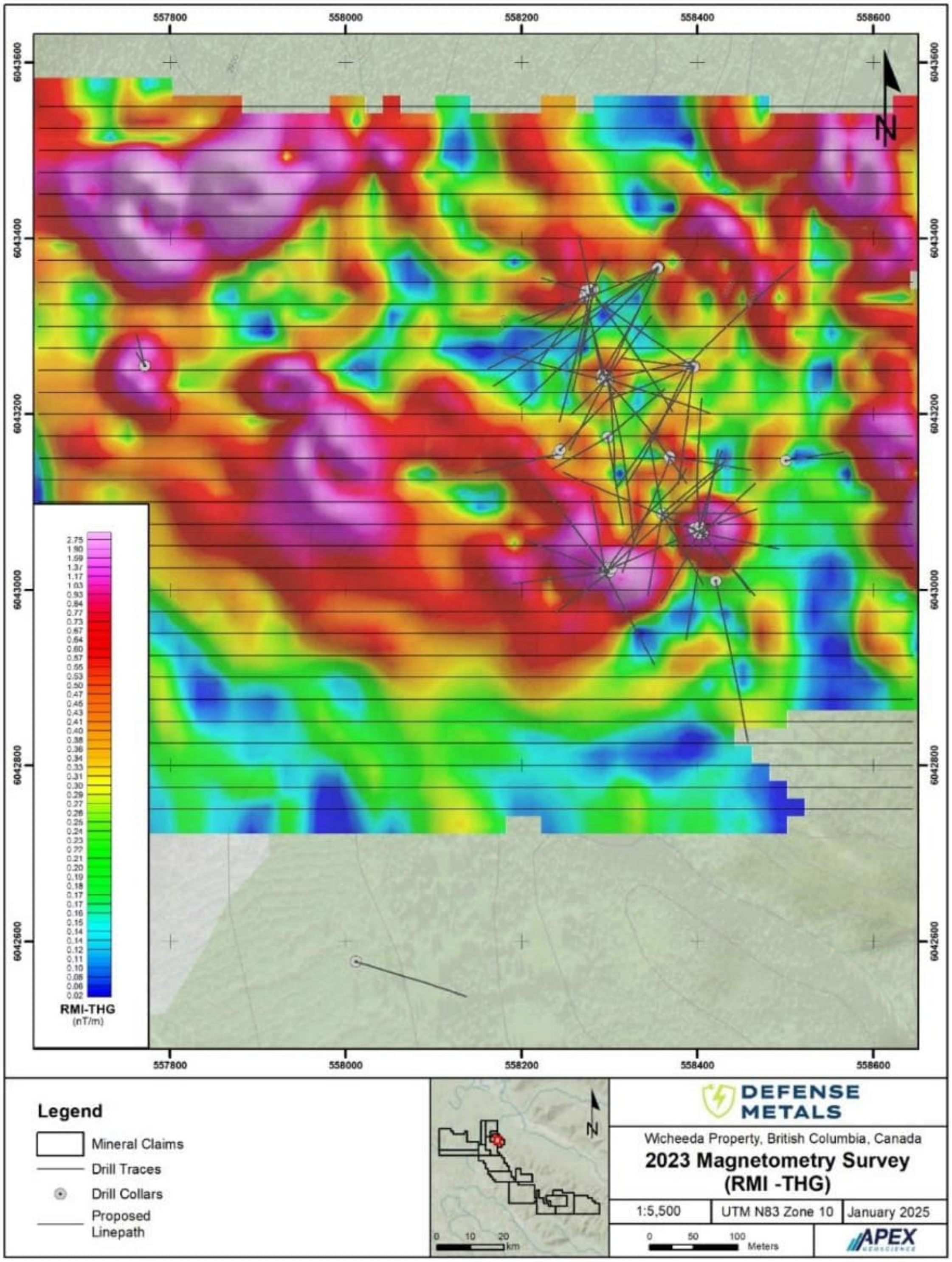
Figure 9-1: Litho-structural Outcrop Mapping - Wicheeda Property



Source: APEX, 2025

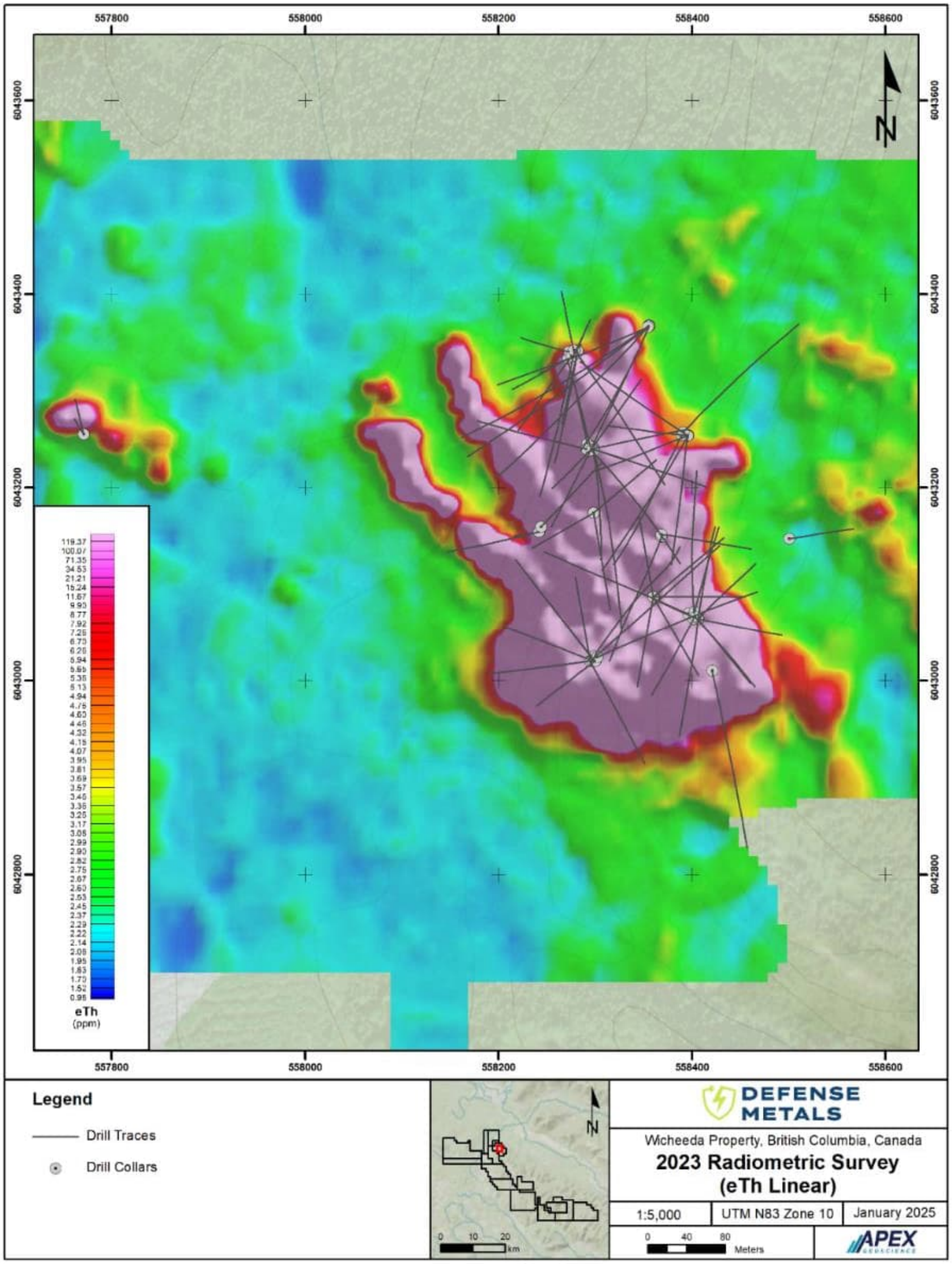
Figure 9-2: Lithological Mapping and Geology Model Projection - Wicheeda Deposit

``



Source: APEX, 2025

Figure 9-3: Ground Geophysics: Magnetometry, (RMI – THG)



Source: APEX, 2025

Figure 9-4: Ground Geophysics: Radiometric Survey, equivalent Thorium (eTH)

``

9.3 Airborne Geophysics

During October, 2023 a heli-borne geophysical survey was carried out over the Wicheeda Property. The survey comprised a total of 503 line-km covering an area of 45.3 km². The survey was flown at 100 m spacing at a heading of 045°/225° and a 1000 m tie lines at a heading of 135°/315°. The proposed survey height was 50 m constant height above ground level (Table 9-1 and Figure 9-5).

The survey was flown with a helicopter mounted Scintrex CS-3 magnetometer. Temporal variations of Earth's magnetic field, particularly diurnal, were monitored and recorded by two GEM GSM-19T base station magnetometers during surveying. The base stations were in an area with low magnetic gradient (i.e., away from electric power transmission lines and ferrous objects) for optimum survey data integrity.

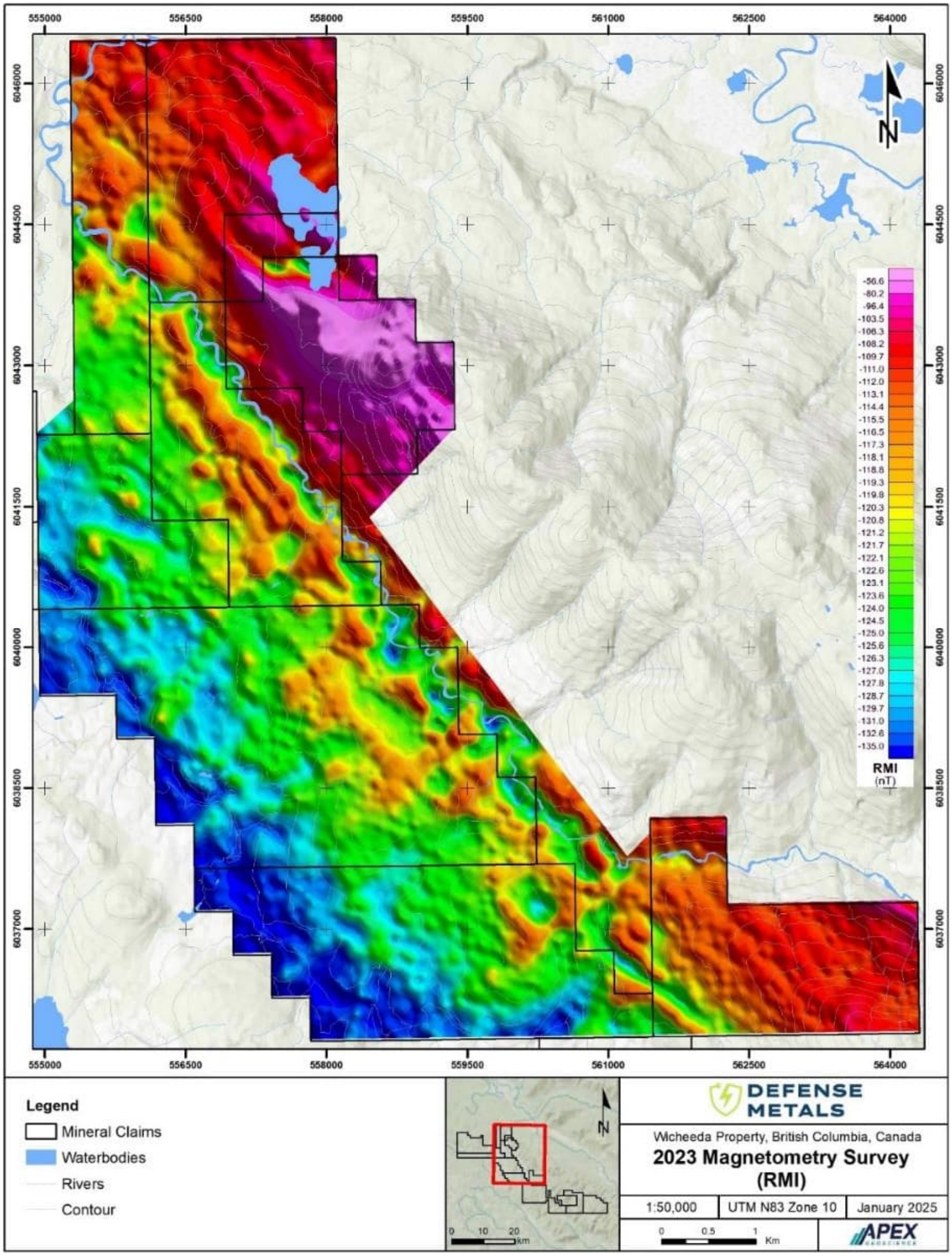
Table 9-1: 2023 Airborne Survey Specifications

Survey Block	Area (km2)	Line Type	Line Orientation (UTM grid)	Line Spacing (m)	Lines Planned	Lines Completed	Total Planned Line km	Total Actual km Flown
Wicheeda	45.3	Survey	045°/225°	100	138	138	453	457
		Tie	135°/315°	1000	8	8	46	46
		Total:			**146**	**146**	**499**	**503**

Source: APEX, 2025

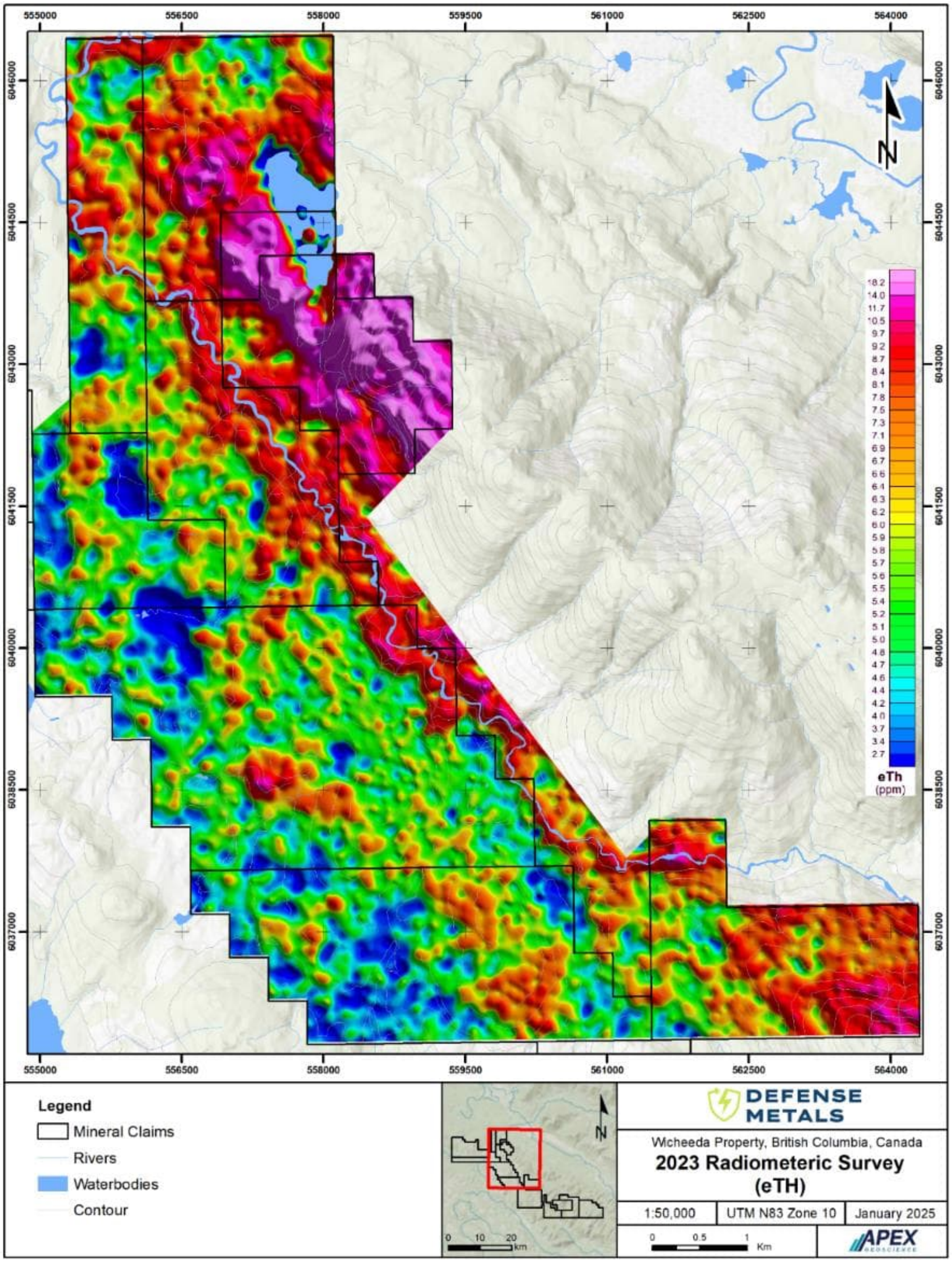
Gamma radiation data were collected by a Medusa GR-820.1 gamma ray spectrometer. The GR-820.1 is a self-calibrating, fully integrated gamma detection system containing downward and upward looking thallium-activated synthetic sodium iodide crystals recording total gamma count, differentiation of individual radioelements (K, U, and Th), cosmic radiation.

Results of the Airborne geophysical survey confirms the anomalous higher magnetic values over the syenite-carbonatite complex. The anomalous radiometric values highlighted the Wicheeda deposit and new radiometric anomalies west and northwest of the Wicheeda deposit. Residual magnetic intensity (RMI) and equivalent thorium (eTH) are illustrated in Figure 9-5 and Figure 9-6 respectively.



Source: APEX, 2025

Figure 9-5: Airborne Geophysics: Magnetometry (RMI - THG)



Source: APEX, 2025

Figure 9-6: Airborne Geophysics: Radiometric Survey, equivalent Thorium (eTH)

``

9.4 2023 LiDaR Survey

During October, 2024, an airborne Light Detection and Ranging (LiDaR) survey was flown over the Wicheeda Project. The LiDaR data were collected at an average rate of ≥ 8 points per square meter with aerial photography for a 15 cm orthophoto. The LiDaR data was captured using the RIEGL LMS-Q1560 mounted in Cessna 206. The total area covered was 74 km^2 on an east – west lines and a single north - south tie line. The flight lines total length is 155 km over 13 lines and 319 photos. The data resolution is 0.1 meter. Calibration was carried out before the start of the survey.

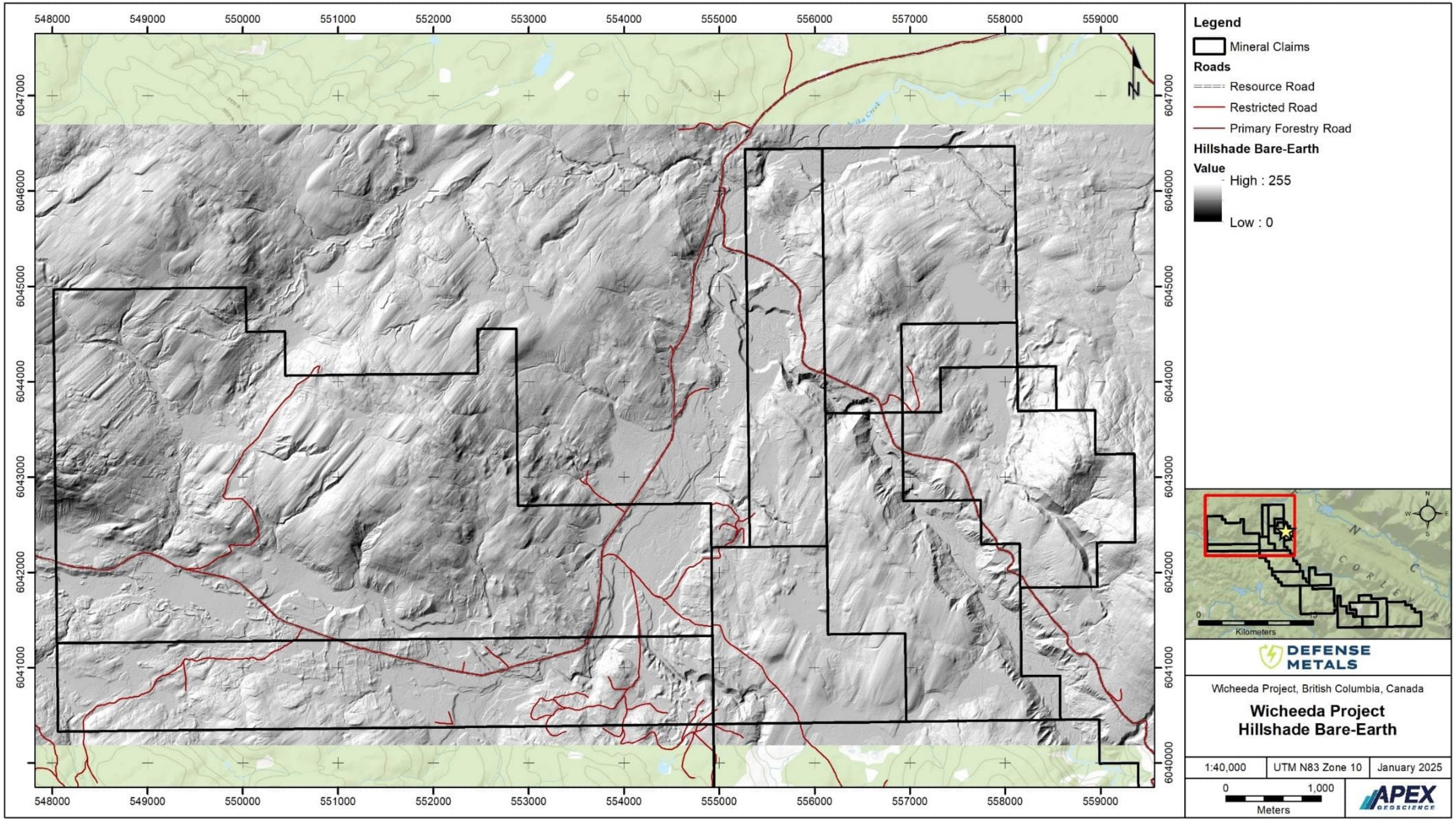
To achieve a minimum of 8 pulses/m^2, a double swathes with > 50% overlap > 4 pulses per meter was flown for an aggregated coverage > 8 pulses per meter across the entire project to ensure a void free dataset.

Using a pre-established tile grid, individual “tiles” of data were clipped out of the larger data set. These files were written in LAS 1.4 format, which is the industry standard binary format. 1km X 1km grid tiles were created for post processing with a 5% overlap which would be clipped to the requested grid size at the end of processing.

The LiDaR derived hillshade bare earth model in Figure 9-7 presents a detailed image of the terrain highlighting drainage, wetlands, man-made structures, and glacial geomorphology present within the Project area.

9.5 QP Opinion

In the opinion of Mr. Reid, the exploration programs conducted on the project, as outlined, are appropriate for the style of mineralization identified. The current degree of geological knowledge and understanding of mineralization is considered to be adequate for a PFS level study.



Source: APEX, 2025

Figure 9-7: 2023 LiDaR Survey – Hillshade Bare-Earth

``

10. Drilling

A description of the historical drilling completed within the Property, as it relates to the current mineral resource estimate with respect to the Wicheeda Property (this Report) is considered relevant. A detailed discussion of historical drilling completed on the Property is included in Section 6.3 and Section 6.6 and it is summarized below.

Historical drilling on the Property has been conducted by Spectrum Mining during 2008 and 2009 (Section 10.1). Diamond drilling completed by Defense Metals during 2019 (Section 10.2), 2021 (Section 10.3), 2022 (Section 10.4) and 2023 (Section 10.5) comprised 16,762 m in 66 drill holes. In total, 85 drillholes, totalling 14,066 m, were completed between 2019 and 2023 on the Wicheeda Property. Table 10-1 provides drill hole locations and header. Information on these drill holes has been compiled into the Project drillhole database. Wicheeda drill hole locations are shown in Figure10-1. Drill hole splits are stored at a core storage facility near Prince George, British Columbia. Half core or all core of some drill holes have been consumed for metallurgical studies.

In the opinion of the author, there are no sampling or recovery factors that could materially impact the accuracy and reliability of the drill results.

The Wicheeda Project diamond drill programs have shown REE-enriched carbonatite rocks of the Wicheeda Deposit are part of a narrow, elongate, northwest-southeast trending intrusive carbonatite-syenite sill complex. The carbonatite is intruded into syenite, mafic dykes, limestone and calcareous sedimentary wall rocks. The Wicheeda REE Deposit has dimensions of approximately 400 m north-south by 100-250 m east-west. Diamond drilling data supports the interpretation of a moderately north-northeast dipping, shallowly north plunging, layered sill complex having syenite at its base, overlain by hybrid matrix to clast-supported limestone or mafic intrusive xenolithic carbonatite, and finally significantly REE-bearing dolomite-carbonatite rocks, which form the main body of the Wicheeda REE Deposit outcropping at surface. This layered sill complex occurs within primarily barren limestone waste rock (Figure 10-2 to Figure 10-5).

Table 10-1: Wicheeda Project Drill Hole Locations

Hole ID	East UTM N83z10	North UTM N83z10	Elevation (m)	Azimuth	Dip	Total Depth (m)	Year	Core Size	Pad	Drill Contractor
WI08-01	558295	6043020	1047.7	152	-50	185.62	2008	NQ	2008-1	Falcon Drilling
WI08-02	558295	6043020	1047.7	0	-90	215.8	2008	NQ	2008-1	Falcon Drilling
WI08-03	558295	6043020	1047.7	48	-54	305.41	2008	NQ	2008-1	Falcon Drilling
WI08-04	558295	6043020	1047.7	350	-55	154.23	2008	NQ	2008-1	Falcon Drilling
WI09-05	558360	6043087	1084	0	-90	56.39	2009	HQ	Site A	Falcon Drilling
WI09-06	558360	6043087	1084	50	-50	147.83	2009	HQ	Site A	Falcon Drilling
WI09-07	558360	6043087	1084	150	-50	145.39	2009	HQ	Site A	Falcon Drilling
WI09-08	558360	6043087	1084	190	-50	146.91	2009	HQ	Site A	Falcon Drilling
WI09-09	558360	6043087	1084	90	-50	148.13	2009	HQ	Site A	Falcon Drilling
WI09-10	558360	6043087	1084	350	-55	148.13	2009	HQ	Site A	Falcon Drilling
WI09-11	558298	6043174	1032	0	-90	146.61	2009	HQ	Site B	Falcon Drilling
WI09-12	558298	6043174	1032	240	-60	146.61	2009	HQ	Site B	Falcon Drilling
WI09-13	558298	6043174	1032	50	-55	147.52	2009	HQ	Site B	Falcon Drilling
WI09-14	558298	6043174	1032	170	-45	144.17	2009	HQ	Site B	Falcon Drilling
WI09-15	557611	6043637	940	100	-90	101.8	2009	HQ	Site D	Falcon Drilling
WI09-16	557611	6043637	940	170	-50	95.71	2009	HQ	Site D	Falcon Drilling
WI09-17	557611	6043637	940	350	-50	148.13	2009	HQ	Site D	Falcon Drilling
WI09-18	557771	6043255	915	328	-70	53.95	2009	HQ	Site C	Falcon Drilling
WI09-19	557771	6043255	915	346	-50	57.91	2009	HQ	Site C	Falcon Drilling
WI19-20	558299	6043020	1051	230	-55	136.4	2019	NQ	2008-1	Falcon Drilling
WI19-21	558299	6043020	1051	290	-55	179.35	2019	NQ	2008-1	Falcon Drilling
WI19-22	558406	6043064	1124	100	-90	127.15	2019	NQ	2019-2	Falcon Drilling
WI19-23	558406	6043064	1124	100	-45	126	2019	NQ	2019-2	Falcon Drilling
WI19-24	558406	6043064	1124	140	-45	122.95	2019	NQ	2019-2	Falcon Drilling
WI19-25	558406	6043064	1124	185	-45	175.65	2019	NQ	2019-2	Falcon Drilling
WI19-26	558406	6043064	1124	295	-65	156.3	2019	NQ	2019-2	Falcon Drilling
WI19-27	558406	6043064	1124	10	-45	139.85	2019	NQ	2019-2	Falcon Drilling

``

Hole ID	East UTM N83z10	North UTM N83z10	Elevation (m)	Azimuth	Dip	Total Depth (m)	Year	Core Size	Pad	Drill Contractor
WI19-28	558406	6043064	1124	45	-45	117.15	2019	NQ	2019-2	Falcon Drilling
WI19-29	558396	6043254	1082	190	-45	184.05	2019	NQ	2019-3	Falcon Drilling
WI19-30	558396	6043254	1082	250	-55	179.5	2019	NQ	2019-3	Falcon Drilling
WI19-31	558396	6043254	1082	275	-55	138.5	2019	NQ	2019-3	Falcon Drilling
WI19-32	558396	6043254	1082	300	-55	224.7	2019	NQ	2019-3	Falcon Drilling
WI21-33	558292	6043245	1017	350.2	-80	274.4	2021	NQ	2021-08	Gateway Drilling
WI21-34	558298	6043242	1017	39.4	-55	150.9	2021	NQ	2021-08	Gateway Drilling
WI21-35	558299	6043241	1017	79.8	-55	172.85	2021	NQ	2021-08	Gateway Drilling
WI21-36	558297	6043240	1016	108.1	-80	197.25	2021	NQ	2021-08	Gateway Drilling
WI21-37	558297	6043240	1016	108.1	-45	175.9	2021	NQ	2021-08	Gateway Drilling
WI21-38	558293	6043238	1015	219.4	-70	148.5	2021	NQ	2021-08	Gateway Drilling
WI21-39	558291	6043240	1009	284.6	-60	224.8	2021	NQ	2021-08	Gateway Drilling
WI21-40	558295	6043243	1015	345	-65	209.15	2021	NQ	2021-08	Gateway Drilling
WI21-41	558278	6043339	993	26.3	-55	68.3	2021	NQ	2021-09	Gateway Drilling
WI21-42	558278	6043337	996	26.3	-70	93	2021	NQ	2021-09	Gateway Drilling
WI21-43	558278	6043339	994	45	-85	124.1	2021	NQ	2021-09	Gateway Drilling
WI21-44	558272	6043334	1001	240	-60	125.6	2021	NQ	2021-09	Gateway Drilling
WI21-45	558272	6043334	1001	240	-75	114.35	2021	NQ	2021-09	Gateway Drilling
WI21-46	558282	6043342	986	190	-50	182.2	2021	NQ	2021-09	Gateway Drilling
WI21-47	558273	6043341	997	280	-60	98.36	2021	NQ	2021-09	Gateway Drilling
WI21-48	558281	6043339	994	145	-45	220.45	2021	NQ	2021-09	Gateway Drilling
WI21-49	558282	6043342	986	190	-70	228.65	2021	NQ	2021-09	Gateway Drilling
WI21-50	558275	6043336	990	215	-50	149.7	2021	NQ	2021-09	Gateway Drilling

``

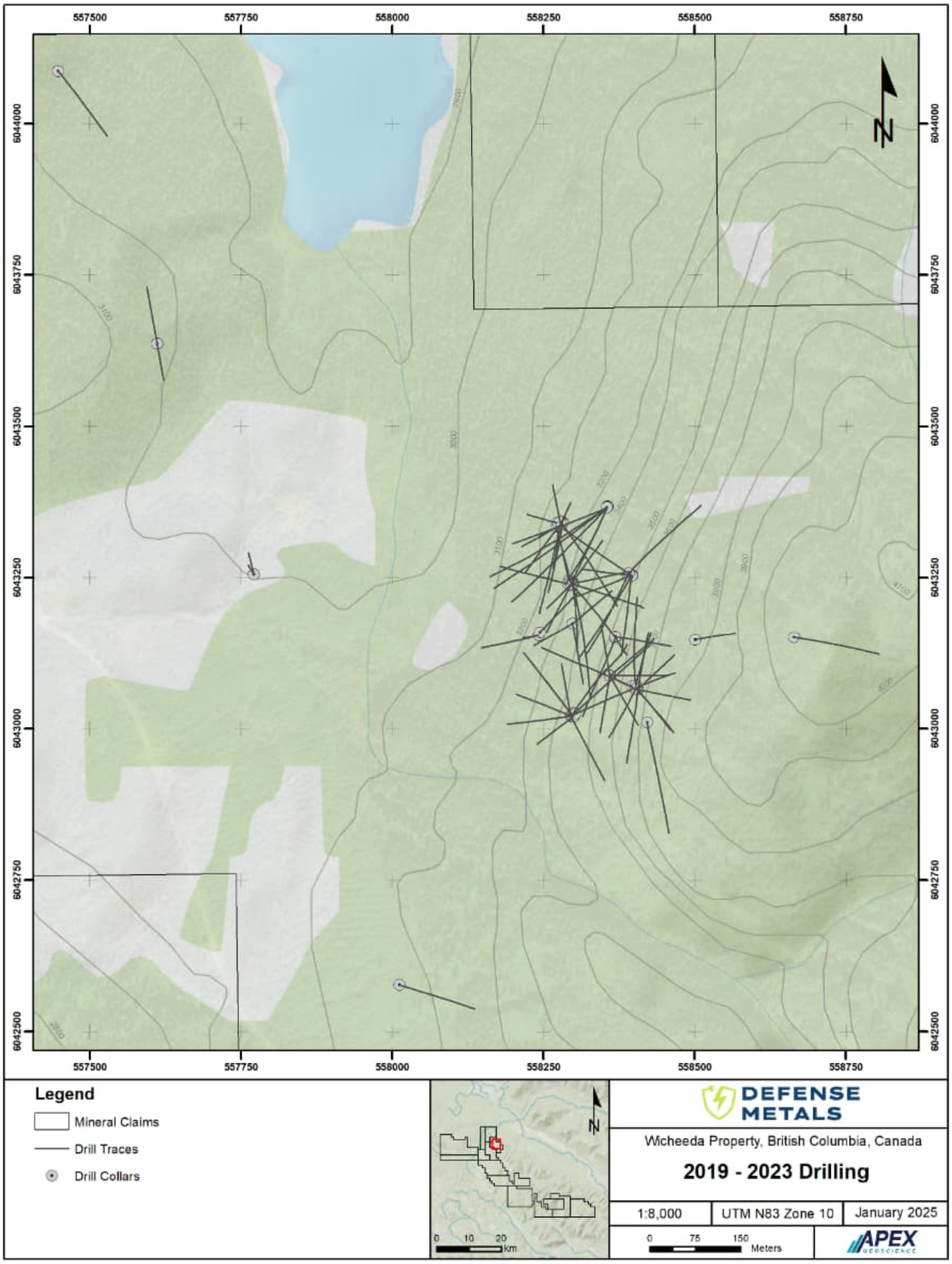
Hole ID	East UTM N83z10	North UTM N83z10	Elevation (m)	Azimuth	Dip	Total Depth (m)	Year	Core Size	Pad	Drill Contractor
WI21-51	558299	6043026	1066	30	-55	290.85	2021	NQ	2008-01	Discovery Drilling
WI21-52	558297	6043021	1064	260	-45	150.9	2021	NQ	2008-01	Discovery Drilling
WI21-53	558297	6043021	1064	260	-65	111.9	2021	NQ	2008-01	Discovery Drilling
WI21-54	558300	6043020	1043	320	-45	187.5	2021	NQ	2008-01	Discovery Drilling
WI21-55	558300	6043020	1043	320	-65	178.95	2021	NQ	2008-01	Discovery Drilling
WI21-56	558302	6043024	1060	65	-45	250	2021	NQ	2008-01	Discovery Drilling
WI21-57	558396	6043070	1125	290	-50	263.9	2021	NQ	2019-06	Discovery Drilling
WI21-58	558402	6043070	1128	355	-60	301.5	2021	NQ	2019-06	Discovery Drilling
WI21-59	558403	6043072	1130	15	-70	268.9	2021	NQ	2019-06	Discovery Drilling
WI21-60	558402	6043064	1128	205	-55	154.9	2021	NQ	2019-06	Discovery Drilling
WI21-61	558397	6043253	1089	210	-50	248.5	2021	NQ	2019-07	Discovery Drilling
WI-22-62	558356	6043367	1026.1	204	-50	340	2022	HQ3	2021-10	Radius Drilling
WI-22-63	558356	6043367	1026.1	204	-60	322	2022	HQ3	2021-10	Radius Drilling
WI-22-64	558390	6043254	1081.7	204	-65	384.3	2022	HQ3	2019-07	Radius Drilling
WI-22-65	558012	6042577	986.8	106	-60	266	2022	HQ3	2022-11	Radius Drilling
WI-22-66	557447	6044087	915.7	141	-60	264.8	2022	HQ3	2022-12	Radius Drilling
WI-22-67	558279	6043342	990.1	197	-60	320	2022	HQ3	2021-09	Radius Drilling
WI-22-68	558355	6043368	1025.6	219	-55	395	2022	HQ3	2021-10	Radius Drilling
WI-22-69	558355	6043368	1023.9	229	-50	353	2022	HQ3	2021-10	Radius Drilling
WI-22-70	558355	6043367	1025.9	235	-55	386	2022	HQ3	2021-10	Radius Drilling
WI-22-71	558298	6043239	1010.1	163	-50	360	2022	HQ3	2021-08	Radius Drilling
WI-22-72	558298	6043238	1010.4	167	-70	374	2022	HQ3	2021-08	Radius Drilling
WI-22-73	558296	6043238	1009	134	-60	308	2022	HQ3	2021-08	Radius Drilling
WI-22-74	558403	6043066	1131.8	139	-65	251.5	2022	HQ3	2019-06	Radius Drilling
WI-22-75	558501	6043147	1160.4	79	-70	199	2022	HQ3	2022-13	Radius Drilling

``

Hole ID	East UTM N83z10	North UTM N83z10	Elevation (m)	Azimuth	Dip	Total Depth (m)	Year	Core Size	Pad	Drill Contractor
WI-22-76	558355	6043366	1026.2	242	-55	284	2022	HQ3	2021-10	Radius Drilling
WI-22-77	558278	6043342	997.7	348	-70	171	2022	HQ3	2021-09	Radius Drilling
WI-22-78	558368	6043149	1083.2	199	-60	300.5	2022	HQ3	2022-14	Radius Drilling
WI-22-79	558368	6043151	1084	95	-65	231	2022	HQ3	2022-14	Radius Drilling
WI23-80	558664	6043151	1255.1	99	50	226	2023	HQ	2023-15	Discovery Drilling
WI23-81	558421	6043010	1142.7	172.1	45	271	2023	HQ	2023-16	Discovery Drilling
WI23-82	558242	6043154	997.6	253.7	55	172	2023	HQ	2019-07	Discovery Drilling
WI23-83	558391	6043257	1084.1	41.7	50	251	2023	HQ	2023-18	Discovery Drilling
WI23-84	558242	6043154	997.6	253.7	80	131	2023	HQ	2023-19	Discovery Drilling
WI23-85	558244	6043159	997.9	45.8	45	131	2023	HQ	2023-20	Discovery Drilling
Total number of drill holes (2008 – 2023)						85				
Total meterage (2008 – 2023)						16,762				

Source: APEX, 2025

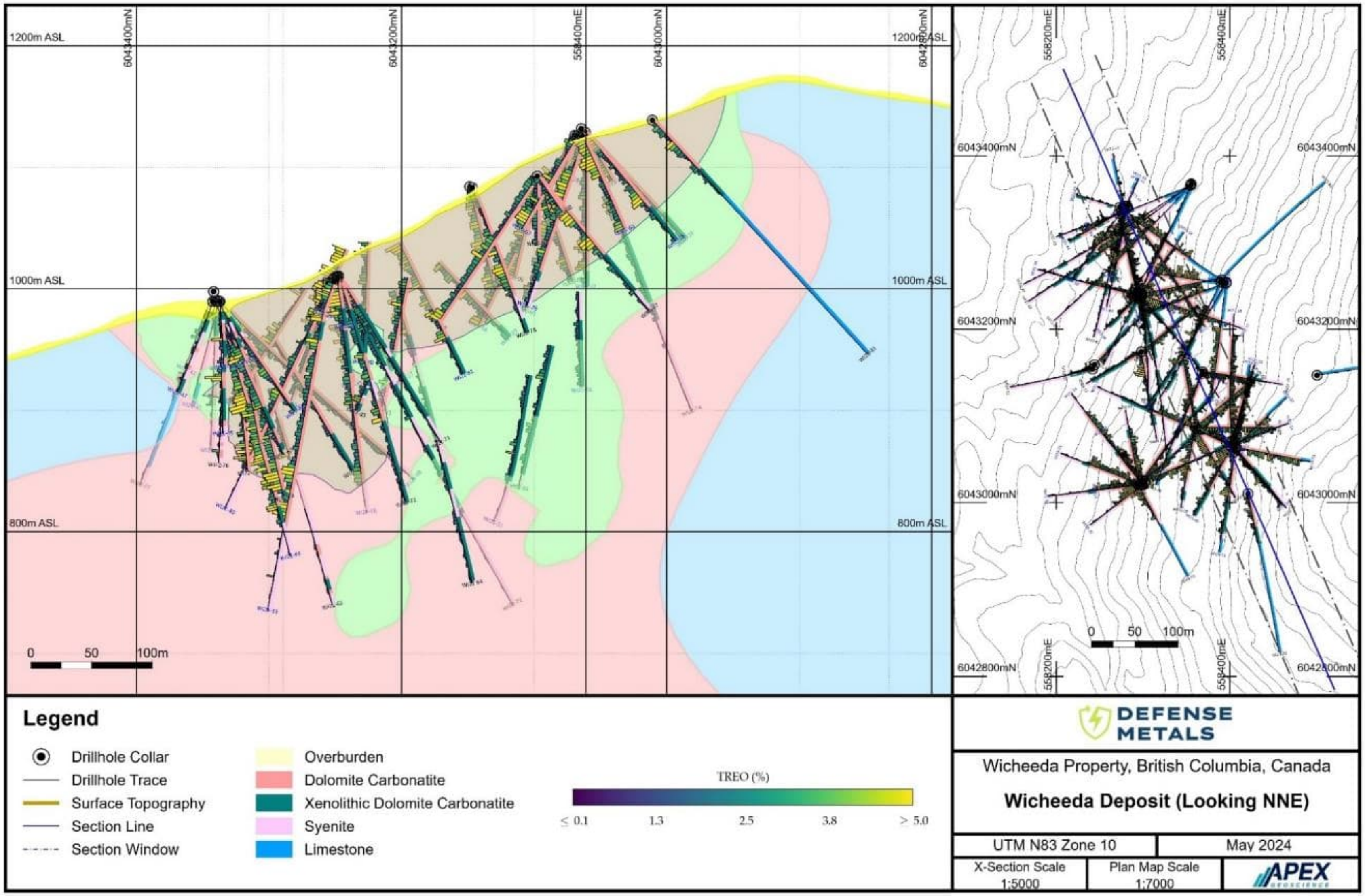
``



Source: APEX, 2025

Figure10-1: Wicheeda Property Drill Hole Locations

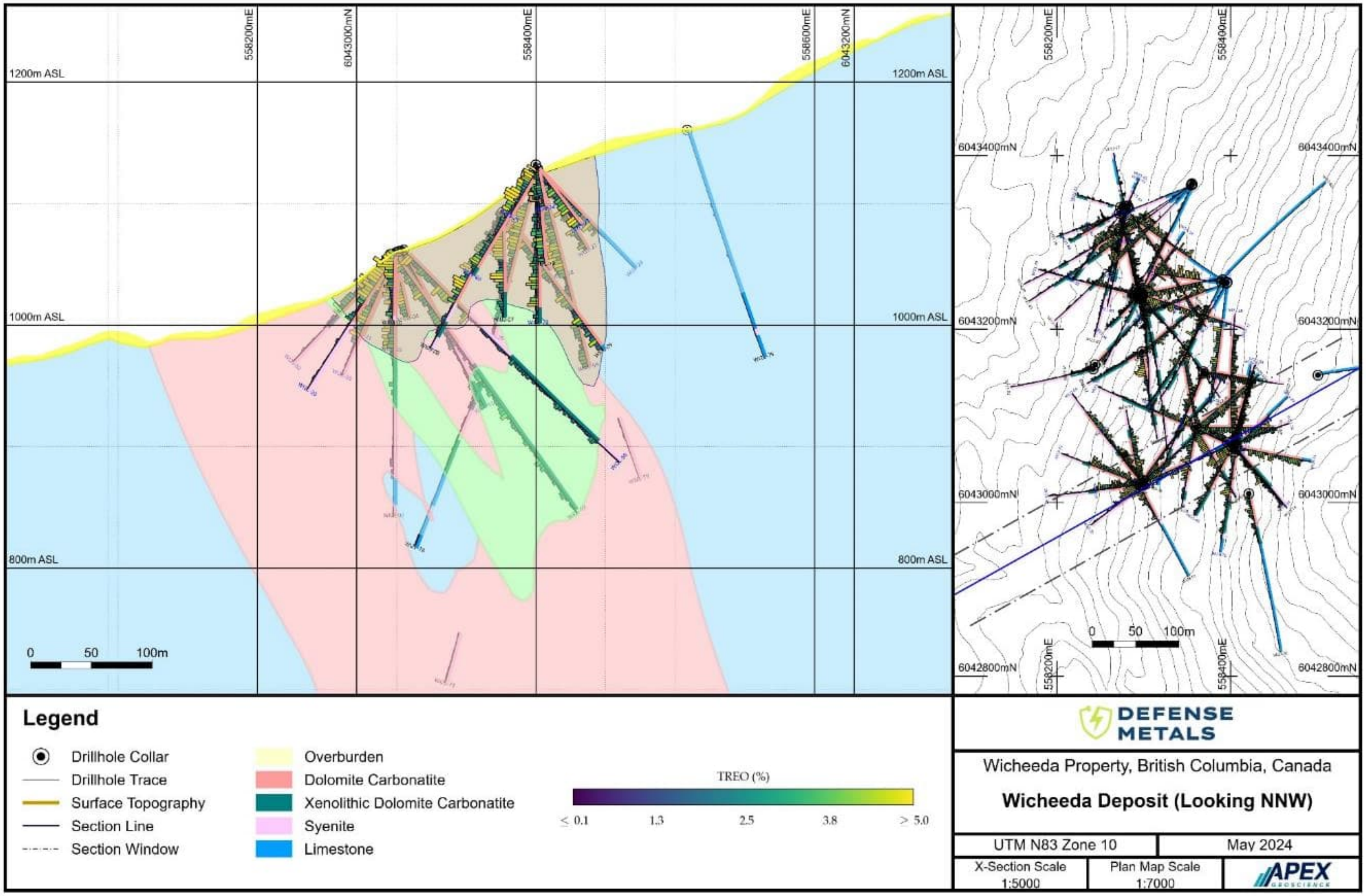
``



Source: APEX, 2025

Figure 10-2: Wicheeda Drill Holes (Section Looking NEE), Eastern section

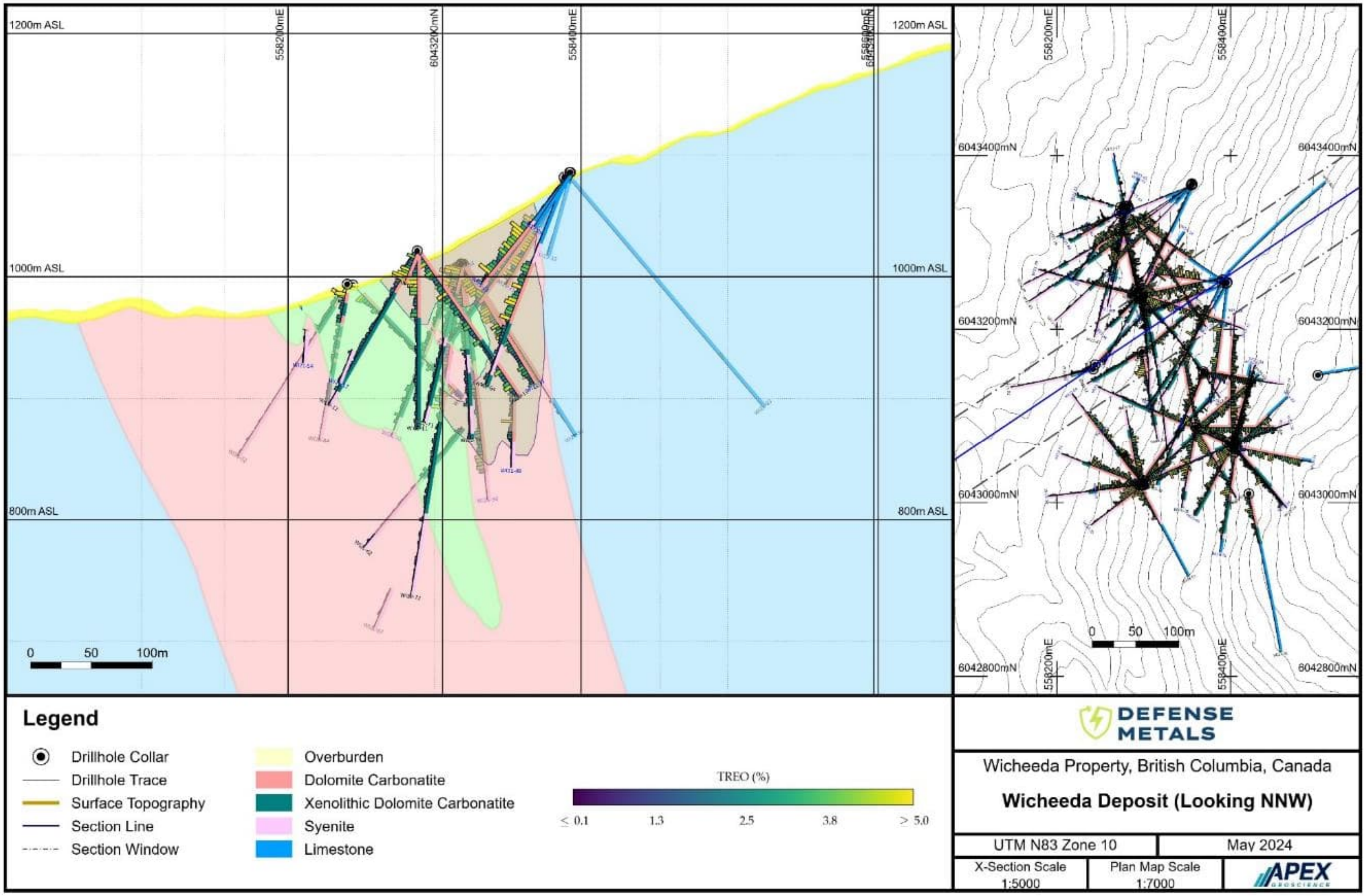
``



Source: APEX, 2025

Figure 10-3: Wicheeda Drill Holes (Section Looking NNW). Southern Section

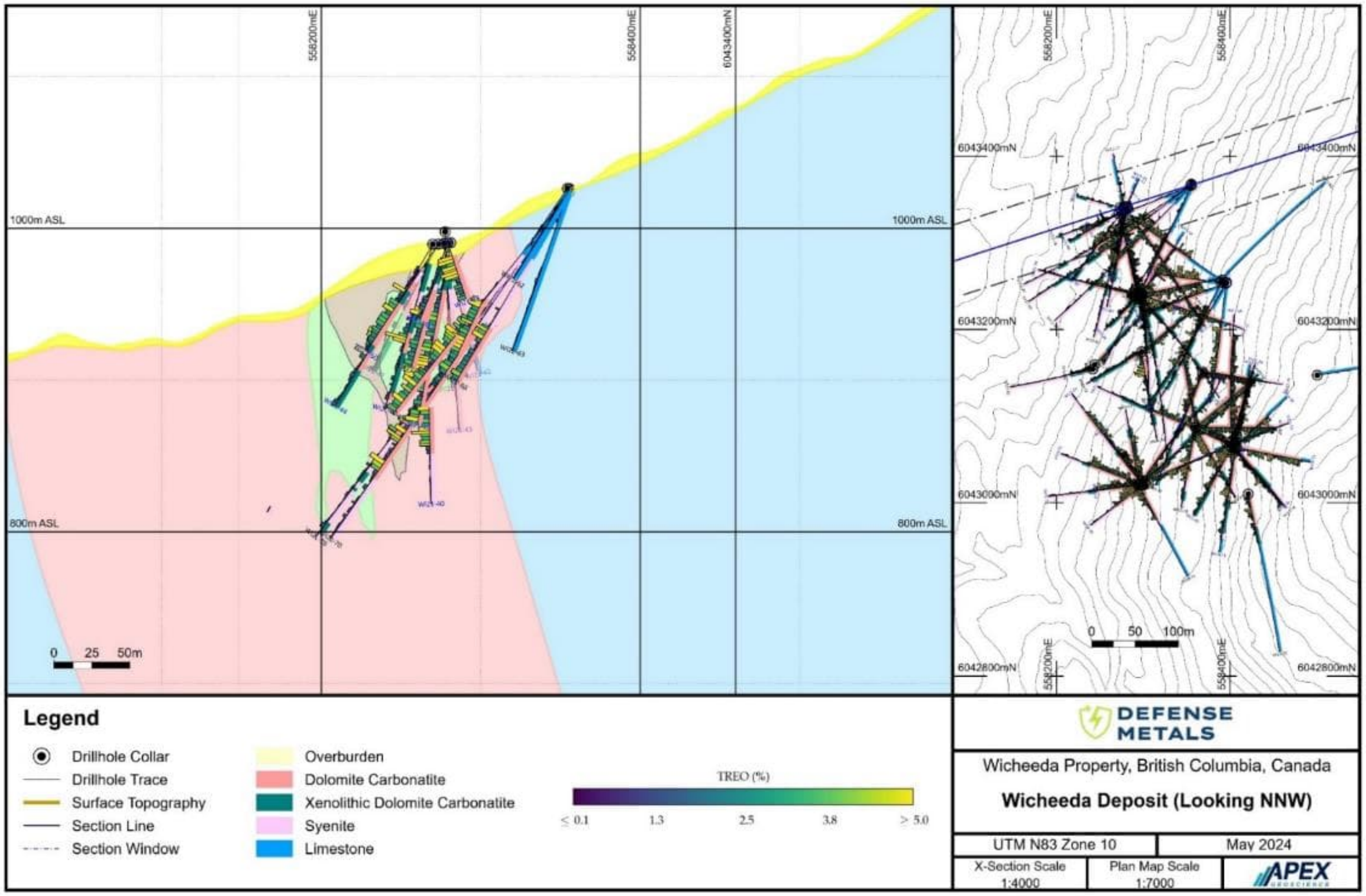
``



Source: APEX, 2025

Figure 10-4: Wicheeda Drill Holes (Section Looking NNW). Central Section

``



Source: APEX, 2025

Figure 10-5: Wicheeda Drill Holes (Section Looking NNW). North Section

10.1 2008 -2009 Historical Drilling

From late September to mid-October 2008, Spectrum completed four diamond drill holes (WI08-01 to WI08-04) with an aggregate length of 866 m (Lane, 2009). The holes were drilled from a single helicopter-supported drill pad and included one vertical hole and three inclined holes drilled on different azimuths. Each drill hole was collared in intrusive carbonatite and confirmed the presence of a LREE-bearing dolomite carbonatite body that outcrops on a west-facing slope one km south of Wicheeda Lake. Due to the limited amount of drilling, the overall geometry of the Wicheeda Carbonatite was not resolved; however, the 2008 campaign established an eastern structural footwall to the zone. The western, northern, southern and depth components remained open (Lane, 2009). The Wicheeda Carbonatite was found to contain significant concentrations of the LREEs cerium (Ce), lanthanum (La), and neodymium (Nd) as well as anomalous concentrations of Nb, Pr, Y, As, Ba, Mo, Mn, Pb, Sr, and Th (Lane, 2009).

In 2009, Spectrum completed 15 additional drill holes (WI09-05 to WI09-20), totaling 1,824 m (Lane, 2010a). Ten holes tested the Wicheeda Carbonatite from two different set-ups (sites A and B), two holes were drilled northwest of previous sites to intersect a small carbonatite dyke that outcrops on a trail leading to Wicheeda Lake (site C), and three holes tested a REE soil anomaly located northwest of site C and southwest of Wicheeda Lake (site D). All ten holes drilled on the Wicheeda Carbonatite intersected significant intervals of REE-bearing dolomite ± calcite carbonatite from surface to variable depths. The highest REE values correlated with dolomite carbonatite, dolomite carbonatite breccia and calcite carbonatite. To a lesser degree, high REE values also occurred in syenite breccia (later recognized as fenite) where dolomite carbonatite, as matrix to clasts of syenite (fenite), formed >50% of the rock mass (Lane, 2010a).

10.2 2019 Diamond Drilling

During 2019, Defense Metals retained APEX Geoscience Ltd. to conduct a diamond drilling exploration program at the Wicheeda Property. The program directive was to test the extent of the Wicheeda deposit where it is still open, and further delineate the relatively higher-grade near surface dolomite unit. Thirteen diamond drill holes, totalling 2,007 m from three different drill pads, tested the southern, central and northern zones of the carbonatite. All drill holes intersected variable lengths of significant REE mineralization, mainly in the carbonatite dolomite body and, to a lesser extent, in the lithologies enveloping the carbonatite deposit. The 2019 diamond drill program was successful in expanding the REE mineralized footprint of the Wicheeda Deposit to the south and north (Raffle and Nichols, 2020).

The drilling program was completed between July 28 and October 22, 2019. Drilling began at the location of the 2019 bulk sample (2008 drill site). Prior to drilling, drill hole collars and drill sites were located by handheld GPS. Falcon Drilling Ltd. of Prince George, British Columbia was contracted to complete the drilling. At each drill site, drill pads were constructed on steep mountain terrains. Once retrieved, drill core was removed from the core tube and placed directly into four row NQ-sized wooden core boxes with standard 1.2 m length. The core boxes were then sealed with wooden lids, strapped tightly and transported by a helicopter to the gravel pit camp site for logging purposes. Then, the core were carefully reconstructed, geotechnical data were recorded (depth markers, core recovery, rock quality designation (RQD), specific gravity, scintillometer), geological observations were recorded (lithology,

``

alteration and weathering, structure, veining, mineralization) and core was then sampled. Once sampled, the core is cut, placed in a sealed polybag and shipped in rice bags to the sample preparation laboratory. Down-hole survey directional data was collected using a Reflex EZ-Shot instrument (Raffle and Nichols, 2020).

All 13 drill holes intersected significant intercepts of REE-mineralized dolomite carbonatite rocks. Drilling at the northern extent delineated and expanded the northern margin of the deposit a 120 m, representing a 50% increase in the strike length of the known Wicheeda Carbonatite at the time. The last hole (WI19-32) of the drilling program intersected a 130 m interval of REE-mineralized dolomite-carbonatite and as a result, the deposit was still open to the north. Infill drilling southeast of the deposit expanded the deposit 40 m beyond the existing limit and delineation drilling in the southwest area of the deposit extended the limit of the dolomite-carbonatite a further 25 m (Raffle and Nichols, 2020).

10.3 2021 Diamond Drilling

During 2021, Defense Metals retained APEX Geoscience Ltd. to conduct a follow up diamond drilling exploration program at the Wicheeda Property. The program directive was to test the extent of the Wicheeda deposit where it was still open to the north and northwest, further delineate the relatively higher-grade near-surface dolomite unit, and to convert the inferred and/or indicated mineral resource into indicated and measured mineral resource. Twenty-nine NQ diameter diamond drill holes, totalling 5,366 m, were completed from five different drill pads, testing the southern, central and northern zones of the carbonatite.

All 29 drill holes crosscut significant intercepts of REE-mineralized dolomite carbonatite. Drilling delineated and expanded the carbonatite body to the north / northwest and marginally around the deposit. Drilling increased the extent of the deposit roughly 30 m along NW/SE strike from 2019 last drill hole (WI19-32) and up to 85 m to the west from drill hole WI19-31.

All drill holes yield fine to coarse-grained REE minerals (monazite and bastnäsite/parasite/synchysite) forming millimetre to centimeter-scale aggregates interstitial to dolomite-ankerite.

10.4 2022 Diamond Drilling

During 2022, Defense Metals retained APEX Geoscience Ltd. to conduct a follow up diamond drilling exploration program at the Wicheeda Property. The program was designed to test the extent of the Wicheeda deposit where it was still open to the north and northwest, further delineate the relatively higher-grade near-surface dolomite unit, and to convert the inferred and/or indicated mineral resource into indicated and measured mineral resource. Eighteen HQ3 diameter diamond drill holes, totalling 5,510 m, were completed from nine different drill pads, testing the southern, central and northern zones of the carbonatite (Figure 10-2 to Figure 10-5).

Fifteen holes intersected variable lengths of significant REE mineralization, mainly in the carbonatite dolomite body and, to a lesser extent, in the lithologies enveloping the dolomite carbonatite deposit. The 2022 drilling program aimed to delineate existing resources further, assessing near deposit exploration targets and geotechnical drilling for the purpose of optimization of open pit slope design, and generating additional REE mineralized material for metallurgical testwork exploration. The delineation program expanded the main carbonatite

``

body to the north and northwest at depth and marginally around the deposit. The drilling program also increased the higher-grade dolomite carbonatite zones and xenolithic dolomite carbonatite throughout the deposit.

The drilling program was supervised by APEX which provided geological and logistical services in the field. Diamond drilling was carried out by Radius Drilling Corp. of Prince George, BC. Pad building was completed by Rugged Edge Holdings, BC. Drill logging and core cutting were carried out by APEX Geoscience Ltd from the Wicheeda camp, 80 km northeast of Prince George, BC.

Prior to drilling, drill pads were sighted by handheld GPS to be built and later, drill hole collars were positioned using a Reflex Azimuth Positioning System (APS) unit. Upon completion of a 3 m drill run, drill core was removed from the core tube and placed directly into four row HQ-sized wooden core boxes with a standard 1.2 m length. Core boxes were sealed with wooden lids, strapped tightly and transported by helicopter to camp. Once in camp, core was carefully reconstructed, then, geotechnical data were recorded (depth markers, core recovery, rock quality designation (RQD), specific gravity, scintillometer), geological observations were recorded (lithology, alteration and weathering, structure, veining, mineralization), core photos were taken and sample intervals were marked for analysis. Once sampled, the core was cut in half, placed in a sealed poly bags marked with a sample number and shipped to the analyzing laboratory. Down-hole survey directional data was collected using a Reflex EZ-Shot instrument at specific depths, averaging at 50 m.

For geotechnical drill holes, once a 3-metre drill run was completed, drill core was removed from the core tube and a mark on the down-hole end of the run was made for the orientation line. This mark from the center axis to the perimeter of the core references the bottom part of the drill hole.

Drillers transferred the core on a half-split tube to be logged on-rig by an APEX geo-technician. The basic logging on-rig was conducted as follows:

1. Core was carefully cleaned and reconstructed.
2. An orientation line was drawn from the driller’s mark and the offset to the previous run was recorded.
3. Depth markers and down-hole arrows were added.
4. Split photos were taken.
5. Fracture types were identified and marked with their respective colours (Joints, Mechanical breaks, Foliation/Bedding, Cemented joints, Core Handling Breaks).
6. Geotechnical intervals in the core run were defined.
7. Basic geotechnical logging was then conducted. This includes:
    a. Total Core Recovery (TCR) logging
    b. Rock Quality Designation (RQD) logging

   c. Weathering conditions on rock fabric
   d. Mechanical and natural rubble zones
   e. Intact Rock Strength (IRS) estimates

The drill run was then placed directly into three row HQ-sized wooden core boxes with a standard 1.2 m length. The core boxes were then sealed with wooden lids, strapped tightly, and transported by helicopter to the Wicheeda Camp. Once in camp, the detailed geotechnical core logging procedure was conducted as follows:

1. Pocket penetrometer testing of weaker material (if applicable)
2. Determining micro-defect intensity and strength
3. Discontinuity Logging:
   a. Recording discontinuity depth
   b. Joint Surface conditions/rating (RMR76 and RMR89): roughness, weathering, aperture, and fill strength.
   c. Alpha and beta angle measurements
4. Major Structure Logging:
   a. Recording interval of structure
   b. Determining geotechnical structure type and geological classification
5. Point load testing.
6. Core box photos.
7. Geotechnical sampling (if any).
8. On-site QAQC on logging conducted at the rig.

After completing geotechnical logging, geological observations were recorded (lithology, alteration and weathering, structure, veining, mineralization) and sample intervals were marked for geochemical analysis. Once sampled, the core was cut in half, placed in a sealed poly bags marked with a sample number and shipped to the analyzing laboratory. Down-hole survey directional data was collected using a Reflex EZ-Shot instrument at specific depths, averaging at 50 m.

10.5 2023 Diamond Drilling

Defense Metals retained APEX Geoscience Ltd. to conduct a follow up exploration and geotechnical diamond drilling program at the Wicheeda Property. Six HQ diameter diamond drill holes, totalling 1,182 m, were completed from five different drill pads (Table 10-1).

The drilling program was supervised by APEX who provided geological and logistical services in the field. Diamond drilling was completed by Discovery Diamond Drilling Ltd. of Stewart, BC. Drill core logging and core cutting were carried out at the Wicheeda camp, 5 km northwest of the deposit area.

Similar to previous exploration programs, rock geotechnical, logging and downhole surveying procedures were used. For geotechnical drill holes, logging procedure implemented during 2022 geotechnical program was also used.

Drilling tested near-deposit exploration targets and successfully characterized pit wall lithologies. Four of the six diamond drill holes intersected variable lengths of REE mineralization. Two diamond drill holes intersected sediments and fenite with trace REE mineralization.

Mineralized intersects comprised fine to coarse-grained REE minerals (monazite and bastnäsite/parisite/synchysite) forming millimetre to centimeter-scale aggregates interstitial to dolomite-ankerite.

Not all geotechnical drill holes were sampled.

10.6 QP Opinion

Mr. Reid has reviewed and accepted the drilling information provided in this section. Details pertaining to the verification of the respective information are contained in Section 12.

11. Sample Preparation, Analyses and Security

11.1 2019 Bulk Sample

This section is included for reference only and does not form part of the PFS study. It was part of the previous PEA study.

11.1.1 *2019 Bulk Sample Collection and Security*

In September 2018, Spectrum received approval from the BC Ministry of Mines for a work permit to collect a 30-tonne bulk sample of rare earth mineralization from Wicheeda. This 30 tonne sample was collected near the 2008 drill site. After collection the sample was placed into one tonne poly-woven duffle top bulk bags and transported via commercial flatbed haul truck to Prince George for secure storage prior to shipment to SGS Minerals Lakefield, ON facility

SGS Minerals Lakefield is an ISO/IEC 17025 and ISO9001:2015 accredited geoanalytical services provider. SGS is independent of Defense Metals and the authors.

11.1.2 2019 Bulk Sample Preparation and Analysis

At SGS the entire 30 tonne bulk sample was jaw crushed to nominal 1 inch, and homogenized/blended via backhoe. A 400 kg sample representative sample was then selected and further homogenized by tumbling and crushed to ½ inch. Half of the 400 kg sample was retained for future testing. The primary 200 kg sample was then crushed to 6 mesh (3.36 mm), homogenized and split into 10 kg charges. Two of the 10 kg charges were combined and split into 2 kg charges, from one of which 150 g was pulverized to 80% passing 75 micron. Major element, and lanthanum and neodymium oxides, and loss on ignition (LOI) were determined by whole rock analysis, via lithium-borate fusion of a 0.5 gram sample analyzed via wavelength dispersion X-ray fluorescence (WD-XRF). The remaining rare earth elements were determined via 0.5 gram sodium-peroxide fusion multi-element ICP-MS.

A sub-sample of the head sample was submitted for abrasion index testing, and the remainder crushed to 100% passing 12.7 mm and 25 kg sample was taken for Bond Rod Mill Work Index (RWI) AND Bond Ball Mill Work Index (BWI) test work. The rest of the sample was further crushed to 100% passing 3.3 mm. The less than 3.3 mm sample was split into 2 kg and 10 kg charges for batch and bulk concentrate flotation production tests. Flotation charges were stage-ground to 100% passing 106 µm or 150 µm based on mineralogical data and SGS's prior experience with REE flotation testwork programs.

For the locked-cycle test, a stability check reveals that reasonable stability was achieved quickly for all elements, and three cycles were deemed suitable for projected mass balance calculation, to simulate the metallurgical performance that would be achieved in a continuous operation.

Head grade, batch, and locked-cycle concentrate products for cerium, lanthanum, neodymium and praseodymium oxides were determined via lithium-borate fusion of a 0.5 gram sample analyzed via wavelength dispersion X-ray fluorescence (WD-XRF). The remaining rare earth elements for the head sample were determined via 0.5 gram sodium-peroxide fusion multi-element ICP-MS.

``

11.1.3 *2019 Bulk Sample Quality Assurance – Quality Control*

The SGS analysis included a quality assurance / quality control (QA/QC) program including the insertion of rare earth element standard and blank samples. The PEA QP detected no significant QA/QC issues during review of the data.

11.2 2008 and 2009 Core Sample Preparation

11.2.1 *2008 and 2009 Sample Collection and Security*

In 2008, four BTW-diameter (∅ 40.7 mm) holes totaling 866.06m were drilled on the Project. In 2008, drill core was transported to Prince George following the completion of all four holes and was logged and sampled in a secure, gated warehouse located on the premises of Allnorth Consulting Ltd. After delivery of the core the driller’s run blocks were converted to metric units, and recovery and RQD were measured prior to logging. The core was logged for geological and geotechnical properties by Jay W. Page, P.Geo. Each section of core to be sampled was clearly identified and then marked with a centre line. All core was photographed, sawn and sampled using a nominal sample interval of 3 m. Core splitting, using a water-cooled diamond saw, was conducted by competent, experienced technicians under the guidance of Page and Lane (Lane, 2009).

Two hundred fifty-five (255) core samples were labelled, cut and bagged. Thirty-three (33) quality control samples (blanks, duplicates and just two standards) were inserted into the sample stream at regular intervals following a prescribed sequence. All of the bagged core samples were recorded on shipment forms, packed in large woven nylon ‘rice’ bags and trucked via independent commercial transport to the Global Discovery Labs (TeckCominco) in Vancouver, BC, for 30 element ICP-AES analysis and for selected light rare-earth element analysis (i.e., lanthanum, cerium and neodymium) and niobium by XRF (pressed pellet) analysis. The lab inserted its own blanks, duplicates and standards into the sample stream and routinely conducted repeat analysis.

Following receipt of the ICP-AES and XRF data, pulps from the upper part of each drill hole, prepared by Global Discovery Labs, were shipped to Activation Laboratories Ltd. in Ancaster, Ontario, for lithium metaborate/tetraborate fusion ICP/MS analysis. A total of 73 sample pulps were analyzed for 43 elements including the light and heavy REE. Nine quality control samples (blanks, standards and duplicates) were inserted into the sample stream at regular intervals.

In 2009, 15 HQ (63.5 mm diameter) holes totaling 1,823 m were drilled on the Project. In 2009, drill core was transported to Prince George following the completion of each hole and was logged and sampled in a secure, gated warehouse located on the premises of Falcon Drilling Ltd. After delivery of the core the driller’s run blocks were converted to metric units, and recovery and RQD were measured prior to logging.

Geological logging in 2009 was performed by geologist Murray Morrison. All core was photographed, sawn and sampled using a nominal sample interval of 3 m. Core splitting, using a water-cooled diamond saw, was conducted by competent, experienced technicians under the guidance of Morrison and Lane (Lane, 2010a).

11.2.2 *2008 and 2009 Sample Preparation and Analysis*

Five hundred eighty-three (583) core samples were labelled, cut and bagged. Seventy-four (74) quality control samples (blanks, standards and duplicates) were inserted into the sample stream at regular intervals following a prescribed sequence. All of the samples were recorded on shipment forms and the samples were trucked to the Global Discovery Labs (which was purchased by Acme Analytical Labs during the year and was subsequently purchased by Bureau Veritas) in Vancouver, BC, for 30 element ICP-AES analysis. Samples from drill holes WI09-05 to WI09-14 were also analyzed for selected light REE (La, Ce, Sm and Nd) and Nb by XRF (pressed pellet) analysis. The lab also inserted its own blanks, duplicates and standards into the sample stream and routinely conducted repeat analysis.

Acme Analytical Labs, Bureau Veritas, and Activation Laboratories Ltd. are all ISO/IEC 17025 and ISO9001:2015 accredited geoanalytical services providers and are independent of Defense Metals.

It is unknown if Global Discover Lab was accredited, but it was independent of Defense Metals.

11.2.3 *2008 and 2009 Quality Assurance – Quality Control*

This section includes a review of control samples (blanks, standards and duplicates) used in the 2008 and 2009 diamond drill programs. The 2008 drilling at the Project did not include adequate control samples as the percentage of certified reference standards fell below the recommended minimum level of 5%, but this was improved in 2009. The control sample insertion rate is presented in Table 11-1 and shows an acceptable overall rate of more than 12%.

Table 11-1: Quality Control Sample Insertion Rate Summary (XRF Data)

Type	2008	2009
Field Blank	16	28
Field Standard	2	28
Field Duplicate	15	18
Primary Samples	255	583
Insertion Rates		
Blanks	6.3%	4.8%
Field Standards	0.7%	4.8%
Field Duplicates	5.9%	3.1%
Primary Samples	87.1%	87.3%

Source: APEX, 2023

Two different field blanks were used (CDN-BL-3 and CDN-BL-4); both gave consistently low values for REE and Nb. The analytical results for the blanks inserted into the 2009 sample stream indicated that there was little to no contamination in the lab. Blank results ranged from 11-65 ppm Ce, 8-64 ppm La, <3-26 ppm Nd, 4-8 ppm Nb and <3-8 ppm Sm.

``

The principal certified reference standard material (SRM) used in the field in 2009 was SY-4. SY-4 is diorite gneiss and is source from the National Research Council Canada(NRC). SY-4 certified values (±95% confidence interval) include: Ce – 122 ± 2 ppm; La – 58 ± 1 ppm, Nb – 13 ± 1 ppm, Nd – 57 ± 1 ppm, and Sm – 12.7 ± 0.4 ppm. SY-4 analytical results were within a relatively narrow range with the exception of one result for La, which was approximately twice that of the other received values for that REE. This result may be spurious or may indicate minor analytical inconsistencies at the lab.

SY-4 is suitable as a standard for background or low grade REE mineralization but is not suitable as a standard for higher grade REE mineralization. This is no longer a concern as Defense Metals has re-assayed the 2008 and 2009 samples with appropriate SRMs as discussed in Section 11.3.

The assay laboratory inserted higher grade SRMs into each batch of Wicheeda samples, but these results cannot be relied upon as they are not considered blind to the assay laboratory.

Core duplicates were prepared by sawing a sawn sample a second time to produce a two ¼ samples for analysis with the remaining half returned to the core box. Duplicates followed the original sample in the sample stream. Duplicate results show reasonable agreement for Ce, La, and Nd.

It is the QP’s opinion that core logging, sampling, assaying, and chain of custody procedures utilized by Spectrum in 2008 and 2009 were generally consistent with industry practices at the time the data was collected.

11.3 2008 and 2009 Drill Core Pulp Re-analysis

11.3.1 2008 and 2009 Re-Sample Collection and Security

During late 2020, APEX personnel travelled to the Prince George secure core storage facility to retrieve original 2008 and 2009 drill core prepared pulps. The prepared pulps were found to be in good condition, having been stored indoors within their original packaging, and affixed with labels consistent with their original sample IDs. A total of 743 samples were sent for analysis, including 91 QA/QC samples.

The samples were palletized in their original containers and transported by an APEX geologist to a commercial shipping company in Prince George, BC. Samples were then shipped via ground service to the ALS Minerals (ALS) laboratory in Vancouver, BC. Upon receiving the samples, the laboratory sorted and checked the samples received against the sample submission form.

APEX did not always monitor the samples during transport; however, the pallet was reportedly undamaged when received by ALS. Therefore, there is no reason to believe that the security of the samples was compromised in any way during transport or once they entered the ALS chain of custody.

11.3.2 2008 and 2009 Re-Sample Preparation and Analysis

Re-analysis of all 2008 and 2009 original drill core pulps (less the 73 samples submitted to Actlabs during 2008 described above for which sample pulps no longer remain) was completed during 2020 and 2021 to reduce the uncertainty regarding the historical incomplete XRF analytical results.

Once received by ALS in Vancouver, the drill core pulps were logged in to the ALS computerized tracking system and assigned bar code labels. The samples were then analyzed using lithium metaborate fusion with an ICP-MS finish (ALS code ME-MS81). Samples returning greater than 10,000 ppm Ce or La, or greater than 1,000 ppm Pr were subject to overlimit analysis of via high grade REE lithium metaborate fusion with an ICP-MS finish (ME-MS81h).

APEX replace the Actlabs results with the ALS lithium metaborate fusion results for all but the 73 results discussed above). In the QP’s opinion, the use of Actlab assay results for these 73 samples does not pose a material risk to the Mineral Resource Estimate.

ALS is an ISO 9001:2008 certified laboratory and is also accredited by the Standards Council of Canada (SCC) and has been found to conform to the requirements of ISO/IEC 17025:2005. ALS is independent of Defense Metals.

11.3.3 2008 and 2009 Re-Sample Quality Assurance – Quality Control

Re-analysis of 2008 and 2009 original drill core pulps was completed during 2020 and 2021 to reduce the uncertainty regarding the historical incomplete XRF analytical results. A total of 743 samples were sent for analysis, including 91 QA/QC samples to ALS.

The current drill hole database used for the MRE comprises REEs determined by fusion ICP-MS analysis at Actlabs (73 samples) and ALS (2008 and 2009 drill core pulp re-analysis), in addition to the 2019-2023 drill core samples (ICP-MS at ALS).

11.3.3.1 Standards

Thirty analytical standards (CDN-RE-1201) were inserted into the 2008 and 2009 drill core pulp re- analysis sample sequence. Assay results show two Nd assays exceeded the acceptable three standard deviation from the certified value (311 ppm ± 27 ppm, and two Ce assay results exceeded the acceptable standard deviation for the certified value (1327 ppm ± 165 ppm). All La results were within the acceptable standard deviation (certified value; 959 ppm ± 161 ppm), and all Pr assays were within the acceptable standard deviation from certified value (112 ppm ± 11 ppm).

11.3.3.2 Blanks

Thirty-five CDN Resource Laboratories (CDN) granitic material pulps blanks (CD-BL-3 and CDN- BL-4) were inserted. For the 35 blanks, no samples exceeded the established cut-off values of 50 ppm Nd, 150 ppm Ce, 100 ppm La, and 20 ppm Pr.

11.3.3.3 Duplicates

Twenty-six quartered drill core samples were collected during the diamond drill program. Results of duplicate samples indicate good overall repeatability of the Nd, Ce, La, and Pr values. This is interpreted to indicate a low “nugget” effect with respect to REE analysis.

``

Excluding primary geological heterogeneity (quarter-core), the data show a homogenous distribution of Ne, Ce, La and Pr values within the Wicheeda drill core.

11.4 Defense Metals Drilling (2019 - 2023)

11.4.1 2019 to 2023 Sample Collection and Security

Drill core samples were sawed in half longitudinally using a diamond bladed core saw. For each sample, one half core was sent for analysis, and the other half was left in the box. Duplicate samples were cut into quarters, where one quarter of the core was used as the "original" sample and the other quarter was used as the "duplicate" sample. The remaining half core was left in the box.

Drill core samples were placed into labelled plastic sample bags along with a sample tag inscribed with the unique sample number. The samples, including requisite QA/QC samples, were placed into woven poly (rice) bags for shipping to the analyzing laboratory. Cable ties were used to securely close the rice bags. Samples were transported by APEX personnel to a shipping company in Prince George, BC. Samples were then shipped via ground service to ALS Minerals ("ALS") laboratory in Kamloops or Langley British Columbia for preparation. Upon receiving the samples, the laboratory sorted and checked the samples received against the sample submission form.

The authors did not monitor the drill core samples at all times during transport; however, the sealed rice bags with unique identifiers (security tags) were intact when received by ALS. The authors have no reason to believe that the security of the samples was compromised in any way during transport or once they entered the ALS chain of custody.

11.4.1.1 2019 Drilling

A total of 717 samples were collected and sent for analysis, including 145 QA/QC samples. Sample intervals were typically between 1 and 3 m. Due to poor core recovery, one sample measured 7.05 m hole length; however, the actual core length was 1.85 m (Raffle and Asmail, 2022). Sample intervals were marked out and tagged by APEX geologists, and the core was then photographed. Standard, blank and duplicate samples were inserted at regular intervals in the sample sequence.

11.4.1.2 *2021 Drilling*

A total of 2171 samples were collected and sent for analysis, including 338 QA/QC samples. Sample intervals were typically between 1 and 3 m. Due to poor core recovery, few samples were over 3 meters: two samples measured 9.15 m; however, the actual core lengths were 1.15 – 1.4 m, and three samples measured 6.1 m hole length; however the actual core length were 0.61 cm – 1.1 m. Sample intervals were marked out and tagged by APEX geologists, and the core was then photographed. Standard, blank and duplicate samples were inserted at regular intervals in the sample sequence.

11.4.1.3 2022 Drilling

A total of 1971 samples were collected and sent for analysis, including 296 QA/QC samples. Sample intervals were typically between 1 m and 3 m. Due to poor core recovery, two samples were over 3 m and not over 4.5 m. Sample intervals were marked out and tagged by APEX geologists, and the core was then photographed. Standard, blank and duplicate samples were inserted at regular intervals in the sample sequence.

``

11.4.1.4 *2023* Drilling

A total of 235 samples were collected and sent for analysis, including 34 QA/QC samples. Sample intervals were typically between 1 m and 3 m. Due to poor core recovery, one sample was over 3 m and not over 3.5 m. Sample intervals were marked out and tagged by APEX geologists, and the core was then photographed. Standard, blank and duplicate samples were inserted at regular intervals in the sample sequence.

11.4.2 2019 to 2023 Sample Preparation and analysis

Once received by ALS, the drill core samples were logged in to the ALS computerized tracking system, assigned bar code labels. The samples were then air dried overnight or oven dried to a maximum of 120ºC. The samples were then weighed, crushed to better than 70% passing 2 mm, and the whole sample homogenized before taking the final split for the pulp. Once the samples were homogenized, a 250 g split was selected to be pulverized to better than 85% passing 75 microns. The prepped samples were then shipped to the ALS facility in North Vancouver, British Columbia, for analysis for high grade REE. The samples were analyzed using lithium metaborate fusion with an ICP/MS finish (ALS code ME-MS81h).

ALS is an ISO 9001:2015 certified laboratory and has received ISO/IEC 17025:2017 accreditation from the Standards Council of Canada (SCC). ALS is independent of Defense Metals.

11.4.3 2019 to 2023 Quality Assurance – Quality Control (QA/QC)

The QA/QC measures employed by APEX geologist comprised inserting field standards, blanks and duplicate samples. Analytical standards were inserted into the sample stream to verify the accuracy of the laboratory analysis. Barren coarse material was used for coarse "blank" samples to monitor potential contamination during the sample preparation procedure. Duplicate samples were collected to assess the repeatability of individual analytical values. QA/QC samples were inserted at a rate of approximately 1 standard, black or duplicate per 20 samples.

Each standard has an accepted concentration as well as known "between laboratory" standard deviations or expected variability. There are two general industry criteria employed by which standards are assigned a "pass" or "reviewable" status. First, a "reviewable" standard is defined as any standard occurring anywhere in a drill hole returning greater than three standard deviations (>3SD) above or below the accepted value for an element. Second, if two or more consecutive standards from the same batch return values greater than two standard deviations (>2SD) above or below the accepted value on the same side of the mean for at least one element, they are classified as "reviewable".

QA/QC samples falling outside established limits are flagged and subject to review and possibly re-analysis, along with the 10 preceding and succeeding samples.

Blank samples were inserted into the sample stream to check for contamination during the sample preparation procedures. Standard coarse-crushed silica blanks were used, sourced from landscaping material.

Duplicate (quartered drill core) samples were collected to assess the repeatability of individual analytical values.

``

The results are summarized in Table 11-2 and Table 11-3. The overall results are acceptable, but the QP recommends including Dy, Tb and Gd in the analysis as the current SRMs are certified for these elements. An analysis for blank and duplicate results for these elements should be possible based on the available data.

Should changes to the process flowing occur in the future, the QA/QC analysis should be adjusted to cover the primary economic elements. As Nd and Pr constitute about 95% of the economic value, the lack of reported QAQC on the remaining elements poses little risk to the Mineral Resources, Mineral Reserves or the mine plan and schedule.

Considering the correlation between the various REE's the impact to current classification is expected to be minimal.

A detailed description of QA/QC results by period is included in the following sections.

Table 11-2: Quality Control Sample Insertion Rate Summary

Year	2019	2021	2022	2023
Core Samples	717	2171	1971	235
Blanks	37	101	100	12
Standards	36	116	97	11
Duplicates	36	108	99	11
Insertion Rates				
Blanks	5.16%	4.65%	5.07%	5.11%
Standards	5.02%	5.34%	4.92%	4.68%
Duplicates	5.02%	4.97%	5.02%	4.68%

Source: APEX, 2025

Table 11-3: Quality Control Sample Summary

Element	Ce (ppm)	La (ppm)	Nd (ppm)	Pr (ppm)
Blanks				
Measured_Mean	59	43	15	5
Standards				
Certified	8110	6508	1573	619
Between Lab 3SD	± 936	± 633	± 151.5	± 48
Between Lab 3SD %	12%	10%	10%	8%
Measured_Mean	8184	6666	1618	611
% Bias	0.9%	2.4%	2.9%	-1.3%
Duplicates				
R2 Coefficient of Determination (primary versus replicate core)	0.96	0.95	0.96	0.95

Source: APEX, 2025

``

11.4.3.1 2019-2023 Standards

The 2019 through 2023 diamond drill program utilized the standard material produced by CDN Resource Laboratories, CDN-RE-1203, and sourced from REE mineralized carbonate drill core from the Carbo Property located adjacent to the Wicheeda Project. The standard was chosen based on its geologic similarly to Wicheeda, in addition the accepted values within the range of mineralized material within the Wicheeda resource as determined buy lithium metaborate fusion with an ICP-MS finish. The recommended accepted values and between laboratory three standard deviation are: Ce (8110 ppm ± 936 ppm), La (6508 ppm ± 633 ppm), Nd (1573 ppm ± 151.5 ppm), Pr (619 ppm ± 48 ppm).

11.4.3.1.1 2019 Standards

A total of 36 standards were inserted into the sample stream. Results show all Nd assay were within the acceptable standard deviation from the certified value, and no Ce assay results fell outside of the acceptable limits. Three La and three Pr results fell outside of the acceptable limits.

11.4.3.1.2 2021 Standards

A total of 129 standards were inserted into the sample stream. Results show one Ce, four Nd, ten La and eight Pr results fell outside of the acceptable limits.

11.4.3.1.3 2022 Standards

A total of 97 standards were inserted into the sample stream. Results show no Ce, five Nd, no La and one Pr result fell outside of the acceptable limits.

11.4.3.1.4 2023 Standards

A total of 11 standards were inserted into the sample stream. Results show no Ce, no Nd, one La and one Pr results fell outside of the acceptable limits.

11.4.3.2 2019-2023 Blanks

11.4.3.2.1 2019 Blanks

Thirty-seven Analytical Solutions Ltd. (ASL) coarse silica blanks were used, sourced from Carboniferous sedimentary rocks of the Maritimes Basin in New Brunswick. For the 37 blanks, only one Nd sample exceeded the cut-off value of 50 ppm, while three Ce assays exceeded the cut-off value of 150 ppm. Five La results exceeded the cut-off value of 100 ppm, and 1 Pr assay exceeded the cut-off value of 20 ppm.

11.4.3.2.2 2021 Blanks

Of the 101 blanks analyzed, six samples exceeded the Nd cut-off value of 50 ppm, while ten Ce assays exceeded the cut-off value of 150 ppm. Fifteen La results exceeded the cut-off value of 100 ppm, and five Pr assay exceeded the cut-off value of 20 ppm.

11.4.3.2.3 2022 Blanks

Coarse-crushed silica blanks were used, sourced from landscaping material. Of the 99 blanks analyzed, one sample exceeded the Nd cut-off value of 50 ppm, while three Ce assays exceeded the cut-off value of 150 ppm. Three La results exceeded the cut-off value of 100 ppm, and one Pr assay exceeded the cut-off value of 20 ppm.

11.4.3.2.4 2023 Blanks

Standard coarse-crushed silica blanks were used, sourced from landscaping material of pure quartz. Of the 12 blanks analyzed, none of the samples exceeded the cut-off value of Ce, La, Nd and Pr.

11.4.3.3 2019-2023 Duplicates

11.4.3.3.1 2019 Duplicates

Seventy-two quartered drill core samples were collected during the diamond drill program. Results of duplicate samples indicate good overall repeatability of the Nd, Ce, La, and Pr values. This is interpreted to indicate a low “nugget” effect with respect to REE analysis. Excluding primary geological heterogeneity (quarter-core), the data show a homogenous distribution of Nd, Ce, La and Pr values within the Wicheeda drill core.

11.4.3.3.2 2021 Duplicates

One hundred and seven duplicate core samples were collected during the diamond drill program. Results of duplicate samples indicate good overall repeatability of the Nd, Ce, La, and Pr values similar to the results of the 2019 duplicate analyses.

11.4.3.3.3 2022 Duplicates

Ninety-nine duplicate core samples were collected during the diamond drill program. Results of duplicate samples indicate good overall repeatability of the Nd, Ce, La, and Pr values similar to previous years results.

11.4.3.3.4 2023 Duplicates

Eleven duplicate core samples were collected during the diamond drill program. Results of duplicate samples indicate good overall repeatability of the Nd, Ce, La, and Pr values similar to previous years results.

``

11.5 Variability Samples for Metallurgy Testwork Sample (2019-2023)

11.5.1 *Sample Collection and Security*

Defense Metals prepared 16 variability samples covering different lithologies, areas of the deposit, and head grades using drill core material. The average mass of each sample was 32.5 kg, with the Total Rare Earth Oxide ("TREO") assays ranging from 1.07% to 4.52% with an average of 2.34% TREO. Drill core material was also used to make a 260 kg Master Composite (MC) sample containing each of the three lithologies in their respective life-of-mine proportions. The MC sample had a head grade assay of 2.49% TREO. All variability samples and the MC sample were shipped to SGS, Lakefield, Ontario. SGS Lakefield is an ISO/IEC 17025 and ISO9001:2015 accredited laboratory. SGS is independent of Defense Metals Corp.

11.5.2 Sample Preparation and Analysis

Once at SGS samples were checked, crushed, and composited. A total of 87 flotation tests were completed to investigate the impact of collector type and dosage, depressant type and dosage, pulp temperature, pulp density, pulp pH, and flotation feed size.

Bulk flotation and other operations were carried out at SGS in order to prepare concentrate samples for continuing hydrometallurgical test work and planned hydrometallurgical pilot plant testing.

Feed samples were analyzed by Inductively coupled plasma mass spectrometry (ICP-MS) and flotation products were analyzed by SGS using wavelength dispersive X-ray fluorescence (WD-XRF) following lithium borate fusion of the sample. The SGS analyses included a quality assurance / quality control (QA/QC) program including the insertion of rare earth element standard and blank samples.

11.5.3 Quality Assurance – Quality Control

The SGS analysis included a quality assurance / quality control (QA/QC) program including the insertion of rare earth element standard and blank samples. The authors detected no significant QA/QC issues during review of the data.

11.6 QP Opinion

It is the QP’s opinion that the sample collection, preparation, security, analytical and QA/QC measures used during the 2019 through 2023 diamond drilling programs were adequate for this stage of exploration at the Wicheeda Property.

The only QA/QC issue detected during review of the SGS testwork report and data was that SGS stated 17 variability samples were evaluated when only 16 were reported in Table I of “An Investigation into Beneficiation Testwork on Low Grade and Variability Samples from the Wicheeda Rare Earth Deposit,” prepared for Defense Metals Corp., Project 17173-06, dated September 26, 2023. Mr. Giuseppe Paventi has accepted these results. Details pertaining to the verification of the respective information are contained in Section 13 of this report.

12. Data Verification

12.1 APEX

APEX managed and oversaw technical aspects of the 2019-2023 Wicheeda diamond drill campaigns, including selection of diamond drill sites, establishing base geotechnical and geological core logging and sampling procedures, in addition to the QA/QC program design and monitoring.

The complete Wicheeda Project drill hole database was reviewed in 2023 by APEX to ensure it was suitable for resource estimation. Validation by APEX and the authors included visual inspection and validation of drill hole collar, downhole survey data, and core recovery; in addition to digital validation for overlapping and missing lithology and sample intervals. Data from the drilling programs between 2019 and 2022 was captured and validated by APEX during each drilling campaign, after which APEX compiled the results with the historical data. Subsequently, the authors created an updated 3D geological model for the Wicheeda Deposit consistent with interpretation as a carbonate-hosted intrusion REE deposit.

In addition to the above, during late 2020, APEX personnel travelled to the Prince George secure core storage facility to retrieve original 2008 and 2009 drill core prepared pulps. The prepared pulps were found to be in good condition, having been stored indoors within their original packaging, and affixed with labels consistent with their original sample IDs. A total of 743 samples were sent for analysis, including 91 QA/QC samples. Details of results and associated QAQC are presented in Section 11.3. Samples were then analyzed using lithium metaborate fusion with an ICP-MS finish (ALS code ME-MS81). Samples returning greater than 10,000 ppm Ce or La, or greater than 1,000 ppm Pr were subject to overlimit analysis of via high grade REE lithium metaborate fusion with an ICP-MS finish (ME-MS81h). This type of analysis has been considered the appropriate analysis for this mineralization style and has been used in subsequent drilling programs in 2019, 2021. 2022 and 2023. Details of results and associated QAQC is presented in Section 11.3.

Based on the results of the data review, verification, validation, and results of the 2019, 2021, 2022 and 2023 diamond drilling programs; APEX determined the Wicheeda drillhole database and 3D geological model to be in good condition and suitable to use in ongoing resource estimation studies.

``

12.2 SRK Site Visits

12.2.1 Mineral Resources

An SRK Principal Consultant completed a site visit to Defense Metals 2021 drill hole core processing facility, managed by APEX on October 5, 2021 and the Wicheeda project area on October 6, 2021. APEX's site geologist, Mr. Mo Asmail, accompanied SRK staff during the site visit. Mr. Asmail provided a site geological overview as well as a summary of the procedures employed at the site. During the site visit, SRK reviewed the drill hole core handing and chain of custody, logging, sampling, QA/QC and found the practices employed by APEX to be according to industry standard practices.

SRK reviewed the electronic logging and sampling with the drillhole core for drill holes WI19-31 and WI19-32 mineralized intersections respectively and found no material differences. Two half core check samples were taken for verification purposes from these two drill holes. An additional four coarse rejects samples from the 2008 and 2009 drilling were also selected for check analyses. SRK sealed the samples and couriered the samples to ALS, Vancouver.

In SRK's opinion there are no obvious bias for either the drill hole core or the coarse rejects' witness samples. The REE original and duplicate results (for main economic elements) are presented in Table 12-1.

Table 12-1: SRK Witness Sample Comparison

Drill Hole	Original ID	Duplicate ID	Type	Depth from (m)	Depth to (m)	Year Drilled	Pr Org	Pr SRK	Nd Org	Nd SRK	Tb Org	Tb SRK	Dy Org	Dy SRK
							(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)
WI19-31	A0797677	SRK1	Core	130.00	133.00	2019	329	349	1,015	987	3.7	4.7	9.7	12.1
WI19-32	A0797671	SRK2	Core	117.00	119.60	2019	140	154	457	460	1.7	1.6	4.0	4.2
WI09-13	214749	SRK3	Coarse reject	130.83	133.83	2009	1210	1100	3,460	3,270	11.2	11.3	29.1	33.1
WI09-07	214508	SRK4	Coarse reject	95.44	98.44	2009	1170	1000	3,410	3,060	11.8	12.0	32.3	33.1
WI08-01	828633	SRK5	Coarse reject	83.25	86.25	2008	49	40	157	124	1.6	1.5	8.0	7.8
WI08-02	828665	SRK6	Coarse reject	71.75	73.75	2008	987	978	2,560	2,650	8.4	8.7	21.6	25.2
						Average	648	603	1,843	1,758	6.4	6.6	17.5	19.2
						Percent Difference	-7%		-5%		3%		10%	

Source: SRK 2024

SRK QP, Douglas Reid, visited the Wicheeda site on October 31 and November 1, 2024. No drilling was active during the site visit. During the visit, Mr. Reid was accompanied by APEX's site geologist, Mr. Mo Asmail. They visited drill pads 2019-1 and 2019-2 and confirmed collar locations for multiple drill holes drilled from these pads. Mr. Reid verified the collar locations with a handheld GPS (Garman etrex10). The differences shown in Table 12-2 are within known accuracy of the GPS unit.

Table 12-2: GPS Collar Verification

GPS Pt.	Easting	Northing	Elev	Hole ID	X	Y	Z_DTM	Pad	Diff X	Diff Y	Diff Z
170	558502	6043148	1168	WI22-75	558500.6	6043147	1157.524		-1.41	-0.83	-10.48
173	558402	6043067	1128	WI19-22	558406	6043064	1128.685	2019-2	4.00	-3.00	0.68
				WI19-23	558406	6043064	1128.685				
				WI19-24	558406	6043064	1128.685				
				WI19-25	558406	6043064	1128.685				
				WI19-26	558406	6043064	1128.685				
				WI19-27	558406	6043064	1128.685				
				WI19-28	558406	6043064	1128.685				
174	558303	6043028	1063	WI21-51	558299.4	6043026	1060.795	2019-1	-3.56	-2.34	-2.20
				WI21-52	558296.7	6043021	1059.778		-6.27	-6.82	-3.22
				WI21-53	558296.7	6043021	1059.778		-6.27	-6.82	-3.22
				WI21-54	558299.8	6043020	1060.642		-3.23	-7.72	-2.36
				WI21-55	558299.9	6043020	1060.677		-3.10	-7.88	-2.32
				WI21-56	558302.2	6043024	1061.724		-0.82	-3.64	-1.28

Source: SRK 2025

Mr. Reid reviewed available core from mineralized portions of WI22-62, 63 and 64, WI22-71, WI22-78 and a higher-grade interval of WI22-79. They examined the primary lithologies and observed both high grade and disseminated styles of mineralization. The intervals containing visible mineralization reported corresponding higher grades of REE. The detailed drillhole logs and assay results were compared against the core, and SRK confirmed the logging was accurate and sufficiently detailed.

12.2.2 Mineral Reserves

Dr. Ebrahimi conducted his site inspection on October 26-27, 2021. He was able to confirm and assess the site physiography in support of designing site and open pit accesses as well as siting various project infrastructure.

12.3 Data Verification

12.3.1 Mineral Resources

Mr. Reid reviewed the drill hole information with regards to logging convention errors i.e., gaps, overlaps, duplicate intervals, and the analytical values were checked for anomalous or switching of values. No obvious errors were encountered.

Mr. Reid compared the drill collar locations, downhole survey data, logged lithology and density data against original documentation. No material differences were noted.

Mr. Reid compared the assay certificates downloaded directly from the assay laboratory to samples recorded in the drillhole database as supplied by Defense Metals. No transcription errors weere encountered.

It is Mr. Reid’s opinion that the data provided is adequate for the purposes used for the MRE.

12.3.2 Mining

Data used in mine planning was derived or verified by other QPs contributing to this technical report. For cost information, Dr. Ebrahimi and Mr. McCarthy relied upon engineering judgement and experience.

13. Mineral Processing and Metallurgical Testing

13.1 General

Metallurgical development programs have been conducted, using samples from the Wicheeda project, between 2011 and 2024, to experimentally determine and optimize the beneficiation and hydrometallurgical flowsheets. Experimental testing was conducted at bench and pilot scales. All testing was conducted by SGS Canada Inc. at its Lakefield site ("SGS"). This work has been used as a basis for the design criteria for the process as outlined in Section 17.

The work covered in this section mainly concerns the metallurgical test work completed from late 2021 onward. The latest studies including comminution, metallurgical, and environmental test work have been conducted with samples (composites and variability) representing the main lithological domains present in the Wicheeda deposit. The work included the following tests:

Mineral Processing

- Mineralogical analysis
- Sample characterization
- Grindability tests
- Heavy liquid separation tests
- Batch flotation tests
- Flotation locked cycle tests
- Rare earth element analysis by ICP-MS
- Solid liquid separation tests

Hydrometallurgical Processing

- Bench scale test for the main hydrometallurgical steps
    - Acid baking, water leaching, primary neutralization, secondary neutralization, uranium ion exchange, solvent extraction, rare earth precipitation, magnesium removal
- Acid baking rotary kiln tests
- Two hydrometallurgical pilot plant campaigns
- Testing of alternative processing routes
- Solid-liquid separation tests

Environmental Testing

- Metal leaching and acid rock drainage (ML/ARD) and radionuclide potential testing.

13.2 Previous Metallurgical Testing

The previous metallurgical testing of Wicheeda samples spans over a period of 11 years from 2011 to 2021. All tests were conducted by SGS. In 2011, Spectrum Mining Corporation contracted SGS to carry out initial exploratory test work to develop a flotation process for recovering the contained rare earth minerals into a flotation concentrate. This work was followed by flotation optimization test work conducted for Defense Metals Corporation[1] in 2019 and flotation pilot plant testing in 2020. Hydrometallurgical test work was conducted on bulk flotation concentrates to demonstrate process requirements to produce a mixed REE precipitate.

Feed material for the 2019 and 2020 metallurgical testing was sourced from the approximately 30-tonnes bulk sample collected from the Wicheeda deposit by Spectrum Mining Corporation during October 2018. The 2019 SGS flotation test work sought to establish a Wicheeda metallurgical process base case by confirming the reproducibility of previous 2011 metallurgical tests. This was followed by process flowsheet optimization including researching the effects of various reagent combinations as well as varying grind size, flotation pulp temperature, and pH. Flotation concentrate samples were subjected to hydrometallurgical testing to determine a suitable process for the recovery of REE in a readily marketable form.

The work conducted for the Wicheeda project over the period 2011-2021 has been discussed in the SRK Report “**Independent Preliminary Economic Assessment for the Wicheeda Rare Earth Element Project, British Columbia, Canada**” Report Number 2CD031.000 January 2022 and is not repeated here.

13.3 Mineral Processing Test Work

All test data summarized in this section is based on the results provided by SGS, as detailed in the report titled 'An Investigation into Beneficiation Testwork on Low Grade and Variability Samples from the Wicheeda Rare Earth Deposit,' prepared for Defense Metals Corp., Project 17173-06, dated September 26, 2023.

13.3.1 Ore Sampling

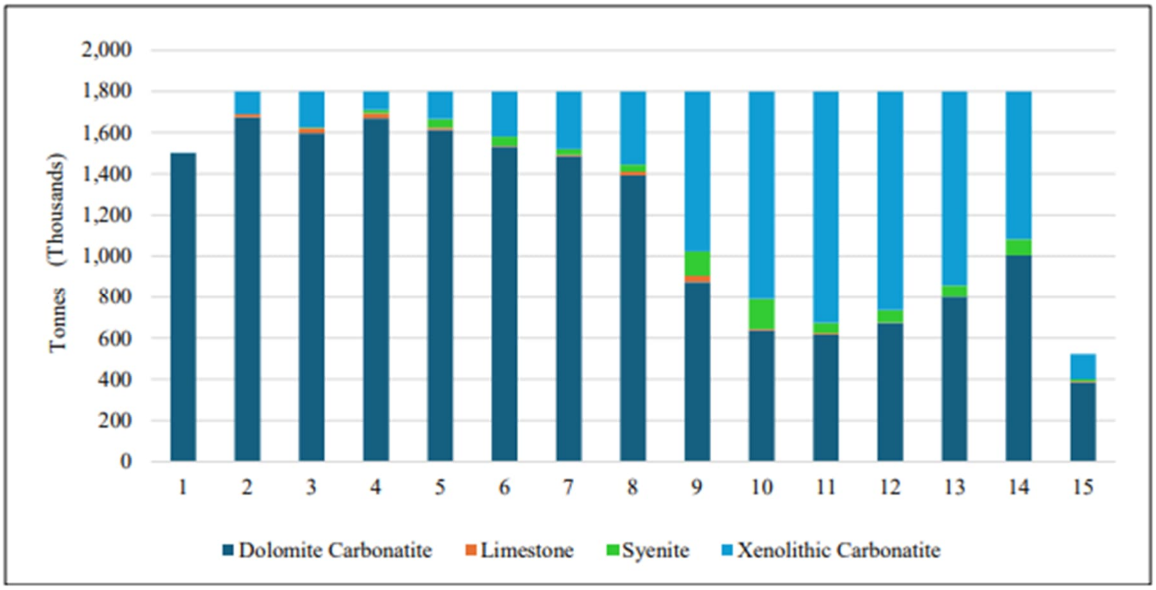
The metallurgical testing was conducted to identify the ore response to comminution, flotation, and hydrometallurgical processing and to define the most suitable process flowsheet taking into consideration lithology, mineralogy, and the mining schedule.

13.3.1.1 Deposit Domaining and Mine Plan

The Wicheeda REE deposit is characterized by three main REE-bearing lithologies: dolomite carbonatite (DC), which is the dominant lithology, Xenolithic Carbonatite (XE), and Syenite (SYN). Limestone (LIM) is the major waste rock lithology. Figure 13-1 shows a general representation of the project mine plan (see Section 16-10), including the ratio of each lithology on a year-by-year basis.

---

[1] In November of 2018, Defense Metals Corporation exercised the option to acquire Spectrum Mining Corporation (the acquisition was finalized in early January 2022).



Source: SRK, 2025

Figure 13-1: Wicheeda Mine Plan

13.3.1.2 *Sample Selection and Sample Preparation*

The metallurgical testing completed during this phase of the project was conducted with 26 samples representing the three major lithology groups defined for the Wicheeda deposit including 10 composite and 16 variability samples. The variability samples represent different locations within the pit limits as well as a range of total rare earth oxides (TREO) grades and processing years in accordance with the mine plan for the project. Most of the samples were produced using half cores from contiguous drill core intervals. Most of the samples were prepared with 15 m of drill core intervals, and the weight of each one varied between 15 kg and 35 kg.

The master composite samples MC and NMC represent the average feed composition for the project's life of mine based on lithology ratios. These samples were composited with several discrete intervals of drill core selected from various locations and depths from each lithology domain. The DC, XE and SYN composites represent the three main lithologies DC-XE2, and DC-XE3 are composite samples representing different ratios of DC and XE lithologies, and DC-SYN1 and DC-SYN2 are composite samples representing different ratios of DC and SYN ore lithologies. In addition, a tenth composite sample, DC_XEN1, was prepared by Defense Metals. This sample is a blend of DC and XE ore and was included in the variability test work program, as shown in Table 13-1.

Table 13-1: Composite Samples Blend Specifications

Sample ID	Sample Composition
Master Comp (MC)	73.7%DC, 22.5%XE, 3.8%SYN (composited by Defense Metals)
NMC Comp (NMC)	73.7%DC, 22.5%XE, 3.8%SYN (composited by SGS)
DC Comp (DC)	Composited from DC02 to DC09
XE Comp (XE)	Composited from DC01, XE1 to XE5
SYN Comp (SYN)	Composited from SYN1 and SYN2
DC-XE2	DC Comp: XE Comp = 1:2
DC-XE3	DC Comp: XE Comp = 2:1
DC-SYN1	DC03: SYN Comp = 2:1
DC-SYN2	DC Comp: SYN Comp = 2:1
DC_XEN1	DC-XE Blend by Defense Metals

Source: Hatch, 2025; SGS Data

A Master Composite (MC) weighing approximately 260 kg was prepared by Defense Metals. This composite was created using several drill core intervals from different locations and depths within the deposit pit limits, ensuring a proportional representation of the three main lithologies present within the Wicheeda deposit.

The preparation of the master composite included a total of 15 core intervals from the DC domain, approximately 79 meters of intervals, three drill core intervals from the XE domain (30 meters of core intervals), and one interval of 6 meters from the SYN domain. These 19 drill core intervals were selected to prepare the master composite (MC) sample.

A second master composite sample (NMC), weighing approximately 247 kg, was prepared by SGS using the remaining variability samples. This sample followed the same lithology makeup ratio as the master composite (MC), specifically 73.7% DC, 22.5% XE, and 3.8% SYN. The NMC sample was rotary split into 10 kg and 25 kg test charges for bulk concentrate production tests.

A second set of batch flotation tests were conducted with 11 samples at SGS from September 2023 to October 2023. The objectives of these tests were to generate additional flotation concentrate for hydrometallurgical test work and to investigate the effect of operating flotation at lower temperatures (55ºC in roughers and 60 ºC in cleaners). From the eleven tests conducted, one test was performed using an NMC composite sample, four tests utilized DC composite samples, one test used an XE composite sample, and one test was conducted with a SYN composite sample. Additionally, four new composite samples were prepared and tested as summarized in Table 13-2.

Table 13-2: New Composite Samples for Extended Flotation Tests

Sample	New Comp Makeup
DC02:DC03 Mix	DC02: DC03 = 1:4
DC04:DC05 Mix	DC04: DC05 = 2.3:1
DC06:DC08 Mix	DC06: DC08 = 1:1
DC-XE4	DC Comp: XE Comp = 1:1

Source: Hatch, 2025; SGS Data

13.3.2 Ore Characterization

To define the ore characteristics, representative samples from each of the composite and variability samples were split from the main samples and sent for head analysis and mineralogy, as indicated in Table 13-3.

13.3.2.1 Head Analysis

A representative subsample was split from each of the samples used in the metallurgical test work and submitted for head assays. These assays included measurements for Total Rare Earth Elements (TREE) and Total Rare Earth Oxides (TREO), as well as specific elements such as lanthanum (La), cerium (Ce), praseodymium (Pr), neodymium (Nd), terbium (Tb), dysprosium (Dy), and yttrium (Y). Each sample was also sent for whole rock assays, which included measurements of SiO_2, Al_2O_3, Fe_2O_3, MgO, and CaO. Additionally, the samples were analyzed for scandium (Sc), uranium (U), thorium (Th), fluorine (F), total sulphur (S), and sulphide sulphur (S^{2-}). A summary of these results together with the whole rock assays results can be seen in Table 13-4.

Compared with the results from the 2019 trial, Table 13-3 shows that the TREO grade in the 2019 bulk sample (DM composite) was relatively high, at 4.8 wt% TREO. The results also indicate that the composite samples prepared by SGS are very similar and generally within the same range, with the NMC composite sample TREO grades being slightly higher than the Master composite (MC).

The analysis of the specific lithology composite samples (DC, XE, and SYN), Table 13-3 shows that the TREO grades of the DC composite are much higher than the XE and SYN composite grades. This trend is consistent in variability samples, where the DC variability samples TREO grades are significantly higher, at least double, compared to the other two lithology domain variability samples.

Table 13-3: Samples Head Assays

Sample ID	REE Assays								
	TREE	TREO	La	Ce	Pr	Nd	Tb	Dy	Y
	wt%	wt%	wt%	wt%	wt%	wt%	g/t	g/t	g/t
2019 DM Head	3.94	4.84	1.41	1.81	0.17	0.46	19	47	90
Master Comp	2.07	2.49	0.72	1.00	0.08	0.23	9	25	<50
NMC Comp	2.43	2.91	0.86	1.15	0.09	0.27	11	25	72
DC Comp	2.82	3.39	1.00	1.35	0.11	0.31	10	37	-
XE Comp	1.20	1.44	0.39	0.57	0.05	0.15	6.70	22	-
SYN Comp	0.96	1.15	0.31	0.45	0.04	0.12	5.80	13	-
DC02	1.68	2.02	0.57	0.81	0.07	0.19	9	33	52
DC03	1.88	2.25	0.62	0.89	0.08	0.23	9	31	49
DC04	2.61	3.14	0.93	1.26	0.10	0.27	10	29	41
DC05	2.40	2.87	0.89	1.15	0.09	0.24	6	19	12
DC06	3.02	3.62	1.08	1.43	0.12	0.33	10	29	29
DC07	2.86	3.44	1.02	1.37	0.11	0.30	12	35	64
DC08	3.76	4.52	1.32	1.79	0.15	0.43	12	37	43
DC09	3.60	4.32	1.30	1.75	0.13	0.34	16	58	114
DC01	1.41	1.70	0.48	0.67	0.06	0.17	5	14	25
XE1	1.06	1.27	0.34	0.50	0.04	0.13	8	25	65
XE2	1.14	1.37	0.36	0.54	0.05	0.15	8	31	78
XE3	1.03	1.24	0.34	0.49	0.04	0.13	6	19	36
XE4	1.30	1.56	0.41	0.61	0.06	0.17	8	21	50
XE5	1.29	1.55	0.43	0.62	0.05	0.16	6	22	40
DC_XEN1	2.25	2.69	0.72	1.06	0.09	0.28	15	55	137
SYN1	1.04	1.25	0.33	0.49	0.04	0.14	6	20	39
SYN2	0.89	1.07	0.30	0.42	0.04	0.11	6	17	28

Source: SGS, 2023

The whole rock assay results depicted in Table 13-4 indicate that the Si and Al content in the XE and the SYN rocks are significantly higher than in the DC rock domain. Conversely, Fe, Mg and Ca content are typically higher in the DC ore, approximately 50% more than in the XE and SYN lithologies (Table 13-4).

The uranium content in the XE and SYN ore is double that of the DC ore. Additionally, the content of Th, F, S, and S^{2-} is greater in the DC domain compared to the XE and SYN domains.

Table 13-4: Whole Rock Assays

Sample ID	Whole Rock Assays					Others					
	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Sc	U	Th	F	S	S^{2-}
	wt%	wt%	wt%	wt%	wt%	g/t	g/t	g/t	wt%	wt%	wt%
2019 DM Head	2.28	0.35	9.25	12.8	27.3	<25	1	801	-	-	-
Master Comp	7.81	2.31	8.82	11.9	24.9	-	4	420	0.14	0.47	0.47
NMC Comp	9.63	2.94	8.48	11.9	23.8	-	5	461	-	-	-
DC comp	5.63	1.63	9.04	12.7	25.3	-	3	507	0.27	-	-
XE Comp	21.5	6.63	6.52	9.47	18.8	-	13	345	0.13	-	-
SYN Comp	29.7	8.94	5.54	7.39	16.1	-	16	309	0.16	-	-
DC02	3.91	1.14	9.59	13	26.6	<25	4	405	0.12	0.34	0.36
DC03	11.4	3.22	7.97	11.8	23.7	<25	4	475	0.14	0.41	0.45
DC04	5.37	1.51	10.2	12.1	25.3	<25	27	416	0.21	0.85	0.77
DC05	11.8	3.23	10.7	11.9	20.9	<25	6	298	0.65	0.87	0.75
DC06	2.95	0.88	8.89	13.5	26.4	<25	<1	529	0.32	0.35	0.32
DC07	3.41	0.84	7.79	14.5	26.7	<25	4	671	0.23	1.58	1.32
DC08	3.69	1.48	8.14	12.9	27.4	<25	1	709	0.39	0.76	0.62
DC09	1.8	0.58	9.12	13.2	26.6	<25	4	579	0.13	0.1	0.11
DC01	25	7.42	5.73	8.49	17.8	<25	17	297	0.15	0.09	0.1
XE1	28.4	8.76	5.99	7.75	15.9	<25	19	275	0.17	0.15	0.16
XE2	15.3	4.56	6.91	11	22.3	<25	10	321	0.13	0.24	0.26
XE3	25.2	7.51	6.17	8.73	17.1	<25	17	263	0.12	0.15	0.14
XE4	15.5	5.99	7.02	11.8	20.7	<25	9	424	0.16	0.3	0.29
XE5	21.1	6.15	7.25	9.13	18.9	<25	12	399	0.07	0.5	0.43
DC_XENI	14.4	4.26	8.06	10.1	22.4	<25	6	470	0.14	1.86	1.57
SYNI	28.1	9.13	5.72	8.34	15.4	<25	16	315	0.2	0.2	0.2
SYN2	31.1	8.73	5.2	6.48	17	<25	15	324	0.12	0.14	0.14

Source: SGS, 2023

13.3.3 Mineralogy Model

The Master Composite and 16 variability samples with an addition DC-XEN1 composite sample were submitted for mineralogical analysis by TIMA-X. The master composite was ground to a P_{80} of approximately 125 µm, while the variability samples were stage ground to a P_{80} of approximately 75 µm. The results of the modal mineralogy are summarized in Table 13-5.

Table 13-5: Minerals Modal Summary

Association / Fraction	Variability Sample																
	Master Comp	DC02	DC03	DC04	DCO5	DC06	DC07	DC08	DC09	DCO1	XE1	XE2	XE3	XE4	XE5	DC_XENI	SYN1	SYN2
	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %	wt. %
Monazite	2.1	1.7	1.3	1.8	0.5	0.4	1.4	0.5	5.7	0.9	1.2	1.0	0.9	1.4	1.8	3.1	1.1	1.3
Synchysite/ Parisite	2.0	2.1	1.6	2.7	1.6	4.9	1.4	4.9	2.3	1.6	0.6	1.0	0.9	0.9	0.8	1.3	0.6	0.6
Bastnasite	0.9	0.5	1.4	2.5	3.3	2.2	4.6	3.5	0.7	1.0	0.5	0.4	0.7	0.8	0.1	0.5	1.0	0.1
Other REM	0.0	0.0	0.2	0.0	0.0	0.1	0.1	0.1	0.1	0.0	0.0	0.1	0.0	0.0	0.0	0.1	0.0	0.0
Thorite	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Pyrochlore	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.1	0.1	0.1	0.2	0.0	0.1	0.0	0.1	0.2
Calcite/ Dolomite	39.4	33.9	45.6	28.3	17.0	47.6	73.6	63.7	39.0	29.1	25.9	41.3	28.7	46.2	24.2	46.0	30.7	27.2
Ankerite/ Siderite	41.3	53.5	30.7	53.0	51.7	38.4	8.3	17.8	48.1	25.3	24.8	28.3	24.6	21.7	34.7	20.4	17.8	21.2
Quartz/ Feldspars	11.1	5.7	16.2	6.7	11.2	4.0	4.5	4.0	1.7	39.4	42.6	23.0	41.1	23.2	34.2	20.9	42.0	44.1
Biotite/ Chlorite	0.6	0.2	0.8	0.8	11.8	1.1	0.7	2.3	1.1	0.6	1.3	0.8	0.7	2.4	0.6	0.7	3.2	2.9
Other Silicates	0.1	0.1	0.3	0.1	0.5	0.1	0.3	0.2	0.1	1.4	1.2	1.1	1.3	1.2	1.3	0.7	1.6	1.3
Apatite	0.2	0.6	0.0	0.0	0.2	0.0	0.1	0.0	0.2	0.0	0.7	1.4	0.1	0.5	0.4	1.8	0.2	0.3
Barite/ Celsian	0.1	0.5	0.1	0.0	0.1	0.2	0.1	0.1	0.3	0.1	0.2	0.0	0.1	0.1	0.1	0.1	0.1	0.1
Pyrite	1.8	1.2	1.5	3.6	1.6	0.8	4.3	2.3	0.4	0.3	0.6	1.0	0.6	1.2	1.5	4.0	0.7	0.5
Other	0.3	0.2	0.3	0.4	0.7	0.2	0.4	0.7	0.3	0.3	0.5	0.5	0.4	0.4	0.3	0.5	1.0	0.4
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Source: SGS, 2023

The dominant rare earth minerals identified are monazite, synchysite / parisite and bastnasite, with their content varying significantly across the variability samples. In the DC domain, synchysite / parisite is the predominant rare earth mineral in most of the samples. However, this is not consistent across all samples; for instance, bastnasite is the primary mineral in samplesDC05 and DC07, while monazite dominates in sample DC09. In the XE and SYN domains, monazite is the main rare earth mineral. The most abundant gangue minerals are calcite/dolomite, ankerite/siderite and quartz/feldspars, collectively representing at least 80% of the total mass of each sample. Similar to the rare earth minerals, the gangue minerals also show significant variability within the samples. Generally, calcite/dolomite and ankerite/siderite are the major components of the DC ore. Quartz/feldspars are the most abundant in the SYN ore, while in the XE domain, the distribution of the three minerals is very even.

13.3.3.1 Mineralogy Liberation

A mineral particle is considered liberated when at least 80% of its surface area is liberated from other minerals. The degree of liberation of a mineral particle depends on factors such as the mineral size and the grinding particle size. Table 13-6 presents the summary of the liberation degree of rare earth minerals (REM) as a function of grinding particle size.

To study the mineral’s liberation, each REM sample was split in three fractions +106 µm, -06 µm to +53 µm, and -53 µm. For the three samples studied, the REM liberation improved from poor to reasonably good when the average particle size was reduced from +106 µm to -106 µm/+53 µm. In the finer particle size fraction (-53 µm), there was a significant improvement in the liberation of monazite. However, this factor did not extend to the liberation of the synchysite / parisite and bastnaesite which only showed slight improvement in the finer fraction.

Table 13-6: Rare Earth Minerals Liberation

Liberation Degree %	Monazite				Synchysite / Parisite				Bastnasite			
		Fraction size range (µm)				Fraction size range (µm)				Fraction size range (µm)		
	Combined	+106	-106/+53	-53	Combined	+106	-106/+53	-53	Combined	+106	-106/+53	-53
≥90	65.7	42.9	53.7	75.6	66.7	57.9	67	70.8	57.9	46.6	60.5	62.1
≥80<90	8.57	9.27	9.05	8.23	8.9	9.59	9.98	8.04	10.9	10.6	10.6	11.2
≥70<80	3.94	2.41	5.2	4.02	4.98	6.73	3.84	4.7	6.53	8.43	3.4	6.97
≥60<70	3.22	3.8	5.49	2.39	3.19	4.28	2.45	3.03	5.3	9.55	3.97	3.85
≥50<60	3.34	5.8	4.03	2.45	2.91	2.46	3.3	2.95	5.12	4.64	6.2	4.89
≥40<50	2.03	1.25	4.27	1.6	2.36	2.68	2.33	2.22	3.19	3.34	3.99	2.78
≥30<40	3.59	9.46	4.76	1.59	2.16	2.67	2.3	1.85	2.41	3.82	1.76	2.02
≥20<30	2.69	6.64	3.15	1.43	2.33	3.46	2.06	1.93	2.59	3.49	2.04	2.41
≥10<20	3.39	8.32	5.41	1.41	2.69	3.87	2.51	2.21	2.71	3.67	3.39	1.96
≥10	3.53	10.1	4.9	1.27	3.75	6.37	4.27	2.23	3.35	5.84	4.18	1.82
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Source: Hatch, 2025; SGS Data

Table 13-7 summarizes the percentage liberated of the three main REM across all samples. For the study, each sample was ground to a P_{80} of approximately 75 µm. At this P_{80} the REM liberation was reasonably good, with the monazite and the synchysite / parisite showing similar results and better than the results shown by bastnasite.

Table 13-7: REM % Liberated in Variability Samples

% Liberated	P8O	Monazite	Synchysite/ Parisite	Bastnasite
	µn	%	%	%
DM Head	200	81.4	85.1	52.8
Master Comp	125	74.3	75.6	68.8
DC02	≈75	83	81.1	69.6
DC03	≈75	79.1	70.7	71.3
DC04	≈75	84.2	78.8	74.3
DC05	≈75	71.3	64	74.7
DC06	≈75	88.6	84.2	72.1
DC07	≈75	78.9	65.3	82.9
DC08	≈75	70.6	76.5	67.8
DC09	≈75	80.3	77.3	76.5
DC01	≈75	83.5	80.3	68.1
XE1	≈75	78.9	78.6	73.9
XE2	≈75	77.3	81.2	71.8
XE3	≈75	82.2	81.6	76.2
XE4	≈75	75.5	77.9	68.3
XE5	≈75	82	84.3	64.8
DC_XEN1	≈75	86.4	79.5	62
SYN1	≈75	72.9	68	69.1
SYN2	≈75	75.2	83	60.6
Average DC Litho		79.5	74.7	73.7
Average XE Litho		79.9	80.7	70.5
Average SYN Litho		74.1	75.5	64.9

Source: Hatch, 2025; SGS Data

13.3.4 Comminution Ore Characterization - Grinding Test Work

Comminution test work during 2023 included 18 samples from the Wicheeda deposit: one master composite sample and 17 variability samples, such as the DC-XEN1 composite. Tests conducted were SMC and Bond ball mill grindability for all samples, with Bond rod mill grindability and Bond abrasion tests performed on the master composite sample only. A summary of the grinding test work results is provided in Table 13-8.

The results indicate that the Wicheeda ore is soft to medium soft and very amenable to semi-autogenous grinding (SAG) and ball mill grinding. Within the pit limits the test work results indicate that the material from the DC domain is much softer than the other two domains (XE and SYN) with respect to the SAG and Ball mill grinding.

By contrast, the ore from the SYN domain is the hardest in terms of SAG and ball mill grinding, although this conclusion is based on only two samples. Additional tests are recommended to confirm these results in the future. The hardness difference between the DC ore and the XE and SYN ores is an important factor to consider when defining the characteristics of the optimum grinding circuit configuration for processing the Wicheeda ore.

Table 13-8: Grinding Test Work Results Summary

	SMC Tests Results						Bond Tests
Sample	**A x b**	**ta**	**Relative**	**Mia**	**Mih**	**Mic**	**BWI @65M**
Name			**Density**	**(kWh/t)**	**(kWh/t)**	**(kWh/t)**	**(kWh/t)**
Master Com (MC)	89.5	0.8	2.88	10.1	6.5	3.4	8.70
DC01	109	0.99	2.85	8.7	5.3	2.7	11.70
DC02	127	1.12	2.92	7.7	4.6	2.4	8.50
DC03	154	1.38	2.89	6.6	3.8	2	8.80
DC04	126	1.1	2.97	7.7	4.6	2.4	8.00
DC05	72.5	0.63	2.98	11.9	8	4.1	9.70
DC06	101	0.88	2.97	9.1	5.7	3	9.10
DC07	103	0.9	2.97	9	5.7	2.9	9.20
DC08	92.9	0.82	2.95	9.8	6.3	3.2	9.20
DC09	114	0.98	3.02	8.3	5.1	2.6	8.00
XE1	91.4	0.86	2.76	10	6.3	3.3	11.90
XE2	95	0.86	2.85	9.7	6.1	3.2	10.10
XE3	82.2	0.76	2.79	10.9	7.1	3.7	11.00
XE4	81.4	0.73	2.87	11	7.1	3.7	10.60
XE5	102	0.95	2.78	9.2	5.7	2.9	11.40
DC_XEN1	79.1	0.69	2.98	11.1	7.4	3.8	9.30
SYN1	73.6	0.68	2.79	12	7.9	4.1	12.80
SYN2	45.5	0.43	2.73	17.7	12.8	6.6	12.20

Source: Hatch, 2025; SGS Data

13.3.5 *Flotation Optimization and Variability Study*

Most of the flotation flowsheet optimization test work was completed with Master Composite representative samples, but the initial flotation tests were conducted on the 2019 DM Head sample with the aim to re-establish a baseline, confirm the findings from previous flotation development studies, and advance the testing schedule.

During this laboratory test campaign, a total of 86 flotation tests including 34 flowsheet development tests and 52 variability tests were completed with the objective of investigating the impact of:

- Primary grind sizes
- Collector types and collector dosages
- Depressant types and dosages
- Use of an activator
- Conditioning times and conditioning intensity schemes
- Rougher, rougher scavenger and cleaner flotation temperatures
- Different pulp densities
- Confirmatory tests with specific domain composite samples (DC, XE, and SYN)
- Locked cycle tests (LCT).

As a preface to the detailed discussion that follows, it should be noted that most of the tests have been done as open circuit test which is standard practice for flotation test work. Open circuit tests generally yield a lower grade-recovery relationship compared to a plant where cleaner tailings are recirculated to recover values from cleaner tailings and maximize extraction. A limited number of locked cycle tests (LCT) which more accurately simulate plant performance were completed.

13.3.5.1 *Batch Flotation Test Work*

The recent flotation test program was completed by SGS in 2023. The flotation feed was prepared by stage grinding to target particle sizes, which included multiple stages of screening and grinding of the screen oversize. The purpose of stage grinding the feed was to minimize slimes generation.

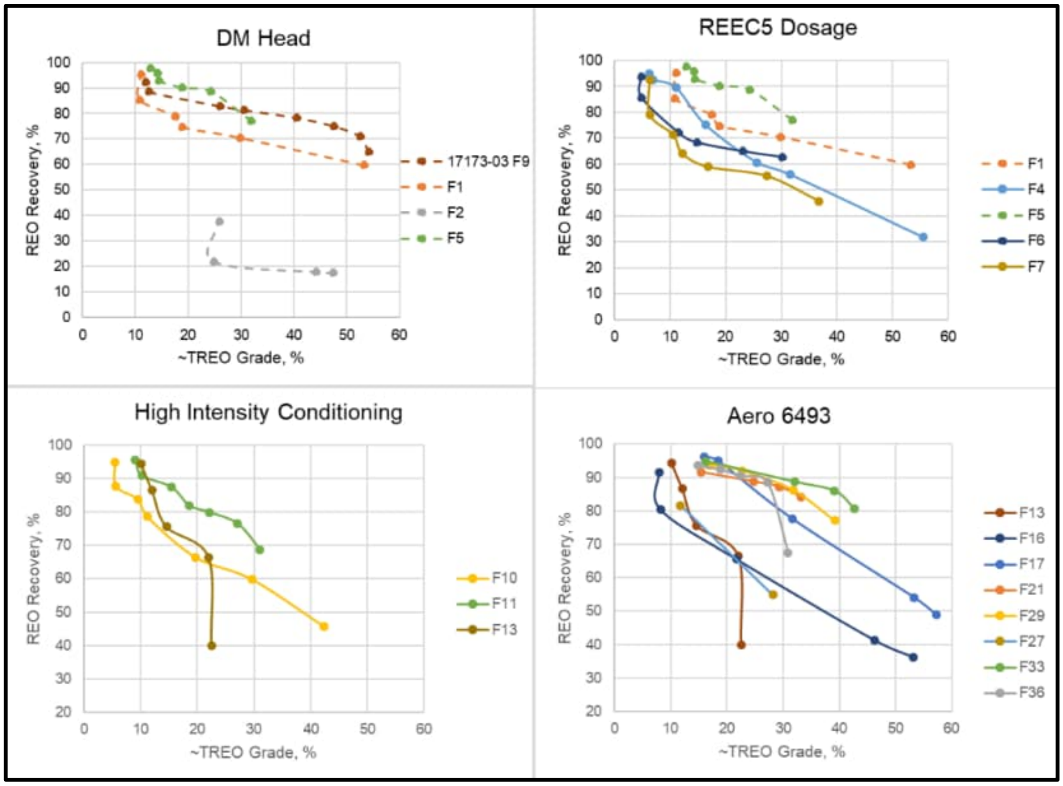
The flotation feed was conditioned at approximately pH 9, at elevated pulp temperature with the range of 50-70°C, and at a high pulp density (~50%). The conditioning stages included pH adjustment with soda ash, followed by conditioning with depressants and dispersants, and lastly collector conditioning. The rougher flotation was performed at 35% pulp density. The rougher and rougher scavenger concentrate was then combined and cleaned in two to three stages. After satisfactory results were achieved with the Master Comp, the flowsheet was evaluated using the variability samples and various composites including specific domain composites samples and specific blends of domain composite samples.

13.3.6 *Flowsheet Development and Optimization*

13.3.6.1 Collector Scheme and Conditioning Intensity

The collector scheme and conditioning intensity were investigated using sub-samples from the Master Composite. Initial batch flotation tests with the 2019 DM Head sample established a baseline, showing similar results to previous tests by SGS in 2020. The REO recovery with the hydroxamate collector Aero 6493 was poor compared to collectors REEC5 and AF3. Increasing collector dosages in the cleaner stages aligned with the 2019 TREO grade/recovery curve but did not produce a high-grade final concentrate.

High-intensity conditioning (HIC) in the rougher and rougher scavengers increased TREO recovery by about 10% without reducing concentrate grade. However, HIC in the cleaner stages resulted in lower TREO grade and recovery, so it was only applied in the rougher stages for subsequent tests. Reducing collector dosages in the rougher and rougher scavenger stages improved performance in the 2nd cleaner, with further reduction to 110 g/t significantly enhancing both TREO grade and recovery. A summary of the test results is shown in Figure 13-2.



Source: SGS, 2023

Figure 13-2: Collector and Conditioning Evaluation Tests Results

13.3.6.2 Depressant Evaluation

This part of the SGS test campaign was to evaluate alternative depressants to replace or complement the reagents, sodium fluorosilicate (Na_2SiF_6) and F220, used in the previous study. The alternatives tested included sodium silicate (Na_2SiO_3), sodium fluoride (NaF), and Calgon, using samples from the Master Composite, DM Head Composite, and DC Composite at grind sizes of -150 µm and -106 µm.

The results showed that sodium fluorosilicate was crucial for maintaining the selectivity of rare earth minerals during flotation. Replacing it with sodium silicate or sodium fluoride resulted in poor flotation responses, and reducing the F220 dosage significantly decreased TREO grade and recovery. Using Aero 6493 and sodium silicate as the sole collector and depressant did not yield favorable results, and adding Calgon in the cleaner stage did not improve outcomes. The flotation conditions of this series of tests are summarized in Table 13-9.

Table 13-9: Depressant Evaluation Test Conditions

Test ID	Sample	F100	Ro + Scav1, g/t							Cleaners, g/t							
		µm	REEC5	AF3	Aero 6493	F200	Na2SIF6	Na2SI03	NaF	REES5	AF3	Aero 6493	F200	Na2SIF6	Na2SIO3	NaF	Calgon
F2	DM Head	150	-	-	200	2200	500	-	-	-	-	-	125	75	-	-	-
F3	DM Head	150	-	-	600	-	-	500	-	-	-	50	-	-	100	-	-
F4	MC	150	200	200	-	2200	500	-	-	15	15	-	350	300	-	-	-
F6	MC	150	140	140	-	2200	500	-	-	20	20	-	450	300	-	-	-
F8	MC	150	170	170	-	2200	-	500	-	25	25	-	400	-	250	-	-
F12	MC	150	170	170	-	2200	-	1100	-	20	20	-	-	-	-	-	50
F13	MC	106	170	170	-	2200	500	-	-	15	15	-	400	250	-	-	-
F14	MC	106	170	170	-	2200	-	-	250	15	15	-	400	175	-	175	-
F22	DC Comp	106	110	110	-	2200	500	-	-	5	10	10	300	125	-	-	-
F23	DC Comp	106	110	110	-	1200	500	-	-	10	15	10	300	225	-	-	-

Source: SGS, 2023

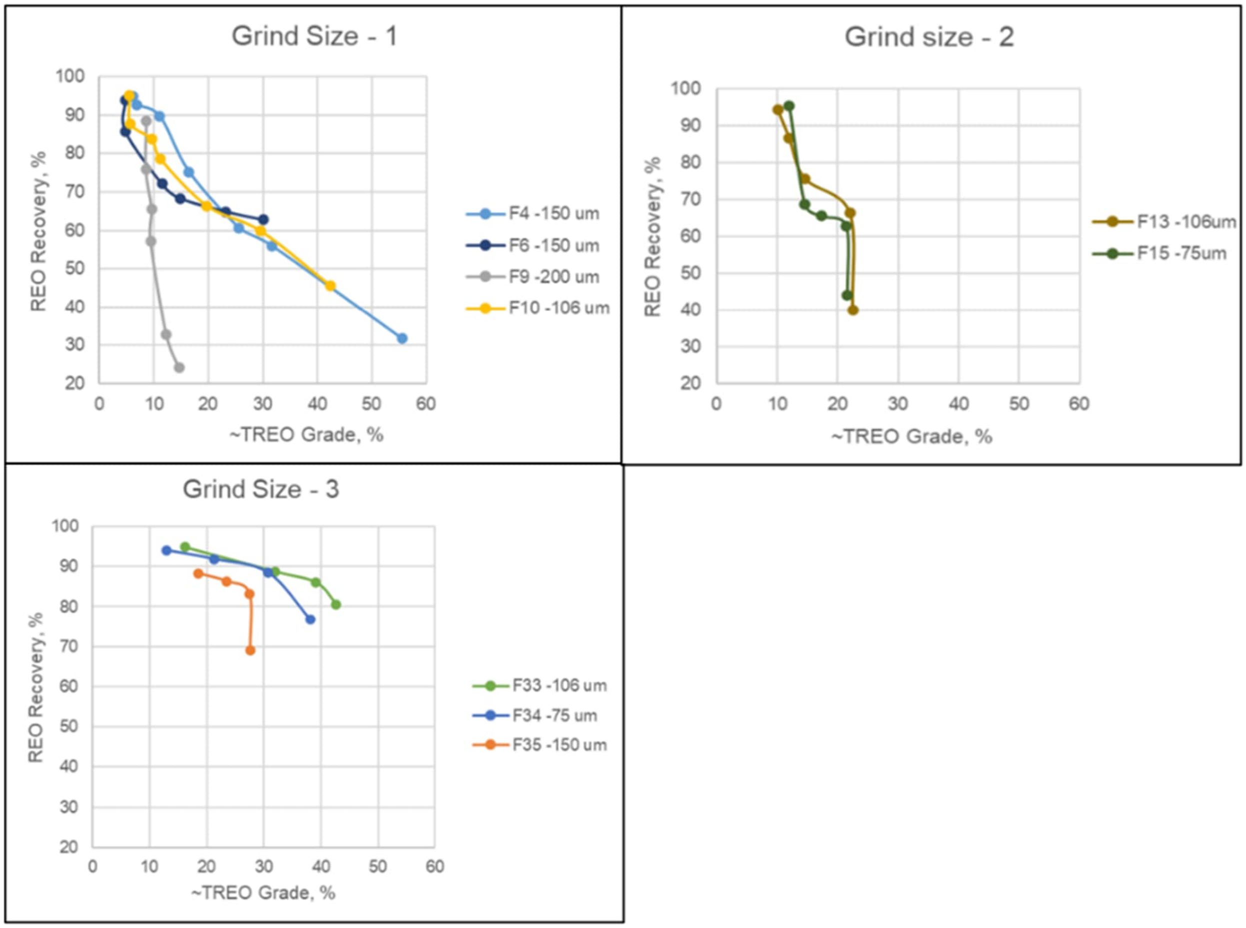
13.3.6.3 Primary Grind Evaluation

The SGS test work evaluated the impact of primary grind particle size on rougher and rougher scavenger flotation using Master Composite split samples. The particle sizes tested were F100 of 75 µm, 106 µm, 150 µm, and 200 µm, and the study was conducted in three phases.

In the first phase, samples ground to F100 of 106 µm, 150 µm, and 200 µm were tested, with collector dosages adjusted for particle surface area changes. The results showed that the finer samples (106 µm and 150 µm) performed similarly, while the 200 µm grind was too coarse, resulting in poor flotation response.

The second phase which compared the sample response to flotation at the finer primary grinds, 75 µm and 106 µm showed similar performance for both grind size but indicated the need for optimized test conditions.

In the final phase, a modified and optimized reagent scheme was used to evaluate the three finer grinds (-75 µm, -106 µm, and 150 µm) under the same conditions. The results demonstrated that the 150 µm grind had a poor flotation response compared to the finer samples. The sample ground to minus 106 µm showed the best results, leading to the use of F100 minus 106 µm for all subsequent tests. Figure 13-3 shows the grind size test results.



Source: SGS, 2023

Figure 13-3: Primary Grind Evaluation Test Results

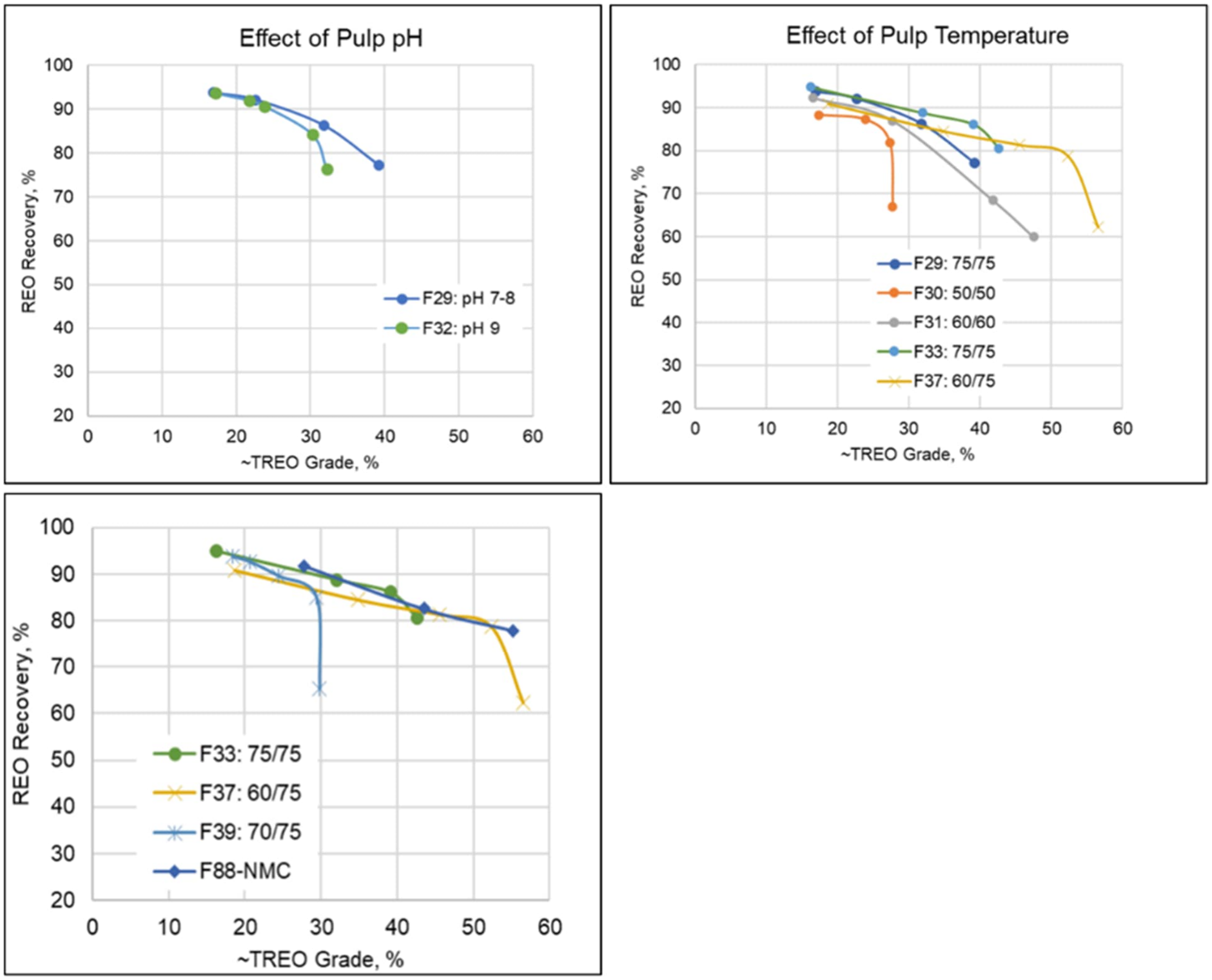
13.3.6.4 Effect of Slurry pH and Temperature

The effects of the pH and the slurry temperature on the TREO flotation performance were investigated. Most tests were maintained at a natural pulp pH between 7 and 8 in the cleaners, with two tests at pH 9 (adjusted with sodium hydroxide) resulting in lower grade concentrate.

The base case flotation temperature was 75°C for rougher, scavenger, and cleaner stages. Alternative temperatures of 30°C, 50°C, and 60°C were tested. At 60°C, the flotation performance was slightly inferior to that at 75°C. At 50°C, rougher flotation showed a poor response and a sharp drop in concentrate grade, indicating that cleaner stages cannot be conducted below 60°C. However, rougher stages could be performed at temperatures below 60°C.

Tests with different temperature schemes, involving lower temperatures for rougher and rougher scavenger stages followed by higher temperatures for cleaner flotation, showed that with the right combination, recoveries of around 80% and final concentrate grades of approximately 50% TREO are achievable. Figure 13-4 presents the test results.

These conditions were used to define the processing parameters for the locked cycle tests.



Source: SGS, 2023

Figure 13-4: pH and Slurry Temperature Flotation Test Results

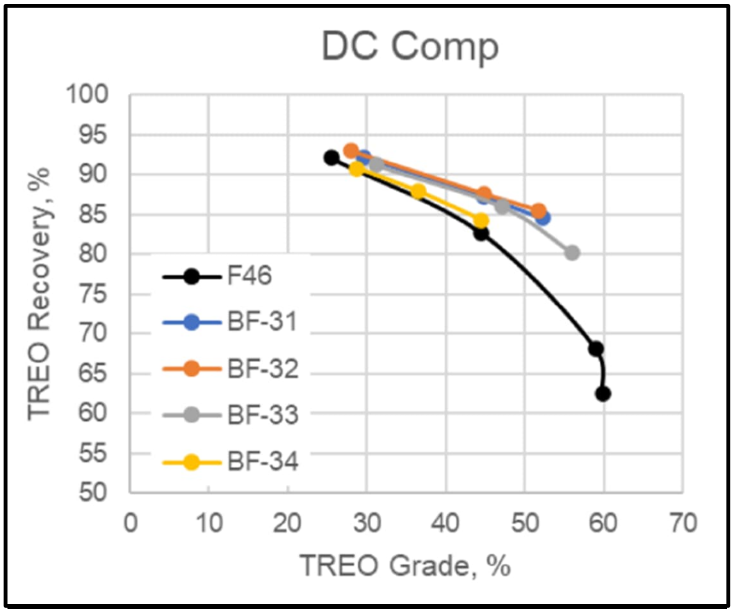
Additional tests conducted from September 2023 to October 2023 with the objective of generating bulk concentrate for hydrometallurgical testing were used to further investigate the effect of the flotation temperature on the ore samples from the Wicheeda deposit. The investigation is detailed in the "**Summary of the Resumed Flotation Tests BF-30 to BF-40**", SGS Canada – Project 17173-06, January 15, 2024.

Following the completion of the flotation flowsheet development program, test F46 was assumed to represent the updated flotation baseline conditions (reagents and temperature) for the DC ore. The tests included evaluations at the baseline temperature and at lower rougher and cleaner flotation temperatures of 55°C and 60°C, respectively. A summary of the test results is depicted in Figure 13-5, and a summary of test condition is provided in Table 13-10.

Table 13-10: Summary of Additional Bulk Flotation Test Conditions

Test ID	Sample	Ro + Ro Scav1, g/t				Cleaners, g/t			
		Pulp Temp °C	REEC5	AF3	Aero 6493	Pulp Temp °C	REEC5	AF3	Aero 6493
F-46	DC Comp	60	110	110	10	75	10	10	10
BF-31	DC Comp	60	110	110	10	75	10	10	5
BF-32	DC Comp BF 31 Repeat	60	110	110	10	75	10	10	5
BF-33	DC Comp	55	110	110	10	60	10	10	5
BF-34	DC Comp BF 33 Repeat	55	110	110	10	60	10	10	5

Source: Hatch, 2025; SGS Data



Source: SGS, 2023

Figure 13-5: Additional Flotation Test for Temperature Optimization

The results indicate that the performance of the two tests conditions is very similar, and in some cases, the test at lower rougher and cleaner temperatures performed slightly better than the baseline test F46. The slightly reduced pulp temperature of 55°C in the rougher and 60°C in the cleaners showed no significantly negative effect on the flotation results compared to tests performed at 60°C in the rougher and 75°C in the cleaners. In both scenarios, the average *grade of* 2nd Cleaner concentrate was around 50% TREO with an approximate recovery of 84%.

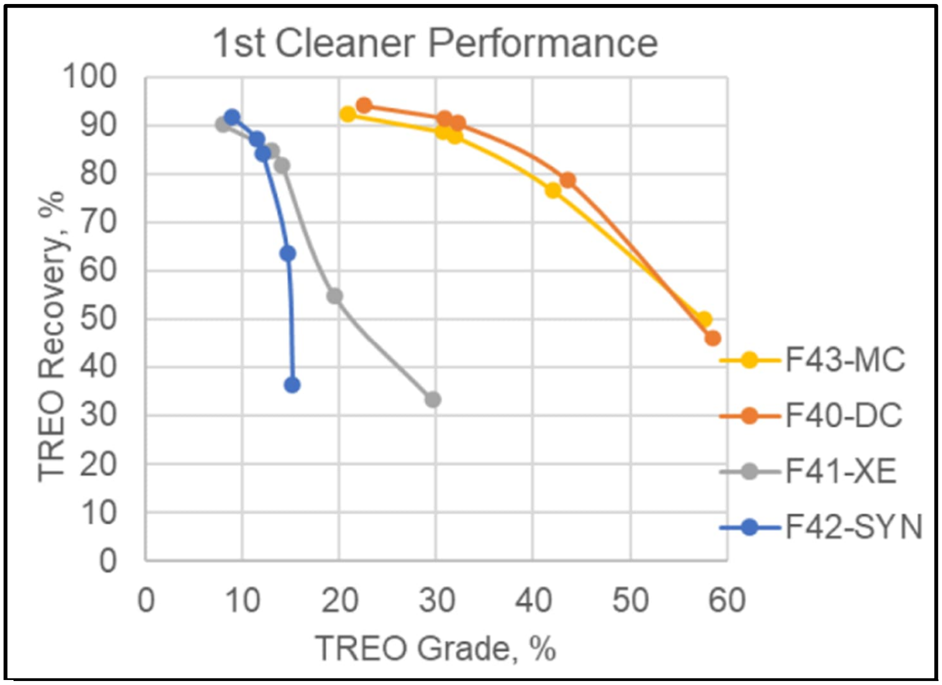
13.3.6.5 Variability Study

The optimum flowsheet and flotation conditions developed with the Master Comp were then used to evaluate the performance of the mini composites that were prepared from different lithologies

13.3.6.6 *Mini Comp Flotation Tests*

The mini composites were DC Comp, XE Comp, and SYN Comp. Various mixtures of the mini composites were also prepared to evaluate the performance of mixtures of DC and XE Comp or SYN Comp. 1st cleaner kinetics tests were also performed to understand flotation recovery and grade as a function of flotation residence time. for each flotation stage. A graphic representation of the test results can be seen in Figure 13-6. It should be noted that in this series of tests, the rougher flotation temperature and cleaner flotation temperature were 60°C and 75°C, respectively.

The DC Comp performed similarly to the Master Comp. Both samples demonstrated the potential to generate a very high-grade REO concentrate associated with a high TREO recovery as well. With longer residence time, the recovery increased significantly at the cost of reduced grade.



Source: SGS, 2023

Figure 13-6: Result Summary of Mini Comp Flotation Tests

The DC Comp yielded slightly better recovery to the flotation concentrate compared to the Master Comp, which could be due to the presence of XE and SYN (of poorer flotation performance) in the MC composite. Both XE Comp and SYN Comp yielded lower grades and recoveries to the flotation concentrates and the initial test results were poor. This is also indicative that the overall flotation performance would be much better during the first 8 years of operation when the feed blend is over 95% DC ore, than later when the content of XE and the SYN in the feed blend are much higher.

By changing the reagent scheme (dosages) it was possible to improve the quality of the second cleaner concentrate when floating XE and SYN composite samples. The improvement for the XE flotation was better than the values achieved for the SYN Comp, but both presented inferior performance to the DC ore.

13.3.6.7 Variability Samples Flotation Tests

The developed flotation flowsheet was evaluated on the 17 variability samples, the flotation conditions for the completion of these test was the baseline defined from the flowsheet development program as shown in Table 13-11.

Table 13-11: Variability Test Work Baseline Conditions

Item	Unit	Value
Grind Size F100	µm	106
Rougher temperature	°C	60
Rougher pH		9
Rougher REEC5 dosage	g/t	110
Rougher AF3 dosage	g/t	110
Rougher Aero 6493	g/t	10
Cleaner temperature	°C	75
Cleaner pH		7 to 8
Cleaner REEC5 dosage	g/t	10
Cleaner AF3 dosage	g/t	10
Cleaner Aero 6493	g/t	10

Source: Hatch, 2025; SGS Data

The collector dosages were adjusted while the depressant dosages were maintained at the same value in most tests. Most of the changes were implemented to the rougher and rougher scavenger stages, while the cleaner, except for some few exceptions, were kept the same for most of the tests. The DC variability samples all generated a high-grade TREO concentrate despite these being open-circuit tests. To achieve a TREO concentrate grade of 50%, the recovery varied from 70 to 85%. If the target TREO concentrate grade was dropped to ~45%, then the overall TREO recovery range increased to 75 to 90%. The results also indicated that the concentrate TREO grade/recovery is lower when the feed grade is lower.

Special attention is needed for sample DC09, a high head grade TREO sample that showed lower grade/recovery results compared to other DC samples with similar head grades. This may be due to its higher monazite content, which should be studied further as it could impact operational targets.

The XE variability samples followed the same trend as the XE mini composite samples, with generally lower grade and recovery compared to DC flotation performance. Similarly, the SYN variability samples showed lower grade and recovery than both XE and DC samples. This issue needs to be addressed to optimize operations, especially when XE and SYN content in the orebody is expected to increase to 40% or more of the total plant feed blend.

13.3.7 Locked Cycle Flotation Tests (LCT)

A total of three flotation locked cycle tests (LCT) were conducted by SGS to identify the best circuit configuration for the processing of the Wicheeda ore. Two of these tests (LCT1 and LCT2) were conducted between July and August 2022 with split samples from the Master composite, while the third test was conducted with representative charges split from the NMC sample and was completed in February 2023.

LCT1 and LCT2 had similar flotation conditions, with LCT1 at 70°C and LCT2 at 75°C. LCT3, using the NMC sample, had slightly different conditions. LCT3 demonstrated a more efficient circuit for processing Wicheeda ore. It achieved a high final concentrate grade of nearly 51% TREO, compared to approximately 34% TREO for LCT1 and LCT2. This efficiency was largely due to less recirculation in the cleaner stages, reducing contamination from gangue minerals. Additionally, LCT3 used a larger charge size of 12 kg, contributing to more stable and efficient flotation performance. The test also reached a steady state more quickly, ensuring consistent results. Overall, LCT3 demonstrated better performance, projecting a second cleaner concentrate grade of 50.7% TREO at 85.4% recovery, making it the best circuit configuration for processing Wicheeda ore.

13.3.7.1 Flotation Circuit Heating

After the completion of the bulk flotation tests, the temperature scheme was changed as follows:

- Conditioning of rougher feed slurry at 55°C
- Rougher floatation at 55°C
- Conditioning of rougher scavenger feed at 55°C
- Rougher scavenger flotation at 55°C
- Conditioning of first cleaner feed at 75°C
- First cleaner floatation at 75°C
- Conditioning of second cleaner feed at 75°C
- Second cleaner flotation at 75°C.

13.4 Hydrometallurgy Test Work

Following the PEA testing, the hydrometallurgical process flowsheet was changed from a caustic crack – acid leach extraction scheme to an acid bake–water leach scheme. For the PFS, SGS conducted various hydrometallurgical bench scale tests as well as two hydrometallurgical pilot plant campaigns to develop and validate this new flowsheet. Testing was conducted using rare earth concentrate samples produced during pilot testing for the concentrator plant (Section 13.3) as well as concentrate samples produced in 2023 from bench-scale flotation tests on variability samples.

Hatch Qualified Person J. Adams visited the SGS facility in April 2023 to witness the operation of the pilot plant.

The hydrometallurgical block flow diagram is shown in Figure 17-2 (in Section 17). The bench and pilot scale tests focused on defining, optimizing, and validating each of the steps in the flowsheet. In this flowsheet, described in more detail in Section 17-1, the concentrate from the beneficiation plant is baked with concentrated sulphuric acid to convert the contained rare earths to water-soluble species that are extracted via water leaching. The leach solution is neutralized in multiple steps using magnesium oxide slurry and subjected to ion exchange to remove impurities. In an earlier version of the flowsheet, the rare earths in the purified process leach solution (PLS) are precipitated from the purified solution, and magnesium is then removed from the process liquor via precipitation. In the current flowsheet, the PLS is treated in a solvent extraction (SX) circuit to extract all the rare earths between Pr and Lu, and Y, and the raffinate is neutralized to both remove magnesium and the unextracted La and Ce. The loaded solvent is stripped and the rare earths precipitated.

The results of the bench-scale test work were used to design and guide the development of the pilot-scale testing (Section 13.4.2). Defense Metals plans to continue testing (after this PFS) for additional flowsheet refinement/development work including validating the untested process steps, further benchmarking different concentrate samples, further characterization of mixed REE products, equipment-specific testing with equipment manufacturers, and other possible process optimizations.

13.4.1 PFS Bench Scale Metallurgical Test work

Bench scale tests were conducted to validate each processing step and optimize processing conditions. Bench scale tests were also conducted to assess the variability in processing different feed concentrate samples, and for validating some process steps that were not included in the pilot testing. The bench scale testing was conducted between Nov 2021. and Jan 2024. The hydrometallurgical pilot plant campaigns (Section 13.4.2.4) were designed based on the results obtained from bench scale testing. Further bench scale tests were performed after the pilot campaigns to optimize the process.

The bench scale test work has been fully documented in the SGS report: **Bench scale development of a hydrometallurgical flowsheet for the extraction and recovery of rare earth elements from the Wicheeda deposit**, 17173-04 – Final Report (April 8, 2024)

13.4.1.1 Testing Feed Materials

The majority of bench scale testing was conducted using two concentrate samples (2021 44.3% REO Conc., Bulk 44.3% REO Conc.) produced during a 2020 flotation pilot plant campaign conducted on a bulk sample of Wicheeda ore. Additional concentrate blends were produced for variability testing from the 2020 flotation pilot plant campaign products. Furthermore, the response to hydrometallurgical processing of feed with different mineralogy was assessed with tests on the concentrates produced from ore samples with different lithologies: DC (pure dolomitic carbonatite), XE (dolomitic carbonatite with xenoliths), and MC (mixed composite containing lithologies over the life of the mine). Additional concentrate samples were also produced to further test the variability of the acid baking – water leaching process.

13.4.1.2 Static Acid Bake – Water Leach (AB)

The acid baking step was tested by heating quartz crucibles containing mixtures of concentrate and 96 wt% sulphuric acid in a muffle furnace. The effectiveness of acid baking was assessed by measuring the extractions observed during water leach tests. In these trials, the tested variables were:

- Acid addition ratio
- Bake temperature
- Retention time
- Reed composition/mineralogy.

A total of approximately 60 static acid bake trials were conducted.

The primary findings from these acid bake tests were that the extraction of REEs is tied to the amount of sulphuric acid available for the REE sulphation reactions (there are evaporative losses of sulphuric acid at high temperatures). Furthermore, the co-extraction of thorium was found to decrease at higher temperatures, and the extractions of neodymium (primary light REE of interest), gadolinium (used as a proxy for the behaviour of samarium, europium, gadolinium, and heavy REEs), and iron do not appear to be significantly affected by changes to the acid strength (93 vs 96 wt% H_2SO_4) or feed concentrate moisture (< 2 vs 5 wt% H_2O).

When concentrate was mixed to a homogeneous paste with approximately 1100 kg/t acid dosage and baked at 300°C for two hours, observed extractions were above 95% for light rare earths and above 90% extractions for heavy rare earths. REE extractions were found to be consistent across multiple feed concentrate compositions and mineralogies (various samples from different locations: pure dolomitic carbonatites and dolomitic carbonate containing xenoliths), although thorium, iron, phosphorus, and fluorine extraction were variable.

13.4.1.3 Tube Furnace Acid Bake – Water Leach (AB)

Two baking tests were also conducted in a small 100 mm diameter tube furnace. These tests compared extractions when baking under induced air flow (TF1), and zero air flow conditions (TF2). The induced airflow resulted in increased sulphuric acid evaporation from 1% to 11%, decreased free acid in the leachate from 20 g/L to 13 g/L, and resulted in a decreased REE extraction (Nd from 95% to 92%, Gd from 88% to 82%). The airflow also resulted in decreased Th extraction from 96% to 0%. Unfortunately, the air flow rate in TF1 was not measured so the relevance of the tests is not exactly known. The mechanism for Th extraction reduction is not clear but likely due to enhanced decomposition of sulphate.

13.4.1.4 *Rotary Kiln Acid Bake – Water Leach (AB)*

The acid baking step was tested under continuous conditions in a 150 mm rotary kiln (including feeding and charge movement down the kiln barrel). These conditions mimic the kiln equipment and conditions employed in the pilot scale tests and the full-scale equipment which will be employed. In these trials, the tested variables were:

- acid addition ratio
- feed acid concentration
- calcine particle size
- retention time.

A total of 16 rotary acid bake trials were conducted. The water leach extractions of various elements were used to measure the effect of baking parameters on extraction efficiency.

The primary findings from these acid bake tests were that rotary kiln extractions were 5% – 10% lower than static acid bake tests (80% to 90% in kiln tests compared to 90% to 100% in the static acid bake tests) due to lower retention time and higher air flow (increased acid evaporation). Increasing temperature to 350°C and a retention time of 1.8 hours improved rare earth extractions to the 85% to 95% range, but increasing acid dosage did not affect extraction. This rotary kiln testing allowed investigations and troubleshooting of issues related to kiln residence time, agglomeration and buildup of material, and sulphuric acid concentration.

13.4.1.5 *Water Leaching (WL)*

As indicated above, water leaching trials were conducted for all the static and kiln acid bake trials. In addition to testing the acid baking parameters, water leach conditions were also assessed in these trials. The tested variables were:

- initial solids/liquid ratio
- leach temperature
- free acid concentration in the leach solution
- the use of gypsum saturated water for leaching.

The primary findings from these water leach tests were that the calcine should be leached at 8 wt% solids to maximize rare earth concentration but reduce gypsum concentration in the residue (to minimize rare earth losses due to co-precipitation with gypsum). The use of gypsum saturated water did not negatively impact REE extractions, meaning that recycled gypsum liquor can be used in the final process to reduce water usage. Furthermore, crushing or grinding the calcine to smaller pieces was found to improve REE extractions.

13.4.1.6 Primary Neutralization (PN)

Following water leaching trials, the primary neutralization step for removal of co-extracted iron, phosphorus, and thorium was tested. The trials were conducted by adjusting the pH of various liquors from water leaching tests with 10 wt% Magnesium oxide slurry within the range of 1.5 – 4.0. The tested variables were:

- Solution pH
- The effect of ferric sulphate addition
- ORP adjustment with hydrogen peroxide.

A total of 21 primary neutralization tests were conducted.

The primary findings from these trials were that Primary Neutralization removed the majority of the iron (>80%), thorium (>80%), and phosphorus (>90%) at pH 1.9 – 2 from the water leach solution. Hydrogen peroxide was employed to oxidize ferrous ions (Fe^{2+}) to ferric (Fe^{3+}), and ferric sulphate was added to maintain the phosphorus balance and prevent rare earth losses as rare earth phosphate. Running neutralization directly after the leach stage without an intermediate filtration step resulted in minor losses of rare earths. Based on the testing results, the target PN pH was set at 1.75.

13.4.1.7 Secondary Neutralization (SN)

Following the Primary Neutralization trials, the Secondary Neutralization step for removal of the remaining iron, thorium, and aluminum was tested. The trials were conducted by adjusting the pH of various filtrates from Primary Neutralization tests with 10 wt% Magnesium oxide slurry.

The tested variable was primarily the solution pH. A total of four secondary neutralization tests were conducted.

The primary findings from these trials were that Secondary Neutralization fully precipitated the remaining iron and thorium. Based on these tests, a pH between 6 and 6.5 was selected for pilot operations, tuning as required once the system reached a steady state, and the SN underflow was recycled to the WL reactor, to recover co-precipitated REEs.

13.4.1.8 Uranium Ion Exchange (UIX)

Following the Secondary Neutralization trials, some preliminary tests of ion exchange as a method for removing uranium from the process liquor were performed. The trials were conducted as 24 h shakeout tests using two different strong base anion exchange resins (UIX1: Purolite A660, UIX2: Dowex 21K XLT, both pre-conditioned with 150 g/L H_2SO_4). This trial demonstrated that UIX removed U from solution without REE losses.

13.4.1.9 Rare Earth Element Precipitation (RP)

Following the uranium ion exchange trials, the rare earth precipitation step for producing solid mixed rare earth products was tested. The trials were conducted by treating the UIX solution with various precipitating agents:

- Magnesium oxide (MgO)
- Oxalic acid ($H_2C_2O_4$)

- Sodium carbonate (Na_2CO_3)
- Ammonium bicarbonate (NH_4HCO_3)
- Sodium sulphate (Na_2SO_4).

A total of 39 rare earth precipitation trials were conducted.

The primary findings from these trials were that conducting rare earth precipitation using 10 wt% MgO slurry at 100% stoichiometric dosage allowed 100% recovery of the REEs; however, this method resulted in magnesium contamination in the final product.

For the pilot testing, both MgO and Oxalic Acid were selected for testing; however, due to concerns around product purity (MgO) and reagent costs (Oxalic Acid), Ammonium Bicarbonate was ultimately selected for the final process design as it achieved ~100% REE recovery and produced a high-purity product (< 1wt% Ca, S concentrations) at a relatively low reagent cost.

13.4.1.10 *Magnesium Removal (MgR)*

Following the Rare Earth Precipitation trials, the Magnesium Removal step was tested for removal of magnesium and ammonium from the liquor prior to recycling to water leaching. The trials were conducted by adjusting the pH of various filtrates from rare earth precipitation tests with 20 wt% calcium hydroxide slurry (slaked lime). The magnesium removal was tested for both liquors with MgO RP (Mg removal only), and NH_4HCO_3 RP (Mg and NH_3 removal). The tested variables were:

- The solution pH
- The temperature
- The lime dosage.

A total of 11 magnesium removal tests were conducted.

The primary findings from these trials were that magnesium hydroxide (along with gypsum) was precipitated using 20 wt% calcium hydroxide slurry at pH 10. Furthermore, increasing the dosage to 120% stoichiometric resulted in 76% ammonium removal (as NH_3 off-gas), alongside 100% Mg precipitation.

13.4.1.11 *LaCe Solvent Extraction (SX)*

Following the pilot plant testing, additional bench-scale testing was conducted to assess the potential to increase the NdPrO content of the final product using solvent extraction (SX).

Several tests were performed in which diluted organic extractant was contacted with UIX liquor from the PP2 pilot trials (Section 13.4.2.3) in an agitated vessel, the phases allowed to settle and then assayed. The solution contained 20,894 mg/L TREE, <0.02 mg/L U and Th, 5,200 mg/L Mg, 872 mg/L Ca, and 209 mg/L of Mn. The free acidity of the liquor was adjusted with sulphuric acid to cover a range from about 3 to about 40 g/L free acid. Nine contacts were made using 10 vol% DEHPA in Orfom SX80 and nine contacts were made using 20 vol% Cyanex 801 in Exxsol D80. In both cases, the solvents were washed with sulphuric acid before use. Different phase ratios were used to cover a range of equilibrium conditions.

The equilibrium data for the selected Cyanex 801 system were plotted as distribution coefficients for each REE versus free acidity and found to be approximately linear and normal. Separation factors were calculated: the value for the critical Ce/Pr separation was 1.7, which is close to published data for the chloride-based system.

Several solvent saponification tests were also performed using MgO and other saponification reagents. The results of this work were used in the proposed SX system design. Defense Metals intends to operate a pilot plant of the solvent extraction system to further develop and validate the SX design.

13.4.2 PFS Pilot Plant Metallurgical Test work

The pilot plant campaigns were conducted to investigate the operability of the flowsheet developed at bench scale under continuous conditions. The campaigns were split into PP1 (a 5-day commissioning and fill campaign, March 27 to March 31, 2023) and PP2 (a 10-day integrated campaign, April 24 to May 3, 2023).

The pilot testing campaigns are documented in the SGS report: **An Investigation into pilot plant testing of the Defense Metals flowsheet**, 17173-05 – Pilot Plant Report (February 1, 2024).

13.4.2.1 *Pilot Plant Description*

Feed material used in the campaigns was generated by blending the flotation pilot plant concentrates produced by SGS flotation pilot testing in 2019/2020. The overall concentrate grade was 47% TREO, with ~84% comprised of Lanthanum and Cerium, and 14% as Praseodymium and Neodymium. The head analysis is presented in Table 13-12.

Table 13-12: Bulk Wicheeda Concentrate Head Analysis

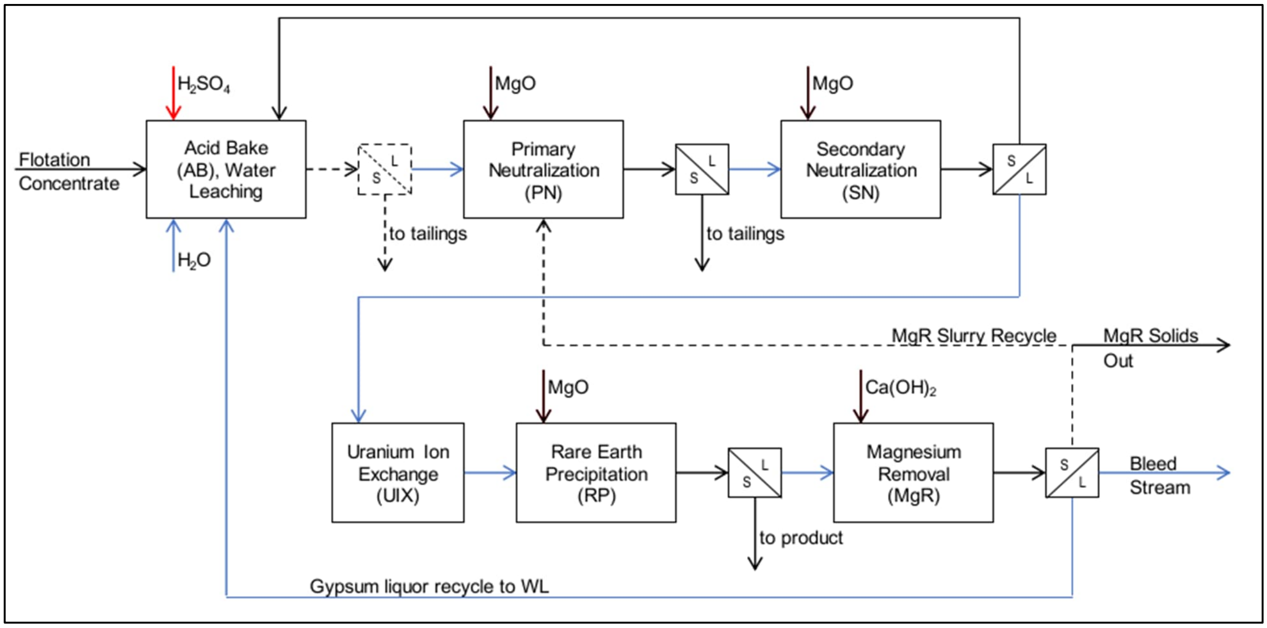
REE (g/t)		Gangue (wt%)	
La	141,000	Si	0.12
Ce	188,000	Al	0.03
Pr	15,400	Fe	2.12
Nd	41,400	Mg	2.5
Sm	4,000	Ca	9.64
Eu	817	Na	0.09
Gd	1,830	K	<0.01
Tb	141	Ti	<0.01
Dy	369	P	2.86
Ho	28	Mn	0.38
Y	639	S	0.03
Er	34.3	F	3.06
Tm	1.4		
Yb	4.1		
Lu	<4		
Th	3,870		
U	1.7		

Source: SGS, 2023

The two pilot campaigns followed the same overall process design showed in Figure 13-7.

1. Acid Bake (AB): Dry concentrate is continuously mixed with acid in a dual auger mixing system to form a homogeneous mass which is fed into a 6” internal diameter indirectly heated rotary kiln. The resulting solid calcine is collected for further processing. The kiln off-gas is directed to a column scrubber operating at pH 9.9-11.7.
2. Water Leach (WL) and Primary Neutralization (PN): Cooled calcine is continuously mixed with water in a series of overflow-equipped stirred tanks which directly flow into a series of neutralization tanks where the slurry is mixed with MgO, $Fe_2(SO_4)_3$, and H_2O_2. The WL and PN steps are operated continuously as a close-coupled system but sampled as separate circuits. The tanks are configured in a gravity overflow arrangement. The PN slurry is discharged to a thickener. The thickener underflow is filtered to collect the filtrate, and the filter solids are displacement-washed with water. The thickener overflow and filter filtrate are fed to the SN system.
3. Secondary Neutralization (SN): Liquor from PN is continuously mixed with MgO, in a series of overflow-equipped stirred tanks. The SN slurry is discharged to a thickener, and the underflow is recycled to the WL step. The thickener overflow is filtered in a cartridge string filter and fed to the UIX system.

4. Uranium Ion Exchange (UIX): The SN liquor is fed through lead/lag ion exchange columns for uranium ion removal. Prior to IX, the liquor is passed through a cartridge string filter (0.5 µm nominal pore size). The columns employ approximately 2 L of Dowex 21K XLT resin (a type I strong base anion resin) and are run in a closed and pressurized configuration.
5. REE Precipitation (RP): UIX effluent is continuously mixed with either MgO (PP1) or oxalic acid (PP2), in a series of overflow-equipped stirred tanks. The RP slurry is discharged to a thickener, the underflow is filtered, and the filtered REE precipitate is displacement-washed with water. The thickener overflow and filter filtrate are fed to the MgR system.
6. Magnesium Removal (MgR): RP liquor is continuously mixed with $Ca(OH)_2$, in a series of overflow-equipped stirred tanks. The RP slurry is discharged to a thickener and filtered.



Source: SGS, 2023

Figure 13-7: Overall Block Flow Diagram of the PP1 Pilot Flow Sheet

The key operating parameters for both pilot campaigns are summarized in Table 13-13. Further description of pilot campaigns and findings are discussed in the sections below.

Table 13-13: Pilot Campaign Key Parameters

Parameter	PP1	PP2
Overall		
Hours of Operation	107 h	211 h
Mass of Concentrate Processed	187 kg	369 kg
Acid Bake (AB)		
Concentrate Feed Rate	1.75 kg/h	
Target Sulphuric Acid Addition	1,100 kg/t	1,000 kg/t raised to 1,100 kg/t
Baking Temperature	350°C lowered to 250°C	250°C raised to 350°C
Water Leach (WL)		
Calcine Feed Rate	2.3 kg/h	
Water Addition Rate	~ 26.4 kg/h	~ 16.9 kg/h
Temperature	Uncontrolled (Ambient)	
Target Pulp Density (relative to calcine)	8%	12%
Residence Time	2.7 h	3.0 h
ORP Target	600 mV	
Primary Neutralization (PN)		
MgO Dosage	Varied to reach pH target	
Ferric Sulphate Dosage	~ 4 mL/min	~ 5 mL/min
Hydrogen Peroxide Dosage	Varied to reach ORP target	
Temperature	50°C	
Residence Time	2.0 h	2.5 h
ORP Target	600 mV	
pH Target	~ 2	
Secondary Neutralization (SN)		
Temperature	50°C to Ambient	Ambient
MgO Dosage	Varied to reach pH target	
Residence Time	3.7 h	4.9 h
pH Target	6 – 6.5	
Uranium Ion Exchange (UIX)		
Resin	Dowex 21K XLT	
Number of Columns	2	
Resin Bed Volume	2 L (per column)	
Rare Earth Precipitation (RP)		
Temperature	Uncontrolled (Ambient)	
Precipitation Reagent	Magnesium Oxide	Oxalic Acid
Residence Time	2.3 h	2.2 h
Wash Water Addition Rate	6.3 kg/h	3.17 kg/h
Magnesium Removal (MgR)		
Temperature	Uncontrolled (Ambient)	
Lime Dosage	Varied to reach pH target	
Residence Time	4.1 h	4.5 h
pH Target	~10	
Recycling to PN	Final 2 Days of Pilot Only	N/A

Source: Hatch, 2025

13.4.2.2 Pilot Plant 1

The first pilot plant (PP1) campaign operated for 5 days with its main goal of filling and commissioning the circuits and identifying potential improvements for the second pilot plant (PP2) campaign. In PP1, the water leach averaged 94% Nd recovery, as well as 94% Pr/La/Ce recoveries. The heavy REE (HREE) had a lower extraction (~78% HREE total, 81% Dy, 84% Tb). Summaries of PP1 recoveries for each process step are shown in the tables below. On the final day of the pilot, the acid bake circuit was shut down first to allow the kiln to empty and cool, while all other circuits were stopped mid-operation with pulps recovered to use as prefill for PP2. Overall summaries for PP1 are shown in Table 13-14, Table 13-15, and Table 13-16.

Table 13-14: PP1 Overall Average Extractions / Precipitations (REE)

Process Step	Average Extraction / Precipitation%*																
	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Y	Er	Tm	Yb	Lu	Th	U
AB/WL Extractions	94	94	94	94	91	89	87	84	81	79	80	78	78	82	53	80	91
PN Precipitation	0	0	0	0	-2	-2	-3	-4	-4	-4	-4	-4	1	-5	1	87	11
SN Precipitation	11	19	23	26	36	36	31	37	39	38	26	40	40	52	39	100	85
RP Precipitation	69	85	90	91	95	96	95	97	97	95	94	94	45	77	72	79	40
MgR Precipitation	99	98	97	97	93	85	88	60	67	46	94	43	28	57	48	93	72

Source: Hatch, 2025; SGS Data

*Negative precipitation values indicate negligible solids formation and are caused by normal measurement variability

Table 13-15: PP1 Average Solids Assays (REE)

Process Step	Average Solids Assays (g/t)																
	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Y	Er	Tm	Yb	Lu	Th	U
Feed – Concentrate	148000	194000	15900	43500	4050	886	1910	142	386	28.5	1000	33	1.6	4.2	<2	4450	2.1
AB – Calcine	92889	123000	10083	27356	2674	565	1220	95.8	249	20.0	436	21.6	1.0	2.5	2	2549	1.6
WL – Residue	31756	42633	3662	10937	1461	378	973	91.9	282	24.3	492	26.6	1.4	3.3	2	3043	1.1
PN – Residue	24544	36967	3209	9276	1143	280	672	63.1	189	16.5	327	17.2	1.0	2.3	2	7733	1.7
SN – Precipitate	94675	239625	24340	77413	11175	2380	4266	384.5	1043	76.15	940	89.1	5.2	16.3	2	3972	18.7
RP – Product	143000	228667	19283	51400	4647	953	2083	150	374	27.4	724	30.4	1.4	2.92	3	11.1	0.5
MgR – Precipitate	7141	3069	153	316	20.9	4.5	8.2	0.9	1.7	0.3	6	0.6	0.3	0.5	0.8	11.6	1.0

Source: Hatch, 2025; SGS Data

Table 13-16: PP1 Average Solid Assays (Non-REE)

Process Step	Average Solids Assays (wt%)												
	Si	Al	Fe	Mg	Ca	Na	K	Ti	P	Mn	S	F	Zn
Feed – Concentrate	0.12	0.02	1.90	2.19	9.15	0.05	<0.01	<0.01	2.88	0.33	0.02	2.9	-
AB – Calcine	0.02	0.02	1.52	1.56	6.2	0.06	0.01	0.01	1.74	0.24	19.0	0.080	-
WL – Residue	0.13	0.04	1.49	0.29	20.3	0.14	0.01	0.01	3.58	0.09	-	-	-
PN – Residue	0.10	0.03	5.88	0.23	17.2	0.10	0.01	0.01	5.25	0.05	-	-	-
SN – Precipitate	0.20	0.35	4.85	1.45	0.29	0.04	0.01	0.01	1.31	0.11	-	-	-
RP – Product	0.08	0.03	0.24	7.66	0.40	0.04	0.01	0.01	0.01	0.18	7.59	-	-
MgR – Precipitate	0.08	0.02	0.05	11.9	18.7	0.02	0.01	0.01	0.01	0.08	-	-	13.5

Source: Hatch, 2025; SGS Data

13.4.2.3 Pilot Plant 2

The second pilot plant (PP2) campaign operated for 10 days and changes were made to the process based on the lessons learned in PP1 as well as results from bench test for alternative RP conditions. The changes made included increasing the pulp density of the water leach circuit to reduce water consumption. As the same kiln discharge throughput was kept, this meant lower volumetric throughput downstream. The same equipment was used from PP1; therefore, overflow ports were lowered to obtain similar retention times as PP1. An additional tank in the water leach circuit was added to allow full target retention time before any recycled solids from SN were added.

The major change for PP2 was switching to oxalic acid to precipitate rare earth oxalate. The physical circuit of RP remained unchanged. As PP1 was performed as a fill campaign, wash solutions recovered from PP1 were introduced in PP2 and shut down pulps collected from each reactor in PP1 were used to refill each tank, except RP tanks.

At the end of PP1, the kiln temperature was lowered to 250°C and no negative effects were observed. Therefore, the kiln temperature at the beginning of PP2 was set at 250°C. The reduced kiln temperature along with lower acid addition resulted in low REE extraction. The REE extractions were as low as 82% for Nd, Pr, La, and Ce. Therefore, on the fourth and fifth day of the campaign, the conditions returned to 350°C kiln temperature and 1100 kg acid/t concentrate acid addition rate. The Nd, Pr, La, and Ce extractions achieved with the new conditions were within the range of 92-94%. The overall PP2 average extraction and precipitation efficiencies are shown in Table 13-17, Table 13-18, and Table 13-19.

Furthermore, additional experimental testing was conducted during PP2 to inform the equipment design and vendor quotation specifications during the PFS: the thickening and filtration system behaviors was assessed by SGS, and the results are described in the following report: “**An Investigation into the Solids-Liquid Separation and Rheology of Four Process Samples from Pilot Plant Campaign No. 2 from Wicheeda Deposit Concentrate prepared for Defense Metals Corporation – Project 17173-05 – DRAFT – Final Report**, September 14, 2023”.

Table 13-17: PP2 Overall Average Extractions / Precipitations (REE)

Process Step	Extraction / Precipitation%*																
	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Y	Er	Tm	Yb	Lu	Th	U
AB/WL Extractions	89	90	90	90	89	86	84	81	79	76	75	81	76	74	39	80	93
PN Precipitation	1	2	3	1	1	0	-1	-2	-3	-2	-1	-2	-2	3	3	90	10
SN Precipitation	22	33	41	47	61	62	57	62	64	63	47	56	69	76	64	100	90
RP Precipitation	92	95	97	97	98	98	98	97	96	94	93	93	43	76	74	57	44
MgR Precipitation	99	99	98	98	97	90	96	75	81	57	99	49	28	56	83	84	73

Source: Hatch, 2025; SGS Data

*Negative precipitation values indicate negligible solids formation and are caused by normal measurement variability

Table 13-18: PP2 Average Solids Assays (REE)

Process Step	Average Solids Assays (g/t)																
	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Y	Er	Tm	Yb	Lu	Th	U
Feed – Concentrate	148000	194000	15900	43500	4050	886	1910	142	386	28.5	1000	33	1.6	4.2	<2	4450	2.1
AB – Calcine	89363	118895	9578	26284	2561	540	1200	89.5	260	18.5	425	22.8	1.1	2.3	2	2676	1.3
WL – Residue	45816	61395	5084	15063	1856	460	1170	105.1	332	27.2	549	31.2	1.6	3.9	3	2630	0.9
PN – Residue	39284	58689	4915	14238	1654	390	925	81.4	254	20.1	406	23.3	1.1	2.7	3	9101	1.6
SN – Precipitate	93658	211158	23132	78079	11908	2567	4661	402	1124	76.3	988	95.6	5.5	17.2	3	2094	17.2
RP – Product	164274	210947	16984	44989	3778	762	1758	117	304	21.2	578	28.7	1.1	2.5	2.8	3.9	0.5
MgR – Precipitate	6504	5172	285	624	50.2	8.3	20.8	2.0	4.9	0.5	24	0.8	0.31	0.53	3.0	4.1	1.0

Source: Hatch, 2025; SGS Data

Table 13-19: PP2 Average Solid Assays (Non-REE)

Process Step	Average Solids Assays (wt%)												
	Si	Al	Fe	Mg	Ca	Na	K	Ti	P	Mn	S	F	Zn
Feed – Concentrate	0.12	0.02	1.90	2.19	9.15	0.05	<0.01	<0.01	2.88	0.33	0.02	2.9	-
AB – Calcine	0.03	0.01	1.37	1.46	5.9	0.06	0.01	0.01	1.72	0.22	18.72	0.2	-
WL – Residue	0.10	0.02	0.61	0.14	20.4	0.11	0.01	0.01	2.59	0.07	-	-	-
PN – Residue	0.14	0.04	7.21	0.11	14.4	0.09	0.01	0.01	5.66	0.05	-	-	-
SN – Precipitate	0.21	0.57	2.54	3.01	0.42	0.04	0.01	0.01	0.87	0.13	-	-	-
RP – Product	0.04	0.02	0.02	0.01	0.27	0.04	0.01	0.01	0.01	0.01	0.09	-	-
MgR – Precipitate	0.09	0.02	0.04	7.9	21.2	0.01	0.01	0.01	0.01	0.25	15.1	-	-

Source: Hatch, 2025; SGS Data

13.4.2.4 Pilot Campaigns Key Findings

A summary of the key findings of the pilot campaigns is given in Table 13-20. Furthermore, this table outlines which elements of the Process Design Criteria for the full-scale plant were derived from the test work.

Table 13-20: Summary of SGS Pilot Plant and Selected Bench-Scale Test Work

Test Description	Major Conclusions	Process Design Criteria
Acid Bake (AB)	**Short Rotary Kiln Tests:** Optimal REE extraction (Nd ext: 91-94%) is achieved with an acid dosage of 1,100 kg/tFeed, a baking temperature of approximately 350°C, and a retention time of 1.8 hours. **PP1:** Material of construction needs to be carefully considered as the corrosive/erosive nature of the sulphuric acid mixture degraded the screw on the mixing auger. Calcine discharged from the kiln as agglomerated lumps, ranging in diameter from <1 cm to >4 cm, agglomerates required a hammer to break into smaller pieces of <2 cm. Lumps were shown to disintegrate under water leach conditions. A screen after the first water leach tank did collect minor coarse material. Material transfer to the auger was complicated as the feed concentrate had tendency to stick and bridge. Kiln temperature started at 350°C then lowered to 250°C on the last 2 days, no major impact on recovery was observed. **PP2**: The screw from the acid mixing auger deteriorated under combined erosive and corrosive forces, the mixing auger had to be disassembled. cleaned out, and sometimes replaced with spares. Acid dosage was returned to 1,100 kg/t acid addition from 1,000 kg/t to achieve higher REE recovery. Kiln temperature started at 250°C then increased back to 350°C midway due to low recoveries. *Note: As acid dosage and kiln temperature were increased back to the PP1 operating conditions at the same time, it is unclear which of the factors impacted the poor extractions.	**AB % Extraction Extents:** REE extraction extents were defined from the REE extractions observed during the short rotary kiln tests (Section 13.4.1.2) at the optimized treatment conditions. **Acid addition:** 1,100 kg/t. **Kiln Internal Temperature:** 350°C. **Evaporation % to Off-Gas:** Sulphuric acid and hydrogen fluoride kiln off-gas flows taken from PP2 mass balance results: H2SO4 to off-gas: 24.2%. HF to off-gas: 98%.

Test Description	Major Conclusions	Process Design Criteria
Water Leach (WL)/ Primary Neutralization (PN)	**PP1**: Negative performance was shown when MgR solids were recycled to the PN circuit; thickener required higher flocculant addition, co-precipitation of REE in PN precipitate was observed. The circuit was able to remove 90-99% Thorium, 70-90% Iron, 90-99% Phosphorus, and limited co-precipitation of REE prior to MgR precipitate recycling. **PP2**: MgR solids recycling to PN was eliminated and replaced with magnesium oxide, co-precipitation of REE remained negligible for the rest of the campaign (<3% LREE). Removal of Thorium, Iron, and Phosphorus averaging 90%. Solid-liquid separation and rheology testing were performed on PN (results were used for producing vendor quote specifications). Use of recycled process water in both PP1 and PP2 had no noticeable negative effects on the water leach circuit.	**WL Dissolution %:** Simulated design assumes complete dissolution (acid baking and water leaching efficiency are coupled). **PN Precipitation %:** REE precipitation percentages were defined from PP2 (average for the second half of the campaign, when the temperature of the kiln was 350°C). Feed liquor to calcine target ratio for WL tanks was taken from PP2: Target Ratio: 7.3 t Liquor/t Acid Baked Material. Neutralization Agent: Magnesium hydroxide slurry. Dosing rate of ferric sulphate and hydrogen peroxide: Dosages were obtained from the second half of PP2: Ferric Sulphate Dosing Rate: 90 g ferric sulphate solution / kg calcine. Hydrogen Peroxide Dosing Rate: 0.78 g H_2O_2 solution / kg calcine. Thickener and filtration: Mass balance inputs were taken from solid liquid separation test work
Secondary Neutralization (SN)	**PP1:** High levels of impurity removal were achieved; concentrations of impurity metals in the remaining liquor were <0.1 mg/L Th (~100%), <2 mg/L Fe (~100%), and <5 mg/L P (94%). Co-precipitation of REE, was up to 30% Nd and 10-20% La/Ce in precipitate. **PP2:** Efforts to further remove thorium by adding additional magnesium oxide and raising pH led to higher co-precipitation, averaging 40-50% Nd/Pr (deemed to be not necessary as thorium levels remained <0.1 mg/L). Solid-liquid separation and rheology testing were performed on SN slurry (test results were used for producing vendor quote specifications). The circuit piloting was successful overall but highlighted the challenge of balancing maximum removal of thorium with excessive co-precipitation of REE.	The design incorporates recycling of the SN residue back to the water leach step to recover the co-precipitated REE. SN Precipitation %: Reaction extents were derived from the maximum extractions of the second half of PP2 results. Thickener mass balance inputs were taken from solid liquid separation test work.

Test Description	Major Conclusions	Process Design Criteria
Uranium Ion Exchange (UIX)	Both campaigns successfully reduced uranium levels from <0.02-0.06 mg/L (in the feed, equivalent to an average of 1.7 g/t_{REE}) to consistently <0.02 mg/L (assay detection limit). This removal corresponded to U content in the REE product of 0.9 mg/kg to <0.5 mg/kg (assay detection limit, equivalent to an average of <1.2 g/t_{REE}) **PP1:** Dowex 21K XLT to selectively remove uranium from solution. Continued gypsum/REE precipitation after SN caused column pressure to increase during operation and accumulation of solids at top of resin bed. **PP2:** Pressure buildup in the column was observed despite attempts to prevent fines advancing (optimization of flocculants, added additional clarifier before IX columns, movement of the UIX feed draw tube to prevent turbulence, etc.).	Residual uranium concentration in liquor was taken from the uranium removal test and is also the assay detection limit: Uranium concentration in liquor: 0.02 mg/L. Resin Type: Dowex 21K XLT or Equivalent.
Solvent Extraction (SX)	**Bench-Scale SX Tests:** In the Cyanex 801 system, distribution coefficients vs free acidity were approximately linear, and the Ce/Pr separation factor was 1.7. Solvent saponification was tested with MgO and other agents.	The design criteria for the SX circuit (inputs, outputs, operating parameters, and equipment sizes) were developed from this equilibrium data by M. Nees and J. Goode (Consultants to Defense Metals).
REE Precipitation (RP)	**PP1:** Magnesium oxide contaminated rare earth product, high magnesium (8% Mg) and sulphur (7.6% S) uptake in final product. RP thickener had to be taken offline due to poor performance. The underflow was too dense and cemented to the bottom of the thickener, direct filtration of the discharge pulps was employed. **PP2:** Use of oxalic acid improved impurity levels (<0.01% Mg and ~0.1% S) and removed need for pH control. Overall successful with average 97% LREE recovery, however use of oxalic acid had tendency to plug or form “bridges” within the reactor addition point. Thickener performance was improved by keeping RP underflow bed level low, not allowing solids to fully compact, and having a larger underflow pipe diameter.	Precipitation Reagent: Ammonium bicarbonate. RP precipitation %: precipitation extents were taken from bench-scale RP testing results.

Test Description	Major Conclusions	Process Design Criteria
	Increased gypsum produced due to the need for neutralization of acid generated during oxalic acid addition. **Bench-Scale RP Tests:** Ammonium bicarbonate was found to achieve ~100% REE recovery and produced a high-purity product (< 1wt% Ca, S concentrations).	
Ammonia Recovery	No test work was performed.	The ammonia recovery area was assumed to be sufficiently similar to the MgR circuit and used the design criteria derived based on the results MgR testing.
Magnesium Removal (MgR)	**PP1:** Similar to the RP circuit, MgR thickener had to be taken offline due to underflow being too dense. Direct filtration was employed. **PP2:** Similar to the RP thickener, MgR thickener performance was improved. Solid-liquid separation and rheology testing were performed on MgR (the test results were used for producing vendor quote specifications). **Bench-scale tests:** Operating MgR at 120% stoichiometric dosage and ambient temperature resulted in 76% ammonium removal (as NH3 off-gas), alongside 100% Mg precipitation. Note: During the five- and ten-days campaigns, gypsum scaling occurred and caused operational upsets.	MgR precipitation %: Expected reaction extents were taken from literature (for the residual REEs and U/Th) (Orhanović, et al., 1966) and bench-scale MgR treatment test work. Thickener and filtration mass balance inputs were taken from solid liquid separation test work:

Source: Hatch, 2025

13.4.2.5 *Mass Balance Accountabilities*

Throughout the pilot campaigns, the mass balance of the various steps was calculated from the collected assays, and the mass balance accountabilities (Ratio of each element measured leaving the unit to the amount that was fed to the unit) were determined.

1. Generally, the light REEs (La-Nd) had acceptable accountabilities (85-100 %) throughout the process, although heavy REEs had more variable accountability due to low concentrations and assay reporting limits. REE losses were noted as being potentially caused by periodic maintenance shutdowns of the mixing/feed auger (decoupling of feed and discharge rates).
2. The mass balance for Fluorine and sulphur in the acid baking step was not closed, since the volatilized amounts reporting to the kiln off-gas scrubber were not measured.

3. One process step that had notably low accountability was the SN step, which may be due to measurement challenges. This step should be investigated in further detail in the next phases of the project.
4. Periods of low accountability in the RP step were observed and attributed to product scaling in the reactors or improperly advancing through the reactor overflows. These issues are potentially indicative of operability risks in the RP step. Given that the use of ammonium bicarbonate as the precipitating agent and the use of purified SX strip solution as the RP feed have not yet been piloted, these potential issues should be investigated in the future piloting activities.
5. Low calcium accountability was reported in the MgR step, which was attributed to gypsum scaling in the circuit. Gypsum scaling has been flagged as an area of concern for the process overall and the MgR step and should be investigated in more detail in the next project phase.

13.4.2.6 Limitations to Pilot Plant Campaigns & Distinctions from PFS Flowsheet

Some aspects of the full-scale plant design were not assessed in the pilot-scale testing due to limitations of the pilot-scale setup, or process changes informed by the pilot tests after they concluded. These limitations and differences will need to be addressed in further testing to be conducted during the next phases of the project.

1. **Off-Gas Treatment:** The capture and treatment of kiln off-gas with calcium hydroxide to prevent SO_2/SO_3 and HF gas emissions has not been tested. Furthermore, acid baking trials using the expected air infiltration rate of full-scale equipment (thus producing an off gas of representative composition) have not been conducted. The design of this unit is based on currently operating and commercial kiln and off-gas treatment systems; future development should include more detailed optimization and validation of this process step with equipment vendors to ensure that the design can conform to all applicable environmental emission regulations.
2. **Ammonium Bicarbonate REE Precipitation:** Conducting the RP step with ammonium bicarbonate has been tested at the bench scale, but it has not yet been tested at the pilot scale within a locked cycle; however, the ammonium bicarbonate precipitation of MREC is a common practice in the REE industry. In particular, the dewatering of rare earth carbonates is known to be difficult, (Chi and Tian, 2008) thus necessitating pilot filtration testing (including membrane squeeze tests) to inform the dewatering equipment design in the next project phase**.**
3. **Ammonium Bicarbonate Regeneration:** The process design specifies the capture and recycling of CO_2 released during REE precipitation using ammonium bicarbonate, NH_3 from the Ammonia Recovery off-gas, and CO_2 from natural gas combustion products, by reacting these off-gas streams to regenerate ammonium bicarbonate. This system configuration has not been validated experimentally and will require piloting in the next project phases; however, it should be noted that ammonium bicarbonate formation/regeneration is a common industrial practice. Furthermore, the Ammonia Recovery step using SX strip solution has not been tested, and specifically the ammonia removal extent has not been measured for this process configuration, and requires

validation (in the process design, the MgR ammonia removal extent was assumed to be valid).

4. **Final Product Drying:** The drying of the final rare earth carbonate product has not yet been characterized by thermogravimetric analysis and neither has a target percentage moisture been established. These data are important for the sizing of the drying system and the capital and operating costs. The drying requirements should be assessed in the next project phase.
5. **Slaking with MgR Liquor:** To reduce water usage, the process design specifies that the lime and magnesia fed to the system are slaked in recycled MgR liquor; however, the effectiveness of this slaking with gypsum-saturated liquor has not been tested yet and should be assessed in the next project phase.
6. **Solvent extraction and ammonia recovery:** The process design specifies the separation of La and Ce from the other REEs via solvent extraction, recovery of NH_3 from the resulting REE precipitation liquor, and the co-precipitation of La and Ce with the MgR solids. This system configuration has not been fully validated experimentally and will require piloting in the next project phases to refine the SX circuit design details.
7. **Full Locked – Cycle Operation:** While the pilot did include the major process recycle, the recirculating MgR liquor, full locked-cycle operation was not observed due to the missing recycle steps which recycle streams to the main process: recycling of ammonium bicarbonate, diversion of spent UIX resin to PN, recycling of reclaimed water from water treatment. Additionally, in the pilot testing, the acid-baked calcine was allowed to cool and manually broken-up prior to being fed to water leach, while in the full process design, the hot calcine is broken-up in an in-line delumper, then dropped hot into the water leach solution, which may result in a different leaching behavior. Furthermore, the campaign had a duration of 10 days – further testing should be conducted to assess the effects, if any, of long-term accumulation of impurities within the circuit and to identify other potential long-term operating challenges.

13.5 Tailings Metallurgical Testing

13.5.1 ML/ARD and Radionuclide Potential

Metal leaching / Acid rock drainage and Radionuclide potential testing is described in Section 18.4.3.2.

14. Mineral Resource Estimate

14.1 Introduction

The mineral resource statement presented herein represents a mineral resource estimate (MRE) prepared for the Wicheeda REE project located near Prince George, BC, Canada., in accordance with National Instrument (NI) 43-101. The Project is classified as a rare earth element enriched carbonatite. Drilling and mapping have confirmed the presence of carbonatite body with continuity over 400 m along a northwest-southeast strike, 220 m east-west width and up to 250 m deep in the central down-dip portion of the body.

A total of 73 diamond drill core holes, totaling 14,592 m have been used to support the MRE. The estimated metals include cerium (Ce), dysprosium (Dy), europium (Eu), gadolinium (Gd), holmium (Ho), lanthanum (La), neodymium (Nd), praseodymium (Pr), samarium (Sm), and terbium (Tb).

The current MRE includes a material change in the geological model compared to the MRE disclosed with the 2022 PEA and considers 45 of a total of 47 new drillholes completed by Defence Metals in 2021 and 2022. The resource estimation workflow and methodologies used remain largely the same, with updates to parameters and settings based on the updated data.

The current MRE was prepared by Mr. Warren Black, M.Sc., P.Geo. and Mr. Tyler Acorn, M.Sc of APEX Geoscience Ltd. under the supervision of the QP, Mr. Michael Dufresne, M.Sc., P.Geol., P.Geo, also of APEX, following CIM Definition Standards. The workflow for calculating the current MRE was completed using commercial mine planning software Micromine v 21.0 (Micromine), Leapfrog Geo v2023.1.1 (Leapfrog), Resource Modelling Solutions Platform v.1.10.2 (RMSP), and Deswik CAD v2022.2 (Deswik). Supplementary data analysis was completed using the Anaconda Python distribution and custom Python packages developed by APEX.

Mr. Douglas Reid, P.Eng. of SRK Consulting (US) Inc. has reviewed the drillhole hole results and composites used for the estimation, the estimation parameters used in the ordinary kriging (OK) process, the estimation results and validations, and has accepted the MRE as he considers them to be in accordance with Industry standard practices. Mr. Reid is acting as Qualified Person (QP) for mineral resources. The effective date of the mineral resource statement is February 28, 2025.

Definitions used in this section are consistent with those adopted by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Council in “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” dated November 29, 2019 and “Definition Standards for Mineral Resources and Mineral Reserves” dated May 10, 2014 and prescribed by the Canadian Securities Administrators' NI 43-101 and Form 43-101F1, Standards of Disclosure for Mineral projects. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves.

The database used to develop the geological model and mineral resource estimates for the Project have been reviewed by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret the geology and mineralization controls of the deposit and that the assay data are sufficiently reliable to support the estimation and classification of mineral resources.

14.2 Drillhole Data Description

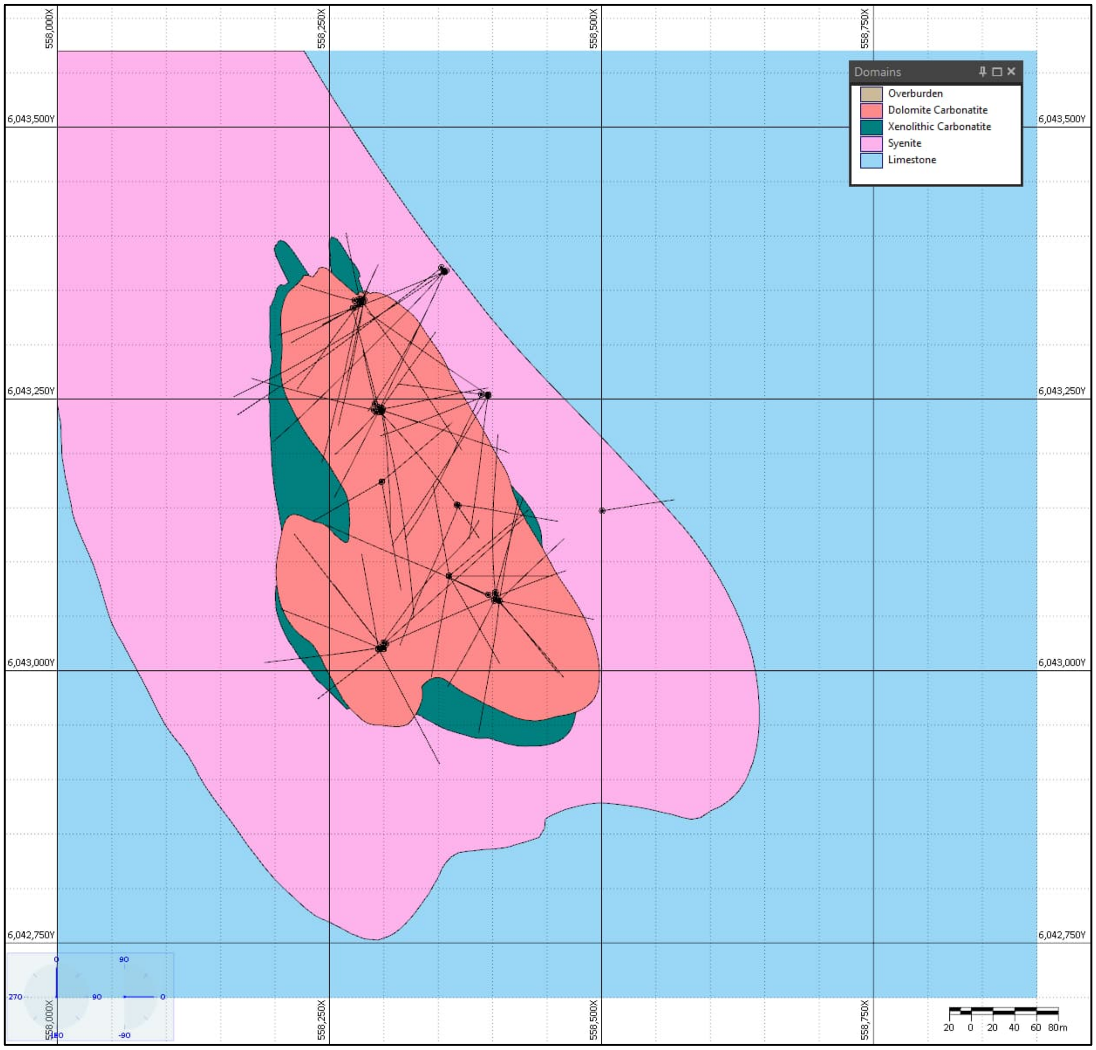
Defense Metals provided APEX with the historical drillhole database for the Wicheeda project that comprised of data collected from 2008 to 2009, before involvement with the program. APEX reviewed the data in 2019 to ensure it was suitable for resource estimation. Data from the recent 2019 drilling program was captured and validated by APEX during the 2019 drilling program, after which APEX compiled the results with the historical data. SRK reviewed and validated the drill hole database in 2024.

The drillhole database used to calculate the MRE is summarized in Table 14-1 and a plan map is shown in Figure 14-1.

Table 14-1: MRE Drill Hole Summary

Year	Number of Drill Holes	Total Length (m)
2008	4	861.1
2009	15	1835.2
2019	13	2007.6
2021	29	5366.3
2022	18	5470.8
Total	79	15540.9

Source: SRK, 2025



Source: Apex, 2025

Figure 14-1: Plan Map - Drill Holes Supporting 2025 MRE

Drillhole collars are snapped to the LiDAR topographic surface. Sample intervals ranged from 0.5 to 7.1 m in length, with 96% of the intervals having a length of 3.0 m or less.

In the drillhole database, 82 intervals were not analyzed. These blank intervals are commonly found at the top of drillholes before mineralization is first encountered or at the end of the drillhole after exiting the mineralized zone. APEX evaluated supporting documents to assess if these blank intervals were either identified as waste material and, therefore, not analyzed on purpose or if insufficient material was returned during drilling to allow the interval to be analyzed. It is essential to distinguish between these two cases as they are treated differently during resource estimation. Intervals classified as “no sample” (NS) are assigned a nominal waste, as Table 14-2 describes. Intervals classified as “insufficient recovery” (IR) are left blank. APEX was conservative when classifying the type of blank interval as NS and IR. If APEX could not confidently determine that a blank interval was IR, it is assumed NS. 28 blank intervals are classified as NS, totalling 405.8 m, and were assigned a nominal waste value. 58 blank intervals are classified as IR, totalling 703.6 m.

All data was validated using the Micromine validation tools at the time the data was imported into the software. No validation errors were encountered. A detailed discussion on the verification of both historical and Defense Metals drillhole data is provided in Section 11 and Section 12 of this report. The APEX authors of this report consider the current Wicheeda drillhole database to be in good condition and suitable for ongoing resource estimation studies.

The QP considers the current Wicheeda drillhole database to be in good condition and suitable to use in ongoing resource estimation studies and APEX's handling of the unsampled intervals reasonable.

Table 14-2: Nominal Metal Values Applied to Intervals Classified as NS

Metal	Nominal Waste Value (ppm)
Ce	0.025
Dy	0.025
Eu	0.01
Gd	0.025
Ho	0.005
La	0.05
Nd	0.05
Pr	0.01
Sm	0.015
Tb	0.005

Source: APEX, 2025

14.3 Estimation Domain Interpretation

14.3.1 Geological Interpretation of Mineralization Domains

REE-enriched carbonatites of the Wicheeda Deposit are part of a narrow, elongate, northwest-southeast trending intrusive carbonatite-syenite sill complex. The carbonatite is intruded into syenite, mafic dikes, limestone and calcareous sedimentary wall rocks. Diamond drilling data supports the interpretation of a moderately north-northeast dipping, shallowly north plunging, layered sill complex having syenite at its base. It is overlain by hybrid matrix to clast-supported limestone or mafic intrusive xenolithic carbonatite (fenite), as well as significantly REE-bearing dolomite-carbonatite rocks, which form the main body of the Wicheeda REE Deposit outcropping at surface. This layered sill complex occurs within an unmineralized limestone waste rock. There is no near-surface oxidized material due to recent glaciation. The primary host, Dolomite-Carbonatite, has dimensions of approximately 450 m north-south by 170-300 m east-west by 100-275 vertically.

The drill pads considered in the 2024 MRE detailed in this report are in areas of very high relief. Most drillholes from them were collared directly into outcrops or minor amounts of talus/rubble material. The westernmost drillholes start in overburden at the base of a slope that dips westward. The 2024 MRE includes an overburden model to account for this.

14.3.2 Estimation Domain Interpretation Methodology

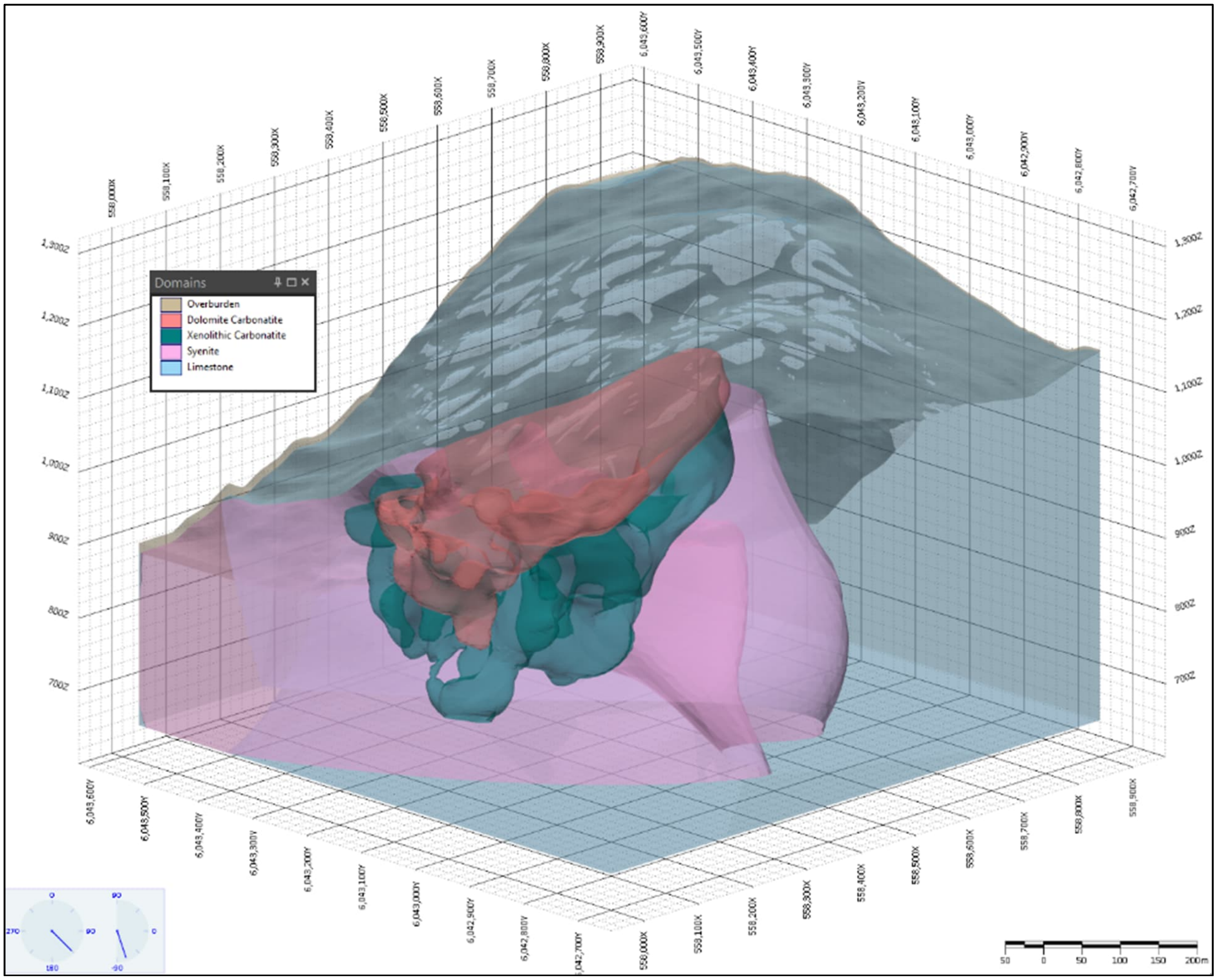
For calculating the 2024 MRE, the syenite-carbonatite complex geology was simplified into four lithologic domains within the 3-D rock model with different mineralization controls and styles that contain varying grades of REE. These domains (and their short name for charts and tables) are:

- Dolomite Carbonatite (DC)
- Xenolithic Carbonatite (Xeno)
- Syenite (Syn)
- Limestone (Lim).

Drillhole intervals were classified as one of the four lithologic domains based on their logged lithology. There are instances where a small interval contained within a dominant domain was simplified to the dominant lithology. Relatively high-grade REE mineralization occurs within Dolomite-Carbonatite (where country rock xenoliths are <20%). Xenolithic Carbonatite represents a hybrid mixed lithology where discontinuous narrow dikes and breccia-zones of dolomite-carbonatite intrude fenitized limestone, syenite, and mafic dike xenoliths comprising between 30-70% of the rock volume. Syenite rocks are interpreted to represent the earliest intrusive phase of the intrusive complex. REE-poor host rocks include fresh and fenitized limestone, calcareous sedimentary rocks, and volumetrically minor mafic dikes.

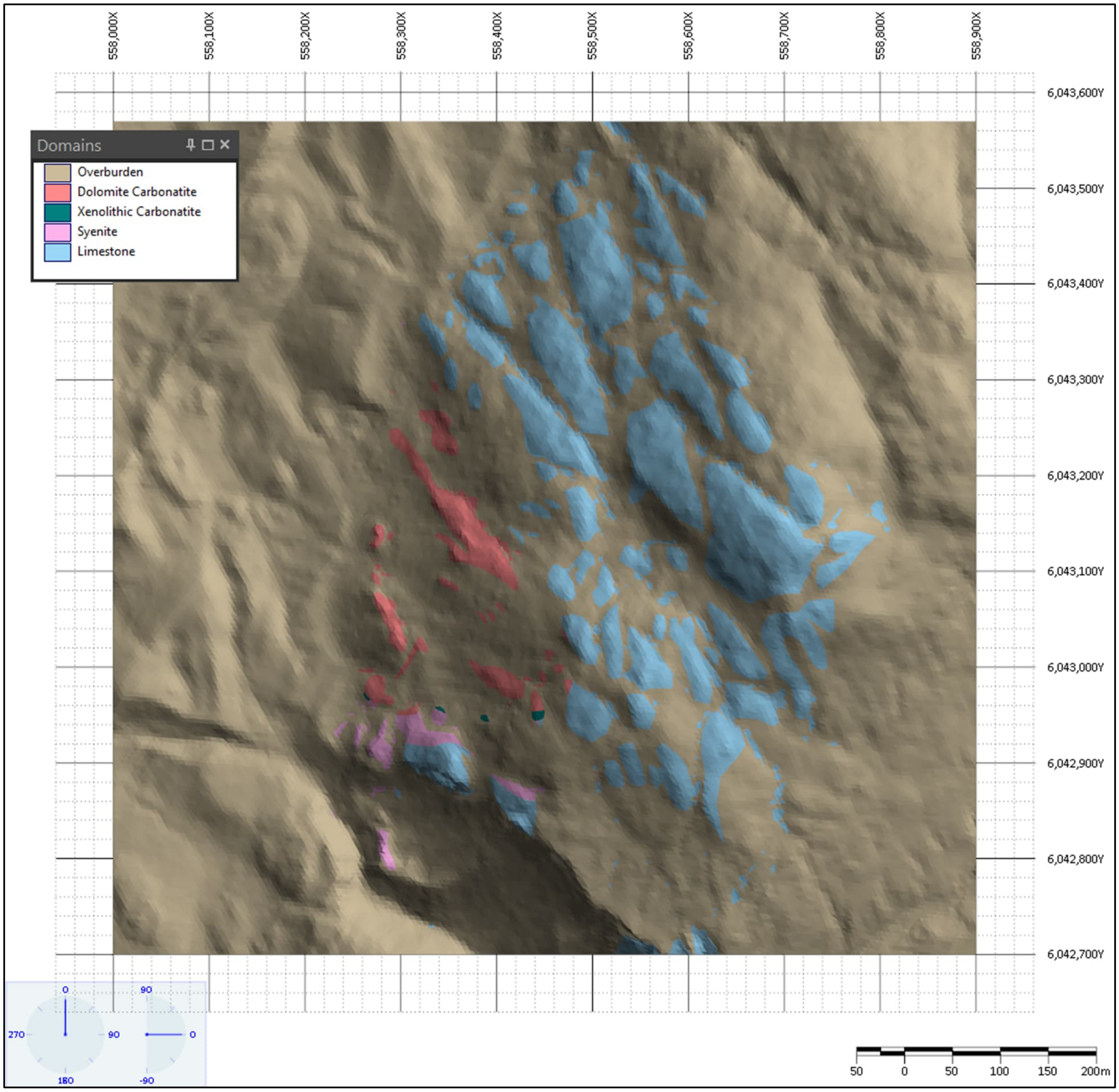
The geological model was constructed using Leapfrog, focusing on a hierarchy of lithological domains from youngest to oldest. Overburden is modelled using drill hole logs, mapped outcrop extents, and the LiDaR. The Dolomite Carbonatite and Xenolithic Carbonatite are modelled as the youngest intrusion package, where Xenolithic Carbonatite represents a mixing of the high-grade dolomite carbonatite and the surrounding syenite. Syenite intrusion is the second youngest unit and is modelled based on drillhole intercepts. Finally, limestone, the oldest rock, is considered the default host and is intruded by the other units. This approach integrates various data sources to represent the geological features comprehensively.

The model’s extent was defined by ensuring that any resource pits would be fully contained within it, allowing waste blocks to be assigned the correct density. 3-D wireframe solids were constructed using the bounding contact surfaces and cut to the LiDaR topography surface. The final 3-D geological model is comprised of a solid of each of the four domains and overburden, totalling five solids. The 3-D rock model was used to discretize drillhole data and the deposit volume into distinct zones (domains) that were treated separately during exploratory data analysis and resource estimation. Figure 14-1, Figure 14-2 and Figure 14-3 represent oblique, plan and sectional views of the geologically modelled domain wireframes.



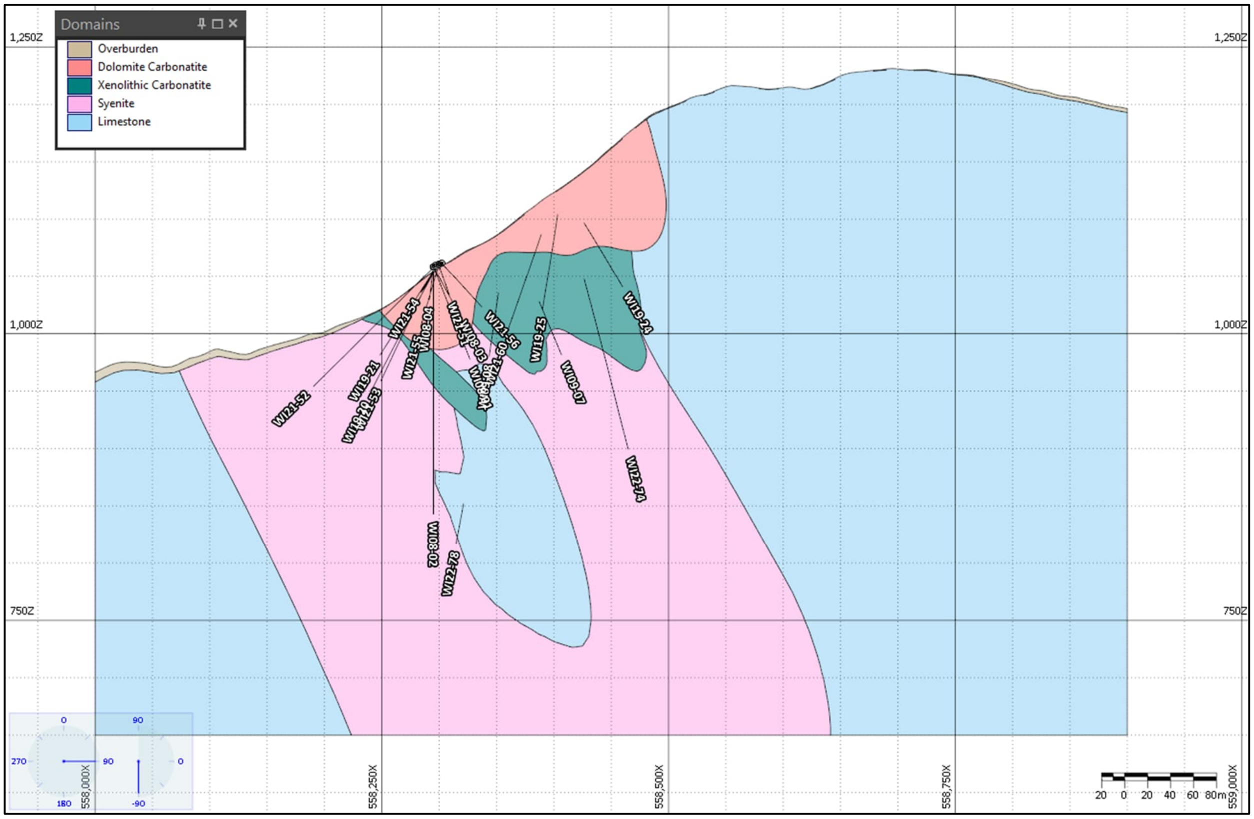
Source: APEX, 2023

Figure 14-2: Oblique view of the domain wireframes looking northeast



Source: APEX, 2023

Figure 14-3: Surficial map of the estimation domain wireframes



Source: APEX, 2023

Figure 14-4: Cross-Section Along 6,043,000E, Looking North Showing Drillhole Traces

14.4 Exploratory Data Analysis and Compositing

14.4.1 Bulk Density

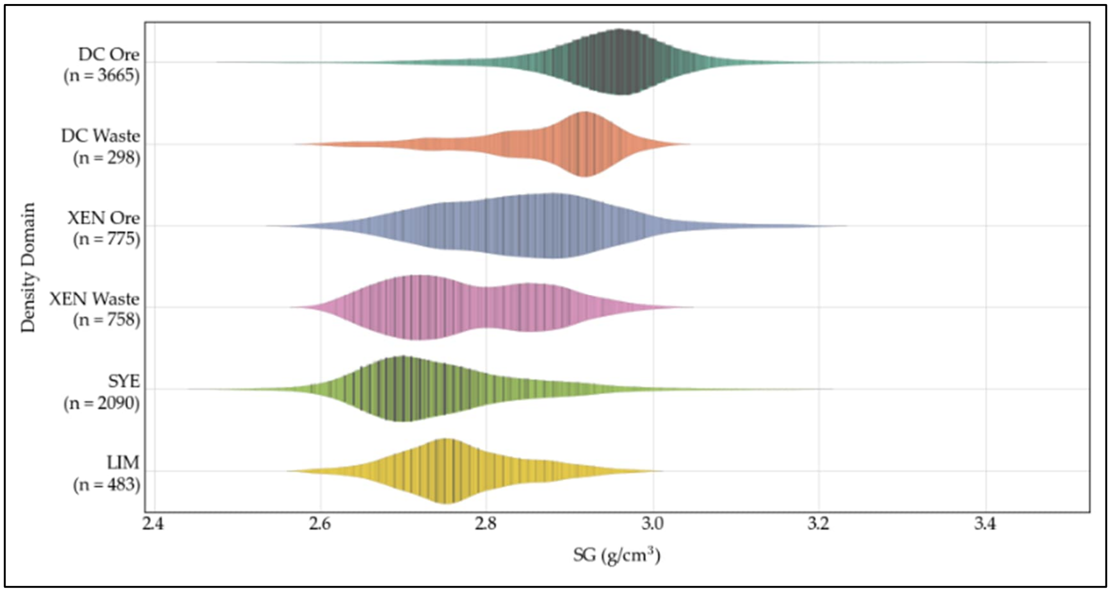
APEX analyzed the available density data to determine what bulk density value to apply to the block model. The Wicheeda project database contains 8,069 density measurements within the estimation domains. Table 14-3 details summary statistics of the measurements categorized by which estimation domain contains each collected sample. Figure 14-5 illustrates the variation in the measurements within each estimation domain. Density measurements from both dolomite-carbonatite and xenolithic-carbonatite are classified as either mineralized or waste based on their total rare earth oxide (TREO) value. Specifically, measurements with a TREO value of 1% or higher are labelled as mineralized, while those below this threshold are categorized as waste.

Median rock densities are supported by 8,069 measurements applied: 2.95 g/cm^3 (mineralized dolomite-carbonatite), 2.90 g/cm^3 (unmineralized dolomite-carbonatite), 2.85 g/cm^3 (mineralized xenolithic-carbonatite), 2.76 g/cm^3 (unmineralized xenolithic-carbonatite), 2.73 g/cm^3 (syenite), and 2.76 g/cm^3 (limestone).

Table 14-3: Summary Statistics of Density Measurements Categorized by Estimation Domain

Domain	Material Type	count	mean	std	min	25%	50%	75%	max
DC	Mineralized	3665	2.94	0.09	2.51	2.9	2.95	2.99	3.44
	Waste	298	2.87	0.09	2.6	2.82	2.9	2.93	3.01
Xeno	Mineralized	775	2.85	0.11	2.58	2.77	2.85	2.92	3.19
	Waste	758	2.78	0.09	2.6	2.7	2.76	2.85	3.01
Lim	Mineralized/Waste	483	2.77	0.08	2.59	2.72	2.76	2.82	2.98
Syn	Mineralized/Waste	2090	2.75	0.1	2.48	2.68	2.73	2.8	3.18

Source: Apex, 2023

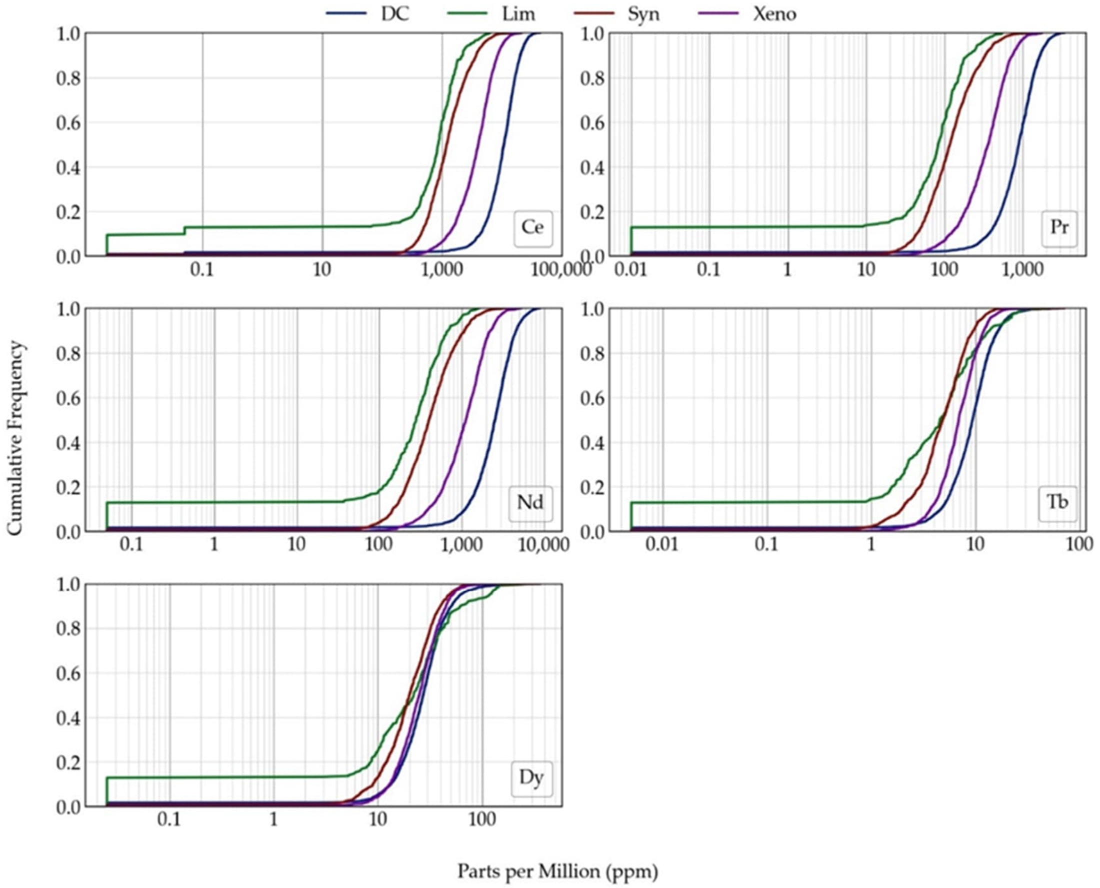


Source: APEX, 2023
Note: Vertical lines represent a single observation.

Figure 14-5: Violin Plot Illustrating the Variation of Density Measurements

14.4.2 Raw Analytical Data

Cumulative histograms and summary statistics for the raw (un-composited) assays from sample intervals contained within the interpreted estimation domains are presented in Figure 14-6 and tabulated in Table 14-4. The assays within each domain generally exhibit a single population for all metals.



Source: APEX, 2025

Figure 14-6: Cumulative Histogram of Each Metal from Sample Intervals in the Estimation Domains

Table 14-4: Summary Statistics of Each Metal from Sample Intervals in the Estimation Domains

	Global	Dolomite Carbonatite	Xenolithic Carbonatite	Syenite	Limestone
Ce (ppm)					
count	4,917	2,372	1,089	1,192	264
mean	7,218	11,851	4,615	1,734	1,080
std	6,683	6,544	3,070	1,684	1,038
var	44,660,030	42,822,556	9,424,214	2,835,043	1,076,747
CV	0.93	0.55	0.67	0.97	0.96
min	0.0025	0.0025	0.0025	0.0025	0.0025
25%	1,675	7,160	2,350	704	447
50%	5,260	10,900	4,050	1,205	870
75%	10,950	15,500	6,130	2,130	1,370
max	44,300	44,300	21,700	18,500	6,480

	Global	Dolomite Carbonatite	Xenolithic Carbonatite	Syenite	Limestone
Dy (ppm)					
count	4,917	2,372	1,089	1,192	264
mean	28.2	30.9	27.4	23.1	30.8
std	20.1	21.1	14	16.7	36
var	405.8	445	197.2	278.4	1,298.30
CV	0.71	0.68	0.51	0.72	1.17
min	0.025	0.025	0.025	0.025	0.025
25%	16.8	19.1	17.6	13.6	10
50%	24.9	27.2	24.9	20.1	22.1
75%	34.6	36.8	34.4	29.7	36.5
max	362	274	158	362	286
Nd (ppm)					
count	4,917	2,372	1,089	1,192	264
mean	1,748.20	2,742.10	1,259.10	528.9	341.3
std	1,483.40	1,453.70	775.6	440.2	295.8
var	2,200,540.70	2,113,202.20	601,526.00	193,746.10	87,482.20
CV	0.85	0.53	0.62	0.83	0.87
min	0.05	0.05	0.05	0.05	0.05
25%	530	1,720.00	683	240	144.9
50%	1,395.00	2,550.00	1,130.00	405	282
75%	2,590.00	3,520.00	1,665.00	658.3	457.8
max	9,030.00	9,030.00	5,360.00	4,490.00	1,655.00
Pr (ppm)					
count	4,917	2,372	1,089	1,192	264
mean	602.6	971.2	404.7	160.8	102.2
std	540.9	533.3	259.2	145	92
var	292,616.10	284,442.40	67,188.50	21,019.20	8,459.20
CV	0.9	0.55	0.64	0.9	0.9
min	0.01	0.01	0.01	0.01	0.01
25%	157	589	213	69.5	44.6
50%	456	893	364	117.8	83.7
75%	899	1,270.00	537	198.1	132.1
max	3,440.00	3,440.00	1,800.00	1,600.00	548
Tb (ppm)					
count	4,917	2,372	1,089	1,192	264
mean	8.26	10.21	7.56	5.47	6.29
std	5.4	5.85	3.6	3.68	6.78
var	29.17	34.25	12.93	13.55	45.9
CV	0.65	0.57	0.48	0.67	1.08
min	0.005	0.005	0.005	0.005	0.005
25%	4.83	6.66	5.16	3.28	1.99

	Global	Dolomite Carbonatite	Xenolithic Carbonatite	Syenite	Limestone
50%	7.36	9.38	6.87	4.85	4.55
75%	10.55	12.41	9.33	7.08	8.16
max	71.3	71.3	34.7	70.6	53.4

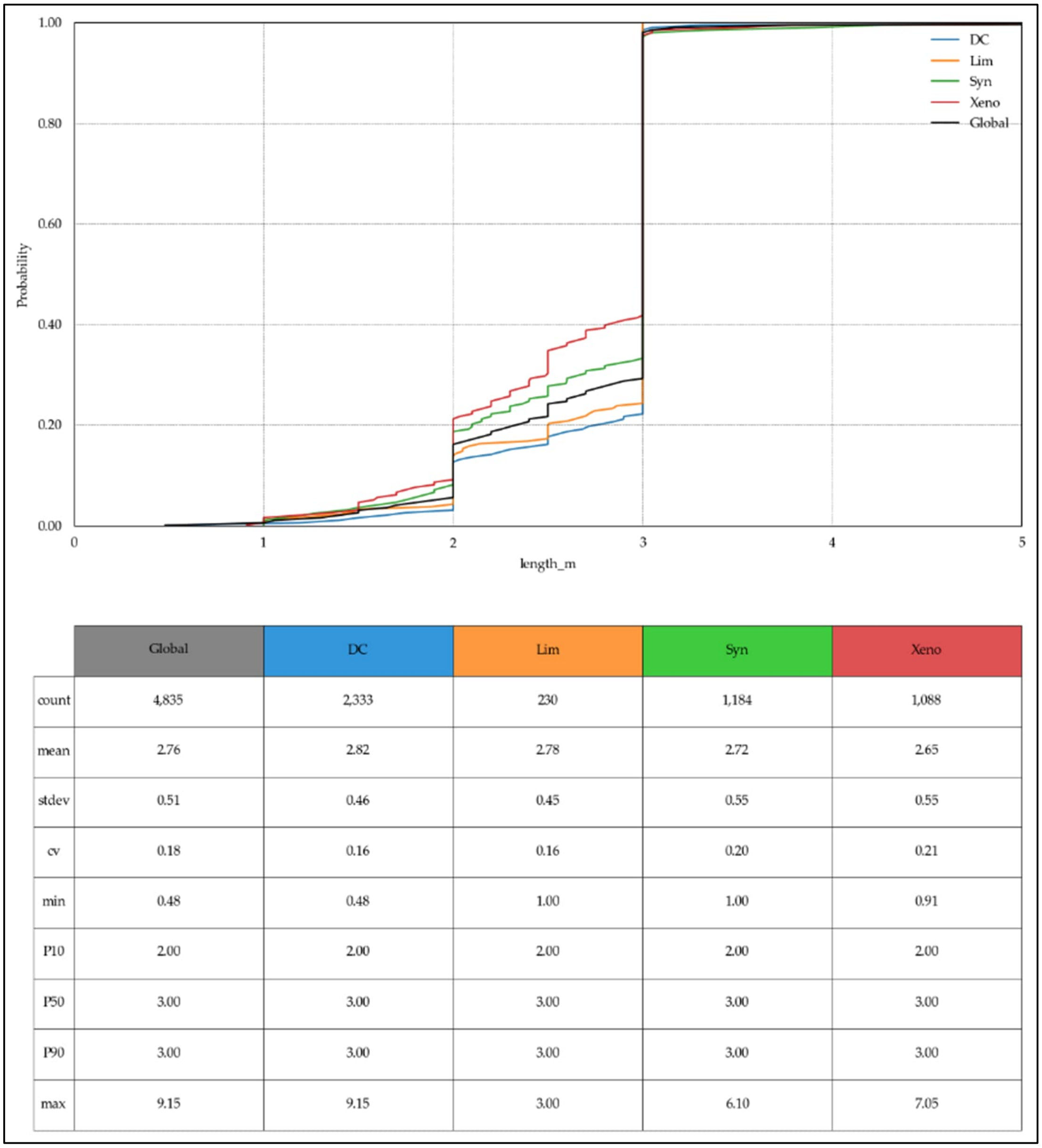
Source: , 2025

14.4.3 *Compositing Methodology*

Downhole sample length analysis shows sample lengths range from 0.48 to 9.15 m, with the dominant sample length being 3.00 m. A composite length of 3.00 m is selected as it provides adequate resolution for mining purposes and is equal to or larger than 98.1% of the drillhole samples (Figure 14-7). Out of 4,835 core samples, 82 exceed the composite length. These longer samples come from areas with poor core recovery, requiring more extensive intervals to ensure the sample contained enough material for analysis.

The length-weighted compositing begins at the drillhole’s top and ends at its bottom. However, composites cannot cross hard boundary domain contacts. So, if a composite hits such a boundary, it is cut short. A new composite starts at that point and continues until it reaches the maximum length, hits another boundary, or reaches the end of the drillhole.

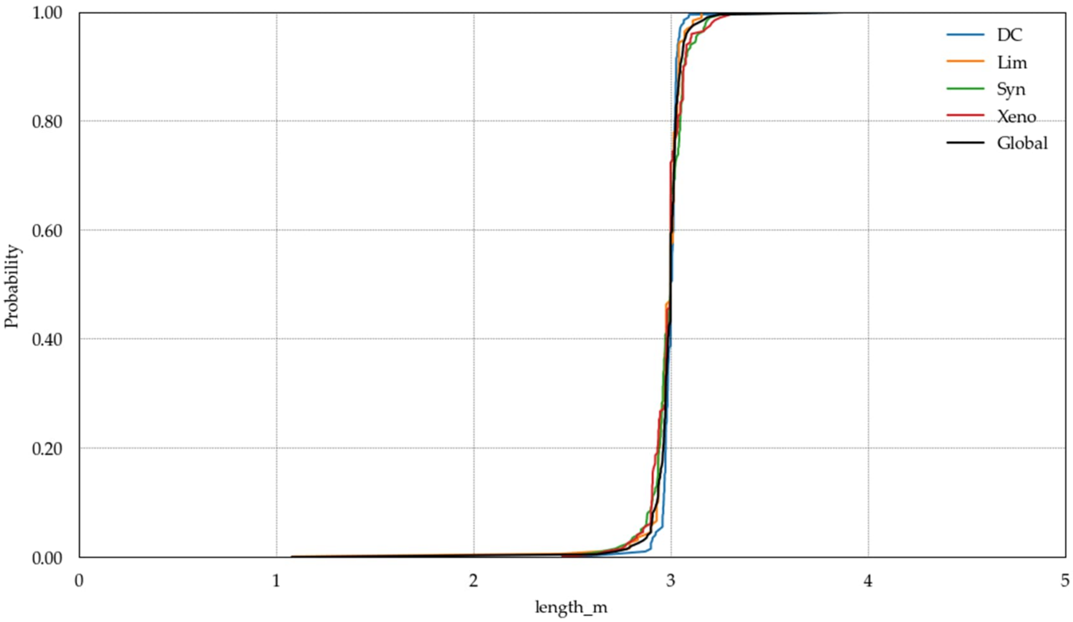
A balanced compositing approach is used. Each continuous section of a drillholes intersection with an estimation domain is examined. The composite length used for each section is adjusted to be uniform, aiming to match a target length and preventing the creation of small, “orphan” composites that standard compositing often produces. A histogram of the composited interval lengths is presented in Figure 14-8. Any composites with a final length of less than 1.5 m are dropped.



	Global	DC	Lim	Syn	Xeno
count	4,835	2,333	230	1,184	1,088
mean	2.76	2.82	2.78	2.72	2.65
stdev	0.51	0.46	0.45	0.55	0.55
cv	0.18	0.16	0.16	0.20	0.21
min	0.48	0.48	1.00	1.00	0.91
P10	2.00	2.00	2.00	2.00	2.00
P50	3.00	3.00	3.00	3.00	3.00
P90	3.00	3.00	3.00	3.00	3.00
max	9.15	9.15	3.00	6.10	7.05

Source: APEX, 2025
Note: Intervals that were not sampled or had insufficient recovery are not illustrated.

Figure 14-7: Cumulative Histogram of Sample Interval Lengths Within the Estimation Domains



	Global	DC	Lim	Syn	Xeno
count	4,824	2,234	488	1,130	972
mean	2.99	3.00	2.98	2.99	2.99
stdev	0.08	0.05	0.11	0.09	0.09
cv	0.03	0.02	0.04	0.03	0.03
min	1.08	2.50	1.08	2.48	2.45
P10	2.93	2.96	2.93	2.91	2.90
P50	3.00	3.00	3.00	3.00	3.00
P90	3.05	3.03	3.04	3.07	3.07
max	4.00	4.00	3.16	3.25	3.30

Source: APEX, 2025
Note: Illustrates composite lengths after compositing but before orphans are dropped.

Figure 14-8: Cumulative Histogram of Composite Interval Lengths Within the Estimation Domains

14.4.4 Declustering

Data collection often focuses on high-value areas, resulting in lower-value areas being underrepresented in the raw composite statistics and distributions, often leading to an inflated mean. Spatially representative (declustered) statistics and distributions are required for accurate validation. Declustering techniques calculate a weight for each datum, giving more weight to data in sparse and less in dense areas. Using the cell sizes described in Table 14-5, APEX applied cell declustering to calculate weights for each drillhole composite across all estimation domains. The declustering weights were used for reporting statistics and for Change of Support studies discussed in Section 14.7.1.1. They were not applied during the estimation process.

Table 14-5: Cell Sizes Used to Calculate Declustering Weight in Each Estimation Domain

Estimation Domain	Cell Declustering Size (m)
Dolomite-Carbonatite	36
Xenolithic-Carbonatite	45
Syenite	55
Limestone	65

Source: APEX, 2025

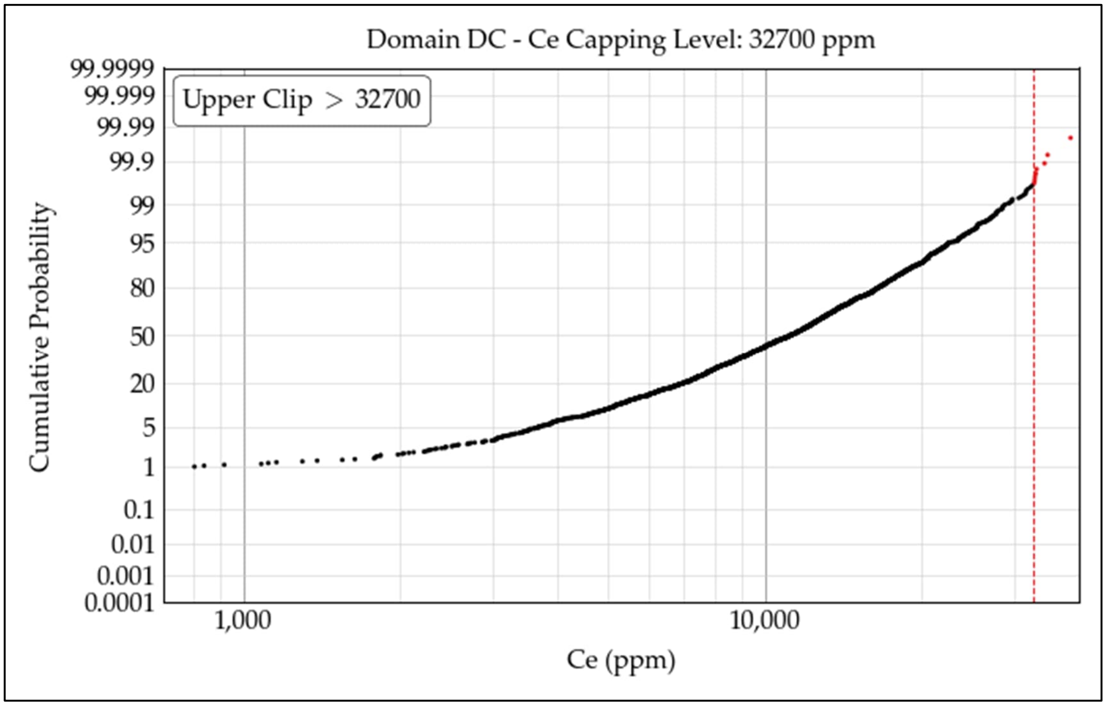
14.4.5 Capping

Composites are capped to a specified maximum value to ensure metal grades are not overestimated by including outlier values during estimation. Probability plots illustrating all values are used to identify outlier values that appear higher than expected relative to the composite population of each metal with the estimation domains. An example of a cumulative probability plot used to select the respective capping levels is shown in Figure 14-9. Visual inspection of the potential outliers revealed that they have no spatial continuity with each other. The capping levels detailed in Table 14-6 were applied to the composites used to calculate the current MRE.

Table 14-6: Capping Levels Applied to Composites Before Estimation

Rock Type	Ce (ppm)	Dy (ppm)	Nd (ppm)	Pr (ppm)	Tb (ppm)
DC	32,700	130	7,900	2,700	44.0
Xeno	15,500	68	3,200	1,150	16.3
Syn	9,500	70	2,400	690	14.8
Lim	3,000	140	1,100	300	24.0

Source: APEX, 2025

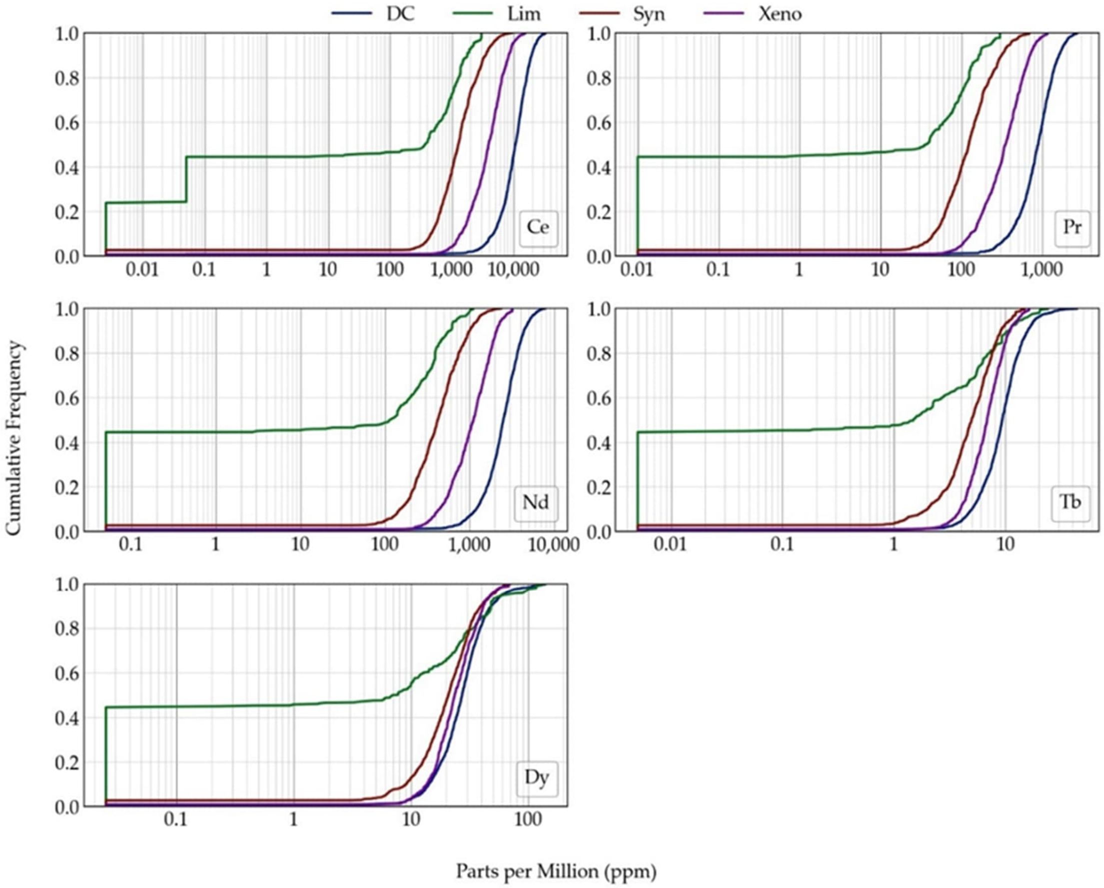


Source: APEX, 2024

Figure 14-9: Example of Cumulative Probability Plot of the Composited Metal Values Used to Determine Capping Level

14.4.6 Final Composite Statistics

Cumulative histograms and summary statistics for the declustered and capped composites contained within the interpreted estimation domains, without orphans <1.5 m, are presented in Figure 14-9 and tabulated in Table 14-7. The assays within each domain generally exhibit a single population for all metals. The large number of waste values within the Limestone domain are due to discretization of large intervals of unsampled rock being that were assigned a nominal into smaller composites.



Source: APEX, 2025
Note: Histograms consider declustering weights, and <1.5 m orphans are removed.

Figure 14-10: Cumulative Histogram of Each Metal from Capped and Declustered Composites

Table 14-7: Summary Statistics of Each Metal from Composites Contained within the Estimation Domains

	Global	Dolomite Carbonatite	Xenolithic Carbonatite	Limestone	Syenite
Ce (ppm)					
count	4,824	2,234	972	488	1,130
mean	6,260	11,547	4,502	636	1,670
std	6,054	5,917	2,799	792	1,378
var	36,654,909	35,012,269	7,834,004	627,579	1,897,860
CV	0.97	0.51	0.62	1.25	0.82
min	0.0025	0.0025	0.0025	0.0025	0.0025
25%	1,413	7,385	2,307	0	731
50%	4,217	10,676	3,920	366	1,271
75%	9,413	14,857	6,014	1,085	2,088
max	32,700	32,700	15,500	3,000	9,500

	Global	Dolomite Carbonatite	Xenolithic Carbonatite	Limestone	Syenite
Dy (ppm)					
count	4,824	2,234	972	488	1,130
mean	26.4	30.6	26.1	17.9	22.9
std	16.8	17.9	11.8	26.2	12.8
var	281.4	321.5	138.7	687.9	164.0
CV	0.64	0.59	0.45	1.46	0.56
min	0.025	0.025	0.025	0.025	0.025
25%	16.4	20.2	17.6	0.0	14.1
50%	24.0	27.2	23.6	7.4	20.8
75%	33.1	36.2	32.8	27.9	29.6
max	140.0	130.0	68.0	140.0	70.0
Nd (ppm)					
count	4,824	2,234	972	488	1,130
mean	1,545.5	2,712.3	1,209.2	206.1	514.1
std	1,361.4	1,321.9	662.6	259.6	371.1
var	1,853,381.2	1,747,457.0	439,036.2	67,390.0	137,678.0
CV	0.88	0.49	0.55	1.26	0.72
min	0.05	0.05	0.05	0.05	0.05
25%	465.5	1,798.4	676.0	0.1	249.2
50%	1,170.4	2,506.8	1,114.5	114.4	419.8
75%	2,286.5	3,452.0	1,636.4	368.6	680.6
max	7,900.0	7,900.0	3,200.0	1,100.0	2,400.0
Pr (ppm)					
count	4,824	2,234	972	488	1,130
mean	527.5	953.7	392.1	61.1	155.5
std	492.5	483.6	228.0	76.0	119.4
var	242,562.2	233,833.0	51,975.5	5,781.4	14,263.9
CV	0.93	0.51	0.58	1.24	0.77
min	0.01	0.01	0.01	0.01	0.01
25%	136.3	617.2	211.8	0.0	72.0
50%	373.5	889.9	350.2	36.5	123.5
75%	790.4	1,225.0	531.5	106.5	194.0
max	2,700.0	2,700.0	1,150.0	300.0	690.0
Tb (ppm)					
count	4,824	2,234	972	488	1,130
mean	7.62	10.16	7.20	3.72	5.39
std	4.74	5.17	2.90	5.08	2.92
var	22.44	26.69	8.42	25.79	8.50
CV	0.62	0.51	0.40	1.37	0.54
min	0.005	0.005	0.005	0.005	0.005
25%	4.59	6.97	5.07	0.01	3.43

	Global	Dolomite Carbonatite	Xenolithic Carbonatite	Limestone	Syenite
50%	7.05	9.35	6.83	1.49	4.97
75%	9.85	12.06	8.89	6.14	7.16
max	44.00	44.00	16.30	24.00	14.80

Source: APEX, 2025

Note: Statistics consider declustering weights, and <1.5 m orphans are removed.

14.4.7 Variography

Experimental semi-variograms for each estimation domain are calculated along the major, minor, and vertical principal directions of continuity that are defined by three Euler angles. Euler angles describe the orientation of anisotropy as a series of rotations (using a left-hand rule) that are as follows:

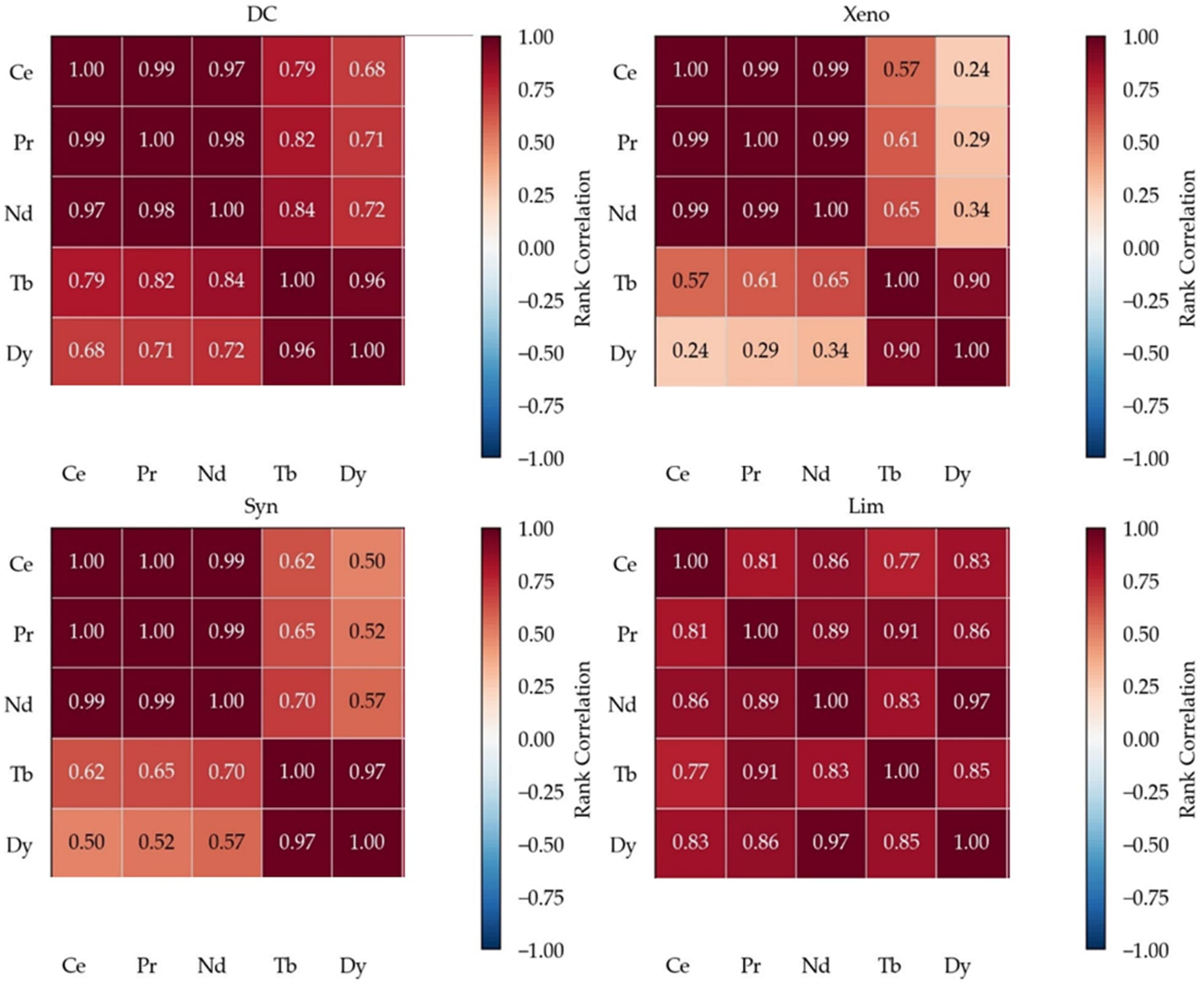
1. A rotation about the Z-axis (azimuth) with positive angles being clockwise rotation and negative representing counter-clockwise rotation
2. A rotation about the X-axis (dip) with positive angles being counter-clockwise rotation and negative representing clockwise rotation
3. A rotation about the Y-axis (tilt) with positive angles being clockwise rotation and negative representing counter-clockwise rotation

APEX calculated experimental semi-variograms for estimated metals in each domain using the correlogram algorithm. Only the dolomite-carbonatite domain yielded stable variograms. Due to the high correlation among REE metals, the modelled variogram structures showed immaterial variation. Hence, a single variogram model based on Ce was used for all metals and domains (See Figure 14-11). Standardized nugget effect and covariance parameters are reported as percentages to allow them to be scaled to each estimation domain variance for kriging purposes (Table 14-9).

Although there is reasonable correlation (Table 14-8) between the various REE to Ce, the QP recommends in future models to consider generating individual variograms, at least for the elements with significant economic contribution.

To compare the fit, the standardized model variogram is plotted against the experimental variograms in Figure 14-12 through Figure 14-15. The fit is reasonable, although improvements could be made. This is not expected to have a material impact on the MRE.

Table 14-8: Correlation Coefficient Summary



DC

	Ce	Pr	Nd	Tb	Dy
Ce	1.00	0.99	0.97	0.79	0.68
Pr	0.99	1.00	0.98	0.82	0.71
Nd	0.97	0.98	1.00	0.84	0.72
Tb	0.79	0.82	0.84	1.00	0.96
Dy	0.68	0.71	0.72	0.96	1.00

Xeno

	Ce	Pr	Nd	Tb	Dy
Ce	1.00	0.99	0.99	0.57	0.24
Pr	0.99	1.00	0.99	0.61	0.29
Nd	0.99	0.99	1.00	0.65	0.34
Tb	0.57	0.61	0.65	1.00	0.90
Dy	0.24	0.29	0.34	0.90	1.00

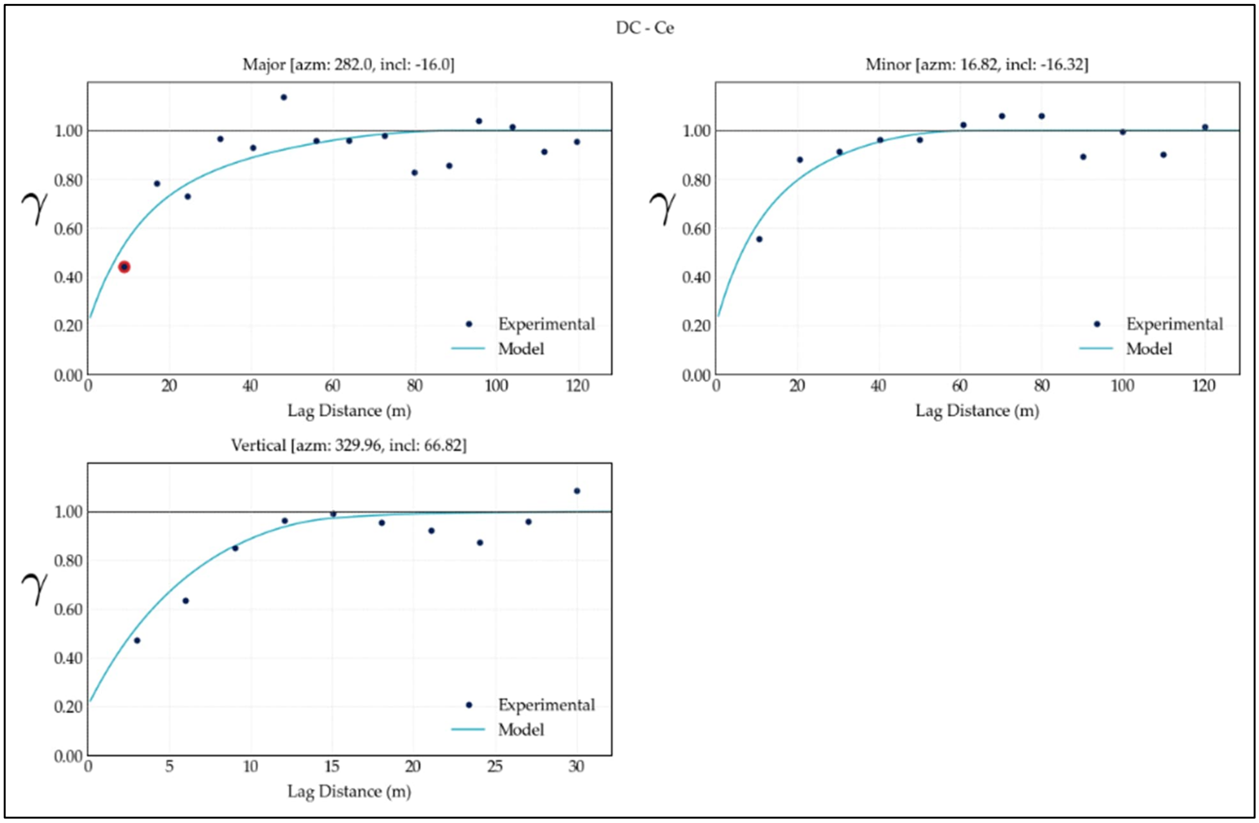
Syn

	Ce	Pr	Nd	Tb	Dy
Ce	1.00	1.00	0.99	0.62	0.50
Pr	1.00	1.00	0.99	0.65	0.52
Nd	0.99	0.99	1.00	0.70	0.57
Tb	0.62	0.65	0.70	1.00	0.97
Dy	0.50	0.52	0.57	0.97	1.00

Lim

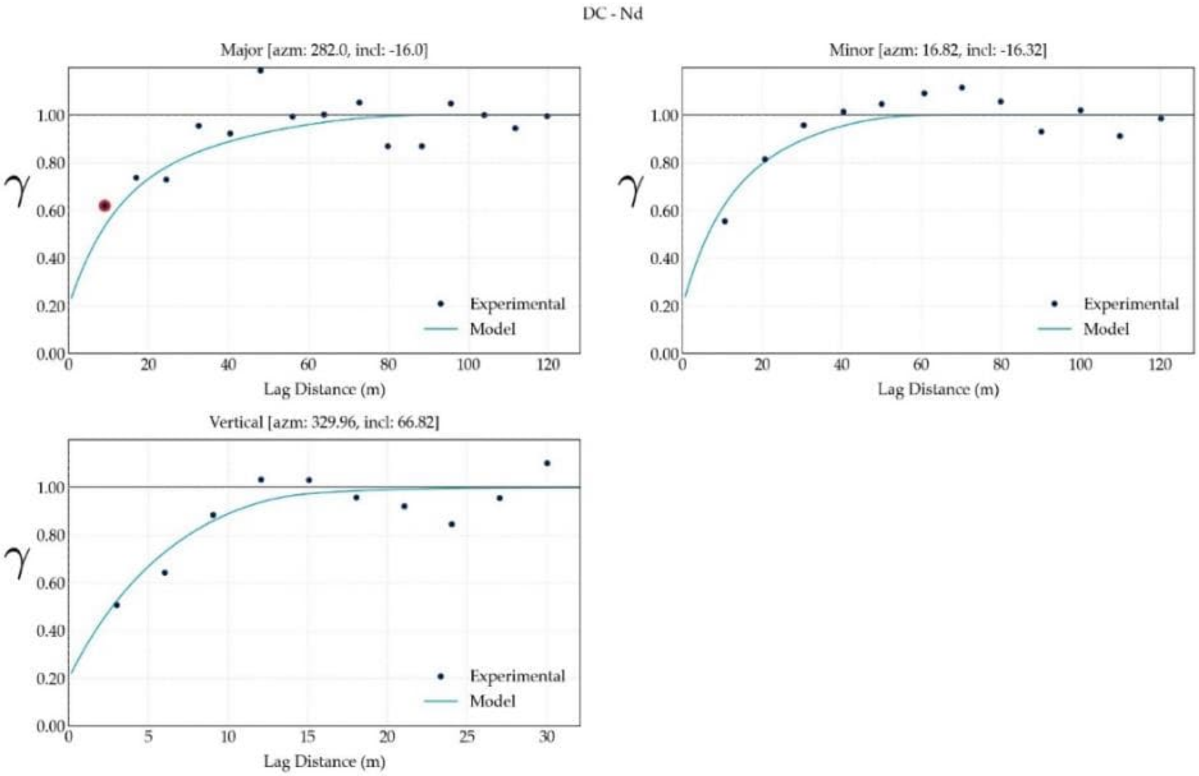
	Ce	Pr	Nd	Tb	Dy
Ce	1.00	0.81	0.86	0.77	0.83
Pr	0.81	1.00	0.89	0.91	0.86
Nd	0.86	0.89	1.00	0.83	0.97
Tb	0.77	0.91	0.83	1.00	0.85
Dy	0.83	0.86	0.97	0.85	1.00

Source : APEX, 2025



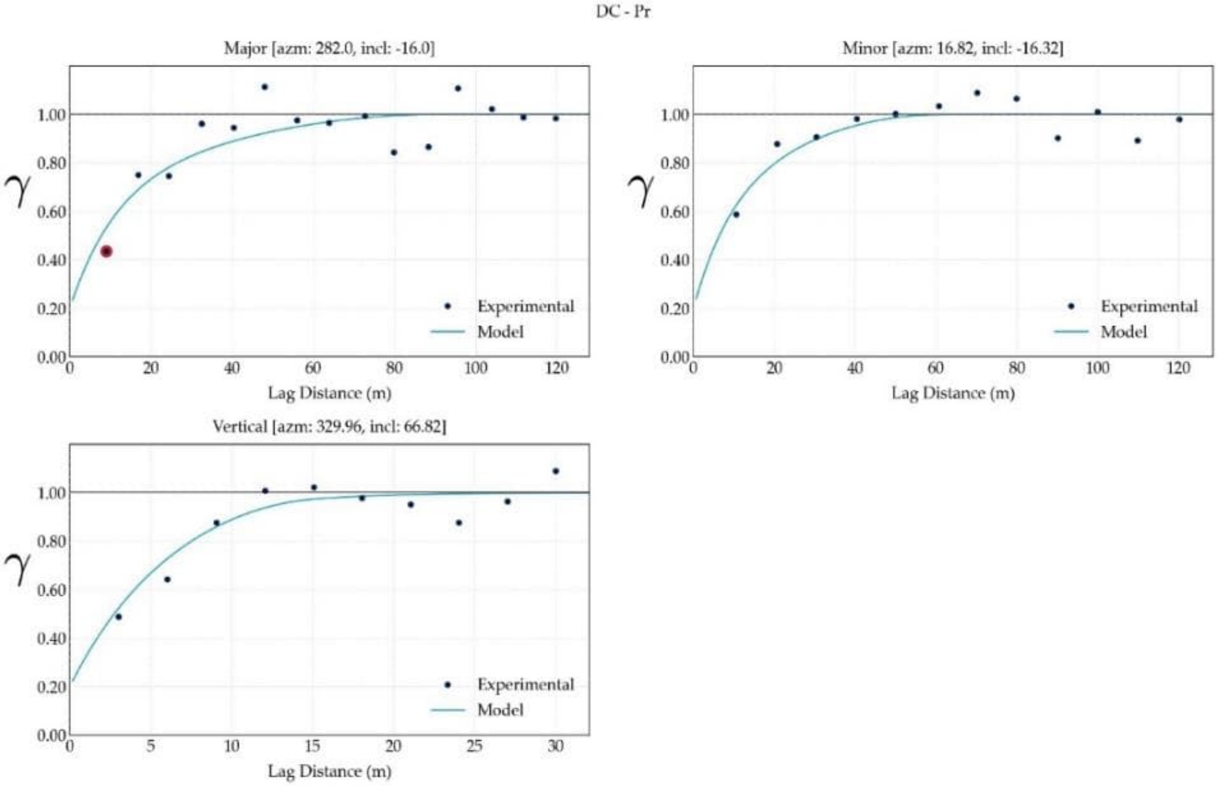
Source : APEX, 2025

Figure 14-11: Standardized Experimental and Modelled Semi-Variogram of the Estimated Metals - Ce



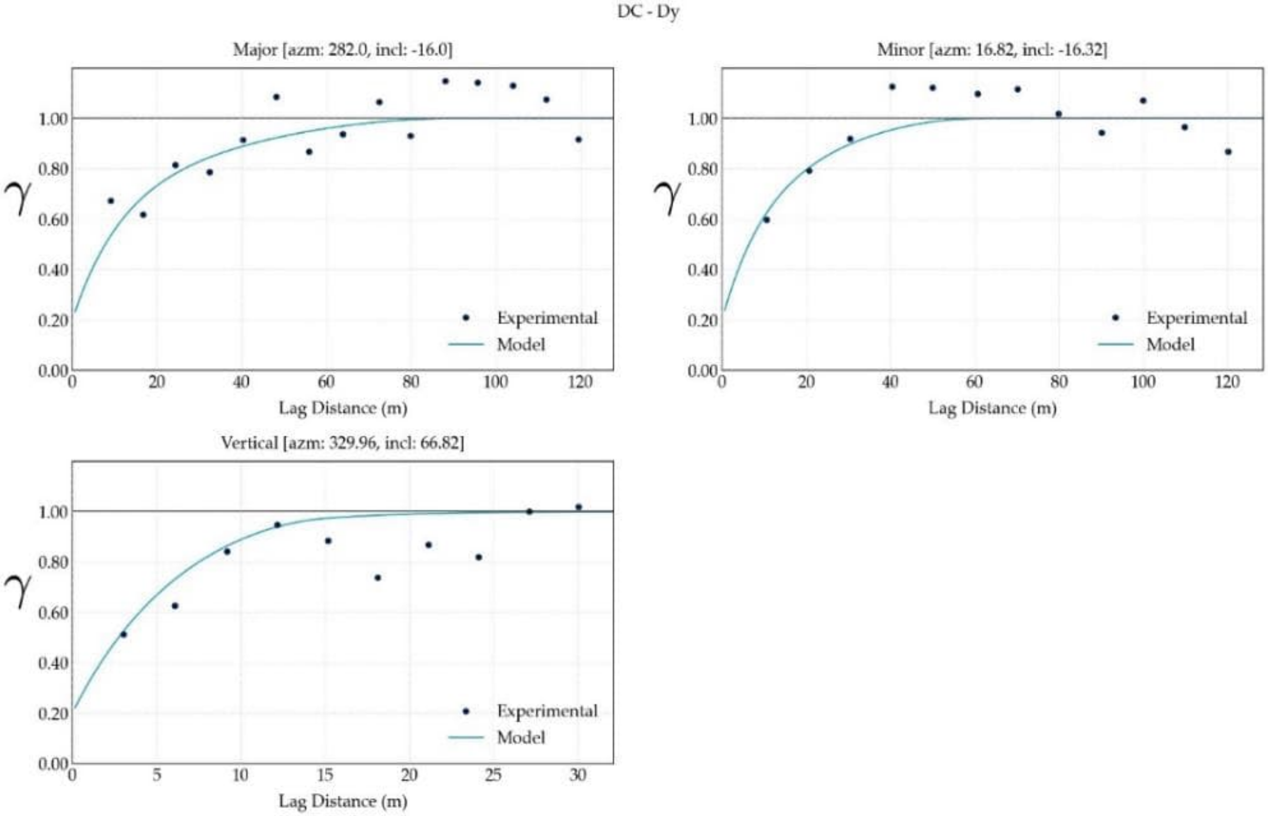
Source : APEX, 2025

Figure 14-12: Standardized Modelled Semi-Variogram Compared to Experimental - Nd



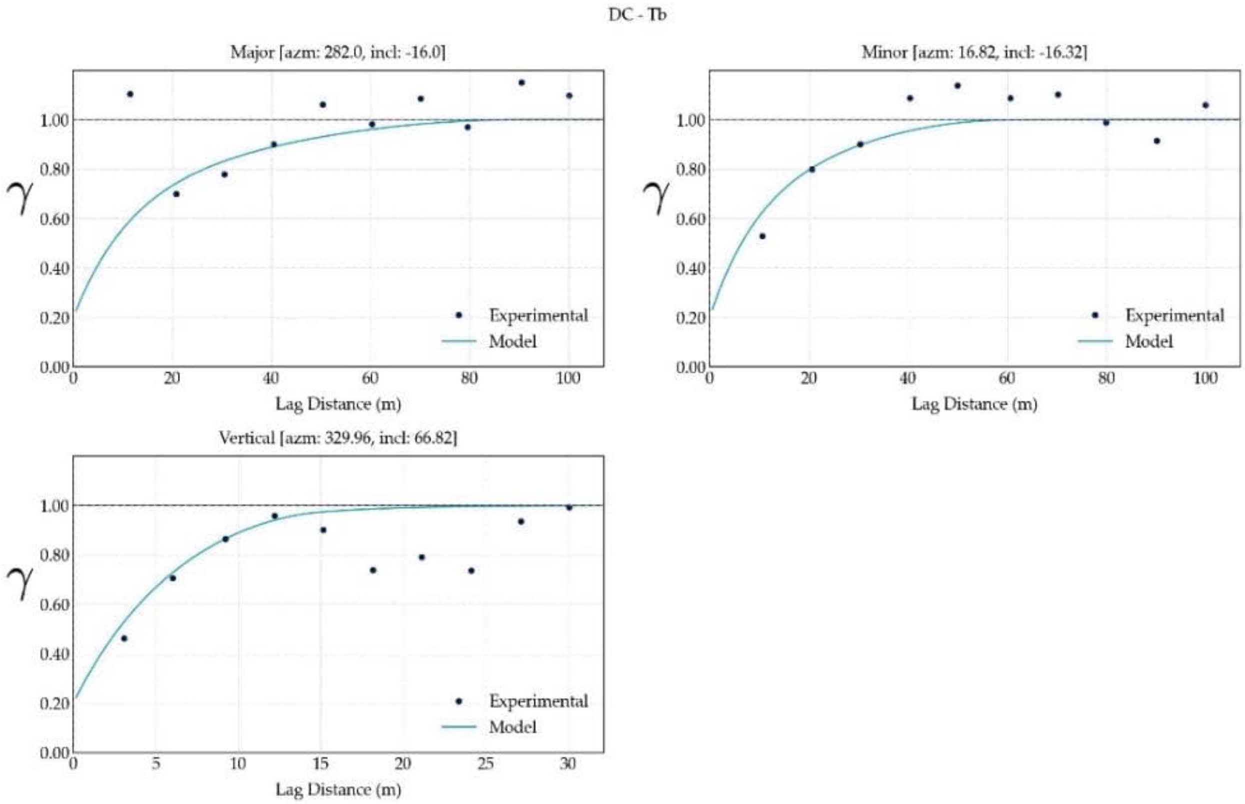
Source : APEX, 2025

Figure 14-13: Standardized Modelled Semi-Variogram Compared to Experimental - Pr



Source : APEX, 2025

Figure 14-14: Standardized Modelled Semi-Variogram Compared to Experimental - Dy



Source : APEX, 2025

Figure 14-15: Standardized Modelled Semi-Variogram Compared to Experimental - Tb

Table 14-9: Standardized Variogram Model Parameters Used by Kriging

Azm[1]	Dip[1]	Tilt[1]	Sill	C0[4]	Structure 1						Structure 2				
					Type[2]	C1[4]	Ranges (m)				Type[3]	C2[4]	Ranges (m)		
							Major	Minor	Vertical				Major	Minor	Vertical
LVA	LVA	LVA	1.00	20%	exp	55%	35	30	15		sph	25%	90	60	15

Source: APEX, 2025

Notes:

[1] LVA represents locally varying anisotropy

[2] exp represents a variogram exponential structure

[3] sph represents a variogram spherical structure

[4] The variogram nugget effect and covariance contribution parameters are indicated as a percentage of the total variance (sill)

14.5 Block Model Parameters

14.5.1 Block Model Parameters

Data spacing and potential mining equipment parameters are the main factors to consider when selecting block size. Block sizes exceeding 25% of the data spacing can introduce estimation errors, especially when using Kriging for grade estimation. The data spacing is approximately 62 m, and truck and shovel open-pit mining methods are assumed. A selective mining unit block size of 3 m satisfies both factors.

In the QP's opinion, the current block size is too small, based on the data spacing, a block size of 15x15x3 would be more appropriate. Studies to optimize the block size should be considered in future models.

Each estimation domain used for the 2024 MRE was populated with a percent model. The percentage of the block within the model is calculated for each domain. No blocks were created outside of the estimation domains. Table 14-10 details the grid definition used.

Table 14-10: Wicheeda 3-D Block Model Size and Extent

Axis	Number of Blocks	Block Size (m)	Minimum Extent (m)	Maximum Extent (m)
X (Easting)	372	3	557863.5	558976.5
Y (Northing)	369	3	6042627.5	6043725.5
Z (Elevation)	205	3	667.5	1279.5

Source: APEX, 2025

14.5.2 *Volumetric Checks*

A comparison of estimation domain wireframe volumes versus block model volumes illustrates there is no considerable over- or under-stating of tonnages (Table 14-11). The calculated block factor for each block is used to scale its volume when calculating the block model's total volume within each estimation domain.

Table 14-11: Estimation Domain Wireframe Versus Block-Model Volume Comparison

Estimation Domain	Wireframe Volume (m^3)	Block Model Volume with Block Factor (m^3)	Volume Difference (%)
Dolomite-Carbonatite	6,226,631	6,227,541	-0.01%
Xenolithic-Carbonatite	6,487,857	6,491,576	-0.06%
Syenite	46,863,912	47,489,275	-1.32%
Limestone	106,073,786	108,023,735	-1.81%
Total	165,652,186	168,232,127	-1.53%

Source: APEX, 2025

14.6 Grade Estimation Methodology

Ordinary Kriging (OK) was used to estimate REE metal grades for the current MRE.

Estimation uses locally varying anisotropy (LVA), which employs different rotation angles to set the variogram model's principal directions and search ellipsoid for each block. LVA was based on the creation of a series of modelled surfaces guided by the observed grade continuity within each geologic domain (DC, XEN, SYN, LIM). Trend surface wireframes assign these angles to blocks within the estimation domain, enabling structural complexities to be captured in the estimated block model.

During grade estimation for each domain, the nugget effect and covariance contributions of the standardized variogram model are scaled to match the variance of the composites within that domain. The ranges used for each mineralized zone are unchanged from the standardized variogram model.

Boundaries between estimation domains and country rock are considered hard boundaries—data from outside a domain can't be used for grade estimation within that domain.

A three-pass estimation method was employed to control Kriging's inherent smoothing and manage the influence of high-grade samples, ensuring accurate volume variance at the chosen block scale. Specific rules for each pass—such as composite limits per drillhole and search sector—are detailed in Table 14-12. The variogram models from Section 14.4 remain unchanged. Kriging settings were optimized for Ce in the dolomite-carbonatite domain. Given the correlation among REE metals, individual tuning yielded negligible differences; hence, a unified search and kriging approach is applied to all metals and domains. Although this method introduces local bias, it enhances the overall accuracy of grade and tonnage estimates above the set cutoff. As the current metallurgical process is now focused on Nd, Pr, Tb, Dy and Gd, each of these elements should be assessed independently. The analysis and estimation should reflect the primary economic elements.

Table 14-12: Estimation Search and kriging Parameters

Pass	Max Variogram and Search Range			Max Comps Per Hole	Min No. Comps	Max No. Comps
	Major	Minor	Vertical			
1	30	15	5	2	1	15
2	60	30	10	3	1	15
3	150	70	10	3	1	15

Source: APEX, 2025

14.7 Model Validation

APEX completed visual and statistical validation to ensure that the estimated block model honors directional trends observed in the composites and that the block model is not over-smoothed or over- or under-estimated. However, as discussed in Section 14.8, the only economical estimation domains are the Dolomite-Carbonatite and Xenolithic-Carbonatite. Because very few blocks from the Syenite and Limestone domains are above the cutoff grade, the validation completed for those domains is not detailed in this section to keep the discussion focused.

14.7.1 Global Bias Check

The QP considers a model to be unbiased if the grade estimate is within ±5% (relative) of the NN grades or declustered composite grades. The biases are ±5% for primary mineralized domains (Table 14-13). The global bias is within ±2%. The QP constructed an independent NN model and the global bias check shows similar results.

Table 14-13: Global Bias Check

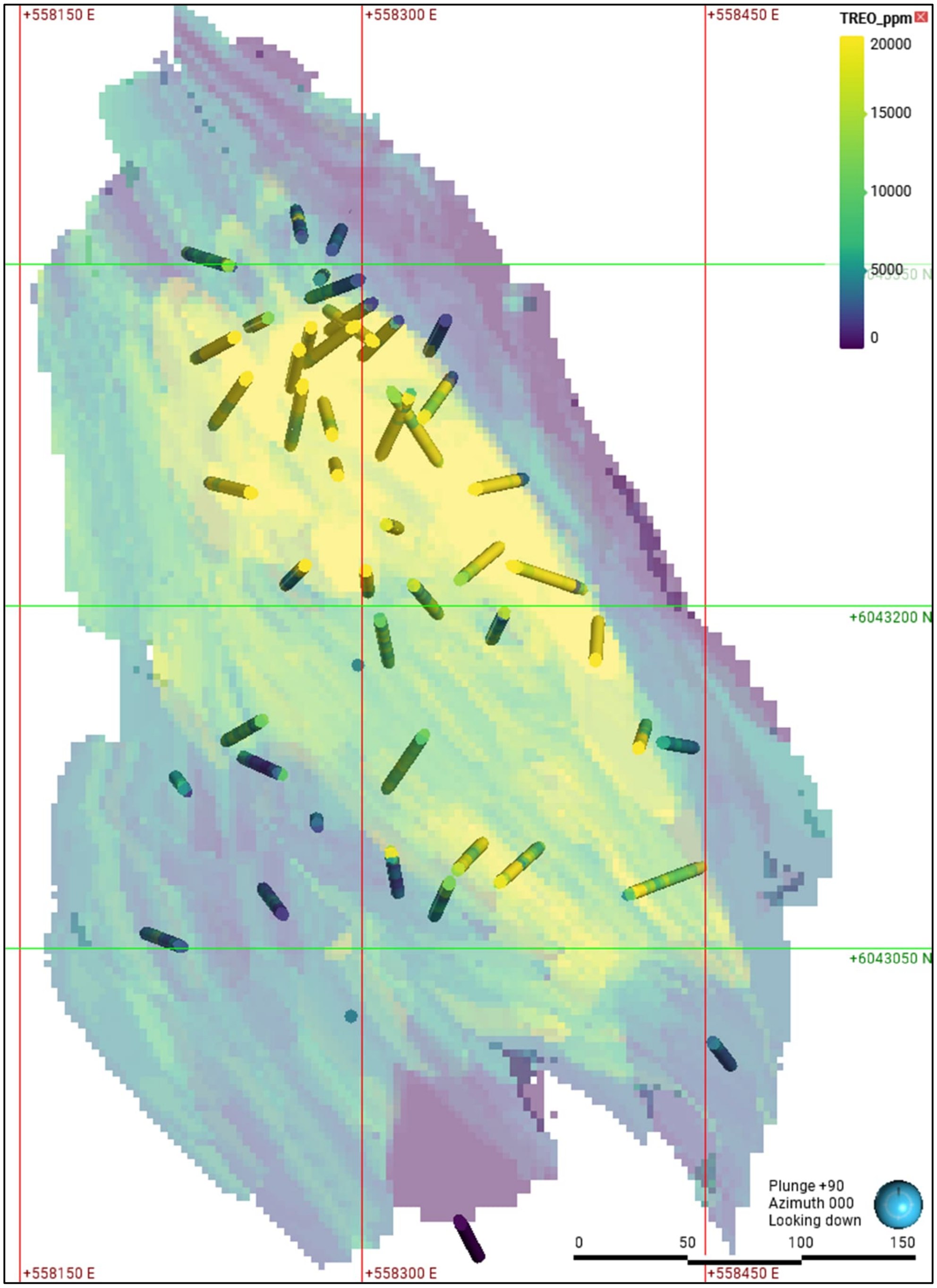
Domain	Element	Composite Mean (ppm)	OK Mean (ppm)	OK-Comp Diff (%)
DC	Ce	11547	11472	-0.66
DC	Pr	954	945	-0.89
DC	Nd	2712	2675	-1.38
DC	Tb	10	10	-1.88
DC	Dy	31	30	-1.72
Xeno	Ce	4502	4559	1.28

Domain	Element	Composite Mean (ppm)	OK Mean (ppm)	OK-Comp Diff (%)
Xeno	Pr	392	399	1.89
Xeno	Nd	1209	1233	2.01
Xeno	Tb	7	7	0.43
Xeno	Dy	26	26	0.15

Source: APEX, 2025

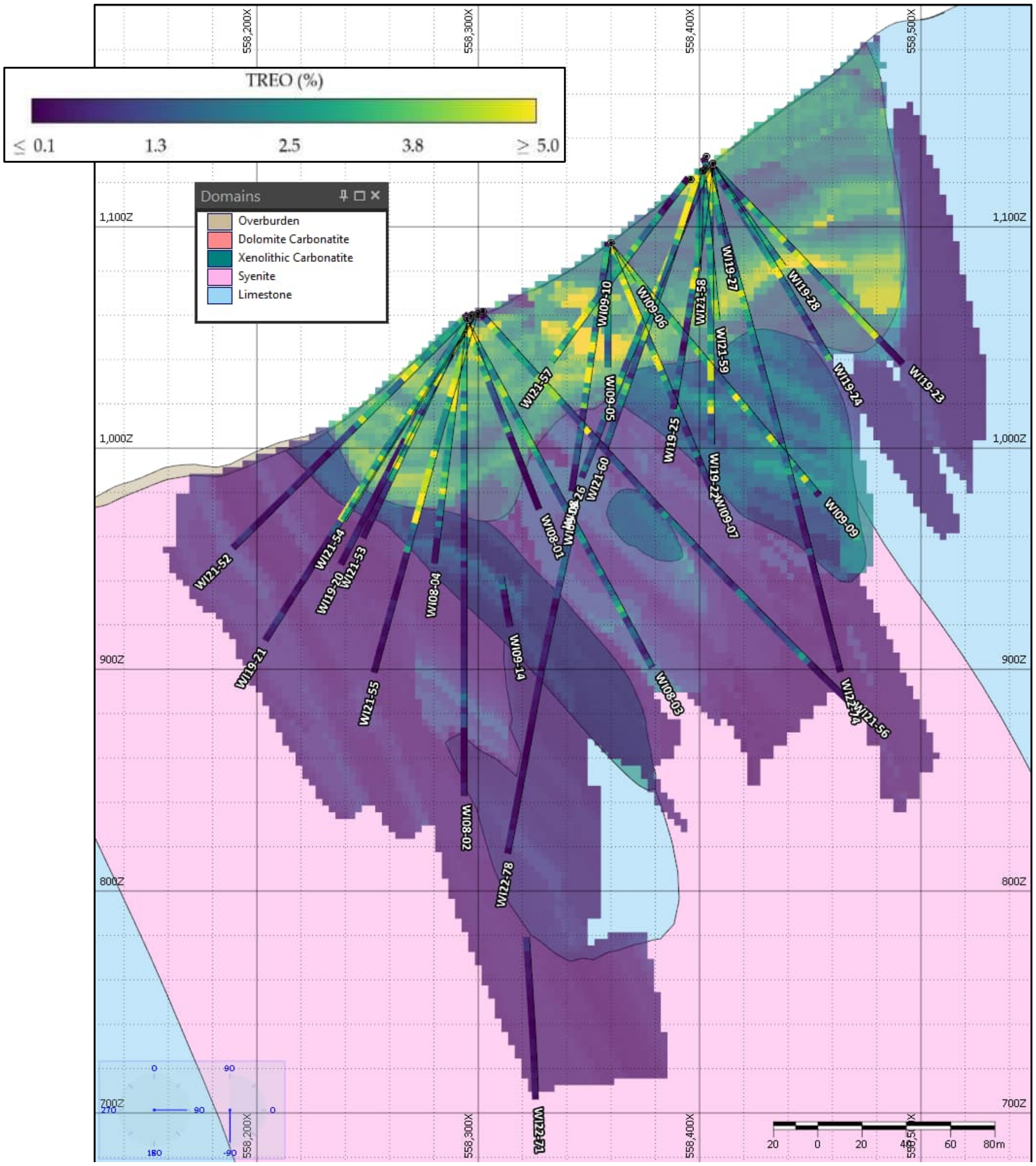
14.7.2 Visual Validation

The block model was visually validated in plan view (Figure 14-16) and in cross-sections (Figure14-17) to compare the estimated metal values versus the conditioning composites. Overall, the model compares well with the composites. There is some local over- and under-estimation observed. Due to the limited number of conditioning data available for the estimation in those areas, this is the expected result. Overall, the estimated block values compare well with composite metal values.



Source: APEX,2025

Figure 14-16: Plan View Showing REE in Composites and Block Model



Source: APEX,2025

Figure14-17: Cross-section Along 6,043,000E, Looking North Showing REE in Composites and Block Model

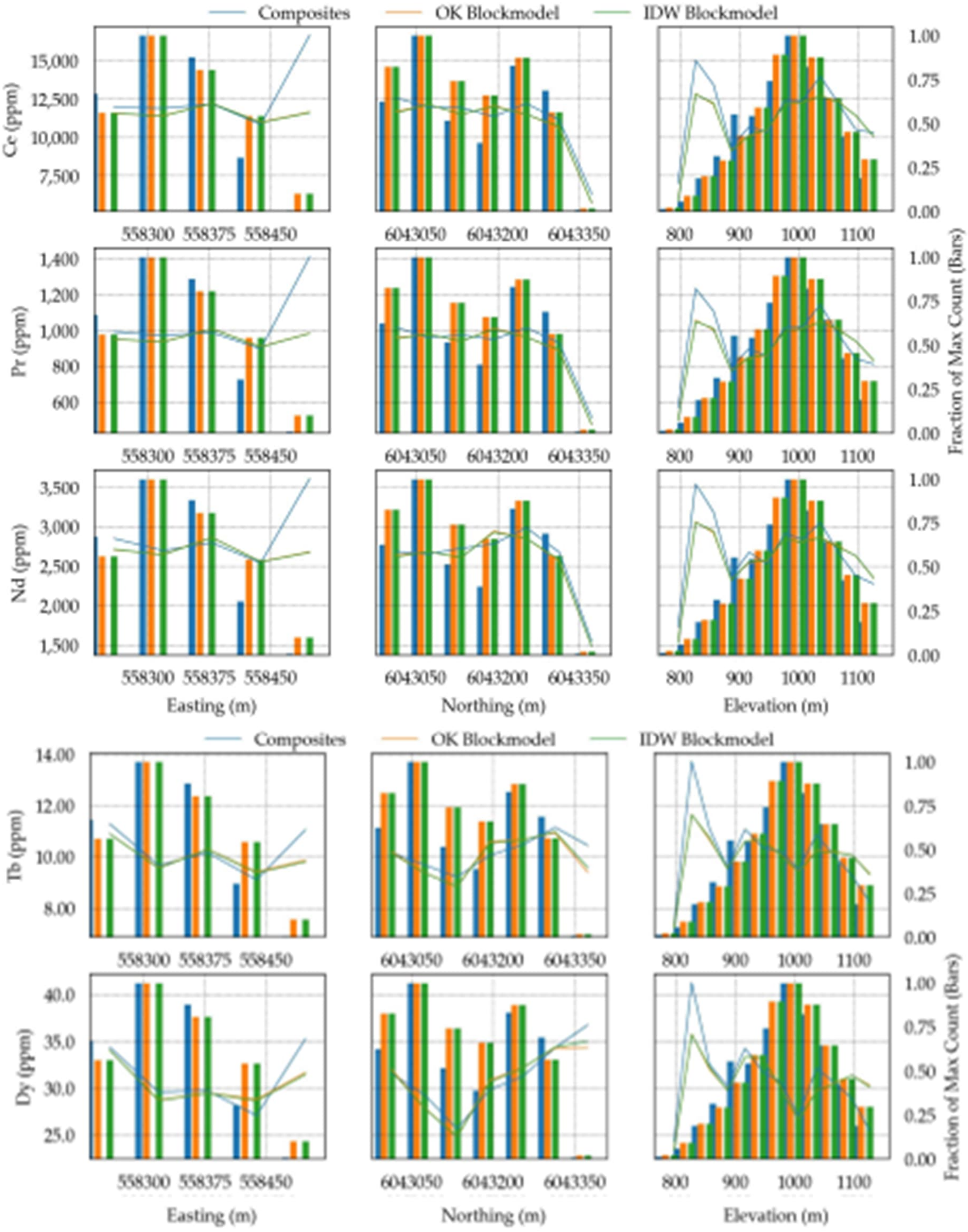
14.7.1 Statistical Validation

14.7.1.1 Swath Plots

Swath plots verify that the estimated block model honors directional trends and identifies potential areas of over- or under-estimation. They are generated by calculating the average metal grades of composites and estimated block models within directional slices. All three directional slices used a window of 30 m.

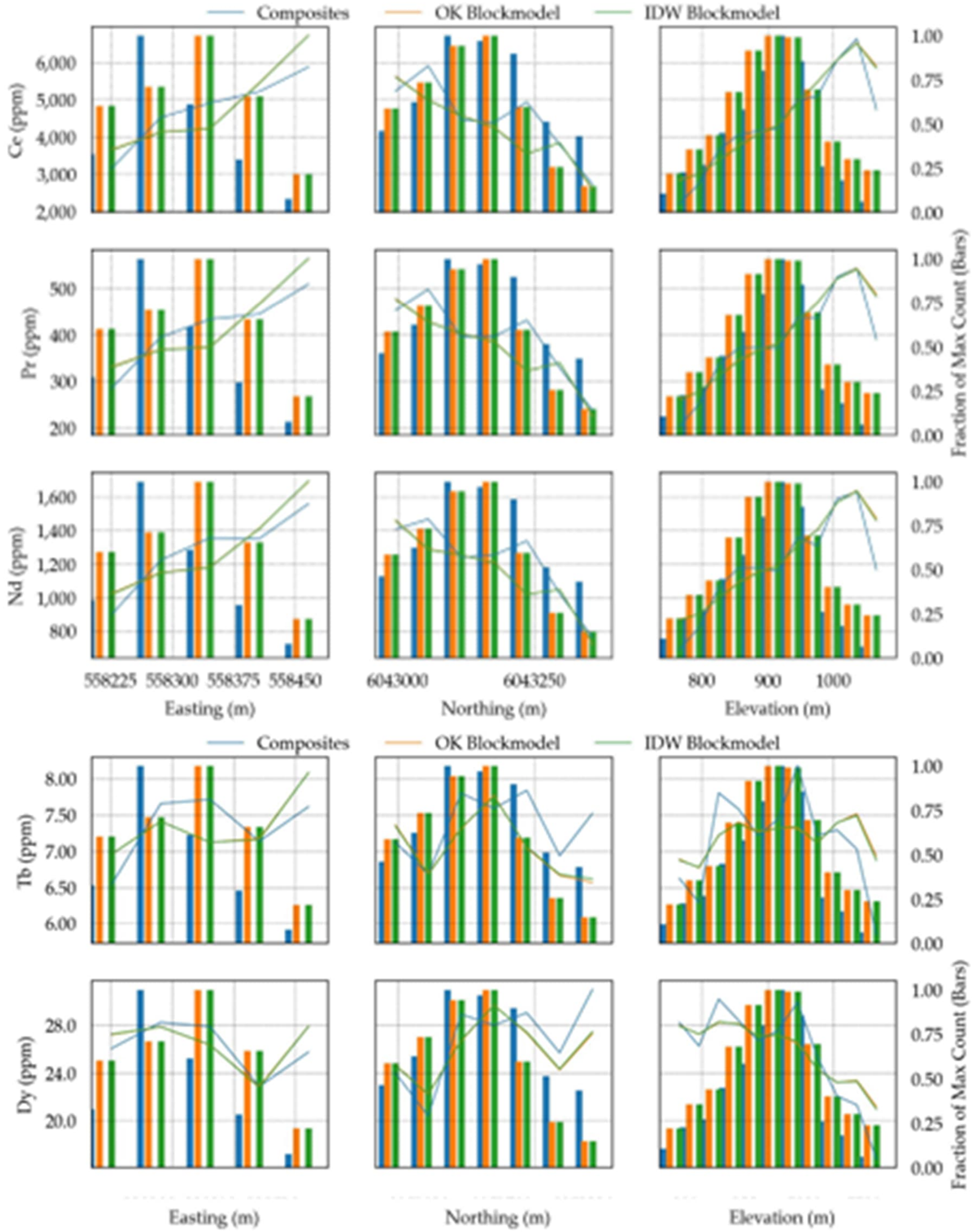
Swath plots for all metals estimates in the Dolomite-Carbonatite and Xenolithic-Carbonatite, are illustrated in Figure 14-18 and and Figure 14-19. Swath plots were generated for the minor syenite and limestone domains but are not shown. There are minor instances of localized over- and under-estimation; however, it is believed to be a product of a lack of conditioning data in those areas and the smoothing effect of kriging. Overall, the block model adequately reproduces the trends observed in the composites in all three directions.

The QP independently created swath plots restricted to Measured + Indicated Resources and did not note any material local biases.



Source: APEX, 2025

Figure 14-18: Swath Plots of Composite Values Versus Estimated Block Model Values in the Dolomite Carbonatite



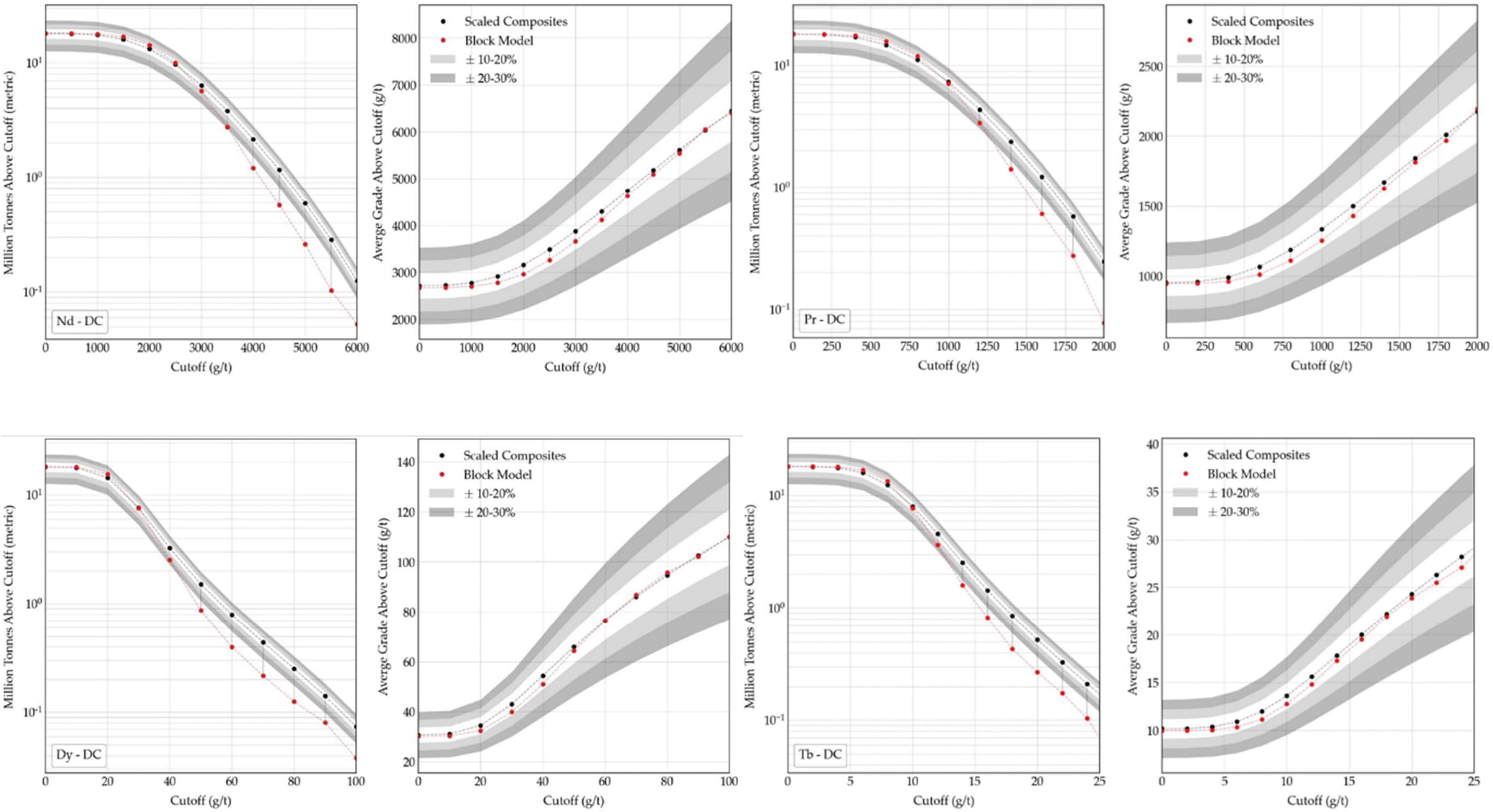
Source: APEX,2025

Figure 14-19: Swath Plots Of Composite Values Versus Estimated Block Model Values in the Xenolithic Carbonatite

14.7.1.1 Volume-Variance Validation – Change of Support

Volume-variance corrections (Figure 14-20) are used to ensure the estimated models are not over-smoothed or too selective, which would lead to inaccurate estimation of global tonnage and grade. To verify that the correct level of smoothing is achieved, grade and tonnage curves using the theoretical histograms that indicate the anticipated variance and distribution of each estimated metal at the selected block model size are calculated and plotted against the estimated final block model. The Dolomite-Carbonatite and Xenolithic Carbonatite domains show acceptable smoothing (within ± 5%) near the average reported resource grades. Continued infill drilling and more refined estimation domain interpretations will help control smoothing in future work.

The QP independently conducted similar checks using Change of Support (COS) Herco methodology. The checks did not suggest issues with smoothing within Measured and Indicated Resources.



Source: APEX,2025

Figure 14-20: Change of Support Analysis in the Dolomitic Carbonatite

14.8 Mineral Resource Classification

14.8.1 Classification Definitions

The 2024 MRE discussed in this report has been classified in accordance with guidelines established by the CIM “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” dated November 29, 2019, and CIM “Definition Standards for Mineral Resources and Mineral Reserves” dated May 10, 2014.

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.

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.

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.

14.8.2 Classification Methodology

According to the CIM Definition Standards, the 2024 MRE update is classified as measured, indicated, and inferred resources. The classification is based on geological confidence, data quality and grade continuity of the data. The most relevant factors considered in the classification process were the following:

- Density of conditioning data
- Level of confidence in drilling results and collar locations
- Level of confidence in the geological interpretation
- Continuity of mineralization

- Level of confidence in the assigned densities
- Metallurgical information to establish recoveries.

A multiple-pass classification strategy consisting of a sequence of runs flagged each block with the run number first meeting a set of search restrictions described in Table 14-14. With each subsequent pass, the search restrictions decrease, representing a decrease in confidence and classification from the previous run. For each run, a search ellipsoid is centered on each block and orientated in the same way described in Section 14.4.7. This process is completed separately from grade estimation. The results were smoothed using an LVA variant of the maximum a posterior selection (MAPS) algorithm developed by APEX. Finally, a small zone was manually upgraded to measured resources to ensure the classification model was adequately continuous.

Mineral resources are not mineral reserves and have not demonstrated economic viability.

Table 14-14: Search Parameters Utilized by the Multiple-Pass Classification Strategy

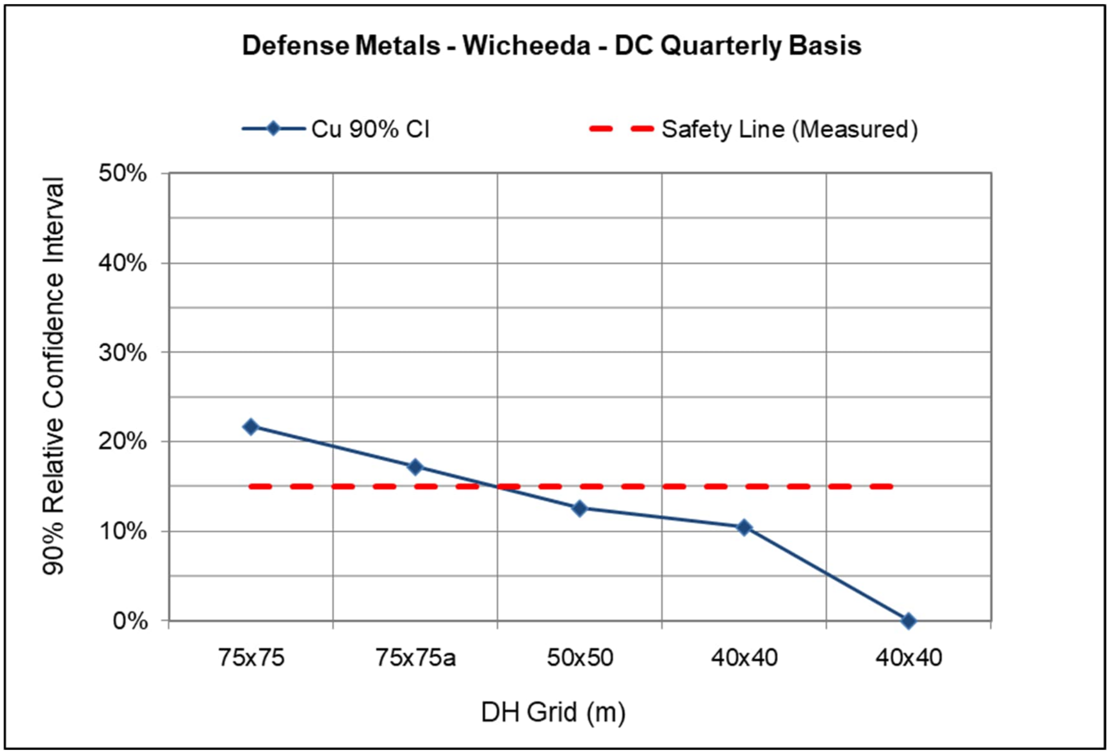
Classification	Pass	Minimum No. of Drillholes	Ranges (m)		
			Major	Minor	Vertical
Measured	1	3	30	30	15
Indicated	2	3	90	60	30
Inferred	3	2	120	120	30

Source: APEX, 2025

The QP conducted an independent drill hole spacing study based on annual and quarterly production increments. Based on a comparison between the two approaches, in the QP’s opinion, the current classification is reasonable.

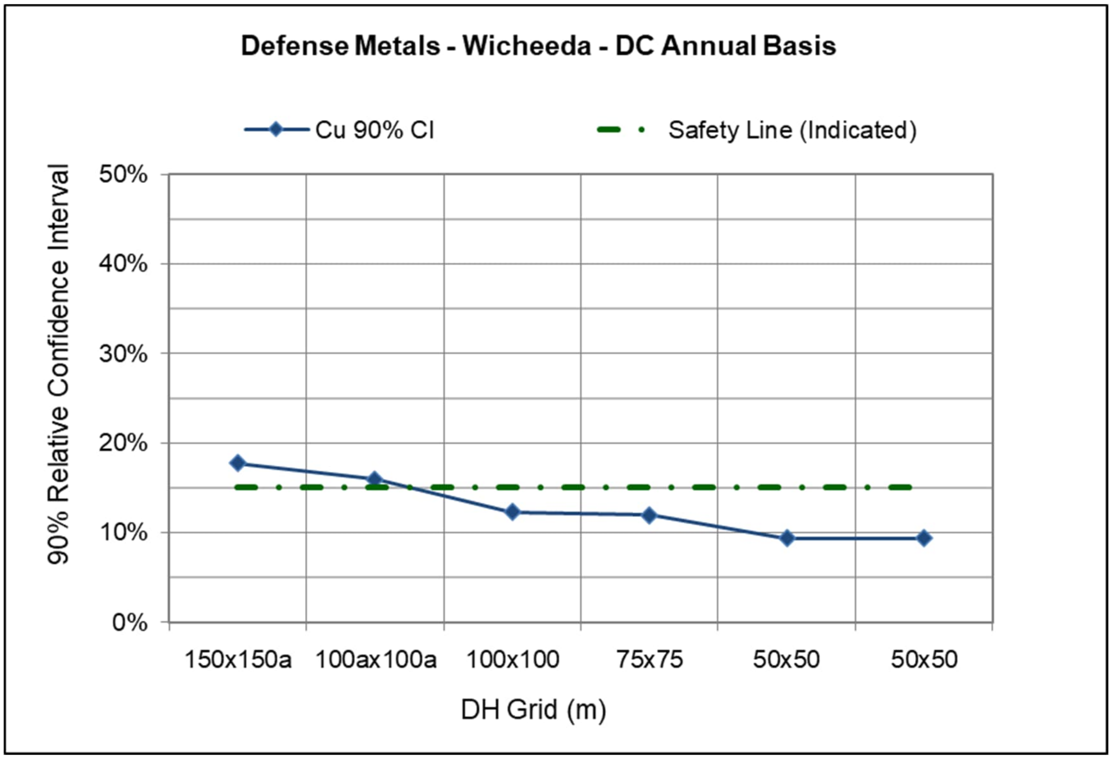
Based on quarterly and annual production increments, a drill hole spacing study suggests a spacing of 65 m is required to support Measured Resources (Figure 14-21) and a spacing of 100 m (Figure 14-22) is required to support Indicated Resources. This study should be refined in future models.

A review of the drill spacing within Measured and Indicated resources found most blocks met the criteria provided by the drill spacing study.



Source SRK, 2025

Figure 14-21: Drill Spacing Summary Dolomite Carbonatite – Quarterly Basis



Source SRK, 2025

Figure 14-22: Drill Spacing Summary Dolomite Carbonatite – Annual Basis

14.9 Evaluation of Reasonable Prospects for Eventual Economic Extraction

The RPEEE requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate economic net smelter return (NSR) taking into account extraction scenarios and processing recoveries. In order to meet this requirement, the QP considers for the purpose of this exercise that the Project is amenable to open pit mining.

To demonstrate reasonable prospects for eventual economic extraction SRK constructed a conceptual constraining pit shell for the Project, based on Measured, Indicated and Inferred mineralized material. The updated mineral resource has been constrained using economic assumptions of surface open pit scenarios. The potentially minable portions of the block model are conceptual in nature.

Input parameter assumptions are provided in Table 14-5.

SRK has defined the proportions of Mineral Resource to have potential for economic extraction for the Mineral Resource based on a NSR. To determine the potential for economic extraction SRK has used the following key assumptions as supplied by Defense Metals, Hatch and Adamas for the portions of the costing, metallurgical recovery and metal prices.

A summary of the key assumptions is shown in Table 14-15. While metal pricing was used for all REO present, for brevity, metal prices are shown only for key economic contributors. The Adamas pricing has been escalated by 15% for resource RPEEE.

Table 14-15: Pit Optimization Key Input Parameters

Description	Units	Value Used
$Nd_2O_3Pr_6O_{11}$ Price	US$/kg	152.6
Tb_4O_7 Price	US$/kg	1567.3
Dy_2O_3 Price	US$/kg	508.8
Operating Cost flotation plant	US$/tonne milled	27.60
Operating Cost – hydrometallurgical plant	US$/tonne concentrate treated	1164.4
Mining Cost	US$/t	4.50
Tailings Cost	US$/t	6.55
Transportation Cost (off-site)	US$/t of precipitate produced	87.76
General & Administration	US$/t	3.67
Flotation Recovery	%	variable
Hydrometallurigcal Recovery	%	variable
Hydrometallurgical Payability	%	95
Slope Angle	Degrees (º)	Variable

Source: SRK, 2025

Using the above parameters, SRK determined a variable NSR was appropriate. The cut-off grade (NSR value) is based on the value factors generated in each block. The revenue and related costs vary based on the composition of different elements in each block. Value of a block is the revenue generated in that block minus the related processing and G&A operating costs. If the value of a block is positive and resides within the LG pit shell, it is considered a resource.

14.10 Sources of Risk and Uncertainty in the Mineral Resource Estimation

Factors that may affect the estimates include metal price and concentrate payable assumptions, changes in interpretations of mineralization geometry, continuity of REE mineralization zones, changes to kriging assumptions, metallurgical recovery assumptions, operating cost assumptions, confidence in the modifying factors, including assumptions that surface rights to allow mining infrastructure to be constructed will be forthcoming, delays or other issues in reaching agreements with regulatory authorities and stakeholders, and changes in land tenure requirements or in permitting requirements.

There are currently no known additional legal, political, title, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the potential development of the mineral resources. As the project develops and economic studies are completed, more information on these factors will become available.

Rare earth oxide (REO) price assumptions are based on the available TREO price information. However, rare earth offtakes are established with long term contracts to a limited number of refineries - primarily in Asia. The likely terms of these contracts are not public information. Offtake term assumptions are indicative only. It is not possible to accurately forecast these assumptions, and there is no guarantee that these terms will be realized. Assumptions on the product sales are indicative of potential market values but moving forward should be confirmed via commercial negotiations with refineries.

With respect to environmental and permitting risk and uncertainty, the area surrounding Wicheeda Lake has been known to have high recreational and ecological values. The lake and surrounding area are currently covered under recreational reserve REC6837 established by FLNR. The northern limit of the LG pit lies approximately 400 m southeast of the southern limit of Wicheeda Lake, and approximately 50 m within REC6837 (Figure 4.2). At present there are no restrictions on mineral exploration activities within REC6837. However, FLNR has requested that Defense Metals take all possible steps to minimize the impacts of exploration to the recreational ecological values associated with Wicheeda Lake.

The SRK QP is unaware of any other risks or uncertainties that could affect the accuracy or confidence of the MRE.

14.11 Mineral Resource Reporting

The updated Wicheeda 2024 MRE is reported in accordance with the Canadian Securities Administrators NI 43-101 rules for disclosure and has been estimated using the CIM “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” dated November 29, 2019 and CIM “Definition Standards for Mineral Resources and Mineral Reserves” dated May 10, 2014.

The resource block model discussed above was regularized to a block size of of 6 m (X) by 6 m (Y) by 3 m (Z) for reporting of Mineral Resources. Mineral reserves, mine planning, and mine scheduling used 6 m (X) by 6 m (Y) by 6 m (Z) blocks.

The 2024 MRE include 29.3 Mt of Measured+Indicated resource at an average grade of 2.27% of TREO and 5.7 Mt of Inferred resource at an average grade of 1.40% TREO. No mining dilution has been incorporated into the Mineral Resources stated below. The Mineral Resources are stated inclusive of Mineral Reserves constrained within an optimized open pit shell.

Cut-off grade is based on the value factors generated in each block. The revenue and related costs vary based on the composition of different elements in each block. The value of a block is the revenue generated in that block minus the related processing and G&A operating costs.

A summary of the surface mineable Mineral Resources by rock type and Resource classification is shown in Table 14-13.

Table 14-16: Summary of the Mineral Resources as of February 28, 2025

Mineral Resource	Rock Type	Ore	TREO	Pr_6O_{11}	Nd_2O_3	Tb_4O_7	Dy_2O_3
		kTonnes	%	ppm	ppm	ppm	ppm
Measured	Dolomite Carbonatite	5,350	2.99	1,161	3,158	12	35
	Xenolithic Carbonatite	300	1.64	662	1,950	11	36
	Syenite	50	1.40	560	1,631	11	40
	Limestone	10	1.96	837	2,310	13	41
	Total	5,720	2.90	1,128	3,079	12	35
Indicated	Dolomite Carbonatite	12,030	2.90	1,139	3,116	12	34
	Xenolithic Carbonatite	10,060	1.32	547	1,618	9	30
	Syenite	1,320	1.07	442	1,331	8	29
	Limestone	160	1.40	569	1,627	11	43
	Total	23,570	2.11	843	2,367	10	32
Measured + Indicated	Dolomite Carbonatite	17,380	2.93	1,145	3,129	12	34
	Xenolithic Carbonatite	10,360	1.33	550	1,628	9	30
	Syenite	1,370	1.08	447	1,343	8	29
	Limestone	170	1.44	588	1,675	11	43
	Total	29,290	2.27	899	2,506	11	33
Inferred	Dolomite Carbonatite	570	2.67	1,072	2,883	12	37
	Xenolithic Carbonatite	3,280	1.42	587	1,712	9	32
	Syenite	1,630	0.90	401	1,229	9	34
	Limestone	210	1.50	600	1,641	9	33
	Total	5,690	1.40	582	1,687	9	33

Source: SRK, 2025

Notes:

- CIM (2014) definitions were followed for Mineral Resources.
- The Qualified Person for the MRE is Doug Reid, P.Eng., EGBC (23347), an SRK employee.
- The effective date of the Mineral Resource is February 28, 2025
- Dollar values herein stated are United States Dollars (US$)

- Mineral Resources are calculated using the values derived from all REEs present in the deposit. Individual REO pricing provided by Adamas Intelligence was escalated by 15% and used for pit optimization. The key REO pricing is as follows:
  - NdPr Oxide 152.6 $/kg REO
  - Tb4O7 1567.3 $/kg REO
  - Dy2O3 508.8 $/kg REO
- Mineral Resources are defined within a pit shell derived from the optimization software, GEOVIA Whittle™
- Cut-off grade is based on the value factors generated in each block. The revenue and related costs vary based on the composition of different elements in each block. The value of a block is the revenue generated in that block minus the related processing and G&A operating costs.
- The base mining costs are assumed to be $4.50/t. The mining costs vary based by the bench and depth of the pit. The average mining costs for the life of mine is calculated to be $4.74/t mined.
- Processing costs consist of flotation plant cost at the mine site and a hydrometallurgical/solvent extraction plant that is off the mine property. The operating cost of the flotation plant is $27.60/t milled and the hydrometallurgical plant operating cost is $1,164.4/t of concentrate treated.
- General and administration costs of the mine site is $3.67/t for ore milled.
- Tailings management and storage cost is $6.55/t of ore.
- Off-site cost (transportation) is $87.76/t of precipitate products produced.
- Processing recoveries are calculated as follows:
  - Flotation recovery for TREO = -11.183*TREO^2 + 67.831*TREO - 20.42194.0%. For ore above 3% TREO, the flotation recovery is set to 82.4%. For grade less than 0.32% TREO, the flotation recovery is set to 0.0%.
  - Flotation recovery for each REE is calculated by multiplying the TREO recovery by that element's recovery factor. For example, the factors for Pr, Nd, Tb, and Dy are 0.995, 0.996, 0.734, and 0.636, respectively.
  - Similarly, hydrometallurgical recoveries are assigned for each REE, which for Pr, Nd, Tb, and Dy are 93.2%, 93.5%, 80.2%, and 73.4%, respectively
- A 95% payability has been applied to the final hydrometallurgical product.
- Bulk density is assigned by lithology.
- No mining dilution has been applied.
- Mineral Resources are reported inclusive of those Mineral Resources converted to Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Figures are rounded to the appropriate level of precision for the reporting of mineral Resources. Due to rounding, some columns or rows may not sum as shown.
- The TREO grade encompasses 15 rare earth elements present in the deposit.
- The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues

The QP, Douglas Reid, does not know of any legal, political, environmental, or other risks that could materially affect the potential development of the mineral Resources. Mr. Reid personally inspected the subject Wicheeda Project on October 31 and November 1, 2024.

14.12 Previous Mineral Resource Estimate

The previous MRE was completed by APEX in 2023 with an effective date of August 28, 2023. There was no additional drilling, changes to the geological model to the estimate grade and classification. The only change from the current MRE to the current MRE was the change in reporting to reflect changes in the processing flowsheet and REE pricing.

The 2023 MRE was reported based on a cut-off of 0.50% TREO, the current MRE is reported based on a NSR approach which considered variable processing costs and related costs on a block-by-block basis. Blocks within the optimized pit shell and a positive revenue after all costs were considered as a resource. This approach may have impacted the optimized pit shell as well.

The current model represents a 15% drop in Measured and Indicated tonnes, a 13% increase in TREO grade and a 4% decrease in contained TREO.

Table 14-17: Wicheeda Mineral Resource (effective date August 28, 2023).

Category	Tonnes	TREO	TREO	CeO2	La2O3	Pr6O11	Nd2O3	Sm2O3	Gd2O3	Eu2O3	Dy2O3	Tb4O7	Ho2O3
	(Million)	(%)	(kt)	(%)	(%)	(%)	(%)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)
Measured	6.37	2.86	183	1.39	1.00	0.11	0.31	312	139	63	35	12	4
Indicated	27.80	1.84	516	0.89	0.62	0.07	0.21	232	111	50	32	10	4
M&I	34.17	2.02	699	0.98	0.69	0.08	0.23	247	116	52	32	10	4
Inferred	11.05	1.02	113	0.50	0.31	0.04	0.13	166	91	38	35	9	5

Source: APEX, 2023

Notes for Resource Table:

- The 2023 MRE is classified according to the CIM “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” dated November 29, 2019, and CIM “Definition Standards for Mineral Resources and Mineral Reserves” dated May 10, 2014.
- The 2023 MRE was prepared by Warren Black, M.Sc., P.Geo. and Tyler Acorn, M.Sc., of APEX Geoscience Ltd under the supervision of the QP, Michael Dufresne, M.Sc., P.Geo. following CIM Definition Standards.
- Mineral Resources that are not mineral reserves have not demonstrated economic viability. There is no guarantee that any part of the mineral resources discussed herein will be converted to a mineral reserve in the future.
- All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- The reasonable prospect for eventual economic extraction is met by reporting the Mineral Resources at a cutoff grade of 0.50% TREO (total rare earth oxide, sum of 10 oxides: CeO2, La2O3, Nd2O3, Pr6O11, Sm2O3, Eu2O3, Gd2O3, Tb4O7, Dy2O3 and Ho2O3), contained within an optimized open pit shell.
- Median rock densities are supported by 8,075 measurements applied: 2.95 g/cm3 (mineralized dolomite-carbonatite), 2.90 g/cm3 (unmineralized dolomite-carbonatite), 2.85 g/cm3 (mineralized xenolithic-carbonatite), 2.76 g/cm3 (unmineralized xenolithic-carbonatite), 2.73 g/cm3 (Syenite), and 2.76 g/cm3 (limestone).
- The reasonable prospect for eventual economic extraction is met by reporting the Mineral Resources at a cutoff grade of 0.50% TREO (total rare earth oxide, sum of 10 oxides: CeO2, La2O3, Nd2O3, Pr6O11, Sm2O3, Eu2O3, Gd2O3, Tb4O7, Dy2O3 and Ho2O3), contained within an optimized open pit shell.
- The cutoff grade is calculated, and the open pit shell is optimized based on the assumption that the hydrometallurgical processing can produce mixed REE carbonate precipitates. The parameters utilized, as in the 2021 MRE, include the following considerations:
  - TREO price: $18.66/kg
  - Exchange rate of 1.30 C$:US$
  - Precipitate production grades of 81.09% of TREO
  - Processing costs include $21.47/t of mill feed for flotation plus a variable cost for hydrometallurgical plant that varies based on the feed grade. The average cost of hydrometallurgical plant is assumed to be $1,204/t of concentrate.
  - Mining cost of C$2.00/t for mill feed and waste
  - G&A Costs of C$3.33/t for mill feed.
  - The overall process recoveries: For TREO>=2.3%, recovery is 69.6%; between 2.3% and 1.5% TREO, recovery is 65.3%; and less than 1.5% TREO, recovery is 52.2%. These assume variable flotation recoveries and a constant 87% hydrometallurgical recovery.
  - Overall pit slope angles vary by zone between 40 and 48 degrees.

15. Mineral Reserve Estimates

15.1 Introduction

Section 15 and Section 1616 of this technical report encompass mine design and associated evaluation-analysis. Section 15 addresses pit optimization, strategic mine planning, and the reserve statement. Section 16 presents the final pit and phase designs, waste dump designs, production scheduling, and mine operations. To gain a comprehensive understanding of the mine planning and design, the author recommends that readers review these two sections of the report together.

REE are a group of 15 elements with similar chemical properties, typically found as oxides in nature. The Wicheeda deposit contains all 15 REEs with different recoverable values. Their contribution to the project's value varies significantly based on price and recovery rates. Neodymium (Nd) accounts for 69.6% of the project's total value, followed by Praseodymium (Pr) at 25.0%. Terbium (Tb), and Dysprosium (Dy) collectively contribute 3.3% of the value. These four elements together represent 97.9% of the project's total value, while the remaining nine elements contribute only 2.1%. For simplicity, only the main four REEs are reported (as REO). Additionally, note that TREO grades in this report represent the sum of all 15 REOs and not just the four main REOs mentioned above.

The Wicheeda deposit will be mined as a conventional open-pit truck and shovel operation. Mining will occur on 6 m benches. Ore will be hauled to a crusher near the pit rim. Crushed ore will be conveyed to the nearby concentrator. Run-of-mine waste will be hauled to a waste storage facility (WSF) located adjacent to the pit.

15.2 Pit Geotechnical Rock Mass Assessment and Slope Design

The geotechnical characteristics of rock types that will be encountered in the Wicheeda pit have been evaluated using the following data:

- Detailed HQ3 oriented core geotechnical holes drilled during the PEA (920 m in 4 holes in 2023, and 1153 m in 5 holes in 2024).
- Four of the 2024 drillholes were probed using Acoustic and Optical Televiewers to collect joint orientations and major structures data.
- All hole orientation surveys were done using two independent methods (magnetic and gyroscopic).
- The geotechnical specific drillholes were tested using industry standard hydraulic injection (packer) testing methods.
- Nested Vibrating Wire Piezometers (VWP) were installed in one of the drillholes.

Drilling and oriented core logging was done for five holes within the centre of the proposed PEA open pit in 2022. Four open pit slope-specific drillholes were designed and drilled in 2023 and logged on-site using a site-specific logging manual by SRK consultants and associates. All data were processed and checked, with some adjustments being required for the 2022 data. All laboratory and downhole geophysics data, including acoustic televiewer

(ATV) interpretation, was incorporated into the geotechnical database. The rock mass rating specific data was imported into Leapfrog Geo™ for processing and visualization, and a preliminary brittle deformation structural model was built.

Standard industry rock logging methods were used to characterize and classify the rock mass. Point load testing and density measurement was done on-site by Defense Metals' contractors (Apex Geoscience). Rock strength testing (uniaxial compressive, triaxial, Brazilian, and shear strength) was done at Queens University laboratories in Ontario, Canada. Packer testing of selected downhole intervals was done using an IPI™ SWiPS (Standard Wireline Packer System).

The total rock mass data has has an average engineered-intact rock strength of around 46 MPa and an RQD (Rock Quality Designation) of 63%. The joint spacing is relatively low for most of the holes, except WI-22-75, WI-23-80, 81, and 83. Joint conditions are FAIR. The RMR76 (Bieniawski Rock Mass Rating, 1976) values by drillhole are, overall, within the FAIR ground class category (Table 15-2), but the fracture frequency, FF/m, suggests the presense of poorer conditions within. These data are also presented by lithology (Table 15-2), rock type (Table 15-3), and geotechnical domain (Table 15-4).

Table 15-1: Rock Mass Summary, by Drillhole (Average Values)

HoleID	Length (m)	n	IRS-eng (MPa)	RQD %	FF/m	J Spacing (m)	JCR76	RMR76
WI-22-74	252	167	44	57	11	0.09	12	46
WI-22-75	199	131	47	46	7	0.14	12	49
WI-22-77	171	107	50	52	15	0.07	12	43
WI-22-78	301	197	47	65	12	0.08	12	47
WI-22-79	231	152	48	76	12	0.08	12	50
WI-23-80	226	75	50	63	6	0.17	12	55
WI-23-81	271	89	41	66	7	0.14	12	49
WI-23-82	172	55	31	57	13	0.07	12	43
WI-23-83	251	83	52	84	6	0.17	12	53
All	-	1,056	46	63	10	0.10	12	48

Source: SRK, 2025

Table 15-2: Rock mass summary, by lithology (average values)

Lithology	Length (m)	TCR(%)	RQD (%)	IRS-eng (MPa)	UCS (MPa)	FF/m	JCR76	RMR76
OVB	26	74	39	34	-	14	9	39
LIM	838	95	67	49	68	7	12	52
DC	377	92	62	45	87	11	11	47
XDC	369	94	66	45	61	10	12	48
SYN	425	92	59	40	101	14	11	45

Source: SRK, 2025

Table 15-3: Rock mass Summary, by Rock Type (Average Values)

Rock Type	Length (m)	TCR(%)	RQD (%)	IRS-eng (MPa)	UCS (MPa)	FF/m	JCR_{76}	RMR_{76}
SED	838	95	67	49	83	7	12	52
INT	1196	92	62	43	68	12	11	46

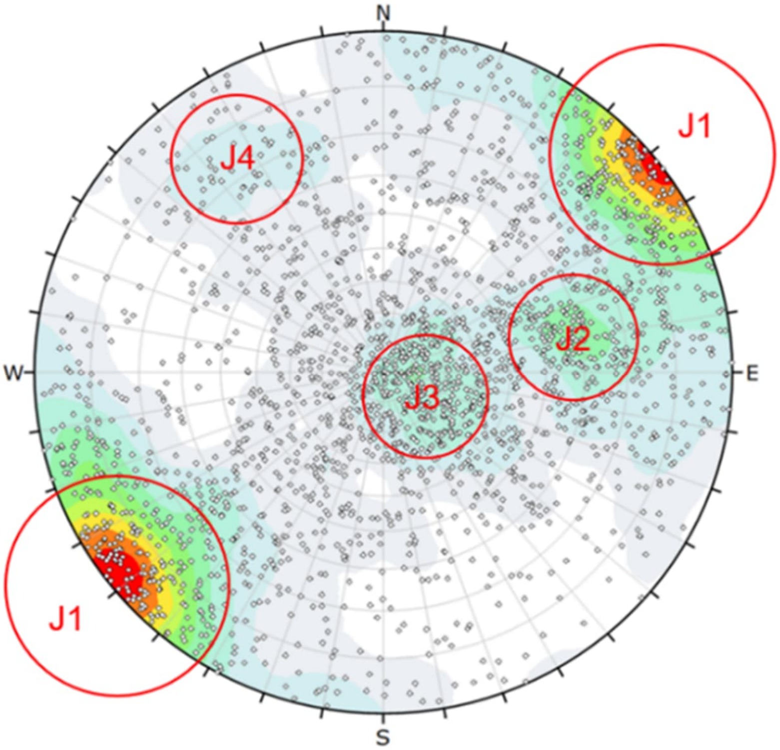
Source: SRK, 2025

Table 15-4: Rock Mass Summary, by Geotechnical Domain (Average Values)

Geotechnical Domain	Length (m)	TCR(%)	RQD (%)	IRS-eng (MPa)	UCS (MPa)	FF/m	JCR_{76}	RMR_{76}
Northeast (NE)	715	96	67	50	106	7	12	52
Southwest (SW)	1319	92	62	44	66	11	11	47

Source: SRK, 2025

Drillholes were oriented perpendicular to the proposed open pit slopes, to reduce the significance of the blind zones. Design joint sets were picked using oriented core and acoustic televiewer (ATV) interpreted joints. Design joint sets illustrated in Figure 15-1 are J1 (88/050 ± 20°), J2 (60/260 ± 15°), J3 (15/310 ± 15°), and J4 (75/155 ± 15°).
J1 prominence is accentuated by the Terzaghi weighting and is probably restricted to the sediments on the north-eastern highwall, while the location within the pit of the J2 set appears to be mostly in the south-west.

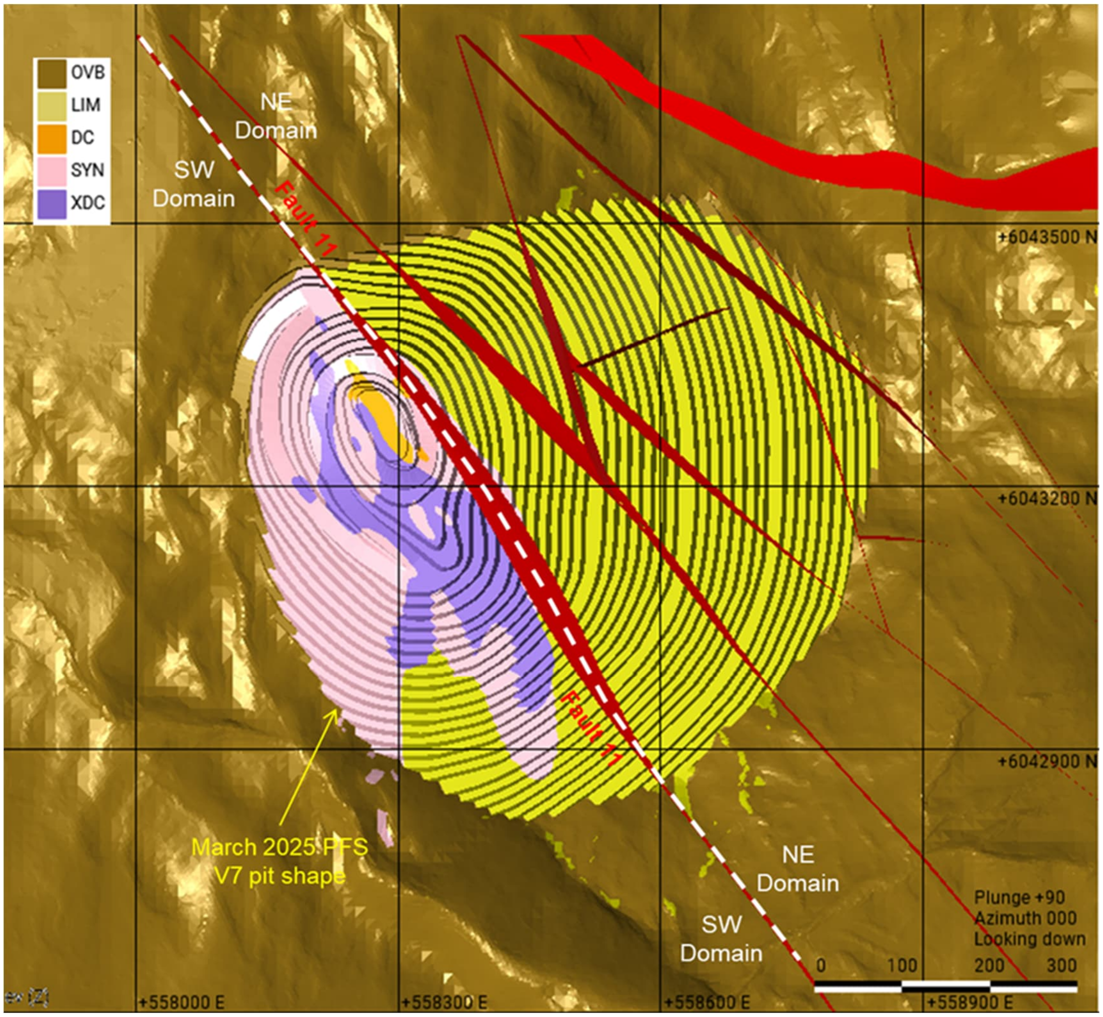


Source: SRK, 2025

Figure 15-1: Oriented core joint poles with design joint sets

Kinematic risk assessments used zero cohesion for the bench and stack scale, with infinitely persistent joints. Data were summarized into intrusive related (INT) and sedimentary (SED) rocks. Preliminary basic cohesion and friction values for INT (c = 200 kPa, φ = 30°) and SED (c = 300 kPa, φ = 33°) were used.

The lithological domains comprise the following rock units – SED: Limestone (LIM) and INT: Dolomite Carbonatite (DC), Xenolithic Dolomite Carbonatite (XDC), and Syenite (SYN). The rock mass geotechnical domains for Wicheeda were established using the litho-structural model as the basis, as illustrated in Figure 15-2.

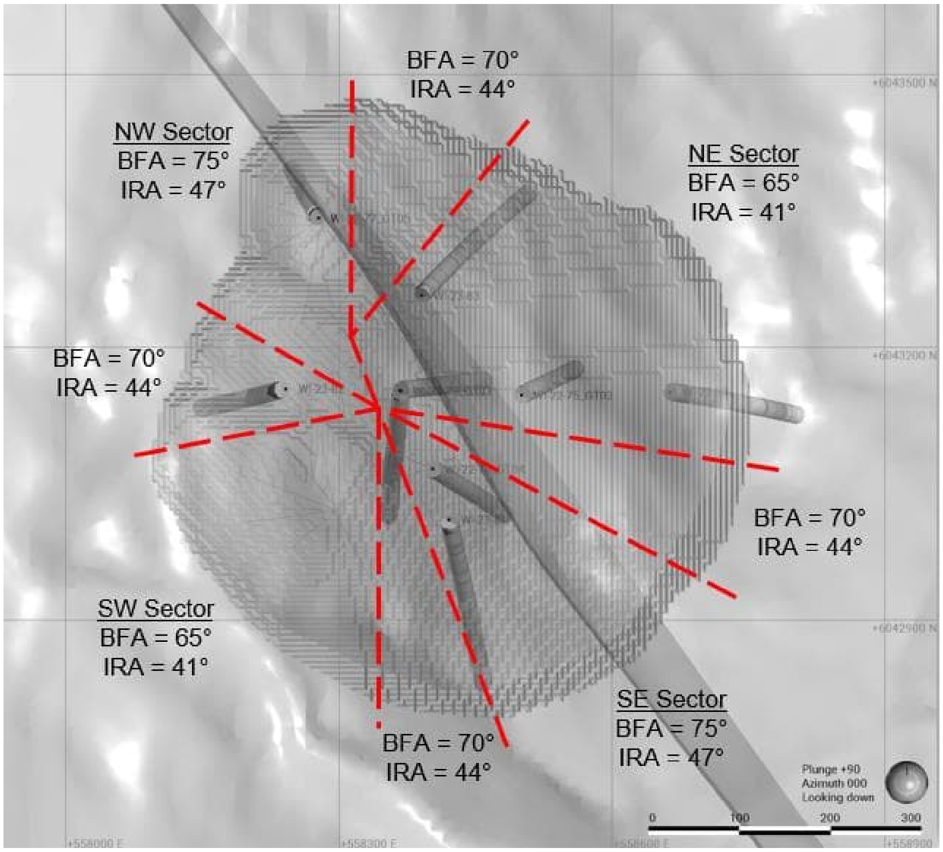


Source: SRK, 2025

Figure 15-2: Open Pit Slope Design Lithologies, Structures and Geotechnical Domains

The structural plane which separates the NE from the SW domain is oriented on an azimuth of approximately 145° (aligned with Fault 11). Rock units within the NE Domain are mostly sedimentary (SED) associated with the regional limestones and siltstone packages. Most joints in this domain are dipping sub-vertical (J1) to intermediate (J2) which overlap into a sub-horizontal set (J3), all which dip in the 210 to 280° dip-direction.In the SW Domain, rocks are mostly intrusive (INT) associated with the carbonatite and country rocks directly influenced by it. Jointing in this domain is dominantly sub-horizontal (J3), but there appears to be a low density cluster of joints which steepen up from J3 in the 010 to 070° dip-direction. A significant bench-face angle (BFA) limiting factor, in both the NE and SW Domains, are localized areas of POOR to FAIR ground conditions.

A 360-degree slope design was constructed, which was split into design sectors, as illustrated in Figure 15-3. The SW and NE Sectors have the highest kinematic risks within the proposed open pit. POOR to FAIR ground conditions have been used for all sectors to arrive at a 12 m high final bench height. Eight-metre-wide catch benches have been used. Geomorphological influence in the form of glacial damage and/or rock mass unloading effects on the upper 50 to 100 m needs further investigation.



Source: SRK, 2025

Figure 15-3: Open Pit Slope Design Sectors with BFA and IRA Annotated

Table 15-5: 360-degree PFS Slope Design Criteria

Slope dip-dir (°)		PFS open pit slope design criteria				Kinematic risk mitigation implemented
From	To	BFA (°)	Bench height (m)	Catch width (m)	IRA (°)	
0	80	65	12	8	41	High planar plus low step-path
80	120	70	12	8	44	Moderate wedge
120	180	75	12	8	47	None implemented
180	220	70	12	8	44	Moderate wedge
220	280	65	12	8	41	High planar, moderate step-path
280	300	70	12	8	44	Moderate planar
300	340	75	12	8	47	None implemented
340	360	70	12	8	44	Moderate wedge

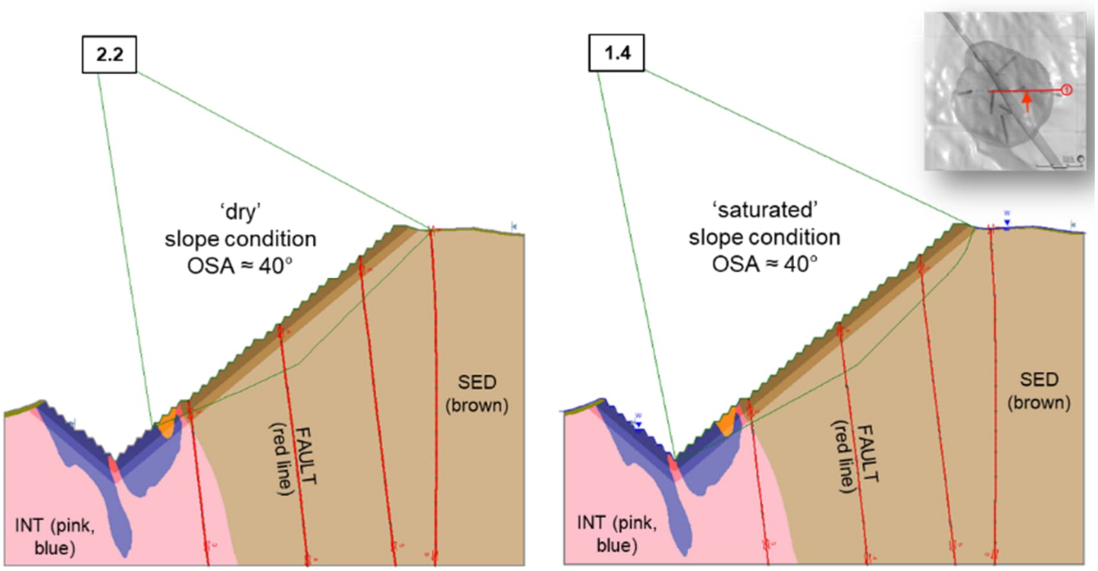
Source: SRK, 2025

Note that:

1) full height pre-shear lines are required on all final faces
2) three-row trim shots should be used between production and final wall blasts
3) cut height is 6 m
4) maximum stack height is 96 m
5) no more than four full bench stacks totaling 384 m height can be placed on a single slope without additional design guidance
6) geotechnical berms of at least 15 m wide should be placed between each 8-bench stack
7) ramps of at least 15 m width can be used in place of geotechnical berms
8) these guidelines assume well drained slopes
9) the double-benching at 12 m height provided in these guidelines may need to be reduced to 6 m in local areas if poorer ground conditions are encountered
10) these guidelines apply to both the NE and SW domains within 100 m of the final PFS ultimate pit design (2025 V7).

The 2025 V7 pit shape was assessed for stability using standard industry limit equilibrium analysis software (Slide2^{TM}). Six vertical sections were cut, at least one in each design sector, and slope stability models were constructed using ‘dry’, ‘realistic’, and ‘saturated’ water conditions. Tension cracks were also introduced into the models as an unfavourable case scenario. The slope stability models, as seen in the example presented in Source: SRK, 2025

Figure 15-4, indicate that the slopes within this V7 pit shape and rock mass achieve the acceptance criteria of a factor of safety of ≥ 1.3.



Source: SRK, 2025

Figure 15-4: Representative Slide2^{TM} Slope Stability Models on the Wicheeda Eastern Highwall

15.3 Pit Optimization Parameters

This section describes the input parameters used for pit optimization, a process whereby an economic pit shell is generated for strategic mine planning and provides guidance for mine design.

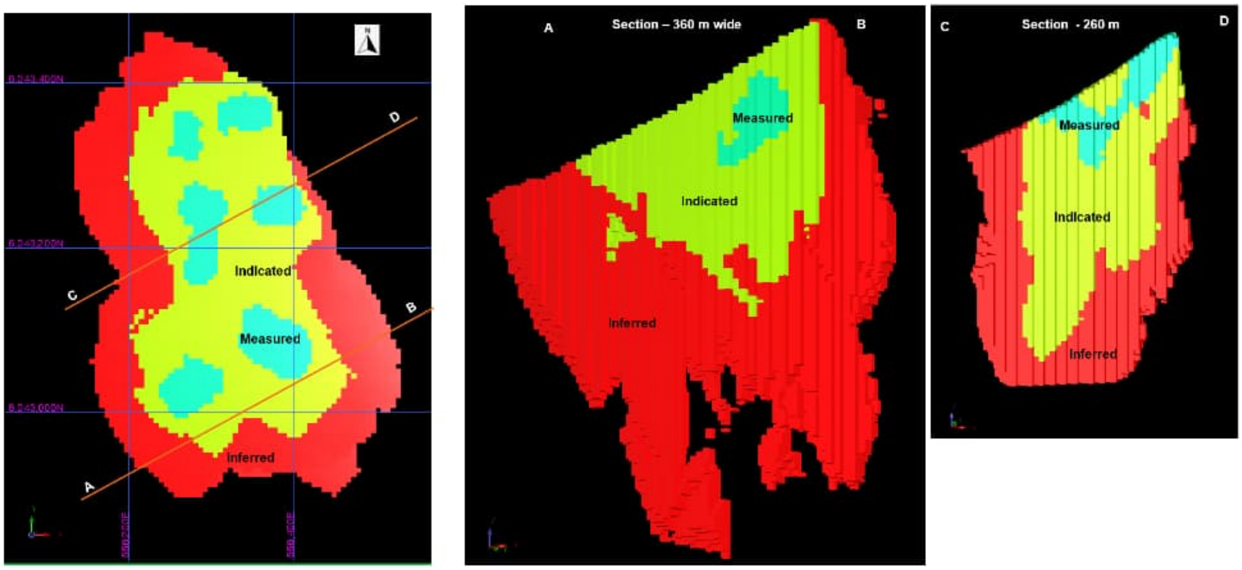
The main input parameters for pit optimization include metal prices, the resource model, geotechnical parameters, operating costs, mineral processing recoveries, and offsite costs and charges. The parameters have been reviewed and provided by QPs in their respective technical areas and are described in the following subsections. Geotechnical parameters were described in Section 15.2.

15.3.1 Commodity Price

15.3.2 Resource Model

A mining model was generated utilizing the resource model described in Section 14, whereby the Mining QP generated additional fields and attributes to facilitate pit optimization, including dilution factors, mining cost adjusting factors, geotechnical zones, recoveries and economic parameters.

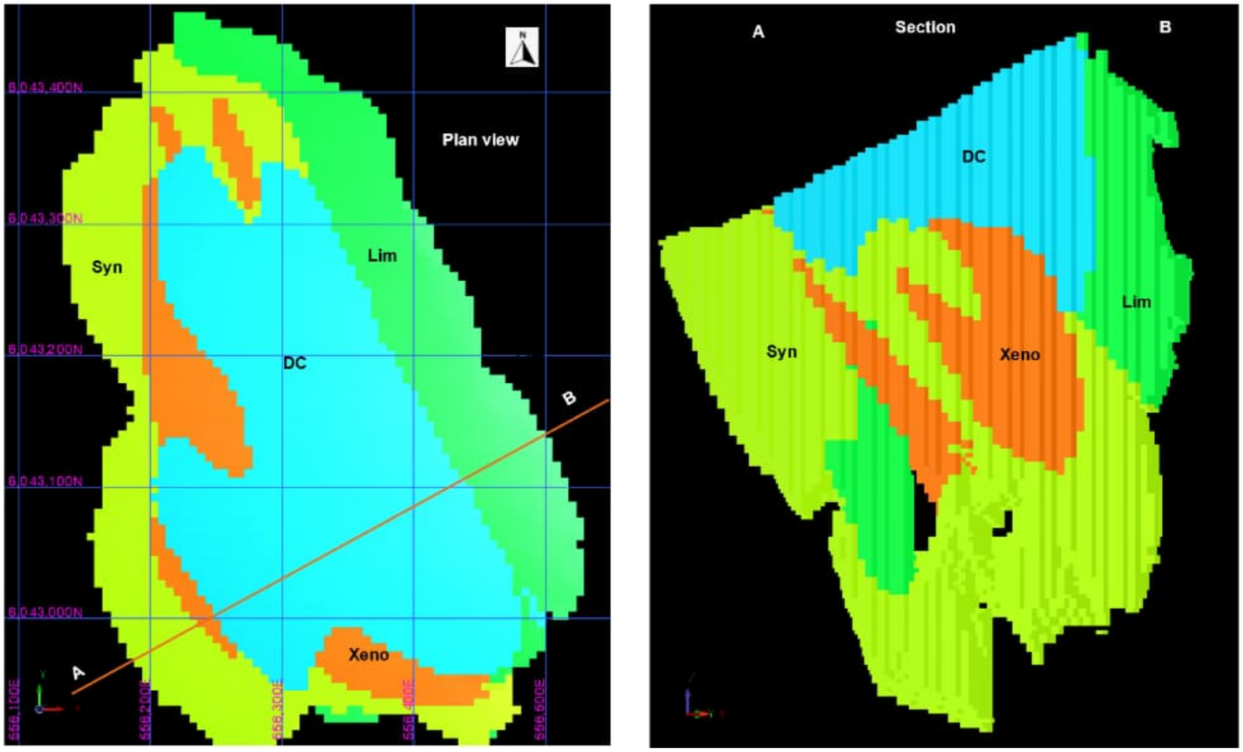
The Wicheeda resource (Figure 15-5) has been classified into Measured, Indicated and Inferred categories (Section 14.8). Inferred resources are not considered ore in reserve calculations and are treated as waste in economic analysis. The Mining QP notes that ongoing exploration work, including in-fill drilling, could result in an adjustment to the classification of material used in this study.



Source: SRK, 2025

Figure 15-5: Resource classification

Figure 15-6 shows the main rock types in the deposit including limestone (Lim), syenite (Syn), xenolithic carbonatite (Xeno), and dolomite carbonatite (DC) – the last being the main unit hosting high-grade REE. Densities were applied based on rock type and by mineralization (Section 14.4).



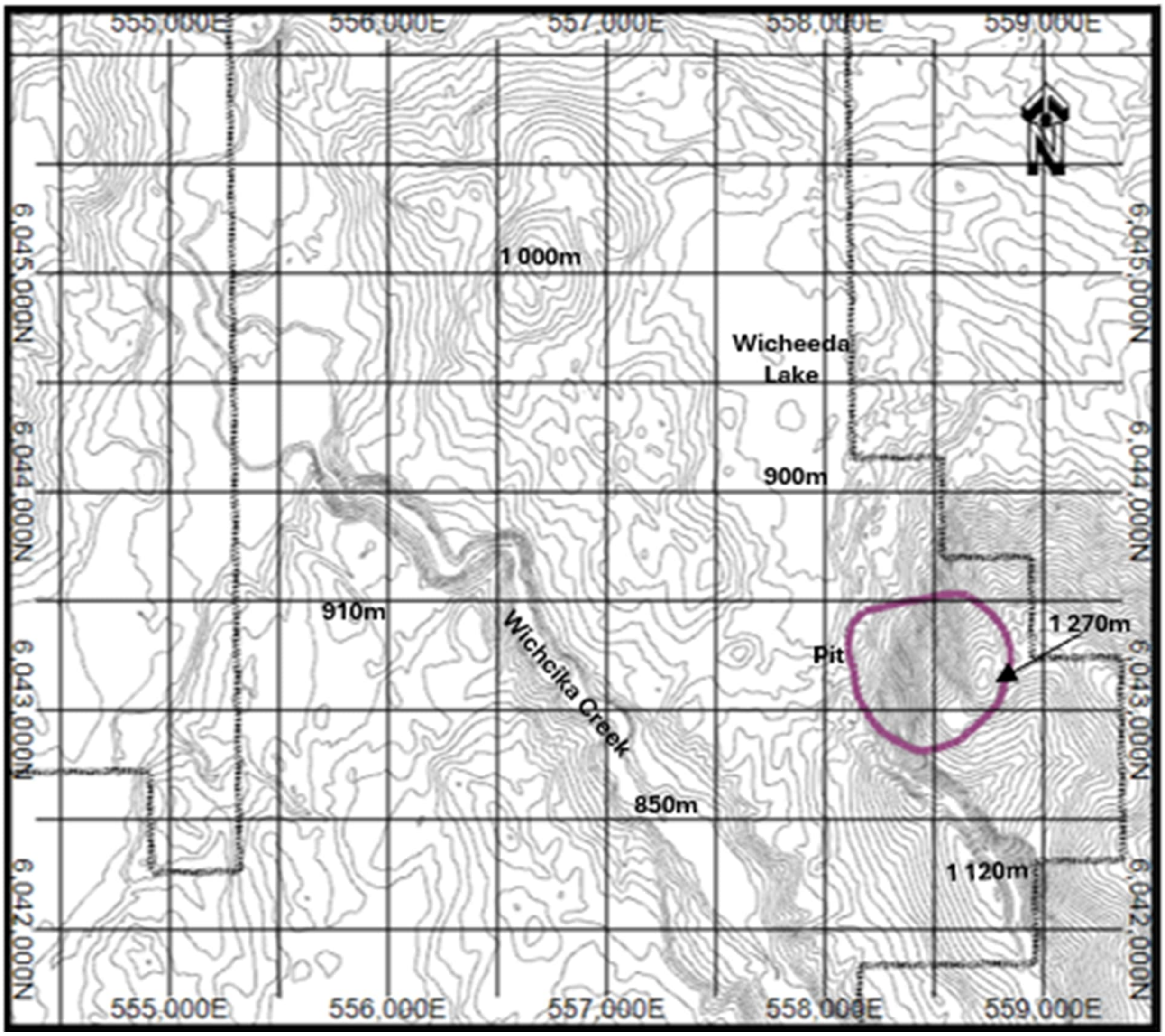
Source: SRK, 2025

Figure 15-6: Rock Types in Minerals Zones

15.3.3 *Topography*

The topography of the Wicheeda property consists of several small peaks on the east and south sides of the property, where the potential pit will be located, plus small hills elsewhere. Wicheeda Creek passes through the middle of the property from the southeast to the northwest, while Wicheeda Lake is located in the northeast of the property. The surface elevation of the pit area varies from 1,270 masl on the east side of the pit to 940 masl on the west.

Figure 15-7 shows a general view of the Wicheeda project topography, with notable elevations and waterways labeled.



Source: SRK, 2025

Figure 15-7: General View of the Topography of Wicheeda Project in the Pit Area

15.3.4 Pit Slope Criteria

There are eight slope design sectors in Wicheeda pit (Figure 15-3). The SRK mining team used the provided inter ramp angles (IRA) together with the haulage road configuration, geotechnical berm requirements, and other design elements, to calculate overall slope angles (OSA). These overall slope angles are used for the pit optimization and analysis. Table 15-6 lists the pit slope design criteria.

Table 15-6: Overall Pit Slope Angles for Wicheeda Project

#	Domain Name	Pit wall azimuth		IRA (degrees)	OSA (degrees)
1	SW Sector	180	260	41	40
2	West	260	300	44	42
3	Northwest	300	360	47	43
4	North	360	400	44	40
5	NE Sector	40	100	41	38
6	East	100	120	44	40
7	SE Sector	120	160	47	43
8	South	160	180	44	40

Source: SRK, 2025

15.3.5 *Processing Method and Recoveries*

The processing methods to produce precipitate products consist of two main stages:

- Stage 1 - Flotation separation methods to produce a concentrate of blended REOs
- Stage 2 - Hydrometallurgical processes to produce a high-grade mixed REE precipitate product

Recovery of REE by flotation varies based on the TREO grade fed to the flotation concentrator. Equation 1 shows the flotation recovery provided by HATCH. TREO grades above 3% are capped at 82.4% recovery.

$$TREO\ Recovery\ (Flotation) = -11.183 * TREO\hat{}2 + 67.831 * TREO - 20.421$$

Equation 1 Flotation Recovery Formula (for Treo<3%)

To calculate the recoveries of individual REEs, the factors listed in Table 15-7 are used. This table provides the factors for both stages of processing flotation and hydrometallurgy. To calculate the final recovery of each REE the overall recovery of TREO is the product of the factors provided in Table 15-7 multiplied by the TREO recovery from Equation 1.

Table 15-7: Individual REE Recovery Factors

Element	Flotation	Hydrometallurgy
La2O3	101.40%	0.00%
CeO2	100.00%	0.50%
Pr6O11	99.50%	93.20%
Nd2O3	99.60%	93.50%
Sm2O3	95.70%	90.80%
Eu2O3*	91.70%	88.30%
Gd2O3	88.50%	84.60%
Tb4O7*	73.40%	80.50%
Dy2O3*	63.60%	73.50%
Ho2O3*	48.30%	71.90%
Y2O3	49.50%	75.50%
Er2O3*	38.30%	68.70%
Tm2O3*	38.90%	88.10%
Yb2O3	12.30%	72.60%
Lu2O3	10.30%	40.40%

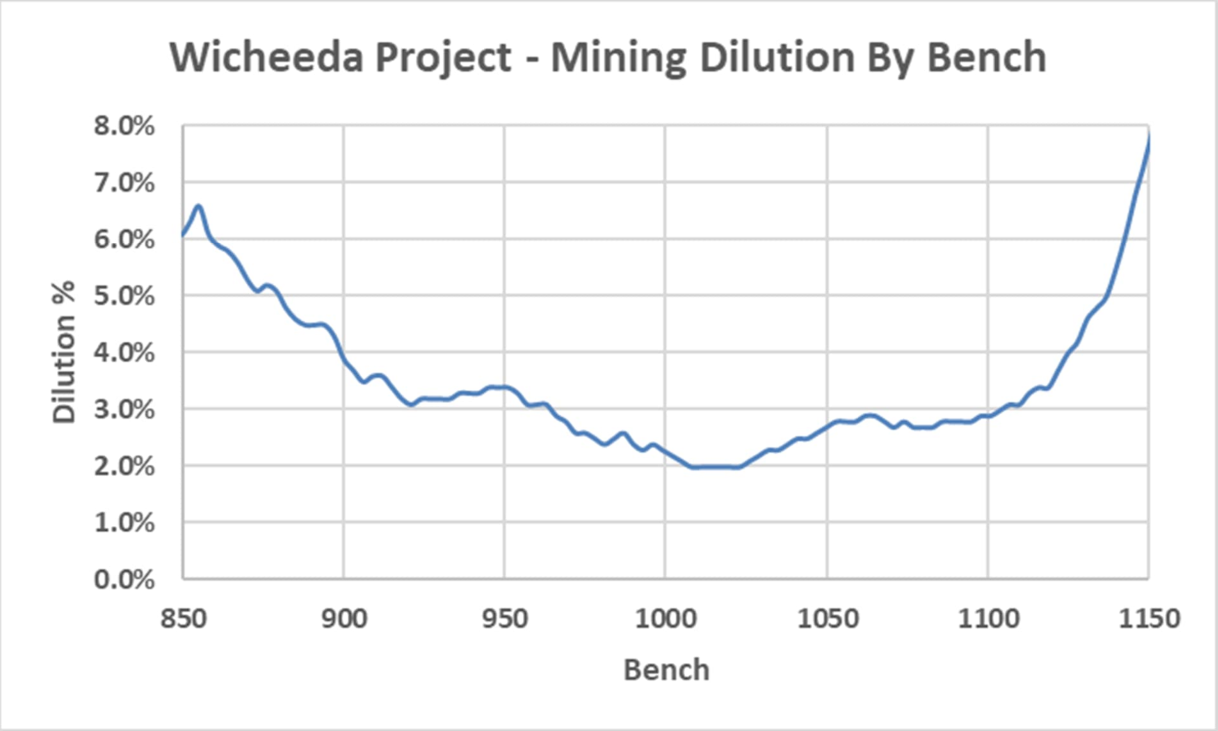
Source: SRK, 2025

15.3.6 Off-Site Costs

The offsite costs for transporting the final product (mixed rare earth concentrate - MREC) and all related expenses are estimated to be a total of US$87.76 per tonne of MREC. The trucking cost for transporting the flotation concentrate from the mine to the hydrometallurgical facility is included in the hydrometallurgical operating costs.

15.3.7 Mining Dilution

Mining dilution is influenced by factors such as commodity price, costs (cut-off grade), the scale of operation, and the shape of the orebody. SRK has calculated the dilution for each block. Dilution is estimated to be 3.1% overall for blocks above the cut-off grade (Section 15.4.2). Figure 15-8 shows the variation of the mining dilution by bench elevation in the final pit.



Source: SRK, 2025

Figure 15-8: Dilution in the Final Pit by Bench

Mineral reserve diluted grades are calculated using the variable dilution factors and the diluting grades for each block.

15.3.8 Mining and Processing Operating Cost Inputs

Mining and filtered tailings operating costs were estimated by SRK, while processing operating costs, offsite costs, and general and administration costs were provided by HATCH.

SRK estimated mining operating costs using a base cost of $5.00 per tonne of material mined and a reference bench elevation of 1050 m – the elevation of the main pit exit. A mining cost adjustment factor (MCAF) was applied, with an additional $0.02 per tonne mined for every 6-meter bench below the reference bench and $0.015 per tonne mined for every 6-meter bench above the reference bench.

The optimization model considers variable processing operating costs based on the TREO grade and the final product (depending on the stage of beneficiation). For instance, for precipitate production, the total processing cost varies by the tonnage of concentrate, which in turn is a function of the feed grade.

Table 15-8 summarizes the operating costs used in the pit optimization assuming a 1.8 Mtpa milling rate and an average feed grade of 2.4% TREO.

Table 15-8: Operating Costs Used in Pit Optimization

Items	Units	Values
Mining cost (Ref bench of 1050, ore and waste)	$/t mined	$5.00
MCAF – 6 m bench above	$/t mined	$0.020
MCAF – 6 m bench below	$/t mined	$0.015
Processing stage 1: Flotation plant	$/t milled	$27.60
Processing stage 2: Hydrometallurgical plant	$/t concentrate	$1,116.40
G&A	$/t milled	$3.67
Filtered tailings operating cost for flotation plant	$/t milled	$6.55

Source: SRK, 2025

15.4 NSR and Cut-off Grades

15.4.1 NSR Calculation

A NSR was used to facilitate pit optimization. In general, the NSR for each block is calculated using Equation 2:

$$\mathbf{NSR} = \sum_{k=0}^{15}(grade^{k} * recovery^{k} * price^{k})\text{- selling cost}$$

Equation 2: NSR Calculation

NSR, recoveries, selling costs and concentrator/hydrometallurgical processing costs are all calculated and stored in the mining model.

15.4.2 Cut-off Grade Calculation

A cut-off grade is where two different courses of action can be taken if the grade is below or above that grade. A milling cut-off grade is the minimum grade that a milled product is profitable after considering all milling and related general and administrative costs.

The Wicheeda project has added complexity, as the total operating cost of processing a tonne of ore varies based on the level of beneficiation and the average feed grade, while revenue is based on the feed grade and the level of beneficiation. This is common for REE projects.

The Mining QP developed NSR values for each block using the input parameters defined above and the optimization scenarios described previously. Block values were then used as the basis for mineable reserve selection considering the related processing and G&A costs. As the total operating costs vary for each block, the TREO cut-off grade also varies accordingly.

15.5 Pit Optimization

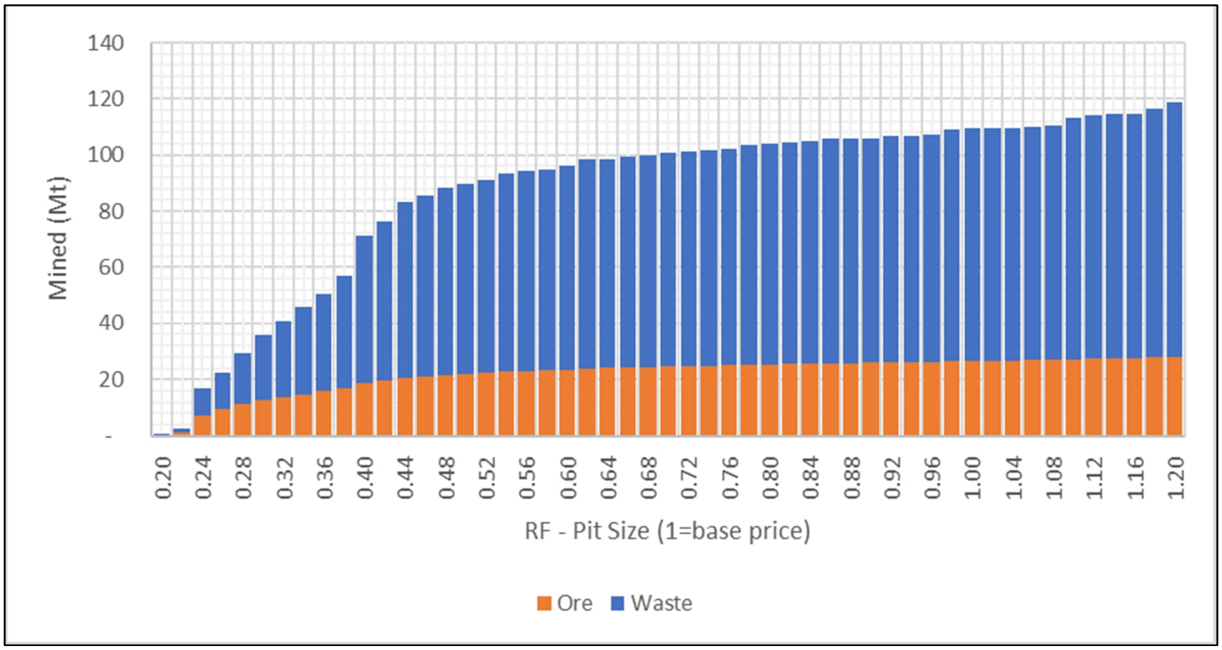
Pit optimization was conducted using Geovia's Whittle™ software and Lerchs-Grossmann and Pseudoflow pit optimization algorithms. The Mining QP used the resource block model along with technical and economic data recommended by other QPs described in previous sections to conduct a pit optimization and strategic mine planning study.

Pit optimization produces nested pit shells using a range of revenue factors. The nested pit shells are used for strategic mine planning economic analysis, and they assist with mine sequencing and production scheduling.

Pit shells were generated based on revenue factors ranging from 0.2 to 1.2 in 0.02 increments, to produce a total of 51 pit shells per run.

15.5.1 Pit Optimization Results

Figure 15-9, shows the tonnages of ore at various revenue factors. At the base case revenue factor (RF 1.00), 26.4 Mt of ore would be mined at an average grade of 2.37% TREO.



Source: SRK, 2025

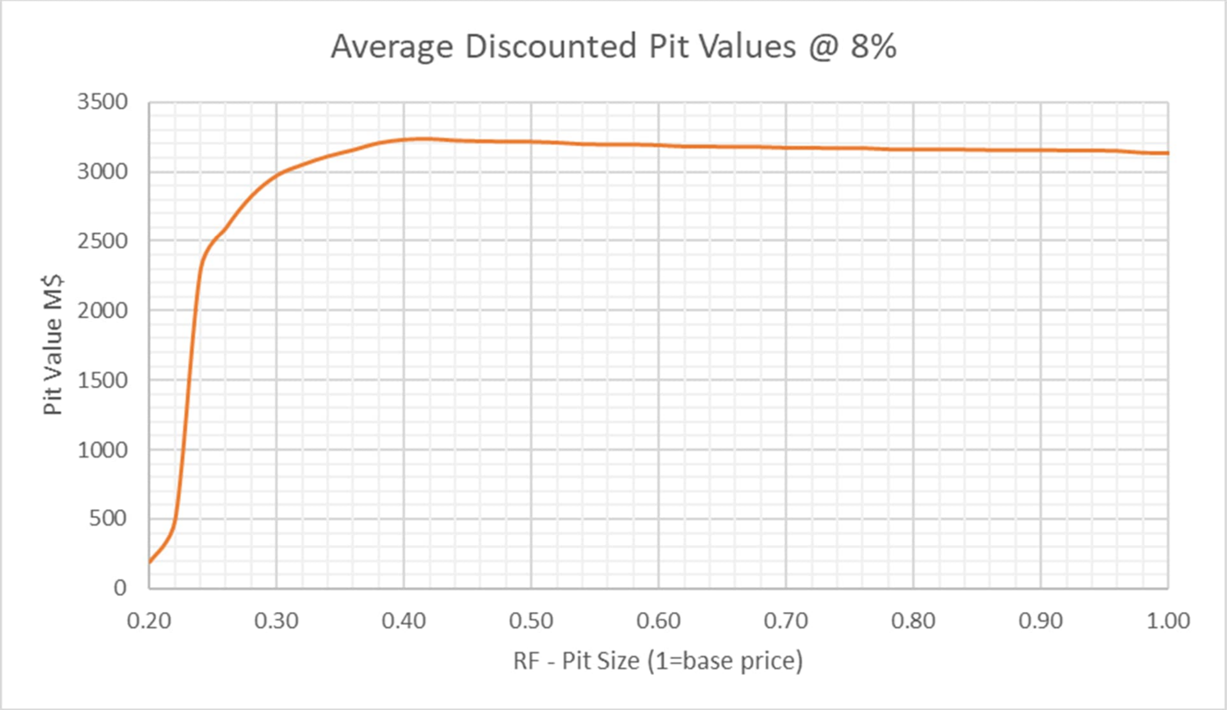
Figure 15-9: Pit optimization results

15.5.2 Ultimate Pit Selection

Pit values (before Capex and tax) are assessed using a range of parameters listed below:

- Discount rate of 8%
- Milling rate of 1.8 Mtpa with one year ramp-up period
- Maximum mining capacity of 9 Mtpa
- Minimum mining width of 50 m
- Considering two major phases for mine sequencing.

Figure 15-10 shows the average estimation of discounted pit value across a range of revenue factors. Pit value increases quickly and peaks at RF 0.42, remaining relatively flat as revenue factors increase. The RF 1.00 pit represents 97% of the maximum value pit.



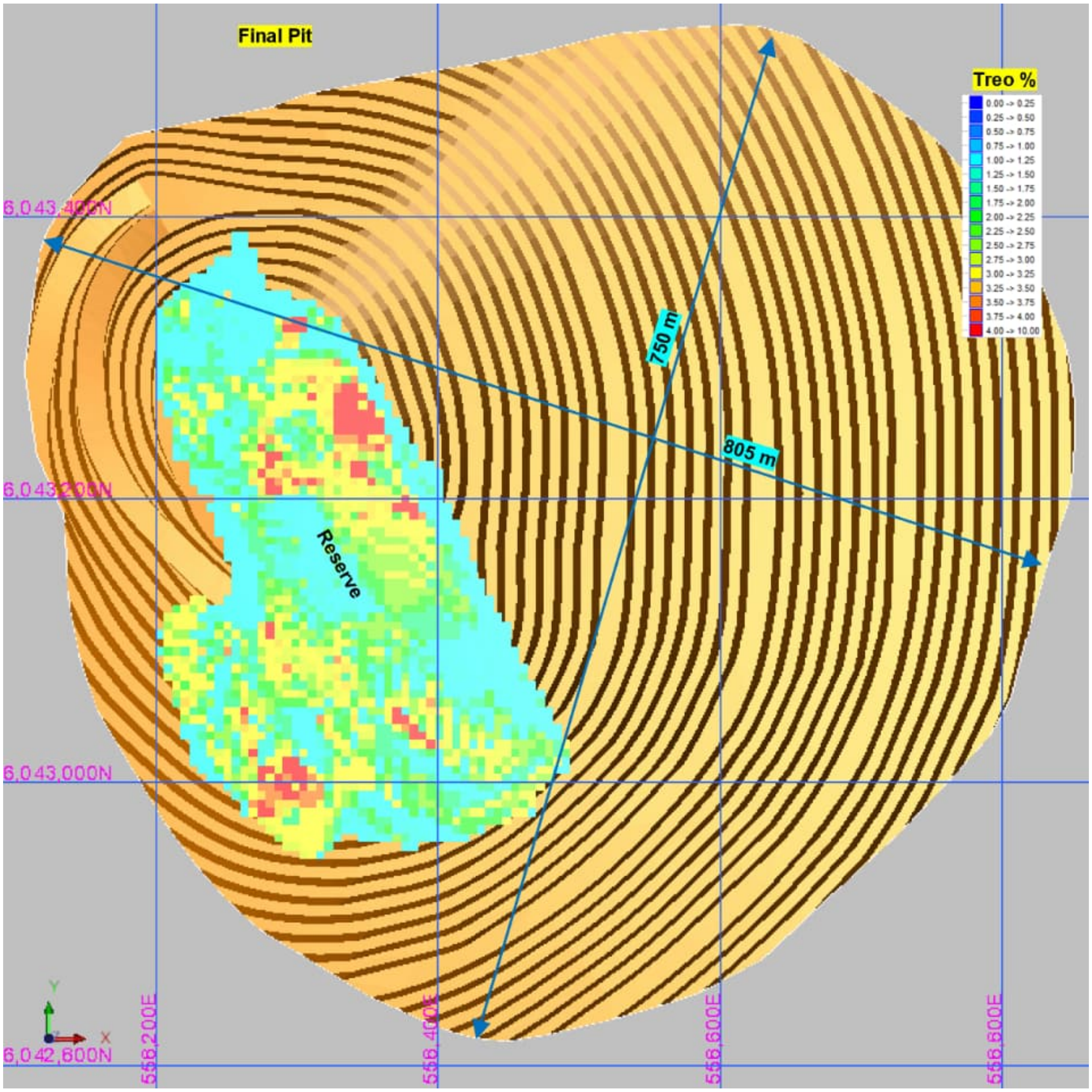
Source: SRK, 2025

Figure 15-10: Discounted Pit Value Versus Revenue Factor (Pit Size)

The flatness of the pit value curve allows for great optionality in pit shell selection to choose a longer mine life pit shell without sacrificing significant discounted value or adding risk of operation. With this consideration, and in consultation with DM, the Mining QP selected the RF 1.00 pit shell to guide the final pit design.

15.6 Ultimate Reserve Pit Design

SRK used the pit shell selected above to design the final pit. Figure 15-11 shows the final reserve pit. The pit is 750 by 805 m wide with maximum depth of 450 m. Details on the pit design can be found in Section 16.



Source: SRK, 2025

Figure 15-11: Final Reserve Pit

15.7 Mineral Reserve Statement

The mineral reserve estimate for the Wicheeda Rare Earth Element Project has been prepared as part of the 2025 PFS in accordance with the CIM Definition Standards adopted May 2014.

The mineral reserves were derived from the mineral resource block model presented in Section 14 of this report. The mineral reserves respective of the open pit are based on Measured and Indicated mineral resources that have been identified as being economically extractable and which incorporate mining losses and mining waste dilution. The mineral reserves include 26.3 Mt of mineable ore from one open pit at an average grade of 2.37% TREO. The mineral reserve includes variable mining dilution, and it is calculated after 1% ore loss.

A summary of the surface mineable mineral reserves by rock type and reserve classification is shown in Table 15-9.

Table 15-9 Summary of the Mineral Reserves as of February 28, 2025

Mineral Reserve	Rock Type	Ore	TREO	Pr6O11	Nd2O3	Tb4O7	Dy2O3
		kTonnes	%	ppm	ppm	ppm	ppm
Proven	Dolomite Carbonatite	5,300	2.96	1,152	3,134	12	35
	Xenolithic Carbonatite	260	1.71	690	2,031	11	37
	Syenite	40	1.44	574	1,658	11	39
	Limestone	10	2.01	858	2,359	12	40
	Total	**5,610**	**2.89**	**1,125**	**3,070**	**12**	**35**
Probable	Dolomite Carbonatite	12,020	2.86	1,120	3,067	12	34
	Xenolithic Carbonatite	7,810	1.38	569	1,674	9	29
	Syenite	760	1.20	482	1,427	8	26
	Limestone	140	1.38	558	1,585	10	38
	Total	**20,730**	**2.23**	**886**	**2,472**	**10**	**32**
Total	Dolomite Carbonatite	17,320	2.89	1,130	3,087	12	34
	Xenolithic Carbonatite	8,070	1.39	573	1,686	9	29
	Syenite	800	1.21	487	1,439	8	27
	Limestone	150	1.42	579	1,639	10	38
	Total	**26,340**	**2.37**	**937**	**2,600**	**11**	**32**

Source: SRK, 2025

Notes:

- The effective date of the Mineral Reserve is February 28, 2025.
- Mineral Reserves are calculated using the values derived from all REEs present in the deposit. Individual REO pricing provided by Adamas Intelligence was used for pit optimization. The key REO pricing is as follows:
  - NdPr Oxide 132.7 $/kg REO
  - Tb4O7 1362.8 $/kg REO
  - Dy2O3 442.5 $/kg REO
- Mineral Reserves are defined within the final pit design guided by pit shells derived from the optimization software, GEOVIA Whittle™

- Cut-off grade is based on the value factors generated in each block. The revenue and related costs vary based on the composition of different elements in each block. Value of a block is the revenue generated in that block minus the related processing and G&A operating costs.
- The base mining costs are assumed to be $5.00/t. The mining costs vary based by the bench and depth of the pit. The average mining costs for the life of mine is calculated to be $5.26/t mined.
- Processing costs consist of flotation concentrator cost at the mine site and a hydrometallurgical plant that is off the property. The operating cost of the flotation concentrator is $27.60/t milled and the hydrometallurgical plant operating cost is $1,164.4/t of concentrate treated.
- General and administration costs of the mine site is $3.67/t for ore milled.
- Tailings cost is $6.55/t of ore.
- Off-site cost (transportation) is $87.76/t of precipitate products produced.
- Processing recoveries are calculated as follows:
    - Flotation recovery for TREO = -11.183*TREO^2 + 67.831*TREO - 20.42194.0%. For ore above 3% TREO, the flotation recovery is set to 82.4%. For grade less than 0.32% TREO, the flotation recovery is set to 0.0%.
    - Flotation recovery for each REE is calculated by multiplying the TREO recovery by that element's recovery factor. For example, the factors for Pr, Nd, Tb, and Dy are 0.995, 0.996, 0.734, and 0.636, respectively.
    - Similarly, hydrometallurgical recoveries are assigned for each REE, which for Pr, Nd, Tb, and Dy are 93.2%, 93.5%, 80.2%, and 73.4%, respectively.
- A 95% payability has been applied to the final hydrometallurgical product.
- Mining dilution varies based on the mining zone. The average mining dilution is calculated to be 2.9%, for the ore delivered to the mill. Tonnages reported as ore includes dilution.
- A 1% ore loss has been applied to the total reserve in each bench.
- Figures are rounded to the appropriate level of precision for the reporting of mineral reserves. Due to rounding, some columns or rows may not sum as shown.
- The overall strip ratio (the amount of waste mined for each tonne of ore) is 3.21 (W:O).
- The mineral reserve is stated as diluted dry metric tonnes.
- The mine plan underpinning the mineral reserves has been prepared by SRK Consulting (Canada) Inc.

Note that the average REO price cited in the reserve statement ($161.77/kg) differs from the long term Upside Case pricing in Section 19. This is due to the distribution of REE in the final reserve pit differing from that used by Adamas; though the individual pricing of REO is the same.

The QP, Dr. Anoush Ebrahimi, does not know of any legal, political, environmental, or other risks that could materially affect the potential development of the mineral reserves. He personally inspected the subject project on October 26, 2021.

16. Mining Methods

16.1 Introduction

The Wicheeda Project is designed as a conventional open pit truck and shovel operation. Mining occurs on 6 m benches. Ore is hauled to a crusher close to the pit rim, and crushed ore is conveyed to the nearby flotation concentrator.

Waste rock is mined and hauled to the waste storage facility (WSF) located adjacent to the pit; although, a small volume of waste rock is utilized in the construction of water management infrastructure.

It is assumed that the Wicheeda mine is owner-operated, with contractor engagement for delivery of explosives to the hole.

16.2 Mine Design

16.2.1 Bench Geometry Inputs

Bench geometry inputs are based on geotechnical recommendations, depending on the different geotechnical domains. Bench geometry inputs are listed in Table 16-1.

Table 16-1: Bench Geometry Inputs

#	Domain Name	Inter-ramp Angles (degrees)	BFA (degrees)	Bench height (m)	Pit depth (m)	Bench width (m)	Geotech Berm Width (m)	Geotech Berm #	Adjusted Overall slope angle
1	SW Sector	41	65	12	440	8	15	0	40
2	West	44	70	12	440	8	15	0	42
3	Northwest	47	75	12	440	8	15	0	43
4	North	44	70	12	440	8	15	4	40
5	NE Sector	41	65	12	440	8	15	4	38
6	East	44	70	12	440	8	15	4	40
7	SE Sector	47	75	12	440	8	15	4	43
8	South	44	70	12	440	8	15	4	40

Source: SRK, 2025

16.2.2 Haul Road Widths and Gradients

Ramp parameters are listed in Table 16-2. Double lane roads are used throughout the mining area, while single lane roads are only for accessing the bottommost benches in the pit.

Table 16-2: Ramp Width Parameters

#	Ramp	Ramp Width (m)	Ramp Gradient
1	Double lane haul roads	20	10%
2	Single lane haul roads	18	12%

Source: SRK, 2025

16.2.3 Ultimate Pit

SRK developed the ultimate pit design from the selected pit shell described in Section 15.5.2. The resulting pit design was presented in Section 15.6 (Figure 15-11).

The ultimate pit has a maximum width of 746 m and a maximum height of 452 m, with the highest and lowest bench elevations at 1262 m and 818 m, respectively.

16.2.4 Pit Phase Designs

The pit design consists of two separate phases to minimize upfront waste stripping costs and to advance the mining of high-grade in the early mine life.

Phase 1 mines out the high-grade area in the center of the deposit while leaving sufficient mining width for Phase 2, deferring waste stripping until later in the mine life.

Table 16-3 summarizes ore and waste rock mined in the Wicheeda pit by phase.

Table 16-3 Wicheeda Pit Phases

Phases	Ore		Waste	Total	SR
	Mt	TREO %	Mt	Mt	W:O
1	13.63	2.8%	28.06	41,684,596	2.06
2	13.00	1.9%	56.83	69,819,854	4.38
Total	**26.62**	**2.4%**	**84.89**	**111,504,450**	**3.19**

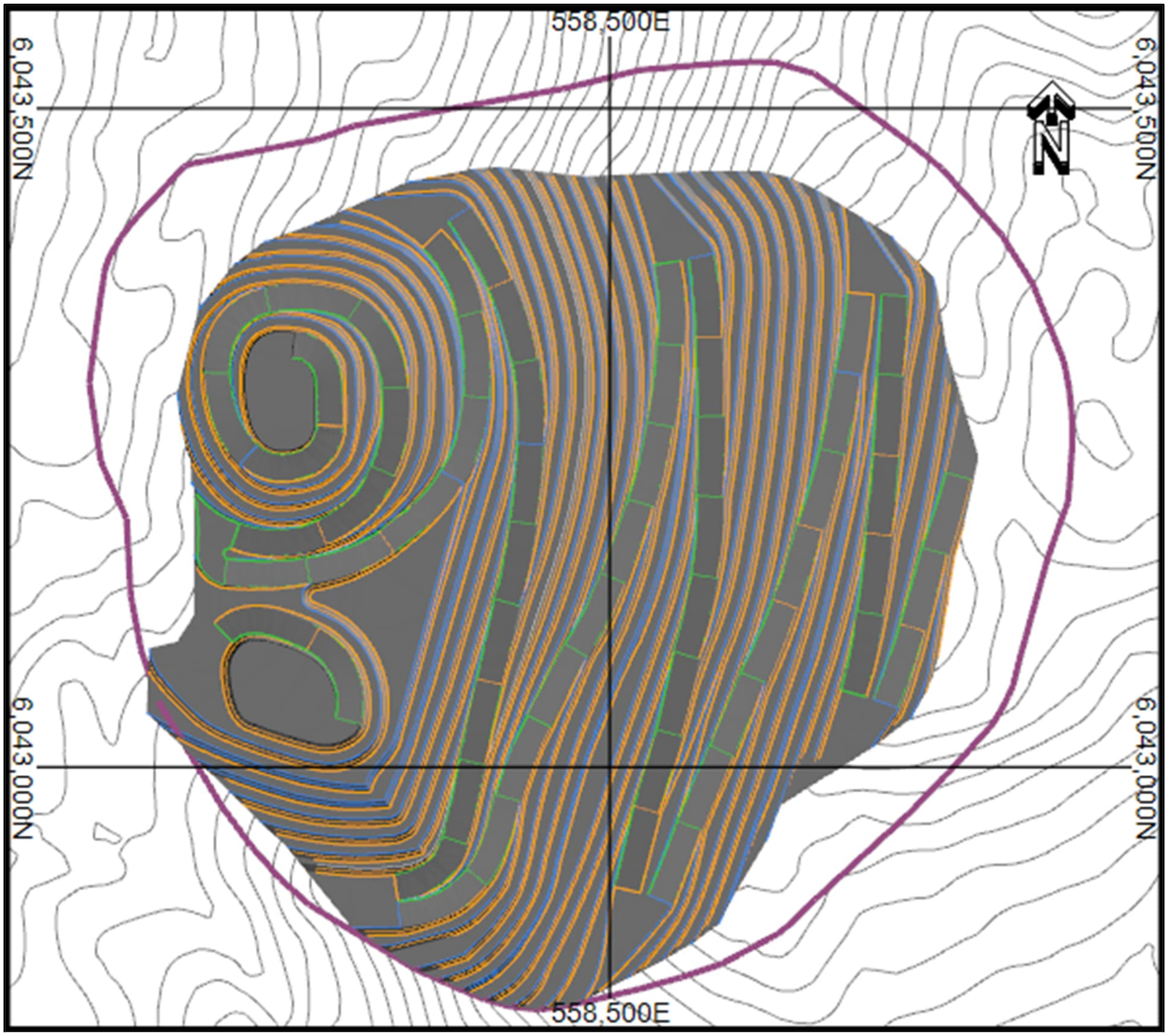
Source: SRK, 2025

16.2.4.1 Phase 1

Phase 1 mines benches from 1250 to 914 m elevation and provides high-grade, mostly dolomite carbonatite ore for the first eight years of mine life. The phase is accessed via a pioneering road which crosses the pit and provides access to the top of the phase. The road is generally cut within the footprint of the pit while the switchbacks are predominantly fill external to the pit. As the phase develops, an interim ramp will be left in the Phase 1 highwall to facilitate Phase 2 mining and haulage. Two pit bottoms are developed at depth accessed by single lane haul roads for the bottom few benches.

Ore is located, near surface, from elevation 1162 m down to the bottom of the phase. Ore will be hauled down to the crusher utilizing the pioneering road. Waste will be hauled to the WSF located adjacent to the pit, accessed via roads which intersect from the pioneering road switchbacks.

Figure 16-1 shows Phase 1 mined out, leaving the interim ramp in its highwall.



Source: SRK, 2025

Figure 16-1: Phase 1 Mined Out

16.2.4.2 *Phase 2*

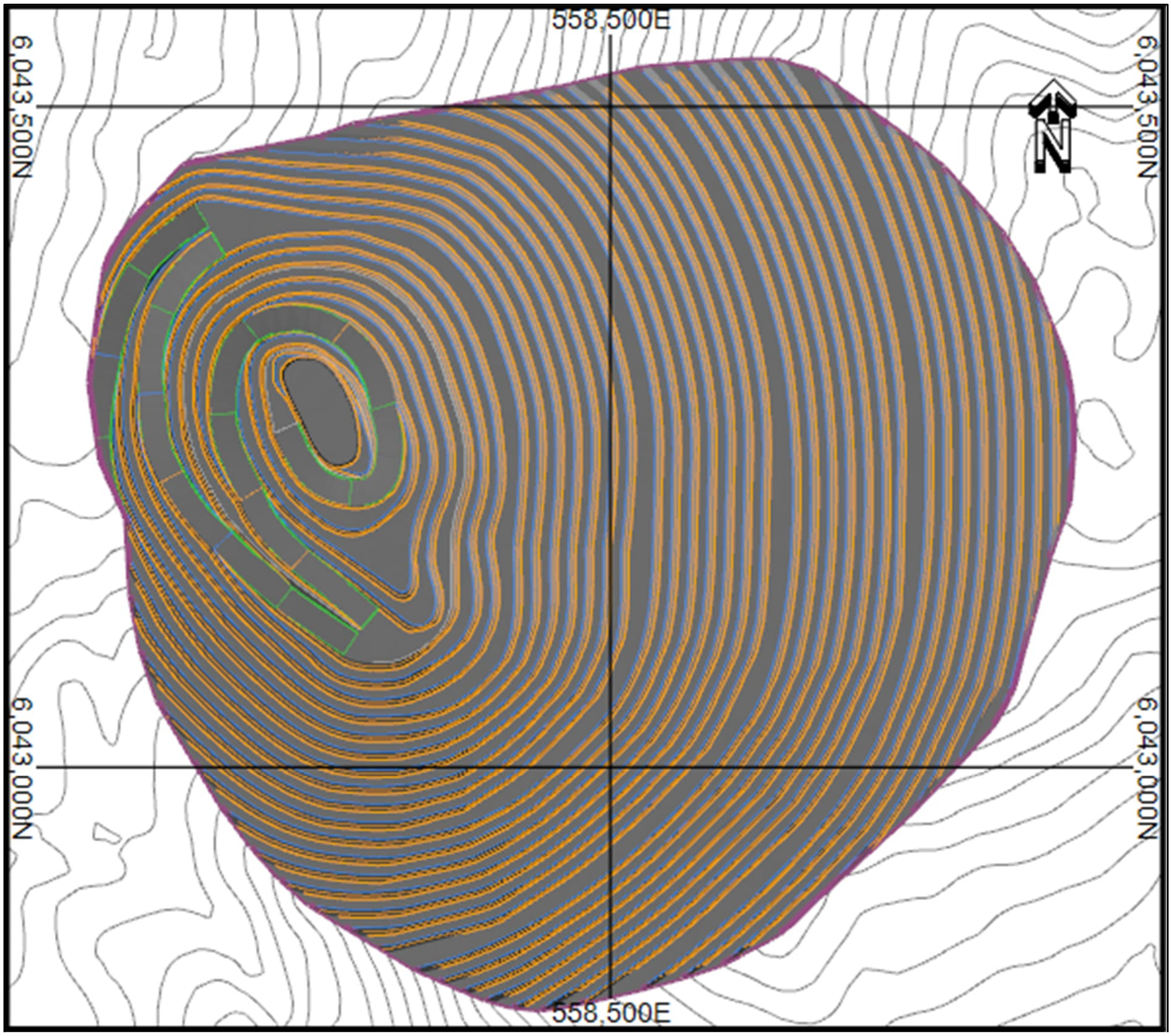
Figure 16-2 shows Phase 2 (the ultimate pit) mined out.

For Phase 2, ore haulage to the crusher will utilize the interim ramp left in the highwall of Phase 1, and waste haulage will be facilitated by similar accesses to Phase 1, tying in at upper elevations, while hauling up the face of the WSF at lower elevations.

At depth, the pit has a single spiral access ramp that continues to a single pit bottom, with one-way road access in the lowest few benches.

Four geotechnical berms will be left in the highwall to adhere to the guidance provided in Section 15.2.

Phase 2 mines from 1262 to 818 m elevation and has a maximum width of 726 m.



Source: SRK, 2025

Figure 16-2: Phase 2 Mined Out

16.3 Waste Storage Facilities

16.3.1 Facility Design

Nearly all waste rock is sent to a waste storage facility located adjacent to the pit, with a small amount of waste used as embankment construction material for the Contact Water Pond (CWP). Overburden and topsoil are temporarily segregated in separate stockpiles located adjacent to the crusher but will be rehandled and utilized in closure and reclamation on the WSF and elsewhere.

A volumetric breakdown of the waste rock destinations is in Table 16-4, which assume a 30% swell factor.

Table 16-4: Waste Destinations

Facility	Waste (Mt)	Volume (M LCMs)
Waste Storage Facility	84.43	41.05
Contact Water Pond Embankment	0.46	0.22
Total	**84.89**	**41.27**

Source: SRK, 2025

The WSF is described here; for details on the Filtered Tailings Storage Facility (FTSF) and CWP, refer to Section 18.

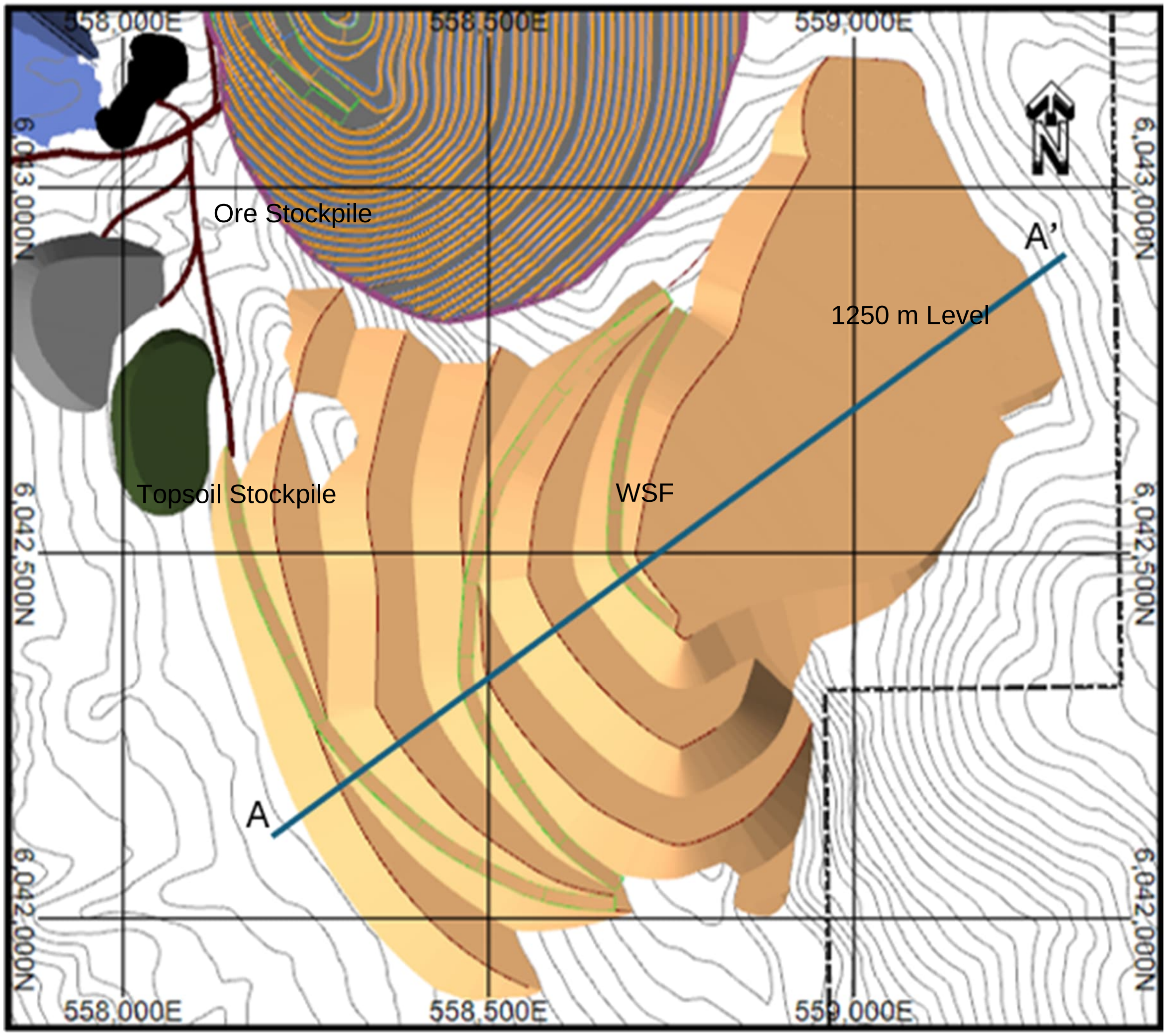
The WSF is located in the valley immediately south of the open pit (Figure 16-3). This location is selected to minimize the haulage cost as well as to minimize the environmental footprint. Face angles of each lift are assumed to be 37° while the WSF is designed to an overall 2.3H:1V slope angle.

The facility will be built in stages in order to minimize haulage distances while adhering to geotechnical guidance (Section 16.3.2). Two lifts at 1250 m and 1200 m will be end-dumped to facilitate early mine-life waste stripping, providing short haulage distances. As both Phase 1 and 2 are mined down, material will be hauled to accesses which tie into the switchbacks on the pioneering or interim ramps located on the southeast side of the pit.

Once the upper lifts are established and mining has progressed to the middle elevations of each phase, bottom-up construction of the WSF will commence in 10 m high lifts, facilitated by a ramp up the face of the facility, until lifts are built up to and merge with the 1250 m elevation end-dump.

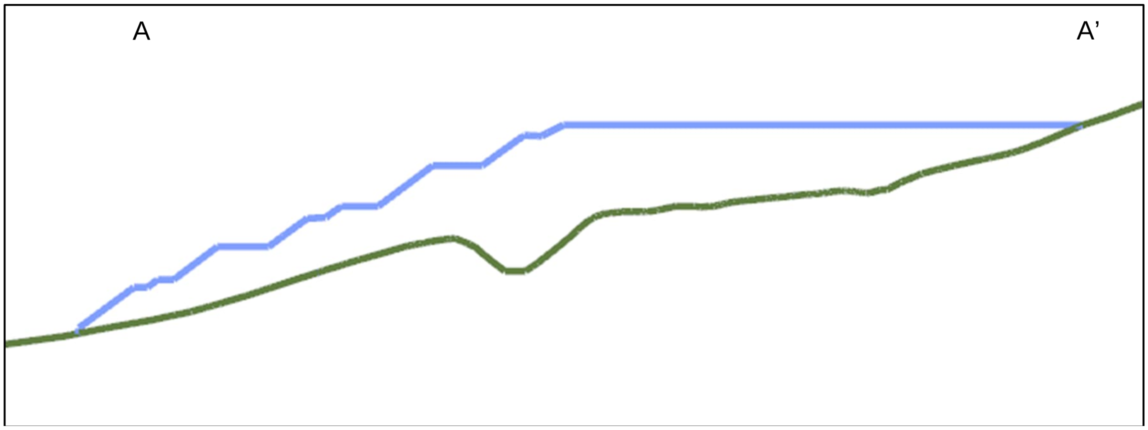
Closure and reclamation activities relating to the WSF include re-sloping and revegetating, adhering to the closure plan outlined in Section 20.

Surface water upstream of the WSF is to be intercepted and directed away from the WSF. Water passing through the WSF will either be captured in the CWP downstream of the WSF or intercepted by a collection ditch at the lower perimeter of the WSF and directed to the CWP. Contact water is to be treated at the CWP before being released. Section 18 provides more details on the site water management plan.



Source: SRK, 2025

Figure 16-3: Ultimate WSF Adjacent to the Pit



Source: SRK, 2025

Figure 16-4: Ultimate WSF Cross-Section

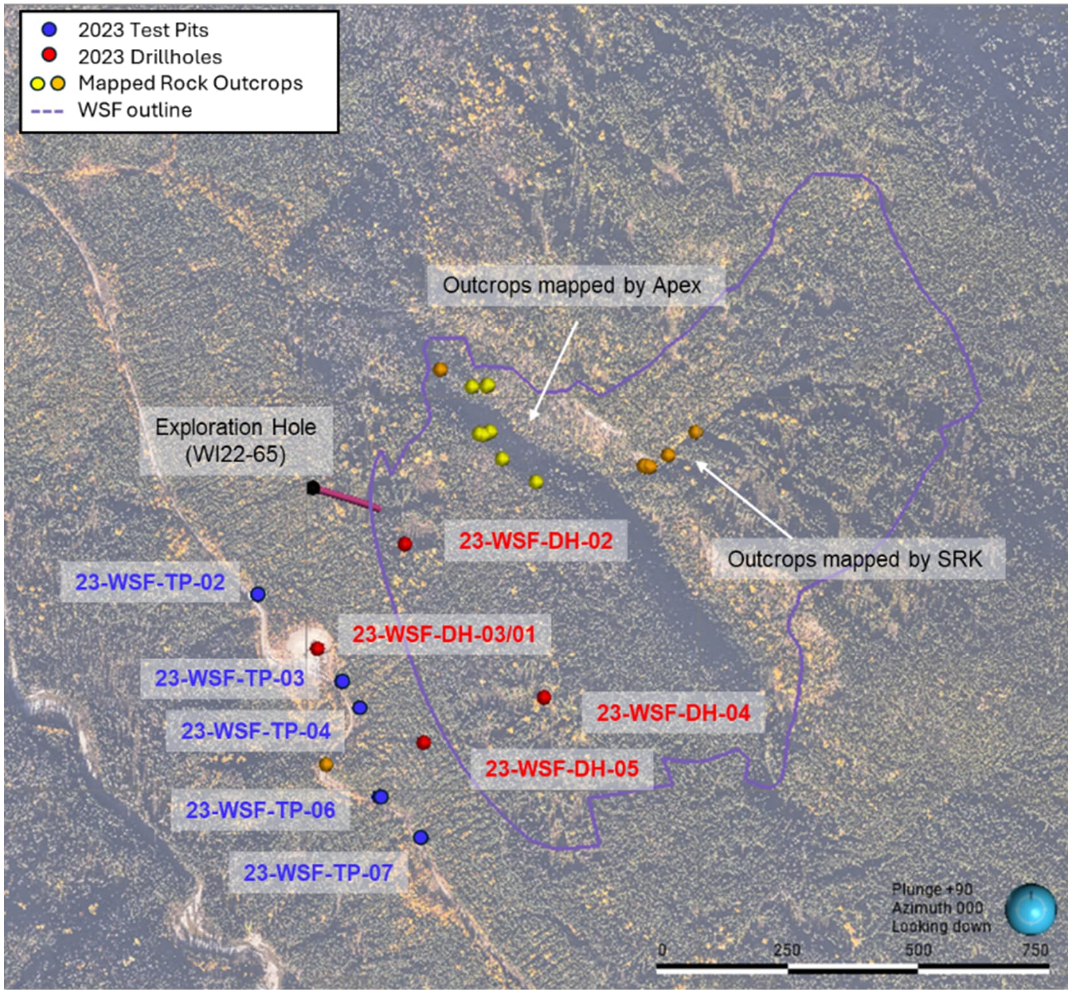
16.3.2 *Waste Storage Facility Geotechnical*

16.3.2.1 Field investigation and Foundation Characterization

In 2023, SRK completed field investigations comprised of five sonic drillholes, five test pits and a terrain/outcrop mapping campaign. In addition, nearby exploration drillholes incorporated into the foundation characterization. A laboratory strength testing program was completed with samples collected from drillholes and test pits.

The field investigations indicated the WSF will be founded on surficial soils overlying bedrock. The surficial soils comprise alluvium generally less than 1.5 m thick and glacial till generally less than 10 m thick. The till is locally up to 28 m in drill hole 23-WSF-DH-03. The alluvium material consists of a silt with little sand and trace gravel, firm to stiff consistency and low plasticity. The till is very dense or hard, non-plastic, and comprises silt with little sand, few gravel and trace cobbles. Groundwater monitoring wells were installed in three drillholes with interpreted water levels at a depth of about 4 m within the till.

The toe of the WSF is to be located on flat to gently sloping topography between 5° and 10°.



Source: SRK, 2025

Figure 16-5: Location Plan Showing 2023 WSF Geotechnical Investigations

16.3.2.2 *Geotechnical Assessment*

A Waste Dump and Stockpile Stability Rating and Hazard Classification (WSRHC) was completed. The WSRHC indicates that the facility is considered a moderate hazard primarily due to the design volume, height of construction and sloping topography.

Design acceptance criteria was established for:

- Physical stability at lower two platforms and overall scale stability
- Set-back from the uppermost erosional scarps along the Wichcika Creek Valley.

Set-back design distances were determined from run-out analyses and empirical relationships for the lower two platforms that are intended to be constructed with bottom-up methods.

Two-dimensional (2D) stability analyses were conducted for the interim and final slope configuration. The analyses evaluated local multi-bench failures and deeper-seated failures along the foundation that could constrain the WSF design and pose a significant hazard. The 2D stability analyses results indicate that the proposed WSF are expected to achieve the minimum design acceptance criteria.

16.3.2.3 *Geotechnical Design*

Geotechnical slope design recommendations are provided in Table 16-5.

Table 16-5: WSF Slope Design Recommendations

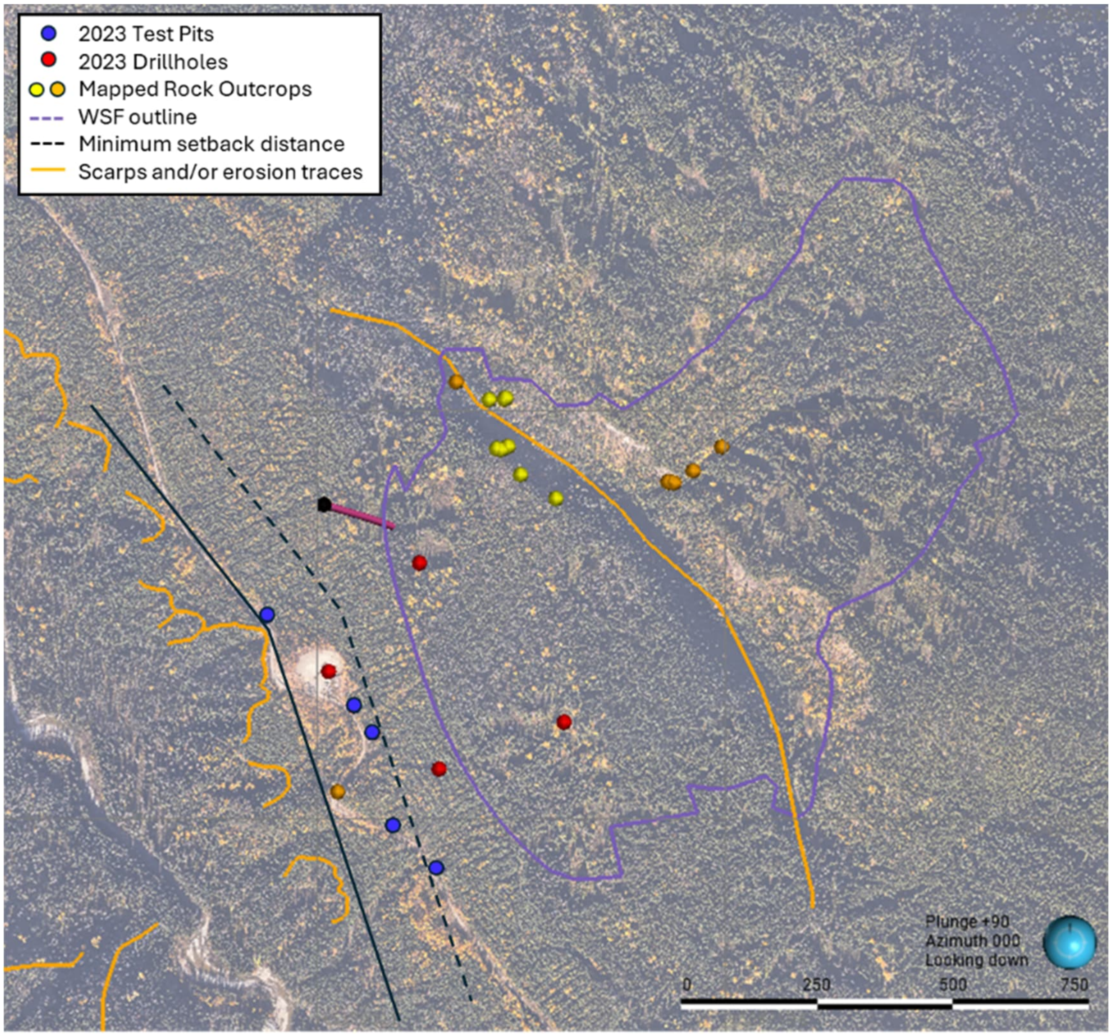
Facility	Face Slope Angle (°)	Maximum Height Lift (m)	Minimum Bench Width (m)	Maximum Overall Slope Angle (°)
WSF	37	50	60	23° (2.3:1)

Source: SRK, 2025

In addition, the following geotechnical design recommendations are required:

- A minimum setback distance of 120 m from the uppermost valley erosion scarps to the toe of the WSF (Figure 16-6.
- The facility will need to be initially constructed over flat to shallow sloping topography.
- Alluvium, soft to firm in composition or saturated, at the toe will need to be removed prior to construction.
- A hybrid construction approach is needed to improve local platform stability. The initial two platforms located at the toe will need to be constructed with bottom-up methods to improve the density of the placed materials. The materials should be placed behind the intended or established platform crest and then pushed utilizing a tracked dozer.
- Platforms away from the toe can be constructed with end-dump methods provided materials are not concentrated and placed in lift heights < 50 m with dumping along contours (i.e., wrap-around).

- Avoid placing overburden, materials with high content of fines, or weak rocks of low durability that may include Dolomite Carbonatite and Syenite in areas that are sensitive to stability.



Source: SRK, 2025

Figure 16-6: WSRF Recommended Setback Behind Areas Along Wichcika Creek

16.4 Stockpiles

16.4.1 Ore Stockpile

An ore stockpile is utilized to smooth total material movement between periods and provide contingency against interruptions in ore production to ensure a constant throughput to the plant.

The majority of the stockpile is established in the early years of mine life and is constrained to lower grade material that not is dolomite carbonatite rock type.

The stockpile is located adjacent to the crusher, has a peak capacity of 0.5 Mt and is fully exhausted in the last year of mine life.

16.4.2 Topsoil and Overburden Stockpiles

Topsoil and overburden are segregated from waste rock and stockpiled in facilities adjacent the plant site and the crusher, totaling 1.0 Mt. This material is re-handled and used for closure and reclamation purposes.

16.5 Mine Scheduling

Mine scheduling was completed in Deswik scheduling software (Interactive Scheduler and Blend modules) while Deswik LHS was utilized to facilitate haulage calculations.

The main driver of scheduling was to forward higher grade TREO material, thereby increasing potential NPV; however, a secondary goal was to smooth haulage distances to reduce costs or spikes in equipment requirements.

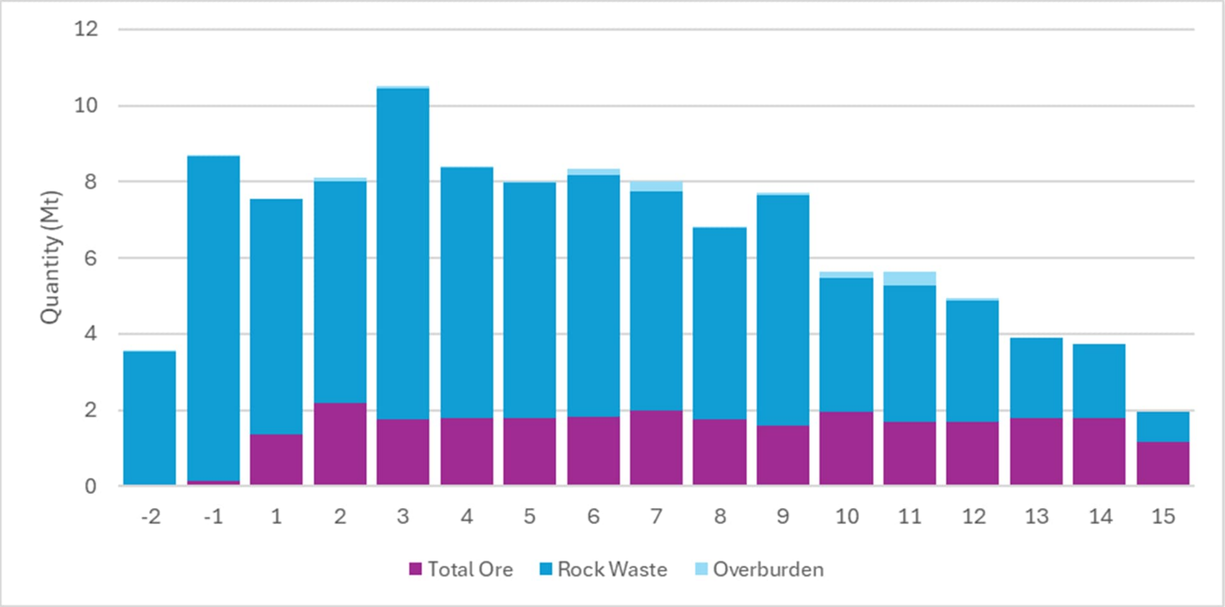
16.5.1 Assumptions and Input Parameters

The following schedule assumptions are made:

- A constant mill feed of 1.8 Mtpa is targeted, with the exception of a ramp up in year 1 targeting 1.5 Mt (83% of steady state production).
- A vertical bench advance rate constraint of 12 benches per annum is set. Consideration is made for extremely small benches at the top of the pit which would likely be dozer pushed instead of mined conventionally.
- 0.46 Mt of waste material will be mined in pre-production to facilitate CWP embankment construction.
- An operational stockpile not exceeding 0.5 Mt is established, consisting of low grade non-DC rock type material.
- The schedule is generated in quarters in pre-production and first two years of production, followed by annual periods until the end of mine life.

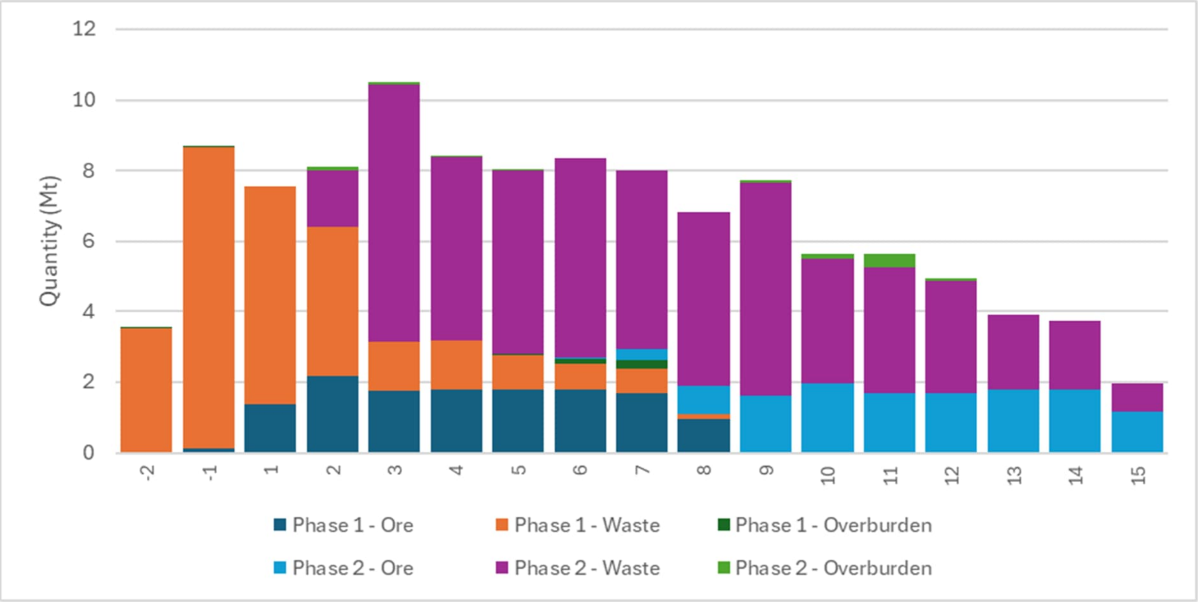
16.5.2 Production Schedule

Figure 16-7 shows total material mined by year, while Figure 16-8 shows total material mined by year and phase.



Source: SRK, 2025

Figure 16-7: Ex-Pit Total Material Movement



Source: SRK, 2025

Figure 16-8: Ex-Pit Total Material Movement by Phase

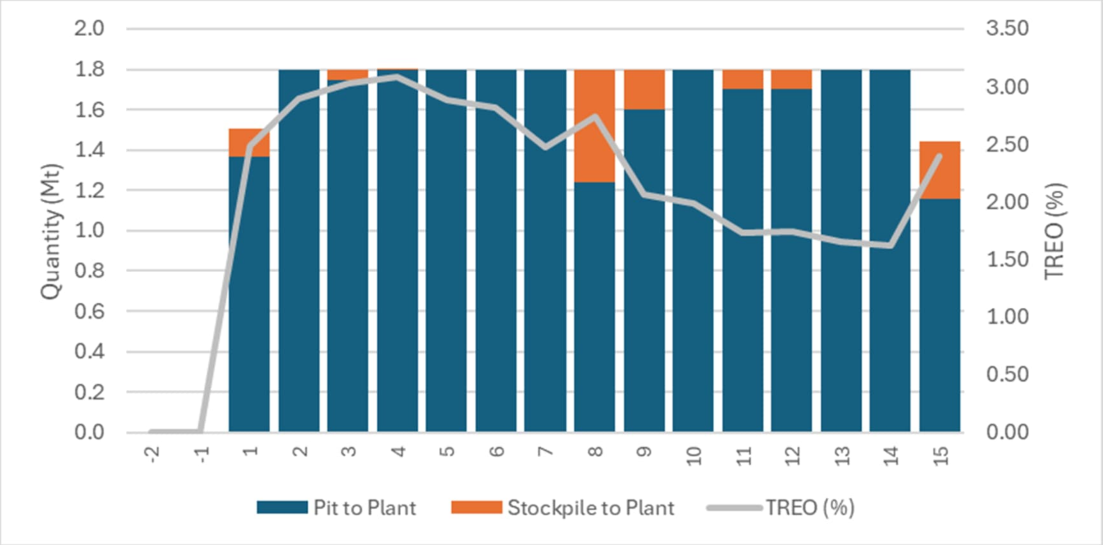
Pre-production mining will be spread over 1.75 years to facilitate waste stripping in Phase 1 to ensure sufficient ore release in Year 1 of the production schedule. Phase 2 commences in Year 2 consisting of low-tonnage benches (waste) constrained by vertical advance limits but continues for another six years of waste removal to ensure adequate ore release as Phase 1 is exhausted in Year 8.

Total material movement peaks in Year 3 at 10.5 Mt when Phase 2 is fully engaged in waste stripping and Phase 1 is still mining ore and waste; however, haul truck cycle times are lessened due to the short haulage opportunities for the Phase 2 waste to the upper lifts of the WSF.

Phase 1 and Phase 2 maintain a vertical separation of at least 100 m during concurrent mining.

16.5.3 *Production*

Production is fed directly from the pit or rehandled from the stockpile and is summarized in Figure 16-8.



Source: SRK, 2025

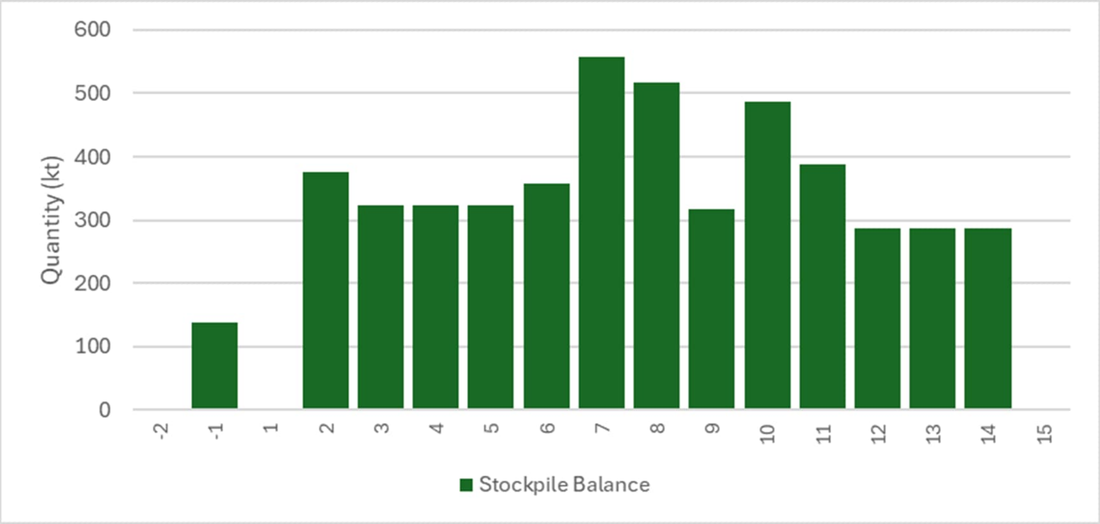
Figure 16-9: Production

Ore is directly fed from Phase 1 in years 1-8; while Phase 2 is in years 9-15 – some material from both phases is occasionally sent to the stockpile throughout the life of mine.

The TREO % in Figure 16-8 shows that the mine plan successfully forwards higher grade ore into the early part of schedule to increase discounted cash flow. The increase in grade in Year 8 is due to stockpile swapping of lower grade ore being mined in that period for higher grade ore residing in the stockpile. The increase in TREO grade in Year 15 however is mostly driven by higher grade DC ore found at depth.

16.5.4 *Stockpile Material Balance*

Figure 16-10 shows the stockpile balance for the life of mine. The operational stockpile allows for smoothing of material handling to reduce spikes in equipment usage and alleviates risk of the mine plan not meeting throughput. The stockpile balance generally sits between 0.3-0.4 Mt which represents 2-3 months of throughput, although it peaks as high as 0.5 Mt in some periods.

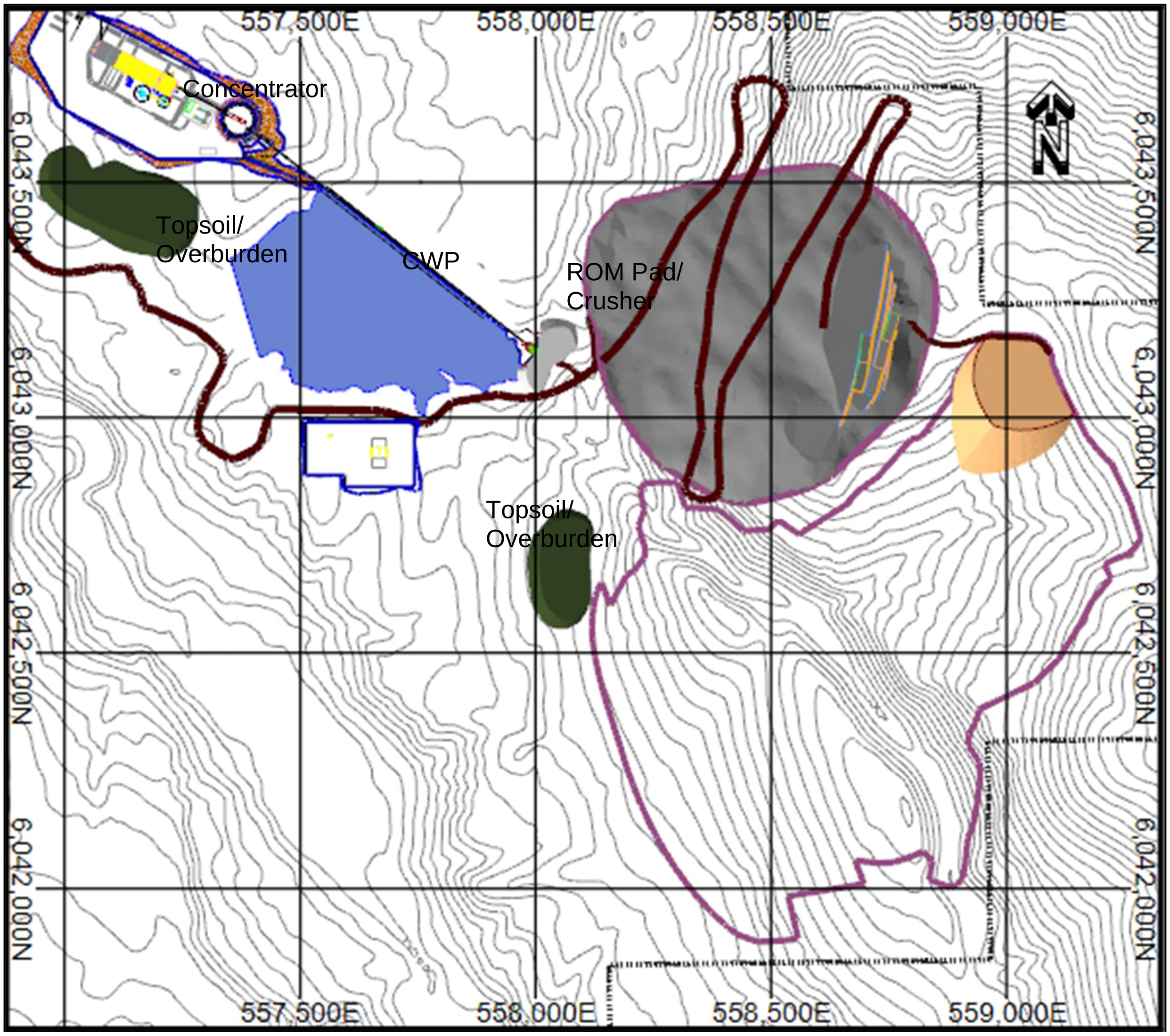


Source: SRK, 2025

Figure 16-10: Stockpile Balance

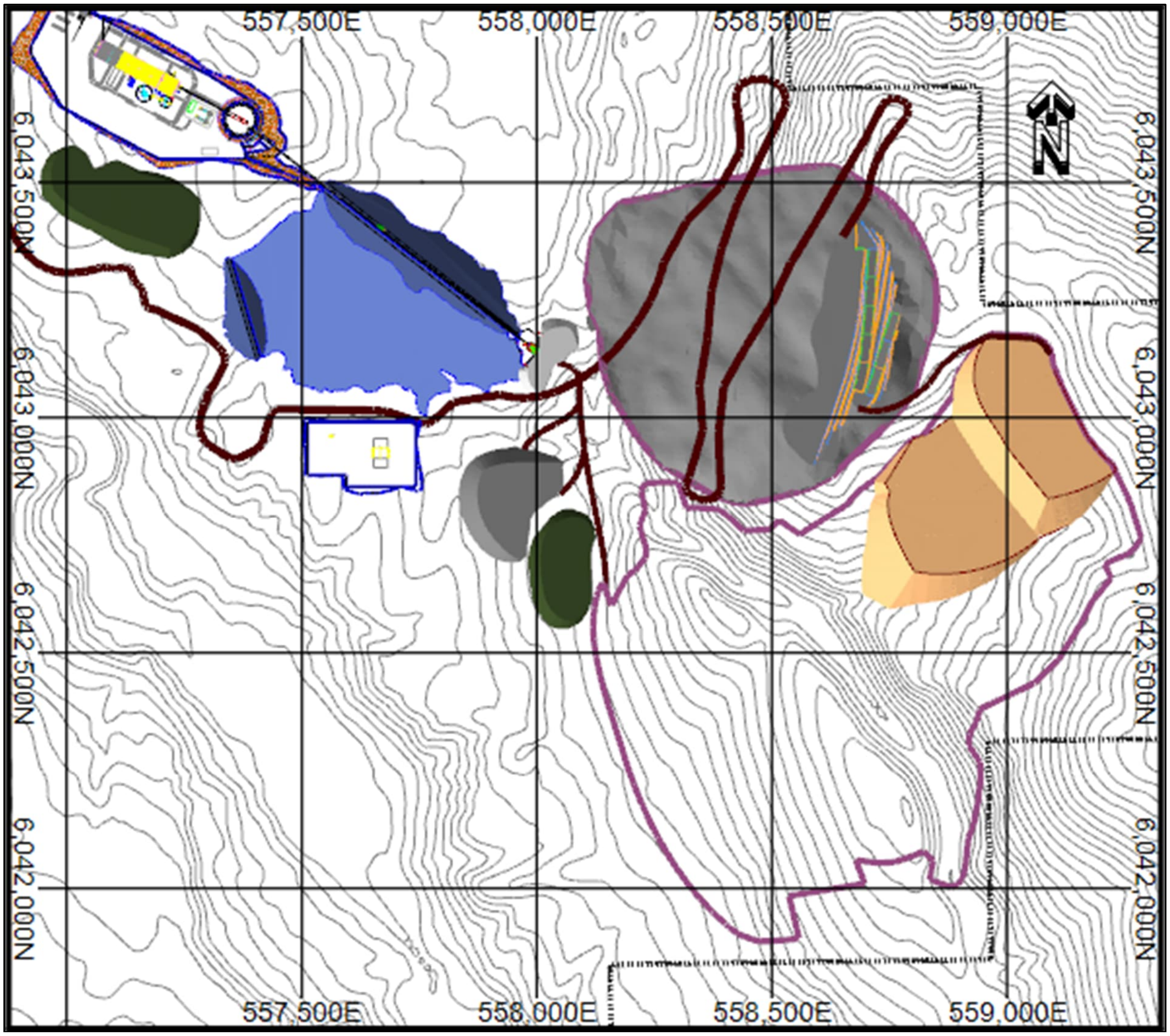
16.5.5 Period Plans

The following are the end of period plans for select points in the LOM production schedule.



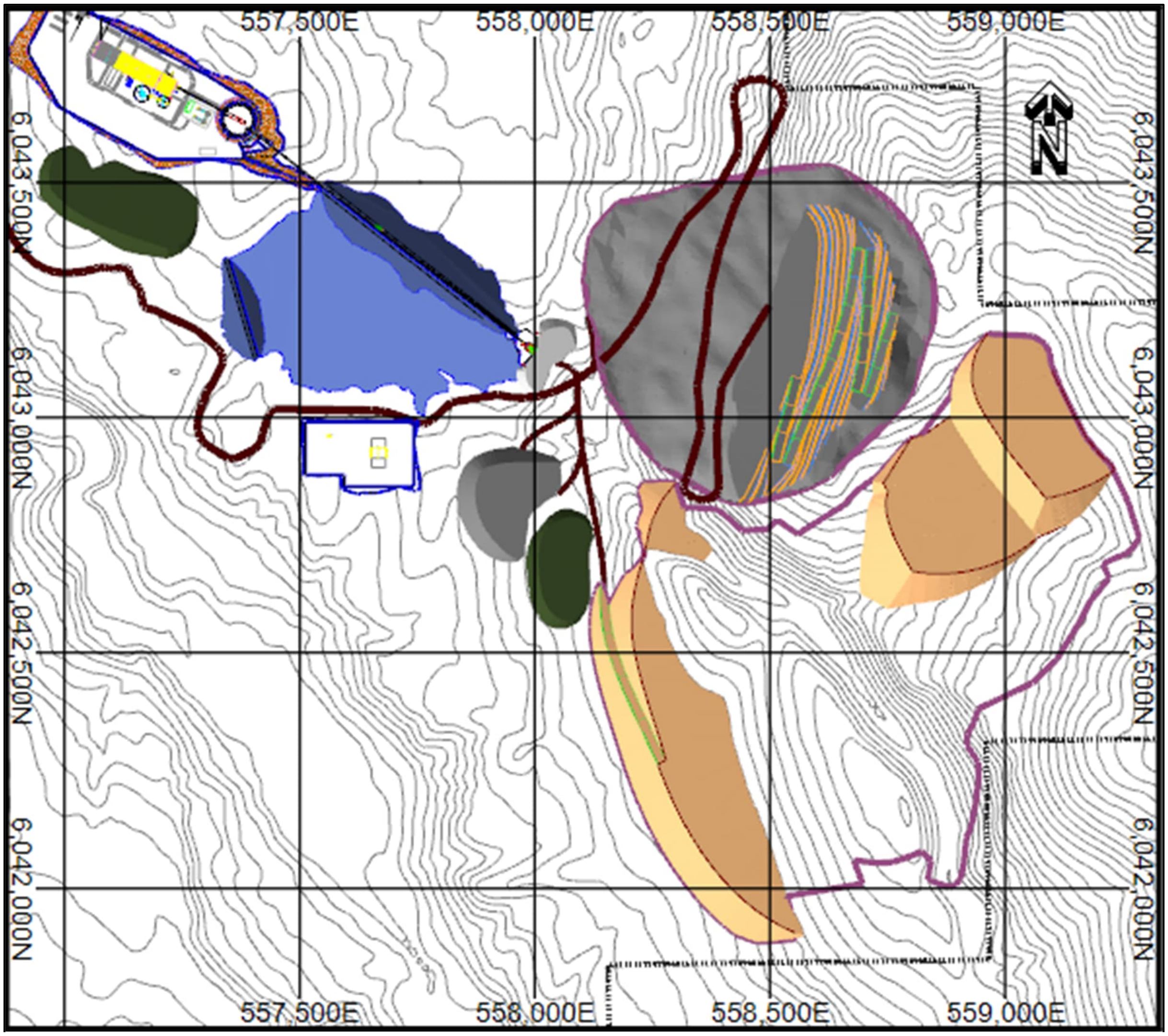
Source: SRK, 2025

Figure 16-11: End of Pre-Production



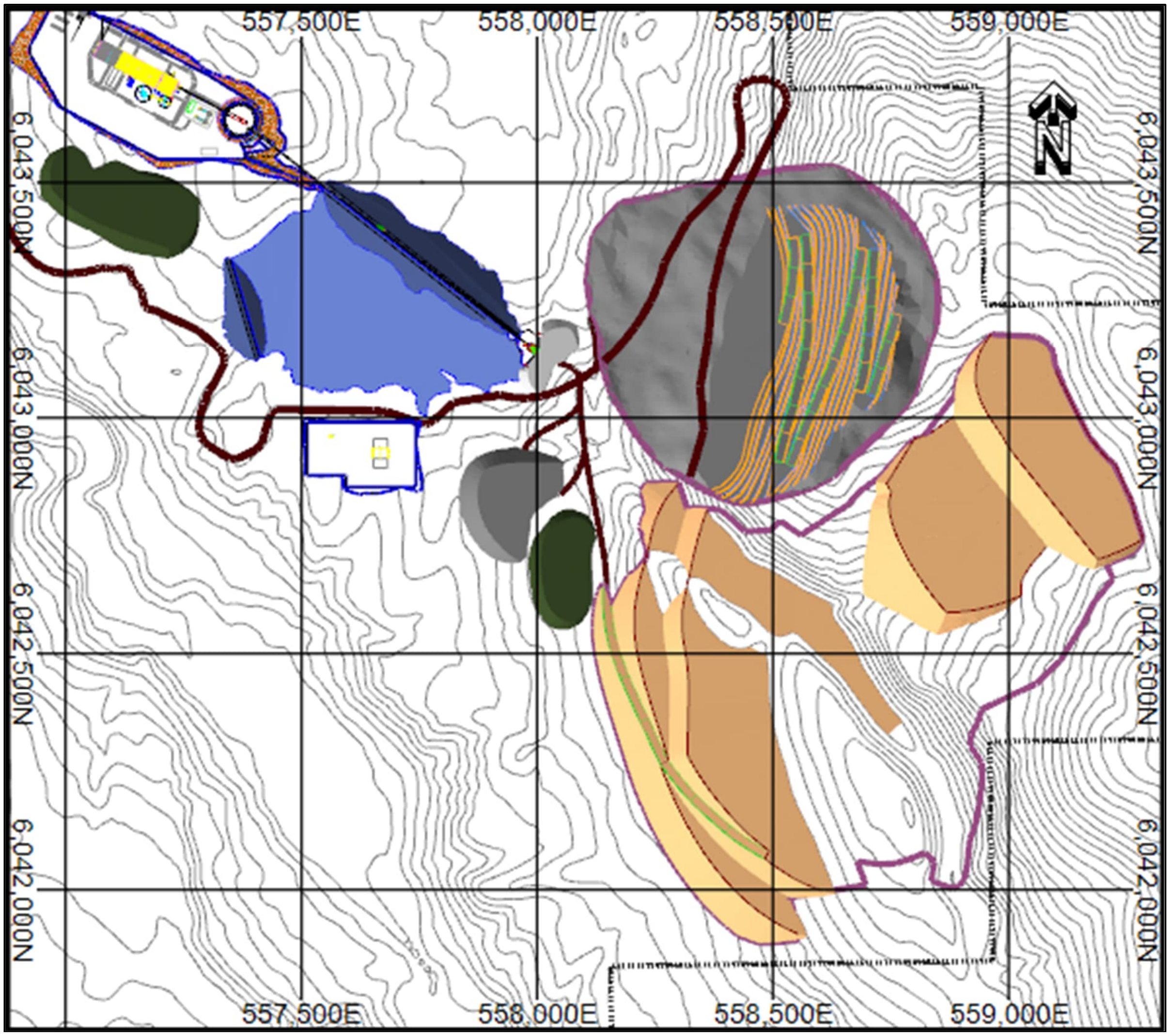
Source: SRK, 2025

Figure 16-12: Year 1



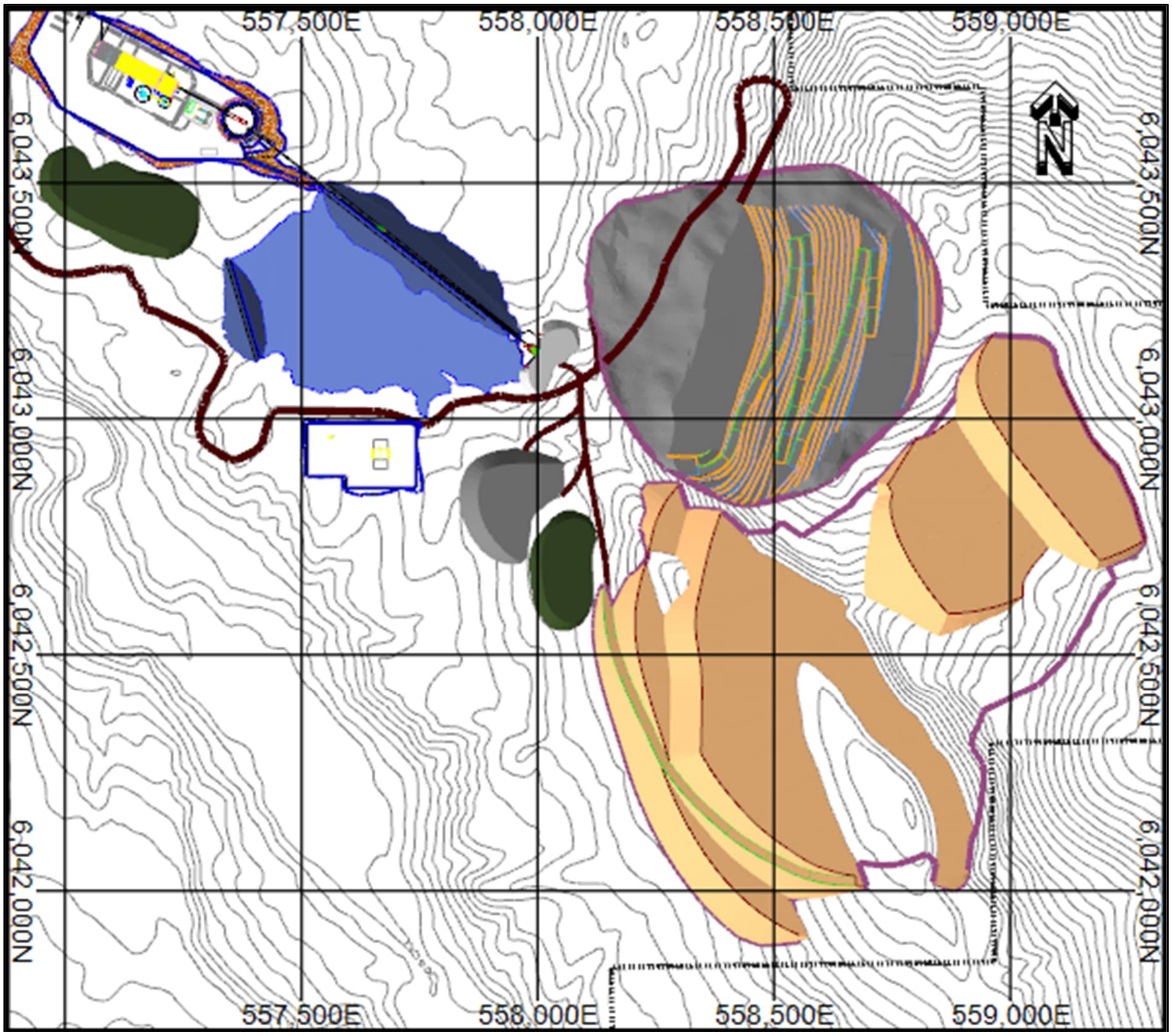
Source: SRK, 2025

Figure 16-13: Year 2



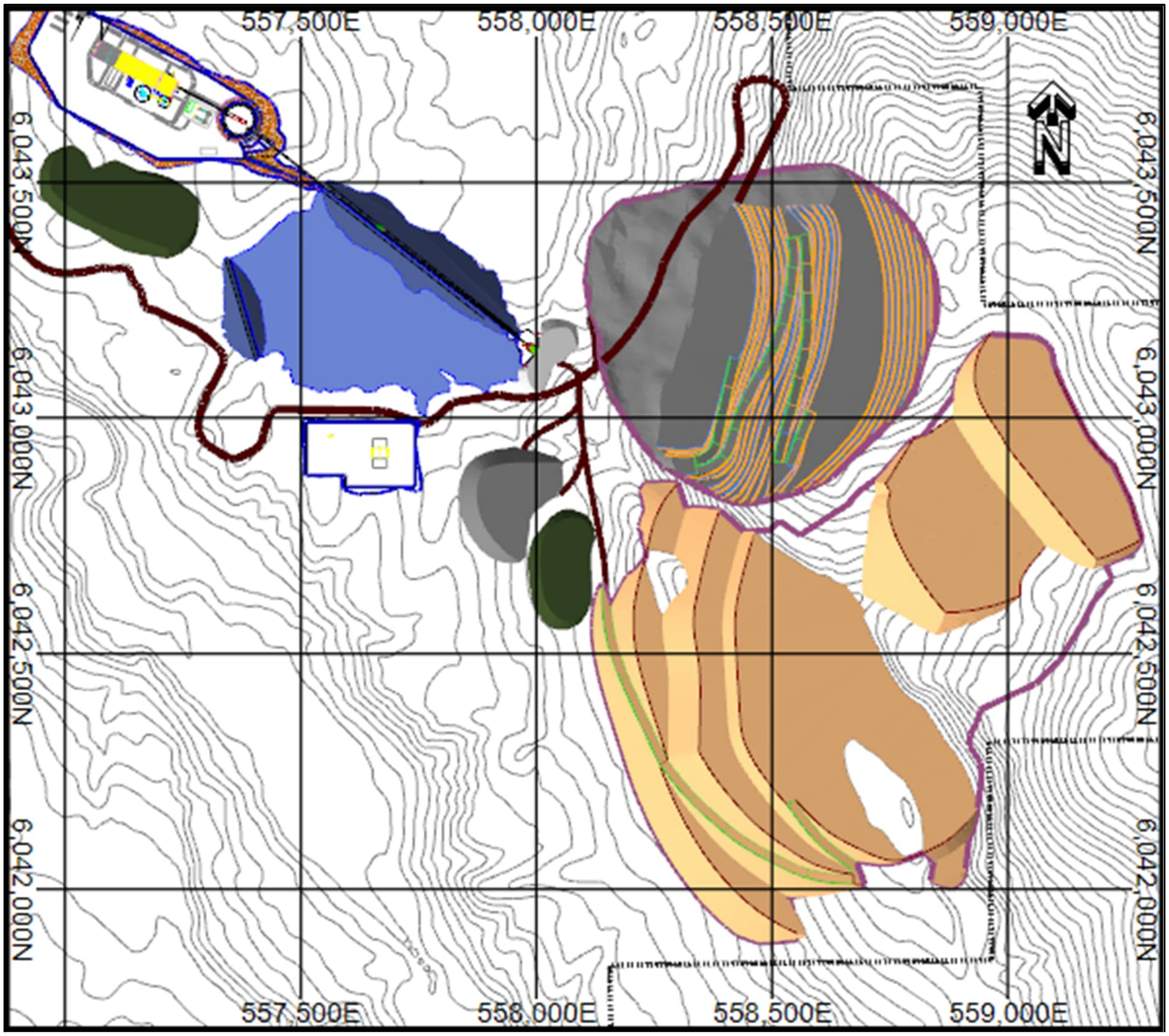
Source: SRK, 2025

Figure 16-14: Year 3



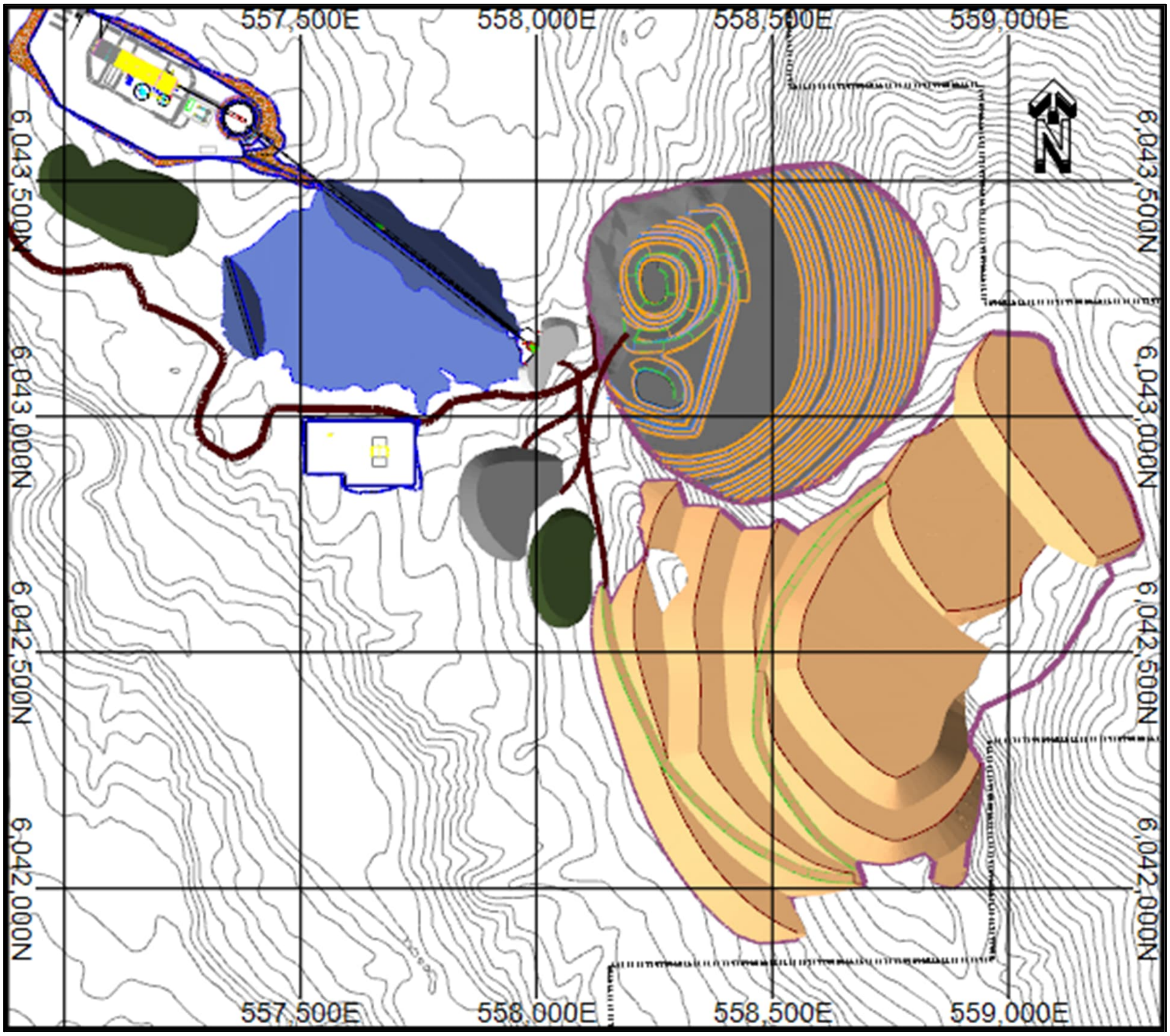
Source: SRK, 2025

Figure 16-15: Year 4



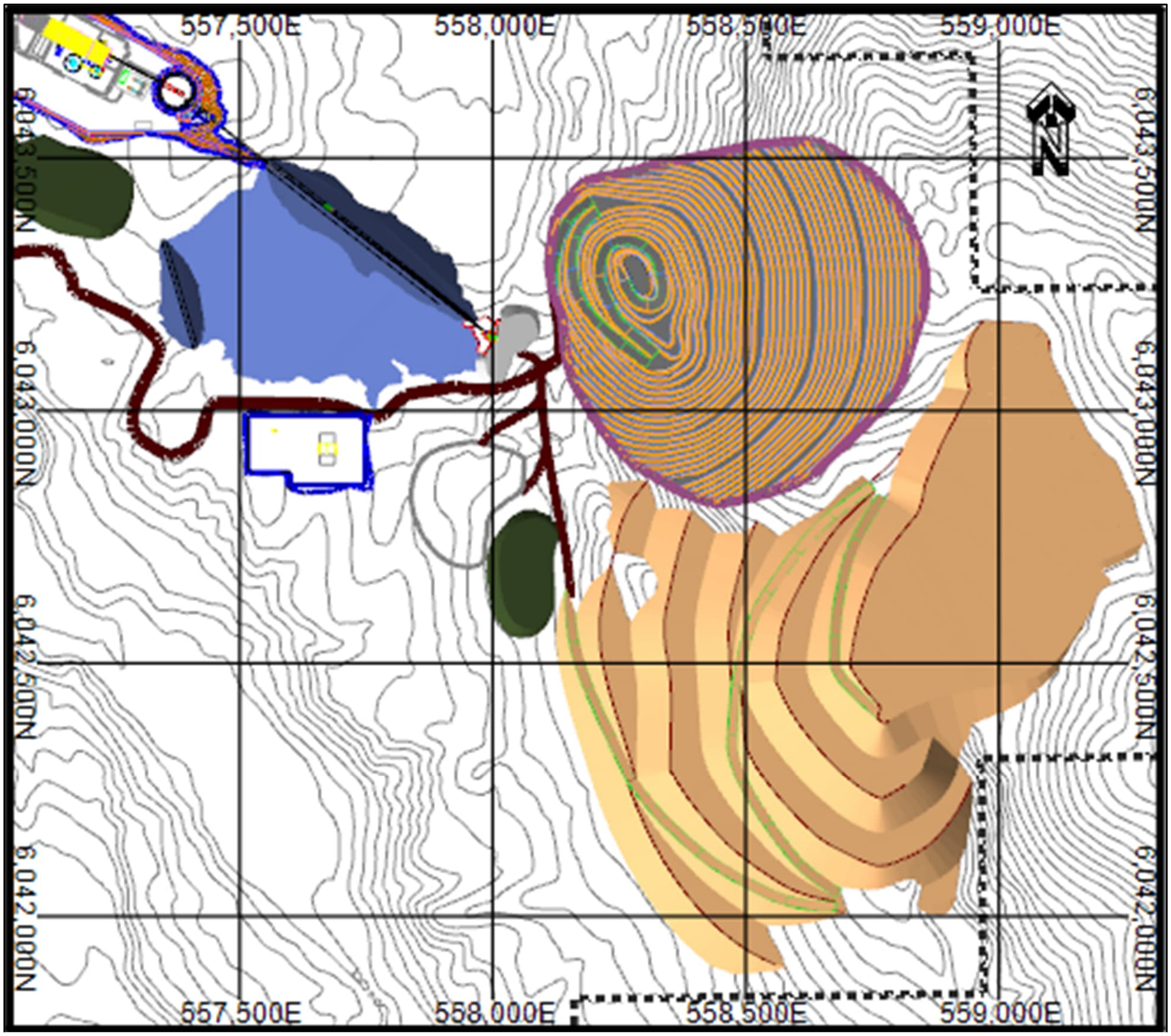
Source: SRK, 2025

Figure 16-16: Year 5 January 2033



Source: SRK, 2025

Figure 16-17: Year 10 January 2038



Source: SRK, 2025

Figure 16-18 End of LOM

16.6 Equipment and Labour Requirements

16.6.1 Approach

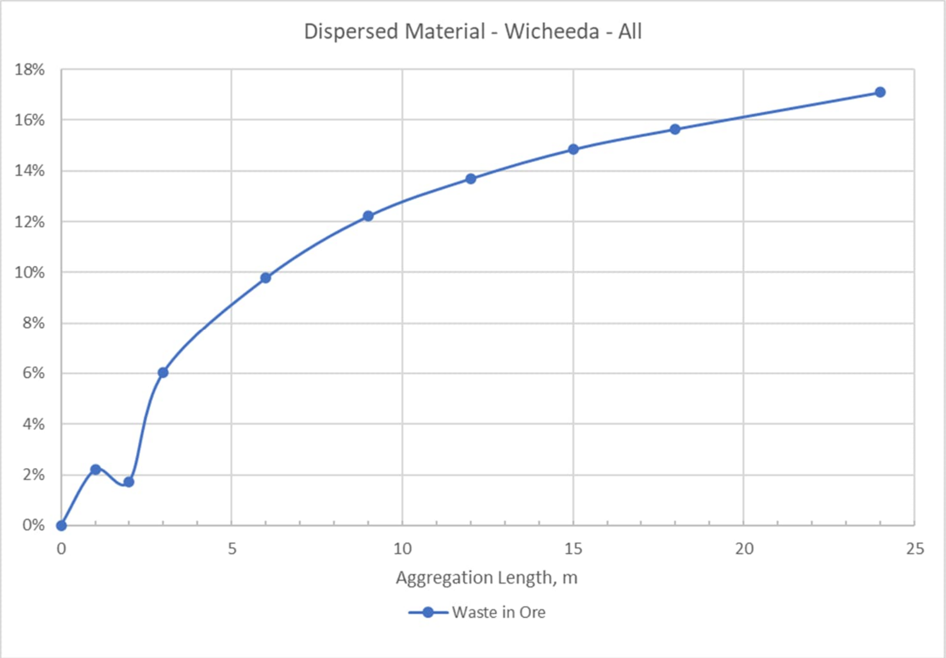
Open pit mining for the Wicheeda Project is to be conducted by conventional truck-and-shovel techniques by the owner. Waste and ore are drilled by percussion drills; blasting is facilitated by explosives contractor loading of blastholes; material is loaded into articulated haul trucks by hydraulic excavators; and the operation is supported by fleets of track dozers, graders, water trucks and smaller miscellaneous equipment.

The equipment and labour requirements for the Wicheeda Project are determined from first principles and engineering judgement. Productivities are estimated for the production equipment and applied against the mined quantities from the production schedule to derive equipment operating hours. Equipment operating hours are converted to fleet size requirements with assumed work schedules, physical availabilities and utilizations. Support equipment are factored against either loading or hauling equipment to derive operating unit requirements.

Labour requirements are based on manning operating equipment for mine operators and industry standard ratios for mine maintenance personnel to operators. Salaried personnel requirements align with regional strategies for supervisory and technical personnel.

16.6.2 Selective Mining Unit Sizing

To guide in equipment selection, SRK conducted a heterogeneity study on exploration drill hole data to assess the amount of internal dilution at different mining scales and thereby facilitate selective mining unit (SMU) sizing, particularly bench height. Figure 16-19 shows the result of this study.



Source: SRK, 2025

Figure 16-19: Heterogeneity and Scale for SMU selection

The chart plots the percent of below cut-off sample intervals within an above cut-off aggregation of samples (“Waste in Ore”) at different aggregation lengths. The analysis is conducted in the vertical direction, so the aggregation lengths are synonymous with bench height.

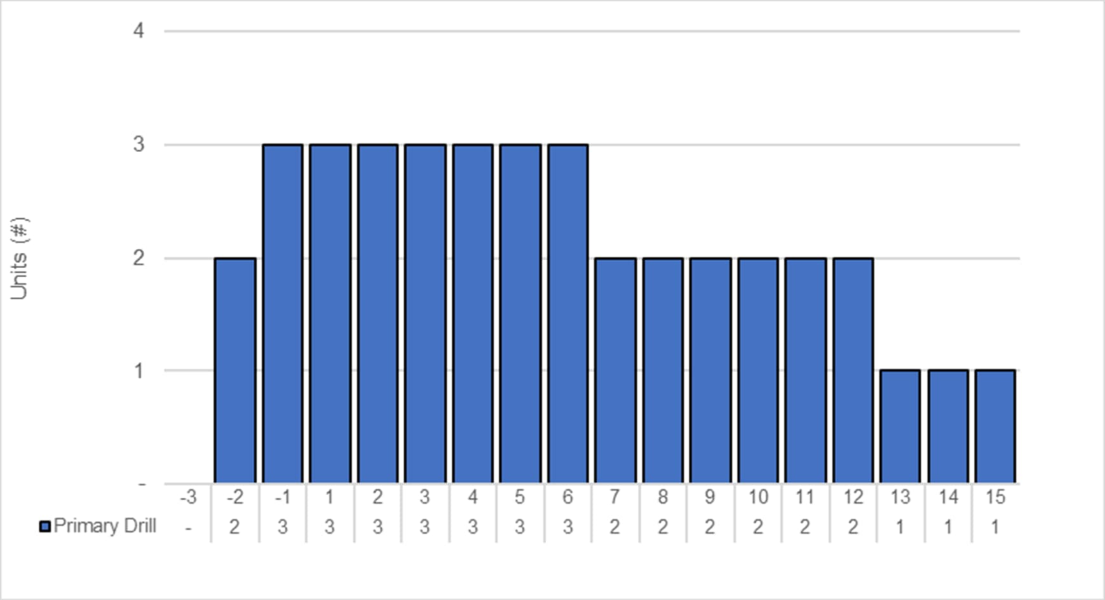
The results show that there is benefit in reducing the bench height to save waste in ore (internal dilution); however, the amount of dilution reduction, moving from 6 to 3 m benches, is less than 4%. This would not be sufficient to offset the extra costs and lower productivity that would arise from mining at such a smaller scale.

To complement this analysis, SRK assessed the loading and truck sizes, and corresponding fleet sizes, needed to meet required production rates. In consideration of this and the heterogeneity analysis, the project has adopted a bench height of six metres for mining in both waste and ore.

16.6.3 Drilling

Drilling is accomplished with down-the-hole percussion drills capable of drilling 127 mm holes for production drilling and 102 mm holes for pre-shear drilling. Drilling is performed on 6 m benches, using pattern sizes of 4.5 x 4.5 m for waste drilling, 4.0 x 4.0 m for ore drilling, and 4.5 x 3.7 m for wall control (cushion) blasting. Pre-shear blasting uses holes spaced 1.1 m around the pit perimeter.

The drill requirements over the life of mine are provided in Figure 16-20.



	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
■ Primary Drill	-	2	3	3	3	3	3	3	3	2	2	2	2	2	2	1	1	1

Source: SRK, 2025

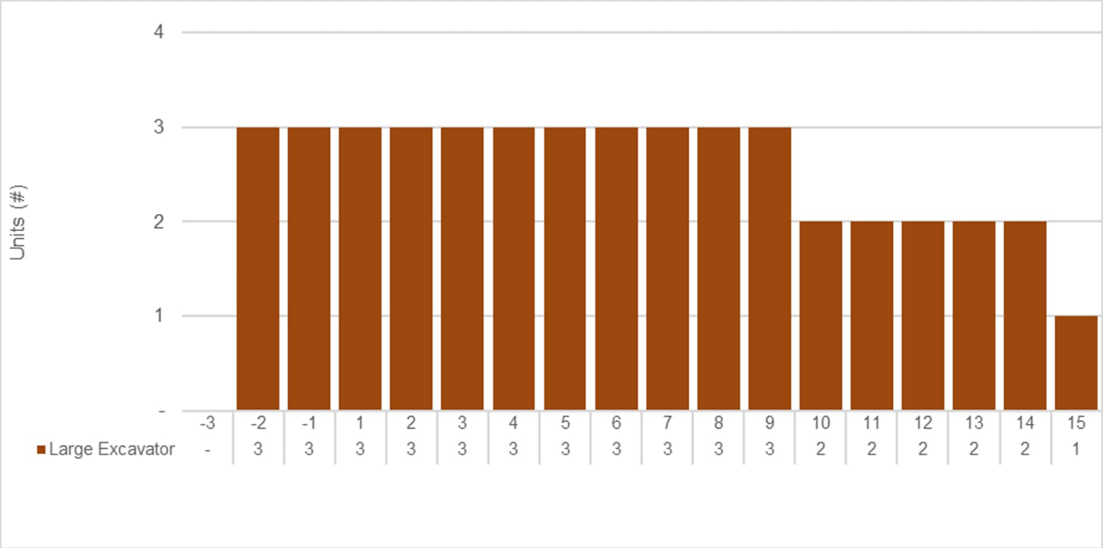
Figure 16-20: Production Drills

16.6.4 Blasting

Blasting is performed by owner blast crews working in conjunction with a contracted explosives supplier who loads the blastholes.

16.6.5 *Loading*

Waste and ore are mined by hydraulic excavators in backhoe configuration, equipped with 8.1 m^3 buckets. The loading unit fleet requirements are provided in Figure 16-21.



	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
■ Large Excavator	-	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	1

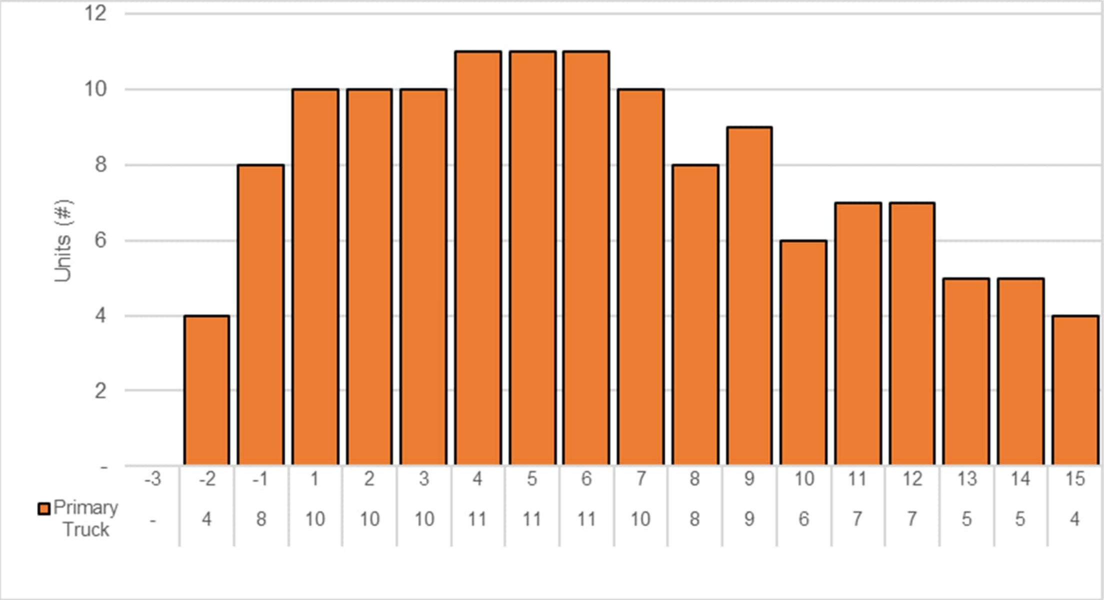
Source: SRK, 2025

Figure 16-21: Loading Units

16.6.6 *Hauling*

Haulage is accomplished by 55 t articulated trucks. Using Deswik.LHS software, haulage cycles times are determined for haul routes between each mining bench and all valid destinations for ore (ROM pad) and waste (WSF). For each period in the scheduling process, the shortest available cycle times are assigned for haulage.

The haul truck fleet requirements are provided in Figure 16-22.



	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Primary Truck	-	4	8	10	10	10	11	11	11	10	8	9	6	7	7	5	5	4

Source: SRK, 2025

Figure 16-22: Haul Trucks

16.6.7 Support

Support equipment requirements are estimated against the needs of primary production equipment. Tracked dozers are factored against the number of loading units, while graders and water trucks are factored against haul trucks.

Table 16-6: Support Equipment Requirements at Peak Production

Equipment Type	Equipment Size	Units
Track Dozer	450 hp	4
Grader	4.9 m blade	2
Water Truck	75,000 L	2
ROM Loader	8.6 m^3 bucket	2
Utility Backhoe	1.6 m^3 bucket	1

Source: SRK, 2025

16.6.8 Ancillary Equipment

Various ancillary equipment are required to support the mine operations, mostly in maintenance functions. These are listed in Table 16-7.

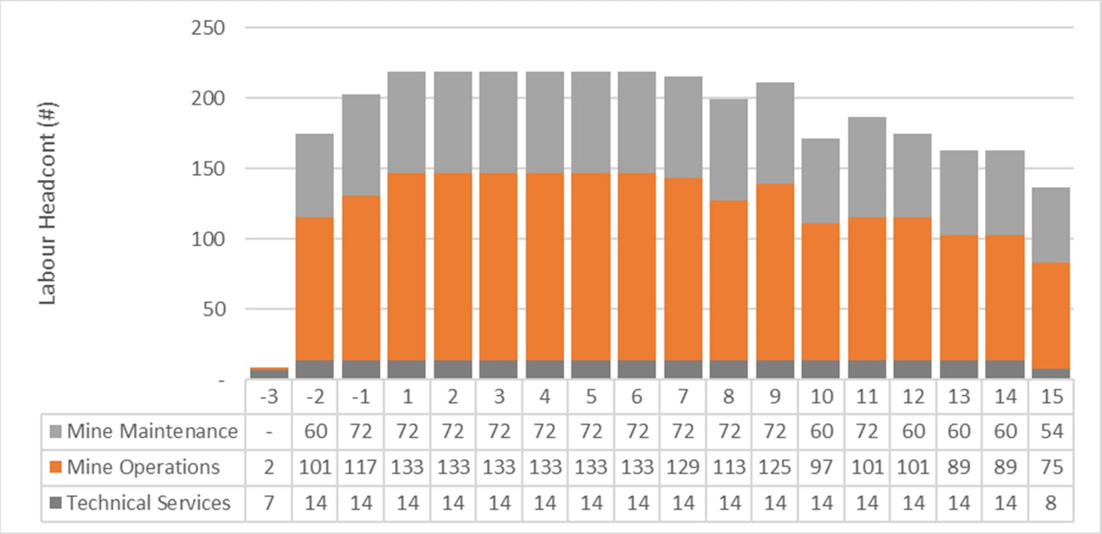
Table 16-7: Ancillary Equipment Requirements at Peak Production

Equipment Type	Units
Fuel/Lube Truck	1
Mechanics Truck	1
Crane Truck	1
Forklift	1
Tire Handler	1
Skid Steer Loader	2
Low Bed Trailer	1
Crew Bus	1
Light Vehicles	11
Lighting Plants	4

Source: SRK, 2025

16.6.9 Labour Requirements

Labour requirements are estimated for Mine Operations, Mine Maintenance, and Technical Services personnel across salaried and hourly ranks (see Figure 16-23). Split by salary and hourly personnel, there are 39 salaried and 180 hourly personnel at peak production.



	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
■ Mine Maintenance	-	60	72	72	72	72	72	72	72	72	72	72	60	72	60	60	60	54
■ Mine Operations	2	101	117	133	133	133	133	133	133	129	113	125	97	101	101	89	89	75
■ Technical Services	7	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	8

Source: SRK, 2025

Figure 16-23: Labour Requirements

17. Recovery Methods

Defense Metals Corporation is developing a REE minerals Concentrator Plant consisting of comminution and flotation facilities, and a Hydrometallurgical Processing Plant to produce a high purity mixed rare earth carbonate (MREC) from the Wicheeda ore. The process will first concentrate the ore by reducing the size of the material particles to a suitable size to enable the separation of the rare earth minerals from the gangue material through a flotation process. The rare earth concentrate will then continue to a Hydrometallurgical Processing Plant employing acid baking and water leaching to extract the rare earths into solution, and purification via selective precipitation, ion exchange, and solvent extraction to obtain a high purity MREC product.

The Concentrator Plant is being located near the mining operation, and the Hydrometallurgical Plant is envisaged to be located in Bear Lake, BC, Canada.

The Concentrator Plant is designed to process 1,800,000 t/a of ore based on the mine production rate developed for the proposed operation by SRK to produce a rare earth flotation concentrate containing 50 wt% TREO, which is then processed in the Hydrometallurgical Plant to a MREC product with an 87 wt% NdPrO rare earth distribution. The beneficiation and hydrometallurgical process plants are discussed in this section.

17.1 Concentrator Plant

The process plant design is based on a combination of metallurgical test work, mine production plan and reference data from industry standards and previous experience. Where necessary, benchmarking has been used to support the design.

- The Wicheeda process plant includes the following unit processes and associated facilities:
  - Primary crushing
  - Overland conveying of crushed process feed
  - Coarse ore stockpile and reclaim
  - SAG / Ball mill (SAB) grinding circuit
  - Flotation comprising a series of conditioning tanks prior to each flotation stage, rougher, scavenger and two stage cleaning at elevated temperatures
  - Concentrate thickening and filtration
  - Concentrate load out and storage (handling)
  - Tailings thickening and filtration
  - Reagents storage and distribution
  - Grinding media storage and addition
  - Water services (including fresh water, fire water, gland water, and process water)

- Potable water supply and distribution
- Air services (including high pressure air and low-pressure process air)
- Plant control room.

17.1.1 Process Design

The following sections outline the basis of process design for the overall plant including key criteria, operating schedule and availability, and throughput.

17.1.1.1 Process Design Basis / Criteria

Key design basis / criteria used in the plant design is summarized in Table 17-1.

The geology of the deposit has identified three main lithologies including Dolomite Carbonatite (DC) (66%), Xenolite Carbonatite (XE) (31%), and the rest mainly Syenite SYN. Based on the project mine plan the first eight (8) years of operation the plant would be processing a blend comprising primarily DC ore only and as from year 9 the blend becomes mainly a combination of DC and XE with a bit of SYN. The beneficiation plant design and equipment sizing criteria were selected based on LOM ore characteristics.

Table 17-1: Key Process Design Basis / Criteria

Description	Unit	Value
Mine Life	years	15
Operating Days in a Year	days	350
Crusher Operating Hours in a Day	h	12
Operating Hours in a Day	h	24
Crusher Plant Operating Factor	%	90
Milling and Flotation Plant Operating Factor	%	92
Filtration Plant Operating Factor	%	85
Annual Operating Hours - Crusher Plant	h	3,780
Annual Operating Hours - Milling and Flotation Plant	h	7,728
Annual Operating Hours - Filtration Plant	h	7,140
Run of Mine (ROM) Ore Annual Feed Rate	tpa	1,800,000
ROM Ore Moisture Content	wt.%	3.0
Ore Specific Gravity - Average	-	2.95
ROM Top Size	mm	1,000
Average Feed Grade (TREO %) Years 2 to 8	%	2.8
Average Feed Grade (TREO %) Years 9 to 15	%	1.92
Run of Mine (ROM) Feed Rate to Crusher Plant	t/h	476
Feed Rate to Milling and Flotation Plant	t/h	233
Feed Rate to Filtration Plant	t/h	252
Forecasted Concentrate Grade (TREO %)	%	50
Forecasted Overall TREO Recovery - Year 1 to 8 of Operation	%	80.9
Forecasted Overall TREO Recovery - Year 9 to 15 of Operation	%	69.3
Flotation Feed Size, (P_{80})	µm	80
Ball Mill Cyclone Overflow Density	% w/w	35

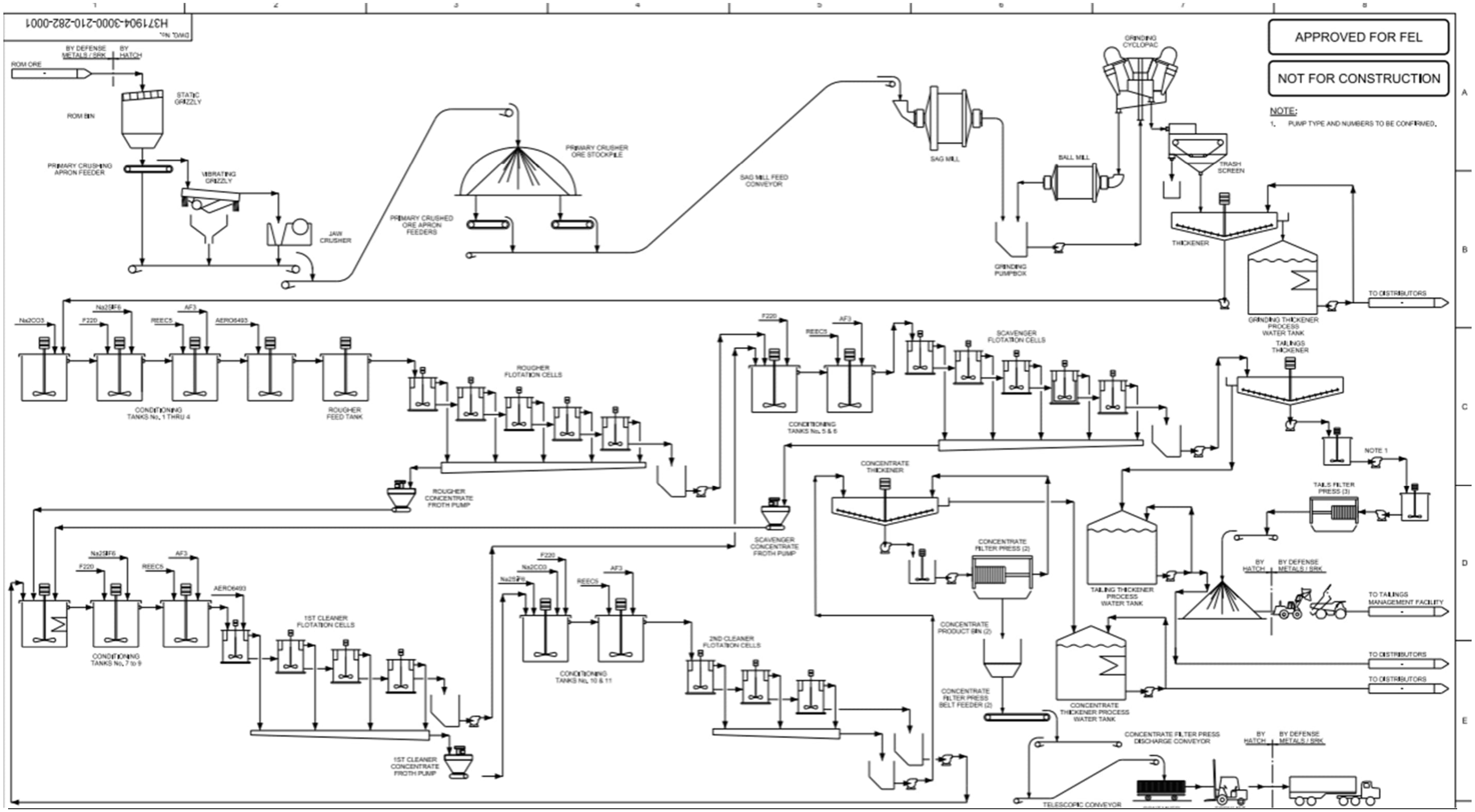
Description	Unit	Value
Grinding Cyclone Overflow Thickener Underflow Density	% w/w	50
Grinding Cyclone Overflow Thickener Underflow Temperature	°C	55
Grinding Cyclone Overflow Thickener Underflow Density	% w/w	50
Rougher / Scavenger Conditioning & Flotation Temperature	°C	55
Cleaner Conditioning & Flotation Temperature	°C	75
Rougher Flotation Density	% w/w	35
Scavenger Flotation Density	% w/w	34
Cleaner Flotation Density	% w/w	21-22
Concentrate Thickener Underflow Density	% w/w	70
Concentrate Filter Cake Moisture	% w/w	7.9
Final Concentrate Solids Specific Gravity	-	4.1
Final Concentrate Particle Size (80% passing – P80)	µm	61-71
Tailings Thickener Underflow Density	% w/w	60
Tailings Filter Cake Moisture	% w/w	9.0

Source: Hatch, 2025

17.1.2 Process Plant Description

The basis of design and description for major plant equipment and unit processes is summarized in the following sub-sections.

A summary flow sheet for the beneficiation plant is shown in Figure 17-1.



Source: Hatch, 2025

Figure 17-1: Simplified Flow Sheet Beneficiation Plant

17.1.2.1 *Primary Crushed Ore Delivery and Storage*

Run of Mine (ROM) material is discharged via haulage truck into a ROM Bin that provides a buffer surge to the crushing plant. ROM oversize and any material bridging that occurs on the static grizzly are cleared via a Rock Breaker.

Material is withdrawn at a controlled rate from the ROM Bin via a variable speed Apron Feeder that discharges the ROM onto a vibrating grizzly screen. Any dribbling material from the apron feeder is collected and conveyed via spillage conveyor. The grizzly screen removes fines that would otherwise increase wear prior to discharging the oversize (+120 mm) into the primary jaw crusher. The crusher operates with a CSS of 150 mm with the product being recombined with the grizzly screens' undersize and conveyed by a short sacrificial conveyor. Tramp magnetic metal is removed by a self-cleaning belt magnet at the head pulley of the sacrificial conveyor and discharged into a Tramp Metal Bin for periodic removal.

A dust extraction system removes dust at various transfer locations in this area and filters it within a bag filter. These include the apron feeder and dribble conveyor discharges, vibrating grizzly screen and jaw crusher feed/discharge.

The crushed feed is transferred to the process plant via an overland conveyor. The overland conveyor discharges to an enclosed coarse ore storage stockpile that provides a buffer surge between the crushing circuit and the process plant. The stockpile will have a live capacity that can support process plant operation for about 24 hours when the crusher circuit is not operating. Dead ore will be recovered by a bulldozer and front-end loader and will be pushed to the feeders if the crusher circuit is unavailable for extended periods.

Ore will be reclaimed from the stockpile and fed to the SAG mill feed conveyors via the variable speed apron feeders. The feeder capacity design will allow each feeder to maintain the required tonnage rate to the SAG mill feed conveyor.

17.1.2.2 *Grinding Circuit*

The grinding line consists of a single fixed speed SAG mill, followed by a single fixed speed ball mill operating in a closed circuit with a cyclone cluster. The product from the grinding circuit (cyclone overflow) has a typical size of 80% passing 80 µm.

The SAG mill feed conveyor discharges mineralized material, and grinding media, into the feed chute of the SAG mill together with mill feed dilution water. The SAG Mill discharge will pass through a trommel equipped with a trommel return system. The SAG discharge trommel undersize will be collected in a pump-box and sent to the secondary grinding (ball mill), while the trommel oversize will be returned to the SAG mill for further size reduction. SAG mill grinding media is also added to the SAG mill feed chute with a 1-t kibble with a false bottom.

The SAG mill trommel undersize gravitates to the cyclone feed pumpbox where it is combined with the discharge from the ball mill. The slurry is transported to a single cyclone cluster using two variable-speed cyclone feed pumps (duty and stand-by).

Dilution process water is added to the cyclone feed pumpbox before the slurry is pumped to the cyclone cluster for classification. Coarse particles report to the cyclone underflow and are directed to the ball mill feed chute. The cyclone overflow stream gravitates to a grinding cyclone overflow thickener via a cross-stream sampler.

SAG mill balls are added via a ball addition system and bunker adjacent to the SAG mill feed conveyor. A separate ball storage bunker is provided for the ball mill which has a dedicated ball charging system.

A SAG mill feed chute removal system, and a ball mill feed chute removal system are used to service the mills. A universal liner handler is provided for use with both the SAG and ball mills.

17.1.2.3 Grinding Cyclone Overflow Thickener

Hydrocyclone overflow will flow by gravity to a trash screen to remove tramp material. The oversize- including rock particles and trash— from the screen will be collected in a bunker and periodically picked up and trucked to the tailings area for disposal. The undersize from the screen as slurry will be collected in a pipe and flow by gravity to a high-rate thickener via a feed tank.

Flocculant and coagulant solutions will be added to the thickener feed to promote the settling of fine solids. The thickener will have a diameter of 25 m, a high-rate feed well design, will produce a thickened slurry of 50% solids to feed the rougher flotation conditioning tanks.

The thickener overflow will be collected in an overflow heated tank via gravity flow, which will recycle back to the process as process water. Both the thickener and the overflow process water will be installed outdoors and will be covered and insulated to maintain temperature at 55°C.

The thickener will have two underflow pumps (one operating and one standby) and transfer the thickened material to the rougher flotation conditioning tanks. The overflow solution from thickener will flow by gravity to the process water grinding thickener heated tank, where it will be recycled at 55°C as needed in the rougher flotation circuit.

17.1.2.4 Flotation Circuit and Conditioning Tanks

The flotation circuit will consist of rougher and scavenger flotation circuits followed by two stages of cleaner flotation to produce a final flotation concentrate containing 50% TREO.

17.1.2.5 Rougher Scavenger Flotation

The rougher scavenger flotation facility will include four rougher conditioning tanks, a rougher feed tank, five rougher flotation cells, two scavenger conditioning tanks, five scavenger flotation cells and a rougher flotation tails pump box.

The slurry from the grinding cyclone overflow thickener underflow, maintained at 55 °C with 50% solids, will be conditioned in a four-stage conditioning process before entering the rougher flotation cells. The first stage conditioner will serve to adjust the slurry pH to 9.5-10 with soda ash, while the second stage conditioner will serve to condition the slurry with depressants for the gangue minerals (F220 and Na_2SiF_6). The third stage conditioner will serve to condition the slurry with the mineral collector and activator (REEC5 and AF3), and the fourth stage conditioner will serve to condition the slurry with the mineral collector (Aero 6493). The conditioned slurry will then gravitate to an agitated tank, where it will be diluted with a controlled quantity of process water to achieve a concentrate of 35% solids before advancing to the rougher flotation circuit.

The concentrate from the rougher cells will gravitate to the 1st cleaner flotation cells. The tails from the rougher cells will be pumped to the scavenger conditioning circuit for further processing.

Tails from rougher flotation stage and 1st cleaner cells will be fed into two-stage conditioning tanks before entering the scavenger flotation cells at 55 °C. The first stage conditioner will condition the slurry with depressants for the gangue minerals (F220), and the second stage conditioner will condition the slurry with the mineral collector and activator (REEC5 and AF3). Tails from the scavenger cells will be pumped to the tailings thickener before being transferred to the tailings filter feed tank.

17.1.2.6 Two Stage Cleaner Flotation

The cleaner flotation facility will include three 1st cleaner conditioning tanks, four 1st cleaner flotation cells, two 2nd cleaner conditioning tanks, three 2nd cleaner flotation cells, and tails and concentrate pump boxes.

Concentrate from rougher and scavenger flotation cells among tails from 2nd cleaner flotation cells will be fed into a three-stage conditioning circuit prior to feeding into the 1st cleaner flotation cells at 75 °C and pH 7.5. Slurry will be heated in the first conditioning tank to 75 °C and be conditioned with depressants for the gangue minerals (F220 and Na_2SiF_6) in the second conditioning tank. In the third conditioning tank, mineral collector and activator (REEC5 and AF3) will be added prior to advancing to the first cleaner stage.

Concentrate from the 1st cleaner cells will be pumped to the 2nd cleaner flotation circuit, while tails from the 1st cleaner cells will be pumped to the scavenger conditioning tanks before feeding to the scavenger flotation stage for further processing.

Concentrates from the 1st cleaner cells will be pumped in two 2nd cleaner conditioning tanks prior to feeding into the 2nd cleaner flotation circuit. The first stage conditioner will serve to condition the slurry with depressants for the gangue minerals (F220, Na_2CO_3, and Na_2SiF_6), and the second stage conditioner will serve to condition the slurry with the mineral collector and activator (REEC5 and AF3).

Concentrate from the 2nd cleaner flotation circuit will be pumped to the concentrate thickener, while tails from the 2nd cleaner cells will be pumped to the 1st cleaner conditioning tanks before feeding to the 1st cleaner flotation cells for further processing.

17.1.2.7 Concentrate Thickening and Filtration Circuit

TREO concentrates are dewatered using a thickener and two filters. The concentrate thickening and filtration circuit consists of a single 5-m diameter high-rate thickener, a heated thickener overflow tank and two pressure filters, one duty and one standby.

The concentrate from the 2nd cleaner flotation cells is pumped into the concentrate thickener. Flocculant and coagulant are added to the thickener feed streams to enhance settling. The thickener overflow reports to concentrate process water tank, which is electrically heated to maintain a 75°C temperature and is then distributed to the process. The thickener TREO concentrate solids settle in the underflow at a density of 70% solids. The thickener underflow stream is pumped to an agitated storage tank using centrifugal pumps (one operating, one standby) The thickener will be installed inside of the concentrator building and be covered and insulated to maintain heat.

The storage tanks provide 4 hours surge capacity allowing filter maintenance to be conducted without affecting process plant throughput. The filter feed is pumped to two pressure filters (one operating, one standby that produces a filter cake of 8% moisture.

Filtrate collected in a filtrate tank, and cloth wash and flushing water is discharged to the filtration area sump pump which returns it to the concentrate thickener.

17.1.2.8 Concentrate Loadout

The flotation concentrate cake at approximately 8 wt.% moisture and approximately 50 wt.% TREO, will be conveyed into half height containers and moved by truck to the hydrometallurgy facility for further processing.

17.1.2.9 Tailing Thickener, and Filtration plant

The final flotation tailing will be thickened to about 60% solids in a 35 m diameter high-rate thickener and then pumped to an agitated surge tank with three-hour storage. Flocculant and coagulant solutions will be added to the thickener feed to promote the settling of fine solids. The thickener overflow will be recycled to a tailings process water tank where it will be pumped to the grinding circuit. The thickener will be installed outside of the concentrator building, due to size, and be covered and insulated to maintain heat.

17.1.2.10 Tails Filtration plant

A thickened slurry with an average of 60 wt% solids from the final tails tank is pumped approximately 3 km to an agitated filter feed tank with one-hour surge capacity in the tailings filtration plant. The slurry is then pumped to three pressure filters (two operational, one standby) that produce filter cake with 9 wt% moisture or less. Each filter press is fed by a centrifugal pump, and the filter cake is discharged onto belt conveyors under each filter unit and stockpiled. Filter cake is transported using trucks to the filtered tailings storage facility for placement and compaction. A front end loader will be used to feed the trucks.

Filtrate from the filters, including cloth wash and flushing water, is collected in a sump pump and returned to the filtrate tank before being pumped to the contact water pond. Compressed air for the filters is supplied by dedicated compressors, with two cake compression and core-blow air compressors and two cake-membrane squeeze-air compressors.

The area is maintained and accessible by mobile equipment for maintenance. Sump pumps return any spilled slurry or solutions to the filter feed tank. Maintenance of filter units is facilitated by an overhead hoist, monorail, and small mobile crane. A forklift is used for sump pump maintenance, and a mobile crane is used for maintaining filter-feed pumps and the filter-feed tank agitator.

17.1.2.11 Reagents and Utilities

17.1.2.11.1 Sodium Carbonate (Na_2CO_3)

Na_2CO_3 or soda ash is used as pH modifier in the flotation circuit. The reagent will come in 1 tonne bulk bags and will be dissolved in water to a concentrate of 5% w/v. The Na_2CO_3 solution is transferred to a storage tank and then metered to the flotation with a dedicated diaphragm type pump.

17.1.2.11.2 F220 (Pionera Lignin Product)

F220 is used as depressant in the flotation circuit. The reagent will come in 1 tonne bulk bags and will be dissolved in water to a concentrate of 10% w/v. The F220 solution is transferred to a storage tank and then metered to the flotation with a dedicated pump.

17.1.2.11.3 REEC5

REEC5 is a mixture of CustoFloat 7080, CustoFloat 7084, tall oil D25LR, diesel fuel, and pine oil and is used as TREO collector in the flotation circuit. Each reagent is delivered as liquid to the site by tanker truck or rail. Individual reagent is metered to a mixing tank with dosing pumps. The reagent mixture is transferred to a storage tank and then pumped to the flotation via dosing pumps.

17.1.2.11.4 AF3

AF3 is a mixture of Texanol ester alcohol, NaF, and octyl phosphonic acid and is used as TREO collector in the flotation circuit. Each reagent is delivered in liquid or 1 tonne bulk bags to the site by tanker truck or rail. These are discharged when required into an agitated mixing tank which has been pre-filled with a known quantity of fresh water. Texanol ester alcohol and octyl phosphonic acid are metered to the mixing tank via dosing pump. NaF is also added in the mixing tank via a variable speed screw feeder. The mixture with a concentrate of 10% w/v is transferred to a storage tank and then pumped to the flotation via dosing pumps.

17.1.2.11.5 Sodium Fluorosilicate (Na_2SiF_6)

Na_2SiF_6 is used as depressant in the flotation circuit. The reagent will come in 1 tonne bulk bags and will be dissolved in water to a concentrate of 2% w/v. The Na_2SiF_6 solution is transferred to a storage tank and then metered to the flotation with a dedicated pump.

17.1.2.11.6 Aero 6493

Aero 6493 is used as TREO collector in the flotation circuit. The reagent is delivered as a liquid in 1.0 m^3 plastic totes. The tote is connected to a fixed manifold and collector is pumped to addition points.

17.1.2.11.7 Flocculant

A packaged flocculant make-up system will be provided to supply flocculant to the thickeners. Flocculant will be supplied in solid form in bags and mixed with fresh water in a mix tank to produce a concentrated solution. The solution will be aged in a mix tank, and then pumped to, and stored in, a holding tank before being pumped to the thickeners. Process water will be added as dilution water to the flocculant solution via an inline mixer at each of the thickeners.

17.1.2.11.8 Coagulant

A packaged coagulant make-up system will be provided to supply coagulant to the thickeners. Coagulant will be supplied in solid form in bags and mixed with fresh water in a mix tank to produce a concentrated solution. The solution will be aged in a mix tank, and then pumped to, and stored in, a holding tank before being pumped to the thickeners. Process water will be added as dilution water to the coagulant solution via an inline mixer at each of the thickeners.

17.1.2.11.9 Fresh Water

Fresh water is stored in a raw water tank that is insulated and traced. It is assumed that the fresh water is clean and can be used in process without further purification.

17.1.2.11.10 Fire Water

Fire water is stored in the same raw water tank as the fresh water. The fire water package (supplied by Vendor) includes main electric firewater pump, diesel backup pump, and electric jockey pump.

17.1.2.11.11 Potable Water

The potable water package (supplied by Vendor) filters the fresh well water to be used for potable water users. The package includes softening, chemical dosing, and filtration.

17.1.2.11.12 Gland Water

Fresh water is filtered to produce gland water that is distributed to slurry pumps. In the next project phase, mechanical seals will be considered, where possible, to reduce water consumption.

17.1.2.11.13 Blower Air

Low pressure air is supplied to the flotation cells by one duty and one standby low-pressure blower.

17.1.2.11.14 Compressed and Instrument Air

Compressed plant air is supplied to the process through an air receiver and distributed to the required process areas. Instrument air is supplied through an air receiver and air dryer to the plant.

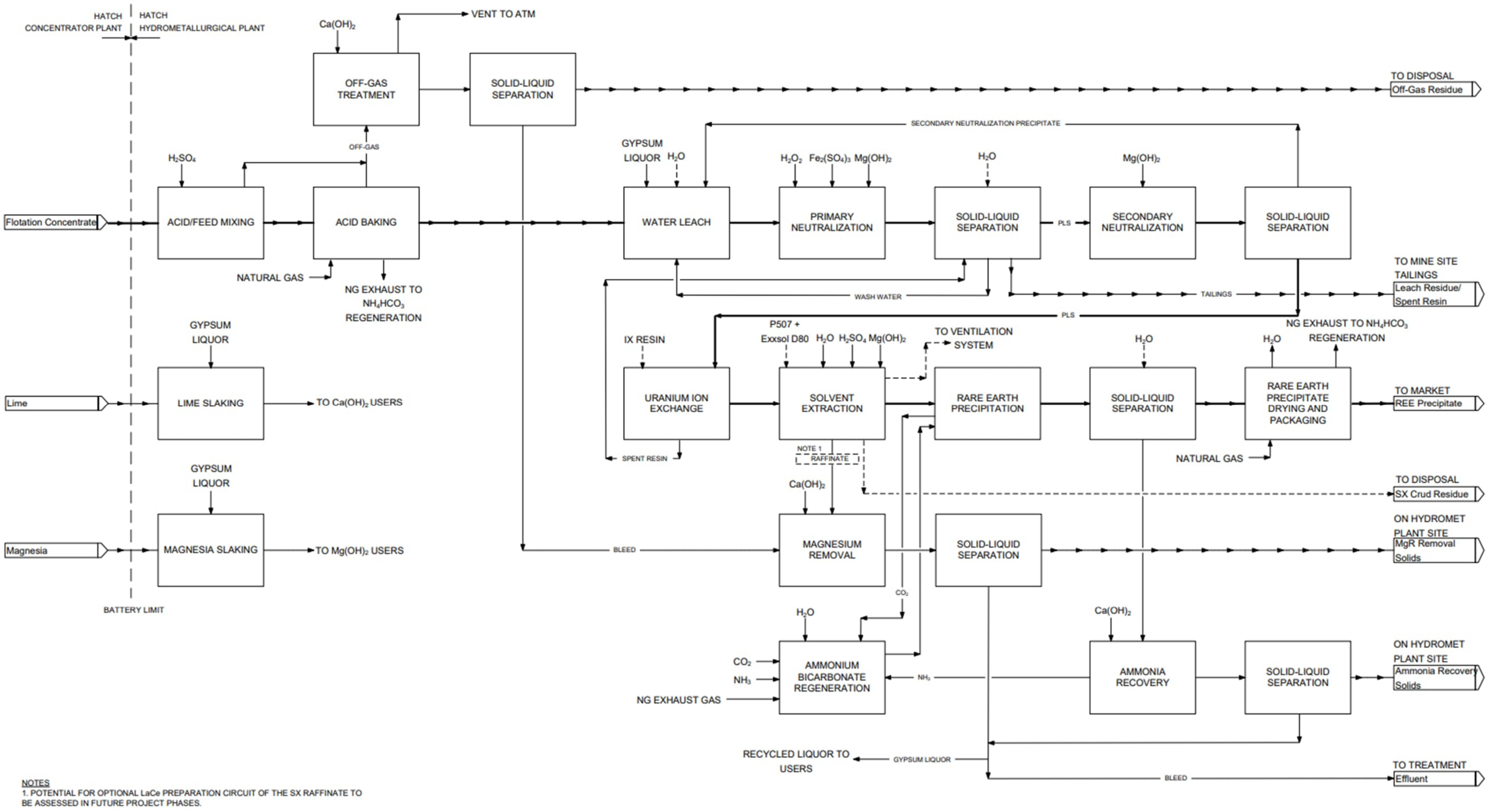
Compressed air for concentrate filter operation is supplied by the filter vendor’s Compressor package. A cake compression air compressor and cake drying air compressor provide air for the concentrate filters.

17.2 Hydrometallurgical Plant

17.2.1 Introduction and Summary

The Wicheeda Hydrometallurgical plant is designed to process 85,800 t/a (dry solids basis) of rare earth concentrate to produce high purity MREC. The proposed flowsheet for the hydrometallurgical plant is shown in Figure 17-2.

The process employs acid mixing/baking and water leaching to extract the rare earths from the concentrate, applies multiple impurity removal steps, one solvent extraction REE separation step to remove lanthanum and cerium, then precipitates the remaining rare earths as a lanthanum and cerium depleted mixed rare earth carbonate. Further description of the process is provided in Section 17.2.4.



Source: Hatch, 2025

Figure 17-2: Hydrometallurgical Plant Block Flow Diagram

17.2.2 Design Basis

17.2.2.1 Concentrate Characteristics

The rare earth composition used for the basis of the hydrometallurgical mass and energy balance (MEB)is shown in Table 17-2. It is based on the design composition of the beneficiation plant concentrate, corresponding to the anticipated concentrate produced during years 2 to 8 of plant operation. Table 17-2 shows the elemental composition which was retrieved from the metallurgical test work. For calculation and simulation purposes, elemental compositions were converted to representative mineral compounds. A moisture content of 7.9 wt% is assumed for the REE concentrate, therefore 0.079 kg H_2O/ kg concentrate is incorporated to the simulation to derive the wet concentrate composition.

Table 17-2: Design Elemental Composition of Rare Earth Concentrate (Year 2 to 8)

Element	g/t	Element	g/t
La	134000	S	200
Ce	178000	F	29400
Pr	14600	Si	1200
Nd	39200	Al	300
Sm	3860	Fe	19600
Eu	769	Mg	23600
Gd	1750	Ca	93500
Tb	124	Na	1200
Dy	352	K	100
Ho	26.1	Ti	100
Er	29.9	Mn	3500
Tm	1.6	U	7.6
Yb	2.6	Th	3890
Lu	2	Ba	8067
Y	600	Sr	2533
P	27900		

Source: Hatch, 2025

17.2.2.2 Process Design Basis and Criteria

The throughput of the plant MEB model is based on a fixed feed rate of flotation concentrate. Rare earth carbonate production basis is determined by the design recoveries and parameters for each processing step derived from metallurgical test work data, literature, or industry best practices. The design criteria for the SX system were developed by J. Goode and M. Nees (Consultants to Defense Metals) using general design data and data generated as part of the Defense Metals program of test work. The model output production rate is 10,750 t/a of rare earth carbonate (60.4%TREO). The major parameters and key design criteria that were used and obtained for the MEB model are listed in Table 17-3. This design criteria should be considered as preliminary and is subject to change in future project phases with additional experimental data and design definition.

Table 17-3: Model Input / Output Major Parameters

Description	Value
Input	
Operating hours per year	7140 hours
Concentrate feed rate (dry basis)	85,800 t/a
Concentrate feed moisture	7.9 wt%
Concentrate feed composition	(see Table 17-2)
Output	
Rare earth carbonate production rate	10,750 t/a
TREO production equivalent	6,490 t/a
Product %NdPr	53 wt%, wet basis 87 wt%, REO basis
Product %TREO	60%, wet basis
Key Design Criteria	
Acid Baking	
Acid addition rate (100% H_2SO_4 basis)	1034 kg $H_2SO_4/t_{concentrate\ (wet\ basis)}$
Temperature	350 °C
Residence time	1.8 h
Off-Gas Treatment	
Sulphuric acid mist removal efficiency	>99.5 %*
Hydrogen fluoride mist removal efficiency	>99.5 %*
Dust discharge concentration	<5 $mg/Nm^{3\ *}$
*The design criteria for the off-gas system are approximate and will be further specified in the next project phase. The off-gas system will be specified to ensure the emissions meet all applicable regulatory limits.	
Water Leaching	
Feed liquid to solids ratio	7.3 t_{Liq}/t_{solids}
Residence time	3.0 h
Temperature	30 °C
Primary Neutralization	
Target pH	2
$Fe_2(SO_4)_3$ dosing	90 $g_{solution\ (60\ gFe/L)}/kg_{solids}$
H_2O_2 dosing	0.78 $g_{solution\ (50\ wt\%)}/kg_{solids}$
Residence time	2.5 h
Secondary Neutralization	
Target pH	6-6.5
Residence time	5.0 h
Uranium Ion Exchange	
Residual U concentration	0.02 mg/L
U concentration on loaded resin	2 g/L
Solvent Extraction	
H_2SO_4 dosing rate	15 $kg_{(93\ wt\%)}/m^3$
$Mg(OH)_2$ dosing rate	2.8 kg/m^3
Water addition rate	0.21 $m^3_{H_2O}/m^3$
Strip liquor production rate	0.08 m^3_{strip}/m^3
Sulphate deportment to strip liquor	51 %
La deportment to raffinate	100 %
Ce deportment to raffinate	99.5 %
Y, Pr-Lu deportment to strip	100 %
Operating temperature	35 °C
Ammonia Recovery	
$Ca(OH)_2$ dosage	20 % excess (for SO_4^{2-} precipitation)
Residence time	4.4 h
RE Precipitation	
NH_4HCO_3 dosage	0.1 % excess (for REE precipitation)
Residence time	2.0 h
Dryer discharge free moisture	0.4 wt% (16.2 wt% water as $RE_2(CO_3)_3{\cdot}5H_2O$)

Description	Value
Magnesium removal	
$Ca(OH)_2$ dosage	20 % excess (for SO_4^{2-} precipitation)
Residence time	4.4 h
Ammonium bicarbonate regeneration	
NH_4HCO_3 outlet concentration	1.89 mol/L
CO_2 capture efficiency	80 %*
NH_3 capture efficiency	99 %*

Source: Hatch, 2025

*The design criteria for the ammonium bicarbonate system are approximate and will be further specified in the next project phase. The ammonium bicarbonate system will be specified to ensure the emissions meet all applicable regulatory limits.

17.2.3 Mass-Energy Balance and Process Flow Diagrams

17.2.3.1 Methodology

A Mass and Energy Balance (MEB) was prepared using the SysCAD software package (Version 9.3, Build 139). The mass balance reflects the selected process flowsheet and is based on inputs from the Process Design Criteria (PDC) and Process Design Basis (PDB).

Component properties were sourced from a combination of the HSC software database (Version 10), the HSC Estimate module (for thermodynamic properties estimation for non-documented species), the model’s internal properties database and publicly available literature sources.

17.2.3.2 Key Inputs to the Mass-Energy Balance

The following key inputs were identified in the development of the MEB:

- The MEB was configured to process 93,200 t/a (85,800 t/a dry feed) of rare earth concentrate which resulted in a flow of 13.05 t/h (12.02 t/h dry feed).

17.2.3.3 Overall Balance

The summary of the overall balance across the entire process can be seen in Table 17-4. The recoveries for each rare earth element are reported in Table 17-5. The largest predicted loss of Nd and Pr in the process is the filter cake from the primary neutralization filter due to incomplete extraction from the feed concentrate and co-precipitation of REEs during primary neutralization.

Table 17-4: Summary Overall Balance Across Entire Process

Description	REE (kg/h)	NdPr (kg/h)	Overall Mass (t/h)
IN	**5,009**	**722**	**151.78**
Mixed Rare Earth Concentrate Feed	5,009	722	13.05
Sulphuric Acid	0	0	16.77
Infiltration Air	0	0	0.11
Sweep Air	0	0	20.29
Burner Air	0	0	13.59
Natural Gas	0	0	0.82
Ingress Air	0	0	6.93
Lime	0	0	9.35
Magnesia	0	0	1.76
Hydrogen Peroxide	0	0	0.01
Fresh IX Resin	0	0	0.002
Ferric Sulphate	0	0	0.84
Ammonia Solution	0	0	0.13
Carbon Dioxide	0	0	0
Magnafloc 10 Slurry	0	0	1.12
Magnafloc 338 Slurry	0	0	0.08
Gland Water	0	0	3.89
Plant Water	0	0	63.02
OUT	**5009**	**722**	**151.78**
Mixed Rare Earth Carbonate Product	773	674	1.51
Treated Paddle Dryer Off-Gas to Atmosphere	0	0	1.91
Off-Gas Residue to Disposal	2.59	0.36	9.95
Treated Off-Gas to Atmosphere	0	0	38.28
Primary Neutralization Residue to Disposal	405	46	11.00
IX Guard Filter Solids	2.57	0.77	0.004
Ammonia Bicarbonate Tower Vent	0	0	17.78
Ammonia Recovery Solids to Disposal	0.18	0.12	2.73
Magnesium Removal Solids to Disposal	3,826	0	33.50
Lime Grit Solids	0	0	0.15
Magnesia Grit Solids	0	0	0.03
Ammonia Storage Vent	0	0	0.01
Cooling Tower Off-Gas	0	0	12.88
Total Effluent to Treatment	0.12	0	22.07
BALANCE	**0.00**	**0.00**	**0.00**
OVERALL RECOVERY TO PRODUCT	**15%**	**93%**	**-**

Source: Hatch, 2025

Table 17-5: Rare Earth Element Recoveries

REE Recoveries (%)														
Y	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu
73	0	0.5	93	93	91	88	85	80	74	72	69	88	76	40

Source: Hatch, 2025

17.2.4 Process Description – Hydrometallurgical Plant

The description of the hydrometallurgical process steps is provided below in the subsequent sections.

17.2.4.1 Acid Mixing and Baking

The feed concentrate is received and treated by sulphuric acid baking to convert the insoluble rare earth minerals to water-soluble rare earth sulphates.

The acid mixing system consists of four pug mill mixers in duty/duty/standby/maintenance configuration. Concentrated sulphuric acid is thoroughly mixed with the concentrate to produce an acid-concentrate paste (the model is configured for 93 wt% H_2SO_4 feed; however, if a sulphuric acid plant is co-located with the plant, the acid concentration can be beneficially increased to 97 wt% H_2SO_4, this will be determined in the next project phase).

Any vapour/off-gas produced during mixing is extracted by a venting system under induced draft. The acid-concentrate mixture is discharged via two screw conveyors to each of the two (duty/duty) acid bake rotary kilns. The kilns operate at 350°C with a 105 min retention time at temperature and produce soluble solid rare earth sulphates, which continue to water leaching. The kiln off-gas passes through a hot cyclone to recover any dust which gets discharged directly to the water leach circuit, before going to off-gas treatment. The kiln is indirectly heated with natural gas burners – the hot exhaust gas from the burners is collected, cooled in air coolers, then directed to the ammonium bicarbonate regeneration unit for CO_2 capture and recycling.

17.2.4.2 Off-Gas Treatment

The acid baking off-gas, containing sulphuric acid fumes (including SO_2 and SO_3), hydrofluoric acid, carbon dioxide, and water vapour is neutralized with slaked lime to prevent the emission of SOx and HF fumes to the environment.

The off-gas is contacted with recirculating water in a venturi/quench system to saturate the vapour and scrub out larger particulate and a cyclonic chamber for crude gas-liquid separation. The gas outlet of the separator is further cleaned in a scrubber, where sulphuric and hydrofluoric acid gases are neutralized with slaked lime. The gas outlet of the scrubber passes through a wet electrostatic precipitator system, then a mist eliminator prior to being vented to the atmosphere.

The scrubber and separator bleeds are fed to a thickener. The thickener overflow goes to a hold-up tank where slaked lime is added before being recirculated back to the off-gas treatment system. The thickener underflow is pumped to a filter feed tank which feeds a filter press where a cake is produced. The solids do not require cake washing but need effective dewatering (membrane squeeze) to produce a cake suitable for disposal. The off-gas residue of calcium sulphate and calcium fluoride solids is disposed of in the hydrometallurgical byproducts storage facility.

17.2.4.3 Water Leaching

The hot kiln discharge containing rare earth sulphates is discharged via gravity into the water leach reactors where the soluble components will dissolve in recirculated gypsum liquor and primary neutralization wash water.

The primary agitated water leach reactors are configured in parallel, such that each kiln discharges into a tank. Additional fresh water is available and may be added to the reactors if required to maintain the target liquid/solids ratio. After the initial pair of water leach reactors, the slurry is cooled indirectly in a heat exchanger, using cooling water to bring down the water leach temperature to 30°C, then returned to the subsequent water leach reactors to ensure sufficient leaching residence time.

At the last reactor, the slurry overflows to the primary neutralization circuit. The water leach circuit recovers any dust and off-gas from vent collection points. The dust and vented gas mixture proceeds to the off-gas treatment area.

17.2.4.4 Primary Neutralization

Magnesium hydroxide, hydrogen peroxide, and ferric sulphate are added to the rare earth sulphate slurry to precipitate thorium, iron, and phosphate impurities.

The primary neutralization circuit reactors overflow in series and are configured such that a reactor can be bypassed for maintenance if required. The water leach solution enters the reactors where it is mixed with magnesium hydroxide slurry to raise pH, hydrogen peroxide to facilitate oxidation of ferrous iron (Fe^{2+}) to ferric iron (Fe^{3+}), and ferric sulphate solution to promote simultaneous removal of iron and phosphorus. Magnesium hydroxide slurry is added to obtain a pH of approximately 2 to precipitate iron, phosphorus, aluminum, and thorium impurities. The air collected from tank vents proceeds to the off-gas treatment area.

The resultant slurry from the last primary neutralization reactor is pumped to a thickener and dewatered. The overflow of the thickener is pumped ahead of the secondary neutralization circuit, and the underflow is pumped to a filter feed tank to a filter press.

Fresh water is used in the filter press to wash the primary neutralization residue in two counter-current washing stages. The filtrate is recycled to the thickener, and the washate is sent to the water leach reactors to recover solubilized rare earth elements and reduce water consumption. The primary neutralization residue containing gypsum, iron and magnesium phosphates, and a small portion of entrained rare earths is collected in portable discharge bins and transferred to the tailings disposal site at the mine. Some rare earth elements are lost in the primary neutralization residue in the form of rare earth hydroxides and as dissolved REEs that are not fully washed from the residue.

17.2.4.5 Secondary Neutralization

The secondary neutralization circuit reactors overflow in series and are configured such that a reactor can be bypassed for maintenance if required. Magnesium hydroxide slurry is added to the reactors to obtain a pH of 6 to precipitate the remaining iron, thorium and aluminum impurities.

The resultant slurry from the last secondary neutralization reactor is pumped to a thickener and dewatered. The overflow from the thickener containing primarily rare earth and magnesium sulphate is pumped to uranium ion exchange and the underflow is recycled to the last water leach reactor to maximize REE recovery, with a portion also recycled to the first secondary neutralization reactor to act as seed.

17.2.4.6 Uranium Ion Exchange

The uranium ion exchange circuit consists of 3 columns, 2 loading columns in Lead-Lag configuration, and a 3rd in resin replacement. The secondary neutralization thickener overflow passes through a cartridge-type polishing filter to capture any entrained solids in solution. The solution passes through the ion exchange columns containing strong base anion resin to remove dissolved uranium from solution. The ion exchange column is operated as a single pass (no resin regeneration) until a maximum residual uranium concentration is reached (breakthrough). Upon breakthrough, the columns switch functions, and uranium-loaded resin is diverted to a spent resin tank and replaced with fresh resin. The spent resin gets pumped to the primary neutralization filter feed tank and filtered out along with the primary neutralization residue. This approach of discarding and replacing loaded resin rather than regenerating the resin avoids the production of a concentrated radionuclide stream which requires costly dedicated disposal.

The uranium-free liquor proceeds to the solvent extraction circuit.

17.2.4.7 Solvent Extraction

The solution from uranium ion exchange (aqueous feed) is sent through a series of mixer-settlers to separate the lanthanum and cerium from the heavier REEs. In the counter-current solvent extraction circuit, the REEs (Y, Pr-Lu) are loaded onto the organic extractant, then stripped with diluted sulphuric acid. The unloaded extractant is then saponified with magnesium hydroxide slurry before recycling to REE loading. The SX circuit includes 43 mixer-settlers, a crud treatment system, a vapour control system, diatomaceous earth and sand filters, and a fire protection foam system. The SX raffinate ($MgSO_4$ solution, containing La and Ce) is sent to the MgR circuit while the strip solution (La and Ce depleted $RE_2(SO_4)_3$) proceeds to rare earth precipitation.

17.2.4.8 Rare Earth Precipitation

The rare earth precipitation circuit reactors overflow in series and are configured such that a reactor can be bypassed for maintenance if required. The strip liquor from SX is reacted with regenerated ammonium bicarbonate in the reactors to precipitate hydrated rare earth carbonates ($RE_2(CO_3)\cdot5H_2O$).

The slurry from the last reactor is pumped to a thickener and dewatered. The thickener overflow of ammonium sulphate solution is pumped through a membrane filter press to recover fine rare earth carbonate particles carried in the overflow solution. The recovered solids are backwashed back to the thickener to increase product recovery, and the filtrate is pumped to the ammonia recovery circuit. The thickener underflow is pumped to a filter feed tank, which feeds the rare earth precipitation filter; a portion of the underflow is recycled to the first rare earth precipitation reactor to act as seed.

Fresh water is used to wash the rare earth precipitate in the filter in two counter-current washing stages. The second stage washate is recycled to the thickener, and the first stage washate is recycled for cake washing. The solids require cake washing to ensure product purity, and effective dewatering (membrane squeeze) to produce a cake suitable for downstream drying.

Ammonium bicarbonate is used as the rare earth precipitant, making carbon dioxide gas a product of the precipitation reaction. Thus, the reactors and thickener must be covered and operated under an induced draft to capture the produced carbon dioxide gas, which is recycled to the ammonium bicarbonate regeneration circuit to reduce the fresh CO_2 requirement.

The rare earth carbonate product is dried in a paddle dryer system, which contains a cooling water-fed product cooler and a dryer vent scrubber. The product exits the dryer system to the automated product bagging system. The product is bagged in 1 tonne heat-resistant lined bulk bags and loaded onto pallets before going to the market. The dryer is indirectly heated with natural gas burners – the hot exhaust gas from the burners is collected, cooled using air coolers, then directed to the ammonium bicarbonate regeneration unit for CO_2 capture and recycling.

17.2.4.9 Magnesium Removal

The solvent extraction raffinate stream, comprising primarily magnesium sulphate, lanthanum sulphate, and cerium sulphate is treated with slaked lime to remove the magnesium as magnesium hydroxide, sulphate as calcium sulphate, lanthanum as lanthanum hydroxide and cerium as cerium hydroxide.

The magnesium removal circuit reactors overflow in series and are configured so that a reactor can be bypassed for maintenance if required. Slaked lime is added to the reactors to raise the pH to approximately 11 to precipitate magnesium hydroxide and calcium sulphate. The slurry from the last reactor containing primarily saturated calcium sulphate solution with suspended gypsum, magnesium hydroxide, and calcium carbonate solids is pumped to a thickener and dewatered.

The thickener overflow solution is cooled with cooling water in a heat exchanger, then stored in the gypsum liquor storage tank. This gypsum liquor is used as a recycled water source for multiple units: Lime slaking, Magnesia slaking, and Water leach. Excess gypsum liquor is directed to the hydrometallurgical effluent treatment plant (Section 17.2.7).

The thickener underflow passes through three parallel filter presses. The filtrate is recycled to the thickener. The solids will not require cake washing but will need effective dewatering (membrane squeeze) to produce a suitable product for disposal. The magnesium removal solids containing calcium carbonate, gypsum, lanthanum hydroxide, cerium hydroxide, and magnesium hydroxide is discharged from each filter and collected in a temporary storage bunker, to be transferred to the hydrometallurgical byproducts storage facility.

17.2.4.10 Ammonia Recovery

The rare earth precipitate thickener overflow solution goes to the ammonia recovery circuit. The circuit is in series and is configured such that a reactor can be bypassed for maintenance if required. Slaked lime is added to the reactors to precipitate calcium sulphate and evolve ammonia gas.

The slurry from the last reactor is pumped to a thickener and dewatered. The reactors and thickener are covered and operated under an induced draft to capture ammonia gas, which is routed to ammonium bicarbonate regeneration. The degassing is conducted at approximately ambient temperature and is assisted by the mechanical agitation of the tanks. The thickener overflow of gypsum-saturated solution, containing remaining entrained ammonia gas, feeds through the ammonia degassing column that strips the residual ammonia off-gas which is also sent to ammonium bicarbonate regeneration. The ammonia-depleted thickener overflow solution is collected in recirculating gypsum liquor tank.

The thickener underflow solids will not require cake washing but will need effective dewatering (membrane squeeze) to produce a suitable product for disposal. This slurry may produce toxic ammonia vapors, and thus the filtration system requires ventilation. The ammonia recovery solids containing mainly gypsum are collected in a temporary storage bunker, to be transferred to the hydrometallurgical byproducts storage facility.

17.2.4.11 Ammonium Bicarbonate Regeneration

Ammonium bicarbonate solution is generated for use in the rare earth precipitation circuit with recycled process gas products and purchased reagents. Gaseous ammonia from ammonia recovery circuit, carbon dioxide gas emitted from the rare earth precipitation circuit, and natural gas combustion products enter the absorption tower along with purchased ammonia solution and fresh water. The absorption tower produces ammonium bicarbonate solution.

17.2.4.12 Reagents and Utilities

17.2.4.12.1 Sulphuric Acid

Sulphuric acid (93 wt% H_2SO_4) is supplied from an across the fence acid plant and stored in two large tanks on site (a H_2SO_4 concentration of 97 wt% may be used, depending on the design of the onsite plant), the acid source will be confirmed in the next project phase.

17.2.4.12.2 Lime

Lime is pneumatically conveyed to the plant from haul trucks or rail. The prepared slurry is stored in an agitator tank before being pumped to the process areas (magnesium removal, off-gas treatment, and ammonia recovery). The grits produced during slaking are directed to the hydrometallurgical byproducts storage facility for disposal.

17.2.4.12.3 Magnesia

The magnesia system is smaller than the lime system but operates very similarly with receipt of powder and slaking with gypsum liquor. The prepared slurry used in the primary neutralization, secondary neutralization, and solvent extraction process areas. The grits produced during slaking are directed to the hydrometallurgical byproducts storage facility for disposal.

17.2.4.12.4 Hydrogen Peroxide

Hydrogen peroxide (50 wt% H_2O_2) is stored in a tank on site. It is supplied by tanker trucks or rail.

17.2.4.12.5 Ammonia

Liquid ammonium hydroxide (29 wt% NH_3 concentration) is delivered to site by tanker trucks or rail. Liquid ammonia is stored in a tank with vent scrubber system to prevent the emission of ammonia vapors to meet environmental emission regulations and ensure worker health and safety. At the volumes required, liquid Ammonia is preferred over anhydrous Ammonia for safety reasons.

17.2.4.12.6 Ferric Sulphate

Ferric sulphate solution (10 wt% Fe(III)) is stored in a tank onsite. It is delivered to site by tanker truck.

17.2.4.12.7 Carbon Dioxide

Carbon Dioxide is kept onsite to be used for ammonium bicarbonate regeneration in upset and start-up conditions. Liquid carbon dioxide is delivered to site by tanker truck or rail. Carbon Dioxide is stored as a liquid in a horizontal refrigerated tank and vaporized to a gas for process use. The vaporizer heats the carbon dioxide with electrical power.

17.2.4.12.8 Solvent Extraction Reagents

Extractant and diluent are delivered to site by truck via isotainers and stored in inventory.

17.2.4.12.9 Flocculants

Magnafloc 10 and Magnafloc 338 flocculants are both supplied in bags and made into concentrated flocculant solution using fresh water. The prepared flocculant solution is stored in a holding tank before being pumped to their respective process areas where they are further diluted with thickener overflow solution to meet the desired concentration.

17.2.4.12.10 Fresh Water

Fresh water is stored in a raw water tank that is insulated. It is assumed that the fresh water is clean and can be used in process without further purification.

17.2.4.12.11 Fire Water

Fire water is stored in the same raw water tank as the fresh water (with priority outlet nozzles). The fire water package includes the main electric firewater pump, diesel backup pump, and electric jockey pump.

17.2.4.12.12 Potable Water

The potable water package filters the fresh water to be used for potable water users. The package includes softening, chemical dosing and filtration.

17.2.4.12.13 Cooling Water

Evaporative cooling tower(s) are located on site to generate cooling water at 20°C. The recirculating cooling water is dosed with minor reagents (to be defined in the next project phase), and a blowdown is bled to hydrometallurgical effluent treatment plant (section 17.2.7) to prevent buildup of impurities in the circuit.

17.2.4.12.14 Gland Water

Fresh water is filtered to produce gland water that is distributed to slurry pumps. In the next project phase, mechanical seals will be considered where possible to reduce water consumption.

17.2.4.12.15 Compressed Air

Compressed plant air is supplied to the process through an air receiver and distributed to the required process areas. Instrument air is supplied from a dedicated compressor system through an air receiver to the plant.

17.2.5 Mechanical Equipment – Hydrometallurgical plant

From the process flow diagrams prepared for the hydrometallurgical plant, a list of all process-related mechanical equipment was produced. For each piece of equipment, the relevant data for preliminary equipment sizing was recorded and used to determine:

1. The equipment power consumption
2. The footprint and building height requirements
3. An estimate of the equipment cost / the necessary design input for a vendor to produce a budgetary quote

A summary of the equipment list is shown in Table 17-6.

Table 17-6: Summary of the hydrometallurgical mechanical equipment list

Equipment Category	Quantity
Tanks	73
Pumps	154
Agitators	35
Bins/Silos/Hoppers	9
Heat Exchangers	6
Fans	8
Conveyors/Feeders	12
Filters	12
Thickeners	6
Mixer-Settlers	43
Settlers	2
Samplers	8
Acid Mixers	4
Rotary Kilns	2
Off-Gas Treatment System	1
Ion Exchange Columns	3
Dryer System	1
Absorption Column	1
Slakers	2
CO2 storage and vaporizer system	1
Cooling tower	1
Potable water system	1
Fire suppression system	1
SX vapour control system	1
Product bagging system	1
Misc. Others	31

Source: Hatch, 2025

17.2.6 Power Requirements – Hydrometallurgical Plant

From the mechanical equipment list, the power consumptions (electrical and natural gas) were compiled for the process equipment in each area. This power usage was used to estimate the utilities cost for the facility and was the basis for the design of the electrical infrastructure. The total electrical power consumption is 6,300 kW, and natural gas consumption is 10,600 kW. A breakdown of the process power consumption is shown in Section 21-19.

17.2.7 Effluent Treatment – Hydrometallurgical plant

The Hydrometallurgical Water Treatment facility is described in Section 18-29.

17.2.8 Air Emissions – Hydrometallurgical plant

The air emissions from the hydrometallurgical plant are treated with various gas treatment systems to ensure their compliance with regulatory emission requirements, then emitted to the atmosphere. Descriptions of the hydrometallurgical air emissions are given in Table 17-7. The design of the gas treatment systems at this project phase is preliminary and will be specified in further detail in the next project phase.

Table 17-7: Summary of Hydrometallurgical Air Emission Streams

Stream	Stream Description	Contaminants removed in treatment	Composition Description
Off-gas Treatment Mist Eliminator – Stack outlet	Acid Kiln off-gas that has had dust and H_2SO_4/HF vapours removed via the off-gas treatment system	H_2SO_4, HF	Air with CO_2 and water vapour
Rare Earth Precipitate Dryer Vent Scrubber – Stack outlet	Product dryer off-gas that has had MREC dust removed via wet scrubbing	MREC particulate	Air with water vapour
Ammonium Bicarbonate Absorption Tower - Stack outlet	Vent gas from the ammonium bicarbonate absorption column. The column is fed with MREC precipitation off-gas, ammonia recovery off-gas, and natural gas combustion exhaust	NH_3	Air with CO_2 and water vapour
Lime Slaking – Wet dust scrubber	Vent air from the Lime slaker, dust has been wet-scrubbed	CaO	Air with water vapour
Magnesia Slaking – Wet dust scrubber	Vent air from the Magnesia slaker, dust has been wet-scrubbed	MgO	Air with water vapour
Ammonia Storage Scrubber – Stack outlet	Vent gas from the ammonia storage tank, equipped with a wet scrubber	NH_3	Air with water vapour
Cooling Tower – Evaporative and drift loss	Evaporative and drift loss from the cooling towers	N/A	Air with water vapour. A small amount of cooling water additives are entrained
Feed Material Dust - Handling system exhaust	Exhaust gas from the systems that handle dust released during concentrate handling/loading.	Concentrate particulate	Air
Product Dust - Handling system exhaust	Exhaust gas from the systems that handle dust released during MREC handling/packaging.	MREC particulate	Air
Organic Vapours From SX plant - Handling system exhaust	Exhaust gas from the systems that handle organic vapours from the SX plant (Mixer-Settlers and tanks are ventilated and connected to a vapour control system to reduce losses)	P507, Exxsol D80	Air

Source: Hatch, 2025

18. Project Infrastructure

The Wicheeda Rare Earths project consists of two project sites: a mine site with a concentrator, and a separate hydrometallurgical plant site.

18.1 Mine Site Off-site Project Infrastructure

18.1.1 Power

Power is supplied via a new high-voltage overhead power line connecting to the BC Hydro 138 kV line (1L 365) west of the project site.

Power line cost has been derived from benchmarks and no detail design has been undertaken. The alignment has not been selected, although a combination of using the access road alignment and more direct routes is likely. Allowances for a site substation and the connection costs at the supply end were made.

At the end of mine life, the power supply will remain in place to support long-term water treatment.

18.1.2 Access Road and Bridge

The mine site is accessed from highway 97 via the Chuchinka FSR 700 RD; 51 km up this road is branch 51A, which provides access to the mine site. The turn off to the Chuchinka FSR is located near Bear Lake, BC. It is assumed that some road upgrading will be required. There are four single lane bridges each with design load rating of 91 t. The last bridge requires repair. A detailed analysis of the road will be completed during the next phase of engineering.

18.1.3 Airports

The closest airport is Prince George Airport (YXS) which is approximately 83 km south of the hydrometallurgical site and a further 52 km to the mine site. Regular bus service will be established from the Prince George Airport to the hydrometallurgical plant and mine site for fly-in/fly-out personnel.

18.2 Mine Site On-site Project Infrastructure

18.2.1 General Site Layout

Refer to Figure 18-1 for the following discussion.

Ore exiting from the pit is sent to either the crusher pad, located west of the pit, or to the operational stockpile located southwest of the pit. The crusher pad facilitates the crushing and transporting of mill feed northwest to the processing plant via a conveyor installed along the crest of the northern embankment of the CWP (Section 18.5.2).

Run-of-mine waste will be hauled to a WSF located adjacent to the pit (Section 16.3).

Three topsoil stockpiles are utilized, storing the topsoil and some overburden stripped during site development. On the east side of Wichcika Creek, there is one adjacent to the operational stockpile and another adjacent and below the concentrator. On the west side of the creek, the third stockpile is situated close to the tailings storage area.

Tailings generated by the flotation concentrator are transported to a filtration plant on the western side of Wichcika Creek via a pipeline and then filtered and stored in the FTSF (Section 18.4).

Explosive storage facilities infrastructure are located north west of the concentrator.

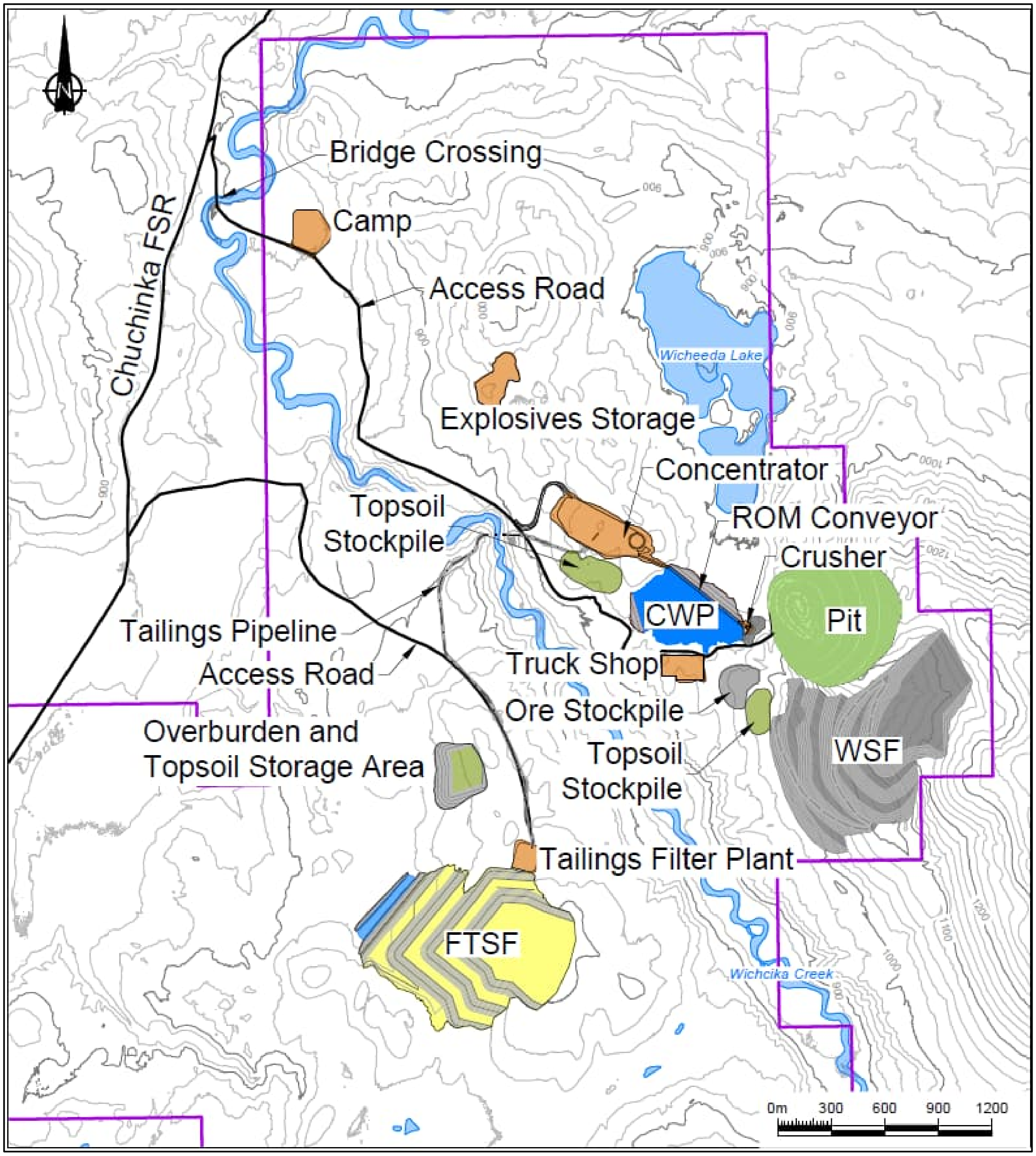
18.2.2 Site Facilities

Mine support infrastructure includes:

- Truck shop / mine dry / warehouse building
- Diesel fuel storage and dispensing facility
- Power line to these facilities.

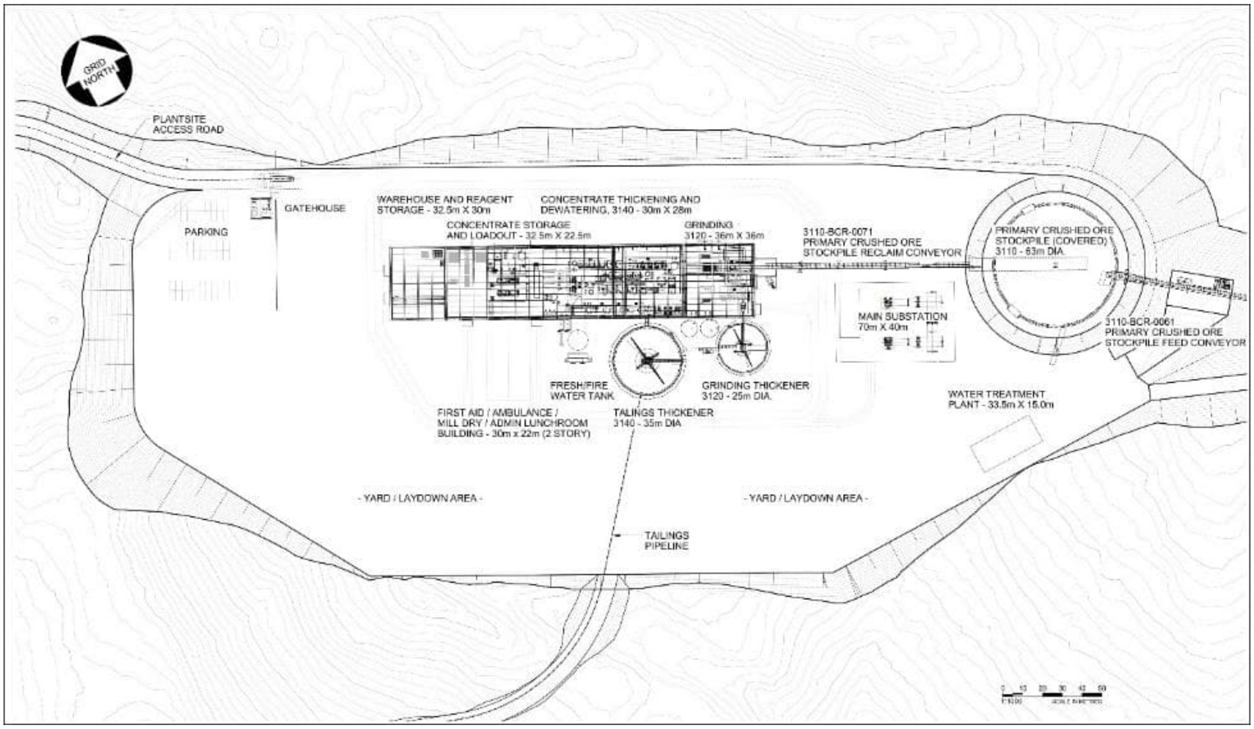
A concentrator plant site is located on the hill northwest of the pit and CWP and west of Wicheeda Lake. The plant site has been designed to ensure water runoff reports to the CWP. A water treatment plant (Section 18.5.6), administration offices and other buildings are also located at the plant site. The facilities include:

- Mill dry
- Offices
- Maintenance building and shops
- Warehousing
- Reagents storage
- Concentrate container load out.
- Gatehouse
- First aid/emergency response
- Administration
- IT infrastructure, communications
- Process, fire, and potable water storage and distribution systems
- Main HV power line and substation and emergency electrical power backup
- Accommodation camps are located off Branch 51A. The installation starts with a pioneer camp, followed by a construction camp, which is converted to an operations camp. A helicopter pad will be located near the camp.



Source: SRK, 2025

Figure 18-1: Mine Site Layout



Source: Hatch 2025

Figure 18-2: Concentrator Plant Layout

During closure mine support infrastructure will be removed, with the exception of buildings and powerlines that are required to support long-term water treatment. The pads associated with the removed infrastructure will be regraded and seeded.

18.2.3 Mine Site Roads

The site layout takes advantage of existing logging road access to the mine site. Logistics studies of the roads will be done during the next phase of the project.

New gravel roads around the mine site are included in the design for access and maintenance purposes.

At closure mine site roads will be scarified and seeded, with the exception of the roads needed to support long term-water treatment and monitoring.

18.2.4 Accommodation

There is a 100-bed pioneer camp and a 400-bed construction camp. The 400-bed construction camp will be renovated and converted to the permanent operations camp after the completion of construction.

In the 400-bed construction camp and future operations camp, each room will have its own washroom and shower. Recreation facilities will be provided as part of the camp.

18.2.5 *Mine Maintenance Facilities*

A mine maintenance building is installed to support maintenance of 60 t haul trucks. The truck shop supports tire changes and regular maintenance of haul trucks. Integrated with the truck shop are the mine dry, mine offices, and mine warehouse. This facility is located close to the mine to reduce haul truck travel time.

18.2.6 *Water Supply*

Process water for the concentrator and mine comes from the CWP, which collects all the process water and mine drainage. The CWP will also capture runoff from the mine and impacted areas. Runoff from the FTSF will be collected and pumped to the CWP.

Potable water is trucked to site and stored in potable water storage tanks.

Two wastewater treatment plants treat domestic sewage prior to release.

18.2.7 *Electricity & Communications*

A new 138 kV power line runs from the McKeown substation near Bear Lake to the mine site. A new substation is installed to distribute power at 13.8 kV for mine site, mineral processing, tailing storage facility, and camp.

An allowance has been included for fibre optic line from the mine site back to the main highway. Allowances have been made for IT fit-out of the mine site.

18.2.8 *Diesel Fuel & Propane Gas*

There is a centralized diesel fuel storage and distribution facility consisting of a 400 m^3 diesel storage tank with containment and fueling station for mine trucks as well as small vehicles. This is located close to the open pit mine.

18.3 Mine Site Waste Rock Storage Facility

18.3.1 *WSF Design*

Run-of-mine waste will be hauled to a WSF located adjacent to the pit. The design of the WSF is described in Section 16.3.

18.3.2 *ML/ARD and Radionuclide Potential of Waste Rock*

Characterization of the metal leaching and acid rock drainage (ML/ARD) potential of waste rock is described in Section 20.1.1. In summary, waste rock has low potential for ARD. It is expected to generate mildly alkaline drainage and have potential for leaching of several parameters above applicable water quality guidelines.

The occurrence of radionuclides in waste rock is described in Section 20.1.1. In summary, the radionuclide content of waste rock combined in the waste rock storage facility exceeds the unconditional derived release limits for diffuse NORM sources for solids. As such, a site review will be required to determine NORM management program requirements.

18.3.3 *Closure Design*

Closure of the WSF occurs progressively during operation and is completed in the active closure phase at the end of mining. The WSF slopes are regraded to an overall slope of 2H:1V to 2.5H:1V, and covered with a growth medium cover in the order of 1 m thick. Cover material is sourced from the overburden stockpile. Once the growth medium is in place the

WSF is revegetated through a combination of hydroseeding, broadcast seeding and seedling planting. No other cover material or liner is assumed to be required.

Seepage from the base of the WSF is collected in a collection ditch and conveyed to the CWP. Water from the CWP is treated prior to discharge.

As in the operations phase, during closure, seepage from WSF is collected and conveyed to the CWP before being treated and discharged. The current closure plan assumes perpetual water management and treatment at the mine.

18.4 Mine Site Filtered Tailings Storage Facility

18.4.1 Overview

Tailings generated in the flotation concentrator are transported via pipeline to a tailings filtration plant and dewatered to a filter cake before being hauled, placed and compacted in a lined FTSF. The filtration plant is located adjacent to the FTSF to minimize haul distances, and both are located west of Wichcika Creek. The water leach (WL) primary neutralization (PN) wastes (collectively termed WL/PN residues) and loaded uranium ion exchange resin produced at the hydrometallurgical plant are also stored in the mine site FTSF. These wastes are dewatered via filtration at the hydrometallurgical plant and hauled by truck to the FTSF before blending with the dewatered flotation tailings.

18.4.2 Site and Technology Selection

A preliminary alternatives assessment was completed to evaluate potential tailings storage sites and tailings storage technologies. Both conventional slurry tailings (‘wet’ options requiring a containment dam) and non-conventional filtered tailings (‘filter stack’ options that are self-supporting) were assessed to bracket the range of tailings management technologies. Potential storage sites within an approximate 10 km radius of the mine were considered, including the wet slurry storage location identified in the 2023 PEA study (SRK, 2023).

At least eight wet tailings options and three filter stack options were identified. These were screened to four potential options as follows:

- Two wet tailings options – one at a newly identified valley storage location approximately 8 km west of the mine, and the other being the PEA wet tailings option on the western side of Wichcika Creek.
- Two filter stack options – one on the hillside close to the proposed concentrator location, and one at the PEA wet tailings option location.

Based on preliminary cost estimates completed as part of the alternatives assessment, the filtered tailings options had been deemed unfavourable due to high OPEX and the wet slurry tailings option 8 km west of the mine had been initially carried forward given its promising economics. However, as the project advanced the costs of the 10 km pipeline to this location were refined and increased while the OPEX of the filtered tailings option at the PEA tailings site were optimized by placing the tailings filtration plant adjacent to the stack. The filtered tailings approach at the PEA location was also evaluated as having several other benefits over the wet tailings option, including: ability to implement progressive closure; reduced post-closure risks (smaller catchment, no dam or spillway), reduced consequences of failure and potential to be viewed more positively by the regulator, community, First Nations and other stakeholders (as building a landform rather than a dam). Based on these outcomes a decision was made by the project team to adopt filtered tailings with a filter stack at the previous PEA TSF site as the preferred option for this PFS.

18.4.3 Tailings Geotechnical Properties

Two samples of flotation tailings have been tested for its geotechnical properties (SGS, 2021, 2024). Results indicate the tailings are a non-plastic sandy silt, with fines content (0.075 mm or smaller) between 64 to 80% of which 4 to 6% are clay-sized (0.002 mm or smaller). A proctor compaction test indicates the material has a maximum dry density of 1.99 t/m^3 at an optimum geotechnical moisture content of 11.2 % by mass. The specific gravity of the material is around 3.0.

Two samples of PN residue were tested for their particle size distribution (SGS, 2024). The results indicate the material is slightly finer than the flotation tailings, with 85 to 87% fines content and 8 to 11% clay-sized particles. Atterberg limits and proctor tests were not completed on this material. Geotechnical strength testing has not been completed on either material.

18.4.4 Tailings ML/ARD and Radionuclide Potential

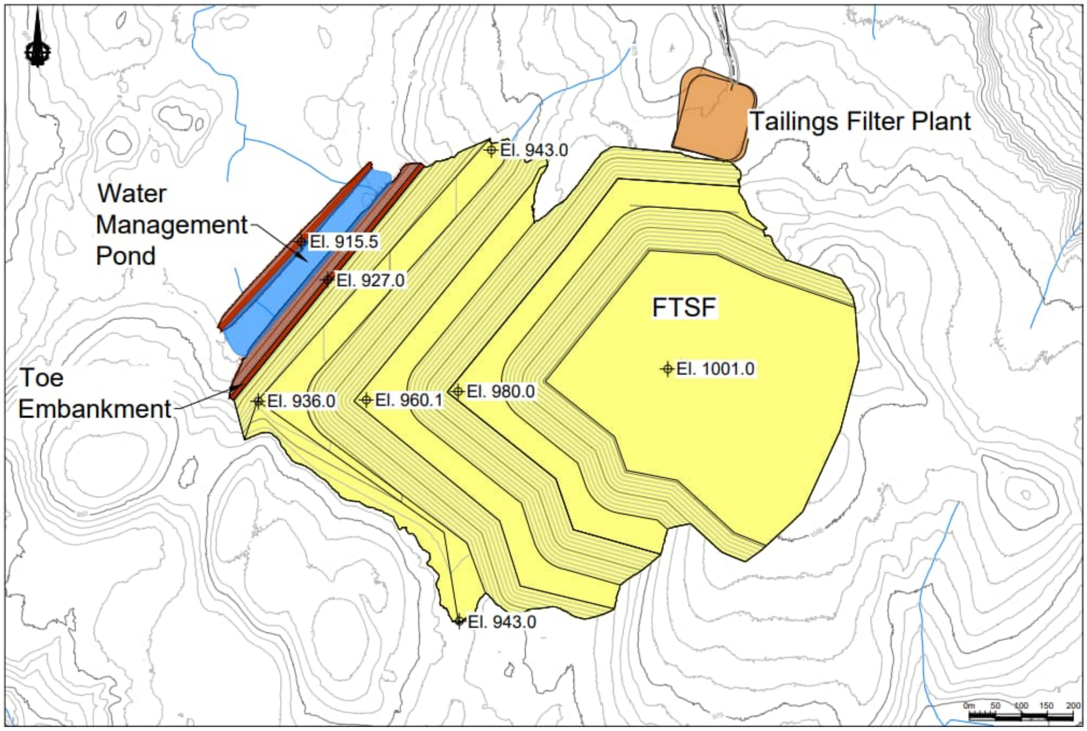
Characterization of the ML/ARD potential of tailings (and co-disposed WL/PN residue) is described in Section 20.1.1. In summary, tailings and co-disposed WL/PN residue have low potential for ARD. Well-blended, they are expected to generate mildly alkaline drainage and have potential for leaching of several parameters above applicable water quality guidelines.

The occurrence of radionuclides in tailings and WL/PN residue is described in Section 20.1.1. In summary, the radionuclide content of tailings and co-disposed WL/PN residue exceeds the unconditional derived release limits for diffuse NORM sources for solids. As such, a site review will be required to determine NORM management program requirements.

The potential for leaching of uranium from loaded uranium ion exchange resin, or its radionuclide content, has not been characterized.

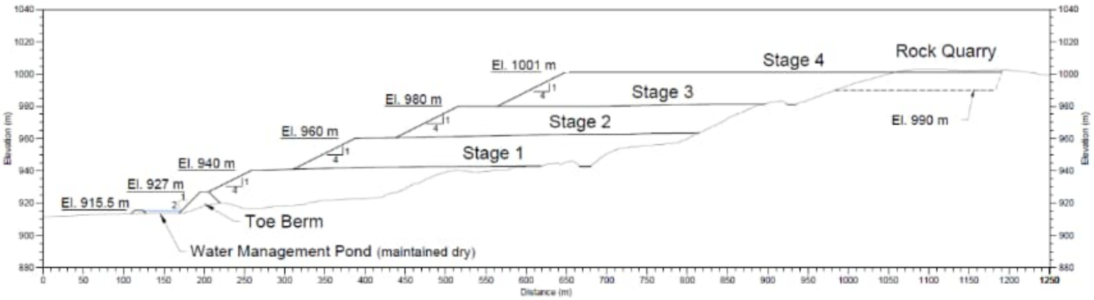
18.4.5 FTSF Design

The FTSF is designed as a valley-fill ‘stack’ consisting of four main levels plus a starter toe embankment and water management pond downstream. Construction commences at the valley bottom and progress up the hillslope, with each level representing a stage as follows: Stage 1 (elevation 943 m), Stage 2 (elevation 960 m), Stage 3 (elevation 980 m), and Stage 4 (elevation 1,000 m). The total height of Stage 1 is approximately 25 m, including the toe embankment with a maximum height above natural ground of 15 m. Each lift is offset from the previous by a 50 m bench width and the slope between benches is 4H:1V. The ultimate height of the stack is approximately 85 m tall from crest to toe, and the overall slope angle is approximately 5.6H:1V or 10 degrees. Figure 18-3 shows a layout of the FTSF design and Figure 18-4 shows a typical section.



Source: SRK, 2025

Figure 18-3: FTSF Layout



Source: SRK, 2025

Figure 18-4: Typical FTSF Section

There is currently very limited data on the foundation conditions of the FTSF area. Based on aerial imagery and preliminary terrain mapping, the footprint of the FTSF is assumed to mostly comprise till, till veneer and bedrock outcrop, which are characteristic of the hillslopes in the area. However, at the toe of the stack is a flat, bare area that's been described as a 'marshland' zone. The area is visually wet in spring and summer. Ground conditions are assumed to be poor, consisting of organics or soft deposits, and their thickness is currently unknown. The relatively conservative slope geometry of the stack was therefore adopted to account for this uncertainty in foundation conditions. The design also assumes foundation removals in this area to an average of 3 m deep but up to 5 m in the toe embankment footprint and replacing up to two-thirds of the excavated material with rockfill to maintain the base of the stack above any groundwater.

The FTSF stack has storage capacity for just over 16 million cubic meters (Mm^3) of compacted tailings. Based on the mine plan, approximately 25 million metric tonnes (Mt) of flotation tailings and 0.8 Mt of WL/PN residue are expected to be produced over the 15-year mine life (on a dry basis). Adopting a conservative placed and compacted dry density of 1.8 t/m^3, the equivalent storage volume required is 14.4 Mm^3. About 90% of the required storage capacity is provided by Stages 1 to 3 with potential optimization to include 100%. Stage 4 is included as contingency and may not be required. The volumes per stage are summarized in Table 18-1.

Table 18-1: Filtered Tailings Storage Facility Storage Volumes per Stage

FTSF Stage	Elevation (masl) at End of Stage	Storage Volume per Stage (Mm^3)	Years of Capacity (assuming full production)
1	943	4.6	4.8
2	960	4.5	4.7
3	980	4.1	4.2
4	1,000	3.2	3.3
Totals		**16.4**	**17.1**

Source: SRK, 2025

Due to the uncertainties in the hydrogeological conditions of the site and the anticipated geochemistry of the tailings solids, including co-disposal with the hydrometallurgical residue, the facility is currently planned to include an impermeable geomembrane liner system in the stack foundation and on the upstream face of the toe embankment. The liner is assumed to be textured and a minimum 1.5 mm (60 mil) thick. The total footprint area to be lined is approximately 70 hectares (ha), with Stage 1 comprising 50% of the total area and each other stage in the order of 15 to 18% each.

The low-permeability liner incorporates both below-liner and above-liner drains to mitigate pore pressure build-up from either groundwater artesian conditions (if present) or excess water in the tailings. The above-liner drains flow to a toe drain along the upstream toe of the toe embankment. This toe drain includes several sumps (above-liner) that are accessed from the toe embankment crest using riser pipes and drained with mobile pumps. Assuming filter performance and compaction in the field generally meets design targets, excess water and seepage in the tailings are expected to be minimal. The liner and above-liner drains provide containment and collection if any excess water and seepage occurs.

A conceptual 2D model of the stack and foundation conditions was developed for preliminary stability assessment. The model included a range of assumptions for the lateral extent, thickness and strength of the marshland zone, as well as the liner interface and a perched phreatic level in the tailings above the liner. The results indicate the stack is stable and meets an appropriate factor of safety to slope failure when the soft/organic marshland layer is less than 5 m thick. However, this assessment is considered screening level only and shall not be relied upon until the foundation conditions are characterized and laboratory strength testing of the tailings, foundation and construction materials (including liner interface shear strength) has been completed. The sizing of the toe embankment and design geometry of the FTSF stack will also need to be re-visited once these conditions are better understood.

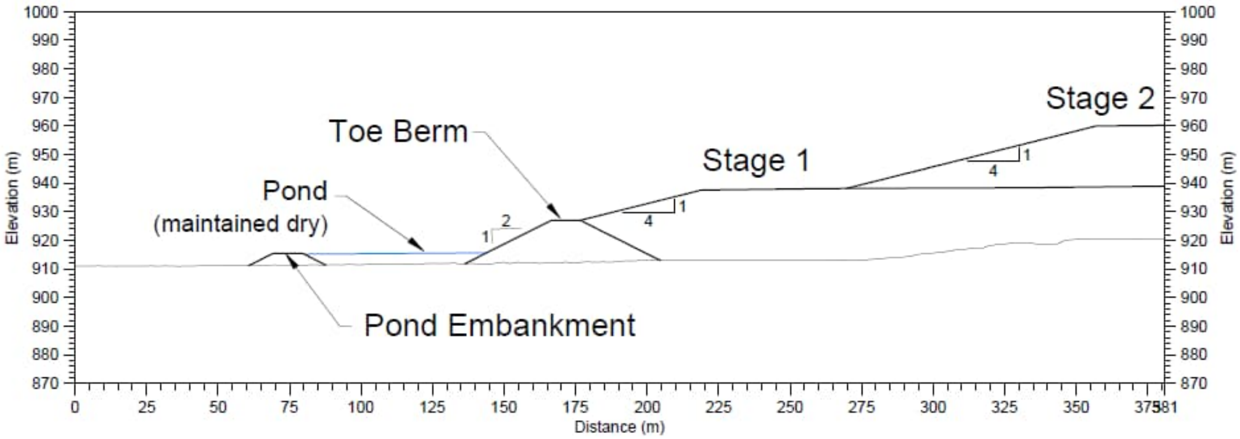
18.4.6 Water Management

The water management design for the FTSF includes surface water channels for diverting non-contact water around the facility, a water management pond at the toe of the facility for collecting runoff from the stack and operating areas, and internal drains for managing groundwater in the foundation or excess water or seepage in the stack. The surface water diversion channels are developed for each stage of the FTSF, as well as permanent channels for closure.

During operation, the surface of the compacted filtered tailings is maintained with a minimum 0.5% grade towards the back of the facility and away from the slopes of the stack. This will enable surface water runoff on the stack to be managed at the back of the facility, where it can be collected in temporary sumps and transferred to the water management pond using mobile pumps. All runoff on the stack is assumed to be ‘contact’ water and may include suspended tailings solids. Contact water runoff and any water collected in the internal drains is pumped to the water management pond using mobile pumps before being pumped to the mine’s CWP for treatment and discharge.

The FTSF’s water management pond provides containment of contact water runoff during storm events. During normal operating conditions the pond is expected to be dry as minimal excess water or seepage is expected in the dewatered tailings. The pond is sized to store an environmental design flood (EDF) equivalent to a 200-year, 72-hour storm falling on the entire FTSF catchment and assuming 100% runoff. The pond is formed by a small embankment (maximum height in the order of 5 m above natural ground) approximately 50 m downstream of the FTSF toe embankment. A typical section through the pond is shown in 18-5. Both the upstream face of the pond embankment and a portion of the FTSF toe embankment downstream slope is lined to contain the pond, with the liner keyed into the foundation. A liner is not currently envisaged for the remaining pond footprint as the foundation is anticipated to be low-permeability material. The return water pipeline and pump from the FTSF water management pond to the CWP is sized assuming a pond at full capacity is removed within 7 to 10 days. An allowance is made for other contact water management requirements (e.g., temporary sumps, mobile pumps).

The design of the CWP and surface water channels is described in Section 18.5.2 and Section 18.5.3, respectively.



Source: SRK, 2025

Figure 18-5: FTSF Water Management Pond Typical Section

18.4.7 Construction and Operation

Construction of the FTSF involves foundation clearing and grubbing, foundation excavation and replacement with competent fill in areas of poor ground conditions, construction of internal drains and surface water diversion channels, installation of a liner system, and construction of the toe and water management pond embankments. A quarry at the top of the hill above the FTSF is developed to provide rockfill for drains and the toe and water management pond embankments, as well as rock cladding during operation. Approximately 1.4 Mm^3 of rockfill is required for the stack over the life of mine, equivalent to 1 M bank cubic meters (BCM). Till may also be used for the embankments depending on its availability and suitability.

Stripped topsoil and unsuitable material from foundation over-excavation are stockpiled in an area approximately 1 km north of the FTSF. The temporary stockpile is located within the same catchment as the FTSF and has capacity for the approximately 1 Mm^3 of excavated material expected over the life of the facility with capacity for more if required. The stockpiled material is used for progressive reclamation and closure of the stack.

During operation of the FTSF, filtered tailings are loaded into trucks at the filter plant and hauled to the stacking area before being dumped, spread with a dozer into approximately 300 mm thick layers, and compacted with a roller. At full production, the total amount of material to be moved per day is in the order of 5,600 tonnes (on a wet basis). Since the tailings will be filtered at or close to their optimum moisture content for compaction, minimal moisture conditioning is expected. During wet periods, filtered tailings may be temporarily stockpiled at the plant or on the FTSF until they can be spread and compacted. In winter, snow clearing is required to prevent ice entrainment. It is currently assumed that all filtered tailings placed in the stack will be compacted to 95% of maximum dry density to achieve a stable, non-liquefiable, self-supporting landform. However, future phases of design and testwork will investigate whether a zoned approach is feasible, whereby the stack could accept zones of less compacted material without compromising stability. The zones of less compacted material would allow for more flexibility and placement when material is not at optimum moisture content.

As the FTSF grows in height, the outer slopes are clad with run-of-quarry rockfill to provide environmental containment and prevent erosion of the face. The rock cladding is assumed to be built as a small berm on the outer face, with a minimum thickness of 1 m perpendicular to the slope and raised continuously as the stack level increases. Crushed rock may also be required for internal haul routes on the stack to assist with dust management. An allowance for crushed rock is made assuming two primary haul routes along the longest length of the stack, with a new road every 1 m in stack vertical height, and a road fill thickness of 0.3 m. The haul routes will be temporary and covered as the tailings are placed. Additional dust control measures such as water trucks and or soil tackifiers or stabilizers may be required.

As described in the FTSF Design (Section 18.4.4), hydrometallurgical residue will be combined with the flotation tailings. Approximately three percent (3%) of the FTSF capacity accounts for hydrometallurgical WL/PN residue (on a dry mass basis). The residue, in the form of moist filter cake, is trucked from the hydrometallurgical plant to the covered temporary storage area at the tailings filter plant and added to the flotation tailings filter cake material before trucking to the FTSF. Given the design moisture content of the WL/PN filter cake is higher than the flotation tailings filter cake (approximately 43% geotechnical moisture content versus 10%), the actual ratio on a total mass basis is closer to 4%. At a 4% total mass ratio, this equates to one loader bucket of WL/PN (about 7 to 8 tonnes) every four to five truck loads (assuming 40-tonne trucks) of flotation tailings. No other mechanical mixing is envisaged; however, trucks carrying a bucket load of WL/PN residue will have restrictions on where they can place the material (e.g., away from the final edges or slopes of the stack).

18.4.8 FTSF Scheduling

Construction of each FTSF stage is assumed to occur in the year prior to it being required for placement of filtered tailings. Therefore, construction activities are spread out over the life of mine and occur every 3 to 4 years. Stage 1 requires the largest construction effort, and for costing purposes it is assumed it will mostly be completed using a contractor. Although construction for subsequent stages is generally more straightforward, it is assumed they will also be completed using a contractor. All lining and development and operation of the quarry is assumed to be done by contractor.

The big drivers of initial CAPEX for the FTSF during pre-production are the foundation preparation requirements (including excavation of unsuitable material, backfill and liner placement) and toe embankment construction for Stage 1. To delay some of these costs, SRK evaluated a smaller ‘starter’ stack upslope of the initial development that avoids most of the area identified as having potentially poor ground conditions. The starter stack could be built in a similar fashion to the ultimate stack i.e., in approximately 20 m high levels with 50 m wide benches and 4H:1V bench slopes. If the starter stack was built to three levels (943 m, 960 m, and 980 m), it could provide approximately 4 to 5 years of initial tailings storage and delay some of the larger Stage 1 CAPEX requirements. The starter stack would still require a toe embankment (albeit smaller than the Stage 1 toe embankment) and rockfill cladding on the slopes, which add to total costs, but the deferral of CAPEX provides a net benefit to the NPV.

The opportunity for the starter stack is currently the basis of the capital cost estimate but will require further assessment to demonstrate it is viable.

18.4.9 Monitoring

The stack includes geotechnical instrumentation to monitor pore pressures and any potential deformation. Instrumentation includes piezometers, shape arrays, and inclinometers as a minimum and will be installed at each stage of the FTSF. It is also anticipated that prior to advancing to the next stack level, a cone penetration test (CPT) program will be completed in the tailings to confirm design parameters are being achieved. Allowances have been made for instrumentation and CPT programs in the cost estimate.

18.4.10 Closure Design

Implementation of the closure design occurs both progressively during operation and in the active closure phase at the end of mining. The FTSF stack surface, including slopes, will be covered with a growth medium cover in the order of 0.5 to 1 m thick, with material sourced from the topsoil stockpile. Note that the slopes will be clad with rockfill during operation, so the final slope cover thickness will be in the order of 1.5 to 2 m thick. It is assumed that benches and the final top of the stack will be covered only with the growth medium. Once the growth medium is in place the stack surface will be revegetated through a combination of hydroseeding, broadcast seeding and seedling planting. No other cover material or liner is assumed to be required.

The final surface of the stack is designed to promote surface water runoff at closure. It is envisaged that the final grading plan will generally be achieved during operation and development of the stack, thereby minimizing the need for regrading at closure. Surface water channels with riprap erosion protection will be constructed in specific locations, including around the stack as well as on benches and some slope faces. Clean runoff from the covered stack post-closure will be directed into the clean water diversions that flow around the water management pond.

As per the operation phase, seepage from the stack is anticipated to be minimal since the tailings are dewatered via filtration to a relatively low moisture content before being placed and compacted. The foundation liner and internal drains will provide containment and the ability to capture any excess water or seepage within the tailings if it occurs. The FTSF water management pond will remain in place to collect runoff from the stack during storm events, with contact water pumped to the mine’s CWP for treatment and discharge. The current closure plan assumes perpetual water management and treatment at the mine, although the FTSF water management pond may be removed if it can be demonstrated that water runoff from the FTSF landform meets water discharge criteria.

Section 20.5 describes the general closure plan for the mine. Water treatment requirements are described in Section 18.5.6.

18.4.11 Tailings Filtration Plant

The tailings filtration plant is situated on relatively flat ground close to the top of the catchment where the FTSF is located and at an elevation near the ultimate height of the FTSF. Siting the plant at this location allows the majority of filtered tailings haul to be downhill loaded as opposed to uphill loaded and minimizes haul costs. There is a small positive pump head from the concentrator to the plant but this is outweighed by the haul cost savings.

The tailings delivery pipeline from the concentrator to the filtration plant will cross Wichcika Creek at an elevation that allows it to be drained to a lined emergency pond during shutdowns for maintenance or emergencies. The creek crossing includes double containment to ensure line leaks will not result in any tailings entering the creek. A new road from Chichunka FSR to the filter plant is developed, utilizing existing unused access roads where possible.

The design of the filtration plant is described in Section 17.1.2.10.

18.5 Mine Site Water Management

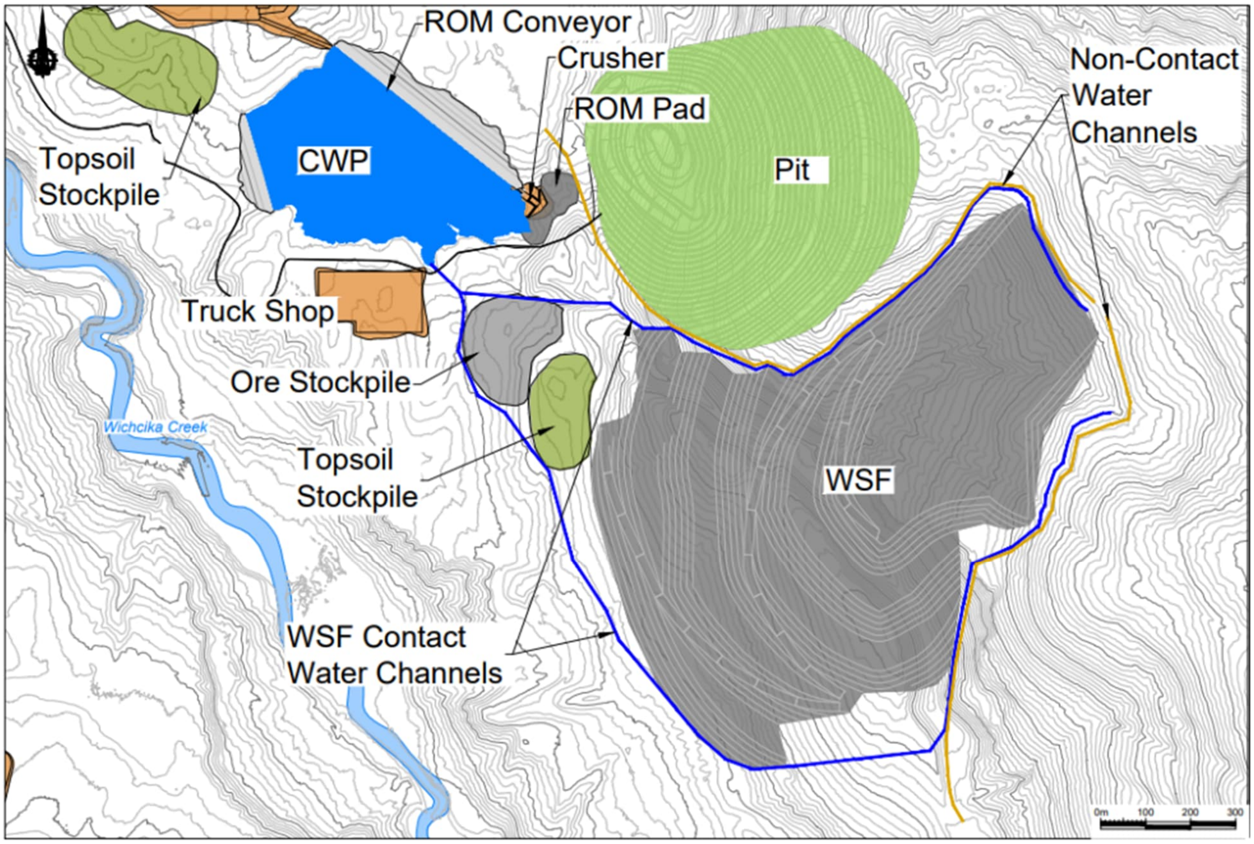
18.5.1 Overview

The Wicheeda project consists of infrastructure on the east and west extents of Wichcika Creek, and upstream of Wicheeda Lake. Water management infrastructure is required to capture the surface water runoff and any potential seepage (hereby termed ‘contact’ water) from the open pit, WSF, FTSF and stockpiles, as well as to divert non-contact water around these facilities where possible.

The CWP is located adjacent to the process plant to store all site contact water before it is sent to the process plant or to the water treatment plant (WTP) for treatment and discharge into Wichcika Creek. Contact water from the waste rock dump and mine stockpiles flows to the CWP via a collection channel as well as natural drainages. Contact water from the pit and the FTSF water management pond is pumped to the CWP.

18.5.2 Contact Water Pond

The WSF is sited to ensure all runoff and seepage water can be captured at a single collection point, downstream of the open pit, WSF and stockpiles. This downstream location is where the CWP is constructed as shown in Figure 18-6.



Source: SRK, 2025

Figure 18-6: Mine Site Water Management

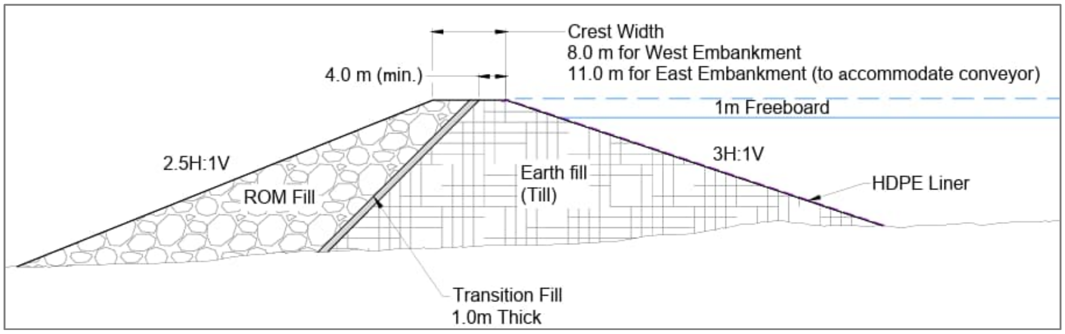
The CWP is formed by two cross-valley embankments; one to the west and one to the east. It is situated at a topographical low immediately below the process plant area and straddles a catchment divide: to the west is Wichcika Creek and to the east is Wicheeda Lake. The West Embankment is approximately 230 m long with a maximum height of 20 m. The East Embankment is 640 m long with a maximum height of 27 m. The southern end of the East Embankment also ties into the proposed mine crusher pad and will incorporate a conveyor from the crusher and along its crest to bring crushed ore up to the process plant. The pond is fully lined with geomembrane up to its crest level, including a berm around the crusher pad. The total pond area is approximately 21 ha.

The CWP is designed to provide storage capacity to manage the combined monthly inflow volumes from the WSF and the FTSF during a 1 in 25-year wet freshet condition, with surge capacity to store the 1 in 200-year 72-hour rainfall storm event from the direct catchment area. The design maximum capacity requirement of the CWP is 1.1 Mm^3, not including freeboard. The CWP embankments are designed to crest elevation 934.5 m and include 1 m of freeboard. The storage capacity at 933.5 m is approximately 1.12 Mm^3 and the capacity up to its crest level is 1.28 Mm^3.

A dam consequence classification of ‘High’ has been cautiously adopted using Canadian Dam Association (CDA) guidelines, although future studies may demonstrate a lower classification is reasonable. For a ‘High’ classification, CDA requires the inflow design flood (IDF) to be one-third between the 1,000-year event and the probable maximum flood (PMF). Runoff from the IDF into the CWP can be contained within the 1 m freeboard allowance. An emergency overflow spillway is currently not envisaged.

Based on geotechnical investigations in the CWP footprint, overburden (till) is relatively thick, ranging from 5 to 20 m. The embankments are designed to maximize local borrow from within the impoundment footprint and minimize the need for mine rock from pit development during pre-production, which has a long haul and higher cost. Borrowing from within the impoundment also adds storage capacity and allows optimization of the embankment volumes.

A typical embankment section is illustrated in Figure 18-7. Local till borrow is utilized for the bulk of the embankment volume and builds the upstream side, with an upstream slope of 3H:1V to facilitate liner installation. Rockfill is required on the downstream slope for added strength and armouring, with a transition layer (minimum 1 m thick) of crushed or select rock between the till and run-of-mine rockfill. The downstream slope is designed at 2.5H:1V. The crest width of the West Embankment is 8 m to allow for light vehicle access, safety berms and pipelines if required. The crest width of the East Embankment is increased to 11 m to facilitate the conveyor.



Source: SRK, 2025

Figure 18-7: CWP Typical Embankment Section

Table 18-2 summarizes the material volumes required for the CWP embankments. The design currently assumes the central high ‘island’ within the impoundment can be excavated relatively flat and be used for borrow. This provides approximately 137,000 m^3 of fill or about 40% of the till volume required. The remainder is sourced from within the impoundment footprint. Transition and run-of-mine rockfill comes from pit development.

Table 18-2: CWP Embankment Construction Volumes

Component	Material Volumes Required (m^3)		
	West Embankment	East Embankment	Totals
Till	46,110	312,950	359,060
Transition Rockfill	2,990	13,420	16,410
Run-of-Mine Rockfill	27,100	187,650	214,750
Totals	**76,200**	**514,020**	**590,220**

Source: SRK, 2025

Inflows to the CWP are pumped to the processing plant or treated and discharged to Wichcika Creek. Water collected in the open pit is also directed to the CWP, along with runoff from the processing plant pad, ore stockpile, waste rock ditches, and dry stack ditch. The dewatering system from the CWP to the water treatment plant was sized based on a monthly water balance and the 1 in 25-year monthly runoff inflows, calibrated to a regional Water Survey of Canada Meteorological station (WSC Station 07EE009 Chuchinka River at the Mouth). A pumping system equipped to discharge up to 520 m^3/h to the water treatment plant is installed with an 18” DR17 HDPE pipeline to the plant site area. Based on this pumping capacity, the 200-year 72-hour storm volume could be dewatered to the treatment plant in 12 days.

The ML/ARD potential of tailings and the chemistry of process water is described in Section 20.1.1. Tailings are expected to have low potential for ARD but some potential for metal leaching. During operations reclaim water from the tailings filter plant is expected to be dominated by process water chemistry. Process water is expected to contain elevated fluoride from processing reagent use, and elevated molybdenum from reaction between process reagents and mineralized rock. As such, the CWP is lined with an HDPE liner to limit infiltration and seepage. Seepage collection sumps are planned at the downstream toe of each earth dam for the CWP. Any seepage water or runoff collected in these sumps is pumped back into the CWP. Three sump stations are planned.

The CWP is expected to remain in place post-closure to support perpetual water treatment of WSF and FTSF seepage.

18.5.3 Filtered Tailings Storage Facility Water Management

Runoff and any potential seepage from the FTSF are collected at a water management pond at the downstream toe of the stack and pumped to the CWP. Further details of the water management plan at the FTSF are described in Section 18.4.6.

18.5.4 Collection Channels and Diversions

Open channels are situated downstream of proposed infrastructure to collect contact water, as well as upstream of infrastructure to divert non-contact water. All channels were designed to convey the 200-year return period 24-hour peak flow with an additional freeboard of 0.3 m using a PCSWMM model to calculate peak flows, assuming a Curve Number (CN) value of 73 with an additional 0.3 m of freeboard.

Channels will have a varying base width from 0.5 m for the most upstream channels, to 2 m for the most downstream; and side slopes of 2:1 H:V. The channels are lined with an HDPE geomembrane, a non-woven geotextile and a layer of riprap erosion protection. Access roads along the downgradient channel bank are constructed for channel inspection and maintenance.

Three channels are planned to manage runoff:

- Contact water from the WSF is conveyed to the CWP. A portion of the facility drains by gravity directly to the pond while the majority drains into the South Collection Channel which follows to the west toe of the WSF and discharges in the CWP.
- Non-contact water upstream of the WSF is diverted in the Waste Rock Diversion Channel, which will discharge towards Wichcika Creek. The Waste Rock Diversion Channel consists of two open-channel segments and a pipeline conveying water through the steeper section of the hillslope. The pipeline section is 160 m in length and consist of two parallel 1000 mm diameter HDPE pipes, which convey water down the steeper section of the hillslope.
- Non-contact water upstream of the FTSF area is diverted in the FTSF Diversion Channel, which will discharge around the FTSF back to the natural drainage.

Upstream non-contact water channels will remain in place and continue to convey water around main features at closure.

18.5.5 Open Pit Dewatering

Inflows to the open pit includes surface runoff and groundwater. All water within the pit footprint is collected in the floor of the pit and pumped to the CWP. The dewatering system was sized to dewater the 100-year 24-hour rainfall event inflow volume over a seven (7) day duration. Costs assume two 230 m^3/h pumping systems are in-place with a 12” DR17 HDPE pipeline to the CWP along the pit access ramp alignment.

Characterization of groundwater inflows to the open pit will be developed in further studies. It is assumed that during the design event for pump sizing, groundwater inflows would represent a small fraction of total inflows and would not significantly change pumping requirements.

18.5.6 Water Treatment Plant

The water quality of process water and water contacting the operational stockpile, WSF, pit walls, and FTSF was estimated through development of geochemical source terms that synthesized data from metallurgical test work, data from static and kinetic geochemical characteristics of each mine waste material, mine waste facility geometries, and climatic information.

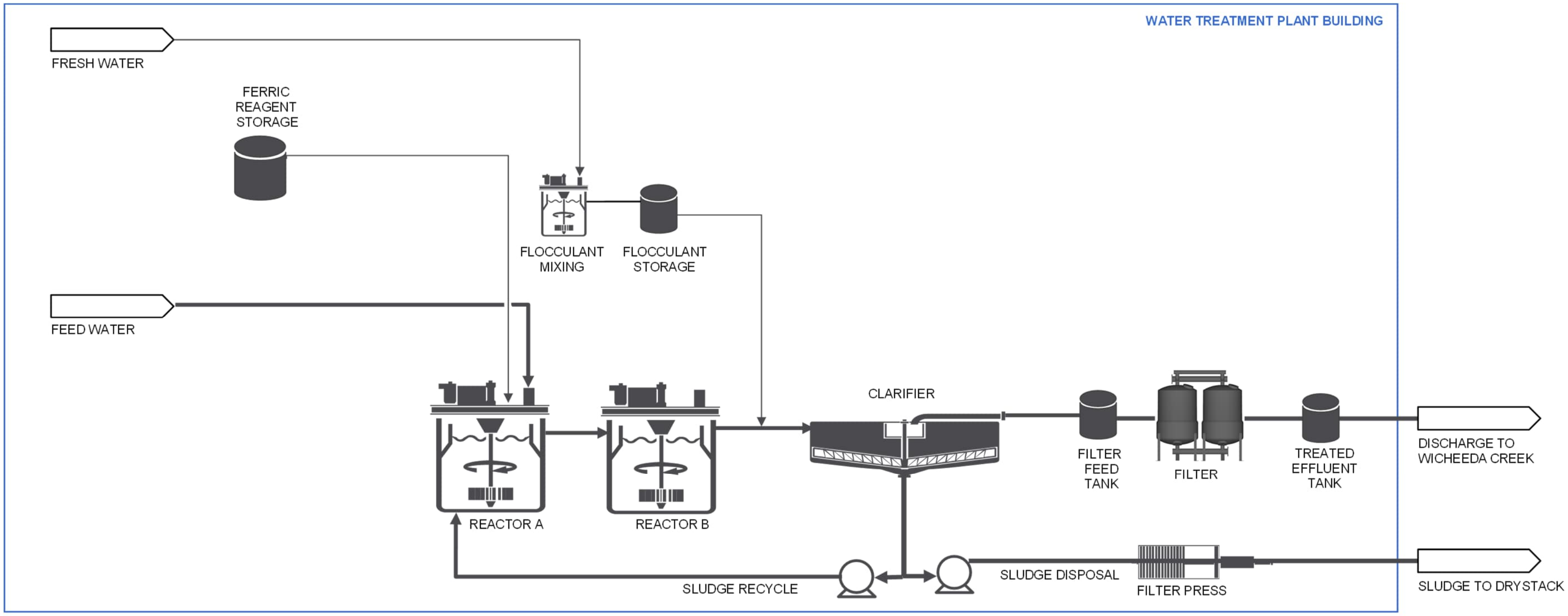
The methods used to develop geochemical source terms for each load source identified are summarized as follows:

- Source terms for flotation plant and hydrometallurgical process water chemistry and the water contacting the Hydrometallurgical waste facility were based on testing of process water samples generated during metallurgical and hydrometallurgical testing.

- Source terms for the FTSF, ore stockpile, WSF, and pit walls were developed using standard methods that scale the results of laboratory kinetic testing to site conditions through a number of steps that include:
    - The rate of mineral weathering and subsequent element or radionuclide release for each mine waste material is measured using laboratory kinetic tests.
    - Laboratory measured weathering rates scaled to field conditions using factors to account for differences in temperature, particle size, and water contact between laboratory and field conditions.
    - Field scale weathering rates integrated with the mass of mine waste material in each facility to determine the mass of each element that is available for leaching. This mass is then dissolved into the volume of water contacting the facility indicated by climatic information to determine potential concentrations in contact waters.
    - Potential concentrations are compared to solubility limits that incorporate mineral precipitation and equilibrium mechanisms to determine final predicted contact water concentrations for each facility.

The geochemical source terms for each facility were integrated within a water and load balance (WLB) model that incorporates the relative flows and chemical loads from each site facility or site component. The WLB model indicated that water treatment for removal of molybdenum and uranium will be required before water from the CWP would be acceptable for discharged to Wichcika Creek. Annual discharge of approximately 1.5 Mm^3/year is expected in a year with average hydrological conditions, and up to approximately 2.5 Mm^3/year in a 1 in 200 wet year.

The water treatment process for removal of molybdenum and uranium is ferric co-precipitation. The process works by adding ferric sulphate ($Fe_2(SO4)_3$) to the process water (CWP water) in an agitated reactor tank (Figure 18-8). The neutral pH of the process water causes ferric (Fe^{3+}) to precipitate as ferric hydroxide ($Fe(OH)_3$), which is a brown amorphous precipitate.



Source: SRK, 2025

Figure 18-8 Process Flow Diagram Ferric Co-Precipitation Water Treatment Process

Both dissolved molybdenum and uranium adsorb to the ferric hydroxide precipitate, which is removed as sludge in a clarifier (thereby removing it from the CWP water). Flocculant is added to the process water as it flows from the reactor tank to the clarifier. A subsequent filtration step ensures that small ferric hydroxide particles that do not readily settle in the clarifier are removed from the treated effluent.

A portion of the sludge collected in the clarifier is pumped back to the reactor tank to increase the mass of ferric hydroxide precipitate in the reactor. Recirculation of sludge improved the removal efficiency, reduces the ferric reagent demand, and increases the density of the sludge, which improves its settling and dewatering characteristics. Sludge is periodically withdrawn from the clarifier, dewatered in a plate and frame filter press and trucked to the TSF for permanent disposal. The sludge consists predominantly of ferric hydroxide solids, which are chemically inert under circum-neutral conditions.

The water treatment plant is expected to operate between April and October each year. The nominal treatment capacity is 520 m^3/h and the maximum capacity is 680 m^3/h. Operating at nominal capacity at a mechanical availability of 95%, the plant can treat approximately 2.5 Mm^3 over a 10-month treatment season, which is sufficient to manage the inventory of site in up to a 200 wet year. However, the allowance for a maximum treatment capacity of 680 m^3/h was made such that the operations can elect to treat and discharge water at a higher rate during the high flow freshet season.

The water treatment plant is constructed as an annex to the mill building for easy access to power, compressed air, and reagent storage.

Water treatment is expected to continue long-term into the post-closure period. The possibility of constructing low-permeability synthetic covers over the waste rock and TSF areas have been evaluated at a preliminary level as an opportunity to eliminate the need for ongoing water treatment. However, this study assumes that water treatment will continue long-term.

18.6 Hydrometallurgical Site On-Site Infrastructure

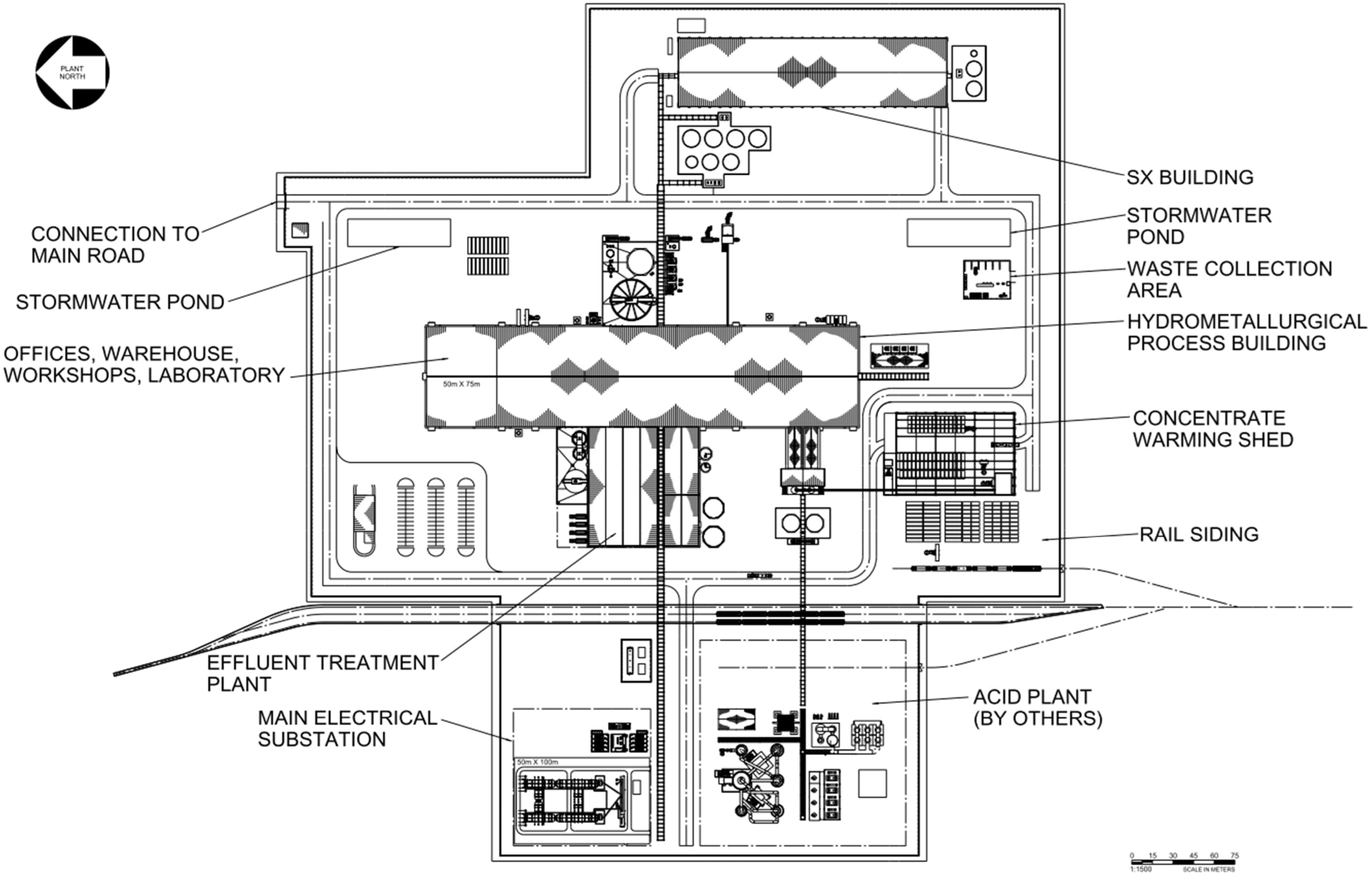
18.6.1 Plant Location

The hydrometallurgical plant is a standalone industrial facility planned to be located in Bear Lake approximately 70 km north of Prince George just off Highway 97. Bear Lake has highway access, access to the Chuckinka FSR, rail access, and is near both electrical power and natural gas supplies. This study assumes that an existing partially developed heavy industrial site will be available, but a specific site has not been selected. Obtaining access to a specific site will be investigated before and during the next phase of the project.

New roads on the site have been included in the estimate. These roads are asphalt paved to minimize dust generation and grading requirements.

The hydrometallurgical plant site facilities include the following:

- Site roads
- Receiving and unloading facilities for the mine concentrate
- Road and rail connections to existing networks
- Receiving, unloading and storing facilities for the plant reagents
- General facilities including, shops, warehouses, first aid/emergency response, administration offices, IT, communications, laboratories, HV Substation, site power distribution, solids waste disposal, emergency power, diesel fuel storage and dispensing, mobile equipment for site
- Treatment of the process wastewater for discharge to suitable receiving water stream
- Municipal potable / plant water connection
- A separate warehouse for managing the incoming concentrate containers and the containers that have been refilled with PN waste are included as the ore and PN waste contain radionuclides. These containers will be mildly radioactive, in addition there will be segregation between the incoming concentrate and out-going PN waste. Empty concentrate containers will be filled with PN waste for shipment back to the mine site.



Source: Hatch, 2025

Figure 18-9: Hydrometallurgical Plant Site Layout

18.6.1 *Hydrometallurgical Plant Railway Access*

The hydrometallurgical plant is connected to the existing CN rail network. The costs for connection spur and on-site spurs are to be borne by CN and are not included in the capital cost.

The hydrometallurgical plant railway spur is assumed to remain in place post-closure, to support potential future redevelopment of the site.

18.6.2 *Accommodation & Maintenance*

The hydrometallurgical plant is located near enough to Prince George that no accommodation camps are planned. The workforce will stay in local towns or Prince George.

18.6.3 *Hydrometallurgical Reagents*

Hydrometallurgical reagents are brought onsite via rail delivery, with truck delivery available as a backup. Sulphuric acid is supplied over the fence via pipeline from a co-located acid plant. Chemtrade Logistics have indicated a willingness to construct an acid plant adjacent to the hydrometallurgical plant; this will be further developed in the next phase.

Details on the reagent storage and distribution systems are given in Section 17.2.4.12.

Ammonia is brought in as a 29% ammonia solution and stored in tanks on site. Anhydrous ammonia was considered, but at the volumes required, the additional safety concerns did not warrant the delivery, storage and handling of anhydrous ammonia.

Other minor reagents are supplied in either totes or super sacks, depending the on material being delivered, and stored in an onsite storage area prior to use.

All storage tanks have bunding with required regulatory containment volumes.

18.6.4 *Water*

The hydrometallurgical plant is suppled with water (process and potable) via a municipal drinking water connection. The plant process water supply is supplemented with recycled water reclaimed from the hydrometallurgical water treatment facility. Further details on the water storage and distribution systems are given in Section 17.2.4.12.

Domestic sewage is pumped to a local community sanitary sewer, which will need to be constructed (the details of the community sewer will be determined in the next project phase).

18.6.5 *Electricity & Communications*

The Hydrometallurgical plant has a138 kV grid connection to a new substation which will supply 13.8 kV power for distribution within the hydrometallurgical plant. This sub-station is assumed to remain in place post-closure to support potential redevelopment of the site.

Allowances have been for IT fit-out of the hydrometallurgical plant.

18.6.6 *Diesel Fuel & Natural Gas*

At the hydrometallurgical plant a diesel fuel storage and dispensing station is contracted to a local fuel supplier. The project provides space and containment for the fuel dispensing. Fuel dispensing will be for on-site mobile equipment.

18.6.7 Hydrometallurgical Waste Streams

The various liquid and solid hydrometallurgical waste streams are described in Table 18-3. The waste streams are disposed by one of three methods: storage in the hydrometallurgical waste storage facility, co-mingling with flotation tailing in the FTSF, and treatment in the hydrometallurgical water treatment facility.

Table 18-3: Summary of Hydrometallurgical Waste Streams

Stream	Description	Design Mass Flow (t/y)	Primary Components	Potential Radioactivity
Off-Gas Treatment Precipitates	**Filter solids from the offgas treatment area**. Comprises primarily gypsum solids formed in the scrubber, with a small amount of CaF2, and dust from the acid baking kilns. This waste material is stacked at the hydrometallurgical waste storage facility.	71,000	Gypsum, CaF2	Minimal, potential for a small amount of radioactive solids
Primary Neutralization Precipitates	**Filter solids from primary neutralization area**. Comprises primarily water leaching residue (gypsum and unreacted concentrate), and the primary neutralization precipitate (magnesium and iron phosphates). This material will periodically contain spent UIX resin (IX resin with adsorbed U ions). This waste material contains nearly all of the radionuclides introduced from the feed concentrate. This material is backhauled to the mine site for co-mingling with the concentrator tailings.	79,000	Gypsum, unleached concentrate, Mg3(PO4)2, FePO4	Contains Normally Occurring Radioactive Materials (Th, U). See Section 20.2.2
Magnesium Removal Precipitates	**Filter solids from the magnesium removal area**. Comprises primarily gypsum solids formed in the MgR units, along with precipitated magnesium, lanthanum and cerium hydroxides. This waste material is stacked at the hydrometallurgical waste storage facility.	239,000	Gypsum, La(OH)3, Ce(OH)3, Mg(OH)2	Non-radioactive*
Ammonia Recovery Precipitates	**Filter solids from the ammonia recovery area**. Comprises primarily high purity gypsum solids. This waste material is stacked at the hydrometallurgical waste storage facility.	19,000	Gypsum	Non-radioactive

Stream	Description	Design Mass Flow (t/y)	Primary Components	Potential Radioactivity
Lime Grit	**Undissolved grits from the supply of lime for slaking**. This waste material is stacked at the hydrometallurgical waste storage facility.	1,100	Lime	Non-radioactive
Magnesia Grit	**Undissolved grits from the supply of magnesia for slaking.** This waste material is stacked at the hydrometallurgical waste storage facility.	220	Magnesia	Non-radioactive
Recirculating Liquor Effluent	**Bleed from the recirculating plant water.** This stream is primarily water with CaSO4 at the saturation limit. The water also contains a small amount of dissolved ammonium sulfate. This liquid effluent is treated at the hydrometallurgical water treatment facility.	139,000	Water, Calcium Sulfate, Ammonium Sulfate	Non-radioactive
Cooling Tower Blowdown	**Bleed from the recirculating cooling tower water**. This stream is primarily water with small amounts of cooling tower chemical additives and any dust/debris which enter the cooling tower. This liquid effluent is treated at the hydrometallurgical water treatment facility.	18,000	Water	Non-radioactive
Ammonia Storage Scrubber Bleed	**Bleed from the recirculating ammonia storage scrubber**. This stream is primarily water with dissolved ammonia. This liquid effluent is treated at the hydrometallurgical water treatment facility.	1	Water	Non-radioactive

Source: Hatch, 2025

*The magnesium removal precipitates will contain a small amount of ^{227}Ac which is a decay product of ^{235}U that chemically partitions according to lanthanum; however, the radiation emitted from this amount of ^{227}Ac is expected to be negligible and does not necessitate special handling of MgR precipitate. (Memorandum to Defense Metals Corp. – Wicheeda Project Radioactivity Management, Arcadis, February 7, 2025)

18.6.8 Hydrometallurgical Waste Storage Facility

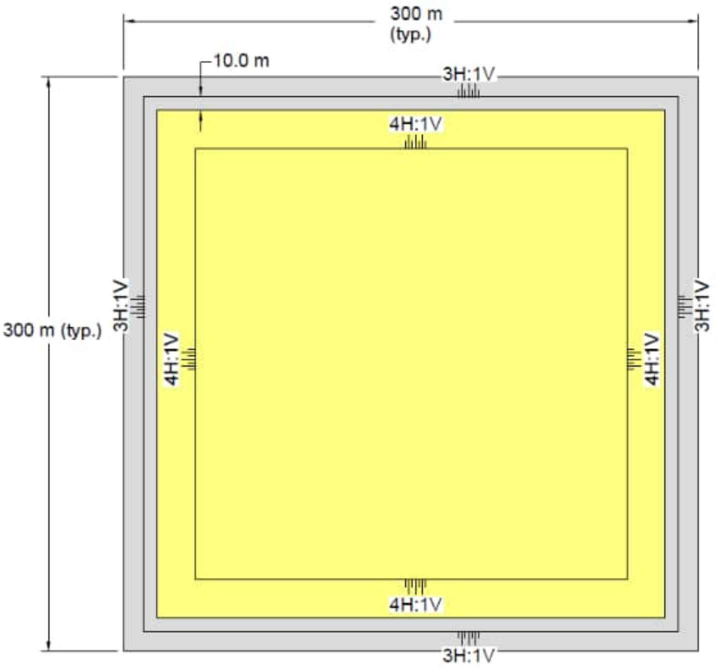
A physical location for the Hydrometallurgical Waste Storage Facility (HWSF) has not yet been identified but it is assumed to be located adjacent to the hydrometallurgical plant at Bear Lake. The storage concept for the hydrometallurgical wastes is to place and stack them in a lined facility. All three hydrometallurgical wastes stored in the HWSF are predominantly gypsum materials and will be filtered to a transportable filter cake and trucked to the facility for placement and compaction. Wet or dry storage of gypsum by-products in ‘gypstacks’ is common practice, particularly phosphogypsum by-products from fertilizer and phosphoric acid production.

The hydrometallurgical waste filter cakes (20-30 wt% moisture) produced in the process plant (see Section 17.2.4) are collected via conveyor belt to a common loadout building to be loaded onto trucks for transport to the HWSF.

The storage facility consists of four equally sized cells, with only one cell being constructed and operated at a time to defer capital cost. The design assumes a square-shaped cell with an outer starter embankment around the full perimeter. Fill for the embankment is assumed to come from within the cell’s footprint to minimize haul and provide additional storage capacity. The height of the starter embankment is 5 m above ground level with excavation within the footprint corresponding to a depth of 2.5 m to balance the fill. The foundation includes a geomembrane liner over the entire cell footprint and up to the crest of the starter embankment. A 0.5 m thick drainage layer is included above the liner which will also serve as liner protection during initial material placement. No under-liner drains are currently included; it is assumed there are no artesian conditions and the facility is placed well above the groundwater table.

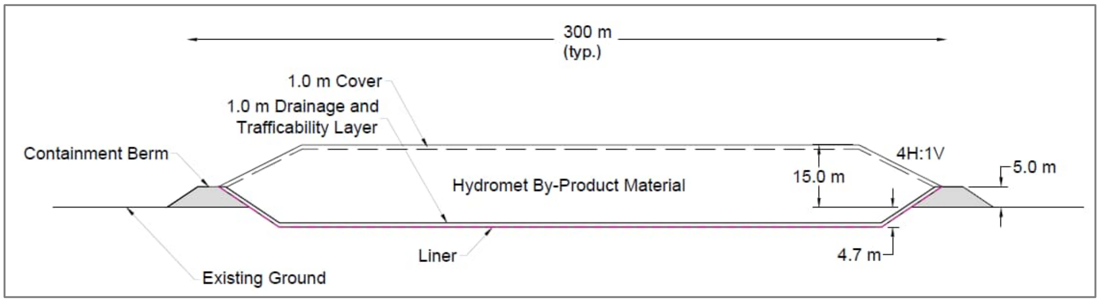
During operation, the filtered waste is loaded into trucks at the hydrometallurgical plant and hauled to the active cell before being dumped, spread with a dozer into approximately 300 mm thick layers, and compacted with a roller. For costing, it is assumed the storage facility is within 500 m of the hydrometallurgical plant. At full production, the total amount of material to be moved per day is in the order of 1,000 tonnes on a wet basis (700 tonnes dry). As the cell is filled, material is eventually placed above the elevation of the perimeter embankment at a slope of 4H:1V until reaching a maximum height of approximately 17 m above ground (i.e., 12 m above the starter embankment). Like the mine site FTSF, the outer slopes are clad with 1 m-thick rockfill to provide environmental containment and prevent erosion of the face.

Each cell is approximately 300 x 300 m as shown in Figure 18-10 and provides storage capacity for 0.7 Mm^3 of material. Figure 18-11 shows a typical section. The anticipated total amount of hydrometallurgical waste produced during operation is approximately 3.5 Mt (0.75 Mt of off-gas, 2.5 Mt of Mg-R, and 0.2 Mt of ammonia recovery). A conservative placed and compacted dry density of 1.4 t/m^3 has been assumed, giving a total storage volume required of 2.5 Mm^3. The four cells together provide 2.8 Mm^3.



Source: SRK, 2025

Figure 18-10: HWSF Cell Layout



Source: SRK, 2025

Figure 18-11: Typical HWSF Section

Runoff water as well as any potential seepage from the facility is collected by a full perimeter channel and is directed to a lined water management pond. The pond is a maximum 3 m deep and approximately 60 x 60 m in surface and sized to accommodate runoff from the facility during a 200-year, 72-hour storm event. A diversion berm is also assumed on the downstream side of the perimeter collection channel to prevent external non-contact water from entering the system.

The closure design assumes the slopes and final top surface will be covered with a 1 m thick growth medium and revegetated. Note this is in addition to the rock cladding on the slopes. Further, the cover includes a geomembrane liner given the potential water solubility of the stored material. Implementation of the closure design occurs both progressively during operation and in the active closure phase at the end of processing.

18.6.9 ML/ARD and Radionuclide Potential of Hydrometallurgical Waste Disposal Facility

The LaCe-MgR (magnesium removal precipitates with lanthanum and cerium hydroxides; Table 18-3) hydrometallurgical waste stream will be disposed at the hydrometallurgical site. Characterization of its ML/ARD potential and radionuclide content is described in Section 20.1.1. In summary, drainage from the hydrometallurgical waste disposal facility is expected to be alkaline and have potential for leaching of sulphate and ammonia. Waste in the hydrometallurgical disposal facility is expected to be within the unconditional derived release limits for diffuse NORM sources for solids.

18.6.10 Hydrometallurgical Water Treatment

The flow and water quality of excess process water and water contacting the HWSF was estimated through development of geochemical source terms that synthesized data from metallurgical test work, current understanding of the static and kinetic geochemical characteristics of each hydrometallurgical waste material, HWSF geometries, and climatic information.

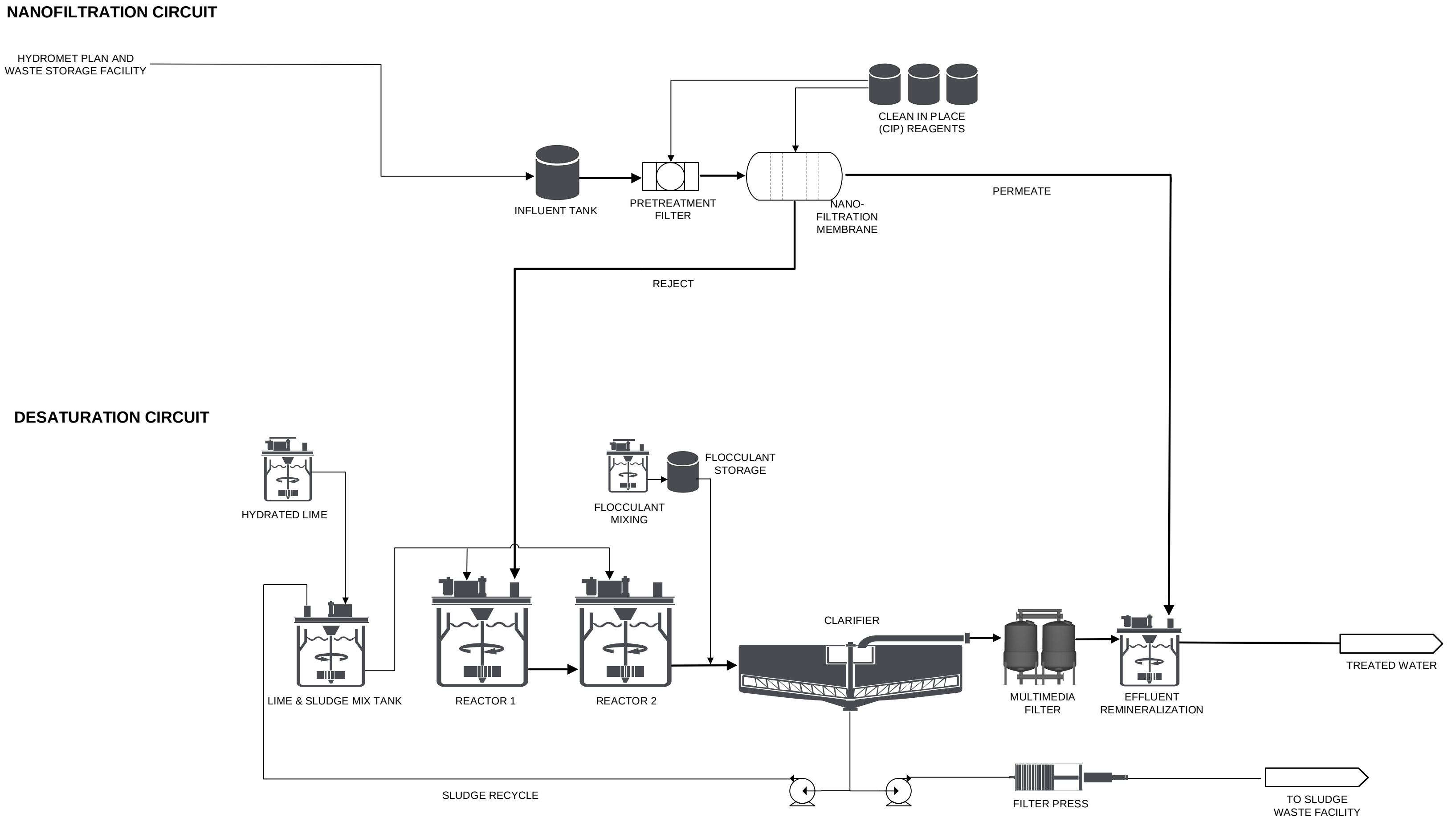
The geochemical source terms were integrated within the site WLB and compared to water quality guidelines to determine if management or treatment of the hydrometallurgical process water or water contacting the hydrometallurgical waste facility would be required.

Excess process water from the hydrometallurgical plant and runoff and seepage from the waste storage facility is primarily gypsum ($CaSO_4$) saturated water, which is expected to have a sulphate concentration between 2,000 and 2,200 mg/L. It was assumed that these contact water sources would have to be reduced to levels below or near the BC water quality guideline of 429 mg/L (for very hard water).

The process selected for removal of sulphate is nano-filtration and desaturation (Figure 18-12). Excess water from the hydrometallurgical plant and water from the HWSF are pumped to and influent tank. From there, the water is pumped through a pre-filter that removes particles larger than 1 µm. After pre-filtration, the process water is pumped through nano-filtration membranes at high pressure.

Nano-filtration membranes allow molecules with no charge (such as H_2O) and smaller ions (such as Na+ and K+) to pass but retains large molecules, including sulphate. The process produces two streams: a clean permeate that has low sulphate concentrations and a brine reject that contains most of the sulphate in the feed stream. The flow rate of the clean permeate stream is approximately 75% of the feed flow while the brine stream is roughly 25% of the flow. This means that the sulphate concentration in the brine stream is concentrated by a factor of 3 to 4 compared to the feed water concentration. As such, the feed water, which was already saturated by gypsum becomes super-saturated.

Clean permeate is pumped to an effluent tank while the supersaturated solution is piped to a desaturation circuit. Here, hydrated lime ($Ca(OH)_2$) is added to the brine in agitated reactors. The addition of calcium causes gypsum to precipitate from the supersaturated solution. The precipitated gypsum is collected as sludge in a conventional clarifier. A portion of the sludge collected in the clarifier is pumped back to the reactor tank to increase the mass of gypsum precipitate in the reactor.



Source: SRK, 2025

Figure 18-12 Process Flow Diagram for the Nano-Filtration and Desaturation Water Treatment Process

Recirculation of sludge improved the desaturation efficiency, reduces the lime demand, and increases the density of the sludge, which improves its settling and dewatering characteristics. Sludge is periodically withdrawn from the clarifier, dewatered in a plate and frame filter press and trucked to the HWSF for permanent disposal. The dewatered gypsum sludge is of a similar composition as the hydrometallurgical waste in the storage facility.

The nominal treatment capacity for the hydrometallurgical water treatment plant is 45 m^3/h, and the maximum capacity is 90 m^3/h. The treatment capacity was based on the hydrometallurgical plan mass balance and estimates runoff and seepage from the HWSF. The treatment plant will be constructed inside the hydrometallurgical plant, although the clarifier for the desaturation circuit may be located outside of the plant building.

The treated water from the effluent tank is recirculated to the plant process water supply to reduce the consumption of fresh water from the municipal source. Details on the recycling of effluent water will be developed in the next project phase.

18.6.11 Hydrometallurgical Plant Site Closure

During active remediation, once the hydrometallurgical waste storage facility cells have been covered and downstream water quality meets water discharge limits, the hydrometallurgical plant and most of the associated infrastructure will be demolished. For the purpose of this study, since the facility is assumed to be built on an industrial site with some prior development history, the hydrometallurgical plant end land use is assumed to be as an industrial site. The railway spur and substation are assumed to remain in place. Disturbed areas will be regraded and vegetated to prevent erosion and growth of invasive species.

19. Market Studies and Contracts

The market study which follows below was provided by Adamas Intelligence, an independent market expert in REEs. It has been reviewed and approved for use in this report by the Qualified Person for Section 19.

19.1 REEs: Critical Enablers of Electric Mobility, Automation and Energy Efficiency

Compared to similarly abundant elements in nature, such as copper, lead, and tin, global annual production of rare earth elements is notably low.

Nevertheless, rare earth elements have become critical enablers of technologies at the heart of clean energy initiatives worldwide, as well as ubiquitous gadgetry and electronics that have pervaded modern society.

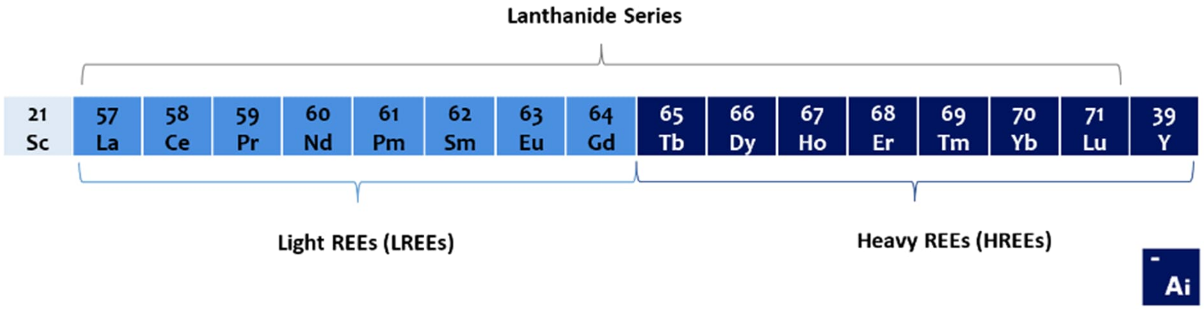
Rare earth elements are used in small, but often necessary, amounts in hundreds of different technologies, materials, and chemicals worldwide for commercial, industrial, social, medical, and environmental applications.

In just a period of decades, rare earth elements have seeped deeply into the fabric of modern technology and industry and have proven exceptionally challenging to duplicate or replace.

19.2 Classification and Terminology

On the Periodic Table of Elements, rare earth elements include the lanthanide series (lanthanum, cerium, praseodymium, neodymium, samarium, europium, gadolinium, terbium, dysprosium, holmium, erbium, thulium, ytterbium and lutetium), plus yttrium and scandium (see Figure 19-1).

Yttrium is classified as a rare earth element because of its similar ionic radius to the lanthanides, as well as its similar chemical properties, whereas scandium is classified as a rare earth element because of its tendency to concentrate into many of the same minerals.



Source: Adamas Intelligence

Figure 19-1: Rare Earth Elements Include the Lanthanide Series Plus Scandium and Yttrium

Rare earth elements are arbitrarily classified as light rare earth elements or oxides (“LREEs” or “LREOs”) or heavy rare earth elements or oxides (“HREEs” or “HREOs”) based on their electron configurations.

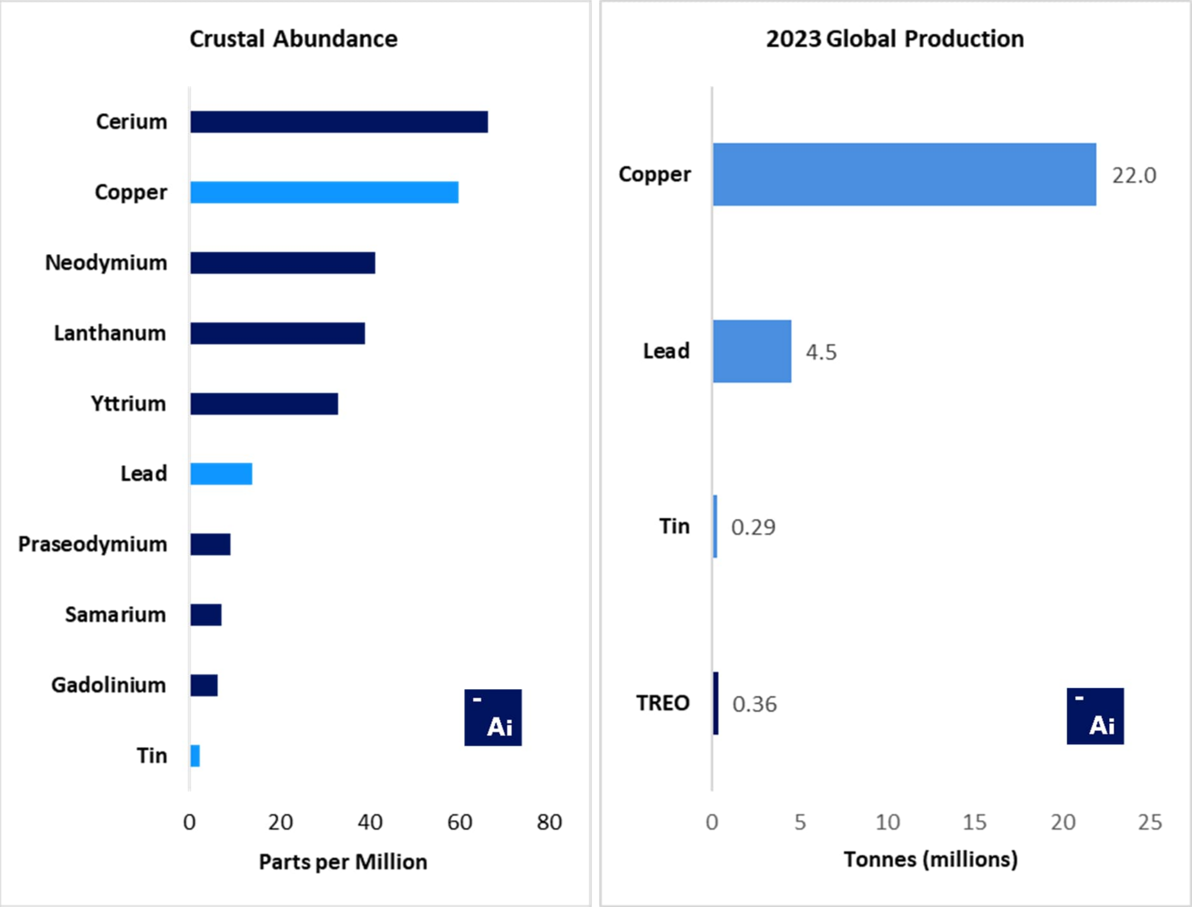
By virtue of having a higher crustal abundance, LREOs collectively make up over 90% of the total rare earth oxide ("TREO") content in a typical rare earth deposit and thereby also make up the vast majority of the world's TREO output each year. Heavy rare earth oxides, on the other hand, are present in the Earth's crust in substantially lower concentrations than LREOs and as such make up a relatively small portion of the world's TREO output each year.

19.3 Rarely Enriched in Nature

Despite the misleading moniker, rare earth elements are not remarkably rare in nature but rather are rarely concentrated into economically significant amounts for extraction and processing owing to certain physical and chemical characteristics that promote their broad dissipation in most rock types.

In fact, cerium is more abundant in the Earth's crust than copper; neodymium, lanthanum and yttrium are more abundant than lead; and praseodymium, samarium and gadolinium are more abundant than tin (see Figure 19-2 – LHS).

Despite this fact, there were only 362,404 tonnes of all 17 REOs combined ("TREO") produced globally in 2023 versus 22.0 million tonnes of copper, 4.5 million tonnes of lead and 290,000 tonnes of tin in the same year (see Figure 19-2 – RHS).



Source: Adamas Intelligence research, USGS, Jefferson Lab

Figure 19-2: Global Production of REEs is Remarkably Low Compared to Similarly Abundant Elements

19.4 Global Reserves

According to the U.S. Geological Survey (“USGS”), known global reserves are host to more than 90 million tonnes of rare earth oxides (and oxide equivalents) in-situ (see Table 19-1).

With 44 million tonnes in the ground, China is host to 48% of the world’s known rare earth reserves, over 80% of which are located at the Bayan Obo mine in Inner Mongolia.

With 21 million tonnes in-situ, Brazil is host to 23% of the world’s known rare earth reserves, much of which is contained in HREE-rich ion-adsorption clay deposits.

Moreover, with 6.9 million tonnes in-situ, India is host to 8% of the world’s known rare earth reserves while Australia, with 5.7 million tonnes in-situ, is host to an additional 6%.

Other nations endowed with known rare earth reserves include Russia, Vietnam, the U.S., Greenland, Tanzania, South Africa and Canada, each holding less than 5% of the global total, according to the USGS.

Notable sources of current rare earth production not shown in the table below are Myanmar, Laos, Madagascar and Nigeria where transparent reserve estimates are lacking.

In addition to reserves, the USGS notes that measured and indicated resources of rare earths are estimated to include 3.6 million tonnes in the U.S. and more than 14 million tonnes in Canada.

Table 19-1: Overview of Known Global Rare Earth Reserves by Country

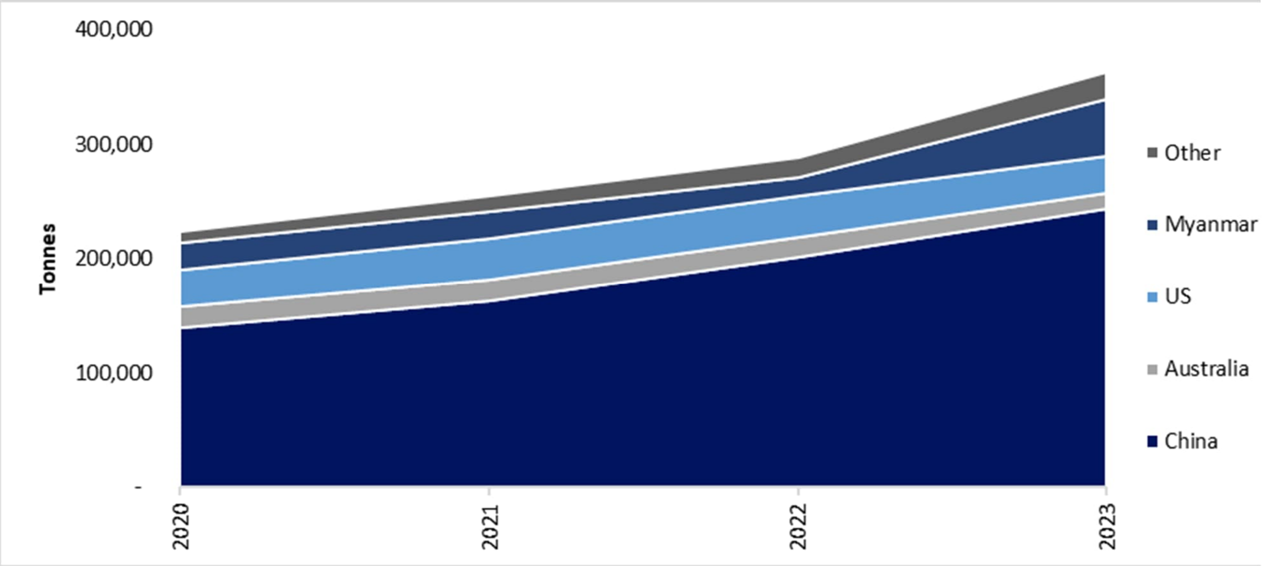
Country	Reserves (TREO tonnes in-situ)	Global Share
China	44,000,000	48%
Brazil	21,000,000	23%
India	6,900,000	8%
Australia	5,700,000	6%
Russia	3,800,000	4%
Vietnam	3,500,000	4%
United States	1,900,000	2%
Greenland	1,500,000	2%
Tanzania	890,000	1%
South Africa	860,000	1%
Canada	830,000	1%
World	90,880,000	100% (of known reserves)

Source: Adamas Intelligence after USGS (2025)

19.5 Historical Production

From 2020 through 2023, Adamas data indicates that global mine production of rare earth oxides (“TREO”) increased at a CAGR of 17.4%, from 224,000 tonnes to 362,000 tonnes, with China responsible for 62% to 70% of output each year.

Other notable mine producers in recent years have included Australia, the US and Myanmar, albeit their combined share of global supply has fallen from 33% in 2020 to 26% in 2023, outpaced by China's production growth.

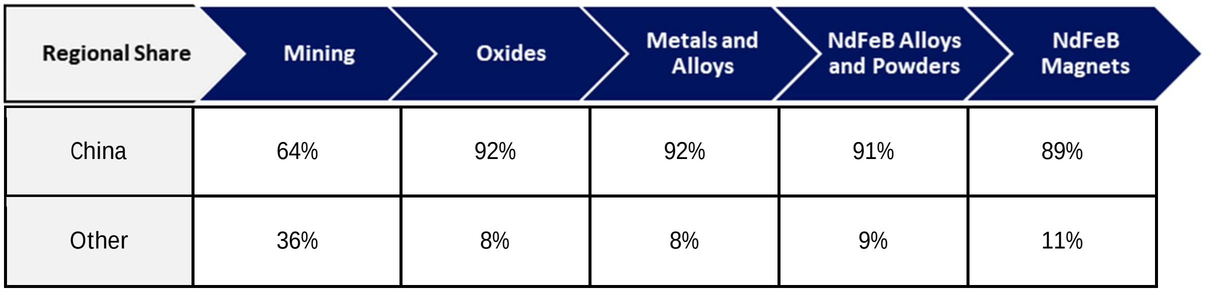


Source: Adamas Intelligence

Figure 19-3: Historical global TREO mine production by country

Since the early 2000s, China has leveraged its dominance of upstream production to cement control over all downstream value adding steps along the mine-to-magnet supply chain.

As of 2023, China was responsible for 64% of global mine production of magnet rare earths and around 90% of each subsequent step, making it the world's dominant producer of refined rare earth oxides, metals and alloys, magnetic alloys, and NdFeB permanent magnets.



Regional Share	Mining	Oxides	Metals and Alloys	NdFeB Alloys and Powders	NdFeB Magnets
China	64%	92%	92%	91%	89%
Other	36%	8%	8%	9%	11%

Source: Adamas Intelligence, 2025

Note: Mining encompasses production of rare earth mineral concentrates and mixed rare earth chemical concentrates (e.g., mixed rare earth carbonate "MREC")

Figure 19-4: Overview of the Global Mine to Magnet Value Chain in 2023, Led by China at every step

19.6 Eight End-Use Categories

Rare earth elements are used in hundreds of unique end-uses and applications that collectively fall into one of eight end-use categories: 1.) Battery Alloys, 2.) Catalysts, 3.) Ceramics, Pigments and Glazes, 4.) Glass Polishing Powders and Additives, 5.) Metallurgy and Alloys, 6.) Permanent Magnets, 7.) Phosphors, and 8.) Other End-Uses and Applications (see Table 19-2).

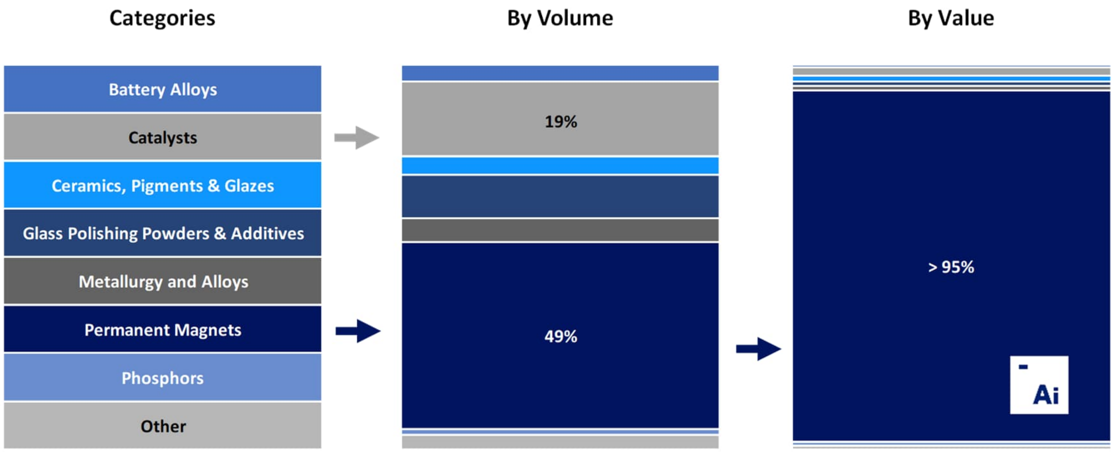
Table 19-2: Rare Earth Applications and End-uses Fall into one of Eight End-use Categories

End-Use Category	Description
Battery Alloys (La, Ce, Pr, Nd)	Rare earth elements are used to produce anode materials for nickel-metal hydride ("NiMH") batteries. NiMH batteries are used in hybrid electric vehicles, consumer electronics, cordless shavers, cordless powertools, baby monitors and other applications of rechargeable batteries.
Catalysts (La, Ce)	Rare earth elements, such as cerium and lanthanum, are used in catalytic converters of gasoline- and diesel-powered vehicles, as well as fuel cracking catalysts and additives used by oil refiners to break down crude oil into lighter distillates, such as gasoline, diesel, kerosene and more.
Ceramics, Pigments and Glazes (La, Ce, Pr, Nd, Y)	Rare earth elements are used to produce decorative ceramics, functional ceramics, structural ceramics, bio ceramics and many other types of ceramics used in everything from jet engine coatings to ceramic cutting tools, dental crowns, ceramic capacitors, ceramic tiles, and more.
Glass Polishing Powders and Additives (Ce, La, Er, Gd, Y)	Rare earth elements, such as cerium, are used to polish optical glass, hard disk drive platters, LCD display screens and gemstones, among a long list of applications. Cerium is also used as an additive in UV-filtering glass and container glass, whereas lanthanum, yttrium and gadolinium are used to produce high quality optical glass used in camera lenses, microscopes and telescopes.
Metallurgy and Alloys (La, Ce, Ho, Gd, Y)	Rare earth mischmetal (a mixture of light REE metals) is used during production of some types of steel, as well as ductile iron making. Rare earth elements are also used to produce a variety of different alloys, such as ferro-cerium, ferro-holmium, ferro-gadolinium and a growing list of others.
Permanent Magnets (Nd, Pr, Dy, Tb, Sm)	Rare earth elements are used to produce high-strength permanent magnets that have enabled the production of ubiquitous gadgets and electronics, such as mobile phones and laptops, as well as power dense energy-efficient electric motors and generators used in electric vehicles, wind turbines, energy efficient appliances and hundreds of other applications.
Phosphors (Ce, La, Y, Tb, Eu)	Rare earth elements are used in phosphors for energy efficient lamps, display screens and avionics, and are added to fiat currency in some nations as an anti-counterfeit measure.
Other (La, Ce, Nd, Dy, Tb, Gd, Lu, Tm)	Aside from the above-described end uses and categories, rare earth elements are used in a long list of other end uses and applications, including many in defense, medicine, health, wellness, aerospace, agriculture, high-tech and chemical industries.

Source: Adamas Intelligence, 2025

19.7 Global Rare Earth Consumption in 2023

By volume, permanent magnets and catalysts were collectively responsible for 68% of global TREO consumption in 2023 (see Figure 19-5). However, by value, permanent magnets alone were responsible for over 95% of the total value of global TREO consumption in 2023 (see Figure 19-5), a share that continues to expand as demand for (and prices of) neodymium, praseodymium, didymium (a compound of neodymium and praseodymium), dysprosium and terbium outperform.

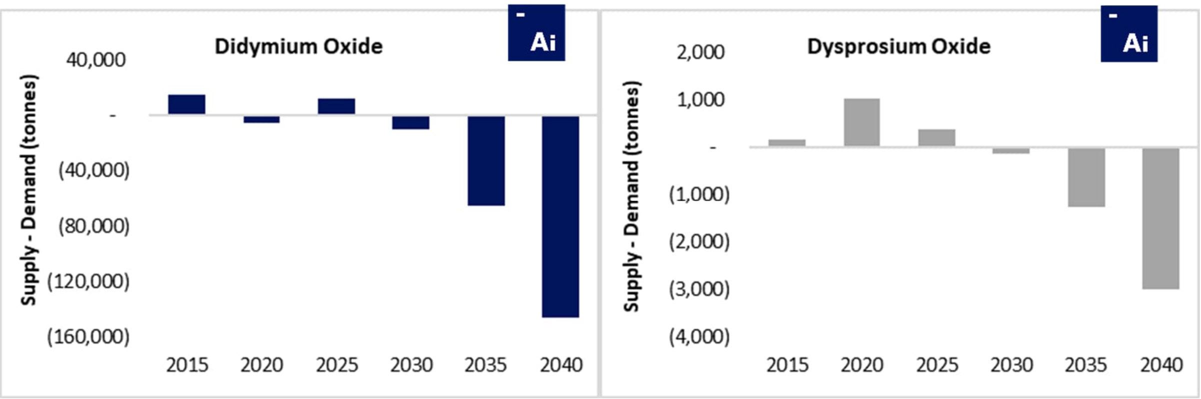


Source: Adamas Intelligence

Figure 19-5: Permanent Magnets and Catalysts are the Largest Rare Earth Demand Drivers

Not only does demand for neodymium, praseodymium, didymium, dysprosium and terbium collectively make up the majority of global value today, but in the years ahead demand for these four rare earth elements will continue to grow faster than demand for all other rare earth elements, challenging the ability of the supply-side to keep up.

As shown in Figure 19-6, Adamas Intelligence forecasts that global annual demand for didymium oxide and dysprosium oxide (or oxide equivalents) will increasingly exceed global annual production post-2025, leading to the depletion of historically accumulated inventories and, ultimately, shortages of these critical magnet materials if substantial additional sources of supply are not developed.



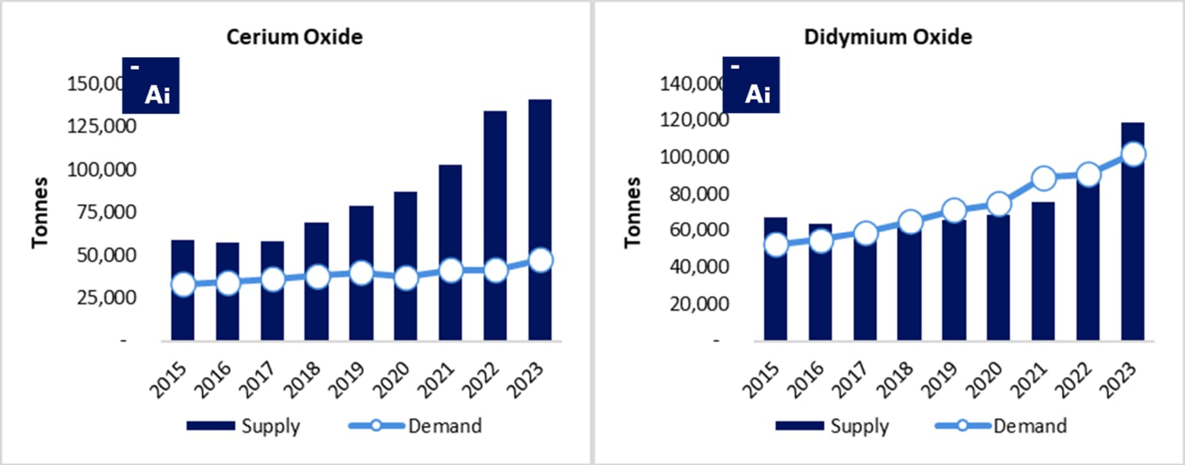
Source: Adamas Intelligence

Note: Didymium Oxide = NdPr Oxide = Oxide compound of 75% Nd2O3 and 25% Pr6O11; NdPr is the main rare earth input material for NdFeB magnet production; Only minor quantities of individual Nd oxide and Pr oxide are produced globally each year.

Figure 19-6: The Supply-Side will Struggle to Keep Up With Rising Demand for Magnet Rare Earths

19.8 Rare Earth Balance Problem

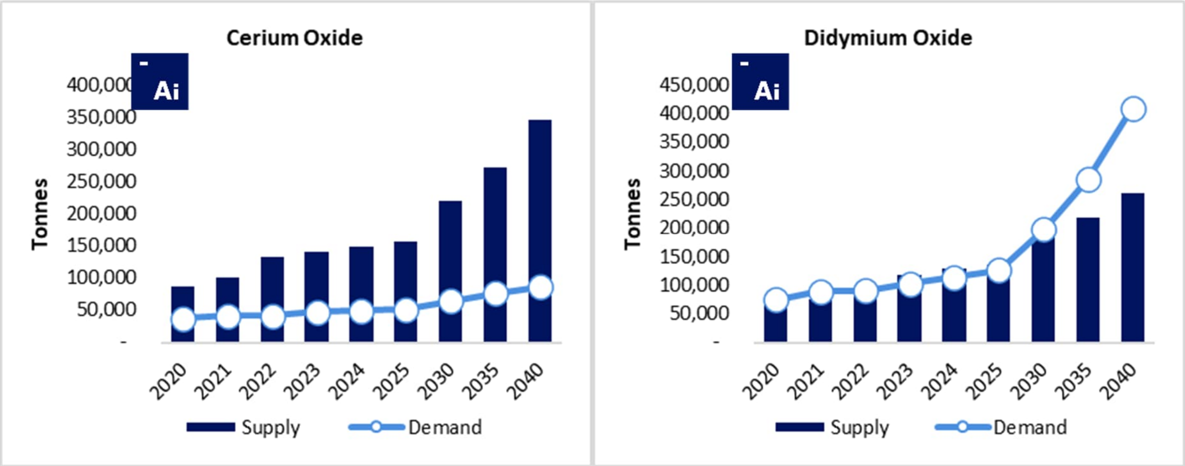
Over the past decade, rare earth producers globally have sacrificially overproduced certain low value rare earth elements, such as cerium (see Figure 19-7 – LHS), to keep up with rapidly growing demand for other high value rare earth elements and compounds, such as didymium (see Figure 19-7 – RHS).



Source: Adamas Intelligence, 2025

Figure 19-7: Sacrificial Over-Production of Cerium Oxide to Satisfy Rapidly Growing Demand for Didymium Oxide

Looking ahead, Adamas forecasts that steadily increasing demand for rare earth permanent magnets will drive global demand for didymium oxide (or oxide equivalent) to new heights (see Figure 19-8 – RHS), exacerbating the imbalance between production and demand of other rare earth elements, such as cerium oxide (see Figure 19-8 – LHS) if the industry continues on a path of business-as-usual.



Source: Adamas Intelligence, 2025

Figure 19-8: Strong Future Demand Growth for Permanent Magnets will Exacerbate the Balance Problem

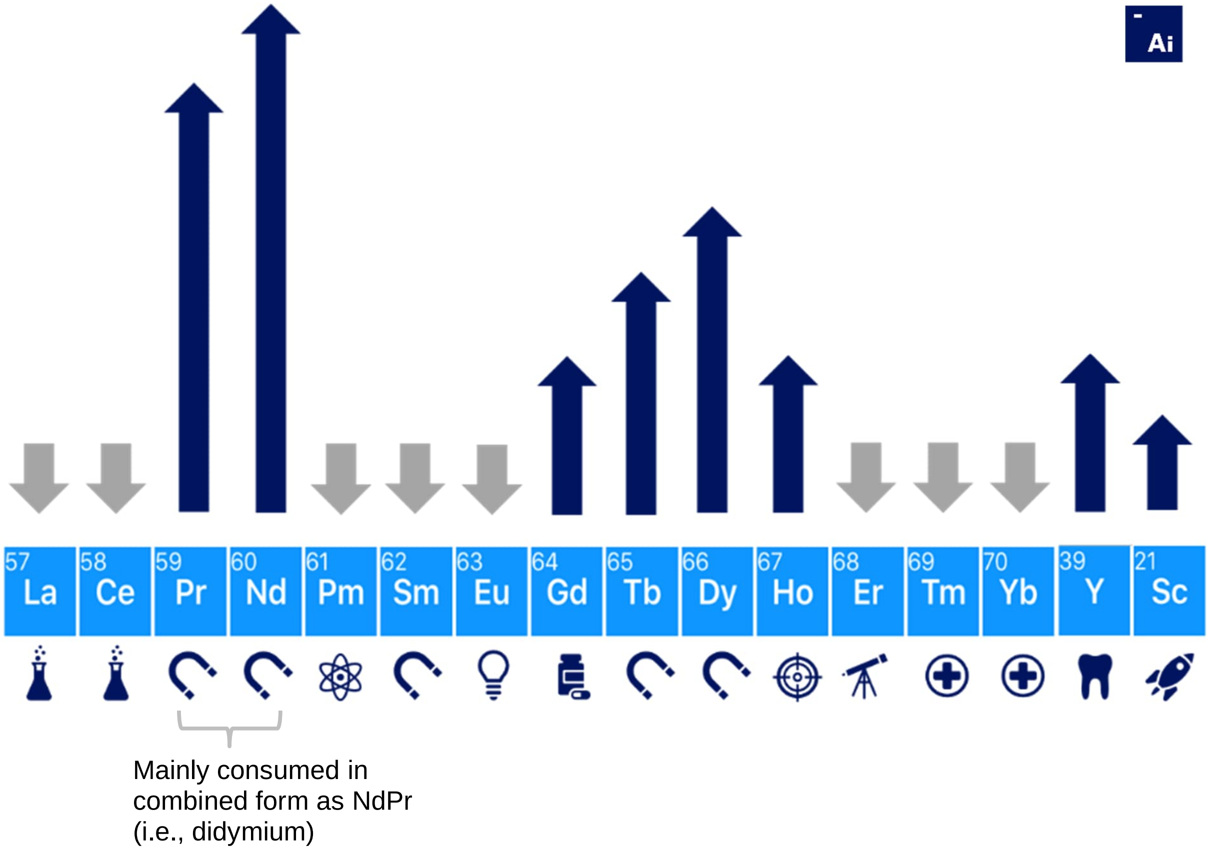
19.9 Implications of the Balance Problem

Unless new end-uses and applications are developed for cerium, lanthanum, and other sacrificially-overproduced rare earth elements in the near-term (see Figure 19-9 – light grey arrows), Adamas forecasts that prices of high-demand elements and compounds, like neodymium, praseodymium, didymium, dysprosium and terbium will stay relatively strong and/or rise accordingly (see Figure 19-9 – dark blue arrows) to compensate for losses that producers are chronically incurring by necessarily over-producing the other unsaleable, surplus rare earths.

The industries that will feel these price increases the most in the coming years are those reliant on use of high-strength NdFeB permanent magnets, such as the automotive industry, the wind power sector, the robotics industry, the defense industry, and many others.

Ultimately, Adamas expects that price increases of magnet input materials may upend the economics of using rare earths in some end-use sectors – pushing some manufacturers to adopt alternatives to rare earth permanent magnets where possible.

However, for the most promising of end-use sectors – such as electric vehicles, wind power, robotics, advanced air mobility, and others – the economics of using rare earth elements are robust and Adamas expects these segments will continue to fuel strong rare earth demand growth into the foreseeable future.



Source: Adamas Intelligence, 2025

Figure 19-9: Prices of Magnet Rare Earths Will Rise to Compensate for Losses Incurred on Other Rare Earths

19.10 Forecasted TREO Demand by End-Use Category

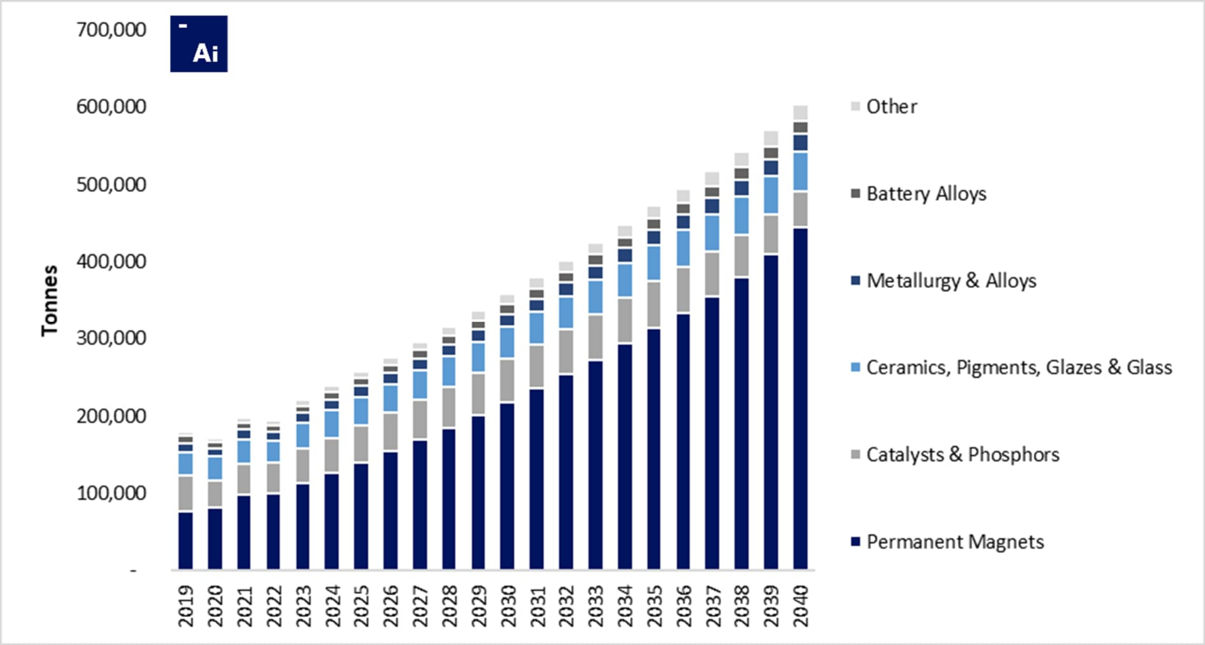
After rising 7.9% last year, Adamas forecasts that global TREO demand will increase at a CAGR of 6.0% going forward, from 238,810 tonnes in 2024 to 604,355 tonnes in 2040, driven primarily by the permanent magnet sector (see Figure 12).

In the years ahead, global TREO demand for permanent magnets is projected to rise at a market leading CAGR of 8.2%, driven by double-digit demand growth for applications involving electric mobility, robotics, advanced air mobility and more (see Figure 19-10).

Conversely, over the forecast period, global TREO demand for all other end-use categories, except for phosphors, are projected to grow at market lagging CAGRs of 0% to 6%, while TREO demand for phosphors is projected to fall at a CAGR of -5.7% (see Figure 19-10).

In the years ahead, the rapid TREO demand growth expected for permanent magnets will lead the end-use category to continuously absorb market share from incumbent categories.

By 2035, Adamas projects that permanent magnets will drive 66% of global TREO demand by volume and over 95% of the market's value each year.



Source: Adamas Intelligence, 2025

Figure 19-10: Forecasted Global TREO Demand by End-Use Category from 2024 Through 2040

Source: Adamas Intelligence

19.11 NdFeB Permanent Magnets: Enablers of Modern Technology

19.11.1 ***What is NdFeB?***

Neodymium-iron-boron (“NdFeB”) is a permanent magnet alloy that was developed and commercialized in the 1980s as an alternative to costly samarium-cobalt (“SmCo”) alloy that was developed and commercialized three decades earlier.

19.11.2 ***What is it made of?***

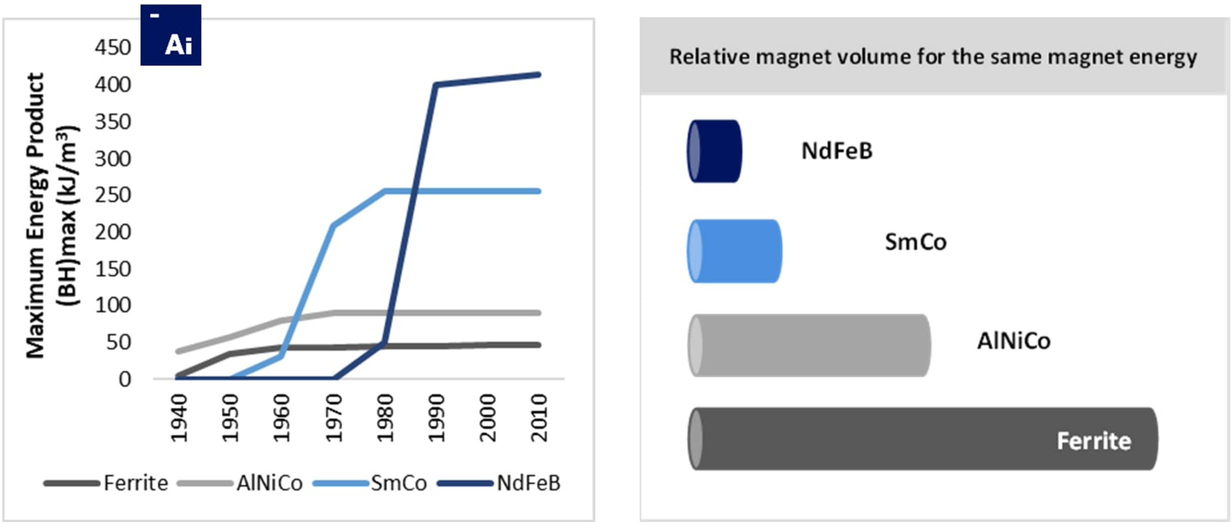
As the name suggests, NdFeB alloy is comprised primarily of neodymium, iron, and boron in a $Nd_2Fe_{14}B$ tetragonal crystalline structure, and often contains lesser concentrations of praseodymium (combined with neodymium in the form of didymium), dysprosium, terbium, cerium, gadolinium, holmium, copper, cobalt, niobium, and other metals to optimize the alloy’s properties for certain applications.

19.11.3 ***Why is it special?***

NdFeB permanent magnet alloy is the strongest type of permanent magnet material commercially available today in terms of maximum energy product (i.e., magnetic flux output per unit volume, measured in megagauss-oersteds (MGOe) or Joules per cubic meter (J/m^3)) (see Figure 19-11).

As such, NdFeB magnets have largely supplanted SmCo, AlNiCo, and ferrite magnets in many size- and weight-sensitive applications since the 1980s, and simultaneously have enabled the conception and miniaturization of a wide array of ubiquitous gadgets and electronics that have pervaded modern society.

NdFeB permanent magnets are used in hundreds of different end-uses and applications – many of which we interact with daily, whether we realize it or not. From mobile phone loudspeakers and vibration motors, to hard disk drives, optical disc drives, electric vehicle traction motors, automotive micromotors, wind power generators, and beyond – NdFeB permanent magnets are literally all around us.



Source: after Kallaste et al. (2012), Adamas Intelligence research

Figure 19-11: NdFeB is the Strongest Permanent Magnet Material Commercially Available Today

19.12 Forecasted TREO Demand for Permanent Magnets by End-Use Category

After an estimated 10.9% last year, Adamas forecasts that global TREO demand for permanent magnets will rise at a CAGR of 8.2% going forward, from 126,000 tonnes in 2024 to 445,000 tonnes in 2040, boosted by strong demand growth from electric vehicles, robotics, advanced air mobility and other applications of NdFeB magnets (see Figure 19-12).

Specifically, from 2024 through 2040 Adamas forecasts that global TREO demand for passenger EV traction motors, commercial EV traction motors and “other e-mobility” applications will collectively increase at a CAGR of 8.8%, together representing the single largest demand driver by 2040 (see Figure 19-12).

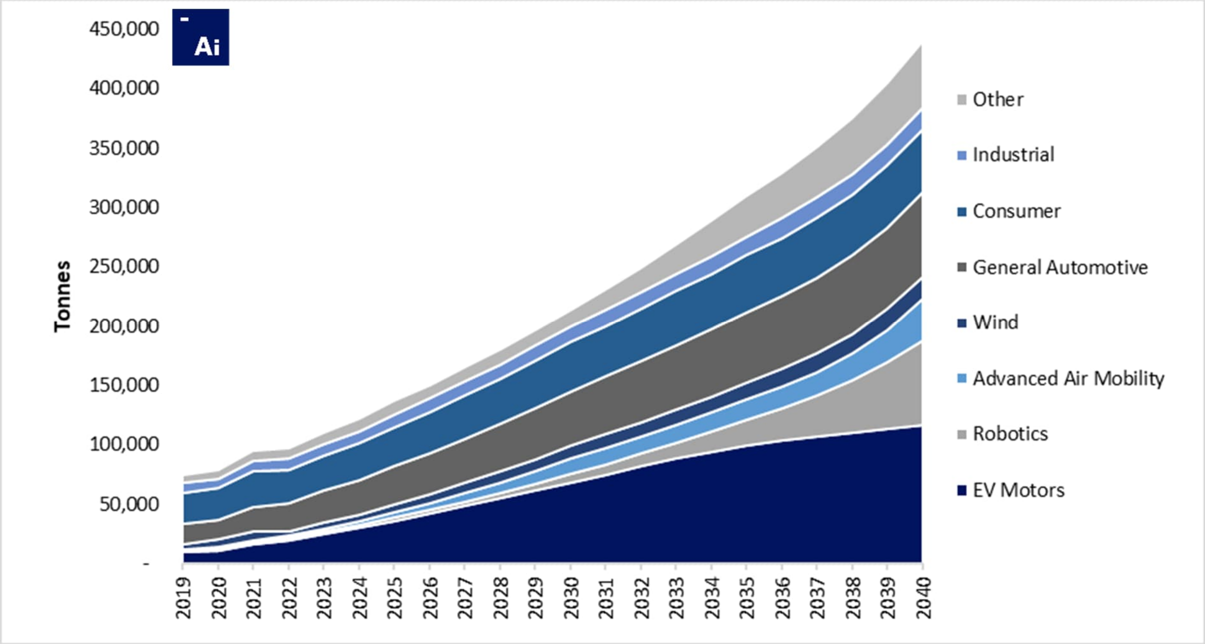
Similarly, from 2024 through 2040 Adamas forecasts that global TREO demand for industrial robots, consumer service robots and professional service robots will collectively increase at a CAGR of 23.5%, poised to collectively overtake EV motors in the decade to follow (see Figure 19-12).

Moreover, from 2024 through 2040 Adamas projects that global TREO demand for applications involving advanced air mobility (“AAM”), including consumer drones, commercial drones, electric vertical takeoff and landing (“eVTOL”) aircraft, and more, will increase at a CAGR of 17.0% to become one of the largest end use categories by the end of the forecast period (see Figure 19-12).

Additionally, from 2024 through 2040 Adamas projects that global TREO demand for direct drive and hybrid direct drive wind power generators for onshore and offshore applications will rise at a CAGR of 7.7% as the competitive economics of wind power generation (and low maintenance of hybrid and direct drive generators) continue to spur growing adoption (see Figure 19-12).

Lastly, from 2024 through 2040 Adamas forecasts that global TREO demand for all other end-uses and applications of NdFeB permanent magnets will increase at CAGRs of 3.4% to

11.0%, leading some sectors to forgo market share to electric vehicles, robotics and other high-growth applications (see Figure 19-12).

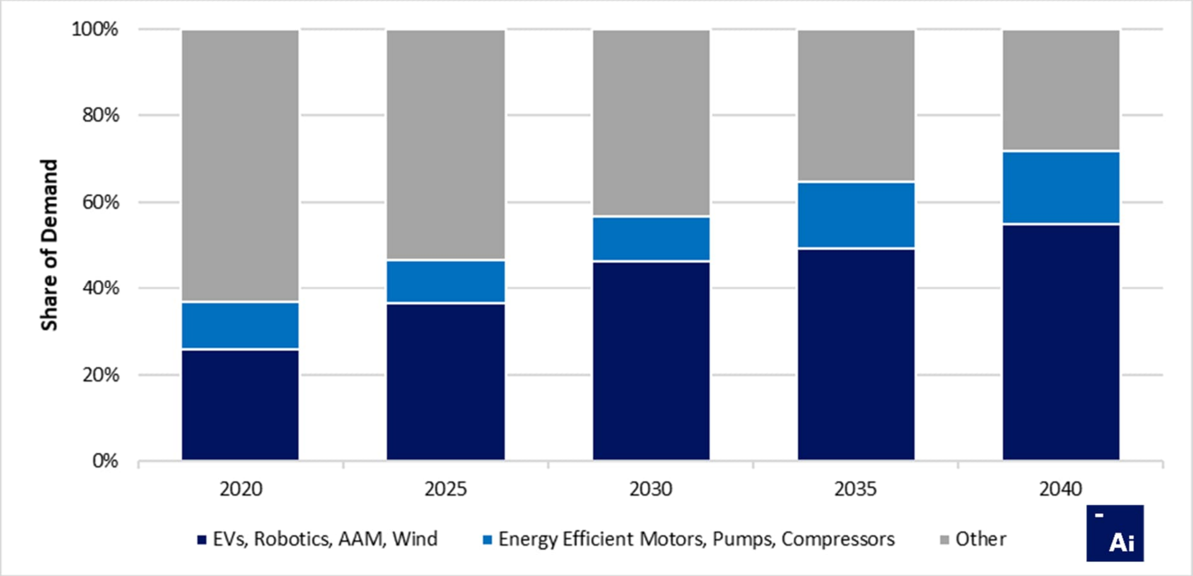


Source: Adamas Intelligence, 2025

Figure 19-12: Historical Global Consumption and Forecasted Demand for Magnet Rare Earth Oxides by End-Use Category

19.13 EVs, Robotics, AAM and Energy Efficient Applications to Drive 65% of Demand by 2035

By 2035, Adamas forecasts that electric vehicles, robotics, advanced air mobility, wind, and other energy efficient motors, pumps and compressors will collectively be responsible for 65% of total global NdFeB magnet demand, up from just over one-third in 2020 (see Figure 19-13 – blue bars).



Source: Adamas Intelligence, 2025

Figure 19-13: Forecasted Share of Global NdFeB Demand Driven by Less Price Sensitive Applications

Adamas believes this evolution is noteworthy as it implies that the future of magnet rare earths demand will be less sensitive to price than that of the past because future demand will be increasingly driven by applications in which the use of rare earth permanent magnets imparts an economic benefit at the system level.

Be it through battery cost thrifting in an electric vehicle, maintenance cost savings in a wind farm or robot fleet, or electricity cost savings in an industrial facility, grocery store or hotel, the economic upsides enabled by using technologies based on rare earth permanent magnets allow for a significant rise in magnet rare earth prices going forward before it would be economically justifiable to switch to a REE-free alternative.

As such – Adamas expects that the future of rare earths demand (at least in the case of didymium, dysprosium and terbium) will be more robust, more resilient and less sensitive to price than demand of the past and present, which is still largely driven by consumer and legacy automotive applications.

19.14 Forecasted Production – Demand Balance for Didymium Oxide to 2040

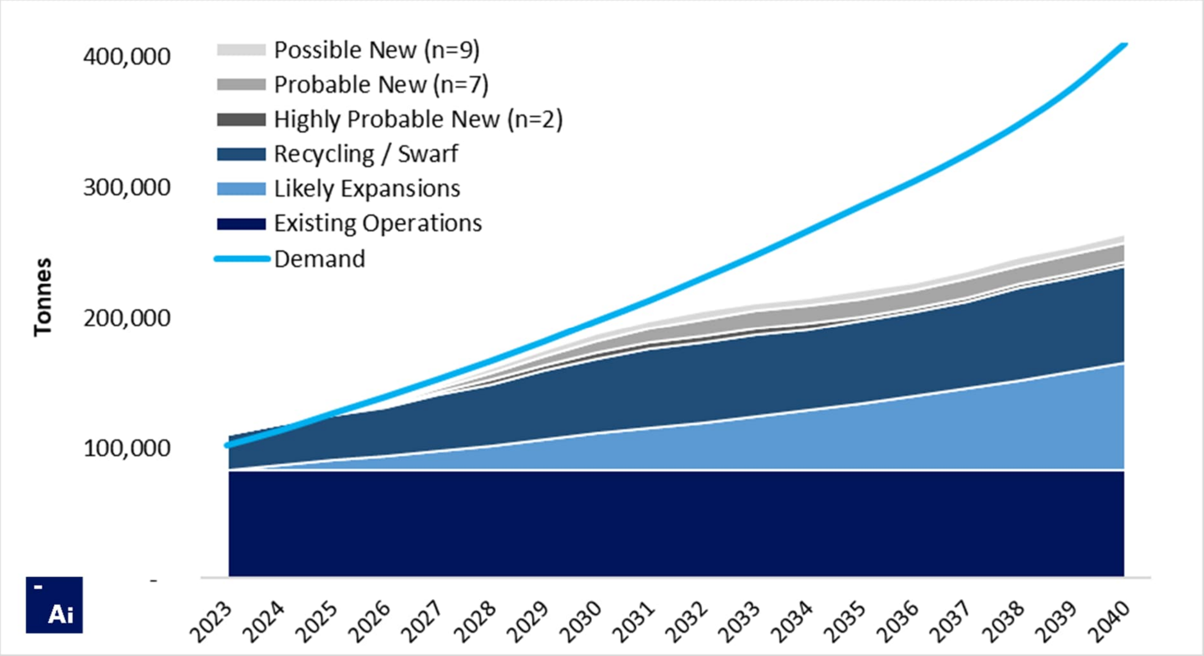
From 2025 on, Adamas projects that all magnet rare earths will experience persistent and rising structural deficits throughout the forecast period as the supply-side of the industry increasingly struggles to keep up with rapidly growing demand for NdFeB magnets for electric vehicle traction motors, wind power generators, robotics, advanced air mobility, industrial applications, consumer applications and more.

These pervasive deficits will lead to the depletion of historically accumulated inventories and, ultimately, rising shortages of magnet rare earths should global supply not increase substantially more than what is currently anticipated.

On the supply side, Adamas expects that expansions of today's existing producers coupled with the start-up of 18 new projects globally will slow growth of the market's deficit between

2025 and 2029 but production will increasingly struggle to keep up with demand growth in the years thereafter.

For China alone to close the growing supply gap projected between 2027 and 2040 would require annual production from its main NdPr oxide mining operations to increase nearly five-fold, massively depleting the country's reserves. Conversely, for ex-China suppliers to close the growing supply gap will require the development of another 20 to 30 modest-scale mines by 2040, over and above those already expected to be developed in Adamas' base case scenario.



Source: Adamas Intelligence, 2025

Figure 19-14: Forecasted Production – Demand Balance for NdPr Oxide to 2040 (Base Case)

19.15 Forecasted REO Prices to 2040

As per its latest "Rare Earth Pricing Quarterly Outlook" report (Q1 2025), Adamas Intelligence forecasted annual average prices for each rare earth oxide to 2040 under three scenarios.

19.15.1 Base Case

In the Base Case, Adamas expects the price of NdPr oxide to increase from US$55-60/kg last year to US$70-110/kg in the late-2020s. In a rational market, Adamas would expect these price increases to induce investment in new production capacity, however, owing to the long lead times to develop new rare earth supplies, Adamas sees potential for pervasive deficits to push prices above required inducement levels (estimated at US$100-150/kg) in the long term.

19.15.2 Upside

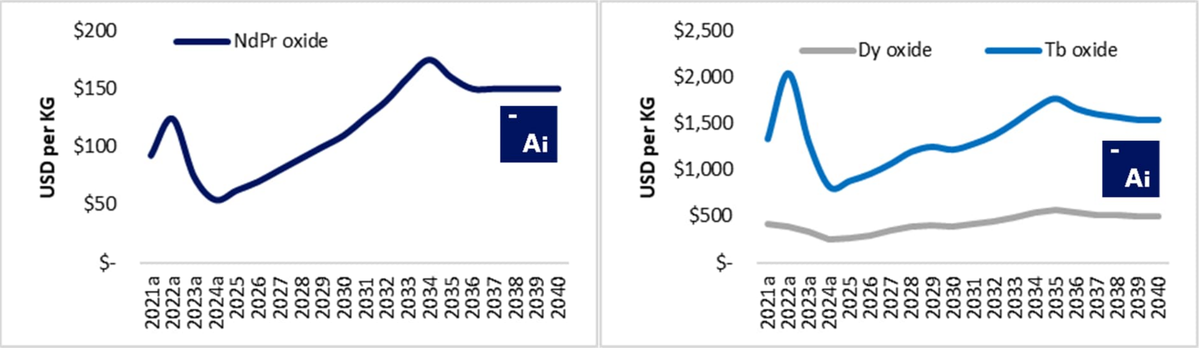
In the Upside forecast scenario, Adamas considered a future in which a strong, steady price environment induces the development of an additional 14 new producers (over and above the 18 already expected in the Base Case) from various corners of the globe.

Adamas believes the enhanced supply-side diversity, transparency and lack of price volatility/swings would help assuage demand-side risk aversion, fostering a more robust demand outlook and higher prices than considered in the Base Case forecast scenario.

19.15.3 Downside

In the Downside forecast scenario, Adamas considered a future in which a low and/or persistently volatile price environment results in fewer new producers coming online than expected in the Base Case.

In this scenario, Adamas believes the reduced supply-side diversity (and resultant strengthening of China's control) coupled with resultant price uncertainty could translate to a weaker demand outlook and lower price environment than considered in the Base Case forecast scenario.



Source**:** Adamas Intelligence, 2025

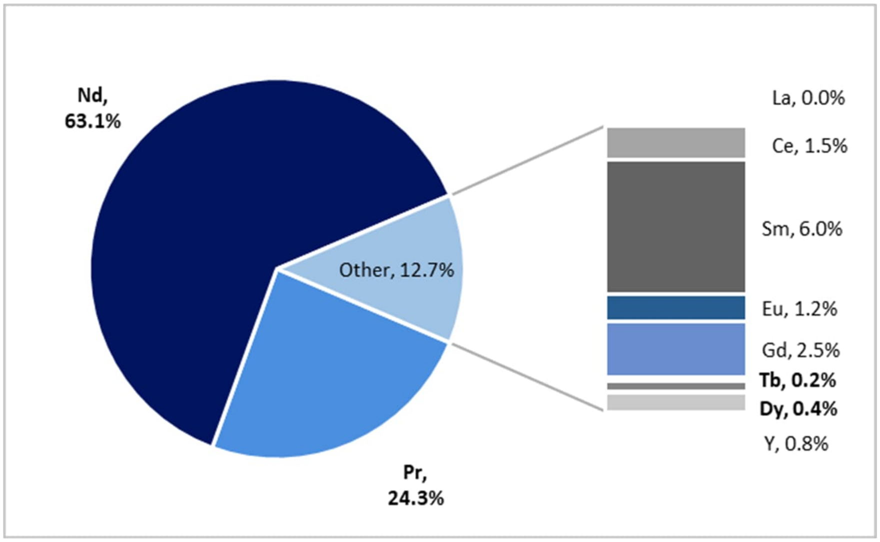
* Forecasted prices are in Real 2025 US dollars and include 13% VAT; If selling into China, VAT should be deducted; if selling ex-China prices above should be taken at face value.

Figure 19-15: Forecasted Magnet Rare Earth Oxide Prices to 2040 (Base Case)

19.16 Forecasted per-REO Contribution to Wicheeda Project Basket Value in 2032

Figure 19-16 shows the relative distribution of rare earth oxides contained in the Wicheeda project production suite. By volume, the four critical magnet rare earth oxides (neodymium, praseodymium, dysprosium and terbium) make up 87.9% of the TREO contained in the product basket.

Oxide	Relative %
La	0.0%
Ce	1.5%
Pr	24.3%
Nd	63.1%
Sm	6.0%
Eu	1.2%
Gd	2.5%
Tb	0.2%
Dy	0.4%
Ho	0.0%
Er	0.0%
Tm	0.0%
Yb	0.0%
Lu	0.0%
Y	0.8%
TREO	100.0%

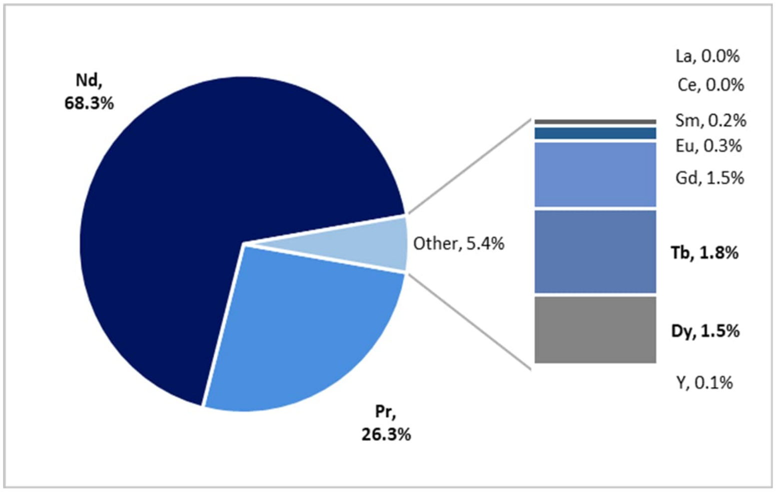


Source**:** Adamas Intelligence analysis, 2025

Figure 19-16: Relative Distribution of Rare Earth Oxides in Wicheeda Project Product Basket

Figure 19-17 shows the forecasted per-REO contribution to the Wicheeda project's TREO basket value in 2032 based on Adamas Intelligence's Base Case scenario. By value, the four critical magnet rare earth oxides (neodymium, praseodymium, dysprosium and terbium) are projected to drive 97.9% of the project's basket value in 2032, a proportion that will go largely unchanged over the forecast period.

Oxide	Relative %
La	0.0%
Ce	0.0%
Pr	26.3%
Nd	68.3%
Sm	0.2%
Eu	0.3%
Gd	1.5%
Tb	1.8%
Dy	1.5%
Ho	0.0%
Er	0.0%
Tm	0.0%
Yb	0.0%
Lu	0.0%
Y	0.1%
TREO	100.0%



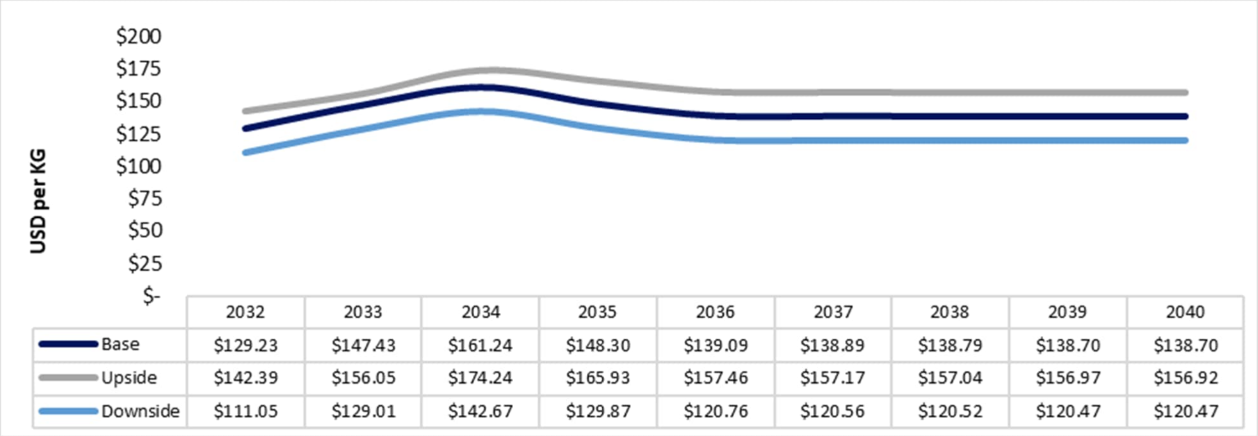
Source**:** Adamas Intelligence Analysis, 2025

Figure 19-17: Per-REO Contribution to Wicheeda Project Basket Value in 2032

19.17 Forecasted Basket Value and Implications for Project

Taking Adamas' Base Case price forecasts into account, along with the relative distribution of rare earth oxides in the Wicheeda project suite (see Figure 19-16), the project basket value (i.e., value of rare earth oxides contained in one kilogram of separated TREO produced from the project) was projected for each year from 2032 through 2040, as shown in Figure 19-18.

In Adamas' Base Case, Upside and Downside price forecast scenarios, the Wicheeda project basket value will increase overall from 2032 through 2040 at a CAGR of 0.9%, 1.2% or 1.0%, respectively.



	2032	2033	2034	2035	2036	2037	2038	2039	2040
Base	$129.23	$147.43	$161.24	$148.30	$139.09	$138.89	$138.79	$138.70	$138.70
Upside	$142.39	$156.05	$174.24	$165.93	$157.46	$157.17	$157.04	$156.97	$156.92
Downside	$111.05	$129.01	$142.67	$129.87	$120.76	$120.56	$120.52	$120.47	$120.47

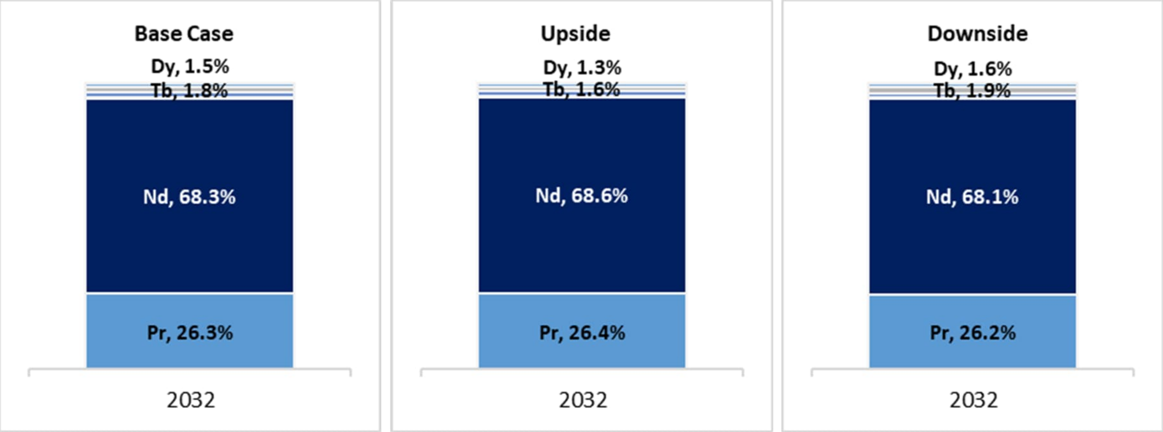
Source: Adamas Intelligence, 2025

* Basket values include 13% VAT; forecasted prices in Real 2025 dollars

* If selling into China VAT should be deducted; if selling ex-China above prices should be taken at face value

Figure 19-18: Forecasted Wicheeda Basket Value from 2032 through 2040

In all scenarios examined, Adamas projects that critical magnet rare earth oxides (i.e., neodymium, praseodymium, dysprosium and terbium) will collectively drive around 98% of the Wicheeda project basket value each year from 2032 through 2040 (see Figure 19-19).



Source: Adamas Intelligence, 2025

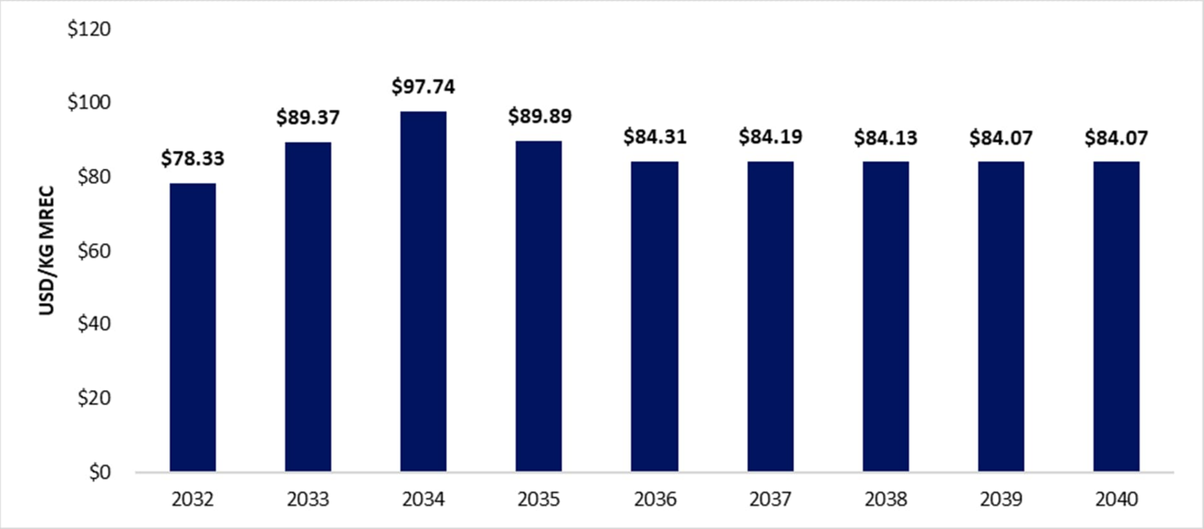
Figure 19-19: Contribution of Magnet Rare Earth Oxides to the Wicheeda Basket Value In Each Scenario

19.18 Forecasted Value of Wicheeda Mixed Rare Earth Carbonate

In Adamas' view, the mixed rare earth carbonate ("MREC") that Defense Metals plans to produce from Wicheeda would be marketable and desirable to existing and emerging rare earth separation facilities globally.

Since the MREC is almost fully devoid of low value lanthanum ("La") and cerium ("Ce"), which typically comprise 50% to 70% of the rare earth contents in a standard MREC, a prospective processor of Wicheeda concentrate would not need to tie up capacity or expend costs to treat La and Ce making the Wicheeda concentrate a premium product in Adamas' view.

Inferring from Chinese processing costs, Adamas believes that from 2032 through 2040 Defense Metals could expect to receive a price for its MREC equal to 95% of the rare earth oxide value it contains (value based on China domestic prices, excluding VAT).



Source: Adamas Intelligence, 2025

* Prices in USD per kilogram of MREC; MREC contains 72.1 weight % TREO

* Value based on forecasted China domestic prices, excluding VAT

* Forecast in Real 2025 dollars

Figure 19-20: Forecasted Value of Wicheeda MREC from 2032 through 2040

19.19 Addressable Market for Mixed Rare Earth Carbonate

Globally, the addressable market for selling mixed rare earth carbonate and other mixed rare earth chemicals continues to grow as new rare earth processing facilities are developed and existing capacity expanded.

As the largest rare earth producer and consumer globally, China is host to an extensive industry of rare earth traders and processors that actively offtake and import rare earth feedstocks from abroad.

Outside China, there is existing processing capacity in Asia, Europe and North America, plus additional capacity under development in Australia, that increase optionality for an emerging supplier of mixed rare earth carbonate.

19.20 International Supply Chain Development

Outside of China and Japan, a wave of incoming NdPr oxide supply is helping de-risk the business case for downstream investments in metals, alloy and magnet production capacity, spurring public and private sectors into action over the past 36 months.

19.20.1 Company Announcements

In the past three years, companies including MP Materials, VAC Group, USA Rare Earth, Neo Performance Materials, Star Group, Noveon Magnetics and others have started production, construction or announced plans to establish NdFeB magnet production capacity in North America and Europe within the near- to medium-term.

These developments, and others yet to come, are a testament to the upstream market's rising diversity and supply security coupled with the downstream market's rapidly increasing demand for sustainable supplies of NdFeB magnets – particularly for electric vehicles, robotics, advanced air mobility and defense sectors.

19.20.2 Alternative Supply Chains are Coming Together

These recent announcements, and others to come, reflect a growing concern in North America and Europe about supply chain sustainability, and signal the coalescing of new alternative international mine-to-magnet supply chains connecting the Americas, Europe, Africa, Asia and Australia.

These emerging supply chains, and the strategically important industries propelling them, stand to fuel rising demand for non-China mined rare earth materials in the decade ahead.

19.20.3 Government Initiatives

Governments in the U.S., Canada, Europe and Australia are actively supporting the development of new mine-to-magnet supply chains, critical for clean energy technologies like electric vehicles and wind turbines, and strategic industries like robotics, advanced air mobility and defense.

In the U.S., newly imposed import tariffs of 20% on all Chinese goods, including rare earths, layered atop existing duties, raise the cost of foreign supply, while tax credits under the Inflation Reduction Act and other federal supports for processors are bolstering domestic production.

In Canada, the Strategic Innovation Fund allocates up to $1.5 billion for critical minerals extraction and processing, fostering innovation and upstream project advancements.

In Europe, the European Commission's Critical Raw Materials Act incentivizes mining, processing and recycling, while the European Raw Material Alliance drives investment in resilient supply chains.

In Australia, the government is providing loans and incentives for companies to expand processing and establish downstream capabilities.

Together, these efforts aim to secure supply, enhance economic competitiveness and support the demands of strategically important downstream industries.

19.21 Current State of the Market and Near-Term Outlook

19.21.1 China Tightening Grip on Domestic Rare Earth Industry

In early 2025, China proposed new regulations aimed at further tightening its grip on its domestic rare earth industry. The regulations include stricter production quotas that align with national strategic interests, not necessarily market dynamics. The regulations also include the establishment of a rare earth traceability database that would allow China to strategically and expeditiously cut off supplies to specific industries or companies it exports to, enhancing its ability to wield rare earths as a geopolitical and economic weapon.

China has a history of using rare earths as leverage. In 2010 during a territorial spat, China halted rare earth exports to Japan, disrupting its electronics and automotive industries. The lack of a traceability system back then made it a blunt, broad cutoff, but it still demonstrated Beijing’s willingness to weaponize supply. Similarly, in 2023 China banned exports of rare earth magnet manufacturing technology and restricted shipments of gallium and germanium, targeting Western tech and defense sectors in response to U.S. semiconductor curbs.

While rare earths continue to be exported from China unabated today, current trade tensions with the West increase the potential that China could impose export restrictions, amplifying the need for alternative sources of supply.

19.21.2 China Becoming Increasingly Reliant On Expensive Concentrate Imports

While conventional wisdom suggests China’s rare earth industry is a super consolidated, monolithic entity – the reality is that most producers and value-adders in the nation are not vertically integrated and, as a whole, China’s industry has become increasingly reliant on foreign sources of feedstock from abroad in recent years.

In 2015, over 90% of all separated light rare earth oxides produced in China were derived from domestically mined feedstock but in 2023 this share was down to nearly 70% as the nation’s reliance on imports expands.

Overall, Adamas believes this increasing reliance on foreign sources of relatively expensive concentrate bodes well for rare earth oxide prices over the near- to medium-term.

19.21.3 Myanmar Supply Disruptions Could Persist

In recent years, Myanmar has become an indispensable supplier of rare earth concentrate to China’s processors and magnet makers. However, with a coup underway in the nation since 2021, flows from Myanmar to China have been volatile and their sustainability uncertain.

In late-2024, resistance forces captured the largest rare earth mining center in Myanmar from the military government leading to the shutdown of most mining operations in Kachin State and a steep drop in exports from Myanmar to China.

With Myanmar responsible for a major share of global magnet rare earth mine production each year (all of which is processed in China), a prolonged shutdown could significantly disrupt China’s supply of these critical elements, adding support for a rise in prices.

19.22 Key Takeaways

- From 2024 through 2040, Adamas forecasts that global TREO demand for permanent magnets will rise at a CAGR of 8.2%, boosted by strong demand growth from electric vehicles, robotics, advanced air mobility and other applications of NdFeB magnets.
- Over the same period, Adamas forecasts that global production of neodymium, praseodymium, dysprosium and terbium (the so-called magnet rare earths) will collectively increase at a slower CAGR of 5.1% as the supply side of the market increasingly struggles to keep up with rapidly growing demand.
- Post-2025 Adamas forecasts that the global rare earth industry will consistently underproduce neodymium, praseodymium, dysprosium and terbium oxides (or oxide equivalents), resulting in the depletion of historically accumulated inventories and, ultimately, shortages of these critical magnet materials if supply is not increased beyond levels currently anticipated.
- The Wicheeda project offers a very high degree of economic exposure to the rare earth permanent magnet sector, which is the fastest-growing end-use category and most in need of additional rare earth supplies, according to Adamas Intelligence. In all scenarios examined, Adamas projects that magnet rare earths will collectively drive around 98% of the Wicheeda project basket value each year from 2032 through 2040.
- In its Base Case scenario, Adamas Intelligence forecasts that the basket value of Wicheeda TREO production will total $129.23 per kilogram in 2032 and will increase to $138.70 per kilogram in 2040.
- Inferring from Chinese processing costs, Adamas believes that from 2032 through 2040 Defense Metals could expect to receive a price for its mixed rare earth carbonate (“MREC”) equal to 95% of the rare earth oxide value it contains (value based on China domestic prices, excluding VAT). Since the MREC is almost fully devoid of low value lanthanum and cerium, which typically comprise 50% to 70% of the rare earth contents in a standard MREC, a prospective processor of Wicheeda MREC would not need to tie up capacity or expend costs to treat lanthanum and cerium making the Wicheeda concentrate a premium product in Adamas’ opinion.
- In its Base Case scenario, Adamas forecasts that the value of Wicheeda MREC will amount to $78.33 per kilogram of MREC in 2032 and will increase to $84.07 per kilogram in 2040.
- In Adamas’ view, the Wicheeda MREC would be marketable and desirable to existing and emerging rare earth separation facilities globally.

19.23 QP Interpretation and Recommendation

The long-term pricing from the Adamas Base Case described in Section 19.15.1 and Figure 19-15 is recommended as the reference price for resources and reserves estimation and the economic analysis in Section 22.

In the Base Case the long-term price for NdPr oxide is $132.74/kg (based on the Adamas price of $150/kg less 13% VAT for sales outside China), while the prices for Tb oxide and Dy oxide are $1,362.8/kg and $442.5/kg, respectively, based on the same VAT deduction. The corresponding long-term prices for the balance of individual REOs are also recommended but are not disclosed individually in this paragraph to satisfy the usage requirements of the Adamas market report.

19.24 Contracts

Defense Metals has not entered into any material commercial agreements related to its products at the time of this report's publication.

20. Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Existing Conditions Studies

Wicheeda project is located in an ecologically rich area that supports diverse fish and wildlife populations. While the immediate project area lacks permanent settlements it holds significant cultural and traditional value for the Indigenous peoples and supports the cultural, recreational, conservation and economical functions of the larger region.

Baseline studies for geochemistry, climate and meteorology, surface water quality, hydrology, and groundwater are described in the following sections. For several environmental areas, baseline conditions were characterized based on publicly available data and will be more fully developed during the 2025 and 2026 field season. Summaries of the demographics and social characteristics of communities proximal to the project are provided. Refined socio-economic baseline studies to support environmental assessment processes will commence in 2026.

20.1.1 Geochemistry

20.1.1.1 Waste Rock

Static and kinetic testing has been conducted on samples of waste rock that may represent material disposed in the WSF or remaining in pit walls.

A total of 231 samples have undergone static testing including 155 sedimentary rock, 41 syenite, 13 dolomite carbonatite, 17 xenolithic dolomite carbonatite, 2 mafic intrusive and 3 igneous breccia. Static testing included: acid base accounting (ABA), multi-element analysis (including REE), and fluorine analysed at ALS Minerals in North Vancouver. A sub-set of samples were analysed for mineralogy (QEMSCAN) at ALS Kamloops, and radionuclides in naturally occurring radioactive materials (NORMs), at Saskatchewan Research Council (SRC) in Saskatoon.

Static results from 231 samples indicated that:

- Acid rock drainage (ARD) potential of waste rock was classified as non-potentially acid generating (non-PAG), except for four sedimentary samples with uncertain potential for ARD. Samples with uncertain ARD potential tended to be strongly fenitized sedimentary rocks with around 3% sulphide (as %S).
- Elements that were commonly enriched in the sedimentary rock samples compared to average crustal abundances were cerium and molybdenum in more than 25% of samples.
- Several elements were commonly enriched in syenite samples compared to average crustal abundances including sulphur, molybdenum and thorium in more than 50% of samples, and arsenic, strontium, lanthanum, and cerium in more than 25% of samples.

- Several elements were enriched in dolomite carbonatite or xenolithic dolomite carbonatite samples compared to average crustal abundances including sulphur, molybdenum, selenium, lanthanum, cerium, and thorium in more than 50% of samples, in addition to arsenic and strontium in more than 25% of xenolithic dolomite carbonatite samples, and arsenic and silver in more than 25% of dolomite carbonatite samples.
- Radionuclide activity was compared to Health Canada’s NORM guidelines for unconditional derived release limits for diffuse NORM sources for solids, for samples that had radionuclide results from SRC, and also for all static samples by calculating radionuclide activity from uranium and thorium ICP-MS results (following 4-acid digestion), and assuming equilibrium between the parent and daughter radionuclides (which is reasonable for rock prior to mineral processing). Where more than one long-lived radionuclide is present, the “sum of ratios” needs to be applied whereby the radionuclides are compared to their specific guideline and these ratios are summed and compared to the sum of ratios limit. For syenite, dolomite carbonatite, and xenolithic dolomite carbonatite, all samples exceeded the sum of ratios limit, whereas 44% of the sedimentary rock samples exceeded the sum of ratios limit.

Humidity cell testing (HCT) is being conducted at ALS Environmental in Burnaby, BC on 18 samples including: 12 sedimentary rock, 3 syenite, 1 dolomite carbonatite, and 2 xenolithic dolomite carbonatite. Leachates from all cells are analysed for major and trace elements, including REEs. Ten of the cells are standard 1 kg HCTs and eight are 2 kg HCTs. The 2 kg cells provide sufficient leachate for analysis of radionuclides through generating 4-week composite samples, that are sent to SRC for analysis. The HCTs have been operating for 48 weeks as of February 2025.

Five on-site kinetic tests (barrel tests), each containing 215 kg to 237 kg of crushed rock (<2.54 cm particle size) were set up near the Wicheeda project site in February 2024. Four tests contain sedimentary rock and one test contains syenite, representing the most abundant waste rock types. Leachates from the tests were sampled five times during 2024 and analyzed for major and trace elements including REEs at ALS Environmental in Burnaby, BC, and radionuclides at SRC in Saskatchewan.

Kinetic test results were used as an input to project water quality predictions as described in Section 18.5.6.

Concentrations of key parameters in the static samples that describe ARD potential (i.e. sulphur, acid potential (AP), neutralization potential (NP), NP/AP); and elements that were most commonly enriched, or expected to be mobile at mildly alkaline pH are shown in Table 20-1. Radionuclide activities compared to the NORM solids guidelines are shown in Table 20-2.

Table 20-1: Summary of Key Static Results for Waste Rock and Metallurgical Testing Samples

Lithology	Statistic	AP	Statistic for NP and NP/AP	NP	NP/AP	Total S	As	Mo	Se	La	Ce	U	Th
		kgCaCO3/t		kgCaCO3/t	Ratio	%	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg
Sedimentary Rocks (n=155)	Median	6.3		299	48	0.22	7.2	8.9	0.075	190	330	1.6	30
	P75	13	P25	197	21	0.45	11	27	0.17	450	750	2.2	54
	P95	69	P5	129	4.4	2.2	36	110	0.6	1100	1800	5.9	170
Syenite (n=41)	Median	16		339	22	0.55	8.2	15	0.32	680	1300	17	150
	P75	26	P25	262	12	0.84	16	33	0.44	950	1700	27	200
	P95	47	P5	174	4.0	1.5	32	200	0.62	1800	2900	38	290
Dolomite Carbonatite (n=13)	Median	25		769	32	0.79	7.3	110	0.6	2900	4600	3.3	370
	P75	58	P25	655	11	1.9	26	190	0.7	4900	7500	6.9	480
	P95	73	P5	617	10	2.4	79	530	1.3	7600	10000	31	560
Xenolithic Dolomite Carbonatite (n=17)	Median	20		632	37	0.65	10	25	0.41	1600	2900	8.2	270
	P75	35	P25	430	11	1.2	18	69	0.54	2000	3000	18	340
	P95	63	P5	360	6.3	2.1	30	200	1.2	3500	5500	40	580
Ore composite (n=1)		22	-	408	19	0.70	42	93	8.3	6500	9200	10	550
Flotation Tailings (n=1)		21	-	360	17	0.69	16	100	1.9	950	1400	8.1	350
Tailings+WL/PN Residue (n=1)		15		343	23	1.00	21	93	3.2	1700	2600	8.3	550
WL/PN Residue (n=2)	Mean	53	-	8.5	**0.16**	15	69	42	36	34500	45500	1.7	7900
LaCe-MgR Residue (n=1)		6.3	-	220	35	12	292	0.2	21	47000	62000	0.37	2.6
Average crustal abundance of shale X10						2.4	130	26	6	920	590	37	120
Average crustal abundance of syenite X10						0.3	14	6	0.5	700	1610	30	130

Source: SRK, 2025

Notes:

1. 'n' is the number of samples tested AP=acid potential, NP=neutralization potential.
2. **Red font** indicates potentially acid generating (PAG), green highlighting indicates non-PAG.
3. As, Mo, Se, La, Ce, U, Th results are from a 4-acid digestion. Average crustal abundances reported in Price (1997). Orange highlighting indicates enriched (i.e., 10 times higher) compared to average crustal abundance of shale, purple highlighting indicates enriched (i.e., 10 times higher) compared to average crustal abundance of syenite

Table 20-2: Summary of Static Radionuclide Results for Waste Rock and Metallurgical Testing Samples

		Th-232 Decay Chain			Uranium-238 Decay Chain				
Lithology	**Statistic**	**Th-232**	**Ra-228**	**Th-228**	**U-238**	**Th-230**	**Ra-226**	**Pb-210**	**Sum of Ratios**
		Bq/g	**Bq/g**	**Bq/g**	**Bq/g**	**Bq/g**	**Bq/g**	**Bq/g**	
Sedimentary Rocks (n=155)	Median	0.12	0.12	0.12	0.02	0.02	0.02	0.02	0.91
	P75	0.22	0.22	0.22	0.03	0.03	0.027	0.027	1.6
	P95	0.68	0.68	0.68	0.07	0.07	0.073	0.073	4.9
Syenite (n=41)	Median	0.62	0.62	0.62	0.21	0.21	0.21	0.21	5.3
	P75	0.83	0.83	0.83	0.34	0.34	0.34	0.34	6.6
	P95	1.2	1.2	1.2	0.47	0.47	0.47	0.47	8.6
Dolomite Carbonatite (n=13)	Median	1.5	1.5	1.5	0.04	0.04	0.041	0.041	10
	P75	2.0	2.0	2.0	0.09	0.09	0.086	0.086	13
	P95	2.3	2.3	2.3	0.39	0.39	0.39	0.39	16
Xenolithic Dolomite Carbonatite (n=17)	Median	1.1	1.1	1.1	0.10	0.10	0.10	0.10	8.6
	P75	1.4	1.4	1.4	0.22	0.22	0.22	0.22	9.6
	P95	2.4	2.4	2.4	0.49	0.49	0.49	0.49	16
Ore composite (n=1)		1.7	2.1	2.2	0.09	<1	0.50	0.20	17
Flotation Tailings (n=1)		1.1	1.5	1.5	0.09	<0.9	0.20	0.09	11
Tailings+WL/PN Residue (n=1)		1.8	2.2	2.4	0.078	<1	0.20	<0.08	16
WL/PN Residue (n=2)	Mean	26	30	32	0.02	<3	<0.45	<0.25	210
LaCe-MgR Residue (n=1)		<0.001	<0.01	<0.004	0.004	<0.3	<0.09	<0.03	0.48
UDRL Diffuse NORM Sources		10	0.3	0.3	10	10	0.3	0.3	1

Source: SRK, 2025

Notes:

1. 'n' is the number of samples tested
2. Pink highlighting indicates exceeded the unconditional derived release limits (UDRL) for diffuse NORM sources (Health Canada 2014).

20.1.1.2 *Tailings and Hydrometallurgical Waste Streams*

Static testing has been conducted on an ore composite, a flotation tailings sample, two samples of the WL/PN hydrometallurgical residue, one sample of WL/PN hydrometallurgical residue blended with flotation tailings (in the approximate proportions to be disposed in the FTSF), and one sample of the LaCe-MgR hydrometallurgical residue.

Testing was conducted for the same analytes as for waste rock described above, but at SGS Lakefield in Ontario for most parameters, and SRC in Saskatoon for radionuclides. Select static results are provided in Table 20-1, with radionuclide results provided in Table 20-2. Static results indicated that:

- Acid rock drainage potential of flotation tailings, WL/PN residue blended with flotation tailings, and LaCe-MgR residue were classified as non-PAG, whereas the WL/PN residue was classified as PAG. The WL/PN residue had average paste pH of 1.2 and therefore was acid generating when tested.
- Similar to waste rock, flotation tailings were enriched in sulphur, arsenic, molybdenum, selenium, lanthanum, and thorium compared to average crustal abundances. The

hydrometallurgical residues were not assessed for element enrichment compared to average crustal abundances as they are not expected to be similar to rock.

- Radionuclide activity was compared to Health Canada’s NORM guidelines for unconditional derived release limits for diffuse NORM sources for solids. Flotation tailings, WL/PN residue blended with flotation tailings, and the WL/PN residue samples exceeded the sum of ratios limit, whereas the LaCe-MgR residue sample was below the sum of ratios limit.

Humidity cell testing is being conducted at SGS Lakefield on the flotation tailings sample and a blend of flotation tailings with WL/PN residue. Leachates from both cells are analysed for major and trace ions including REEs. The cells are 2 kg HCTs and provide sufficient leachate for analysis of radionuclides through generating 4-week composite samples, that are sent to SRC for analysis. The HCTs have been operating for 43 weeks as of February 2025. The kinetic results were used as an input to project water quality predictions as described in Section 18.5.6

20.1.2 *Climate and Meteorology*

Climate and meteorology baseline studies have been initiated for the project and are ongoing to ensure high-quality data sets for water, climate and wind dispersion modelling. In 2020, an initial climate station was installed at Wicheeda Lake. The Wicheeda Lake climate station is 10 meters in height and equipped with sensors for air temperature, relative humidity, wind speed and direction, barometric pressure, solar radiation, all-weather precipitation, and snow depth.

In 2021, an additional climate station was installed (Wicheeda Alpine) to capture and record higher-elevation meteorological data, as it is expected that the weather and climate data will represent the Project's entire footprint. The Wicheeda Alpine station is 3 meters in height and equipped with air temperature, relative humidity, wind speed/direction, barometric pressure, rain, and snow depth sensors.

To understand snowpack dynamics and improve the accuracy of future hydrological models, a snow survey was established at each station during the 2022-23 winter season, and snow surveys were conducted during the 2023-24 and the 2024-25 winter season to capture snow water equivalent and snow density.

Quality long-duration regional data exists and will be corelated with site data and utilized as a long period of record dataset. Site meteorological baseline studies will be continued throughout the entire project life and a climatological assessment will be conducted to support the modelling for environmental assessment and engineering design. Upgrades to the precipitation collection equipment are required and will be completed in 2025.

The need for a meteorological monitoring station at the hydrometallurgical plant location will be determined and mobilized prior to the initiation of refined studies in that location, if required.

20.1.3 Soils and Vegetation

In advance of conducting baseline studies, readily available information has been summarised to provide a preliminary description of the soils and vegetation setting in the vicinity of the Project site.

The mine site area elevation ranges from approximately 1,320 masl to 810 masl and includes a mix of terrains from gentle slopes to moderate hills. Forests and logged areas cover most of the land. The Project primarily overlaps the Fraser Basin ecoregion, which is located on tertiary and volcanic bedrock covered with glacial deposits. The soils of the Fraser Basin are mostly Gray Luvisols and Dystric Brunisols. Within this ecoregion, the Project overlaps two biogeoclimatic zones: the Sub-boreal Spruce (SBS) and Engelmann Spruce-Subalpine Fir (ESSF) zones. The SBS zone overlaps most of the project and is characterized as having many wetlands and hybrid white spruce and subalpine fir forests.

The ESSF overlaps a small area of the eastern portion of the Project area. The ESSF zone is characterized by Engelmann spruce and subalpine fir forest. Lodgepole pine can be found across both biogeoclimatic zones in wetter areas. The project overlaps three subzone variants: SBS very wet cool (SBSvk), SBS wet cool 1 (SBSwk1), and ESSF wet cool 2 (ESSFwk2).

Publicly available 1 Ha resolution predictive mapping of soil parent types within the mine site area indicates that there are areas of weathered bedrock and colluvium within the WSF and open pit area and lacustrine soils near to the base of the FTSF. The areas adjacent to Wichcika Creek have continuous glaciofluvial and fluvial soils.

Vegetation and soils baseline studies, planned to commence in 2026, will refine soils characterization and will identify plant species and plant communities with any potential overlaps with the lifecycle habitats of federally and provincially listed species at risk. These findings will inform the environmental assessment's analysis of the project's potential to interact with such species and guide the development of appropriate mitigation measures if required.

20.1.4 Wildlife and Wildlife Habitat

A preliminary description of the setting for wildlife and wildlife habitat in and around the project area, including a desktop assessment of potentially occurring species of conservation concern has been conducted in advance of refined field baseline studies for wildlife and wildlife habitat. Field wildlife studies will commence in the spring of 2026.

The mature and old-growth coniferous forest of the SBS biogeoclimatic zone provides habitat for many wildlife species, including ungulates, furbearers, birds, amphibians, and reptiles. For example, moose (Alces alces) is an ungulate species that commonly occurs in the SBS zone, and mountain caribou can be found at higher elevations.

The project footprint overlaps with mapped critical habitat (i.e., matrix range) in the Hart Ranges for the threatened mountain caribou (Southern Mountain population; Rangifer tarandus). A proposed agreement for the conservation of the Southern Mountain population, including the Hart Ranges Subpopulation that may utilize the project area for part of their lifecycles, has been developed between the MLIB and the federal government. The Hart Ranges subpopulation is the largest population of the southern group of Southern Mountain caribou. Wolf predation was identified as a limiting factor, and five years of wolf reduction efforts conducted by the provincial government have yielded population increases. The current population estimate for the Hart Ranges is 628, and the provincial government will be conducting another population survey in March 2025. It is accepted that mine sites deter caribou within an approximate 2 km zone when they are active, and a mitigation strategy will need to be proposed and accepted to ensure impacts are limited and an overall benefit to the species is achieved.

Furbearers that can be found near the project include fisher (Pekania pennanti), martin (Martes americana), ermine (Mustela erminea), and the grey wolf (Canis lupus). A wide variety of bird species use the SBS zone for their entire life cycle. Bird guilds that occur in the project area include raptors, upland game birds, songbirds, waterfowl, shorebirds, and corvids. Migratory bird species also spend certain life stages in the project area.

Amphibians that occur in the project area include wood frog (Lithobates sylvaticus) and western toad (Anaxyrus boreas), which breed in aquatic environments but spend the remainder of their life cycle primarily in terrestrial environments. Reptile species that may occur in the area include common garter snake (Thamnophis sirtalis), terrestrial garter snake (Thamnophis elegans), and western painted turtle (Chrysemys picta bellii).

Based on the desktop assessment, there are 22 listed wildlife species (i.e., Schedule 1 of the Species at Risk Act, the Committee on the Status of Wildlife in Canada [COSEWIC], and/or provincially Red- or Blue-listed) that may occur in the project area. This includes 14 birds, seven mammals, one amphibian, and one reptile (Table 20-3.). The presence of these species will be verified through baseline surveys planned for 2026.

Table 20-3. Federally and Provincially Listed Wildlife Species that have the Potential to Occur within the Project Area

Common Name	Scientific Name	SARA Status1	COSEWIC1	BC List2
Amphibian				
Western Toad	Anaxyrus boreas	Special Concern	Special Concern	Yellow
Bird				
Black Swift	Cypseloides niger	Endangered	Endangered	Blue
Bank Swallow	Riparia riparia	Threatened	Threatened	Yellow
Barn Swallow	Hirundo rustica	Threatened	Special Concern	Yellow
Short-eared Owl	Asio flammeus	Special Concern	Threatened	Blue
Common Nighthawk	Chordeiles minor	Special Concern	Special Concern	Blue
Rusty Blackbird	Euphagus carolinus	Special Concern	Special Concern	Blue
Evening Grosbeak	Coccothraustes vespertinus	Special Concern	Special Concern	Yellow
Long-billed Curlew	Numenius americanus	Special Concern	Special Concern	Yellow
Olive-sided Flycatcher	Contopus cooperi	Special Concern	Special Concern	Yellow

Common Name	Scientific Name	SARA Status1	COSEWIC1	BC List2
Swainson's Hawk	Buteo swainsoni	No status	No status	Red
American Bittern	Botaurus lentiginosus	No status	No status	Blue
Eared Grebe	Podiceps nigricollis	No status	No status	Blue
Sharp-tailed Grouse, columbianus subspecies	Tympanuchus phasianellus columbianus	No status	No status	Blue
Northern Goshawk, atricapillus subspecies	Accipiter gentilis atricapillus	No status	Not at risk	Blue
Mammal				
Little Brown Myotis	Myotis lucifugus	Endangered	Endangered	Blue
Northern Myotis	Myotis septentrionalis	Endangered	Endangered	Blue
Caribou (Southern Mountain Population)	Rangifer tarandus pop. 1	Threatened	Endangered	Red
Grizzly Bear	Ursus arctos	Special Concern	Special Concern	Blue
Wolverine, luscus subspecies	Gulo gulo luscus	Special Concern	Special Concern	Blue
Hoary Bat	Lasiurus cinereus	No status	Endangered	Blue
Reptile				
Western Painted Turtle – Intermountain – Rocky Mountain Population	Chrysemys picta bellii	Special Concern	Special Concern	Blue

Source: SRK, 2025

[1] Endangered: a species that is facing imminent extirpation or extinction. Threatened: a species that is likely to become an endangered species if nothing is done to reverse the factors leading to its extirpation or extinction. Special Concern: A species that may become a threatened or an endangered species because of a combination of biological characteristics and identified threats.

[2] Red: species identified as either Endangered or Threatened. Blue: species identified as Special Concern. Yellow: species not at risk.

20.1.5 Surface Water

The Project is located in the upper reaches of the Peace River watershed, upstream of the WAC Bennet Dam and the Williston Reservoir. The Project mine site area footprint is situated in the Parsnip Watershed and the Crooked/Pack Watershed that drains to the northwest into the Williston Reservoir and is part of the Mackenzie (Arctic) drainage system.

The proposed mine is located in the Wichcika Creek Watershed. Wichcika Creek is a 5th order stream with a total drainage area of approximately 182 km^2 (Hagen and Gantner 2020). Wichcika Creek drains generally north and discharges into the Parsnip River, which then drains into the Williston Reservoir. Wicheeda Lake, which drains into Wichcika Creek, is located to the northeast of the mine site infrastructure. The hydrometallurgical plant is within the Crooked River Watershed, also a tributary of the Mackenzie drainage system.

Quarterly water sampling was completed between 2020 through 2022. In 2023 and until September 2024, baseline surface water quality and flow monitoring for the Wicheeda property was collected monthly in accordance with British Columbia environmental assessment guidelines of data for a period of 2 years, including a 5-in-30 sampling program. Water quality sampling was paused after November 2024, to allow for a compilation of data and redesign of the monitoring network to capture the evolving mine site area infrastructure footprint. The water quality sampling program will recommence in Q2 2025.

Surface water data collected will be used to characterize baseline conditions, which supports the development of environmental management strategies. While not required under provincial guidance, routine analysis of REEs forms part of the surface water quality program; given that the Project is focused on mining REE, Defense aims to establish an appropriate baseline dataset to monitor water quality during future operations.

Currently, eight hydrometeorological stations are established for the Project with preliminary rating curves developed (except for the Wicheeda Lake site) to construct discharge hydrographs, which allow for peak flow and 7-day low-flow periods to be identified. As the location of the FTSF was not determined at the inception of the surface water program and hydrometeorological program, additional sites will be developed to characterize the area west of Wichcika Creek. Additional years of baseline monitoring are planned.

20.1.6 Groundwater

The groundwater baseline program requirements under the Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators (British Columbia Ministry of Environment 2016) include one year of quarterly sampling.

Twelve hydrogeological monitoring wells have been developed at the mine site and additional ones will be required for coverage of site infrastructure areas which changed since 2023 when the well locations were chosen. Additional locations will also be chosen to reflect the need for sentinel wells for life of mine monitoring. The monitoring wells, including those additions, are planned for installation in 2025 and will be sampled at quarterly intervals to provide inputs to the conceptual and numerical hydrogeological models required for the assessment and permitting processes as well as ensure that a monitoring network remains for life of mine.

The baseline groundwater monitoring program commenced in May 2024.

20.1.7 Fisheries and Aquatics

A high-level assessment of fisheries and fish habitat, including aquatic invertebrates, was conducted in April 2024 using publicly available literature and government databases.

Additional data around fish habitat and fish passage in the Parsnip watershed was conducted by the forest licensee, consultants and MLIB under the guidance of the provincial government with the purpose being fish passage restriction identification and the ultimate goal of fish passage restoration. Numerous fish passage barriers and restrictions were identified on Wichcika Creek and Wichcika Creek tributaries.

Fish species that have been historically captured in waterbodies in the vicinity of the project are identified in Table 20-4. In general, the fish species found in the project area are typical of the diverse community in the upper Peace River Watershed.

Conservation statuses of the identified fish species are presented in Table 20-5 at both provincial and federal levels. Most species are apparently secure, with ‘Yellow’ provincial classification and no status assigned by COSEWIC. Bull Trout and Brassy Minnow (Western Arctic populations) are both on the provincial ‘Blue’ list and are classified as ‘Special Concern’ by COSEWIC (BC CDC 2024, COSEWIC 2012, 2022). The Fisheries and Oceans Canada Aquatic Species at Risk Map identifies potential Bull Trout habitat throughout the Upper Peace Watershed, including the Wichcika Creek and Chuchinka Creek Watersheds (DFO 2024). Brassy Minnow has not been historically captured in the watersheds in the immediate vicinity of the Project.

Table 20-4. Historical Fish Capture Results in Waterbodies Within and Downstream of the Project Area

Species	Wicheeda Lake	Wichcika Creek	Parsnip River	Chuchinka Creek	Angusmac Creek	Williston Watershed[1]
Arctic Grayling		✓	✓			✓
Brassy Minnow						✓
Brook Trout			✓			
Bull Trout[2]		✓	✓		✓	✓
Burbot		✓	✓	✓	✓	✓
Kokanee			✓			✓
Lake Chub			✓	✓	✓	✓
Lake Trout						✓
Lake Whitefish			✓			✓
Largescale Sucker			✓			✓
Longnose Dace			✓		✓	✓
Longnose Sucker			✓	✓	✓	✓
Mountain Whitefish		✓	✓		✓	✓
Northern Pikeminnow			✓			✓
Peamouth Chub			✓			✓
Prickly Sculpin			✓		✓	✓
Pygmy Whitefish			✓			✓
Rainbow Trout	✓	✓	✓	✓	✓	✓
Redside Shiner			✓		✓	✓
Slimy Sculpin		✓	✓			
White Sucker			✓			✓

Source: British Columbia Conservation Data, 2024

[1] This includes fish historically found in the following waterbodies: Angusmac Creek, Crooked River, Redrocky Lake, Kerry Lake, McLeod Lake, Pack River, and Williston Reservoir.

[2] Dolly Varden, rather than Bull Trout, are identified as present in historical capture data, but this likely a misidentification of Bull Trout due to similarities of distinguishing characteristics.

Table 20-5: Provincial and Federal Conservation Status of Fish Species Found Within the Project Area

Common Name	Scientific Name	BC Status[1]	COSEWIC Status[2]
Arctic Grayling	Thymallus arcticus	Yellow	No Status
Brassy Minnow	Hybognathus hankinsoni	Blue	Special Concern (Western Arctic population)
Brook Trout	Salvelinus fontinalis	Exotic	No Status
Bull Trout	Salvelinus confluentus	Blue	Special Concern (Western Arctic population)
Burbot	Lota lota	Yellow	No Status
Kokanee	Oncorhynchus nerka	No listing	No Status
Lake Chub	Couesius plumbeus	Yellow	No Status
Lake Trout	Salvelinus namaycush	Yellow	No Status
Lake Whitefish	Coregonus clupeaformis	Yellow	No Status
Largescale Sucker	Catostomus macrocheilus	Yellow	No Status
Longnose Dace	Rhinichthys cataractae	Yellow	No Status
Longnose Sucker	Catostomus catostomus	Yellow	No Status
Mountain Whitefish	Prosopium williamsoni	Yellow	No Status
Northern Pikeminnow	Ptychocheilus oregonensis	Yellow	No Status
Peamouth Chub	Mylocheilus caurinus	Yellow	No Status
Prickly Sculpin	Cottus asper	Yellow	No Status
Pygmy Whitefish	Prosopium coulterii	Yellow	Not at Risk
Rainbow Trout	Oncorhynchus mykiss	Yellow	No Status
Redside Shiner	Richardsonius balteatus	Yellow	No Status
Slimy Sculpin	Cottus cognatus	Yellow	No Status
White Sucker	Catostomus commersonii	Yellow	No Status

Sources: British Columbia Conservation Data Centre 2024, ECCC 2024

[1] Yellow listing indicates that the species is ‘Apparently Secure’. Blue listing indicates that the species is considered to be of ‘Special Concern’ and at risk, but not extirpated, endangered, or threatened.

[2] No Status indicates that the species is not listed in SARA. ‘Special Concern’ indicates that the species may become a threatened or endangered species because of a combination of biological characteristics and identified threats.

20.1.8 Indigenous Knowledge and Traditional Land and Resource Use Information

Defense Metals entered into a Co-Design Agreement with the MLIB in January 2024 to support ongoing development of the Project. The Co-Design Agreement is intended to facilitate MLIB’s participation in and development of the environmental assessment process for the project, including in the planning and design of certain technical studies relating to the project. Accordingly, Defense Metals envisions that MLIB will be integrally involved in future environmental studies, including providing perspectives informed by MLIB’s Indigenous knowledge and traditional land and resource use information. Defence Metals also remains open to collaborating on environmental studies with other Indigenous nations that could be adversely impacted by the Project.

20.1.9 Next steps

Baseline studies for surface water quality, hydrology, groundwater, and meteorology and climate commenced in 2020 and were paused in Q4 2024 to allow for a compilation of data and redesign of the monitoring network to capture the evolving mine site area infrastructure footprint and ensure appropriate coverage. The water quality program will recommence in Q2 2025. Planning for baseline studies for wildlife and wildlife habitat, soils and vegetation, fish and fish habitat, and aquatic resources began in early 2024, with fish and aquatic resources field studies commencing during the 2024 field season. Fish and fish habitat and aquatic resources work compiled to date will be utilized to refine the workplan for additional work in the 2025 and 2026 field seasons. Wildlife and wildlife habitat, and soils and vegetation planning will recommence in 2025 with field work anticipated in 2026. Data collection and reporting will be conducted in accordance with relevant guidance documents published by the Government of British Columbia. Baseline studies will incorporate existing local and regional data and Indigenous knowledge with a view toward establishing a robust baseline for consideration in the environmental assessment for the Project.

20.2 Waste Management and Disposal, Water Management, and Site Monitoring

20.2.1 Tailings and Waste Rock

As described in detail in Section 18, the proposed WSF and FTSF are mine waste management facilities that will be progressively developed and operated over the life of the mine. The FTSF design considers management of all mine flotation tailings in a single, lined facility on the western side of Wichcika Creek. The FTSF will also store WL/PN residue and loaded uranium ion exchange resin from the hydrometallurgical plant. The non-WL/PN hydrometallurgical waste residues will be stored in the HWSF, adjacent to the hydrometallurgical plant. All tailings and hydrometallurgical waste residues will be dewatered using filter presses or membrane filter presses (see Section 17) and placed and compacted in their respective facilities.

The WSF has been located on the southern edge of the open pit. Waste rock will be used to the extent possible in construction activities including roads, berms, infrastructure pads and embankments.

Defense Metals will continue to evaluate all aspects of the mine design, including alternatives for mine waste management, informed by future technical and environmental studies, and ongoing collaboration with Indigenous rightsholders.

20.2.2 NORM Management

Mineral occurrences of REEs typically contain measurable concentrations of NORMs – e.g., potassium, uranium, thorium. The principal elements of interest for the project are uranium and thorium, and radioactive decay products, such as radium (Table 20-2).

20.2.2.1 Uranium

Uranium (U) is commonly found in almost all soils, rocks, as well as in groundwater and sea water. The U concentrations in soils and rocks are typically low, e.g., 2-10 mg/kg; in sea water at 3 parts per billion. These levels are termed “background” levels. The earth’s crust contains about 2.5 mg/kg U on average.

The average uranium concentration in the Wicheeda ore has been measured to be 11 mg/kg, a level that can be described as within or close to natural background concentrations. Uranium concentrations in samples representing waste rock are provided in Table 20-1, and vary by rock type. Sedimentary waste rock has levels similar to background (typically 0.3-6 mg/kg), whereas syenite and xenolithic dolomite carbonatite waste rock have slightly higher uranium concentrations (typically 3-40 mg/kg). Flotation tailings and WL/PN residue blended with flotation tailings have around 8 mg/kg uranium. The WL/PN residue has 1.7 mg/kg uranium, whereas the LaCe-MgR residue sample has 0.37 mg/kg uranium.

20.2.2.2 Thorium

Thorium, like uranium, is typically found in many rocks and soils but background levels are typically higher than for uranium with the average thorium content of the earth’s crust of about 11 mg/kg and a range of about 5 to 15 mg/kg. Thorium is relatively immobile in rocks and soils and does not readily migrate in the natural environment or to mobilize in fresh or salt water.

Thorium concentrations in the Wicheeda ore vary based on rock type and REE grade with averages of 52 to 324 mg/kg thorium in low grade ores and averages of 176 to 482 mg/kg thorium in high grade ores. Concentrations in waste rock (Table 20-1) are typically 10-170 mg/kg in sedimentary rocks, and 50-300 mg/kg in syenite waste rock. Flotation tailings have 350 mg/kg thorium, WL/PN residue blended with flotation tailings have 550 mg/kg thorium, whereas uncombined WL/PN residue has 7900 mg/kg thorium (Table 20-1). The LaCe-MgR residue has 2.6 mg/kg thorium.

20.2.2.3 NORM Activities

Radionuclide activities of uranium (U-238) and thorium (Th-232) and their long-lived decay products are provided in Table 20-2, compared to Health Canada (2014) solids guidelines for unconditional derived release limits for diffuse NORM sources. For samples representing syenite, dolomite carbonatite, and xenolithic dolomite carbonatite waste, all samples exceeded the limits, whereas 44% of sedimentary waste rock samples exceeded the limits. Flotation tailings, WL/PN residue blended with flotation tailings, and the WL/PN residue samples exceeded the limits, whereas the LaCe-MgR residue sample was below the limits. Not all hydrometallurgical waste streams have been tested, including the loaded uranium ion exchange resin (which is periodically blended with WL/PN residue).

A site radiation risk assessment will be required to determine NORM management project requirements.

20.2.2.4 NORM Risk Management

The project operations will be expected to include radiation risk management, radiation monitoring and a radiation management plan to reduce radiation dose to workers to a level that is as low as reasonably achievable.

Health Canada (2020) has outlined a scaled approach for NORM management as shown in Table 20-6. Based on the data available, it could be expected that while some project workers in select roles might fall in the Dose Management Category, it is anticipated that most workers would fall in the NORM Management category. The category (0.3<Dose<1.0 mSv/y) could be expected to apply to a well-engineered Wicheeda concentrator and hydrometallurgical plant (Arcadis, 2023).

Table 20-6: NORM Radiation Program Classifications (Health Canada, 2014)

Annual Dose (above background) - mSv/y	Radiation Protection Program REE Mineral Processing and Hydrometallurgicalallurgical Facilities
Dose <0.3	No Requirements for Dose Management
0.3<Dose<1.0	NORM Management Radiation Surveys of Work Areas
1.0 < Dose < 5.0	Dose Management Dose estimates via radiation surveys and worker exposure times. Worker dose to be reported to National Dose Registry. Expert advice recommended.
Dose > 5.0	Radiation Management Formal radiation protection program and the use of TLD's for worker dose measurement. Expert advice should be obtained.

Source: Health Canada, 2014

20.2.3 *Water Management*

Defense Metals intends to adopt several water management objectives for the Project, including managing site water so that sufficient water is available for material processing, minimizing the potential for storm flows to cause damage to mine infrastructure, and minimizing the risk of adverse effects to water quality in the downstream receiving environment. As described in detail in Section 18, the strategies applied to achieve these objectives are to divert noncontact water around the project wherever possible to keep it clean, use and recycle the water from the CWP for processing, and manage sediment mobilization and erosion through Best Management Practices through the life of the mine.

Diversion ditches will be constructed upslope of the WSF and the FTSF to divert clean run-off from the upper catchments around the facilities. Groundwater and precipitation will be dewatered from the open pit throughout mine life, with dewatering flows from the pit sumps pumped to the CWP for use in processing. Filtrate from the tailings filter plant will be returned directly to the process plant for reuse.

Seepage and runoff from the WSF will be collected in downslope seepage collection ditches and subsequently diverted to the CWP. At the FTSF, a water management pond is designed at the downstream toe to collect runoff from the facility, as well as any potential seepage through the stack. Given the tailings will be dewatered and placed and compacted to achieve density requirements, minimal seepage is anticipated. Contact water collected in the FTSF pond will be pumped to the CWP.

Water treatment for removal of molybdenum and uranium will be required before water from the CWP can be acceptable for discharge to Wichcika Creek. The water treatment process for removal of molybdenum and uranium is ferric co-precipitation. The water treatment plant is expected to operate seasonally, between April and October each year, at a nominal treatment capacity of 520 m^3/h, and is expected to continue long-term into the post-closure period.

The HWSF run-off and hydrometallurgical liquid effluent are collected and treated in the hydrometallurgical water treatment facility (see section 18.6.11). Water is reclaimed from these effluents for recycling to the hydrometallurgical process plant to reduce consumption of water from the local municipal water source.

20.3 Social Setting

The mine site area and the hydrometallurgical plant area are in the Omineca Natural Resource Region of the Prince George Natural Resource District and within the Regional District of Fraser-Fort George.

The healthcare provider for the region is Northern Health Services, one of five health authority regions in British Columbia. The project is located within the Northern Interior Service Delivery Area, where the keystone facility is the University Hospital of Northern British Columbia (UHNBC). Small health stations, administered by British Columbia Emergency Health Services (BCEHS) are located in the communities of Bear Lake and Mackenzie.

20.3.1 Regional District of Fraser-Fort George

The Regional District of Fraser-Fort George encompasses a vast area comprising four municipalities and seven electoral areas. Prince George is the largest urban center and the administrative hub of the regional district. The regional district administers emergency services, waste management, land use planning and zoning, and plays a role in addressing issues such as waste transportation and environmental protection in areas that lack a municipal government.

The Regional District of Fraser-Fort George, Provincial Electoral Area G (Crooked River-Parsnip) is the specific sub-area within the regional district that encompasses the mine area and the hydrometallurgical plant. Electoral Area G has a population of approximately 365 and includes Summit Lake, Bear Lake, McLeod Lake, and areas outside Mackenzie. The labour force in the district is mainly comprised of workers in manufacturing, construction, accommodation and food services, with a substantial percentage working seasonally or part-time.

The hydrometallurgical plant is within the Crooked River-Parsnip Office Community Plan (OCP) that encompasses the rural communities along HWY 97 between Prince George and Mackenzie. The OCP includes mining, minerals processing and related industry within the definition of Heavy Industrial Land Use and service industries incidental to mineral extraction within the Intermediate Industrial Land Use designation. Within Heavy Industrial Land Use designation, the use of land also includes additional residential accommodation for staff if the use location is relatively remote from a residential settlement. The Mineral Resource Management Objective is “to support opportunity for mineral resource industry in a manner reflecting the importance of environmental protections, complementary to other industry in the region”.

The Fraser-Fort George Regional District is served by School District #57. School aged children in rural areas and in the small communities along Highway 97 are bused to schools in larger centres. The students from Bear Lake are transported for an hour each way to

Prince George area schools, and the students from McLeod Lake are bused to Mackenzie, where there are both elementary and secondary schools.

20.3.2 Communities

20.3.2.1 McLeod Lake Indian Band

The Wicheeda property lies in the traditional territory of MLIB who are part of the Tse'Khene group of the Indigenous, Athapascan-speaking, people of northern Canada. Its traditional territory spans approximately 108,000 km^2, with the main community located on McLeod Lake Indian Reserves #1 and #5, which are situated alongside the northern shoreline of McLeod Lake. MLIB's registered 2025 population (as defined under the *Indian Act*) is approximately 618, of whom about 97 members reside on-reserve and 521 members reside off-reserve[2]. In 2000, MLIB adhered to Treaty 8.

Treaty 8 was originally signed in 1899 between the Crown (as represented by the federal and provincial governments) and several First Nations groups. There are eight First Nations in British Columbia that are signatories to Treaty 8, including MLIB, and the remainder of which are situated east of the Rocky Mountains within the Peace River Watershed.

Treaty 8 covers an area of 840,000 km^2, spanning from northeast BC, across northern Alberta, and into Saskatchewan and the Northwest Territories. The southern edges of the mine site infrastructure are between 3.5 km and 4 km from the southern boundary of Treaty 8 and upper boundary of Lheidli T'enneh First Nation territory.

Treaty 8 had the effect of vesting title to the land in the Crown, while guaranteeing the rights of Indigenous signatories to continue to use the land to hunt, trap, fish, and gather. Today, the rights guaranteed by Treaty 8 are constitutionally protected.

MLIB plays a significant role in supporting and stimulating activity in the local and regional economies. They have active agreements with the provincial government related to mineral resource revenue sharing and decision making and have, through their company Sekani Forest Products Ltd. directly purchased forestry rights. MLIB is one of the largest forestry license holders in the Omineca region.

20.3.2.2 City of Prince George

The City of Prince George, BC, known as the "Northern Capital of British Columbia," has a population of 76,700. Prince George could serve to provide a ready, nearby workforce for the project. All goods and services required by the project, including industry services such as laboratory services, mining equipment, drilling contractors, skilled labour, and supply dealerships, are available in Prince George. In 2022, the supply of goods and services to the mining and smelting sector generated $237 million in economic activity for Prince George (MABC 2022).

The major regional hospital is in Prince George. It is the largest acute care facility in the region, and a hub for specialized health services. Critical transfers and emergencies are serviced by the air and ground ambulance services, which include critical care paramedics, two fixed-wing airplanes and one helicopter stationed in Prince George.

---

[2] First Nation Profiles (aadnc-aandc.gc.ca) Accessed March 28, 2025.

Public schools for children are numerous in Prince George with programs offered at specialty schools that include French Immersion, Francophone, Montessori, Indigenous Choice and Art Programs. Secondary school choices include a career technical centre where students can advance a career in a trade while completing high school graduation credits.

One university and three colleges provide diverse education opportunities from university degrees to college diplomas and certificates and trades training.

Daily commercial air service is available between Prince George and Vancouver, BC, with multiple flights available each week between Prince George and several other destinations including Victoria, BC, Calgary, AB, and Edmonton, AB. Helicopter charter services are also available year-round in Prince George.

20.3.2.3 McLeod Lake Community Profile

McLeod Lake is situated 135 kilometres north of Prince George and 65 km north of Bear Lake along Highway 97. The primary development is concentrated at the lake's northern end, encompassing the community of McLeod Lake and the McLeod Lake Indian Reserve. This area features a mixed-use environment with residential, commercial, and industrial sectors.

Community amenities include a general store, church, resort, the McLeod Lake Indian Band Office, and the Fort McLeod Historic Site. Services available to residents include natural gas in select areas and single-phase hydro power along the John Hart Highway.

20.3.2.4 Bear Lake Community Profile

Bear Lake, situated 50km north of Prince George's along Highway 97, is an unincorporated community with a population of 150 people. It is the closest community to the mine site area and the hydrometallurgical plant area. The community features a mix of residential, commercial, and industrial sectors, with a strong emphasis on the forestry industry.

The community's development plan encourages further expansion, identifying designated areas for future residential, industrial, and commercial growth. The Crooked River-Parsnip Area G, OCP has delineated areas for Heavy and Intermediate Industrial development.

Bear Lake is governed by a Community Commission comprising four locally elected Commissioners and the Director for Electoral Area G. This Commission oversees essential services, such as water system management, street lighting, recreational facilities, ambulance station operations and fire protection services.

Furthermore, the Commission acts as an advisory body to the Regional Board on land use planning and community matters.

20.3.2.5 Summit Lake Community Profile

- Summit Lake is located approximately 30 kms north of Prince George. It is an unincorporated community with a small year-round population and a larger seasonal population. The majority of the development is situated between HWY 97 and the eastern shore of Summit Lake. The islands are well developed with seasonal cottages.
- Summit Lake facilities include a community hall, public boat launches, a community campground, and park land. There is a ski hill with a tube park. Services available to the Summit Lake area include natural gas available in some areas and three-phase hydro power through the Summit Lake community and along HWY 97.

20.3.3 Recreational Users

Wicheeda Lake and surrounding areas are currently covered under Recreational Reserve REC6837. As of May 20, 2020, the BC Ministry of Forests Land and Natural Resource (FLNR) indicated that, given its current priorities and capacity, there is no intent to establish a recreation site at Wicheeda Lake in the near future.

At present there are no restrictions on mineral exploration activities within REC6837. However, FLNR has requested that Defense Metals take all possible steps to minimize the impacts of exploration to the recreational ecological values associated with Wicheeda Lake.

Wicheeda Lake is a managed rainbow trout sport fishery that is closed to winter angling and free from ice fishing pressure. A proposal to change the daily quota of rainbow trout from 3 to 1 with no fish over 40 cm is currently in process with the BC government.

There are several registered traplines in the vicinity of the project; two of which overlap the mine site area entirely, and the wildlife management unit (7-16) that overlaps the project is subject to provincial administration of limited-entry hunting zones. Sport hunters utilize the area for limited entry hunting of deer, moose, elk, bear and mountain goat as well as avian species. There are limitations on the use of snowmobiles for hunting specific species.

The upper boundary of a commercial recreation licence for heli-skiing terrain is within 4 km of the mine project area to the south.

20.3.4 Other Interest Holders

There is an outstanding silviculture obligation to the east of the FTSF, another to the north of the process plant and another along the Chuchinka FSR 3 km south of the bridge

An active guide outfitter active license overlaps the entirety of the mine site area and 70 km to both the east and west.

The mine site area and the hydrometallurgical plant area are both within the Prince George Timber Sales business area which is part of BC Timber Sales (BCTS) areas of management of the public timber supply.

There are multiple old-growth deferral areas within the mine site area; one which directly transect the WSF and another on the western perimeter of the open pit. Other old growth deferral areas are dispersed within the general mines site area at a similar consistency as the other areas within the region. Logging deferrals are viewed as temporary measures utilized to allow time for planning and decision-making about the long-term management of old-growth forests. Outcomes post-deferral are decided collaboratively with Indigenous rightsholders, consider both ecological and economical values and can include: permanent protections, modified management, inclusion in land use plans or a return to standard forest management.

The Prince George Timber Supply Area has a legal order of Landscape Biodiversity objectives that partially overlaps the mine site; FTSF, CWP, processing plant and parts of the WSF and open pit. This legal objective was developed within the BC Strategic Land and Resource Planning processes and establishes non-spatial old-growth forest targets that must be maintained by commercial forest harvesters within specific natural disturbance units. If the required tree age distribution cannot be reasonably obtained, a recruitment strategy must be developed that is consistent with meeting the objectives for the unit in the shortest time as practicable. This order applies to commercial forestry in the area but will also apply to the mine site area when an occupant license to cut is required for pre-strip.

A hunting camp on a commercial recreation license, occupation tenure is approximately 3.7 km from the westerly edge of the FTSF.

Defense Metals' mineral tenures also overlap with a First Nations Crown Grant owned by a member of MLIB and a parcel of unsurveyed Crown land held by MLIB, both of which were transferred following a land settlement claim between MLIB and the Crown.

The project infrastructure footprint does not directly impact these land parcels but ongoing analysis of the impacts on these interest holders will be undertaken as the project progresses, and appropriate actions to mitigate or compensate will be taken, if required.

20.3.5 Agreements and Negotiations

20.3.5.1 MLIB and Defense Metals – Co-Design Agreement and Equity Partnership

In January 2024, MLIB and Defense Metals entered into a first-of-its-kind Co-Design Agreement. The effect of the agreement is to create a process whereby both parties will co-develop the inputs and outputs of the environmental assessment for the project, including baseline studies, the initial project description, and the assessment of the project's potential impacts on the environment and Indigenous rights.

The agreement solidifies MLIB's and Defense Metals' joint commitment to the successful advancement of the Project and empowers MLIB to play an integral part in the feasibility design and decision-making processes for the technical, social, engineering, and environmental aspects of the Project. Specifically, MLIB will have the opportunity to provide input into project design, such as in planning the layout of the mine's infrastructure and evaluating available technologies and associated impacts. The integration of MLIB's technical input, Indigenous knowledge, and traditional land and resource use information into project plans is expected to support MLIB's interests and priorities and, in turn, contribute to a more efficient and robust environmental assessment process.

In January 2024, MLIB and Defense Metals also announced a strategic Equity Partnership wherein MLIB has taken a 1% equity position in the Project, demonstrating the parties' joint and long-term commitment to the Project. The advances that Defense Metals has made in securing MLIB as a strategic partner has positioned the Project as a key initiative to support federal and provincial priorities for critical minerals development and reconciliation with Indigenous peoples.

Prior to the Co-Design Agreement and Equity Partnership, Defense Metals committed to early engagement with MLIB, which commenced in September 2020. The parties entered a Mineral Exploration Agreement (MEA) in August 2022. The MEA emphasized the parties' commitment to maintain open and transparent communication of information regarding Project exploration plans and schedules, as well as potential impacts on MLIB's rights and interests.

20.3.5.2 Treaty 8 Agreement

The Treaty 8 is an agreement between the Government of Canada and Cree, Beaver, Chipewyan and other Indigenous Peoples (recognized today as McLeod Lake Indian Band (signatory to adhesion), West Moberly First Nations, Doig River First Nation, Blueberry River First Nations, Fort Nelson First Nation, Halfway River First Nation, Prophet River First Nation, and Saulteau First Nations] that states that Indigenous signatories have rights enshrined within the oral and written terms of Treaty No. 8, as well as rights recognized and affirmed by section 35(1) of the *Constitution Act, 1982*.

20.4 Environmental Assessment and Permitting

20.4.1 Site Investigation Permitting

The Property exploration work to date has been conducted under a Multi-Year Permit, issued by the Ministry of Mining and Critical Minerals (MCM) to Spectrum on September 18, 2008, and amended on February 26, 2019 and November 16, 2023 (Permit MX-13-168). The permit was valid until December 31, 2024 and has expired.

Reclamation security funds totalling $24,300 were posted by Defense Metals to be held under Permit MX-13-168 by the British Columbia Minister of Finance. While this permit has lapsed, the reclamation funds are held until all the reclamation conditions of the permit are met in a manner satisfactory to the Chief Inspector of Mines.

A new Multi-Year, Area-Based permit for continued site investigations was submitted in May, 2024 and has been processed by MCM through government agency review and Indigenous consultation. A request for reclamation security of $157,000 CDN has been received. MCM will issue a decision on the permit upon submittal of a letter of credit for the reclamation security. The permit authorization will be for five years and will include drill sites, test pits sites, staging areas, new exploration trails, fuel storage, temporary bridges, water supply and camps. An occupant license to cut issued by the Ministry of Forests, Lands, Natural Resources Operations and Rural Development (FLNRORD) will be required for select tree removal to facilitate the site investigations.

20.4.2 Environmental Assessment

Environmental Assessment (EA) and permitting frameworks for metal mining in Canada are well established. Federal and provincial assessment processes have similar procedural frameworks, but their scopes are defined by separate jurisdictions. The federal process examines impacts within federal jurisdiction, focusing on areas like fisheries, migratory birds, federal lands, and effects on Indigenous rights. The provincial EA, conversely, covers a broader range of impacts within provincial boundaries, including environmental, social, economic, cultural, and health effects.

20.4.2.1 *Federal Impact Assessment Applicability*

The mine is planned to have a production capacity of 1,800,000 tonnes of ore per year; therefore, a federal Impact Assessment (IA) will be required. If mine production capacity were to change to less than 912,500 tonnes/year, an IA would not be automatically triggered, although the project could still be designated by the Minister for review by the IAAC pursuant to section 9(1) of the IAA. Section 9(1) grants the Minister of Environment and Climate Change the authority to designate a project for an IA based on the potential for adverse effects within federal jurisdictions, including fish, aquatic species, migratory birds, and Indigenous peoples.

20.4.2.2 *Provincial Environmental Assessment Applicability*

The British Columbia *Environmental Assessment Act*, SBC 2018, chapter 51 (EAA), and the *Reviewable Projects Regulation*, BC Reg 243/2019, states that a new mine facility that will have a production capacity of greater than or equal to 75,000 tonnes/year of mill feed requires a provincial EA.

20.4.2.3 *Environmental Assessment Process*

As the production capacity of the project exceeds the non-assessment threshold limit for both the provincial EAA and federal Impact Assessment Act (IAA) requirements and both provincial and federal assessments are required, the project will proceed through a concurrent EA process. This could be either through a substituted or coordinated approach between the federal Impact Assessment Agency of Canada (IAAC) and BC’s Environmental Assessment Office (EAO). An agreement, known as the Impact Assessment Cooperation Agreement, exists between the two governments that provides the framework for cooperation on the EA to avoid duplication while retaining their respective powers to approve a given project. A determination of the applicable process will be made once both the IAAC and EAO have been notified of the project with the submission of an Initial Project Description (IPD).

Although timelines are defined in the EA, the project may deviate from the outlined timelines due to the complexity of the project, the requirements for robust engagement, traditional knowledge integration, the need to address stakeholder concerns or changes to project design. The Indigenous involvement in the project will be integrated into submissions as outlined in the Co-design Agreement, which increases confidence in regulatory success and has the potential to minimize timelines for the EA process.

As part of a 2018 revision to the EAA and IAA processes, a decision point was added to determine whether a project should proceed to an environmental assessment as a gatekeeping step. The decision is made after an IPD has progressed to a Detailed Project Description (DPD), and a summary of stakeholder and rightsholder issues has been compiled. Options for the readiness decision include requiring a revised DPD, proceeding to agency-led environmental assessment, proceeding to assessment for review by panel, recommending the minister exempt the project from environmental assessment, or recommending the minister terminate the project from the process.

20.4.2.4 Assessment - Ministerial Decisions

The EA process will culminate in a provincially issued Environmental Assessment Certificate (EAC) and a federal Decision Statement prior to entering a licensing/permitting phase to authorize the various activities required to develop, operate and close the mine. An EAC is received after a positive project decision by the Ministry of Environment and Parks (EP) and the MCM. It is legally binding and includes conditions that must be followed to mitigate potential impacts. The federal process concludes with the Decision Statement, which outlines the final decision, the public interest determination, and the legally binding conditions. These conditions, along with the EAC, set the requirements for subsequent permits.

20.4.2.5 Concurrent Approval Regulation

While full permitting cannot occur during the environmental assessment, efforts will be made to coordinate and align information requirements and reviews to streamline the post-EA permitting phase. An application for concurrent review will be submitted to the EAO under the Concurrent Approval Regulation. This allows for concurrent review of other provincial approvals (e.g., licences and permits) while the project is still undergoing an environmental assessment. This will allow for the timely issuance of other required approvals if an environmental assessment certificate is granted. Where EAO allows for the concurrent review of permit applications, authorizations can be issued within 60 days of the issuance of the environmental assessment certificate.

20.4.2.6 Critical Minerals Strategy

In January 2024, the Province of British Columbia announced its Critical Minerals Strategy (CMS), designed to enhance the contribution of mineral exploration and mining to the province's economy by developing critical minerals. The project is a provincially recognized critical minerals project and will benefit from the strategy. The CMS has three main goals: (1) to expand partnerships with First Nations, enhance shared decision-making, and promote reconciliation; (2) to increase business certainty and attract investment; and (3) to establish funding partnerships to advance critical mineral projects.

A key initiative of the CMS is the implementation of the Critical Minerals Office, which offers a 'concierge-like' service with dedicated support to navigate regulatory processes, advance funding opportunities, and expedite solutions to issues.

20.4.3 Permitting Requirements

A number of permits, licenses, and authorizations are expected to be required from both federal and provincial regulators to advance the project to construction and operations.

20.4.3.1 Provincial Requirements

In addition to the requirement to obtain an EAC the project will require a mining lease to convert mineral claims to a mining lease pursuant to section 42 of the *Mineral Tenure Act*.

All mines in BC must hold a permit, issued by the Chief Inspector of Mines, under the *Mines Act*, and in accordance with Part 10 of the Health, Safety and Reclamation Code (HSRC) for Mines in BC. Permitting under the *Mines Act* is applicable for all onsite mining activities and considers detailed designs for all project components and phases of mine life, including construction, operation, reclamation, and closure. *Mines Act* permitting must also consider proposed project activities and mitigations such as the management of water quality, waste and metal leaching/acid rock drainage, and geotechnical design. A *Mines Act* permit does not expire and can only be closed by the Chief Inspector once the reclamation obligations of the permit have been fulfilled.

Discharges of waste from mining activities to the environment require authorization under the EMA. A permit to authorize the ongoing discharge of waste for a mining project is required for:

- Effluent discharges (e.g., FTSF supernatant, mine-influenced run-off, water treatment discharge and sewage)
- Air emissions (e.g., refuse incinerator emissions, emissions from milling and processing, greenhouse gases etc.)
- Solid wastes (e.g., mill tailings, water treatment plant sludge, refuse, etc.).

An *EMA* permit will set the terms and conditions for the waste discharge, with the goal of ensuring the protection of human health and the environment. The terms and conditions of the permit may include limiting the quantity and quality of waste contaminants that are allowed to be discharged and monitoring the discharge and the receiving environment.

Additional permits, licenses, authorizations, and approvals that may be required from various provincial regulatory bodies are noted in Table 20-7.

Table 20-7: Provincial Permit Requirements for the Construction Phase and Project Operations

Legislation	Authorization	Phase Permit is Required	
		Construction Phase	Project Operations
Drinking Water Protection Act	Drinking water permits	X	X
Environmental Management Act	Air Emissions Discharge Permit	X	X
	Solid Waste Discharge Permit	X	X
	Hazardous Waste Registration	X	X
	Fuel Storage Registration	X	X
	Sewage Registration	X	X
Forest Act and Forest & Range Practices Act	Occupant License to Cut	X	
	Forest Service Roads Use Permit	X	X
	Special Use Permit	X	X
Heritage Conservation Act	Heritage Inspection Permit	X	
	Site Alteration Permit	X	X
Land Act	Various land tenures	X	
Mineral Tenure Act	Mineral Claims	X	
	Mineral Lease	X	
Mines Act	Notice of Work for exploration	X	
	Explosives Storage and Use Permit	X	X

Legislation	Authorization	Phase Permit is Required	
		Construction Phase	Project Operations
Public Health Act	Industrial Camp Regulation	X	X
	Sewerage System Regulation	X	X
Transportation of Dangerous Goods Act	Training certificates or equivalency certificates	X	X
Water Sustainability Act	Stream Change Approval	X	X
	Water Licenses	X	X
Wildlife Act	Authorization permits for general works	X	X

Source: SRK, 2025

20.4.3.2 CleanBC Roadmap 2030

The CleanBC Roadmap is the overarching climate action plan developed by the BC government to reduce gas emissions. Revisions are periodically made to the plan to reflect policy changes and evolving circumstances such as the March 2025 removal of a scheduled increase to the consumer carbon tax. It is expected that both the mine site and hydrometallurgical plan will interact with these requirements.

20.4.3.2.1 Greenhouse Gas Industrial Reporting and Control Act

The net-zero policy objectives relevant to mineral extraction and processing are defined in amendments and additions to the Greenhouse Gas Reporting and Control Act (GGIRCA). GGIRCA sets the requirement that the project must demonstrate, through a credible plan, how it will align with the Net-Zero Industry Policy targets and sector-specific goals, contribute to the province's emissions reduction goal and align with the emissions intensity of world-leading facilities of the same class. This will be undertaken and reviewed while the project is undergoing EA. Early consideration of designing to reduce emissions during project planning will ensure approvals and competitiveness.

In accordance with the February 2024 amendment to GGIRCA, the project will also need to meet the requirements of the British Columbia Output Based Pricing System. This necessitates the establishment and adherence to operation-specific emission limits, with potential carbon pricing applied to emissions exceeding those limits. The project will also be eligible to utilize BC carbon offset units and earned credits to meet compliance obligations.

20.4.3.3 British Columbia Utilities Commission

BC Utilities Commission (BCUC) review and approval are required to connect directly to the main provincial power grid. There are specific pre-application consultations, detailed application processes, and BCUC reviews that culminate in a decision document issued by a review panel, which approves, denies, or approves the project with specific conditions. The BCUC approvals process focuses on public utility services, economic feasibility, and the technical aspects of power delivery whereas the EA addresses environmental and broader socio-economic impact assessments. For major mine projects, the EA typically occurs first, and the EAO and the BCUC may coordinate consultation processes to ensure that the BCUC can consider the findings of the Environmental Assessment during their review, thereby avoiding the re-examination of the same issues.

20.4.4 *Federal Permits, Licenses, Authorizations and Approvals*

The project will also be reviewed for potential impacts to fish and fish habitat, in accordance with the federal Fisheries Act. If the project is likely to cause the death of fish or the harmful alteration, disruption, or destruction of fish habitat, a section 34 or 35 authorization from the Department of Fisheries and Oceans Canada will be required.

Refined fisheries investigations will be utilized to confirm that the FTSF final footprint area does not overprint a waterbody frequented by fish. It is not expected, given the terrain and topography of the selected FTSF location, that fish will frequent the area and therefore is not likely that an authorization in the form of a Schedule 2 amendment to the Metal and Diamond Mine Effluent Regulations under Section 36 of the Fisheries Act will be required.

Additional permits, licenses, authorizations, and approvals that may be required from various federal authorities for project operations are noted in Table 20-8.

Table 20-8: Federal Permit Requirements for Project Operations

Legislation	Authorization
Environmental Protection Act	Environmental Emergency Regulation
Explosives Act	Manufacturing License
Migratory Birds Convention Act	Damage or Danger Permit
Canadian Navigable Waters Act	Application for Approval
Nuclear Safety and Control Act	License for use of nuclear substances and radiation devices
Radiocommunication Act	Authorization for the use of radio equipment
Species at Risk Act	Species at Risk Permit

Source: SRK, 2025

20.5 Mine Closure

20.5.1 *Reclamation and Closure*

Closure of the project will primarily be regulated by MCM under the BC *Mines Act* and HSRC. Closure measures and strategy will also be developed with consideration for the EMA, which regulates the discharge of wastes.

A preliminary conceptual reclamation and closure plan was developed to support the Study. The detail of this plan will increase as the project advances and stakeholder and rightsholders' interests are incorporated. A conceptual reclamation and closure plan, a detailed five-year mine reclamation plan, and associated reclamation cost estimates will be developed to support the Mines Act and Environmental Management Act permit applications.

The closure plan must be updated every five years after that, in support of permit amendments, and twelve months before the planned date of mine closure.

20.5.2 *Closure Principles and Objectives*

The HSRC indicates that the main objective of reclamation and closure is to return areas disturbed by mining activities to a physically and geochemically stable state with end land use capability the same as the average pre-mining land capability, unless other end land uses have been approved.

Specific closure objectives will be developed that aim to return disturbed areas to conditions consistent with an agreed upon end land use. The closure and reclamation plan, closure objectives and end land use will be developed in consultation with the project team, Indigenous rightsholders, interested parties and appropriate regulatory agencies.

The Mine Site is a largely undisturbed natural area, and it is assumed for this study that the objective of reclamation will be to restore the area to a similar natural state and end land use. The physical location for the hydrometallurgical plant facility, has not been identified, so the existing conditions of the site are not known. For the purpose of this study it is assumed that the facility is an industrial site with some prior development history. As such the end land use is assumed to be as an industrial site, therefore, the railway spur and substation are assumed to remain in place. Disturbed areas will be regraded and vegetated to prevent erosion and growth of invasive species.

Principles for reclamation and closure planning for the project include:

- Design for closure.
- Incorporate climate change considerations.
- Reduce impacted water.
- Include source (contaminant) control.
- Undertake progressive reclamation.
- Minimize long-term activities, where practicable.
- Integrate disturbed lands into the surrounding landscape and restore the natural appearance of the area, to the extent possible.
- Establish a self-sustaining vegetative cover consistent with end land uses.
- Plan for long-term monitoring and maintenance.

Closure planning will be carried out concurrently with various stages of project development and design to integrate closure objectives into the design, construction and operation of mine infrastructure and activities.

The following closure considerations are incorporated into the project design, so the mine infrastructure is designed for expected closure and reclamation practices.

- Bottom-up waste rock storage facility construction, to allow for progressive reclamation.
- Filtered tailings storage facility, built in- stages, which will allow for progressive reclamation, and the filtered tailings simplify cover placement and long-term water management.
- Topsoil salvage.
- Diversion channels designed for operations and closure.

The closure prescriptions for the infrastructure are described in Section 18, while over arching closure activities are listed below:

- Construction of a protective berm around the open pit.
- Removal and proper disposal of pipelines, structures and equipment not required beyond the end of mine life.
- Reclamation of disturbed areas, including regrading, cover and/or topsoil placement, as needed, and revegetation.
- Placement of covers on the waste rock and tailings.
- Long-term monitoring and treatment of seepage water from the waste rock and tailings facilities are assumed.

20.5.3 Closure Monitoring

The purpose of the active closure and post-closure monitoring and maintenance program is to evaluate that the site is stable, safe and meeting the closure performance objectives. The closure monitoring program builds on the operational monitoring program for the project. The active closure period includes regular monitoring, maintenance, and reporting. Post-closure monitoring will continue until reclamation objectives are met; when monitoring indicates that closure performance objectives have been met, monitoring and maintenance frequency will be reduced. Continuous long-term monitoring is expected at the Mine Site due to the planned perpetual water treatment, it is expected that monitoring frequencies at the hydrometallurgical plant will be reduced once monitoring indicates performance objectives have been met.

Active closure and post-closure monitoring are expected to include:

- Physical and chemical stability of engineered facilities
- Surface and groundwater quantity and quality
- Reclamation and wildlife use
- Aquatic resources (benthic invertebrates and fish).

20.5.4 Mine Closure Requirements and Financial Assurance

The reclamation and closure cost estimate for the study is based on the preliminary conceptual reclamation and closure plan developed to support the project design. The reclamation and closure costs include landforming, cover placement and revegetation, engineering and administration, equipment and structure removal, monitoring and maintenance, and water treatment costs. The reclamation estimate includes a credit for salvage of building steel and assumes a 15% contingency.

For the Mine Site the water treatment costs include operational costs in perpetuity and capital costs for water treatment plant replacement every 50 years. For the hydrometallurgical plant, water treatment operational costs are included for two years after placement of the cover over the HWSF, after which it is assumed that water treatment will no longer be necessary.

The closure cost estimates assume that active closure will occur over three years directly following the end of mining. Post-closure monitoring, maintenance and water treatment costs were then estimated for 100 years post closure.

The reclamation security required by the Mines Act, will be developed and calculated with increasing refinements as the project progresses through engineering and assessment processes. It will be based on the net present value of the peak estimated liability during the five years addressed in an approved closure plan. Liability estimates are calculated based on 100 years and discounted according to the categories of liability held. The required content of a liability cost estimate is comprehensive. It includes reclamation costs for land forming and revegetation, engineering and administration, equipment and structure removal, water treatment capital and operating costs, maintenance and monitoring, labour rates based on third-party contractors and a default contingency. Special approvals are required to allow for the use of salvage value or the value of any other assets or revenue stream in offsetting the reclamation liability amounts.

There are mechanisms in place to secure the reclamation liability against the remaining mineral reserves, dependent on the remaining life of the mine up to when only 5 years of economically viable reserves remain, at which time 100% of reclamation liability is required for the remaining life of the mine.

21. Capital and Operating Costs

21.1 Capital Costs

All costs are expressed in U.S. dollars. An exchange rate of 1.40 CAD = 1.00 USD was considered for this cost estimate. Costing was conducted on a Q1 2025 basis.

Capital cost estimates for the project includes the mine, both processing plants, waste management facilities, and closure costs. The overall initial project capital cost is $1,441M, with sustaining capital, closure costs, and post closure costs bringing the total to $2,007M. The capital costs are summarized in Table 1-3.

Table 21-1: Summary of Capital Costs

Area	**Capital costs (US$M)**				
	Initial	**Sustaining**	**Closure**	**Post closure**	**Total**
Mining	96.88	55.6	-	-	152.48
Concentrator	450.86	-	-	-	450.86
Hydromet	614.49	-	-	-	614.49
Mine Tailings	19.84	45.67	-	-	65.51
Hydromet Waste	10.66	14.04	-	-	24.70
Contact Water Pond	11.76	-	-	-	11.76
Mine Site Water Management	1.59	-	-	-	1.59
Mine Site Water Treatment	10.03	-	-	-	10.03
Hydrometallurgical Water Treatment	6.61	-	-	-	6.61
Closure	-	-	57.35	325.12	382.47
Contingency	217.85	20.62	7.17	40.64	280.96
Total	**1,441**	**136**	**65**	**366**	**2,007**

Source: Hatch, 2025

Note: Cost estimates do not consider cost escalation resulting from the imposition of new tariffs, counter-tariffs, import and/or export duties, or other similar charges applicable to raw, semi-finished or finished materials and/or other products.

The initial capital costs are spread over a three-year construction period in the economic model. The earliest that construction is expected to start is Q1 2030 due to environmental permitting timelines. This would enable the concentrator and hydrometallurgical process plants to start up Q1 2033.

21.1.1 Mining Capital Cost Estimate

21.1.1.1 Pre-Production Activities

Pre-production activities are key components of the mining capital cost expenditure (Capex) estimate and consist of the following:

- Logging and grubbing of pit, WSF, CWP, and FTSF areas
- Topsoil salvage for reclamation purposes of pit, WSF, CWP, and FTSF areas
- Capitalized mining expenses prior to the commencement of processing operations

Pre-production activities for logging, grubbing, and topsoil salvage are estimated at $8.4M and are expended in Years -3 to Year -1. Further pre-production activities from Year 1 onward are estimated at $2.5M.

Capitalized mining expenses captured from Year -3 to Year -1 total $57.4M.

Contingency on pre-production activities is assigned at 25%, except for capitalized mining expense, which has no contingency allocated.

21.1.1.2 Mine Equipment

The mine equipment capital cost is estimated for both primary and ancillary equipment. The primary equipment includes items such as drills, loading equipment, haul trucks, track dozers, graders, and water trucks. The ancillary equipment includes light vehicles, service vehicles, and lighting equipment.

The primary equipment requirement estimate is based on the mine schedule quantities, determinations of productivities and therefore equipment requirements. Costs are derived from vendor quotations and recent SRK studies.

The ancillary equipment capital cost estimate is based on benchmark costs for individual equipment types.

The Capex estimate for mine equipment is summarized in Table 21-2. New equipment costing has been assumed for the full mine production period. Initial Capex is calculated for the equipment required up to the end of the pre-production period. Freight and commissioning costs are included in the equipment pricing. Equipment purchases are timed for the year in which the equipment is required.

Contingency on equipment costs is assigned at 10%.

Table 21-2: Mining Equipment Capital Cost

Parameter	Initial (US$M)	Sustaining (US$M)	Total (US$M)
New/Additional Equipment Capital	30.4	7.1	37.6
Equipment Replace/Rebuild	0.0	44.7	44.7
Capital Costs	30.4	51.9	82.3

Source: SRK, 2025

21.1.1.3 Miscellaneous Mining Capex

Additional mining Capex is included for dewatering equipment (pumps, valves, pipelines, etc.), pre-production contractor site preparation, and survey equipment. These total $0.6M for initial Capex (to end of pre-production), $1.3M for sustaining capital, and $1.9M LOM total.

21.1.1.4 Infrastructure

Mine operations infrastructure for the Wicheeda project includes:

- Mine maintenance shop
- Mine office/dry
- Refueling station for mine equipment

The costing for the above is included in the overall site infrastructure estimate (Section 21.1.2)

Explosives storage and garage facilities will also be required; however, since explosive loading is a contracted service, the explosives provider will be responsible for the construction of these facilities.

21.1.1.5 Summary of Mining Capital Costs

The summary of mining capital costs is provided in Table 21-3.

Table 21-3: Summary of Mining Capital Costs

Item	Initial (US$M)	Sustaining (US$M)	Total Cost (US$M)
Pre-Production	65.8	2.5	68.3
Mine Equipment	30.4	51.9	82.3
Miscellaneous	0.6	1.3	1.9
Subtotal	96.9	55.6	152.5
Contingency	5.2	5.9	11.1
Total Mining Capital	**102.1**	**61.5**	**163.6**

Source: SRK, 2025

21.1.2 Concentrator Capital Cost Estimate

The concentrator plant capital cost is estimated to be M$537, and the cost details are summarized in Table 21-4. The estimate considers all process and non-process equipment and infrastructure within the plant area (see Sections 17 and 18), the battery limits considered for this estimate were:

1. Input ROM ore delivery to primary crusher.
2. Output of concentrator tailings filter cake at the discharge of the tailings filtration unit.
3. Output of concentrate at the concentrator plant gate.

The estimate was prepared by combining unit rates for equipment, materials, labour, and subcontracts with unit quantities from the following engineering deliverables: mechanical equipment list, electrical equipment list, building list, material takeoffs for earthworks, concrete, and steel, and the piping takeoff (for the tailings line). The unit costs were based on a combination of vendor budget quotes, reference costs from similar projects, factored costs and allowances. Process plant piping was factored from the mechanical equipment cost. Indirect costs were factored from the direct cost or itemized from reference costs as required. The cost includes a 20% contingency and 6% owner’s cost.

Table 21-4: Concentrator Capital Cost by Trade

Trade	Description	Total ($USD)
Direct Cost		
A	Site Development	6,599,670
C	Concrete	37,401,577
D	Roadworks, Drainage & Paving	14,407,369
E	Earthworks	38,777,922
F	Architectural	43,079,806
J	Control & Instrumentation	15,601,638
L	Electrical Equipment	40,835,076
M	Mechanical Equipment	73,694,947
N	Mech Platework	876,629
O	Mobile Equipment	7,368,666
P	Pipework & Fittings	14,698,149
S	Structural Steel	35,073,924
W	Wire & Cable	9,153,942
Subtotal Direct Cost		337,569,315
Indirect Cost		
Y	Indirects	93,040,449
Subtotal Indirect Cost		93,040,449
Total Direct + Indirect Cost		430,609,763
Provisions		
Z	Contingency	86,121,953
V	Owners Costs	20,254,159
Total Cost		**536,985,875**

Source: Hatch , 2025

21.1.3 *Hydrometallurgical Plant Capital Cost Estimate*

The Hydrometallurgical plant capital cost is estimated to be M$732, and the cost details are summarized in Table 21-4. The estimate considers all process and non-process equipment and infrastructure within the plant fence except for the effluent treatment plant (see Section 21.1.6), the battery limits considered for this estimate were:

1. Input concentrate at the hydrometallurgical plant gate
2. Sulfuric acid tie-in at the fence line (the capital cost for the sulfuric acid plant was excluded from the estimate since the payback cost is included in the acid purchase price)
3. Output of rare earth precipitate at the plant gate
4. Output of the primary neutralization residue at the plant gate
5. Output of hydrometallurgical waste product to the hydrometallurgical waste storage facility.
6. Output of hydrometallurgical effluent at the tie-in to the hydrometallurgical effluent treatment plant

The estimate was prepared using the same methodology described for the concentrator plant (Section 21.1.2).

Table 21-5: Hydrometallurgical Plant Capital Cost by Trade

Trade	Description	Total ($USD)
Direct Cost		
A	Site Development	1,786,484
C	Concrete	48,098,981
D	Roadworks, Drainage & Paving	4,359,026
E	Earthworks	24,912,102
F	Architectural	63,904,534
J	Control & Instrumentation	19,349,243
L	Electrical Equipment	20,845,807
M	Mechanical Equipment	135,278,141
N	Mech Platework	3,909,196
O	Mobile Equipment	3,494,286
P	Pipework & Fittings	34,180,858
S	Structural Steel	58,176,424
W	Wire & Cable	14,647,656
Subtotal Direct Cost		432,942,738
Indirect Cost		
Y	Indirects	155,574,918
Subtotal Indirect Cost		155,574,918
Total Direct + Indirect Cost		588,517,656
Provisions		
Z	Contingency	117,703,531
V	Owners Costs	25,976,564
Total Cost		**732,197,751**

Source: Hatch , 2025

21.1.4 *Tailings*

Initial capital costs for the FTSF and HWSF are approximately M$20.0 and M$10.5, respectively. The estimate was developed using design quantities and assumed unit rates from SRK's experience with similar projects as well as cost database resources (e.g., RSMeans construction cost data). Costs are summarized in Table 21-6.

For the FTSF, the development of Stage 1 was optimized to defer initial capital, mainly to avoid the area believed to have the highest foundation excavation risk and to minimize the initial footprint to reduce upfront liner costs. Other significant cost items include the purchase of a mobile equipment fleet to haul and place tailings during operation.

For the HWSF, it is assumed one of the four cells will be constructed prior to operation. Cost-driving items include foundation preparation (which includes a liner and internal drainage layer along the base) and purchase of a mobile equipment fleet for operation.

Sustaining capital costs for the FTSF are for foundation preparation, liner installation and surface water diversion channels as the facility's footprint expands. Haul trucks are also replaced halfway through the LOM. For the HWSF, sustaining costs are for construction of the three remaining cells.

Table 21-6: Tailings Storage Facility Capital Costs

Facility	Item	Initial capital cost (US$M)	Sustaining Capital Cost (US$M)
FTSF	Toe Embankment	1.6	45.7
	Foundation Preparation	2.4	
	Foundation Liner	5.8	
	Surface Water Management (inc. diversion channels & water management pond)	1.9	
	External Roads	1.0	
	Mobile equipment fleet	6.5	
	Engineering, General Management & Other Support	0.6	
HWSF	Foundation Preparation (inc. Liner)	3.4	14.0
	Starter Embankment	0.8	
	Roads, Channels, Ponds	0.4	
	Mobile equipment fleet	5.7	
	Engineering, General Management & Other Support	0.3	
Subtotal		30.5	59.7
Contingency % (weighted average)		19%	25%
Contingency amount		5.8	14.7
Initial and Sustaining Capital Totals		36.3	74.4
Total		**110.7**	

Source: SRK, 2025

21.1.5 Mine Site Water Management and Treatment

Water management costs include building the CWP, costs for contact and non-contact surface water collection, diversion channels around the WSF, and diversion channels around the process plant. The CWP costs include the costs to construct the two embankments using a combination of local till borrow and run-of-mine rock, and costs to strip the impoundment area and line it. Costs for diversions around the FTSF are included as part of FTSF capital costs.

Table 21-7: Mine Site Water Management Capital Costs

Category	Item	Capital Cost (US$M)
Contact water pond	West Embankment	0.8
	East Embankment	4.7
	Impoundment	6.0
	Engineering, General Management & Other Support	0.3
Surface water controls	WSF diversion channel	0.5
	WSF collection channel	0.9
	Process Plant/Shop channels	0.2
Subtotal		13.4
Contingency amount		3.0
Total		**16.4**

Source: SRK, 2025

The water treatment plant estimate includes all water treatment required for the ferric co-precipitation process, including all water treatment equipment, reagent and flocculant handling, influent piping and building costs. The estimate was developed by scaling the design throughput of the mine water treatment plant, based on in-house estimates and benchmarks from SRK's internal database.

Table 21-8: Mine Site Water Treatment Capital Costs

Category	Item	Capital cost (US$M)
Mine Site Water Treatment Direct	Equipment costs	2.4
	Installation costs	2.4
	Shipping/Transportation	0.2
	Site Preparation	0.3
	Civil Foundations	0.3
	Piping	0.5
	Electrical	0.5
	Controls	0.3
	Plant Services and Utilities	0.3
Subtotal		7.2
Mine Site Water Treatment Indirect	EPCM	1.5
	Temporary Construction Services	0.3
	Insurance	0.1
	Spares	0.4
	Cold Commissioning	0.2
	Cost for Settling Pond, Initial CAPEX	0.3
Subtotal		2.8
Total		**10.0**

Source: SRK, 2025

21.1.6 Hydrometallurigical Facility Water Treatment

The water treatment plant for the hydrometallurgical facility cost is presented in Table 21-9. The estimate includes all water treatment equipment, including reagent and flocculant handling, influent piping, and building costs. The estimate was developed by scaling the design throughput of the hydrometallurgical water treatment plant, based on in-house estimates and benchmarks from SRK’s internal database.

Table 21-9: Hydrometallurgical Water Treatment Plant Cost

Direct/Indirect	Item	Capital Cost (US$M)
Direct	Equipment costs	2.14
	Installation costs	1.07
	Shipping/Transportation	0.16
	Site Preparation	0.23
	Civil Foundations	0.23
	Piping	0.32
	Electrical	0.32
	Controls	0.23
	Plant Services and Utilities	0.24
Indirect	EPCM	0.99
	Temporary Construction Services	0.20
	Insurance	0.10
	Spares	0.25
	Cold Commissioning	0.15
Total Hydromet Water Treatment Capex	**Total**	**6.61**

Source: SRK, 2025

21.2 Operating Costs

The operating costs for mining, the concentrator plant, the hydrometallurgical plant, and the waste management were estimated. The costs are summarized in Table 1-4. Some costs were fixed over the project life while others are variable due to changes in the mine production. The total life of mine operating cost was estimated at $2,566M, which corresponds to an equivalent unit production cost of $38.42/kg$_{\text{NdPrO}}$.

Table 21-10: Operating Costs Summary

Operating Costs	LOM ($M)	LOM avg ($M/y)	LOM ($/kg NdPrO equivalent in MREC)
Mining	552	36.8	8.27
Concentrator	744	49.6	11.14
Hydrometallurgical Facility	999	66.6	14.95
Mine Site & Hydrometallurgical Plant G&A	135	9.0	2.03
Mine Site Tailings	94	6.3	1.41
Hydrometallurgical Waste	20	1.4	0.31
Contact Water Pond	2	0.2	0.04
Mine Site Water Treatment	11	0.7	0.16
Hydrometallurgical Plant Water Treatment	8	0.5	0.12
Total	**2,566**	**171**	**38.42**

Source: Hatch, 2025

Operating personnel costs for water treatment are included in the operating costs for the concentrator and hydrometallurgical facilities.

21.2.1 Mining Operating Cost Estimate

The mining operating costs are organized into mining activity and presented in Table 21-11 in terms of total dollars over the life-of-mine plan and the relevant unit costs. Equipment operating costs are derived from vendor-provided quotes and assumed diesel pricing and, for ancillary equipment, SRK's industry benchmark costs.

The blasting cost was estimated from first principles with a vendor quotation for explosives, accessories, and contract services.

Labour costs are derived from a labour model and rates from the region and are categorized into hourly and salaried cost. A burden rate of 40% is applied to all employees.

The operating cost of the equipment was estimated based on equipment productivities, capacities, working hours and efficiencies to estimate the operating hours to which hourly rates were applied. A fuel price of C$1.25 per litre was used to estimate the fuel cost.

Table 21-11: Mine Operating Costs

LOM Operating Costs - Activity	M	552.2
Drilling	US$M	28.8
Blasting	US$M	71.8
Loading	US$M	62.7
Hauling	US$M	122.1
Support Equipment	US$M	93.9
Ancillary Equipment	US$M	86.6
Management & General	US$M	51.1
Technical Services	US$M	21.9
Water Management	US$M	9.2
Contract Services	US$M	4.2
Unit Operating Costs - Activity	**US$/t**	**5.54**
Drilling	US$/t	0.29
Blasting	US$/t	0.72
Loading	US$/t	0.63
Hauling	US$/t	1.23
Support Equipment	US$/t	0.94
Ancillary Equipment	US$/t	0.87
Management & General	US$/t	0.51
Technical Services	US$/t	0.22
Water Management	US$/t	0.09
Contract Services	US$/t	0.04

Source: SRK, 2025

These mining costs exclude costs associated with FTSF construction. That cost is captured under the FTSF cost estimate.

21.2.2 Concentrator Operating Cost Estimate

Concentrator plant operating expenditure (OPEX) includes costs associated with all process plant areas, plant maintenance, and plant technical services (metallurgy and engineering).

The estimation methodology was varied by cost component, but was primarily built from first principles relying on a combination of:

- Mass balance and process design criteria
- Power consumption estimates per motor
- Hatch in-house knowledge of similar operations
- Supplier reagent and consumables quotes.

Major categories include the following, which collectively result in a processing cost estimate for each category:

- Labor
- Reagents
- Consumables

- Utility
- Maintenance Materials
- Miscellaneous.

Process plant facility operating costs were estimated for the 1.8 M tonnes per year of process plant feed, and the following operating hours for each unit operation:

- Crushing: 3,780 hrs/yr
- Milling, flotation, and thickening circuit: 7,728 hrs/yr
- Filtration circuit: 7,140 hrs/yr.

21.2.2.1 Base Date and Exchange Rates

The concentrator OPEX estimate is in real terms on an annual basis, excluding any consideration for inflation or escalation. The estimate base date is Q1 2025. The base currency of the OPEX is USD dollars ($).

21.2.2.2 Summary of Estimate

The operating cost has been estimated on a yearly basis. The forecast average yearly operating cost estimate for the concentrator plant is provided in Table 21-12. The annual operating cost to process one tonne of plant feed is $28.10.

Table 21-12: Summary of Concentrator OPEX

Item	Total Cost ($/Year)	$/t of Plant Feed
Labor	11,069,293	6.15
Reagent	24,975,980	13.88
Consumable	3,069,976	1.71
Utility	5,989,212	3.33
Maintenance	5,470,462	3.04
Total	**50,574,923**	**28.10**

Source: Hatch, 2025

21.2.2.3 Labour – Production and Plant Maintenance

Labour staffing was estimated from first principles, and agreed to by Defense Metals. The labour costs incorporate requirements for plant operation, such as management, metallurgy, operations, maintenance, and site services staff. The total operational labour averages 111 employees. Their salaries were estimated based on public information on trade rates, union contracts, and in-house data for similar operations. Table 21-13 summarizes the costs of mill operation staff, and maintenance labour requirements based on total annual salaries.

Table 21-13: Concentrator Operation & Maintenance Labour Summary

Labour List	# of positions per shift	# of shifts worked	Rotation Shift	Annual Total Cost ($/Year)
Mill Operation and Technical Staff				
Technical Staff				
Plant Manager	1	1	No	233,405
Senior Metallurgist	1	1	No	154,206
Metallurgist	1	2	Yes	462,617
Met Lab Technician	2	1	No	197,576
Mill Operations				
Mill Foreman	1	1	Yes	266,005
Shift Supervisor	1	2	Yes	491,531
Mill Clerk	1	1	No	67,465
Crushing Operator	1	2	Yes	398,469
Grinding Operators	1	2	Yes	398,469
Flotation and dryer Operators	1	2	Yes	398,469
Thickening, and Dewatering Operators	1	2	Yes	346,499
Assistance Safety/Training Officer Mill	1	1	No	115,654
Concentrate Loadout	2	1	Yes	346,499
Reagents Make-up Laborer	1	2	Yes	311,733
Labour General	1	2	Yes	311,733
Day Shift Laborers	2	1	Yes	311,733
Mill and Surface Maintenance				
Maintenance Manager	0.5	1	No	90,769
Electrical Foreman	1	1	Yes	266,005
Mill and Surface Maintenance Foreman	1	1	Yes	266,005
Instrumentation Foreman	1	1	Yes	266,005
Mill Maintenance Planner	1	1	Yes	136,064
Millwright	4	1	Yes	868,981
Filtration Operators (Tails Filter Plant)	1	2	Yes	346,499
Laborers (Tails Filter Plant)	1	2	Yes	311,733
Instrument Technician	1	2	Yes	434,490
Mill Electricians	1.5	2	Yes	686,144
Welder	1	1	Yes	194,396
Site Services Supervisor	1	2	Yes	491,531
Site Services Operators	3	2	Yes	1,029,282
Administration				
IT and Communications.	1	1	Yes	176,571
First Aid Attendant/gate house	2	1	Yes	294,286
Total				**11,069,293**

Source: Hatch, 2025

21.2.2.4 *Reagents*

Annual operating consumables expenses were calculated from the required reagent consumption (kg/t processed).

Individual reagent consumption rates were estimated based on the metallurgical test work results, Hatch’s in-house database and experience, industry practice, and peer-reviewed literature. Reagent unit costs ($/t) were obtained through vendor quotations or recent in-house data. Reagents represent approximately 49.4% of the total process operating costs.

Table 21-14 depicts the estimated rates and cost for the reagents used in the process plant.

Table 21-14: Summary of Concentrator Plant Reagents OPEX

Reagent	Consumption (t/Year)	Annual Cost (CAD/Year)
Flocculant Magnafloc 336	90	355,500
Coagulant Magnafloc 504	1,440	5,889,600
Soda Ash	5,423	3,254,040
F220	4,410	10,363,500
Sodium Fluorosilicate	1,125	3,656,250
REEC5	158	677,643
AF3	158	523,173
Aero 6493	23	256,275
Total		**24,975,980**

Source: Hatch, 2025

21.2.2.5 Consumables

The maintenance consumables include crusher and ball mill liners, grinding media for the ball mill, and filter cloths. The consumption rates were estimated using:

- Metallurgical testing results (abrasion)
- Vendor budgetary quotes
- Hatch in-house calculation methods
- Forecasted total power consumption.

The unit costs were obtained through vendor quotations or recent Hatch in-house data. Table 21-15 depicts the various consumption rates and associated operating costs for the concentrator plant.

Table 21-15: Summary of Concentrator Plant Consumables OPEX

Consumable	Total cost ($/Year)
Jaw Crusher Liner consumption	144,200
Grinding Mill Liner Consumption	1,099,038
SAG Mill Media	421,200
Ball Mill Media	700,560
Filter Cloth	704,978
Total	**3,069,976**

Source: Hatch, 2025

21.2.2.6 Utilities

The utility consumption rate was estimated using:

- The processing power draw was based on the average power utilization of each motor on the equipment list for the process plant and services and building lighting and air conditioning system. Electrical heater power draw was calculated based on predicted average monthly usage. Annual power consumption was estimated to be approximately 149,000 MWh. The annual power costs were calculated using a unit price of $ 0.040 / kWh estimated based on information received from BC Hydro.

Table 21-16 summarizes the utility estimate for the concentrate facility.

Table 21-16: Summary of Concentrator Utility OPEX

Item	Total Cost ($/Year)
Power including building & Camp	5,988,164
Total	**5,988,164**

Source: Hatch, 2025

21.2.2.7 Maintenance

Maintenance cost for the concentrator plant were calculated based on total equipment and material costs for each area using a weighted average factor of between 2.7% and 4%. A factor of 10% was applied on total mobile equipment cost to estimate mobile equipment maintenance cost.

21.2.2.8 Assay Laboratory

Concentrator laboratory cost was included in the hydrometallurgical plant laboratory cost per Defense Metals direction. Samples will be taken the hydrometallurgical plant for processing

21.2.3 Hydrometallurgical Plant Operating Cost Estimate

The annual operating cost estimate (OPEX) for the hydrometallurgical plant is presented in Table 21-17. Details for each cost components are provided in the subsections. This OPEX comprises both fixed costs (independent of production throughput) and variable costs (scale with throughput) – the basis for the values shown here is a throughput of 85,800 t/a of dry concentrate (5670 t/a NdPrO). The resulting hydrometallurgical plant operating cost (at the design conditions) is $14.7 USD/kg$_{\text{NdPrO}}$. For the life of mine, given the variable plant throughput, the overall operating cost is 15.0 USD/kg$_{\text{NdPrO}}$.

Reagent costs are based on the mass balance consumption rates and recent unit reagent pricing (as-delivered basis). Consumables costs are based on expected consumption rates and recent prices, and in-house data from similar projects. Utilities costs are based on the electrical load list, and the mass and energy balance, alongside utilities unit costs for British Columbia. Labour costs are based on a preliminary staffing plan and typical wages for the area. Maintenance costs are factored from the equipment supply costs. The contract laboratory and environmental services costs are taken from in-house data from similar projects. The hydrometallurgical costs exclude general and administrative costs as these costs are project-wide and are accounted for elsewhere.

Table 21-17: Hydrometallurgical Plant Operating Cost Estimate

Cost Description	Estimated Annual Expenditure (USD$/year)			
	Annual Cost ($M)	% of Total Cost	Cost / t Concentrate (dry)	Cost / t NdPrO Equiv.
Reagents	$56.28	67.6%	$656	$9,925
Consumables	$1.68	2.0%	$20	$296
Utilities (includes Power)	$8.65	10.4%	$101	$1,525
Labour	$12.67	15.2%	$148	$2,235
Maintenance	$3.83	4.6%	$45	$675
Contract Services	$0.10	0.1%	$1	$17
Total	**$83.20**	**100%**	**$969**	**$14,673**

Source: Hatch, 2025

21.2.3.1 Basis of Estimate

The OPEX estimate is prepared based on the following parameters:

- OPEX has been estimated for the plant for a single year operating at full nominal capacity during the first 8 years of operation, given the assumed average feed rate, grade of ore, and recovery rate as per the Process Design Criteria (PDC).
- Reagent, consumables, and utility costs are exclusive of tax or duties.
- No consideration is given to long term variation or averages for reagent/utility pricing or labour costs.
- The OPEX estimate does not include any contingency/miscellaneous costs.

The estimate is based on the following inputs.

Table 21-18: Overall Inputs

Description	Input	Source
Feed Input	85,824 t/a dry flotation concentrate	Process Design Basis
Feed Moisture Content	8 wt%	Process Design Basis
Operating Availability[1]	85%	Process Design Basis
Operating Hours per year	7140 hours per year	Process Design Basis
Canadian Dollar per US Dollar	$1.40 CAD / $1.00 USD	OANDA Business Information & Services Inc. (2024-11-30)

Source: Hatch, 2025

Note 1: operating availability is the amount of time the facilities are running at design throughput.

21.2.3.2 Variable and Fixed Costs

For the economic analysis, the hydrometallurgical operating costs were split into Variable and Fixed Costs as described in Table 21-19.

Table 21-19: Operating costs Variable/Fixed Categories

Cost Description	Variable/Fixed	Cost Basis (USD)
Reagents	Variable	$655.77/t concentrate (dry)
Consumables	Variable	$19.54/t concentrate (dry)
Utilities (non-process)	Fixed	M$0.96 /y
Utilities (process)	Variable	$89.55/t concentrate (dry)
Labour	Fixed	M$12.67 /y
Maintenance	Fixed	M$3.83 /y
Contract Services	Fixed	M$0.10 /y
Total Variable	**Variable**	**$764.86/t concentrate (dry)**
Total Fixed	**Fixed**	**M$17.56 /y**

Source: Hatch, 2025

21.2.3.3 *Reagent Costs – Hydrometallurgical Plant*

Reagent consumptions were calculated based on the mass and energy balance. The reagent consumptions provided are based on the nominal consumption. Most reagent unit pricing was provided by various vendors via budgetary quotation in Q4 2024/Q1 2025, with some costs escalated from Q1 2024 prices (3%); other reagent costs were taken from internal databases of reference costs from similar projects. The quoted costs include transportation to the Bear Lake site. A summary of the total reagent consumption and cost is provided in Table 21-20.

High consumption of sulphuric acid is required for the acid mixing and baking step to liberate and solubilize the rare earth minerals. High consumption of lime is also used to raise the pH of the system including in the off-gas treatment, magnesium removal, and ammonia recovery systems.

Table 21-20: Hydrometallurgical plant reagent costs

Cost Description	Annual Cost (M$USD /y)	Consumption (t/y)	Unit Cost ($USD/t)	Unit Cost Reference
Sulphuric Acid (93% H2SO4)	$21.88	119752	$183	Vendor quote – over the fence supply
Quicklime (CaO)	$23.01	66793	$344	Vendor quote – as delivered
Magnesia (MgO)	$7.66	12549	$611	Vendor quote – as delivered
Hydrogen Peroxide (50% H2O2)	$0.09	102	$921	Vendor quote – as delivered
Ferric Sulfate (50% Fe2(SO4)3)	$2.86	5980	$478	Internal Cost Database
Ammonia (29% NH3)	$0.30	876	$339	Vendor quote – as delivered
Carbon Dioxide (CO2)	$0.00	0	$441	Vendor quote – as delivered
Magnafloc 10	$0.09	24	$3,850	Vendor quote – as delivered
Magnafloc 338	$0.04	10	$3,850	Vendor quote – as delivered
P507 Extractant - Annual Replacement (m3)	$0.19	22	$8,455	Vendor quote – as delivered
Exxsol D80 Diluent - Annual Replacement (m3)	$0.17	88	$1,899	Vendor quote – as delivered
Total	**$56.28**			

Source: Hatch, 2025

21.2.3.4 *Consumables Costs – Hydrometallurgical Plant*

Consumables costs are included to account for regular changes of process equipment consumables such as filter cloths and packaging bags. The consumables are based on the mechanical equipment list, vendor data, and internal databases. Future test work is needed to verify some consumptions and replacement rates in the next phase of the project.

Allowances for some consumable costs were estimated based on a percentage of the associated equipment cost or from internal databases of reference costs from projects comprising of equipment of similar size and complexity. A summary of the total assumed consumable quantity and cost is provided in Table 21-21.

Table 21-21: Hydrometallurgical plant consumables costs

Cost Description	Annual Cost (M$USD /y)	Consumption	Unit Cost ($USD)	Unit Cost Reference
Filter Cloths	$0.16	6572 m2/y	$24.28	Internal Cost Database
Gland Water Filter Cartridges	$0.001	unit	$1,159	Factored from equipment cost
Dust Collector Bags	$0.005	unit	$4,583	Factored from equipment cost
Cooling Tower Chemicals	$0.15	27.6 t/y	$5,512	Internal Cost Database
Laboratory Costs	$0.18	unit	$183,929	Internal Cost Database
IX Resin Replacement	$0.19	22 m3/yr	$8,816	Vendor quote
IX Cartridge Guard Filter Unit	$0.01	unit	$6,917	Factored from equipment cost
Polypropylene Lined Bags	$0.26	10749 bags/year	$24.62	Vendor quote
Pallets	$0.35	10749 pallet/year	$32.16	Vendor quote
Vehicle Operating Cost	$0.37	unit	$367,857	Internal Cost Database
Total	**$1.68**			

Source: Hatch, 2025

21.2.3.5 Utility Costs – Hydrometallurgical Plant

The process utility usage was obtained from the mass and energy balance, and the Electrical Load List. Internal reference data from similar projects, vendor provided data, and public utility rates for British Columbia have been used to determine the utilities unit costs. The utility demands provided are based on the nominal consumption. The plant water usage excludes fire water and does not consider savings from recovered fresh water from the hydrometallurgical effluent plant. A summary of the total utilities demand and cost is provided in Table 21-22.

Table 21-22: Hydrometallurgical Plant Utilities Costs

Cost Description	Annual Cost (M$USD /y)	Consumption	Unit Cost ($CAD)	Unit Cost Reference
Electricity		(kW)	($/kWh)	
Area 1300 - Administration and Maintenance Buildings	$0.43	1499	$0.040	BCHydro – electricity cost including carbon tax
Area 4100 - Concentrate Handling and Treatment	$0.13	470		
Area 4200 - Off-Gas Treatment	$0.22	759		
Area 4300 - Water Leach and Impurity Removal	$0.30	1033		
Area 4410/4420 - Rare Earth Production	$0.07	235		
Area 4430/4450 - Solvent Extraction	$0.22	758		
Area 4440 - Ammonia Recovery	$0.05	169		
Area 4500 - Depleted Liquor Recycling	$0.51	1765		
Area 4600 - Reagents Storage, Preparation and Distribution	$0.08	269		
Area 2300 - Site Utilities & Services	$0.25	863		
HVAC and Mobile Equipment	$0.17	585		
Natural Gas		(kW)	($/GJ)	
HVAC	$0.37	1611	$8.83	Fortis Inc. Industrial Rates for Businesses, including carbon tax.
Area 4100 - NG-fired kiln	$2.19	9651		
Area 4400 - NG-fired dryer	$0.21	939		
Water Supply		(m^3/h)	($/$m^3$)	
Total Water Supply	$0.11	69	$0.23	Internal Cost Database
Material Transportation Costs		(truck weeks/y)	$/week/truck	
Transportation Cost	$2.25	170	$13,243	Vendor quote - Transport of Concentrate to Bear Lake and PN residue to the mine site
Residue Disposal				
Residue Handling	$1.10	unit	$1,103,571	Internal Cost Database
Total	**$8.65**			

Source: Hatch, 2025

21.2.3.6 *Labour Cost – Hydrometallurgical Plant*

The preliminary staffing plan and labour rates were sourced from public information on trade rates, union contracts, and in-house data. The following methodology and assumptions were used to quantify labour requirements for the hydrometallurgical plant:

- The labour estimate includes only positions for the process plant staff and excludes general and administrative labour such as executive management, human resources, clerks, etc.
- It assumed that all staff live locally to the Hydrometallurgical plant and travel to site daily.
- The labour personnel were estimated based on the assumption of 2 x 12 hour shifts per day (total of 4 shift crews). Positions were classified as Day/Night shift (4/position), Day shift (2/position), and Weekday non-shift (1/position).
- Salaries shown are all-inclusive, and include the base salary, benefits, bonuses, and overtime.

The OPEX labour costs are summarized below in Table 21-23.

Table 21-23: Hydrometallurgical Plant Staffing Plan and Labour Cost

Cost Description	Annual Cost (M$USD /y)	Staff/position	Unit Cost ($USD/y)
Plant Operations			
Mill Foreperson	$0.266	2	$133,002
Shift Supervisor	$0.477	4	$119,304
Mill Clerk	$0.067	1	$67,465
Control Room Operator	$0.797	8	$99,617
Acid Bake Kiln Operator	$0.398	4	$99,617
Water Leach Operator	$0.346	4	$86,625
Ion Exchange and Residue Operator	$0.346	4	$86,625
SX Area Operator	$0.693	8	$86,625
Precipitation Operator	$0.346	4	$86,625
Mg and Bicarbonate area operator	$0.346	4	$86,625
Packaging Operator	$0.272	4	$68,032
Reagent Operator	$0.312	4	$77,933
Labourers	$0.408	6	$68,032
Mill and Surface Maintenance			
Maintenance Manager	$0.182	1	$181,538
Electrical Foreperson	$0.266	2	$133,002
Mill and Surface Maintenance Foreperson	$0.266	2	$133,002
Instrumentation Foreperson	$0.266	2	$133,002
Mill Maintenance Planner	$0.135	2	$67,465
Mill Mechanics	$1.033	12	$86,086
Instrument Technician	$0.344	4	$86,086
Mill Electricians	$0.689	8	$86,086

Cost Description	Annual Cost (M$USD /y)	Staff/position	Unit Cost ($USD/y)
Welder	$0.172	2	$86,086
Site Services Operators	$1.033	12	$86,086
Pipefitter	$0.457	4	$114,357
Technical Staff			
Plant Manager	$0.233	1	$233,405
Senior Metallurgist	$0.299	2	$149,714
Chief Assayer	$0.096	1	$95,911
Metallurgist	$0.449	4	$112,286
Assay Technicians	$1.156	12	$96,324
Met Lab Technician	$0.172	2	$86,086
Shift met technician	$0.344	4	$86,086
Total	**$12.67**	**134**	

Source: Hatch, 2025

21.2.3.7 Maintenance Cost – Hydrometallurgical Plant

Maintenance costs, accounting primarily for material spares, changeouts, etc., are estimated as a percentage of the mechanical equipment supply cost. This percentage is typically between 1 – 5% depending on the equipment type. Given the highly erosive and corrosive conditions for the acid mixing and acid baking units, a higher maintenance factor was assumed. Similarly, mobile equipment was assumed to require a higher maintenance cost. Costs associated with contract crewing for the annual maintenance shutdown is excluded. The maintenance costs in the estimate are assumed to cover sustaining capital cost. Contract laboratory testing and environmental services are based on internal reference data from similar projects. A summary of the maintenance cost is provided in Table 21-24.

Table 21-24: Hydrometallurgical Plant Maintenance Cost

Cost Description	Annual Cost (M$USD /y)	Maintenance Factor (%)	Equipment Supply Cost (M$USD)
Factored Maintenance			
Mechanical (excluding area 4100, 4430, and 4450)	$1.39	3%	$46.4
Mechanical (area 4100)	$1.17	9%	$13.0
Mechanical (area 4430 and 4450)	$0.92	3%	$30.6
Mobile Equipment	$0.35	10%	$3.5
Contract Services			
External lab testing services	$0.02		
Environmental Services	$0.07		
Total	**$3.92**		

Source: Hatch, 2025

21.2.4 General and Administration Costs

Site general and administration costs (G&A) were estimated at $6M per year.

An allowance of $710 per year and per person for other expenses (i.e., training, safety clothing, any computer and software licenses cost, etc.) was included in the overall mine site cost.

Camp cost was included for the mine site operating cost estimate A camp cost of $57 per day and per person was estimated for the concentrator plant camp cost

21.2.5 Infrastructure Operating Cost

21.2.5.1 Off-Site Infrastructure

No specific provisions for operating costs for off-site infrastructure were made. It is expected to be minimal and assumed to be included in G&A operating costs.

21.2.5.2 On-Site Infrastructure

No specific provisions for operating costs for on-site infrastructure were made. It is expected to be minimal and assumed to be included in G&A operating costs.

21.2.6 Tailings Operating Cost

This section includes the operating cost estimate starting from loading filtered tailings or hydrometallurgical waste off a stockpile until final deposition; it excludes filtering costs. Table 21-25 provides a summary of FTSF and HWSF operating costs per year. There are three years of additional operating costs beyond the LOM (i.e., Year 16 to 18). These are costs associated with monitoring and inspections, instrumentation installation, and contact water management activities that are completed during normal operation and are assumed to be required for at least three years following the end of operation. These costs are not included as part of the closure costs.

Further details are provided in Section 21.2.6.1 and Section 21.2.6.2.

Table 21-25: Tailings Operating Costs by Year

Year	FTSF operating costs (US$M)	HWSF operating costs (US$M)	Total (US$M)
1	5.5	1.1	6.6
2	6.2	1.1	7.3
3	6.5	1.3	7.9
4	7.7	1.6	9.2
5	6.2	1.3	7.5
6	6.0	1.1	7.1
7	6.0	1.3	7.3
8	6.1	1.3	7.4
9	6.7	1.4	8.1
10	6.4	1.3	7.7
11	5.9	1.3	7.3
12	5.9	1.3	7.3
13	6.5	1.5	8.0
14	7.4	1.5	8.9

Year	FTSF operating costs (US$M)	HWSF operating costs (US$M)	Total (US$M)
15	4.0	1.4	5.4
16	0.4	0.2	0.6
17	0.3	0.1	0.4
18	0.3	0.1	0.4
LOM Total	**94.0**	**20.5**	**114.4**

Source: SRK, 2025

21.2.6.1 Mine Site Tailings Operating Cost

Operating costs for the FTSF at the mine site include tailings load, haul and placement costs, costs for rockfill, and other supporting costs. These are discussed as follows:

21.2.6.1.1 Tailings Load, Haul and Placement

Average haul costs were calculated for each of the four FTSF stages assuming a haul route from the filter plant to the centroid of the stack stage. The average haul distance and elevation difference changes per stage. Haulage costs were calculated based on assumed equipment productivities and haul times. Load and haul unit rates ranged between $0.8-1.0 per metric tonne of moist filter cake material for the four FTSF stages.

Dozer spreading costs were calculated based on tailings production, equipment productivity and average push distances. The average unit rate for the dozer is estimated to be $0.35 per metric tonne of moist filter cake. Tailings compaction costs using a vibratory roller were assumed from the RSMeans cost database at $0.70 per cubic meter of placed and compacted material.

21.2.6.1.2 Rockfill

Costs for rockfill include rock cladding on outer slopes of the stack and for building temporary internal roads. All rock will be sourced from the proposed quarry.

21.2.6.1.3 Other Supporting Costs

Other supporting costs include site management and monitoring, engineering support, field testing and site investigation, and management of contact water.

Table 21-26: Mine Site Tailings (FTSF) Operating Costs by Year

Year	Tailings Load, Haul & Placement Costs ($M)	FTSF Rockfill Requirements During Operation ($M)	Other (Engineering, General Management & Other Support) ($M)	Contingency ($M)	Total ($M)
1	2.6	0.7	0.9	1.3	5.5
2	3.1	0.7	0.9	1.4	6.2
3	3.1	1.0	0.9	1.5	6.5
4	3.1	1.6	1.3	1.8	7.7
5	3.1	0.6	1.1	1.4	6.2
6	3.1	0.6	0.9	1.4	6.0
7	3.1	0.6	0.9	1.4	6.0
8	3.1	0.7	0.9	1.4	6.1
9	3.1	0.7	1.3	1.5	6.7

Year	Tailings Load, Haul & Placement Costs ($M)	FTSF Rockfill Requirements During Operation ($M)	Other (Engineering, General Management & Other Support) ($M)	Contingency ($M)	Total ($M)
10	3.1	0.7	1.1	1.5	6.4
11	3.0	0.7	0.8	1.4	5.9
12	3.0	0.7	0.8	1.4	5.9
13	3.0	0.7	1.3	1.5	6.5
14	3.4	1.3	1.0	1.7	7.4
15	2.7	1.3	0.8	1.4	6.2
16	-	-	0.3	0.1	0.4
17	-	-	0.2	0.1	0.3
18	-	-	0.2	0.1	0.3
LOM Total	44.0	12.6	15.7	21.7	94.0

Source: SRK, 2025

21.2.6.2 Hydrometallurgical Waste Operating Cost

Operating costs for the HWSF include the same items as the FTSF i.e., load, haul and placement costs, costs for rockfill, and other supporting costs. Given a physical location for the facility is yet to be determined, haulage costs assume a constant haul distance of 500 m between the hydromet plant and HWSF.

Table 21-27: Hydrometallurgical Waste (HWSF) Operating Costs by Year

Year	Waste Load, Haul & Placement Costs ($M)	HWSF Rockfill Requirements During Operation ($M)	Other (Engineering, General Management & Other Support) ($M)	Contingency ($M)	Total ($M)
1	0.5	0.1	0.3	0.3	1.1
2	0.5	0.1	0.3	0.3	1.1
3	0.5	0.2	0.3	0.3	1.3
4	0.5	0.2	0.5	0.4	1.6
5	0.5	0.1	0.4	0.3	1.3
6	0.5	0.1	0.3	0.3	1.1
7	0.5	0.2	0.3	0.3	1.3
8	0.5	0.2	0.3	0.3	1.3
9	0.5	0.1	0.5	0.3	1.4
10	0.5	0.1	0.4	0.3	1.3
11	0.5	0.2	0.3	0.3	1.3
12	0.5	0.2	0.3	0.3	1.3
13	0.5	0.1	0.5	0.3	1.5
14	0.5	0.3	0.4	0.3	1.5
15	0.5	0.3	0.3	0.3	1.4
16	-	-	0.1	0.04	0.2
17	-	-	0.1	0.02	0.1
18	-	-	0.1	0.02	0.1
LOM Total	7.4	2.5	5.8	4.7	20.5

Source: SRK, 2025

21.2.7 *Mine Site Water Management and Treatment Operating Cost*

Water management operating costs include costs to manage and monitor the CWP, as well as costs for engineering and consultant support (e.g., annual inspections, performance reviews, etc.).

Table 21-28: Mine Site Water Management Operating Annual Costs

Item	Annual Cost ($M)
Site management & monitoring	0.06
Engineering/Consultant support	0.06
Contingency	0.03
Total	**0.15**

Source: SRK, 2025

The operating cost for the mine site water treatment plant involves costs associated with power and heating, reagents, external contractors, and lab fees. The cost of power and heating was calculated to be 1,500,000 kWh, and mostly comes from power required for pumping. The cost of reagent is based on 380 tonnes per year of 50% Ferric Sulphate, and 4 tones per year of flocculant. An allowance based on SRK internal benchmarks were used for external contractors, and the lab fees are based on 730 samples per year.

Table 21-29: Mine Site Water Treatment operating annual costs

Item	Annual Cost ($M)
Power and Heating	0.07
Reagents	0.19
Repair, Maintenance and Contractors	0.25
Lab Fees and Consumables	0.21
Total	**0.72**

Source: SRK, 2025

21.2.8 *Hydrometallurgical Water Treatment Operating Cost*

The operating cost for the mine site water treatment plant involves costs associated with power and heating, reagents, external contractors, and lab fees. The cost of power and heating was calculated to be 1,350,000 kWh, and mostly comes from power required for pumping. The cost of reagent is based on 60 tonnes per year of lime, and 600 kg per year of flocculant, 15 tonnes per year of HCl (32%), as periodic membrane antiscalant and replacement. An allowance based on SRK internal benchmarks were used for external contractors, and the lab fees are based on 730 samples per year

Table 21-30: Hydrometallurgical Water Treatment Operating Costs by Year

Item	Annual cost ($M)
Power and Heating	0.10
Reagents	0.11
Repair, Maintenance and Contractors	0.11
Lab Fees and Consumables	0.21
Total	**0.52**

Source: SRK, 2025

21.3 Closure Cost

21.3.1 Mine Site

Undiscounted progressive-closure, closure and post-closure costs for the mine site are summarized in Table 21-31. These costs were developed using the costing assumptions described in Section 20.5.

Progressive-closure costs are closure costs that are assumed to occur during mine operations and consist of cover placement on the lower portions of the filtered tailings storage facility. Closure costs are assumed to be incurred over the three year active closure period. Post-closure costs are long-term monitoring, maintenance and water treatment costs for 100 years post-closure.

Indirect costs consist of mobilization/demobilization, engineering design and construction planning, contractor overhead and profit and construction management.

Table 21-31: Mine Site Progressive-Closure, Closure and Post-Closure Costs

Item	Progressive-Closure Costs (US$M)	Closure Costs(US$M)	Post-Closure Costs (US$M)
Waste Storage Facility	0.0	3.5	0.0
Filtered Tailings Storage Facility	1.6	1.5	0.0
Yards/Surface Disturbances/Roads	0.0	0.2	0.0
Buildings/Demolition	0.0	11.3	0.0
Water Treatment	0.0	4.1	176.9
Monitoring and Maintenance	0.0	0.6	8.1
Construction Management and Indirect Costs	0.3	9.6	38.3
Contingency	0.2	3.8	27.9
Total	**2.1**	**34.6**	**251.3**

Source: SRK, 2025

21.3.2 Hydrometallurgical Plant Site

Undiscounted progressive-closure, closure and post-closure costs for the mine site are summarized in Table 21-32. These costs were developed using the costing assumptions described in Section 20.5.

Progressive-closure costs for the hydrometallurgical plant site consist of covering all but the last cells in the hydrometallurgical waste storage facility.

Table 21-32: Hydrometallurgical Plant Site Progressive-Closure, Closure and Post-Closure Costs

Item	Progressive-Closure Costs (US$M)	Closure Costs (US$M)	Post-Closure Costs (US$M)
Hydrometallurgical Facility	8.6	4.6	0.0
Yards/Surface Disturbances/Roads	0.0	0.1	0.0
Buildings/Demolition	0.0	0.4	78.5
Water Treatment	0.0	3.4	2.3
Monitoring and Maintenance	0.0	0.4	4.0
Construction Management and Indirect Costs	1.7	5.5	17.0
Contingency	1.3	1.8	12.7
Total	**11.6**	**16.2**	**114.5**

Source: SRK, 2025

22. Economic Analysis

22.1 Summary

The economic analysis is based on a discounted cash flow model in real terms. The model includes the 15-year production plan, operating costs, capital costs, and market assumptions discussed in this report, in addition to financial assumptions introduced in this section. Project returns are calculated in the model before and after taxes, including net present value (NPV), internal rate of return (IRR), and payback period.

Returns are sensitive to input assumptions and should be viewed in the context of the sensitivity analysis provided in this section as well as the stated accuracies for items such as capital costs.

The base case assumes long term prices of NdPr oxide $132.7/kg, Tb oxide $1,362.8/kg, and Dy oxide $442.5/kg which result in a basket value of $121.9/kg TREO after including the balance of REOs. The base case also assumes a 95% payability for the TREO in MREC which gives an average realized price of $115.8/kg TREO equivalent. At these prices the project achieves a positive NPV at an 8% real discount rate. A summary of key indicators is shown in Table 22-1.

Table 22-1: Key Indicators Summary

Financial Metrics	Units	Value
Pre-tax NPV @ 8%	US$M	1,746
After-tax NPV @ 8%	US$M	957
Pre-tax IRR	%	24.2%
After-tax IRR	%	18.6%
Pre-tax payback period from start of production	Years	3.2
After-tax payback period from start of production	Years	3.7
Initial capital expenditure	US$M	1,441
Average annual operating cost	US$M per annum	171.1
Average annual operating cost	US$/kg NdPrO equivalent in MREC	38.4
NdPrO price	US$/kg	132.74
MREC realized price	US$/kg TREO equivalent	115.8
Gross revenue (LOM)	US$M	9,062
EBITDA margin	%	71

Source: Hatch, 2025

22.2 Assumptions and Inputs

22.2.1 General

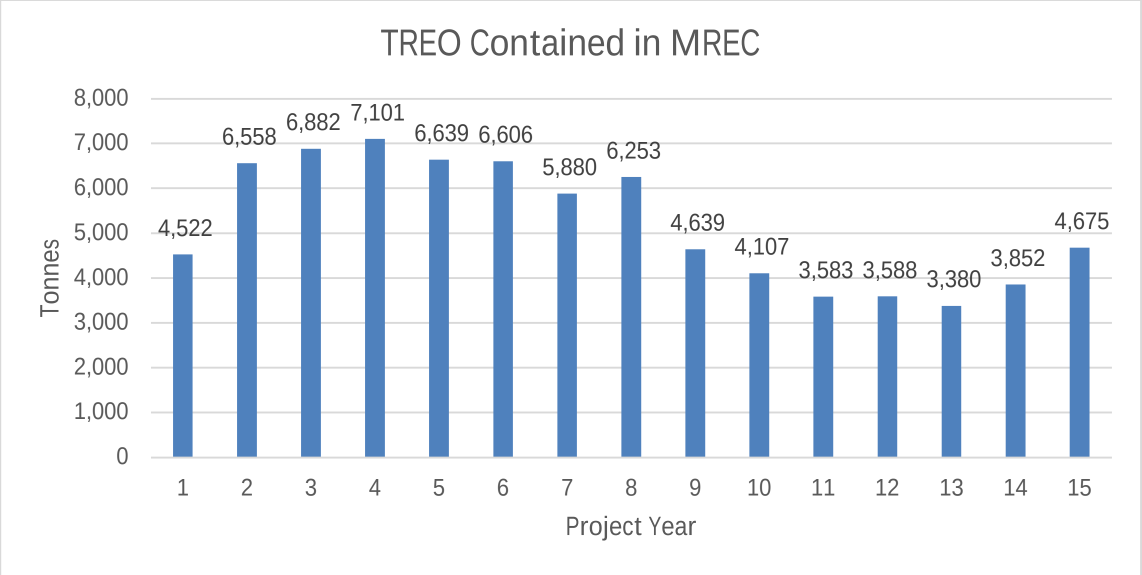
The following general assumptions form part of this analysis:

- The currency basis is real 2025 USD with no inflation.
- 100% equity financing.
- 1.4 C$/US$ exchange rate.
- Mid-year discounting for NPV calculation.

22.2.2 Production Schedule

The main elements of the production schedule are summarized in Table 22-2 and based on the following inputs and assumptions:

- **Concentrator feed:** Concentrator feed production is based on the production plan from Section 16.5.3. The plan delivers 1.5 Mtpa ore in Year 1, 1.8 Mtpa ore in Years 2-14, and 1.4 Mtpa ore in Year 15. The Year 1 feed is constrained to 1.5 Mtpa ore by the mill ramp up equal to 83.5% of steady-state annual capacity.
- **Concentrator feed grade**: Annual concentrator feed grades from the production plan from Section 16.5.3 are applied and are shown in Table 22-2. These result in Year 1-8 and Year 9-15 average TREO grades of 2.8% and 1.9%, respectively, corresponding to the Process Design Basis Table 17-1.
- **REE Recoveries:** Overall recoveries are the product of concentrator recoveries and hydrometallurgical recoveries. Concentrator recoveries vary year by year. The Year 1-8 and Year 9-15 average concentrator recoveries are 81% and 70%, respectively, corresponding to the Process Design Basis Table 17-1. The hydrometallurgical recoveries do not vary in the model and are taken from Table 17-5.
- **TREO Contained in MREC:** The average TREO contained in MREC is 5,218 tpa. The profile varies annually and is shown in Figure 22-1. Year 2-6 production exceeds the process design basis value of 6,490 tpa TREO in Table 17-3 somewhat due to varying concentrator feed grades but this is not a concern as it is still within the hydrometallurgical plant design margin.
- **MREC Production:** MREC production varies proportionally with the TREO production in MREC and has a grade of 60.4% (wet basis). The average annual MREC production is 8.6 ktpa (wet).



Source: Hatch, 2025

Figure 22-1: TREO Contained in MREC

22.2.3 *Product Pricing*

The long term REO prices recommended in Section 19 are applied. The main prices are NdPr oxide, Dy oxide, and Tb oxide as they account for ~98% of the basket value. The following values are applied along with the other elements (not shown) to arrive at a basket value of $121.9/kg of TREO in MREC:

- NdPr oxide at $132.7/kg
- Tb oxide at $1,362.8/kg
- Dy oxide at $442.5/kg.

The calculated basket value is in line with Section 19 after adjusting for feed grade variations and VAT.

A 95% payability is applied to the TREO in the MREC in line with Section 19.

22.2.4 *Transportation Costs*

A product transportation cost of $92.5/t (wet) of MREC is applied based on shipping to East Asia.

22.2.5 *Royalties*

The model applies a royalty of 1% of NSR plus a C$1M (US$0.71M) cost to buy out the other 1% of NSR as discussed in Section 4.2.

22.2.6 *Site Operating Costs*

The site operating costs are taken from Section 21. The model applies fixed and variable costs which lead to a variable cost profile and average annual cost of $171M.

22.2.7 Changes in Net Working Capital

Working capital is based on 90 days of accounts receivable, 60 days of accounts payable, and 30 days of inventory.

22.2.8 Capital Costs

The $1,441M initial capital cost estimate and sustaining capital cost estimates from Section 21 which average $9.1M/year are applied. Initial capital costs are spread over a three-year construction period. Closure costs of $430M from Section 21 are applied throughout, and past, the project life with $65M incurred up to and during the 3-year closure period and $366M being incurred throughout the 81-year post closure period.

22.2.9 Taxes

BC mineral tax includes a 2% tax on net current proceeds and a 13% tax on net revenues after the current expenditures account balance becomes positive. An investment allowance rate based on the Bank of Canada rate is applied on the current expenditures account.

Federal and provincial corporate taxes are based on a 15% federal and 12% provincial tax rate. Capital cost allowance (CCA) Class 41 depreciation at 25% is applied and tax losses are carried forward.

22.3 Cash Flow

The annual cash flow summary for the PFS base case is shown in Table 22-2 along with key economic indicators: NPV, IRR, and payback period. The NPV is positive at an 8% real discount rate.

Table 22-2: Annual Cash Flow Summary

Item	Units	Total/ Average	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	Post closure period
Material Mined	Mt	111.5	-	3.6	8.7	7.5	8.1	10.5	8.4	8.0	8.4	8.0	6.8	7.7	5.7	5.6	5.0	3.9	3.7	1.9	-	-	-	-
Effective Strip Ratio	t:t	3.2	-	-	61.8	4.5	2.7	5.0	3.7	3.4	3.6	3.0	2.9	3.8	1.9	2.3	1.9	1.2	1.1	0.7	-	-	-	-
Concentrator Feed	Mt	26.3	-	-	-	1.5	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.4	-	-	-	-
TREO Grade	%	2.4%	-	-	-	2.5%	2.9%	3.0%	3.1%	2.9%	2.8%	2.5%	2.7%	2.1%	2.0%	1.7%	1.7%	1.6%	1.6%	2.4%	0.0%	0.0%	0.0%	0.0%
Pr6O11 Grade	%	0.09%	-	-	-	0.10%	0.11%	0.12%	0.12%	0.11%	0.11%	0.10%	0.11%	0.08%	0.08%	0.07%	0.07%	0.07%	0.07%	0.09%	0.00%	0.00%	0.00%	0.00%
Nd2O3 Grade	%	0.26%	-	-	-	0.27%	0.30%	0.31%	0.32%	0.30%	0.30%	0.27%	0.30%	0.23%	0.22%	0.20%	0.21%	0.20%	0.20%	0.27%	0.00%	0.00%	0.00%	0.00%
Tb4O7 Grade	%	0.0011%	-	-	-	0.0011%	0.0012%	0.0012%	0.0012%	0.0011%	0.0011%	0.0012%	0.0012%	0.0009%	0.0009%	0.0009%	0.0010%	0.0010%	0.0010%	0.0011%	0.0000%	0.0000%	0.0000%	0.0000%
Dy2O3 Grade	%	0.0032%	-	-	-	0.0033%	0.0034%	0.0034%	0.0033%	0.0030%	0.0033%	0.0036%	0.0037%	0.0028%	0.0027%	0.0030%	0.0033%	0.0034%	0.0033%	0.0035%	0.0000%	0.0000%	0.0000%	0.0000%
TREO Recovery	%	12.5%	-	-	-	12.1%	12.6%	12.6%	12.8%	12.8%	13.0%	13.2%	12.7%	12.5%	11.5%	11.5%	11.4%	11.4%	13.2%	13.5%	-	-	-	-
Pr6O11 Recovery	%	71.6%	-	-	-	71.0%	76.0%	76.4%	76.4%	76.4%	76.4%	74.8%	73.7%	69.6%	64.7%	61.1%	59.6%	57.7%	67.6%	75.8%	0.0%	0.0%	0.0%	0.0%
Nd2O3 Recovery	%	71.7%	-	-	-	71.3%	76.2%	76.7%	76.7%	76.7%	76.7%	75.0%	73.9%	69.8%	64.9%	61.3%	59.8%	57.9%	67.9%	76.1%	0.0%	0.0%	0.0%	0.0%
Tb4O7 Recovery	%	61.2%	-	-	-	61.3%	65.6%	66.0%	66.0%	66.0%	66.0%	64.6%	63.6%	60.1%	55.9%	52.8%	51.5%	49.9%	58.4%	65.5%	0.0%	0.0%	0.0%	0.0%
Dy2O3 Recovery	%	55.7%	-	-	-	56.0%	59.9%	60.3%	60.3%	60.3%	60.3%	59.0%	58.1%	54.9%	51.1%	48.2%	47.0%	45.5%	53.3%	59.8%	0.0%	0.0%	0.0%	0.0%
TREO in MREC	t	78,264	-	-	-	4,522	6,558	6,882	7,101	6,639	6,606	5,880	6,253	4,639	4,107	3,583	3,588	3,380	3,852	4,675	-	-	-	-
Pr6O11 in MREC	t	17,682	-	-	-	1,043	1,534	1,625	1,673	1,560	1,519	1,312	1,406	1,029	914	767	758	704	807	1,033	-	-	-	-
Nd2O3 in MREC	t	49,105	-	-	-	2,844	4,110	4,313	4,465	4,188	4,178	3,688	3,927	2,911	2,579	2,238	2,237	2,100	2,391	2,934	-	-	-	-
Tb4O7 in MREC	t	171.7	-	-	-	9.7	13.7	13.9	13.8	12.6	13.2	13.4	13.9	10.3	9.0	8.8	9.2	9.1	10.3	10.6	-	-	-	-
Dy2O3 in MREC	t	476.4	-	-	-	27.8	36.9	36.5	35.5	32.2	35.6	38.4	38.4	27.3	24.7	25.8	27.6	27.8	31.3	30.3	-	-	-	-
MREC produced	kt (wet)	129.5	-	-	-	7.5	10.9	11.4	11.8	11.0	10.9	9.7	10.3	7.7	6.8	5.9	5.9	5.6	6.4	7.7	-	-	-	-
TREO % in MREC	%	60.4%	-	-	-	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	60.4%	0.0%	0.0%	0.0%	0.0%
Basket Value	$/t REO	121.9	-	-	-	122.5	122.3	122.3	122.3	122.3	122.3	121.9	121.9	121.4	121.5	121.1	121.1	120.9	120.9	121.7	-	-	-	-
Payable %	%	95%	-	-	-	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Revenue Total	$M	9,061.6	-	-	-	526.4	762.1	799.6	824.8	771.2	767.2	680.9	724.2	535.0	474.2	412.4	412.7	388.2	442.5	540.3	-	-	-	-
Product transportation	$M	(26.3)	-	-	-	(1.5)	(2.2)	(2.3)	(2.4)	(2.2)	(2.2)	(2.0)	(2.1)	(1.6)	(1.4)	(1.2)	(1.2)	(1.1)	(1.3)	(1.6)	-	-	-	-
Royalty	$M	(90.4)	-	-	-	(5.2)	(7.6)	(8.0)	(8.2)	(7.7)	(7.6)	(6.8)	(7.2)	(5.3)	(4.7)	(4.1)	(4.1)	(3.9)	(4.4)	(5.4)	-	-	-	-
Site operating costs	$M	(2,566.1)	-	-	-	(163.2)	(190.1)	(198.1)	(200.8)	(194.0)	(192.2)	(182.2)	(184.0)	(169.9)	(157.0)	(152.2)	(149.5)	(145.1)	(150.3)	(136.0)	(0.6)	(0.4)	(0.4)	-
Mining	$M	(552.2)	-	-	-	(41.0)	(39.3)	(43.2)	(43.3)	(42.9)	(42.8)	(41.7)	(38.4)	(40.8)	(32.9)	(35.0)	(33.0)	(30.5)	(30.5)	(17.0)	-	-	-	-
Concentrator	$M	(744.3)	-	-	-	(44.1)	(50.6)	(50.6)	(50.6)	(50.6)	(50.6)	(50.6)	(50.6)	(50.6)	(50.6)	(50.6)	(50.6)	(50.6)	(50.6)	(42.7)	-	-	-	-
Hydrometallurgical facility	$M	(998.8)	-	-	-	(61.0)	(82.5)	(86.0)	(87.2)	(82.6)	(81.2)	(72.2)	(77.2)	(60.0)	(55.5)	(48.9)	(48.3)	(45.7)	(49.9)	(60.5)	-	-	-	-
Mine & hydromet G&A	$M	(135.4)	-	-	-	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	(9.0)	-	-	-	-
Mine & hydromet waste	$M	(114.4)	-	-	-	(6.6)	(7.3)	(7.9)	(9.2)	(7.5)	(7.1)	(7.3)	(7.4)	(8.1)	(7.7)	(7.3)	(7.3)	(8.0)	(8.9)	(5.4)	(0.6)	(0.4)	(0.4)	-
Mine water treatment	$M	(13.1)	-	-	-	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.0)	(0.0)	(0.0)	-
Hydrometallurgical water treatment	$M	(7.9)	-	-	-	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	-	-	-	-
EBITDA	$M	6,378.9	-	-	-	356.4	562.1	591.2	613.4	567.3	565.2	489.9	530.9	358.2	311.0	254.9	257.8	238.1	286.5	397.4	(0.6)	(0.4)	(0.4)	-
∆Net working capital	$M	-	-	-	-	(125.8)	(55.8)	(8.9)	(6.1)	12.8	0.9	20.6	(10.3)	45.3	14.6	14.7	(0.1)	5.8	(13.0)	(24.0)	129.4	(0.0)	-	-
Initial capital	$M	(1,440.6)	(339.6)	(589.1)	(511.9)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital	$M	(135.9)	-	-	-	(7.7)	(4.7)	(5.6)	(32.0)	(22.4)	(2.2)	(6.3)	(13.5)	(10.0)	(9.9)	(4.1)	(10.4)	(7.1)	-	-	-	-	-	-
Closure & reclamation	$M	(430.3)	-	-	-	-	-	-	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.2)	(1.2)	(1.2)	(1.2)	(17.6)	(18.1)	(15.1)	(365.8)
Royalty buy-out	$M	(0.7)	(0.7)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pre-tax cash flow	$M	4,371.4	(340.3)	(589.1)	(511.9)	222.9	501.7	576.6	574.2	556.6	562.7	503.1	506.0	392.3	314.6	264.3	246.1	235.6	272.3	372.2	111.2	(18.5)	(15.6)	(365.8)

Item	Units	Total/ Average	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	Post closure period
BC mineral tax	$M	(596.6)	-	-	-	(7.3)	(11.4)	(12.0)	(15.0)	(72.0)	(74.3)	(63.9)	(68.3)	(46.0)	(39.8)	(33.1)	(32.7)	(30.5)	(37.8)	(52.4)	-	-	-	-
Corporate tax	$M	(1,141.5)	-	-	-	-	(5.3)	(117.5)	(131.1)	(109.0)	(113.1)	(100.1)	(112.9)	(74.5)	(65.1)	(53.2)	(55.2)	(51.2)	(63.2)	(90.1)	-	-	-	-
After-tax cash flow	$M	2,633.2	(340.3)	(589.1)	(511.9)	215.6	485.0	447.1	428.1	375.6	375.3	339.1	324.7	271.9	209.7	177.9	158.2	153.9	171.3	229.7	111.2	(18.5)	(15.6)	(365.8)
Pre-tax NPV @ 8%	$M	1,746																						
After-tax NPV @ 8%	$M	957																						
Pre-tax IRR	%	24.2%																						
After-tax IRR	%	18.6%																						
Pre-tax payback	op. years	3.2																						
After-tax payback	op. years	3.7																						

Source: Hatch, 2025

22.4 Cash Flow Summary

Table 22-3 summarizes the life-of-project cash flow discounted and undiscounted, as well as the cash flow on a unit basis in $/kg NdPrO equivalent in MREC.

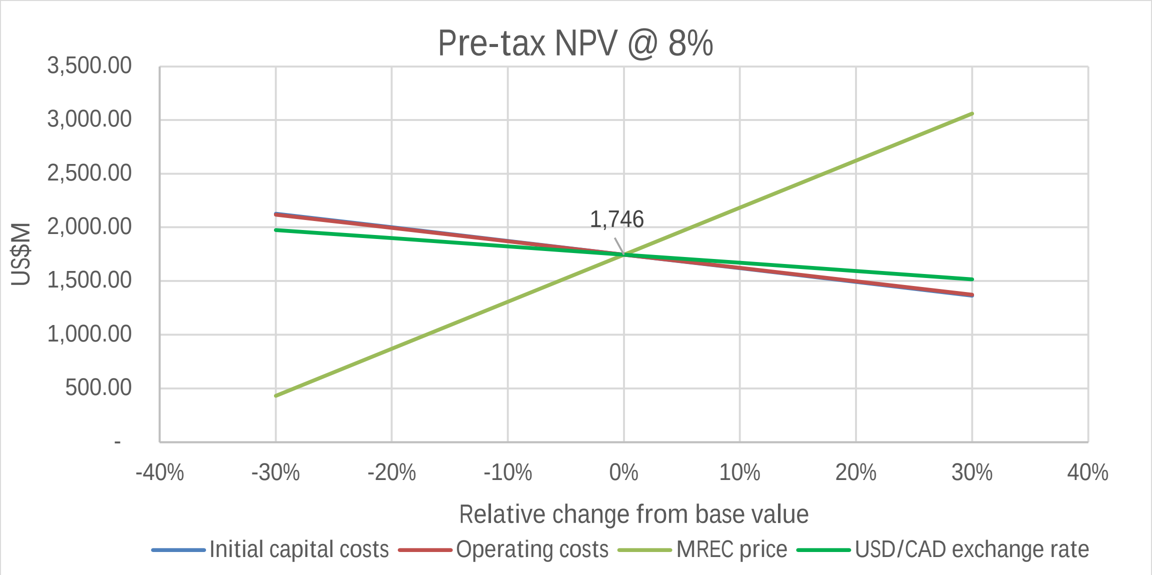
Table 22-3: Cash Flow Summary for Life of Project

Item	Undiscounted		Discounted
	LOM ($M)	Unit Average ($/kg NdPrO contained in MREC)	LOM ($M)
Gross Revenue	**9,062**	**135.68**	**4,513**
Operating Costs	(2,566)	(38.42)	(1,244)
Product Transportation	(26)	(0.39)	(13)
Royalties	(90)	(1.35)	(45)
EBITDA	**6,379**	**95.51**	**3,211**
Changes in Net Working Capital	-	-	(80)
Initial Capital Cost	(1,441)	(21.57)	(1,274)
Sustaining Capital Cost	(136)	(2.04)	(70)
Closure and Reclamation Bond Cost	(430)	(6.44)	(41)
Royalty buy-out	(1)	(0.01)	(1)
Pre-Tax Cash Flow	**4,371**	**65.45**	**1,746**
BC mineral tax	(597)	(8.93)	(261)
Corporate tax	(1,142)	(17.09)	(528)
After-Tax Cash Flow	**2,633**	**39.43**	**957**

Source: Hatch, 2025

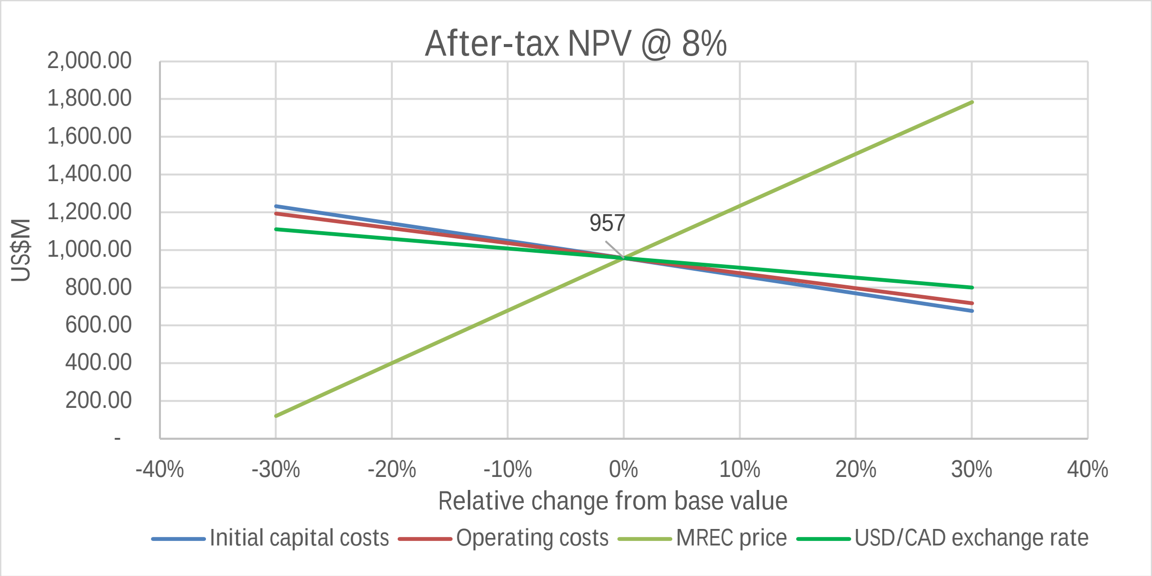
22.5 Sensitivity Analysis

A sensitivity analysis was conducted to show the impact of key variables on project returns. The product price, exchange rate, capital cost, and operating cost were each varied independently on an annual basis and the resulting variations in NPV @ 8% and IRR are shown in Figure 22-2 through Figure 22-5 before and after taxes. NPV is most sensitive to product price. Initial capital cost, operating cost, and exchange rate have a smaller impact on NPV. For clarity, variations in the exchange rate impact capital and operating costs originating in Canadian dollars, such as labour.



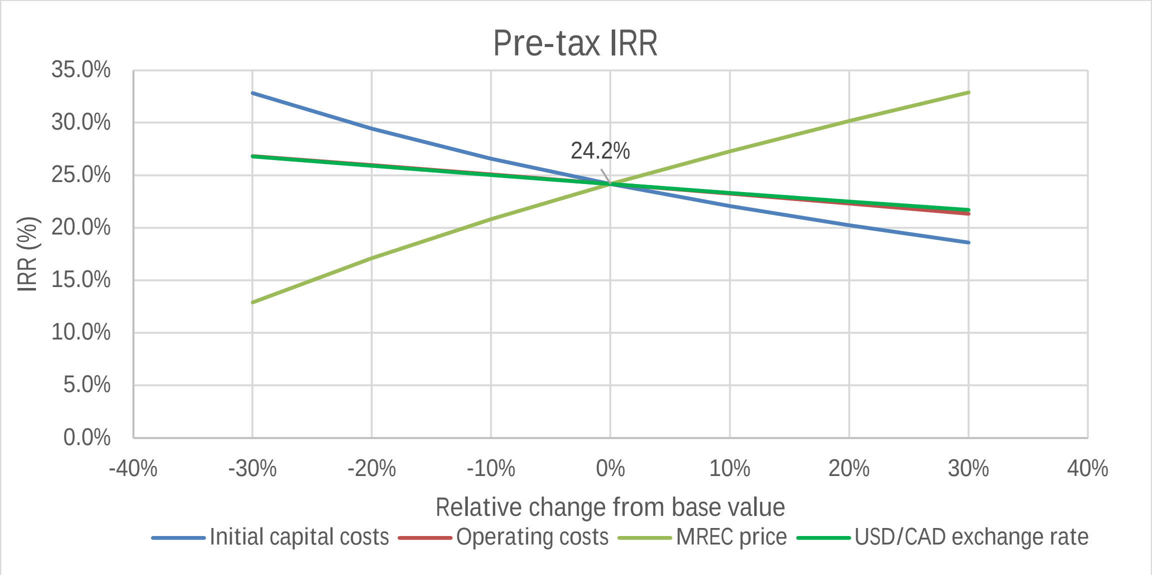
Source: Hatch, 2025

Figure 22-2: NPV @ 8% Pre-Tax Sensitivity



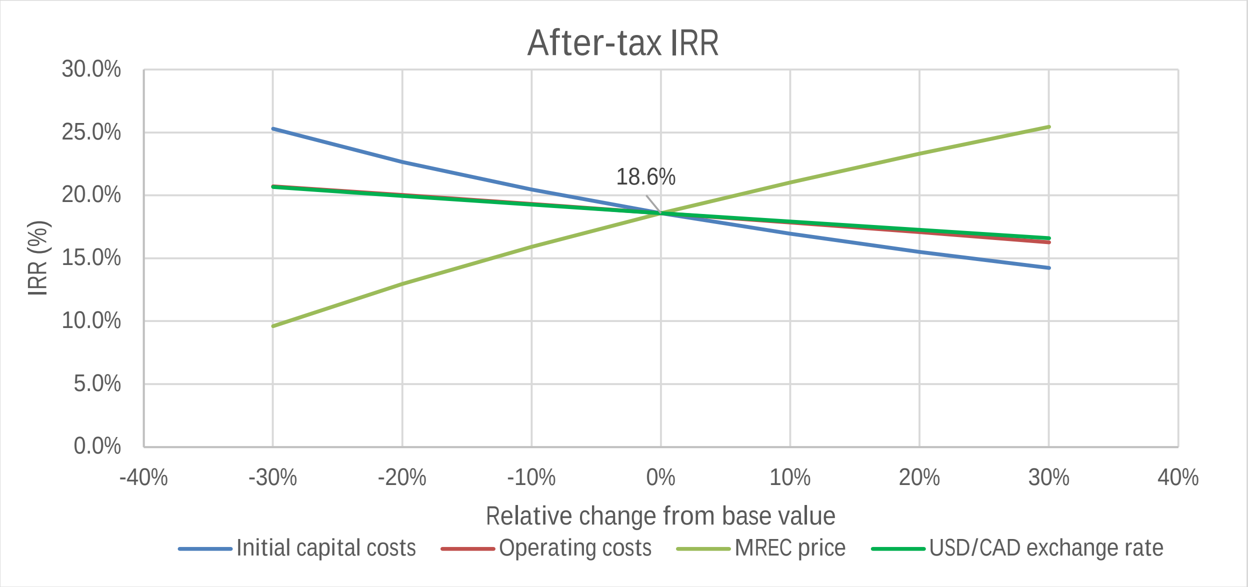
Source: Hatch, 2025

Figure 22-3: NPV @ 8% After-Tax Sensitivity



Source: Hatch, 2025

Figure 22-4: IRR Pre-Tax Sensitivity



Source: Hatch, 2025

Figure 22-5: IRR After-Tax Sensitivity

The NPV sensitivities to the discount rate are shown in Table 22-4.

Table 22-4: NPV sensitivities to discount rate

Discount rate	5%	6%	7%	8% (base)	9%	10%	15%
NPV Pre-tax, $M	2,518	2,233	1,976	1,746	1,538	1,352	658
NPV After-tax, $M	1,474	1,284	1,112	957	817	691	220

Source: Hatch, 2025

Finally, the NPV reaches $0 at an NdPrO price of $83.9/kg while the IRR reaches 0% at an NdPrO price of $66.0/kg, all other things equal.

23. Adjacent Properties

The QP has not been able to verify the following information nor is it indicative of the mineralization encountered within the Wicheeda Project.

Adjacent properties to the Project include the Wicheeda East group of properties of Neotech Metals Corp. (D1-D2 claims), Eagle Bay Resources, G. R. Delorme, M.J. Specogna and Trevor Alexander Rabb. (previously belonged to the Carbonatite Syndicate property of Canadian Carbon Resources). Records of drilling appear in the assessment reports of these properties. No significant intersections were presented for the Carbonatite Syndicate property drilling campaign in 2011 (Churchill et al., 2011).

Additional properties surrounding Defense Metals' Wicheeda project include Wicheeda north group of properties, Power One Resources Corp., D. H. Earl, C. N. Delorme and E. Black, N. W. Perk as well as a few other mineral claims of different owners (Figure 23-1).

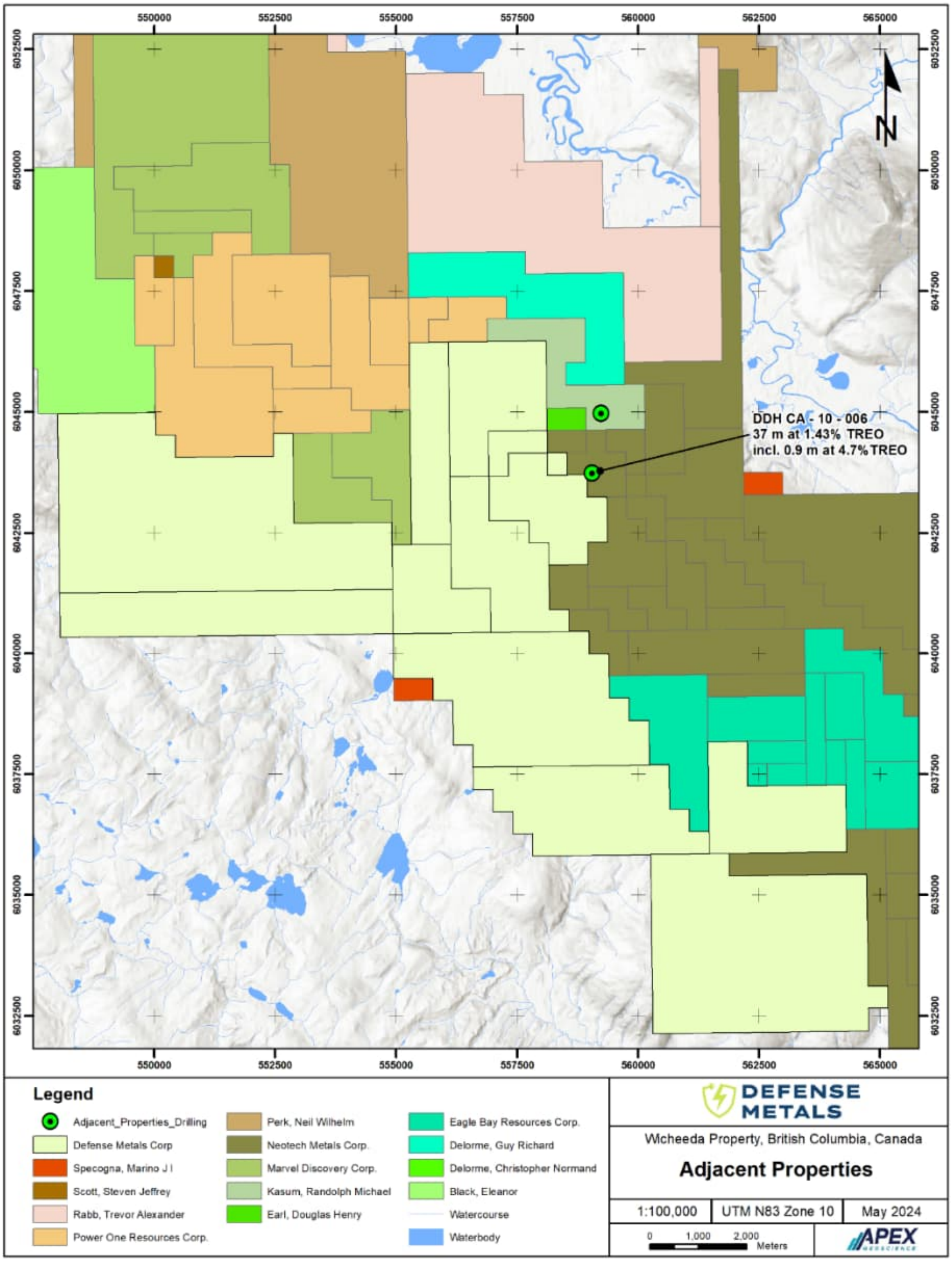
23.1 D1 Claim

The D1 claim (claim number 1106061), located to the East of the Wicheeda deposit covers 112.6 ha. Records of drilling have been reported for the D1 claim in the past while it was considered part of the Carbo 1 property of Canadian International Minerals INC. ("CCE"), which were originally staked by Jody Dahrouge in 2005 and 2006 (Bruland, 2011).

After positive geophysical airborne and soil sampling geochemical campaigns drilling took place Carbo 1 property claim in 2010. The drilling intersected REE mineralization in carbonatite dykes and a network of carbonatite/calcite veins that intruded the Upper Cambrian to Lower Ordovician Kechika Group bedded sediments. (Bruland, 2011).

The drilling returned Light Rare Earth Elements (LREE; Ce, La, Nd) and praseodymium Pr ranging from 4.7% Total Rare Earth Oxide (TREO) over 0.9 m to 1.4% TREO over 37.3 m in carbonatite dykes. REE minerals identified are parisite (of variable composition), bastnaesite, burbankite, monazite, and aeschynite with parisite the most common. Sulphide mineralization (pyrite, pyrrhotite, sphalerite, galena, arsenopyrite, chalcopyrite) is unrelated to the REE mineralization as is the Nb (Nb-rutile) (Bruland, 2011).

The Carbo 1 property lapsed and was re-staked by different entities. The D1 block is currently part of Neotech Metals Corp. with an expiry date of April 27, 2027.



DDH CA - 10 - 006
37 m at 1.43% TREO
incl. 0.9 m at 4.7% TREO

Source: APEX, 2025

Figure 23-1: Adjacent Properties

23.2 Regional Projects

Additional projects, in various stages of development , although not rare earth element related, include the following:

Giscome Quarry and Lime Plant- Giscome Quarry and Lime Plant, 50 km SSW of mine site area, owned by Graymont Western Canada Incorporated, consists of a lime processing facility and quarry with a production capacity of 600,000 tonnes per year and an estimated mine life of 25 years. Environmental Assessment Certificate Granted in 2016, not substantially started, granted an extension until December 14, 2026 (no more extension time allowed under the Act)

Summit Lake LNG Canada Ltd. JX LNG Canada Ltd. proposes to develop Summit Lake PG LNG, a liquid natural gas processing plant located approximately 30 kilometers north from Prince George, B.C. and 50 km southwest of the Project site. The proposed project is anticipated to produce approximately 10,060 thousand cubic metres per day of natural gas and 2.7 million tonnes per year of liquid natural gas. It would include incoming gas facilities, gas treatment facilities, liquefaction facilities, LNG storage facilities, product loading facilities, cooling systems, flare systems, fire and gas systems, and firefighting facilities.

Vitreo Minerals Ltd. proposes to develop the Angus silica sand mine located approximately 60 kilometres north of Prince George near Bear Lake. The proposed project is anticipated to produce approximately two million tonnes per year of silica sand which is used in hydraulic fracturing associated with crude oil and natural gas extraction.

The Nilhts'I Ecoener Project (a partnership between Ecoener, a Spanish company and Lheidli T'enneh First Nation) is expected to start construction in 2029 and is projected to produce 140MW from eighteen wind turbines. They have a 30 year power purchase agreement with BC Hydro. The project is located east of Hixon, 60 km from Prince George.

In 2023, MLIB and BC signed a memorandum of understanding that lay the foundation for the Tse'Knene Energy Transition Hub. The project is a carbon negative project being led by MLIB in partnership with Saulteau First Nation and is proposed for the MLIB Kerry Lake East Reserve south of McLeod Lake. Two projects are currently under development; the first is a green hydrogen production facility and the second is a natural gas liquids extraction plant and fractionator. The natural gas liquids extraction plant, also called a straddle plant, has undergone initial technical feasibility with AltaGas and has received confirmation from BCHydro of power availability. The hydrogen plant has progressed similarly and a formal working group has been set up with BC's Clean Energy Major Project Office.

24. Other Relevant Data and Information

No other relevant data or information is required.

25. Interpretation and Conclusions

25.1 Geology, Mineralization and Exploration

The descriptions of geology, mineralization and exploration contained in this report (Sections 7, 8, 9) and the recommendations for future work that follow in Section 26.1 are based on results from diamond drilling, geophysical surveys, geochemistry, and geological mapping. The QP foresees no specific risks that would impact continued exploration and development on the property.

25.2 Mineral Resource

The MRE reported herein is the similar to the MRE calculated and disclosed in 2023. The only change from the 2023 MRE to the current MRE was the change in reporting to reflect changes in the processing flowsheet and REE pricing.

The 2023 MRE was reported based on a cut-off of 0.50 TREO, the current MRE is reported based on a NSR approach which considered variable processing costs and related costs on a block-by-block basis. Blocks within the optimized pit shell showing a positive revenue, after all operating costs, were considered as a resource. This approach may have impacted the optimized pit shell as well. The 2024 model represents a 15% drop in Measured and Indicated tonnes, a 13% increase in TREO grade and a 4% decrease in contained TREO.

Factors that may affect the estimates include metal price and concentrate payable assumptions, changes in interpretations of mineralization geometry, continuity of REE mineralization zones, changes to kriging assumptions, metallurgical recovery assumptions, operating cost assumptions, confidence in the modifying factors, including assumptions that surface rights to allow mining infrastructure to be constructed will be forthcoming, delays or other issues in reaching agreements with regulatory authorities and stakeholders, and changes in land tenure requirements or in permitting requirements.

There are currently no known additional legal, political, title, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the potential development of the mineral resources. As the project develops and economic studies are completed, more information on these factors will become available.

Rare earth oxide (REO) price assumptions are based on the available TREO price information provided by Adamas Intelligence, an industry expert. However, rare earth offtakes are established with long term contracts to a limited number of refineries - primarily in Asia. The terms of these contracts are not public information. Offtake term assumptions are indicative only. It is not possible to accurately forecast these assumptions, and there is no guarantee that these terms will be realized. Assumptions on the product sales are indicative of potential market values but moving forward should be confirmed via commercial negotiations with refineries.

With respect to environmental and permitting risk and uncertainty, the area surrounding Wicheeda Lake has been known to have high recreational and ecological values. The lake and surrounding area are currently covered under recreational reserve REC6837 established by FLNR. At present there are no restrictions on mineral exploration activities within REC6837. However, FLNR has requested that Defense Metals takes all possible steps to minimize the impacts of exploration to the recreational ecological values associated with Wicheeda Lake.

The mine site area is within identified critical habitat for endangered southern mountain caribou. Prior to project development, a caribou mitigation plan will need to be developed collaboratively with Indigenous rightsholders that aligns with caribou recovery objectives.

It is established that MLIB holds aboriginal and treaty rights within the project area and that MLIB supports the continued evaluation of the project. Other rightsholders may be identified or self-identify as the project progresses particularly in relation to socio-economic or effluent related impacts.

The SRK QP is unaware of any other risks or uncertainties that could affect the accuracy or confidence of the MRE.

- Several sources of uncertainty exist with regards to the current MRE These include the following:
    - The current estimation methodology is largely based on correlation of all elements to Ce rather than based on individual elements. While Ce represents a significant percentage of the TREE, it provides little economic value in the current metallurgical flowsheet. SRK recommends future estimates be based on individual variograms for each element (or at a minimum, the major economic elements) within the dolomitic and xenolithic carbonatite units.
    - In SRK's opinion, the current block size is too small based on the current drill spacing. Studies to determine a better block size should be completed prior to future estimates.

However, the QP considers that these sources of uncertainty are minor and are unlikely to significantly impact the current MRE.

25.3 Mining

25.3.1 Pit Geotechnical

The following summarizes the opinion of SRK's pit geotechnical QP, Mr. Thomas, for pit geotechnical:

- Rock mass and structures data for the proposed pit envelope have been collected from HQ3 oriented core and televiewer data over two field campaigns. VWP installations and packer testing and was conducted for hydrogeological characterization.
- The primary structures model indicates a dominant NE to SW trend of high angle faults, none of which dip into/towards the pit. There are no modelled structures within an envelope extending 200 m beyond the final pit limits.
- Groundwater modeling indicates that phreatic surfaces will draw down as the pit is mined and conditions of elevated water pressure are unlikely to prevail.
- Structural fabric and rock block kinematic mechanisms are expected to be the primary control on achievable bench and slope configurations.

- A kinematics-based open pit slope design was developed, and the resulting pit shape slope stability analyzed using 2D limit equilibrium (LE) methods.
- LE stability analyses of vertical sections in each pit sector (six total), indicate factors of safety greater than the minimum acceptance criteria of 1.3.

25.3.2 Mine Planning

The Wicheeda project is an open-pit mining operation designed to supply an on-site flotation concentrator with ore at a rate of 1.8 Mtpa. The resulting flotation concentrate is transported off-site for further processing at a dedicated hydrometallurgical plant. The mine planning outlined in this PFS demonstrates that the mineral reserves can be effectively extracted using conventional truck-and-shovel mining methods. Additionally, the economic analysis confirms economic viability of the project based on the reserve.

Key risks and opportunities related to the mining aspects of the Wicheeda project are outlined below.

25.3.2.1 Mining Risks

- The project’s topography presents some challenge and safety concerns – vertical advance rate is a constraint and as such, if not achieved operationally, could slow the schedule.
- The two-phase pit design has mining in Phase 1 undertaken whilst Phase 2 commences above. Consequently, appropriate short –term mine planning particularly for blasting and loading practices near the front portion of Phase 2 need to be in place to ensure safe operations.
- There are identified Inferred mineral resources within and adjacent to the open pit that may warrant a pit expansion. In case of potential pit expansion, there may be the need for an additional WSF.

25.3.2.2 Mining Opportunities

- A substantial amount of inferred resource is located within and adjacent to the final pit. Upgrading these inferred resources to measured or indicated through additional exploration presents a strong opportunity for potential pit expansion.

25.3.3 Waste Storage Facility Geotechnical

The field investigations indicated the WSF will be founded on alluvium and glacial till surficial soils overlying bedrock. The surficial soils comprise alluvium generally less than 1.5 m thick and glacial till generally less than 10 m thick. The toe of the WSF is to be located on flat to gently sloping topography between 5 and 10°. The upper WSF portions are founded on steeper slopes.

The proposed WSF is located upslope of Wichcika Creek, and therefore, the geotechnical design and the adopted construction methodology is important to reduce the potential for an instability leading to consequential run-out event. A minimum setback distance of 120 m is established from the upper-most valley erosion scarps to the toe of the WSF (about 200 m from the main valley crest). The geotechnical design incorporates the following important elements to increase the physical WSF stability:

- Foundation preparation works and initial construction over flat to shallow sloping topography
- A hybrid approach with the initial two 50 m high platforms constructed with bottom-up methods
- The use of durable, strong rock in areas important to achieving the design stability conditions

The WSF geotechnical work carried out at the PFS has attempted to mitigate the identified risks, however, there other geotechnical aspects that need to be further investigated and/or evaluated to advance the design and mine plan to a feasibility level. These aspects include:

- Foundation behaviour under loading conditions representing the waste rock placement
- Potential geohazards and their interaction with the geotechnical design
- Run-out potential and setbacks to the river valley crest and infrastructure
- Local and overall stability conditions

Alternative waste rock placement options should be evaluated should there be a requirement to further setback the facility from the river valley crest, as identified during the FS.

25.4 Waste Management

25.4.1 ML/ARD Potential and Radionuclide Activity

Static and kinetic testing has indicated that acid rock drainage (ARD) potential of waste rock (including pit walls) is low due to high carbonate content, but there is some potential for metal leaching under mildly alkaline pH conditions, particularly for fluoride, molybdenum and uranium.

Static and kinetic testing has indicated that the ARD potential of tailings that contains well-blended WL/PN residue is low due to high carbonate content. There is some potential for metal leaching under mildly alkaline pH conditions, particularly for sulphate, molybdenum, and uranium.

Fluoride is expected to be elevated in process water due to flotation process reagent use.

Static testing of the LaCe MgR residue to be disposed at the hydrometallurgical plant has indicated low potential for ARD, but potential for elevated concentrations of dissolved sulphate.

Uranium and thorium abundances and radionuclide data for NORM indicate that waste rock, and tailings with blended WL/PN residue, exceed Health Canada’s unconditional derived release limits (UDRL’s) for diffuse NORM sources and therefore NORM management program requirements will need to be determined for these wastes.

Radionuclide results for the LaCe MgR residue to be disposed at the hydrometallurgical plant indicate this waste stream is within the UDRL's for diffuse NORM sources.

Risks and uncertainties that remain for characterization of ML/ARD potential and radionuclide activity of waste rock and process wastes are:

- The site review to determine NORM management program requirements may result in the need for thicker and/or more robust closure covers on the WSF.
- The lack of drilling into the sedimentary rocks that will represent the bulk of the waste rock. Additional drilling will be needed to acquire samples to characterize this waste.
- Uncertainty on how the WL/PN residue can be effectively blended with flotation tailings, and whether heterogeneity may cause an increased risk of metal leaching from the FTSF.
- Uncertainty on how periodically adding loaded uranium ion exchange resin to the FTSF may affect metal leaching potential and cause point sources of radionuclide activity that require additional management.
- For wastes that will be disposed at the hydrometallurgical facility, the LaCe MgR residue has undergone static testing for characterization of ML/ARD potential and radionuclide activity, but the other waste streams, dominantly gypsum and which will be stored with the LaCe MgR residue, lack static testing. None of the wastes have undergone kinetic testing therefore their behaviour upon weathering is unknown.

25.4.2 Mine Site Tailings

Tailings generated in the flotation concentrator will be transported via pipeline to a tailings filtration plant on the western side of Wichcika Creek and dewatered to a filter cake before being hauled, placed and compacted in a lined FTSF. The WL/PN residues and loaded uranium ion exchange resin produced at the hydrometallurgical plant will also be stored in the mine site FTSF. These wastes will be dewatered via filtration at the hydrometallurgical plant and hauled by truck to the FTSF before blending with the dewatered flotation tailings.

SRK has developed a preliminary design concept for the FTSF. It includes a foundation liner and drainage system, a starter toe embankment and a water management pond. The tailings stack will be built from the bottom up and rockfill cladding will be placed on the final outer slopes before they are progressively reclaimed.

The tailings storage risks currently include:

- The tailings alternatives assessment is preliminary. As part of permitting, a more detailed assessment may be required, one that includes multiple stakeholders and a more systematic method of evaluating and ranking options (e.g., a multiple accounts analysis). Future alternatives assessments could yield different results or changes to the preferred tailings option adopted for the PFS.
- There is very limited data on the foundation conditions of the FTSF area and ground conditions have been assumed based on preliminary terrain mapping, air photos and nearby investigation. Allowances have been made for foundation excavation removals

and the stack design concept is assumed to meet stability requirements. Should ground conditions be worse than those currently assumed, additional foundation improvements or changes to the design may be required which will impact cost. Uncertainty in foundation conditions should be addressed by future investigations and design phases.

- Hydrogeological site conditions at the FTSF are not well understood. Groundwater levels, seasonal fluctuation, ground hydraulic conductivity and occurrence of ephemeral springs need to be characterized at the site. This may change the configuration of the groundwater management system.
- It is assumed that the site bedrock is suitable for quarry development; however, its physical and chemical characteristics have not been studied. If nearby rock is unsuitable, the design philosophy might change and there would be potentially significant cost implications. Waste rock from the mine could be used but at increased cost due to the long haul required to the other side of Wichcika Creek.
- Preliminary analysis indicates that the planned covers should be thick enough to attenuate radiation and otherwise limit exposure of or to the tailings. The site review to determine NORM management program requirements may result in the need for thicker and/or more robust closure covers on the tailings.
- The design assumes tailings are filtered at or close to optimum moisture content for compaction. In the event of prolonged poor filter performance, a change in the design and operating strategy would be required, impacting cost.

Given the preliminary nature of the FTSF design, there is opportunity for future optimization, including:

- SRK has adopted a cautious approach in the design given the foundation uncertainties (e.g., large foundation excavation removal assumptions, relatively large toe embankment, overly wide bench widths). If these poor foundation conditions do not materialize, the stack design may be optimized to reduce costs.
- SRK has adopted a relatively robust FTSF groundwater management design, inclusive of a low permeability liner and below- and above-liner drains. Once the site groundwater conditions have been characterized, the system might be optimized.
- Unit rates for construction, except for load, haul and placement of tailings, are assumed to be contractor unit rates based on similar, recent projects. Unit rates could potentially be refined in future studies using actual quotes once designs are advanced.
- The closure concept for the FTSF assumes perpetual water treatment of FTSF seepage and runoff. Further evaluation of more robust closure covers could eliminate or reduce the water treatment requirements.

25.4.3 Hydrometallurgical Residues

A physical location for the Hydrometallurgical Waste Storage Facility (HWSF) has not yet been identified but is assumed to be located adjacent to the hydrometallurgical plant at Bear Lake. The storage concept for the hydrometallurgical wastes is to place and stack them in a lined facility. All three hydrometallurgical wastes stored in the HWSF are predominantly gypsum materials and will be filtered to a transportable filter cake and trucked to the facility for placement and compaction.

There are currently several significant uncertainties associated with the design of the facility given the lack of a physical site and the lack of geotechnical characterization, strength and compaction testwork for the three waste streams.

Risks and uncertainties include:

- Once a site is confirmed, various conditions including ground conditions, environmental, stakeholder, geological, topographical, etc. could present challenges not currently accounted for. The design currently assumes no adverse foundation or groundwater conditions. Likewise, the water management strategy includes a relatively simple design assuming little to no upstream catchments to manage.
- If a viable site is identified greater than 500 m from the Hydrometallurgical plant, haulage costs will increase.
- The design filter cake process moisture content for the three streams is 30% (43% geotechnical moisture content). This is relatively high and without further testing it is unclear if this is close to optimum for compaction or if significant drying in the field is required to achieve compaction densities. The high moisture could also make the material difficult to traffic. However, based on industry examples, specifically gypsum drystack residue storage systems elsewhere, it is presumed that the hydrometallurgical residues are trafficable and stackable. Further test work will be required in future studies to demonstrate the stacking concept for these materials is viable.
- Furthermore, in the event of prolonged poor filter performance, a change in the design and operating strategy would be required, impacting cost.

Opportunities include:

- It is currently assumed that four cells are built independently of each other. Once a site is confirmed, there may be potential for design optimization, for example, utilizing a portion of the starter embankment for adjoining cells.
- Strength and compaction testwork could show currently assumed dry density and maximum facility height restrictions are overly cautious, allowing for further design optimization.
- Geochemical and water solubility concerns should be further investigated to demonstrate if the geomembrane liner cover is required for closure, or not.

- It is possible that a portion of the hydrometallurgical plant residue could be marketed, reducing the mass to be stored. Specifically, some of the gypsum might be suitable for wallboard production or agricultural use, and some or all of the LaCe might be marketed.
- Alternative storage options (e.g., 'wet' wastes stored behind dams) can be considered if future studies encounter serious issues with the filtered waste stacking concept.Water Management / Treatment.

25.4.4 Mine Site

The following risks have been identified related to water management and treatment for the project:

- Radioactivity of the sludge from the water treatment plant needs to be modelled, as radioactivity does not conserve in a water and load balance, which could affect sludge management disposal options and costs.
- The duration of water treatment after mine closure is unknown. A water quality model for closure is required, combining geochemical source terms with expected water volumes across the mine site, to evaluate long-term treatment requirements.
- Erosion and sediment transport during both construction, operations and closure phases could impact downstream water quality. A detailed sediment and erosion control plan is required to further upgrade currently proposed sediment control features and add more where appropriate.
- Opportunities related to water management and treatment include:
  - Parameters identified for water treatment are based on water and load balance modelling. Further dilution from natural runoff contributions into the CWP could prove that treatment for some parameters is not required. In addition, a mixing zone in Wichcika Creek could also reduce on-site water treatment requirements after consideration of in-stream dilution.
  - Advancing the geochemical source terms of pit wall and waste rock runoff and the predictive water quality model may provide justification for segregating different types of water to reduce water treatment requirements. If waste rock runoff, for example, was found to require sediment control only, the size of the water treatment plant could be reduced.
  - The assumed perpetual water treatment requirement could be removed or reduced if it can be demonstrated that seepage from the WSF meets water discharge criteria or further evaluation of more robust closure covers indicate that seepage from the WSF would be expected to meet water discharge criteria.

25.4.5 Hydrometallurgical Site

The same risks as described for the HWSF generally apply for site-wide water management and treatment risks at the Hydrometallurgical Site, given the lack of a physical location. Once a site is confirmed, various conditions including ground conditions, environmental, stakeholder, geological, topographical, etc. could present challenges not currently accounted for in the water management and treatment approach. However, the designs have been developed using conservative assumptions and there may be opportunities for optimization once site conditions are confirmed.

25.5 Metallurgy

25.5.1 Flotation Mineral Processing

The key outcome of the laboratory-scale metallurgical test work performed during 2021-2024 is the successful production of a 50% TREO concentrate with an average recovery of 82% TREO. This result was confirmed through batch testing, variability samples, and locked cycle tests on the master composite sample.

Further work is required to investigate the effects of lower ore grades and ore blending ratios based on the latest mine plan, particularly for years 9-15 of plant operation. Additionally, the impact of heating during conditioning and each flotation stage needs to be further examined to reduce energy requirements.

25.5.2 Hydrometallurgical Processing

The experimental testing and validation of the hydrometallurgical portions of the proposed flowsheet were conducted at the bench scale and at the pilot scale. The bench scale testing examined the major steps in the process, investigating both the effect of changing feed material properties, and the impacts of various operating parameters. The pilot-scale testing demonstrated the behaviour of the overall process circuit over extended operating durations, including the recirculation of the process liquor throughout the units. The pilot testing generated performance data which were used for the preparation of the full-scale process mass and energy balance, and equipment sizing and specifications. This experimental testing also produced mixed rare earth carbonate material, allowing the determination of the final product properties and characteristics, and its suitability for use by off-take partners (rare earth separation facilities).

In these tests, REE extractions of up to 95% from feed concentrate was achieved, with < 3% loss of REEs in impurity removal, and production of a high-purity MREC product (< 1wt% Ca, S).

While the main process flow was successfully tested in a continuous configuration, some components of the final process design have not been tested at the pilot scale. The following steps of the process require further validation by experimental trials:

- Solvent extraction and ammonia recovery: separation of La and Ce via solvent extraction, recovery of NH_3 from the REE precipitation liquor, and co-precipitation of La and Ce with the MgR solids.
- Ammonium bicarbonate RE precipitation: The use of ammonium bicarbonate as the RE precipitating agent within the context of the overall integrated circuit.

- Ammonium bicarbonate regeneration: The recycling of off-gas from the RE precipitation step, the ammonia recovery step, and natural gas combustion products to regenerate ammonium bicarbonate.
- Final product drying: The full thermogravimetric drying characteristics of the final product.
- Lime and Magnesia slaking with recycled liquor: The use of lime and magnesia slaked with recycled process liquor within the context of the overall integrated circuit.
- Water treatment: The treatment of liquid waste produced by the process to return it to safe disposal or recycle conditions.

Furthermore, some aspects of the process require further testing and investigation – in particular, the acid mixing and acid baking steps of the process experienced considerable complications related to corrosion and erosion of the mixing equipment, agglomeration of material, and material sticking to the internals of the equipment. Additionally, to assist in the design of vendor-supplied equipment, such as filters and solids-handling units, testing of real process material by equipment OEMs should be conducted, since the current designs are preliminary, based on data collected during the pilot tests, and experience from previous studies.

Corrosion studies are required to finalize the materials of construction selection throughout the plant, and further characterization is required for the various waste materials produced in the process. The magnesium removal solids, off-gas residue, and ammonia recovery solids, which will be impounded in the hydrometallurgical waste storage facility, require an assessment of structural and chemical stability. Similarly, the gas products (treated kiln off gas and ammonium bicarbonate column vent) should be further characterized to inform the design of treatment systems to ensure they meet release limits.

Risks and uncertainties include:

- The performance of the un-piloted process steps listed above may differ at commercial scale from observed bench-scale performance and design assumptions. The resulting inaccuracy in the design criteria, if not addressed by additional testing and piloting, can lead to inefficient process performance, and additional costs and schedule delays to redesign and adjust the system.
- Incomplete experimental data for the equipment design, as noted above for acid mixing/baking, solids handling, filtration, corrosivity, and others, can lead to increased maintenance costs, process upsets and downtime, and start-up delays if this data is not collected.

Opportunities include:

- Additional bench test, pilot testing, and equipment testing can enable further adjustments and optimizations to the operating parameters, such as reagent dosage rates and residence time requirements, thus potentially enabling reduced operating and equipment costs.

25.6 Recovery Methods

25.6.1 Concentrator Plant Design

A preliminary process flowsheet, mass and energy balance, and equipment design developed based on a combination of metallurgical test work, mine production plan and reference data from industry standards and previous experience. Where necessary, benchmarking has been used to support the design. This design offers a stable framework for process equipment, supporting in cost estimation and further system design, including control systems, start-up and shut-down operations, and safety and environmental studies. This preliminary design needs further refinement and testing to avoid risks such as delays, equipment failures, inadequate infrastructure, health, safety, and environmental issues. It is recommended in the next phase to perform a trade off study to review the most efficient way to maintain heat within the flotation circuit and maximize flotation recovery.

25.6.2 Hydrometallurgical Plant Design

A process flowsheet, mass and energy balance, and preliminary equipment designs were developed for the hydrometallurgical plant. The basis of this plant design was the results from the pilot and bench scale test work, supplemented by published chemical and physical property data, thermodynamic simulations, and input from equipment vendors. This plant design provided a preliminary steady-state design for the process equipment to support cost estimating, and the further design of the system, including control systems, unsteady state operation (start-up, shut-down, upsets, etc.), and hazard and environmental studies.

Risks and uncertainties include:

- This plant design is preliminary, and requires further refining and derisking, as informed by additional testing. The main process risk is that the process design inputs are insufficient or otherwise inappropriate for the sizing of equipment and specification of process inputs and outputs – leading to potential delays in commissioning and ramp-up, excessive equipment failures and maintenance, insufficient supply infrastructure, and Health/Safety/Environmental risks.
- Furthermore, additional development is required for the various auxiliary systems around the process, such as the off-gas, dust, and organic vapours management systems, the SX fire protection system, and the effluent treatment systems. Insufficient definition in the design criteria, if not addressed by additional design development prior to construction, can lead to inefficient process performance, safety and environmental risks, and additional costs and schedule delays to redesign and adjust the system.

Opportunities include:

- Multiple potential optimizations to the recovery flowsheet were identified, potentially enabling reduced capital and operating costs, and potential additional revenue. These opportunities are further discussed in Section 26.6.2.

25.7 Capital and Operating Cost

The capital cost estimation has been prepared to meet the AACE Class 3 standard. The total initial project capital cost is $1,441M, with the sustaining capital, closure costs, and post closure costs bringing the total to $2,007M.

The impact of tariffs and other recent trade related developments are not included in the study as they started after the PFS work was complete and the future impacts are unknown.

The operating costs for the concentrator facility were estimated at $50.6 million USD per year, corresponding to $28.1 per tonne of concentrator plant feed processed. The main opportunities for optimizing costs lie in investigating the heating requirements and exploring alternative heating methods, which should be conducted in future project phases.

The operating costs for the hydrometallurgical facility were estimated at $83.2 M USD per year, corresponding to $969 per tonne of dry concentrate feed processed, and $14.67 per kg of NdPrO equivalent in the final mixed RE carbonate. This operating cost intensity is on a similar order of magnitude to comparable RE processing facilities.

Overall, the operating costs are driven by the chemical/metallurgical characteristics of the process, and the location of the two project sites.

The main risks for the operating costs are:

- Changes in the costs of reagents (primarily H_2SO_4 and CaO, which represent 54% of the total hydrometallurgical OPEX), labour, and energy can have significant impacts on the operating costs.
- The acid supply cost estimate is based on an over-the-fence supply concept, but no agreement is currently in place with a supplier.

The main opportunities for optimizing the costs are:

- Addition pilot testing and engineering optimization of the process equipment, which will be conducted in future project phases, can potentially reduce reagent and energy consumptions
- Optimizing materials of construction informed by future corrosion testing
- modularization of process plant unit operations to reduce site installation costs and construction personnel requirements
- the optimization of the hydrometallurgical plant location, which should be conducted in the future project phases, can potentially reduce reagent and product transport costs.

25.8 Infrastructure

No “fatal flaws” have been identified with respect to on-site or offsite project infrastructure for both sites. The engineering and costing assumptions are at a PFS level and will require additional engineering and estimating precision to confirm the assumptions and develop the design, in the next phase, for a feasibility-level study.

The hydrometallurgical site infrastructure is designed based on having an existing, flat, partially developed industrial site available. A specific hydrometallurgical site must be determined before the feasibility study commences.

The main risks for the infrastructure (concentrator and hydrometallurgical) are:

- Obtaining timely access to a specific hydrometallurgical site and all required utilities such as power and gas.
- Change to geotechnical assumptions
- NORM management impact on systems such as ventilation (e.g. dust containing radionuclides)
- Accommodation camp costs are based on purchasing a good quality used camp. It is possible that one is not available at the required time.

The main opportunities for the infrastructure are:

- Optimizing hydrometallurgical site location
- Optimizing plant infrastructure designs, including modularization of utility subsystems.
- Cost sharing of FSR road maintenance with other users of the FSR.

25.9 Environmental and Social

The environmental and social aspects of the project are well understood for this stage and are assessed under established and well understood Environmental Assessment processes. Continued baseline studies, results of which will be used to identify receptors, assess potential project effects and used as inputs for detailed engineering, have been started or are planned.

The main risks are:

- Indigenous rightsholder oppositions
- Potentially impacted communities opposition to project
- Conflicting exercise of established or asserted rights in relation to the project or project area
- Mitigation and offsetting plan development is determined to not fully meet the objectives relating to endangered caribou recovery.
- Uncertainty about proven mitigation strategies to reduce or minimize adverse impacts associated with REE mining.

25.10 Economic Analysis

The economic analysis of the estimated cashflows for the project indicates the potential for an economic project across a broad range of input assumptions. The NPV calculated at an 8% discount rate is positive, and the Internal Rate of Return calculated for the project is within a favourable range.

The main risks are:

- Product prices and payabilities could be materially different than modeled. China can influence the market through government intervention which can drastically affect prices and the Wicheeda product payability is preliminary and not yet supported by any customer agreements or detailed marketing studies. There is also uncertainty around the future structure and function of the markets for mixed rare earth carbonate given ongoing global trade disputes. There is no definitive timeline for project development and it is likely that the structure of the market will continue to evolve during the next project development phase.
- The modeled production of TREO in MREC exceeds the nominal design TREO production capacity up to 9% in some of the early years of the project due to variations in the concentrator feed grade. This production excess is not expected to be an issue as there is sufficient plant design margin to account for this expected variation.

26. Recommendations

The following is recommended to advance the Wicheeda RE project to the next (feasibility) stage.

26.1 Exploration and Mineral Resource

Mr. Reid recommends the following actions and considerations for future development of the Wicheeda project, by improving the confidence in the MRE:

- Incorporate QAQC results from Dy, Tb and Gd in the analysis as the current SRMs are certified for these elements. An analysis for blank and duplicate results for these elements should be possible based on the available data.
- Continue with geotechnical and additional studies as recommended by SRK and other consultants.
- Continue with proposed drilling to improve increase confidence in areas inside the pit currently classified as Inferred to increase classification to Indicated.
- Estimate the concentration of deleterious elements such as thorium in the MRE.
- Develop variograms from the current and additional drillhole results to assist in the kriging process of economic elements.
- Drill hole core density measurements should be routinely taken to build-up a density database that can be utilized to assign a representative density values, to the different waste rock and mineralized lithologies.
- Conduct studies to optimize the block size to be better aligned with the drill spacing.

Defense Metals has a proposed budget for 20 holes totaling between 2,500 m to 3,000 m attempting to upgrade the most of the in pit inferred resources to indicated. Based on an all-inclusive cost of $500/m for exploration drilling costs, the budget would total between 1.5 to 2.0 M USD.

26.2 Mining

26.2.1 Pit Geotechnical

The SRK pit geotechnical QP provides these recommendations for future studies:

- Six additional oriented HQ3 diamond core drilling should be done to advance the database to the FS Level of study.
- The detailed oriented core holes should include industry standard rock logging (using the existing PFS Rock Logging Manual) and downhole geophysics for rock mass and hydrogeological characterization for each of the six holes.
- Field strength and density, with laboratory (UCS, triaxial and direct shear) testing of recovered samples should be done at regular (30 m) downhole intervals.

- Targeted hydrogeological testing (like air-lift and packer testing) for characterization and definition of hydraulic properties and installations (like standpipe piezometers, vibrating wire piezometers) for groundwater monitoring, should be completed in the detailed oriented core holes.
- The structural model needs to be improved to include all new exploration and detailed geotechnical data acquired for the post-PFS programs.
- The minimum catch-bench requirement of eight-metre width, regardless of bench height (within British Columbia), should be investigated relative to existing internationally accepted rock fall catch width models.

The proposed open pit rock mass geotechnical characterization and FS slope design and stability analysis cost would be between USD1,000,000 and USD1,100,000 – which includes core drilling, field lodging and transport, geological logging, and downhole geophysics contractor expenses.

26.2.2 Mining Studies

The SRK mining QP recommends that:

- Due to global pressure on REE market and changes in supply and demand, SRK recommends monitoring the REE prices frequently. Mineral reserves and related designs should be modified to reflect the changes in the market and operating costs.
- Upon further exploration and development of an updated mineral resource estimate (MRE) that converts more Inferred resource to Indicated, an updated mineral reserve estimation should be undertaken as part of a FS on the Wicheeda project.
- Consider the need for additional WSF capacity for an expanded open pit.
- In addition to exploration and MRE update, other areas of required data collection and analysis commensurate for a FS are described in this Section 26. The suggested budget for the mining portion of an FS, after all related data has been gathered, is USD2,000,000-2,500,000.

26.2.3 Waste Rock Storage Geotechnical

The following field-based recommendations are provided to advance the waste rock geotechnical characterization to an FS level:

- Four sonic drillholes and eight test pits located within the proposed footprint
- Specialist soil testing on in-situ and remolded sampled collected from the alluvium and till units to investigate strengths and potential undrained behaviour
- Detailed terrain and geohazard analyses, including the historical landform processes near the river valley.

Following the completion of the field investigation works, a geotechnical stability and design study will need to be completed. The design study will need to incorporate the following:

- Specific foundation preparation works
- Foundation behaviour under loading conditions representing the waste rock placement
- Potential geohazards and their interaction with the geotechnical design
- Run-out potential and setbacks to the river valley crest and infrastructure, including run-out mitigation strategies
- Design integration with water management infrastructure
- Local and overall stability conditions.

Importantly, alternative or additional waste storage options should be evaluated should there be a requirement to further setback the facility from the river valley crest, or if concentrated loading rates are temporary high resulting in the need to have flexibility in the rock placement plan.

Expected costs for these recommendations, including field programs is USD250,000-USD350,000.

26.3 Waste Management

26.3.1 Geochemistry

As the project advances, additional characterization of the waste rock, tailings, and hydrometallurgical waste streams will be required. This information will be used to support feasibility study, mine planning, and environmental permitting.

The following will require static testing for ML/ARD potential, and radionuclide activity where appropriate, with a subset of samples to undergo mineralogical and kinetic testing, depending on the static results:

- Approximately 100 to 300 samples of waste rock, from approximately 10 additional drill holes through the sedimentary waste rock, with the number of samples depending on variability in the sedimentary rocks recorded during drill core logging
- Ore composites developed for FS level metallurgical testing
- Flotation tailings representing various stages of the mine life
- All hydrometallurgical process wastes
- Combinations of flotation and hydrometallurgical wastes that are being considered for co-disposal for the FS
- Glacial till and/or other borrow sources planned to be excavated or used for process plant or embankment construction.

Expected costs for these recommendations (not including the drill program) are USD 550,000 (of which USD 250,000 are lab fees).

26.3.2 Tailings Management

The following investigations are recommended for tailings management:

- Perform a field investigation of the foundation conditions for the FTSF as well as other nearby infrastructure areas such as the overburden stockpile, filter plant, access roads and tailings pipeline corridor. Characterization of the proposed rock quarry is also required. Given the potential soft ground conditions and access challenges near the toe of the FTSF, alternative investigation methods such as geophysical techniques should be considered.
- Perform geotechnical characterization and behaviour (e.g., strength, compaction) testing on flotation tailings (with and without hydrometallurgical residue).
- Perform filtration testing to understand the variability in filter cake moisture content and filtration effort to achieve target moisture content.
- Conduct a formal tailings alternatives assessment with appropriate stakeholders. While a high level trade-off study between slurry and filtered tailings showed preference to filtered tailings to reduce risk, input from other stakeholders is required for permitting.

The same general comments apply for the hydrometallurgical waste facility once a preferred site is identified i.e., site investigation of the facility’s foundation conditions, and geotechnical testing of the wastes and any potential borrow materials. Expected costs for these recommendations for both the FTSF and Hydrometallurgical waste facility, including field programs, is USD 500,000-USD1,000,000.

26.4 Water Management / Treatment

The following studies are recommended to develop the water management designs for the project:

- Hydrometeorological baseline characterization to refine climatic and hydrologic inputs for infrastructure design and water quality modeling. The baseline program should include installation of local hydrometric stations for streamflow characterization and local meteorological stations for climatic characterization.
- Climate change assessment to evaluate effects to climate and associated parameters (i.e., streamflow and evaporation rates) and the associated impacts on infrastructure, mine operations and closure planning
- Baseline water quality monitoring program to characterize the baseline concentrations in the receiving environment
- Mixing zone assessment in the receiving environment of Wicheeda Creek to establish baseline conditions and available dilution capacity for project discharge
- Advance the geochemical characterization to establish expected and worst-case water quality estimates from mining and waste storage facility areas including the pit walls, waste rock, tailings and hydrometallurgical residues.

- Further develop the predictive water and load balance model to develop expected and worst-case water quality and water volumes in the Contact Water Pond under average, wet and dry conditions. Modeling should consider flow rates and water quality in the receiving environment and have the ability to simulate climate change conditions.
- Hydrogeological model to establish potential effects of the project on the regional groundwater system and magnitude of inflows to the open pit
- Dam breach assessment to establish the dam classifications for the CWP dams.

The expected costs for the noted recommendations as required for a FS is USD 300,000-USD 500,000.

26.5 Metallurgical Testing

A 1-year ramp-up period is targeted for this processing facility. To meet this ramp-up objective, the project should meet the following industry-accepted criteria: the use of mature technology, equipment with similar size and duty to earlier successful examples, thorough piloting of potentially risky unit operations, a well-understood resource, and an experienced corporate management and project team. (McNulty and Parameswaran, 2024) As such, future development efforts should focus on addressing these items with further pilot testing, engagement with equipment manufacturers, and testing of a variety of feed materials.

26.5.1 Concentrator Plant

- The concentrator bench and pilot testing of the proposed Wicheeda flowsheet was extensive and was in-line with expected industry standards for a pre-feasibility study; however, additional testing is required for a subsequent feasibility study. The following testing recommendations should enable the development of the concentrator flowsheet:
- Investigate the impact of lower ore grades and different ore blending ratios on the metallurgical process, especially during the later years of plant operation (Years 9-15). This will ensure consistent performance and recovery rates over the mine's lifespan.
- Examine the heating requirements during conditioning and each flotation stage to identify opportunities for reducing energy consumption. This includes exploring alternative heating methods and optimizing temperature settings to enhance efficiency.
- Additional solid/liquid separation tests with representative samples are needed to confirm the equipment sizing criteria for dewatering equipment
- Bulk material handling test work is required to determine the appropriate sizing criteria for bulk material handling equipment, such as bins, chutes, and conveyors.

26.5.2 *Hydrometallurgical Processing*

The hydrometallurgical bench and pilot testing of the proposed Wicheeda flowsheet was extensive and was in-line with expected industry standards for a pre-feasibility study; however, additional testing is required for a subsequent feasibility study. The following metallurgical testing recommendations should enable the development of the hydrometallurgical processing flowsheet:

- The list of limitations with the completed hydrometallurgical pilot testing and the distinctions between the tested flowsheet and the PFS process have been outlined in Section 13.4.2. To further validate and derisk the process additional experimental testing, focused around closing these untested gaps in the flowsheet, should be conducted. In particular, flowsheet steps which were added after the previous pilot testing, including SX for LaCe removal, REE precipitation with ammonium bicarbonate, and ammonium bicarbonate regeneration, should be tested. This pilot testing is recommended to demonstrate the operation of the integrated flowsheet, and de-risk the complex segments, such as SX separation and acid baking steps.
- While some hydrometallurgical testing has been conducted with variability samples, representing the different expected feed lithologies, additional variability testing is required to assess the expected bounds of operation.
- Testing and investigation around the acid mixing and acid baking step should be continued, potentially in conjunction with equipment manufacturers, to address the challenges around handling the acid bake calcine in the kiln, which tends to agglomerate and stick to the kiln walls. The optimization of this acid baking step is critical since, when operating at scale, frequent interruption of the kiln’s operation to clear out agglomerated and stuck material from the kiln would incur considerable operating costs.
- To support the more detailed design of process equipment, equipment-manufacturer-driven tests, such as filtration testing, drying tests, and material handling tests, should be conducted to quantify the various material properties and parameters that determine equipment design. Such testing can be conducted in conjunction with the general pilot testing that should be conducted.
- Corrosion testing will be required to enable a detailed materials of construction selection.
- The magnesium removal solids, off-gas residue, and ammonia recovery solids, which will be impounded in the hydrometallurgical waste storage facility, require an assessment of structural and chemical stability to validate the proposed disposal methods.
- The gas products (treated kiln off gas and ammonium bicarbonate column vent) should be further characterized to inform the design of treatment systems to ensure they meet release limits.
- Future pilot testing should aim to produce sufficient rare earth carbonate product to enable testing by potential off-take partners (rare earth separation facilities) to assess the compatibility of the material with their systems, and the accordance with their feed specifications and requirements.

The expected costs for the additional metallurgical testing program (for both the concentrator and the hydrometallurgical plant) is USD 5 million.

26.6 Recovery Methods

26.6.1 Concentrator Plant

The following potential optimizations were identified for investigation in the next phase:

- Optimizing the conditioning tank stage and sizing involves investigating the effects of solids density and heating requirements to enhance reagent mixing and slurry conditioning, thereby improving flotation performance.
- Revisit the equipment sizing based on the latest mine plan.
- Obtain formal budgetary bids for key process equipment such as the grinding circuit, flotation cells and dewatering equipment to improve the overall layout, operability and maintainability of equipment, while increasing the accuracy of the capital cost of the plant.
- Progress the dust handling design in the concentrator plant including the primary crusher circuit and stockpile.

26.6.2 Hydrometallurgical Processing

To advance the project, the level of engineering definition around the process should be advanced to the feasibility study level, including (but not limited to):

- Updating pre-feasibility process deliverables with data from additional pilot testing and input from equipment vendors and producing new feasibility-level engineering deliverables.
- Development of mass and energy balance models for various process conditions, such as unsteady-state operating conditions (start-up, shut-down, process upsets, etc.) and different feeds corresponding to different ore compositions in the mine plan.
- Conducting engineering studies, such as: capacity/availability analysis, hazard identification studies, corrosion studies, and process/equipment optimization investigations.

The plant design incorporates major vendor-designed sections, such as the off-gas treatment, and ammonium bicarbonate regeneration sections. As part of the engineering in the next phase, vendors/manufacturers should be engaged to develop feasibility-level designs and deliverables for these process sections.

The following potential optimizations were identified for future investigation prior to the Feasibility study:

- **Acid Mixing and Baking Equipment:** Pilot testing revealed some materials handling challenges with the Wicheeda ore during acid baking and mixing. A trade-off of different equipment types for these steps can address these challenges

- **Product Drying Optimization:** A trade-off of drying extents (no-drying vs anhydrous, etc.) different drying technologies (Paddle dryer vs Fluidized bed, Rotary Drum, etc.).
- **Additional SX separation steps:** the opportunities and challenges of including additional SX steps to further separate the target REEs and thus produce higher value products should be assessed
- **Lanthanum and Cerium Recovery:** the separate precipitation of LaCe to make a value-added product (vs co-precipitation with MgR).
- **Collection and Separate Disposal of UIX Resin / Regenerative UIX:** potential alternatives for investigation are the separate disposal of loaded UIX resin (avoiding the comingling of radioactive IX resin with PN residue), sale of loaded resin to uranium producing off-take partners, or regenerative UIX (reduces IX resin loss).
- **Optimization of MgR Solids Recycle to PN Step:** preliminary investigations into the recycle of $Mg(OH)_2$-containing MgR residue to the primary neutralisation step to reduce magnesia consumption were conducted, but not further pursued. This option can be revisited.

This update to the process design should be conducted alongside and in support of the development of the non-process components of the project (the expected costs are included in the costs of the metallurgical test work and feasibility study development).

26.7 Infrastructure

In the next project phase advance design, engineering, and cost estimation in parallel with the mining and processing feasibility study work.

A project execution plan will be developed to confirm how the project will developed within the envisioned schedule.

26.7.1 Mine Site Off-Site Infrastructure

Survey the mine site access route to confirm its suitability and define any required upgrades. Commence more detailed engineering design and cost estimation for the road upgrade and bridge repairs.

Commence electrical power line design, including early contractor involvement, route optimization, and BC Hydro connections requirements

26.7.2 Mine Site Infrastructure

In the next project phase develop all the required mine site infrastructure including but not limited to:

- Mine support infrastructure such as truck shop
- Accommodation complex
- Site facilities such as offices, warehouses, etc.
- Develop the process related infrastructure, such as site preparation and buildings, and including any requirements to address NORM management such as ventilation and dust control systems, vehicle wash facilities, etc.

- Develop any other NORM management requirements such as exposure controls for workers.
- The development of the infrastructure design and costs shall include vendor and contractor inputs to improve design and capital cost accuracy.

26.7.3 Hydrometallurgical Plant Location

Location trade-off study to consider the best location, considering Bear Lake, and other sites in BC or North America. The hydrometallurgical site location shall be finalized as an input into the next stage feasibility study.

26.7.4 Hydrometallurgical Plant Infrastructure Design

Once the location is finalized, the off-site and on-site infrastructure design and updated costs will be developed in the feasibility study, including obtaining budget quotations and early contractor input.

- Develop a supply agreement with a sulphuric acid provider for an over-the-fence supply.

The expected costs for the Feasibility Study work for processing and infrastructure, not including the mining, tailings and water treatment is USD 12 million. The expected cost for the various bridging studies to provide inputs to the Feasibility Study is USD 250,000.

26.8 Environmental and Social

Develop and implement an engagement plan to advance discussions and obtain and integrate the concerns and traditional knowledge of the rightsholders for the immediate and regional areas associated with the proposed project. This will build social capital for the project and meet the requirements of both provincial and federal environmental assessments.

All components of a comprehensive environmental and social baseline characterization program should be continued and/or initiated to support the project’s environmental assessment requirements. The development and implementation of these studies should include recommendations obtained during the early engagement sessions with the project’s Indigenous rightsholders. The participation of MLIB representatives in the field environmental baseline program be continued and expanded.

Development of a single document that captures all regulatory processes and requirements to support permitting discussions and ensure a timely and successful permitting process, is recommended for de-risking the permitting process.

The establishment of a local office that allows stakeholders and rights holders to have real time connection, could enhance opportunities project engagement, including education on aspects of the project, and help to mitigate social risks.

26.9 Economics

The positive results of the economic analysis over a reasonably wide range of assumptions support the Wicheeda project proceeding the next project development stage. While the capital and operating costs will become more accurate in the next study, the economic assessment should include quarterly scheduling for mining, processing and initial capital expenditures. The tax model should also be updated to reflect the detailed tax environment in BC and incorporate the existence of any opening balance for tax losses, prior expenditure and depreciation balances.

Technical marketing studies that focus on the expected product marketability and payability can support more accurate product value forecasting.

26.10 Summary

The various investigations described herein, as well as the budgets estimated by Hatch and SRK QPs, are listed in Table 26-1. Where budget ranges have been suggested, the range averages are in Table 26-1.

Table 26-1: Estimated Cost for Proposed Recommendations

Description	USD ($M)
Exploration drilling	1.5 - 2.0
Open pit rock mass geotechnical characterization and FS slope design and stability analysis	1.0 - 1.1
Feasibility study – mining component	2.0 - 2.5
Waste rock storage geotechnical studies	0.25 - 0.35
Waste material geochemistry studies	0.55
Tailings management studies	0.5 - 1.0
Water management / treatment studies	0.3 - 0.5
Concentrator and hydrometallurgical plant bench and pilot testing programs	5.0
Processing and infrastructure bridging studies	0.25
Feasibility study - processing and infrastructure	12.0
Subtotal	23.3-25.2
10% Contingency	2.3-2.5
TOTAL	**25.6-27.7**

Source: Hatch and SRK, 2025

27. References

1. APEX Geoscience. (2023). Technical report on the Wicheeda Property, British Columbia, Canada.
2. Arcadis. (2025). Memorandum to Defense Metals Corp. – Wicheeda Project radioactivity management.
3. B.C. Conservation Data Centre. (2024). *BC Species and Ecosystems Explorer.* B.C. Government, Victoria B.C. Available: http://a100.gov.bc.ca/pub/eswp/ (date accessed: March 2024)
4. British Columbia Ministry of Environment and Climate Change Strategy. (2016). Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators. Retrieved from https://www2.gov.bc.ca/assets/gov/environment/waste-management/industrial-waste/industrial-waste/water_air_baseline_monitoring.pdf
5. B.C. Ministry of Water, Land and Resource Stewardship, Caribou Recovery Program. (2024, June). *Predator reduction to support caribou recovery: 2023-2024 summary*. https://www2.gov.bc.ca/assets/gov/environment/plants-animals-and-ecosystems/wildlife-wildlife-habitat/caribou/predator_reduction_to_support_caribou_recovery_2023-2024.pdf
6. Bulmer, C., Schmidt, M. G., Heung, B., Scarpone, C., Zhang, J., Filatow, D., Finvers, M., Berch, S., & Smith, S. DOI: 10.1007/978-981-10-0415-5_24
7. Bulmer, et al. (2016). Improved soil mapping in British Columbia, Canada with legacy soil data and random forest. In *Digital soil mapping across paradigms, scales and boundaries*. B.C. Map Hub. (Updated Feb 7, 2024). ISBN: 978-981-10-0414-8.
8. Chi, R., & Tian, J. (2008). *Weathered crust elution-deposited rare earth ores*. Nova Science Publishers.
9. Government of British Columbia. (2023). Woodland Caribou Plan for the Hart Ranges Subpopulation, Phase One Draft. Victoria, BC.
10. Government of Canada. (2025). *First Nation profiles. Registered population, McLeod Lake, Number 618*. https://fnp-ppn.aadnc-aandc.gc.ca/fnp/Main/index.aspx?lang=eng
11. Hagen, John & Gantner, Nikolaus. (2020). Abundance and Trend of Arctic Grayling (Thymallus arcticus) in Index Sites of the Parsnip River Watershed, 1995-2019.
12. Health Canada. (2014). Canadian guidelines for the management of naturally occurring radioactive materials (NORM). Revised 2011, published April 2014.
13. McNulty, T., & Parameswaran, K. (2024). McNulty ramp-up curves: An update. In *Proceedings of the 63rd Conference of Metallurgists* (COM 2024).
14. Mining Association of British Columbia (MABC) & Mining Suppliers Association of BC (MSABC). (2024). *One province, one economy: Benefits of British Columbia's mining supply chain*. https://mining.bc.ca/wp-content/uploads/2024/01/OPOE-Report-2022-Web-Version-2.pdf

15. Orhanović, Z., et al. (1966). Precipitation and hydrolysis of metallic ions. III. Studies on the solubility of yttrium and some rare earth hydroxides. *Croatica Chemica Acta, 38*(4), 269-276.
16. Price, W. A. (1997). Draft guidelines and recommended methods for the prediction of metal leaching and acid rock drainage at mine sites in British Columbia. British Columbia Ministry of Employment and Investment, Energy and Minerals Division.
17. Province of British Columbia, Ministry of Energy, Mines and Low Carbon Innovation. (2024). *Consideration of climate change in mine reclamation planning* (Version 1.0). https://www2.gov.bc.ca/assets/gov/farming-natural-resources-and-industry/mineral-exploration-mining/documents/reclamation-and-closure/consideration_of_climate_change_in_reclamation_planning_guidance.pdf
18. Province of British Columbia, Ministry of Energy, Mines and Low Carbon Innovation. (2024). *Major mine reclamation liability cost estimate (RLCE) guidance*. http://www2.gov.bc.ca/assets/gov/farming-natural-resources-and-industry/mineral-exploration-mining/documents/reclamation-and-closure/reclamation-security/major_mine_rlce_guidance.pdf
19. Province of British Columbia, Ministry of Energy, Mines and Low Carbon Innovation. (2022). *Major mines reclamation security policy (Interim)*. https://www2.gov.bc.ca/assets/gov/farming-natural-resources-and-industry/mineral-exploration-mining/documents/reclamation-and-closure/major_mines_reclamation_security_policy_interim_v1_05apr2022.pdf
20. Regional District of Fraser-Fort George. (n.d.). *Electoral Area 'G' OCP Bylaw No. 2975*. https://www.rdffg.ca/building-and-development/zoning-and-land-use/official-community-plans
21. SGS Canada Inc. (2023). An investigation into beneficiation testwork on low grade and variability samples from the Wicheeda Rare Earth Deposit (Project No. 17173-06).
22. SGS Canada Inc. (2023). An investigation into the solids-liquid separation and rheology of four process samples from Pilot Plant Campaign No. 2 from Wicheeda Deposit Concentrate prepared for Defense Metals Corporation (Project No. 17173-05 – Draft – Final Report).
23. SGS Canada Inc. (2024). An investigation into pilot plant testing of the Defense Metals flowsheet (Project No. 17173-05 – Pilot Plant Report).
24. SGS Canada Inc. (2024). Bench scale development of a hydrometallurgical flowsheet for the extraction and recovery of rare earth elements from the Wicheeda deposit (Project No. 17173-04 – Final Report).
25. SGS Canada Inc. (2024). Summary of the resumed flotation tests BF-30 to BF-40 (Project No. 17173-06).
26. SRK Consulting. (2022). Independent preliminary economic assessment for the Wicheeda Rare Earth Element Project, British Columbia, Canada (Report No. 2CD031.000)

27. Statistics Canada. (2021). *Census profile, 2021 census of population: Bear Lake, designated place, British Columbia*. https://www12.statcan.gc.ca/census-recensement/2021/dp-pd/prof/details/page.cfm?Lang=E&SearchText=Bear%20Lake&DGUIDlist=2021A0006590114&GENDERlist=1,2,3&STATISTIClist=1&HEADERlist=0
28. Statistics Canada. (2021). *Census profile, 2021 census of population: McLeod Lake 1, Indian reserve, British Columbia*. https://www12.statcan.gc.ca/census-recensement/2021/dp-pd/prof/details/page.cfm?Lang=E&SearchText=McLeod%20Lake%201&DGUIDlist=2021A00055953802&GENDERlist=1,2,3&STATISTIClist=1&HEADERlist=0
29. Statistics Canada. (2016). *Census profile, 2016 census: Fraser-Fort George, regional district [Census division], British Columbia*. https://www12.statcan.gc.ca/census-recensement/2016/dp-pd/prof/details/page.cfm?Lang=E&Geo1=CSD&Code1=5953050&Geo2=PR&Code2=59&SearchText=Fraser-Fort%20George%20G&SearchType=Begins&SearchPR=01&B1=All&GeoLevel=PR&GeoCode=5953050&TABID=1&type=0
30. Verbaan, N., Liu, J., Johnson, M., Gunning, C., Aghamirian, M., MacNeil, T., Wallster, D., & Taylor, C. (2020*). Beneficiation and extraction of REE from Defense Metals' Wicheeda Deposit in BC.* SGS Canada Inc. & Defense Metals Corp.

28. Acronyms and Abbreviations

Acronym	Description
AAM	advanced air mobility
AB	acid baking
ABA	acid base accounting
ADC	analog to digital converter
AP	acid potential
APS	azimuth positioning system
ARD	acid rock drainage
ASL	Analytical Solutions Ltd.
ATV	acoustic televiewer
BCEHS	British Columbia Emergency Health Services
BCM	bank cubic meter
BCTS	BC timber sales
BCUC	BC Utilities Commission
BFA	bench-face angle
BWI	Bond work index
CAD	Canadian dollar
CAGR	compound annual growth rate
CAPEX	capital expense
CCA	capital cost allowance
CDA	Canadian Dam Association
CDN	CDN Resource Laboratories
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CMS	critical minerals strategy
CN	Canadian National railway
COS	change of support
COSEWIC	Committee on the Status of Wildlife in Canada
CPT	cone penetration test
CSS	close side setting
CWP	contact water pond
DC	dolomite carbonatite
DEHPA	di(2-ethylhexyl)phosphoric acid
DFO	Fisheries and Oceans Canada
DM	Defense Metals
DPD	detailed project description
EA	environmental assessment
EAA	environmental assessment act
EAC	environmental assessment certificate
EAO	environmental assessment office

Acronym	Description
EBITDA	earnings before interest, taxes, depreciation, and amortisation
EDF	environmental design flood
EGBC	Engineers and Geoscientists BC
EP	Ministry of Environment and Parks
EPCM	engineering, procurement, and construction management
ESSF	Engelmann spruce-subalpine fir
ESSFwk2	ESSF wet cool 2
EV	electric vehicle
FLNR	Forests Land and Natural Resource
FLNRORD	Ministry of Forests, Lands, Natural Resources Operations and Rural Development
FSR	Forest Service Road
FTSF	filtered tailings storage facility
G&A	general and administration
GGIRCA	Greenhouse Gas Reporting and Control Act
GPS	global positioning system
HCT	humidity cell testing
HDPE	high-density polyethylene
HIC	High-intensity conditioning
HREE	heavy rare earth element
HSRC	Health, Safety and Reclamation Code
HVAC	heating, ventilation, and air conditioning
HWSF	hydrometallurgical waste storage facility
IA	impact assessment
IAA	impact assessment act
IAAC	impact assessment agency of Canada
ICP-AES	inductively coupled plasma - atomic emission spectrometry
ICP-MS	inductively coupled plasma - mass spectrometry
IDF	inflow design flood
IEC	International Electrotechnical Commission
INT	intrusive related
IPD	initial project description
IRA	inter ramp angles
IRR	internal rate of return
IRS	intact rock strength
ISO	International Organization for Standardization
IUGS	International Union of Geological Sciences
LCT	locked cycle test
LHS	left hand side
LIM	limestone
LOI	loss on ignition
LOM	life of mine

Acronym	Description
LREE	light rare earth element
LREO	light rare earth oxide
LVA	locally varying anisotropy
masl	meters above sea level
MC	master composite
MCAF	mining cost adjustment factor
MCM	ministry of mining and critical minerals
MEA	mineral exploration agreement
MEB	mass and energy balance
MgR	magnesium removal
MLIB	McLeod Lake Indian Band
MRE	mineral resource estimate
MREC	mixed rare earth carbonate
NdPrO	Nd2O3 + Pr6O11
NMC	second master composite
NORM	naturally occurring radioactive materials
NP	neutralization potential
NPV	net present value
NRC	National Research Council Canada
NSR	net smelter return
OCP	office community plan
OPEX	operating expense
ORP	oxidation-reduction potential
OSA	overall slope angles
OVB	overburden
PAG	potentially acid generating
PDB	process design basis
PDC	process design criteria
PEA	preliminary economic analysis
PFS	pre-feasibility study
PGE	platinum group element
PLS	process leach solution
PMF	probable maximum flood
PN	primary neutralization
PP	pilot plant
QAQC	quality assurance / quality control

Acronym	Description
REE	rare earth element
REM	rare earth mineral
REO	rare earth oxide
RHS	right hand side
RMI	residual magnetic intensity
RMSP	Resource Modelling Solutions Platform
ROM	run of mill
RP	rare earth precipitation
RPEEE	reasonable prospects for eventual economic extraction
RQD	rock quality designation
RWI	rod mill work index
SAB	SAG / ball mill
SAG	semi-autogenous grinding
SARA	species at risk act
SBS	sub-boreal spruce
SBSvk	SBS very wet cool
SBSwk1	SBS wet cool 1
SCC	Standards Council of Canada
SED	sedimentary
SMC	SMC Test®
SMU	selective mining unit
SN	secondary neutralization
SRC	Saskatchewan research council
SRM	reference standard material
SX	solvent extraction
SYN	syenite
TCR	total core recovery
THG	total horizontal gradient
TIMA-X	tescan integrated mineral analyzer
TREE	total rare earth element
TREO	total rare earth oxide
UCS	uniaxial compressive strength
UDRL	unconditional derived release limits
UHNBC	University Hospital of Northern British Columbia
UIX	uranium ion exchange
US$M	1,000,000 USD
USD	United States dollar
USGS	United States geological survey

Acronym	Description
VAT	value-added tax
VWP	vibrating wire piezometer
WL	water leaching
WLB	water and load balance
WSF	waste storage facility
WSRHC	waste dump and stockpile stability rating and hazard classification
WTP	water treatment plant
XDC	xenolithic dolomite carbonatite
XE	xenolithic carbonatite
XRF	X-ray fluorescence
YXS	Prince George Airport

Source: SRK and Hatch, 2025

Attachment 1
Qualified Person Certificates

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9

+1 604 681 4196 office
+1 778 508 3584 fax

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS” prepared for Defense Metals Corporation (the “Issuer”) dated April 4, 2025, with an effective date of February 28, 2025 (the “Technical Report”).

I, Andy Thomas, do hereby certify that:

1. I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.
2. I am a graduate of the University of Adelaide (2004, B.Sc. Geology), University of Adelaide (2004, B.E. Civil & Environmental Engineering) and University of British Columbia in 2014, M.Eng Geotechnical Engineering.
3. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia (EGBC), license number: #44961.
4. I have visited the project site on 17-18 June 2022.
5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of National Instrument 43-101.
6. I, as a Qualified Person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
7. I am responsible for Sections 15.2, 25.3.1, 26.2.1and accept professional responsibility for these sections of this Technical Report.
8. I have had no prior involvement with the subject property.
9. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.

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

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Andy Thomas, P.Eng, M.Eng.

srk consulting

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9

+1 604 681 4196 office
+1 778 508 3584 fax

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS” prepared for Defense Metals Corporation (the “Issuer”) dated April 4, 2025, with an effective date of February 28, 2025 (the “Technical Report”).

I, Anoush Ebrahimi, do hereby certify that:

1. I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

2. I am a graduate of the University of Kerman (1991, B.Sc. Mining), University of P.T Tehran (1994, M.Sc. Mining) and University of British Columbia in 2004, Ph.D. Mining.

3. I am professional mining engineer who has worked for open pit mines, consulting companies, and universities since 1991. I designed multiple small and large open pit mines in Canada and abroad, including copper, gold, molybdenum, poly metal, coal, manganese, phosphate, bauxite, and oil sand projects. I worked on mining projects from conceptual studies through to construction. I specialize in strategic mine planning, mine evaluation, open pit optimization and design, reconciliation and dilution studies, production scheduling, and mine layout optimization. scheduling, mine layout optimization.

4. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia (EGBC), license number: 30166

5. I have personally inspected the subject project October 26, 2021.

6. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of National Instrument 43-101.

7. I, as a Qualified Person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.

8. I am a contributing author of the Technical Report and am responsible for Sections 1.7, 2.7, 15 (except for 15.2), 16, 25.3.2, 25.3.3, 26.2.2, 26.2.3 and accept professional responsibility for these sections of the Technical Report.

9. I was the QP (mining sections) for preliminary economic assessment technical report (PEA) published in January 2022.

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

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

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Anoush Ebrahimi P.Eng, Ph.D.

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9

+1 604 681 4196 office
+1 778 508 3584 fax

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS” prepared for Defense Metals Corporation (the “Issuer”) dated April 4, 2025, with an effective date of February 28, 2025 (the “Technical Report”).

I, Bob McCarthy, do hereby certify that:

1. I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

2. I am a graduate with a Bachelor of Applied Science (B.A.Sc.) degree in Engineering obtained from the University of British Columbia in 1984 and a Master of Business Administration (MBA) degree from Athabasca University in 2005. I have practiced my profession continuously since 1984.

3. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia (EGBC), license number: 136877

4. I have not visited the project site.

5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of National Instrument 43-101.

6. I, as a Qualified Person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.

7. I am responsible for Section(s) 1.1, 1.2, 2, 4.1, 4.2, 5, 21.1.1, and 21.2.1 and accept professional responsibility for these sections of this Technical Report.

8. I have had no prior involvement with the subject property.

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

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

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Robert McCarthy, P.Eng., MBA

srk consulting

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9

+1 604 681 4196 office
+1 778 508 3584 fax

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS" prepared for Defense Metals Corporation (the "Issuer") dated April 4, 2025, with an effective date of February 28, 2025 (the "Technical Report").

I, Christina James, do hereby certify that:

1. I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

2. I am a graduate of the University of British Columbia (2001, B.Sc. Combined Honours Oceanography and Physics) and University of British Columbia (2004, M.A.Sc. Civil Engineering Environmental Fluid Mechanics).

3. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia (EGBC), license number: 59919

4. I have not visited the project site.

5. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of National Instrument 43-101.

6. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.

7. I am responsible for Sections 1.10, 4.3, 4.4, 20, 25.10 and 26.8.

8. and accept professional responsibility for these sections of this technical report.

9. I have had no prior involvement with the subject property;

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

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

Signed and sealed this 2nd day of April 2025

[Original signed and sealed]

Christina James, M.A.Sc.

srk consulting

SRK Consulting (U.S.), Inc.
Suite 400
999 Seventeenth Street
Denver, CO 80202

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Douglas Reid, P. Eng. do hereby certify that:

1. I am Principal Consultant of SRK Consulting (U.S.), Inc., 999 Seventeenth Street, Suite 400, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS” with an Effective Date of 28 February 2025 (the “Technical Report”).
3. I graduated with a degree in a Bachelor of Science in Geological (Geophysics) Engineering from the University of Saskatchewan in 1986. I am a P. Eng. (23347) of the Engineers and Geoscientists British Columbia. I have worked as a Geological Engineer for a total of 35 years since my graduation from university. My relevant experience includes developing and reviewing resource models and mineral resource estimation for mineral projects in North and South America and Africa since 1994.
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 Wicheeda property on 31 October 2024 for 2 days.
6. I am responsible for Geology and Mineral Resources, Sections 6 through 12, Section 14, Section 23, and portions of Sections 1, 2, 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.

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Douglas Reid, P.Eng.

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9

+1 604 681 4196 office
+1 778 508 3584 fax

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS" prepared for Defense Metals Corporation (the "Issuer") dated April 4, 2025, with an effective date of February 28, 2025 (the "Technical Report").

I, Ignacio Garcia Schmidt, do hereby certify that:

1. I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

2. I am a graduate of the Catholic University Chile (2004, B.Eng.Sc. Civil), Catholic University Chile (2008, Civil Engineering Professional Degree) and Catholic University Chile (2008, M.Sc. Geotechnical Engineering).

3. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia (EGBC), license number: 210782

4. I have not visited the project site.

5. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of National Instrument 43-101.

6. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.

7. I am responsible for Sections 1.9.2, 1.9.5, 18.4 (except 18.4.4, 18.4.6 and 18.4.11), 18.6.8, 21.1.4, 21.2.6, 25.4.2, 25.4.3 and 26.3.2 and accept professional responsibility for these sections of this technical report.

8. I have had no prior involvement with the subject property;

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

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

Signed and sealed this 3rd day of April 2025

[Original signed and sealed]

Ignacio Garcia Schmidt, P.Eng.

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9

+1 604 681 4196 office
+1 778 508 3584 fax

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS” prepared for Defense Metals Corporation (the “Issuer”) dated April 4, 2025, with an effective date of February 28, 2025 (the “Technical Report”).

I, Kirsty Ketchum, do hereby certify that:

1. I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

2. I am a graduate of the University of Wales, with a Bachelor of Science (B.Sc.) in Geology in 1992 and Doctor of Philosophy (Ph.D, in geochemistry) from the University of Portsmouth in 1996.

3. I am a Professional Geoscientist registered with the Professional Engineers and Geoscientists of British Columbia (EGBC), license number: 58390.

4. I have visited the project site on June 24, 2023, and the project drill core stored in Prince George on June 20 to 23, 2023.

5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of National Instrument 43-101.

6. I, as a Qualified Person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.

7. I am responsible for Sections 18.3.2, 18.4.4, 18.6.9, 20.1.1, 20.2.2, 25.4.1, 26.3.1 and accept professional responsibility for these sections of this Technical Report.

8. I have had no prior involvement with the subject property.

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

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

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Kirsty Ketchum, P.Geo., Ph.D.

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9

+1 604 681 4196 office
+1 778 508 3584 fax

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS" prepared for Defense Metals Corporation (the "Issuer") dated April 4, 2025, with an effective date of February 28, 2025 (the "Technical Report").

I, Mauricio Herrera, do hereby certify that:

1. I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

2. I am a graduate of the Universidad del Norte (Colombia, 1997, BS Civil Engineering), Universidad de los Andes (Colombia, 1998, M.Sc. Water Resources Engineering) and University of Guelph (2009, Ph.D. Water Resources Engineering).

3. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia (EGBC), license number: 34942

4. I have not visited the project site.

5. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of National Instrument 43-101.

6. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.

7. I am responsible for Sections 1.9.3, 1.9.6, 18.4.6,18.5 (except 18.5.6), 21.1.5, 21.2.7, 25.4.4, 25.4.5, 26.4 and accept professional responsibility for these sections of this technical report.

8. I have had no prior involvement with the subject property;

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

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

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Mauricio Herrera, PhD, P.Eng

srk consulting

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9

+1 604 681 4196 office
+1 778 508 3584 fax

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS" prepared for Defense Metals Corporation (the "Issuer") dated April 4, 2025, with an effective date of February 28, 2025 (the "Technical Report").

I, Megan Miller, do hereby certify that:

1. I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

2. I am a graduate of the University of the Alberta (2009, B.Sc. in Civil Engineering) and University of British Columbia (2023, M.Eng. in Civil Engineering).

3. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia (EGBC), license number: 39873

4. I have not visited the project site.

5. I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of National Instrument 43-101.

6. I, as a Qualified Person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.

7. I am responsible for Section(s) 18.3.3, 18.6.11, 21.3 and accept professional responsibility for these sections of this Technical Report.

8. I have had no prior involvement with the subject property.

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

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

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Megan Miller, P.Eng., M.Eng.

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9

+1 604 681 4196 office
+1 778 508 3584 fax

vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS" prepared for Defense Metals Corporation (the "Issuer") dated April 4, 2025, with an effective date of February 28, 2025 (the "Technical Report").

I, Soren Jensen, do hereby certify that:

1. I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office located at 2600–320 Granville St., Vancouver, BC V6C 1S9, Canada.

2. I am a graduate of the University of British Columbia (2002, B.A.Sc. Chemical Engineering) and McGill University (2004, M.Eng. Chemical Engineering).

3. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia (EGBC), license number: 32876

4. I have not visited the project site.

5. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of National Instrument 43-101.

6. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.

7. I am responsible for Sections 1.9.3 (treatment), 18.5.6, 18.6.10, 21.1.5 (treatment), 21.1.6, 21.2.7 (treatment), 21.2.8, 25.4.4 (treatment), 25.4.5 (treatment), 26.4 (treatment) and accept professional responsibility for these sections of this technical report.

8. I have had no prior involvement with the subject property;

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

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

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Soren Jensen, P.Eng

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS” prepared for Defense Metals Corporation (the “Issuer”) dated April 4, 2025, with an effective date of February 28, 2025 (the “Technical Report”).

I, Jeffrey (Jeff) Adams, do hereby certify that:

1. I am a Consultant, Hydrometallurgy with the firm of Hatch Ltd. with an office located at 15 Allstate Parkway, Suite 300, Markham, ON, Canada.
2. I am a graduate of the University of Toronto, where I obtained a BASc in Chemical Engineering in 1995, an MASc in Chemical Engineering in 1999, and a PhD in Chemical Engineering in 2004.
3. I have practiced my current profession continuously since 2005. My principal experience is in the areas of process engineering plant design in hydrometallurgy. This includes experience in testwork program analysis, engineering design and cost estimation on numerous projects and commodities for various clients.
4. I am a Professional Engineer registered with PEO in the province of Ontario, member #100129529, APEGS in the province of Saskatchewan, member #67944, and EGBC in the province of British Columbia, member #60546;
5. I have not personally inspected the subject property. I inspected the Wicheeda pilot plant operation at SGS in Lakefield, Ontario on April 29th, 2023.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Sections 1.5.2, 1.8.2, 13.1, 13.2, 13.4, 17.2, 21.2.3, 25.6.2, 25.7.2, 26.5.2, 26.6.2 and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Jeff Adams, P. Eng

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS" prepared for Defense Metals Corporation (the "Issuer") dated April 4, 2025, with an effective date of February 28, 2025 (the "Technical Report").

I, Stefan Joseph Hlouschko, do hereby certify that:

1. I am a Senior Engagement Manager, Advisory, with the firm of Hatch Ltd. with an office located at 2800 Speakman Drive, Mississauga, Ontario, Canada.
2. I am a graduate of Queen's University, Canada, where, in 2006 I obtained a Bachelor of Science in Engineering (Engineering Physics) degree through the Faculty of Applied Science; and I am a graduate of Queen's University, Canada, where, in 2008 I obtained a Master of Science in Engineering (Mechanical Engineering) degree through the School of Graduate Studies;
3. I have practiced my current profession continuously since 2013. My principal experience is in the areas of project economic evaluation, market analysis, and due diligence for over ten years.
4. I am a Professional Engineer registered with PEO in the province of Ontario, member #100185785 and with EGBC in the province of British Columbia, member #62511
5. I have not personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Sections 1.12, 19, 22, 25.11, 26.9, and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Stefan Joseph Hlouschko, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS” prepared for Defense Metals Corporation (the “Issuer”) dated April 4, 2025, with an effective date of February 28, 2025 (the “Technical Report”).

I, Giuseppe (Joe) Paventi, do hereby certify that:

1. I am a Principal Engineer in Mineral Processing with the firm of Hatch Ltd. with an office located at 2800 Speakman Drive, Mississauga, On, Canada.
2. I am a graduate of the McGill University, where, in 1996 I obtained a Bachelor of Engineering – Metallurgical. In 2011, I obtained a Master of Project Management (MPM) from Penn State University.
3. I have practiced my current profession continuously since 1996. I have worked as a metallurgist for several mining companies and as a metallurgical consultant, study manager, project engineer and engineering manager involved with numerous projects from scoping study through to detailed engineering for base and precious metals. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia and the Professional Engineers of Ontario.
4. I have not personally inspected the subject property.
5. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
6. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
7. I am a contributing author of the Technical Report and am responsible for Sections 1.5.1, 1.8.1, 11.5,13.1, 13.2, 13.3, 17.1, 21.2.2, 25.6.1, 25.7.1, 26.5.1, 26.6.1 and accept professional responsibility for these sections of the Technical Report.
8. I have had no prior involvement with the subject property.
9. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Giuseppe (Joe) Paventi, P.Eng. MPM

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “N.I. 43-101 Technical Report Wicheeda Rare Earths Project PFS” prepared for Defense Metals Corporation (the “Issuer”) dated April 4, 2025, with an effective date of February 28, 2025 (the “Technical Report”).

I, Gerald (Gerry) Schwab, do hereby certify that:

1. I am a Project Manager with the firm of Hatch Ltd. with an office located at 1066 West Hastings Street, Suite 400, Vancouver, BC, Canada.
2. I am a graduate of the University of British Columbia, where, in 1984 I obtained a BASc in Mechanical Engineering through the Engineering Department.
3. I have practiced my current profession continuously since 1984. My principal experience is in the areas of process plant design, and project management.
4. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia;
5. I have not personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Sections 1.1, 1.9.1, 1.9.4, 1.9.7, 1.11, 1.13, 2 (except 2.7), 18.1, 18.2, 18.4.11, 18.6 (except 18.6.9-12), 21.1.2, 21.1.3, 21.2.4, 21.2.5, 24, 25.8, 25.9, 26.7, 26.10 and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 4th day of April 2025

[Original signed and sealed]

Gerry Schwab, P.Eng.