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PRELIMINARY ECONOMIC ASSESSMENT ON THE VIKEN ENERGY METALS PROJECT, JÄMTLAND COUNTY, SWEDEN

LATITUDE 63° 4' 35" N, LONGITUDE 14° 16' 48" E
WGS84 UTM 33V 463,600 m E, 6,994,300 m N

FOR
DISTRICT METALS CORP.

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FORWARD-LOOKING INFORMATION

This Technical Report contains certain statements that may be considered "forward-looking information" with respect to the Company within the meaning of applicable securities laws. In some cases, but not necessarily in all cases, forward-looking information can be identified by the use of forward-looking terminology such as "plans", "targets", "expects" or "does not expect", "is expected", "an opportunity exists", "is positioned", "estimates", "intends", "assumes", "anticipates" or "does not anticipate" or "believes", or variations of such words and phrases or statements that certain actions, events or results "may", "could", "would", "might", "will" or "will be taken", "occur" or "be achieved" and any similar expressions. In addition, any statements that refer to expectations, predictions, indications, projections or other characterizations of future events or circumstances contain forward-looking information. Statements containing forward-looking information are not historical facts but instead represent the Authors' expectations, estimates and projections regarding future events. Forward-looking information in this Technical Report relating to the Company include, among other things, statements relating to uranium and Alum Shale mining regulation in Sweden; the economic parameters of the PEA and the Viken Deposit; potential of the Viken Deposit; targets for further exploration; mineral resource estimates; the cost and timing of any development of the Viken Deposit; the proposed mine plan and mining methods; mining recoveries; processing method and rates; production rates; projected metallurgical recovery rates; infrastructure requirements; capital and operating cost estimates; the projected LOM and other expected attributes of the Viken Deposit; the NPV, IRR and payback period of capital; the uranium industry and uranium prices; government regulations and permitting; estimates of reclamation obligations and closure costs; requirements for additional capital; expectations with respect to project development and permitting, construction and operational processes; availability of services to be provided by third parties; future development methods and plans; and other activities, events or developments that are expected, anticipated or may occur in the future.

These statements and other forward-looking information are based on opinions, assumptions and estimates made by the Authors in light of its experience and perception of historical trends, current conditions and expected future developments, as well as other factors that the Authors' believe are appropriate and reasonable in the circumstances, as of the date of this Technical Report, including, without limitation, the reliability of exploration and drill results; reliability of data and the accuracy of publicly reported information regarding current, past and historical mines in the Bergslagen district and in respect of the Swedish properties; uranium and Alum Shale exploration and mining regulation in Sweden; the Company's ability to raise sufficient capital to fund planned exploration activities, maintain corporate capacity; stability in financial and capital markets; the Company's ability to complete its planned exploration programs; the absence of adverse conditions at mineral properties; no unforeseen operational delays; no material delays in obtaining necessary permits; the price of metals remaining at levels that render mineral properties economic.

Forward-looking information is necessarily based on a number of opinions, assumptions and estimates that, while considered reasonable by the Authors as of the date such statements are made, are subject to known and unknown risks, uncertainties, assumptions and other factors that may cause the actual results, level of activity, performance or achievements to be materially different from those expressed or implied by such forward-looking information, including but not limited to risks associated with the following: uranium exploration and mining regulation in Sweden; the reliability of historical data on District's properties; the Company's ability to raise sufficient capital to finance planned exploration; the Company's limited operating history; the Company's negative operating cash flow and dependence on third-party financing; the uncertainty of additional funding; the uncertainties associated with early stage exploration activities including general economic, market and business conditions, the regulatory process, failure to obtain necessary permits and approvals, technical issues, potential delays, unexpected events and management's capacity to execute and implement its future plans; the Company's ability to identify Mineral Resources and Mineral Reserves; the substantial expenditures required to establish Mineral Reserves through drilling and the estimation of Mineral Reserves or Mineral Resources; the uncertainty of estimates used to calculated mineralization figures; changes in governmental regulations; compliance with applicable laws and regulations; competition for future resource acquisitions and skilled industry personnel; reliance on key personnel; title matters; conflicts of interest; environmental laws and regulations and associated risks, including climate change legislation; land reclamation requirements; changes in government policies; volatility of the Company's share price; the unlikelihood that shareholders will receive dividends from the Company; potential future acquisitions and joint ventures; infrastructure risks; fluctuations in demand for, and prices of metals; fluctuations in foreign currency exchange rates; legal proceedings and the enforceability of judgments; going concern risk; risks related to the Company's information technology systems and cyber-security risks; and risk related to the outbreak of epidemics or pandemics or other health crises. These factors and assumptions are not intended to represent a complete list of the factors and assumptions that could affect the Company. These factors and assumptions, however, should be considered carefully. Although the Authors have attempted to identify factors that would cause actual actions, events or results to differ materially from those disclosed in the forward-looking information or information, there may be other factors that cause actions, events or results not to be as anticipated, estimated or intended. Also, many of such factors are beyond the control of the Authors. Accordingly, readers should not place undue reliance on forward-looking information. The forward-looking information is made as of the date of this Technical Report, and the Authors assume no obligation to publicly update or revise such forward-looking information, except as required by applicable securities laws.

CAUTIONARY NOTE REGARDING NON-GAAP FINANCIAL MEASURES

Alternative performance measures in this Technical Report such as "cash cost" and "AISC" are furnished to provide additional information. These non-GAAP performance measures are included in this Technical Report because these statistics are used as key performance measures used to monitor and assess performance of the Viken Deposit, and to plan and assess the overall effectiveness and efficiency of mining operations. These performance measures do not have a standard meaning within International Financial Reporting Standards ("IFRS") and, therefore, amounts presented may not be comparable to similar data presented by other mining companies. These performance measures should not be considered in isolation as a substitute for measures of performance in accordance with IFRS.

Cash Costs

Cash costs include site operating costs (mining, processing, site G&A), refinery costs and royalties, and excludes head office G&A and exploration expenses. While there is no standardized meaning of the measure across the industry, the Company believes that this measure is useful to external users in assessing operating performance.

All-In Sustaining Cost

Site level AISC includes cash costs and sustaining and expansion capital and excludes head office G&A and exploration expenses. The Company believes that this measure is useful to external users in assessing operating performance and the Company's ability to generate free cash flow from potential operations.

1.0 SUMMARY

1.1 INTRODUCTION

P&E Mining Consultants Inc. ("P&E") was retained by District Metals Corp. ("District Metals" or the "Company") to prepare an independent National Instrument 43-101 Preliminary Economic Assessment ("PEA" or "Report") on the Viken Project in Jämtland County, central Sweden. Vanadium (V), uranium (U), sulphate of potash (SOP), molybdenum (Mo), nickel (Ni), and zinc (Zn) mineralization occur in the Alum Shale, a distinctive black shale unit on the Viken Property.

Input to this PEA was also provided by METS Engineering Group Pty Ltd ("METS") for mineral processing and metallurgy, surface infrastructure, and product marketing and pricing. This PEA considers the mineralization at the Viken Energy Metals Property (the "Property" or the "Project") that is amenable to open pit mining and has an effective date of June 26, 2026. The Property is 100% owned by District Metals.

The Swedish moratorium on uranium exploration and mining was lifted effective January 1, 2026. On June 15, 2026, the Swedish Parliament voted to abolish a Municipal veto on uranium extraction that goes into legislative effect on July 15, 2026. A key aspect of the legislation is that the extraction and processing of uranium is now treated in a manner consistent with other metals and minerals within Sweden's overall permitting framework.

On June 11, 2026, the Swedish Government presented the terms of reference for a new inquiry that will investigate ways to enhance local and municipal influence for Alum Shale deposits. The inquiry mentions that it will include concerned parties from the private sector in their stakeholder dialogue. It will also provide an analysis of commercial consequences of any proposal made and will take into account Sweden's commitments according to the Critical Raw Materials Act ("CRMA").

The Viken Deposit is currently being reviewed by the Geological Survey of Sweden ("SGU") as a deposit of National Interest (Riksintresse), and the SGU's review is expected to be completed within Q3 2026.

The mine plan in this PEA accounts for approximately 3% of the current Mineral Resource and is intended to illustrate the first phase of a potential multi-phase Project. A PEA is preliminary in nature, and it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as Mineral Reserves, and there is no certainty that the PEA will be realized.

1.2 PROPERTY LOCATION AND MINERAL TENURE

The Viken Property is located in the central region of Sweden, approximately 520 km northwest of the Capital City of Stockholm and approximately 600 km south of the Arctic Circle. Viken is situated approximately 25 km southwest of the Town of Östersund in the Municipality of Berg and Åre, Jämtland County. It is centred at Latitude 63°04' North and Longitude 14°16' East (WGS84 UTM 33V 463,600 m East, 6,994,300 m North).

The Viken Property consists of eight contiguous licenses that cover a total area of 29,857 hectares ("ha") and are 100% owned by Bergslagen Metals AB, a wholly-owned Swedish subsidiary of District Metals. The eight licences are in good standing as of the effective date of this Report. The current Mineral Resource Estimate is covered by the Viken Property licenses Viken nr 1, Norra Leden, Norr Viken and Lill Viken.

1.3 ACCESS, CLIMATE, LOCAL RESOURCES, PHYSIOGRAPHY

The Property is easily accessible via highway E14 from Sundsvall, and then by local highways E45 and 321 that connect to the Property area through Svenstavik on the southern end of Lake Storsjön. The Östersund Region has a subarctic climate (Kopper: Dfc). Moderate temperatures in the summer months and sub-zero temperatures in the winter months are normal. The topography of the area is dominated by rounded hills and broad valleys at elevations in the range of 313 to 370 masl. Mixed coniferous and deciduous forests occur between open pasture lands. Mixed farming and small-scale forestry operations constitute the main land use. Most exploration activities can be conducted year-round.

Transportation systems are well developed in the area with daily air service and rail and truck freight services. Electrical power and modern communications are readily available in the area. The major mining centres in Sweden are Kiruna (~850 km), Malmberget and Aitik (~700 km), and the Skellefte District, approximately 450 km towards northeast. Likewise, in the Bergslagen District with Garpenberg (~400 km) and Zinkgruvan, approximately 600 km to the south, each of which are sources of mining-related services. The Viken Property has sufficient surface area for future exploration or mining and processing operations.

1.4 HISTORICAL OWNERSHIP, EXPLORATION AND DRILLING

The Geological Survey of Sweden ("SGU") carried out work on the Viken Alum Shales in the late 1970s that included completion of 19 drill holes within and in the vicinity of the Viken Deposit. In 2005, Continental Precious Minerals ("CPM") purchased mineral licenses that covered prospective Alum Shales and completed 26,293 m in 122 drill holes from 2006 to 2012 to delineate the Viken Deposit. CPM subsequently completed a Mineral Resource Estimate ("MRE") and PEA in 2010, and completed an updated MRE and PEA in 2014. No work has been completed on the Property since 2014. From 2023 to early 2025, District Metals consolidated 100% ownership of the Viken Property and acquired the 2% net smelter return ("NSR") royalty over the four mineral licenses covering the Viken Deposit. As a result, District Metals' 100% owned Viken Deposit is now only subject to a 0.2% state mineral fee payable to the landowner (0.15%) and Swedish State (0.05%).

1.5 GEOLOGY AND EXPLORATION

The Viken Deposit is hosted within the Cambrian age Viken Shale, which is regionally extensive in Sweden and referred to as the Alum Shale. The Alum Shale is enriched in metals such as vanadium, uranium, potassium, molybdenum, zinc and nickel. It is valued locally as bituminous shale with recoverable hydrocarbons.

The stratigraphy across the Viken Property consists of upper Middle and Upper Cambrian age Alum Shale occurring as both *in situ* and fault-detached blocks. The latter having greater potential for economic mineralization, due to imbrication of mineralized blocks. The Alum Shale is mostly exposed at surface and is underlain by Proterozoic granites and gneisses thrust eastward over Archean granitoid basement rocks. The thickness of the Alum Shale host rock has been tectonically increased from 20 to 30 m to approximately 180 m by thrusting and folding during the Silurian Age.

Mineralization of potential economic significance is hosted in Middle and Upper Cambrian Alum Shale, with the Upper Cambrian Age strata more enriched in vanadium and uranium than the Middle Cambrian. Vanadium is held in the lattice of a mica mineral named roscoelite. Uranium values are predominantly associated with sub-micrometric uraninite crystals. Nickel, molybdenum and zinc are present as sulphides.

The Viken Deposit is an extensive, very large and low-grade uranium polymetallic black shale hosted mineral deposit.

The database supplied by District Metals contains data for 152 drill holes totalling 28,667 m, of which 122 drill holes were within or adjacent to the Viken Deposit.

A high-resolution airborne MobileMT Survey was flown over the entire Viken Property by Expert Geophysics Limited in May-June 2025, with the aim to identify potential mineralized Alum Shale for follow-up confirmation by drilling. The MobileMT survey covered ~2,350-line km at 200 m spacing and was completed in two phases. Phase 1 captured the low resistivity signature of the mineralized Alum Shale that hosts the Viken Deposit, and Phase 2 covered the remainder of the Viken Property and identified additional low resistivity signatures resembling those of the Viken Deposit. High-priority target areas were selected by District Metals based on the shallow depth and thickness of the mineralized Alum Shale. The relatively flat-lying Alum Shale is rich in graphite and sulphides, making it low resistivity (high conductivity) and easily detectable using the airborne MobileMT system.

In all, nine target areas, labelled A to I, were identified outside of the Viken Deposit. Three of these target areas, A to C, show conductivity responses that are larger and stronger than those observed over the Viken Deposit itself. With these results, the Company plans to prioritize the targets for drill testing in 2026.

The Authors have recognized that the Viken Energy Metals Deposit contains Targets for Further Exploration with a potential range of 980 to 1,040 Mt at grade ranges of 140 to 180 ppm U_3O_8, 2,170 to 2,740 ppm V_2O_5 and 210 to 260 ppm Mo. These Targets for Further Exploration are based on the estimated strike length, depth and width of the mineralization, as supported by intermittently-spaced drill holes and observations of mineralized outcrops. The Targets for Further Exploration are located adjacent to the margins of the current Mineral Resource.

The potential quantities and grades of the Targets for Further Exploration are conceptual in nature. There has been insufficient work done by a Qualified Person to define these estimates as Mineral Resources. The Company is not treating these estimates as Mineral Resources, and readers should not place undue reliance on these estimates. Even with additional work, there is no certainty that these estimates will be classified as Mineral Resources. In addition, there is no certainty that these estimates will prove to be economically recoverable.

1.6 SAMPLE ANALYSES AND DATA VERIFICATION

It is the Author's opinion that sample preparation, security and analytical procedures for the Viken Project were adequate, and that the data are of suitable quality and satisfactory for use in the current Mineral Resource Estimate. Future drill core sampling at the Project should include the insertion of certified reference materials of appropriate grades, for all elements of interest, into the drill core sample stream on-site before shipping to the lab, the insertion and monitoring of field and coarse reject duplicates, and to umpire sample 5 to 10% of all future drill core samples at a reputable secondary laboratory.

Verification of the Viken Project data, used for the current Mineral Resource Estimate, was undertaken by the Authors, and included multiple site visits, due diligence sampling, drill hole collar and outcrop location verification, verification of drilling assay data, and assessment of the available QA/QC data for the drilling data. The Authors consider that there is excellent correlation between the U, V, K, Ni, Zn and Mo assay values in the Company's database and the 2013 independent verification samples collected by the Authors and analyzed at AGAT Laboratories. The Authors consider that sufficient verification of the Project data has been undertaken and that the supplied data are of suitable quality and satisfactory for use in the current Mineral Resource Estimate.

1.7 METALLURGICAL TESTING

Extensive metallurgical testwork programs were undertaken between 2007 and 2026 to assess the processing characteristics of mineralized shale from the Viken Deposit and support evaluation of potential processing routes. The testwork investigated mineral behaviour, metallurgical response, recovery performance, and process suitability across a range of development scenarios. As part of the PEA, METS completed an independent review of all available historical metallurgical investigations and used the results, together with comparable technical studies and benchmarking from similar projects, to establish process flowsheets, recovery assumptions and design criteria adopted for this Report.

The available testwork indicates that the Viken mineralized shale is a complex polymetallic material that is amenable to multiple processing approaches. Historical mineralogical and metallurgical investigations indicate that uranium and molybdenum show relatively favourable response under certain leaching conditions, while vanadium recovery is more technically challenging and is likely to require thermal activation or more aggressive process conditions such as a pug roast circuit to achieve commercially meaningful extraction. The mineralization includes potassium, nickel and zinc, providing potential opportunities for by-product recovery and additional value generation. However, the technical and commercial basis for recovery of these secondary products remains less developed and defined than for the primary uranium and vanadium processing pathways. Consequently, an extensive program of confirmatory, variability, and optimization testwork will be required to improve confidence in the proposed flowsheets, validate recovery assumptions, reduce technical uncertainty, and support future process design, engineering development, and cost estimation.

Overall metallurgical recoveries for the pug roast and leach option are estimated at 91% U_3O_8, 68% V_2O_5, 70% SOP, 32% Ni, 47% Zn and 52% Mo.

1.8 MINERAL RESOURCES

The Mineral Resource Estimate for uranium oxide, vanadium oxide, potassium oxide, molybdenum, nickel and zinc are summarized in Table 1.1. Additional elements reported (but not contributing to the NSR value) are copper, phosphorous pentoxide, cesium oxide, yttrium oxide, and lanthanum oxide. Many of these Mineral Resource elements are listed by the European Union ("EU") as Critical Raw Materials. In comparison to the 2014 Mineral Resource Estimate, the tonnage for Inferred Mineral Resources has increased to 4.333 billion tonnes and for Indicated Mineral Resources has increased to 456 million tonnes. There are no significant changes in grades from the 2014 Mineral Resource Estimate. Mineral Resources have been reported using an average internal (processing plus G&A) US$22/t NSR cut-off value.

The Mineral Resource Estimate is prepared in accordance with Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019), is effective as of June 26, 2026, and takes into account the results from a total of 122 drill holes completed by previous operators between 2006 and 2012 on the Viken Property. The spacing of the drill holes ranges from 30 to 380 m and averages approximately 300 m. The Alum Shale extends under the majority of the Viken Property and beyond its boundaries.

TABLE 1.1
PIT-CONSTRAINED MINERAL RESOURCE ESTIMATE FOR THE VIKEN DEPOSIT (1-7)

Classification	Tonnes (M)	U_3O_8 (ppm)	V_2O_5 (ppm)	Mo (ppm)	Ni (ppm)	Cu (ppm)	Zn (ppm)	P_2O_5 (ppm)	Ce_2O_3 (ppm)	Y_2O_3 (ppm)	La_2O_3 (ppm)	K_2O (%)
Indicated	456	175	2,836	257	330	113	411	2,461	88	492	7	3.84
		(Mlb)						(Mt)				
Indicated	Contained Metal	176	2,851	258	332	114	413	1.12	0.04	0.22	0.00	17.53
Classification	**Tonnes (M)**	**U_3O_8 (ppm)**	**V_2O_5 (ppm)**	**Mo (ppm)**	**Ni (ppm)**	**Cu (ppm)**	**Zn (ppm)**	**P_2O_5 (ppm)**	**Ce_2O_3 (ppm)**	**Y_2O_3 (ppm)**	**La_2O_3 (ppm)**	**K_2O (%)**
Inferred	4,333	161	2,543	240	321	118	417	2,541	88	528	7	3.70
		(Mlb)						(Mt)				
Inferred	Contained Metal	1,538	24,295	2,293	3,067	1,127	3,984	11.01	0.38	2.29	0.03	160.27

Notes:

1. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2. *The Inferred Mineral Resource in this Estimate has a lower level of confidence that that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
3. *The Mineral Resources in this Report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
4. *The Mineral Resource Estimate was based on March 2025 approximate Consensus Economics forecast US$ metal prices of $72/lb U_3O_8, $5/lb V_2O_5, $17/lb Mo, $8.50/lb Ni, $4.25/lb Cu and $1.30/lb Zn with process recoveries of 80%, 80%, 70%, 70%, 50% and 75%, respectively.*
5. *Overburden, waste and mineralized US$ mining costs per tonne mined were respectively $2.00, $2.50 and $3.00.*
6. *Processing and G&A US$ costs per tonne processed were respectively $20 and $2.*
7. *Constraining pit shell slopes were 45°.*

1.9 MINING METHODS

The proposed mine plan for the Viken Deposit comprises a conventional open pit operation developed as a single pit. The pit extends from an exit elevation of approximately 345 masl to a lowest elevation of 200 masl, with an average depth of 145 m. Mining is planned over one pre-production year followed by 13 years of active mining and processing, for a total duration of 14 years. Pre-production includes limited pre-stripping of 4,500 tonnes of waste material and construction of mine roads and site preparation.

The operation is planned to be owner operated. Mining will employ Surface Excavation Machines capable of cutting 0.3 m into the Alum Shale, supported by conventional loading and haulage equipment. No drilling or blasting is planned. Material movement will use 11.5 m^3 wheel loaders loading 90 tonne off-highway haul trucks, supported by dozers, graders, excavators, dewatering pumps, service equipment, and dry stack tailings handling equipment. Mining and support activities are planned on a continuous 24 hour per day, seven day per week basis.

Life-of-mine production totals 127.36 Mt of process plant feed and 23.49 Mt of waste rock for a strip ratio of 0.18:1. Average feed grades are 197 ppm U_3O_8, 3,714 ppm V_2O_5, 293 ppm Mo, 395 ppm Ni, 479 ppm Zn, and 4.12% K_2O, with an overall NSR of US$118/t. Mining dilution and mining losses are assumed to be zero since the potentially mineable blocks have no edge contact with the large Mineral Resource wireframe. Process throughput targets 10 Mtpa or 28 ktpd, with ramp-up to 75% capacity in Year 1 and full capacity from Year 2 onward. Process plant feed includes approximately 77% Indicated Mineral Resources and 23% Inferred Mineral Resources. The mine plan and PEA includes Inferred Mineral Resources that are too speculative geologically to have the economic considerations applied to them to be classified as Mineral Reserves, and there is no certainty that the PEA will be realized. Waste rock will be stored both externally with dry stack tailings and in-pit during later years of operation.

1.10 RECOVERY METHODS

Four different recovery methods were studied and economic analysis determined that the pug roast leach method with an on-site acid production plant resulted in the highest net present value ("NPV") and internal rate of return ("IRR") for the Viken Deposit. The process feed is crushed, grinded and floated before being mixed with concentrated sulphuric acid in a pug mill and thermally roasted to convert vanadium, uranium, potassium and associated metals into water soluble sulphate species. This configuration targets high leach recoveries and allows a more liberal flotation regime, which can enhance recovery of by products such as mixed metals precipitate ("MMP") and sulphate of potash ("SOP"). Approximately 35% of the total V_2O_5 production will be directed to vanadium electrolyte production, with the remaining V_2O_5 allocated to FeV production.

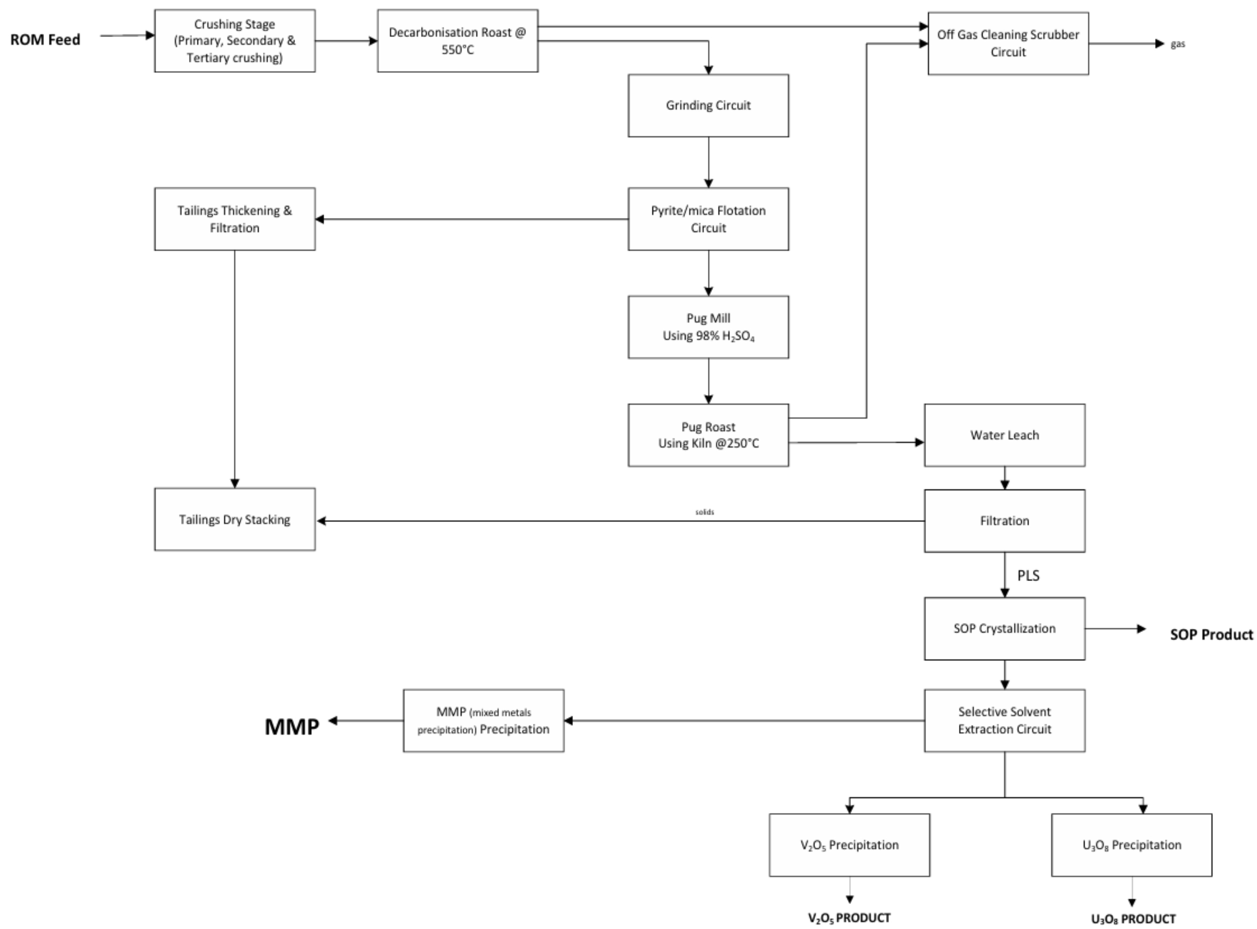
The beneficiation and purification techniques are:

- Crushing
- Grinding
- Flotation
- Pug Roasting

- SOP Crystallization
- Ion Exchange
- Solvent Extraction
- Uranium and Vanadium Precipitation
- Mixed Metals Precipitation
- Residue Handling.

A description of the pug roast flowsheet is provided in Figure 1.1.

FIGURE 1.1 PUG ROAST BLOCK FLOW DIAGRAM



Source: *METS (2026)*

1.11 PROJECT INFRASTRUCTURE

Project infrastructure for the Viken Project is based on existing regional services supplemented by site-specific facilities. Access will be provided via Highway E14 with connections through E45 and Road 321 via Svenstavik, supported by a new approximately 600 m two-way gravel access road and approximately 650 m of internal site roads connecting the open pit, process facilities, waste storage areas, workshops, and administration facilities. Regional rail infrastructure through Östersund provides freight access to Sundsvall and Trondheim ports, which are intended to support inbound reagents and outbound products. Air access is available through Östersund Airport located approximately 42 km by road from site.

Primary electrical supply is planned from the regional grid through a new overhead distribution line connected to the existing low voltage network located within approximately 3 km of site, supported by a site substation, transformers, distribution lines, motor control centres, and backup power systems. Water supply is proposed from Lake Storsjön or Abassen Creek with recycling of recovered process water and pit dewatering water. LNG delivered by road tanker and stored in cryogenic tanks is planned for process and utility requirements.

Site facilities will include administration buildings, workshops, warehouses, assay laboratory, maintenance facilities, fuel storage and dispensing, water treatment facilities, communications systems, and support services. The process plant will include crushing, grinding, flotation, pug roasting, ion exchange, solvent extraction, product packaging, and storage to produce vanadium products, U_3O_8, sulphate of potassium and a mixed metal concentrate. Waste management includes, septic systems, hazardous waste controls, and a dry stacked tailings and waste rock storage facility.

1.12 MARKET STUDIES AND CONTRACTS

The objective of the marketing assessment is to identify the target markets and likely customers for the proposed Viken Project products, develop preliminary long-term price assumptions for use in the PEA financial model, and assess the market context for product sales with reference to prevailing and forecast supply and demand conditions. The assessment also considers key opportunities and constraints associated with product specification, marketability, logistics and pricing for each proposed product stream.

The Viken Project is primarily focused on vanadium products, with vanadium electrolyte and ferrovanadium (FeV80) identified as the key saleable products. Commercial-grade V_2O_5 flake is treated as an intermediate product and may also be sold directly, subject to market conditions and customer requirements. In addition to vanadium products, the Project is expected to produce the following secondary product streams: Uranium oxide (U_3O_8), Sulphate of Potash (“SOP”) and Mixed Metal Sulphides (“MMS”) containing nickel, zinc and molybdenum.

For the purposes of this PEA, preliminary long-term price assumptions, expressed in United States dollars are presented in Table 1.2.

TABLE 1.2
COMMODITY PRICES (US$)

Commodity	Unit	Price
Ferrovanadium (FeV80)	kg	38
Vanadium Electrolyte	L	9
Uranium Oxide	lb	85
Sulphate of Potash (for 50% K_2O SOP granules)	t	650
Nickel	lb	7.71

TABLE 1.2 COMMODITY PRICES (US$)		
Commodity	**Unit**	**Price**
Zinc	lb	1.45
Molybdenum	lb	27.22

There are currently no material contracts in place for the Viken Project.

1.13 ENVIRONMENTAL STUDIES, PERMITS AND SOCIAL OR COMMUNITY IMPACTS

Environmental and social evaluation for the Viken Project is at a conceptual stage and additional baseline studies are required to support future assessment and permitting. Historical and recent public concerns have focused primarily on potential impacts to water quality in Lake Storsjön, land disturbance, and uranium extraction activities. Anticipated environmental management measures include control of dust from crushing, management of gaseous emissions from calcining and roasting, recycling of treated water where possible, neutralization of barren process solutions with lime, and co-disposal of gypsum and metal hydroxide water treatment by-products in a dedicated chemical solids storage facility. Flotation tailings and process residues are proposed to be dry stacked and progressively covered using an impervious multi-zone soil cover.

Project development will require permitting under the Swedish Minerals Act and Swedish Environmental Code, including preparation of an Environmental Impact Assessment, public consultation, municipal approvals, and potential consideration under the Nuclear Activities Act and radiation protection legislation. Hydrogeological characterization and groundwater studies are identified as key permitting requirements. Community engagement has included information initiatives and landowner meetings. Closure planning contemplates progressive reclamation, pit flooding with water treatment, engineered waste storage, long-term monitoring, and inclusion of closure surety requirements as part of the permitting and licensing process.

1.14 CAPITAL AND OPERATING COSTS

All costs are presented in Q2 2026 US dollars. No provision has been included in the cost estimates to offset future escalation. A contingency of approximately 20% has been added to all capital costs (“CAPEX”). No contingency is added to operating costs (“OPEX”).

The total initial CAPEX of the Viken Project is estimated at $876M. Sustaining capital costs incurred during the 13 production years and one year of rehabilitation are estimated to total $158M. Total OPEX over the life-of-mine (“LOM”) is estimated at $5.9B, averaging $46.58/t processed.

1.14.1 Capital Costs

The initial CAPEX of the Viken Project includes engineering, procurement, construction and start-up of an open-pit mine, a process plant and acid plant facility capable of handling 10 Mtpa, and associated ancillary surface facilities such as offices and dry stack tailings storage. Initial CAPEX is estimated at $876M and is presented in Table 1.3.

TABLE 1.3 CAPITAL COST SUMMARY			
Capital Costs (US$ Millions)	**Initial**	**Sustaining**	**Total**
Mine	60	44	105
Process Plant & Infrastructure & Indirect Costs	657	79	736
Rehabilitation	0	8	8
Contingency	159	27	185
Total Capital Costs[1]	876	158	1,034

***Note:** [1] Totals may not sum due to rounding.*

Sustaining CAPEX during the production years is estimated at $158M primarily for equipment replacement at both the mining operation and process plant. Reclamation costs for the Project at the end of the LOM are estimated at $8M.

1.14.2 Operating Costs

The OPEX estimate includes mining, processing, waste management, and G&A services. The LOM totals and average operating costs for the Viken Project are presented in Table 1.4. The mining cost per tonne processed is based on a unit cost of $2.95/t mined at a strip ratio during production years of 0.18:1.

TABLE 1.4 UNIT OPERATING COSTS		
Item	**Unit Cost ($/t processed)**	**LOM Total ($M)**
Open Pit Mining	3.50	445
Processing	37.43	4,767
Dry Stack Tailings	1.81	231
General & Administrative	1.33	169
SOP Freight	2.51	320
Total Operating Cost[1]	**46.58**	**5,932**

***Note:** [1] Totals may not sum due to rounding.*

The electrical power cost is estimated at $0.08/kWh for power from the national grid. The diesel cost delivered to site was based on a reliable diesel price website at $2.58/L.

An annual Swedish state mineral fee in the amount of 0.15% of gross value of concession minerals is due to the landowners within the concession area and 0.05% is due to the State. Over the LOM the total fee to the landowners is estimated at $21.9M and the State fee is estimated at $7.3M for a total of $29.2M.

Cash costs over the LOM are estimated to average negative US$121/lb U_3O_8 net of by-product credits. All-In Sustaining Costs ("AISC") over the LOM are estimated to average negative US$118/lb U_3O_8 net of by-product credits.

1.15 ECONOMIC ANALYSIS

Cautionary Statement - The reader is advised that the PEA summarized in this Technical Report is preliminary in nature and is intended to provide only an initial, high-level review of the Project potential and design options. The PEA mine plan and economic model include numerous assumptions and the use of Inferred Mineral Resources. Inferred Mineral Resources are considered to be too speculative to have the economic considerations applied to them that would enable them to be classified as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

The Viken Project economic evaluation conclusions are summarized in Table 1.5. At base case metal/commodity prices of US$38/kg FeV80, US$9/L vanadium electrolyte, US$85/lb U_3O_8, US$650/t SOP, US$7.71/lb Ni, US$1.45/lb Zn and US$27.22/lb Mo the Project has an estimated US$2.88B after-tax net present value ("NPV") at an 8% discount rate ("NPV8%"), and an after-tax internal rate of return ("IRR") of 46%. The after-tax payback period is estimated to be 2.1 years from the start of production.

TABLE 1.5
ECONOMIC EVALUATION SUMMARY

Item	Pre-Tax	After-Tax
NPV0% ($M)	7,592	6,014
NPV5% ($M)	4,796	3,765
NPV8% ($M)	3,698	2,881
NPV10% ($M)	3,126	2,421
IRR (%) (base case)	52.6	45.9
Payback period (years) (base case)	1.9	2.1

Table 1.6 provides further details on the Project LOM cash flow.

TABLE 1.6 **PROJECT CASH FLOW SUMMARY**					
Parameter	**Unit**	**Value**	**Parameter**	**Unit**	**Value**
Process Plant Feed	Mt	127.4	Net Revenue	US$M	14,587
Waste Rock Mined	Mt	23.4	Initial Capital	US$M	876
Strip Ratio	waste:feed	0.18:1	Sustaining Capital	US$M	158
V_2O_5 Grade	ppm	3,714.53	Mining Costs	$/t Feed	3.50
U_3O_8 Grade	ppm	196.57	Processing Costs	$/t Feed	37.43
K_2O Grade	%	4.12	G&A Costs	$/t Feed	1.33
Ni Grade	ppm	394.71	Dry Stack Tailings Costs	$/t Feed	1.81
Zn Grade	ppm	479.22	SOP Freight	$/t Feed	2.51
Mo Grade	ppm	292.87	Operating Costs	$/t Feed	46.58
Payable FeV	Mkg	76.2	Operating Cash Cost Net By-Products	US$/lb U_3O_8	(-) $121
Payable V Electrolyte	ML	474.4	All-in Sustaining Cost Net By-Products	US$/lb U_3O_8	(-) $118
Payable U_3O_8	Mlb	41.8	After-Tax NPV (8% discount)	US$M	2,881
Payable SOP	Mt	3.2	Pre-Tax NPV (8% discount)	US$M	3,698
Payable Ni	Mlb	20.9	After-Tax IRR	%	45.9
Payable Zn	Mlb	27.0	Pre-Tax IRR	%	52.6
Payable Mo	Mlb	58.3	After-Tax Payback Period	years	2.1

Table 1.7 presents a metal/commodity price sensitivity analysis and Table 1.8 presents an OPEX and CAPEX sensitivity analysis.

TABLE 1.7 **METAL PRICE SENSITIVITY NPV, IRR AND PAYBACK**							
Sensitivity	**-30%**	**-20%**	**-10%**	**Base Case**	**+10%**	**+20%**	**+30%**
FeV Price (US$/kg)	27	30	34	**38**	42	46	49
V Electrolyte Price (US$/L)	6	7	8	**9**	10	11	12
U_3O_8 Price (US$/lb)	60	68	77	**85**	94	102	111
SOP Price (US$/t)	455	520	585	**650**	715	780	845
After-Tax NPV8% (US$M)	1,023	1,642	2,262	**2,881**	3,498	4,115	4,732
After-Tax IRR (%)	24.3	32.2	39.3	**45.9**	52.0	57.8	63.3
After-Tax Payback (years)	3.7	2.9	2.5	**2.1**	1.9	1.8	1.6

Table 1.8 **Capital and Operating Cost Sensitivity of NPV and IRR**							
Sensitivity	**-30%**	**-20%**	**-10%**	**Base Case**	**+10%**	**+20%**	**+30%**
Operating Costs – NPV8% (US$M)	3,639	3,386	3,134	**2,881**	2,627	2,373	2,118
Operating Costs – IRR (%)	53.8	51.3	48.6	**45.9**	43.1	40.2	37.3
Capital Costs – NPV8% (US$M)	3,108	3,033	2,957	**2,881**	2,805	2,728	2,651
Capital Costs – IRR (%)	60.0	54.4	49.8	**45.9**	42.6	39.7	37.2

The after-tax base case NPV's and IRR's are most sensitive to metal/commodity prices followed by operating costs and capital costs.

The FeV:V Electrolyte ratio base case has been set at 35:65 to reflect estimated market demand over the long term. If the amount of vanadium electrolyte was to be reduced, the Project economics could potentially adjust as per the sensitivity analysis set out in Table 1.9.

Table 1.9 **Vanadium Product Sensitivity**			
Sensitivity Item	**FeV:V Electrolyte**		
	35:65 base case	**80:20**	**90:10**
FeV (LOM Mkg)	**76.2**	93.8	105.6
V Electrolyte (LOM ML)	**474.4**	271.1	135.6
V Electrolyte (Average ML/year)	**38.8**	22.2	11.1
After-Tax NPV8% (US$M)	**2,881**	2,390	2,062
After-Tax IRR (%)	**45.9**	41.0	37.4
After-Tax Payback (years)	**2.1**	2.4	2.5

1.16 ADJACENT PROPERTIES

Aura Energy Limited (ASX: AEE, AIM: AURA) has a comparable, uranium polymetallic deposit, named Häggån, immediately west adjacent to the Viken Property (Aura, 2023). Häggån is a preliminary level project.

1.17 RISKS AND OPPORTUNITIES

The Project risk assessment identified key technical, environmental, commercial, regulatory, and execution uncertainties associated with the proposed development and Pug Roasting process route.

Principal risks include variability in mineralization characteristics and metallurgical performance, limited validation of recoveries and reagent consumption assumptions, open pit geotechnical uncertainty, dry stacked tailings performance, environmental compliance, permitting timelines, infrastructure execution, occupational health and safety, and social acceptance associated with uranium processing and water management. Financial risks reflect preliminary cost estimates with an estimated accuracy of ±50%, together with exposure to inflation, exchange rates, and commodity market conditions.

Opportunities include the scale of the multi-commodity Mineral Resource and associated long mine life potential, diversified revenue sources from uranium, vanadium, sulphate of potash, nickel, zinc, and molybdenum products, potential Mineral Resource growth, process optimization and recovery improvements, modular development approaches, and environmental performance enhancements through dry stack tailings, water recycling, and progressive rehabilitation.

1.18 CONCLUSIONS

The open pit mine plan is scheduled over a production period of 13 years. Total material mined is 150.8 Mt, comprising 127.4 Mt of mineralized material that averages 197 ppm U_3O_8, 3,714 ppm V_2O_5, 293 ppm Mo, 395 ppm Ni, 479 ppm Zn, and 4.12% K_2O, with an overall NSR of $118/t, and 23.4 Mt of waste material, resulting in a strip ratio of 0.18:1. Payable metal over the LOM is estimated at 76.2 Mkg FeV, 474.4 ML vanadium electrolyte, 41.8 Mlb U_3O_8, 20.9 Mlb Ni, 27.0 Mlb Zn, 58.3 Mlb Mo and 3.2 Mt SOP. Approximately 77% of the Mineral Resource is classified as Indicated with 23% as Inferred.

Based on the work undertaken to date, as summarized in this Technical Report, and the conclusions listed in Section 25, the base case operating plan is a viable and attractive development opportunity. Overall Project risks are perceived as moderate.

Current base case economic analysis indicates that the Project is forecast to be viable. It is recommended that District Metals continue the Project development plans, and implement the recommendations set out in Section 26 of this Technical Report.

1.19 RECOMMENDATIONS

The Authors conclude that the Viken Project contains a very large vanadium, uranium, potassium and polymetallic Mineral Resource that merits further exploration and evaluation.

The results for this PEA, using the base case assumptions, indicate that the Viken Deposit has both technical and financial merit. The Project's next steps are recommended to include:

- Drill testing of airborne MobileMT anomalies;
- Drilling to convert Inferred Mineral Resources to the Indicated classification;
- An extensive program of confirmatory, variability, and optimization testwork to improve confidence in the proposed process flowsheets, validate recovery assumptions,

reduce technical uncertainty, and support future Pre-Feasibility level process design, engineering development, and cost estimation;

- Geotechnical and hydrogeological drilling and studies;
- An environmental baseline study to characterize the existing features of the air, water and soil both on the Viken Property and in the surrounding area; and
- A Pre-Feasibility Study ("PFS").

The estimated budget to complete the recommended work program is approximately $7.3M and is presented in Table 1.10. The work should be completed in the next 12 to 18 months.

TABLE 1.10
RECOMMENDED WORK PROGRAM

Activity	Units	Cost Estimate (US$)
Drill testing of airborne MobileMT anomalies	3,000 m x $300/m	900,000
Interpretation, Modelling, Reporting		100,000
Diamond Drilling: Metallurgy and Inferred to Indicated Mineral Resource Conversion	6,000 m x $300/m	1,800,000
Metallurgical Testwork		900,000
Environmental Baseline Study		300,000
Geotechnical and Hydrogeological Studies		600,000
Pre-Feasibility Study		1,500,000
Subtotal		**6,100,000**
Contingency (20%)		1,220,000
Total		**7,320,000**

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 TERMS OF REFERENCE

The purpose of this Report is to provide an independent, NI 43-101 Preliminary Economic Assessment ("PEA" or the "Report") on the Districts Metals Corp. ("District Metals" or the "Company") Viken Project in Jämtland County, central Sweden (the "Project"). This PEA considers the mineralization at the Viken Energy Metals Property (the "Property") that is amenable to open pit mining. The Property is 100% owned by District Metals. The Alum Shale, a distinctive black shale unit on the Viken Property, hosts mineralization such as vanadium (V), uranium (U), sulphate of potash (SOP), molybdenum (Mo), nickel (Ni), and zinc (Zn).

The mine plan in this PEA accounts for approximately 3% of the current Mineral Resource and is intended to illustrate the first phase of a potential multi-phase Project. A PEA is preliminary in nature, and it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as Mineral Reserves, and there is no certainty that the PEA will be realized.

This Report was prepared by P&E Mining Consultants Inc. ("P&E") at the request of Mr. Garrett Ainsworth, President, CEO, and Director of District Metals. Input to this PEA was also provided by METS Engineering Group Pty Ltd ("METS") for mineral processing and metallurgy, surface infrastructure, and product marketing and pricing.

District Metals is a reporting issuer listed on the TSX Venture Exchange trading under the symbol DMX, and is also listed on the Nasdaq First North Growth Market in Sweden under the symbol DMXSE SDB. The corporate address of District Metals is:

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This Report is prepared in accordance with the requirements of National Instrument 43-101 ("NI 43-101") and in compliance with Form 43-101F1 of the Canadian Securities Administrators ("CSA"). The Mineral Resource Estimate is prepared in accordance with the CIM Definitions and Standards on Mineral Resources and Mineral Reserves, which are in force as of the effective date of this Report.

2.2 EFFECTIVE DATE

The effective date of this Report is June 26, 2026.

The Authors are satisfied that there has been no material change to the Property between the effective date and the signing date of this Report.

2.3 INDEPENDENT SITE VISIT

Mr. David Burga, P.Geo., an independent Qualified Person as defined in NI 43-101, completed a site visit to the Viken Property on March 20, 2025. The purpose of the site visit was to meet with District Metals technical staff on-site, complete a surface tour, check the location of selected surface drill hole collars, and confirm that there has been no new development work since the previous site visit in 2013.

Previously, Mr. Eugene Puritch, P.Eng, an independent Qualified Person as defined in NI 43-101, completed a site visit to the Property on June 18, 2013. A data verification sampling program was completed as part of that site visit.

2.4 RECENT TECHNICAL REPORT

A Technical Report was completed by P&E Mining Consultants for District Metals Corp. on June 13, 2025 ("P&E 2025") titled Updated Mineral Resource Estimate and Technical Report on the Viken Energy Metals Project, Jämtland County, Sweden. The Authors were: Puritch, Eugene; Barry, Jarita; Brown, Fred; Burga, David; Feasby, Grant; and Stone, William. The P&E 2025 Report had an effective date of April 25, 2025. The P&E 2025 Report and Updated Mineral Resource Estimate ("MRE") form the basis for this PEA. Several sections of this PEA have been based on the P&E 2025 Report, such as Sections 4 to 12, 14 and 23.

2.5 HISTORICAL TECHNICAL REPORTS

The following historical Technical Reports have been prepared under NI 43-101 on the Viken Property:

- Puritch, Eugene, Rodgers, K., Sutcliffe, R., Orava, D., Salari, D., Lawrence, R., Nodwell, M., Armstrong, T., Burga, D., and Brown, F. 2014. Updated Technical Report, Resource Estimate and Preliminary Economic Assessment on the Viken MMS Project, Sweden. Prepared by P&E Mining Consultants Inc. for Continental Precious Minerals Inc. Dated February 27, 2014. 148 pages.

- Puritch, E., Hayden, A., Partsch, A., Harron, G. and Brown, F. 2010. Preliminary Economic Assessment on the Viken MMS Project, Sweden, for Continental Precious Minerals Inc., dated October 19, 2010, NI 43-101 Technical Report filed on SEDAR.

- Harron, G.A., Brown, F.H. and Puritch, E. 2009. Third Updated Technical Report on Viken MMS Licence, Jämtland, Kingdom of Sweden for Continental Precious Minerals Inc., dated March 19, 2009.

- Harron, G.A., Brown, F.H. and Puritch, E. 2008. Second Updated Technical Report on Viken MMS licence, Jämtland, Kingdom of Sweden for Continental Precious Minerals Inc., April 11, 2008.

- Harron G.A . 2007. Technical Report on Viken MMS Licence, Jämtland, Kingdom of Sweden for Continental Precious Minerals Inc., March 31, 2007.

- Harron G.A., Brown, F.H. and Puritch, E. 2007. Updated Technical Report on Viken MMS Licence, Jämtland, Kingdom of Sweden for Continental Precious Minerals Inc., August 28, 2007.

2.6 SOURCES OF INFORMATION

This Report is based, in part, on internal company technical reports, and maps, published government reports, company letters and memoranda, and public information listed in the References section (Section 27) of this Report. As referred to throughout this Report, the sources of information include:

- Data supplied by District Metals;
- Observations made during the independent site visits by the Authors;
- Review of various data and reports from District Metals;
- Review of technical papers presented in various journals;
- Discussions with District Metals management and staff familiar with the Property; and
- Professional knowledge of black shale hosted mineral deposits.

The Authors have assumed, and relied on the fact, that all the information and existing technical documents listed in the References section of this Report are accurate and complete in all material aspects, and have taken all appropriate steps in their professional judgement to confirm same and are not disclaiming any responsibility for this Report except as permitted under applicable securities laws. The Authors reserve the right, but will not be obligated, to revise the Report and conclusions if additional information becomes known to the Authors subsequent to the effective date of this Report.

Select technical data, as noted in this Report, were provided by District Metals and the Authors have relied on the integrity of such data. A draft copy of the Report has been reviewed for factual errors by District Metals and the Authors have relied on District Metals' knowledge of the Property in this regard. All statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the effective date of this Report.

The Authors and Co-authors of each section of the Report (Table 2.1), who acting as independent Qualified Persons as defined in NI 43-101, take responsibility for those sections of the Report as outlined in Section 28 - Certificates of Author. The Authors acknowledge the helpful cooperation of District Metals' management and consultants who addressed all data requests and responded openly and helpfully to all questions and requests for material.

TABLE 2.1
QUALIFIED PERSONS RESPONSIBLE FOR THIS TECHNICAL REPORT

Qualified Person	Contracted by	Report Sections
Andrew Bradfield, P.Eng.	P&E Mining Consultants Inc.	2, 3, 15, 16, 22 and Co-author 1, 21, 24, 25, 26, 27
Jarita Barry, P.Geo.	P&E Mining Consultants Inc.	11 and Co-author 1, 12, 25, 26, 27
Fred Brown, P.Geo.	P&E Mining Consultants Inc.	Co-author 1, 14, 25, 26, 27
David Burga, P.Geo.	P&E Mining Consultants Inc.	Co-author 1, 12, 25, 26, 27
Grant Feasby, P.Eng.	P&E Mining Consultants Inc.	20 and Co-author 1, 25, 26, 27
Eugene Puritch, P.Eng., FEC, CET	P&E Mining Consultants Inc.	Co-author 1, 12, 14, 25, 26, 27
William Stone, Ph.D., P.Geo.	P&E Mining Consultants Inc.	4 to 10, 23 and Co-author 1, 25, 26, 27
Damian Connelly, FAusIMM (CP) Met. MMSA. FIEAust	METS Engineering Group Pty Ltd	13, 17, 18, 19 and Co-author 1, 21, 24, 25, 26, 27

2.7 UNITS AND CURRENCY

Metric units of measure are used in this Report, except references to metal concentrations to reflect the fact that uranium is traditionally traded in Imperial units. References to dollars in the report are to the US currency, unless otherwise indicated.

2.8 GLOSSARY AND ABBREVIATION OF TERMS

Table 2.2 lists the abbreviations for technical terms and Table 2.3 lists unit measurements used throughout the text of this Report.

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
$	dollar(s)
°	degree(s)
°C	degrees Celsius
<	less than
>	greater than
%	percent
µm	micrometre, micron
3-D	three-dimensional
Actlabs	Activation Laboratories Ltd.
ADU	ammonium diuranate
Ag	silver

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
AGAT	AGAT Laboratories Ltd.
Ai	(Bond) Abrasion index
AISC	all-in sustaining costs
Al	Aluminium
ALS or ALS Chemex	ALS Laboratories Limited or ALS Chemex Laboratories, part of ALS Global, ALS Limited
AMV	ammonium metavanadate
As	arsenic
Atomenergi AB	Atomic Energy Company
Au	gold
Authors	authors of this Technical Report
Bi	Bismuth
BRWi	Bond Rod Mill Work index
BWi	Bond Ball Mill Work index
C	carbon
C$	Canadian dollar
c/s	counts per second
Ca	calcium
$CaCO_3$	calcium carbonate
CAGR	compound annual growth rate
CaO	calcium oxide, lime, quicklime
CAPEX	capital costs
CCD	counter-current-decantation
CCTV	closed-circuit television
Ce	cerium
Ce_2O_3	cerium (III) oxide or cerium oxide
CFB	circulating fluidized bed
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetre
Co	cobalt
CO_2	carbon dioxide
Co-authors	co-authors of this Technical Report
Company, the	District Metals Corp.
CPM	Continental Precious Metals Inc.
Cr	chromium
CRM	certified reference material
CRMA	Sweden's Critical Raw Materials Act
CSA	Canadian Securities Administrators

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
Cu	copper
CWi	(Bond) Crushing Work index
D2EHPA	di(2-ethylhexyl)phosphoric acid
District Metals	District Metals Corp.
EIA	Environmental Impact Assessment
EPCM	Engineering, Procurement and Construction Management
ESIA	Environmental and Social Impact Assessments
EU	European Union
Expert	Expert Geophysics Limited
Fe	iron
FEED	Front-End Engineering Design
FeV	ferrovanadium
FeV80	ferrovanadium alloy 80% vanadium
FS	Feasibility Study
g	gram
g/L	grams per litre
g/t	grams per tonne
G&A	General and administrative
Ga	billion(s) of years
Geoforum	Geoforum Scandinavia AB
GET	ground-engaging tools
GPS	Global Positioning System
H_2SO_4	sulphuric acid
ha	hectare(s)
Hatch	Hatch Ltd.
HLS	heavy liquid separation
HMS	heavy media separation
HPGR	high-pressure grinding rolls
HSLA	high-strength low-alloy
ICP-MS	inductively coupled plasma mass spectrometry
IRR	internal rate of return
ISO	International Organization for Standardization
ISO/IEC	International Organization for Standardization/ International Electrotechnical Commission
ISR	in-situ recovery
IX	ion exchange
k	thousands

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
K	potassium
K_2O	potassium oxide
K_2SO_4	sulphate of potash
$KAl(SO_4)_2 \cdot 12H_2O$	potassium alum, potassium aluminum sulphate dodecahydrate
kg	kilogram
km	kilometre(s)
ktpd	thousands of tonnes per day
L	litre
La	lanthanum
La_2O_3	lanthana, lanthanum trioxide
lb	pounds
LKAB	Luosavaara-Kirunavaara Aktiebolag
LNG	liquefied natural gas
LOM	life of mine
M	millions
LPG	liquefied petroleum gas
m	metre(s)
m^3	cubic metre
Ma	millions of years
masl	metres above sea level
MCC	motor control centre
METS	METS Engineering Group Pty Ltd
Mg	magnesium
MgO	magnesium oxide
MIBC	methyl isobutyl carbinol
MIS	Mining Inspectorate of Sweden
Mkg	millions of kilograms
ML	millions of litres
Mlb	millions of pounds
mm	millimetre
Mm^3	millions of cubic metres
MMS	mixed metal sulphides
MMP	mixed metals precipitate
Mn	manganese
Mo	molybdenum
MoO_3	molybdenum trioxide
MoO_4^2	molybdate

Table 2.2
Terminology and Abbreviations

Abbreviation	Meaning
MOP	muriate of potash
MRE	Mineral Resource Estimate
Mt	million tonnes
Mtpa	million tonnes per annum/year
Myr	Myrviken
Na	sodium
NaCl	sodium chloride
$NaClO_3$	sodium chlorate
Na_2CO_3	sodium carbonate, soda ash
$NaHCO_3$	sodium hydrogen carbonate, sodium bicarbonate
NaOH	sodium hydroxide
NCF	net operating cash flows
Ni	nickel
NI 43-101	National Instrument 43-101 – Standards of Disclosure for Mineral Projects
NORM	naturally occurring radioactive materials
NPR	net profit royalty
NPV	net present value
NPVS	NPV Scheduler™ software
NSG	Sweden Bureau of Mines
NSR	net smelter return
O_2	oxygen
OK	Ordinary Kriging
OPEX	operating costs
P_{80}	passing 80%
P_2O_5	phosphorus pentoxide
P&E	P&E Mining Consultants Inc.
PAL	pressure acid leach
PAX	potassium amyl xanthate
Pb	lead
Pd	palladium
PEA	Preliminary Economic Assessment
P.Eng,	Professional Engineer
PFS	Pre-Feasibility Study
P.Geo.	Professional Geoscientist
PLS	pregnant leach solution
POL	pressure oxidative leach
POX	pressure oxidation

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
ppm	part per million
Project, the	Viken Energy Metals Project
Property, the	Viken Energy Metals Property
Pt	platinum
Purchase Agreement	definitive purchase agreement for the four mineral licenses covering the Viken Deposit
Q1, Q2, Q3, Q4	first quarter, second quarter, third quarter, fourth quarter
QA/QC	quality assurance/quality control
QC	quality control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy
QMS	quality management system
Rb	rubidium
Re	rhenium
Report, the	this Technical Report
RF	revenue factors
RM	reference material
ROM	run of mine: unprocessed mine product
S	sulphur
Sc	scandium
Se	selenium
SEDAR	System for Electronic Document Analysis and Retrieval
SEK	Swedish Krona (currency)
SEMs	Surface Excavation Machines
SFS number	refers to specific parts of the Swedish government regulatory acts
SG	specific gravity
SGAB	Sveriges Geologiska AB
SGU	Geological Survey of Sweden
skaelsta	composite shale reference material
SKB	Svensk Kärnbränsleförsorjning AB (Swedish Nuclear Fuel and Waste Management Company)
SMC test	Steve Morrell Comminution test
SMR	small modular reactor
Sn	tin
SO_2	sulphur dioxide
SO_3	sulphur trioxide
SO_4	sulphate

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
SOP	sulphate of potash
SRC	SRC Geoanalytical Laboratories, part of the Saskatchewan Research Council
SX	solvent extraction
t	tonnes
t/m^3	tonne(s) per cubic metre
Ta	tantalum
TBP	tributyl phosphate
TBS	teetered bed separator
Ti	titanium
TOF-SIMS	time-of-flight secondary ion mass spectrometry
tpa	tonnes per annum, year
TSX	Toronto Stock Exchange
U	uranium
UCS	unconfined compressive strength
UHF/VHF	ultra high frequency/very high frequency
UO_2	uraninite, uranium dioxide
U_3O_8	tri-uranium octoxide
UOC	uranium oxide concentrate
US$	United States dollar
UTM	Universal Transverse Mercator
V	vanadium
V_2O_5	vanadium pentoxide
Vendor, the	arm's length vendor selling the four mineral licenses covering the Viken Deposit
VO_2	vanadium dioxide
VRFB	vanadium redox flow battery
WGS84	World Geodetic System 1984
Y	yttrium
Y_2O_3	yttrium oxide or yttria
Zn	zinc

TABLE 2.3
UNIT MEASUREMENT ABBREVIATIONS

Abbreviation	Meaning	Abbreviation	Meaning
>	greater than	m^3/d	cubic metre per day
<	less than	m^3/h	cubic metre per hour
°	degree	m^3/s	cubic metre per second
µm	microns, micrometre	m^3/y	cubic metre per year
$	dollar	mØ	metre diameter
$/t	dollar per metric tonne	m/h	metre per hour
%	percent sign	m/s	metre per second
% w/w	percent solid by weight	Mt	million tonnes
¢/kWh	cent per kilowatt hour	Mtpy	million tonnes per year
°C	degree Celsius	min	minute
cm	centimetre	mL	millilitre
d	day	mm	millimetre
ft	feet	MV	medium voltage
GWh	Gigawatt hours	MVA	mega volt-ampere
g/t	grams per tonne	MW	megawatts
h	hour	oz	ounce (troy)
ha	hectare	Pa	Pascal
hp	horsepower	pH	Measure of acidity
k	kilo, thousands	ppb	part per billion
kg	kilogram	ppm	part per million
kg/t	kilogram per metric tonne	s	second
km	kilometre	t or tonne	metric tonne
kPa	kilopascal	tpd	metric tonne per day
kV	kilovolt	t/h	metric tonne per hour
kW	kilowatt	t/h/m	metric tonne per hour per metre
kWh	kilowatt-hour	$t/h/m^2$	metric tonne per hour per square metre
kWh/a	kilowatt-hour per annum	t/m	metric tonne per month
kWh/t	kilowatt-hour per metric tonne	t/m^2	metric tonne per square metre
L	litre	t/m^3	metric tonne per cubic metre
L/s	litres per second	T	short ton
lb	pound(s)	tpy	metric tonnes per year
M	million	V	volt
m	metre	W	Watt
m^2	square metre	wt%	weight percent
m^3	cubic metre	yr	year

3.0 RELIANCE ON OTHER EXPERTS

The Authors of this Report have assumed that all the information and technical documents listed in the References section of this Report (Section 27) are accurate and complete in all material aspects. The Authors reserve the right, but will not be obligated, to revise this Report and its conclusions if additional information becomes known subsequent to the effective date of this Report.

Although copies of the licenses, permits and work contracts were reviewed, the Authors have not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties, and have relied on the efficacy of the legal due diligence process conducted by legal counsel(s) to District Metals. The Authors have fully relied on information provided by District Metals and legal experts retained by District Metals for the foregoing information through the following document:

- Wahlin. 2026. District Metals Corp. - Title Opinion, Viken Property. Prepared for District Metals Corp. June 26, 2026. 3 pages plus 2 Appendices.

A draft copy of this Report has been reviewed for factual errors by District Metals. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading as of the effective date of this Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

The Viken Energy Metals Property (the “Property) is located in the central region of Sweden, approximately 520 km northwest of the Capital City of Stockholm and approximately 600 km south of the Arctic Circle (Figure 4.1). The Viken Property is situated approximately 25 km southwest of the Town of Östersund in the Municipality of Berg and Åre, Jämtland County. It is centred at Latitude 63° 04’ 35” North and Longitude 14° 16’ 48” East (WGS84 UTM 33V 463,600 m East, 6,994,300 m North).

4.2 MINERAL TENURE

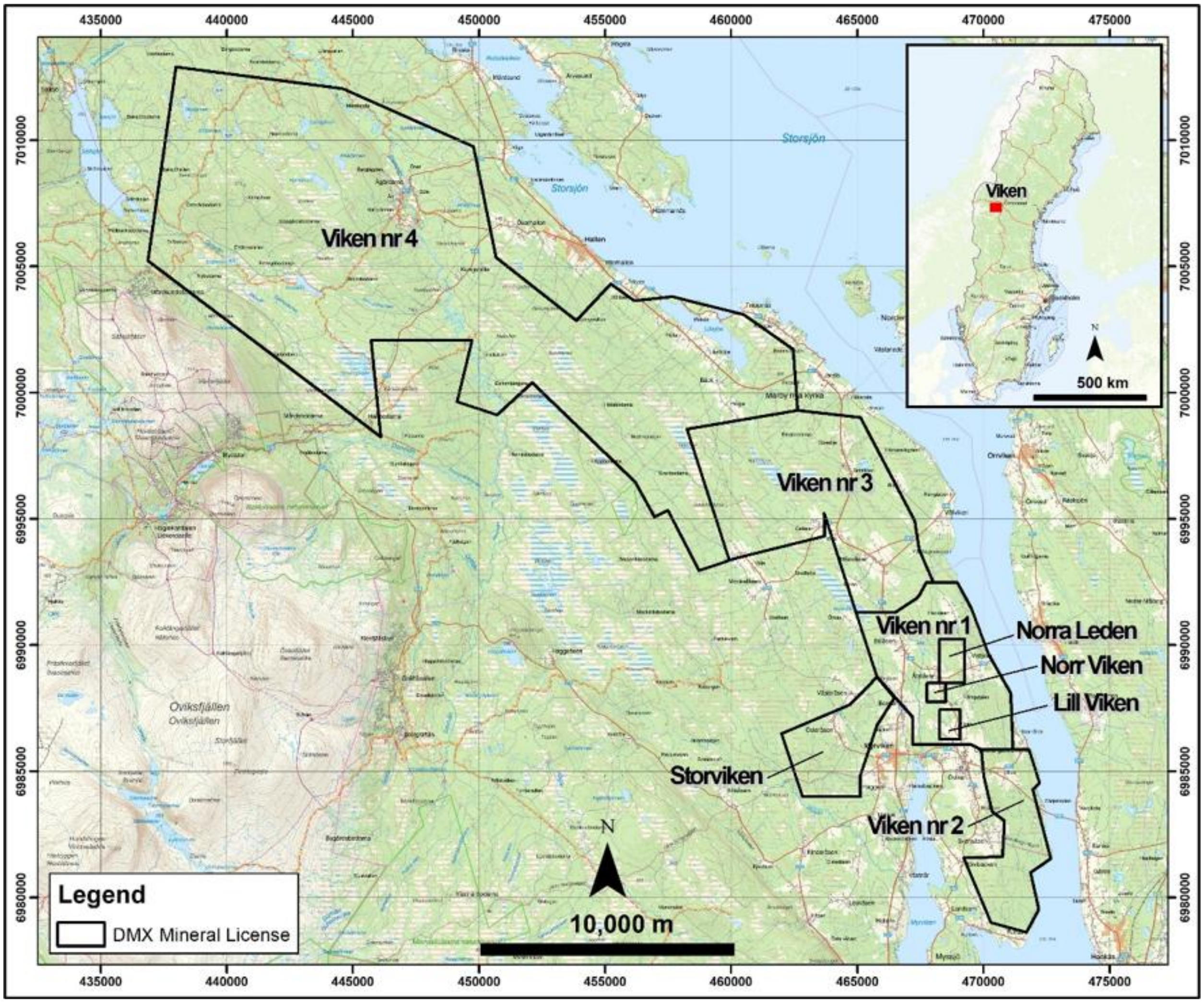
The Viken Property consists of eight contiguous licenses that cover a total area of 29,857 hectares (“ha”). Details of the Viken Property licenses are shown in Figure 4.2 and listed in Table 4.1.

As of the effective date of this Report, all eight licences are in good standing and 100% owned by Bergslagen Metals AB, a wholly-owned Swedish subsidiary of District Metals. In connection with a recently approved extension application, Viken 3 has been reduced in size. Viken 4 has also been reduced in size through a separate application and subsequent decision by the Mining Inspectorate. The current Mineral Resources are covered by the Viken Property licenses Viken nr 1, Norra Leden, Norr Viken and Lill Viken (Figure 4.3).

FIGURE 4.1 GENERAL LOCATION MAP

Source: *District Metals (June 2025)*

FIGURE 4.2 VIKEN PROPERTY LICENCES

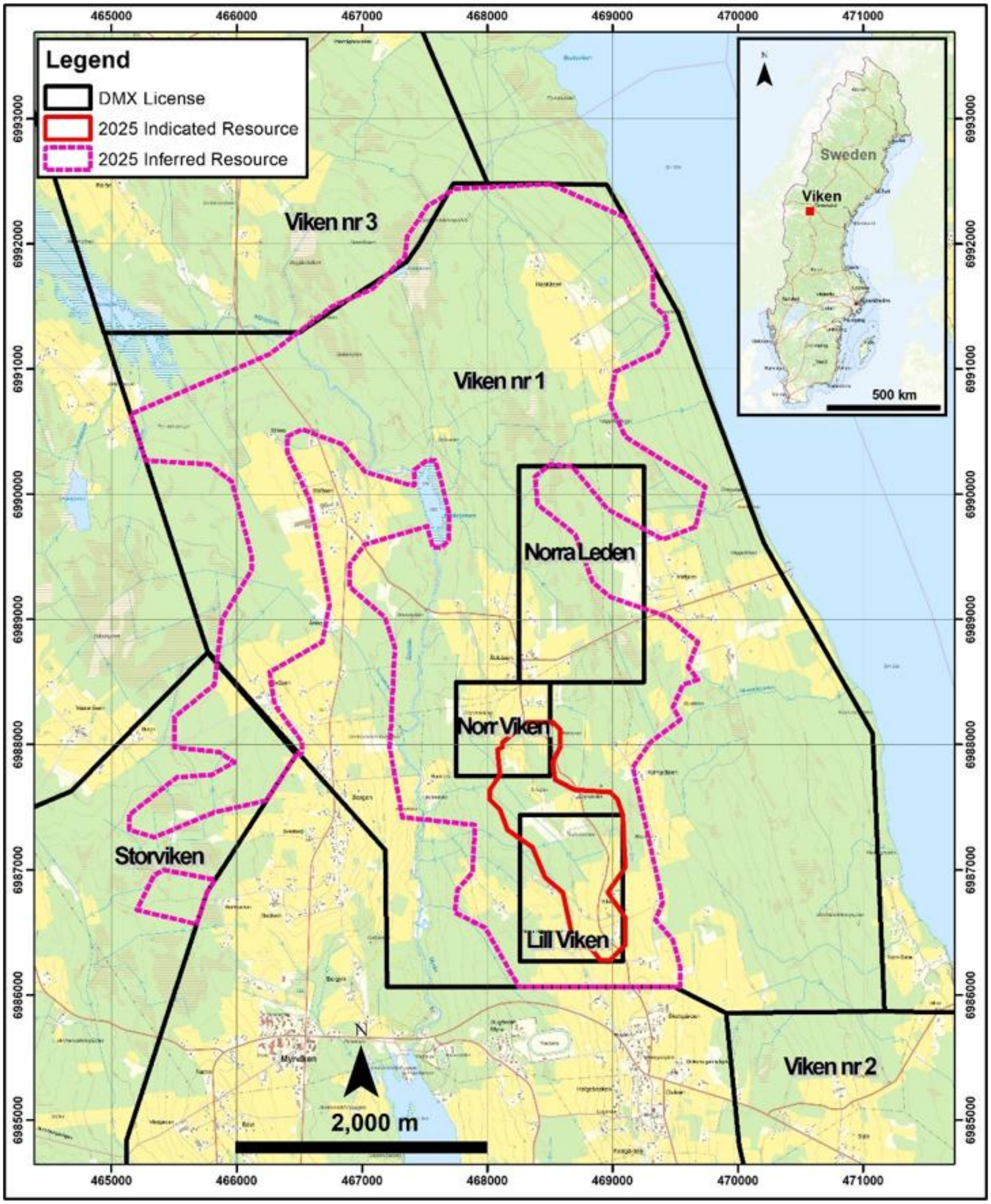


Source: *District Metals (June 2026)*

Table 4.1 Viken Property Exploration Licenses*							
Mineral License Name	**Mineral License ID**	**Owner (100%)**	**Date Granted**	**Expiry Date**	**Municipality**	**Metals/Mineral**	**Area (ha)**
Viken nr 1	2023:44	Bergslagen Metals AB	2023-04-03	2029-04-03	Åre and Berg	La, Mo, Ni, Sc, V, Y, Zn	2,301.50
Viken nr 2	2023:58	Bergslagen Metals AB	2023-04-27	2029-04-27	Berg	La, Mo, Ni, Sc, V, Y, Zn	1,419.61
Viken nr 3	2023:59	Bergslagen Metals AB	2023-04-27	2029-04-27	Åre and Berg	La, Mo, Ni, Sc, V, Y, Zn	4,609.66
Viken nr 4	2024:65	Bergslagen Metals AB	2024-04-11	2027-04-11	Åre	La, Mo, Ni, Sc, V, Y, Zn	20,080.25
Storviken	2023:19	Bergslagen Metals AB	2023-02-21	2029-02-21	Berg	Cu, Mo, Ni, V, pyrite	1,121.03
Norra Leden	2023:06	Bergslagen Metals AB	2023-01-17	2029-01-17	Berg	Cu, Mo, Ni, Rb, V, pyrite	172.50
Norr Viken	2022:79	Bergslagen Metals AB	2022-11-23	2028-11-23	Berg	Cu, Mo, Ni, Ta, V	56.25
Lill Viken	2022:78	Bergslagen Metals AB	2022-11-23	2028-11-23	Berg	Cu, Mo, Ni, Ta, V, pyrite	96.04
Total							**29,856.84**

Source: *Wahlin. 2026. District Metals Corp. - Title Opinion, Viken Property. Prepared for District Metals Corp. 3 pages plus 2 Appendices.*
Note: ** Mineral License information effective June 26, 2026.*

FIGURE 4.3 VIKEN NR 1 MINERAL LICENSE AND VIKEN DEPOSIT OUTLINE



Source: *District Metals (June 2026)*

4.3 MINERAL RIGHTS IN SWEDEN

In Sweden, mineral rights are primarily governed by the Minerals Act (1991:45) and Mineral Ordinance (1992:285), which regulate exploration and exploitation of particular minerals. Mineral rights are distinct from land ownership, which means that the State can grant permits for exploration and mining even if the land is privately owned. However, landowners and those with right of use to the land are entitled to compensation for damages and a share of the profits from any extracted minerals.

The following, more detailed information on mineral rights in Sweden is summarized from the following sources:

- https://www.sgu.se/en/mining-inspectorate/legislation/mineral-act-199145/, and
- from Harron *et al.* (2009).

All the Viken mineral licences were acquired in accordance with the Minerals Act (1991:45) since 2022, and are valid for a three-year term. Areas excluded from licence applications include lands occupied by private dwellings, state infrastructure, industrial plants, town sites, public buildings, cemeteries, military installations, historical sites and national parks or protected areas. A part of the application process requires that all the property owners in the licence area be notified that an exploration licence has been granted.

The Minerals Act (1991:45) governs the ownership and exploitation of mineral substances on private and public land. Mineral substances (concession minerals) are divided into three categories: 1) metallic substances (arsenic, beryllium, bismuth, cesium, chromium, cobalt, copper, gold, iridium, iron ore, lanthanum and lanthanide series, lead, lithium, manganese, mercury, molybdenum, nickel, niobium, osmium palladium, platinum, rhodium, rubidium, ruthenium, scandium, silver strontium, tantalum, thorium, tin, titanium, tungsten, uranium, vanadium, yttrium, zinc, and zirconium); 2) industrial minerals (alum shale, andalusite, apatite, barite, brucite, refractory and clinkering clay, coal, fluorspar, graphite, kyanite, magnesite, nepheline syenite, pyrite, pyrrhotite, salt and other similar salt deposits, sillimanite and wollastonite); and 3) oil, gaseous hydrocarbons and diamonds. Other minerals not captioned above belong to the landowner.

There are no restrictions on foreign ownership of licences. Exploration and exploitation licences are transferable subject to review by the Mining Inspectorate of Sweden ("MIS"). An exploration licence is granted for a specific area where there is some likelihood of a successful discovery of a concession mineral being made. A licence may be withheld if it is obvious that the applicant has not the possibility or intention to conduct appropriate exploration or has earlier shown unsuitability to conduct exploration work.

An exploration licence is valid for a period of three years from the date of issue and can be extended by another period of up to a maximum of three years (years four to six) provided that acceptable exploration work has been carried out on the licence. An extension may also be granted if the licence holder has valid reasons for non-performance of work and indicates that exploration will be conducted during the extension of time. In exceptional circumstances the licence may be extended for another maximum of four years (years seven to 10), and in extreme cases the licence may be extended by another maximum of five years (years 11 to 15). Fees attached to the acquisition and maintenance of an exploration licence include an application fee of SEK 500 per application and an exploration fee of SEK 20 per hectare for the first three-year period. Fees for the period four to six years are SEK 21 annually per hectare. Higher negotiated fees are attached to time periods beyond six years. All fees are paid in advance subject to partial reimbursement if necessary.

Prior to commencing invasive and systematic exploration activities, like drilling, the licence holder is required to file a work plan detailing proposed activities, timetable, and an assessment of the impact on private rights and public interests. The work plan needs to be communicated to all landowners and other interested parties covered by the work plan area. The work plan can be executed if no objections are raised. Failing initial approval, mediation between the licence holder and the objecting party is undertaken, and failing mutual agreement the MIS can set up conditions for the work plan to proceed. Guaranteed security deposits are also required to cover possible damage and encroachment from exploration work.

District Metals' management has provided a compilation on legal title of the eight Viken Property mineral licences. Inspection of these documents indicates that the licences are in good standing until at least April 2027.

Environmental and planning permits are not required for geological mapping and manual geochemical surveying. Permits are required for systematic till sampling, mechanical trenching and drilling programs.

When an exploration licence is terminated without the granting of an exploitation licence, the explorer is required to submit a report detailing the work performed, including the collected raw data (field observations, geophysical surveys, analytical data).

An application fee of SEK 6,000 is charged for each exploitation licence. Exploitation licences are granted for a period of 25 years unless the applicant requests a shorter period of time. Licences are automatically extended for 10-year terms if exploitation is in progress. An annual state mineral fee in the amount of 0.15% of gross value of concession minerals is due to the landowners within the concession area and 0.05% is due to the State.

However, the decision to issue an exploitation licence is dependent upon the proponent obtaining approval of the host Municipality. In addition to an exploitation licence, permits under the Swedish Environmental Code (1998:808), the Swedish Nuclear Activities Act are required.

4.4 SURFACE RIGHTS

District Metals does not control any surface rights at Viken. Mineral license holders in Sweden are entitled to explore for and develop mineral deposits in accordance with the Minerals Act /Ordinance ("Minerallagen" SFS 1991:45, and "Mineralförordningen" SFS 1992:285, and SFS 2005:943). Permissions for access to the license areas and to execute investigative work programs are governed by Bergsstaten, the Swedish Mining Inspectorate (www.sgu.se/bergsstaten).

4.5 PROPERTY APPROVALS AND ACQUISITION AGREEMENTS

4.5.1 Approval of Viken nr 1

In a press release dated January 5, 2023, District Metals announced that Bergslagen Metals AB (a 100% owned Swedish subsidiary of District Metals) had applied for a 2,302 ha mineral license (Viken nr 1) to explore for vanadium, nickel, molybdenum, zinc and other elements, covering approximately 68% of the polymetallic Viken Deposit. The application was approved by the Bergsstaten (Mining Inspectorate) in April 2023. Viken nr 1 covers 68% of the polymetallic Viken Deposit.

4.5.2 Approval of Viken nr 2 and Viken nr 3

In a press release dated March 7, 2023, District Metals announced that Bergslagen Metals AB (a 100% owned Swedish subsidiary of District Metals) had applied for two additional mineral licenses covering the area to the south (Viken nr 2) and north (Viken nr 3) of the Viken nr 1 application. The application was approved by the Bergsstaten (Mining Inspectorate) in May 2023. The approval of the Viken nr 2 and 3 licenses increased the area of the Viken Property from 2,302 ha to 9,367 ha.

4.5.3 Acquisition of Norra Leden, Norr Viken, Lill Viken and Storviken

In a press release dated January 15, 2024, District Metals announced that it has closed acquisition of the four mineral licenses covering the Viken Deposit that the Company did not previously control. Three of the licenses, namely Norra Leden, Norr Viken and Lill Viken, are entirely enclosed with the Viken nr 1 license. The fourth license, namely Storviken, is located southwest of Viken nr 1. Following such acquisition, the Company controlled 100% of the mineral licenses covering the Viken Deposit.

Pursuant to the definitive purchase agreement (the ‘Purchase Agreement”), District Metals acquired these four mineral licenses from an arm’s length vendor (the “Vendor”) on the following terms:

- C$50,000 cash paid to the Vendor on closing;
- C$50,000 cash payable to the Vendor within 30 days following the moratorium on uranium exploration and mining in Sweden being lifted;
- 1,000,000 District Metals shares issued to the Vendor on closing;
- 3,500,000 District Metals shares to be issued to the Vendor within 30 days following the moratorium on uranium exploration and mining in Sweden being lifted. These District Metals shares will be subject to a voluntary lock-up pursuant to which 500,000 will be released after four months after issuance, 500,000 will be released after six months after issuance, 1,000,000 will be released after twelve months after issuance, 1,000,000 will be released after 18 months after issuance, and 500,000 will be released twenty-four months after issuance; and
- A 2% net smelter return (“NSR”) royalty granted to the Vendor on closing that can be bought back in its entirety at any time for a value of C$8,000,000 where the first 1% NSR royalty may be purchased for C$2,000,000.

Pursuant to applicable securities laws, the District Metals shares issued to the Vendor on closing are subject to a four-month and one day hold period from the closing.

4.5.4 Application for Viken nr 4 Mineral License

In a press release dated May 21, 2024, District Metals reported that Bergslagen Metals AB (a 100% owned Swedish subsidiary of District Metals) had received final approvals from the Bergsstaten (Mining Inspectorate) for the Viken nr 4 mineral licence application to explore for vanadium, nickel, molybdenum, zinc, rare earth elements and other elements.

The approval of Viken nr 4 increased the area of the Viken Property from 9,367 a to 38,657 ha. The license is in good standing to 2027. Renewal for an additional three years will require payment of mineral license fees to the Bergsstaten, and the completion of some geological, geochemical, or geophysical work on the mineral license before the three-year term expires.

4.5.5 District Metals Acquires 2% NSR

In a press release dated February 3, 2025, District Metals announced acquisition of the 2% NSR royalty over the four mineral licenses covering the Viken Deposit (Norra Leden, Norr Viken, Lill Viken and Storviken). As a result, Districts Metals' 100% owned Viken Deposit is now completely free of any NSR royalty.

Pursuant to the definitive royalty purchase agreement, District Metals acquired the 2.0% NSR from an arm's length vendor for a purchase price of 500,000 common shares of District Metals (the "Consideration Shares"). The Consideration Shares are subject to a hold period of four months and one day, from the date of issue.

4.6 ENVIRONMENT AND PERMITTING

Environmental and planning permits are not required for geological mapping and manual geochemical surveying. Permits are required from the authorities for systematic till sampling, mechanical trenching and drilling programs. The Mining Inspectorate of Sweden is responsible for issuing permits for exploration and mining.

More details on environment and permitting are given in Section 20 of this Report.

4.7 OTHER SIGNIFICANT FACTORS

The Swedish moratorium on uranium exploration and mining was lifted effective January 1, 2026. On June 15, 2026, the Swedish Parliament voted to abolish a Municipal veto on uranium extraction that goes into legislative effect on July 15, 2026. A key aspect of the legislation is that the extraction and processing of uranium is now treated in a manner consistent with other metals and minerals within Sweden's overall permitting framework.

On June 11, 2026, the Swedish Government presented the terms of reference for a new inquiry that will investigate ways to enhance local and municipal influence for Alum Shale deposits. The inquiry mentions that it will include concerned parties from the private sector in their stakeholder dialogue. It will also provide an analysis of commercial consequences of any proposal made and will take into account Sweden’s commitments according to the Critical Raw Materials Act (“CRMA”).

The Viken Deposit is currently being reviewed by the Geological Survey of Sweden (“SGU”) as a deposit of National Interest (Riksintresse), and the SGU’s review is expected to be completed in Q3 2026.

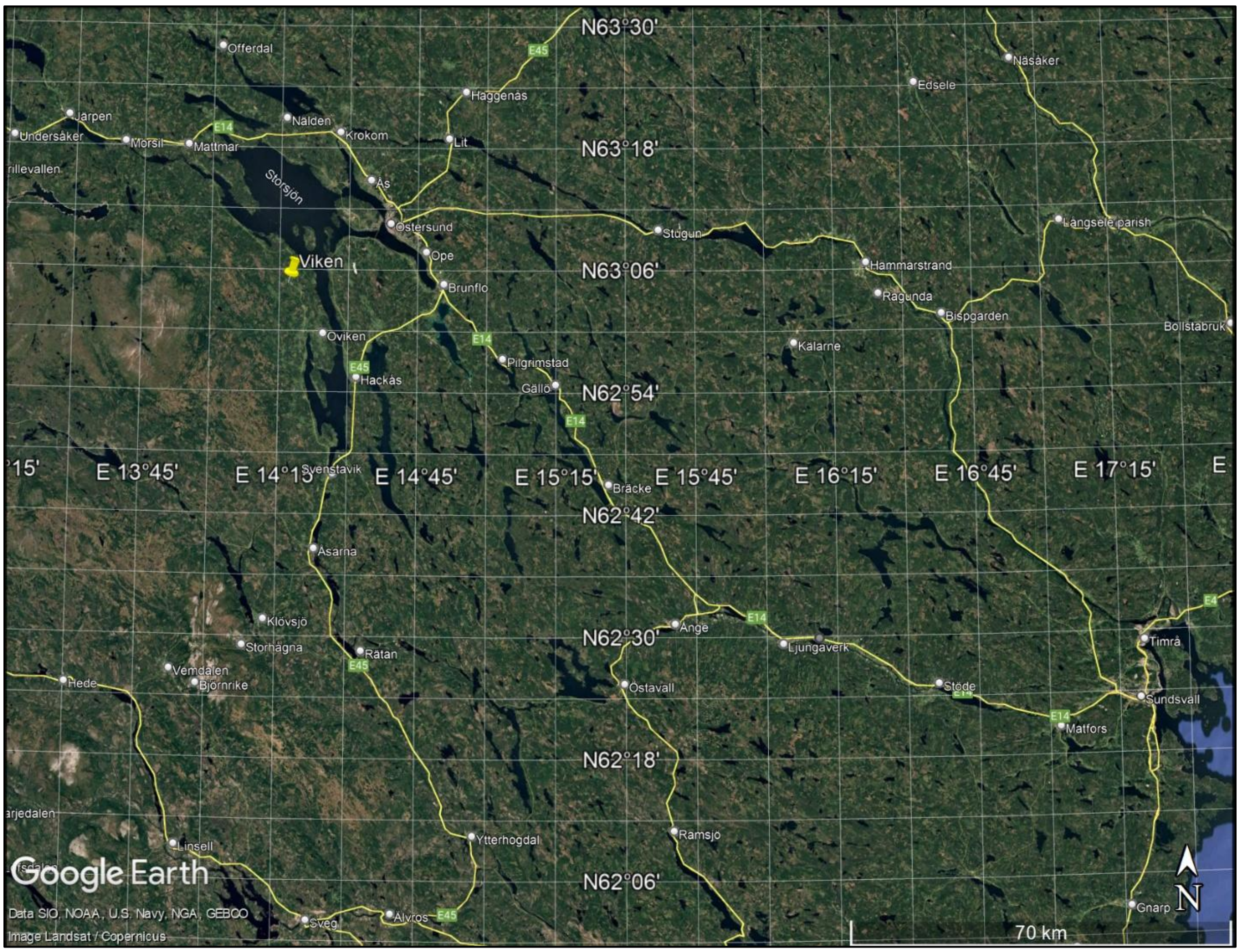
The Author is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the Property.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, PHYSIOGRAPHY AND INFRASTRUCTURE

5.1 ACCESS

The Viken Property is easily accessible via highway E14 from Sundsvall, and then by local highways E45 and 321 that connect to the Property area through Svenstavik at the southern end of Lake Storsjön. The Town of Östersund, with a population of 31,548 people (2025), is located approximately 40 km by road to the northeast (Figure 5.1).

FIGURE 5.1 ACCESS TO THE VIKEN PROPERTY



Source: *P&E (2025)*

5.2 CLIMATE AND PHYSIOGRAPHY

The Östersund region has a subarctic climate (Kopper: Dfc). Moderate temperatures in the summer months and sub-zero temperatures in the winter months are normal. Climate data are summarized in Figure 5.2.

The topography of the area is dominated by rounded hills and broad valleys at elevations in the range of 313 to 370 masl. Mixed coniferous and deciduous forests occur between open pasture lands. Mixed farming and small-scale forestry operations constitute the main land use.

Most exploration activities can be conducted year-round.

FIGURE 5.2 CLIMATE DATA FOR ÖSTERSUND

Climate data for Åre Östersund Airport (Frösön), 2002–2020; precipitation in Tullus 2002–2020; extremes since 1901 [hide]

Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year
Record high °C (°F)	9.8 (49.6)	10.3 (50.5)	17.8 (64.0)	20.5 (68.9)	26.6 (79.9)	32.0 (89.6)	33.0 (91.4)	31.7 (89.1)	25.0 (77.0)	19.7 (67.5)	12.2 (54.0)	10.8 (51.4)	33.0 (91.4)
Mean maximum °C (°F)	5.0 (41.0)	4.9 (40.8)	8.1 (46.6)	15.0 (59.0)	21.8 (71.2)	24.7 (76.5)	26.4 (79.5)	24.6 (76.3)	19.2 (66.6)	12.8 (55.0)	8.2 (46.8)	5.9 (42.6)	27.5 (81.5)
Mean daily maximum °C (°F)	−2.9 (26.8)	−2.2 (28.0)	1.4 (34.5)	7.2 (45.0)	12.6 (54.7)	16.8 (62.2)	19.5 (67.1)	17.9 (64.2)	12.8 (55.0)	6.3 (43.3)	1.1 (34.0)	−1.0 (30.2)	7.5 (45.4)
Daily mean °C (°F)	−5.9 (21.4)	−5.2 (22.6)	−2.1 (28.2)	3.2 (37.8)	8.2 (46.8)	12.5 (54.5)	15.3 (59.5)	14.1 (57.4)	9.8 (49.6)	4.0 (39.2)	−1.1 (30.0)	−3.7 (25.3)	4.1 (39.4)
Mean daily minimum °C (°F)	−8.8 (16.2)	−8.2 (17.2)	−5.6 (21.9)	−0.8 (30.6)	3.8 (38.8)	8.2 (46.8)	11.1 (52.0)	10.3 (50.5)	6.7 (44.1)	1.6 (34.9)	−3.3 (26.1)	−6.3 (20.7)	0.7 (33.3)
Mean minimum °C (°F)	−20.9 (−5.6)	−20.4 (−4.7)	−16.4 (2.5)	−7.2 (19.0)	−1.8 (28.8)	2.7 (36.9)	6.1 (43.0)	5.0 (41.0)	1.1 (34.0)	−5.3 (22.5)	−10.6 (12.9)	−16.8 (1.8)	−24.6 (−12.3)
Record low °C (°F)	−38.0 (−36.4)	−34.6 (−30.3)	−32.5 (−26.5)	−22.0 (−7.6)	−9.0 (15.8)	−3.0 (26.6)	−1.5 (29.3)	−0.8 (30.6)	−5.2 (22.6)	−17.7 (0.1)	−25.2 (−13.4)	−38.1 (−36.6)	−38.1 (−36.6)
Average precipitation mm (inches)	32.1 (1.26)	20.6 (0.81)	22.2 (0.87)	23.0 (0.91)	48.1 (1.89)	62.3 (2.45)	76.3 (3.00)	80.9 (3.19)	55.1 (2.17)	44.1 (1.74)	33.5 (1.32)	36.0 (1.42)	534.2 (21.03)
Average extreme snow depth cm (inches)	39 (15)	47 (19)	46 (18)	27 (11)	1 (0.4)	0 (0)	0 (0)	0 (0)	0 (0)	7 (2.8)	17 (6.7)	30 (12)	56 (22)
Mean monthly sunshine hours	30	70	150	202	244	259	257	208	127	83	40	22	1,692

Source 1: SMHI Temperature Data[15]

Source 2: SMHI Precipitation[16]

Sources: *www.wikipedia.org (March 2025) after SMHI Temperature Data (2021) and SMHI Precipitation (2021)*

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

Transportation systems are well developed in the area with daily air service and rail and truck freight services. Electrical power and modern communications are readily available in the area. The major mining centres in Sweden are Kiruna (~850 km), Malmberget and Aitik (~700 km) and the Skellefteå District (~450 km) to the northeast. Likewise, in the Bergslagen area with Garpenberg (~400 km) and Zinkgruvan (~600 km) to the south, each of which are sources of mining-related services. The Viken Property has sufficient surface area for future exploration or mining operations, including potential tailings storage areas, waste rock disposal areas and process plant sites.

6.0 HISTORY

6.1 PRE-2006 EXPLORATION

The following summary of historical exploration is derived from the 2009 Technical Report (Harron *et al*., 2009) and the 2010 PEA Report (Puritch *et al.*, 2010).

Mining of the Alum Shale started in 1637 at Andrarum in Skåne for the recovery of alum (potassium aluminum sulphate), used mainly for the preservation of hides and in the textile industry for the fixation of colours. The presence and potential usage of the kerogen component of the Alum Shale was recognized in the 1800s and many attempts were made to extract and refine the hydrocarbons using conventional retorting methods. Oil was recovered from deposits at Kinnekulle, Närke and Östergötland during World War II, largely for military purposes. However, with the renewed import of conventional oil after the war, the extraction plants ceased to be economically viable and production dwindled and finally ceased in 1966. The identification of high concentrations of uranium in kolm lenses (very high organic content rock) dates back to 1893. Mining proved unsuccessful, due to the erratic distribution of the kolm lenses.

After World War II, systematic exploration for uranium by the SGU on behalf of the Swedish Oil Shale Company, and later by the Atomic Energy Company ("Atomenergi AB") focused in the Västergötland and Närke areas. A bed of Alum Shale 3.5 m thick containing approximately 300 ppm U identified over a large area led to the establishment of a uranium extraction and research facility at Ranstad, with a nominal capacity of 120 tonnes of uranium per year. The process plant operated from 1965 to 1969 at approximately half capacity, due to the lower market price for uranium and lagging domestic demand. The recovery process consisted of milling the shale, removal of carbonate rock and acid leaching. Recoveries of approximately 67% were subsequently increased to approximately 80% prior to closing.

The presence of abnormally high concentrations of vanadium in the Alum Shale in the Skåne area was recorded in 1940 and led to various attempts to extract that element. The extraction technique involved roasting followed by sulphuric acid leaching. Small amounts of vanadium pentoxide and alum were produced during World War II.

Prior to changes in the Minerals Act in 1992-1993, most exploration work in Sweden was carried out by the SGU and state-controlled exploration companies, namely Sweden Bureau of Mines ("NSG"), Sveriges Geologiska AB ("SGAB"), and Luosavaara-Kirunavaara Aktiebolag ("LKAB"). In general, the state-controlled companies had separate geographical areas and metals of interest, and foreign companies were discouraged by the mandatory 50% State interest in all projects.

The Swedish State, in the form of AB Atomenergi (Anon, 1990), carried out exploration for uranium in the Alum Shale from the 1950s to 1967, after which the SGU took over uranium exploration. The SGU carried out work under contract for LKAB, in the Alum Shale in 1977-1978. This work resulted in the completion of 17 drill holes around Myrviken ("Myr") in the vicinity of the Viken nr 1 Licence. Two of these drill holes, Myr_78-002 and Myr_78-005 are in CPM's drill hole database. When drilled, assay values returned from Myr_78-005 drill core were 0.41 lb per ton U_3O_8 (205 ppm U_3O_8), 5.36 lb per ton V_2O_5 (2,680 ppm V_2O_5), 0.91 lb per ton MoO_3

(455 ppm MoO_3) and 0.08 lb per ton Ni (40 ppm Ni) over 182 m true thickness. This thickness is one of the largest ever recorded in the Alum Shale and coupled with the metal concentrations, indicated a target area for follow-up exploration.

In 1977, the Swedish Government commenced a 5-year exploration and evaluation self-sufficiency program for uranium, financed by the Svensk Kärnbränsleförsorjning AB (Swedish Nuclear Fuel and Waste Management Company, or "SKB"). The Swedish Government stopped its uranium exploration program in 1981 following an unfavourable referendum on nuclear power, though SKB continued work on selected targets until 1985, using SGAB (the exploration arm of SGU at the time) as the operator until 1982. There has not been any state-supported uranium exploration in Sweden since 1985 (Anon, 2002).

6.2 CPM 2006 TO 2012

6.2.1 Ownership

CPM entered into a purchase agreement with Geoforum Scandinavia AB ("Geoforum") on March 21, 2005 for the acquisition of the Viken Deposit area licences. Under the terms of the agreement, CPM issued 300,000 common shares to Geoforum and paid $40,000 cash, with Geoforum retaining a 5% net profit royalty ("NPR").

CPM reduced Geoforum's 5% NPR to 1% on March 7, 2012, in order to facilitate negotiations with potential joint venture partners. Geoforum received 5,000,000 common shares of CPM (representing 8.8% of the issued and outstanding shares of CPM following issuance) and $57,735 cash. CPM had the right of first refusal on the sale of the first 1,000,000 common shares by Geoforum following the transfer of the first 2,500,000 shares.

6.2.2 Exploration

CPM completed further geophysical surveying at the Viken Deposit and surrounding exploration licences following the 2010 PEA (Puritch *et al.*, 2010). Radiometric measurements were undertaken at the Koborgen and Viken licences and radon measurements were made.

The drill rig was modified in order that it could take surface bedrock, base of till and other overburden samples to expedite surface drill mapping. Base-of-till samples were taken on many of the exploration licences surrounding the main Viken licence.

Trenching and excavation work was also undertaken to pass through a thin layer of limestone and penetrate the underlying black shale following the 2011 drilling.

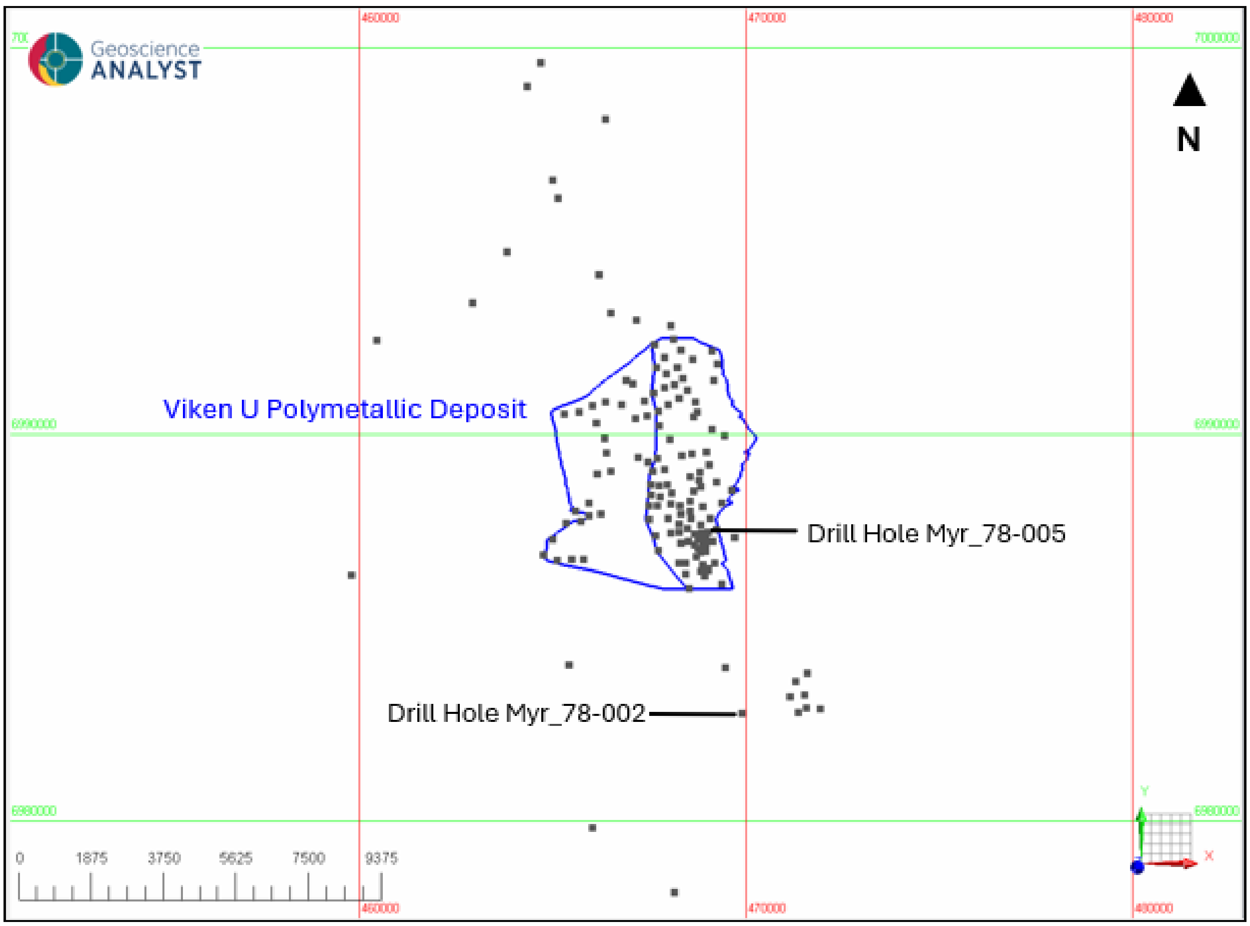
Ten pits were excavated down to bedrock (depths ranging from 2 to 5 m) with the aim to define and analyze the characteristics of the soil stratigraphy and to place vertical drainage "well" pipes for sampling ground water within some properties in the Viken Licence.

Topographic profiles of the Project area were also studied in greater detail over numerous licences, in order to delineate prospective lithology.

6.2.3 Drilling

Drilling completed by CPM from 2006 to 2012 is the main support for the current Mineral Resource Estimate presented in Section 14 of this Report. This work by CPM involved completion of 150 drill holes totalling 28,208 m in the Viken Deposit and surrounding area (Figure 6.1). In addition, two of the SGU 1978 drill holes (Myr_78-005 and Myr_78-002) totalling 459 m are included in the CPM database. The CPM drilling results for 2006 to 2008 and 2010 to 2012 are described below. A drill hole location surface plan is included in Appendix A.

FIGURE 6.1 CPM AND SGU DRILLING IN THE VIKEN DEPOSIT AREA



Source: *P&E (2025)*
Note: *The symbols represent drill hole locations.*

6.2.3.1 CPM 2006 to 2008 Drilling Programs

The following information on the 2006 to 2008 drilling by CPM is taken largely from Harron *et al.* (2009).

Drilling on the Viken Licence commenced in August 2006 and 11 drill holes (Myr_06-001 to Myr_06-011) for a total of 1,953 m were completed by year-end. Drilling resumed in February

2007 with two drill rigs and to December 31, 2007, 65 additional diamond drill holes were completed for a total of 12,779 m. In 2008, 55 drill holes for 10,872 m were completed. To December 31, 2008, a combined total of 131 diamond drill holes had been completed for 25,605 m. All drill collars were sited using GPS instruments and capped casings left in place pending final collar surveying with high precision GPS survey systems. All diamond drill holes were vertical holes, yielding true thickness intercepts. Drill hole trajectory surveys were not undertaken, due to the short length of the drill holes and the friable nature of the Alum Shale.

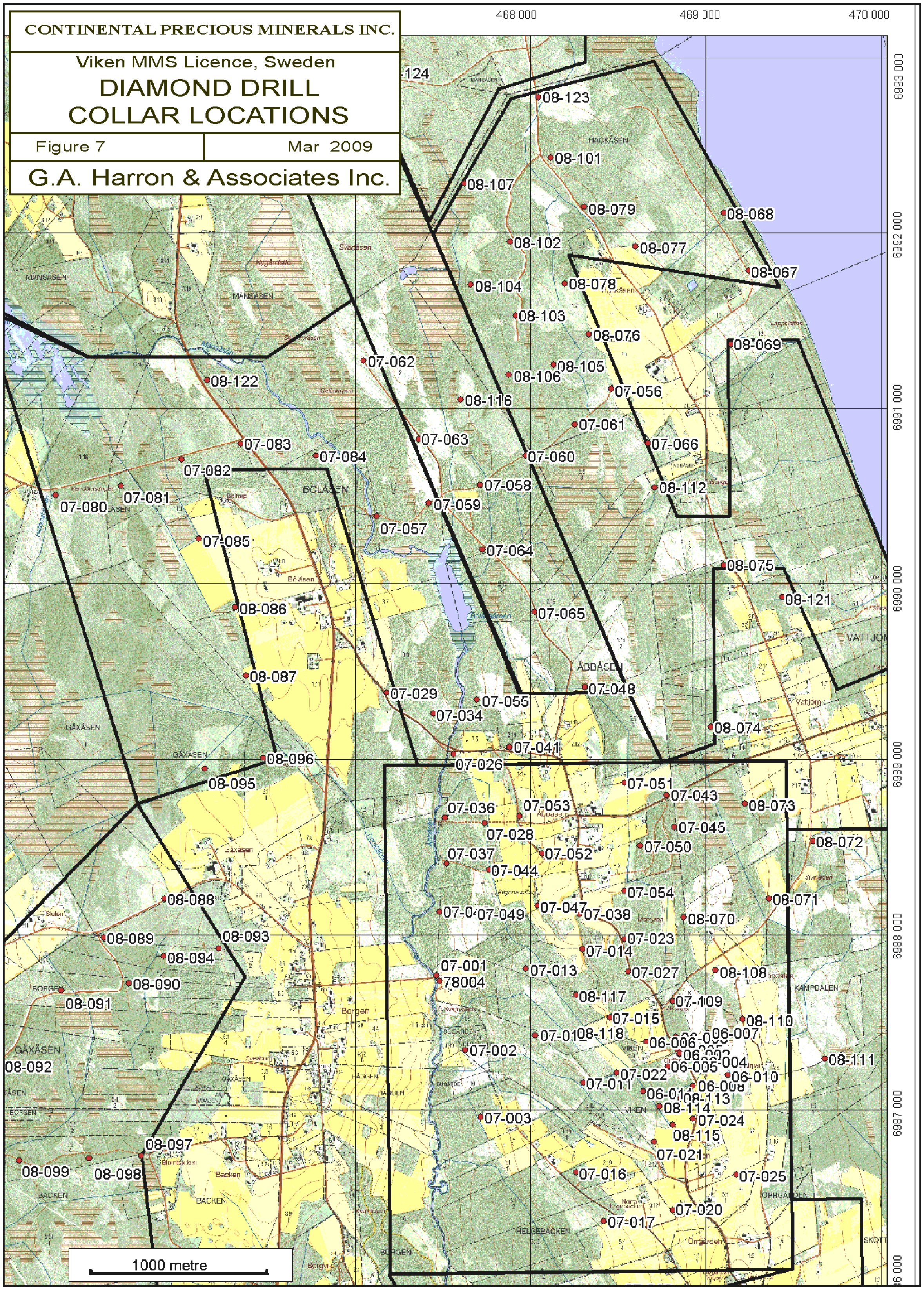
The 2006 diamond drilling was contracted to Taiga Exploration Drilling AB of Malå, and the 2007 and 2008 drilling was contracted to T.G.B. Borrteknik AB of Gråbo and Ludvika Borrteknik HB of Ludvika. All three drill contractors used Atlas Copco track mounted drills that recovered BQ size drill core.

Spatial distribution of the drill holes used to in the estimation of Inferred and Indicated Mineral Resources is illustrated in Figure 6.2. The locations and details of the drill holes are listed in Table 6.1.

Drill holes Myr_06-001 and Myr_06-002 were completed 2 m from each other and approximately 30 m south of drill hole Myrviken_78-005 (254 m in length), in order to verify the results of the previous drilling by the SGU in 1978 and to acquire sufficient mineralized material to complete additional analytical tests. The lithologies in Myr_06-001 and Myr 06-002 are almost identical, and therefore only the analytical data from Myr_06-002 are discussed further.

Drill holes Myr_06-001 and Myr_06-002 intersected a true thickness of 153.1 m of Alum Shale, whereas drill hole Myrviken_78-005 (~30 m distant) intersected a true thickness of 193.5 m. The analytical results for organic carbon and selected oxides in each of the two drill cores are very similar (Table 6.2). This correlation suggests an acceptable level of accuracy associated with the current analytical results.

FIGURE 6.2 2006 TO 2008 DIAMOND DRILL HOLE COLLAR LOCATIONS



Source: *Harron et al. (2009)*
Note: *The mineral license boundaries shown (dark lines) are as they were in 2009.*

TABLE 6.1 2006 TO 2008 DIAMOND DRILL HOLE LOCATIONS AND DETAILS						
Drill Hole ID	**Easting (m)**	**Northing (m)**	**Elevation (masl)**	**Length (m)**	**Azimuth (°)**	**Dip (°)**
Myr_06-001	468,849	6,987,322	366.09	247.1	360	90
Myr_06-002	468,850	6,987,323	366.34	200.4	360	90
Myr_06-003	468,828	6,987,417	367.30	224.3	360	90
Myr_06-004	468,940	6,987,281	363.71	226.3	360	90
Myr_06-005	468,782	6,987,249	356.07	193.7	360	90
Myr_06-006	468,659	6,987,393	359.29	206.0	360	90
Myr_06-007	469,010	6,987,439	362.75	101.8	360	90
Myr_06-008	468,926	6,987,140	359.31	210.3	360	90
Myr_06-009	468,623	6,990,440	370.00	97.8	360	90
Myr_06-010	469,122	6,987,198	366.29	47.8	360	90
Myr_06-011	468,641	6,987,105	349.46	197.9	360	90
Myr_07-001	467,464	6,987,770	309.72	191.2	360	90
Myr_07-002	467,627	6,987,343	310.16	200.0	360	90
Myr_07-003	467,714	6,986,962	312.17	207.0	360	90
Myr_07-011	468,300	6,987,157	341.93	196.3	360	90
Myr_07-012	468,027	6,987,426	338.32	199.7	360	90
Myr_07-013	467,972	6,987,807	335.03	194.3	360	90
Myr_07-014	468,294	6,987,920	348.19	188.9	360	90
Myr_07-015	468,449	6,987,528	351.26	209.0	360	90
Myr_07-016	468,258	6,986,646	336.44	203.3	360	90
Myr_07-017	468,413	6,986,368	334.63	218.1	360	90
Myr_07-018	468,514	6,985,982	337.12	199.4	360	90
Myr_07-020	468,810	6,986,431	353.72	224.2	360	90
Myr_07-021	468,703	6,986,817	350.19	200.8	360	90
Myr_07-022	468,491	6,987,214	344.44	200.3	360	90
Myr_07-023	468,536	6,987,978	362.04	234.0	360	90
Myr_07-024	468,926	6,986,951	360.33	224.2	360	90
Myr_07-025	469,172	6,986,635	369.27	206.8	360	90
Myr_07-026	467,558	6,989,031	322.37	213.3	360	90
Myr_07-027	468,558	6,987,788	364.52	225.5	360	90
Myr_07-028	467,736	6,988,638	322.81	198.0	360	90
Myr_07-029	467,180	6,989,382	333.33	217.5	360	90
Myr_07-031	471,573	6,983,804	308.65	95.7	360	90
Myr_07-032	471,503	6,983,228	309.35	101.2	360	90
Myr_07-033	471,124	6,983,179	315.90	112.7	360	90
Myr_07-034	467,443	6,989,262	325.01	203.8	360	90
Myr_07-035	471,259	6,983,591	316.69	111.4	360	90

TABLE 6.1
2006 TO 2008 DIAMOND DRILL HOLE LOCATIONS AND DETAILS

Drill Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Length (m)	Azimuth (°)	Dip (°)
Myr_07-036	467,511	6,988,666	320.45	187.0	360	90
Myr_07-037	467,521	6,988,407	321.34	189.7	360	90
Myr_07-038	468,279	6,988,118	361.21	228.3	360	90
Myr_07-039	471,550	6,982,904	309.04	92.9	360	90
Myr_07-040	471,884	6,982,889	300.96	82.9	360	90
Myr_07-041	467,881	6,989,071	331.07	209.8	360	90
Myr_07-042	467,479	6,988,132	317.35	184.6	360	90
Myr_07-043	468,775	6,988,796	358.43	221.8	360	90
Myr_07-044	467,763	6,988,369	326.13	199.8	360	90
Myr_07-045	468,818	6,988,616	360.14	220.3	360	90
Myr_07-047	468,040	6,988,169	342.8	213.3	360	90
Myr_07-048	468,307	6,989,412	349.38	211.1	360	90
Myr_07-049	467,672	6,988,122	327.02	200.9	360	90
Myr_07-050	468,624	6,988,510	357.83	216.0	360	90
Myr_07-051	468,537	6,988,868	355.56	220.5	360	90
Myr_07-052	468,064	6,988,465	345.33	214.9	360	90
Myr_07-053	467,936	6,988,678	334.34	206.7	360	90
Myr_07-054	468,534	6,988,249	359.72	219.0	360	90
Myr_07-055	467,694	6,989,343	321.26	188.8	360	90
Myr_07-030	469,459	6,983,940	305.00	103.1	360	90
Myr_07-046	471,321	6,982,786	311.88	92.3	360	90
Myr_07-056	468,460	6,991,113	343.67	217.7	360	90
Myr_07-057	467,120	6,990,386	325.76	208.3	360	90
Myr_07-058	467,708	6,990,566	334.78	244.1	360	90
Myr_07-059	467,419	6,990,461	325.29	226.6	360	90
Myr_07-060	467,970	6,990,732	341.33	242.3	360	90
Myr_07-061	468,252	6,990,912	346.70	228.4	360	90
Myr_07-062	467,045	6,991,273	337.25	220.5	360	90
Myr_07-063	467,360	6,990,825	334.35	191.4	360	90
Myr_07-064	467,725	6,990,195	328.04	209.5	360	90
Myr_07-065	468,022	6,989,840	338.48	192.0	360	90
Myr_07-066	468,668	6,990,805	343.12	211.3	360	90
Myr_07-080	465,290	6,990,508	338.04	254.8	360	90
Myr_07-081	465,663	6,990,558	331.11	194.4	360	90
Myr_07-082	466,008	6,990,711	337.45	122.5	360	90
Myr_07-083	466,347	6,990,799	339.42	245.7	360	90
Myr_07-084	466,773	6,990,734	329.76	224.4	360	90
Myr_07-085	466,106	6,990,257	340.72	241.0	360	90

TABLE 6.1
2006 TO 2008 DIAMOND DRILL HOLE LOCATIONS AND DETAILS

Drill Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Length (m)	Azimuth (°)	Dip (°)
Myr_07-109	468,809	6,987,623	368.07	224.8	360	90
Myr_08-067	469,242	6,991,787	307.87	189.3	360	90
Myr_08-068	469,100	6,992,115	302.54	182.7	360	90
Myr_08-069	469,143	6,991,365	325.19	192.7	360	90
Myr_08-075	469,102	6,990,108	347.66	233.2	360	90
Myr_08-076	469,102	6,991,423	344.72	206.7	360	90
Myr_08-077	469,102	6,991,925	327.24	219.0	360	90
Myr_08-078	468,194	6,991,711	340.64	243.9	360	90
Myr_08-079	468,303	6,992,152	328.54	230.8	360	90
Myr_08-086	466,314	6,989,869	343.33	232.5	360	90
Myr_08-087	466,377	6,989,479	342.74	233.7	360	90
Myr_08-088	465,910	6,988,208	359.59	140.2	360	90
Myr_08-089	465,565	6,987,985	352.34	213.0	360	90
Myr_08-090	465,709	6,987,723	350.68	129.8	360	90
Myr_08-091	465,322	6,987,685	342.63	210.0	360	90
Myr_08-092	464,980	6,987,253	342.06	124.0	360	90
Myr_08-093	466,220	6,987,924	361.85	175.5	360	90
Myr_08-094	465,907	6,987,877	355.19	123.0	360	90
Myr_08-095	466,140	6,988,945	347.10	225.0	360	90
Myr_08-096	466,473	6,989,019	341.64	226.4	360	90
Myr_08-097	465,778	6,986,738	337.58	228.0	360	90
Myr_08-098	465,482	6,986,728	332.52	202.0	360	90
Myr_08-099	465,084	6,986,715	333.27	244.5	360	90
Myr_08-100	464,736	6,986,844	327.58	181.7	360	90
Myr_08-101	468,114	6,992,431	323.27	84.5	360	90
Myr_08-102	467,881	6,991,952	338.36	180.9	360	90
Myr_08-103	467,914	6,991,531	342.77	191.8	360	90
Myr_08-104	467,659	6,991,706	336.93	236.7	360	90
Myr_08-105	468,133	6,991,250	345.68	239.7	360	90
Myr_08-106	467,874	6,991,193	340.36	230.7	360	90
Myr_08-107	467,620	6,992,285	331.15	159.3	360	90
Myr_08-112	468,707	6,990,551	343.73	222.6	360	90
Myr_08-116	467,598	6,991,050	334.67	209.8	360	90
Myr_08-119	466,187	6,994,105	318.93	195.0	360	90
Myr_08-120	464,982	6,996,543	332.15	261.0	360	90
Myr_08-121	469,435	6,989,926	333.77	209.6	360	90
Myr_08-123	468,044	6,992,778	310.45	209.5	360	90
Myr_08-124	467,140	6,992,925	322.49	164.8	360	90

TABLE 6.1 2006 TO 2008 DIAMOND DRILL HOLE LOCATIONS AND DETAILS						
Drill Hole ID	**Easting (m)**	**Northing (m)**	**Elevation (masl)**	**Length (m)**	**Azimuth (°)**	**Dip (°)**
Myr_08-125	464,668	6,999,588	302.77	184.3	360	90
Myr_08-126	464,313	6,998,964	332.78	172.8	360	90
Myr_08-128	462,900	6,993,367	332.90	179.7	360	90
Myr_08-129	466,021	6,979,807	349.79	89.5	360	90
Myr_08-130	465,410	6,984,018	319.26	76.6	360	90
Myr_78-002*	469,863	6,982,769	354.12	204.8	360	90
Myr_78-005*	468,834	6,987,345	366.20	254.4	360	90
Total	**120**			**23,278.0**		

Source: *Harron et al. (2009)*
Notes: ** Historical drill holes completed by SGU in 1978.*

TABLE 6.2 COMPARISON OF ALUM SHALE IN MYR_78-005 AND MYR_06-002 DRILL CORE										
Drill Hole ID	**True Thick-ness (m)**	**C_{org} (%)**	**U_3O_8 (lb/ton)**	**U_3O_8 (ppm)**	**V_2O_5 (lb/ton)**	**V_2O_5 (ppm)**	**Mo (lb/ton)**	**Mo (ppm)**	**Ni (lb/ton)**	**Ni (ppm)**
Myr_06-002	153.1	12.1	0.41	205	6.45	3,225	0.94	470	0.81	405
Myr_78-005	193.5	~11.1	0.41	205	5.40	2,700	0.92	460	n.a.	n.a.

Source: *Harron et al. (2009)*

The increases in the content of organic carbon and vanadium oxide are probably a result of improved analytical techniques since 1978. The close correlation of the analytical values from the two drill holes indicates that the SGU data is credible and can be integrated into the Viken Project database.

The initial drill pattern was designed to drill test the Alum Shale at distances of approximately 50 to 150 m from drill hole Myrviken_78-005. This objective was accomplished with drill holes Myr_06-002, Myr_06-003, and Myr_06-005. The next two diamond drill holes, Myr_06-006 and Myr_06-007, tested the Alum shale at a distance of 200 m from drill hole Myrviken_78-005. Step-out drill holes Myr_06-008 and Myr_06-009 were sited 250 m away from Myrviken_78-005, and drill holes Myr_06-010 and Myr_06-011 were sited 300 m distant from Myrviken_78-005.

Satisfied with the geological and grade continuity results from the 2006 drilling, CPM commenced grid drilling at 400 m centres with reference to grid base line oriented at 345°, which is essentially the strike direction of the exposed (or thinly covered) Alum Shale nappes.

Drilling recommenced in February of 2007 and continued until December 31, 2007. Results of this drilling indicated that the most metalliferous and thickest portion of the outcropping Alum Shale trends northwest along strike from the historical drill hole Myrviken_78-005. The Alum Shale was also encountered at depths >50 m in the vicinity of drill holes Myr_07-001, Myr_07-002 and Myr_07-003.

Assay results for the mineralized intervals in the 2006 to 2008 drilling are listed in Table 6.3.

TABLE 6.3
SUMMARY OF MINERALIZED INTERVALS IN 2006 TO 2008 DRILLING PROGRAMS

Drill Hole Number	From (m)	To (m)	Length (m)	U (ppm)	U_3O_8 (ppm)	V (ppm)	V_2O_5 (ppm)	Zn (ppm)	Mo (ppm)	Ni (ppm)
Myr_06-001	12.72	209.90	162.78	180	212	1,881	3,358	329	287	289
Myr_06-002	15.63	200.40	153.12	172	203	1,808	3,228	436	312	403
Myr_06-003	42.19	211.87	167.68	161	190	1,590	2,838	333	270	264
Myr_06-004	4.44	216.09	157.37	169	199	1,595	2,847	290	301	276
Myr_06-005	4.02	190.91	185.11	134	158	2,191	3,911	432	237	271
Myr_06-006	8.76	201.37	192.42	129	152	1,895	3,383	431	222	336
Myr_06-008	5.46	194.51	188.51	193	228	1,721	3,072	427	316	421
Myr_06-011	7.25	190.28	182.78	143	169	1,761	3,144	429	264	360
Myr_07-001	51.14	162.87	104.00	132	156	1,368	2,442	377	186	197
Myr_07-003	160.15	200.30	40.16	45	53	420	750	286	70	81
Myr_07-011	42.32	180.25	80.44	124	146	1,439	2,569	447	223	312
Myr_07-012	128.69	179.06	23.50	146	172	1,214	2,167	436	238	321
Myr_07-013	6.94	189.79	182.79	130	153	1,453	2,594	386	229	301
Myr_07-014	2.77	188.90	183.80	140	165	1,341	2,394	376	254	297
Myr_07-015	5.40	163.32	155.90	125	147	1,434	2,560	383	200	307
Myr_07-016	146.47	203.30	46.00	136	160	1,042	1,860	418	235	296
Myr_07-017	149.78	212.35	57.35	111	131	950	1,696	310	183	193
Myr_07-020	6.44	218.75	212.25	149	176	2,007	3,583	446	239	253
Myr_07-021	4.04	174.29	166.60	162	191	1,645	2,937	444	284	375
Myr_07-022	5.19	181.24	175.50	139	164	1,306	2,331	388	216	294
Myr_07-023	86.53	211.25	111.92	133	157	1,253	2,237	381	221	291
Myr_07-024	9.42	209.24	195.34	163	192	1,703	3,040	461	289	383
Myr_07-025	113.40	199.16	51.79	158	186	1,385	2,473	396	299	378
Myr_07-026	0.31	193.16	166.65	108	127	1,654	2,953	440	217	315
Myr_07-027	105.03	218.28	94.46	137	162	1,461	2,608	422	248	325
Myr_07-028	7.58	187.48	179.78	134	158	1,909	3,408	465	247	345
Myr_07-029	149.16	207.81	58.65	111	131	542	968	204	183	175
Myr_07-034	117.62	195.19	77.57	125	147	1,002	1,789	298	183	231
Myr_07-036	37.63	179.90	135.28	137	162	1,642	2,931	375	239	325
Myr_07-038	3.79	219.23	186.67	158	186	1,480	2,642	426	266	344
Myr_07-041	50.65	190.49	132.95	128	151	1,417	2,530	372	205	281
Myr_07-042	78.81	178.02	99.21	139	164	1,305	2,330	433	241	307

TABLE 6.3
SUMMARY OF MINERALIZED INTERVALS IN 2006 TO 2008 DRILLING PROGRAMS

Drill Hole Number	From (m)	To (m)	Length (m)	U (ppm)	U_3O_8 (ppm)	V (ppm)	V_2O_5 (ppm)	Zn (ppm)	Mo (ppm)	Ni (ppm)
Myr_07-043	53.02	213.09	155.53	174	205	1,438	2,567	400	305	396
Myr_07-044	2.93	191.55	127.17	142	167	1,629	2,908	390	236	330
Myr_07-045	144.86	213.70	68.84	181	213	1,504	2,685	420	320	404
Myr_07-047	4.05	208.11	204.07	174	205	1,922	3,431	432	300	411
Myr_07-048	68.72	200.95	131.00	141	166	1,522	2,717	437	249	338
Myr_07-049	3.61	191.88	167.86	111	131	1,271	2,269	349	192	259
Myr_07-050	81.67	203.54	118.48	161	190	1,824	3,256	478	274	385
Myr_07-051	119.29	212.12	84.40	140	165	1,737	3,101	439	245	355
Myr_07-052	6.26	207.88	195.83	145	171	1,736	3,099	454	249	352
Myr_07-053	11.09	198.98	186.37	144	170	1,802	3,217	416	241	349
Myr_07-054	94.82	213.70	104.48	140	165	1,234	2,203	409	283	320
Myr_07-055	58.83	179.45	99.94	107	126	1,105	1,973	323	185	235
Myr_07-056	2.29	211.87	185.60	113	133	1,228	2,192	433	204	270
Myr_07-057	139.78	199.18	53.85	126	149	1,829	3,265	478	207	319
Myr_07-058	169.75	237.71	67.96	125	147	871	1,555	452	199	253
Myr_07-059	133.18	220.01	86.83	169	199	1,721	3,072	412	294	381
Myr_07-060	66.32	240.54	154.34	116	137	1,378	2,460	424	212	281
Myr_07-061	27.00	217.99	190.7	110	130	1,283	2,290	392	208	263
Myr_07-062	113.50	206.64	42.99	126	149	1,609	2,872	471	265	332
Myr_07-064	140.67	193.16	48.95	116	137	1,048	1,871	351	218	255
Myr_07-065	79.75	177.43	77.70	157	185	1,854	3,310	472	265	381
Myr_07-084	142.53	212.58	70.01	119	140	1,448	2,585	372	191	279
Myr_07-085	148.32	230.31	77.09	102	120	806	1,439	329	200	204
Myr_07-109	109.44	218.51	109.08	170	200	1,632	2,913	402	301	390
Myr_08-067	124.69	177.56	52.81	149	176	1,624	2,899	512	257	351
Myr_08-068	83.69	175.84	92.15	143	169	1,176	2,099	486	229	280
Myr_08-069	122.39	188.73	66.34	158	186	1,378	2,460	451	271	341
Myr_08-070	197.10	223.51	26.41	157	185	1,571	2,805	446	289	362
Myr_08-071	135.92	200.29	64.37	192	226	1,624	2,899	432	312	407
Myr_08-073	126.22	188.20	61.38	172	203	1,671	2,983	413	324	423
Myr_08-075	47.43	223.67	164.58	140	165	1,391	2,483	506	267	308
Myr_08-076	40.36	206.66	166.30	115	136	1,446	2,581	418	204	289
Myr_08-077	12.98	207.71	194.67	156	184	1,598	2,853	478	270	358
Myr_08-078	29.22	233.01	203.79	127	150	1,137	2,030	398	229	296
Myr_08-079	6.57	212.88	194.61	99	117	1,049	1,873	411	157	227
Myr_08-086	138.63	226.26	87.63	119	140	870	1,553	370	174	245
Myr_08-087	164.98	215.39	50.42	147	173	1,250	2,232	472	241	316

TABLE 6.3 **SUMMARY OF MINERALIZED INTERVALS IN 2006 TO 2008 DRILLING PROGRAMS**										
Drill Hole Number	**From (m)**	**To (m)**	**Length (m)**	**U (ppm)**	**U_3O_8 (ppm)**	**V (ppm)**	**V_2O_5 (ppm)**	**Zn (ppm)**	**Mo (ppm)**	**Ni (ppm)**
Myr_08-095	150.79	217.84	67.01	122	144	1,161	2,073	472	209	291
Myr_08-096	212.64	214.99	2.35	172	203	1,600	2,856	408	296	393
Myr_08-102	26.01	159.57	86.45	86	101	773	1,380	558	143	204
Myr_08-103	67.37	191.75	110.85	91	107	815	1,455	340	182	197
Myr_08-104	50.6	221.41	148.71	118	139	1371	2,448	341	196	271
Myr_08-105	14.07	193.58	133.08	90	106	1017	1,816	273	171	222
Myr_08-106	87.79	220.82	133.03	91	107	815	1,455	315	200	192
Myr_08-107	17.70	147.11	128.20	78	92	903	1,612	227	139	197
Myr_08-112	48.27	209.72	113.02	128	151	1,631	2,912	372	218	324
Myr_08-113	6.81	217.15	210.15	154	182	1,958	3,495	455	266	396
Myr_08-114	4.23	193.59	189.29	161	190	1,844	3,292	472	263	377
Myr_08-115	5.60	202.77	197.17	156	184	1,762	3,146	443	274	368
Myr_08-116	118.29	202.43	63.12	113	133	1,450	2,589	372	215	287
Myr_08-117	6.67	181.58	174.91	117	138	1,111	1,983	329	221	245
Myr_08-118	8.26	158.69	141.64	104	123	744	1,328	269	162	190
Myr_78-005	16.45	214.09	197.64	176	208	1,531	2,733	NS	307	NS

Source: *Harron et al. (2009)*

Drill core logging indicates that only four bedrock lithologies were encountered in the drilling. In the uppermost structural position, a unit of mixed grey to black fine-grained shales occur (unit 1). Underlying the shale is a light to medium grey coloured limestone (unit 2), which overlies the Alum Shale. The Alum Shale (unit 3) underlies the limestone and consists of aphanitic black fissile shale partly converted to coal. Pyrite aggregates measuring a few mm to 10 cm with limestone interlayers measuring cm to 0.9 m and thin deformed calcite veins are common in the shale. Locally, dark grey fine-grained limestone bodies resembling reefs (stink stone) occur within the Alum Shale (unit 3). The base of the Alum Shale is recorded where massive medium grey mudstone becomes the dominant lithology (unit 4). All analytical data are derived from samples of Alum Shale (unit 3), since the other lithologies are not mineralized.

As a consequence of the tectonic evolution of the area, the rocks are highly brecciated, deformed and foliated, which obliterated features such as pre-existing faults and primary depositional structures.

An interpretation of the drill results indicated that the most metalliferous and thickest portion of the outcropping Alum Shale trends north-northwest. Maximum thickness is approximately 200 m and the width of the mineralized zone is approximately 1,000 m.

6.2.3.2 2010 to 2012 Drilling Programs

From 2010 to 2012, CPM completed 19 drill holes totalling 2,603 m at the Viken Deposit and surrounding exploration licences.

Drill collars were initially sited with the aid of a handheld GPS and later surveyed with a high precision Topcon Hiper Network RTK GPS by Mätservice i Jämtland AB, when the drill holes were completed and casings capped. Downhole surveys were not undertaken, due to the short length of the drill holes and the friable nature of the Alum Shale.

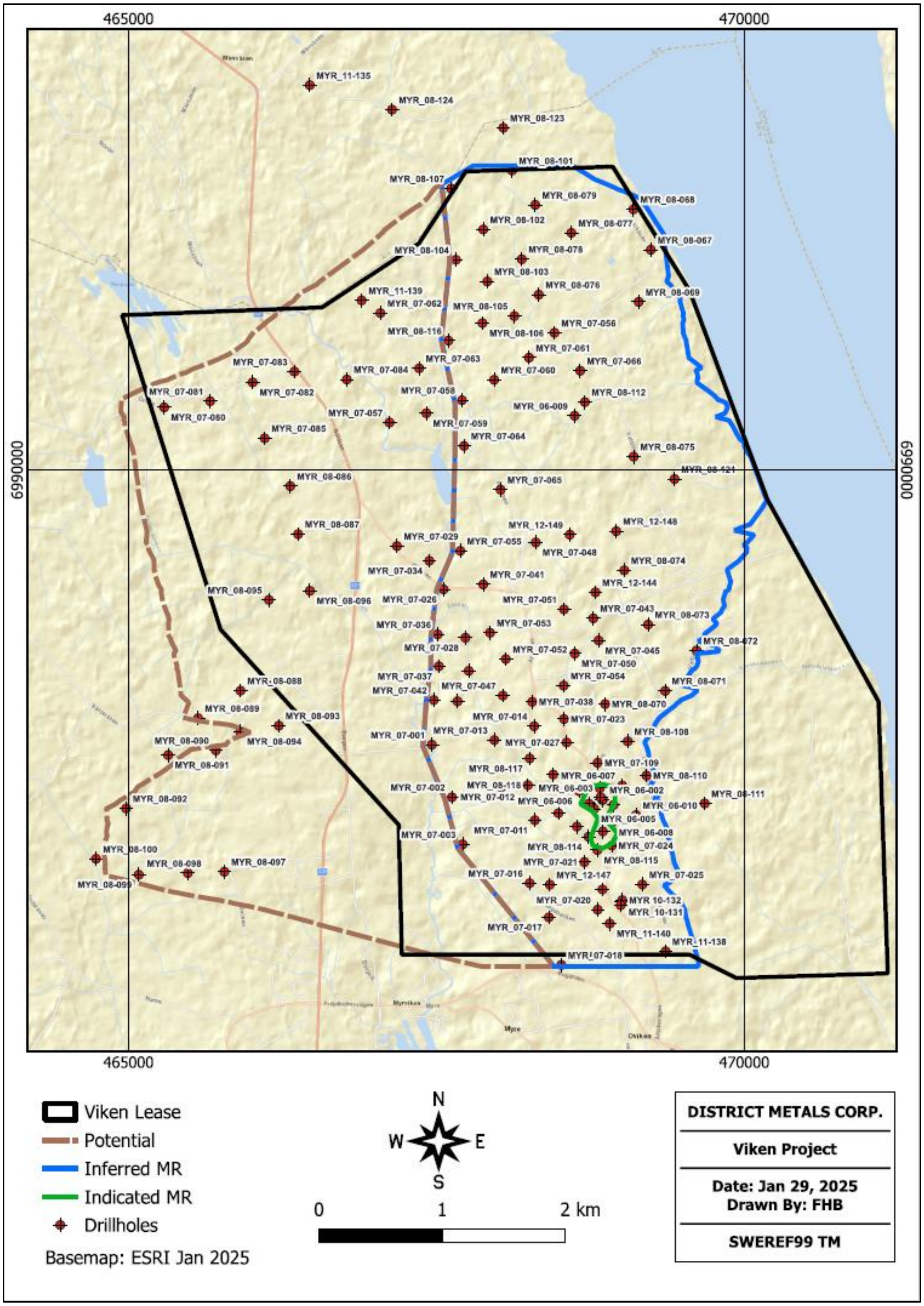
Diamond drilling from 2010 to 2012 was completed by T.G.B. Borrteknik AB of Gråbo and Ludvika Borrteknik HB of Ludvika. Drill core diameter was BQ-sized for all holes.

The drill holes completed in 2010 were shallow vertical holes undertaken to test the near-surface black shale. Those completed in 2011 were to supply samples for tests and analytical work.

Drill holes MYR_11-140, MYR_11-141 and MYR_11-142 were metallurgical drill holes and remaining drill holes were exploration drill holes and infill drill holes located in the North and South Pit areas as defined in the 2010 PEA. Drill holes MYR_12-143 and MYR_12-149 were completed 600 and 850 m, respectively, northeast of the proposed 2010 PEA pit limits, and returned significant mineralized intervals. Drill holes MYR_10-137 and MYR_11-143, completed 9.5 and 8.0 km, respectively, north of the 2010 Mineral Resources, also intersected significant mineralization.

The CPM 2010 to 2012 drill hole locations, in addition to those from 2006 to 2008, in the Viken Energy Metals Deposit area are shown in Figure 6.3. CPM 2010 to 2012 drill hole locations and details are listed in Table 6.4. The assay highlights from the CPM 2010 to 2012 drilling programs are summarized in Table 6.5. A drill hole location surface plan is included in Appendix A.

FIGURE 6.3 LOCATION OF THE 2006 TO 2012 DRILL HOLES ON THE VIKEN DEPOSIT



Source: *District Metals (2025)*

TABLE 6.4
2010 TO 2012 DRILL HOLE LOCATIONS AND DETAILS

Drill Hole ID	Easting (m)	Northing (m)	Elev-ation (masl)	Length (m)	Azimuth (°)	Dip (°)
MYR_10-131	469,004	6,986,493	369.99	203.13	360	-90
MYR_10-132	469,010	6,986,508	370.26	62.85	360	-90
MYR_10-133	468,997	6,986,467	367.54	210.30	360	-90
MYR_11-134	468,127	6,978,132	297.17	36.60	360	-90
MYR_11-135	466,472	6,993,124	334.29	26.65	360	-90
MYR_11-136	463,812	6,994,687	346.95	101.35	360	-90
MYR_11-137	466,344	6,998,119	318.97	114.05	360	-90
MYR_11-138	469,362	6,986,092	371.06	70.90	360	-90
MYR_11-139	466,893	6,991,377	334.58	183.60	360	-90
MYR_11-140	468,908	6,986,318	358.84	222.40	360	-90
MYR_11-141	468,849	6,986,595	357.04	199.50	360	-90
MYR_11-142	468,738	6,987,291	354.69	190.70	360	-90
MYR_11-143	465,117	6,996,089	317.39	183.60	360	-90
MYR_12-144	468,792	6,989,007	356.37	213.25	360	-90
MYR_12-145	459,772	6,986,341	359.00	88.45	360	-90
MYR_12-146	460,448	6,992,406	379.00	51.10	360	-90
MYR_12-147	468,422	6,986,634	340.10	89.75	360	-90
MYR_12-148	468,961	6,989,503	353.12	126.00	360	-90
MYR_12-149	468,584	6,989,475	359.02	229.20	360	-90

TABLE 6.5
SIGNIFICANT MINERALIZED INTERVALS FROM THE 2010 TO 2012 DRILLING PROGRAMS

Drill Hole ID	Target	From (m)	To (m)	Thick-ness (m)	U (ppm)	U_3O_8 (ppm)	V (ppm)	V_2O_5 (ppm)	Mo (ppm)	Ni (ppm)	Zn (ppm)	Cu (ppm)
MYR_10-131	Infill drilling	17.00	182.00	165.00	194	229	1,661	2,965	353	434	446	135
MYR_10-136	Northern extension of Viken Deposit	38.70	81.80	43.10	154	182	2,256	4,027	263	419	541	122
MYR_10-137	Northern extension of Viken Deposit	7.90	34.30	26.40	201	237	1,652	2,949	318	447	528	130
MYR_10-139	Northern extension of Viken Deposit	31.50	180.90	149.50	168	198	1,729	3,087	286	387	428	118
MYR_11-140	Metallurgical drill hole from 2010 PEA Pit	4.80	217.70	213.00	161	190	1,959	3,497	294	406	542	138
MYR_11-141	Metallurgical drill hole from 2010 PEA Pit	4.60	191.40	186.80	153	180	1,834	3,274	270	382	545	137
MYR_11-142	Metallurgical drill hole from 2010 PEA Pit	2.00	170.00	168.00	158	186	2,073	3,701	260	398	482	482
MYR_11-143	Northern extension of Viken Deposit	35.30	80.00	44.70	168	198	1,450	2,589	260	371	488	123
MYR_12-144	Northern extension of Viken Deposit	33.10	199.00	165.90	202	238	1,559	2,783	330	438	494	128
MYR_12-149	Northern extension of Viken Deposit	48.50	219.40	171.00	181	213	1,611	2,876	291	394	400	116

Note: *True thicknesses are estimated to be >95% drilled widths.*

6.3 RECENT HISTORICAL MINERAL RESOURCE ESTIMATES

Historical Mineral Resource Estimates of the Viken Deposit were made in 2007, 2008, 2009, 2010, and 2014. Further to the Company press release dated April 29, 2025, the 2010 and 2014 historical Mineral Resource Estimates are summarized in Tables 6.6 and 6.7.

TABLE 6.6
2010 HISTORICAL MINERAL RESOURCE ESTIMATE (1-8)

Classification	Tonnes (k)	Grade				Contained Metal			
		V_2O_5 (%)	U_3O_8 (%)	Mo (%)	Ni (%)	V_2O_5 (Mlb)	U_3O_8 (Mlb)	Mo (Mlb)	Ni (Mlb)
Indicated	23,610	0.313	0.019	0.028	0.032	162.8	9.9	14.7	16.5
Inferred	2,830,757	0.268	0.017	0.024	0.032	16,716	1,038	1,517	2,016

Source: *Puritch et al. (2010)*
Notes *for Table 6.6:*
1. *The Company views the 2010 Viken Report as relevant and reliable;*
2. *Weighting of composite samples by linear Ordinary Kriging was used for the estimation of block grades. Kriging parameters were based on the grade-element variography derived from the mineralized shale domain. A block discretization level of 5 x 5 x 2 m was used during kriging. The mineralized shale domain was treated as a hard boundary, and data used during estimation were limited to composite samples located within the mineralized shale domain wireframe. Only blocks wholly or partially within the mineralized shale domain were estimated.*
3. *During the first pass, four samples from each of three drill holes within 110 m of the block centroid were required. All block grades estimated during the first pass were classified as Indicated;*
4. *During the second pass, blocks not populated during the first pass were estimated. A minimum of three and a maximum of six samples from one or more drill holes within 330 m of the block centroid were required. All block grades estimated during the second pass were classified as Inferred;*
5. *An internal break-even cut-off grade of US$7.50/t was used in reporting this historical estimate;*
6. *The historical estimate classification used the previous definition standards and do not match the current classification used for NI 43-101;*
7. *The Company would need to complete an exploration program that includes twinning of historical drill holes, in order to verify the Viken Deposit historical estimate as a current Mineral Resource; and*
8. *The mineral resource estimate is considered to be an "historical estimate" under NI 43-101 and a Qualified Person has not done sufficient work to classify the historical estimate as a current Mineral Resource and District Metals is not treating the historical estimate as a current Mineral Resource.*

Table 6.7 2014 Historical Mineral Resource Estimate (1-8)									
Class-ification	**Tonnes (k)**	**Grade**				**Contained Metal**			
		U_3O_8 (%)	**Ni (%)**	**Cu (%)**	**Zn (%)**	**U_3O_8 (Mlb)**	**Ni (Mlb)**	**Cu (Mlb)**	**Zn (Mlb)**
Indicated	43,000	0.019	0.034	0.01	0.041	18.0	32.0	10.0	38.0
Inferred	3,019,000	0.017	0.034	0.012	0.042	1,145.0	2,230.0	799.0	2,802.0

Source: *Puritch et al. (2014)*

Notes *for Table 6.7:*

1. *The Company views the 2014 Viken Report as relevant and reliable;*
2. *Block grades were estimated using Ordinary Kriging of capped composite samples. Only blocks wholly or partially within the mineralized shale domain were estimated, and between six and fifteen samples from two or more drill holes within 660 m of the block centroid were used for estimation. A small area in the southern portion of the Deposit with an average drill hole spacing of approximately 120 m has been classified as an Indicated Mineral Resource;*
3. *An internal break-even cut-off grade of US$11.00/t was used in reporting this historical estimate;*
4. *The historical estimate was classified under the previous definition standards and do not match the current classification under NI 43-101;*
5. *The Company would need to complete an exploration program that includes twinning of historical drill holes, in order to verify the Viken Deposit historical estimate as a current Mineral Resource; and*
6. *The Mineral Resource Estimate is considered to be an "historical estimate" under NI 43-101, and a Qualified Person has not done sufficient work to classify the historical estimate as a current Mineral Resource and District Metals is not treating the historical estimate as a current Mineral Resource.*

The Mineral Resource Estimates presented in these tables are each considered to be a "historical Mineral Resource" under NI 43-101. A Qualified Person has not done sufficient work to classify these historical Mineral Resources Estimates as current Mineral Resource Estimates and District Metals is not treating the historical Mineral Resource Estimates as current Mineral Resource Estimates. The current Mineral Resource Estimates are presented in Section 14 of this Report.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The information in this section is summarized mainly from Harron *et al.* (2009) and Anderson *et al.* (1985).

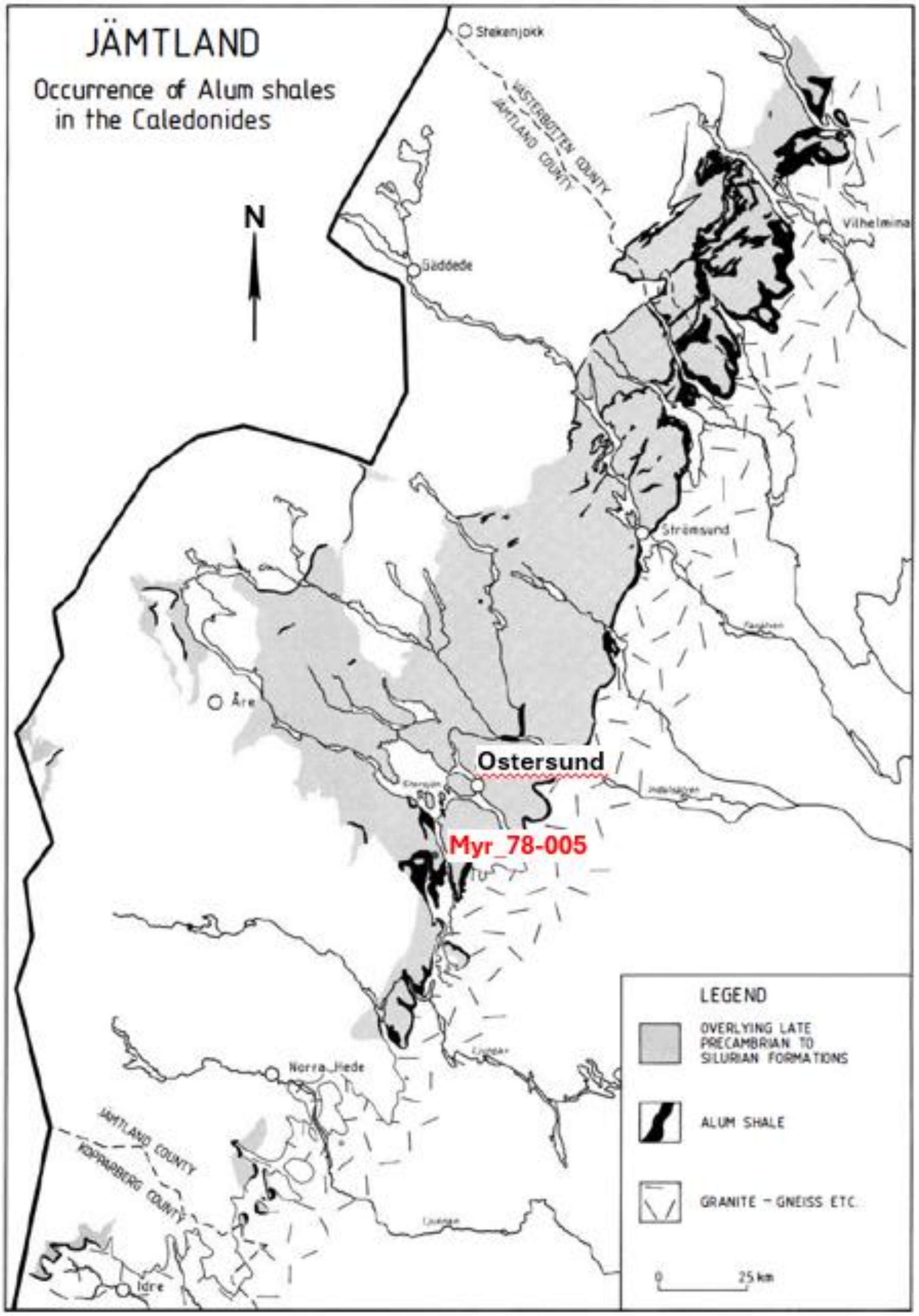
7.1 REGIONAL GEOLOGY

Archean age rocks from the Fennoscandian Shield age dated at between 2.8 and 2.6 Ga outcrop in northern Sweden. Paleoproterozoic age rocks dated at between 2.0 and 1.65 Ga overlie the Archean rocks and outcrop in the northern, western and southern parts of Sweden. Deformation and metamorphism accompanying the Svecokarelian Orogeny affect both the Archean and Proterozoic age basement rocks. Rocks affected by late Svecokarelian deformation after 1,850 Ma exhibit north-northeast trending folds with crenulation cleavage overprinted on older structures. Svecokarelian deformation is also associated with northwest trending ductile transcurrent shear zones that deform earlier folds into open S and Z shapes.

During the Neoproterozoic (>570 Ma), rift basins preserved on the Baltic side of the Iapetus Ocean were filled with coarse clastic rocks prior to passive plate margin development in Cambrian times. The passive margin was characterized by shallow marine sandstones and with later bituminous Alum Shale deposited in a stable epicontinental sea. Final closure of the Iapetus Ocean during the Silurian resulted in easterly-directed thrusting of continental basinal rocks (allochthonous) ranging in age from Precambrian to Silurian onto the Baltic Shield area covering autochthonous Alum Shale and older lithologies (Figure 7.1). The Viken area is situated within allocthonous Middle and Upper Cambrian and Ordovician rocks in the foreland of the Caledonide Mountains, near Östersund and around historical drill hole Myr_78-005.

On a regional scale, lithofacies development in Cambrian age rocks is similar throughout northern, central and southern Sweden. Lower Cambrian sandstones underlie phosphorite-bearing conglomerates, which in turn are overlain by Middle Cambrian age grey-green siltstones. Upper Middle, Upper Cambrian and lower Ordovician Alum Shale overlie this sedimentary rock sequence.

FIGURE 7.1 REGIONAL GEOLOGY MAP



Source: *Modified by P&E (2025) from Anderson et al. (1985)*

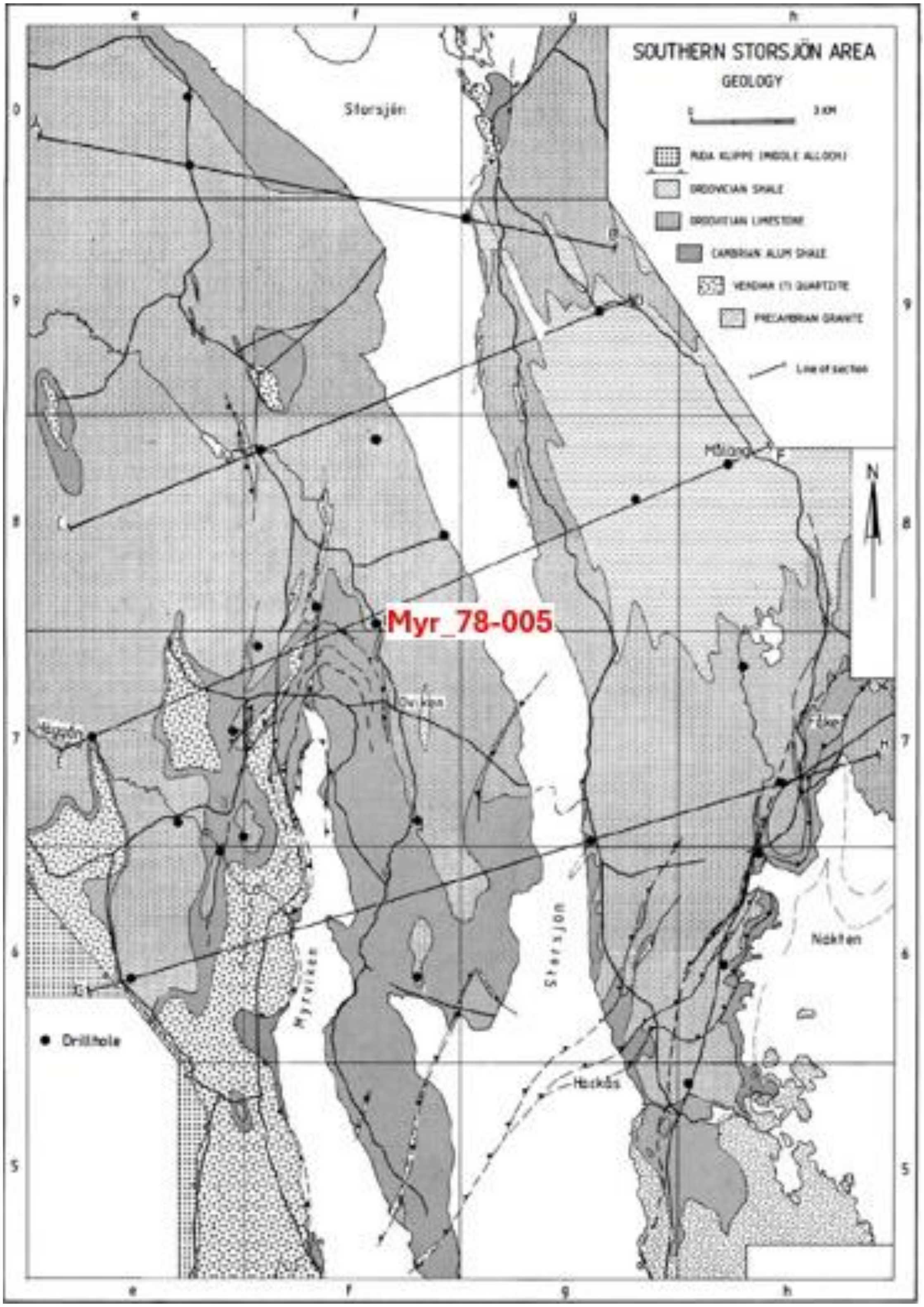
Figure 7.1 Description*: Distribution of the Alum Shales in western Jämtland, northernmost Kopparberg and southernmost Västerbotten Counties. Areas are also marked where the alum shales are covered by Ordovician and Silurian strata and (or) thrust sheets of Upper Proterozoic/Lower Cambrian sandstones and quartzites. The Alum Shales probably also occur farther west at greater depths beneath the higher nappes.*

7.2 PROPERTY GEOLOGY

The Viken Property stratigraphy consists of upper Middle and Upper Cambrian age black shales interlayered with subordinate quartzites, limestones and bituminous limestone (‘stinkstones’). This assemblage is overlain in parts of the licence area by Ordovician limestones (Figure 7.2). The black shales are generally underlain by Proterozoic granites and gneisses thrust eastward over Archean granitic basement rocks. The shales occur as both autochthonous (in situ) and allochthonous (fault detached) blocks, the latter having greater potential for economic mineralization, due to imbrication of mineralized blocks (Figure 7.3). The thickness of the Upper Cambrian black shale stratigraphy hosting the syngenetic deposits has been increased tectonically from a general 20 to 30 m by Silurian thrusting and folding to approximately 180 m near the village of Myrviken. The allochthonous blocks can be subdivided into those belonging to the Middle Cambrian and Upper Cambrian, with the higher grades of mineralization occurring in the Upper Cambrian blocks.

Structurally the Viken Property is situated on the eastern flank of a major north-northwest trending anticline, which is manifested by a broad ridge trending in the same direction. The Alum Shale exhibits a strong penetrative foliation and has been subjected to metamorphic temperatures of 200 to 300°C, converting black shale into semi-anthracitic to anthracitic grade “coal”.

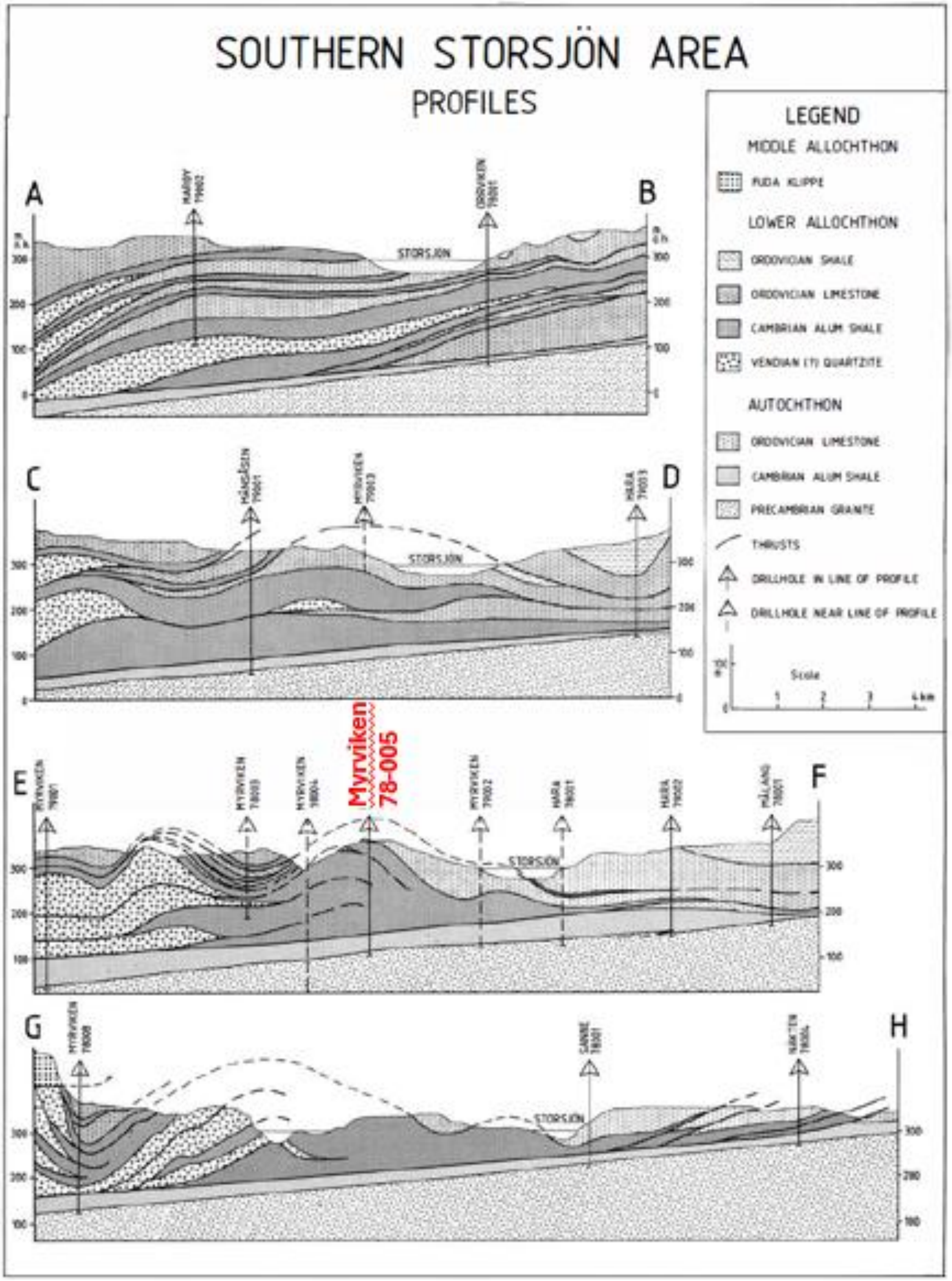
FIGURE 7.2 LOCAL GEOLOGY OF THE VIKEN POLYMETALLIC AREA



Source: *Modified by P&E (2025) from Anderson et al. (1985)*

Figure 7.2 Description*: Bedrock geology of the area of southern Storsjön, Jämtland (based on Gee et al. 1982, unpublished report). The lines A-B , C-D , E-F and G-H mark the location of the cross sections in Figure 7.3.*

FIGURE 7.3 GEOLOGICAL CROSS SECTIONS OF THE VIKEN POLYMETALLIC AREA



Source: *Modified by P&E (2025) from Anderson et al. (1985)*

Figure 7.3 Description*: Cross sections through the southern Storsjön area, Jämtland, (taken from Gee et al. 1982 unpublished report) illustrating the thick development of the Alum Shales where they are tectonically repeated in the drill core of the Myrviken Antiform. Note that the allochthonous cover of sedimentary rocks is thrust over a passive largely undisturbed autochthon of Cambrian shales (locally with Lower Ordovician limestones) and underlying Precambrian crystalline rocks.*

7.3 DEPOSIT GEOLOGY AND MINERALIZATION

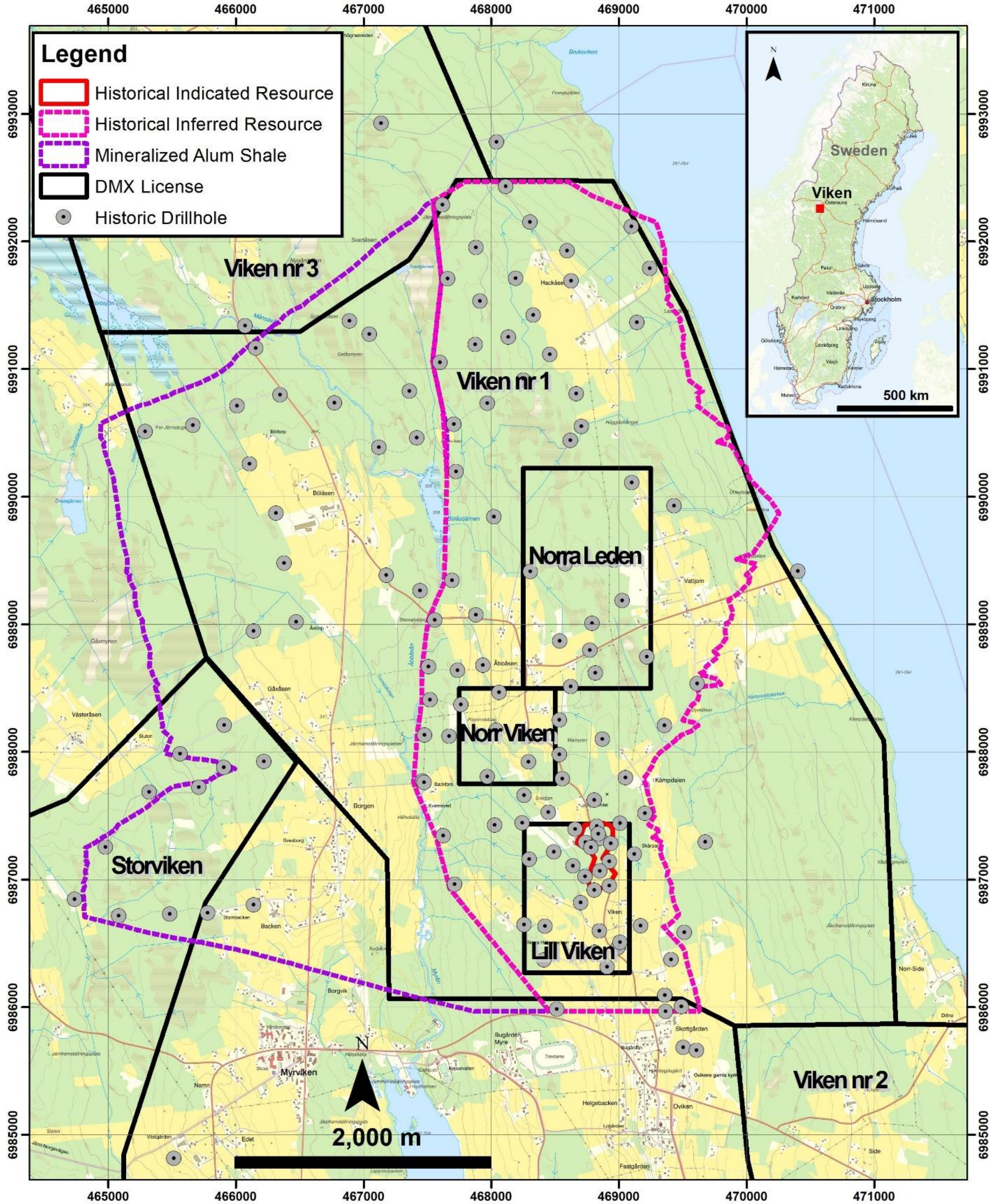
Diamond drill information suggests that a broadly north-trending zone >1,000 m wide of mineralized Alum Shale traverses the entire Viken Property (Figure 7.4). Visual examination of drill core does not reveal the mode of occurrence of the mineral species associated with analytical results, which reflects the extremely fine-grained nature of the mineralization.

Mineralization of potential economic significance is hosted in black shales of the Alum Shale, with the Upper Cambrian age strata more enriched in uranium and vanadium than the Middle Cambrian (Andersson *et al.,* 1985). Locally the black shale contains up to 15% organic carbon. Higher uranium contents are associated with organic C contents >10%. The association of U and C is demonstrated in drill hole Myr_78-005 (Figure 7.5).

Uranium values are predominantly associated with sub-µm size uraninite (UO_2) crystals, the result of precipitation of uranium from aqueous uranium complexes entering the chemically reducing environment of the Alum Shale. Some uranium is reported to be present as organo-metal complexes (Andersson *et al.*, 1985).

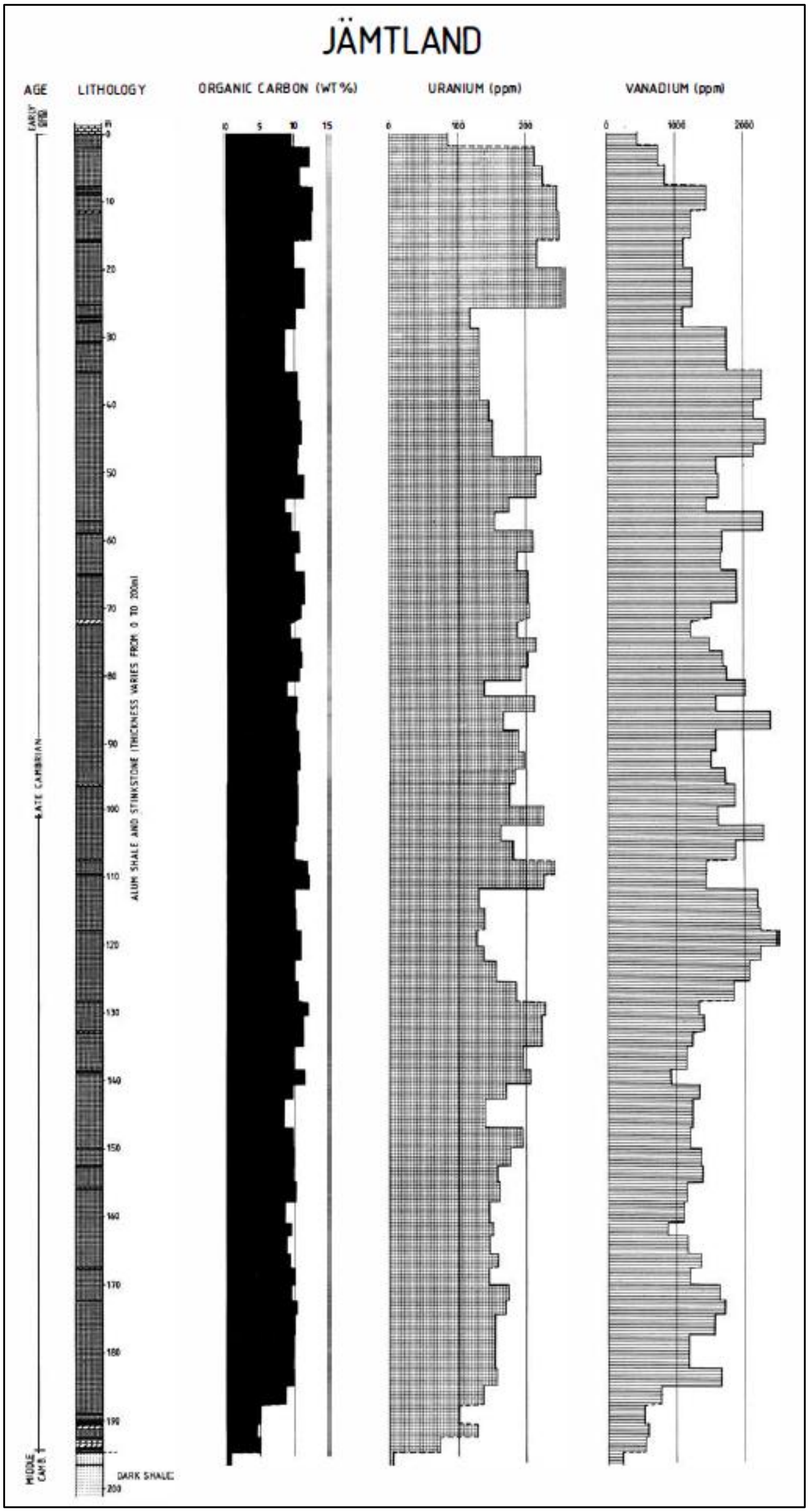
Vanadium, molybdenum and nickel mimic the abundance of uranium, suggestive of similar modes of occurrence. Vanadium occurs within the lattice of the mica mineral roscoelite. Nickel, molybdenum, copper and zinc are present as sulphides.

FIGURE 7.4 PLAN MAP SHOWING THE EXTENT OF THE MINERALIZED ALUM SHALE IN THE VIKEN LICENSE AREA



Source: *District Metals (June 2026)*
Note: *The round symbols represent drill hole locations.*

FIGURE 7.5 C, U AND V DISTRIBUTION IN DRILL CORE MYR_78-005 FROM THE ALUM SHALE



Source: *Anderson et al. (1985)*

8.0 DEPOSIT TYPES

The following information is derived largely from Harron *et al*. (2009).

Black shales comprise a widely distributed unique rock suite, particularly in pre-Jurassic time. Many black shales are enriched in various metals and hydrocarbons (Swanson, 1961; Vine and Tourtelot, 1970; Grauch and Leventhal, 1989; Johnson *et al*., 2017). The metals suite includes Cu, Ni, Zn, Pb, Ag, Zn, Co, Mo, Re, V, U, Se, Sn, Bi, Au, Pt, Pd, etc.) with variable proportions of kerogen and sulphide phases. Nearly all black shales contain distinctive fossils (Bulman, 1955), lack signs of bioturbation, and are characteristically thinly laminated. Most black shales accumulated in oxygen-poor to anoxic environments (Pettijohn, 1949). The mineralogical composition of the black shale sedimentary rock records the provenance of the rock. The abundance of quartz, feldspars and clay minerals records a granitic source area (Snäll, 1988). The suites of metals contained in black shale are also related to the chemical composition and weathering history of the source area. Concentrations of U, V, and Mo reflect a granitoid source area, such as the Baltic Craton. Metals liberated during weathering of the source terrain are transported as oxidized species in surface and ground water into anoxic environments. Reducing conditions cause the metals to bond with organic matter or, in the case of uranium, precipitate as uraninite.

The rate of sedimentation probably exercised an important control on the concentration of some trace elements. In the Ranstad area of Sweden, an inverse relationship exists between uranium content and shale thickness in the Upper Cambrian facies (Andersson *et al.*, 1985). The organic carbon content of the Alum Shale generally shows a sympathetic relationship with the concentrations of uranium, vanadium, molybdenum and nickel, which indicates that organic carbon sequesters some of the metals as organo-metallic complexes.

Metal contents of all species present in black shale are rather uniform over large distances, which reflects the large and rather uniform composition of the source areas. Vertical variations in metal species within the shale reflects the opening and closing of the Iapetus Ocean with a concomitant change in chemical composition of the source areas.

Not all black shales contain the same suite of anomalous metals and (or) kerogen contents. Quinby-Hunt *et al*. (1989) included analyses of Upper Cambrian black shales from the Alum Shale in Sweden and Norway in an update of the compilation by Vine and Tourtelot (1970). This revised compilation of metal contents in an "average" black shale is compared to the metal contents of Alum Shale in the Viken Project area (Table 8.1). The comparison data suggest that the Alum Shale in the Jämtland area may be relatively enriched in U, V, and Mo.

TABLE 8.1 COMPARISON OF THE CHEMISTRY OF JÄMTLAND BLACK SHALE AND AVERAGE BLACK SHALE				
Element	**Uranium-Rich Black Shale Southern Storsjön Jämtland[1]**	**Average Black Shale Composite[2]**		**Average Black Shale No. Samples**
		Mean	**Mode**	
Al (%)	6.3	8.21	7.0	287
Fe (%)	3.4	3.68	5.0	287
Mg (%)	0.58	1.04	1.0	284
Ca (%)	2.3	1.71	0.26	134
Na (%)	0.06	0.53	0.7	287
K (%)	3.8	2.99	2.0	286
Ti (%)	0.39	0.43	0.50	286
Mn (%)	0.25	0.04	0.02	287
C_{org} (%)	14.2	n.a.	n.a.	
U (ppm)	245	15.2	3.0	287
V (ppm)	1600	500	150	187
Mo (ppm)	460	65	1.8	118
Ni (ppm)	440	n.a.	n.a.	

Source: *Harron et al. (2009)*
Notes: *[1] Gee and Snäll (1981); [2]Quinby-Hunt et al. (1989); C org = organic carbon; n.a. = not analyzed.*

The Alum Shale is regionally extensive in Sweden and has been previously mined for U at the Ranstad Deposit in the Mount Billingen area of southern Sweden. At Mount Billlingen, Laznicka (2010) estimates the Alum Shales average 292 ppm U.

Metalliferous black shales are common throughout the world in Cambrian-Ordovician and Silurian marine sedimentary sequences. A notable example is the “Kupferschiefer”, a lithological formation that extends from England to Poland, where exploitable Cu reserves represent only 0.2% of the total area, notably at the southern edge of the Zechstein Basin. To date, >2 Mt of copper have been produced from this geological formation along with minor amounts of noble and rare metals.

Black shales other than the Alum Shale, that have received considerable attention in the past as possible hydrocarbon sources, include the Mancos Shale Formation of the Green River Basin in Colorado and Utah, U.S.A.

Another global example of a metal-rich black shale is the Ogcheon Fold Belt in South Korea (Chi and Yun, 2006). Several low-grade uranium deposits near Geosan, Boeun and Geumsan, with a maximum content of 300 to 400 ppm U_3O_8 were discovered by drilling. Uranium-bearing black shale consists of quartz, micas, graphite, calcite dolomite, muscovite, vanadium-bearing mica, chlorite barite, barium-rich feldspars, apatite, iron oxides, sulphides, and uranium minerals.

In these deposits, uranium occurs mainly as secondary minerals (autunite, meta-autunite, torbernite, and francevillite) in the form of scattered grains and thin films along fractures in "coaly" beds. Primary uraninite is rare and occurs as very fine-grained disseminations and associated with organic materials.

Black shale deposits are stratigraphic formations that cover very large areas and are not necessarily exposed at ground surface. Deposits located at or near the present-day ground surface are preferred since they are associated with more favourable exploitation costs. Therefore, shallow diamond drilling is the most effective method of exploration when used together with stratigraphic and structural analyses of the host geology.

9.0 EXPLORATION

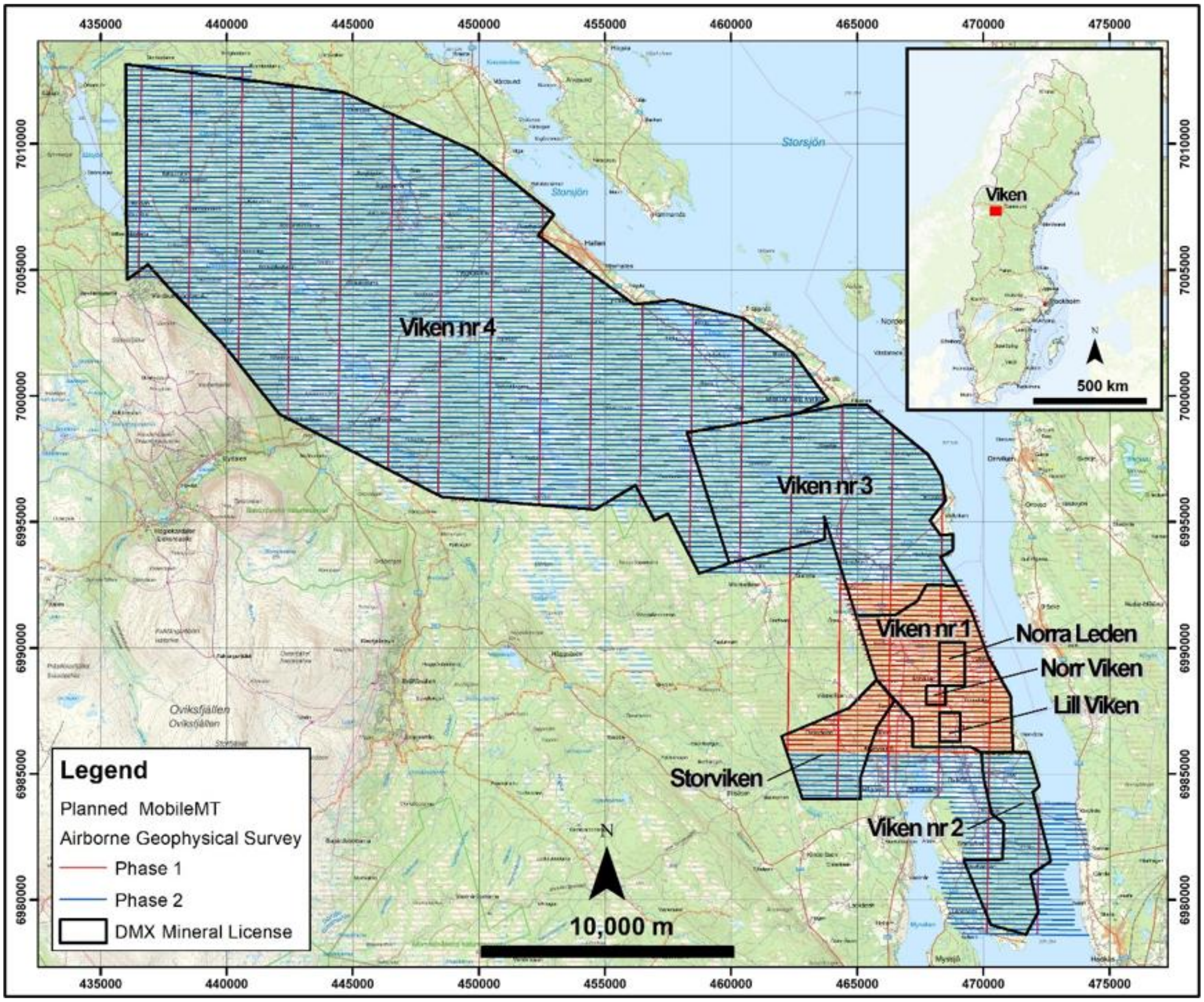
District Metals completed an airborne MobileMT survey over the Viken Property in May-June 2025, which is summarized below. Exploration completed by previous operators is summarized in Section 6 of this Report.

9.1 AIRBORNE GEOPHYSICS

A high-resolution airborne MobileMT Survey was flown over the entire Viken Property by Expert Geophysics Limited in May-June 2025, with the aim to identify potential mineralized Alum Shale for follow-up confirmation by drilling. The MobileMT survey covered ~2,350-line km at 200-m spacing and was completed in two phases. Phase 1 captured the low resistivity signature of the mineralized Alum Shale that hosts the Viken Deposit, and Phase 2 covered the remainder of the Viken Property and identified additional low resistivity signatures resembling those of the Viken Deposit (Figure 9.1). High-priority target areas were selected by the District Metals based on the shallow depth and thickness of the mineralized Alum Shale (Figures 9.2 and 9.3). The relatively flat-lying Alum Shale is rich in graphite and sulphides, making it low resistivity (high conductivity) and easily detectable using the airborne MobileMT system.

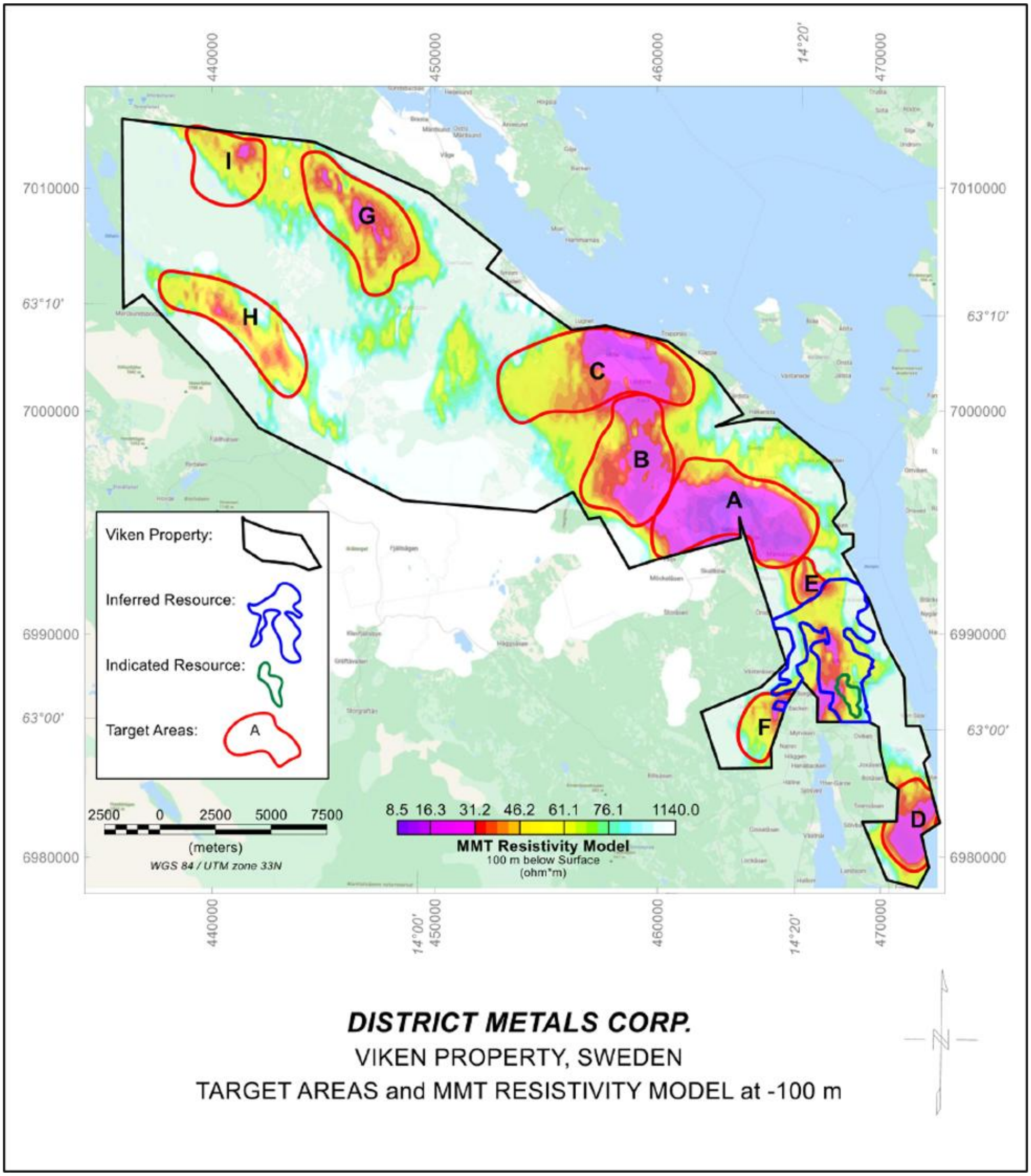
In all, nine target areas were identified outside of the Viken Deposit. These target areas are labelled A to I in Figure 9.2. Three of the target areas, A to C, show conductivity responses that are larger and stronger than those observed at the Viken Deposit itself. With these results, the Company plans to prioritize the targets for drill testing in 2026.

FIGURE 9.1 AIRBORNE MOBILEMT SURVEY FLIGHT LINES



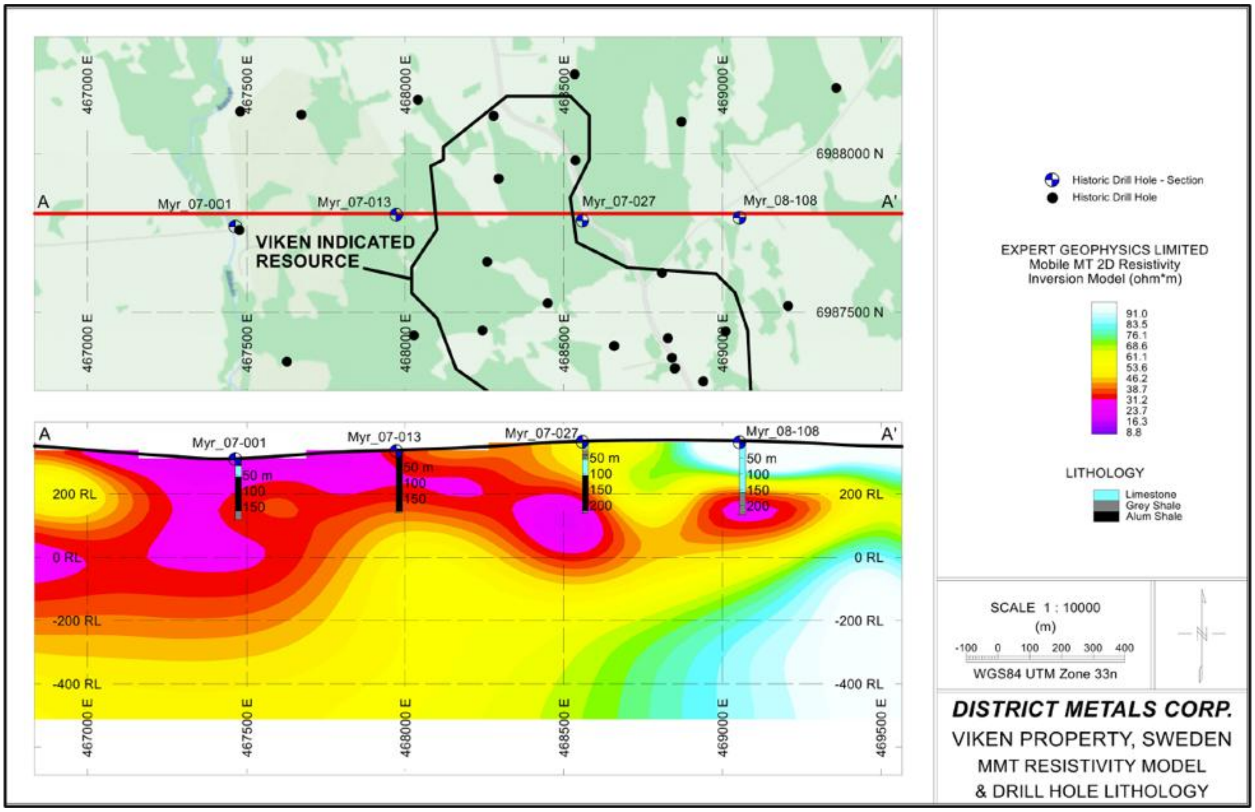
Source: *District Metals (June 2025)*

FIGURE 9.2 PLAN VIEW OF VIKEN MOBILEMT SURVEY RESULTS



Source: *District Metals press release dated September 24, 2025*

FIGURE 9.3 CROSS-SECTIONAL VIEW OF THE MOBILEMT SURVEY RESULTS



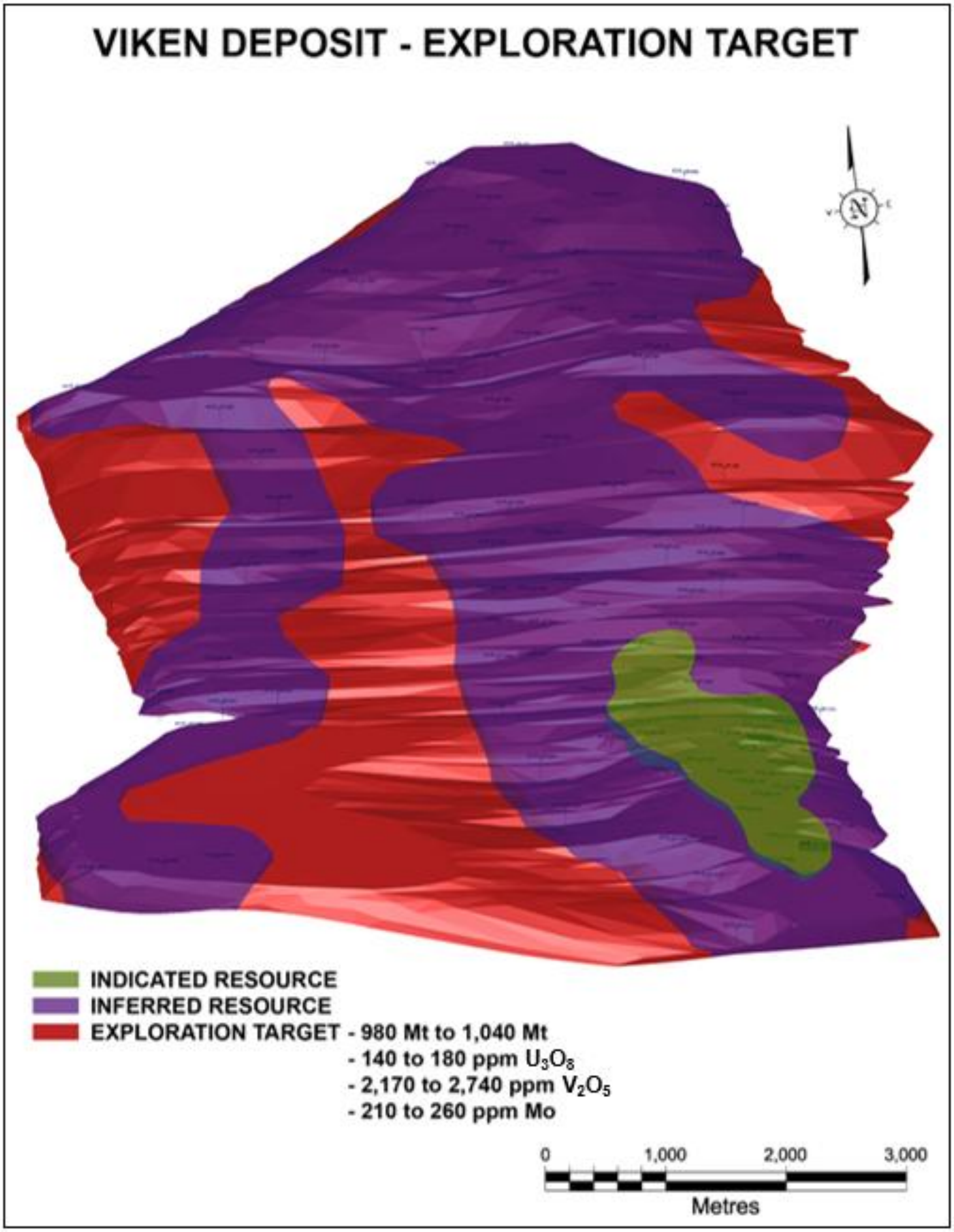
Source: *District Metals press release dated September 24, 2025*

9.2 TARGETS FOR FURTHER EXPLORATION

The Author established that the Viken Deposit contains Targets for Further Exploration with a potential range of 980 Mt to 1,040 Mt at grade ranges of 140 to 180 ppm U_3O_8, 2,170 to 2,740 ppm V_2O_5, and 210 to 260 ppm Mo. These Targets for Further Exploration are based on the estimated strike length, depth and width of the mineralization, as supported by intermittently-spaced drill holes and observations of mineralized outcrops. The Targets for Further Exploration are located adjacent to the margins of the current Mineral Resources (Figure 9.4).

The potential quantities and grades of the Targets for Further Exploration are conceptual in nature. There has been insufficient work done by a Qualified Person to define these estimates as Mineral Resources. The Company is not treating these estimates as Mineral Resources, and readers should not place undue reliance on these estimates. Even with additional work, there is no certainty that these estimates will be classified as Mineral Resources. In addition, there is no certainty that these estimates will prove to be economically recoverable.

FIGURE 9.4 VIKEN EXPLORATION TARGET



Source: *P&E (2025)*

10.0 DRILLING

District Metals has not completed any drilling on the Viken Property. All drilling was completed by historical operators, mainly CPM, and is described in Section 6 of this Report.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following section discusses drill core sampling carried out by CPM at the Viken Property from 2006 to 2012.

11.1 SAMPLE PREPARATION AND SECURITY

Drill core samples were logged, sawn and sampled by CPM personnel from 2006 to 2012, except for the initial stages of the 2006 drill program when drill core splitting was contracted to Taiga Exploration Drilling AB.

All drill core logging and drill core splitting was completed in a secure drill core handling facility. The distance between the depth markers added by the drill personnel was measured to check for misplaced markers and for lost drill core. All logging information was recorded directly onto paper logging forms. Drill core intervals identified for sampling were marked with wax crayons and sample tags were placed at the start of a sample interval. The entire Alum Shale intercept was sampled. Sample lengths ranged from 0.19 to 10.4 m, with an average of 2 m, and individual samples were taken such as to not cross lithologic contacts or abrupt changes in mineralization. Drill core was quarter split using a diamond saw. Where possible, contiguous sample tag series were used for drill core logging. Sample intervals were recorded on sample tag books, marked on drill core boxes, and subsequently recorded in computerized drill logs. The quarter-split drill core samples were then placed into plastic sample bags, sample numbers were recorded, shipping bags were sealed, and shipment particulars noted.

Security of the drill core samples was maintained by using a secure drill core handling facility located on fenced private land. Samples awaiting dispatch to the assay laboratory were stored in this secured building and accessible only to authorized personnel. Individual samples were placed in shipping bags, which in turn were sealed with plastic tie straps and the shipping bags remained sealed until they were opened by assay laboratory personnel.

11.2 SAMPLE PREPARATION AND ANALYSES

Analyses of the 2006 to 2012 samples were performed at one of the ALS laboratories in either Luleå, Öjebyn, or Piteå, Sweden. Sample management at all facilities utilized bar coding to track samples and thereby maintaining the integrity of the samples. Sample preparation procedures included crushing the entire sample to 70% passing a 9 mesh or 2 mm screen, followed by splitting of an approximately 200 to 250 g sample for pulverization to 80 or 85% passing a 200 mesh or 75 µm screen. Crushing and pulverization apparatus were cleaned between each sample with a quartz sand wash. A 5 g sample was split from the pulverized sub sample and digested in a four-acid solution to accomplish a near total dissolution.

Determination of an array of elements, including 12 major oxide elements and sulphur, was accomplished by 4-acid digest with ICP-MS finish (ME-MS61), with carbon (total) (C-IR07), carbon (organic) (C-GAS05), sulphur and "Loss on Ignition" determined by using a Leco Furnace. Carbonate carbon was determined by coulometric titration. Pulps and rejects were stored in accordance with client requests. As part of the standard protocol of the laboratory, a series of blanks, certified reference materials "CRM" and duplicates were inserted on a regular basis for quality control purposes.

ALS is independent of CPM, District Metals and the Author, and has developed and implemented strategically designed processes and a global quality management system at each of its locations. The global quality program includes internal and external inter-laboratory test programs and regularly scheduled internal audits that meet all requirements of ISO/IEC 17025:2017 and ISO 9001:2015. All ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2017 for specific analytical procedures.

Drill holes MYR-12-140, MYR-12-141, and MYR-12-142 were metallurgical drill holes that were used to evaluate the potential viability of bio heap leaching for the production of uranium, nickel, zinc and copper. Assay samples from these three drill holes were analyzed at SRC Geoanalytical Laboratories ("SRC"), in Saskatoon, SK, with samples run for multi-element geochemical analysis using a total digestion with ICP-MS finish. Inorganic carbon and organic carbon, and total sulphur content were also analyzed. SRC is an independent laboratory whose quality management system and selected methods are ISO/IEC 17025:2005 accredited by the Standards Council of Canada. The laboratory is also compliant to ASB, Requirements and Guidance for Mineral Analysis Testing Laboratories and participates in regular inter-laboratory tests for many of its package elements.

11.3 BULK DENSITY DETERMINATIONS

A total of 314 bulk density measurements from drill hole core were supplied by District Metals. The pulp samples were taken from drill core recovered in 2007 and 2008 and were submitted to ALS for bulk density determinations (OA-GRA08b method). From the pulverized samples, 3.0 g were weighed into an empty pycnometer. The pycnometer was then filled with a solvent (either methanol or acetone) and weighed. From the weight of the sample and the weight of the solvent displaced by the sample, the bulk gravity was calculated. A mean bulk density value of 2.57 t/m^3 was assumed for the mineralized shale.

Independent verification sampling of Viken drill core was undertaken during the June 2013 site visit. A total of nine due diligence samples were taken and subsequently measured independently at AGAT of Mississauga, ON, by the pycnometer method. The samples returned a mean value of 2.64 t/m^3.

11.4 QUALITY ASSURANCE/QUALITY CONTROL PROGRAM

The Quality Assurance/Quality Control ("QA/QC" or "QC") program implemented from May 2008 to 2012 involved the insertion of a composite shale reference material (called "skaelsta") that contained low uranium, and pulps sent to a secondary lab for assay checks. The number of skaelsta samples analyzed with the 2011 to 2012 drilling totalled 12, and the number of pulps sent to a secondary lab totalled 45. CPM relied mainly on ALS' internal lab CRMs, blanks and pulp duplicates for accuracy, absence of contamination and precision.

11.4.1 2006 to 2010 Quality Assurance/Quality Control

The shale reference material ("RM") was not available for use from 2006 until May 2008 and CPM relied on the internal reference materials utilized by the laboratories to monitor analytical performance. QA/QC procedures up to May 8, 2008 consisted of submitting every 25th sample to a second laboratory for re-analysis. Analyses were performed by ALS in Öjebyn and Luleå in Sweden. Commencing with drill hole Myr-08-115, completed in early May 2008, Actlabs of Ancaster, Ontario provided an umpire check on analyses performed by the ALS labs in Sweden. Scatter plots of 13 duplicate pair samples indicated a strong correlation between the analyses of the original and duplicate pulp samples, indicating a relatively high precision of analytical techniques used at the ALS laboratory.

Data verification of a previously completed drill hole at Viken was undertaken by completing a twin drill hole (Myr 06-002) 30 m south of historical drill hole Myrviken 78-005. Comparison of the results showed excellent correlation for both spatial distribution and values for uranium, indicating that the historical uranium data were reliable. Inter-laboratory accuracy was also explored since two laboratories were used to generate analytical results. A total of 128 samples were analyzed at each laboratory. The resultant data indicated no significant bias exists for U_3O_8 and V_2O_5 values. A significant bias exists, however, for MoO_3 (10.5%) and Ni values (45.2%).

11.4.2 2011 to 2012 Quality Assurance/Quality Control

11.4.2.1 Performance of Certified Reference Materials

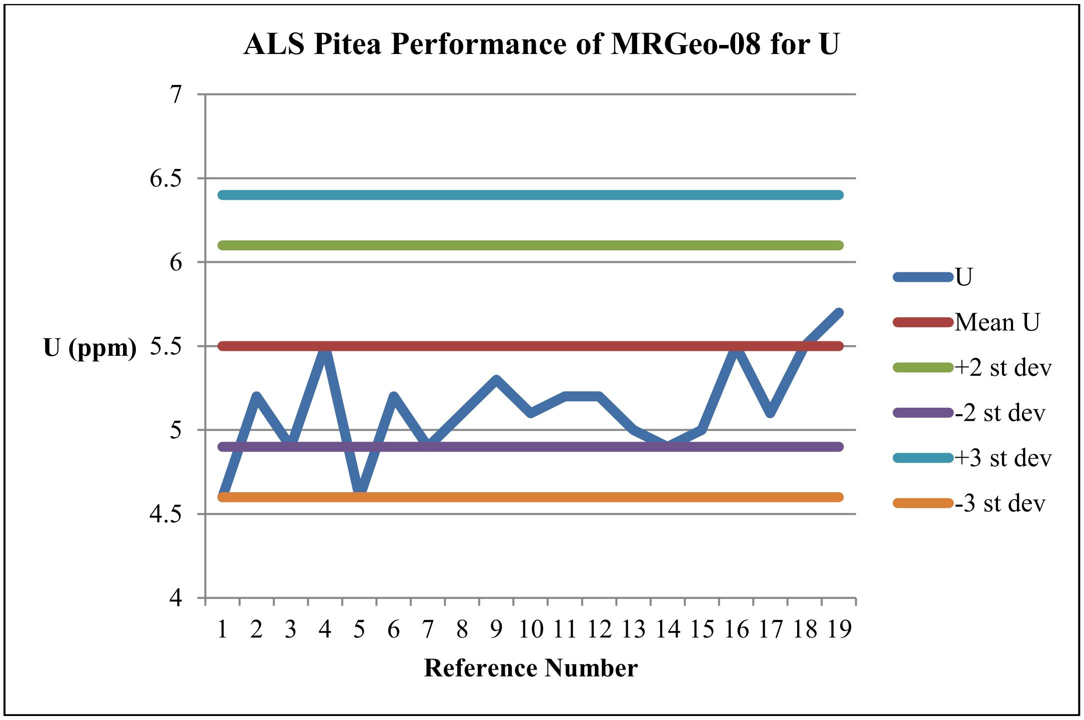
The principal ALS lab in Pitea, Sweden, used four different CRMs to monitor the accuracy of the uranium, vanadium, nickel, copper, zinc and molybdenum analyses. One of the CRMs was certified for all six elements, two were certified for copper, nickel and zinc, and the fourth was certified for only copper and zinc. In addition, CPM supplied the custom-made skaelsta shale RM to the lab, which was inserted routinely into the sample stream. Skaelsta was not certified; however, a round robin of 82 samples was completed at two labs and the mean and standard deviation from the 82 samples was used to evaluate performance.

There were a total of 48 CRMs inserted by the lab for the drill programs. CRM performance was considered acceptable with only two failures noted as exceeding minus three standard deviations from the mean. The two failures were for U for the MRGeo08 CRM (Figure 11.1). Performance charts are presented in Figures 11.1 to 11.6.

There were 12 data points for the skaelsta RM to review and all 12 values remained within ± two standard deviations from the calculated mean for uranium, vanadium, nickel, copper, zinc and molybdenum.

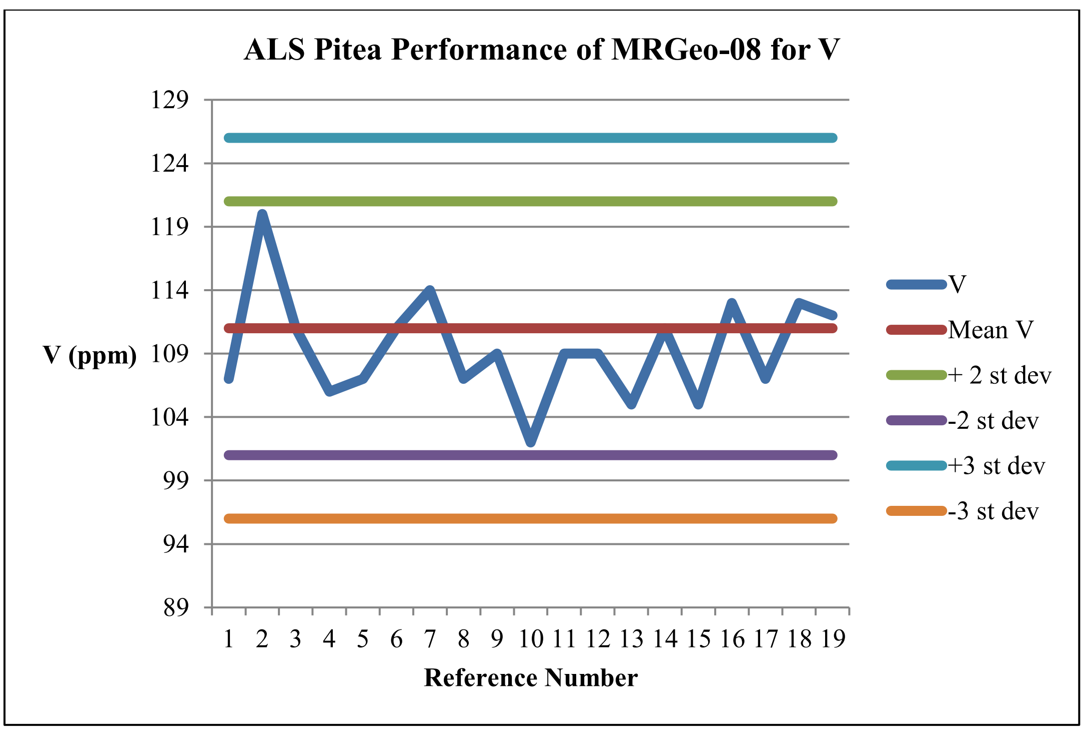
The Author recommends inserting suitable CRMs in the field before shipping samples to the lab, for future drilling at the Project. CRMs should be carefully selected to match grades to those of the Mineral Resources (particularly uranium, which was very low-grade and close to detection levels for both the skaelsta and MRGeo08 reference materials).

FIGURE 11.1 PERFORMANCE OF CRM MRGEO-08 FOR URANIUM*



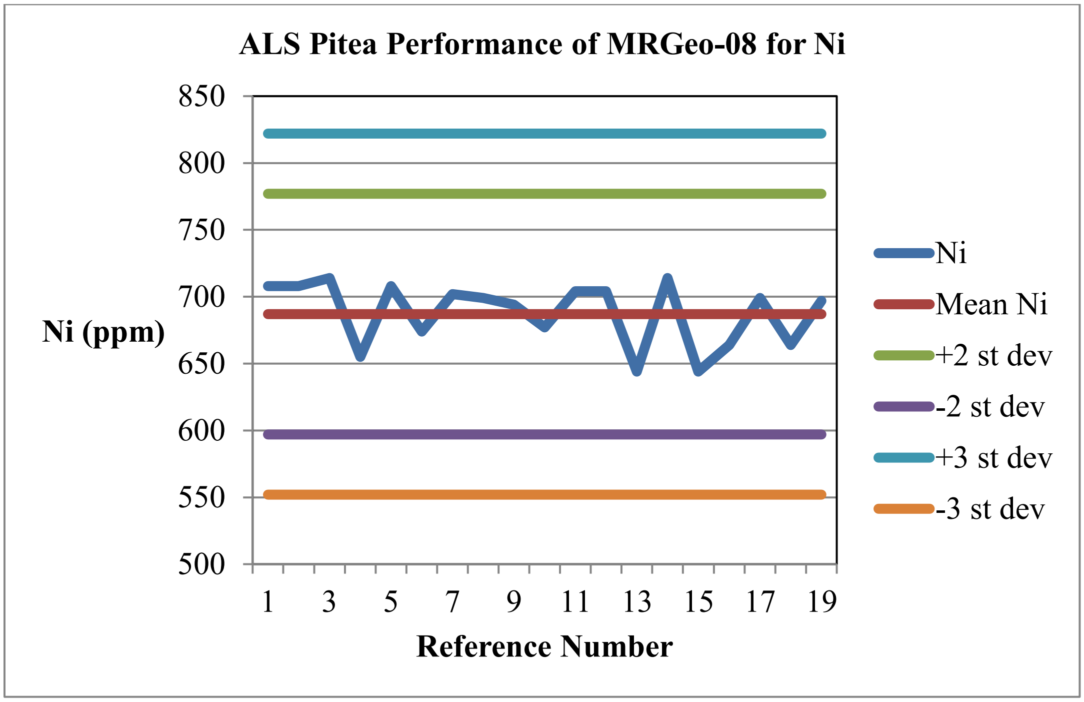
***Source:** P&E (2014)*
***Note:** *Two failures for U on this CRM, and the only failures of a total of 48 CRMs.*

FIGURE 11.2 PERFORMANCE OF CRM MRGEO-08 FOR VANADIUM



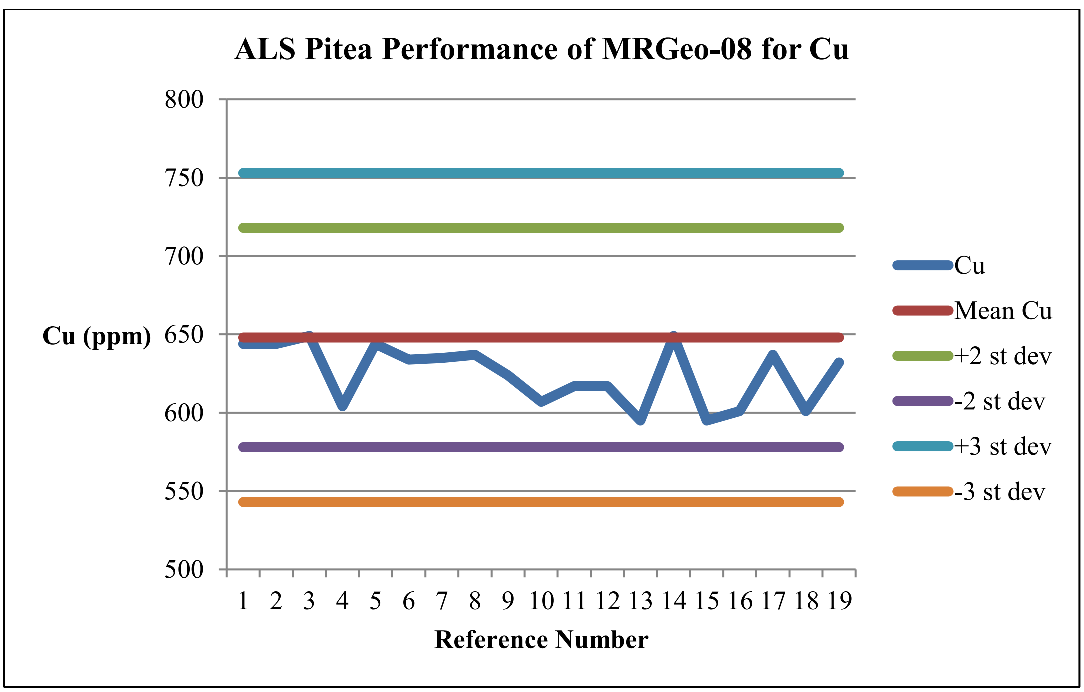
***Source:** P&E (2025)*

FIGURE 11.3 PERFORMANCE OF CRM MRGEO-08 FOR NICKEL



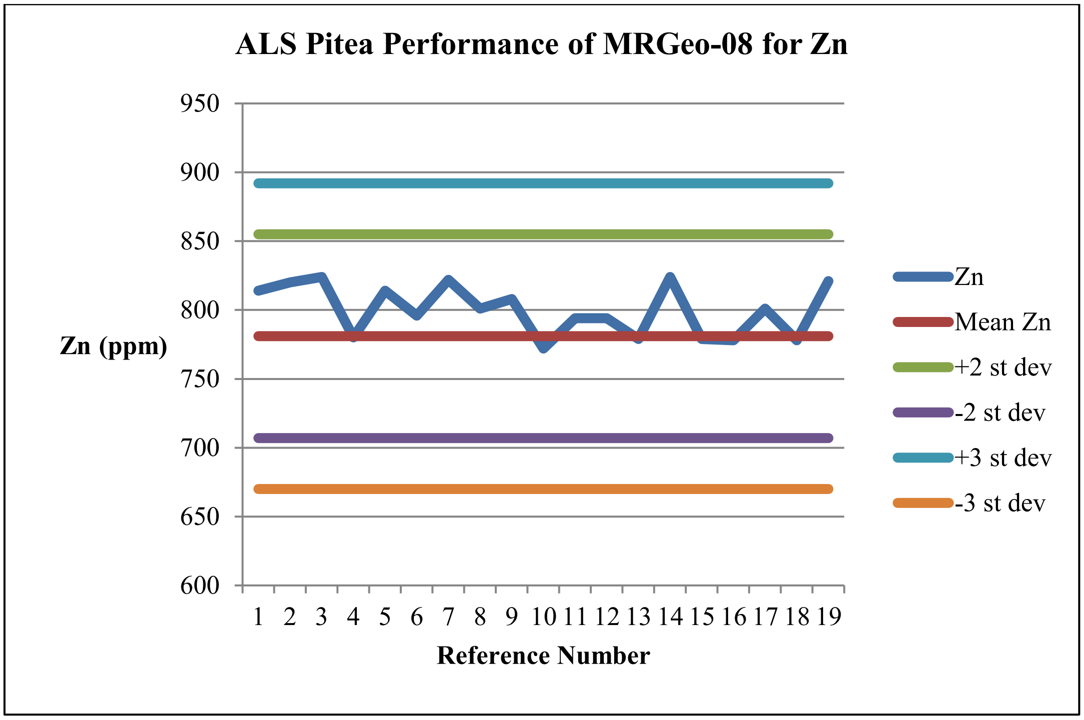
Source: *P&E (2014)*

FIGURE 11.4 PERFORMANCE OF CRM MRGEO-08 FOR COPPER



Source: *P&E (2014)*

FIGURE 11.5 PERFORMANCE OF CRM MRGEO-08 FOR ZINC



Source: *P&E (2014)*

FIGURE 11.6 PERFORMANCE OF CRM MRGEO-08 FOR MOLYBDENUM

Source: *P&E (2025)*

11.4.2.2 Performance of Blank Material

There were 29 blanks inserted by ALS Pitea, which were used to monitor contamination for uranium, vanadium, nickel, copper, zinc and molybdenum. All 29 blanks reported values less than five times the detection limit for all elements in the Mineral Resource Estimate, indicating an absence of contamination at the analytical level.

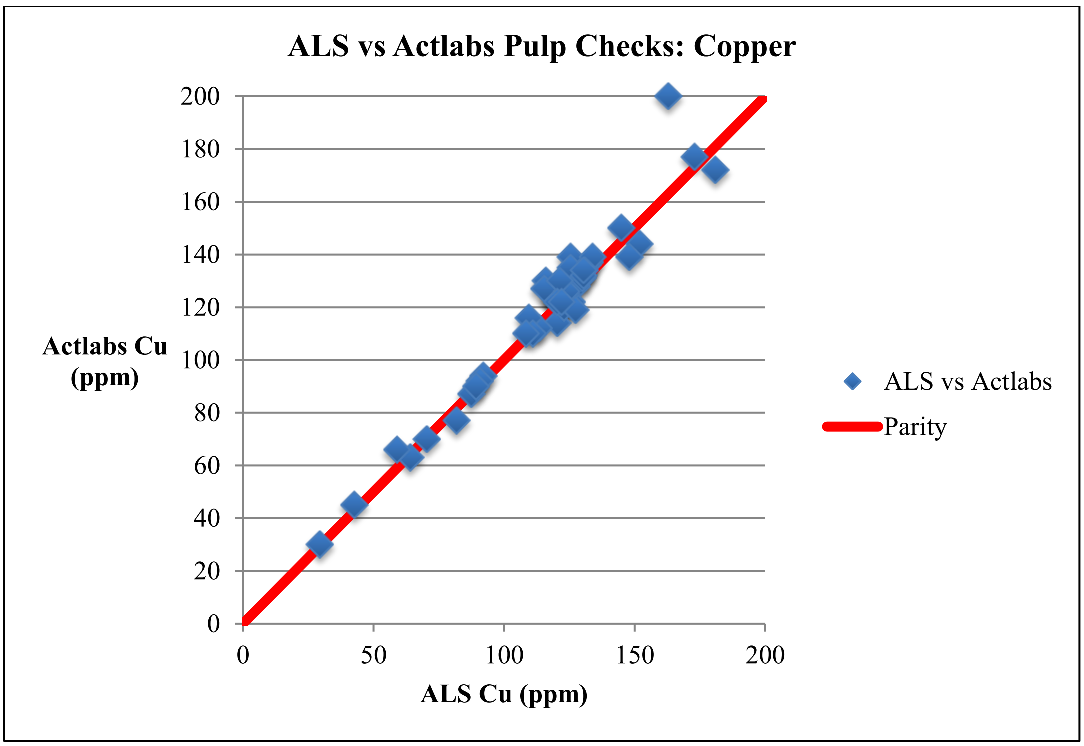
11.4.2.3 Performance of ALS Internal Pulp Duplicates

There were 12 pulp duplicates prepared and analyzed at the lab for the drill program, and all pulp pairs for uranium, vanadium, nickel, copper, zinc and molybdenum essentially had ratios of 1:1, indicating excellent precision at the pulp level.

11.4.2.4 Performance of Secondary Lab Pulp Duplicates

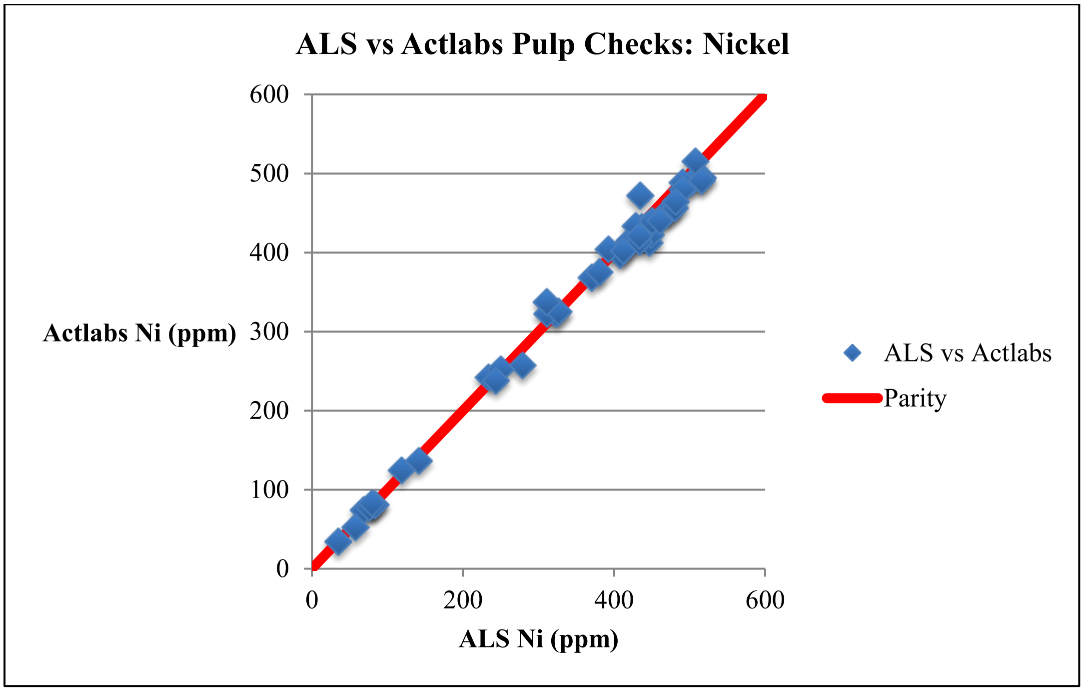
A total of 45 pulps that had been prepared at ALS were sent to independent laboratory Actlabs, Ancaster, Ontario, as a check on the principal lab. Performance was very satisfactory, and results of the checks are presented in Figures 11.7 to 11.10.

FIGURE 11.7 ALS VERSUS ACTLABS PULP CHECKS FOR COPPER



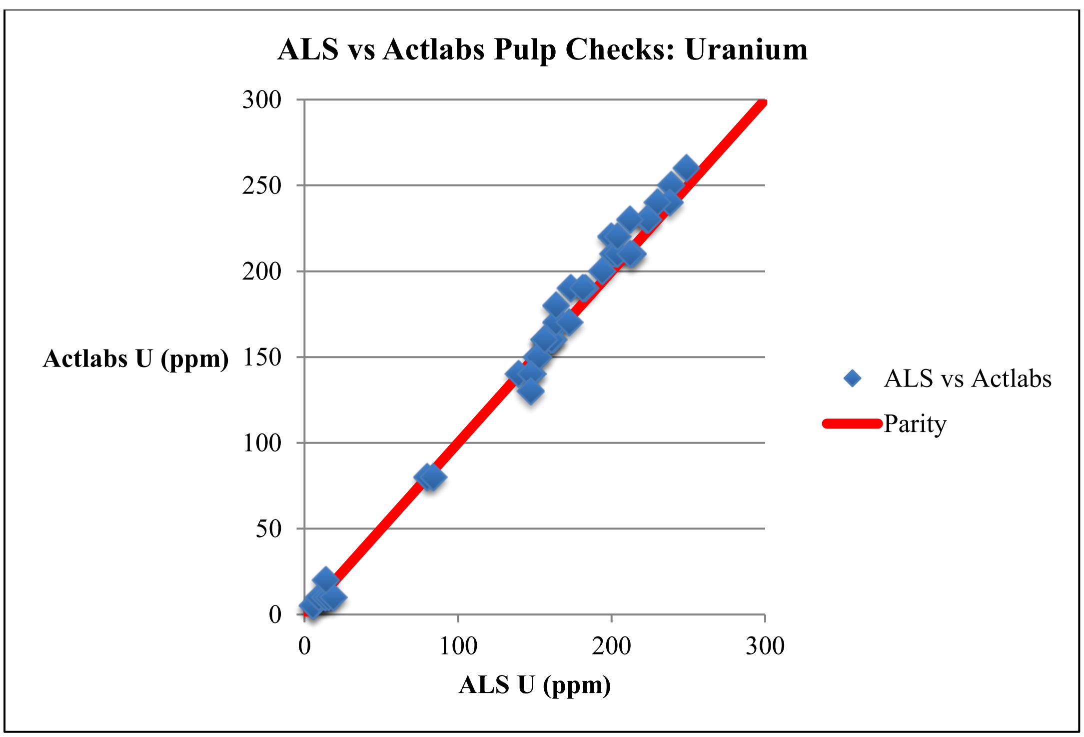
Source: *P&E (2014)*

FIGURE 11.8 ALS VERSUS ACTLABS PULP CHECKS FOR NICKEL



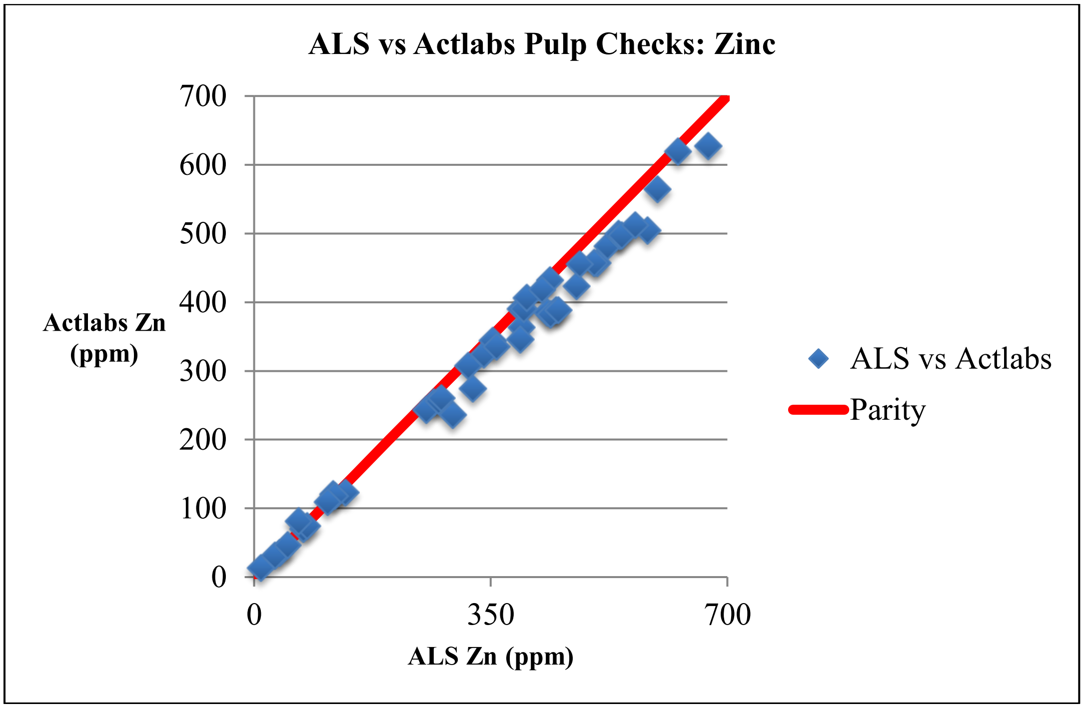
Source: *P&E (2014)*

FIGURE 11.9 ALS VERSUS ACTLABS PULP CHECKS FOR URANIUM



Source: *P&E (2014)*

FIGURE 11.10 ALS VERSUS ACTLABS PULP CHECKS FOR ZINC



Source: *P&E (2014)*

11.5 CONCLUSIONS QUALITY CONTROL

The Author is of the opinion that the data have been collected and analyzed using industry best practices, and the results are satisfactory for use in the Mineral Resource Estimate as presented in Section 14 of this Report.

The Author recommends the following be undertaken during future sampling at Viken:

1. Insert CRMs of appropriate grades for all elements of interest into the drill core sample stream on-site before shipping to the lab;
2. The routine insertion of field and coarse reject duplicates into the sampling stream; and
3. Check analyses of 5 to 10% of drill core samples taken at the Project, ensuring to include adequate QC samples to monitor umpire laboratory performance.

It is the Author's opinion that sample preparation, security and analytical procedures for the drilling at the Viken Project were adequate and examination of QA/QC results indicates no significant issues with accuracy, contamination or precision in the data.

The Author considers the data to be of suitable quality and satisfactory for use in the current Mineral Resource Estimate.

12.0 DATA VERIFICATION

12.1 DRILL HOLE DATABASE VERIFICATION

12.1.1 Assay Verification

Verification of drill hole assay data entry was performed by the Author on assay intervals for U, V, Ni, Cu, Zn and Mo. Data from drill holes completed in 2006, 2008, 2010, 2011 and 2012 were verified. A total of 967 intervals for U, V, Ni, Cu and Zn, and 287 intervals for Mo, were checked against original digital assay laboratory certificates provided directly to the Authors by ALS. The 967 checked assays represent 11.4% of the entire database (967 out of 8,499 samples), and 15.3% of the constrained data (772 out of 7,164 samples). The 287 intervals checked for Mo represented approximately 4% of the constrained drill holes assays. Very few minor errors of no material impact were encountered in the data during the verification process.

12.1.2 Drill Hole Data Verification

The Author validated the Mineral Resource database in GEMS™ by checking for inconsistencies in analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, survey and missing interval and coordinate fields. A few minor errors were identified and corrected in the database.

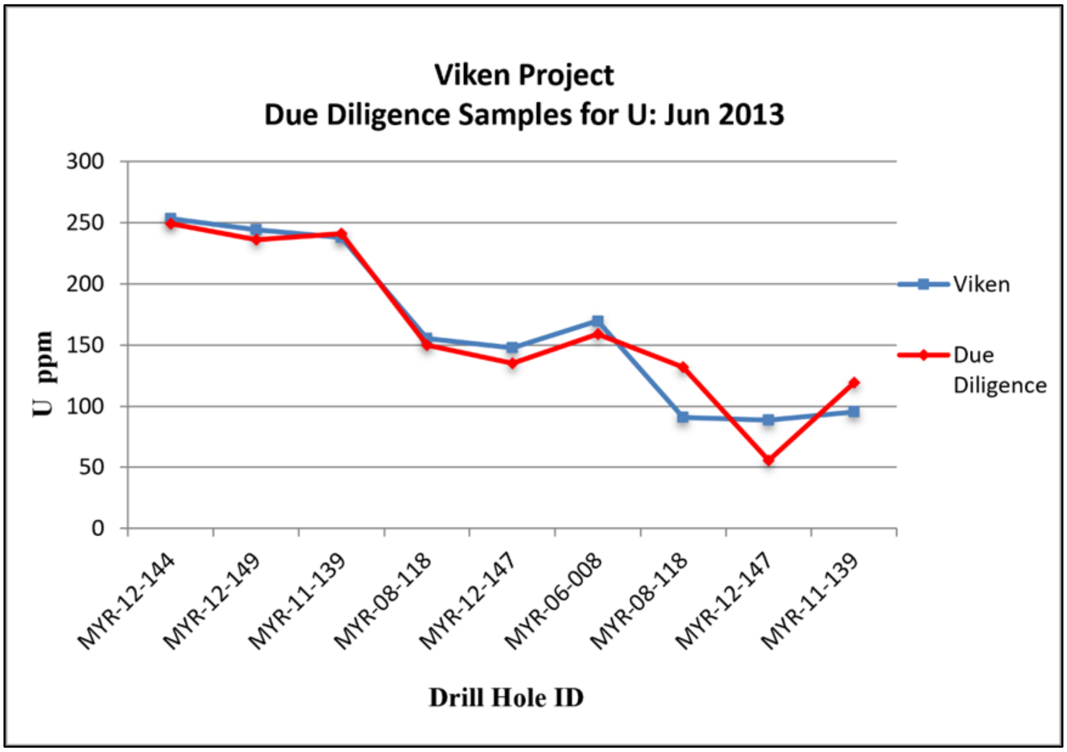
12.2 2013 SITE VISIT AND INDEPENDENT SAMPLING

The Viken Project was visited by Mr. Eugene Puritch, P.Eng., of P&E, on June 18, 2013, for the purposes of completing a site visit and independent sampling program.

Mr. Puritch toured the secure drill core facilities, and the Project site, and details of the Project were discussed with site personnel. Mr. Puritch collected nine samples of drill core from eight drill holes by sawing a quarter drill core from the remaining half drill core in the box. Samples were selected from a range of grades and placed in a plastic bag with a unique sample tag. Each bag was sealed, and when all the samples had been collected, they were placed in polypropylene bags and brought to the offices of P&E in Brampton, ON by Mr. Puritch. From there, the samples were sent via courier to AGAT Labs in Mississauga, ON, for analysis. At no time were any officers, directors or employees of CPM notified as to the location of the samples to be collected.

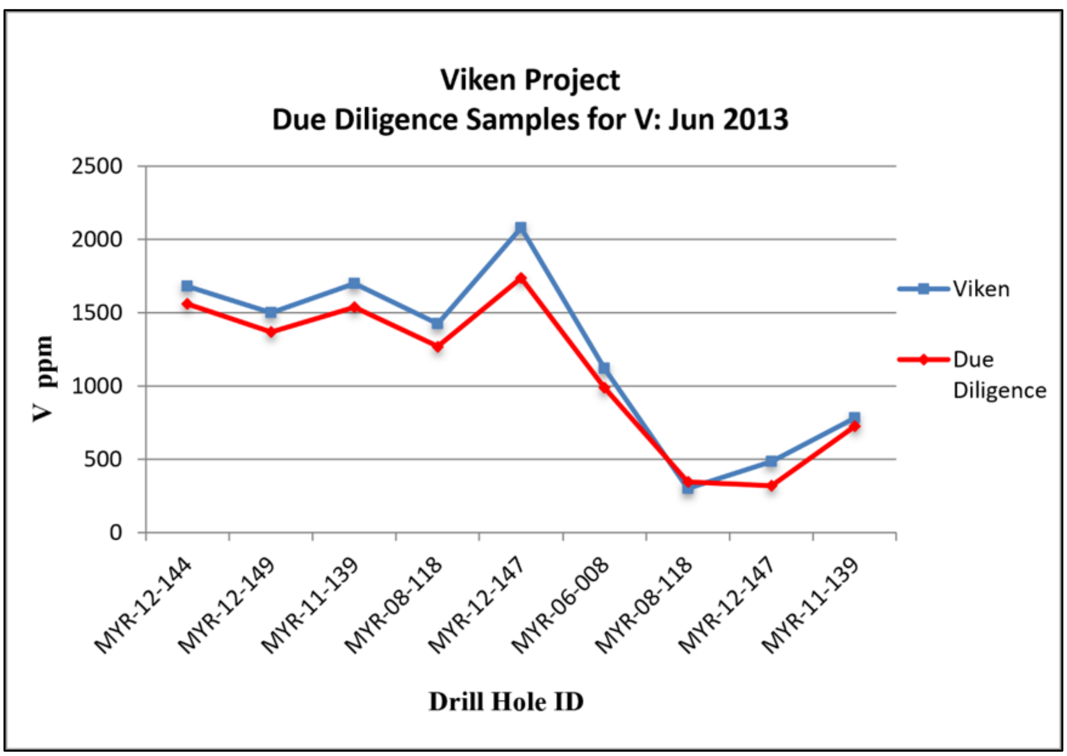
Samples at AGAT were analyzed for a suite of 48 elements, including U, V, Ni, Cu, Zn and Mo, by four-acid digest with an ICP-MS finish. AGAT has developed and implemented at each of its locations a Quality Management System ("QMS") designed to ensure the production of consistently reliable data. The system covers all laboratory activities and takes into consideration the requirements of ISO standards. AGAT maintains ISO registrations and accreditations (ISO 9001:2015 and ISO/IEC 17025:2017). Results of the 2013 Viken independent sampling program are shown in Figures 12.1 to 12.6.

FIGURE 12.1 2013 DUE DILIGENCE SAMPLE RESULTS FOR URANIUM



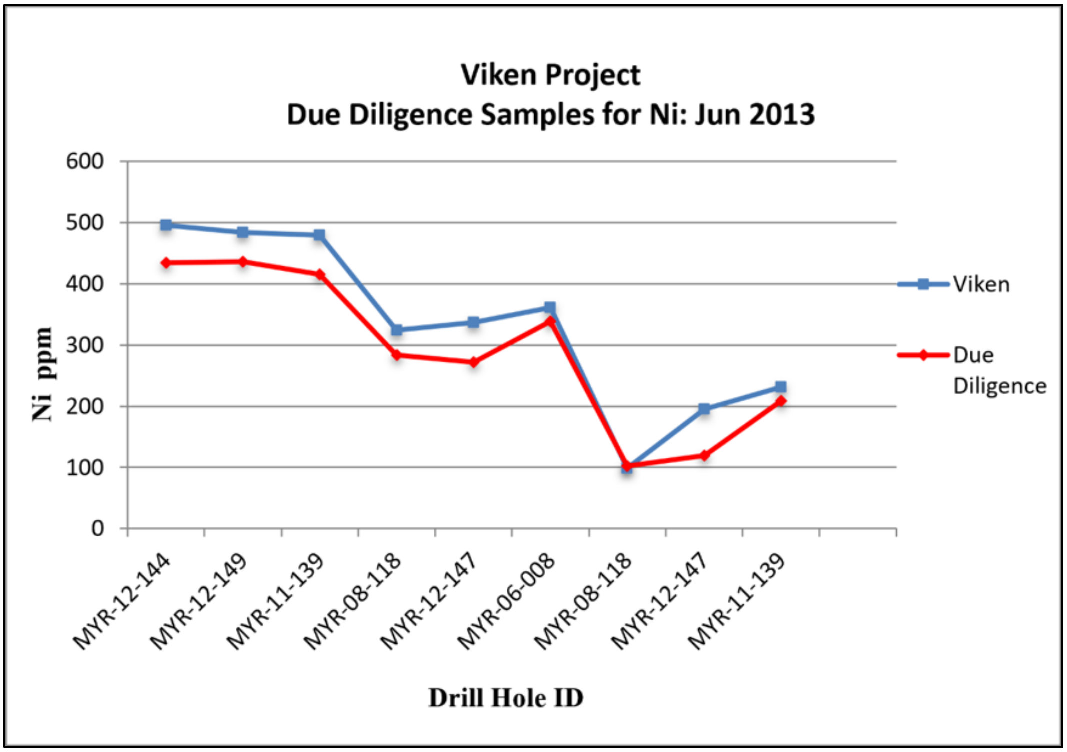
Source: *P&E (2014)*

FIGURE 12.2 2013 DUE DILIGENCE SAMPLE RESULTS FOR VANADIUM



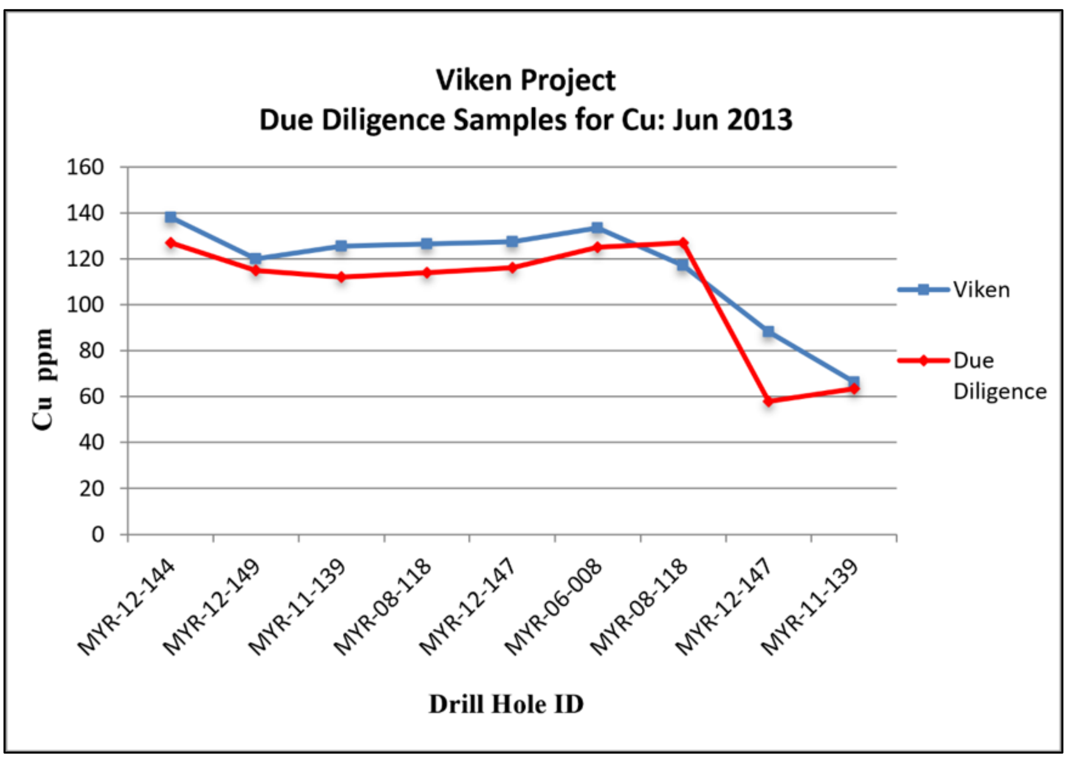
Source: *P&E (2014)*

FIGURE 12.3 2013 DUE DILIGENCE SAMPLE RESULTS FOR NICKEL



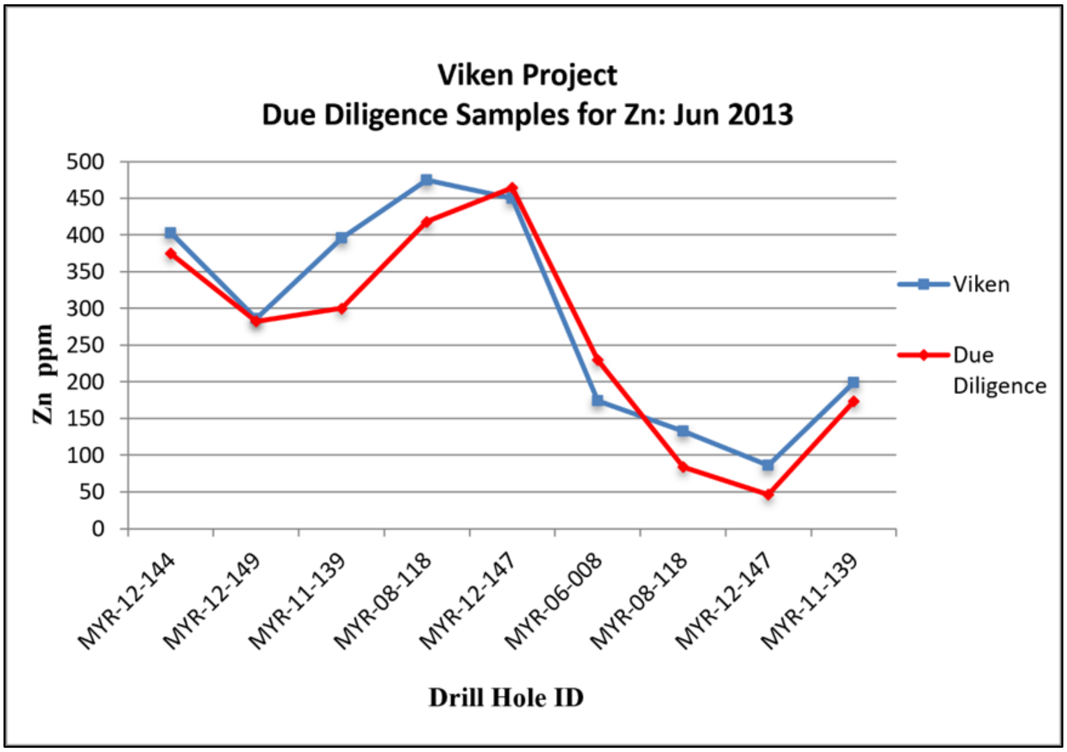
Source: *P&E (2014)*

FIGURE 12.4 2013 DUE DILIGENCE SAMPLE RESULTS FOR COPPER



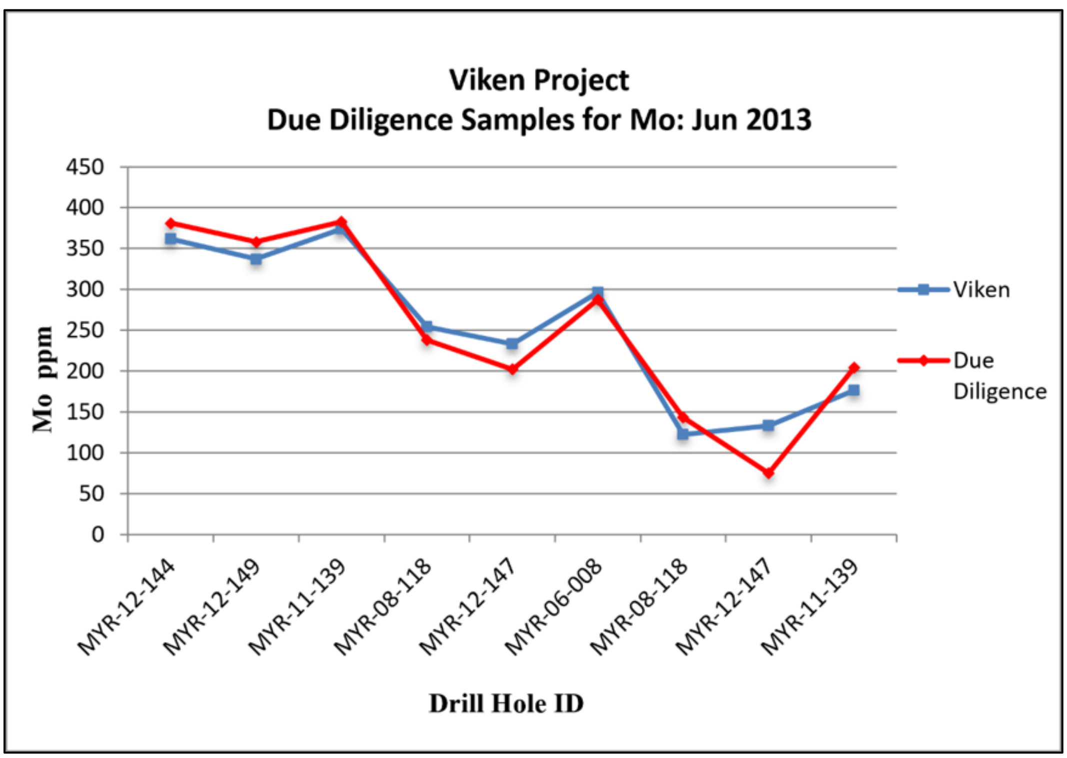
Source: *P&E (2014)*

FIGURE 12.5 2013 DUE DILIGENCE SAMPLE RESULTS FOR ZINC



Source: *P&E (2014)*

FIGURE 12.6 2013 DUE DILIGENCE SAMPLE RESULTS FOR MOLYBDENUM



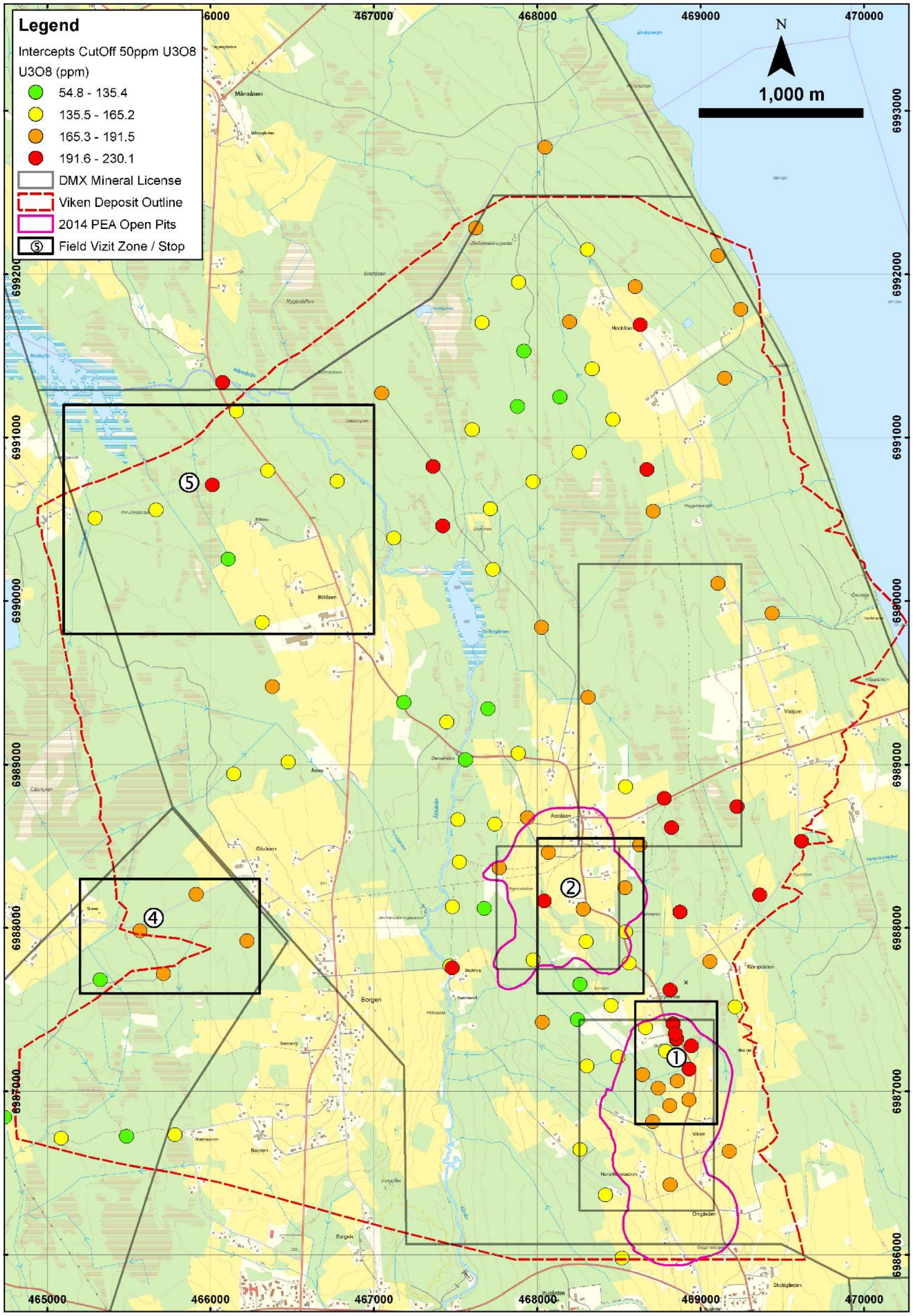
Source: *P&E (2014)*

12.3 2025 SITE VISIT AND INDEPENDENT SAMPLING

The Viken Project was visited by Mr. David Burga, P.Geo. of P&E, on March 20, 2025, for the purpose of completing a site visit that included technical discussions, viewing drill hole collar sites and outcrops, and taking GPS location verifications and scintillometer readings.

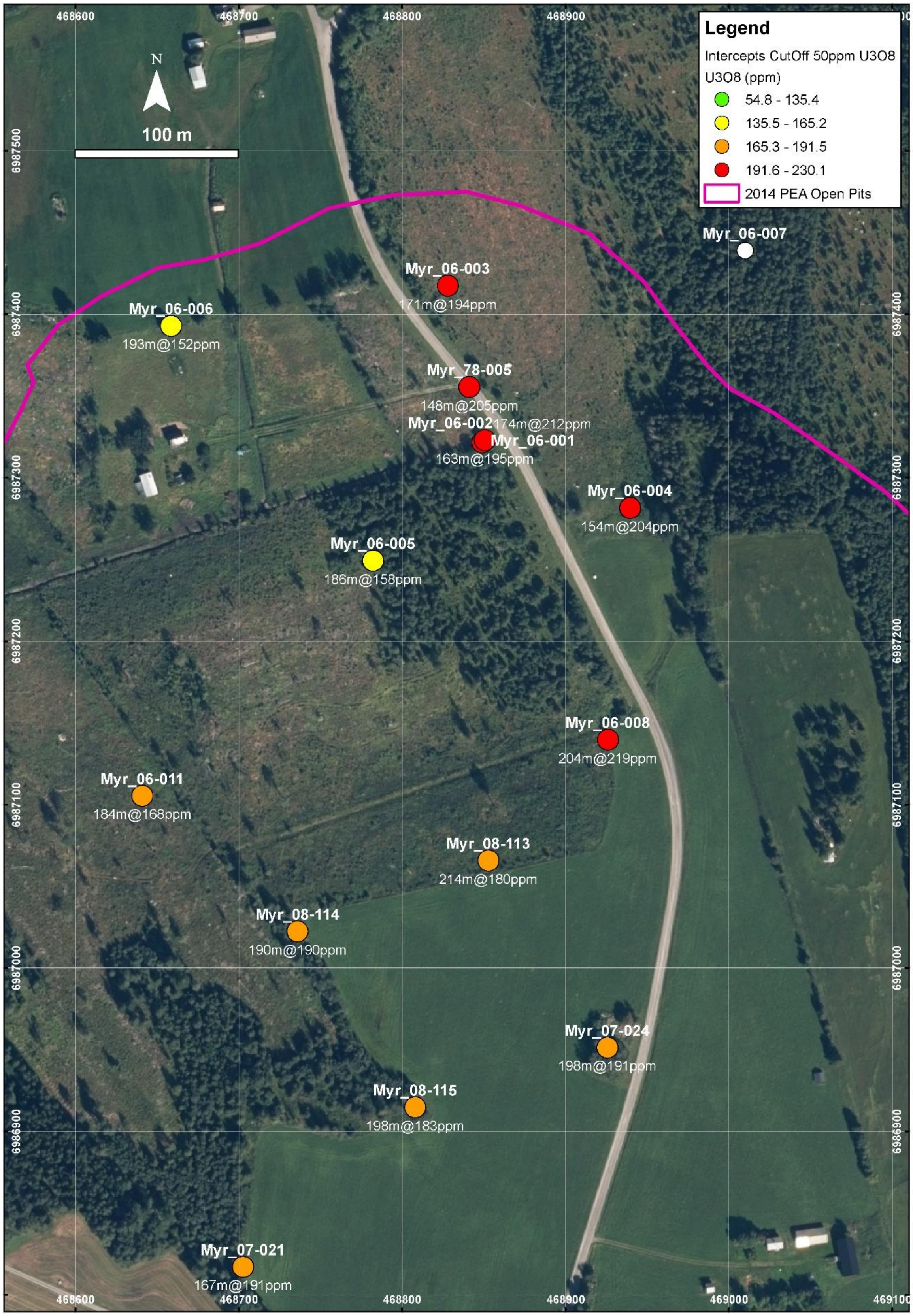
Mr. Burga's tour of the Property included stopping at four different locations throughout the visit (Stops 1, 2, 4 and 5, as shown on Figure 12.7). Stop 1 was near drill hole Myr_07-024 (bottom right corner of Figure 12.8) and stop 2 included spotting the post mark for drill hole Myr_07-23 (Figures 12.9 and 12.10 and Table 12.1). Viken tour Stop 4 involved stopping at several drill hole collars, including drill holes Myr_08-089, Myr_08-090 and Myr_08-091, and shale outcrop 1 where a scintillometer reading of 1,562 counts per second ("c/s") was taken (Figure 12.11 to 12.13 and Table12.1). None of the recent drill hole collars were able to be located at Viken tour Stop 5 (Figure 12.14), due to snow cover. However, a steel pole marking historical drill hole 79001 was encountered (Figure 12.15 and Table 12.1). A scintillometer reading of 1,478 c/s was taken at shale outcrop 2 (Figure 12.16 and Table 12.1).

FIGURE 12.7 VIKEN PROPERTY TOUR LOCATIONS 1, 2, 4 AND 5



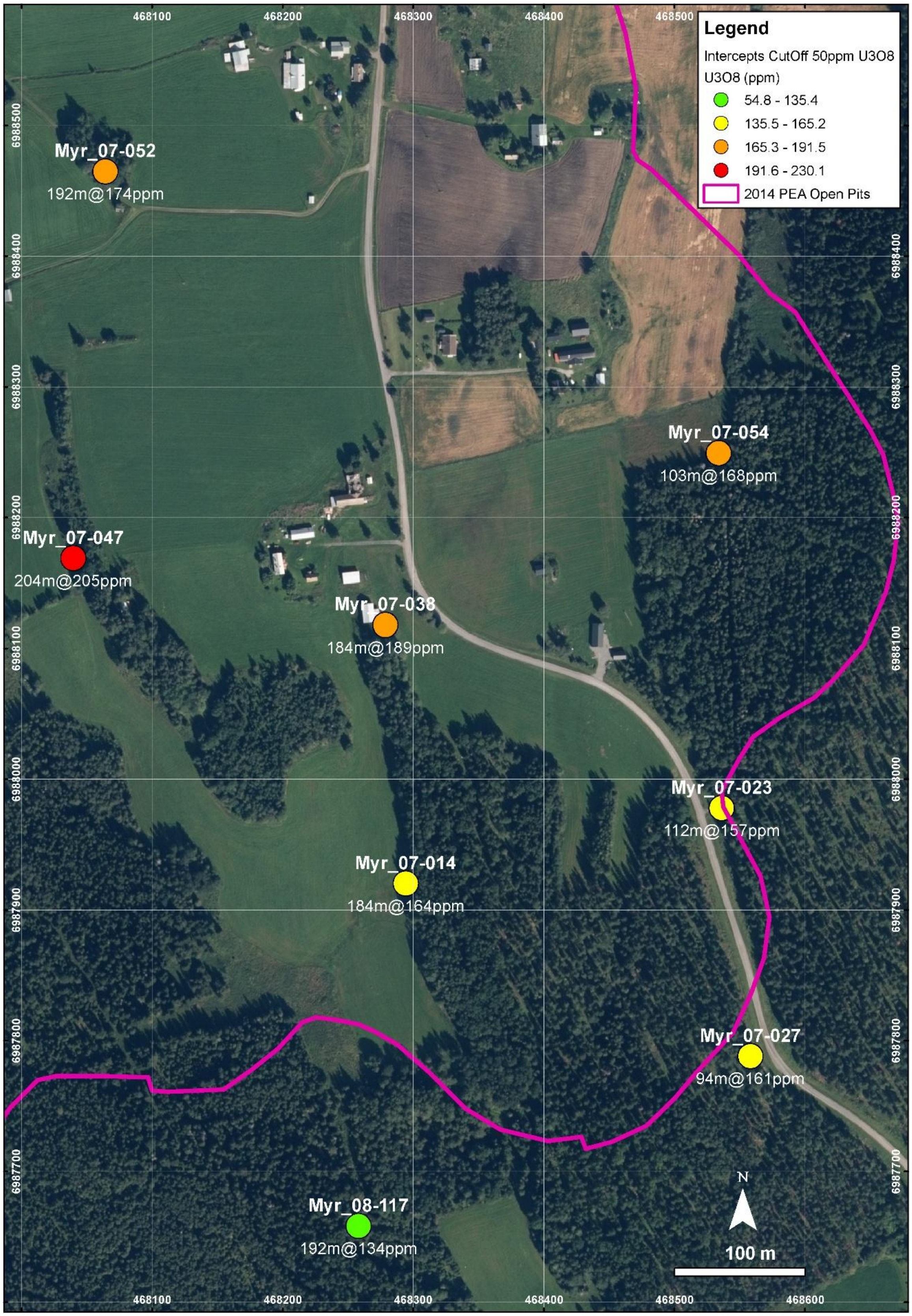
Source: *District Metals (June 2025)*

FIGURE 12.8 VIKEN TOUR STOP 1: NEAR DRILL HOLE MYR-07-024



Source: *District Metals (June 2025)*

FIGURE 12.9 VIKEN TOUR STOP 2: NEAR DRILL HOLE MYR-07-023



Source: *District Metals (June 2025)*

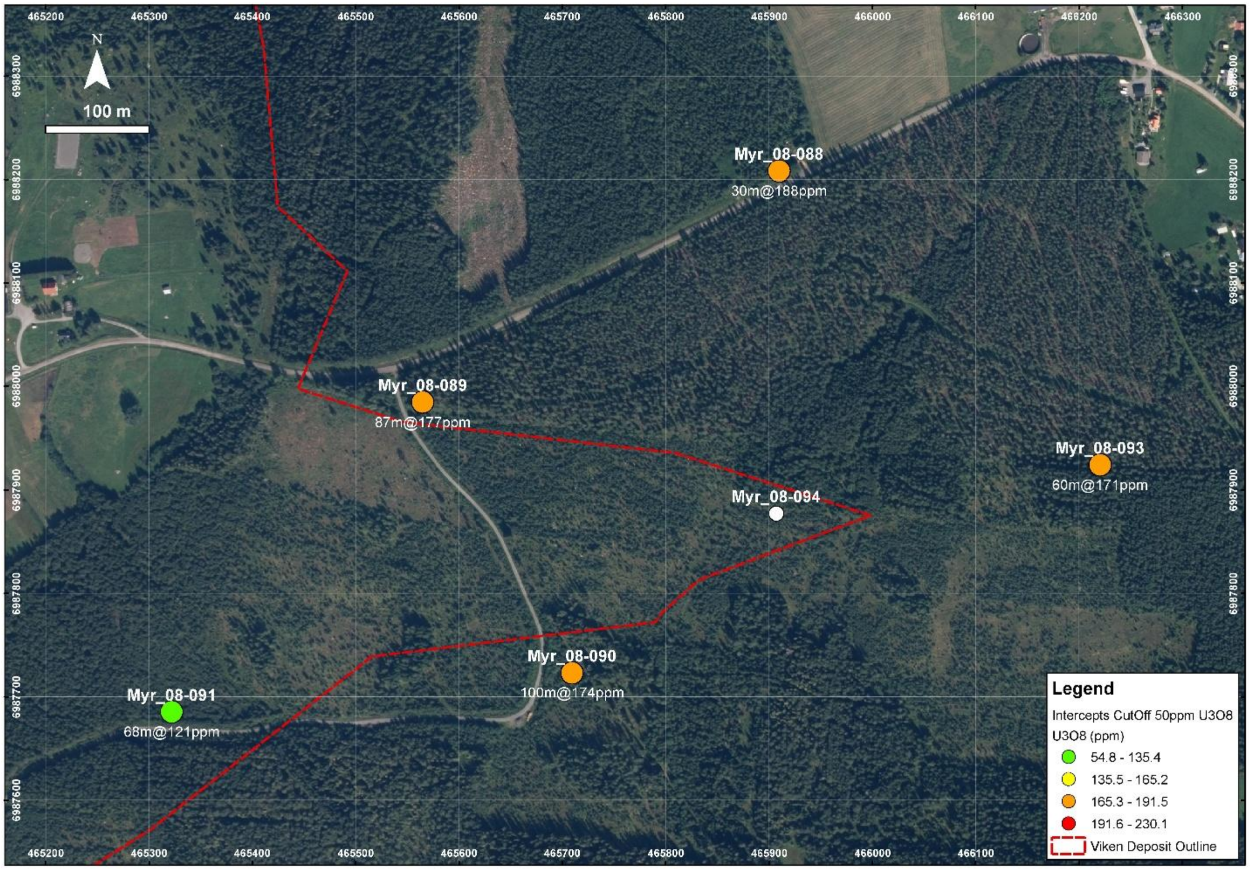
FIGURE 12.10 VIKEN TOUR STOP 2: POST MARKER FOR DRILL HOLE MYR-07-23

Source: *P&E (2025)*

TABLE 12.1 UTM ZONE 33V COORDINATE LOCATIONS TAKEN DURING VIKEN SITE VISIT		
Drill Hole or Outcrop ID	**Easting (m)**	**Northing (m)**
Myr-07-023	468,538	6,987,976
Myr-08-089	465,568	6,987,984
Myr-08-090	465,710	6,987,721
Myr-08-091	465,323	4,987,681
M66-07-080	465,289	6,990,504
Historical Hole 79001	466,075	6,991,318
Shale Outcrop 1	465,504	6,987,678
Shale Outcrop 2	465,879	6,991,320
Black Shale Outcrop	465,896	6,984,515

Source: *P&E (2025)*

FIGURE 12.11 VIKEN TOUR STOP 4: DRILL HOLES MYR-08-089, MYR-08-090, MYR-08-091 AND SHALE OUTCROP 1



Source: *District Metals (June 2025)*

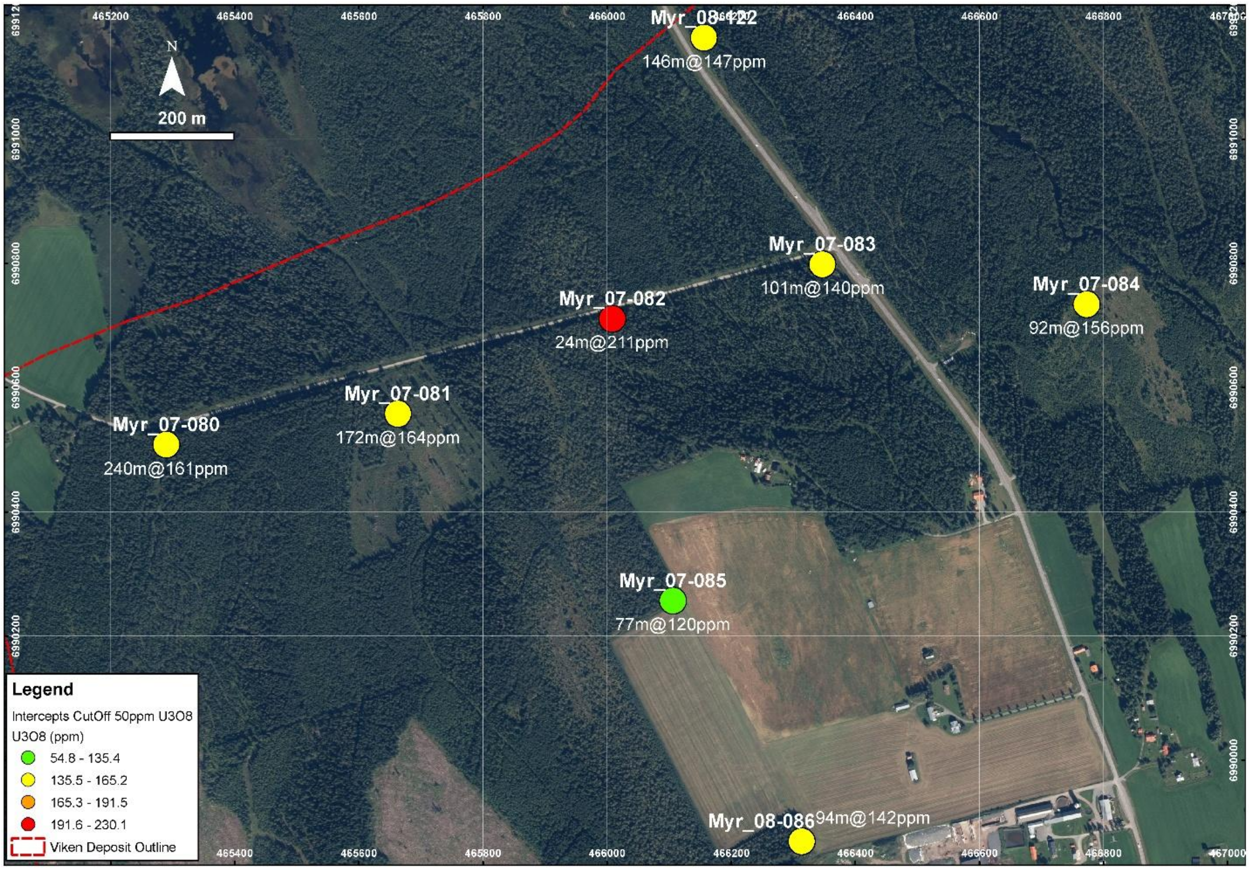
FIGURE 12.12 VIKEN TOUR STOP 4: SHALE OUTCROP 1

Source: *P&E (2025)*

FIGURE 12.13 SCINTILLOMETER SHOWING READING TAKEN AT SHALE OUTCROP 1

Source: *P&E (2025)*

FIGURE 12.14 VIKEN TOUR STOP 5: HISTORICAL DRILL HOLE 79001 AND SHALE OUTCROP 2



Source: *P&E (2025)*

FIGURE 12.15 VIKEN TOUR STOP 5: POST MARKER FOR HISTORICAL DRILL HOLE 79001

Source: *P&E (2025)*

FIGURE 12.16 STOP 5: SCINTILLOMETER SHOWING READING TAKEN AT SHALE OUTCROP 2

Source: *P&E (2025)*

12.4 ADEQUACY OF DATA

Verification of the Viken Project data, used for the current Mineral Resource Estimate, was undertaken by the Authors, and included multiple site visits, due diligence sampling, drill hole collar and outcrop location verification, verification of drilling assay data, and assessment of the available QA/QC data for the recent drilling data. The Authors consider that there is excellent correlation between the U, V, Ni, Cu, Zn and Mo assay values in the Company's database and the independent verification samples collected by the Authors and analyzed at AGAT. The Authors consider that sufficient verification of the Project data has been undertaken and that the supplied data are of suitable quality and satisfactory for use in the current Mineral Resource Estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

13.1.1 Purpose and Scope of Metallurgical Gap Analysis

Numerous metallurgical testwork programs were conducted between 2007 and 2026 to evaluate the processing characteristics of the mineralized shale from the Viken Deposit and to support the selection of appropriate processing routes. As part of this PEA, METS conducted an independent review of all relevant historical metallurgical testwork. METS developed the recovery routes and assumptions for the PEA study based on a combination of historical testwork results, comparable technical studies, and its experience from similar projects.

13.1.2 Source Documents and Historical Studies Reviewed

The testwork was undertaken by a range of independent commercial laboratories and academic institutions, including Process Research Ortech (Mississauga, Ontario, Canada), Luleå University of Technology (Luleå, Sweden), Saskatchewan Research Council (Saskatoon, Saskatchewan, Canada), SGS (Lakefield, Ontario, Canada), Alberta Innovates – Technology Futures (Edmonton, Alberta, Canada), and Inspectorate Exploration and Mining Services Ltd. (Richmond, British Columbia, Canada). The results of these programs indicate that the mineralized shale is amenable to multiple processing approaches, reflecting its complex and polymetallic nature. This section summarizes the relevant metallurgical testwork conducted to date, the associated outcomes for the Viken Deposit and recommendations for the next stage of testwork for the Project.

Metallurgical reviews and testwork results have been compiled from the following key reports:

- Analytical data and results generated between 2007 and 2008 by Actlabs, ALS Analytica, and ALS Chemex.
- 2007 Drill hole analysis data.
- 2010, 2014 and 2018 Viken PEA Reports.
- 2009 Third Updated Technical Report.
- 2025 Updated Mineral Resource Estimate and Technical Report.
- 2017 Pressure Oxidation of Vanadium-Bearing Carbonaceous Shale Testwork Report.
- 2010 Hydrocarbon Recovery, Gasification, and Biological Leaching for Metals Extraction Report.
- 2013 Hatch Viken Project Scoping Study FEL 1 Report.

- 2013 Hatch Scoping Study Attachments: Agglomeration leaching Testwork, Column Bio-leach Testwork and results, Column leach assays, Equipment Sizing, Mass Balance and Power Cost.

- 2023 Aura Energy Haagen Scoping Study.

- Miscellaneous historical metallurgical data and testwork results of Alum Shale processing and mineralogy referenced.

13.2 MINERALOGY AND CHEMICAL CHARACTERISATION

13.2.1 Resource Chemistry and Key Components

The Viken Mineral Resource contains five groups of substances that are of interest:

1. Carbon: on average (mean) approximately 11% of the black shale samples tested as organic carbon. This could be a source of thermal heat required in a process or the carbon could be converted to synthetic petroleum.

2. Alum: $KAl(SO_4)_2 \cdot 12H_2O$. Alum is a water-soluble compound and could be converted to a potassium- based fertilizer e.g., K_2SO_4. The equivalent of approximately 4% K_2O is present in the Mineral Resource.

3. Vanadium: a valuable metal – could be extracted and converted from an oxide to a chloride for water solubility. V_2O_5 content is listed present as 0.13%.

4. Heavy metals: copper, iron, molybdenum, nickel and zinc – all, except iron, are present at low concentrations – 100s of ppm, whereas iron concentration exceeds 6%. Their presence as sulphides in the Viken Deposit is assumed.

5. Uranium: moderately low concentration – 150 ppm, however, potentially more valuable than the heavy metals. Uranium has been reported to be present in the black Alum Shales with the carbon, phosphorous, or as tiny crystals of UO_2.

Elements of potential environmental or health concern – arsenic and cadmium are moderate and low at 85 ppm and 7 ppm, respectively, and would need to be monitored for potential reporting to any metal concentration product. In addition, uranium and uranium daughter radionuclides, particularly the distribution of Ra-226, would need to be monitored and potentially controlled, especially in a potassium fertilizer product and in tailings.

13.2.2 Historical Assay Datasets

Between 2006 and 2008, a diamond drilling program was conducted to support the estimation of historical Inferred and Indicated Mineral Resources for the Viken Deposit. Drill core from this program was logged, split, and sampled, with the resulting dataset compiled and reported in the 2009 "Third Updated Technical Report". Analytical testwork on the 2006–2007 samples were undertaken at two laboratories, ALS Analytica AB in Luleå, Sweden, and ALS Chemex in Öjebyn, Sweden. Four grade-elements were defined for the Viken resource: uranium (U) which ranged between 3.7 to 287 ppm with a mean of 137.5 ppm, vanadium (V) which ranged between 30 to 3,290 ppm with a mean of 1,473.1 ppm, molybdenum (Mo) which ranged between 1.3 to 2,130 ppm with a mean of 238.8 ppm, and nickel (Ni) which ranged between 10 to 637 ppm with a mean of 308.6 ppm.

13.2.3 2010 – 2013 PEA

In the October 2010 "CPM Preliminary Economic Assessment (PEA)", the average payable metal grades within the conceptual pit were reported as approximately 2,003 ppm vanadium (V), 155 ppm uranium (U), and 270 ppm molybdenum (Mo). Most analysis was conducted on a composite shale sample derived from three drill holes within the Viken Deposit. Analysis was conducted at several independent laboratories including ALS Chemex (Luleå), SGS, AMML, and ARC. A composite of this material was made and used for bio-heap leach testwork conducted at Luleå University between 2011 and 2012. The key assay data from the historical datasets were compared with the composite (SRC) assay as shown in Table 13.1.

TABLE 13.1
ASSAY COMPARISON EXTRACT

Location		Southern pit	Southern pit		2012 Lulea
Drill holes		140 141 142	MYR 113114115	Pit shell	MYR 113114115
Report		SRC composite	Gap analysis	2010 PEA	bio-heap test
U	ppm	148	157.3	155	151
V	ppm	1770	1815	2003	2000
Mo	ppm	258	273.8	270	246
Ni	ppm	369	442.6	NR	342
Zn	ppm	500	461.4	NR	527
Cu	ppm	130	187.4	NR	115
C (organic)	%	11.4	11.4	NR	10.94
C (inorganic)	%	0.52	0.9	NR	0.81
S	%	4.1	4	NR	4.6
Fe	%	4.33	4.5	NR	4.4
Ca	~~%~~	3.94	2.9	NR	3
Al	%	6.49	6.4	NR	6.3
K	%	3.6	3.2	NR	3.3
Mg	%	0.68	0.7	NR	0.6
Ti	%	0.38	0.4	NR	NR
P	ppm	1135	961.4	NR	NR
Mn	ppm	774	605.2	NR	770
Co	ppm	23	24.9	NR	21

A subsample of the head sample was sent from the SRC to the SGS Lakefield lab for the "Hatch Scoping Study" in 2013. The QEMSCAN confirmed the presence of Fe, Mn, Ni, Co, Cr, Cu and Zn as sulphides. It showed that Ca and Mg are primarily associated with carbonates whilst the Mo was more difficult to characterize since it appeared to be only partially present in the sulphide phase. V and Ti are associated mainly with the aluminosilicate (phyllosilicate) phase. Historically V and other metals often appear as substitutional elements within a phyllosilicate matrix with a small number of free vanadium oxide grains present. The QEMSCAN identified very little U in a distinct mineral phase, and it was suspected that the U was absorbed on the carbon. To investigate the carbonaceous phase further, a sample of material underwent TOF-SIMS analysis on the surfaces of -300 µm particles, as well as cross-sections of coarser particles. It was not possible to separate a pure carbonaceous particle due to the fine grain structure. It was determined that the U and C are indeed positively spatially associated, and U is negatively correlated with all other species.

13.2.4 2014 PEA

In 2014 a head sample was taken and analyzed from a composite formed for the Bioleach testwork program (2014 PEA). The results were consistent with a polymetallic alum shale system, characterized by elevated vanadium and moderate concentrations of uranium and base metals. Vanadium was the dominant payable element at approximately 1,524 ppm (0.15%), indicating strong potential for recovery. Uranium was present at 132 ppm, which is within the expected range for alum shale deposits and providing the opportunity for a secondary value stream. Molybdenum (265.7 ppm), nickel (291.1 ppm), zinc (359 ppm), and copper (151.7 ppm) occurred at moderate levels, suggesting potential for by-product recovery. Iron content (3.92%) and sulphur content (4.03%) were relatively high indicating the presence of sulphide minerals, which are typical for reducing environments associated with alum shales.

Results from the head assay taken and reported in the 2014 PEA are shown in Table 13.2.

Table 13.2
Head Sample Analysis in ppm for Inspectorate Bio-leach Testwork

U (ppm)	Mo (ppm)	Ni (ppm)	V (ppm)	Mn (ppm)	Zn (ppm)	Cu (ppm)	Fe (%)	S (%)
132.3	265.7	291.1	1,524	592	359	151.7	3.92	4.03

13.2.5 Mineralogical Findings

Mineralogical characterisation underpins understanding of comminution behaviour, flotation response, leach performance and ultimate product quality. Without it, process selection and engineering design rely on high level assumptions and therefore carry greater technical uncertainty. While a number of chemical characterisation programs have been completed since 2007, additional mineralogical work is required to adequately capture the spatial variability of the Deposit.

13.2.6 Gaps and Recommendations

- Implement a domain-based automated mineralogy program (QEMSCAN/TIMA) on fresh, representative drill core to quantify mineral assemblages, liberation, and element deportment for U, V, Mo, Ni, K, Fe, Ca, Mg, As, and radionuclides.

- Tie mineralogical domains explicitly to the resource model, so that comminution, flotation and leach testwork can be planned on meaningful composites rather than generic "average" material.

13.3 METALLURGICAL TESTWORK – HISTORICAL REVIEW

Over the past decade, the Viken Deposit has undergone several changes in ownership and has been subject to multiple metallurgical testwork programs. During this period, processing technologies have evolved and improved, and the regulatory framework governing mineral processing and environmental compliance in Sweden have also changed. These developments have influenced the scope and focus of processing options for the Deposit. This section summarises and incorporates the metallurgical testwork studies conducted to date, together with the associated results for the Viken Deposit and recommendations for the next stage of testwork.

13.3.1 Comminution

13.3.1.1 Historical Hatch Testwork and Results

The 2013 Hatch scoping study details crushing testwork performed on full diameter drill core samples at the Saskatchewan Research Council ("SRC") lab in Saskatoon. A variety of hardness tests were performed primarily to determine the power required for comminution and to size the required crushing and/or grinding equipment. The testwork provided estimated a Bond Ball Mill Work index ("BWi") of 14.7 kWh/t and a Bond Rod Mill Work Index ("BRWi") of 12.5 kWh/t indicating that the mineralized material is moderately hard to hard material. A SMC Test was also conducted recording a Mia of 7.8 kWh/t, Mih of 4.5 kWh/t, and Mic of 2.3 kWh/t. Additionally, HPGR testwork was conducted, followed by particle size distribution analysis using screening. Size distributions and assays by size were measured after each step of crushing to evaluate mineral liberation and degree of size reduction. The results indicated that approximately 30% of the sample reported to the 2.36 mm fraction, while 9.2% reported to the pan fraction (<0.5 mm). Table 13.3 shows the average size distribution of the products from six secondary HPGR runs.

TABLE 13.3
SIZE DISTRIBUTION OF SECONDARY HPGR PRODUCT EXTRACT

Screen	Mass	Cumulative
mm	*%*	*%*
12.5	1.6	1.6
8.0	19.4	21.1
5.6	18.4	39.5
2.36	30.0	69.4
1.18	13.8	83.2
0.5	7.5	90.8
Pan	9.2	100.0
Source: **SRC Testwork**		

13.3.1.2 Implications for Comminution and Early Rejection

The existing results indicate that the calcite phase within the mineralized material exhibits relatively high hardness and commonly occurs as visible veins. This mineralogical characteristic suggests the potential for selective rejection of calcite-rich material through screening following coarse crushing. Testwork indicates that optimum liberation of coarse calcite particles occurs within the 8 to 16 mm particle size range, which is achievable using conventional High Pressure Grinding Rolls ("HPGR"). In contrast, the organic-rich shale fraction that hosts many of the key payable metals appears to have comparatively lower hardness. This difference in hardness between calcite and the organic shale matrix may present opportunities for selective comminution and early-stage gangue rejection.

13.3.1.3 Gaps and Recommendations

Conduct a comprehensive comminution testwork program on representative diamond drill core samples from each major metallurgical domain to determine mineralization variability:

- Bond Crushing Work Index ("CWi")
- Bond Ball Mill Work Index ("BWi")
- Bond Abrasion Index ("Ai")
- Unconfined Compressive Strength ("UCS")
- SMC Test
- HPGR Particle Sizing.

13.3.2 Gravity Separation

13.3.2.1 Coarse Density Tests (HLS and HMS)

Heavy liquid separation testwork was performed in 2013 for the Hatch Scoping Study. The testwork was conducted at six different densities for crushed mineralization screened to 2 to 5 mm and 5 to 10 mm, respectively. The heavy liquid tests showed sharp separation of a very high sulphide sink at a specific gravity ("SG") of 3.1. However, by mass split, only 20 to 40% of the S in these size fractions were recovered to sinks. This quantifies the mass distribution of coarse nodular versus disseminated sulphide within the organic phase. Optimal nodular sulphide liberation occurs in the 2 to 5 mm range.

Heavy media separations were performed in cyclones using ferrosilicon media, as is standard at a SG greater than 2.0. The separation was performed in two stages, with the first stage separating at 3.1 SG for the separation of sulphides, and overflow from the first stage feeding the second at 2.56 SG for the separation of the calcite phase. The feed was screened into 2 to 5 mm and 5 to 10 mm size fractions. The 3.1 SG separation effectively concentrated sulphides into a small, high-density fraction (4 to 7% mass pull), demonstrating that sulphide minerals are strongly associated with the highest density material. Organic carbon shows strong enrichment in the overflow fraction, suggesting association with lighter shale material. Uranium predominantly reported to the low-density overflow fraction in both size classes (78 to 80% recovery), indicating that uranium is not primarily associated with the high-density sulphide fraction. The high mass pull to overflow (~73 to 77%) suggests limited mass rejection opportunity unless targeting sulphide removal specifically.

13.3.2.2 Fine Size Gravity Tests (Knelson, TBS)

Two gravity concentration approaches were evaluated on the medium–fine size fraction to assess the potential for density-based upgrading of uranium and sulphides:

- Knelson centrifugal concentration (laboratory-scale unit)
- Teetered Bed Separator (“TBS”).

The test material was screened at 500 µm, with the undersize directed to the Knelson concentrator and the oversize treated in the TBS.

The Knelson concentrator produced:

- Concentrate mass pull: 7.2%
- Uranium recovery to concentrate: 5.0%
- Sulphur recovery: 13.0%
- Organic carbon recovery: 7.2%.

The TBS operates using hindered-bed fluidisation to establish density separation within an upward water flow. Lighter particles overflow, while denser particles are withdrawn as underflow.

TBS produced:

- Underflow mass pull: ~22 to 26%
- Sulphur recovery to underflow: 18 to 30%
- Uranium recovery to underflow: ~22 to 30%
- Organic carbon recovery to overflow: ~23 to 24%.

Uranium did not demonstrate strong association with the highest-density fractions in either gravity device. Both Knelson and TBS showed some sulphide concentration, however, concentration gradients were modest. Organic carbon consistently reported to lighter fractions, reflecting its low-density association.

13.3.2.3 Conclusions and Limitations

Gravity separation testwork was undertaken to evaluate the potential for upgrading uranium-bearing material and rejecting gangue minerals using density-based techniques. The results indicate that although some degree of mineral separation was achieved, the overall upgrading ratios and recoveries were limited and are unlikely to justify the implementation of centrifugal or teetered-bed gravity concentration in a commercial processing circuit without further optimization. Testwork results showed that a significant proportion of uranium reported to the overflow fraction, suggesting that density-based separation methods such as heavy media separation ("HMS") may not be effective as a primary upgrading technique for uranium-bearing material. While theoretical heavy liquid separation tests suggested the potential for selective separation of calcite and sulphide phases, practical test results demonstrated that the achievable separation efficiencies are insufficient to justify the inclusion of an HMS circuit to remove calcite or sulphides for pre-concentration of the organic shale fraction.

13.3.2.4 Recommendations

Gravity concentration should not be included as a primary upgrading stage for uranium.

13.3.3 Flotation

13.3.3.1 Historical Sulphide Flotation (2010)

A single test directed towards the flotation of sulphides was conducted to scope the possibility of removing sulphides from the mineralized material in 2010.The results of the testwork are shown in Table 13.4 2010 PEA Flotation Results. The results were poor due to carbon floating strongly together with the sulphides. Sulphide removal was effective, however, flotation was not selective.

TABLE 13.4
2010 PEA FLOTATION RESULTS

Fraction	Distribution (%)					
	Mass	$C_{(tot)}$	S^{2-}	Mo	U	V
Rougher Conc.1	55.6	84.9	61.4	88.9	71.3	60.3
S^{2-} Rougher Conc.1	19.3	8.2	36.8	9.1	11.4	15.0
Rougher Conc. 2	1.1	0.6	0.6	0.5	0.8	1.4
Metallic Mineral Float Conc.	1.8	0.6	0.4	0.2	1.2	2.1
Rougher Tailings	22.2	5.8	0.7	1.3	15.3	21.2

Note: *C(tot) = total carbon.*

13.3.3.2 Historical Carbon / Coal Style Flotation (2013)

Bench-scale froth flotation tests were conducted to evaluate the floatability of the mineral phases using two different reagent schemes:

- Coal-style flotation, targeting concentration of the organic carbon phase (which hosts most of the uranium).

- Sulphide flotation, targeting concentration of sulphide minerals.

The feed sample was screened at 2 mm and ground in a laboratory ball mill to:

- P_{80} ~100 µm for sulphide flotation.
- ~50 µm for coal-style flotation.

The coal flotation used kerosene as the collector and MIBC as the frother, while the sulphide flotation used PAX as the collector and MIBC as the frother. Table 13.5 shows the results reported for the tests.

TABLE 13.5
HATCH FLOTATION RESULTS EXTRACT

Component	Feed	Conc.	Tailings	Recovery
Coal-style flotation	*wt%*	*wt%*	*wt%*	*%*
Total	100.00	30.20	-	30.20
Organic Carbon	15.16	24.60	9.67	59.70
Inorganic Carbon	0.49	0.06	0.74	4.50
Sulphur	4.66	4.71	4.63	37.60
Sulphide flotation				
Total	100.00			6.80
Sulphur	5.50	5.08	5.53	7.90
Source: **SRC Testwork**				

The sulphide flotation test results showed limited upgrading of sulphur with similar sulphur grades being reported in the concentrate (5.08%) and tailings (5.53%). These results indicate poor flotation response of sulphides. The organic carbon flotation tests showed stronger results, with a mass pull of 30.2% and a total recovery of 59.7%, resulting in an overall upgrade of 198%. Although the organic carbon flotation shows greater selectivity than the sulphide flotation, a total recovery of 59.7% in a 30.2% mass pull is poor, however, the potential for reagent and operating condition optimization is present.

13.3.3.3 Potential Roles for Flotation in the Flowsheet

The mineral grades achieved do not justify flotation as a primary upgrading method for metal recovery. However, the testwork suggests several potential processing opportunities, such as:

- The selective removal of calcite into a low-tonnage stream, which could reduce acid consumption in downstream hydrometallurgical circuits. In this context, flotation may be more appropriately applied as a secondary or intermediate beneficiation stage rather than as the primary upgrading method.

- The production of an organic carbon-enriched residue that could potentially be utilised as a fuel source in an ancillary combustion unit for steam and electricity generation, thereby contributing to process energy requirements.

13.3.3.4 Assumed Mica Flotation Recoveries for PEA

Based on historical testwork and benchmarking against previous projects and METS testwork programs, the flotation recoveries used for the PEA are presented in Table 13.6 at an assumed 50% mass recovery with mica flotation removing 80% of muscovite from the stream.

TABLE 13.6
ASSUMED MICA FLOTATION RECOVERIES

Type	Uranium (%)	Vanadium (%)	Potassium (%)	Nickel (%)	Molybdenum (%)	Iron (%)	Calcium (%)
Recoveries	90	95	70	90	90	90	15

13.3.3.5 Gaps and Recommendations

A comprehensive flotation testwork program is required for the Deposit. The program should include rougher, bulk, cleaner flotation paired with reagent, grind and residence time variability testwork.

13.3.4 Leaching

Leaching tests were conducted for the 2010 PEA under varying conditions. The tests indicated that up to 91% extraction of uranium and 99% extraction of molybdenum can be obtained by alkaline leaching while vanadium extraction is essentially zero despite the varying conditions.

13.3.4.1 Historical Alkaline and Acid Leach Results (2010, 2013)

Both alkaline and acid leaching beaker tests were done to determine the amenability of uranium to two leaching methods: acid and alkaline leaching for the “Hatch Scoping Study” in 2013.
The alkaline tests were conducted with these baseline conditions:

- P_{80}: 100 µm
- Residence Time:72 hours
- Temperature: 90ºC
- Reagent Additions: 40 g/L Na_2CO_3 10 g/L $NaHCO_3$, pH 9.6, no oxidant addition.

13.3.4.2 Behaviour of U, V, Mo, Ni and Reagent Demand

Test results are shown in Table 13.7. Uranium recovery is relatively high at ~70% with molybdenum moderately high at ~45% showing that the U and Mo in the mineralization are receptive to alkaline leaching.

TABLE 13.7
RECOVERY OF URANIUM AND MOLYBDENUM IN ALKALINE LEACH TESTS EXTRACT

Component	Mean Recovery		Median Recovery	
	feed-residue	leachate-residue	feed-residue	leachate-residue
Uranium	70%	72%	68%	72%
Molybdenum	47%	49%	42%	49%
Source: **SRC Testwork**				

The acid tests were conducted using these baseline conditions:

- P_{80}: 300 µm
- Residence Time: 48 hours
- Temperature: 40ºC,
- Reagent Additions: sulphuric acid to pH 1.7, 4 g/L of Fe^{3+} (as ferric sulphate), no other oxidant addition.

The results indicate that the majority of uranium in the mineralized material is readily leachable in the presence of a moderate ferric iron concentration. Increasing the ferric dosage, or adding supplementary oxidants such as $NaClO_3$, only marginally improved uranium extraction by enhancing oxidation. Average sulphuric acid consumption during these tests was approximately 50 kg per tonne of mineralization. Results are shown in Table 13.8.

TABLE 13.8
URANIUM ACID LEACHING RECOVERIES EXTRACT

Component	Mean Recovery		
	feed-residue	leachate-residue	Average
Uranium	63%	61%	62%
Molybdenum	7%	2%	4%
Vanadium	9%	1%	5%
Nickel	20%	18%	19%
Zinc	35%	33%	34%
Source: **SRC Testwork**			

The direct leaching testwork conducted on the samples indicates that vanadium exhibits strong resistance to direct alkaline leaching, suggesting that a prior thermal activation step such as roasting may be required to improve metal solubility and overall extraction efficiency. This behaviour is consistent with alum shale deposits, where vanadium is often associated with organic matter or refractory mineral phases, which limit direct dissolution under standard leaching conditions.

Testwork results also show that nickel extraction improves with longer leach residence times, indicating slower dissolution kinetics for nickel-bearing phases. This suggests that any future hydrometallurgical process design may require extended leaching durations, aggressive leaching conditions or using leaching as a secondary process to achieve acceptable recoveries.

13.3.4.3 Assumed Leach Recoveries for PEA Options

Based on historical testwork and benchmarking against previous projects and METS testwork programs, the leaching recoveries for each developed process option are presented in Table 13.9.

TABLE 13.9
ASSUMED LEACHING RECOVERIES

Process Option	Uranium (%)	Vanadium (%)	Potassium (%)	Nickel (%)	Molyb-denum (%)	Copper (%)	Zinc (%)
3. Agitated Tank Leach	70	60	45	40	40	40	40
4. HPGR to Direct Agitated Tank Leach	60	50	35	40	40	40	40

13.3.4.4 Gaps and Recommendations

Metals such as vanadium and uranium are frequently associated with organic carbon and fine-grained silicate minerals, which can limit metal liberation and reduce leaching efficiency without adequate pre-treatment or fine grinding. The available leaching testwork suggests that more aggressive leaching pathways should be considered such as a pressure oxidative leach ("POL") or pug roast. Since the metals, particularly uranium and vanadium, are very low, ion exchange followed by solvent extraction should be investigated as a process step.

13.3.5 Roasting and Thermal Treatment

13.3.5.1 Historical Roasting Results (2010, 2013)

Several small-scale roasting tests directed toward establishing conditions suitable for subsequent vanadium recovery were conducted. Both single stage and two-stage roasting were evaluated at a range of temperatures, with and without additives in a muffle furnace and a small fluid bed roaster. Vanadium extractions of up to approximately 90% were obtained and this level was selected for use in the 2010 PEA. No results tables or worksheets were found in the dataset to support the findings stated in the 2010 PEA.

Vanadium roasting and leaching tests were conducted as a secondary stage after uranium leaching was performed in the 2013 "Hatch Scoping Study". The overall goal of the testwork program was to understand the roasting behaviour of the uranium-leached residues, and subsequent vanadium leaching of the roasted material. Vanadium leaching of alkaline leach residue was conducted. The base case roasting conditions were:

- Temperature: 850ºC
- Residence Time: 1.5 hours
- Reagent Addition: 6% NaCl by weight.

Most of the leach tests were performed with sodium hydroxide and sodium carbonate at pH 13. The test results are shown in Table 13.10.

TABLE 13.10
VANADIUM ROASTING AND LEACHING AFTER ALKALINE LEACHING OF URANIUM EXTRACT

Test #	V	Mo	Conditions
	Recovery	Recovery	
#1	61%	68%	Base
#2	75%	83%	Base
#3	18%	58%	water leach
#4	26%	23%	gradual cooling
#5	46%	121%	no salt roast
#6	71%	77%	high salt roast
#7	63%	66%	875°C
#8	68%	97%	825°C
#9	67%	73%	long roast (3h)

Source: **SRC Testwork**

One test using conventional salt roasting, followed by pure-water leaching (no reagents) achieved 18% V extraction and 58% Mo extraction, indicating an opportunity to recover metal values with reduced operating cost. Gradual cooling was clearly detrimental to both V and Mo recovery, yielding outlier results of 26% and 23%, respectively. The vanadium recovery behaviour is not yet well understood, and additional testwork is required to better characterize key variables. Tests using strong caustic leaching produced relatively scattered results, with V extractions ranging from 46% to 75%. Molybdenum recoveries ranged from 58% to 97%, suggesting that Mo is readily recoverable after roasting under a wide range of conditions.

Vanadium leaching of acid leach residue was conducted. The baseline conditions were:

- Temperature: 850°C
- Residence time: 1.5-hour with rapid quenching.

The results are shown in Table 13.11.

TABLE 13.11
VANADIUM ROASTING AND LEACHING AFTER ACID LEACHING OF URANIUM EXTRACT

Test #	V	Mo	Conditions
	Recovery	Recovery	
#1	5%	51%	base
#2	16%	62%	base
#3	15%	61%	gradual cooling
#4	2%	67%	no salt roast
#5	14%	66%	high salt roast
#6	12%	67%	base
#7	12%	56%	875°C
#8	13%	62%	825°C
#9	22%	73%	long roast (3h)
#10	24%	66%	long roast (3h)
Source: **SRC Testwork**			

Roasting without salt resulted in almost no vanadium extraction (2%) in the water leach. Except for one outlier, tests conducted using the standard 6% salt plus 1% per 1% vanadium roast achieved V recoveries between 12% and 24%, comparable to alkaline residue test number three.

13.3.5.2 Summary

Testwork indicates that the mineralization is amenable to roasting, particularly for the extraction of vanadium and uranium. However, salt roasting does not appear to be the most favourable approach. Based on historical testwork and benchmarking against comparable projects and METS testwork programs, alternative methods such as pug roasting and pressure oxidation ("POX") may offer more suitable processing routes. The breakdown of alum shale requires relatively high temperatures and/or pressures, highlighting the energy-intensive nature of these options.

Testwork to date demonstrates that vanadium recovery from alum shale remains challenging and is highly sensitive to both roasting conditions and subsequent leach chemistry. Strong caustic leaching of roasted residues has the potential to improve vanadium extraction; however, the associated reagent consumption is significant and may materially impact operating costs. While longer roasting residence times have been shown to enhance vanadium recovery, such conditions may not be economically viable where roaster capacity is a key capital cost driver.

13.3.5.3 Assumed Recoveries for Roast Based PEA Options

Molybdenum recovery from roasted residues is higher and less sensitive to process conditions. This indicates that molybdenum recovery is likely to be achievable across broader operating ranges, whereas vanadium recovery will require further optimisation and remains a key technical uncertainty. Based on historical testwork and benchmarking against previous projects and METS testwork programs, the leaching recoveries for each developed process option are presented in Table 13.12.

Table 13.12 Assumed Recoveries							
Process Option	**Uranium (%)**	**Vanadium (%)**	**Potassium (%)**	**Nickel (%)**	**Molyb-denum (%)**	**Copper (%)**	**Zinc (%)**
1. Pug Roast Leach	90	80	60	80	80	80	80
2. Pressure Acid Leach	90	80	60	80	80	80	80

13.3.5.4 Gaps and Recommendations

Additional testwork is needed to refine roasting parameters, evaluate cost-effective leaching strategies, and improve confidence in the overall vanadium extraction pathway. The available leaching testwork is limited and insufficient to fully define an optimal hydrometallurgical processing route, and further systematic roasting and leaching studies would be performed to properly evaluate metal recovery behaviour for the Deposit.

13.4 DOWNSTREAM PROCESSING AND PRODUCT RECOVERY (CONCEPTUAL)

13.4.1 Potassium Sulphate Production (SOP)

No dedicated testwork has been completed on potassium sulphate production for this PEA, and information on downstream product recovery is limited. The crystallisation process for SOP production is well established within the industry, with standard operating conditions widely documented. Based on publicly available data and industry practice, it was assumed that potassium sulphate could be recovered to meet market specifications. The potassium sulphate product specification requirements are:

- $K_2O > 50\%$
- S > 17%
- Cl < 1%.

In the initial stages, potassium typically crystallises as potassium alum, making aluminium the principal impurity of concern, with calcium also important as it is used to separate aluminium. Based on benchmarking against previous projects and METS testwork programs, SOP recovery for each developed process option is presented in Table 13.13. Due to the higher free acid required to perform a pressure acid leach, it was assumed that the recovery of potassium in the PAL flotation circuit would be lower.

TABLE 13.13 SOP OVERALL RECOVERY	
Process Options	**SOP Crystallization (%)**
1. Pug Roast Leach	70
2. Pressure Acid Leach	48
3. Agitated Tank Leach	45
4. HPGR to Direct Agitated Tank Leach	35

13.4.1.1 Recommendations

Detailed testwork should be undertaken to establish the most appropriate operating conditions and circuit configurations for the process. In particular, optimization of precipitation conditions for final products will require bulk samples to generate sufficiently representative process solutions. The testwork will require up-front planning and representative drill core. Test plans will need to be prepared to estimate the weight of drill core required and details on comminution, extraction, and variability testwork. The testwork will likely take six months given the number of unit operations, then a summary report will be prepared. It is estimated that a nine-month period will be required to complete this work before a PFS could be undertaken.

13.4.2 Uranium and Vanadium IX/SX

No ion exchange or solvent extraction testwork has been carried out on Viken leach solutions within the available metallurgical dataset, and information on downstream precipitation and product recovery is also limited. Solvent extraction ("SX") is a liquid–liquid separation process used to selectively transfer dissolved metals from an aqueous solution into an immiscible organic phase containing a specific extractant. In this process, the metal ions present in the leach solution chemically interact with the extractant in the organic phase, allowing them to be selectively extracted while most impurities remain in the aqueous raffinate. The loaded organic phase is then contacted with a stripping solution to recover the metal into a purified aqueous stream suitable for downstream processing or precipitation. Solvent extraction is widely used in hydrometallurgical operations due to its high selectivity, scalability, and ability to produce high-purity metal solutions.

For low tenor leach solutions such as those anticipated for this project, solvent extraction is typically deployed after ion exchange ("IX"). In this configuration, IX is used first to capture and pre concentrate metals from dilute liquors, and the resulting enriched eluate is then treated by solvent extraction ("SX") to further purify and upgrade the metal concentration ahead of final product recovery. Ion exchange is a reversible process in which dissolved ions (for example calcium, magnesium or other contaminants) are exchanged for more desirable ions (such as sodium or hydrogen) of the same charge on a solid resin matrix, and it is widely applied in water softening, purification, deionization and industrial effluent treatment. Solvent extraction is a mature, conventional hydrometallurgical technology with broad industrial application. Drawing on benchmarking from previous projects and METS' experience, the PEA adopts the following uranium and vanadium performance assumptions at this stage of the circuit:

- 95% Extraction efficiency
- No Scrubbing required
- 97% Stripping efficiency
- 100% Ion Exchange efficiency.

13.4.2.1 Gaps and Recommendations

Targeted uranium, vanadium ion exchange and solvent extraction and stripping testwork should be undertaken to establish the most suitable extractants, operating conditions, and circuit configurations.

13.4.3 Metal Precipitation

No data was available for metal precipitation testwork within the metallurgical dataset reviewed for this study. In addition, there is limited information regarding downstream precipitation and product recovery, including the conditions required to produce uranium and vanadium products that meet marketable purity specifications. Metal precipitation represents a key stage in the hydrometallurgical flowsheet, as it directly influences product quality, impurity rejection, and overall recovery efficiency.

Precipitation and separation of molybdenum, nickel, and zinc are well-established processes with standard operating conditions documented in industry. Conventional conditions and recoveries have been adopted for the purposes of this PEA. Based on historical testwork and benchmarking against previous projects and METS testwork programs, the precipitation recoveries for each developed process option are presented in Table 13.14.

TABLE 13.14
METAL PRECIPITATION RECOVERIES

Process Options	Nickel (%)	Molybdenum (%)	Zinc (%)
Pug Roast Leach	32	52	47
Pressure Acid Leach	32	52	47
Agitated Tank Leach	20	26	27
HPGR to Direct Agitated Tank Leach	25	29	30

13.4.3.1 Gaps and Recommendations

It is recommended that dedicated metal precipitation testwork should be conducted for the MMP stream. Testwork should investigate precipitation reagents, operating conditions, impurity behaviour, and product washing requirements to determine achievable product grades and recovery efficiencies. The results will provide key design criteria for the product recovery circuits, support flowsheet development, and improve the reliability of capital and operating cost estimates for the Viken Project.

13.4.4 Hydrocarbon and Energy Recovery

Some hydrocarbon testwork was reported in the 2013 “Hatch Scoping Study” as shown in Table 13.15.

TABLE 13.15
HYDROCARBON TESTWORK RESULTS EXTRACT

Property	Result	ASTM Analytical Method
Density	1076 kg/m^3	D4052
Pour Point	-6 °C	D97
Total Sulphur	0.21 wt%	D4239
Carbon	91.72 wt%	D5291
Hydrogen	8.03 wt%	D5291
Nitrogen	<0.01 wt%	D5291
Oxygen	<0.1 wt%	D5291
Source: AITF Report: “Viken Project Hydrocarbon Processing Test Runs”		

Overall, the available hydrocarbon testwork indicates limited potential for direct shale oil recovery. Future studies should focus on evaluating energy recovery or other value-adding pathways for the organic component of the shale i.e. the use of Circulating Fluidized Bed (“CFB”) combustion technology, which could utilise the organic carbon in the shale as a fuel source for power generation. Integration of a CFB unit into the process flowsheet may therefore provide a value-added opportunity, potentially contributing to on-site energy generation and improving the overall project economics.

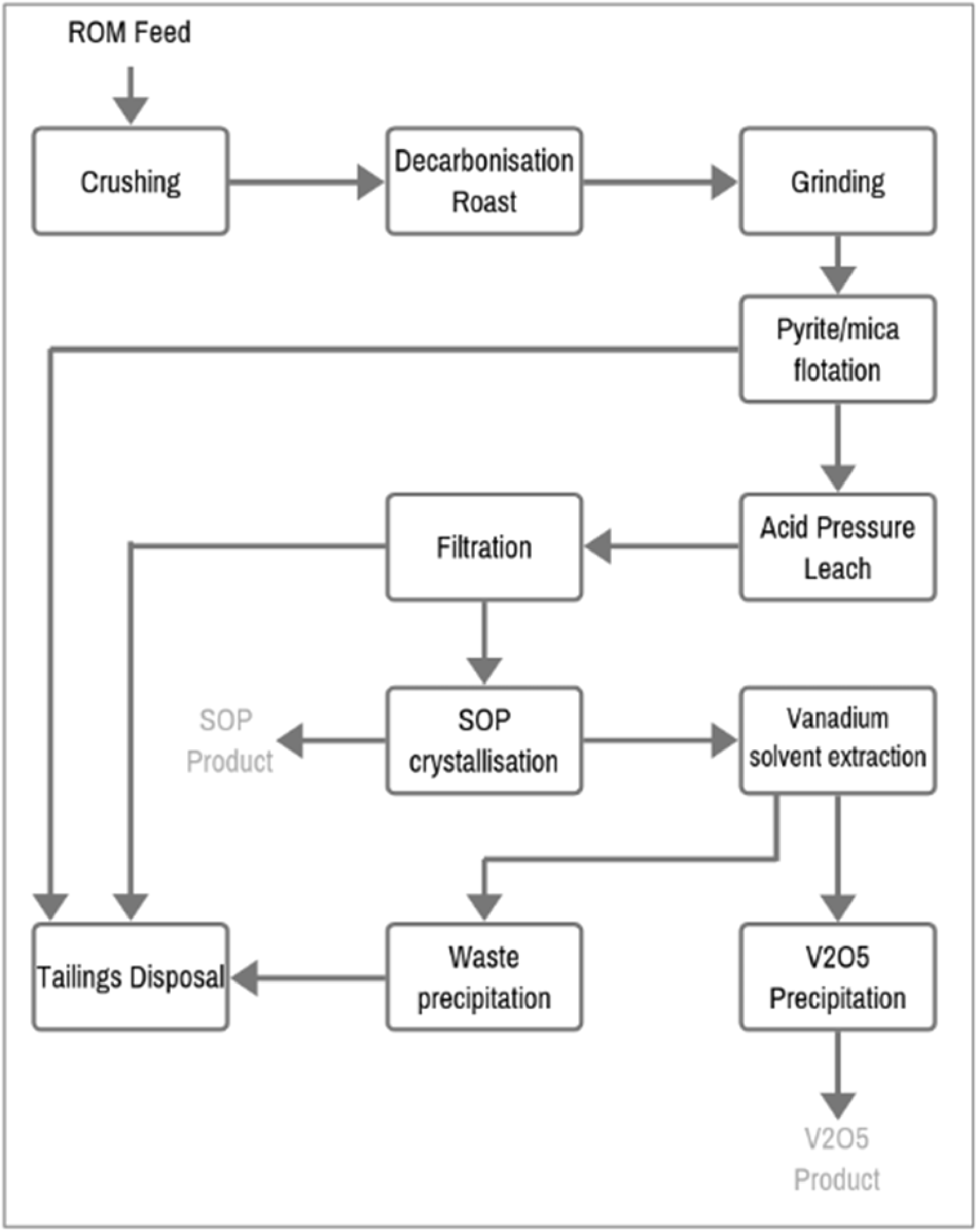
13.4.5 Aura Energy Public Domain Reports ASX

In 2023 Aura Energy published a Scoping Study for their Häggån project in Sweden. The Häggån Deposit is comprised of alum shale similar to the Viken Mineral Resource, hence its metallurgical testwork was reviewed for this PEA.

13.4.5.1 Haggan Processing Route

The flowsheet developed for the scoping study is shown in Figure 13.1.

FIGURE 13.1 HÄGGÅN PROCESS FLOWSHEET



Source: *Aura (2023)*

The process involves crushing run-of-mine mineralization, followed by a decarbonisation roast and grinding to a fine size using a ball mill. A concentrate rich in pyrite and vanadium bearing mica is then recovered by selective froth flotation, with much of the calcite rejected to the flotation tailings residue stream. The concentrate, containing 85% of the vanadium in 50% of the run of mine ("ROM") mass, is then leached at elevated temperature and pressure in an autoclave with sulphuric acid. This results in the breakdown of the mica, releasing vanadium and other valuable products into solution. This stage of the process is referred to as pressure acid leaching.

The discharge slurry from the pressure acid leach is dewatered in a thickener from which pregnant solution reports to the overflow and remaining solid material reports to the underflow and is discharged to waste, with excess solution removed through a filter press. The vanadium-rich solution passes to a potassium salt crystalliser for recovery of SOP, followed by sulphide precipitation to produce a Mixed Sulphide Precipitate containing Ni, Mo and Zn. The pH of the vanadium-rich solution is adjusted using limestone sourced from waste rock at the site, allowing iron to undergo reduction and be removed.

Vanadium is extracted from the purified solution by solvent extraction and precipitated as ammonium metavanadate, before being further purified by rotary calcination and converted to vanadium pentoxide (V_2O_5) flake product. The barren solution from solvent extraction undergoes a process of calcium phosphate precipitation that will encapsulate impurity metals remaining in solution, importantly including uranium, as stable and safe compounds, which can then be disposed of in the dry tailings' storage facility. At the time of the study, Sweden had removed uranium from the mining act making its production illegal.

The testwork results and assumptions including comparable results from technical and project studies used in the scoping study are shown in Table 13.16.

TABLE 13.16
HÄGGÅN RECOVERY RATES EXTRACT

	Scoping Study Assumption		
	V_2O_5	K_2SO_4	Ni, Mo, Zn
Flotation V_2O_5 recovery	85%	85%	85%
Pressure Leaching V_2O_5 recovery	96%	83%	96%
Solvent extraction V_2O_5 recovery	98%	-	
SOP crystallisation	-	85%	
MSP precipitation	-	-	98%
Overall V_2O_5 recovery	**80%**	**60%**	**80%**

13.4.5.2 Uranium Processing

A subsequent scoping study was carried out as part of the 2023 Scoping Study exploring the production of Uranium Oxide Concentrate ("UOC"), or Yellowcake from solution using solvent extraction. The Alum Shale making up the Häggån Mineral Resource contains a large endowment of uranium at consistent but low grade. The uranium concentration averages 140 ppm U_3O_8, with uranium characterized as amorphous inclusions hosted with the carbon matrix of the Alum Shale. The flowsheet was modified by replacing the calcium phosphate precipitation circuit with a uranium precipitation and purification circuit. This included solvent extraction, precipitation of uranium as ammonium diuranate, purification and final precipitation as U_3O_8 powder.

13.4.5.3 Recommendations

It is recommended that metallurgical data and process design information from the Häggån Scoping Study can be used to support the development of the Viken Deposit PEA. Following the assumption that the Häggån Deposit is similar in mineralogy to the Viken Deposit the processing routes evaluated in the scoping study are applicable. As a result, the available metallurgical performance data, process assumptions, and conceptual flowsheet provide a reasonable technical basis for preliminary evaluation at the PEA stage.

13.4.6 Overall Recoveries

Based on historical testwork and benchmarking against previous projects and METS testwork programs, the key overall recoveries for each developed process option are presented in Table 13.17.

TABLE 13.17
KEY OVERALL RECOVERIES

Process Options	U_3O_8 (%)	V_2O_5 (%)	SOP (%)	Nickel (%)	Molybdenum (%)	Zinc (%)
1. Pug Roast Leach	91	68	70	32	52	47
2. Pressure Acid Leach	90	67	48	32	52	47
3. Agitated Tank Leach	67	51	45	20	26	27
4. HPGR to Direct Agitated Tank Leach	61	44	35	25	29	30

14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The Mineral Resource Estimate presented herein has been prepared following the guidelines of the Canadian Securities Administrators' NI 43-101 and Form 43-101F1, and in conformity with generally accepted "CIM Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. Mineral Resources have been classified in accordance with the "CIM Standards on Mineral Resources and Reserves: Definition (2014) and Best Practices (2019)" as adopted by CIM Council.

> *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.*

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the Mineral Resource will be converted into a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration. The estimate of

Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant factors.

The Authors consider that the Mineral Resource Estimate and Mineral Resource classification represent a reasonable estimation of the global mineralization for the Viken Project with regard to compliance with generally accepted industry standards and guidelines, the methodology used for estimation, the classification criteria used and the implementation of the methodology in terms of Mineral Resource estimation and reporting.

All Mineral Resource estimation work reported herein was carried out or reviewed by Fred Brown, P.Geo. and Eugene Puritch, P.Eng., FEC, CET, both independent Qualified Persons as defined in NI 43-101 by reason of education, affiliation with a professional association and past relevant work experience. This Mineral Resource Estimate is based on information and data supplied to and verified by the Authors.

Mineral Resource modelling and grade estimation were carried out using Gemcom GEMS™ software. Variography was carried out using Snowden Supervisor™. Open-pit optimization was carried out using NPV Scheduler™ software.

This Report and updated Mineral Resource Estimate supersede all previous Technical Reports and Mineral Resource Estimates for the Viken Project. The effective date of this Mineral Resource Estimate is June 26, 2026.

14.2 DATABASE

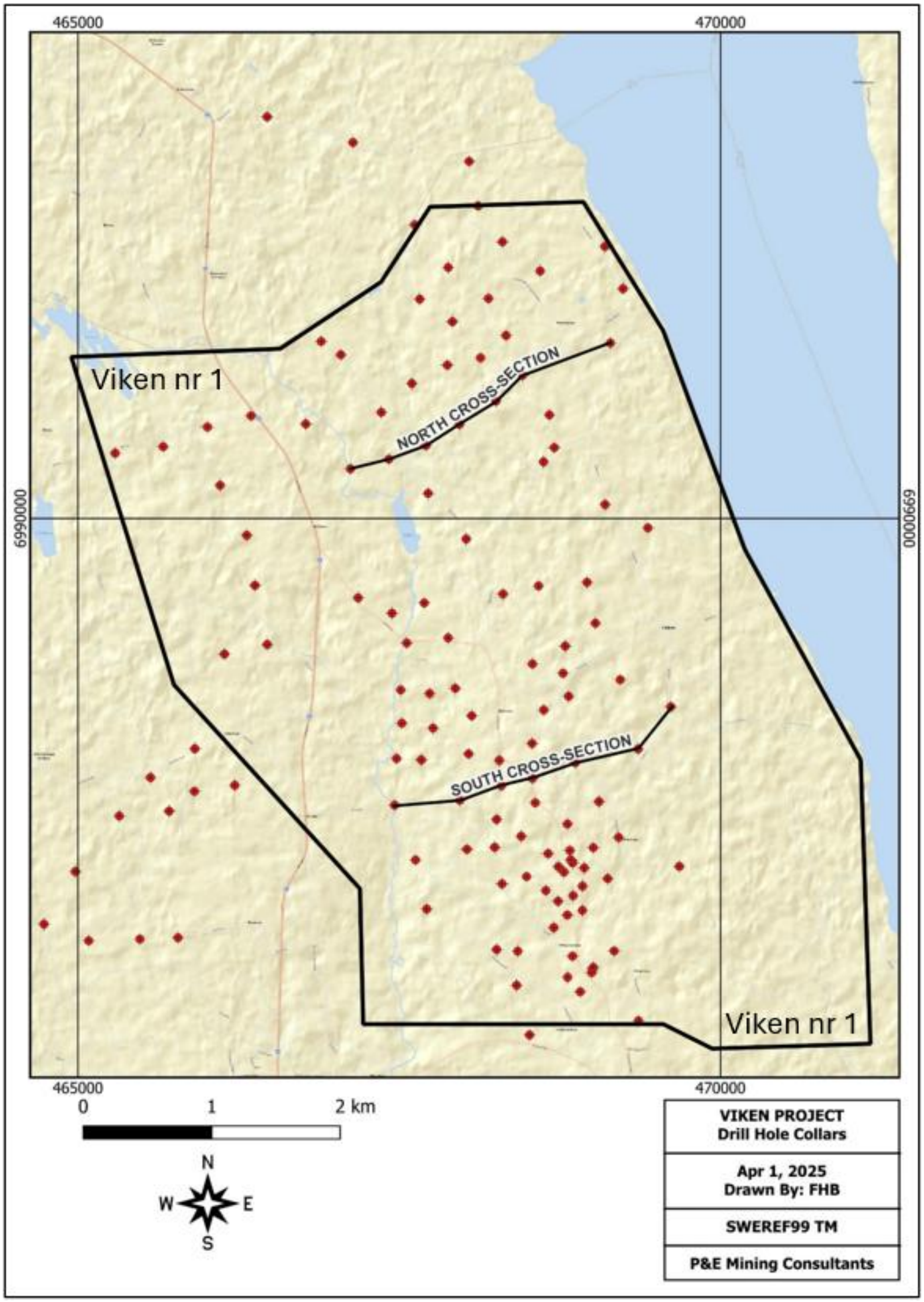
Data were supplied in the form of Excel spreadsheets and ALS Chemex assay certificates, and takes into account a total of 152 drill holes on the Viken nr 1 mineral licence and adjacent licences (Figure 14.1).

Data supplied include collar easting, northing and elevation coordinates, lithological information, assay data, and bulk density data. All assay data were provided in parts-per-million ("ppm"). Collar coordinates were surveyed by Mätservice i Jämtland AB using a Topcon Hiper Network RTK GPS with the elevation corrected using Swen01L. A 10 m contour digital elevation model relative to SWEREF99 TM was also supplied, and Property and licence boundary coordinates of the Project area.

Of the 152 drill holes available, a total of 122 drill holes within or adjacent to the Viken nr 1 licence limits were used for grade estimation and modelling of the Mineral Resources. The average nearest neighbour collar distance within the Project area is 422 m.

Collar data, lithological interval data, bulk density data and assay interval data were imported into an Access format Gemcom™ database and validated, with no significant discrepancies noted.

FIGURE 14.1 VIKEN DRILLING*



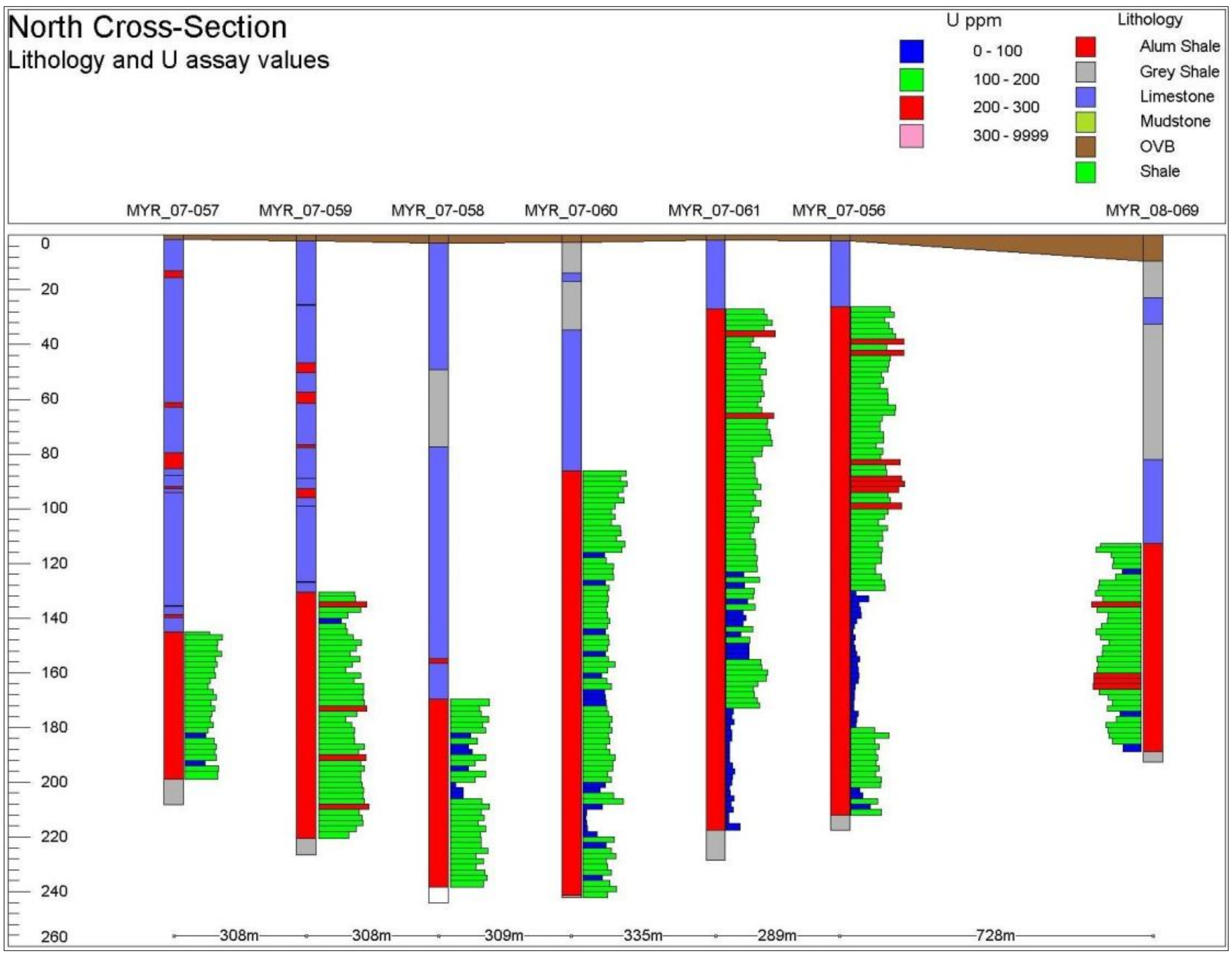
Source: *P&E (2025)*

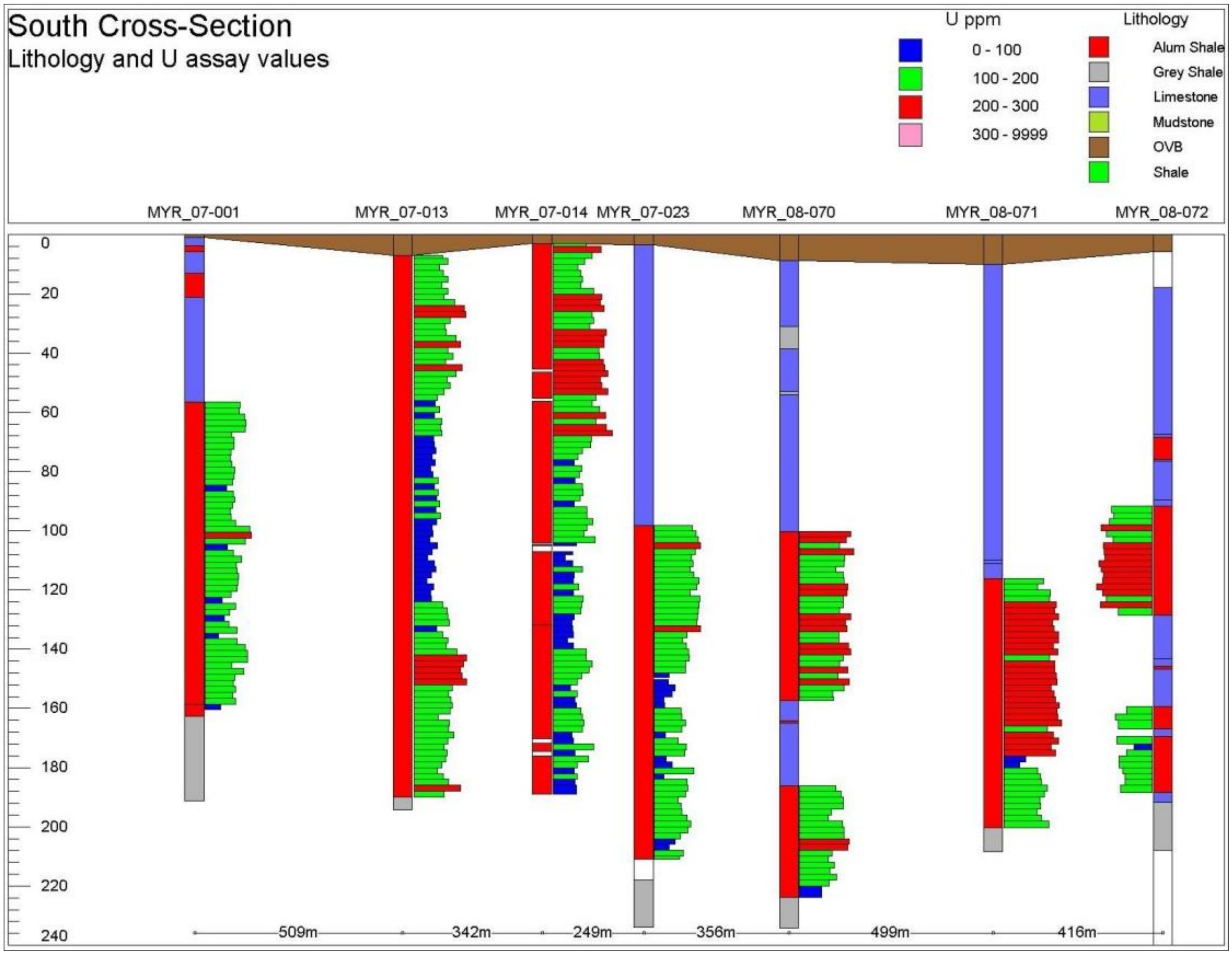
Note: **Only the Viken nr 1 mineral license is shown. Other licenses not shown for clarity. See Figure 4.3 for all license areas.*

14.3 LITHOLOGICAL MODEL

A 3-D geological model was constructed by the Authors based on lithological units as defined by the supplied drill hole logs. The Black Alum Shale and Central Alum Shale were combined into a single mineralized shale domain (Figure 14.2). Lithological units were delineated on east-to-west cross-sections spaced every 20 m across the Viken Property. Polygonal nodes were snapped directly to drill hole contacts, in order to generate a true 3-D representation of the Viken Deposit geology. Wireframe models and surfaces representing the topography, overburden, basement and the mineralized shale were constructed.

FIGURE 14.2 LITHOLOGICAL CROSS-SECTIONS - NORTH AND SOUTH





Source: *P&E (2025)*

14.4 ASSAY SUMMARY STATISTICS

Assay data available for the Viken Mineral Resource Estimate include copper (Cu), molybdenum (Mo), nickel (Ni), uranium (U), vanadium (V) and zinc (Z). A reduced number of assays are available for potassium (K), phosphorus (P), lanthanum (La), cerium (Ce) and yttrium (Y). Summary statistics (Table 14.1) were calculated for assay data sets within the modelled mineralized shale.

Table 14.1 Assay Summary Statistics within the Mineralized Shale Domain						
Statistic	**U (ppm)**	**Ni (ppm)**	**Cu (ppm)**	**Zn (ppm)**	**Mo (ppm)**	**V (ppm)**
Sample Count	6,991	6,922	6,815	6,922	6,991	6,991
Minimum	0.5	10	3	7	0.24	18
Maximum	296	637	213	7120	2,130	3,740
Mean	137	312	115	409	238	1446
Standard Deviation	52.11	122.86	25	228.2	98.13	731.23
Coefficient of variation	0.38	0.39	0.22	0.56	0.41	0.51
Statistic	**K (%)**	**P (ppm)**	**La (ppm)**	**Ce (ppm)**	**Y (ppm)**	
Sample Count	5,158	5,158	5,158	5,158	5,158	
Minimum	0.29	220	3.60	6.73	7	
Maximum	5.02	10,000	64.10	170.50	7,120	
Mean	3.12	1,114	38.57	75.98	406	
Standard Deviation	0.54	783	5.01	10.44	233	
Coefficient of variation	0.17	0.70	0.13	0.14	0.57	

14.5 DATA CONDITIONING

The average assay sample length within the mineralized shale domain is 2.03 m, with 95% of the assays having a sample length of 2.0 m. Assay values were therefore composited to 2.0 m length-weighted intervals. Length-weighted composites were calculated within the mineralized shale domain wireframe, starting at the first point of intersection between the drill hole and the mineralized shale wireframe, and halted upon exit from the mineralized shale wireframe. A small number of unsampled intervals were treated as null values. Composites that were <1.0 m in length were discarded. The mineralized shale wireframe was also used to back-tag the appropriate rock code field into the assay and composite drill hole workspaces. The composite data were then extracted to Gemcom™ files for statistical analysis and grade estimation. The compositing process did not add any significant bias to the assay data (Table 14.2).

Table 14.2 Composite Summary Statistics						
Statistic	**U (ppm)**	**Ni (ppm)**	**Cu (ppm)**	**Zn (ppm)**	**Mo (ppm)**	**V (ppm)**
Sample Count	7,096	6,999	6,895	6,999	7,096	7,096
Minimum	1	10	5	7	0.2	18
Maximum	278	635	203	5958	2,130	3,682
Mean	138	314	114	411	239	1,455
Standard Deviation	51	120	24	206	94	712
Coefficient of variation	0.5	0.38	0.21	0.5	0.39	0.49

Table 14.2 **Composite Summary Statistics**						
Statistic	**K (%)**	**P (ppm)**	**La (ppm)**	**Ce (ppm)**	**Y (ppm)**	
Sample Count	5,140	5,140	5,140	5,140	5,140	
Minimum	0.71	220	3.60	6.73	7	
Maximum	4.59	9,610	58.33	144.50	5,958	
Mean	3.12	1,113	38.59	76.00	406	
Standard Deviation	0.50	683	4.61	9.67	209	
Coefficient of variation	0.16	0.61	0.12	0.13	51	

Correlations between metal grades were also calculated for the assay data set. Mo, Ni, V and U display a strong correlation (Table 14.3). A weaker correlation with Cu was also observed.

Table 14.3 **Pearson Correlation Between Assay Values**									
Element	**Ce**	**Cu**	**K**	**La**	**Mo**	**Ni**	**P**	**U**	**V**
Cu	-0.13								
K	0.53	0.11							
La	0.91	-0.01	0.55						
Mo	-0.20	0.56	0.13	-0.09					
Ni	-0.05	0.66	0.40	0.07	0.78				
P	-0.13	-0.05	-0.52	-0.19	0.13	-0.22			
U	-0.07	0.46	0.15	0.01	0.52	0.57	-0.05		
V	-0.05	0.54	0.41	0.07	0.61	0.88	-0.23	0.37	
Y	0.11	0.15	0.05	0.12	0.13	0.15	0.06	0.79	0.04

Extreme values were identified for each grade-element within the mineralized shale domain from the examination of composite histograms and log-probability plots. The values derived were then used as thresholds on composite values in order to reduce the impact of high-grade outlier values during linear block grade estimation. Prior to grade estimation, composite grades were capped to the values listed in Table 14.4.

Table 14.4 **Capping Levels**	
Grade-Element	**Capping Level**
Cu	160 ppm
Mo	370 ppm
Ni	480 ppm
U	220 ppm
V	2,700 ppm

TABLE 14.4 CAPPING LEVELS	
Grade-Element	**Capping Level**
Zn	1,300 ppm
K	4.4%
P	8,000 ppm
La	5.5 ppm
Y	2,000 ppm

14.6 GRADE CONTINUITY

Isotropic experimental spherical semi-variograms derived from the composite data within the mineralized shale domain were modelled for each individual grade-element. The modelled semi-variograms were used to define the search and classification limits used during grade estimation (Table 14.5).

The low coefficient of variation of the grade sample populations and the quality of the modelled semi-variograms demonstrate the observed grade continuity across the Deposit and suggest that drilling on a 200 x 200 m grid would be sufficient to raise the overall confidence level of the Deposit to Inferred.

TABLE 14.5 EXPERIMENTAL SEMI-VARIOGRAM PARAMETERS					
Element	**C0**	**C1**	**Range 1**	**C2**	**Range 2**
Cu	0.1	0.2	20	0.7	50
Mo	0.2	0.2	15	0.6	180
Ni	0.1	0.2	30	0.7	180
U	0.1	0.2	40	0.7	220
V	0.2	0.3	40	0.5	180
Zn	0.3	0.4	10	0.3	70
K	0.04	0.38	9	0.6	70
P	0.20	0.56	6	0.24	40
La	0.09	0.40	9	0.51	90
Y	0.27	0.53	10	0.20	0

14.7 BULK DENSITY

A total of 314 bulk density measurements taken from drill hole core were provided. Bulk density sample intervals were back-tagged to the appropriate lithological domain and used to assign a bulk density value for Mineral Resource estimation (Table 14.6). A bulk density value of 1.80 t/m^3 was assumed for the overburden.

TABLE 14.6 BULK DENSITY VALUES		
Domain	**Sample Count**	**Bulk Density (t/m^3)**
Overburden	n.a.	1.80
Mineralized Shale	271	2.57
Waste + Basement	43	2.70

14.8 MINERAL RESOURCE ESTIMATION AND CLASSIFICATION

A block model size of 20 m x 20 m x 10 m was implemented, with the model extended far enough beyond the limits of the mineralized shale domain to accommodate a pit shell (Table 14.7).

TABLE 14.7 BLOCK MODEL SETUP				
Item	**Minimum (m)**	**Maximum (m)**	**Count**	**Size (m)**
Easting	464,700	470,700	300	20
Northing	6,985,400	6,993,400	400	20
Elevation	50	450	40	10
Rotation	0°			

Separate block model attributes were developed for rock type, bulk density, classification and grade estimates. A volume percent model was used to accurately represent the volume and tonnage of the mineralized shale domain within a block.

Weighting of composite samples by linear Ordinary Kriging ("OK") was used for the estimation of block grades. Kriging parameters were based on the grade-element variography derived from composites within the mineralized shale domain. A discretization level of 4 x 4 x 2 m was used for block kriging. The mineralized shale domain was treated as a hard boundary, and data used during grade estimation were limited to composite samples located within the mineralized shale domain wireframe. Only grade blocks wholly or partially within the mineralized shale domain were estimated.

Between six and 15 samples from two or more drill holes within 660 m of the block centroid were used for grade estimation. Classification was based on the observed geological continuity and the range of the U semi-variogram. A block within 220 m of one drill hole was classified as Inferred. Blocks within 220 m of three or more drill holes were classified as Indicated. Subsequent to the initial classification, block classifications were consolidated using a best-fit polygon, in order to remove isolated artifacts.

14.9 MINERAL RESOURCE ESTIMATE

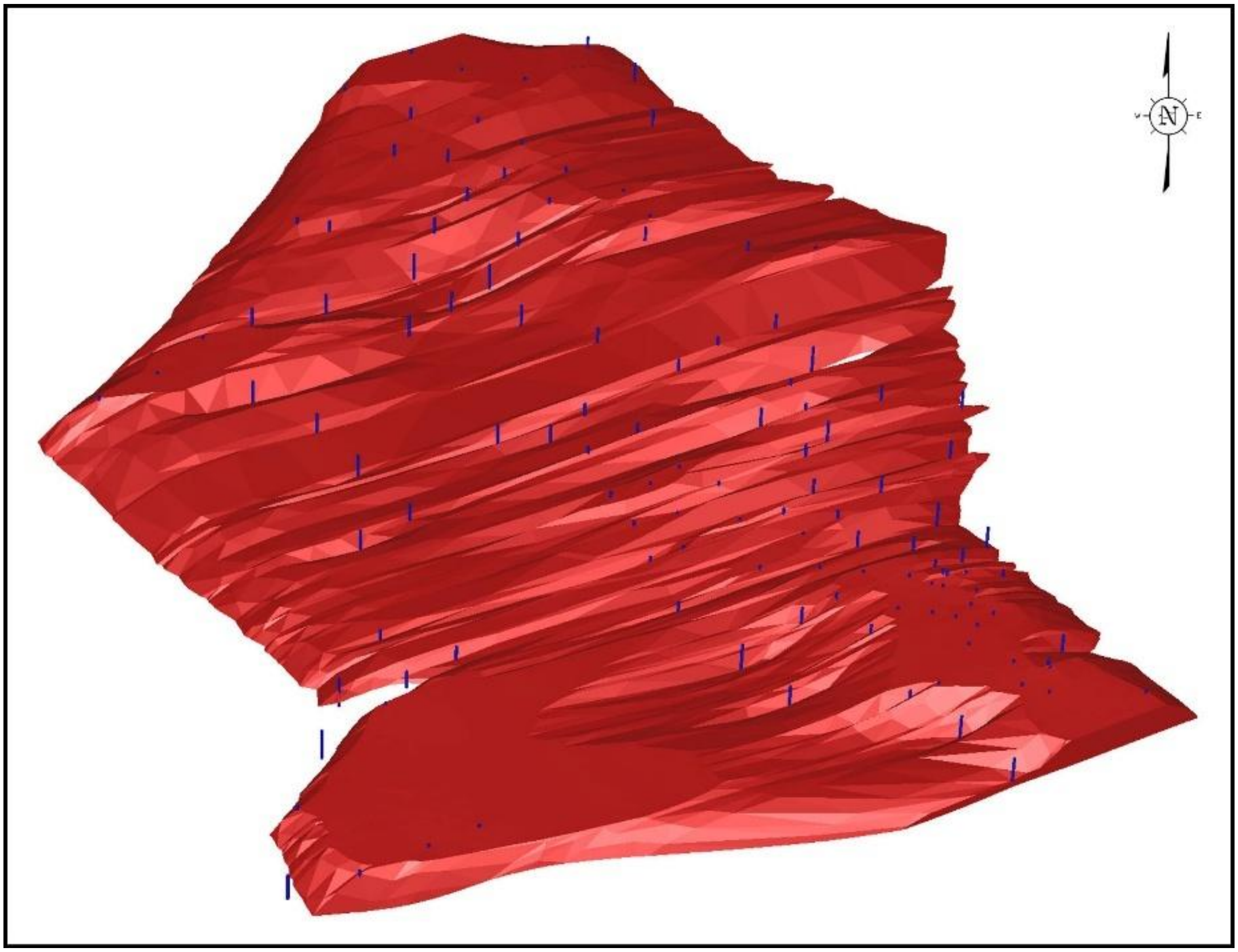
Each grade-element was estimated separately, and a NSR variable was subsequently calculated based on the block values for U, Cu, Ni, Zn, Mo and V using the preliminary parameters given in Table 14.8. The MRE was based on March 2025 Consensus Economics long-term forecast US$ metal prices.

TABLE 14.8
NSR PARAMETERS

Parameter	U_3O_8	Ni	Cu	Zn	Mo	V_2O_5
Price (US$ per lb)	72	8.50	4.25	1.30	17	5
Recovery (%)	80	70	50	75	70	80
Payable (%)	100	85	95	90	95	95
Refining	0	$0.50/lb	$0.10/lb	0	$1.00/lb	0

For the current updated Mineral Resource Estimate, potentially economic Mineral Resources were defined as those blocks falling within an optimized pit shell derived from the economic parameters listed in Table 14.8. Costs were based on a general knowledge of mining, processing and G&A costs from similar open-pit mining operations. The pit shell was further constrained to remain within the licence Property boundaries and has been set back from cultural landmarks. A 3-D perspective view of the mineralized shale domain is shown in Figure 14.3 and Appendix B, and the optimized pit-shell is shown in Figure 14.4 and Appendix F.

FIGURE 14.3 3-D PERSPECTIVE VIEW OF THE MINERALIZED SHALE DOMAIN



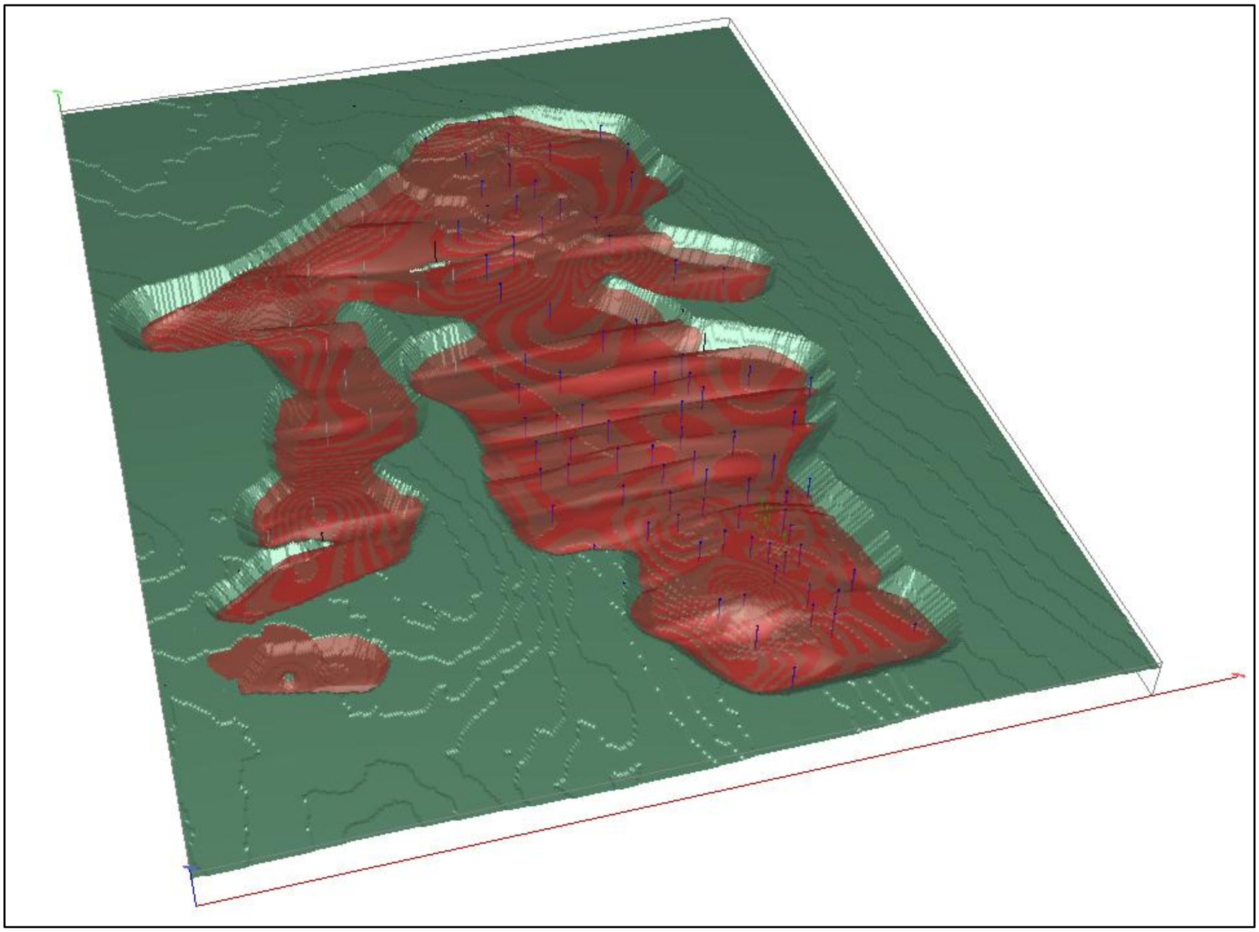
Source: *P&E (2025)*

An internal NSR cut-off grade of US$22/t was used for Mineral Resource reporting, based on the costs listed in Table 14.9.

TABLE 14.9
PIT SHELL OPTIMIZATION PARAMETERS

Parameter	Amount
Processing Cost	US$15/t
Handling Cost	US$5/t
G&A Cost	US$2/t
Mining Cost	US$3/t
Pit Slopes	45°
NSR Cut-off	US$22/t

FIGURE 14.4 OPTIMIZED MINERAL RESOURCE PIT SHELL



Source: *P&E (2025)*
Notes: *red = current MRE.*

Estimated grades were converted stoichiometrically to elemental oxides for reporting purposes. The updated Mineral Resource Estimate is listed in Table 14.10.

Table 14.10 **Pit-Constrained Mineral Resource Estimate for the Viken Deposit (1-7)**												
Classification	**Tonnes (M)**	**U_3O_8 (ppm)**	**V_2O_5 (ppm)**	**Mo (ppm)**	**Ni (ppm)**	**Cu (ppm)**	**Zn (ppm)**	**P_2O_5 (ppm)**	**Ce_2O_3 (ppm)**	**Y_2O_3 (ppm)**	**La_2O_3 (ppm)**	**K_2O (%)**
Indicated	456	175	2,836	257	330	113	411	2,461	88	492	7	3.84
		(Mlb)						**(Mt)**				
Indicated	Contained Metal	176	2,851	258	332	114	413	1.12	0.04	0.22	0.00	17.53
Classification	**Tonnes (M)**	**U_3O_8 (ppm)**	**V_2O_5 (ppm)**	**Mo (ppm)**	**Ni (ppm)**	**Cu (ppm)**	**Zn (ppm)**	**P_2O_5 (ppm)**	**Ce_2O_3 (ppm)**	**Y_2O_3 (ppm)**	**La_2O_3 (ppm)**	**K_2O (%)**
Inferred	4,333	161	2,543	240	321	118	417	2,541	88	528	7	3.70
		(Mlb)						**(Mt)**				
Inferred	Contained Metal	1,538	24,295	2,293	3,067	1,127	3,984	11.01	0.38	2.29	0.03	160.27

Notes:

1. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2. *The Inferred Mineral Resource in this MRE has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
3. *The Mineral Resource in this MRE was estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
4. *The MRE was based on March 2025 Consensus Economics forecast US$ metal prices of $72/lb U_3O_8, $5/lb V_2O_5, $17/lb Mo, $8.50/lb Ni, $4.25/lb Cu and $1.30/lb Zn with process recoveries of 80%, 80%, 70%, 70%, 50% and 75%, respectively.*
5. *Overburden, waste and mineralized material US$ mining costs per tonne mined were $2.00, $2.50 and $3.00, respectively.*
6. *Processing and G&A US$ costs per tonne processed were $20 and $2, respectively.*
7. *Constraining pit shell slopes were 45°.*

14.10 VALIDATION OF ESTIMATE

The block model was validated visually by the inspection of successive cross-section lines in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values. In addition, summary statistics for the block grade estimates were calculated and compared to the global composite grades (Table 14.11). No significant discrepancies were noted. Block model vertical cross sections and plans are shown in Appendix C for U and Appendix D for V. Classification block model vertical cross-sections and plans are shown in Appendix E.

TABLE 14.11
COMPARATIVE MEANS

Statistic	U (ppm)	Ni (ppm)	Cu (ppm)	Zn (ppm)	Mo (ppm)	V (ppm)
Assay Average	137	312	122	409	238	1,446
Composite Average	138	314	114	411	239	1,455
Capped Composite Average	137	309	112	403	237	1,450
Block Average	132	307	114	407	230	1,363

15.0 MINERAL RESERVE ESTIMATES

NI 43-101 Mineral Reserves currently do not exist for the Project. Any reference to historical non-compliant reserve estimates are summarized in Section 6 of this Report. This section is not applicable to this Report.

16.0 MINING METHODS

16.1 INTRODUCTION

The Viken Deposit will be mined using conventional open pit mining methods. A single open pit consisting of two north and south pit bottoms connected at the top benches will be developed. The open pit exit point is at approximately 345 masl and the lowest point is at 200 masl, with an average pit depth of 145 m.

A mine design and production schedule have been developed for the Project that has been used in the financial analysis. This production schedule utilizes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them to be classified as Mineral Reserves, and there is no certainty that the PEA will be realized.

The open pit will require the excavation of two different materials:

- Waste rock, primarily limestone (co-placed into a waste rock storage facility along with the dry stack tailings); and

- Process feed (treated through the process plant).

The development of the mine production schedule entailed several sequential steps:

- Perform pit optimizations to select the optimal ultimate pit shell;

- Design an operational pit (with ramps and benches) based on the selected optimal ultimate pit shell in the previous step;

- Design intermediate pit phases (pushbacks) to smooth the annual production schedule; and

- Develop a life-of-mine ("LOM") production schedule.

16.2 PIT OPTIMIZATION

Pit optimization was performed using Datamine NPV Scheduler™ software ("NPVS"). A series of optimized pit shells was generated at different revenue factors ("RF") to assess the sensitivity of the open pit size to metal prices, select the optimum base case pit shell for further detailed pit designs and guide the LOM planning process by generating possible intermediate mining phases. An RF applies the same percent change in metal prices to all payable metals and hence to the entire net smelter return ("NSR") value simultaneously.

The unit revenue for each individual block is represented by the NSR in $/t as described in Section 14.9. Pit optimization was completed based on the NSR values provided in the block model and the pit optimization parameters shown in Table 16.1.

A series of optimized pit shells were generated by changing the RF between 0.35 and 0.9 at 0.01 intervals, where a RF of 1.0 corresponds to the base case metal prices. The optimization results are shown graphically in Figure 16.1 (undiscounted and discounted net operating cash flows "NCF") and Figure 16.2 (potential process feed and waste tonnages). The NCF numbers in Figure 16.1 are indicative merit measures used to compare the different optimized pit shells and are based on revenues and operating costs that are calculated using the pit optimization parameters. These measures do not incorporate the detailed operating cost estimations reported in Section 21, capital costs (initial or sustaining), taxes, or closure costs, and therefore are not directly comparable to the actual Project economic merit measures (reported in Section 22). These results provide an estimate for the potentially economic portion of the Mineral Resource process feed for each revenue factor as well as potential strip ratio. Figure 16.2 shows that the NSR declines sharply and the waste tonnage increases rapidly by pit number. The goal was to maximize value and minimize footprint of the starter pit, which will be followed by staged expansions in future studies. The target process feed tonnage was set to 125 Mt to provide a reasonable starter pit LOM. The optimized pit shell corresponding to pit number 4 was selected as the optimal ultimate pit shell to form the basis for further detailed final pit design.

The tonnages shown in Figure 16.2 represent the potentially economic portion of the Mineral Resource contained in the optimized pit shell; however, the tonnages that will be reported in the production schedule will be derived from the operational final pit design.

TABLE 16.1
PIT OPTIMIZATION PARAMETERS

Parameter	Unit	Value
NSR	$/t	See Table 14.8
Process Recovery	%	See Table 13.17
Waste Rock Mining Cost	$/t mined	2.50
Process Feed Mining Cost	$/t mined	3.00
Processing Cost	$/t processed	20.00
G&A Cost	$/t processed	2.00
Pit Slopes for Optimization		
Overburden	degrees	30º
Mineralization	degrees	45º

FIGURE 16.1 PIT OPTIMIZATION NET OPERATING CASH FLOWS

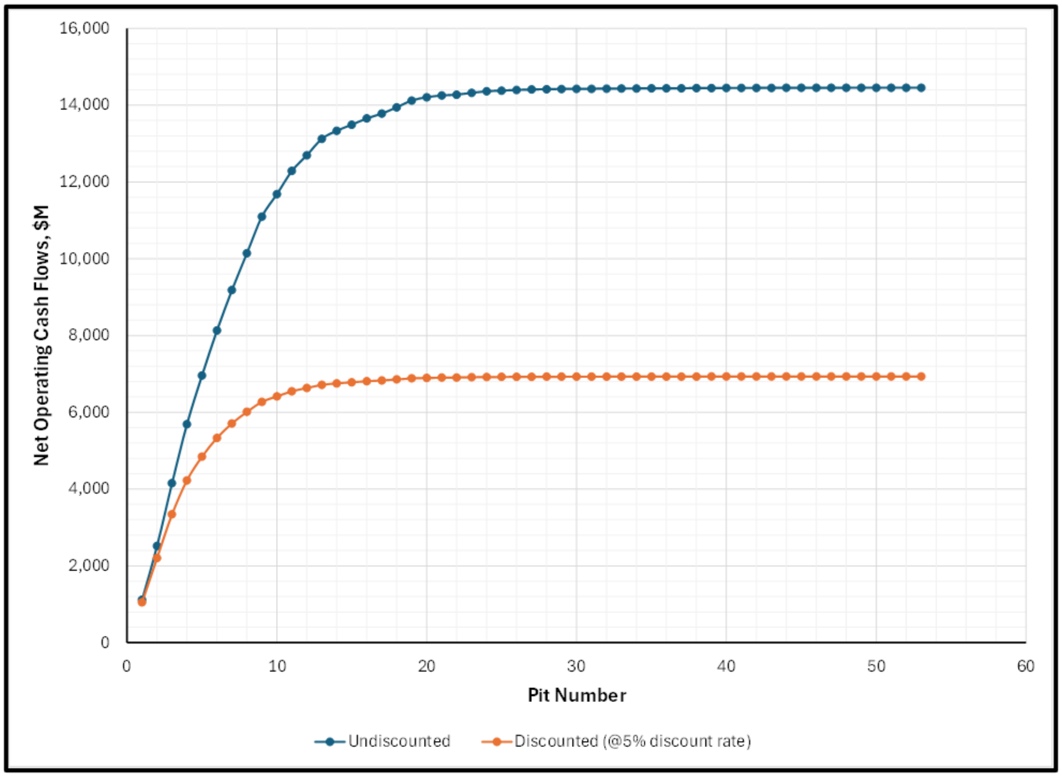
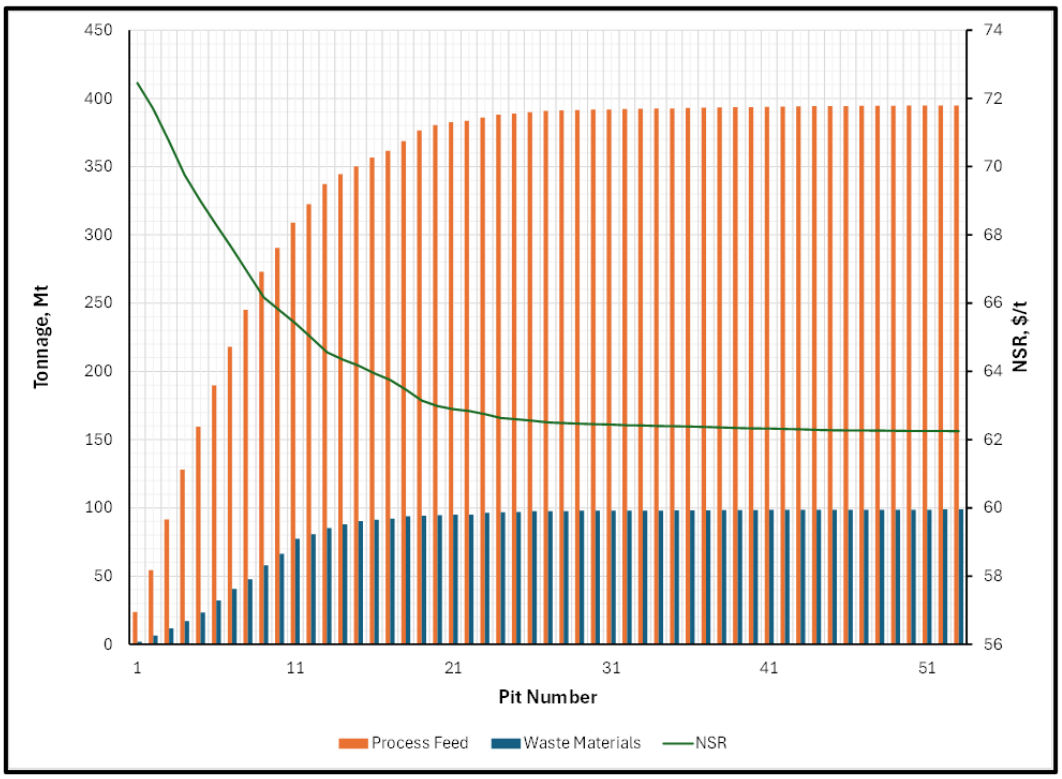


FIGURE 16.2 PIT OPTIMIZATION TONNAGES AND STRIP RATIO



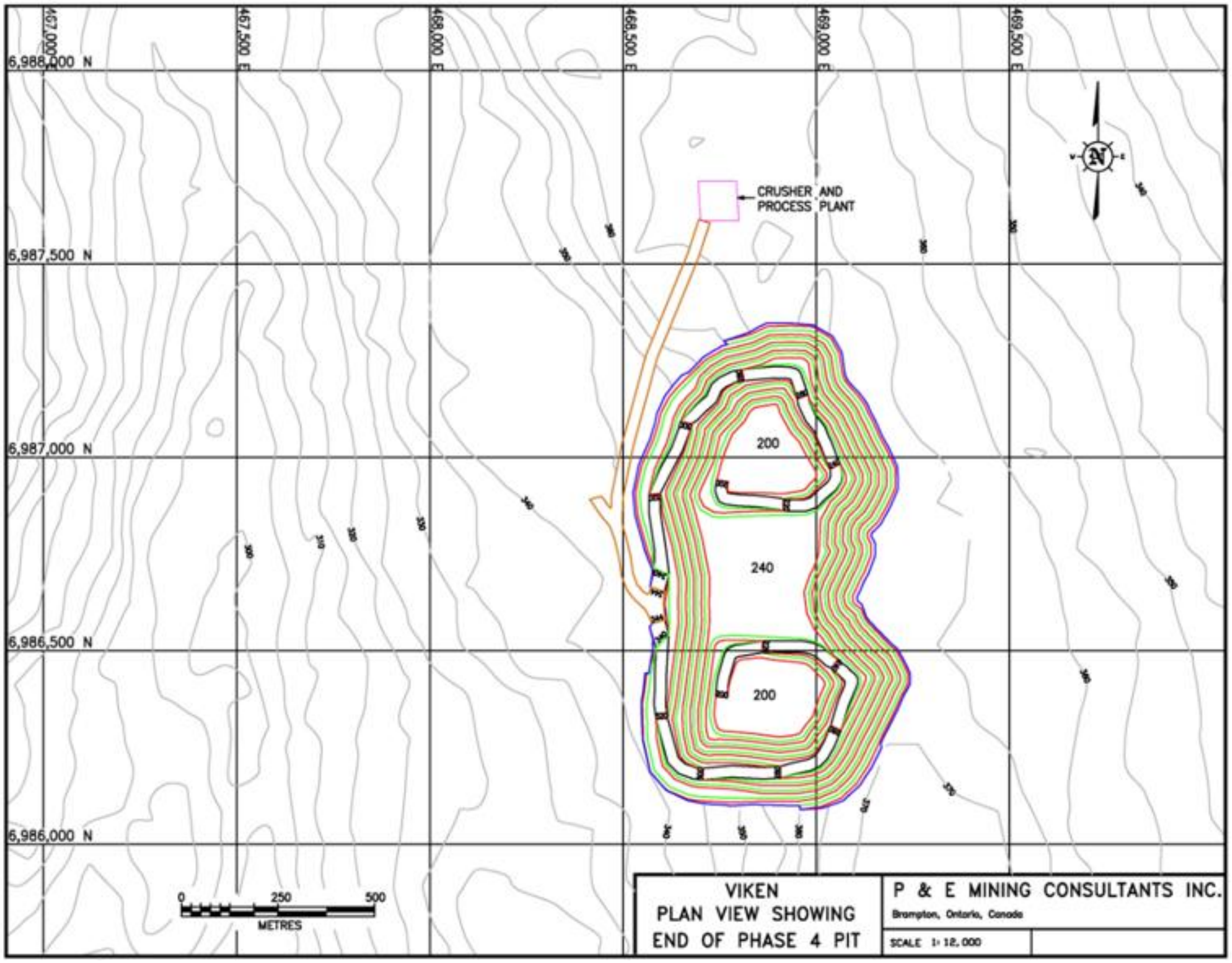
16.3 OPEN PIT DESIGN

An operational open pit design was created using the selected optimized pit shell. Benches and haul roads were added, according to the guidelines shown in Table 16.2. Figure 16.3 presents a plan view of the final pit design.

TABLE 16.2
PIT DESIGN PARAMETERS

Item	Unit	Value
Haul Road Width (double lane)	m	30
Haul Road Width (single lane)	m	20
Haul Road Grade (maximum)	%	10
Pit Wall Slope Assumptions (Overburden)		
Bench Height	m	20
Bench Face Angle	degrees	40
Catch-bench Width	m	10.8
Inter-ramp Angle	degrees	30
Pit Wall Slope Assumptions (Rock)		
Bench Height (triple bench)	m	20
Bench Face Angle	degrees	60
Catch-bench Width	m	8.5
Inter-ramp Angle	degrees	45

FIGURE 16.3 FINAL OPEN PIT DESIGN



16.4 GEOTECHNICAL AND HYDROGEOLOGICAL STUDIES

No geotechnical or hydrogeological studies have been completed for this PEA to assess open pit wall slopes or groundwater conditions.

16.5 MINING DILUTION AND MINING LOSSES

The mineralized material selected for the mine design and production schedule consists of Mineral Resource model blocks that are well inside the bounds of the constraining Mineral Resource wireframe. Since the potentially mineable blocks have no edge contact with the wireframe, dilution and losses are expected to be zero.

16.6 POTENTIALLY MINEABLE PORTION OF THE MINERAL RESOURCE

Table 16.3 summarizes the tonnages and grades of the potentially mineable portion of the Mineral Resource as well as the waste tonnages within the designed final pit. These tonnages are used as the basis for the production schedule.

TABLE 16.3
POTENTIALLY MINEABLE PORTION OF THE MINERAL RESOURCE

Item	Unit	Value
Total Material in Pit	Mt	150.85
Waste Rock	Mt	23.49
Process Plant Feed (diluted)	Mt	127.36
Strip Ratio	waste:feed	0.18
Average Grades & NSR		
U_3O_8	ppm	197
V_2O_5	ppm	3,714
Mo	ppm	293
Ni	ppm	395
Zn	ppm	479
K_2O	%	4.12
NSR	$/t	118

The total open pit process plant feed of 127.36 Mt consists of 77% Indicated Mineral Resource and 23% Inferred Mineral Resource (Table 16.4).

TABLE 16.4
PROCESS PLANT FEED CLASSIFICATION

Classification	Tonnes (k)	U_3O_8 (ppm)	V_2O_5 (ppm)	Mo (ppm)	Ni (ppm)	Zn (ppm)	K_2O (%)	NSR ($/t)
Indicated	97,916	198	3,667	294	396	476	4.13	117
Inferred	29,445	192	3,871	290	392	490	4.08	119

16.7 PIT PHASES

A series of intermediate pit phases were designed for LOM production scheduling purposes. The phases are used to enhance Project economics by distributing waste tonnages over time and allowing mining to start in an area near surface with limited waste rock to strip. In this respect, the pit was sub-divided into four phases. The plan views of intermediate phases 1 to 3 are shown in Figure 16.4 to Figure 16.6. Phase 4 shown in Figure 16.7 represents the materials left in the final pit design after all three intermediate mining phases are mined out. The tonnages and grades contained within each phase are shown in Table 16.5.

FIGURE 16.4 PLAN VIEW OF PHASE 1

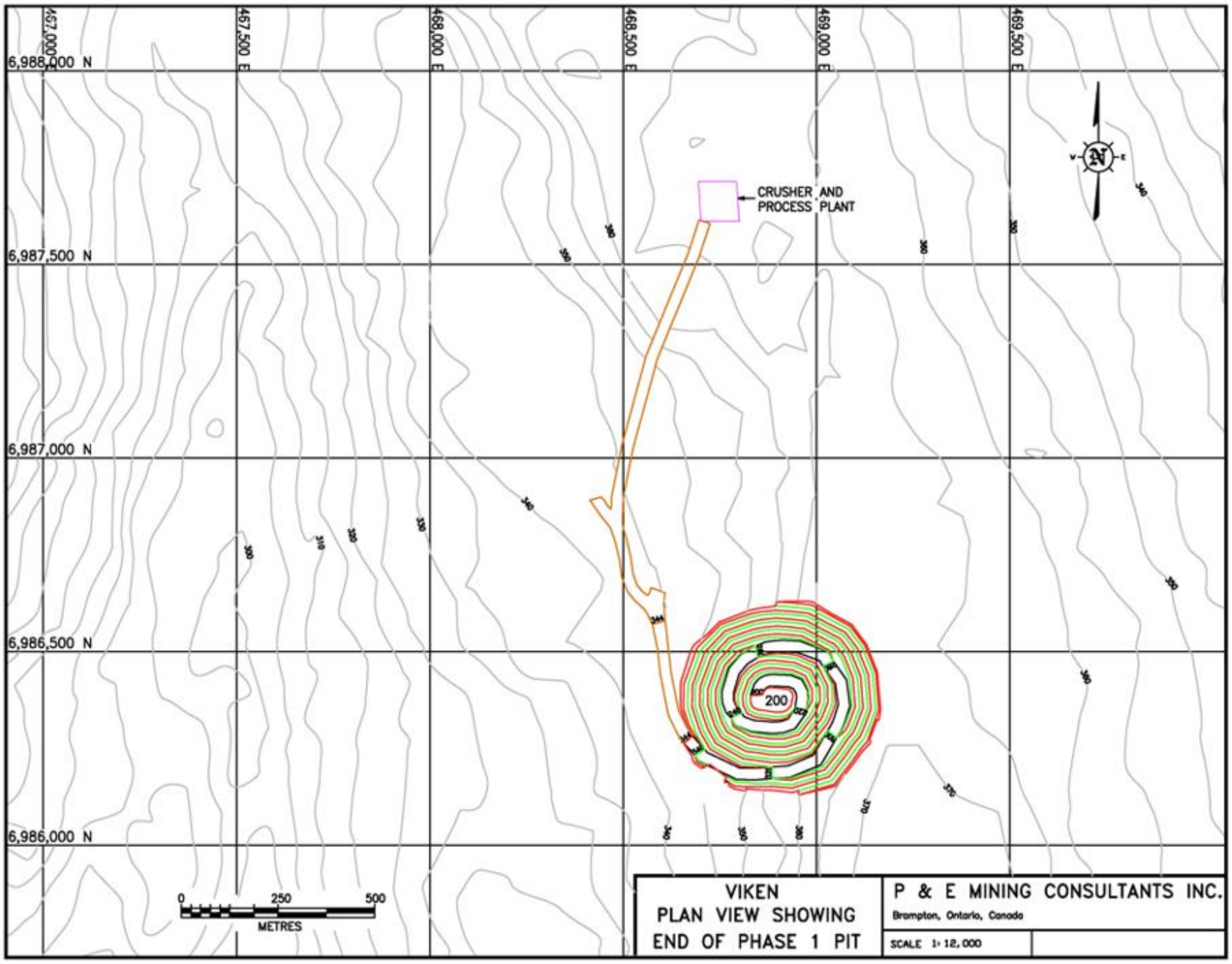


FIGURE 16.5 PLAN VIEW OF PHASE 2

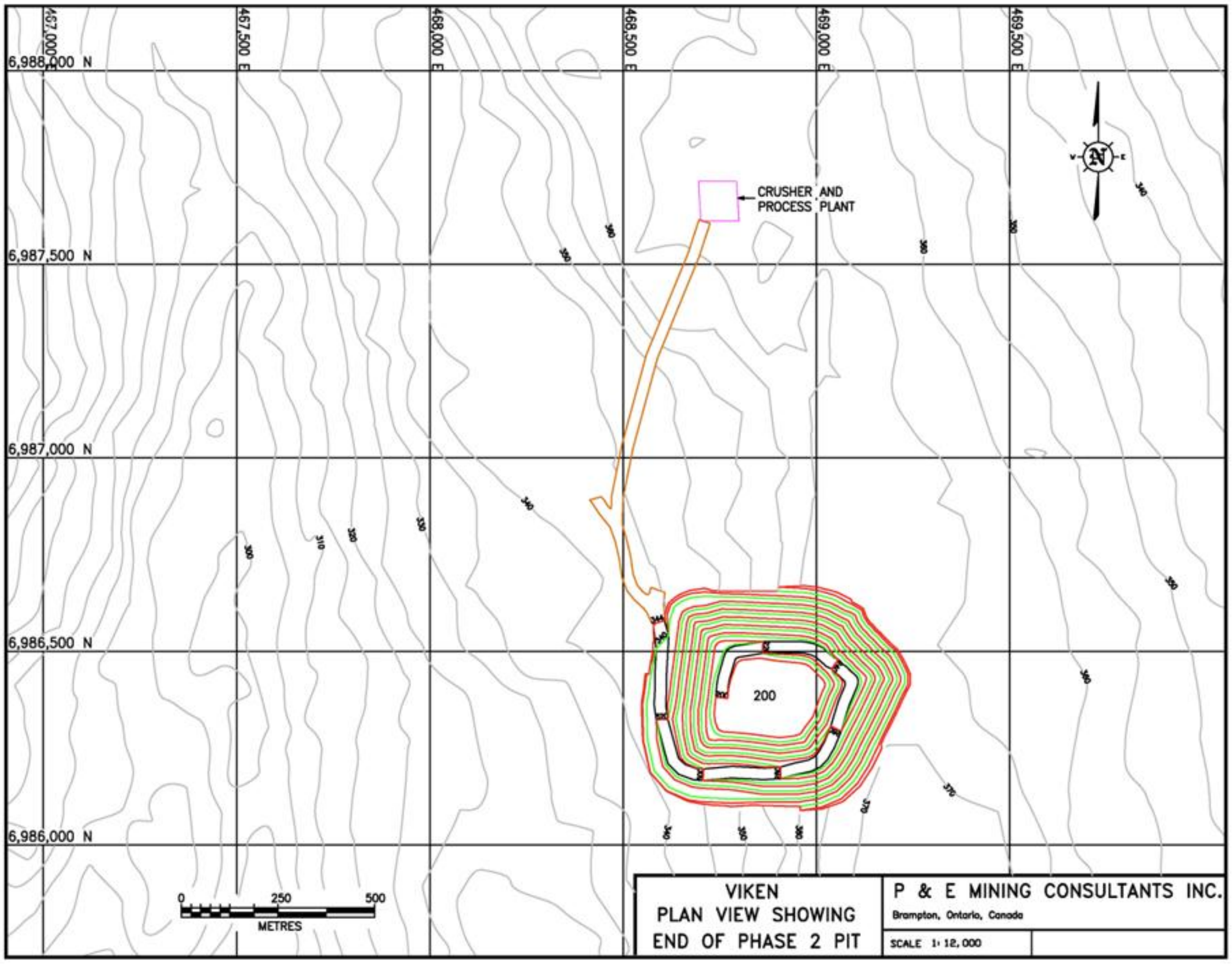


FIGURE 16.6 PLAN VIEW OF PHASE 3

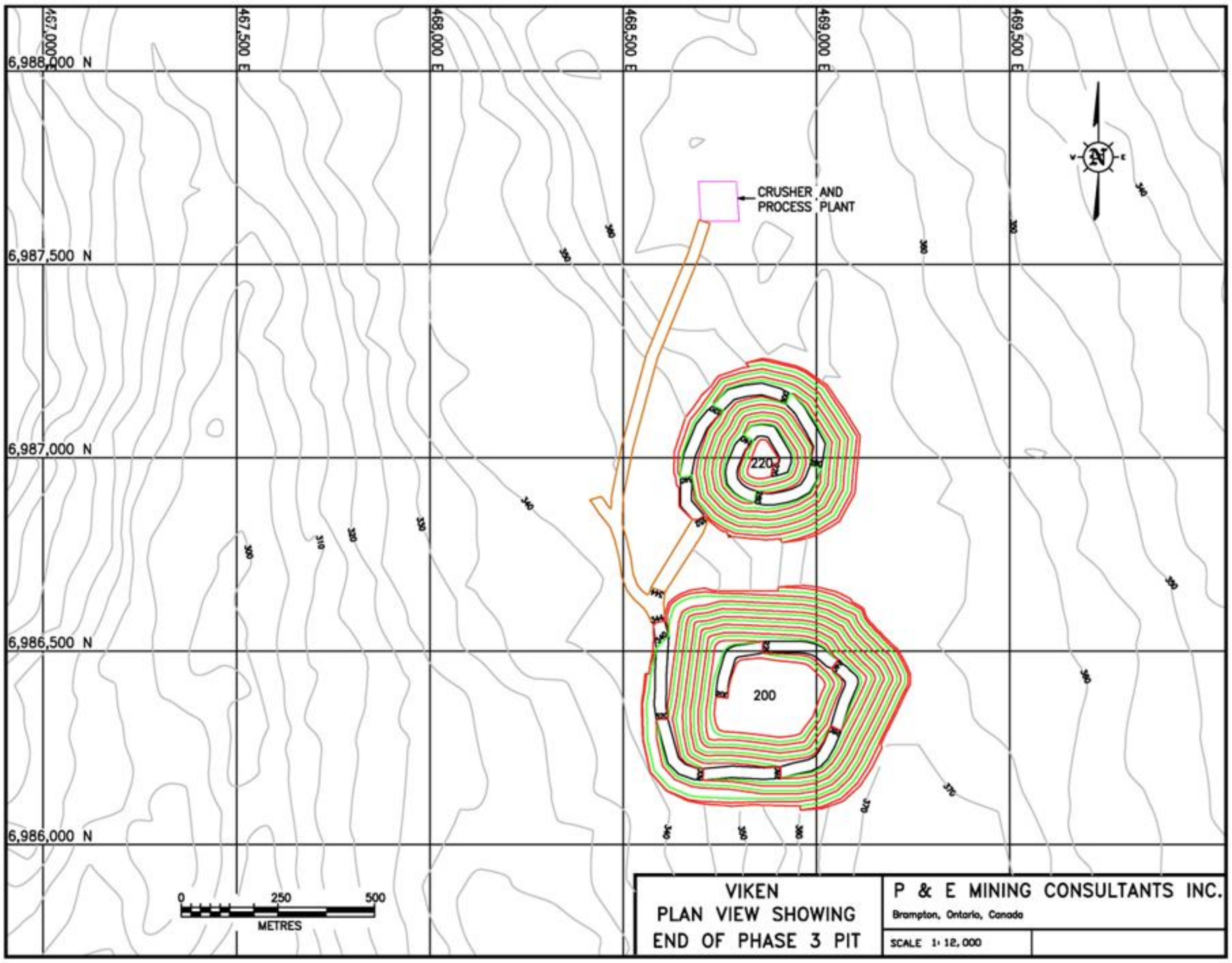


FIGURE 16.7 PLAN VIEW OF PHASE 4

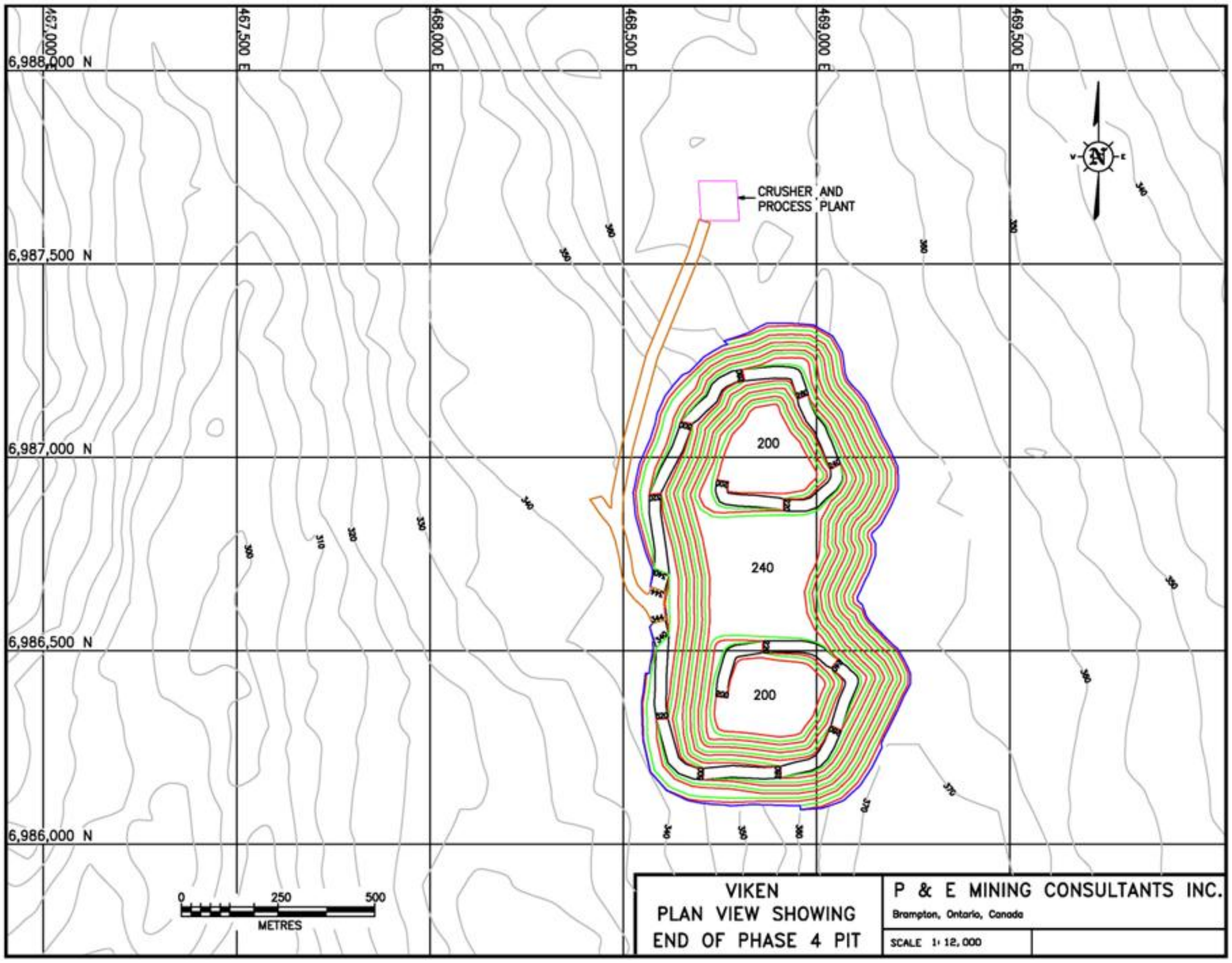


TABLE 16.5
PIT PHASE TONNAGES

Item	Unit	Ph. 1	Ph.2	Ph. 3	Ph. 4	Total
Total Material in Pit	Mt	34.63	30.77	25.49	59.96	**150.85**
Waste Rock	Mt	4.59	5.85	4.45	8.60	**23.49**
Process Plant Feed	Mt	30.04	24.92	21.05	51.36	**127.36**
Strip Ratio	waste:feed	0.15	0.23	0.21	0.17	**0.18**
Average Grades and NSR						
U_3O_8	ppm	194	197	196	198	**197**
V_2O_5	ppm	4,062	3,644	3,762	3,524	**3,714**
Mo	ppm	293	293	291	294	**293**
Ni	ppm	389	383	411	397	**395**
Zn	ppm	502	490	464	466	**479**
K_2O	%	4.16	4.02	4.20	4.12	**4.12**
NSR	$/t	123	116	119	115	**118**

Note: *Ph. = Phase.*

16.8 PRODUCTION SCHEDULE

The LOM production schedule was generated on an annual basis. As shown in Table 16.6 and presented in Figure 16.8, the mine production schedule consists of one pre-production year (pre-stripping) and 13 years of active mining and processing operations, for a total of 14 years. Only 4,500 tonnes of waste material are mined during pre-production. The pre-production period will be mainly for constructing mine roads and preparing the site for active mining.

The target processing rate is 10 Mtpa, or 28 ktpd. Process plant ramp up is 75% in the first production year and reaches full capacity starting in the second production year.

Over the LOM, the pit will produce 127.4 Mt of process plant feed material grading averages of 197 ppm U_3O_8, 3,714 ppm V_2O_5, 293 ppm Mo, 395 ppm Ni, 479 ppm Zn and 4.12% K_2O, for an overall NSR of $118/t process feed. Total waste material mined is 23.5 Mt with a LOM strip ratio of 0.18:1.

FIGURE 16.8 MINE PRODUCTION SCHEDULE

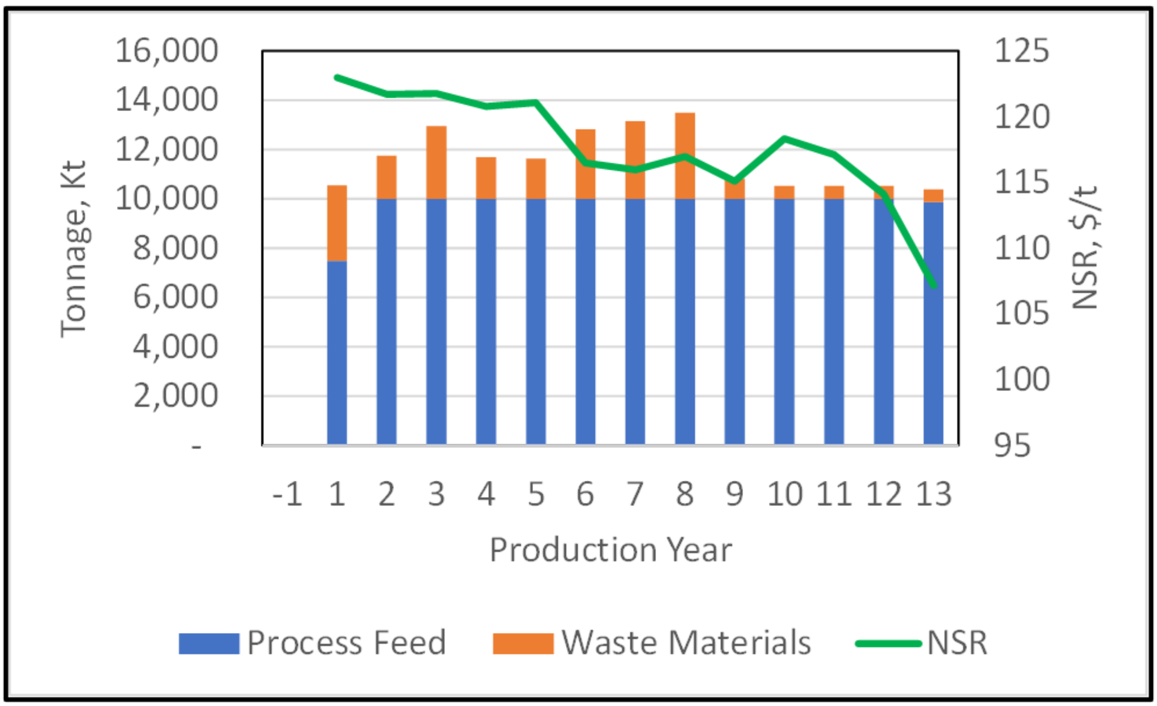


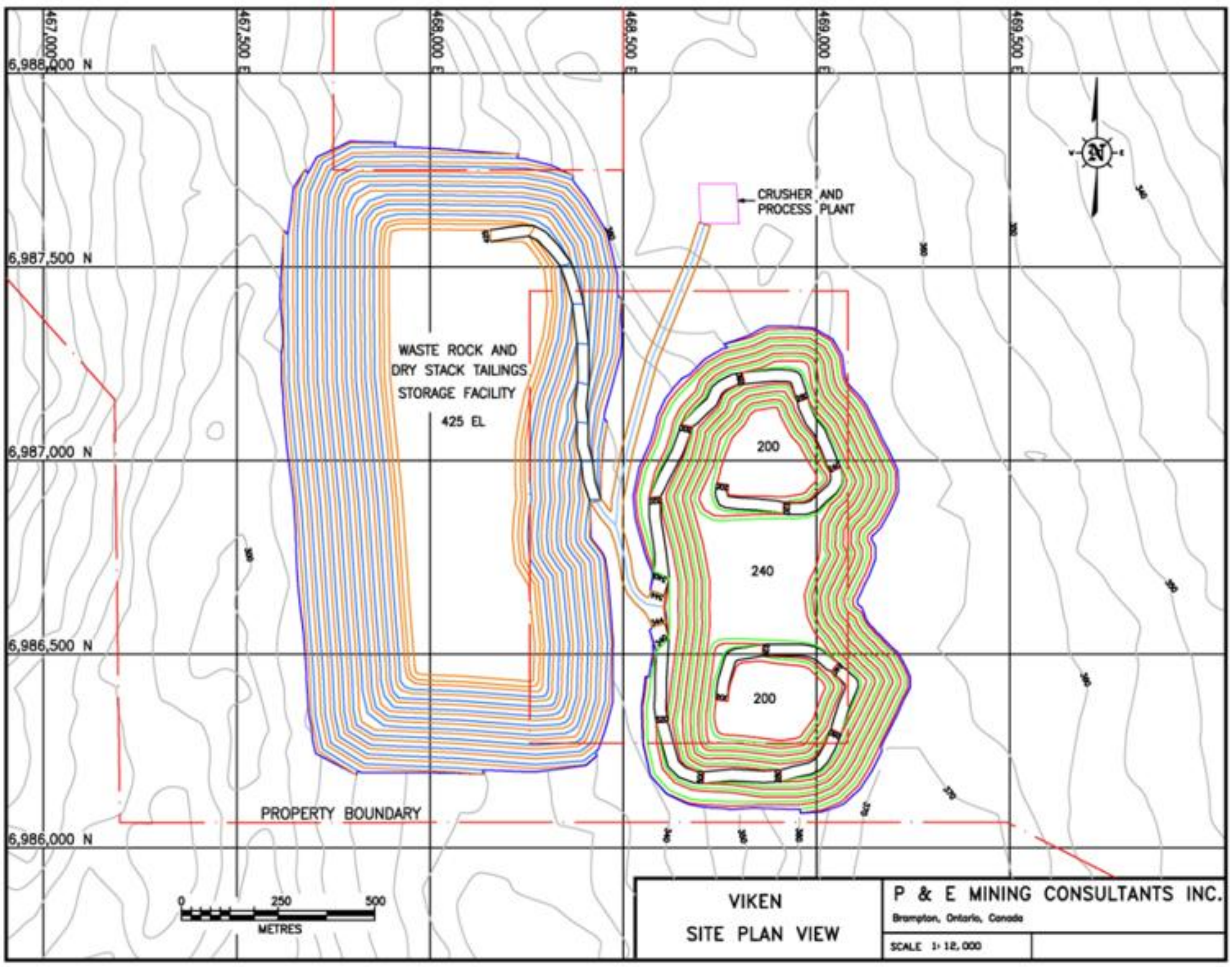
TABLE 16.6
MINE PRODUCTION SCHEDULE

Production Year	Mined				Processed							
	Total Mined (kt)	Feed (kt)	Waste Material (kt)	Strip Ratio waste:feed	Process Feed (kt)	U_3O_8 (ppm)	V_2O_5 (ppm)	Mo (ppm)	Ni (ppm)	Zn (ppm)	K_2O (%)	NSR ($/t)
-1	4	-	4	-	-	-	-	-	-	-	-	-
1	10,553	7,500	3,056	0.41	7,500	182	4,196	283	387	469	4.19	123
2	11,748	10,000	1,754	0.18	10,000	187	4,069	284	380	498	4.18	122
3	12,953	10,000	2,961	0.30	10,000	179	4,165	275	385	536	4.20	122
4	11,683	10,000	1,692	0.17	10,000	201	3,863	299	392	484	4.11	121
5	11,625	10,000	1,634	0.16	10,000	205	3,837	308	404	510	4.04	121
6	12,803	10,000	2,812	0.28	10,000	214	3,459	307	401	471	4.10	116
7	13,142	10,000	3,152	0.32	10,000	189	3,734	281	386	473	4.05	116
8	13,494	10,000	3,503	0.35	10,000	195	3,693	289	393	462	4.18	117
9	10,808	10,000	817	0.08	10,000	185	3,713	273	381	472	4.16	115
10	10,516	10,000	526	0.05	10,000	192	3,796	292	401	466	4.16	118
11	10,516	10,000	526	0.05	10,000	206	3,551	305	406	464	4.20	117
12	10,516	10,000	526	0.05	10,000	209	3,356	306	414	465	4.11	114
13	10,486	9,861	524	0.05	9,861	209	2,966	303	399	458	3.97	107
Total LOM	**150,848**	**127,361**	**23,488**	**0.18**	**127,361**	**197**	**3,714**	**293**	**395**	**479**	**4.12**	**118**

16.9 OPEN PIT MINING PRACTICES

The Viken Project consists of a relatively shallow deposit and a mine plan that will consist of a single open pit mined with Surface Excavation Machines ("SEMs") capable of cutting 0.3 m deep into the Alum Shale, augmented by conventional loading and hauling equipment. Figure 16.9 shows the mine site plan. It is assumed that the Viken Mine will be owner operated. The SEMs will be used in both mineralized and waste materials. No drilling or blasting operations will be required.

FIGURE 16.9 MINE SITE PLAN



The Owner's mining team would undertake all SEM cutting operations, loading, hauling, and mine site maintenance activities. The Owner will also be responsible for technical services, such as mine planning, grade control, geotechnical, and surveying services.

It is anticipated that the mining operations would be conducted 24 hours per day and seven days per week throughout the entire year.

16.9.1 Mine Equipment Fleet and Personnel

The mine equipment listed in Table 16.7 includes the requirements for both mining operations and the requirements for loading, hauling, spreading and compacting the dry stack tailings.

Waste and mineralized materials cutting operations will be performed using the T1255 Single Side Direct Drive SEM with a 3.7 m width of cut and a depth of cut of 25 to 30 cm.

Loading operations will be performed using front-end loaders with an 11.5 m^3 bucket. The loaders will load 90-tonne off-highway haul trucks in a 5-pass loading match.

The primary mining operations will be supported by a fleet of support equipment consisting of PR776 size class bulldozers with ripper attachments, Caterpillar 14 M class graders as well as a Caterpillar 814 class wheel dozer, water truck, maintenance vehicles, and service vehicles.

Table 16.7 summarizes the mining equipment fleet requirements.

TABLE 16.7
MINE EQUIPMENT FLEET

Item	Year													
	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
T1255-DD SS Surface Miner	2	3	3	3	3	3	3	3	3	2	2	2	2	2
Wheel Loader 11.5 m^3 (Komatsu WA800-8)	1	3	3	3	3	3	3	3	3	3	3	3	3	3
Haul Truck 90 t (CAT 777)	1	6	8	9	9	10	10	10	10	9	10	10	10	10
Personnel Van	-	2	2	2	2	2	2	2	2	2	2	2	2	2
Crane, Grove 40 t	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel Dozer CAT 814	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Dozer PR776	-	3	3	3	3	3	3	3	3	3	3	3	3	3
Mechanic & Welding Truck	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator, 2.3 m^3 (CAT 336)	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel & Lube Truck	-	2	2	2	2	2	2	2	2	2	2	2	2	2
Grader CAT 14	-	2	2	2	2	2	2	2	2	2	2	2	2	2
Flat Deck with Hiab	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Light Plant	-	6	6	6	6	6	6	6	6	6	6	6	6	6
Tire Manipulator	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Truck and Trailer, 200 t	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Pickup Truck	-	8	8	8	8	8	8	8	8	8	8	8	8	8
Pit Water Pumps	-	2	2	2	2	3	3	3	3	4	4	4	4	5
Forklift	-	1	1	1	1	1	1	1	1	1	1	1	1	1

Table 16.7 Mine Equipment Fleet														
Item	**Year**													
	-1	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**
Wheel Loader 4 m^3	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Tractor Massey Ferguson 375/4WD	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck CAT 745	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Dozer CAT D8*	-	-	2	2	2	2	2	2	2	2	2	2	2	2
Vibratory Compactor*	-	2	2	2	2	2	2	2	2	2	2	2	2	2

Note: ** For dry stack tailings handling and storage.*

The mining personnel will peak in year 7 and include operators, maintenance, supervision, and technical staff. The workforce breakdown by role is presented in Table 16.8.

Table 16.8 Peak Mine Personnel	
Category	**Number**
Cutter Operator	10
Truck Driver	39
Wheel Loader Operator	6
Small Excavator Operator	2
Heavy Duty Mechanic	26
Pit Services (dewatering)	2
Grader Operator	5
Dozer Operator	14
Water/Sand Truck Operator	3
Utility Operators	9
Mine Superintendent	1
Mine Foremen	4
Mine Clerk	1
Equipment Trainer	4
Maintenance Foreman	2
Maintenance Clerk	1
Planner	1
Welder	2
Gas Mechanic	2
Fuel and Lube Person	2
Tireman	2
Partsman	1
Labourer	2

TABLE 16.8
PEAK MINE PERSONNEL

Category	Number
General Maintenance	2
Chief Mine Engineer	1
Senior Pit Engineer	2
Mine Engineer	1
Project Engineer	1
Geologist	2
Surveyor	1
Survey Technician	2
Mine Technician	1
Grade Control Technician	2
Geotechnical Engineer	1
Total	**157**

16.9.2 Pit Dewatering

No quantitative information was available to adequately predict the expected water inflow into the open pit. Skid or trailer mounted centrifugal pumps will be staged up the side of the pit to remove any water from the pit sump locations during pit development.

16.9.3 Waste Material Storage

The open pit will require the development of a waste rock storage facility. Over the LOM, the pit will generate 23.5 Mt of waste rock. Most of the waste rock generated will be co-stored with the dry stack tailings in an external waste storage facility located west of the open pit. In production year 8, the Southern area of the pit will be completely mined out and will provide 3 Mm^3 of in-pit waste storage space. Therefore, some of the waste rock materials mined out from the Northern area of the pit in productions years 8 to 13 will be stored in-pit in the Southern area. Over the LOM, a total of 17 Mt will be stored in the external waste storage facility, and 6.5 Mt will be stored in-pit in the South pit bottom.

At this PEA stage, the waste storage facilities are not designed in detail. Potential locations are identified, and field reconnaissance and geotechnical investigations will be undertaken at the next stage of study to confirm suitability and design specifications.

16.9.4 Support Facilities

The Viken Mine will require mine offices, change house facilities, maintenance facilities, warehousing and storage areas. A mine office will be provided for mine management, engineering, geology and survey services. These are part of the Project infrastructure described in Section 18.

A maintenance shop which will provide open pit support services. The mine maintenance facility will consist of a truck shop which will include a wash facility, welding equipment, maintenance bays and a dedicated preventive maintenance bay. The facility will have adjoining indoor parts storage and tool crib. A fuel and lube station will be conveniently located near the maintenance facility and main haul road for equipment access. A mobile truck mounted fuel and lube system will be available to service less mobile equipment in the field.

17.0 RECOVERY METHODS

17.1 PUG ROASTING

Four different recovery methods were studied and economic analysis determined that the pug roast leach method with an on-site acid production plant resulted in the highest net present value (“NPV”) and internal rate of return (“IRR”) for the Viken Deposit (refer to Section 24.1 for analysis). The process feed is crushed, grinded and floated before being mixed with concentrated sulphuric acid in a pug mill and thermally roasted to convert vanadium, uranium, potassium and associated metals into water-soluble sulphate species. This configuration targets high leach recoveries and allows a more liberal flotation regime, which can enhance recovery of by-products such as mixed metals precipitate (“MMP”) and sulphate of potash (“SOP”). Refer to Section 18.7 for details of the process plant infrastructure. A description of each process plant circuit is below.

17.1.1 Pug Roast Circuit

The thickened/filtered flotation concentrate is mixed with concentrated sulphuric acid (98%) in a pug mill (intensive mixer) to produce a uniformly acidulated feed (similar to a clay mix dough). The material is then fed to a rotary kiln or multiple-hearth furnace for controlled sulphation roasting. Sulphation roasting occurs at a temperature range of 220 to 250°C converting vanadium, potash, other metals and uranium into water-soluble sulphate compounds, allowing subsequent extraction via simple water leaching.

Key Sulphation Reactions

Uranium Sulphation Reaction:

$$UO_2 + H_2SO_4 + \frac{1}{2}O_2 \rightarrow UO_2SO_4 + H_2O$$

Vanadium Sulphation Reaction:

$$V_2O_3 + 3H_2SO_4 + \frac{1}{2}O_2 \rightarrow V_2(SO_4)_3 + 3H_2O$$

Iron sulphation (side reaction):

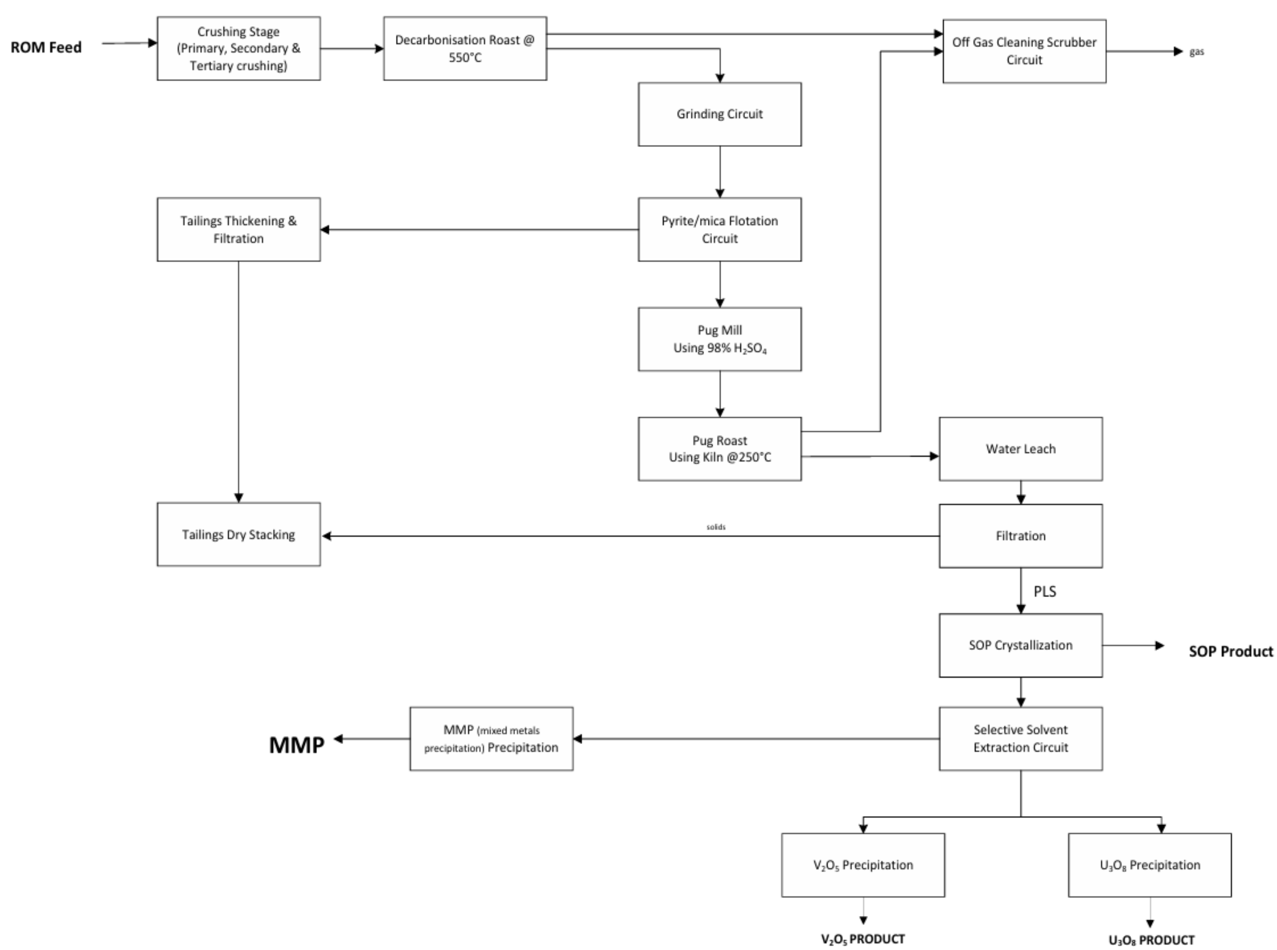
$$Fe_2O_3 + 3H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 3H_2O$$

The roasting temperature is carefully controlled to avoid sulphate decomposition. Roasting produces: SO_2, SO_3, acid mist and water vapour as off gas which is treated in a wet scrubber. The roasting temperature is controlled to prevent sulphate decomposition, with the resulting off-gas (SO_2, SO_3, acid mist and water vapour) treated in a wet scrubber, while the cooled calcine is transferred to agitated leach tanks where water at 60 to 90°C dissolves the water-soluble sulphate salts formed during roasting.

After leaching the slurry is sent to a counter-current-decantation ("CCD") thickening circuit. The leached slurry then proceeds to solid–liquid separation where it is washed and pressure filtered with the residue being dry stacked and the filtrate, pregnant leach solution ("PLS"), is passed to a potassium salt crystalliser for the recovery of sulphate of potash, K_2SO_4 ("SOP").

Ion exchange and solvent extraction are carried out to concentrate and separate the uranium and vanadium ions. The separate solutions containing vanadium and uranium are precipitated, forming V_2O_5 and U_3O_8, respectively. The remaining spent PLS is treated to precipitate any remaining metals to form a bulk concentrate. A description of the pug-roast flowsheet is provided in Figure 17.1.

FIGURE 17.1 BLOCK FLOW DIAGRAM PUG ROAST



Source: *METS (2026)*

17.1.2 Crushing Circuit

Run of mine ("ROM") will be delivered to the process plant via trucks. The ROM particle size determines the number of crushing stages required (primary, secondary, tertiary). Increasing the number of crushing stages increases capital expenditure ("CAPEX") due to additional equipment, conveyors, and infrastructure. If feed carbonate content exceeds ~10%, the crushed feed will undergo decarbonisation roasting (calcination) in a kiln to reduce acid consumption during downstream hydrometallurgical processing. The process will be performed in a kiln or fluidised bed roaster (fine feed) at a temperature of ~550°C (OPEX driver). The breakdown of the carbonate minerals generates CO_2 emissions which will require off-gas treatment, a scrubbing system.

17.1.3 Grinding and Flotation Circuit

The feed will then undergo grinding for further size reduction to a fine size of P_{80} 75 microns (µm). Grinding is extremely energy-intensive which affects operational costs but is crucial for mineral liberation before flotation and leaching. The grinding mill discharge will then pass through a series of classification cyclones. Cyclone overflow will report to flotation whilst the underflow returns to the mill. Selective froth flotation will remove some of the acid-consuming gangue, iron minerals and residual carbonates while concentrating the V-bearing mica (muscovite) and uranium minerals. Most calcite and gangue report to the tailings stream. Flotation tailings will be pumped to a thickener and pressure filtered for dry stacking. Pressure filtration is used due to the high degree of dryness required for dry stacking.

17.1.4 SOP Crystallization (Potassium Recovery)

The pregnant leach solution ("PLS") is first conditioned with calcite to neutralise acidity and precipitate metals and sulphate as gypsum, with the underflow filtered to tailings and the overflow directed to an iron-removal stage. The clarified solution then has its pH adjusted with limestone before evaporative crystallisation, where water removal and cooling increase SOP supersaturation and promote K_2SO_4 crystal growth. The SOP product is recovered by centrifuging or filtering, washed to remove residual liquor and impurities, and dried prior to bulk storage.

17.1.5 Ion Exchange (IX)

Following SOP crystallisation, the PLS still contains low concentrations of dissolved uranium and vanadium, along with residual acid and impurities. To recover these metals efficiently from the dilute solution, an ion exchange ("IX") circuit is employed ahead of final purification and precipitation, with the PLS passed through resin columns that selectively load uranium and, to a lesser extent, vanadium.

Key Ion Exchange Reactions

Under acidic sulphate conditions, uranium is present mainly as uranyl sulphate complexes, for example:

$$UO_2^{2+} + 3SO_4^{2-} \rightarrow [UO_2(SO_4)_3]^{4-}$$

These complexes are loaded onto the resin via anion exchange:

$$R^+Cl^- + [UO_2(SO_4)_3]^{4-} \rightarrow R^+[UO_2(SO_4)_3]^{4-} + Cl^-$$

Vanadium may occur as vanadate species under oxidising conditions, such as:

$$VO_2^+ + 2H_2O \rightarrow H_2VO_4^- + 3H^+$$

These anions can also be partially loaded onto the resin, depending on solution chemistry and resin selectivity.

Once the resin is loaded, uranium and vanadium are stripped using an appropriate eluant to produce a concentrated metal solution, and the regenerated resin is returned to the adsorption stage. The eluate then advances to the solvent extraction circuit for further purification and separation of uranium and vanadium.

17.1.6 Solvent Extraction (SX)

Vanadium and uranium are recovered from the PLS by selective solvent extraction in mixer–settler circuits, operated in two main stages: extraction and stripping. The aqueous phase contacts an organic phase containing a tertiary amine extractant (dissolved in kerosene with a modifier), which selectively loads uranyl sulphate complexes into the organic.

Key uranium SX reactions

Extraction (amine system):

$$R_3NH^+ + UO_2(SO_4)_3^{4-} \rightarrow (R_3NH)_4UO_2(SO_4)_3$$

Carbonate stripping (example):

$$UO_2^{2+} + 3CO_3^{2-} \rightarrow UO_2(CO_3)_3^{4-}$$

Loaded organic is stripped using ammonium sulphate or sodium carbonate solutions to produce a concentrated uranium strip liquor, and the regenerated organic returns to extraction.

Vanadium SX

Vanadium is extracted as V(V) species using an acidic extractant, represented generically as:

$$VO_2^+ + \text{Extractant} \rightarrow \text{Loaded organic}$$

Stripping is carried out with an alkaline solution such as sodium carbonate, for example:

$$VO_2^+ + CO_3^{2-} \rightarrow VO_3^- \text{(in alkaline strip)}$$

A scrubbing stage can be included to remove co-extracted impurities. The SX raffinate, depleted in U and V, is then directed to the mixed metals precipitation (“MMP”) circuit.

17.1.7 Uranium and Vanadium Precipitation

Following solvent extraction and stripping, uranium and vanadium are recovered from their respective strip liquors by controlled precipitation and calcination.

For uranium, ammonium diuranate (ADU, $(NH_4)_2U_2O_7$) is precipitated by adding ammonia to raise the pH to approximately 6.5–8.0 at 30–60°C under agitation, for example:

$$2UO_2^{2+} + 6NH_3 + 3H_2O \rightarrow (NH_4)_2U_2O_7$$

The ADU slurry is filtered and dried, then calcined in an oxidising atmosphere at approximately 500 to 800°C to produce uranium oxide concentrate (U_3O_8), with release of ammonia and water:

$$(NH_4)_2U_2O_7 \rightarrow U_3O_8 + NH_3 + H_2O$$

The cooled U_3O_8 product is then packaged as final uranium concentrate.

For vanadium, ammonium metavanadate (AMV, NH_4VO_3) is precipitated from the alkaline strip liquor by adding ammonium salts or ammonia to adjust the pH to near neutral at approximately 50 to 80°C:

$$VO_3^- + NH_4^+ \rightarrow NH_4VO_3$$

The AMV is filtered, washed and calcined at roughly 500 to 700°C in an oxidizing atmosphere to form vanadium pentoxide flake, with evolution of ammonia and water:

$$2NH_4VO_3 \rightarrow V_2O_5 + 2NH_3 + H_2O$$

The molten V_2O_5 is cooled, then crushed or flaked to produce the final V_2O_5 product. Approximately 35% of the total V_2O_5 production will be directed to vanadium electrolyte production, with the remaining V_2O_5 allocated to FeV production. The Viken process plant will be able to adjust production of vanadium electrolyte and ferrovanadium depending on market demand and pricing.

17.1.8 Mixed Metals Precipitation (MMP)

The barren SX raffinate typically contains residual base metals (Cu, Zn^{2+}, Ni^{2+}, MoO_4^{2-}), sulphuric acid and high sulphate levels. The mixed metals precipitation ("MMP") circuit is designed to recover these metals in saleable or intermediate form while producing a neutralized effluent suitable for recycle or discharge.

Molybdenum is removed first to avoid contaminating downstream zinc and nickel products. Lime (CaO) is added in a conditioning tank to raise the pH to approximately 2 to 4 at 30 to 60°C, precipitating calcium molybdate:

$$MoO_4^{2-} + Ca^{2+} \rightarrow CaMoO_4 \downarrow$$

After Mo removal, zinc and nickel remain in solution as Zn^{2+} and Ni^{2+}. The pH is then increased to roughly 8 to 10 with lime at 30 to 50°C, promoting hydroxide precipitation, for example:

$$Zn^{2+} + 2OH^{-} \rightarrow Zn(OH)_2 \downarrow$$

$$Ni^{2+} + 2OH^{-} \rightarrow Ni(OH)_2 \downarrow$$

The resulting Mo, Zn and Ni precipitates are recovered by filtration, with the filter cakes dried for storage or further treatment. Detailed testwork will be required to confirm selectivity, optimal pH ranges and final product specifications.

17.1.9 Tailings Handling

Tailings are dewatered in a high-rate thickener and then filtered in a pressure filter to a target moisture of approximately 7 to 10%, suitable for dry stacking. The filter cake is trucked to the dry stack, placed in layers and compacted, while recovered water is returned to the grinding and flotation circuits to minimise freshwater demand. As an alternative, a paste tailings system may be adopted, in which tailings are thickened to roughly 65 to 75% solids and pumped as paste to a contained storage facility.

18.0 PROJECT INFRASTRUCTURE

The Viken Property is in central Sweden, approximately 520 km northwest of Stockholm and 600 km south of the Arctic Circle. The Project is situated approximately 25 km southwest of the Town of Östersund in the Municipality of Åre and Berg, Jämtland County. The region is well serviced by existing infrastructure, including road, rail, and air transport networks, which will support both construction and operational activities for the Project. Daily air services operate in the region, and both rail and truck freight services are available for the transport of reagents, consumables, and product materials. Electrical power supply and modern telecommunications infrastructure are already available in the area, which will reduce the extent of new project specific infrastructure.

The Project is located within a well-established mining jurisdiction with several major mining centres in northern and central Sweden, including Kiruna, Malmberget, Aitik, and the Skellefteå mining district, as well as the Bergslagen mining region in southern and central Sweden. These established mining regions provide access to experienced mining contractors, equipment suppliers, maintenance services, laboratories, and other mining related support services, which will be beneficial during both construction and operations. The licence area has adequate space for the development of process plant facilities, tailings, waste rock storage areas, stockpile areas, and associated site infrastructure.

Although the region has good existing infrastructure, the Project will still require the development of site-specific infrastructure to support mining and processing operations. Infrastructure additions are expected to include:

- Site clearing and earthworks for process plant, waste, and stockpile areas.
- Construction of a new mine access road and internal site road network.
- Upgrading of existing local roads where required to accommodate Project traffic.
- Electrical power supply infrastructure, including site substation, transformers, and distribution.
- Water supply and water management infrastructure, including process and potable water systems.
- Communications systems, including data, voice, and site security.
- Workshops, warehouses, and maintenance facilities.
- Administration and general services buildings.
- Reagent and fuel storage, handling, and distribution facilities.

The workforce will be able to commute to the Project from nearby communities.

18.1 TRANSPORTATION

Table 18.1 shows a summary of the distances of major centres from the Project site.

TABLE 18.1
DISTANCES TO MAJOR CENTRES

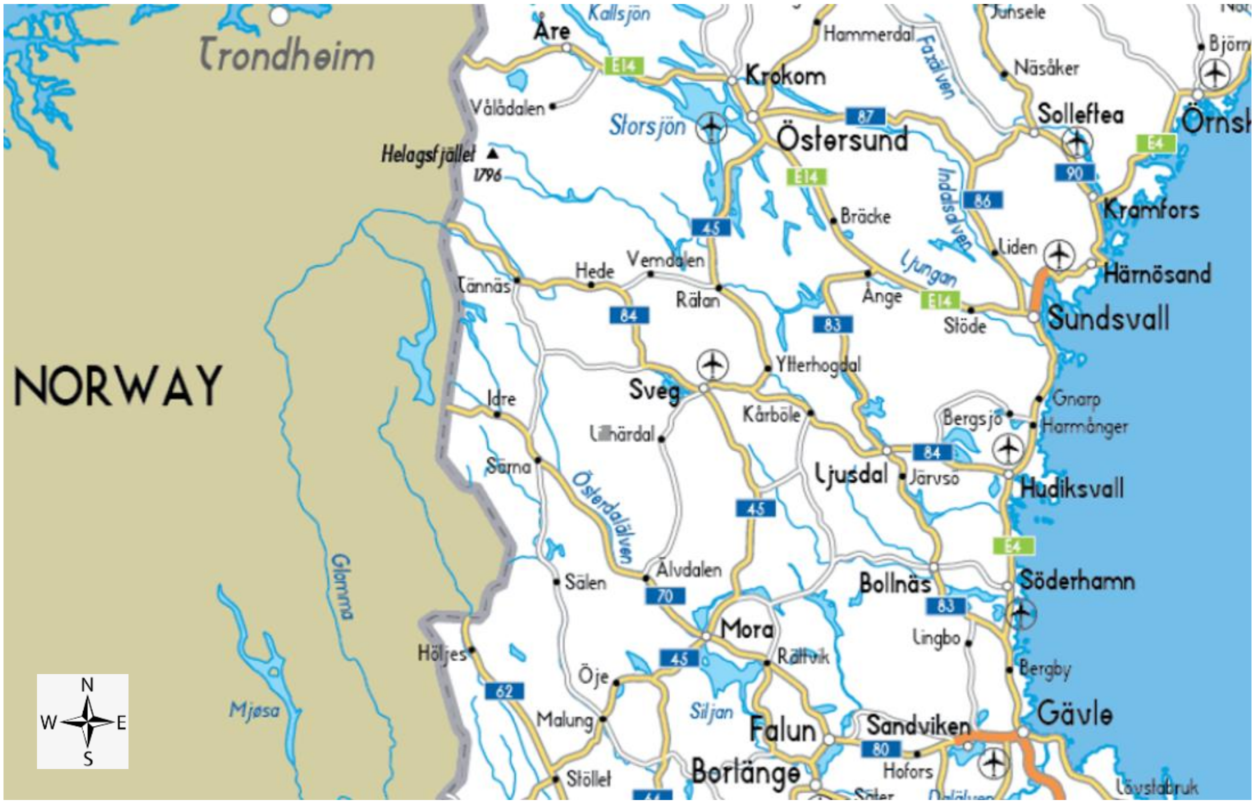
Centre Locations	By Road	By Rail
Östersund to Trondheim, Norway, a port city on the Norwegian Sea	261 km	268 km
Östersund to Sundsvall, a port city on Gulf of Bothnia connected to the Baltic Sea	188 km	196 km
Östersund to Stockholm, capital city of Sweden	564 km	540 km

18.1.1 Roads

The Viken Property is easily accessible via existing regional and national road infrastructure. The Project can be accessed from Highway E14 from Sundsvall, which connects to regional highways, E45 and Road 321, connected to the Project via Svenstavik, located at the southern end of Lake Storsjön. A road map is shown in Figure 18.1. The Town of Östersund - with a population of approximately 31,500 people - is located approximately 40 km by road to the northeast of the Project site and serves as the main regional service centre.

The Project area is bordered by Fröjdholmsvägen to the south and Borgviksvägen to the east, with these roads intersecting near the small community of Myrviken. A 37 km two-way paved road connects Myrviken to Östersund. Major road connections exist from Östersund to Sundsvall (approximately 188 km) and Trondheim, Norway (approximately 261 km), providing regional and international transport links for equipment, reagents, consumables, and product transport.

FIGURE 18.1 SWEDEN ROAD MAP



Source: *Worldometer.com (2026)*

Due to the Project's connection to major highways and established regional roads, the overall road quality approaching the mine site is high. This reduces transport risks associated with oversized loads during the construction phase and improves reliability and safety of access to and from the site during both construction and operations. Access to the Project site will be controlled through a manned security gate at the site entrance, and perimeter fencing will be installed around key operational areas.

Some rural roads exist within the broader Project area; however, the proposed mine development area itself contains no significant previous mining infrastructure and generally limited existing buildings or infrastructure. Depending on the final layout a connecting internal road network will be constructed across the site. A new approximately 600 m long two-way gravel access road will be constructed from the existing local road network into the mine property. Approximately 650 m of internal site roads will be constructed to connect the access road to the open pit, processing facility, waste rock and tailings storage areas, workshops, and administration buildings. Mine site roads will be constructed using locally available waste rock and overburden material generated during pre-stripping activities, which will reduce construction costs and material haulage requirements. These roads will be designed to accommodate heavy mining equipment, haul trucks, and delivery vehicles.

18.1.2 Marine Ports

Marine transport for the Viken Project is supported by access to established ports located at Sundsvall in Sweden and Trondheim in Norway. The Port of Sundsvall provides access to shipping routes along the Baltic Sea and can be utilized for the import of reagents, consumables, and equipment, as well as the export of potential products (Figure 18.2). Trondheim, located west of Östersund, hosts a significantly larger port with access to international shipping routes via the Norwegian Sea, offering greater capacity and flexibility for bulk material handling. The availability of two port options is a strategic advantage for the Project, allowing for optimization of supply chains, particularly for bulk materials such as sulphur, sulphuric acid, fuel, and process reagents.

FIGURE 18.2 GEOGRAPHICAL LOCATION OF ÖSTERSUND AND SUNDSVALL

Source: *Googlemaps (2026)*

18.1.3 Airports

Air transport access to the Viken Project is provided via the airport located in Östersund, which is approximately 42 km by road from the Project site. The airport provides regular commercial flights connecting Östersund to Stockholm and other regional centres, supporting efficient movement of personnel, contractors, and specialist consultants during the construction and operational phases of the Project (Figure 18.3).

FIGURE 18.3 SWEDEN AIRPORT MAP

Source: *Centreforaviation.com (2026)*

18.1.4 Railway

Sweden has a well-developed railway network that extends throughout the country and provides reliable freight and passenger transport connections between major cities, industrial centres, and ports. The Viken Project benefits from its proximity to established rail infrastructure, which can be used for the transport of reagents, consumables, equipment, and potentially product materials.

Railway lines connect Östersund to the coastal port city of Sundsvall, the Norwegian port city of Trondheim, and the capital city of Stockholm. The railway station at Östersund serves as the main regional rail hub and provides access to both domestic and international freight routes. These rail connections are particularly important for bulk transport such as sulphur, sulphuric acid, fuel, reagents, and concentrate or product transport to export ports.

The rail connections to the ports of Sundsvall and Trondheim provide potential export routes for Project products and import routes for reagents and major equipment during both construction and operation. Rail transport is expected to reduce trucking distances, lower transport costs for bulk materials, and reduce road traffic impacts associated with the Project. Figure 18.4 shows a rail map of Sweden, with the main operational rail lines shown in red, illustrating the connectivity between Östersund, major cities, and port facilities.

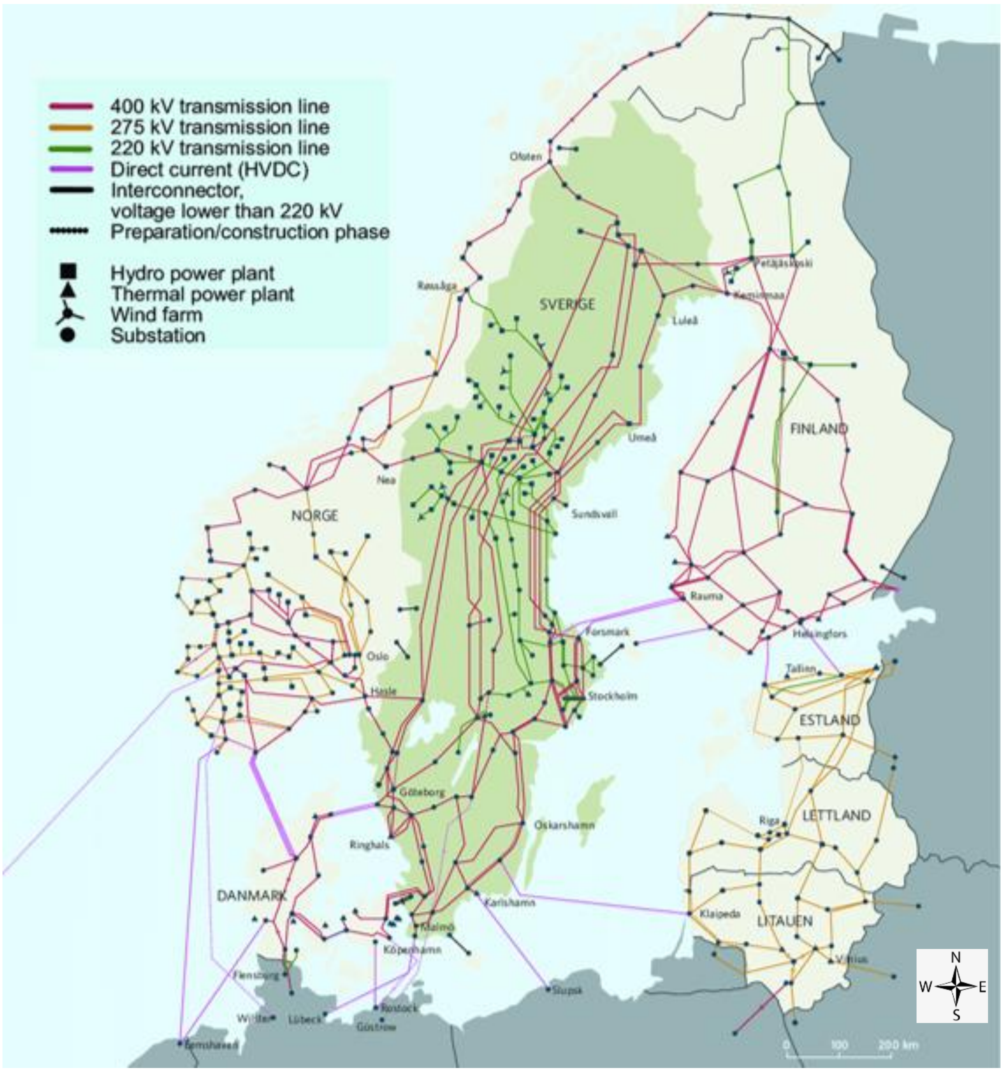
FIGURE 18.4 SWEDEN RAILWAY MAP

Source: *Happyrail.com (2026)*

18.2 POWER SUPPLY

Sweden has a reliable and well-developed power supply system, with electricity generation primarily based on hydroelectric and nuclear power, supplemented by wind and other renewable energy sources. This provides a stable and relatively low carbon electricity supply for industrial projects such as the Viken Project. Local low voltage power lines are available within approximately 1 km of the proposed process plant site, providing a potential nearby connection point for the site power distribution system. In addition, high-capacity transmission lines from the Swedish national grid, rated at approximately 220 kV to 300 kV, are located approximately 40 km from the Project area (Figure 18.5) and could provide access to higher capacity supply in future expansion scenarios.

FIGURE 18.5 SWEDEN ELECTRICITY TRANSMISSION GRID



Source: *Google maps (2026)*

Due to the presence of combustible carbonaceous material within the Viken Deposit, there is potential for supplementary on site power generation using this material as an energy source within the process plant. This concept is at a preliminary stage and will require further metallurgical, energy balance, and environmental studies; however, it may offer an opportunity to reduce external power demand and operating costs depending on the final process design. For the purposes of this PEA, it has been assumed that primary electrical power will be supplied from the regional grid via a new overhead distribution line connecting the site substation to the existing low voltage network located within approximately 3 km of the Project site. Based on current power demand estimates for the proposed mining and processing operation, connection to the high voltage transmission network is not expected to be required at this stage of Project development.

The following major electrical infrastructure components are envisaged for the Project at a conceptual level:

- Site substation;
- Transformers;
- Power distribution lines;
- Motor control centres;
- Backup power systems for critical loads;
- Electrical supply to the process plant, workshops, administration buildings, and water management systems; and
- Power distribution to other site infrastructure as required.

The power supply concept outlined above forms the basis for PEA level capital and operating cost estimates related to electrical infrastructure. Detailed electrical design, load flow studies, and confirmation of grid connection arrangements will be completed during subsequent study phases in consultation with the regional power utility and relevant Swedish regulatory authorities.

The preliminary peak power demand for the pug roasting plant is estimated at 91.91 MW and the peak power demand for the plant inclusive of vanadium electrolyte and FeV production is 92.55 MW. The all-inclusive estimated power consumption is 583 million kWh/a.

18.3 WATER SUPPLY

There are two potential sources of fresh water for the Viken Project: Lake Storsjön and Abassen Creek. At this stage of the Project, neither of these water sources has been fully verified with respect to available water volume, seasonal variability, water quality, permitting requirements, or environmental constraints. Further hydrological and environmental studies will be required during later study phases to confirm the suitability of these sources for long-term Project water supply.

Process water for mining and processing operations will be sourced from Lake Storsjön or Abassen Creek, supplemented where possible by water recovered from open pit dewatering operations. Pit dewatering water may provide a significant portion of process water depending on groundwater inflows and pit water management requirements. The process plant will be designed to maximize water recycling to minimize freshwater intake requirements. Recycled process water will be recovered from the dry stacking operation and returned to the process plant for reuse in grinding, flotation, and other processing operations. This will reduce both freshwater demand and the volume of water requiring discharge.

Excess water from the site water management system, including runoff, seepage collection, and surplus process water, will be collected, treated if necessary, and discharged in accordance with environmental regulations and permit conditions. The preference will be to use water as process water rather than discharge to the environment. Water treatment requirements will depend on water quality, including potential dissolved metals, suspended solids, and process reagents.

For processing at 10 Mtpa the net water requirement is estimated at 186.9 m^3/h.

18.4 NATURAL GAS SUPPLY

Natural gas will be used to support process plant operations, heating systems, drying processes, reagent preparation, and potential on-site power generation. Sweden has an established energy infrastructure; however, the national natural gas grid is limited and primarily located in the southern and western parts of the country. As a result, it is unlikely that a direct connection to a major natural gas pipeline will be available at the Project site. Therefore, alternative natural gas supply options will need to be considered for the Project in the form of delivery of liquefied natural gas ("LNG") by road tanker from regional LNG supply terminals. LNG will be delivered to site, stored in cryogenic storage tanks, and regasified on site for use in boilers, heaters, dryers, or gas-fired generators.

The natural gas supply infrastructure at the site would typically include:

- LNG storage tanks;
- Regasification units;
- Gas pressure regulation systems;
- Gas distribution pipelines within the process plant;
- Metering and control systems; and
- Safety systems including gas detection and emergency shutdown systems.

18.5 FUEL SUPPLY

Fuel will be used to support mining operations, mobile equipment, construction activities, backup power generation, and general site infrastructure. Fuel will be delivered to site by road tanker and stored in designated on-site fuel storage facilities. The fuel storage area will be located within a bunded fuel farm designed to contain spills or leaks and prevent environmental contamination. Bunding capacity will typically be designed to contain at least 110% of the volume of the largest storage tank in accordance with environmental and safety standards.

The fuel storage and distribution system will typically include:

- Bulk diesel storage tanks;
- Bunded containment area;
- Fuel transfer pumps;
- Fuel dispensing stations for heavy vehicles;
- Separate fuel dispensing area for light vehicles;
- Fuel management and metering systems;
- Spill containment and cleanup equipment;
- Fire protection systems; and
- Grounding and lightning protection.

Mobile refuelling trucks may also be used to refuel mining equipment in the pit and other remote areas of the site to reduce equipment downtime and improve operational efficiency. Fuel handling procedures will be implemented to ensure safe storage, handling, and transfer of fuel. These procedures will include spill prevention measures, emergency response procedures, safe refuelling practices, and routine inspection of tanks, pumps, hoses, and valves.

18.6 BUILDINGS

The Project will require several supporting process plant area buildings, facilities and infrastructure which includes, but not limited to:

- Administration buildings;
- Changing rooms and ablution facilities;
- Cafeteria;
- Assay laboratory;
- Workshops and maintenance facilities;
- Warehouses and storage facilities;
- Fuel storage and dispensing station;
- Motor Control Centre (“MCC”) and switch rooms;
- Water treatment facility;
- Parking area for employees and visitors; and
- Other support facilities (IT room, medical and training facilities).

All buildings will be designed and constructed in compliance with applicable Swedish building codes, occupational health and safety regulations, and project design standards. The layout and sizing of buildings will be refined during subsequent study phases to align with the final plant layout, workforce requirements, and operational philosophy.

18.6.1 Accommodation

No provisions have been made for permanent employee housing or Company-provided transportation to and from the site. It has been assumed that employees will reside in nearby communities and will arrange their own transport to work, with on site parking provided for employee vehicles. The availability of a local workforce in the region is expected to be sufficient to staff the operation, supplemented by targeted recruitment where necessary.

Where required, training will be provided to develop and qualify local personnel for higher skill positions on site. Training plans and skills development initiatives will be further detailed during subsequent study phases as part of the overall human resources and operations readiness planning.

18.7 PROCESS PLANT

An on-site process plant will be constructed as part of the Viken Project to treat mined material and produce V_2O_5, U_3O_8 and sulphate of potash (“SOP”) as primary products. The process plant will be designed to process feed from the mining operation through a series of conventional mineral processing and hydrometallurgical stages.

The process flowsheet incorporates successive stages of crushing, grinding, flotation, pug roasting and downstream purification using ion exchange and solvent extraction processes. The process plant will include maintenance facilities to support the repair and servicing of process equipment.

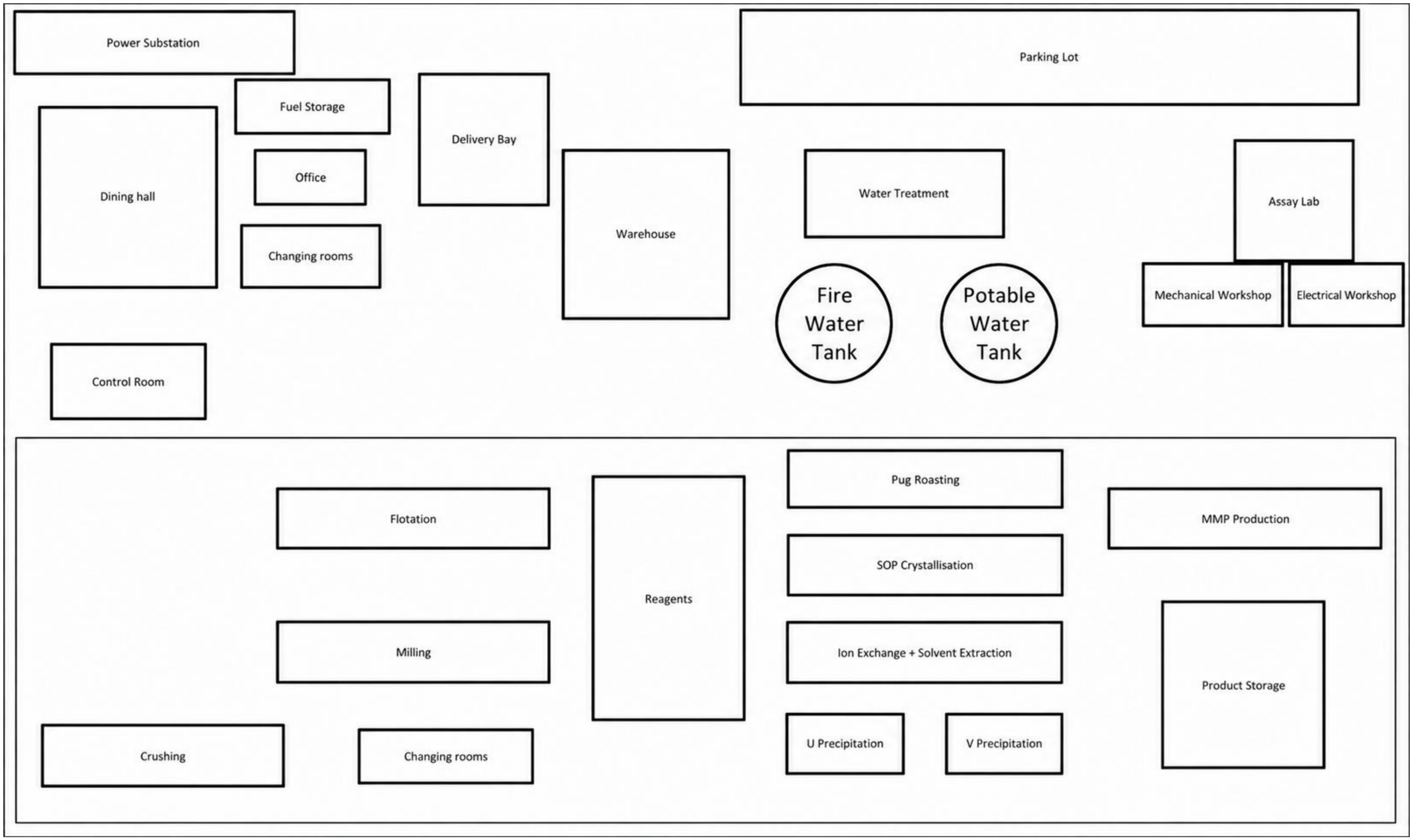
A sulphuric acid production plant will be constructed so that the flotation concentrate can be mixed with the acid prior to pug roasting. Sulphuric acid will also be used as the primary leaching and dissolution reagent within the vanadium electrolyte production process.

Process products will be packaged on-site and stored in a covered warehouse prior to transportation. Storage facilities will be designed to protect the product from environmental exposure and to ensure safe handling and inventory management. Final products will be transported off-site via highway trucks to appropriate rail or port facilities for onward distribution to customers. Packaging and handling of the products will be carried out in accordance with Swedish regulatory requirements and applicable international standards to ensure safety, quality, and compliance during storage and transport.

18.7.1 Conceptual Process Plant Layout

The conceptual process plant layout is shown in Figure 18.6. The layout illustrates the arrangement of major process areas, including crushing, grinding, flotation, pug roasting, solvent extraction and ion exchange circuits, reagent preparation, product packaging, and associated infrastructure. The layout is at a conceptual level and will be refined during subsequent study phases to optimize equipment placement, materials handling, and integration with site infrastructure.

FIGURE 18.6 CONCEPTUAL PROCESS PLANT LAYOUT



Source: *METS (2026)*

18.8 COMMUNICATIONS

Communication systems will be required to support operational coordination, safety systems, monitoring and control systems, corporate data transfer, and general site communications. Satellite telecommunications systems are considered a reliable communication solution for mining operations, particularly in remote or semi-remote locations, and have a strong track record across the mining industry. A satellite communication system will be implemented to provide reliable internet connectivity, corporate network access, email, data transfer, and remote monitoring capabilities. This system will support process plant control systems, reporting systems, maintenance systems, and communication with corporate offices and external stakeholders.

In addition to data communications, the site will require security and operational communication systems. Closed-circuit television ("CCTV") systems will be installed across key areas of the site including the process plant, workshops, fuel storage areas, reagent storage areas, warehouses, access gates, and administration buildings to enhance site security, monitor operations, and support incident investigations if required. A UHF/VHF two-way radio communication system will also be installed to enable communication between mobile equipment operators, process plant operators, maintenance personnel, supervisors, and emergency response teams across the site. Radio communication is essential for day-to-day operations, particularly for mobile equipment coordination, maintenance activities, and emergency response situations.

18.9 WASTE MANAGEMENT

18.9.1 Sanitary Waste

Sanitary waste generated from sinks, lavatories, toilets, and showers will be managed through septic systems installed on site. These systems will either be dedicated to individual buildings or designed to service groups of ancillary facilities that share a common septic tank and leach field, depending on the location and number of facilities. Sewage and grey water produced on site will be treated locally through these septic systems before being discharged. The septic systems will be designed to accommodate the expected workforce numbers and site facilities, including administration buildings, workshops, processing facilities, and accommodation or ablution facilities where applicable.

The design, installation, and permitting of septic systems will be undertaken in accordance with Swedish environmental and public health regulations governing wastewater treatment and disposal. This will include appropriate system sizing, soil permeability considerations, setback distances from water bodies, and groundwater protection requirements. Sludge from septic tanks will be removed periodically by licensed waste contractors and disposed of at approved municipal or licensed treatment facilities.

18.9.2 General/Process Related Waste Disposal

Sinks and drains associated with chemical handling and reagent areas will discharge into the tank farm sump system. The sump will be manually pumped out as required and the collected liquid will be transferred to a dedicated chemical containment tank for temporary storage and management. These containment tanks will be maintained and managed by licensed hazardous materials handling personnel in accordance with relevant state and local government regulations governing hazardous material storage, handling, and disposal.

Solid waste generated on site will be collected in approved waste containers and stored in designated waste collection areas prior to removal from site by a licensed solid waste contractor. Waste disposal will be carried out in accordance with applicable environmental regulations and waste management requirements. During the construction phase, any excess construction materials, packaging, and construction debris will be removed from site by the contractor responsible for generating the waste. Contractors will be required to manage and dispose of their waste in accordance with Project environmental and waste management procedures.

Non-hazardous recyclable materials generated during operations will be segregated and placed into appropriate recycling containers. These materials may include scrap metal, paper and cardboard, used oil, batteries, and wood products. These recyclable materials will be collected and removed from site by approved recycling contractors and processed through appropriate recycling vendors where possible. Hazardous materials and hazardous waste, including contaminated greases, chemicals, paints, solvents, and process reagents, will be collected, stored, and managed in designated hazardous waste storage areas. Wherever feasible, hazardous materials will be recycled or recovered. If recycling is not possible, hazardous waste will be transported off-site by licensed hazardous waste contractors for approved destruction, treatment, or disposal at licensed facilities.

18.9.3 Process Tailings

Tailings produced by the process plant will be dewatered using filtration to reduce the moisture content to typically below 8 to 15% moisture, producing an unsaturated, solid filter cake. At this moisture content, the tailings are not suitable for pipeline transport and will instead be handled by conveyors or haul trucks (costs in this PEA are based on trucking). The filtered tailings will be transported to a designated tailings storage area located on previously cleared land. This is a dry stacked tailings concept. Once there is space in a mined-out open pit, the dry stack tailings will be stored in-pit.

At the storage area, tailings will be deposited in layers, spread, and compacted to form an unsaturated tailings landform. This approach creates a stable storage facility that typically requires minimal or no large, high-risk retention embankments, thereby eliminating the need for conventional large tailings dams. In-pit tailings disposal may also be implemented, subject to the outcomes of detailed geotechnical, hydrogeological, and environmental assessments to confirm the viability and long-term safety of this option.

The conceptual filtered tailings and storage strategy described here forms the basis for PEA level capital and operating cost estimates for tailings management infrastructure. Further engineering design, stability analysis, water balance modelling, and permitting work will be completed during subsequent study phases.

19.0 MARKET STUDIES AND CONTRACTS

19.1 OVERVIEW

The Viken Project is primarily focused on vanadium products, with vanadium electrolyte and ferrovanadium (FeV 80%) identified as the key saleable products for this PEA base case. Commercial-grade V_2O_5 flake is treated as an intermediate product and may also be sold directly, subject to market conditions and customer requirements. In addition, the Project is expected to produce FeV80, vanadium electrolyte, uranium oxide (U_3O_8), sulphate of potash (SOP), and a mixed sulphide by-product containing nickel, zinc and molybdenum, providing additional revenue streams where marketable specifications and payability terms can be achieved. These product assumptions are preliminary and will be refined as further technical and marketing studies are completed.

U_3O_8 production will provide an additional revenue stream through recovery of uranium contained within the Deposit. SOP production is expected to target the agricultural fertilizer market, while nickel, zinc, and molybdenum products will be marketed to the broader base metals industry. Product packaging and logistics assumptions are preliminary and based on standard bulk and containerized shipment for comparable products.

19.1.1 Key Marketing Strategies

19.1.1.1 Multi-Product Strategy

The Viken Project has been designed as a multi-product critical minerals operation capable of producing vanadium pentoxide (V_2O_5) flake, ferrovanadium (FeV), vanadium electrolyte, uranium oxide (U_3O_8), sulphate of potash (SOP), and a mixed sulphide by-product containing nickel, zinc, and molybdenum. This diversified product suite differentiates Viken from many existing vanadium operations, which are typically focused on a single end product and are therefore more exposed to fluctuations within a specific market segment.

By maintaining exposure to both traditional vanadium markets through ferrovanadium and emerging energy storage markets through vanadium electrolyte, the Project is positioned to benefit from multiple long-term demand drivers. Additional exposure to the nuclear energy sector through uranium production, the agricultural sector through SOP, and base metal markets through the mixed sulphide by-product further broadens the Project's revenue base and reduces dependence on any single commodity or end-use market. This provides resilience against commodity price cycles and changing market conditions. In particular, the ability to participate in steel, aerospace, specialty alloy, energy storage, nuclear energy, agriculture, and industrial metals markets provides a level of diversification that is uncommon among vanadium projects globally.

19.1.1.2 Integrated Product Pathway

A key feature of the Viken Project is the production of vanadium pentoxide (V_2O_5) as a common intermediate product. The upstream beneficiation, extraction, purification, and refining circuits are designed to produce high-purity V_2O_5 irrespective of the ultimate product. Commercial-grade V_2O_5

flake may be sold directly into the global vanadium chemicals market, subject to market conditions and customer requirements. Alternatively, the V_2O_5 can be further processed into ferrovanadium and vanadium electrolyte through dedicated downstream circuits. This product-agnostic upstream design provides a high degree of commercial flexibility. As market conditions change, the Project can adjust the allocation of V_2O_5 between the various downstream products without requiring substantial reinvestment in mining, beneficiation, leaching, purification, or V_2O_5 production infrastructure.

19.1.1.3 Downstream Product Flexibility

The downstream production split between ferrovanadium and vanadium electrolyte can be adjusted through operational parameters within the downstream processing facilities. Production rates can be influenced by modifying furnace batch frequency and, where justified by long-term market demand, furnace sizing and diameter during future expansions or plant upgrades. Alternatively, the downstream facilities may be designed with sufficient flexibility to accommodate future modifications and feed allocation adjustments, enabling the Project to respond to evolving market conditions and customer requirements. This flexibility allows the Project to optimize product allocation based on prevailing market conditions, customer contracts, and long-term demand outlooks. For example, periods of strong steel sector demand may favour increased ferrovanadium production, whereas rapid deployment of vanadium redox flow batteries may support a greater allocation of V_2O_5 towards electrolyte manufacture.

19.1.1.4 Integrated Operation

Viken is expected to be one of the few integrated vanadium operations globally capable of producing all three principal vanadium products—V_2O_5 flake, ferrovanadium, and vanadium electrolyte. The integrated structure offers several advantages over single-product producers:

- Exposure to multiple vanadium end-use markets;
- Reduced dependence on a single customer segment;
- Ability to optimize product mix over the life of the operation;
- Improved resilience to market cyclicality;
- Enhanced opportunities to establish strategic partnerships across the steel and energy storage sectors; and
- Greater value capture through participation in downstream processing rather than selling only intermediate products.

Another key advantage is the ability of the operation to manage product quality variations within the integrated flowsheet. Off-specification vanadium products from the electrolyte circuit can be redirected to the ferrovanadium production circuit, where impurity tolerances are generally less restrictive. This provides an additional recovery pathway and reduces the risk of material becoming a low-value by-product or requiring reprocessing. This capability represents a meaningful advantage over standalone electrolyte producers, which may have limited options for managing off-specification material and therefore face higher quality control and disposal costs.

19.1.1.5 High Purity Electrolyte Opportunity

As the vanadium redox flow battery ("VRFB") market continues to expand, electrolyte quality is becoming an increasingly important differentiator. High-purity vanadium electrolyte can improve electrochemical stability, reduce the risk of membrane fouling and electrolyte degradation, and contribute to improved battery performance, reliability, and component life. These benefits can lower the total cost of ownership for battery operators through reduced maintenance requirements and longer operating life.

The global vanadium electrolyte market is currently dominated by Chinese supply, creating an opportunity for alternative Western sources of high-quality, traceable electrolyte. Through the production of high-purity vanadium pentoxide (V_2O_5) and downstream electrolyte processing, the Viken Project is well positioned to supply premium electrolyte products to the growing energy storage sector. Consequently, customers may be willing to pay a premium for high-quality, traceable electrolyte sourced from producers operating under stringent environmental, quality assurance, and supply chain standards. In addition to providing geographic diversification of supply, Viken has the potential to support customers seeking secure, transparent, and high-quality vanadium products, strengthening the Project's competitive position within the rapidly growing VRFB market.

19.1.1.6 Uranium Production

In addition to its vanadium products, the Viken Project contains a significant uranium inventory that is recoverable as a by-product. Uranium production has the potential to reach approximately 3.3 million pounds of U_3O_8 per annum. At this production level, the Project has the capability to supply the full uranium requirements of Sweden's current nuclear reactor fleet. Sweden currently operates six commercial nuclear reactors, which provide a substantial portion (~30%) of the country's electricity generation. Despite its long history of nuclear power generation, Sweden does not currently have a domestic uranium mining industry and imports uranium and nuclear fuel from international suppliers. Uranium feedstock is sourced from countries including Canada, Australia, Namibia, Kazakhstan and others, while enrichment services and fuel fabrication are performed through international supply chains. The ability of Viken to produce uranium domestically could therefore provide an important contribution to Sweden's and Europe's long-term energy security objectives. In addition to reducing reliance on imported uranium feedstock, domestic production would support greater supply chain resilience for a nuclear fleet that currently depends on international mining, conversion, enrichment, and fuel supply networks.

19.2 COMMODITY PRICES

The long-term price assumptions in Table 19.1 have been used in this PEA, expressed in United States dollars.

TABLE 19.1 COMMODITY PRICES (US$)		
Commodity	**Unit**	**Price**
Ferrovanadium (FeV80)	kg	38
Vanadium Electrolyte	L	9
Uranium Oxide	lb	85
Sulphate of Potash (for 50% K_2O SOP granules)	t	650
Nickel	lb	7.71
Zinc	lb	1.45
Molybdenum	lb	27.22

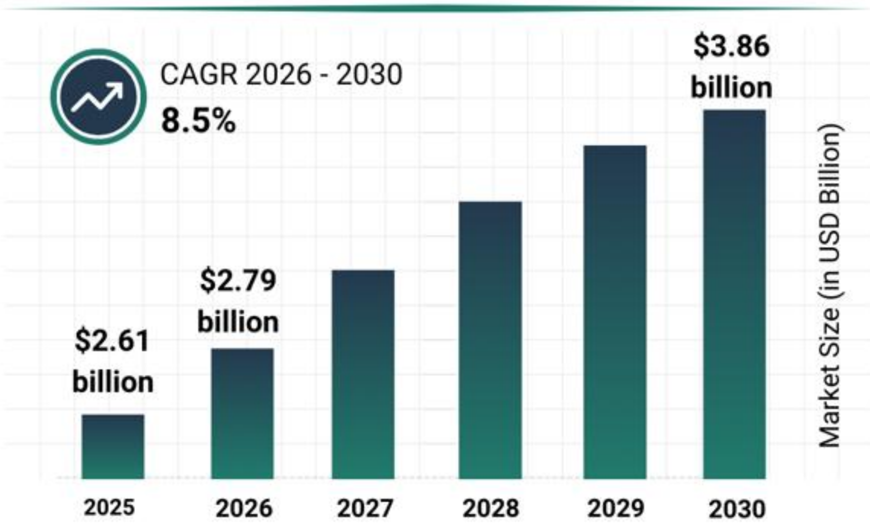
19.3 MARKET STUDIES

19.3.1 Vanadium Market

The vanadium market has grown steadily in recent years and is expected to continue expanding through to 2030, supported by structural demand from both the steel industry and the emerging energy storage sector. The growing preference for sustainable and low-emission materials is expected to be a key driver of future vanadium demand, particularly through increased use in high-strength steel applications that improve material efficiency and reduce overall carbon intensity in infrastructure and construction.

The global vanadium market was estimated at approximately US$2.61 billion in 2025 and is projected to reach ~US$3.86 billion by 2030, representing a compound annual growth rate (CAGR) of approximately 8.5% from 2026 to 2030 (Figure 19.1). Market growth is primarily attributed to increasing crude steel production, supported by rising demand from the construction, automotive, machinery, infrastructure, and transportation sectors. Long-term growth is also expected from the expanding adoption of vanadium redox flow batteries ("VRFBs") and long-duration energy storage applications associated with the global energy transition.

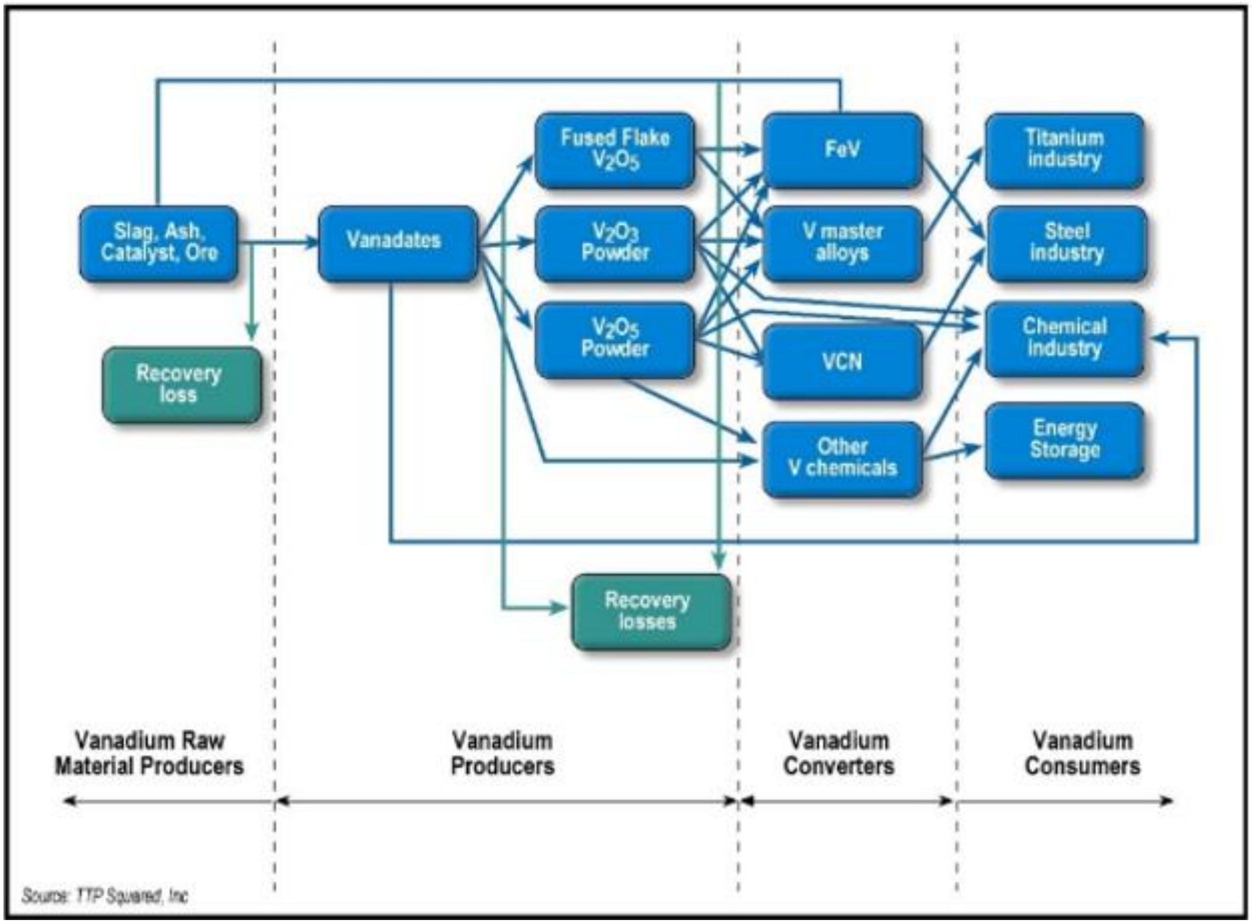
FIGURE 19.1 VANADIUM MARKET



Source: *thebusinessresearchcompany.com*

Figure 19.2 provides an overview of the vanadium industry and identifies the main vanadium raw materials and intermediate products in the supply chain as well as the primary consumer industries.

FIGURE 19.2 VANADIUM PRODUCTION CHAIN



Source: *TTP Squared Inc.*

The Viken Project focuses on the production of vanadium pentoxide - flake, ferrovanadium and vanadium electrolyte as its vanadium products. The Project's process flow has been designed around the production of high-purity V_2O_5 as an intermediate product, which serves as the feedstock for the downstream production of ferrovanadium and vanadium electrolyte.

Ferrovanadium (FeV) is an iron–vanadium alloy primarily used as an additive in steel and specialty alloys. Its primary uses are:

- High-strength low-alloy ("HSLA") steel (largest market);
- Rebar and structural steel (construction sector); and
- Tool steels, aerospace and titanium alloys.

Vanadium electrolyte is a battery-grade liquid solution containing dissolved vanadium salts used in Vanadium Redox Flow Batteries ("VRFBs"). It is the core energy storage medium in VRFB systems whose primary uses are:

- Grid-scale energy storage (long-duration storage, 4 to 12+ hours); and
- Renewable energy integration, microgrids and off-grid systems.

For Viken, exposure to both traditional steel-related vanadium demand (via FeV) and emerging VRFB electrolyte markets provides diversified revenue potential. However, the Project's competitiveness will depend on achieving cost effective V_2O_5 production, securing offtake in a relatively concentrated market, and managing price volatility linked to steel cycles and VRFB deployment rates.

Global vanadium production is heavily concentrated, with more than 80% of supply originating from China, Russia, South Africa, and Brazil. China is the dominant producer, accounting for approximately two-thirds of global vanadium supply. Vanadium is primarily recovered as a co-product or by-product from titaniferous magnetite ores during steelmaking processes, resulting in a supply chain that is closely linked to global steel production cycles.

Global vanadium production increased from approximately 109,400 tonnes in 2020 to 129,000 tonnes in 2024 before easing to approximately 122,700 tonnes in 2025. Despite recent production growth, the global vanadium market remains structurally concentrated due to the limited number of primary vanadium operations and the industry's dependence on steel-related co-product production. As a potential primary producer located outside the major existing vanadium-producing regions, Viken could offer geographic and geopolitical diversification to end users, although its competitiveness will depend on operating costs, product quality and logistics relative to other suppliers.

The global vanadium market remains relatively concentrated on both the supply and demand sides, with market dynamics strongly influenced by steel production activity, Chinese consumption patterns, and emerging demand from the energy storage sector. Historically, the vanadium market has experienced periods of supply deficit and surplus due to the limited number of primary vanadium producers and the heavy reliance on co-product production from steelmaking operations.

Vanadium supply has generally remained sufficient to meet traditional steel-related demand, however, the accelerating growth of the VRFB sector may place additional pressure on future supply availability. This is particularly significant given the concentrated nature of global production and the relatively small pipeline of new primary vanadium projects outside major producing regions such as China and Russia. The long-term supply-demand balance is expected to depend on the successful development of new primary vanadium projects, expansion of electrolyte manufacturing capacity, increased recycling and secondary recovery initiatives, and improvements in extraction efficiency from existing steelmaking streams.

Ferrovanadium prices have historically tracked V_2O_5 prices, with a conversion premium reflecting alloying and processing costs (~US$5 to 10/kg V). Unlike V_2O_5, FeV pricing is based on the contained vanadium units (US$/kg V) and the alloy grade (typically FeV80 vs FeV75). Ferrovanadium is typically sold as FeV (50 to 80% V content), with FeV80 being the most common grade. Pricing is typically quoted per kilogram of contained vanadium, not per tonne of the alloy. The cost of FeV experienced a sharp increase from 2018 to 2019, when prices surged to multi-year highs exceeding US$100/kg (US$45/lb) in most markets. This shift was driven primarily by the introduction of stricter rebar quality standards in China, which increased vanadium consumption in steel production. Following the peak, prices moderated during 2020 to 2021 as supply chains adjusted to the higher demand environment and market conditions stabilized. Between 2022 and 2024, FeV prices experienced fluctuations in line with broader global economic conditions, with inflationary pressures contributing to cost variability. A price decline was experienced in the period between 2020 to 2025 with prices stabilizing around US$30 to 50/kg V. This was driven by weak global steel demand and oversupply of vanadium units. Spot prices are currently showing an upward trend due to ongoing supply concerns in China and tightened environmental checks. In Q4 of 2025 and Q1 of 2026 prices have shown an early recovery trend in line with V_2O_5 supported by production curtailments and gradual increase in steel demand. Current indicative price range for 2026 is US$32 to 45/kg V (FeV basis) which is equivalent to US$14.5 to 20.5/lb V.

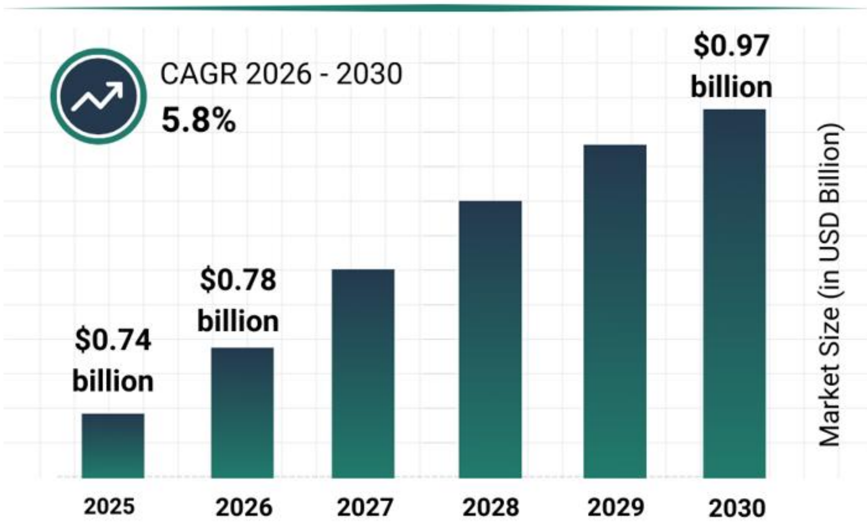
Vanadium electrolyte pricing is derivative based, not an independent commodity market. It is derived from V_2O_5 prices, with additional costs for processing, purification, and electrolyte preparation. The historical price trend follows that of V_2O_5. Global demand for vanadium electrolyte is currently estimated at approximately 52 million liters per annum, with demand expected to increase to around 170 million litres per annum by 2031, representing a CAGR of approximately 15 to 20%. The growth is being driven primarily by increasing deployment of VRFBs. In comparison, the ferrovanadium market is significantly larger and more mature, with current global consumption estimated at approximately 120 to 130 million kilograms per annum, largely supported by demand from the steel sector. Market forecasts indicate continued growth in ferrovanadium demand, with projected CAGR of 4 to 6% over the next decade. Recent industry reports estimate the global ferrovanadium market value will increase from approximately US$8.6 billion in 2024 to US$15.4 billion by 2034, highlighting sustained demand growth across both traditional steelmaking and emerging energy storage sectors.

19.3.2 Uranium Market

The uranium-yellowcake market has strengthened significantly in recent years and is expected to maintain positive growth through to 2030, supported by renewed global interest in nuclear energy as part of decarbonization and energy security strategies. Increasing electricity demand, commitments to reduce greenhouse gas emissions, and the extension and restart of nuclear generating capacity are expected to remain the primary drivers of future uranium demand. Additional growth is anticipated from the development of advanced reactor technologies and small modular reactors (“SMRs”), which are expected to expand the long-term nuclear fuel market. Supply constraints arising from years of underinvestment, production discipline by major producers, and the lengthy development timelines required for new uranium projects are also expected to support favourable market conditions. Overall, the uranium market is anticipated to experience sustained expansion as governments pursue low-carbon baseload power generation and diversify energy supply sources. Notwithstanding this constructive outlook, uranium market conditions remain sensitive to nuclear policy decisions, project execution risks, changes in secondary supply availability and the timing of SMR deployment. The PEA price assumptions should therefore be regarded as preliminary.

The global uranium ore market was valued at approximately US$0.74 billion in 2025 and is projected to grow to approximately US$0.97 billion by 2030 (Figure 19.3), representing a CAGR of approximately 5.8%. Market growth is expected to be driven by additional nuclear generating capacity being commissioned globally, increasing energy security requirements, and the transition toward low-carbon power generation. Further demand growth is also anticipated from the development of SMRs, which are expected to increase future uranium consumption requirements.

FIGURE 19.3 URANIUM ORE MARKET



Source: *thebusinessresearchcompany.com*

The uranium market differs from most metals due to:

- Most uranium is sold via long-term contracts;
- Spot price is less important than contract price; and
- The market is influenced by government policy and nuclear energy programs.

Uranium is commonly traded as uranium oxide concentrate (U_3O_8), more commonly referred to as "yellowcake." Yellowcake is an intermediate uranium product typically packaged in sealed steel drums for transport to downstream conversion and enrichment facilities prior to nuclear fuel fabrication. Yellowcake (U_3O_8) will be the uranium product produced in the Viken Project, assumed to be packaged in sealed steel drums for shipment to conversion facilities. Yellowcake from Viken is expected to be sold into the conventional U_3O_8 market for conversion and enrichment, targeting utilities or trading houses under long term contracts typical for uranium supply.

The world has approximately 5.9 million tonnes of known, recoverable uranium resources. Global uranium production stands at roughly 60,000 to 65,000 tonnes of uranium annually, supplying approximately 85-90% of global commercial reactor demand. Global uranium production is concentrated among a relatively small number of producing countries, with Kazakhstan, Canada, Namibia, Australia, and Uzbekistan accounting for the majority of worldwide uranium supply. Kazakhstan is the dominant global producer, primarily through low-cost in-situ recovery ("ISR") operations, and typically contributes more than 40% of annual global uranium production.

Global uranium production has experienced significant volatility over the past decade, driven by fluctuating nuclear generation demand, geopolitical developments, supply curtailments, and sustained underinvestment following the Fukushima Daiichi nuclear disaster. Despite improving market fundamentals in recent years, uranium supply growth has remained relatively constrained, with several major producers adopting production discipline strategies to support price stability and market rebalancing. For the purposes of this PEA, it is assumed that Viken's yellowcake production would be sold predominantly under long term contracts, consistent with prevailing industry practice.

A structural supply deficit currently exists within the uranium market, with global mine production estimated to fall short of annual reactor consumption by approximately 35 to 40 million pounds of uranium per year. This deficit is currently bridged through secondary supply sources, including down-blended military stockpiles, reprocessed nuclear fuel, commercial inventories, and other legacy supply streams. However, reliance on these secondary sources is expected to become increasingly constrained over the longer term, further supporting the need for additional primary uranium production capacity.

Demand for uranium is primarily driven by the nuclear power generation sector, where yellowcake serves as the key feed material for nuclear fuel production. Global demand is closely linked to reactor operating capacity, new nuclear construction programs, reactor life extensions, and national energy security strategies. The global transition toward low-emission energy systems and decarbonization policies has reinforced nuclear power as a stable baseload energy source, supporting sustained uranium demand growth. China remains the most significant driver of future demand growth due to its large-scale nuclear expansion program, while additional growth is expected from India, the Middle East, and reactor restarts in other regions. Emerging technologies such as SMRs are also expected to contribute incremental long-term demand growth with more than 30 countries committing to triple global nuclear capacity by 2050. Annual uranium requirements are projected to increase significantly over the coming decades, with long-term demand estimates reaching approximately 238 to 250 million pounds of uranium per year.

Global uranium consumption is currently estimated at approximately 145 to 155 million pounds per annum (equivalent to ~65,000 to 70,000 tonnes of uranium), supplied by around 440 operating nuclear reactors worldwide. A typical 1,000 MW reactor requires approximately 400 to 600 tonnes of uranium per year, meaning large reactor fleets collectively underpin substantial and stable baseline demand. The United States is the largest single consumer, with approximately 40 million pounds per year, supported by a fleet of approximately 93 reactors. Other major consuming regions include France, China, South Korea, and Japan, which collectively account for most of the remaining global demand due to their reliance on nuclear power for low-carbon baseload electricity. Industry forecasts indicate that uranium consumption could rise above 180 Mlb per year by 2030, driven by new reactor builds, life extensions and SMR deployment. These projections are indicative and subject to change as nuclear policy, project delivery and technology evolve.

Uranium prices were relatively low during the period from 2011 to 2020, generally ranging between US$20 and US$30/lb of U_3O_8. Between 2021 and 2024, uranium prices increased significantly due to supply constraints, reduced production from major producers, and renewed global interest in nuclear energy as a low-carbon power source. Since then, prices have been considerably higher, with spot prices ranging between US$63 and US$83/lb of U_3O_8. Long-term contract prices are typically higher and more stable than spot prices, ranging between US$80 and US$105/lb of U_3O_8 depending on contract terms, delivery schedules, and market conditions. Some industry commentators project a cumulative supply deficit on the order of 100 Mlb over the next decade, reflecting increased nuclear capacity and growing electricity demand, including from data centres. Under such scenarios, U_3O_8 spot prices could exceed US$100/lb; however, these projections are uncertain and should be treated as indicative rather than deterministic. As of April 2026, uranium yellowcake (U_3O_8) spot prices are approximately US$84 to 88/lb, declining slightly from previous highs above US$100/lb as a result of increased global supply entering the market.

19.3.3 Sulphate of Potash (SOP) Market

The sulphate of potash ("SOP") market has demonstrated steady growth and is expected to continue expanding, driven by increasing demand for premium and chloride-free fertilizers in high-value agriculture. Growth is expected to be supported by rising global food consumption, expansion of horticultural crops, greenhouse farming, precision agriculture and increasing cultivation of fruits, vegetables, nuts and specialty crops that benefit from low-chloride nutrient sources. Another important trend influencing the SOP market is the increasing focus on sustainable agricultural practices and improved nutrient efficiency, encouraging greater use of specialty fertilizers. Technological improvements in fertilizer production processes, irrigation systems and controlled-environment agriculture are also expected to support market development. Overall, the SOP market is expected to maintain positive growth due to increasing agricultural intensification, higher crop quality requirements and continued adoption of advanced farming practices.

At the same time, SOP market development is exposed to risks including competition from muriate of potash ("MOP") plus specialty blends, potential over capacity if multiple new projects are developed concurrently, and sensitivity to energy and freight costs. Policy changes affecting fertilizer use, environmental regulation and trade flows may also influence regional pricing and demand patterns. The SOP price and demand assumptions used in this PEA should therefore be regarded as preliminary and will be revisited as more detailed marketing engagement is undertaken. For this PEA, SOP is treated as a secondary revenue source, with economics contingent on achieving suitable product specifications and transport/logistics costs.

The global SOP market was valued at approximately US$56.1 billion in 2024 and is forecast to reach around US$82.9 billion by 2032, implying a CAGR of approximately 5%. SOP is a chloride-free potash fertilizer containing potassium and sulphur, typically sold at a premium to MOP due to its suitability for chloride sensitive and high value crops. Key applications include premium fertilizers for fruits, vegetables, tobacco, nuts and other high value agricultural products, as well as organic and intensive horticulture sectors.

The global SOP market is characterized by a relatively concentrated supply base and limited production capacity compared with standard potash fertilizers such as MOP. SOP is primarily produced via two routes: the Mannheim process, which converts potassium chloride and sulphuric acid into SOP, and natural brine or salt lake extraction from sulphate-rich deposits.

China dominates global SOP production, accounting for approximately 60% of global capacity, supported by both Mannheim-based facilities and natural brine operations, with total national capacity of around 6.6 Mtpa. Globally, SOP production capacity is estimated at approximately 11.2 Mtpa, although actual output is lower at around 6.6 to 7.3 Mtpa due to plant utilization rates and prevailing market conditions. Production outside China is distributed across Europe, North America, South America and the Middle East, with a mix of chemical-based and brine extraction operations.

Viken's SOP production is expected to be based on brine/solution/chemical processing of process liquors and would be sold as granular product. This positions the Project outside the traditional Mannheim-based supply but subject to similar feedstock and energy cost sensitivities. At a production rate estimated at 250,000 tonnes per year SOP as fertilizer, the Project would be capable of supplying approximately 16% of Europe's annual SOP demand that is currently import-dependent.

Supply growth remains constrained due to high production costs, limited natural resource availability, and reliance on key feedstocks such as potassium chloride and sulphuric acid. As a result, new capacity additions have primarily focused on lower-cost brine-based projects and selective expansions aimed at servicing premium agricultural markets. Overall supply conditions remain sensitive to energy pricing, feedstock availability, logistics constraints, geopolitical factors and environmental regulations. These factors are particularly important given SOP's focus on high value, chloride sensitive agricultural markets where consistent quality and delivery are critical.

Global SOP demand is driven primarily by the agricultural sector, which accounts for approximately 80 to 95% of total consumption, reflecting SOP's role as a premium chloride-free potassium fertilizer for high-value and chloride-sensitive crops. Global SOP demand is estimated at approximately 6.5 to 7.3 Mtpa based on various industry sources, significantly smaller than the broader MOP market due to SOP's higher production costs and premium pricing.

Asia-Pacific represents the largest consuming region, accounting for approximately 50 to 60% of global demand, led by China, while Europe and North America remain important markets due to intensive horticulture, fertigation systems and precision agriculture practices. Future demand growth is expected to be supported by increasing global food consumption, expansion of greenhouse and hydroponic farming, rising cultivation of high-value crops, and greater adoption of water-efficient agricultural systems. Demand growth is also expected to benefit from the increasing use of specialty fertilizers aimed at improving crop quality, nutrient efficiency and soil management.

SOP prices were relatively stable during the 2015 to 2020 period, ranging between US$450 and US$550/t depending on region and product quality. Between 2021 and 2023, SOP prices increased significantly due to global fertilizer supply shortages, rising energy costs, and disruptions to supply chains, particularly from major potash-producing regions. During this period, SOP prices in some markets exceeded US$900/t. Since 2024, prices have moderated, however, remain above long-term historical averages, typically ranging between approximately US$500 and US$700/t depending on region, transport costs, and contract terms. Long-term contract prices for SOP are generally more stable than spot prices and are commonly used for project economic evaluations in prefeasibility and feasibility studies. Most industry forecasts indicate that long term demand for SOP should remain robust, and prices are expected to remain broadly stable in real terms with potential for moderate growth. These outlooks are indicative and subject to uncertainty regarding fertilizer demand, energy costs and supply from new projects.

19.3.4 Mixed Metal Sulphides

The market outlook for mixed metal sulphides containing nickel, zinc and molybdenum remain positive, supported by favourable long-term demand fundamentals across industrial, infrastructure and energy transition sectors. Nickel demand is expected to continue increasing due to its role in stainless steel production and expanding use in battery chemistries for electric vehicles and energy storage systems. Zinc demand is anticipated to remain closely linked to infrastructure development, galvanized steel consumption and renewable energy construction, while molybdenum is expected to benefit from growing requirements for high-strength and corrosion-resistant alloys used in energy, aerospace and industrial applications. Technological advancements in battery materials, electrification infrastructure, advanced manufacturing and process optimization are expected to further support market growth.

Overall, demand for nickel, zinc and molybdenum is expected to remain robust through to 2030, underpinned by stainless steel and alloy applications, infrastructure investment and the energy transition. However, prices for these commodities have historically been cyclical and volatile, and are sensitive to changes in mine supply, policy, technology and macroeconomic conditions. In addition, the MMS product contemplated for Viken is an intermediate material that will require confirmation of smelter acceptance and commercial terms. The price and payability assumptions used in this PEA should therefore be regarded as preliminary and subject to further validation as additional testwork and market engagement are undertaken.

Global supply of nickel, zinc and molybdenum is supported by well-established mining and refining industries, with production distributed across several major jurisdictions. Given the scale of the global nickel, zinc and molybdenum markets and the established smelting and refining capacity in Europe and other regions, Viken's MMS production is expected to represent a small incremental feed source that could be blended into existing smelter circuits, subject to achieving acceptable impurity levels and logistics costs. Identification of suitable smelter partners and confirmation of indicative terms will be a key focus for subsequent study phases.

19.4 CONTRACTS

There are currently no material contracts in place for the Viken Project.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 BACKGROUND

The Viken Project is a based on a moderately low-grade uranium polymetallic deposit, hosted in the Alum Shale. The Project is envisaged to include open pit mining and on-site processing operations to produce vanadium oxide (V_2O_5) refined into FeV and vanadium electrolyte products, uranium as "yellowcake" (U_3O_8), as well as a potassium salt for conversion to a fertilizer, and a mixed metals sulphide concentrate of molybdenum, zinc, and nickel.

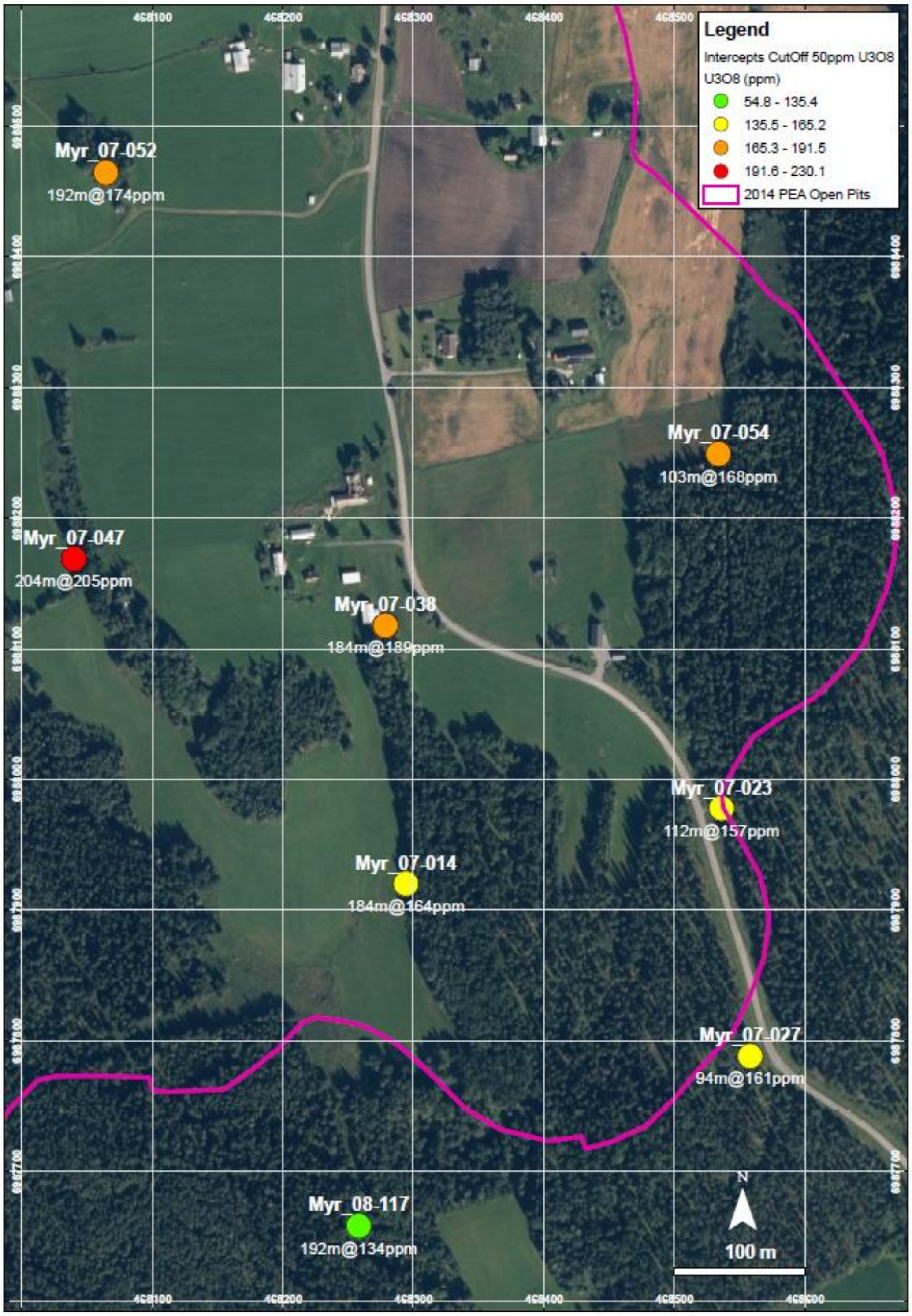
The Viken Project had been considered several years ago. Some public concerns and oppositions had arisen as early as 2013, one of which was the potential impact on water quality in Lake Storsjön, a large lake located to the east of the Viken Project area (Figure 20.1).

FIGURE 20.1 VIKEN PROJECT LOCATION

Source: *P&E (2025)*

The Viken Deposit location is thinly populated and in an agricultural and coniferous-forest covered region, as shown in Figure 20.2. Indicated uranium content at depth determined by drilling is shown in this figure.

FIGURE 20.2 2025 VIKEN SITE VISIT STOP 2 AREA



Source: *District Metals (June 2025)*

Opposition has emerged to potential mining at the Viken Project, as outlined in a recent April 2025 article entitled "Uranium Mining Resurfaces in Swedish Village"[1]. Landowners, rural residents and local Oviken village persons, including health care professionals, have expressed strong opposition to any uranium-based mining project in the area. The potential for water contamination and land disturbance are among the recent concerns.

20.2 CURRENT VIKEN PROJECT SCOPE

An outline of the scope of the Project for the purposes of this PEA includes open pit Alum Shale mining, grinding, flotation, pug roasting, calcining, metal concentrating and hydrometallurgical processing, and associated waste management activities. The Viken Project would produce vanadium oxide refined into FeV and vanadium electrolyte products, uranium yellowcake, sulphate of potash ("SOP"), as well as mixed metal sulphides. These metals are closing associated with organic black carbon-rich zones of the Alum Shale. The potential exists for the SOP to be converted into a fertilizer. Several process options could be considered, however, initially a potential overall process named the Pug Roasting Process has been identified and is illustrated in Section 17 of this Report.

Some anticipated Project details would include:

- Large scale (>30 ktpd) open pit black shale mining. Surface limestone would be excavated and transferred to a waste rock storage location. The underlying mineralized Alum Shale material would be excavated and hauled to a primary crusher. The multi-stage crusher discharge would be calcined at >550°C to remove CO_2 from carbonates.

- The calcined material would be finely ground and subjected to froth flotation to concentrate the vanadium-bearing mica and uranium and other heavy metal containing mineralization.

- The flotation concentrate would be mixed with sulphuric acid followed by a sulphation roast at approximately 250°C.

- The roasted material would subject to a water leach process. The valuable components of the Pregnant Leach Solution ("PLS") would conceptually be recovered by factional crystallization (potassium sulphate) and by ion exchange/solvent extraction to separately recover vanadium and uranium. The barren solution is expected to contain dissolved base metals such as zinc, nickel and molybdenum.

From air management environmental perspectives, dust emissions from crushing, gaseous emissions from the calcining and roasting will need management and control. The flotation tailings combined with other residues will be "dry stacked" and covered on a continuous basis with an impervious multi-zone soil cover. Barren PLS would be neutralized with lime and the resulting gypsum and metal hydroxide water treatment by-products would be co-disposed in a special chemical solids storage facility. Treated water would be recycled as much as possible.

[1] Resolve.media April 4, 2025, "Uranium Mining Resurfaces in Swedish Village"

In general, until additional studies are completed, the Viken Project is at the conceptual stage and there is a possibility that process stages would be further modified and improved to meet economic and environmental targets.

20.3 REGULATORY FRAMEWORK AND PERMITTING PROCESS

20.3.1 Regulatory Framework

The key aspects are:

- The Swedish Minerals Act and Minerals Ordinance (SFS 1991:45, SFS 1992:285 and amendments) which would apply to mine development and operation in Sweden. This is linked to the Swedish Environmental Code, the Planning and Building Act and the Cultural Heritage Management Act;

- The Swedish Environmental Code (SFS 1998:808 and amendments) and relevant ordinances and regulations. This includes reference to the Nuclear Activities Act (1984:3) and plants for the extraction of uranium-bearing materials. The Swedish Radiation Safety Authority of the Ministry of the Environment has a nuclear safety, radiation protection mandate;

- The Minerals Act (1991:45) and Minerals Ordinance; and

- The Radiation Protection Act (SFS 1988:220) and the Nuclear Activities Act (SFS 1984:3), which have been amended to conform to European Community Legislation.

- In a press release dated December 20, 2024, the Swedish Ministry of Climate and Enterprise published the results of its 2024 inquiry into lifting the uranium moratorium. In the press release, the Swedish Government re-iterated its intention to lift the 2018 ban on uranium mining and restore the legislation as it was beforehand. In summary:

    - The current ban in the Environmental code is to be overturned, and it shall be possible to mine uranium in Sweden; and
    - Uranium will be regulated as a concession mineral within the Swedish Minerals Act.

  In addition, the inquiry into lifting the uranium moratorium listed the next steps:

    - A written consultation on the inquiry will be conducted until March 20, 2025;
    - A legislative proposal is to be brought to Parliament after March 20, 2025; and
    - The legislative changes are proposed to go into effect by January 1, 2026.

The legislative proposal to lift the uranium ban was approved in Parliament on November 5, 2025. The Swedish moratorium on uranium exploration and mining was lifted effective January 1, 2026. On June 15, 2026, the Swedish Parliament voted to abolish a Municipal veto on uranium extraction that goes into legislative effect on July 15, 2026. A key aspect of the legislation is that the extraction and processing of uranium is now treated in a manner consistent with other metals and minerals within Sweden's overall permitting framework.

On June 11, 2026, the Swedish Government presented the terms of reference for a new inquiry that will investigate ways to enhance local and municipal influence for Alum Shale deposits. The inquiry mentions that it will include concerned parties from the private sector in their stakeholder dialogue. It will also provide an analysis of commercial consequences of any proposal made and will take into account Sweden's commitments according to the Critical Raw Materials Act ("CRMA").

The Viken Deposit is currently being reviewed by the Geological Survey of Sweden ("SGU") as a deposit of National Interest (Riksintresse), and the SGU's review is expected to be completed in Q3 2026.

20.3.2 Permitting Requirements

The envisaged Project would be subject to Swedish regulatory requirements. Two principal permits would be required under the Minerals Act (Exploration Permit and Exploitation Concession) and the Environmental Code (Environmental Permit). The Viken Project may also be subject to provisions under the Nuclear Activities Act, which regulate the right to acquire, possess or deal with minerals containing materials to supply the nuclear power facilities in Sweden.

A Permit for Exploitation of Project will include an Environmental Impact Assessment ("EIA") compliance requirement, which is described in the Swedish Environmental Code EIA requirements under the Environmental Code. These requirements include public consultations and the approval of local municipal councils under the Environmental Code. The technical and social aspects of the Project would be expected to be critically reviewed during an EIA procedure and in the permitting/licensing process. It is expected that the Project would be designed to provide a high-level overview of potential adverse impacts to the public, the Project workers, and the existing ecology.

It is anticipated that extensive baseline data and information collection and assessment work would be required to support the development of an EIA for the Project. Hydrogeological characterization work, including pump tests, bedrock testing, and till characterization, are required to produce a detailed ground water model that would be needed for preliminary pit designs and ensure protection of groundwater and lake water.

If an extraction project is defined as a 'strategic project' within the EU Critical Raw Materials Act then this entitles 'fast-track' processing which could potentially be completed within a maximum of 27 months from the application date. The timeline applies to the entire permit-granting process, including environmental assessments, once the application is considered complete. Most Swedish mining projects require 4 to 10 years from initial permitting to final approval. The timeline is dependent on appeals, land-use conflicts, and environmental reviews.

20.4 COMMUNITY CONSULTATIONS AND ENGAGEMENT

Local support is important for the Project and the licensing process has recently been adjusted with further reviews underway. On June 15, 2026, a legislative proposal was approved to remove the municipal veto on uranium extraction and processing, which goes into legislative effect on July 15, 2026. In parallel, on June 11, 2026, the Swedish Government published the terms of reference for a new inquiry to examine measures aimed at enhancing local and municipal influence over the development of Alum Shale deposits. The inquiry states that it will engage with stakeholders, including representatives from the private sector, as part of its consultation process. It will also assess the commercial implications of any proposed measures while considering Sweden's obligations under the Critical Raw Materials Act ("CRMA").

Concerns about the potential impacts of a medium-scale mining project in the area can be considered to be normal. Impacts on land use, ecology, and water can be considered significant but manageable, given modern engineering practices to obtain successful reclamation. Concerns can be expected to be expanded for the Project due to the presence of low-level radioactivity and the potential production of a uranium concentrate.

The Viken Project proponent has established an initiative managed by the District Metals' Swedish subsidiary, Bergslagen Metals AB, to provide information for landowners, municipalities and other stakeholders. One-on-one meetings with landowners associated with exploration drilling permit applications are examples of to-date community engagement.

Typically, new mining projects now incorporate procedures and controls to protect human health and the ecology from potential impacts to the land, air, surface and ground water quality. Domestic animals and wildlife, vegetation, valued ecosystem components and most importantly people in the region and at locations outside of the region could potentially be affected by the Project. While environmentally and socially acceptable mining and mineral processing activity has been and continues to be located in many areas of Sweden, there has not been a significant history of mining in the Project area, and therefore detailed discussions and explanations of the Project aspects are necessary.

While the uranium content of the Viken Deposit is low, the control and management of radioactivity can be expected to be a focus of Project development. Perceived and actual success in control and management of the low-level uranium-based radioactivity will depend on extensive public involvement in determining the risk and the success in the application of robust radiation dose management methodology, as well as the limitation of the dispersion of two important uranium "daughters" – radium and radon.

The potential impacts of the project and environmental protection controls will need to be established through an extensive environmental and social impact review process. Areas where additional field study and assessment focus may be required for this Project over the foreseeable future include potential impacts to groundwater quality and uses, surface water quality and uses, air quality, fisheries and aquatic life, land access, and use and valued ecosystem components.

20.5 CLOSURE

The following components of a Closure Plan for the Project are expected to be developed. This Plan will likely include:

- As part of the permitting/licensing process under the Swedish Environmental Code, any operator seeking a permit for mining must provide financial security to ensure that all closure, remediation, and aftercare obligations can be fulfilled;

- Dry stack tailings and waste rock will be placed in mined-out open pits as mining progresses on the Property so that the land is returned to a "natural" state. There may be an allowance for the last mined-out pit to naturally flood, with in-pit water treatment (e.g. slaked lime dispersion as well as natural water treatment at the depth of the pit water). Deep pit water can provide an anoxic heavy metal trap as well as collect suspended solids from incidental surface erosion;

- In the early years of closure, any pit "lake" level would be depressed by pumping to ensure conservation of groundwater quality;

- Flattening pit wall slopes above any eventual pit lake level;

- Placing the limestone waste rock, non-mineralized shale and poorly mineralized shale storage stockpiles in a stable physical condition in storage piles or in mined-out pits;

- Under the current concept, the leached concentrate and flotation tailings blending and storing in a multi-segment engineered facility that is progressively closed out with a multi-zone impervious soil cover;

- Reclaiming the mine and process plant sites and rehabilitating to a "natural" state; and

- Insuring that the Project would have comprehensive waste management procedures as part of an integrated environmental management system that would detail the Project's monitoring program during development, operations, closure works, and post-closure.

21.0 CAPITAL AND OPERATING COSTS

21.1 SUMMARY

All costs are presented in Q2 2026 US dollars. No provision has been included in the cost estimates to offset future escalation. A contingency of approximately 20% has been added to all capital costs ("CAPEX"). No contingency is added to operating costs ("OPEX").

The total initial CAPEX of the Viken Project is estimated at $876M. Sustaining capital costs incurred during the 13 production years and one year of rehabilitation are estimated to total $158M. Total OPEX over the life-of-mine ("LOM") is estimated at $5.9B, averaging $46.58/t processed. The following subsections provide details of these costs.

21.2 CAPITAL COSTS

21.2.1 Initial Capital Costs

Table 21.1 presents the CAPEX incurred over the Viken Project LOM.

TABLE 21.1
CAPITAL COST SUMMARY

Capital Cost Item	Initial ($M)	Sustaining ($M)	Total ($M)
Mine	60.4	44.2	104.6
Process Plant & Infrastructure & Indirect Costs	657.2	79.2	736.4
Rehabilitation	0	8.3	8.3
Contingency	158.5	26.4	184.9
Total Capital Costs[1]	**876.1**	**158.1**	**1,034.2**

Note: *[1] Totals may not sum due to rounding.*

21.2.1.1 Mine

The initial requirements of open pit mining equipment are planned to be purchased during the two year, pre-production period. Major equipment is comprised of surface excavation machines, haul trucks and front-end wheel loaders. The rest of the open pit mining equipment is considered support equipment and consists of items such as dozers, graders, small excavators, maintenance equipment and service vehicles. The initial CAPEX for open pit mining equipment during the pre-production period is estimated at $44.7M. The major equipment capital costs are based on recent budgetary quotes from Caterpillar and Komatsu.

Haul road construction and site development in preparation for mining are additional initial CAPEX estimated at $15.7M.

The total initial capital cost for open pit mining equipment, pre-stripping and set up of open pit operations is estimated at $60.4M as shown in Table 21.2.

Table 21.2 Open Pit Mining Capital Cost Estimate			
Item	**Initial Capital ($M)**	**Sustaining ($M)**	**Total LOM ($M)**
Mining Equipment	44.7	44.2	88.9
Pre-stripping and Site Development	15.7	0.0	15.7
Total[1]	**60.4**	**44.2**	**104.6**

Note: *[1] Totals may not sum due to rounding.*

21.2.1.2 Process Plant and Infrastructure

A cost estimate based on a 10 Mtpa acid process plant pug roast flowsheet was developed for the Viken Project. This provides substantiated costs for the Project infrastructure. The CAPEX estimation was produced by the Authors utilizing estimating procedures and systems in accordance with AACE Class 5 Estimate methodology and accuracy. Table 21.3 presents the estimated CAPEX for the process plant and acid plant at $657.2M. The largest cost drivers are the acid plant, which costs $37M to produce enough acid to operate at 10 Mtpa, electrical power connection to the national grid with site distribution which costs $31M, the dry stack tailings facility which costs $18M, the ball mill which costs $16M, rougher flotation cells at $12M, and the pug mill which costs $8M.

Table 21.3 Process Plant and Infrastructure Capital Cost Summary				
Capital Cost Item	**Equipment ($M)**	**Directs ($M)**	**Indirects ($M)**	**Total ($M)**
10 Mtpa Pug Roast with Acid Plant	247.5	198.0	151.9	597.4
FeV Production Circuit	13.7	10.9	8.4	33.0
Vanadium Electrolyte Production Circuit	0.8	0.6	0.5	1.9
Miscellaneous Infrastructure	10.0	8.0	7.0	25.0
Total[1]	**272.0**	**217.5**	**167.8**	**657.2**

Note: *[1] Totals may not sum due to rounding.*

Process Plant and Infrastructure Equipment Costs

Table 21.4 provides the total equipment costs for each area of the 10 Mtpa pug roast process plant with an acid plant. The total CAPEX is estimated at $247.5M.

TABLE 21.4 PROCESSING EQUIPMENT AND INFRASTRUCTURE COSTS		
Area Code	**Area**	**Cost ($M)**
100	Crushing	6.5
200	Milling	25.5
300	Flotation	32.2
400	Leaching	20.3
500	SOP Crystallization	9.0
600	Ion Exchange and Solvent Extraction	8.3
700	Precipitation	2.0
800	MMP Production	23.8
900	Dry Stacking	26.6
1000	Reagents	5.3
1100	Utilities and Services Incl. Power	88.0
Total Equipment Costs[1]		247.5

Note: *[1] Totals may not sum due to rounding.*

Process Plant and Infrastructure Direct Costs

Table 21.5 provides a summary of the direct costs factored from the total equipment value. Direct costs are estimated at $198.0M.

TABLE 21.5 DIRECT COSTS	
Direct Cost Area	**Cost ($M)**
Earthworks	12.4
Concrete	4.9
Structural Steelwork	24.7
Mechanical Installation	86.6
Pipework	24.7
Electrical and Instrumentation	17.3
Roads	4.9
Freight	22.3
Total Direct Costs[1]	**198.0**

Note: *[1] Totals may not sum due to rounding.*

Process Plant and Infrastructure Indirect Costs

Table 21.6 provides a summary of the indirect costs factored from equipment value and direct costs. Indirect costs are estimated at $151.9M.

TABLE 21.6 INDIRECT COSTS	
Indirect Cost Area	**Cost ($M)**
Working Capital	44.5
Insurance	13.4
EPCM	44.5
Owner's Costs	13.4
Commissioning	22.3
Workforce accommodation & meals, temp services	8.9
Spares and tools	4.9
Total Indirect Costs[1]	**151.9**

Note: *[1] Totals may not sum due to rounding.*
EPCM *= engineering, procurement and construction management.*

FeV Production Circuit

The FeV Production Circuit CAPEX is primarily driven by the smelting package associated with the aluminothermic reduction and DC arc furnace operations. The FeV crushing circuit represents a comparatively small proportion of the total capital cost due to the use of conventional crushing and screening equipment. Indirect costs contribute significantly to the total CAPEX due to EPCM, commissioning and Owner's cost allowances applied at the PEA level. FeV production and smelting equipment CAPEX is estimated at $13.5M, and crushing is estimated at $0.2M. The total cost of the circuit is estimated at $33.0M.

Vanadium Electrolyte Production Circuit

The Vanadium Electrolyte Production Circuit CAPEX is primarily driven by the electrolyte production area, which includes the dissolution, reduction, filtration and electrolyte conditioning equipment. The reagents and utilities area represents a smaller proportion of the total capital cost due to the relatively simple reagent storage and handling infrastructure. Indirect costs contribute significantly to the total CAPEX due to EPCM, commissioning and Owner's cost allowances applied at the PEA level. Reagents and utilities equipment costs are estimated at $0.3M, and vanadium electrolyte production equipment costs are estimated at $0.5M. The total cost of the circuit is estimated at $1.9M.

Miscellaneous Infrastructure

Infrastructure CAPEX of $26.0M is estimated for items not included in the process plant costs above. A breakdown of the infrastructure items is presented in Table 21.7.

TABLE 21.7 MISCELLANEOUS INFRASTRUCTURE COSTS	
Infrastructure Item	**Cost ($M)**
Maintenance shop	10
Offices for all G&A	5
Warehouse	5
Diesel farm	4
Fencing	2
Total[1]	**26**

Note: *[1] Totals may not sum due to rounding.*

21.2.1.3 Contingency

Contingency costs on initial CAPEX are estimated to total $158.5M which equates to approximately 22%. Mining capital items have been allocated a 20% contingency cost. Process plant and infrastructure contingency is based on 30% of direct CAPEX.

21.2.2 Sustaining Capital Costs

Total sustaining CAPEX over the production years is estimated at $158.1M and is presented above in Table 21.1.

21.2.2.1 Mine

Mine sustaining CAPEX consists primarily of mining equipment replacement ($35.3M) with additional costs for equipment rebuilds and overhauls ($8.9M), and is estimated to total $44.2M.

21.2.2.2 Process Plant

Sustaining CAPEX for the process plant is based on 3% of direct costs, or $6.3M per annum before contingency. Over the 13 production years this totals $79.2M after accounting for $4.2M in the last year.

21.2.2.3 Rehabilitation

The cost to rehabilitate the site after 13 production years of mining and processing is estimated at $8.3M. A complete closure cost has not been estimated since the Deposit contains a vast Mineral Resource and it is likely that a new future mining area will be established and that the feed will be treated in the existing process plant.

During each year of operation, dry stack tailings will be sequentially closed out, consisting of approximately 10 Mt of tailings (10 Mt of ROM, less 0.5 Mt of volatiles plus 0.5 Mt of reagents), and will be covered with multi-zone low permeability soils. Seepage drains with a central drain collection system will be established when the dry stack tailings storage facility is constructed and will be in service throughout operations.

During operations, the collected drainage will be treated and discharged to the local surface water environment. The treatment process will be dedicated to tailings characteristics. Due to high levels of dissolved salts in this water, no recycling to the process plant is anticipated, however, this will be reviewed in future studies.

Similarly, during operations, waste rock materials will be stockpiled with the dry stack tailings. The waste rock is mainly limestone and is assumed that it will not be metal leaching (however, could include radioactive elements) and is unlikely to be acid generating. Tests would need to be conducted for design and treatment during operations.

A water treatment plant will be constructed to treat open pit water. The treated water would likely be pumped to the process plant as reclaim water. Once the open pit is dry at the end of operations, it would be allowed to flood with natural ground water.

Decontamination of any equipment to be sold would be necessary, and any scrap materials would need to be disposed appropriately.

Additional site fencing will be installed so that the entire perimeter of the mine site is enclosed.

21.2.2.4 Contingency

A 20% contingency has been applied to all sustaining CAPEX and is estimated to total $26.4M over the production years and rehabilitation year.

21.3 OPERATING COSTS

Operating costs over 13 years of production are estimated to average $46.58/t processed as presented in Table 21.8. The operating costs have been estimated from first principles and consumable quotes, with factoring and estimates from the Author's experience at other mines.

TABLE 21.8
UNIT OPERATING COSTS

Item	Unit Cost ($/t processed)	LOM Total ($M)
Open Pit Mining	3.50	445.4
Processing	37.43	4,767.1
Dry Stack Tailings	1.81	231.0
General & Administrative	1.33	169.0
SOP Freight	2.51	319.9
Total Operating Cost[1]	**46.58**	**5,932.4**

Note: *[1] Totals may not sum due to rounding.*

21.3.1 Open Pit Mining

A breakdown of LOM average open pit mining unit operating costs by activity and by element is presented in Table 21.9. LOM total costs are also shown.

Mine operating costs are derived from a combination of first principle calculations with an in-house equipment database for major and supporting equipment operating parameters, and include fuel, consumables, labour ratios, and general parts costs. The average open pit mine operating cost is estimated at $2.95/t mined ($3.50/t processed) over the 13 production years.

Surface excavation machine capacity is based on guidelines provided by the manufacturer. Annual mineralized material tonnes, waste rock tonnes and loading and hauling hours are calculated based on the capacities of the loading and hauling fleet. These tonnes and hours provide the basis for support fleet inputs.

The diesel fuel price delivered to site is estimated at $2.58/L based on details provided by GlobalPetrolPrices.com and reviewing recent prices in Sweden.

All equipment costs are based on estimated fuel consumption rates, consumables costs, ground-engaging tools ("GET"), and general parts and preventative maintenance costs on a per-hour or per-metre unit basis.

Operating labour man-hours are categorized for the different labour categories such as operators, mechanics, electricians, etc. The mining cost also includes all mine salaried staff, technical consumables and software.

TABLE 21.9
OPEN PIT MINING OPERATING COST ESTIMATE

Item	Unit Cost ($/t material)	LOM Total Cost ($M)
By Activity		
Surface Excavation	0.54	81.0
Loading	0.25	37.8
Hauling	1.06	159.7
Services/Roads/Waste Storage	0.74	110.9
General/Supervision/Technical	0.23	34.7
5% Allowance	0.14	21.2
Total [1]	**2.95**	**445.4**
By Element		
Operating Labour	0.43	64.3
Maintenance Labour	0.19	28.2
Supervision and Technical	0.22	32.7

TABLE 21.9 **OPEN PIT MINING OPERATING COST ESTIMATE**		
Item	**Unit Cost ($/t material)**	**LOM Total Cost ($M)**
Non-Energy Consumables and Parts	0.72	108.0
Fuel	1.25	188.7
Outside Services	0.02	2.3
Allowance	0.14	21.2
Total [1]	**2.95**	**445.4**

[1] Totals may not sum due to rounding.

21.3.2 Processing

This section outlines the operating cost estimate prepared for the Viken Deposit PEA. Process plant operating costs for the main cost categories—reagents, labour, consumables and power—have been developed from multiple sources and consolidated into a total operating cost estimate. Where practicable, these costs have been benchmarked against comparable projects and alternative data points. All values are considered accurate to within ±50%. The cost estimates and underlying assumptions are described in the following subsections.

21.3.2.1 Estimate Basis and Scope

Process operating costs have been prepared to a PEA Study level. The overall operating cost estimate was consolidated using METS estimating procedures and systems. All monetary values are presented in United States Dollars, with conversions from Swedish Krona (“SEK”) provided for consumables sourced locally. The conversion rate used is 9.28 SEK:US$1.00.

Limitations, Exclusions and Qualifications

The operating cost estimate has been prepared for the process plant only and excludes the following items:

- Currency fluctuation impacts;
- Bonds and other financial guarantees;
- Corporate and head office support services;
- Royalties and all forms of taxation;
- Mining and mine-site operating costs;
- Operating cost contingencies; and
- Insurance other than that explicitly included elsewhere in this PEA.

21.3.2.2 OPEX Development Methodology

The operating cost estimate was developed using a combination of assumptions and indicative estimates for the principal cost drivers, namely:

- Power;
- Consumables;
- Reagents;
- Labour;
- Maintenance;
- Processing General and Administrative Costs (G&A); and
- Water Costs.

21.3.2.3 Key Technical and Economic Assumptions

The key assumptions used to build the OPEX estimate are:

1. The OPEX was developed using a Mineral Resource split of V_2O_5 at 40%, U_3O_8 at 35%, SOP at 10% and MMP at 25%.

2. Annual process plant throughput (tpa) used was 10,000,000 tonnes.

3. Feed grades were assumed constant throughout the year at:
    - 4% for K
    - 0.2% for V
    - 0.016% for U.

4. Product recoveries were based on the mass balance.

5. Product recoveries were assumed constant throughout the year.

6. Only major reactions were considered when determining reagent costs.

7. Acid consumption was extracted from the mass balance based on reaction stoichiometry.

8. An OPEX estimate was developed based on the Pug Roast Leach inclusive of an onsite acid plant scenario.

21.3.2.4 Process Plant Operating Cost Summary

A summary of the overall operating cost estimates for a 10 Mtpa processing plant based on the Pug Roast process route is provided in Table 21.10.

TABLE 21.10 PROCESS PLANT OPERATING COST ESTIMATE						
Item	**$M/year**	**$/t of ROM**	**$/lb of V_2O_5**	**$/lb of U_3O_8**	**$/t of SOP**	**$/t of MMP**
Reagents	259.0	25.90	3.16	28.60	98.34	15.56
Labour	13.3	1.33	0.16	1.47	5.04	0.80
Consumables	14.2	1.42	0.17	1.57	5.38	0.85
Power	46.8	4.68	0.57	5.17	17.78	2.81
Maintenance	14.2	1.42	0.17	1.57	5.38	0.85
G & A	3.7	0.37	0.05	0.41	1.42	0.23
Water Costs	3.9	0.39	0.05	0.43	1.47	0.23
Total	**355.0**	**35.50**	**4.34**	**39.22**	**134.81**	**21.33**

21.3.2.5 Processing Cost Components

Reagents Consumption and Cost Basis

Reagent costs have been derived from METS' previous projects and cost database, supplemented by vendor quotations and publicly available pricing sources. The total reagent costs include an allowance of 10% for freight. Reagent consumption rates were taken from the process mass balance, which was developed using historical Viken testwork, benchmark data from prior METS projects, and engineering calculations. The reagent prices applied in the estimate are summarized in Table 21.11.

TABLE 21.11 REAGENT PRICES	
Reagent	**Cost ($/t)**
Flocculant	4,106
Collector	3,236
Frother	3,340
Sodium Carbonate	388
Iron Powder	518
Magnesium Oxide	272
Phosphoric Acid	971
Calcite	13
Sodium Hydroxide	362
TBP	5,566
D2EHPA	5,955
Kerosene	712
Anhydrous Ammonia	712
Sulphur	142

TABLE 21.11 REAGENT PRICES	
Reagent	**Cost ($/t)**
Oxygen	64
LPG	1.52
Sulphuric Acid	150

Reagents Notes:

- Flocculant, collector and frother are used in the flotation circuit.
- Na_2CO_3, MgO, NaOH and $CaCO_3$ are used for pH control and neutralization.
- H_2SO_4 is used as the primary leaching agent. Process options that include an acid plant assume on site combustion of sulphur to produce sulphuric acid.
- Iron powder and oxygen are applied as reducing and oxidizing agents, respectively.
- Phosphoric acid is used for downstream impurity control.
- TBP, D2EHPA and kerosene are used in the solvent extraction (SX) circuits.
- IX reagents are not included in this study.
- Anhydrous ammonia and magnesium oxide are used for product precipitation.
- A thermal efficiency (heat loss) of 40% has been assumed.
- Calcite is assumed to be mined and processed on site with the cost used for the study being made up of the mining and processing fees to produce it.
- LPG is treated as a consumable reagent in this study. The large volumes required for kiln heating and calcination significantly influence overall reagent costs, and a detailed fuel source assessment and heat and energy balance should be undertaken in the next project phase to identify the most suitable energy source for kiln and drying operations.

Process Plant Labour and On-Costs

The salary scheme applied in this PEA was developed using a combination of information from previous METS projects, internal databases, and publicly available sources. Salary data were primarily sourced from Paylab.com, which provides current annual salary benchmarks for a range of technical and operational roles within a specific region. Where discrepancies in salary levels were observed due to differences in seniority or role definitions, average salary values were adopted to maintain consistency within the estimate. The labour costing covers all process plant employees, including managers, metallurgists, operators and other staff. Operational rosters were developed based on 12-hour shift patterns, consistent with common practices for large-scale processing facilities. The workforce is assumed to consist predominantly of locally sourced labour, which contributes to comparatively lower labour costs within the operating expenditure. The salary on-costs for personnel were calculated as a percentage of their basic salary. The percentages for the on-costs are shown in Table 21.12.

TABLE 21.12 ON-COST PERCENTAGES	
On-Cost Item	**Percentage of Basic Salary**
Pension fund	10.2
Survivors Pension	1.17
Health Insurance	4.35
Occupational Injury Insurance	0.3
Parental Insurance	2.6
Unemployment Insurance	2.91
Payroll Contribution	9.88
Total	**31.42**

Plant Consumables

Operating consumables are assumed to be 5% of the equipment cost extracted from the CAPEX.

Electrical Power Demand and Tariff Assumptions

The PEA assumes that electrical power is supplied entirely from the state power grid with the cost of connection to the grid included in the capital cost estimate. The cost of power was charged at 8 cents per kWh based on the METS database (previous Sweden project). Power requirements were extracted from the equipment list based on the process flowsheets developed by METS. The power draws were calculated using the utilization and load factors provided in Table 21.13.

TABLE 21.13 PROCESS POWER UTILIZATION AND LOAD FACTOR		
Area	**Utilization (%)**	**Load Factor (%)**
Area 100 Crushing	90	80
Area 200 Milling	90	80
Area 300 Flotation	90	80
Area 400 Leaching	90	80
Area 500 SOP Crystallization	90	80
Area 600 Solvent Extraction	90	80
Area 700 Precipitation	90	80
Area 800 MMP Production	90	80

TABLE 21.13 PROCESS POWER UTILIZATION AND LOAD FACTOR		
Area	**Utilization (%)**	**Load Factor (%)**
Area 900 Dry Stacking	90	80
Area 1000 Reagents	90	60
Area 1100 Utilities and Service	90	80

Electrical power consumption for each area is calculated as Installed kW × Utilization × Load Factor. The peak power demand for the pug roasting plant is estimated at 91.91 MW and the peak power demand for the process plant inclusive of vanadium electrolyte and FeV production is 92.55 MW. The all-inclusive estimated power consumption is 583 million kWh/a.

Maintenance Strategy and Cost Allowances

Maintenance costs were estimated at 5% of total equipment costs extracted from the CAPEX.

Processing Related General and Administrative Costs

The Processing General & Administrative (G&A) costs include items not captured in the other operating cost categories, such as:

- Safety/Medical Clinic Labour/Equipment/Medicines costs assumed $100,000/annum;
- Camp Management/Catering/ Housekeeping costs assumed based on $40/person/day;
- Communication Costs assumed at $200,000/annum;
- Laboratory cost assumed at $200,000/annum;
- Contingency for mobile equipment lease. Assumed fixed cost of $1,500,000/annum; and
- Security requirements for the site and camp. Assumed fixed cost of $500,000/annum.

Process Water Supply, Usage and Cost

The water cost was estimated based on similar projects in the METS database at $2.59 per tonne. Water consumption was extracted from the mass balance.

21.3.2.6 FeV Production OPEX Estimate

A summary of the overall operating cost estimates for FeV production based on the Pug Roast process route is provided in Table 21.14.

TABLE 21.14 FEV PRODUCTION OPERATING COST ESTIMATE			
Item	**$M/year**	**$/t of V_2O_5 Feed**	**$/lb of Contained V**
Reagents	11.18	1,157.41	0.70
Labour	1.45	149.66	0.09
Consumables	0.68	70.72	0.04
Power	0.30	30.86	0.02
Maintenance	0.68	70.72	0.04
G & A	0.68	70.15	0.04
Total	**14.96**	**1,549.52**	**0.94**

Key Technical and Economic Assumptions for the FeV Production Circuit

The following key assumptions were used to develop the FeV Production Circuit operating cost estimate:

- The OPEX estimate was developed based on approximately 65% of the total V_2O_5 production being directed to FeV production, with the remaining V_2O_5 allocated to vanadium electrolyte production;
- The annual V_2O_5 feed to the FeV circuit was assumed to be approximately 9,655 tpa, producing approximately 7,211 tpa of FeV product;
- The FeV product grade was assumed to be approximately 75% vanadium; and
- The FeV production process was based on aluminothermic reduction of V_2O_5 using aluminium reductant, iron addition and lime flux within a DC arc furnace configuration.

FeV Production Reagents Consumption and Cost Basis

The reagent prices applied in the estimate are summarized in Table 21.15.

TABLE 21.15 REAGENTS FOR FEV PRODUCTION COST SUMMARY	
Reagent	**Cost ($/t)**
Scrap aluminum	1,821.00
Scrap iron	400.00
Quicklime (CaO)	150.00

Reagents Notes:

- Aluminium reductant is used as the primary reducing agent within the aluminothermic FeV production process;
- Iron addition is used during smelting to achieve the target ferrovanadium alloy composition and support furnace operation;
- Lime flux is added to assist slag formation and impurity control within the DC arc furnace;
- Process consumptions were developed based on the preliminary mass balance and major reaction stoichiometry prepared for the study;
- Only the major process reactions and primary consumables were considered when estimating reagent consumption and operating costs; and
- Detailed optimization of reagent consumptions, furnace energy efficiency and reductant utilization should be undertaken during future study phases.

FeV Production Related General and Administrative Costs

The Processing General & Administrative (G&A) costs include items not captured in the other operating cost categories, such as:

- Safety/Medical Clinic Labour/Equipment/Medicines costs assumed $100,000/annum, with 20% allocated to the FeV production circuit;
- Camp Management/Catering/ Housekeeping costs assumed based on $40/person/day for the FeV operational workforce, with 100% allocated to the FeV production circuit;
- Communication Costs assumed at $200,000/annum, with 15% allocated to the FeV production circuit;
- Laboratory cost assumed at $200,000/annum, with 30% allocated to the FeV production circuit;
- Mobile equipment lease. Assumed fixed cost of $1.5M/annum, with 25% allocated to the FeV production circuit; and
- Security requirements for the site and camp. Assumed fixed cost of $500,000/annum, with 10% allocated to the FeV production circuit.

21.3.2.7 Vanadium Electrolyte Production OPEX Estimate

A summary of the overall operating cost estimates for Vanadium Electrolyte production based on the Pug Roast process route is provided in Table 21.16.

TABLE 21.16
VANADIUM ELECTROLYTE PRODUCTION OPERATING COST ESTIMATE

Item	$M/year	$/t of V_2O_5 Feed	$/t of Electrolyte	$/lb of Contained V
Reagents	2.65	510.21	56.72	911.10
Labour	0.75	143.49	15.95	256.23
Consumables	0.04	7.73	0.86	13.80
Power	0.03	5.27	0.59	9.40
Maintenance	0.04	7.73	0.86	13.80
G & A	0.60	116.17	12.91	207.45
Water Costs	0.09	17.80	1.98	31.79
Total	**4.20**	**808.40**	**89.86**	**1,443.58**

Key Technical and Economic Assumptions for the Vanadium Electrolyte Production Circuit

The following key assumptions were used to develop the Vanadium Electrolyte Production Circuit operating cost estimate:

- The OPEX estimate was developed based on approximately 35% of the total V_2O_5 production being directed to vanadium electrolyte production, with the remaining V_2O_5 allocated to FeV production;
- The annual V_2O_5 feed to the vanadium electrolyte circuit was assumed to be approximately 5,199 tpa, producing approximately 46,770 tpa of vanadium electrolyte product;
- The vanadium content within V_2O_5 was assumed to be approximately 56%, with contained vanadium within the electrolyte estimated at approximately 2,911 tpa; and
- The vanadium electrolyte production process was based on dissolution and reduction of V_2O_5 using sulphuric acid, oxalic acid and process water, followed by filtration and electrolyte conditioning.

Vanadium Electrolyte Production Reagents Consumption and Cost Basis

The reagent prices applied in the estimate are summarized in Table 21.17.

TABLE 21.17 REAGENTS FOR VANADIUM ELECTROLYTE PRODUCTION COST SUMMARY	
Reagent	**Cost ($/t)**
Sulphuric acid	150
Oxalic acid	600

Reagents Notes:

- Sulphuric acid is used as the primary leaching and dissolution reagent within the vanadium electrolyte production process;
- Oxalic acid is used as the reducing agent for conversion of vanadium to the required electrolyte oxidation state;
- Process consumptions were developed based on the preliminary mass balance and major reaction stoichiometry prepared for the study; and
- Only the major process reactions and primary consumables were considered when estimating reagent consumption and operating costs.

Vanadium Electrolyte Production Related General and Administrative Costs

The vanadium electrolyte production General & Administrative (G&A) costs include items not captured in the other operating cost categories, such as:

- Safety/Medical Clinic Labour/Equipment/Medicines costs assumed $100,000/annum, with 20% allocated to the vanadium electrolyte production circuit;
- Camp Management/Catering/ Housekeeping costs assumed based on $40/person/day for the vanadium electrolyte operational workforce, with 100% allocated to the vanadium electrolyte production circuit;
- Communication Costs assumed at $200,000/annum, with 15% allocated to the vanadium electrolyte production circuit;
- Laboratory cost assumed at $200,000/annum, with 30% allocated to the vanadium electrolyte production circuit;
- Mobile equipment lease. Assumed fixed cost of $1.5M/annum, with 25% allocated to the vanadium electrolyte production circuit; and
- Security requirements for the site and camp. Assumed fixed cost of $500,000/annum, with 10% allocated to the vanadium electrolyte production circuit.

21.3.3 Dry Stack Tailings Costs

The OPEX to load and truck haul dry stack tailings from the process plant to the storage facility, then spread and compact the tailings, is estimated at $1.81/t processed as presented in Table 21.18.

TABLE 21.18 DRY STACK TAILINGS OPERATING COST ESTIMATE		
Item	**Unit Cost ($/t processed)**	**LOM Total Cost ($M)**
Loading	0.26	32.4
Hauling	1.21	154.2
Services/Roads/Waste Storage	0.26	33.3
5% Allowance	0.09	11.0
Total	**1.81**	**230.9**

21.3.4 General and Administrative Costs

General and Administration ("G&A") costs are estimated at $13M annually, as summarized in Table 21.19, and include 67 staff. This equates to an average G&A unit operating cost of $1.33/t process plant feed over the LOM.

TABLE 21.19 GENERAL AND ADMINISTRATION COSTS		
Item	**Number**	**Annual Cost ($M)**
Administration Personnel	67	7.2
Environmental Monitoring	Lump sum	0.5
Community Relations	Lump sum	0.4
Office Supplies, Communication	Lump sum	0.4
Insurance	Lump sum	0.8
Legal, Audits, Outside Services	Lump sum	0.3
Travel	Lump sum	0.2
Health & Safety, Medical	Lump sum	0.4
Surface Support & Maintenance	Lump sum	0.6
Contingency @ 20%		2.2
Total		**13.0**

21.3.5 SOP Freight Costs

It is assumed that SOP production will require railway transportation within a 1,000 km distance from the Viken site to reach markets in Sweden and Europe. An estimated rail transport cost of

$0.10/t, including loading and off-loading, would yield a freight cost of $100/t SOP, which results in an average unit cost of $2.51/t processed over the LOM.

21.4 STATE MINERAL FEE

An annual Swedish state mineral fee in the amount of 0.15% of gross value of concession minerals is due to the landowners within the concession area and 0.05% is due to the State. Over the LOM the total fee to the landowners is estimated at $21.9M and the State fee is estimated at $7.3M for a total of $29.2M.

21.5 CASH COSTS AND ALL-IN SUSTAINING COSTS

Cash costs over the LOM are estimated to average negative US$121/lb U_3O_8 net of by-product credits. All-In Sustaining Costs (“AISC”) over the LOM are estimated to average negative US$118/lb U_3O_8 net of by-product credits.

21.6 SITE MANPOWER

Peak year site manpower is estimated at 355 Company personnel, consisting of 157 open pit mining, 131 process plant and 67 G&A. Maintenance personnel are included in the mining and process plant numbers. The work schedule for hourly personnel is planned at two 12-hour shifts per day for seven days a week.

22.0 ECONOMIC ANALYSIS

22.1 SUMMARY

Cautionary Statement - The reader is advised that the PEA summarized in this Technical Report is preliminary in nature and is intended to provide only an initial, high-level review of the Project potential and design options. The PEA mine plan and economic model include numerous assumptions and the use of Inferred Mineral Resources. Inferred Mineral Resources are considered to be too speculative to have the economic considerations applied to them that would enable them to be classified as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

A discounted cash flow model was prepared using the mine production schedule described in Section 16 and the cost estimates described in Section 21 of this Technical Report. The PEA cash flow model was developed on a pre-tax and after-tax basis. The cash flow model is assumed to commence from the time a production decision is made and does not include time or costs for additional studies after this PEA.

The Viken Project economic evaluation conclusions are summarized in Table 22.1. At base case commodity prices of US$38/kg FeV80, US$9/L vanadium electrolyte, US$85/lb U_3O_8, US$650/t SOP, US$7.71/lb Ni, US$1.45/lb Zn and US$27.22/lb Mo the Project has an estimated US$2.88B after-tax net present value ("NPV") at an 8% discount rate ("NPV8%"), and an after-tax internal rate of return ("IRR") of 46%. The after-tax payback period is estimated to be 2.1 years from the start of production.

TABLE 22.1
ECONOMIC EVALUATION SUMMARY

Item	Pre-Tax	After-Tax
NPV0% ($M)	7,592	6,014
NPV5% ($M)	4,796	3,765
NPV8% ($M)	**3,698**	**2,881**
NPV10% ($M)	3,126	2,421
IRR (%) (base case)	52.6	45.9
Payback period (years) (base case)	1.9	2.1

22.2 BASIC ASSUMPTIONS

A discounted cash flow analysis of the Viken Project was prepared based on technical and cost inputs developed by the Authors.

The discounted cash flow analysis was performed on a stand-alone project basis with annual cash flows discounted. The financial evaluation uses a discount rate of 8%, discounting back to the commencement of construction (Year -2) of the Project. Mining and processing have been planned for 13 years, followed by one year of rehabilitation.

All currency values are expressed in US dollars unless otherwise noted.

Metal/Commodity Price Assumptions

All metal/commodity prices remain constant throughout the LOM of the Project. Base case metal/commodity prices of US$38/kg FeV80, US$9/L vanadium electrolyte, US$85/lb U_3O_8, US$650/t SOP, US$7.71/lb Ni, US$1.45/lb Zn and US$27.22/lb Mo were assumed. The sensitivity of the Project return to variations in the assumed prices was also examined.

Metallurgical Recoveries

The Viken Project's pug roast process with an acid plant metallurgical overall recovery assumptions are:

U_3O_8 = 91%
V_2O_5 = 68%
SOP = 70%
Ni = 32%
Zn = 47%
Mo = 52%

In Year 1 of production it is forecast that the process plant will achieve 75% of throughput capacity, and the FeV and vanadium electrolyte circuits will achieve 25% of metal production capacity. Full production capacity for all circuits is assumed as of the beginning of Year 2.

Capital Costs

Total capital costs during the LOM are estimated to be $1.02B as outlined in the Capital and Operating Cost Section 21. The initial capital costs are incurred over a 24-month construction period and are estimated to be $876M. Sustaining capital costs during the 13 production years and one rehabilitation year are estimated at $158M.

Salvage value and closure costs were not estimated since the Deposit contains a vast Mineral Resource and it is likely that a new future mining area will be established and that the feed will be treated in the existing process plant.

Previous Expenses Provision

An amount of $1.7M was considered as a prior expense pool and these monies were deducted from income in production Year 1 when determining the taxable income.

Income Tax Rate

The Swedish corporate income tax is levied at a rate of 20.6% on the net taxable income. Over the LOM income taxes are estimated at $1.58B.

State Mineral Fee

An annual Swedish state mineral fee in the amount of 0.15% of gross value of concession minerals is due to the landowners within the concession area and 0.05% is due to the State. Over the LOM the total fee to the landowners is estimated at $21.9M and the State fee is estimated at $7.3M for a total of $29.2M.

22.3 CASH FLOW SUMMARY

Using base case metal/commodity prices the Project has an after-tax IRR of 46% and a 2.1-year payback of initial pre-production capital costs. The Project is estimated to realize an after-tax NPV of US$2,881M at a discount rate of 8%.

The estimated annual production and LOM cash flows for the Viken Project are summarized in Table 22.2 and annual cash flows are presented in Figure 22.1. An annual summary of the cash flow model is provided in Table 22.3.

TABLE 22.2
PROJECT CASH FLOW SUMMARY

Parameter	Unit	Value	Parameter	Unit	Value
Process Plant Feed	Mt	127.4	Net Revenue	US$M	14,587
Waste Rock Mined	Mt	23.4	Initial Capital	US$M	876
Strip Ratio	waste:feed	0.18:1	Sustaining Capital	US$M	158
V_2O_5 Grade	ppm	3,714	Mining Costs	$/t Feed	3.50
U_3O_8 Grade	ppm	197	Processing Costs	$/t Feed	37.43
K_2O Grade	%	4.12	G&A Costs	$/t Feed	1.33
Ni Grade	ppm	395	Dry Stack Tailings Costs	$/t Feed	1.81
Zn Grade	ppm	479	SOP Freight	$/t Feed	2.51
Mo Grade	ppm	293	Operating Costs	$/t Feed	46.58
Payable FeV	Mkg	76.2	Operating Cash Cost Net By-Products	US$/lb U_3O_8	(-) $121
Payable V Electrolyte	ML	474.4	All-in Sustaining Cost Net By-Products	US$/lb U_3O_8	(-) $118
Payable U_3O_8	Mlb	41.8	After-Tax NPV (8% discount)	US$M	2,881
Payable SOP	Mt	3.2	Pre-Tax NPV (8% discount)	US$M	3,698
Payable Ni	Mlb	20.9	After-Tax IRR	%	45.9

Table 22.2 Project Cash Flow Summary					
Parameter	**Unit**	**Value**	**Parameter**	**Unit**	**Value**
Payable Zn	Mlb	27.0	Pre-Tax IRR	%	52.6
Payable Mo	Mlb	58.3	After-Tax Payback Period	years	2.1

Figure 22.1 Annual Cash Flow Profile

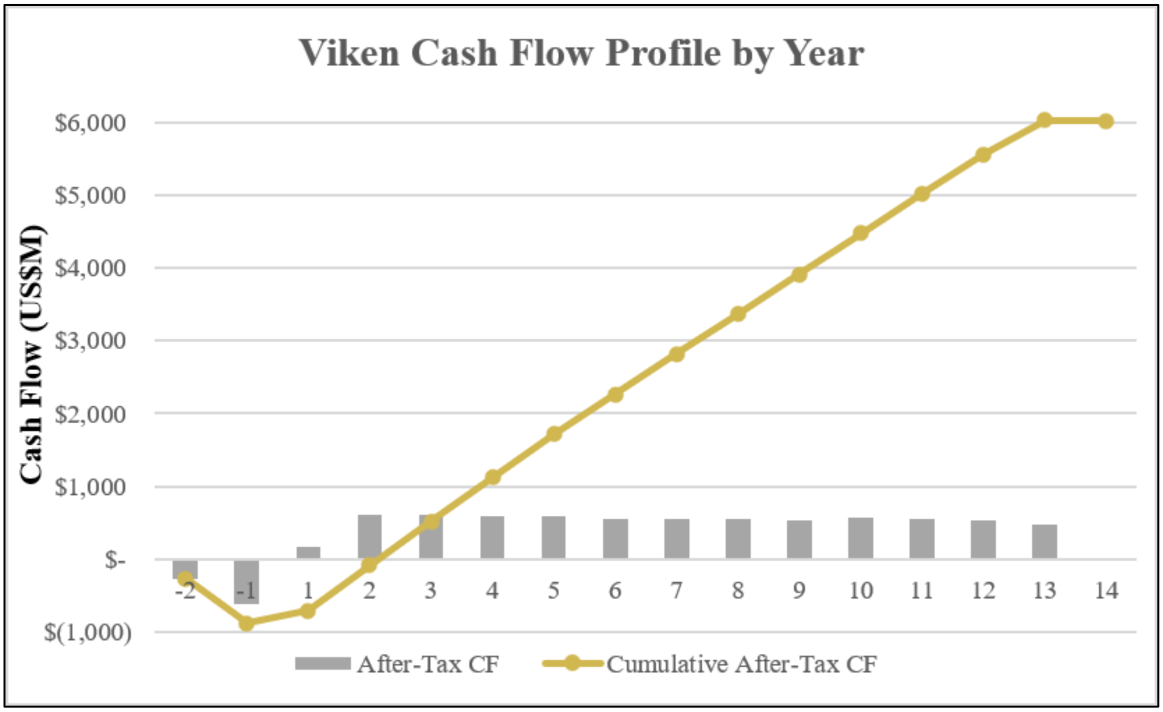


TABLE 22.3
CASH FLOW MODEL SUMMARY

Item	Units	Total	Year															
			-2	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Waste Tonnage	Mt	**23.5**	-	-	3.1	1.8	3.0	1.7	1.6	2.8	3.2	3.5	0.8	0.5	0.5	0.5	0.5	-
Process Plant Feed Tonnage	Mt	**127.4**	-	-	7.5	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	9.9	-
Total Open Pit Tonnage	Mt	**150.8**	-	-	10.6	11.8	13.0	11.7	11.6	12.8	13.2	13.5	10.8	10.5	10.5	10.5	10.4	-
Strip Ratio	w:o	**0.18**	-	-	0.4	0.2	0.3	0.2	0.2	0.3	0.3	0.4	0.1	0.1	0.1	0.1	0.1	-
V_2O_5 Grade	ppm	**3,714**	-	-	4,196	4,069	4,165	3,863	3,837	3,459	3,734	3,693	3,713	3,796	3,551	3,356	2,966	-
U_3O_8 Grade	ppm	**197**	-	-	182	187	179	201	205	214	189	195	185	192	206	209	209	-
Ni Grade	ppm	**395**	-	-	387	380	385	392	404	401	386	393	381	401	406	414	399	-
Zn Grade	ppm	**479**	-	-	469	498	536	484	510	471	473	462	472	466	464	465	458	-
Mo Grade	ppm	**293**	-	-	283	284	275	299	307	307	281	289	273	292	305	306	303	-
K_2O Grade	%	**4.12**	-	-	4.19	4.18	4.20	4.11	4.04	4.10	4.05	4.18	4.16	4.16	4.20	4.11	3.97	-
Payable FeV	Mkg	**76.25**	-	-	1.33	6.90	7.07	6.55	6.51	5.87	6.33	6.26	6.30	6.44	6.02	5.69	4.96	-
Payable V Electrolyte	ML	**474.46**	-	-	8.30	42.95	43.97	40.77	40.51	36.51	39.42	38.98	39.20	40.07	37.49	35.43	30.87	-
Payable U_3O_8	Mlb	**41.80**	-	-	2.28	3.13	2.98	3.36	3.43	3.57	3.15	3.25	3.08	3.20	3.43	3.48	3.43	-
Payable Ni	Mlb	**20.93**	-	-	1.21	1.58	1.60	1.63	1.68	1.67	1.61	1.64	1.59	1.67	1.69	1.72	1.64	-
Payable Zn	Mlb	**27.00**	-	-	1.56	2.20	2.37	2.14	2.26	2.08	2.09	2.04	2.09	2.06	2.05	2.06	2.00	-
Payable Mo	Mlb	**58.31**	-	-	3.31	4.43	4.31	4.67	4.80	4.80	4.39	4.52	4.27	4.56	4.77	4.79	4.67	-
Payable SOP	Mt	**3.20**	-	-	0.19	0.25	0.26	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.26	0.25	0.24	-
Revenue	**$M**	**14,587.5**	**-**	**-**	**545.5**	**1,216.4**	**1,216.9**	**1,207.1**	**1,210.0**	**1,164.1**	**1,158.8**	**1,169.2**	**1,150.4**	**1,182.55**	**1,170.42**	**1,140.41**	**1,055.74**	**-**
(-) Operating Cost	$M	**(5,932.4)**	-	-	(354.3)	(461.3)	(465.9)	(463.9)	(464.8)	(466.9)	(467.0)	(468.3)	(463.4)	(464.5)	(466.1)	(466.2)	(459.8)	-
(-) State Mineral Fee	$M	**(29.2)**			(1.1)	(2.4)	(2.4)	(2.4)	(2.4)	(2.3)	(2.3)	(2.3)	(2.3)	(2.4)	(2.3)	(2.3)	(2.1)	-
(-) Capital Spending	$M	**(1,024.2)**	(261.8)	(614.5)	(17.8)	(10.6)	(7.7)	(10.8)	(11.2)	(17.3)	(8.2)	(13.5)	(9.2)	(12.3)	(16.5)	(7.7)	(5.2)	-

TABLE 22.3
CASH FLOW MODEL SUMMARY

Item	Units	Total	Year															
			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
(-) Rehabilitation	$M	**(10.0)**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(10.0)
Pre-Tax Cash Flow	**$M**	**7,591.7**	**(261.8)**	**(614.5)**	**172.3**	**742.0**	**740.9**	**730.0**	**731.6**	**677.6**	**681.2**	**685.1**	**675.6**	**703.4**	**685.5**	**664.2**	**588.6**	**(10.0)**
(-) Income Tax	$M	**(1,577.5)**	-	-	(1.8)	(124.6)	(129.5)	(132.3)	(136.2)	(128.9)	(130.2)	(133.8)	(132.5)	(140.0)	(137.9)	(132.6)	(117.4)	-
After-Tax Cash Flow	**$M**	**6,014.2**	**(261.8)**	**(614.5)**	**170.6**	**617.4**	**611.4**	**597.8**	**595.4**	**548.7**	**551.0**	**551.3**	**543.1**	**563.4**	**547.6**	**531.6**	**471.2**	**(10.0)**
Cumulative After-Tax Cash Flow	$M	**-**	(261.8)	(876.2)	(705.7)	(88.3)	523.1	1,120.9	1,716.3	2,265.0	2,816.0	3,367.3	3,910.4	4,473.8	5,021.4	5,553.0	6,024.2	6,014.2
Disc After-Tax Cash Flow (8%)	**$M**	**2,881.1**	**(251.9)**	**(547.5)**	**140.7**	**471.6**	**432.5**	**391.5**	**361.0**	**308.1**	**286.5**	**265.4**	**242.0**	**232.5**	**209.3**	**188.1**	**154.4**	**(3.0)**
Disc Cumulative After-Tax Cash Flow	$M	**-**	(251.9)	(799.4)	(658.6)	(187.1)	245.4	636.9	997.9	1,306.0	1,592.4	1,857.8	2,099.9	2,332.4	2,541.6	2,729.7	2,884.1	2,881.1

22.4 SENSITIVITIES

The Viken Project economics were examined with a sensitivity analysis for several key variables. The results of the sensitivity analyses on the after-tax NPV with an 8% discount rate are shown in Tables 22.4 and 22.5, and in Figure 22.2.

TABLE 22.4
METAL PRICE SENSITIVITY NPV, IRR AND PAYBACK

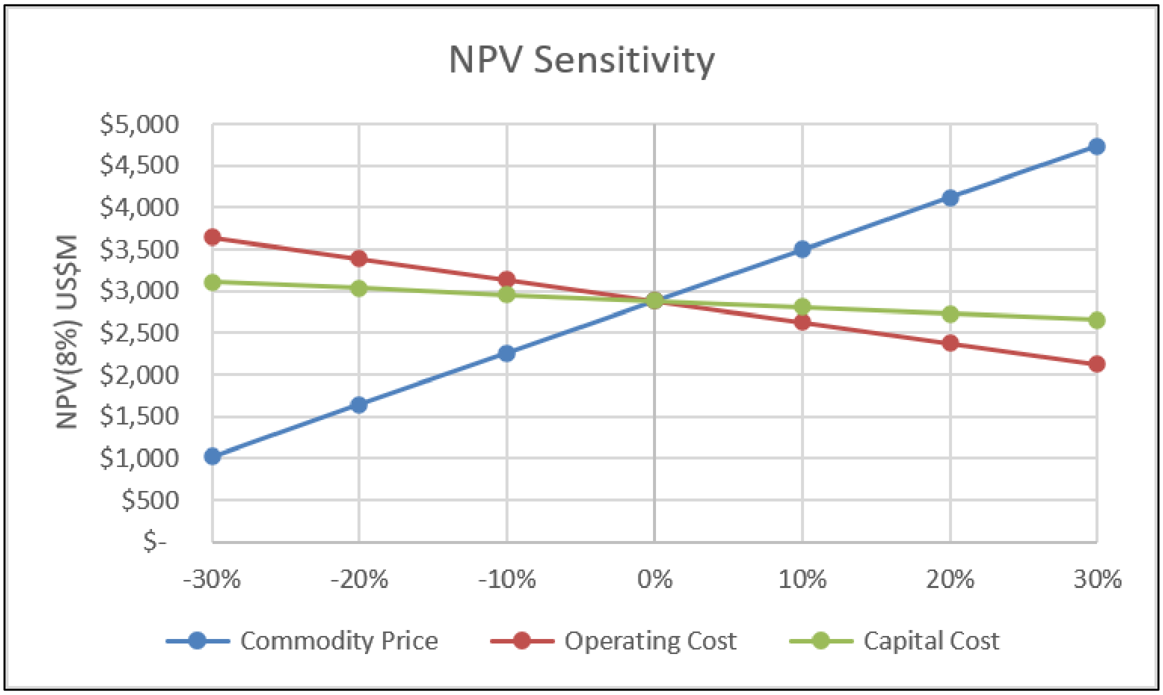
Sensitivity	-30%	-20%	-10%	Base Case	+10%	+20%	+30%
FeV Price (US$/kg)	27	30	34	**38**	42	46	49
V Electrolyte Price (US$/L)	6	7	8	**9**	10	11	12
U_3O_8 Price (US$/lb)	60	68	77	**85**	94	102	111
SOP Price (US$/t)	455	520	585	**650**	715	780	845
After-Tax NPV8% (US$M)	1,023	1,642	2,262	**2,881**	3,498	4,115	4,732
After-Tax IRR (%)	24.3	32.2	39.3	**45.9**	52.0	57.8	63.3
After-Tax Payback (years)	3.7	2.9	2.5	**2.1**	1.9	1.8	1.6

TABLE 22.5
CAPITAL AND OPERATING COST SENSITIVITY OF NPV AND IRR

Sensitivity	-30%	-20%	-10%	Base Case	+10%	+20%	+30%
Operating Costs – NPV8% (US$M)	3,639	3,386	3,134	**2,881**	2,627	2,373	2,118
Operating Costs – IRR (%)	53.8	51.3	48.6	**45.9**	43.1	40.2	37.3
Capital Costs – NPV8% (US$M)	3,108	3,033	2,957	**2,881**	2,805	2,728	2,651
Capital Costs – IRR (%)	60.0	54.4	49.8	**45.9**	42.6	39.7	37.2

The after-tax base case NPV's and IRR's are most sensitive to metal/commodity prices followed by operating costs and capital costs.

FIGURE 22.2 NPV SENSITIVITY GRAPH



The FeV:V Electrolyte ratio base case has been set at 35:65 to reflect estimated market demand over the long term. If the amount of vanadium electrolyte was to be reduced, the Project economics could potentially adjust as per the sensitivity analysis set out in Table 22.6.

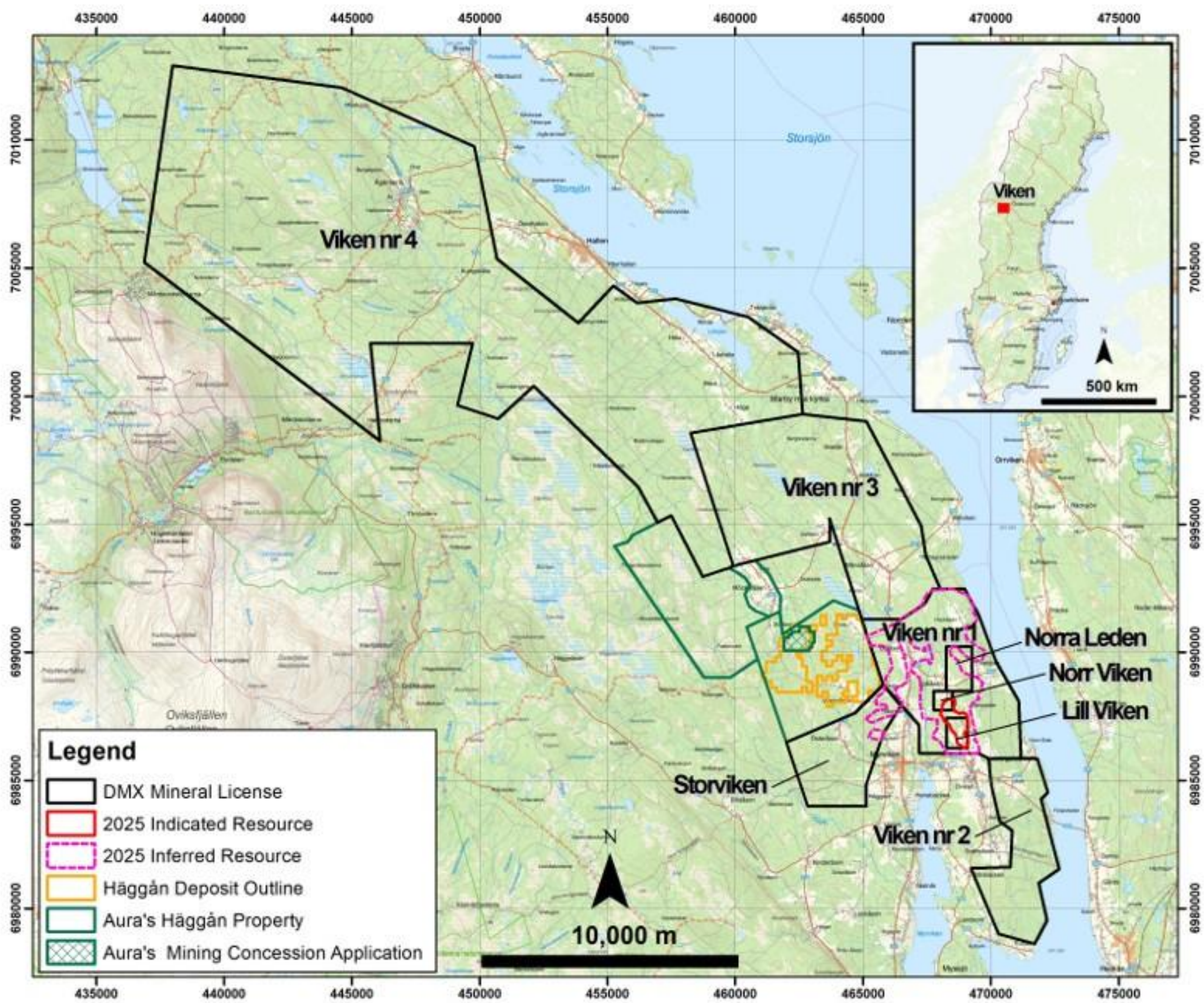
TABLE 22.6 VANADIUM PRODUCT SENSITIVITY			
Sensitivity Item	**FeV:V Electrolyte**		
	35:65 Base Case	**80:20**	**90:10**
FeV (LOM Mkg)	**76.2**	93.8	105.6
V Electrolyte (LOM ML)	**474.4**	271.1	135.6
V Electrolyte (Average ML/year)	**38.8**	22.2	11.1
After-Tax NPV8% (US$M)	**2,881**	2,390	2,062
After-Tax IRR (%)	**45.9**	41.0	37.4
After-Tax Payback (years)	**2.1**	2.4	2.5

23.0 ADJACENT PROPERTIES

Aura Energy Limited (ASX: AEE, AIM: AURA) has a comparable, uranium polymetallic deposit, named Häggån, immediately west adjacent to the Viken Property (Aura, 2023) (Figure 23.1). Häggån is a preliminary level project.

The Author has been unable to verify the information regarding the Häggån project and such information is not necessarily indicative of the mineralization on the Property.

FIGURE 23.1 HÄGGÅN PROJECT ADJACENT TO VIKEN



Source: *District Metals (2026)*

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROCESS OPTION SELECTION

This section outlines the processing options evaluated for recovery of vanadium, uranium, sulphate of potash (“SOP”) and mixed metal precipitate (“MMP”) as part of the Viken PEA. It is intended to describe and compare the main flowsheet configurations and key unit operations at a conceptual level to support economic assessment, and does not provide detailed engineering design, equipment specifications or final design criteria, which will be developed in subsequent study phases. OPEX and CAPEX are compared at a process plant throughput of 4.0 Mtpa to determine if one option is more economic than the others.

Based on prior testwork completed, and prior jobs completed by METS, four process options were identified for process and financial modelling. The four options are summarized in Table 24.1.

TABLE 24.1
OVERVIEW OF PROCESS OPTIONS

Option	Main Leach Method	Flotation ?	HPGR ?	Relative CAPEX	Relative OPEX	Key Advantages	Key Risks / Uncertainties
1	Pug roast + water leach	Yes	No	High	High	High recovery, liberal flotation	High acid/LNG usage, roasting complexity, emissions
2	Pressure acid leach	Yes	No	High	Medium	High recovery, well-established	Autoclave complexity, high free acid requirement
3	Agitated tank leach	Yes	No	Medium	Medium –High	Lower CAPEX, simpler leach	Lower recovery, higher acid in solution
4	HPGR + agitated leach	No	Yes	Lowest	Lowest (target)	Simple front end, lower energy	Leach performance vs testwork not yet demonstrated

The beneficiation and purification techniques common to all options are:

- Crushing
- Grinding
- Flotation
- Circuit for one of Options 1 to 4
- SOP Crystallization
- Ion Exchange
- Solvent Extraction
- Uranium and Vanadium Precipitation

- Mixed Metals Precipitation
- Residue Handling.

Brief descriptions of the four process options are presented below.

24.1.1 Option 1: Pug Roast

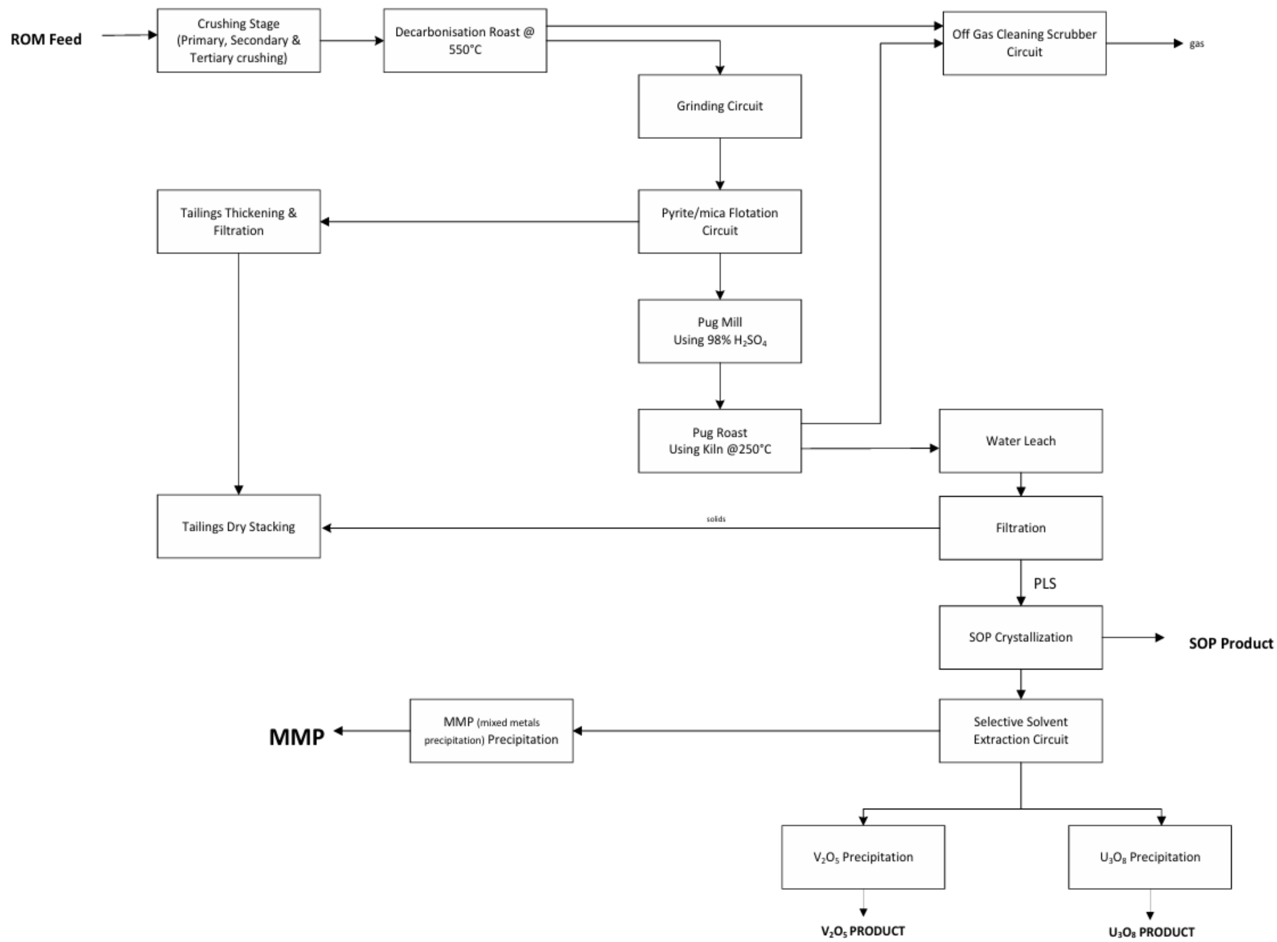
For a pug roast leach, mineralized feed is crushed, grinded and floated before being mixed with concentrated sulphuric acid in a pug mill and thermally roasted to convert vanadium, uranium, potassium and associated metals into water soluble sulphate species. This configuration targets high leach recoveries and allows a more liberal flotation regime, which can enhance recovery of by products such as mixed metal precipitate ("MMP") and sulphate of potash ("SOP").

24.1.1.1 Pug Roast Circuit

The thickened/filtered flotation concentrate is then mixed with concentrated sulphuric acid (98%) in a pug mill (intensive mixer) to produce a uniformly acidulated feed (similar to a clay mix dough). The material is then fed to a rotary kiln or multiple-hearth furnace for controlled sulphation roasting. Sulphation roasting occurs at a temperature range of 220 to 250°C converting vanadium potash, other metals and uranium into water-soluble sulphate compounds, allowing subsequent extraction via simple water leaching.

The pug roast circuit details can be found in Section 17 of this PEA. Figure 24.1 shows the block flow diagram for the process.

FIGURE 24.1 BLOCK FLOW DIAGRAM PUG ROAST



Source: *METS (2026)*

Tables 24.2 and 24.3 present the CAPEX and OPEX estimates for the pug roast option with an acid plant at 4 Mtpa.

TABLE 24.2
PUG ROAST PROCESS PLANT CAPEX AT 4 MTPA

Capital Costs (US$M)	Equipment	Directs	Indirects	Total
4 Mtpa Pug Roast with Acid Plant	133.2	106.5	153.7	**393.4**

Table 24.3 4 Mtpa Pug Roast Process Plant OPEX		
Item	**\$M/year**	**\$/t of ROM**
Reagents	104.1	26.02
Labour	15.4	1.51
Consumables	6.7	1.66
Power	23.2	5.80
Maintenance	6.7	1.66
G & A	3.9	0.98
Water Costs	1.6	0.39
Total	**161.5**	**38.04**

24.1.2 Option 2: Acid Pressure Leach

In the pressure acid leach ("PAL") flowsheet, the process feed is crushed, grinded and floated. Flotation concentrate is treated in an autoclave at elevated temperature and pressure using sulphuric acid and oxygen to oxidize and dissolve the target metals. The exothermic leach reactions largely supply the required heat, limiting external energy demand. PAL generally offers high recoveries and robust performance, however, this requires high free-acid conditions and careful removal of acid-consuming gangue ahead of leach, increasing front-end complexity and capital intensity.

24.1.2.1 Acid Pressure Leach Circuit

The flotation concentrate produced is then pressure leached at an elevated temperature (180 to 250°C) and pressure in an autoclave with oxygen (O_2) as the oxidant and concentrated sulphuric acid (H_2SO_4) as the primary leaching agent. H_2SO_4 provides the acidic environment required for O_2 to oxidize V(III)/V(IV) to soluble V(V) species (VO_2^+/vanadate) and U(IV) to soluble U(VI) (UO_2^{2+}).

Key Oxidation Reactions

Vanadium oxidation:

$$V^{3+}/V^{4+} + O_2 \rightarrow V^{5+}$$

Uranium oxidation:

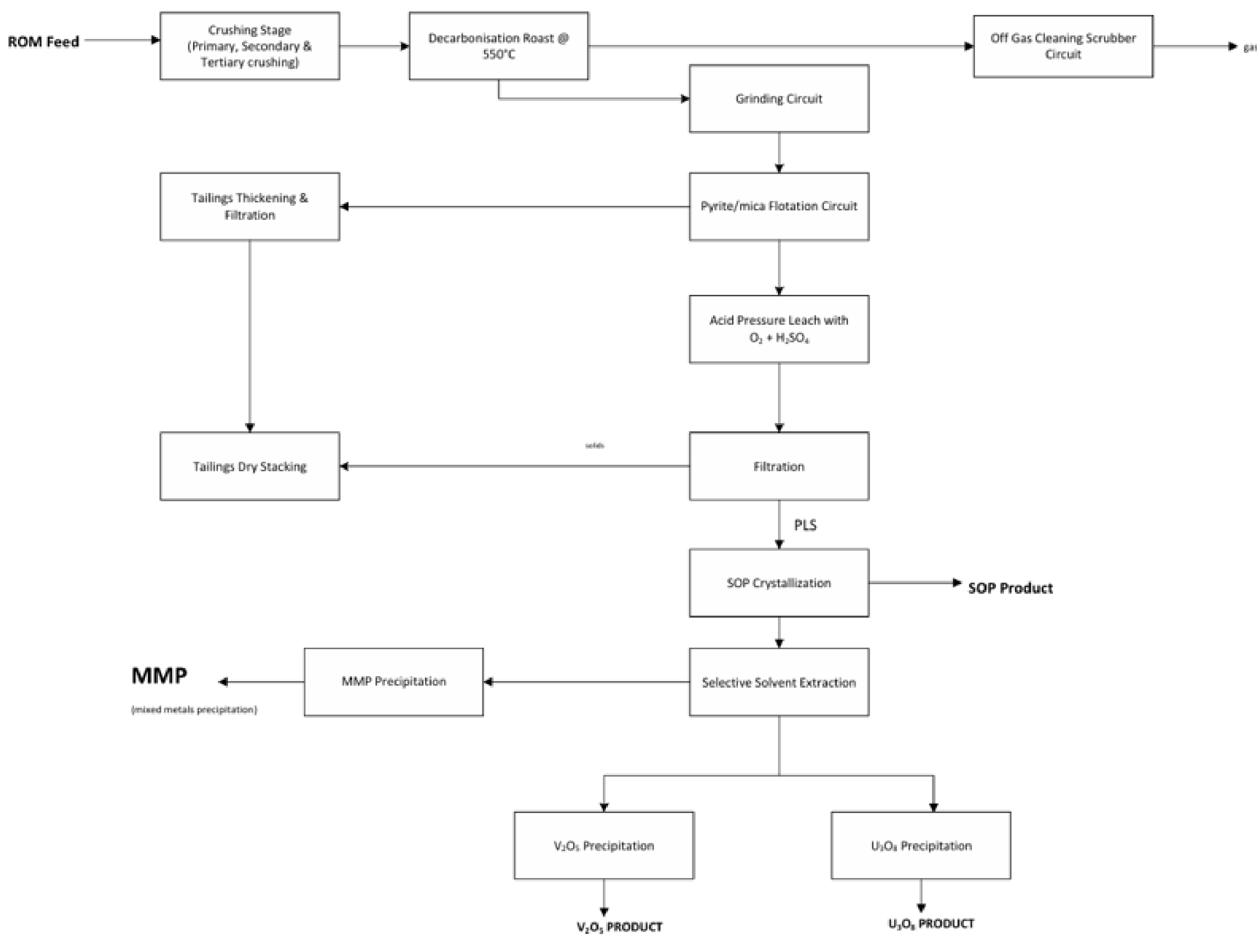
$$UO_2 + \frac{1}{2}O_2 + 2H^+ \rightarrow UO_2^{2+} + H_2O$$

The leaching oxidizes sulphides and organic matter, liberating V and U into solution. Following pressure leaching, the slurry will be flashed to atmospheric pressure, with the hot steam recycled back to heat the POX feed.

The cooled solution is directed to solid–liquid separation using pressure filtration. The residue is washed, dried and dry stacked. Pregnant leach solution ("PLS") is passed to a potassium salt crystallizer for the recovery of sulphate of potash, K_2SO_4 (SOP).

Ion exchange and solvent extraction are carried out to concentrate and separate the uranium and vanadium ions. The separate solutions containing vanadium and uranium are precipitated, forming V_2O_5 and U_3O_8, respectively. The remain spent PLS is treated to precipitate any remaining metals to form a bulk concentrate. Figure 24.2 shows the block flow diagram for the process.

FIGURE 24.2 BLOCK FLOW DIAGRAM ACID PRESSURE LEACH



Source: *METS (2026)*

Tables 24.4 and 24.5 present the CAPEX and OPEX estimates for the acid pressure leach option with an acid plant at 4 Mtpa.

TABLE 24.4 ACID PRESSURE LEACH PROCESS PLANT CAPEX AT 4 MTPA				
Capital Costs (US$M)	**Equipment**	**Directs**	**Indirects**	**Total**
4 Mtpa Acid Pressure Leach with Acid Plant	173.5	138.8	200.2	**512.5**

TABLE 24.5 4 MTPA ACID PRESSURE LEACH PROCESS PLANT OPEX		
Item	**$M/year**	**$/t of ROM**
Reagents	70.3	17.58
Labour	16.4	1.51
Consumables	8.7	2.17
Power	21.3	5.32
Maintenance	8.7	2.17
G & A	4.0	1.01
Water Costs	1.6	0.39
Total	**131.0**	**30.15**

24.1.3 Option 3: Agitated Tank Leach

In the agitated tank leach flowsheet, the process feed is crushed, grinded and floated. Flotation concentrate is pre-treated in a high concentration sulphuric acid conditioning tank, before subsequent water leach tanks are used to put the vanadium, uranium, potassium and other desirable metals in solution. Offering a lower capital cost than the pressure acid leach, agitated tank leaches possess a lower required capital to begin operations in exchange for lower recoveries.

24.1.3.1 Agitated Tank Leaching Circuit

The flotation concentrate is then thickened increasing the slurry density to 40 to 50% solids before leaching. This will reduce the tank volume requirements. The thickener underflow will flow into a condition tank where sulphuric acid (H_2SO_4) will be added as the primary leaching agent before flowing into a series of agitated tanks. Leaching will occur at a pH range of 0.5 to 1.5, temperatures of 60 to 90°C with air as an oxidant. Uranium and vanadium will dissolve into soluble sulphate complexes.

Uranium Dissolution:

$$UO_2 + H_2SO_4 + \frac{1}{2}O_2 \rightarrow UO_2^{2+} + SO_4^{2-}$$

Uranyl forms soluble sulphate complexes:

$$UO_2^{2+} + 3SO_4^{2-} \rightarrow UO_2(SO_4)_3^{4-}$$

Vanadium Dissolution:

$$V_2O_3 + 3H_2SO_4 + \frac{1}{2}O_2 \rightarrow 2VO_2^{+} + 3SO_4^{2-}$$

Vanadium is maintained in V(V) state for downstream SX efficiency.
Acid-consuming gangue reaction:

$$CaCO_3 + H_2SO_4 \rightarrow CaSO_4 + CO_2 + H_2O$$

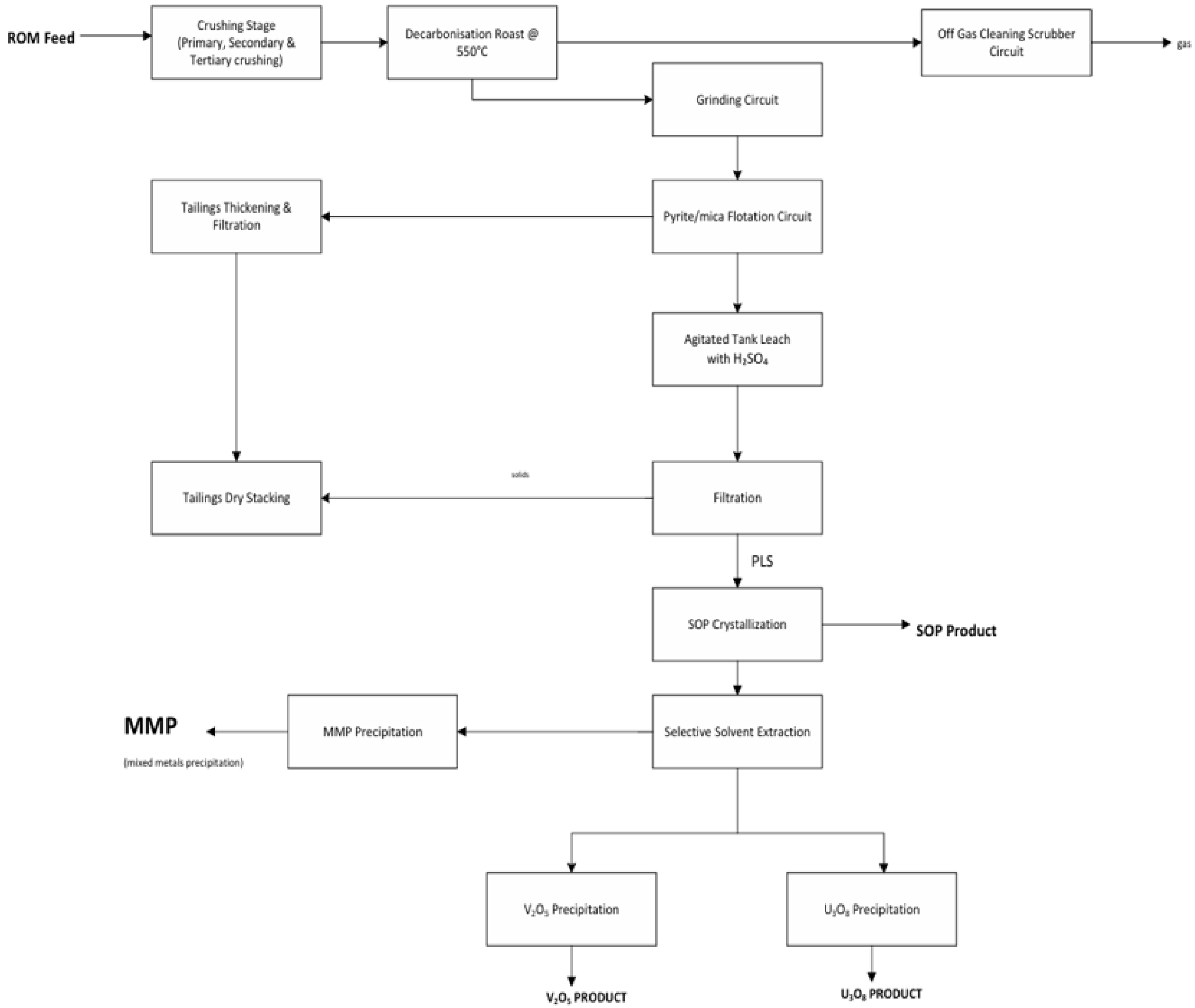
Iron dissolution:

$$Fe_2O_3 + 3H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 3H_2O$$

After leaching the slurry is sent to a counter-current-decantation ("CCD") thickening circuit. The leached slurry then proceeds to solid–liquid separation where it is washed and pressure filtered with the residue being dry stacked and the filtrate, PLS, is passed to a potassium salt crystallizer for the recovery of sulphate of potash, K_2SO_4 (SOP).

Ion exchange and solvent extraction are carried out to concentrate and separate the uranium and vanadium ions. The separate solutions containing vanadium and uranium are precipitated, forming V_2O_5 and U_3O_8, respectively. The remaining spent PLS is treated to precipitate any remaining metals to form a bulk concentrate. A description of the option is shown in Figure 24.3.

FIGURE 24.3 BLOCK FLOW DIAGRAM AGITATED TANK LEACH



Source: *METS (2026)*

Tables 24.6 and 24.7 present the CAPEX and OPEX estimates for the agitated tank leach option with an acid plant at 4 Mtpa.

TABLE 24.6
AGITATED TANK LEACH PROCESS PLANT CAPEX AT 4 MTPA

Capital Costs (US$M)	Equipment	Directs	Indirects	Total
4 Mtpa Agitated Tank Leach with Acid Plant	126.9	101.5	146.4	**374.8**

TABLE 24.7 4 MTPA AGITATED TANK LEACH PROCESS PLANT OPEX		
Item	**$M/year**	**$/t of ROM**
Reagents	59.8	14.95
Labour	16.4	1.51
Consumables	6.3	1.59
Power	21.3	5.33
Maintenance	6.3	1.59
G & A	4.0	1.01
Water Costs	1.6	0.40
Total	**115.8**	**26.37**

24.1.4 Option 4: Direct HPGR Agitated Tank Leach

Option 4 incorporates a high-pressure grinding rolls ("HPGR") before an agitated tank leach circuit. HPGR has a lower energy consumption compared to conventional milling which reduces the OPEX. It also induces internal fractures in minerals, improving acid penetration and leach kinetics in downstream processes. Due to the microfractures produced in HPGR, beneficiation techniques such as flotation are unnecessary. Instead, the process feed is crushed and sent directly to a HPGR circuit. For a project that desires the lowest startup capital, HPGR into a direct agitated leach is typically a strong choice.

24.1.4.1 High Pressure Grinding Rolls

The feed will undergo grinding using HPGR for further size reduction. The micro cracking from HPGR is well documented to enhance leaching. The HPGR product is screened and classified to achieve target size (typically $P_{80} \approx 1{,}000$ µm) for effective leaching in the next stage. This option reduces energy consumption but will need to be demonstrated with testwork, particularly the leaching.

24.1.4.2 Agitated Tank Leaching Circuit

The HPGR product is then mixed with water and thickened increasing the slurry density to 40 to 50% solids before leaching. This will reduce the tank volume requirements. The thickener underflow will flow into a condition tank where sulphuric acid (H_2SO_4) will be added as the primary leaching agent before flowing into a series of agitated tanks. Leaching will occur at a pH range of 0.5 to 1.5, temperatures of 60 to 90°C with air as an oxidant. Uranium and vanadium will dissolve into soluble sulphate complexes.

Uranium Dissolution:

$$UO_2 + H_2SO_4 + \frac{1}{2}O_2 \rightarrow UO_2^{2+} + SO_4^{2-}$$

Uranyl forms soluble sulphate complexes:

$$UO_2^{2+} + 3SO_4^{2-} \rightarrow UO_2(SO_4)_3^{4-}$$

Vanadium Dissolution:

$$V_2O_3 + 3H_2SO_4 + \frac{1}{2}O_2 \rightarrow 2VO_2^{+} + 3SO_4^{2-}$$

Vanadium is maintained in V(V) state for downstream SX efficiency.

Acid-consuming gangue:

$$CaCO_3 + H_2SO_4 \rightarrow CaSO_4 + CO_2 + H_2O$$

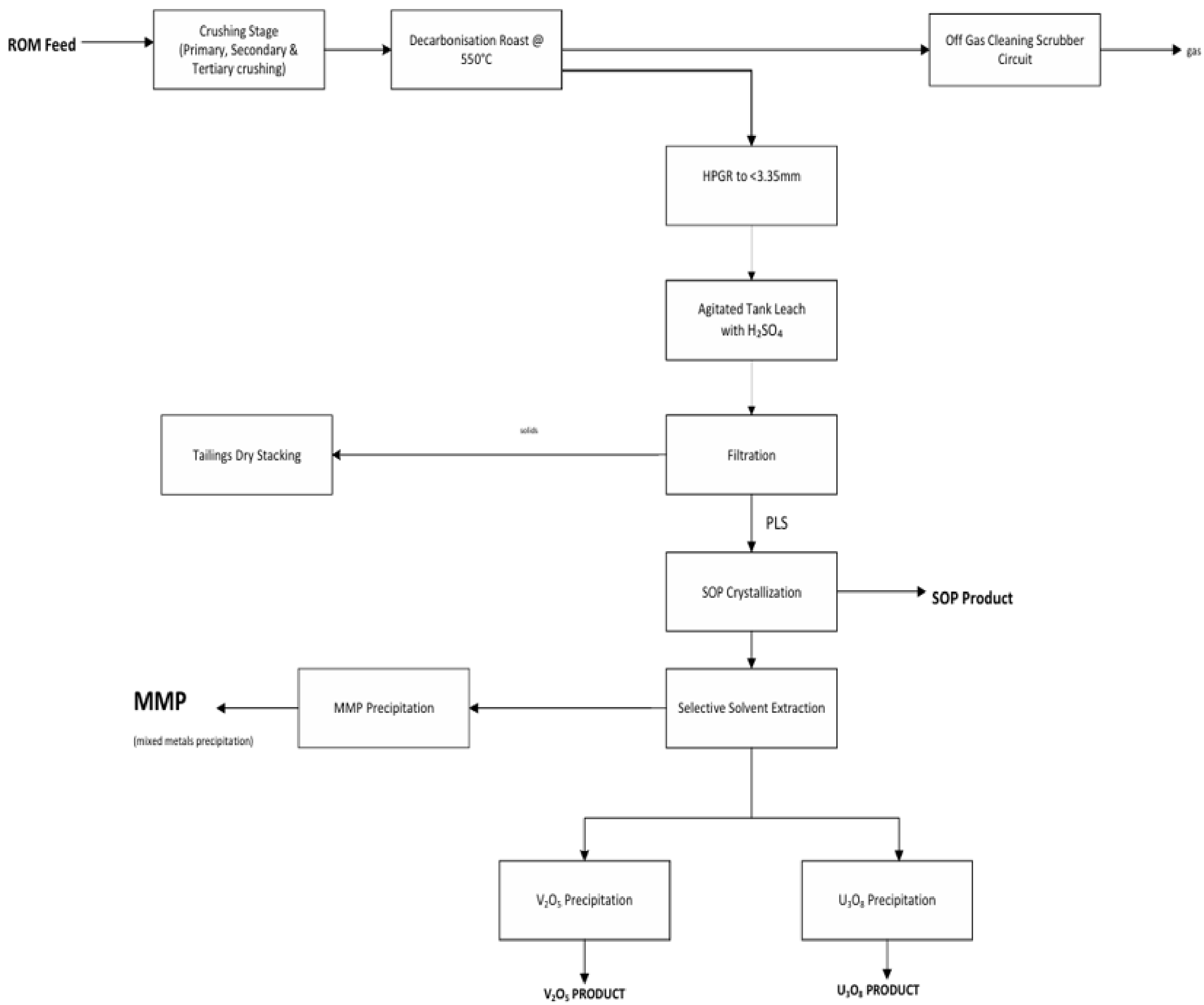
Iron dissolution:

$$Fe_2O_3 + 3H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 3H_2O$$

After leaching the slurry is sent to a CCD thickening circuit. The leached slurry then proceeds to solid–liquid separation where it is washed and pressure filtered with the residue being dry stacked and the filtrate, PLS, is passed to a potassium salt crystallizer for the recovery of sulphate of potash, K_2SO_4 (SOP).

Ion exchange and solvent extraction are carried out to concentrate and separate the uranium and vanadium ions. The separate solutions containing vanadium and uranium are precipitated, forming V_2O_5 and U_3O_8, respectively. The remain spent PLS is treated to precipitate any remaining metals to form a bulk concentrate. A flowsheet of the option is shown in Figure 24.4.

FIGURE 24.4 BLOCK FLOW DIAGRAM DIRECT HPGR AGITATED TANK LEACH



Source: *METS (2026)*

Tables 24.8 and 24.9 present the CAPEX and OPEX estimates for the HPGR agitated tank leach option with an acid plant at 4 Mtpa.

TABLE 24.8
HPGR AGITATED TANK LEACH PROCESS PLANT CAPEX AT 4 MTPA

Capital Costs (US$M)	Equipment	Directs	Indirects	Total
4 Mtpa HPGR Agitated Tank Leach with Acid Plant	114.6	91.6	132.2	**338.4**

TABLE 24.9 4 MTPA HPGR AGITATED TANK LEACH PROCESS PLANT OPEX		
Item	**$M/year**	**$/t of ROM**
Reagents	57.9	14.48
Labour	14.4	1.51
Consumables	5.7	1.43
Power	15.6	3.90
Maintenance	5.7	1.43
G & A	3.8	0.95
Water Costs	1.6	0.40
Total	**104.7**	**24.10**

24.1.5 Financial Analysis and Summary

The four process plant options were analyzed using preliminary estimates for process recovery (refer to Table 13.17) and mass balance spreadsheets developed by METS to determine payable metal at a throughput of 4 Mtpa. CAPEX and OPEX estimates were generated for each option (Table 24.10). An acid production plant was assumed to be built on site for each option since trade-off studies indicated it was less expensive to produce acid rather than purchase it. Site OPEX in Table 24.10 included estimates for mining, processing, G&A and SOP freight. The OPEX margin per tonne of feed was then calculated. A production schedule was generated for each option based on 4 Mtpa processing capacity and average mining grades, and pre-tax NPV8% and IRR were determined. The pug roast option resulted in the highest pre-tax NPV and IRR, and it was selected as the base case option for further study within this PEA.

TABLE 24.10 PROCESS PLANT OPTION ANALYSIS						
Option	**CAPEX (US$M)**	**OPEX (US$M)**	**Site OPEX (US$/t)**	**OPEX Margin (US$/t)**	**Pre-Tax NPV8% (US$M)**	**Pre-Tax IRR (%)**
Pug Roast	393.4	161.5	47.7	29.16	**759.2**	**25.3**
Pressure Leach	512.5	131.0	38.9	30.60	710.5	21.0
Tank Leach	374.8	115.8	35.5	18.33	353.8	16.9
HPGR Leach	338.4	104.7	32.3	16.76	324.7	17.0

24.2 RISKS AND OPPORTUNITIES

The Authors performed a high-level risk assessment to highlight the processing and infrastructure risks associated with the Project that could impact its development and operation. The preliminary risk assessment has been undertaken based on the available technical data and the proposed processing option - Pug Roasting. The primary risks associated with the Project are occupational health and safety, mineralization variability, metallurgical performance, reagent consumption, marketing, infrastructure requirements, environmental compliance, project execution and tailings management. Social and environmental risks were also considered for this section due to the scale of the Project and its potential interaction with surrounding communities and regional resources.

24.2.1 Risk Assessment

Risk is defined in the Risk Management - Guidelines ISO 31000:2018 international standard, as "effect of uncertainty on objectives".

Risk has two characteristics that need to be understood to be managed:

- It has a focus on future events; therefore, it deals in uncertainty.

- It generally focuses on unfavourable events, although the process can be used to identify and manage opportunities.

24.2.1.1 Risk Assessment Methodology

The methodology used for the risk assessment encompass the following activities:

- Risk identification and definition;
- Risk analysis;
- Risk evaluation;
- Risk mitigation strategies;
- Residual risk evaluation; and
- Communicate risks and review of action plans.

At this stage of the Project only risk identification and definition, risk analysis, risk evaluation and risk mitigation strategies have been listed.

24.2.1.2 Scope of Risks

The scope of risks considered in this assessment includes those associated with the proposed processing route, namely the Pug Roast flowsheet incorporating SOP, vanadium, uranium, and mixed metal sulphide recovery circuits, together with risks related to supporting infrastructure and project execution. In addition to process, operational, and engineering risks, the assessment also considers safety, environmental, permitting, regulatory, financial, market, community, and social risks that may influence project development and overall performance.

24.2.1.3 Risks and Mitigation Strategies

Mineral Resource Estimate and Geology

There is a risk that the geological characteristics, mineralization distribution, and grade continuity of the Viken Mineral Resource may vary from current interpretations and estimates as additional drilling, sampling, and Mineral Resource definition activities are completed. The large, mineralized shale sequence contains multiple geological, metallurgical domains and variability in mineral distribution, which can potentially influence metallurgical performance, process recoveries, reagent consumption, product quality, and overall flowsheet performance. Limited availability of large-scale representative samples from certain domains further increases uncertainty associated with process scale-up and recovery assumptions.

If mineralization characteristics are not adequately defined during metallurgical investigations, the selected processing flowsheet may not achieve the targeted operational performance. Variability in mineralogy and behaviour may affect flotation selectivity, pug roasting efficiency, leaching performance, impurity deportment, and overall recoveries. In addition, limited variability testwork across the Deposit introduces uncertainty in comminution energy requirements, process plant throughput, reagent demand, and operating cost estimates, potentially affecting process performance and Project economics.

Risk mitigation should include continued infill and extensional drilling across all major geological and metallurgical domains, expanded variability sampling programs, preparation of additional representative composites. Further metallurgical testwork to improve mineralization characterization and validate recovery assumptions is recommended.

Open Pit Mining

There are currently no geotechnical drilling or studies on the open pit wall slopes. Conservative estimates of inter-ramp angles of 45° for mineralization and waste rock with 30° for overburden have been used. Pit slope geotechnical studies could impact favorably or negatively on the open pit design.

Metallurgical/Processing Risks

The proposed Viken processing flowsheet comprises a complex multi-product circuit involving recovery of uranium (yellowcake), sulphate of potash (SOP), vanadium products (ferrovanadium and electrolyte), and by-products including nickel, zinc, and molybdenum in a mixed metal precipitate. The integration of these circuits creates technical risk due to the interdependence between individual recovery stages, where performance variations in one circuit may influence downstream recovery, product quality, reagent consumption, and overall process plant performance. The current metallurgical database remains limited and does not fully support all aspects of the proposed process configuration, particularly as the selected process route has not yet been comprehensively validated through dedicated laboratory, pilot-scale, and integrated flowsheet testwork. Existing metallurgical information is largely derived from historical programs, introducing uncertainty regarding process performance across the full range of mineralization types and domains expected to feed the processing facilities.

There is therefore a risk that metallurgical recoveries achieved during operation may be lower than those assumed in the PEA due to limited variability testing, reliance on historical data, and incomplete validation of the selected flowsheet. Failure to achieve projected process recoveries could materially affect Project economics through reduced production of uranium, vanadium, SOP, nickel, zinc, and molybdenum products while simultaneously increasing reagent, acid, power, and operating costs. Although benchmarking against comparable vanadium projects indicates that recovery assumptions remain relatively conservative, further laboratory investigations, variability programs, and pilot-scale campaigns are required to validate recoveries, optimize process conditions, and reduce technical uncertainty prior to future study stages.

Several process-specific risks remain associated with the proposed flowsheet and require further validation and optimization:

- Flotation performance risk: Variability in mineralogy, liberation characteristics, and reagent selection may reduce flotation selectivity, resulting in lower concentrate grades, reduced recoveries, and increased reagent consumption;

- Solvent extraction risk: Recovery of uranium and vanadium through solvent extraction may be affected by organic reagent compatibility, extraction conditions, impurity loading, phase separation behavior, and circuit stability, potentially reducing overall metal recoveries and product quality;

- SOP product quality risk: The ability to consistently produce SOP meeting target purity specifications remains uncertain due to the limited quantity of product samples currently available for characterization, impurity assessment, and market evaluation;

- Product specification and marketability risk: Product quality risks extend beyond SOP and include vanadium products (electrolyte and ferrovanadium), uranium concentrate, and mixed metal products. Failure to achieve required specifications may affect market acceptance, product pricing, and overall revenue generation;

- Pug roast acid consumption risk: Acid consumption within the pug roast circuit may be higher than current estimates, as assumptions adopted in the study were primarily based on reaction stoichiometry rather than large-scale metallurgical or pilot-scale testwork;

- Mixed metal precipitate recovery risk: Recovery and precipitation efficiencies for nickel, zinc, and molybdenum from the mixed metal sulphide stream may be lower than expected if precipitation conditions, reagent selection, impurity control, and operating parameters are not adequately optimized;

- Process integration risk: Interactions between flotation, roasting, leaching, solvent extraction, precipitation, and product finishing circuits may influence overall process plant performance, requiring further integrated testwork to validate flowsheet operability and recovery assumptions; and

- Scale-up risk: Limited pilot-scale validation increases uncertainty regarding reagent consumption, residence times, circuit stability, product quality, and recoveries under continuous operating conditions.

Risk mitigation should include expanded laboratory-scale and integrated pilot-scale metallurgical programs covering all major mineralized domains and process routes, together with variability testing across representative composites. Dedicated testwork should also be completed to validate acid consumption, reagent demand, comminution parameters, and power requirements.

Tailings Management Risks

Tailings management represents a significant environmental and operational risk for the Viken Project due to the substantial volumes of residue expected to be generated from processing the Alum Shale. The current Project concept proposes the use of dry stacked tailings to maximize water recovery, minimize water consumption, and reduce the environmental and geotechnical risks typically associated with conventional slurry tailings storage facilities. While dry stacking provides advantages in terms of stability and water management, successful implementation will depend on the physical, geochemical, and mineralogical characteristics of the tailings material.

Alum Shale contains elevated levels of organic carbon and fine-grained carbonaceous material associated with the shale matrix. Residual carbon remaining in the process tailings may influence tailings dewatering performance, filtration efficiency, moisture retention, and long-term storage behaviour. The Alum Shale host sequence may contain sulphides, trace metals, uranium-bearing minerals, and other potentially reactive constituents that require detailed characterization to understand long-term environmental behaviour, leachability, and closure requirements. Failure to adequately characterize and manage tailings storage could result in environmental impacts, increased water management requirements, regulatory constraints, higher closure liabilities, and potential reputational impacts for the Project.

Risk mitigation should include comprehensive tailings characterization programs covering, but not limited to, tailings mineralogy, residual carbon content, radionuclide deportment, leach characteristics, static and kinetic geochemical behaviour, dewatering performance, settling characteristics, thickening response, filtration efficiency, and dry-stack placement requirements. Additional geochemical, environmental, and variability testing should also be undertaken to assess long-term storage behaviour, environmental performance, water quality impacts, and closure requirements. Development of an integrated tailings and water management strategy, including water recovery, seepage control, monitoring programs, and closure planning, will be required to support future stages of Project development.

Market Risks

The economic evaluation for the Viken Project is based on commodity price, product specification, and market assumptions that are considered reasonable at the time of the PEA and reflect current market conditions and projected long-term growth trends. However, there is a risk that these assumptions may change over the life of the Project due to market shifts arising from geopolitical developments, inflationary pressures, changes in global supply and demand balances, energy transition policies, trade restrictions, technological developments, and broader macroeconomic conditions. Variations in these factors may significantly influence future commodity pricing, market demand, and Project economics.

Diversification across multiple commodities provides protection against adverse market conditions by reducing reliance on a single revenue stream, however, producing multiple commodities results in commercial complexity and price volatility risk. Each commodity operates under different demand cycles, market drivers, customer bases, and pricing mechanisms, creating uncertainty in revenue forecasting and long-term cash flow generation. Project economics are also sensitive to final product quality and purity. Revenue assumptions depend on achieving acceptable

SOP grade and purity, vanadium electrolyte quality, ferrovanadium specifications, uranium product quality, and marketable nickel, zinc, and molybdenum by-products. Variations in product grade, impurity levels, recoveries, or final product quality may affect marketability, pricing premiums, and customer acceptance. Lower recoveries combined with increased operating costs, particularly for reagents, power, and acid consumption, could materially reduce Project profitability and financial returns.

Risk mitigation should include ongoing market studies and periodic updates to commodity price forecasts throughout future study phases, together with sensitivity analyses covering price, recovery, product specification, and operating cost variations. Long-term offtake agreements, staged development strategies, and progressive product qualification programs should be considered to further reduce commercial risk.

Financial Analysis

The financial evaluation completed for the PEA is subject to several uncertainties associated with the early stage of Project development. The processing capital and operating cost estimates prepared for the study have been developed to a preliminary level with an estimated accuracy of ±50%, which is consistent with the level of confidence typically applied to a PEA. As a result, there is a risk that actual Project costs, operating expenditures, infrastructure requirements, production rates, and overall Project economics may differ from the values adopted in the study as additional engineering, metallurgical work, and Project definition activities are completed. Additionally, the Project is exposed to exchange rate fluctuations, inflation, changes in labour and construction costs, commodity market conditions, and broader macroeconomic factors. Variations in exchange rates between SEK and US$ may affect capital costs, operating costs, Project cash flows, and economic returns.

Risk mitigation should include periodic updating of capital and operating cost estimates as engineering and metallurgical confidence improves, together with refinement of cost accuracy ranges during future study phases. Financial models should also be updated using current exchange rates, inflation assumptions, and market conditions, with sensitivity analyses undertaken for foreign exchange movements, commodity prices, operating costs, recoveries, and capital expenditure. Scenario analysis and staged development options may further reduce financial exposure and improve flexibility as the Project advances.

Infrastructure and Execution Risks

The Viken Project is exposed to infrastructure and execution risks associated with its location, project scale, and the complexity of the proposed multi-product processing operation. Northern climate conditions, including seasonal temperature variations, snow cover, freeze–thaw cycles, and winter operating constraints, may affect construction scheduling, earthworks, civil activities, water management systems, logistics, and year-round operations. These conditions may also reduce construction productivity and increase project execution costs.

The proposed development involves large-scale processing facilities incorporating multiple recovery circuits. The integration of these product streams increases engineering complexity, capital intensity, commissioning requirements, and start-up risk, particularly where process interactions may influence overall process plant performance. In addition, the Project may be exposed to supply chain constraints for specialized processing equipment, long lead items, reagent supply, filtration systems, roasting equipment, and solvent extraction components, potentially affecting procurement schedules and project timelines.

Risk mitigation should include staged engineering development, early identification and procurement of long-lead equipment, construction sequencing aligned with seasonal conditions, and development of robust logistics and execution plans. Additional pilot-scale testing and phased commissioning strategies should also be considered to reduce process integration risks and improve process plant start-up performance. Contingency allowances for schedule, procurement, and capital costs should be incorporated to manage uncertainties associated with large-scale project delivery.

Community, Local and Social Risks

The Project may face local community and political opposition associated with water use, environmental footprint, radiological concerns, land disturbance, visual impacts, and uncertainty regarding economic benefits to local communities and the broader region. Public perception of uranium processing and management of Alum Shale residues may increase stakeholder scrutiny and create challenges in maintaining the social licence to operate.

Although Sweden supports mining and has a growing focus on critical minerals required for the energy transition, including vanadium, nickel, and battery materials, community acceptance remains a key development factor. Concerns regarding environmental performance, employment opportunities, and long-term regional benefits may influence Project support.

Risk mitigation should include early and continuous stakeholder engagement programs, transparent communication of environmental performance and Project benefits, community consultation processes, and development of local employment and procurement initiatives. Clear demonstration of regional economic benefits, alignment with Sweden’s critical minerals strategy, and implementation of robust environmental management practices will be important to strengthen social acceptance and reduce community-related risks.

Permitting and Regulatory

Sweden is a mining jurisdiction with a well-established and stringent environmental regulatory framework and a long history of mining operations. Mining projects are subject to comprehensive permitting and environmental assessment requirements under the Swedish Environmental Code, with strong emphasis placed on water management, biodiversity protection, land rehabilitation, waste management, and long-term environmental stewardship. As a result, the Project is exposed to permitting risks including potential approval delays, extended assessment periods, additional technical requirements, and possible Project modifications required to satisfy regulatory expectations. Uranium development in Sweden carries increased regulatory and political sensitivity due to concerns regarding radiological safety, environmental impacts, and long-term waste management. Changes to Swedish mining, environmental, radiation, or critical mineral regulations may also influence Project development schedules, processing requirements and economics.

District Metals does not control any surface rights at Viken. Permissions for access to the license areas and to execute investigative work programs are governed by Bergsstaten, the Swedish Mining Inspectorate (www.sgu.se/bergsstaten).

Risk mitigation should include early engagement with regulators and permitting authorities, phased environmental studies, progressive permitting strategies, and completion of comprehensive Environmental and Social Impact Assessments ("ESIA"). Regulatory risks should be managed through ongoing monitoring of legislative changes and incorporation of conservative design criteria aligned with Swedish and EU standards.

Environmental Risks

Processing of Alum Shale material presents environmental risks associated with the presence of naturally occurring radioactive materials ("NORM"), trace metals, sulphides, residual carbon, and process residues. Potential environmental impacts may include dust emissions, radioactive exposure, process gas emissions from roasting operations, water contamination and elevated acid consumption. Roasting and thermal processing stages may also generate gaseous emissions including sulphur-bearing compounds, carbon dioxide, and fine particulate matter, requiring appropriate emission control systems. Tailings and residue streams may require additional management to control long-term geochemical behaviour, metal leaching, radionuclide stability, and water quality impacts. Failure to adequately manage these aspects could result in environmental liabilities, regulatory penalties, increased closure obligations, and reputational impacts.

Risk mitigation should include adoption of dry-stack tailings, closed-loop water circuits, process gas capture systems, acid recovery where applicable, dust suppression measures, filtration and water treatment systems, radiation monitoring programs, and progressive rehabilitation plans. Additional geochemical, environmental, and tailings testwork should also be completed to support long-term storage and closure strategies.

Occupational Health and Safety

The pug roasting and solvent extraction circuits introduce occupational health and safety risks that must be carefully managed throughout the design and operational phases of the Project. The pug roasting process involves the use of concentrated sulphuric acid for vanadium and uranium leaching, which is a highly corrosive chemical that poses risks to workers and the surrounding environment if not properly managed.

Potential hazards include acid exposure, spills during reagent handling, environmental contamination, burns, exposure to high temperatures (up to 700°C) and the release of hot acidic concentrates. Strict acid management procedures, automated dosing systems, high temperature-rated insulated PPE, operator training programs, and emergency response protocols will be required to minimize these risks. The pug mill must also be designed with corrosion-resistant materials, comprehensive monitoring and robust control systems.

Solvent extraction introduces additional safety considerations due to the use of organic chemicals. Reagent spills and environmental contamination are prevalent safety risks in a solvent extraction circuit if engineering controls are not properly implemented. Mixer-settler systems must therefore be designed with, organic-phase-resistant materials, and monitoring systems – with an emphasis on flowrate and level control systems - to ensure safe operation.

Additional health and safety risks may arise from the handling of other reagents used in flotation, including lime, and flotation reagents. Dust generation during crushing and grinding operations may also present respiratory hazards for workers if adequate dust suppression and ventilation systems are not installed. Implementation of occupational health and safety management plans, providing proper PPE, regular training programs, and continuous monitoring will be essential to ensure safe operation of the processing facility.

24.2.1.4 Opportunities

Mineral Resource Scale and Multi-Commodity Opportunity

The Viken Project represents a significant large-scale multi-commodity Mineral Resource with potential for long mine life and staged development options. The presence of uranium, vanadium, sulphate of potash (SOP), nickel, zinc, molybdenum and other possible by-products provides an opportunity to diversify revenue streams and reduce dependence on a single commodity market. Additional metallurgical work may identify further recoverable metals or saleable products not currently included in this PEA, potentially improving Project economics and increasing overall Mineral Resource value. The scale of the Mineral Resource also supports phased development strategies, allowing progressive expansion of mining and processing capacity while reducing initial capital exposure and improving Project flexibility. Additional exploration drilling may potentially increase the size and quality of the Mineral Resource.

Critical Minerals and Strategic Supply Opportunity

The Project provides exposure to several critical minerals that align with European and Swedish energy transition policies and growing demand for battery materials and strategic metals. Vanadium, nickel and molybdenum are increasingly important for energy storage systems, alloy production and industrial applications, while uranium remains a strategic energy commodity supporting low-carbon power generation.

The Project has the potential to contribute significantly to regional supply security and reduce European dependence on imported critical minerals and fertilizer products. Given Sweden's continued use of nuclear energy generation, domestic uranium production may provide strategic supply benefits and reduce reliance on imported uranium feedstocks, subject to future regulatory approvals and market conditions. The Project may also support Sweden's broader battery and energy transition strategy through integration into downstream processing chains and future battery material supply initiatives.

Process Optimization and Technology Opportunities

The current PEA flowsheet remains at an early stage and presents significant opportunities for optimization. Additional laboratory scale, pilot-scale and integrated metallurgical programs may improve recovery, reduce reagent consumption, simplify processing routes, lower capital costs, lower operating costs and provide the Project with clear additional product recovery pathways. Recovery of additional by-products may further enhance Project value, especially if the process methods to get there require few extra steps. Flowsheet simplification opportunities may include optimization of flotation performance, leaching conditions, leaching efficiencies, solvent extraction circuits, impurity rejection and precipitation stages. Advanced technologies may also provide opportunities for improved environmental and operating performance. For example, integration of modern sulphuric acid production technologies, such as Oxley-type acid plants or equivalent acid regeneration systems, may improve acid supply reliability, increase sulphur recovery efficiency, reduce reagent costs, and lower emissions associated with external acid transport and procurement. Further optimization of heat recovery, waste energy utilization, water recycling systems, and dry-stack tailings technology may also improve overall Project sustainability and reduce operating costs.

The Viken process plant will be able to adjust production of vanadium electrolyte and ferrovanadium depending on market demand and pricing. In practice, most vanadium operations are configured to produce either ferrovanadium or vanadium electrolyte, but rarely both. While the upstream V_2O_5 production circuit is common, the downstream infrastructure, equipment, quality control requirements, product handling systems, and customer qualification processes differ significantly between the two products. Consequently, few operations currently have the installed infrastructure required to support both production routes simultaneously. Viken can potentially be one of the few operations to support both production streams.

Infrastructure, Construction and Development Opportunities

Modular construction strategies present opportunities to reduce site construction effort, improve quality control, shorten schedules, and reduce overall execution risk. Use of modular processing units and staged process plant construction may also allow progressive commissioning and future throughput expansion. Alternative equipment sourcing strategies, including use of lower-cost international suppliers such as Chinese manufacturers where technically appropriate and compliant with Project requirements, may provide opportunities to reduce capital expenditure. Early engagement with equipment suppliers and optimization of procurement strategies may further improve Project economics.

Environmental and Sustainability Opportunities

The Project presents opportunities to incorporate advanced environmental management practices from the early stages of development, including dry-stack tailings, closed water circuits, progressive rehabilitation, heat integration, and emissions reduction technologies. Adoption of these approaches may reduce environmental footprint, improve social acceptance, and support alignment with Swedish and European sustainability objectives.

Financial Analysis

Sensitivity analysis of commodity prices indicates that the Project remains viable at a 30% decrease of all commodity base case prices. The Project also remains viable at a 30% increase in LOM OPEX or CAPEX.

24.3 PROJECT EXECUTION

The Viken Project is being advanced at the PEA level to evaluate potential processing routes for the recovery of vanadium, uranium and potassium products from Alum Shale mineralization. The PEA aims to establish a preliminary technical and economic basis for the development of the Project.

As part of this assessment, multiple processing routes have been evaluated to determine technically viable options for the treatment of the mineralization. A consistent evaluation approach has been applied across all routes, based on high-level technical assumptions and order-of-magnitude cost estimates.

The current phase of work focuses on the comparative assessment of these processing routes, including the development of conceptual process flowsheets, high-level mass balances, and preliminary capital and operating cost estimates. The outcomes of this study provide a basis for comparison and support progression of the Project to subsequent study phases.

24.3.1 Metallurgical Testwork

The metallurgical assessment of the Viken Project forms a critical component in the advancement of the Project from early-stage evaluation towards more detailed engineering studies.

As part of the current PEA assessment, multiple processing routes have been evaluated to understand the potential for recovery of vanadium, uranium, and potassium products from Alum Shale mineralization. This evaluation is based on conceptual process design, supported by available historical data.

The key differentiating factor between the processing routes is the leaching approach adopted, which governs metal recovery, reagent consumption, and overall process performance. While the current study provides a preliminary technical and economic basis for comparison, the existing dataset is not sufficiently developed to support detailed process design or the selection of a single preferred processing route.

To support project advancement, a structured metallurgical testwork program will be required. This program should be designed to generate representative and reliable data applicable across all processing routes and to address key technical uncertainties identified in the current PEA.

The testwork program is expected to include:

- Mineralogical and chemical characterization, including definition of metal deportment, mineral associations, and variability across the Deposit;

- Comminution testwork, to define hardness characteristics, variability, and design parameters for crushing and grinding circuits;

- Flotation testwork, to assess the potential for selective removal of gangue minerals or upgrading of specific fractions where applicable;

- Leaching investigations, representing the primary focus of the program, including evaluation of different leaching conditions, optimization of recovery, kinetics, and reagent consumption across the alternative processing routes;

- Solution processing testwork, including ion exchange and solvent extraction, to validate metal recovery and downstream processing performance; and

- Residue and environmental testwork, including residue stability, water quality, and tailings management considerations.

The outcomes of this program will be critical in refining process design, validating key assumptions, supporting process selection, and reducing technical and economic risk as the Project progresses to subsequent study phases.

24.3.2 Engineering Studies

Engineering studies are undertaken to progressively define the technical and economic viability of the Project. These studies are iterative in nature and increase in level of detail and confidence as the Project advances.

The current PEA forms part of the early-stage evaluation of the Viken Project, focusing on the assessment of alternative processing routes and the development of a preliminary technical and economic basis.

The current PEA represents a preliminary assessment focused on the evaluation and comparison of alternative processing routes to establish a preliminary technical and economic basis for the Project. The study is based on conceptual process design, high-level mass balances, and order-of-magnitude cost estimates, and is intended to determine whether the Project demonstrates sufficient potential to justify further development.

Progression to the Pre-Feasibility Study (“PFS”) stage will require refinement of the selected processing route with increased engineering definition and improved cost accuracy, supported by additional metallurgical testwork to validate key process parameters.

A subsequent Feasibility Study (“FS”) will further develop the Project through detailed engineering design, equipment selection, and comprehensive cost estimation, supporting investment decision-making and reducing technical and operational risks.

After a FS has been completed, the Front-End Engineering Design (“FEED”) stage represents the transition to execution, providing the detailed engineering definition required to support procurement, construction planning, and Project implementation.

24.3.3 Environmental and Social Impact Assessment

The Environmental and Social Impact Assessment (“ESIA”) evaluates the potential environmental and social impacts associated with the development of the Project.

At the current stage, environmental and social considerations are identified at a high level to support early Project planning and risk identification. Key aspects include potential impacts associated with processing activities, reagent use, water management, and waste handling.

As the Project advances, more detailed environmental and social studies will be required, including baseline data collection, impact assessment, and the development of mitigation measures.

The ESIA process will be critical to obtaining regulatory approvals and ensuring the Project maintains an appropriate social licence to operate.

24.3.4 Project Implementation

Project implementation represents the transition from engineering design to execution through the integration of engineering, procurement, construction, and commissioning activities.

This phase includes development of project schedules, procurement strategies, and construction planning, supported by cost control and risk management. Project scheduling defines execution timelines and key milestones, forming the basis for project control. Procurement and contracting involve sourcing of equipment, materials, and services, including supplier selection and contract management.

Construction and implementation involve development of processing facilities and associated infrastructure, including installation of equipment and system integration. Operational readiness and handover ensure a controlled transition to operations, including development of procedures, systems, and workforce preparedness.

The Author is not aware of any other data or information that is relevant to the conclusions presented in this Report.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

The Authors note the following interpretations and conclusions in their respective areas of expertise for this Technical Report.

25.2 LOCATION AND OWNERSHIP

The Viken Property is situated in the province of Jämtland, approximately 520 km northwest of the Capital City of Stockholm, and is easily accessibile via highway E14 from Sundsvall, and then by local highways E45 and 321 that connect to the Property area through Svenstavik at the southern end of Lake Storsjön.

The Viken Property consists of eight contiguous licenses that cover a total area of 29,857 ha and are 100% owned by Bergslagen Metals AB, a wholly-owned Swedish subsidiary of District Metals. The licences are in good standing as of the effective date of this Report. The current Mineral Resource Estimate is covered by the Viken Property licenses Viken nr 1, Norra Leden, Norr Viken and Lill Viken.

The Swedish moratorium on uranium exploration and mining was lifted effective January 1, 2026. On June 15, 2026, the Swedish Parliament voted to abolish a Municipal veto on uranium extraction that goes into legislative effect on July 15, 2026. A key aspect of the legislation is that the extraction and processing of uranium is now treated in a manner consistent with other metals and minerals within Sweden's overall permitting framework.

On June 11, 2026, the Swedish Government presented the terms of reference for a new inquiry that will investigate ways to enhance local and municipal influence for Alum Shale deposits. The inquiry mentions that it will include concerned parties from the private sector in their stakeholder dialogue. It will also provide an analysis of commercial consequences of any proposal made and will take into account Sweden's commitments according to the Critical Raw Materials Act ("CRMA").

The Viken Deposit is currently being reviewed by the Geological Survey of Sweden ("SGU") as a deposit of National Interest (Riksintresse), and the SGU's review is expected to be completed in Q3 2026.

25.3 GEOLOGY AND EXPLORATION

The Viken Deposit is uranium-vanadium polymetallic shale-hosted mineralization contained within the Cambrian Viken Shale, which regionally is referred to as the Alum Shale. The Alum Shale is enriched in vanadium, uranium, potassium, molybdenum, nickel, zinc, and copper. Information and data used for the updated Mineral Resource Estimate were supplied by District Metals, and take into account a total of 152 drill holes on the Viken licence and adjacent licences. Spacing between drill holes varies between approximately 30 and 380 m, randomly distributed across the Property area. Of the 152 drill holes (totalling 28,667 m), 122 within or adjacent to the Viken Licence limits were used for the grade estimation and modelling of the Mineral Resources.

A high-resolution airborne MobileMT Survey was flown over the entire Viken Property in May-June 2025, with the aim to identify potential mineralized Alum Shale for follow-up confirmation by drilling. The relatively flat-lying Alum Shale is rich in graphite and sulphides, making it low resistivity (high conductivity) and easily detectable using the airborne MobileMT system. The MobileMT survey covered ~2,350-line km at 200 m spacing and was completed in two phases. Phase 1 captured the low resistivity signature of the mineralized Alum Shale that hosts the Viken Deposit, and Phase 2 covered the remainder of the Viken Property and identified additional low resistivity signatures resembling those of the Viken Deposit. High-priority target areas were selected by District Metals based on the shallow depth and thickness of the mineralized Alum Shale. The relatively flat-lying Alum Shale is rich in graphite and sulphides, making it low resistivity (high conductivity) and easily detectable using the airborne MobileMT system.

In all, nine target areas, labelled A to I, were identified outside of the Viken Deposit. Three of these target areas, A to C, show conductivity responses that are larger and stronger than those observed over the Viken Deposit itself. With these results, the Company plans to prioritize the targets for drill testing in 2026.

The Authors have recognized that the Viken Deposit contains Targets for Further Exploration with a potential range of 980 Mt to 1,040 Mt at grade ranges of 140 to 180 ppm U_3O_8, 2,170 to 2,740 ppm V_2O_5 and 210 to 260 ppm Mo. These Targets for Further Exploration are based on the estimated strike length, depth and width of the mineralization, as supported by intermittently-spaced drill holes and observations of mineralized outcrops. The Targets for Further Exploration are located adjacent to the margins of the current Mineral Resource.

The potential quantities and grades of the Targets for Further Exploration are conceptual in nature. There has been insufficient work done by a Qualified Person to define these estimates as Mineral Resources. The Company is not treating these estimates as Mineral Resources, and readers should not place undue reliance on these estimates. Even with additional work, there is no certainty that these estimates will be classified as Mineral Resources. In addition, there is no certainty that these estimates will prove to be economically viable.

25.4 SAMPLING, ANALYSES AND VERIFICATION

It is the Author opinion that sample preparation, security and analytical procedures for the Viken Project drill program were adequate, and that the data are of good quality and satisfactory for use in the current Mineral Resource Estimate.

It is the Authors' opinion that the mineralized shale analysis grades can be reasonably reproduced, suggesting that the assay results reported by the primary assay laboratories are sufficiently reliable for the estimation of Mineral Resources.

On completion of data validation procedures, the Authors conclude that the digital database for the Viken Deposit is reliable for Mineral Resource estimation.

25.5 METALLURGICAL TESTING

Extensive metallurgical testwork programs were undertaken between 2007 and 2026 to assess the processing characteristics of mineralized shale from the Viken Deposit and support evaluation of potential processing routes. The testwork investigated mineral behaviour, metallurgical response, recovery performance, and process suitability across a range of development scenarios. As part of the PEA, an independent review of all available historical metallurgical investigations was completed, and the results were used together with comparable technical studies and benchmarking from similar projects to establish process flowsheets, recovery assumptions and design criteria adopted for the study.

The available testwork indicates that the Viken mineralized shale is a complex polymetallic material that is amenable to multiple processing approaches. Historical mineralogical and metallurgical investigations indicate that uranium and molybdenum show relatively favourable response under certain leaching conditions, while vanadium recovery is more technically challenging and is likely to require thermal activation or more aggressive process conditions to achieve commercially meaningful extraction. The mineralization includes potassium, nickel and zinc, providing potential opportunities for by-product recovery and additional value generation. However, the technical and commercial basis for recovery of these secondary products remains less developed and defined than for the primary uranium and vanadium processing pathways. Consequently, an extensive program of confirmatory, variability, and optimization testwork will be required to improve confidence in the proposed flowsheets, validate recovery assumptions, reduce technical uncertainty, and support future process design, engineering development, and cost estimation.

25.6 MINERAL RESOURCE ESTIMATE

The Authors reviewed and audited the exploration data available for the Viken Deposit and are of the opinion that the exploration data are sufficiently reliable to interpret with confidence the boundaries of the mineralization and support evaluation and classification of Mineral Resources in accordance with generally accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines and CIM Definition Standards for Mineral Resources and Mineral Reserves.

The Mineral Resource Estimate contains uranium oxide, vanadium oxide, potassium oxide, molybdenum, nickel, copper and zinc. Additional elements reported (but not contributing to the NSR value) are phosphorous pentoxide, cesium oxide, yttrium oxide, and lanthanum oxide. Many of these Mineral Resource elements are listed by the European Union ("EU") as Critical Raw Materials. Inferred Mineral Resources are estimated at 4.333 billion tonnes averaging 161 ppm U_3O_8, 2,543 ppm V_2O_5, 240 ppm Mo, 321 ppm Ni, 118 ppm Cu, 417 ppm Zn and 3.70% K_2O. Indicated Mineral Resources are estimated at 456 million tonnes averaging 175 ppm U_3O_8, 2,836 ppm V_2O_5, 257 ppm Mo, 330 ppm Ni, 113 ppm Cu, 411 ppm Zn and 3.84% K_2O. Mineral Resources have been reported using an average internal (processing plus G&A) US$22/t NSR cut-off value.

The Mineral Resource Estimate is effective as of June 26, 2026, and takes into account the results from a total of 122 drill holes completed by previous operators between 2006 and 2012 on the

Viken Property. The spacing of the drill holes ranges from 30 to 380 m and averages approximately 300 m. The Alum Shale underlies the entire Viken Property and extends beyond its boundaries.

25.7 MINING METHODS

The Viken Deposit is near surface and lends itself to conventional open pit mining methods. The Company will undertake all surface excavation, loading, hauling, and mine site maintenance activities, along with mine management and technical services. Drilling and blasting will not be required. Mining will typically be accomplished on 10 m high benches, using Surface Excavation Machines capable of cutting 0.3 m into the Alum Shale, supported by 11.5 m^3 wheel loaders loading 90 tonne off-highway haul trucks. Mining dilution and mining losses are assumed to be zero since the potentially mineable blocks have no edge contact with the large Mineral Resource wireframe. Life-of-mine production totals 127.36 Mt of process plant feed and 23.49 Mt of waste rock for a strip ratio of 0.18:1. Average feed grades are 197 ppm U_3O_8, 3,714 ppm V_2O_5, 293 ppm Mo, 395 ppm Ni, 479 ppm Zn, and 4.12% K_2O, with an overall NSR of $118/t. Waste rock will be stored initially in an external waste storage facility and later in the mined-out pit, with 17 Mt placed ex-pit and 6.5 Mt stored in-pit. Mineralized material will be hauled to the primary crusher located to the north of the open pit.

25.8 RECOVERY METHODS

The recovery method selected for the Viken Deposit is a pug roast leach. The process feed is crushed, grinded and floated before being mixed with concentrated sulphuric acid in a pug mill and thermally roasted to convert vanadium, uranium, potassium and associated metals into water soluble sulphate species. This configuration targets high leach recoveries and allows a more liberal flotation regime, which can enhance recovery of by products such as mixed metal precipitate ("MMP") and sulphate of potash ("SOP"). Approximately 35% of the total V_2O_5 production will be directed to vanadium electrolyte production, with the remaining V_2O_5 allocated to FeV production.

The beneficiation and purification techniques are: Crushing; Grinding; Flotation; Pug Roasting; SOP Crystallization; Ion Exchange; Solvent Extraction; Uranium and Vanadium Precipitation; Mixed Metals Precipitation; and Residue Handling. Overall metallurgical recoveries for the pug roast and leach option are estimated at 91% U_3O_8, 68% V_2O_5, 70% SOP, 32% Ni, 47% Zn and 52% Mo.

25.9 PROJECT INFRASTRUCTURE

Primary electrical supply is planned from the regional grid through a new overhead distribution line connected to the existing low voltage network located within approximately 3 km of site, supported by a site substation, transformers, distribution lines, motor control centres, and backup power systems. Water supply is proposed from Lake Storsjön or Abassen Creek with recycling of recovered process water and pit dewatering water. LNG delivered by road tanker and stored in cryogenic tanks is planned for process and utility requirements.

Site facilities will include administration buildings, workshops, warehouses, assay laboratory, maintenance facilities, fuel storage and dispensing, water treatment facilities, communications systems, and support services. The process plant will include crushing, grinding, flotation, pug roasting, ion exchange, solvent extraction, product packaging, and storage to produce vanadium products, U_3O_8, sulphate of potassium and a mixed metal concentrate. Waste management includes, septic systems, hazardous waste controls, and a dry stacked tailings and waste rock storage facility.

25.10 MARKET STUDIES AND CONTRACTS

The Viken Project is primarily focused on vanadium products, with vanadium electrolyte and ferrovanadium (FeV80) identified as the key saleable products. Commercial-grade V_2O_5 flake is treated as an intermediate product and may also be sold directly, subject to market conditions and customer requirements. In addition to vanadium products, the Project is expected to produce the following secondary product streams: Uranium oxide (U_3O_8), Sulphate of Potash ("SOP") and Mixed Metal Sulphides ("MMS") containing nickel, zinc and molybdenum.

Base case metal/commodity prices in US$ are: Ferrovanadium (FeV80) $38/kg; Vanadium Electrolyte $9/L; Uranium Oxide ($U_3O_8$) $85/lb; Sulphate of Potash (for 50% K_2O SOP granules) $650/t; Nickel $7.71/lb; Zinc $1.45/lb; and Molybdenum $27.22/lb.

There are currently no material contracts in place for the Viken Project.

25.11 ENVIRONMENTAL, SOCIAL IMPACTS

Environmental and social evaluation for the Viken Project is at a conceptual stage and additional baseline studies are required to support future assessment and permitting. Historical and recent public concerns have focused primarily on potential impacts to water quality in Lake Storsjön, land disturbance, and uranium recovery activities. Anticipated environmental management measures include control of dust from crushing, management of gaseous emissions from calcining and roasting, recycling of treated water where possible, neutralization of barren process solutions with lime, and co-disposal of gypsum and metal hydroxide water treatment by-products in a dedicated chemical solids storage facility. Flotation tailings and process residues are proposed to be dry stacked and progressively covered using an impervious multi-zone soil cover.

Project development will require permitting under the Swedish Minerals Act and Swedish Environmental Code, including preparation of an Environmental Impact Assessment, public consultation, municipal approvals, and potential consideration under the Nuclear Activities Act and radiation protection legislation. Hydrogeological characterization and groundwater studies are identified as key permitting requirements. Community engagement has included information initiatives and landowner meetings. Closure planning contemplates progressive reclamation, pit flooding with water treatment, engineered waste storage, long-term monitoring, and inclusion of closure surety requirements as part of the permitting and licensing process.

25.12 CAPITAL AND OPERATING COSTS

All costs are presented in Q2 2026 US dollars. No provision has been included in the cost estimates to offset future escalation. A contingency of approximately 20% has been added to all capital costs ("CAPEX"). No contingency is added to operating costs ("OPEX").

The total initial CAPEX of the Viken Project is estimated at $876M and addresses the engineering, procurement, construction and start-up of the Viken Project, with a two-year construction period to develop an open pit mine, build a process plant capable of processing 10 Mtpa, and install associated ancillary surface facilities. Sustaining capital costs incurred during the 13 production years and one year of rehabilitation are estimated to total $158M and are primarily for mining and processing equipment replacement.

Total OPEX over the life-of-mine ("LOM") is estimated at $5.9B, averaging $46.58/t processed. LOM average open pit mining is estimated at $2.95/t mined ($3.50/t processed), processing averages $37.43/t, dry stack tailings averages $1.81/t processed, G&A is estimated at $1.33/t processed, and SOP freight averages $2.51/t processed.

Cash costs over the LOM are estimated to average negative US$121/lb U_3O_8 net of by-product credits. All-In Sustaining Costs ("AISC") over the LOM are estimated to average negative US$118/lb U_3O_8 net of by-product credits.

25.13 ECONOMIC ANALYSIS

Payable metal over the LOM is estimated at 76.2 Mkg FeV, 474.4 ML vanadium electrolyte, 41.8 Mlb U_3O_8, 20.9 Mlb Ni, 27.0 Mlb Zn, 58.3 Mlb Mo and 3.2 Mt SOP. The Viken Project economic evaluation at base case metal/commodity prices of US$38/kg FeV80, US$9/L vanadium electrolyte, US$85/lb U_3O_8, US$650/t SOP, US$7.71/lb Ni, US$1.45/lb Zn and US$27.22/lb Mo has an estimated US$2.88B after-tax net present value ("NPV") at an 8% discount rate ("NPV8%"), and an after-tax internal rate of return ("IRR") of 46%. The after-tax payback period is estimated to be 2.1 years from the start of production. The after-tax base case NPVs and IRRs are most sensitive to metal/commodity prices followed by operating costs and capital costs.

25.14 RISKS AND OPPORTUNITIES

Overall Project risks are perceived by the Authors as moderate. Principal risks include variability in mineralization characteristics and metallurgical performance, limited validation of recoveries and reagent consumption assumptions, open pit geotechnical uncertainty, dry stacked tailings performance, environmental compliance, permitting timelines, infrastructure execution, occupational health and safety, and social acceptance associated with uranium processing and water management. Financial risks reflect preliminary cost estimates with an estimated accuracy of ±50%, together with exposure to inflation, exchange rates, and commodity market conditions.

Opportunities include the scale of the multi-commodity Mineral Resource and associated long mine life potential, diversified revenue sources from uranium, vanadium, sulphate of potash, nickel, zinc, and molybdenum products, potential Mineral Resource growth, process optimization and recovery improvements, modular development approaches, and environmental performance enhancements through dry stack tailings, water recycling, and progressive rehabilitation.

26.0 RECOMMENDATIONS

The Authors conclude that the Viken Project contains a very large vanadium, uranium, potassium and polymetallic Mineral Resource that merits further exploration and evaluation.

The Authors' recommendations for advancing the Project are as follows:

- Drill testing of airborne MobileMT anomalies on the Viken Property;
- Interpretation, geological modelling and reporting;
- Diamond drilling to convert Inferred to Indicated Mineral Resources and for metallurgical samples;
- Metallurgical testwork;
- Geotechnical and hydrogeological drilling and studies; and
- A Pre-Feasibility Study ("PFS").

Future drill core sampling at the Project should include the insertion of certified reference materials of appropriate grades, for all elements of interest, into the sample stream on-site before shipping to the lab, the insertion and monitoring of field and coarse reject duplicates, and to umpire analyze 5 to 10% of all future drill core samples at a reputable secondary laboratory.

The proposed metallurgical testwork program should include expanded laboratory-scale and integrated pilot-scale metallurgical programs covering all major mineralized domains and process routes, together with variability testing across representative composites. Dedicated testwork should also be completed to validate acid consumption, reagent demand, comminution parameters, and power requirements. Specific items for validation and optimization include flotation performance, solvent extraction recoveries, SOP product quality, product specification and marketability for all products, pug roast acid consumption, mixed metal precipitate recovery, process integration of the various circuits, and scale-up viability.

District Metals should also commence initial geotechnical testwork of mineralized shale and waste rock regarding open pit wall slopes and extraction techniques, and hydrogeological studies of the rivers, creeks and wetlands in the Project area.

It is further recommended that an environmental baseline study be completed to characterize the existing features of the air, water and soil both on the Viken Property and in the surrounding area. District Metals should commence lab testing to characterize the acid generation or consuming and metal leaching potential of the geologic materials that could be exposed.

District Metals should also commence a community engagement program with local communities, government agencies, and other interested groups.

A PFS should follow this PEA targeting a similar mining scenario with a life of mine of 12 to 14 years.

The estimated budget to complete the recommended work program is approximately $7.3M and is presented in Table 26.1. The work should be completed in the next 12 to 18 months.

TABLE 26.1
RECOMMENDED WORK PROGRAM

Activity	Units	Cost Estimate (US$)
Drill testing of airborne MobileMT anomalies	3,000 m x $300/m	900,000
Interpretation, Modelling, Reporting		100,000
Diamond Drilling: Metallurgy & Inferred to Indicated Mineral Resource Conversion	6,000 m x $300/m	1,800,000
Metallurgical Testwork		900,000
Environmental Baseline Study		300,000
Geotechnical and Hydrogeological Studies		600,000
Pre-Feasibility Study		1,500,000
Subtotal		**6,100,000**
Contingency (20%)		1,220,000
Total		**7,320,000**

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28.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

ANDREW BRADFIELD, P. ENG.

I, Andrew Bradfield, P. Eng., residing at 5 Patrick Drive, Erin, Ontario, N0B 1T0, do hereby certify that:

1. I am an independent mining engineer contracted by P&E Mining Consultants.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment on the Viken Energy Metals Project, Jämtland County, Sweden", (The "Technical Report") with an effective date of June 26, 2026.
3. I am a graduate of Queen's University, with an honours B.Sc. degree in Mining Engineering in 1982. I have practiced my profession continuously since 1982. I am a Professional Engineer of Ontario (License No.4894507). I am also a member of the National CIM.

   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.

   I have practiced my profession continuously since 1982. My summarized career experience is as follows:
   - Various Engineering Positions – Palabora Mining Company, 1982-1986
   - Mines Project Engineer – Falconbridge Limited, 1986-1987
   - Senior Mining Engineer – William Hill Mining Consultants Limited, 1987-1990
   - Independent Mining Engineer, 1990-1991
   - GM Toronto – Bharti Engineering Associates Inc, 1991-1996
   - VP Technical Services, GM of Australian Operations – William Resources Inc, 1996-1999
   - Independent Mining Engineer, 1999-2001
   - Principal Mining Engineer – SRK Consulting, 2001-2003
   - COO – China Diamond Corp, 2003-2006
   - VP Operations – TVI Pacific Inc, 2006-2008
   - COO – Avion Gold Corporation, 2008-2012
   - Independent Mining Engineer, 2012-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 2, 3, 15, 16, 22 and co-authoring Sections 1, 21, 24, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had no prior involvement with the Project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 26, 2026
Signed Date: July 17, 2026

{SIGNED AND SEALED}
[Andrew Bradfield]

Andrew Bradfield, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

WILLIAM STONE, PH.D., P.GEO.

I, William Stone, Ph.D., P.Geo, residing at 4361 Latimer Crescent, Burlington, Ontario, do hereby certify that:

1. I am an independent geological consultant working for P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment on the Viken Energy Metals Project, Jämtland County, Sweden", (The "Technical Report") with an effective date of June 26, 2026.
3. I am a graduate of Dalhousie University with a Bachelor of Science (Honours) degree in Geology (1983). In addition, I have a Master of Science in Geology (1985) and a Ph.D. in Geology (1988) from the University of Western Ontario. I have worked as a geologist for a total of 40 years since obtaining my M.Sc. degree. I am a geological consultant currently licensed by the Professional Geoscientists of Ontario (License No. 1569).

   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.

   My relevant experience for the purpose of the Technical Report is:

• Contract Senior Geologist, LAC Minerals Exploration Ltd.	1985-1988
• Post-Doctoral Fellow, McMaster University	1988-1992
• Contract Senior Geologist, Outokumpu Mines and Metals Ltd.	1993-1996
• Senior Research Geologist, WMC Resources Ltd.	1996-2001
• Senior Lecturer, University of Western Australia	2001-2003
• Principal Geologist, Geoinformatics Exploration Ltd.	2003-2004
• Vice President Exploration, Nevada Star Resources Inc.	2005-2006
• Vice President Exploration, Goldbrook Ventures Inc.	2006-2008
• Vice President Exploration, North American Palladium Ltd.	2008-2009
• Vice President Exploration, Magma Metals Ltd.	2010-2011
• President & COO, Pacific North West Capital Corp.	2011-2014
• Consulting Geologist	2013-2017
• Senior Project Geologist, Anglo American	2017-2019
• Consulting Geoscientist	2020-Present

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 4 to 10, and 23, and co-authoring Sections 1, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Property that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled: "Updated Mineral Resource Estimate and Technical Report on the Viken Energy Metals Project, Jämtland County, Sweden", dated June 13, 2025.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 26, 2026
Signed Date: July 17, 2026

{SIGNED AND SEALED}
[William Stone]

____
William E. Stone, Ph.D., P.Geo.

CERTIFICATE OF QUALIFIED PERSON

FRED H. BROWN, P.GEO.

I, Fred H. Brown, P.Geo., of PO Box 332, Lynden, WA, USA, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc. and have worked as a geologist continuously since my graduation from university in 1987, specialising in gold, silver, base metals, PGEs, diamonds, industrial minerals and other commodities.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment on the Viken Energy Metals Project, Jämtland County, Sweden", (The "Technical Report") with an effective date of June 26, 2026.
3. I graduated with a Bachelor of Science degree in Geology from New Mexico State University in 1987. I obtained a Diploma in Datametrics in 1993 from the University of South Africa, a Graduate Diploma in Engineering (Mining) in 1997 from the University of the Witwatersrand, and a Master of Science in Engineering (Civil) from the University of the Witwatersrand in 2005. In 2015 I obtained a Citation in Applied Geostatistics from the University of Alberta, and a Geographic Information Systems Certificate from the University of California San Diego in 2016. I am registered with the Association of Professional Engineers and Geoscientists of British Columbia as a Professional Geoscientist (171602) and the Society for Mining, Metallurgy and Exploration as a Registered Member (4152172).

   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.

   My relevant experience for the purpose of the Technical Report is:
   - Mineral Resource Manager, AngloGold Corporation, South Africa 1988-1997
   - Chief Geologist, De Beers Consolidated Mines, South Africa 1997-2004
   - Consulting Geologist 2004-2015 &2016-Present
   - Senior Geostatistician, Twin Creeks & Phoenix Mines, Nevada 2015-2016
   - P&E Mining Consultants Inc. – Sr. Associate Geologist 2008-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for co-authoring Sections 1, 14, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Property that is the subject of this Technical Report. I was a "Qualified Person" for Technical Reports titled: "Updated Mineral Resource Estimate and Technical Report on the Viken Energy Metals Project, Jämtland County, Sweden", dated June 13, 2025; "Updated Technical Report, Resource Estimate and Preliminary Economic Assessment on the Viken MMS Project, Sweden", dated February 27, 2014; "Preliminary Economic Assessment on the Viken MMS Project, Sweden", dated October 19, 2010; "Third Updated Technical Report on Viken MMS Licence, Jämtland, Kingdom of Sweden", dated March 19, 2009; "Second updated technical report on Viken MMS licence, Jämtland, Kingdom of Sweden", dated April 11, 2008; "Updated Technical report on Viken MMS Licence, Jämtland, Kingdom of Sweden", dated August 28, 2007.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 26, 2026
Signed Date: July 17, 2026

{SIGNED AND SEALED}
[Fred H. Brown]

Fred H. Brown, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

JARITA BARRY, P.GEO.

I, Jarita Barry, P.Geo., residing at 9052 Mortlake-Ararat Road, Ararat, Victoria, Australia, 3377, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment on the Viken Energy Metals Project, Jämtland County, Sweden", (The "Technical Report") with an effective date of June 26, 2026.
3. I am a graduate of RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology. I have worked as a geologist for over 20 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by Engineers and Geoscientists British Columbia (License No. 40875), Professional Engineers and Geoscientists Newfoundland & Labrador (License No. 08399), Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License No. L3874), and I am registered as a Temporary Registrant with Professional Geoscientists Ontario (Registration No. 3888). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397);

   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.

   My relevant experience for the purpose of the Technical Report is:
   - Geologist, Foran Mining Corp. 2004
   - Geologist, Aurelian Resources Inc. 2004
   - Geologist, Linear Gold Corp. 2005-2006
   - Geologist, Búscore Consulting 2006-2007
   - Consulting Geologist (AusIMM) 2008-2014
   - Consulting Geologist, P.Geo. (EGBC/AusIMM) 2014-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Section 11 and co-authoring Sections 1, 12, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Property that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled: "Updated Mineral Resource Estimate and Technical Report on the Viken Energy Metals Project, Jämtland County, Sweden", dated June 13, 2025.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 26, 2026
Signed Date: July 17, 2026

{SIGNED AND SEALED}
[Jarita Barry]

____
Jarita Barry, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

DAVID BURGA, P.GEO.

I, David Burga, P. Geo., residing at 3884 Freeman Terrace, Mississauga, Ontario, do hereby certify that:

1. I am an independent geological consultant contracted by P & E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment on the Viken Energy Metals Project, Jämtland County, Sweden", (The "Technical Report") with an effective date of June 26, 2026.
3. I am a graduate of the University of Toronto with a Bachelor of Science degree in Geological Sciences (1997). I have worked as a geologist for over 20 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by the Association of Professional Geoscientists of Ontario (License No. 1836).

   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.

   My relevant experience for the purpose of the Technical Report is:

   - Exploration Geologist, Cameco Gold 1997-1998
   - Field Geophysicist, Quantec Geoscience 1998-1999
   - Geological Consultant, Andeburg Consulting Ltd. 1999-2003
   - Geologist, Aeon Egmond Ltd. 2003-2005
   - Project Manager, Jacques Whitford 2005-2008
   - Exploration Manager – Chile, Red Metal Resources 2008-2009
   - Consulting Geologist 2009-Present
4. I have visited the Property that is the subject of this Technical Report on March 20, 2025.
5. I am responsible for co-authoring Sections 1, 12, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Property that is the subject of this Technical Report. I was a "Qualified Person" for Technical Reports titled: "Updated Mineral Resource Estimate and Technical Report on the Viken Energy Metals Project, Jämtland County, Sweden", dated June 13, 2025; and "Updated Technical Report, Resource Estimate and Preliminary Economic Assessment on the Viken MMS Project, Sweden", dated February 27, 2014.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 26, 2026
Signed Date: July 17, 2026

{SIGNED AND SEALED}
[David Burga]

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David Burga, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

D. GRANT FEASBY, P. ENG.

I, D. Grant Feasby, P. Eng., residing at 12,209 Hwy 38, Tichborne, Ontario, K0H 2V0, do hereby certify that:

1. I am currently the Owner and President of:
   FEAS - Feasby Environmental Advantage Services
   38 Gwynne Ave, Ottawa, K1Y1W9
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment on the Viken Energy Metals Project, Jämtland County, Sweden", (The "Technical Report") with an effective date of June 26, 2026.
3. I graduated from Queens University in Kingston Ontario, in 1964 with a Bachelor of Applied Science in Metallurgical Engineering, and a Master of Applied Science in Metallurgical Engineering in 1966. I am a Professional Engineer registered with Professional Engineers Ontario. I have worked as a metallurgical engineer for over 50 years since my graduation from university.

   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.

   My relevant experience for the purpose of the Technical Report has been acquired by the following activities:
   - Metallurgist, Base Metal Processing Plant.
   - Research Engineer and Lab Manager, Industrial Minerals Laboratories in USA and Canada.
   - Research Engineer, Metallurgist and Plant Manager in the Canadian Uranium Industry.
   - Manager of Canadian National Programs on Uranium and Acid Generating Mine Tailings.
   - Director, Environment, Canadian Mineral Research Laboratory.
   - Senior Technical Manager, for large gold and bauxite mining operations in South America.
   - Expert Independent Consultant associated with several companies, including P&E Mining Consultants, on mineral processing, environmental management, and mineral-based radiation assessment.
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Section 20 and co-authoring Sections 1, 25, 26, and 27 of this Technical Report.
6. I am independent of the issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Property that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled: "Updated Mineral Resource Estimate and Technical Report on the Viken Energy Metals Project, Jämtland County, Sweden", dated June 13, 2025.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 26, 2026
Signed Date: July 17, 2026

{SIGNED AND SEALED}
[D. Grant Feasby]

D. Grant Feasby, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

EUGENE PURITCH, P. ENG., FEC, CET

I, Eugene J. Puritch, P. Eng., FEC, CET, residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:

1. I am an independent mining consultant and President of P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment on the Viken Energy Metals Project, Jämtland County, Sweden", (The "Technical Report") with an effective date of June 26, 2026.
3. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition, I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for a Bachelor's degree in Engineering Equivalency. I am a mining consultant currently licensed by the: Professional Engineers and Geoscientists New Brunswick (License No. 4778); Professional Engineers, Geoscientists Newfoundland and Labrador (License No. 5998); Association of Professional Engineers and Geoscientists Saskatchewan (License No. 16216); Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252); Professional Engineers of Ontario (License No. 100014010); Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912); and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (No. L3877). I am also a member of the National Canadian Institute of Mining and Metallurgy.

   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.

   I have practiced my profession continuously since 1978. My summarized career experience is as follows:

   - Mining Technologist - H.B.M.& S. and Inco Ltd., 1978-1980
   - Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd., 1981-1983
   - Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine, 1984-1986
   - Self-Employed Mining Consultant – Timmins Area, 1987-1988
   - Mine Designer/Resource Estimator – Dynatec/CMD/Bharti, 1989-1995
   - Self-Employed Mining Consultant/Resource-Reserve Estimator, 1995-2004
   - President – P&E Mining Consultants Inc, 2004-Present

4. I have visited the Property that is the subject of this Technical Report on June 18, 2013.
5. I am responsible for co-authoring Sections 1, 12, 14, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Property that is the subject of this Technical Report. I was a "Qualified Person" for Technical Reports titled: "Updated Mineral Resource Estimate and Technical Report on the Viken Energy Metals Project, Jämtland County, Sweden", dated June 13, 2025; "Updated Technical Report, Resource Estimate and Preliminary Economic Assessment on the Viken MMS Project, Sweden", dated February 27, 2014; "Preliminary Economic Assessment on the Viken MMS Project, Sweden", dated October 19, 2010; "Third Updated Technical Report on Viken MMS Licence, Jämtland, Kingdom of Sweden", dated March 19, 2009; "Second updated technical report on Viken MMS licence, Jämtland, Kingdom of Sweden", dated April 11, 2008; "Updated Technical report on Viken MMS Licence, Jämtland, Kingdom of Sweden", dated August 28, 2007.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 26, 2026

Signed Date: July 17, 2026

{SIGNED AND SEALED}
[Eugene Puritch]

____

Eugene Puritch, P.Eng., FEC, CET

CERTIFICATE OF QUALIFIED PERSON

DAMIAN CONNELLY, FAUSIMM (CP) MET. MMSA. FIEAUST

I, Damian Connelly, FAusIMM (CP) Met. MMSA FIEAust., residing at 13 Colin St, West Perth, Australia, do hereby certify that:

1. I am an independent consulting engineer contracted by METS Engineering Group Pty Ltd.
2. This certificate applies to the Technical Report titled “Preliminary Economic Assessment on the Viken Energy Metals Project, Jämtland County, Sweden”, (The “Technical Report”) with an effective date of June 26, 2026.
3. I am a graduate of Adelaide University, with a degree in Extractive Metallurgy in 1972. I have practiced my profession continuously since 1972. I am a Consulting Engineer in Metallurgy.

   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.

   My summarized career experience is as follows:

   Principal Consulting Engineer managing a multidisciplinary engineering team with extensive experience in the gold, copper, lead, zinc, uranium, nickel, lithium, vanadium and iron ore industries with particular emphasis on gold. Experience has been gained in plant operations feasibility studies, detailed design, construction and commissioning, and all unit operations. Am an internationally recognized specialist in Mineral Processing and Hydrometallurgy having worked in North and South America, South East Asia, Africa and Europe and am highly sought after as an International Expert. The last +38 years working as a Principal Consulting Engineer on many resource projects and for mining, banking and engineering companies has provided a broad range of experience covering operations, engineering, many commodities and on many continents. Am a Registered Expert Witness and have been engaged on many cases. Am the author of over 230 technical papers on mineral processing, hydrometallurgy and pyrometallurgy.
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 13, 17, 18, 19 and co-authoring Sections 1, 21, 24, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had no prior involvement with the Project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 26, 2026
Signed Date: July 17, 2026

{SIGNED AND SEALED}
[Damian Connelly]

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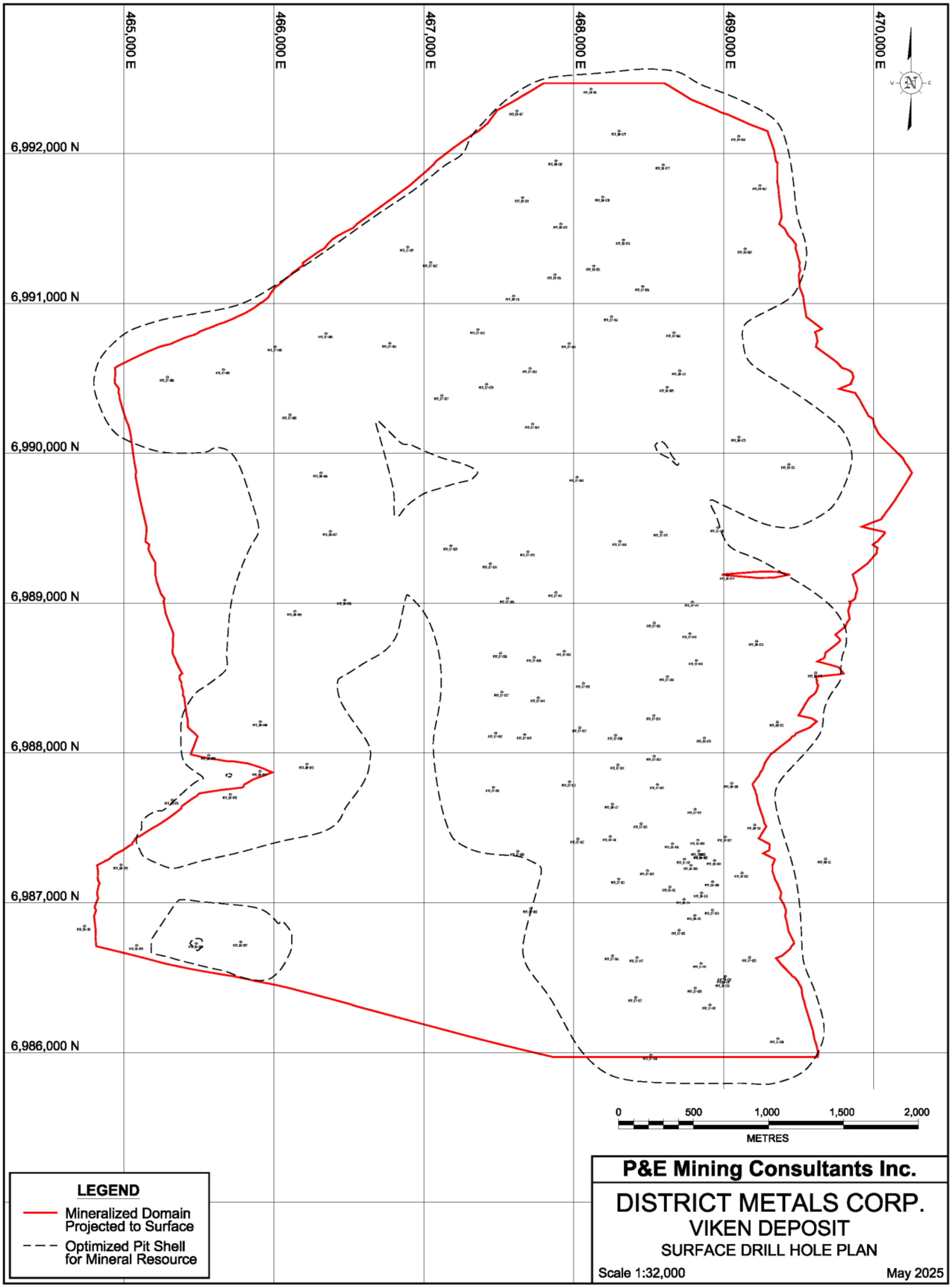
Damian Connelly, FAusIMM (CP) Met. MMSA FIEAust.

APPENDIX A DRILL HOLE PLAN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6,986,000 N
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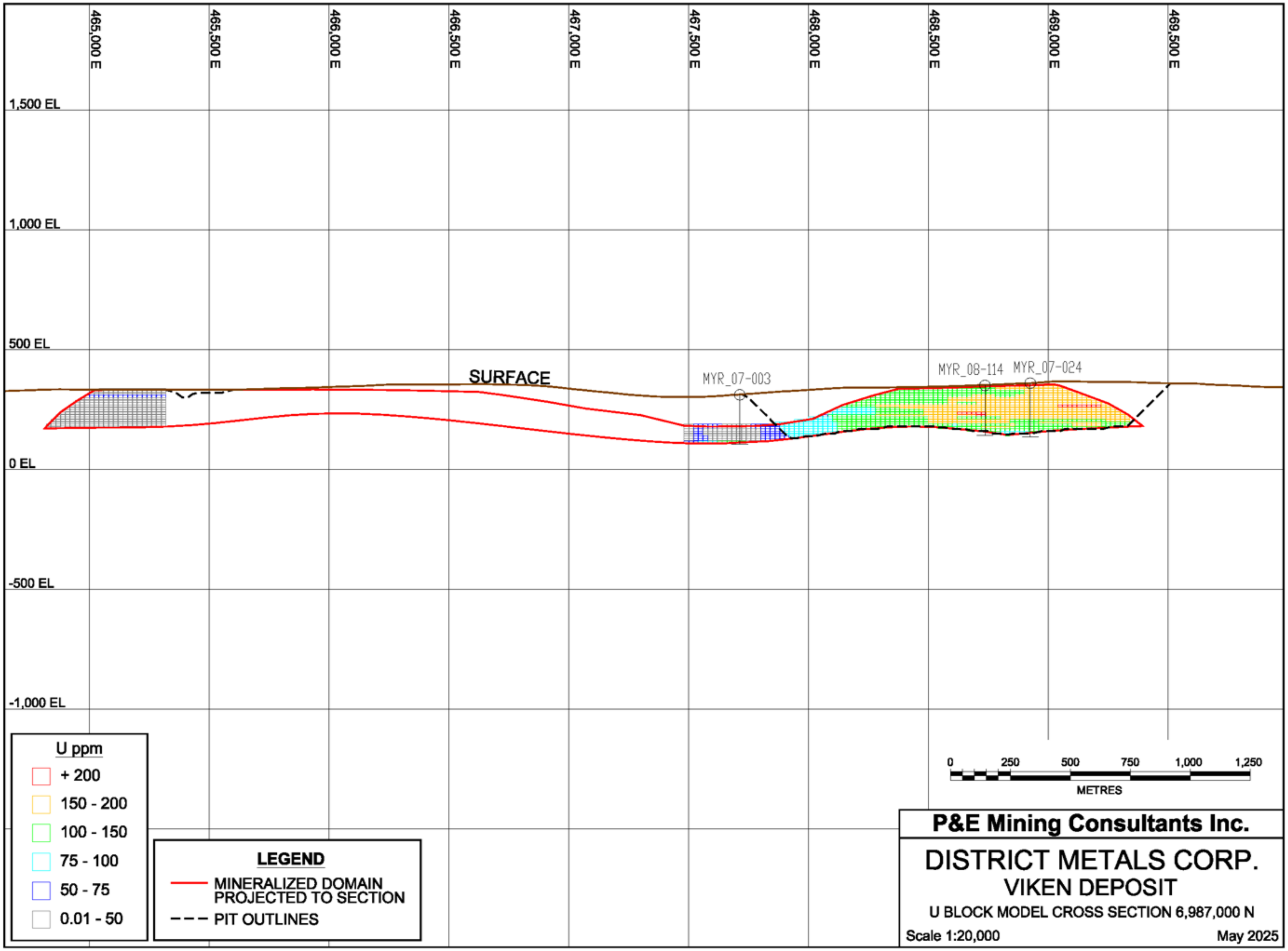
METRES
LEGEND
Mineralized Domain Projected to Surface
Optimized Pit Shell for Mineral Resource
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
SURFACE DRILL HOLE PLAN
Scale 1:32,000
May 2025

APPENDIX B 3-D DOMAIN

VIKEN DEPOSIT - 3D DOMAIN

APPENDIX C U BLOCK MODEL CROSS-SECTIONS AND PLANS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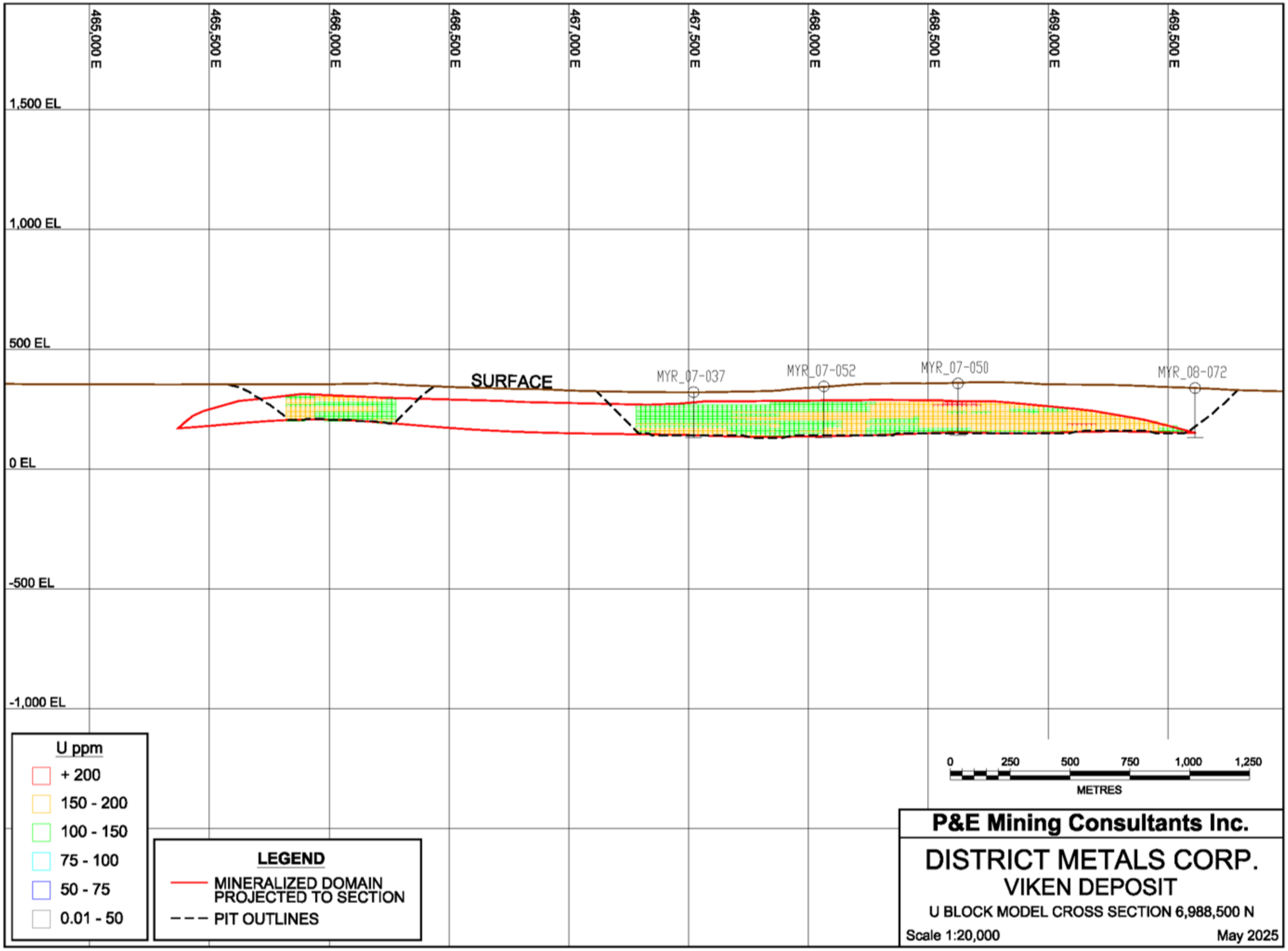
SURFACE
MYR_07-003
MYR_08-114
MYR_07-024
U ppm
+ 200
150 - 200
100 - 150
75 - 100
50 - 75
0.01 - 50
LEGEND
MINERALIZED DOMAIN PROJECTED TO SECTION
PIT OUTLINES
0
250
500
750
1,000
1,250
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
U BLOCK MODEL CROSS SECTION 6,987,000 N
Scale 1:20,000
May 2025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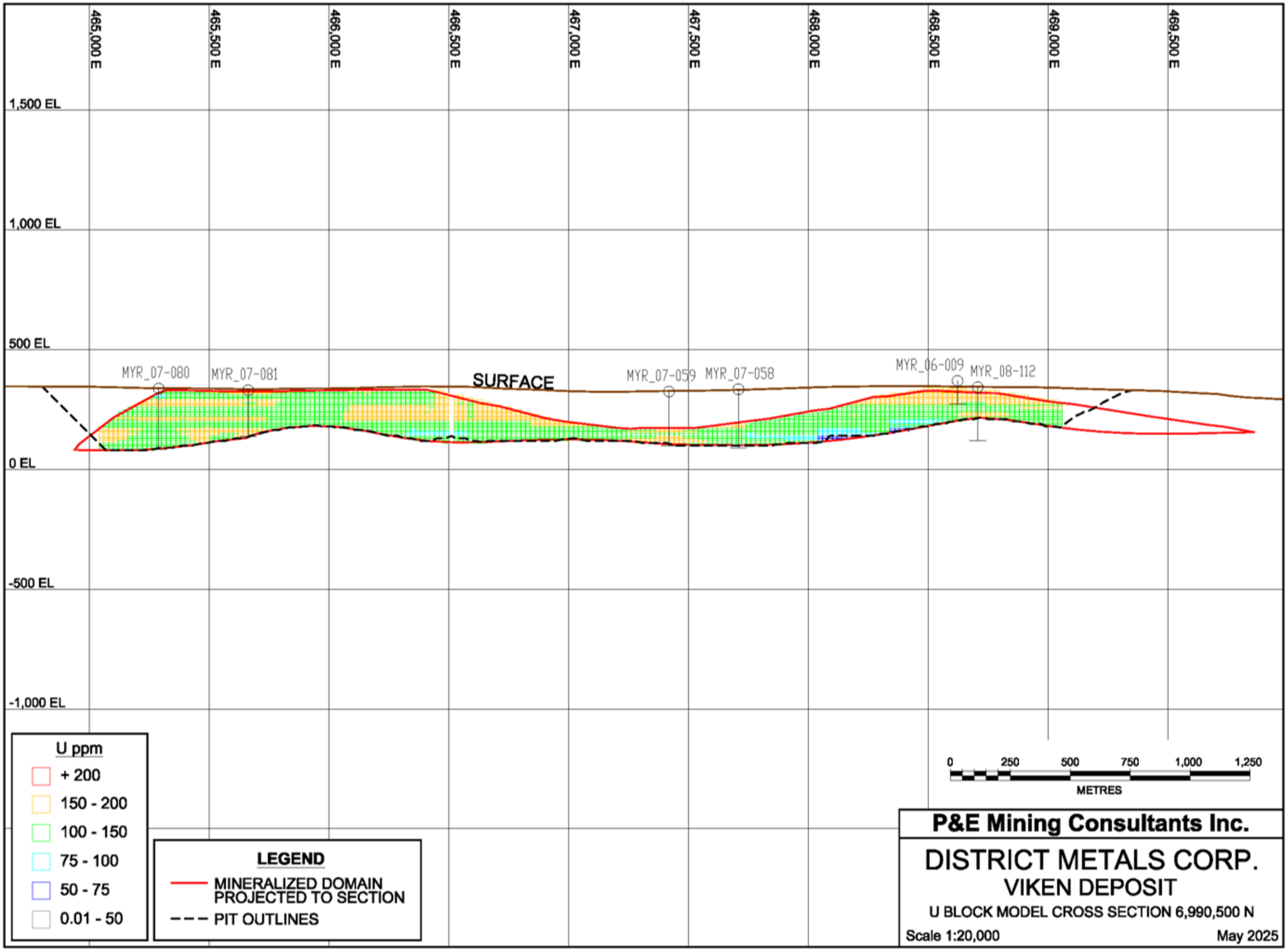
SURFACE
MYR_07-037
MYR_07-052
MYR_07-050
MYR_08-072
U ppm
+ 200
150 - 200
100 - 150
75 - 100
50 - 75
0.01 - 50
LEGEND
MINERALIZED DOMAIN PROJECTED TO SECTION
PIT OUTLINES
0
250
500
750
1,000
1,250
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
U BLOCK MODEL CROSS SECTION 6,988,500 N
Scale 1:20,000
May 2025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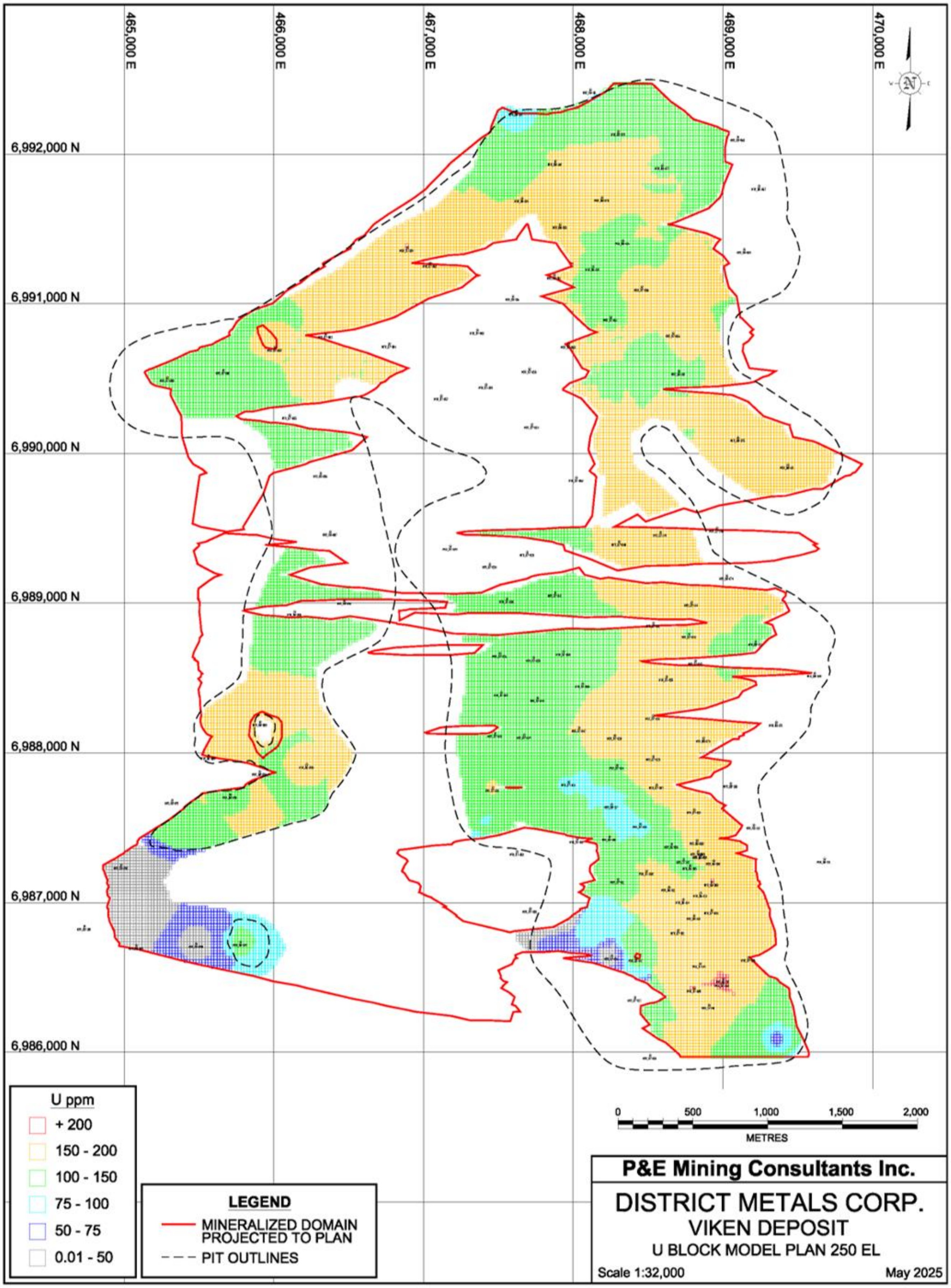
SURFACE
MYR_07-080
MYR_07-081
MYR_07-059
MYR_07-058
MYR_06-009
MYR_08-112
U ppm
+ 200
150 - 200
100 - 150
75 - 100
50 - 75
0.01 - 50
LEGEND
MINERALIZED DOMAIN PROJECTED TO SECTION
PIT OUTLINES
0 250 500 750 1,000 1,250
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
U BLOCK MODEL CROSS SECTION 6,990,500 N
Scale 1:20,000
May 2025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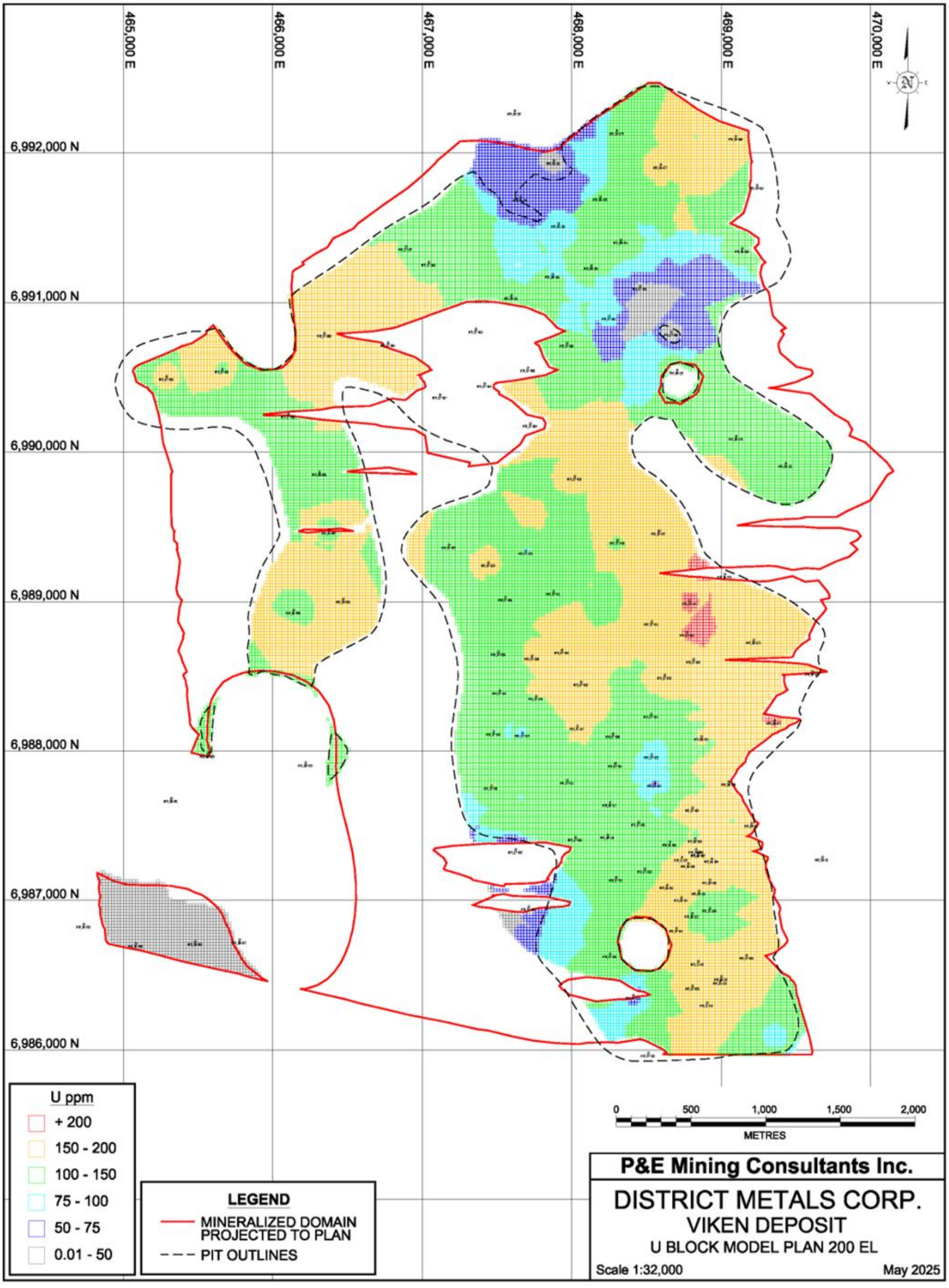
METRES
U ppm
+ 200
150 - 200
100 - 150
75 - 100
50 - 75
0.01 - 50
LEGEND
MINERALIZED DOMAIN PROJECTED TO PLAN
PIT OUTLINES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
U BLOCK MODEL PLAN 250 EL
Scale 1:32,000
May 2025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U ppm
+ 200
150 - 200
100 - 150
75 - 100
50 - 75
0.01 - 50
LEGEND
MINERALIZED DOMAIN PROJECTED TO PLAN
PIT OUTLINES
0
500
1,000
1,500
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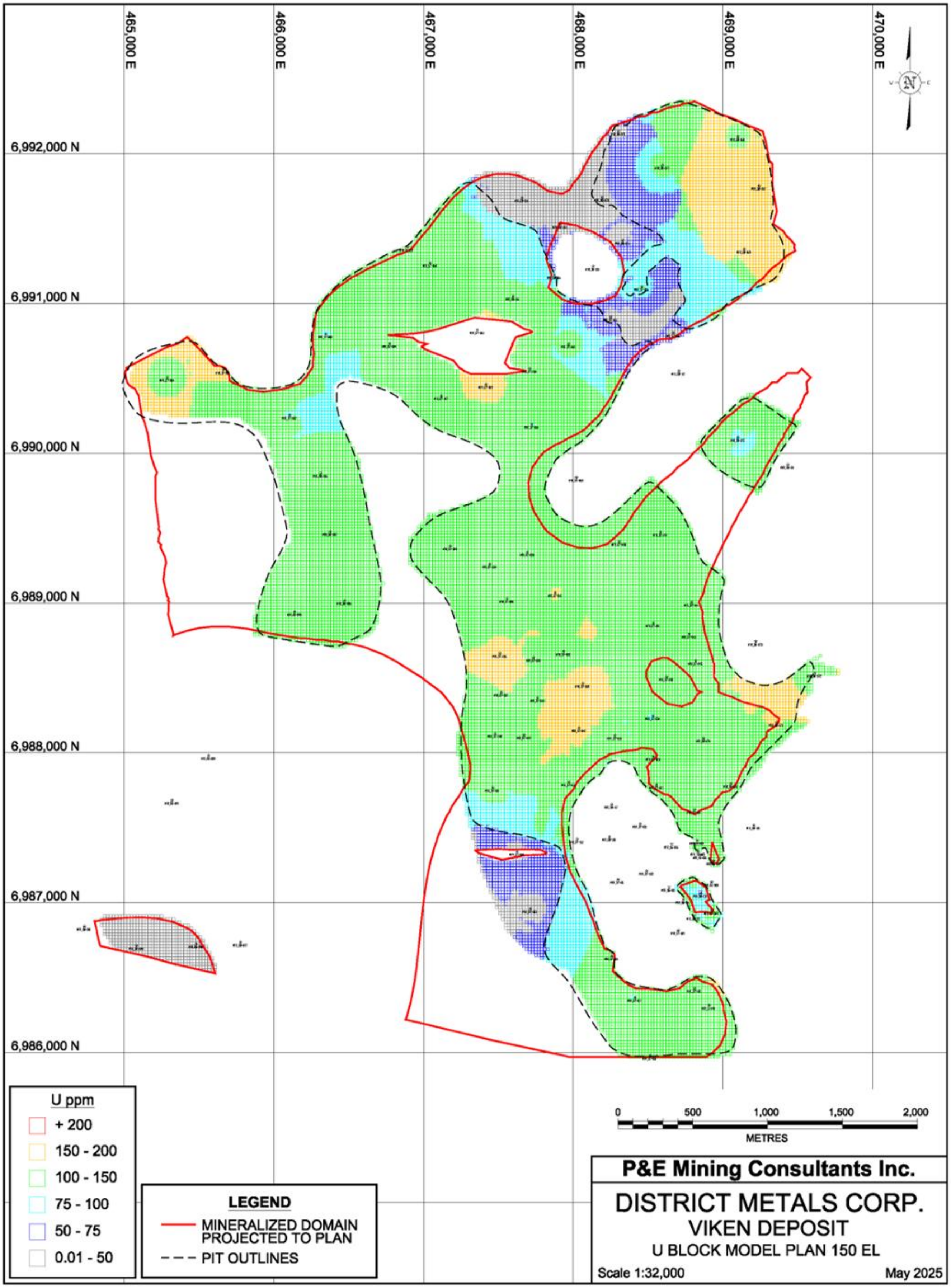
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
U BLOCK MODEL PLAN 200 EL
Scale 1:32,000
May 2025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U ppm
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100 - 150
75 - 100
50 - 75
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LEGEND
MINERALIZED DOMAIN PROJECTED TO PLAN
PIT OUTLINES
0
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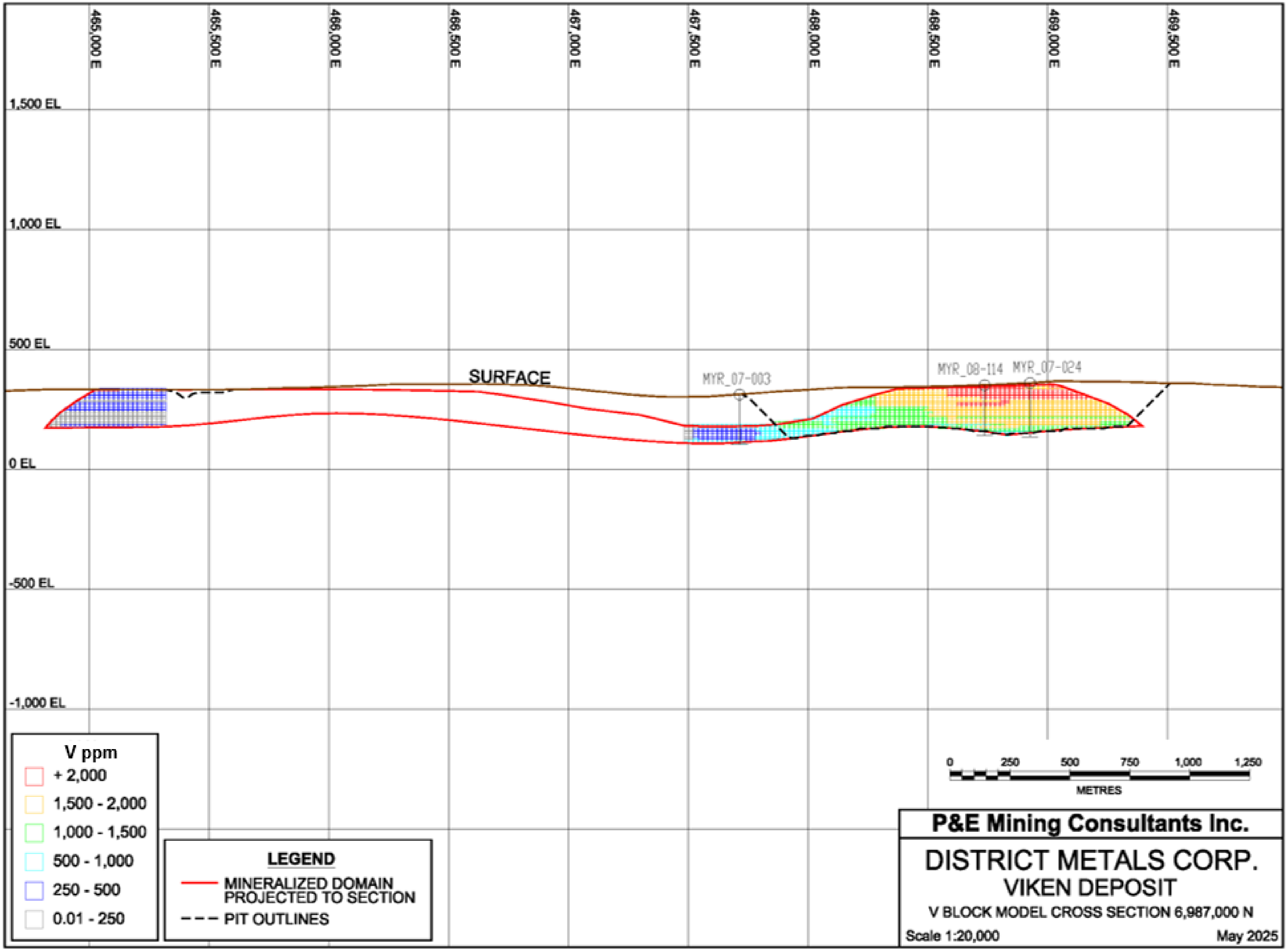
2,000
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
U BLOCK MODEL PLAN 150 EL
Scale 1:32,000
May 2025

APPENDIX D V BLOCK MODEL CROSS-SECTION AND PLANS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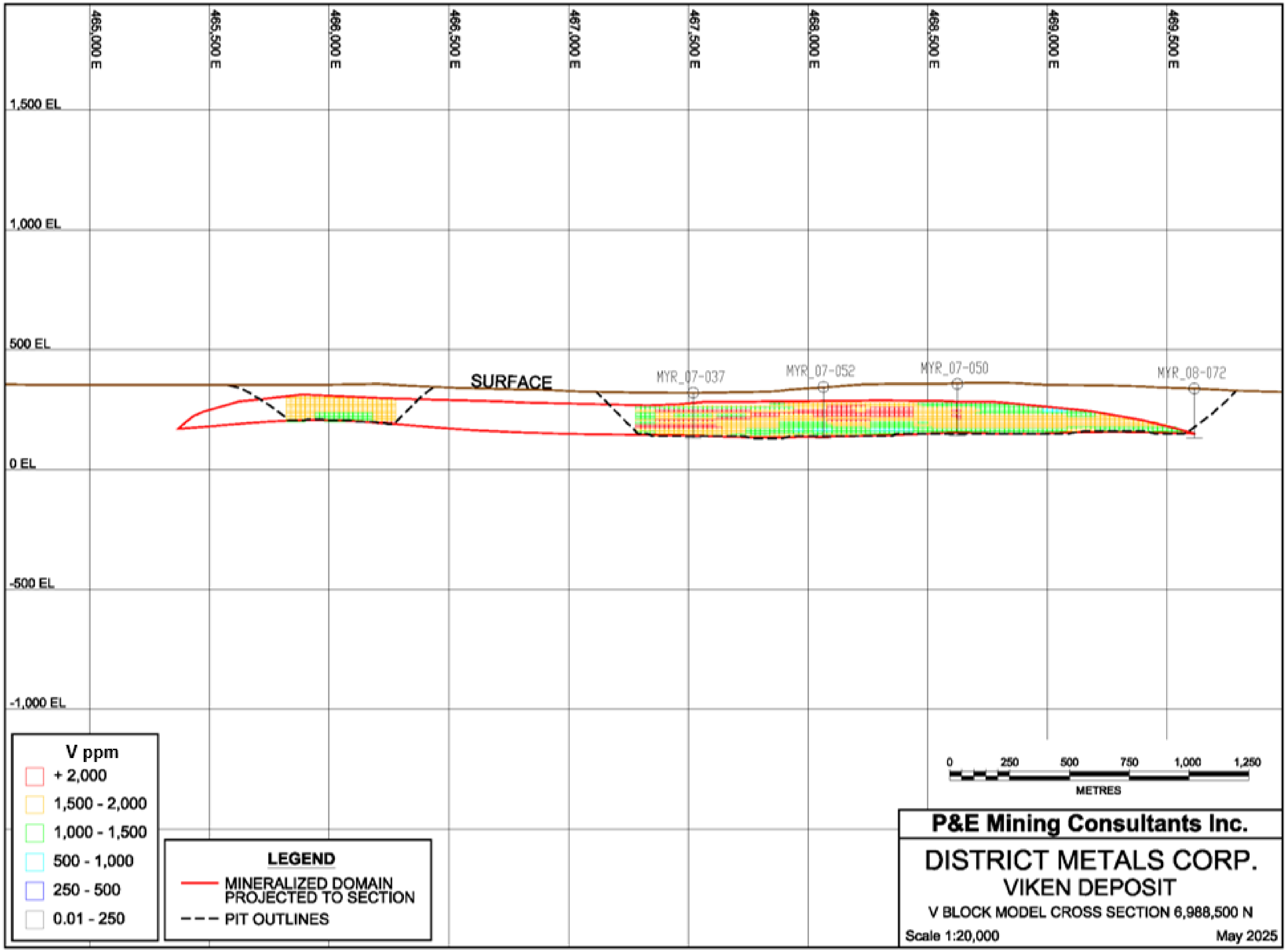
SURFACE
MYR_07-003
MYR_08-114
MYR_07-024
V ppm
+ 2,000
1,500 - 2,000
1,000 - 1,500
500 - 1,000
250 - 500
0.01 - 250
LEGEND
MINERALIZED DOMAIN PROJECTED TO SECTION
PIT OUTLINES
0
250
500
750
1,000
1,250
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
V BLOCK MODEL CROSS SECTION 6,987,000 N
Scale 1:20,000
May 2025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SURFACE
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MYR_07-052
MYR_07-050
MYR_08-072
V ppm
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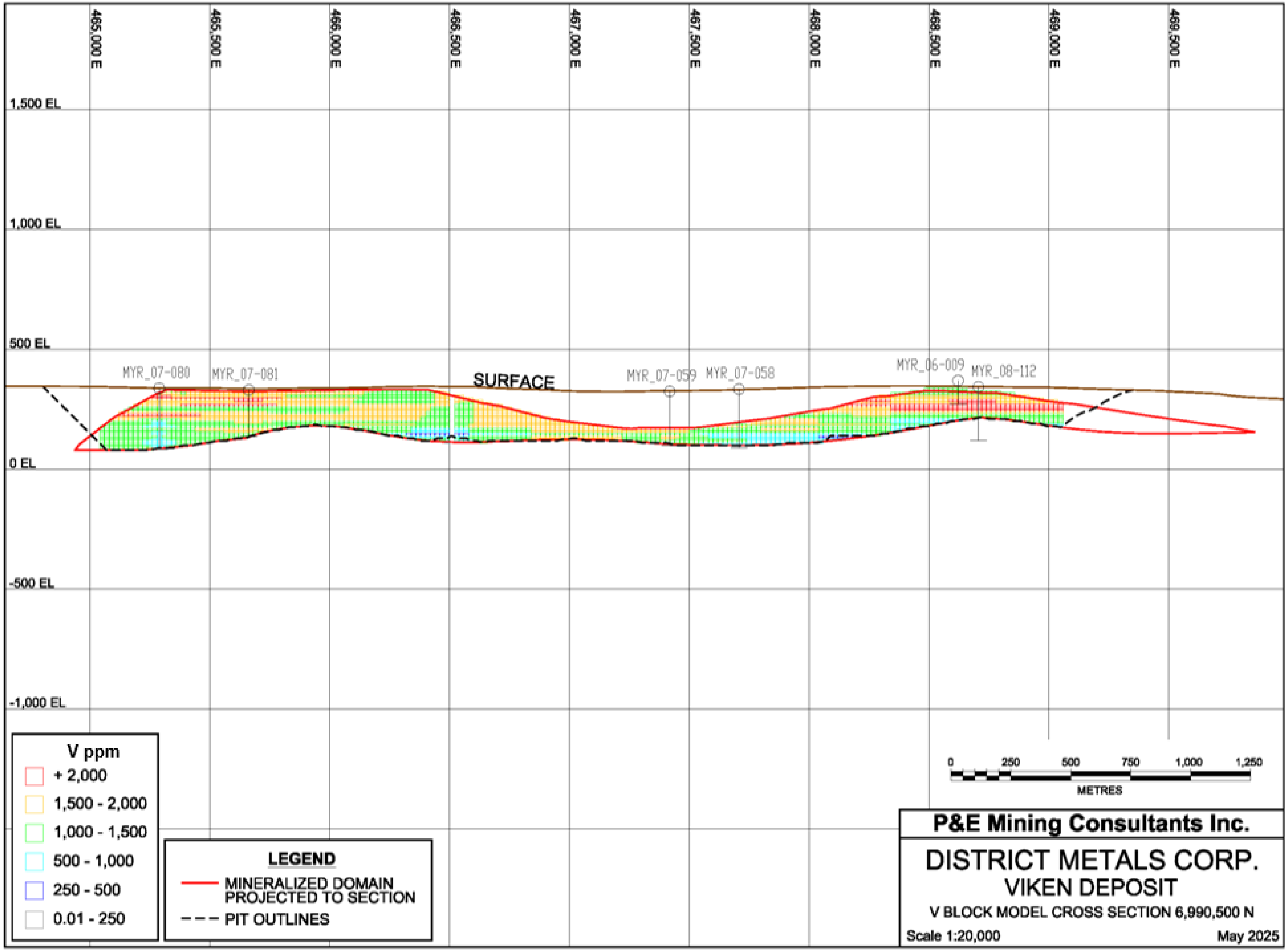
LEGEND
MINERALIZED DOMAIN PROJECTED TO SECTION
PIT OUTLINES
0 250 500 750 1,000 1,250
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
V BLOCK MODEL CROSS SECTION 6,988,500 N
Scale 1:20,000
May 2025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SURFACE
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MYR_07-058
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LEGEND
MINERALIZED DOMAIN PROJECTED TO SECTION
PIT OUTLINES
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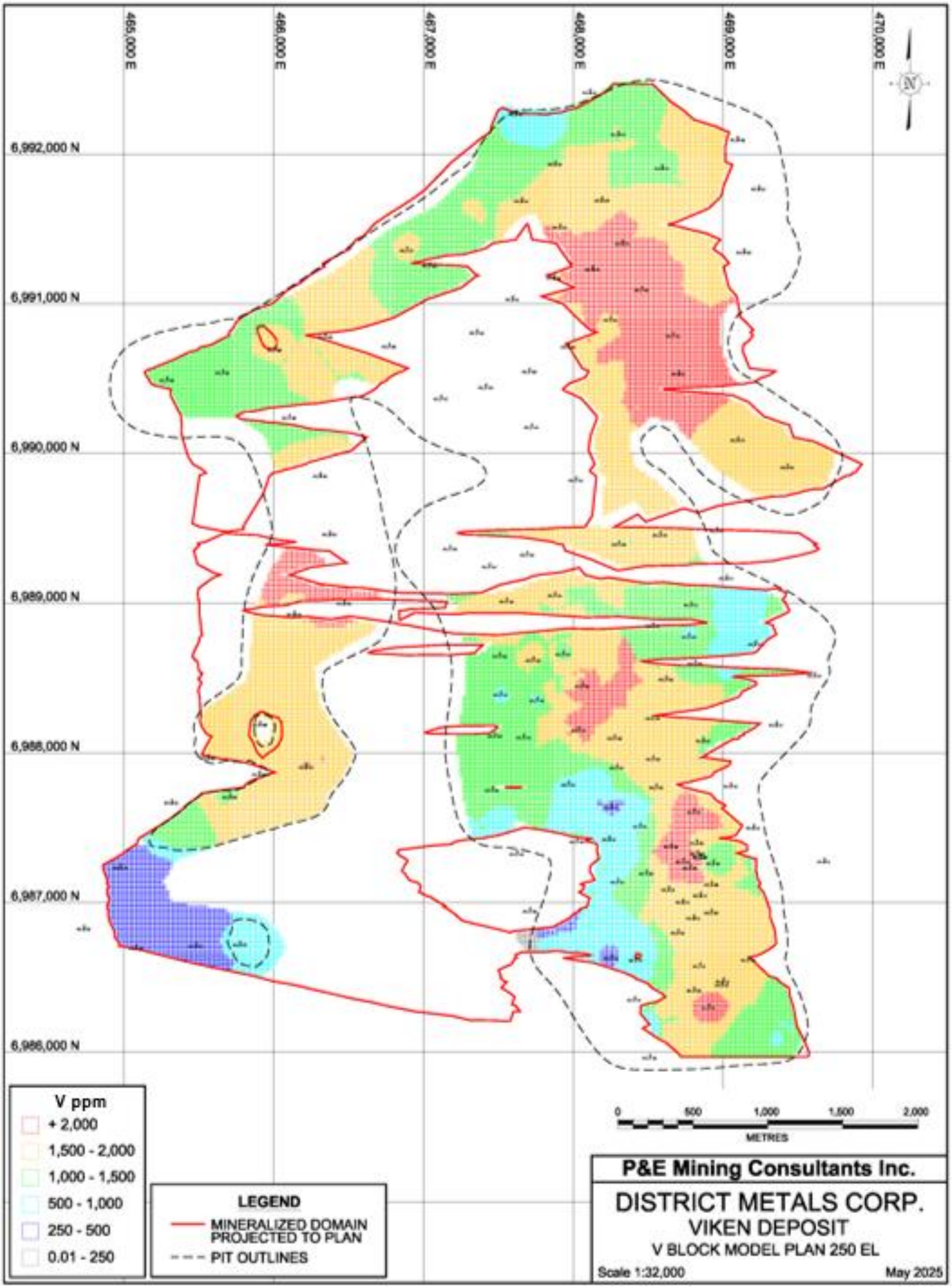
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
V BLOCK MODEL CROSS SECTION 6,990,500 N
Scale 1:20,000
May 2025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V ppm
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250 - 500
0.01 - 250
LEGEND
MINERALIZED DOMAIN PROJECTED TO PLAN
PIT OUTLINES
0
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1,500
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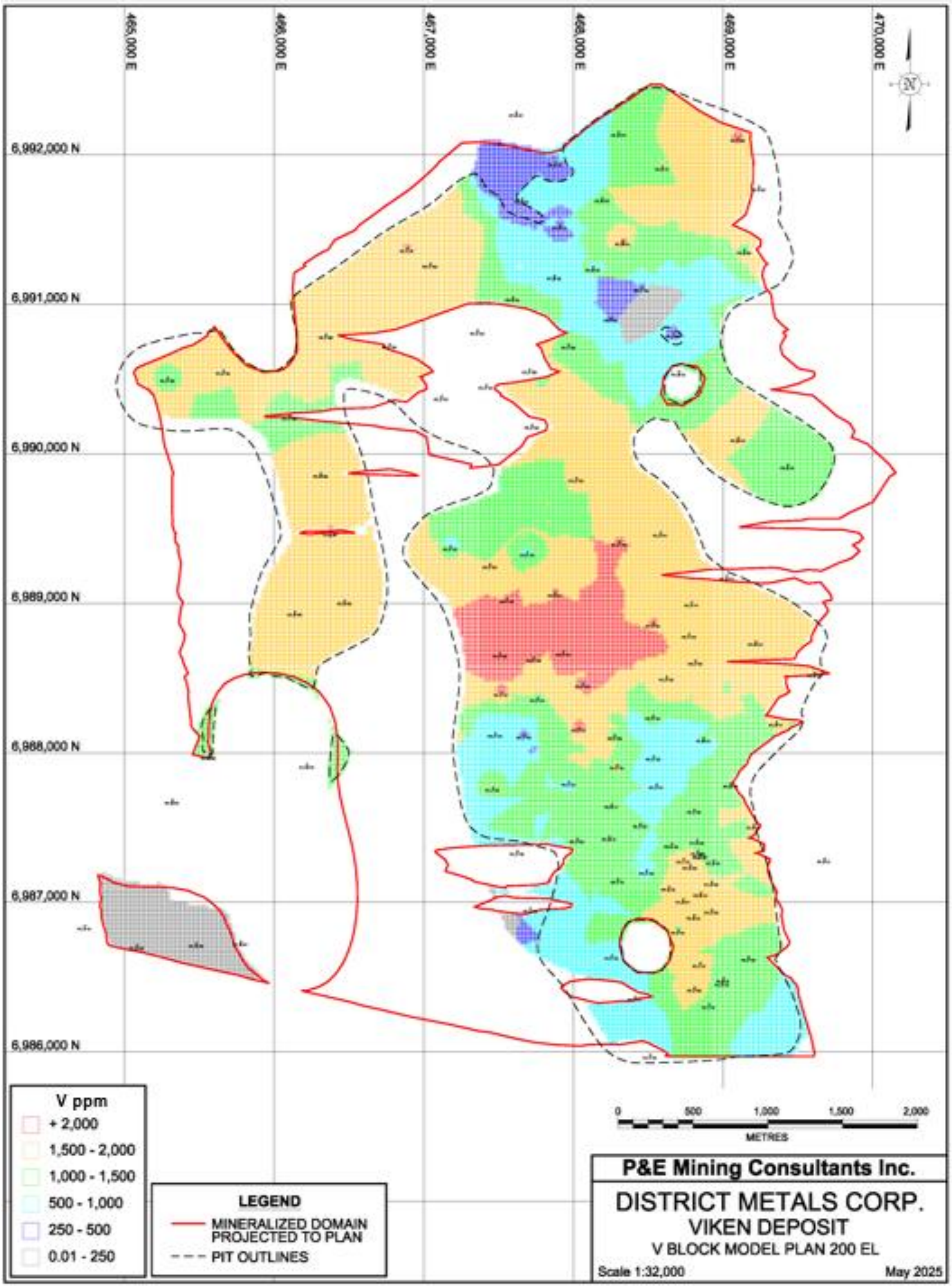
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
V BLOCK MODEL PLAN 250 EL
Scale 1:32,000
May 2025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LEGEND
MINERALIZED DOMAIN PROJECTED TO PLAN
PIT OUTLINES
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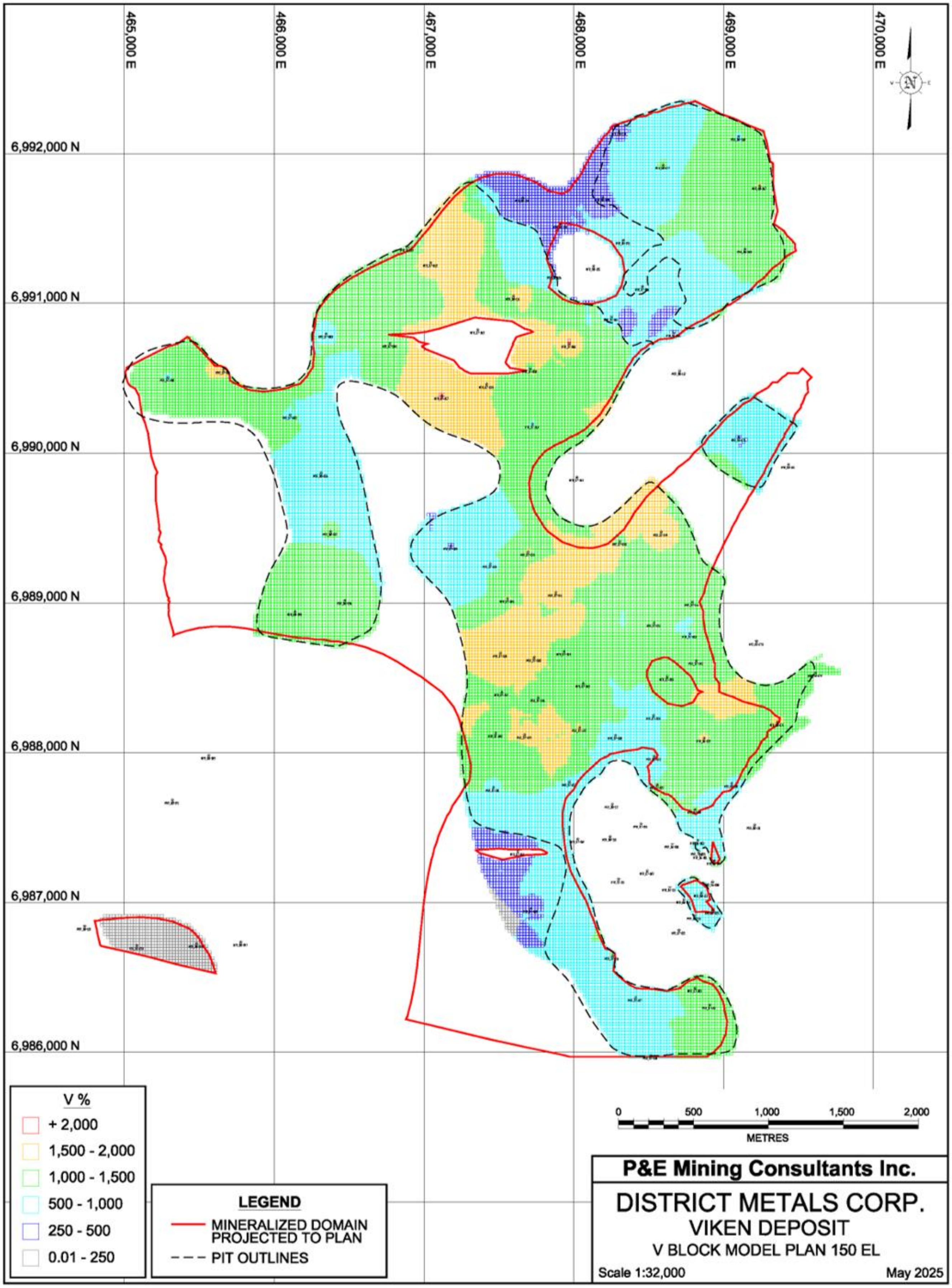
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
V BLOCK MODEL PLAN 200 EL
Scale 1:32,000
May 2025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LEGEND
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PIT OUTLINES
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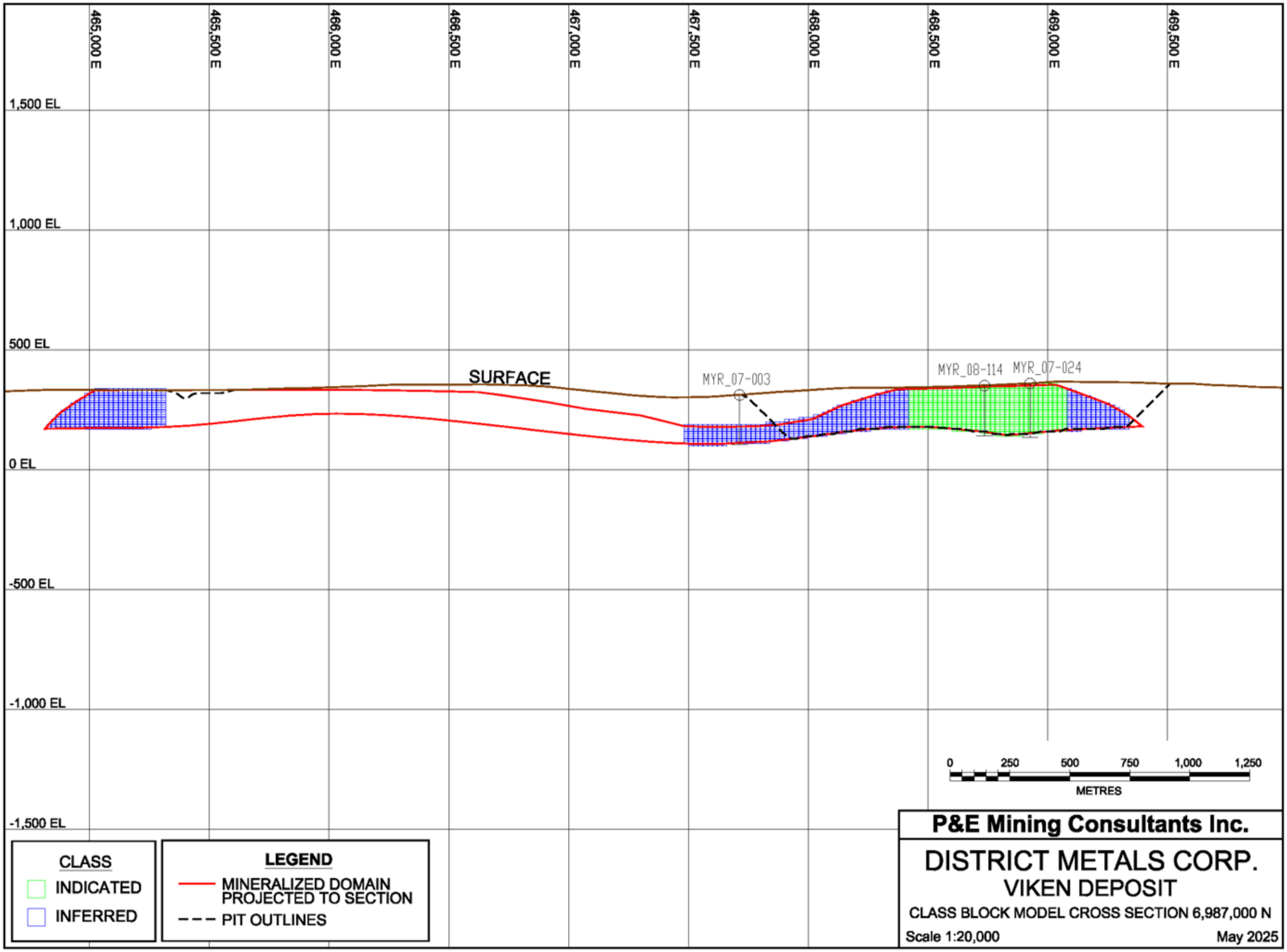
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
V BLOCK MODEL PLAN 150 EL
Scale 1:32,000
May 2025

APPENDIX E CLASSIFICATION BLOCK MODEL CROSS-SECTIONS AND PLANS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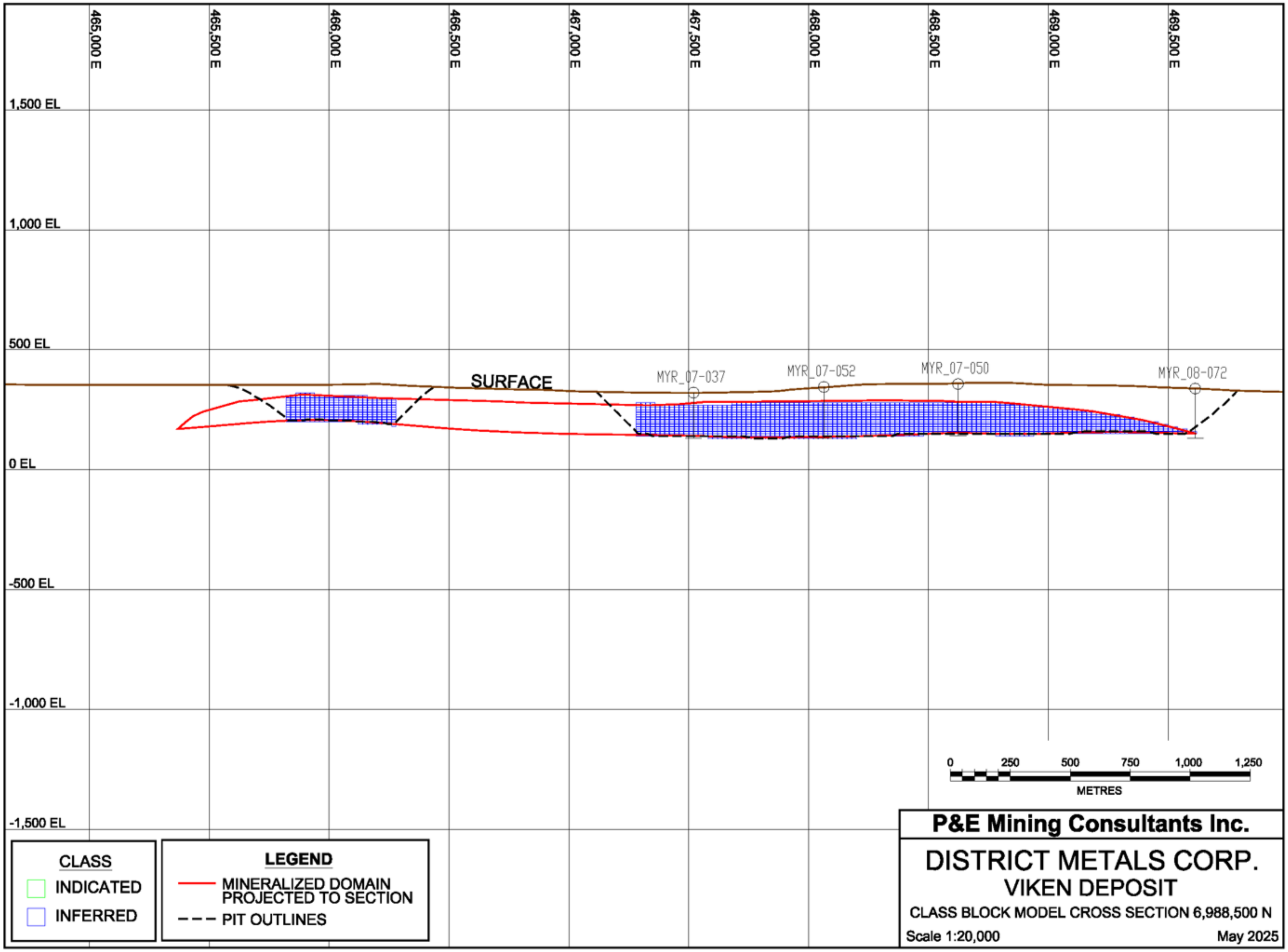
500 EL
0 EL
-500 EL
-1,000 EL
-1,500 EL
SURFACE
MYR_07-003
MYR_08-114
MYR_07-024
0
250
500
750
1,000
1,250
METRES
CLASS
INDICATED
INFERRED
LEGEND
MINERALIZED DOMAIN PROJECTED TO SECTION
PIT OUTLINES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
CLASS BLOCK MODEL CROSS SECTION 6,987,000 N
Scale 1:20,000
May 2025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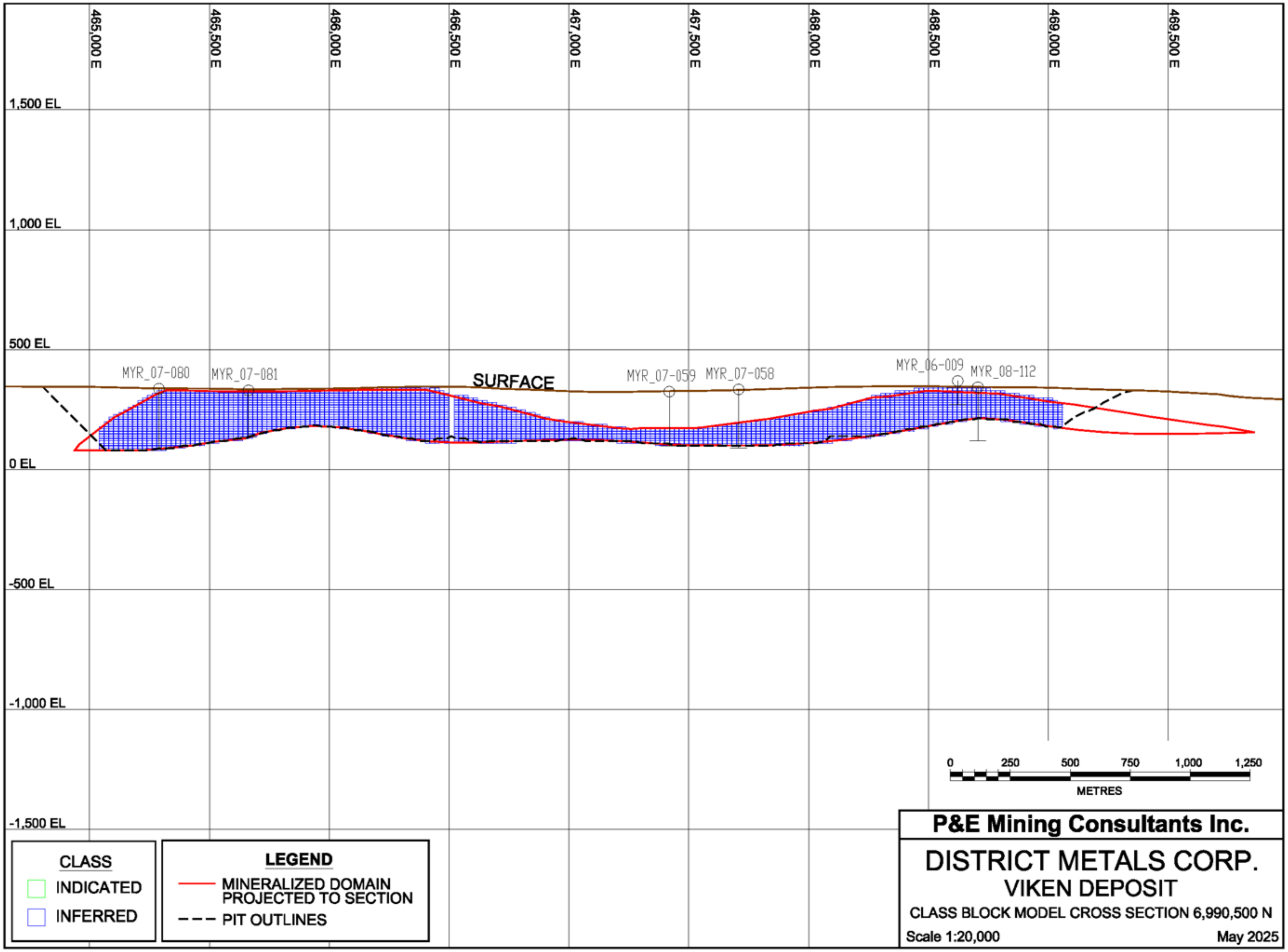
-1,500 EL
SURFACE
MYR_07-037
MYR_07-052
MYR_07-050
MYR_08-072
0
250
500
750
1,000
1,250
METRES
CLASS
INDICATED
INFERRED
LEGEND
MINERALIZED DOMAIN PROJECTED TO SECTION
PIT OUTLINES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
CLASS BLOCK MODEL CROSS SECTION 6,988,500 N
Scale 1:20,000
May 2025



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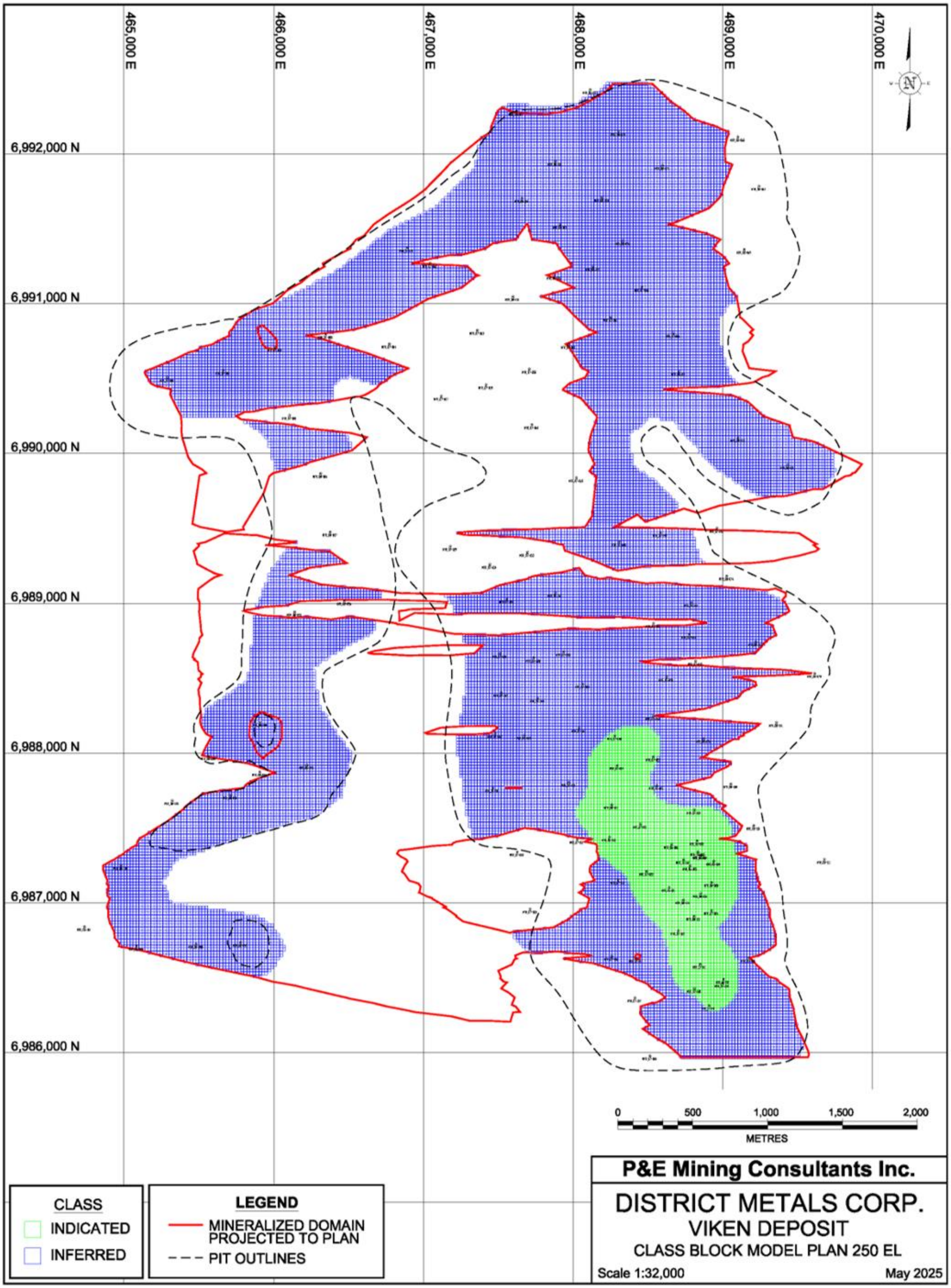
1,000 EL
500 EL
0 EL
-500 EL
-1,000 EL
-1,500 EL
MYR_07-080
MYR_07-081
SURFACE
MYR_07-059
MYR_07-058
MYR_06-009
MYR_08-112
0
250
500
750
1,000
1,250
METRES
CLASS
INDICATED
INFERRED
LEGEND
MINERALIZED DOMAIN PROJECTED TO SECTION
PIT OUTLINES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
CLASS BLOCK MODEL CROSS SECTION 6,990,500 N
Scale 1:20,000
May 2025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467,000 E
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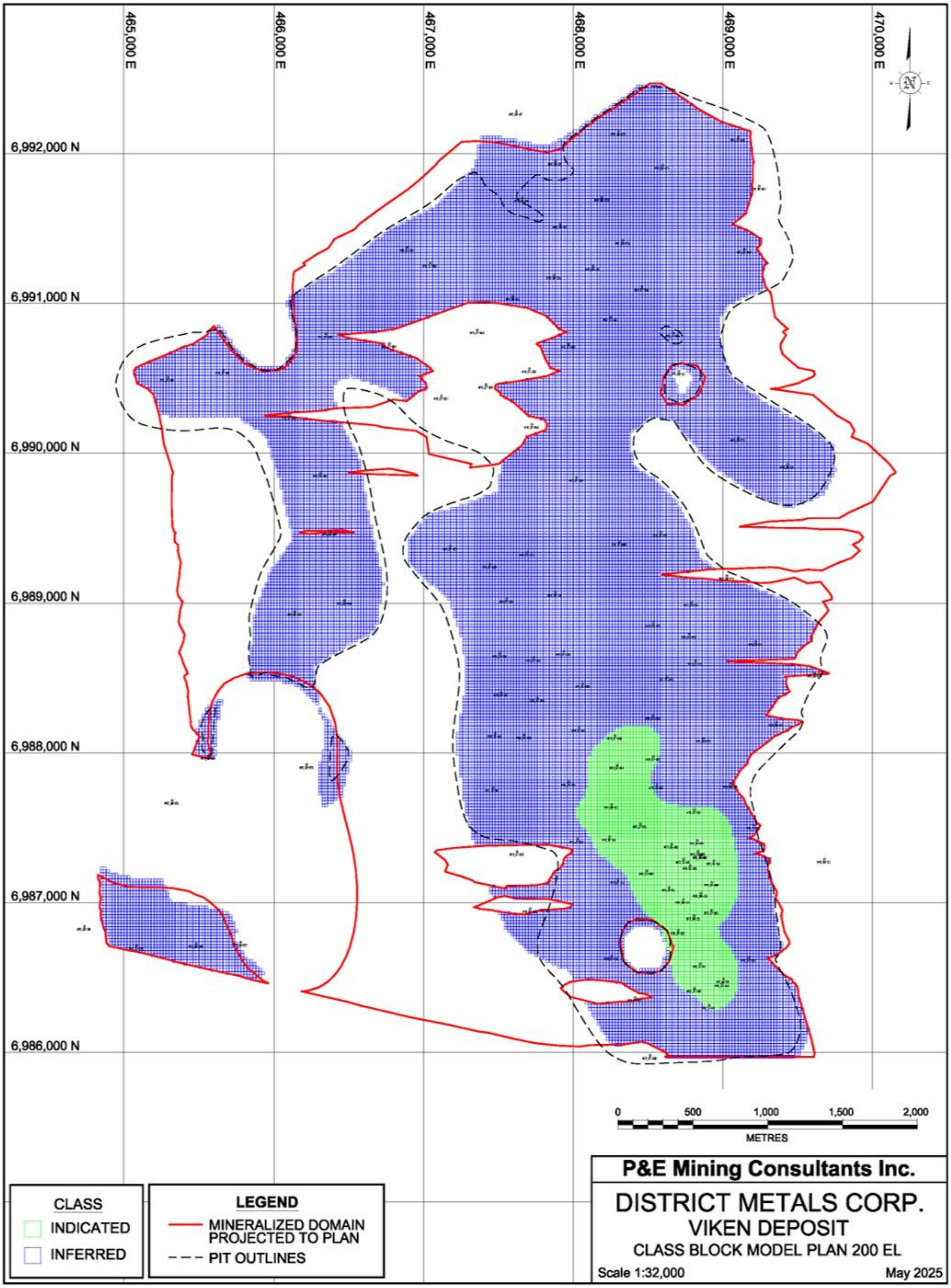
469,000 E
470,000 E
6,992,000 N
6,991,000 N
6,990,000 N
6,989,000 N
6,988,000 N
6,987,000 N
6,986,000 N
N
0
500
1,000
1,500
2,000
METRES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
CLASS BLOCK MODEL PLAN 250 EL
Scale 1:32,000
May 2025
CLASS
INDICATED
INFERRED
LEGEND
MINERALIZED DOMAIN PROJECTED TO PLAN
PIT OUTLINES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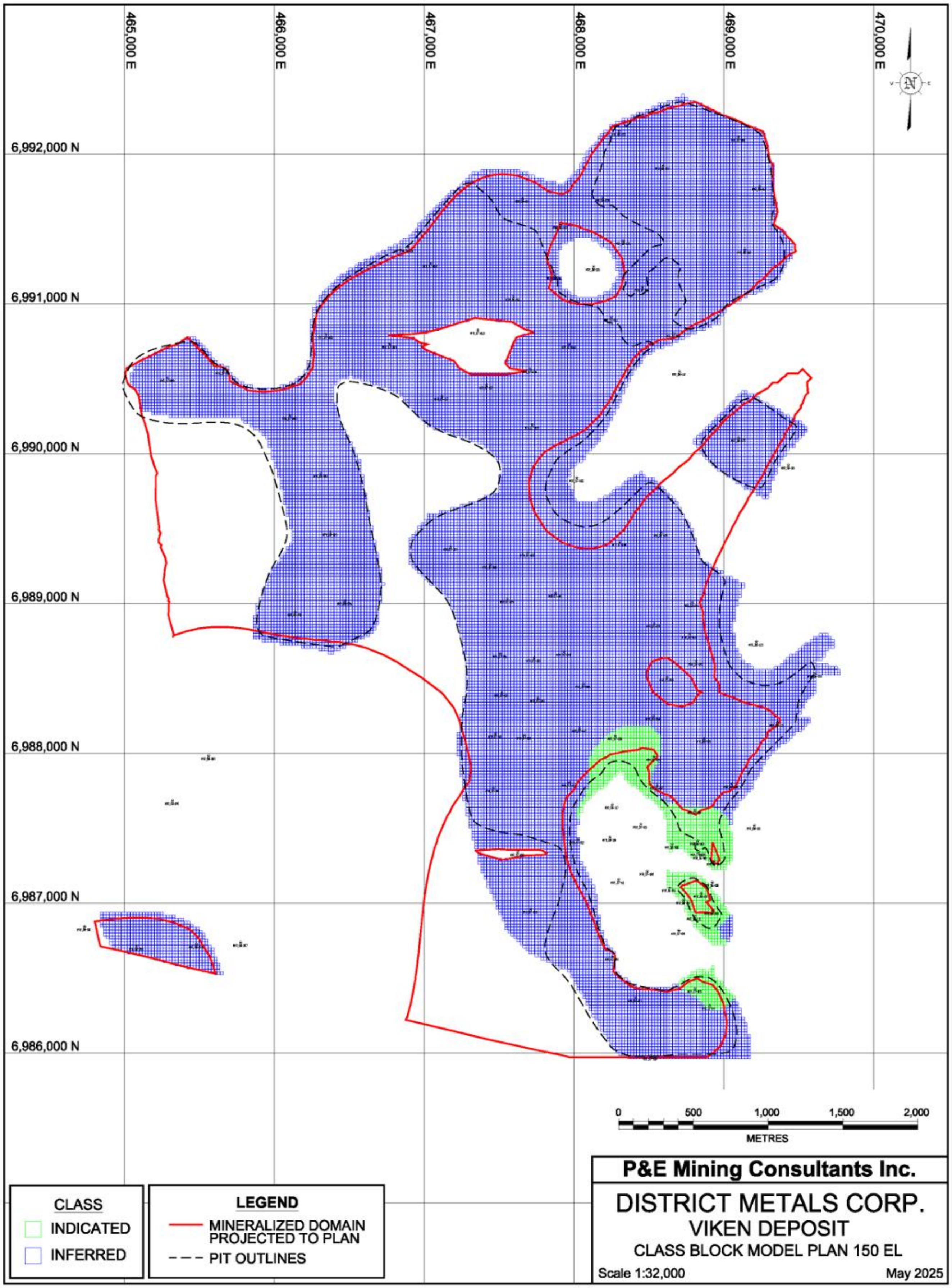
6,988,000 N
6,987,000 N
6,986,000 N
0
500
1,000
1,500
2,000
METRES
CLASS
INDICATED
INFERRED
LEGEND
MINERALIZED DOMAIN PROJECTED TO PLAN
PIT OUTLINES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
CLASS BLOCK MODEL PLAN 200 EL
Scale 1:32,000
May 2025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466,000 E
467,000 E
468,000 E
469,000 E
470,000 E
6,992,000 N
6,991,000 N
6,990,000 N
6,989,000 N
6,988,000 N
6,987,000 N
6,986,000 N
N
0
500
1,000
1,500
2,000
METRES
CLASS
INDICATED
INFERRED
LEGEND
MINERALIZED DOMAIN PROJECTED TO PLAN
PIT OUTLINES
P&E Mining Consultants Inc.
DISTRICT METALS CORP.
VIKEN DEPOSIT
CLASS BLOCK MODEL PLAN 150 EL
Scale 1:32,000
May 2025

APPENDIX F OPTIMIZED PIT SHELL

VIKEN DEPOSIT - OPTIMIZED PIT SHELL