



NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland

Effective Date: January 9, 2026

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Report prepared by Micon International Co Limited

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1 SUMMARY

1.1 OVERVIEW

This Technical Report (the Report) has been prepared for Lumina Metals Corp. (Lumina) in accordance with the disclosure requirements set forth under National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101). Lumina, a British Columbia corporation formerly operating as Miedzi Copper Corporation, holds 100% ownership of the Nowa Sól Copper-Silver Project (the Project or the Property) through its wholly owned subsidiary Zielona Góra Copper sp. z o.o.

Lumina engaged Micon International Co Limited (Micon) to provide a Preliminary Economic Assessment (PEA) Technical Report in accordance with NI 43-101 on the Project located within Lubuskie Province, Poland. Nowa Sól is an advanced-stage copper and silver exploration project.

The purpose of this Report is to provide an independent technical assessment of the Project's potential economic viability, including a summary of historical exploration, current geological data, and Mineral Resource Estimate (MRE). No Mineral Reserves are presented in this Report. Based on the results within this Report and demonstrated potential project economic viability, the authors recommend that the Project advances a focused program of technical, environmental and engineering studies to support progression toward a Pre-Feasibility Study (PFS).

1.2 INTRODUCTION

The Qualified Persons (QPs) responsible for the preparation of this Report are Liz de Klerk, Pr.Sci.Nat., QMR (Micon Managing Director); Dr. Ryan Langdon, Ph.D, CGeol; Richard Gowans P.Eng; Garth Matti Liukko, P.Eng.; Ben Cottrell P.Eng; Alex Zaitchenko P.Eng; Peter Stevens CGeol; Justin Taylor, P.Eng.; Becky Humphrey CEnv, MIMMM; and Christopher Jacobs CEng. MIMMM. All the QPs meet the competency and independence criteria required under NI 43-101 and have confirmed that no conflict of interest exists with Lumina or the Property. A site inspection was completed by Liz de Klerk from February 12 to 14, 2025, including visits to key deposit areas, drill core storage facilities, and meetings with Lumina personnel.

The Report is based on a comprehensive review of historical and current technical data provided by Lumina and its representatives. Where necessary, publicly available regional geological information and reports from previous owners were used. Although several historical MREs are cited (most recently from 2024) these are not treated as current MREs under NI 43-101 due to the absence of adequate verification by the QPs.

All information included herein is considered valid and accurate as of the effective date of the Report. However, Micon acknowledges that future exploration, operational, regulatory, and market factors may affect the forward-looking nature of certain statements. The QPs believe that all opinions and interpretations contained in this Report are reasonable, objective, and aligned with industry best practices.

1.3 RELIANCE ON OTHER EXPERTS

The QPs have relied on information provided by Lumina concerning legal matters relevant to the Report including permitting and concession status. In connection with these matters, the QPs have relied upon official concession details for two contiguous exploration concessions, namely Nowa Sól II (9/2025/p) and Konotop (12/2025/p), obtained from the Ministry of Climate and Environment (MCE) dated October 28, 2025 for Nowa Sól and December 9, 2025 for Konotop, and this information is used in Section 20.1.4. of this Report.

1.4 PROPERTY DESCRIPTION AND LOCATION

The Project is an advanced-stage copper and silver exploration project located in western Poland, within the Lubuskie Province. The Property lies approximately 320 km west of the capital city of Warsaw and is situated within the 120 km² Nowa Sól II exploration concession which spans the administrative districts of Nowa Sól, Wschowa, and Zielona Góra. The Project area is characterized by lowland forested plains ranging from 60 to 110 metres above sea level (masl), with favorable logistical proximity to urban centres and infrastructure.

1.1 Mineral Tenure and Regulatory Framework

Mineral exploration and extraction in Poland are governed by the Geological and Mining Law of June 9, 2011, as amended. Mineral rights are distinct from surface rights and are administered through concessions issued by the MCE. The system recognizes separate concessions for exploration and mining, with additional legal instruments including the mining usufruct agreement, geological documentation, exclusivity rights to geological data, and priority rights to apply for production concessions. Concessions are time-bound, renewable, and subject to regulatory compliance and public consultation.

Under this legal regime, geological documentation of a project must attain at least a C₁ resource classification for a project to proceed toward an application for a mining concession. The geological documentation for the Project was approved by the MCE in February 2020, with Addendum No. 1 accepted in January 2023. Addendum No. 2, submitted in December 2024 to upgrade the classification to C₁, was also approved on April 28, 2025. As a result, Lumina holds a five-year exclusivity and priority right to apply for a mining concession.

1.2 Ownership and Exploration History

Lumina, through its Polish subsidiaries, holds 100% of the Project. Exploration was conducted across four contiguous concessions: Nowa Sól, Zatonie, Jany, and Wilcze, covering a combined area of 354 km². While three of these concessions expired in 2022, the Nowa Sól concession remained active until the end of 2024. This concession has been replaced by a new exploration concession, Nowa Sól II, covering 120 km² which was granted on October 28, 2025 for a period of 3.5 years. This concession now forms the core of the Project, in addition to a contiguous exploration concession located to the northeast, named Konotop, which covers 40 km² and was granted on December 9, 2025 for a period of six years. Exploration efforts between 2011 and 2024 led to the delineation of a sediment-hosted copper-silver deposit. The mineralization discovered within this area has since been classified as C₁ under Polish regulations, aligned with the Measured Mineral Resources category under CRIRSCO standards.

1.3 Surface and Water Rights

Lumina has secured temporary surface access rights through lease agreements with private, public, and corporate landowners. Future mining operations will require secure land titles for infrastructure development. Water use will require a separate permit, to be secured after technical studies have been completed and once the mining concession is granted.

1.4 Environmental and Legal Considerations

There are currently no royalties applied to the Project; however, any future production will be subject to standard exploitation fees and Mineral Exploitation Tax (MET) as mandated under Polish law. The Project does not currently have any material contracts in place.

There are several international and nationally designated Protected Areas in proximity to the Project, which will be avoided by design and are therefore not within the proposed surface footprint. A full review of the potential environmental and social impacts will be undertaken as part of the future Environment Impact Assessment (EIA) process. All former drill sites have been remediated, and Lumina holds no outstanding environmental liabilities.

1.5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

1.5 Accessibility

The Project is located approximately 320 km west of the Warsaw capital, 100 km northwest from the city of Wrocław, 85 km east of the border with Germany and 120 km north of the border with Czechia. The western boundary of the concession is located approximately 25 km from the city of Zielona Góra, the primary transportation and service hub of the region. The Project area is accessible all year-round via a well-maintained network of public roads. The S3 expressway that crosses the county to the west of the Project, links the region to key seaports. Rail links provide both passenger and freight connections to Zielona Góra, seaports and other regional hubs in Poland and Europe, and a decommissioned railroad (Line No. 371 Wolsztyn-Żagań) crosses the northern portion of the Property. The Babimost (IEG) international airport, is situated 20 km from the Property offering connectivity with Warsaw, the capital city. Other international airports located in the relative vicinity of the Project are: Poznań (POZ) 100 km away, Wrocław (WRO) 100 km away, Berlin (BER) 180 km away, and Warsaw (WAW) 320 km away, offering a wider range of direct commercial flights.

1.6 Climate, Physiography and Drainage

The climate in the Lubuskie Province is classified as temperate oceanic, with distinct seasons and average temperatures ranging from around -3°C in winter (January) to 25°C in summer (July). Rainfall occurs year-round and peaks in summer months, with average annual totals of around 600 mm to 650 mm. Conditions are relatively mild compared to other areas of Poland, and Project operations can be conducted year-round.

The area surrounding the Project is characterized by rivers, lakes and forests. The Project is located in the middle reaches of the Odra River basin and the river is 1 km from the western concession boundary at its closest point. The topography is relatively flat, with gently undulating lowlands that form part of the larger Polish Plain and elevations ranging from 60 masl to 110 masl.

The Nowa Sól area is integrated into Poland's national energy grid. Two major power lines run north-to-south near the Project: a 220 kV line to the west, plus a 110 kV line to the east. The Odra River located adjacent to the west of the deposit area offers potential for inland shipping.

The region borders the Polish Copper Mining District (*Zagłębie Miedziowe*, Legnica-Głogów Mining District (LGOM)) to the south. This area hosts KGHM Polska Miedź S.A. (KGHM) a global copper mining and refining company that operates three mines, two smelters, and extensive supporting infrastructure. The nearest smelter is located in the city of Głogów, approximately 25 km south of the Project.

Nowa Sól is also part of the Kostrzyn-Słubice Special Economic Zone, offering preferential regulations and corporate income tax exemptions to investors. In addition, the region also benefits

from EU funding for regional development, which supports projects in infrastructure, education, and innovation.

1.6 HISTORY

The Project is located within the Kupferschiefer (copper-bearing black shale) ore-bearing sequence of southwestern Poland, a formation of considerable historical mining significance. While documented copper mining in the historic region of Lower Silesia dates back to the 14th century, archaeological evidence suggests prehistoric extraction activities may have occurred during the Bronze Age. However, no previous metal mining has taken place specifically on the Project area or in the current Lubuskie Province.

1.7 Historical Drilling and Sampling

All historical exploration on the Property has been conducted through diamond drilling due to the significant depth of mineralization, which begins at approximately 1,700 m below the surface. No surface-based sampling, trenching, or artisanal prospecting has been undertaken. Sampling of historical drill core involved continuous sampling of quarter core that had been cut along its axis, across the mineralized interval, which included the top of the Weissliegend, the basal limestone, the copper-bearing shale, and the base of the Zechstein limestone.

Only one historical drill hole, Bojadła 1 (drilled in 1986), intersected the mineralized horizon. Core sampling was conducted across the Rotliegend–Zechstein stratigraphic boundary, including the principal mineralized interval. The sampling program was designed to reflect lithological boundaries, mineralization styles, and oxidation features observed in the drill core.

Systematic continuous quarter-core sampling was undertaken within the main mineralized zone regardless of visible mineralization, with sample lengths typically ranging from 5 cm to 20 cm and reduced where required to preserve geological resolution. In weakly mineralized or barren intervals, longer continuous samples or representative point and composite samples were collected in accordance with lithological homogeneity and geological confidence.

Sampling protocols ensured minimum analytical masses of 50 g for routine analyses and 35 g for gold and platinum-group element determinations. All sample preparation and analytical testing were completed at the accredited Central Chemical Laboratory of the Polish Geological Institute – National Research Institute (PIG-PIB) in Warsaw, in compliance with ISO/IEC 17025 standards.

1.8 Historical Resource Estimation

Between 2014 and 2024, Lumina undertook systematic drilling and geological evaluation resulting in a series of historical MREs. These were prepared under the Polish Resource Classification System, which is not recognized under NI 43-101 or aligned with CIM Definition Standards, but which is a state-regulated, statutory system governed by the MCE. The historical milestones include:

- May 3, 2014: Initial discovery at Nowa Sól with hole NS_C1.
- 2014 to 2020: completion of 35,416.20 m of drilling within the Nowa Sól area.
- February 12, 2020: Initial C₂ classification approved by the MCE.
- January 13, 2023: Approval of Addendum No. 1, expanding and partially upgrading resources to C₁.
- 2024: Additional drilling (holes C27, C27st, C29, C31, C31st) targeting a full upgrade to C₁ classification.

- April 28, 2025: Approval of Addendum No. 2, officially classifying the resource as C₁ and granting Lumina the associated exclusivity and priority rights.

No historical Mineral Reserve Estimates have ever been declared for the Property.

1.7 GEOLOGICAL SETTING AND MINERALIZATION

The Project is located within the Polish segment of the Central European Basin, specifically within the Fore-Sudetic Monocline, a geotectonic setting known for hosting significant stratabound copper-silver mineralization within the Zechstein Group. The basin comprises a complex sequence of Permian-age sedimentary and volcanic units, which include the Lower Rotliegende redbeds and volcanics and the overlying Upper Permian Zechstein marine evaporites.

1.9 Regional and Stratigraphic Geology

The Central European Basin formed during Late Carboniferous to Early Permian extensional tectonism, followed by extensive sedimentation in arid to semi-arid environments. Marine transgression during the Upper Permian led to the formation of the Zechstein Sea, which deposited a cyclic stratigraphy of carbonates and evaporites. The lowermost Zechstein cyclothem, the Werra Cycle (PZ1), hosts the copper-silver mineralization within the Kupferschiefer horizon and adjacent lithologies. In the Nowa Sól area, the mineralized sequence is thin, typically under 10 m, and includes the Upper Rotliegende sandstone, Kupferschiefer black shale, and Zechstein limestone.

1.10 Local Geology and Mineralization

The Nowa Sól deposit is a concealed stratiform copper-silver deposit located ~30 km northwest of the LGOM. Mineralization is hosted within three principal lithologies: the Weissliegende and Grauliegende sandstones (sandstone ore), the Kupferschiefer shale (shale ore), and the Zechstein carbonate units (carbonate ore). These host rocks developed under differing depositional environments, ranging from arid continental to shallow epicontinental marine conditions. The ore assemblage includes chalcocite, bornite, chalcopyrite, djurleite, and covellite, with accessory sulphides and oxides including galena, sphalerite, tennantite pyrite, and native silver. The highest copper grades occur proximal to redox boundaries at the base of the Kupferschiefer horizon. Mineralization is zoned vertically and laterally and is associated with a classic metal zonation sequence transitioning from copper to lead, zinc, and pyrite-bearing distal zones.

1.11 Paragenesis and Metallogenesis

Mineralizing fluids were derived from deep-seated sources mobilized during multiple tectonic events. Initially, metal-charged fluids migrated through the Rotliegende sediments and were trapped beneath the Kupferschiefer black shale aquiclude, where they precipitated metals due to chemical interactions with reactive host rocks. A secondary mineralizing event during the Palaeogene (~53 Ma) introduced localized overprinting by high-grade breccia and vein-type mineralization associated with magmatic activity and fault reactivation.

1.12 Deposit Genesis and Age

The Kupferschiefer deposits are interpreted as classic diagenetic-replacement type systems, formed under reducing conditions within confined sedimentary basins. Recent paleomagnetic dating constrains the primary mineralization at ~149 Ma (Late Jurassic), with a secondary overprint event dated at ~53 Ma. These findings post-date traditional Triassic interpretations and reflect a complex, multi-phase metallogenic evolution.

1.8 DEPOSIT TYPES

The Nowa Sól deposit is classified as a Kupferschiefer-type system, representing a distinctive subset of Sediment-Hosted Stratiform Copper Deposits. This deposit type is characterized by laterally extensive, thin mineralized zones localized at redox interfaces within sedimentary basins. The deposit model is defined by metal-bearing fluids interacting with stratified sedimentary sequences, particularly at the Lower–Upper Permian transition.

Kupferschiefer-type deposits, including Nowa Sól, are developed within the Central European Basin, a post-orogenic intracratonic basin, and are typically constrained to an arcuate belt overlying the Mid-European Crystalline High. These deposits form through multistage processes, beginning with early diagenetic mineralization followed by later hydrothermal overprints driven by tectonic reactivation. The genesis involves both deep convective flow and lateral migration of oxidized, metal-rich chloride brines, with deposition occurring at chemical traps such as redox fronts.

The mineralization style is zoned, with proximal copper-silver cores grading outward into lead, zinc, and distal iron assemblages. Enrichment in gold and platinum group elements can occur near the oxidized boundary (Rote Fäule), where redox gradients are sharpest. Importantly, the deposit architecture is stratigraphically controlled, with mineralization confined to a narrow interval straddling the Permian unconformity and capped by impermeable evaporite seals.

Comparable Kupferschiefer-type deposits include the historical Mansfeld and Spremberg districts in Germany, as well as Poland’s Lubin, Rudna, and Polkowice-Sieroszowice mines. The consistent structural setting, fluid evolution pathways, and metal zonation patterns reinforce the applicability of this deposit model to the Nowa Sól system.

1.9 EXPLORATION

Since entering Poland, Lumina has implemented a systematic, large-scale exploration program aimed at identifying sediment-hosted copper-silver deposits within the Fore-Sudetic Monocline. It was the largest privately funded metal exploration program nationwide, which continues until today. The approach emphasized predictive modelling, integration of historical datasets, and deep-target exploration technologies to identify concealed mineral systems.

The initial exploration phase involved extensive desktop studies, including the analysis of 411 historical drill holes, reprocessing of 1,693 line-km of seismic data, and acquisition of a 3D geophysical survey. Complementary datasets, including magneto-telluric surveys, gravity data from 23,754 stations, and geological information from national geological archives, were consolidated into a GIS-based platform. This multi-disciplinary review culminated in the selection of eight high-priority target areas, including the Nowa Sól deposit. The exploration information reviewed by Lumina is a regional data set and included exploration results from outside of the Nowa Sól licence area. The use of the historical information as an input to the MRE presented in this Report is discussed in Sections 10, 14 and 23.

Exploration at the Project has been uniquely tailored to the depth of mineralization (1,700 m to 2,200 m), precluding conventional surface techniques. Instead, Lumina used gravity and seismic data interpretation to define stratigraphic markers and infer favourable depositional environments. Key stratigraphic surfaces such as the Uppermost Rotliegende and Top Zechstein were mapped using KINGDOM ADVANCED software. A palaeo-topographic model corrected for tectonic deformation was developed to refine subsurface targeting.

No surface exploration was undertaken on the Property due to the depth of the orebody. Instead, Lumina’s exploration strategy focused on leveraging structural, stratigraphic, and geophysical indicators to define high-probability zones for drill testing.

In the opinion of the QPs, the exploration work completed by Lumina has been technically sound, geologically appropriate, and consistent with best practices for deep-seated sediment-hosted mineral systems. The integration of historical data with modern geophysical analysis provided a reliable framework for drill targeting and resource delineation.

1.10 DRILLING

Drilling undertaken by Lumina between 2014 and 2024 has been central to the delineation of the Nowa Sól copper-silver deposit. The drilling strategy was tailored to target deep stratiform mineralization hosted in the Kupferschiefer horizon, typically encountered between 1,790 m and 2,167 m below the surface.

1.13 Lumina Drilling Campaign (2014 to 2024)

Lumina completed 26 deep drill holes totaling 51,413 m, using both rotary and diamond core drilling techniques. Three independent contractors (EXALO Drilling S.A. (Exalo), GEOPS Deep Drilling, and G-Drilling S.A.) executed the campaigns utilizing high-capacity rigs suitable for depths exceeding 2,500 m. The drilling program was supported by real-time monitoring and geotechnical controls and included site remediation and environmental compliance following completion.

Diamond core drilling targeted the mineralized sequence from the base of the Lower Anhydrite (A1d) into the Lower Permian sandstones. Core recovery was consistently high, averaging 98%. Where core recovery in the mineralized horizon fell below acceptable thresholds, re-entry and sidetracking were executed to ensure complete recovery.

1.14 Sampling and Logging

Systematic core logging was performed at Lumina's Kraków facility, in Poland, supported by detailed photographic records, XRF screening, and standardized digital logging templates. Sample intervals were lithologically constrained, ranging from 0.04 m to 1.36 m. Quarter-core samples were prepared for laboratory analysis, while the remaining core was retained for archiving. All sampling procedures adhered to regulatory standards, including submission of material to the Polish Geological Institute.

Collar and downhole surveys were conducted by certified contractors using high-precision Global Positioning System (GPS) and wireline geophysical tools. Positional accuracy was within ± 0.005 m horizontally and ± 0.010 m vertically. Downhole surveys included gamma, azimuth, inclination, and temperature data, enabling precise spatial control of the mineralized intercepts.

1.15 Drilling Results and Interpretation

Copper and silver mineralization was encountered in 23 of the 26 drill holes, with mineralized intervals ranging from 0.71 m to 4.97 m. All drill holes were collared vertically and based on the low-angle dip of the host lithologies, the reported intercepts are considered to represent true thicknesses. No material sampling or recovery issues are known to exist, and the data are regarded as accurate and reliable.

1.16 QP Opinion

The QPs have reviewed the drilling methodology, recovery, logging, and sampling protocols and are of the opinion that all practices align with industry best standards. There are no known drilling or sampling factors that could materially affect the accuracy or reliability of the data as of the effective date of this Report.

1.11 SAMPLE PREPARATION, ANALYSES AND SECURITY

1.11.1 Sampling of Historical Drill Core

Historical core from holes both inside and outside the Nowa Sol licence area was sampled by Lumina staff between 2011 and 2012, with analyses conducted at Acme Analytical Laboratories Ltd (Acme Labs) in Vancouver. Quality Assurance / Quality Control (QA/QC) procedures included insertion of in-house standards, blanks and duplicates. One historical drill hole, Bojadła 1, within the Project license boundary was incorporated into the MRE, with Lumina purchasing official analytical results from *Państwowy Instytut Geologiczny*, Polish Geological Institute (PIG) in March 2015, having been unable to perform their own sampling for analytical work. The sample preparation, analysis and security of Bojadła_1 is described in Section 10.

1.11.2 Sample Security and Chain of Custody

During drilling, Lumina geologists oversaw all operations, maintaining core security and a full chain of custody. Core boxes were stored in locked steel containers and transported by Lumina staff to secure storage in Kraków. Samples for analysis were hand-delivered to Acme Labs under supervision, ensuring sample integrity. Assay rejects and pulps, as well as selected sulphide-bearing intervals, were preserved in a secure facility for future studies.

1.11.3 Sample Preparation and Analysis

Sample preparation was conducted at Acme Labs in Kraków and analytical work at Bureau Veritas in Vancouver, both accredited laboratories. Procedures included oven drying, crushing, splitting, and pulverising to industry standards. Analytical methods comprised Aqua Regia Inductively Coupled Plasma Mass Spectrometry (ICP-MS), four-acid digestion Inductively Coupled Plasma Optical Emission Spectroscopy (ICP-OES), and fire assay. QA/QC was maintained through laboratory-inserted blanks, standards and duplicates. Lumina also conducted in-house density measurement using the Waxed Air/Water method, supporting resource estimation.

1.11.4 QA/QC

Lumina implemented a comprehensive QA/QC program to ensure the reliability and accuracy of the analytical data used for resource estimation. This program included the routine insertion of Certified Reference Materials (CRMs), blanks, and duplicate samples. These QA/QC samples were inserted approximately every tenth sample, resulting in an overall QA/QC insertion rate of 14.6%, as detailed in Table 1.1.

Table 1.1: Summary of QA/QC Sample Insertion Rates

QA/QC Sample Type	No. Samples	Insertion Rate (%)
Regular Drilling Samples	1,090	-
Blank	50	4.6
Standard	71	6.5
Duplicate	38	3.5
TOTAL QA/QC SAMPLES	1,249	14.6

Source: Micon, 2025.

Blanks, composed of quartzite sourced from a quarry at Bukowa Góra, were inserted to monitor potential contamination during sample preparation and analysis. However, some blanks returned elevated values, particularly for copper, lead, and zinc, due to trace levels of these elements in the blank material itself rather than contamination introduced during processing. Table 1.2 summarizes the blank performance.

Table 1.2: Summary of Blank Sample Results

Element	Number of Samples	Detection Limit (ppm)	Number of Samples Above 3 x Detection Limit	Percentage of Samples Above 3 x Detection Limit
Copper	50	0.1	50	100%
Silver	50	0.1	4	8%
Lead	50	0.1	50	100%
Zinc	50	1	22	44%

Source: Micon, 2025

While the QPs typically consider blank failures to be concerning, the absolute blank values remained below the economic cut-offs used to define the mineralized domains in the MRE. Consequently, these results were considered acceptable, although the QPs recommend sourcing a more inert blank material for future programs.

CRM was also inserted randomly into the sample stream to monitor analytical accuracy. Three CDN CRMs (ME-6, ME-11, and ME-17) were selected to span low, medium, and high-grade ranges for copper, silver, lead, and zinc. Most CRM results fell within the acceptable $\pm 2SD$ and $\pm 3SD$ ranges around their certified mean values, as summarized in Table 1.3.

Table 1.3: Summary Table of CRM Performance

CRM	Element	Number of Samples	Number of Samples Outside 2SD	Percentage of Samples Outside 2SD	Number of Samples Outside 3SD	Percentage of Samples Outside 3SD
CDN-ME-6	Copper	17	0	0%	0	0%
	Silver	16	1	6%	0	0%
	Lead	19	0	0%	0	0%
	Zinc	19	2	11%	0	0%
CDN-ME-11	Copper	30	0	0%	0	0%
	Silver	29	0	0%	0	0%
	Lead	30	0	0%	0	0%
	Zinc	30	0	0%	0	0%
CDN-ME-17	Copper	22	0	0%	0	0%
	Silver	19	1	5%	0	0%
	Lead	22	0	0%	0	0%
	Zinc	22	3	14%	0	0%

Source: Micon, 2025.

Duplicate analyses were conducted on 38 fine duplicate samples to assess analytical precision and reproducibility. Results demonstrated excellent agreement, with R^2 values exceeding 0.99 and most paired results falling within $\pm 20\%$ relative difference. Slightly greater variability was observed at lower grades, consistent with typical nugget effects and heterogeneity in lower-grade mineralization. Overall, the duplicate results confirmed the reliability of the analytical data.

In summary, the QA/QC program was designed and implemented in line with industry best practices. Despite minor blank material limitations, the overall QA/QC dataset provides a high degree of confidence in the analytical data used to support the MRE. The QP concluded that the QA/QC procedures were appropriate and recommended the selection of a more geochemically inert blank material for future drilling campaigns.

1.12 DATA VERIFICATION

Liz de Klerk, an author and QP of this Report, visited the Property between February 12, 2025, and February 14, 2025, accompanied by Matt Ball (Micon Resource Geologist) and Andrzej Zibrow and Tomasz Bieńko, Lumina personnel.

Drill hole data was provided to the QPs by Lumina. The QPs have not conducted any drilling, collection of samples, or independent assaying of material from the Project. The QPs have reviewed the methods used for logging, sampling, and assaying. The Project drill hole database is maintained on-site in a Microsoft Excel and csv files. The data was collated into a single Microsoft Excel spreadsheet by the QPs, and the database was validated and inspected to ensure that the data is acceptable and in a suitable condition for use in modelling and resource evaluation.

Only one historical drill hole within the Nowa Sól license, Bojadła 1, reached sufficient depth to intersect the mineralized interval of the Kupferschiefer horizon. Lumina drill hole Jany_C2 was drilled 1.5 km southeast of Bojadła 1 and allows the best comparison of historical and Lumina data. Downhole plots of the two drill holes show they have comparable grades and grade profiles with mineralization in Jany_C2 being wider than Bojadła 1. This observation is consistent with mineralization thickening to the southeast. Overall, the drill holes show consistent results considering the drill hole spacing. Four additional historical drill holes, which were not drilled into the Lower Permian depths were useful for seismic analysis and target generation prior to the initial drilling by Lumina.

The QPs have reviewed the sample collection, analytical methodologies, QA/QC program, and drill hole database and they are to current industry standards and permit a meaningful investigation of the mineralization at the Project for the purpose of the MRE under the CIM Guidelines (CIM, 2019). The database is also sufficiently reliable to be used as the basis for the conclusions and recommendations reached in this Report.

1.13 MINERAL PROCESSING AND METALLURGICAL TESTING

No historical metallurgical testwork has been completed using mineralized samples from the Project.

In August 2025 XPS Industry Relevant Solutions (XPS) of Sudbury, Canada, were engaged by Lumina to undertake a preliminary metallurgical testwork program using mineralized samples from Nowa Sól. The objective of the XPS testwork program was to develop a conceptual process flowsheet and provide preliminary process design criterion to support a PEA. The test program comprized mineralogical and chemical characterisation of the composite feed sample, standard Bond Ball Mill Work Index (BBWi) determination, preliminary flotation circuit development and characterisation studies of final tailings and concentrate samples generated from the concluding confirmatory tests.

In addition to the XPS testwork, Lumina employed Wrocław University of Science and Technology (Wrocław Tech) in Poland to undertake a similar program of work which included mineralogical characterisation studies and preliminary flowsheet development testwork. Wrocław Tech is considered an expert in the beneficiation of copper mineralization originating from the Legnica-Głogów Copper District, Poland, and has considerable experience with the metallurgical development and optimization of similar type deposits.

Three composite samples were selected by Lumina geologists and prepared by technical staff from the Wrocław Tech. Two of the composite samples were used for characterisation and flowsheet development studies while the third was used for comminution testing. Approximately 50% of the original composites were retained by Wrocław Tech for it's independent parallel testwork program.

The preliminary mineralogical study on the composite sample of Nowa Sól mineralization showed that the primary Cu-bearing mineral was chalcocite, accounting for 80% of the contained copper, followed by covellite and chalcopyrite. At a 99 µm grind, chalcocite was 38% liberated, 5% associated with Cu sulphides, 30% with silicates, and 23% with two or more minerals forming complex particles. The measured 80% passing (P_{80}) size for the chalcocite in the feed was 47 µm, liberated grains P_{80} 74 µm and non-liberated grains P_{80} 34 µm.

It is the QP's opinion that the composite samples prepared for the metallurgical testwork program are representative of the typical mineralization found at the Project. Although the samples were not freshly drilled there was no obvious signs of weathering and despite the average copper grade of the composites being significantly higher than the average mineral resource grade, the QP believes that the samples were suitable to be used for a preliminary testwork program that could support a PEA level technical report.

A single standard (BBWi) test using a 150 mesh of grind was performed by SGS at the Lakefield facility. The work index derived was 15.6 kWh/t.

The preliminary metallurgical testwork undertaken by XPS comprised a series of flowsheet development tests based on conventional single product flotation. Only open circuit bench scale tests were completed by XPS, there was insufficient sample available to undertake locked cycle tests.

The preliminary flowsheet developed from the testwork program is considered conventional and comprises primary grinding to a P_{80} size of 60 µm, rougher flotation, rougher concentrate regrinding to approximately 11 µm and two stages of cleaning.

The average rougher copper recovery from the final confirmation flotation tests was 88.6% into a rougher concentrate containing 7.3% Cu while the copper recovery into the Cleaner 2 Concentrate grading 26.2% Cu was 79.4%.

The results from the metallurgical test program conducted by Wrocław Tech were similar to the XPS test results. The rougher stage copper and silver recoveries tended to be higher than XPS (92% to 94% for copper, and 94% to 98% for silver) albeit at higher weight recoveries and lower grades, but recoveries to the Cleaner 2 concentrate were generally lower and lower grade, due to the absence of a rougher concentrate regrind step.

In interpreting the metallurgical open circuit test results, Wrocław Tech observed that the mineralization/chemical characteristics and rougher flotation efficiency parameters for the Nowa Sol sample were similar to comparable mineralization from a nearby operating mine that was previously tested. Assuming that the selectivity of copper and silver enrichment from Nowa Sól copper mineralization was at the level of the industrial selectivity of the comparable material, Wrocław Tech estimate metallurgical recoveries of 88.75% for copper and 86.64% for silver.

The final copper concentrate produced from the XPS confirmation tests contained 26% Cu and significant payable silver (1,330 g/t Ag). Potential deleterious elements include organic carbon, arsenic, lead and zinc.

1.14 MINERAL RESOURCE ESTIMATES

The MRE was completed independently by the QPs with limited input from Lumina geological personnel. Liz de Klerk, Pr.Sci.Nat., QMR and Dr. Ryan Langdon, Ph.D, CGeol, who are independent of Lumina and are QPs within the meaning of NI 43-101, are responsible for the MRE disclosed in this Report. Mineral Resources are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and were prepared using the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

A database was compiled from Microsoft Excel spreadsheets supplied by Lumina for use in the MRE. The database contains a total of 30 drill holes of which 23 are inside the license boundary. Within the license, 22 were drilled by Lumina between 2014 and 2024 and one historical drill hole completed in 1986. Historical drill holes exterior to the license were included in the MRE database to limit the extrapolation of grades at the margins of the license.

All geological modelling was performed in Leapfrog 2024.1.2. Grade domains were modelled individually for Cu, Ag, Pb and Zn. The depth of mineralization with respect to the known stratigraphy is controlled by the depth of the Rote Fäule (the strongly oxidized redox front) and its proximity to the T1 Kupferschiefer unit. Due to the fact that mineralization is not constrained to a single lithological unit and the grades show a normal shaped distribution either side of the T1 Kupferschiefer hosted in different lithologies a grade shell approach was used for modelling.

Statistical analysis was performed to select appropriate cut-off grades for grade domain modelling and were selected based on observed bi-modal or multi-modal populations and inflection points in log-probability plots. The domains were modelled as veins in Leapfrog Geo.

A total of 387 dry density measurements were taken on drill cores from Lumina drill holes and are well distributed vertically through the different stratigraphic units above, within, and below the mineralized orebody. Both visually and statistically, there is a clear correlation between decreasing density and stratigraphic depth. Four density domains were modelled at different density cut-offs and median density values were applied to each domain.

The core samples exhibited a wide range of original sample lengths. Considering the minimum mining height of 1.8 m applied at the neighbouring KGHM operation, a block size of 0.6 m was deemed appropriate. This selection reflects a balance between the modelled thicknesses of the ore material domains and the computational efficiency required for block model processing. Based on these considerations, a fixed composite length of 0.6 m was applied to the assay dataset for use in the MRE.

Exploratory Data Analysis (EDA) was undertaken for assay data to define and validate stationary estimation domains, identify the nature of the contacts between domains, and to inform outlier management. All the contact plots demonstrated hard boundaries, indicating that they should be interpolated as separate estimation domains. Histograms and log probability plots were created to understand where the grade distributions deviated from a normal distribution, indicating the presence of outliers. Outlier grades that required capping before interpolation to limit their influence were identified for the Pb ore domain only. The ore domains contained outlier values that were mitigated by using a restricted search neighbourhood. The restricted search neighbourhood was set to 50% of the size of the first pass search ellipse, and assay grades outside of this neighbourhood were capped.

A total of 13 estimation domains with hard boundaries were modelled for the Nowa Sól deposit. These are the ore, waste above and waste below grade domains for Cu, Ag, Pb and Zn, and the additional high-grade Ag grade domain. A stratigraphic transformation was applied to each grade

domain. This was deemed an appropriate approach for the Project considering the laterally continuous mineralization that shows vertical variations in grade. Furthermore, the mineralized host rocks are not flat due to topographic variation at the time of deposition as well as the effects of differential compaction and subsequent structural deformation.

Variography was performed on transformed data and the calculation of the variograms followed the approach for tabular deposits. The modelled variograms all show long range structures in the horizontal plane with short ranges in the vertical plane. This is consistent with the observed down hole grade variability and the lateral continuity of grade and the orebody geometry between drill holes.

Assay grade data were interpolated using Ordinary Kriging (OK) for all domains. The block models were constructed with a block size of 20.0 m (X), 20.0 m (Y) and 0.6 m (Z). The block size was chosen to ensure that the geological boundaries of the modelled domains could be accurately represented while ensuring processing times remained reasonable. The vertical block size is also well suited to the operational mining constraints of a minimum mining height of 1.8 m that is utilized at the neighbouring KGHM operation.

The domains were estimated with hard boundaries in two passes with increasingly relaxed constraints. For an increasing estimation pass number, the size of the search ellipse was increased, and the minimum number of samples was decreased to ensure all blocks were interpolated. The search ellipse size and anisotropy were related to the modelled variogram for the respective domain.

To assess the quality of the block model estimate, multiple methods of validation were performed, including visual inspection, statistical comparison, decluster plots, and swath plots. Block model validation was limited to data inside the Nowa Sól license boundary. The results of the block model validation were satisfactory, and the results were suitable for use in the MRE.

The MRE was classified in accordance with NI 43-101. Mineral Resources were classified as Measured, Indicated, and Inferred. A measure of data spacing, which is directly linked to the drill hole spacing, was used to inform the Mineral Resource classification. The data spacing thresholds used for Mineral Resource classification were based on the distance to the nearest drill hole. Blocks where the nearest drill hole was > 1,600 m were classified as Inferred. Measured blocks were classified as being < 1,200 m to the nearest drill hole where the drill holes had been drilled on a regular spaced grid, in the south of the license. All other blocks were classified as Indicated.

Only material that meets reasonable prospects for eventual economic extraction (RPEEE) was considered in the MRE, assuming inputs derived from metallurgical testwork and the operation, or similar operations.

In accordance with the recommendation of the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines 2019 (CIM, 2019), mining envelopes were defined to identify spatially continuous mineralization within potentially mineable shapes using reasonable assumptions based on neighbouring operations and long-term price trends.

The objective of the underground envelope definition process is to convert a block model into a practical, mineable shape that reflects both economic and geometric constraints. The starting point for the process was the definition of a target head grade for the mineable envelope. A pure pay limit calculation for copper results in a 0.70% Cu grade. The terms used to calculate this were established from consensus price forecasts and process recoveries and smelter terms from comparable benchmarks. The operational costs are also based on the benchmark operations with some adjustments for efficiency savings.

Applying the calculated pay limit to the model will not define what can be mined because the block model defines the raw spatial distribution of grade and would not result in a minable shape. Further work was required to determine the optimal cut-off using the same input parameters. The maximum Net Present Value (NPV) is achieved at a cut-off of 1.0% Cu. The target head was selected at 0.95% Cu to ensure inclusion of the maximum mineable material into the mining envelopes. Also, ensuring the low-grade material included in the envelopes is greater than the pay limit. Blocks classified as Measured, Indicated, and Inferred were considered in the optimization using Datamine Studio OP.

A minimum mining height of 1.8 m was applied with a room width of 6.0 m. No additional dilution from the footwall or hanging wall was applied in the optimization, as it is assumed that any dilution can be effectively contained within the defined minimum mining height. Mining envelopes were generated across the entire license area.

The mining envelopes were used to constrain the block model and identify material that meets RPEEE and were considered for inclusion in the MRE. A mining loss factor of 15% was applied to the reported Mineral Resources inside the defined mining envelopes. This adjustment accounts for material that is proven to exist geologically, but cannot be physically recovered in the actual mining operation and therefore does not have RPEEE such as sterilisation of material due to geotechnical barriers and pillars and support that will not be mined and processed.

A sensitivity analysis was conducted by generating grade tonnage data at a variety of cut-offs within the mining envelopes used to constrain the Mineral Resources. As with the Mineral Resources a mining loss factor of 15% was applied to account for material that is proven to exist geologically, but cannot be physically recovered in the actual mining operation.

As of the Report effective date, the QP responsible for the MRE, is not aware of any known current environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that might materially affect the MRE that are not discussed in this Report. However, these factors could impact the Mineral Resources and if any such risk materialise the affected areas must be re-evaluated to confirm changes in the Mineral Resources.

Table 1.4: Nowa Sól Mineral Resource Statement, Effective Date January 9, 2026

Resource Category	Tonnage (kt)	Grade					Contained Metal			
		Cu (%)	Ag (g/t)	Pb (%)	Zn (%)	Cu _{eq} (%)	Cu (kt)	Ag (Moz)	Pb (kt)	Zn (kt)
Measured	183,991	1.37	34.53	0.03	0.00	1.75	2,526.8	204.2	59.4	6.9
Indicated	419,943	1.18	39.93	0.18	0.07	1.62	4,972.1	539.2	760.8	302.4
Measured & Indicated	603,934	1.24	38.29	0.14	0.05	1.67	7,498.9	743.4	820.2	309.4
Inferred	111,855	1.08	28.91	0.25	0.09	1.40	1,206.7	104.0	284.6	99.5

Source: Micon, 2026.

Notes:

- The MRE has been prepared in accordance with NI 43-101 with an effective date of January 9, 2026. Mrs. Liz de Klerk, Pr.Sci.Nat., QMR and Dr. Ryan Langdon, Ph.D, CGeol, of Micon are the QPs responsible for the MRE.
- Cu_{eq} was calculated using the formula $(Cu\% \times 2204.62 \times Cu_{rec} \times Cu_{price}) + (Ag_{ppm} \times 0.032 \times Ag_{rec} \times Ag_{price}) / 2204.62 \times Cu_{rec} \times Cu_{price}$. Where % = Cu grade, ppm = Ag grade, rec = recovery, price = price.
- To RPEEE, mining envelopes used to identify spatially continuous mineralization within potentially mineable shapes using reasonable assumptions based on neighbouring operations and long-term price trends to calculate a pay limit copper grade. These include metal prices of US\$/lb 4.75 for Cu and process recoveries of 89%, payability of 96.5%, transport charges of US\$/wmt 10, treatment charges of US\$/t 50, and refining charges of US\$/t 0.05 for Cu. Operational expenditure costs (OPEX) were determined with reference to comparable operations with some deductions for efficiency savings. These include costs for mining of US\$/t 54.5, processing of US\$/t 8.5, and General and Administrative (G&A) of US\$/t 5.5.
- A target head grade was selected at 0.95% Cu to generate the mining envelopes and was based on the optimal cut-off grade to maximise a theoretical NPV.
- For the mining envelopes a minimum mining height of 1.8 m was used, with a room width of 6.0 m. No additional dilution from the footwall or hanging wall was applied.
- A mining loss factor of 15% was applied to the reported Mineral Resources inside the defined mining envelopes. This adjustment accounts for material that is proven to exist geologically, but cannot be physically recovered in the actual mining operation and therefore does not have RPEEE.
- Diluted tonnages and grades are reported based on the defined mining envelopes. Contained Pb and Zn inside these shapes with RPEEE were included in the Mineral Resources.
- Mineral Resources are not Mineral Reserves and have not demonstrated economic viability. There is no certainty that all or any part of the estimated Mineral Resources will be converted into Mineral Reserves.
- Grade interpolation was by Ordinary Kriging (OK) with a block size of 20 m (X) by 20 m (Y) by 0.6 m (Z). A stratigraphic transform was applied to the data to restore the original continuity, and interpolation was performed in this space before back transforming to real-world space.
- Density domains were modelled, and median density values were assigned: low density 2.25 t/m³, medium density 2.53 t/m³, high density 2.76 t/m³, and very high density 2.93 t/m³.
- The MRE was classified in accordance with NI 43-101 (CIM, 2014). Mineral Resources were classified as Measured, Indicated, and Inferred. Blocks where the nearest drill hole was >1,600 m were classified as Inferred, Measured blocks were classified as being <1,200 m to the nearest drill hole where the drill holes had been drilled on a regular spaced grid, all other blocks were classified as Indicated.
- The totals presented in this table reported from the Mineral Resource models, are subject to rounding, and may not total exactly.

1.15 MINERAL RESERVE ESTIMATES

Lumina has not classified any Mineral Reserve Estimates for the Property.

1.16 MINING METHODS

The Project is proposed to be developed as a large-scale underground mining operation accessed via two independent vertical Shaft Complexes, referred to as the North and South Shaft Complexes. Each complex is designed to support a conceptual production rate of approximately 11.5 Mt/a. The Shaft Complexes are planned to operate as standalone systems with no operational interdependencies, providing flexibility and redundancy at the project level. Each Shaft Complex is made up of three shafts, a production shaft, a service shaft and a ventilation shaft.

For the purposes of this Report, a Room and Pillar mining method has been selected as the base case. This method is well established in the region, currently being used at the neighbouring KGHM mines. Alternative mining methods, including Drift and Fill, have been considered at a high level to compare against the base case.

The mine design, production scheduling, and equipment selection are supported by a geotechnical assessment completed at a level appropriate for a PEA. While this work supports the preliminary selection of mining methods and layouts, additional site-specific data collection and analysis will be required to confirm design parameters in future study phases.

Mineable envelopes were defined by applying an economic cut-off and optimization to the block model to generate contiguous, practical and economic mining shapes. A target head grade of approximately 0.95% Cu was selected to balance inclusion of mineable material while prioritising higher-grade zones during scheduling. These envelopes form the basis for the conceptual mine design and Life of Mine (LoM) schedule developed for the PEA.

Access to the mining horizon is planned via the vertical shafts, with all shafts requiring freezing and tubbing through the upper saturated sediments to depths of approximately 400 m. Conventional drill and blast shaft sinking methods are proposed below the frozen zone, with allowances included for more complex ground conditions in salt and anhydrite units. Shaft sinking rates, sequencing, and associated infrastructure have been estimated at a conceptual level and are subject to refinement in subsequent studies.

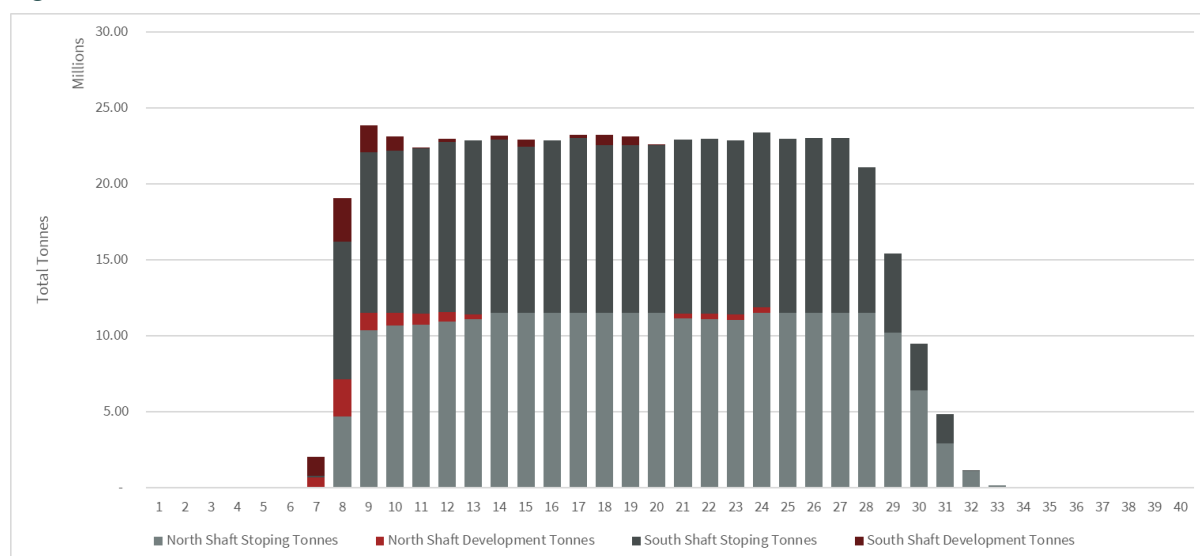
The production schedule assumes steady-state mining at approximately 32,000 t/d per Shaft Complex, with development, stoping, and ventilation layouts designed to support continuous operations. Inferred Mineral Resources have been included in the mine plan for the purposes of this PEA. There is no certainty that these results will be realized, and the mining plan and production schedule are subject to the inherent uncertainties associated with this level of study.

The proposed underground mining method and associated designs present a technically plausible development concept. Additional work, including infill drilling, detailed geotechnical and hydrogeological investigations, optimization of mining methods, and refinement of infrastructure and scheduling assumptions, will be required to advance the Project to a PFS level.

1.16.1 Production Schedule

The production schedule is based on 32,000 t/d (11.5 Mt/a) run of mine (RoM) feed to the processing plant for each Shaft Complex. The production schedule assumes 6 working days in a 7-day cycle, with a maximum of 48 hours per week resulting in three 8-hour shift per day. Multiple blasts are required to reach the development target of 3.5 m/d in the decline system. A stoping width of 5 m and height of 2 m has been used in the underground design. The production schedule is shown in Figure 1.1.

Figure 1.1: Production Schedule



The mineral resources used in the mine plan presented in Table 1.5 are derived from NI 43-101 compliant Mineral Resources and are used solely to demonstrate potential extractable grades and tonnes for the purposes of mine planning, production scheduling, and Discounted Cash Flow (DCF) analysis within this PEA. The mineral resources used in the mine plan are a sub-set of the Measured, Indicated and Inferred Resources declared in Section 14.11 and do not constitute NI 43-101 compliant Mineral Reserves.

Table 1.5: Mineral Resources used in the Mine Plan

Parameter	Tonnage (kt)	Grade				Contained Metal			
		Cu (%)	Ag (ppm)	Pb (%)	Zn (%)	Cu (kt)	Ag (kt)	Pb (kt)	Zn (kt)
North Shaft									
Measured	0	0.00	0.00	0.00	0.00	0.0	0.0	0.0	0.0
Indicated	209,336	1.11	46.74	0.24	0.10	2,324.7	9.8	507.3	212.3
Inferred	49,535	1.14	31.65	0.35	0.16	565.6	1.6	172.2	79.9
Total	258,871	1.12	43.85	0.26	0.11	2,890.3	11.4	679.5	292.2
South Shaft									
Measured	131,821	1.26	36.07	0.03	0.00	1,658.2	4.8	39.0	4.5
Indicated	93,311	1.12	26.26	0.03	0.00	1,047.5	2.5	28.2	3.6
Inferred	26,510	1.38	33.85	0.21	0.02	365.1	0.9	55.6	4.6
Total	251,642	1.22	32.20	0.05	0.01	3,070.9	8.1	122.7	12.7
Combined									
Measured	131,821	1.26	36.07	0.03	0.00	1,658.2	4.8	39.0	4.5
Indicated	302,647	1.11	40.43	0.18	0.07	3,372.1	12.2	535.5	215.9
Inferred	76,046	1.22	32.42	0.30	0.11	930.8	2.5	227.8	84.5
Total	510,513	1.17	38.11	0.16	0.06	5,961.2	19.5	802.2	304.9

Inferred Mineral Resources have been included in the production schedule derived from the minerals resources used in the mine plan for the purposes of this PEA; however, there is no certainty that the results will be realized, and the inclusion of Inferred Mineral Resources is subject to the inherent geological uncertainty associated with this level of confidence.

1.16.2 Geotechnical Assessment

The Nowa Sól deposit is hosted within Permian stratigraphy at depths of 1.9 km to 2.1 km, resulting in the deepest proposed copper-silver mine in Poland. These conditions may create a markedly different stress environment compared with neighbouring KGHM mines which are mined at depths of 0.8 km to 1.4 km. Operational and safety risks associated with high stress and seismicity should be recognized as one of the major risks for the Project.

The geotechnical study by Global Mine Design (GMD) supports the QPs mine planning assessment and evaluates the geotechnical risks and design criteria for the PEA mine plan. This geotechnical study integrates available exploration datasets from Nowa Sól including some laboratory test data, borehole records, and supporting geophysical interpretations. It further benchmarks findings against KGHM's established Room and Pillar operations, while recognising that Nowa Sól will encounter higher stresses and associated risks with mining at greater depth.

Lumina has not yet carried out detailed geotechnical data collection, and this will need to be addressed in the PFS project phase. Available data suggests significant contrasts between key lithological units: dolomites and limestones exhibit very high strength but elevated burst potential, while sandstones are weaker, highly stress-sensitive, and display pronounced strength loss when saturated. A detailed stratigraphic interpretation is required for future planned shaft locations, and in particular halite (salt) units are expected to be highly problematic and prone to shaft squeezing.

Initial analyzes suggest that higher mineral recoveries (of approximately 90%) assumed for Nowa Sól in historical scoping studies are not realistic at acceptable safety margins; recoveries in the range of 50% to 70% for Room and Pillar mine panels are more defensible when high stress productivity risks are considered. Stratigraphic differences have been noted at Nowa Sól relatively to KGHM's operations, including thinner roof dolomite stratigraphy, and lower cover of anhydrite prior to intersecting the halite rock salt stratigraphy. These stratigraphy differences can have implications to Room and Pillar design, as well as shaft and ventilation design. Production constraints associated with delays due to stress management and seismicity should be incorporated into the PFS mine planning scenarios as these could have significant negative impacts to mine economics.

Drift and Fill mining demonstrates potential opportunities for mechanisation and automation but faces similar geotechnical challenges as Room and Pillar including; large effective spans, rapid stress build-up during final pillar extraction, and significant dependency on tight paste backfill. Ore panel recoveries of 60% to 80% are assumed achievable depending on configuration and rock mass quality, though additional productivity constraints will be imparted to Drift and Fill due to reduced working faces and backfill processes. Increased ground support intensity such as dynamic ground support and use of secondary (cable) support, and cribbing-post support should be considered in operating costs for both Drift and Fill and Room and Pillar mine methods.

Temporary and permanent safety pillars are required in the mine layout for excavation stability and stress management. Permanent safety pillars are required to protect shafts, main infrastructure, and long-life excavations. Temporary safety pillars, particularly within production panels, will be needed to manage local instability risks and control sequencing under both Room and Pillar and Drift and Fill methods. Temporary and permanent safety pillars will impact ore recoveries, although temporary pillars can be assumed partially recoverable. Preliminary guidance has been provided in the Report for pillar dimensioning but will need to be assessed in more detail in future study phases.

Shaft stability analysis suggests severe to very severe squeezing in halite units, with closures ranging from ~500 mm to more than 1.5 m depending on depth and diameter. Mudstone and sandstone intervals may also yield substantially, particularly where water-bearing. Preliminary analysis indicates that shaft spacing of ≥ 150 m and shaft offsets from mine panels of 500 m are required to mitigate induced deformation and seismic impacts. The shaft pillar configuration assuming an approximately 1 km x 1 km arrangement influence mine layout and sterilisation. Shaft placement also requires parametric assessment to balance ore sterilisation, early production, minimising halite intersections, avoiding faults, and social-economic constraints.

The PFS will require enhanced field investigations, detailed laboratory testing, refined geological modelling, and advanced 3D geomechanical analyses. A series of trade-off analyses will be required to better define shaft arrangements and sizes, and to hone-in on preferred mining methods, sequencing, ore recoveries, production rates, and operating costs. Various recommendations have been outlined in the geotechnical report for investigation requirements and follow-up study.

1.16.3 Backfill Study

This section of the Report presents a conceptual PEA level assessment of backfill requirements and associated tailings handling infrastructure for the Project, considering two mining method scenarios: Room and Pillar and Drift and Fill. The primary driver for backfill and key Project constraint is that any tailings reporting to surface must be managed as dry stack, and a conventional wet tailings storage facility (TSF) will not be constructed.

A screening review of credible backfill options was completed. Hydraulic fill was assessed and excluded due to its reliance on cyclone classification and the generation of fine tailings streams that typically require wet tailings management. Cemented paste backfill has been selected as the preferred backfill method. Paste backfill provides low permeability performance, predictable strength development, and a non-segregating material suitable for pipeline transport to depth. Importantly, the same filtration circuit that supports paste production also produces filtered tailings suitable for dry stack disposal on surface, enabling an integrated filtration-led tailings strategy consistent with Project constraints.

Conceptual strength criteria were established based on mining method requirements. For Room and Pillar mining, paste backfill is assumed to require 0.2 MPa to 0.5 MPa Unconfined Compressive Strength (UCS) at 28 days, achievable with a nominal 2% binder dosage, with backfill primarily serving confinement and deformation control functions at depth. For Drift and Fill mining, higher structural performance requirements are assumed, with 1.5 MPa to 2.0 MPa UCS at 28 days and a nominal 5% binder dosage to support sequential extraction and working platform stability. These criteria are appropriate for PEA level and will be confirmed through paste mix and UCS testwork during the PFS.

LoM material balances were developed for the North and South Shaft Complexes using annual void creation derived from the mine schedule and applying practical fill factors (90% for Room and Pillar and 97% for Drift and Fill). For the Room and Pillar scenario, steady-state paste backfill production rates of approximately 1,050 t/h (dry solids) for the North and 1,003 t/h (dry solids) for the South are well aligned with the void creation profiles, with backfill placement capable of tracking void creation through the steady-state production period without reliance on deferring backfilling into late mine life. Surplus filtered tailings not used for underground placement are directed to surface dry stack tailings facilities.

A conceptual hydraulic design has been developed for paste delivery to an anticipated mining depth of approximately 2,000 m. Static head is approximately 20 MPa (water equivalent). Allowing for friction losses, local losses and operating margin, a peak design discharge pressure of approximately 22 MPa has been adopted for preliminary equipment selection. Positive displacement pumping has been selected as the conservative PEA level basis of design due to the high static head and the need for stable volumetric flow control. Pressure management options, including intermediate booster stations or alternative strategies, have been identified for evaluation during the PFS.

A PEA level cost estimate has been prepared for a single paste backfill and filtration plant serving either the North or South Shaft Complexes. The estimate includes thickening, pressure filtration, paste mixing, positive displacement pumping, and associated surface and underground reticulation infrastructure. Total capital cost (including 25% contingency) is estimated at approximately US\$91.9 million per plant, with annual operating costs of approximately US\$56.0 million per plant. The dominant cost drivers are pressure filtration capacity, high-head paste pumping, binder consumption, and electrical power demand.

Overall, the PEA level assessment demonstrates that paste backfill is technically credible and well aligned with project constraints, providing an integrated filtration-based approach that supports underground backfilling and surface dry stack tailings management while avoiding the need for a wet TSF. The conceptual design basis and cost estimate provide a suitable platform for progression to the PFS, where tailings testwork, hydraulic modelling, geomechanical assessment and vendor engagement will be used to confirm and refine assumptions, equipment selections and operating parameters.

1.17 RECOVERY METHODS

The design of the recovery method for the Project is based on conventional processing technologies and a proven flowsheet that has been implemented at many other comparable projects.

Fresh material from underground will be crushed and fed to a stockpile from which it will be conveyed to the grinding circuit, consisting of a single Semi-Autogenous Grinding (SAG) and ball mill, with a pebble crusher. The discharge slurry from both mills will be pumped to a cyclone cluster, with the overflow feeding the flotation circuit and cyclone underflow returning to the ball mill.

The copper flotation circuit comprises of rougher and scavenger flotation stages, with regrinding of the combined rougher and scavenger concentrate, plus three stages of cleaning. Preliminary testwork suggests that only two stages are required to produce a 26% copper concentrate, but an additional stage has been included to provide an element of conservatism.

The final flotation concentrate will be dewatered using a thickener and pressure filter, then stored for bulk loading and transportation to market.

Tailings from the flotation circuit will be thickened, filtered and either transported to a Dry Stack Tailings Facility (DSTF), or fed to the tailings backfill plant, where it is mixed with cement and water before being pumped underground.

1.18 PROJECT INFRASTRUCTURE

The Project is at an advanced exploration stage, and project infrastructure requirements have therefore been analysed at a conceptual level only. Desktop research has been carried out to identify local utility and transport infrastructure requirements and there has only been limited engagement with statutory suppliers and other authorities. This engagement resulted in the inclusion of a dedicated substation for the Project into development plans of the national state grid operator (*Polskie Sieci Elektroenergetyczne S.A./ PSE*) on a newly projected high voltage (HV) line directly adjacent to the Project site.

As a greenfield site, the Project will require site preparation and the provision of access and utilities, as well as the construction of a full suite of facilities necessary to carry out large scale mining activities. However, the Project is well positioned to leverage high-quality regional infrastructure with available spare capacity. The presence of major highways and rail corridors connected to the European Transit Network, HV transmission infrastructure integrated into the national grid, as well as strong human capital and established logistics services, supports favourable and cost-effective Project development.

1.19 MARKET STUDIES AND CONTRACTS

1.19.1 Market Studies

Copper is a critical industrial metal underpinning modern infrastructure due to its high electrical conductivity, corrosion resistance, and versatility. Global demand is driven by its widespread application in construction, power transmission, transportation, defense, and digital technologies. The Nowa Sól Copper-Silver Project is strategically located north of the established LGOM in southwestern Poland, a region with over five decades of copper and silver mining history.

For the purposes of this PEA, the Project is assumed to follow a similar production model to KGHM's nearby operations. This includes the production of approximately 23% Cu copper concentrate, processed into 99.99% purity Grade A London Metal Exchange (LME) registered copper cathodes, alongside the recovery of silver and various metallurgical by-products. Silver is typically refined into bars and granules, with annual production in the region exceeding 40 Moz. The assumption that the Project will produce a comparable copper concentrate containing payable copper and silver is conceptual and subject to validation through future metallurgical testing, process design, and engineering studies.

Global copper supply is primarily sourced from mine production, supplemented by a meaningful and growing contribution from recycling, which currently accounts for approximately 30% of annual global demand. In 2024, global copper mine output totalled approximately 23 Mt, with Chile remaining the largest producer, followed by the Democratic Republic of Congo, Peru, China and the United States. Silver supply is also derived from both primary mining and recycling; however, unlike copper, a substantial portion of silver production is generated as a by-product of copper, lead, zinc and gold mining. In 2024, global silver mine production was approximately 820 Moz, led by Mexico, China, Peru and Chile, with overall output influenced by trends in both primary silver operations and base-metal mining activity.

Both copper and silver markets face notable supply-side constraints. Many existing mines are mature and experiencing declining grades, while the pipeline of new brownfield and greenfield projects is comparatively limited relative to prior development cycles. Increasing capital intensity, extended permitting timelines, and heightened environmental and stakeholder requirements are expected to further challenge new supply development. Recent disruptions, including operational issues and project delays, have contributed to periodic supply deficits. Looking ahead, independent

forecasts suggest the potential for more persistent supply shortfalls later in the 2020s, particularly for copper, while silver supply is expected to remain relatively tight and less responsive to price signals due to its reliance on broader base-metal production trends.

Copper demand is underpinned by long-term structural trends including electrification, urbanisation, infrastructure expansion, and the global transition to lower-carbon energy systems, alongside rising defence and digital infrastructure spending. The energy transition is increasing copper intensity across renewable power, electric vehicles, and grid infrastructure, driving higher copper use per unit of output than historical norms. Long-term projections indicate that global copper demand could rise by approximately 70% to exceed 50 Mt per annum by 2050, with growth expected to accelerate to around 2.6% annually through 2035, compared to 1.9% over the past 15 years.

Silver demand is supported by both industrial and investment drivers. Industrial applications account for a significant and growing share of consumption, particularly in electronics, solar photovoltaic (PV) panels, and chemical processes, with solar PV emerging as a key structural source of demand growth. In addition, investment demand through bullion, coins, and exchange-traded products remains an important component of the market, particularly during periods of economic uncertainty. Over the medium to long term, industrial demand linked to energy transition technologies is expected to remain the primary driver of silver consumption growth.

1.19.2 Contracts

As the Project remains in the advanced exploration stage, no formal contracts have been finalized for land use agreements, infrastructure development, mining, offtake arrangements, smelting, refining, transportation, sales, marketing, or other commercial agreements.

1.20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Exploration and mining activities in Poland are governed by the Geological and Mining Law and overseen by the MCE. The mine permitting process depends on achieving a resource classification of C₁ category or higher, which then entitles the project developer to a five-year ‘exclusivity right’ to use the geological information and a ‘priority right’ to establish a contract with the State for mining operations and apply for a mining concession.

Following a C₁ resource designation, the overall mine permitting process is estimated to take at least 48 months. Key components of the process include preparation of a deposit development plan, undertaking an Environmental Impact Assessment (EIA) and obtaining an environmental decision, local and regional spatial planning decisions, and the establishment of a mining usufruct agreement with the State.

Lumina holds valid exploration concessions for Nowa Sól II (120 km²) and Konotop (40 km²) through their Polish subsidiary Zielona Góra Copper sp. z o.o. A C₁ resource classification for the Nowa Sól deposit was granted in April 2025 by the MCE.

The Project is located in a rural area of the Lubuskie Province in western Poland and the surrounding area is characterized by rivers, lakes and forests. The region has a rich variety of flora and fauna and there are a number of EU Natura 2000 sites close to the Project, notably the Nowa Sól Odra Valley, Special Area of Conservation and the Sławskie Lake District Special Protected Area for Birds. Some nationally designated sites such as nature reserves are present within the concession and will be avoided by design. There are no protected areas within the proposed surface Project footprint.

The Nowa Sól II exploration concession, which replaced the previously established Nowa Sól concession in the same area, is centered around the village of Lubięcín and spans five administrative municipalities within three larger administrative districts. There are several other villages within the concession, and the nearest urban area is the town of Nowa Sól, approximately 6 km southwest of the concession boundary. Lumina has a proactive approach to stakeholder engagement and has hired a local community relations officer.

A full review of the potential environmental and social impacts will be undertaken as part of the future EIA process. Risks identified at this stage of the Project include disturbance from air quality, noise, vibration and artificial lighting; habitat and wildlife loss; changes to water quality, river dynamics and groundwater conditions; traffic management; and permitting risks. The Project is anticipated to have a positive impact on the local and regional economy, through creation of direct and indirect jobs and associated training opportunities. There is no need for community resettlement.

Lumina intends to undertake progressive rehabilitation where possible and will follow appropriate national and international guidelines. A preliminary total estimate of US\$283 million has been budgeted for closure costs, including accumulated annual payments to the legislated decommissioning fund.

1.21 CAPITAL AND OPERATING COSTS

1.21.1 Capital Costs

The estimated initial capital expenditures required to bring the Project into production are summarized in Table 1.6. Both Shaft Complexes are independent of each other and can be developed simultaneously or in stages, offering flexibility on the speed of development and initial capital requirements.

The estimate has been compiled by the QP from information provided by other authors (QP's) involved in this Report. The bases of estimate for the principal components of the estimate are given in the following sections. The estimate is expressed in United States dollars (US\$) as of January 2026.

Table 1.6: Initial Capital Cost Summary

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Mining	2,148.3	2,159.7	4,308.0
Surface Infrastructure	83.0	82.4	165.4
Process Plant and Infrastructure	248.9	248.9	497.7
Backfill/Paste Plant	64.4	64.4	128.8
Owner's Costs	15.0	15.0	30.0
Contingency	636.4	639.1	1,275.5
Grand Total	3,195.9	3,209.5	6,405.4

Mining Costs

The estimate includes the cost of sinking three (production, ventilation and service) shafts at each of the North and South Shaft Complexes, mining-related surface and underground static equipment and haulage infrastructure, and the upfront driving of development haulage and access ways.

Information prepared by DMC Mining Services Ltd (DMC) for the estimate cost of shaft sinking, refrigeration and ventilation, has been reviewed by Micon and included with Micon's estimated costs for lateral development, mining equipment and underground infrastructure. The balance of

the estimate comprises provisional amounts estimated by Micon based on benchmarking against similar works on other projects.

Shaft costs dominate upfront mining capital expenditure, totalling US\$3,790 million, or approximately 88% of total mining capital costs of US\$4,308 million.

Surface Infrastructure Costs

The infrastructure costs of US\$165 million are split into five main areas, each for the North and South Shaft Complexes, namely civils/structural/building works (50%), provision of utilities (25%), offsite works (9%), construction preliminaries (2%), sundries (0.4%) and backfill/tailing costs (13%).

Process Costs

The Initial Capital Costs for the Process Plant total US\$497.7 million, split equally between the two Shaft Complexes.

Main mechanical item requirements have been developed from the Process Design Criteria (PDC) document. Thereafter the equipment supply costs have been estimated based on current costs and the Micon database.

Support costs were then based upon different proportions of the total Mechanical cost.

Construction costs for mechanical installation, Electrical Instrumentation and Control (EI&C), piping and Heating, Ventilation, and Air Conditioning (HVAC) have been estimated as a proportion of the plant mechanical costs.

The costs for temporary facilities, such as labour camp, site facilities and all related supervision required to manage the works are not included above as they are included in the infrastructure cost.

Indirect costs: Engineering service cost (i.e. Engineering, Procurement, and Construction Management (EPCM)) has been estimated based upon a proportion of the total direct cost (i.e. mechanical item costs + construction cost).

Paste Backfill Plant

The QPs have provided an estimate for construction (and operation) of a paste preparation and backfill process plant and distribution system for the underground mine as part of this Report.

The cost estimate is based on Q4 2025 costs and totals US\$128.8 million split evenly between the two Shaft Complexes.

Total capital cost has been derived from:

- Obtaining budget quotations from equipment manufacturers and vendors.
- Percentage factors for some direct and indirect costs.
- Micon cost database.
- Indirect costs were estimated as a percentage of total direct costs.

Owner's Costs

Micon has estimated the Owner's Costs based on benchmarking against similar projects. The total estimated Owner's Cost is US\$30 million, split evenly between the two Shaft Complexes.

Contingency

Table 1.7 shows the contingency amounts that are included in each area of the Project, as described above. Total contingency amounts to approximately 25% of the estimated total construction cost.

Table 1.7: Initial Capital Cost – Contingency

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Mining	537.1	539.9	1,077.0
Infrastructure	20.7	20.6	41.3
Process Plant	62.2	62.2	124.4
Backfill/Paste Plant	16.1	16.1	32.2
Owner's Costs	0.3	0.3	0.5
Total Contingency	636.4	639.1	1,275.5

Sustaining Capital

Sustaining capital of US\$2,854 million is provided for over the operating period of the Project.

1.21.2 Operating Costs

North Shaft Complex

Table 1.8 summarizes the LoM operating costs for the North Shaft Complex, taking into account the by-product (silver) revenues.

Table 1.8: Operating Cost – LoM Total - North Shaft Complex

Area	Total (US\$ million)	Unit Cost (US\$/t Treated)	Unit Cost (US\$/lb Cu)
Mining Costs	10,231	39.70	1.88
UG Backfill Costs	614	2.38	0.11
Ore Processing Costs	4,382	17.00	0.80
G&A	1,547	6.00	0.28
Cash Operating Costs	16,774	65.08	3.07
Product Transport, etc.	987	3.83	0.18
Royalties	-	-	-
LESS By-Product Credits	(10,920)	(42.37)	(2.00)
Total Cash Costs	6,840	26.54	1.25

South Shaft Complex

Table 1.9 summarizes the LoM operating costs for the North Shaft Complex, taking into account the by-product (silver) revenues.

Table 1.9: Operating Cost – LoM Total - South Shaft Complex

Area	Total (US\$ million)	Unit Cost (US\$/t Treated)	Unit Cost (US\$/lb Cu)
Mining Costs	9,932	39.50	1.68
UG Backfill Costs	581	2.31	0.10
Ore Processing Costs	4,275	17.00	0.72
G&A	1,509	6.00	0.25
Cash Operating Costs	16,297	64.81	2.75
Product Transport, etc.	1,025	4.08	0.17
Royalties	-	-	-
LESS By-Product Credits	(7,538)	(29.98)	(1.27)
Total Cash Costs	9,784	38.91	1.65

Mining Costs

Underground mine operating costs are based on standard rates for drilling, blasting, loading and hauling in each type (size) of development and from stopes, based on a benchmarked cost for similar operations of US\$41.00/t stoped. However, total milled includes development tonnage, for which the mining cost is capitalized, hence the average cost per tonne treated is slightly less than US\$41.00/t.

Provision is made for the owner's supervision and technical services manpower and associated costs for mine planning, grade control, and survey measure, etc. in G&A costs.

Processing Costs

Processing costs were estimated by the QPs, comprising consumables, labour, spares and maintenance, and waste management. These costs are the same at each Shaft Complex.

Table 1.10 gives a breakdown of the annual and unit costs for processing.

Table 1.10: Operating Cost – Processing

Area	Annual Total (US\$'000)	Treated (US\$/t)
Process Management and Administrative	454	0.04
Plant Operations	2,268	0.20
Plant Maintenance	2,803	0.24
Chemical Laboratory	540	0.05
Operating Supplies	40,221	3.49
Surface Tailings Management	7,225	0.63
Maintenance Supplies	5,300	0.46
Electrical Power	56,537	4.91
Contingency	80,493	6.99
Process Operating Costs	195,840	17.00

General and Administrative Costs

G&A costs were estimated by the QPs, with a forecast annual total of US\$69.1 million comprising labour (US\$22.3 million), and US\$46.8 million per year of fixed costs (Table 1.11).

Table 1.11: Operating Cost – G&A

Area	Annual Total (US\$ million)	Treated (US\$/t)
G&A Costs (Labour)	22.28	1.93
G&A Costs (Fixed)	46.84	4.07
G&A Operating Costs	69.12	6.00

1.22 ECONOMIC ANALYSIS

The economic analysis demonstrates potential economic viability with a pre-tax and pre- Polish MET NPV₇ of US\$8,326 million and 20.5% Internal Rate of Return (IRR) in a base case scenario, and a break-even copper price of US\$2.82/ lb Cu in a base case scenario. These parameters were estimated using the copper price of US\$4.75/lb Cu and silver price of US\$37.50/oz. The after-tax and after-MET NPV₇ is US\$1,604 million and 10.8% IRR.

The economic analysis is preliminary in nature and includes Inferred Mineral Resources, which are considered too speculative geologically to have economic considerations applied to them that would enable them to be classified as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

The economic evaluation was prepared by the QPs using a DCF model to assess the potential economic viability of the Project and to estimate key economic indicators including NPV, IRR, and payback period. The base case cash flow model reflects the mining, processing, production scheduling, capital cost, operating cost, and marketing assumptions described elsewhere in this Report. Sensitivity analyses were conducted to evaluate the impact of variations in key parameters, including commodity prices, capital costs, operating costs, and discount rate, on project economics.

Macro-Economic Assumptions

All economic results are reported in constant Q1 2026 United States dollars (US\$), with no allowance made for inflation or escalation. The Project was evaluated on an all-equity basis, and a real discount rate of 7.0% was applied to the base case cash flows. This discount rate reflects an assumed real risk-free rate of approximately 2.0%, assumed beta of 1.0, and an equity risk premium of 5.0%, taking into account both copper and silver, consistent with industry practice for projects at the PEA level. The sensitivity of the Project to changes in the discount rate was also examined.

Royalty and Taxation Regime

Polish corporate taxation has been calculated at a rate of 19% of profits before tax, after allowing for depreciation of capital assets over 10 years on a straight-line basis.

Polish MET was established on April 18, 2012, pursuant to the Act of the Tax on Extraction of Certain Minerals enacted on March 2, 2012. The MET is a production-based tax calculated on the quantity of copper and silver contained in concentrate, using statutory formulae linked to reference metal prices, with copper and silver assessed separately and subject to statutory caps. The MET is not deductible for corporate income tax purposes. Effective January 1, 2026, the MET framework was amended to introduce investment relief deductions, allowing 40% of qualifying investment expenditures to be credited against future MET liabilities for a period of up to 15 years, subject to a limitation that the deduction applied in any given year may not exceed 30% of the MET payable for that year. The impact of Polish MET has been shown separately as it is anticipated that an amendment to the current legislation would be required for the Project to proceed to a construction phase. Nevertheless, if MET were applied as currently legislated, the Project would generate positive, though less attractive, economic returns.

Expected Metal Prices

Project revenues will be generated from the sale of a copper concentrate with significant silver credits. Commodity prices used in the cash flow model were US\$4.75/lb Cu and US\$37.50/oz Ag.

The spot market price of copper and silver have shown considerable growth over the past year. In January 2026, spot prices averaged around US\$5.94/lb Cu and over US\$90/oz Ag. At that time, the trailing 12-month averages were close to US\$4.66/lb Cu and US\$41.60/oz Ag, respectively.

Technical Assumptions

Technical assumptions used as inputs into the DCF outlined in Section 21 of this Report.

Initial capital expenditure for the Project, made up of the North and South Shaft Complexes, is estimated at approximately US\$6.41 billion, including contingencies, with total LoM capital expenditure of approximately US\$9.26 billion, inclusive of sustaining capital, closure costs, and net of salvage values. Shaft development represents the largest component of the initial capital costs.

LoM operating costs are estimated at approximately US\$33.1 billion on a combined basis, comprising mining, underground backfill, ore processing, general and administrative costs, and product transport. Operating costs are partially offset by significant by-product silver credits, resulting in estimated LoM C₁ cash costs of approximately US\$1.46/lb Cu and All-In Sustaining Costs (AISC) of approximately US\$1.73/lb Cu on a consolidated basis.

Over the LoM, the Project is forecast to process approximately 509 Mt of ore at a consolidated average grade of 1.19% copper and 36.9 g/t silver. Payable metal production is estimated at approximately 5.2 Mt of copper and 492 Moz of silver, for LoM total copper-equivalent production of 6.9 Mt Cu_{eq}. Metallurgical recoveries of 88.75% for copper and 86.64% for silver are assumed, producing a copper concentrate grading approximately 26.2% Cu. All concentrate is assumed to be sold on a Delivered-at-Place (DAP) basis to a local smelter, using industry-standard treatment, refining, and transport charges.

Base Case Evaluation

Figure 1.2 presents a summary of the annual cash flows for the North Shaft Complex and Figure 1.3 presents the same information for the South Shaft Complex.

Figure 1.4 presents a summary of the annual net cash flow, the cumulative cash flow and the cumulative DCF for the North Shaft Complex and Figure 1.5 presents the same information for the South Shaft Complex.

Table 1.12 summarizes the LoM production and cash flows and unit costs for the Project. This includes the average payable Cu, Ag and Cu_{eq} for the first 10 years.

Figure 1.2: Annual Cash Flows – North Shaft Complex

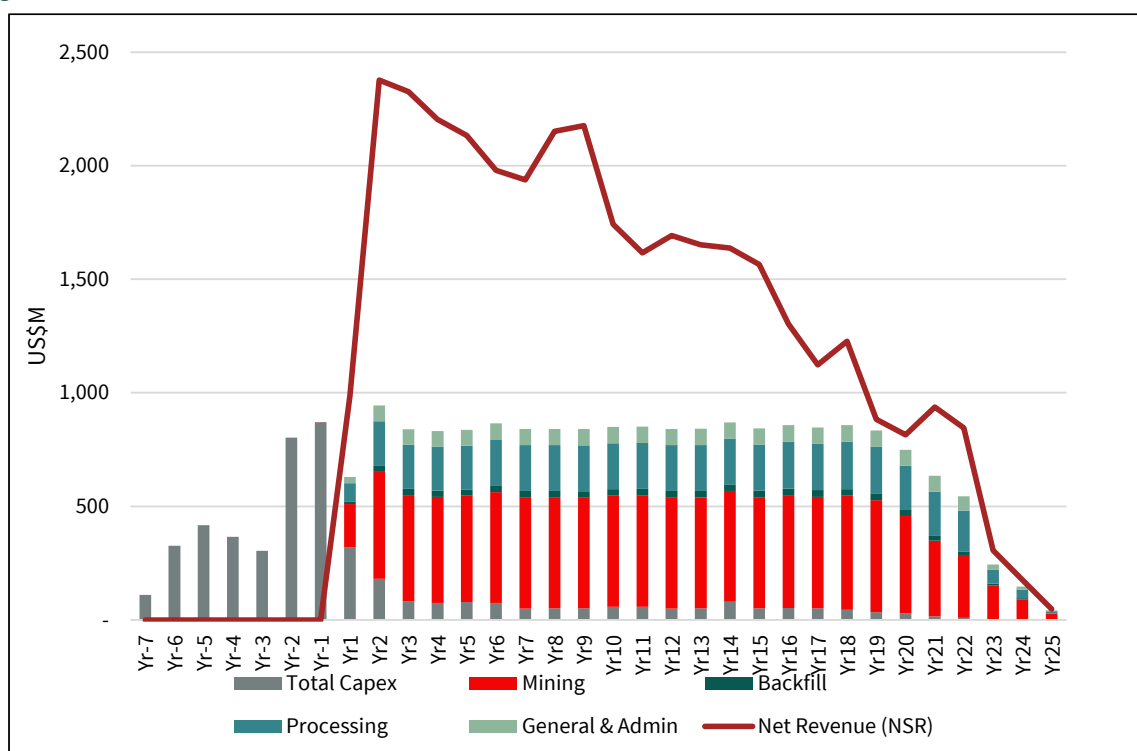


Figure 1.3: Annual Cash Flows – South Shaft Complex

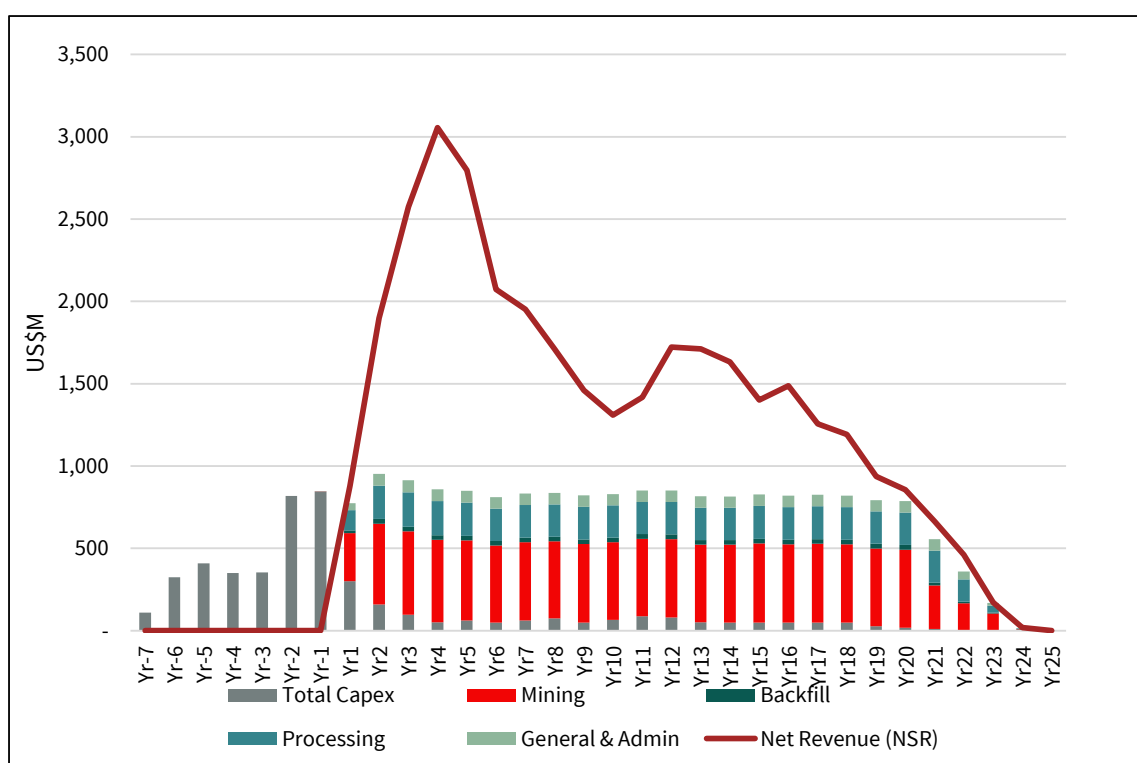


Figure 1.4: Cumulative and Discounted Cash Flows – North Shaft Complex

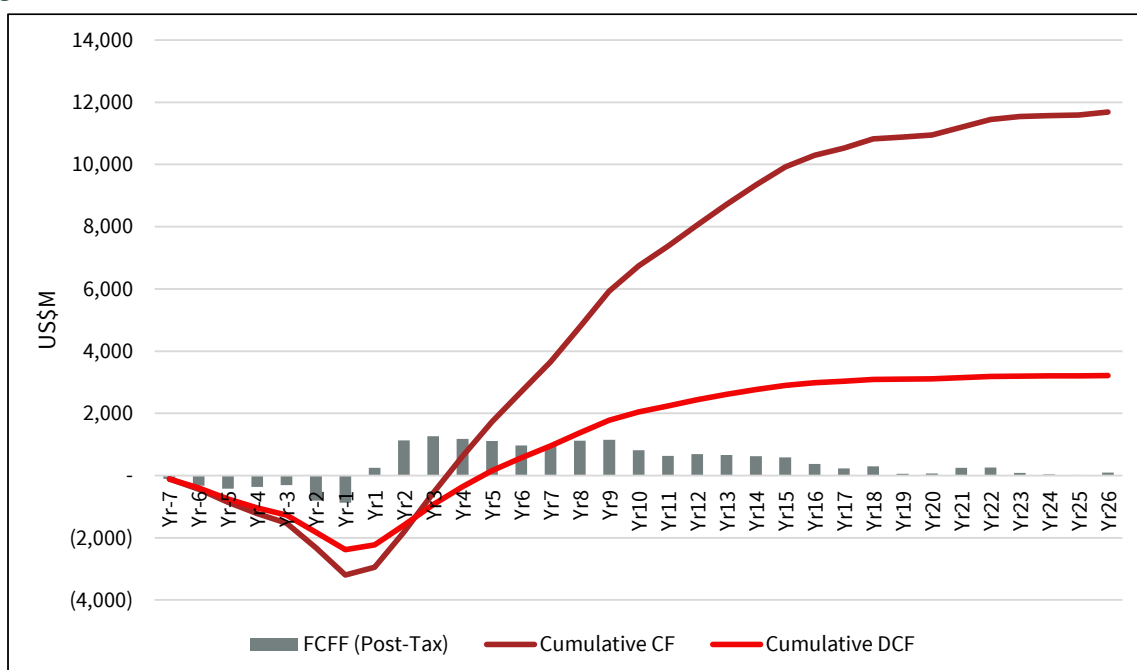
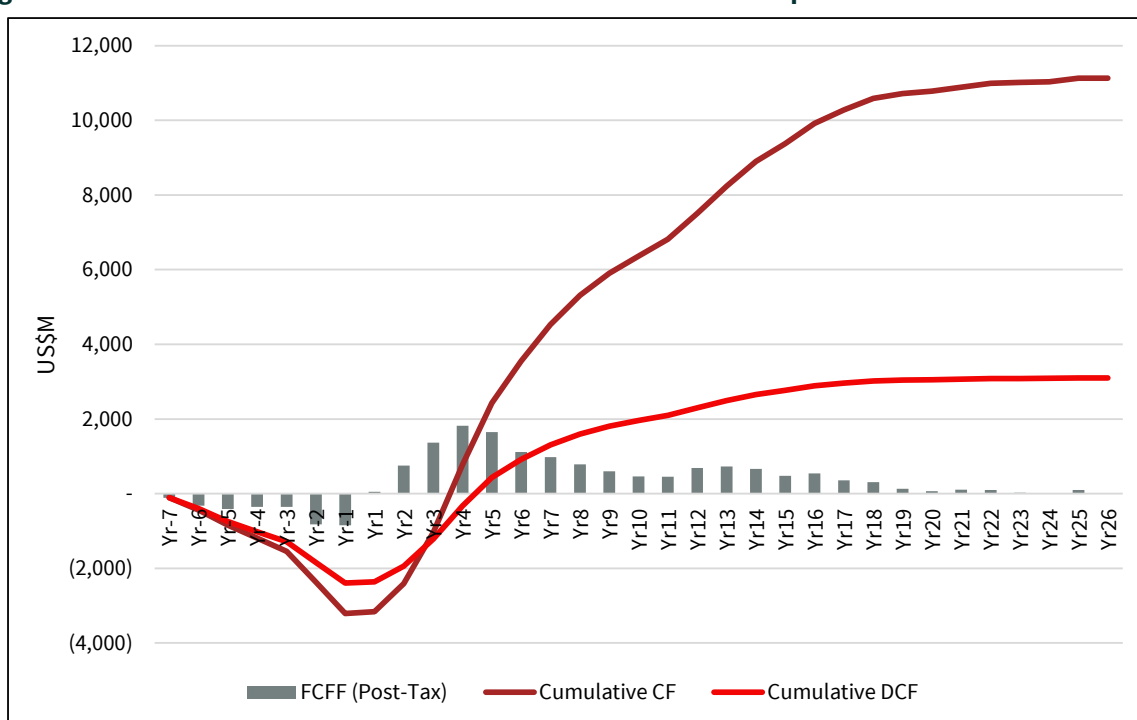


Figure 1.5: Cumulative and Discounted Cash Flows – South Shaft Complex



Note: The PEA is preliminary in nature; it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized.

Table 1.12: LoM Production and Cashflow Summary

Parameter	Unit	North	South	Consolidation
Tonnes Milled	kt	257,739	251,449	509,188
Copper Grade	Cu %	1.12%	1.25%	1.19%
Silver Grade	g/t Ag	43.25	30.39	36.90
Payable Copper Production	Cu kt	2,474	2,686	5,160
Payable Silver Production	Ag koz	291,209	201,009	492,218
Payable Copper Equivalent Production	Cu _{eq} kt	3,517	3,406	6,923
Net Revenue	US\$ million	35,845	34,638	70,483
Operating Costs	US\$ million	16,774	16,297	33,071
EBITDA	US\$ million	19,071	18,341	37,412
Total Pre-Production Capex	US\$ million	3,196	3,210	6,405
Total Sustaining Capex (incl. UG development, closure, net of salvage)	US\$ million	1,454	1,400	2,854
Total Capex	US\$ million	4,650	4,609	9,260
Corporate Taxation*	US\$ million	2,732	2,602	5,334
Net Cash Flow before Polish MET	US\$ million	11,689	11,130	22,819
MET	US\$ million	7,001	6,908	13,909
Net Cash Flow after Polish MET	US\$ million	4,688	4,222	8,910
NPV (Pre-Tax) at 7% Disc.	US\$ million	4,233	4,093	8,326
IRR (Pre-Tax)	%	20.6%	20.4%	20.5%
Payback (pre-tax, years after production startup)	Yrs	3.0	3.3	3.2
NPV (After Corporate Income Tax*) at 7% Disc.	US\$ million	3,214	3,101	6,315
IRR (After Corporate Income Tax*)	%	18.1%	17.9%	18.0%
Payback* Years after Production Startup)	Yrs	3.5	3.6	3.5
NPV After Corporate tax and Polish MET at 7% Disc.	US\$ million	866	738	1,604
IRR After Corporate Tax and Polish MET	%	11.1%	10.6%	10.8%
Payback (years after production startup)	Yrs	5.1	4.6	4.8
C₁ Cost Net of By-Product Credits	US\$/lb Cu	1.25	1.65	1.46
AISC Net of By-Product Credits	US\$/lb Cu	1.54	1.90	1.73
Break-Even Copper Price (NPV ₇ =0)	US\$/lb Cu	2.70	2.93	2.82
Average payable copper production (First 10 years)	kt Cu/a	140	150	290
Average payable silver production (First 10 years)	koz Ag/a	15,711	12,210	27,921
Avg. copper equivalent production (First 10 years)	kt Cu _{eq} /a	196	194	390
AISC net of by-product credits (First 10 years)	US\$/lb Cu	0.97	1.35	1.17
Average Annual EBITDA (First 10 years)	US\$ million/a	1,272	1,220	2,492
Capital intensity (US\$ per Cu _{eq} t/a) (First 10 years)	US\$/t/a	16,268	16,566	16,416
Net Cash flow before MET (First 10 years)	US\$ million/a	994	958	1,952
Net Cash flow after MET (First 10 years)	US\$ million/a	645	609	1,254
Avg. payable copper production (First 20 Years)	Cu kt/a	116	129	245
Average Payable Silver Production (20 Year)	Ag koz/a	13,635	9,767	23,402
Average Copper Equivalent Production (20 Year)	Cu _{eq} kt/a	165	164	328
Capital Intensity (US\$ per Cu _{eq} t/a) (20 Year)	US\$/t/a	19,427	19,593	19,509
NPV ₇ /Initial Capex (after corporate tax*)	US\$/US\$	1.0	1.0	1.0

* Excluding Copper and Silver Mineral Extraction Tax (Polish MET). Note: AISC = All-In Sustaining Costs

Under the base case assumptions, on a pre-tax basis the consolidated NPV at a discount rate of 7.0% (NPV₇) is estimated at approximately US\$8.33 billion, with a pre-tax IRR of approximately 20.5%. Both the North Shaft Complex and the South Shaft Complex generate positive economic returns in the base case, with the North Shaft Complex demonstrating marginally higher NPV, IRR, and a shorter payback period relative to the South Shaft Complex. Both Shaft Complexes are independent

of each other and can be developed simultaneously or in stages, offering flexibility on the speed of development and initial capital requirements.

After deducting Polish corporate income tax, applied at a rate of 19% after allowing for straight-line depreciation of capital assets over ten years, but excluding the MET, the consolidated Project generates a post-corporate tax NPV₇ of approximately US\$6.32 billion, with a post-tax IRR of approximately 18.0% and an estimated payback period of approximately 3.5 years from the commencement of production.

The impact of MET has been shown separately as it is anticipated that an amendment to the current legislation would be required for the Project to proceed. If MET were applied as currently legislated, the Project would nevertheless generate an NPV₇ of US\$1.60 billion and IRR of 10.8% which, whilst significantly reduced from the pre-MET results, still reflect a positive result under the base case assumptions of this PEA Report.

The economic results presented herein are forward-looking and are subject to a number of known and unknown risks and uncertainties, including, but not limited to changes in commodity prices, capital and operating cost estimates, metallurgical performance, geotechnical and hydrogeological conditions, permitting outcomes, environmental and social considerations, taxation and royalty regimes, and the ability to obtain financing. These factors could cause actual results to differ materially from those presented in this assessment.

Sensitivity Study

The sensitivity of the Project's NPV₇ and IRR to changes in prices, operating costs and capital costs were tested over a range of 40% above and below base case values. As is typical, both NPV₇ and IRR are most sensitive to copper prices and grade or recovery. Only at a 40% reduction in copper price (or grade) is the South Shaft NPV₇ reduced to marginally below zero, while the North Shaft NPV₇ remains positive across the range tested. The silver price exhibits moderate sensitivity, and the Project is least sensitive to changes in OPEX and CAPEX. At US\$6.00/lb Cu and US\$90/oz Ag, after tax NPV₇ and IRR are US\$17.7 billion and 29%, respectively, while after-MET NPV₇ and IRR remain strongly positive at US\$9.3 billion and 22%, respectively

Thus, the Project base case demonstrates reasonable economics within the ranges tested.

1.23 ADJACENT PROPERTIES

The Nowa Sól Property is situated within a well-established mining district in southwestern Poland and lies approximately 40 km northwest of the Deep Głogów (*Głogów Głęboki Przemysłowy*, GGP) an underground copper-silver mining area; the northernmost extension of the operations in the Copper Basin owned and operated by KGHM. The GGP mining area is a major contributor of ore produced by the Rudna and Polkowice-Sieroszowice mines, within the LGOM, extracting Kupferschiefer-type mineralization at depths near 1,385 m. KGHM's total throughput capacity is approximately 30 Mt/a with 401 kt/a of mined copper production, and 571 kt/a of refined copper production in 2025. In addition, KGHM's operated smelters in Głogów and Legnica import significant quantities of copper concentrates from overseas.

Although no direct comparison can be made, and the QPs have not verified the information relating to the GGP mining area, and other KGHM operations, aspects of its development (such as underground mining methods, infrastructure design, and operating parameters) may offer valuable context for future conceptual or economic assessments of the Project. In particular, regional infrastructure such as KGHM's smelters at Głogów and Legnica could represent potential processing options, subject to feasibility and commercial negotiations, should the Project proceed to production.

Additionally, Lumina maintains exclusive exploration rights to the Mozów and Sulmierzyce Projects, that target similar sediment-hosted, stratiform copper-silver mineralization within the Permian red-bed sequences of the Fore-Sudetic Monocline. Both properties host historic copper and silver Polish System resources of the same style of mineralization as the Project.

It is emphasized that any information concerning properties not owned by Lumina has not been independently verified by the QPs and is provided for informational purposes only. Such information should not be interpreted as indicative of the mineral potential or economic viability of the Project.

1.24 OTHER RELEVANT DATA AND INFORMATION

For the purposes of the PEA, a Room and Pillar mining method has been selected as the base case for the Nowa Sól Copper-Silver Project. However, as part of the PEA, a mining method trade-off study was conducted using a Drift and Fill mining methodology. The trade-off study does not represent an alternate mining scenario.

The Drift and Fill method involves mining individual drifts using mechanized equipment, followed by backfilling with cemented paste backfill before adjacent drifts are extracted. Residual rock pillars are retained for ground support. A conceptual panel (290 m wide by 580 m long) was modelled in Deswik using the same equipment assumptions as the Room and Pillar case to evaluate comparative productivity and scheduling performance.

The trade-off study indicates that Drift and Fill is less productive overall and has a slower ramp-up due to stricter sequencing and fewer available working faces, resulting in lower average production rates (1,100 t/d versus 1,300 t/d) and a longer extraction schedule (an additional 177 days). However, it offers improved panel recovery (approximately 70% compared to 61.5% for Room and Pillar) and is better suited to automation and mechanisation, with potential safety and productivity benefits. Further optimization of sequencing and incorporation of geotechnical considerations are recommended for future evaluations.

1.25 INTERPRETATION AND CONCLUSIONS

1.25.1 Conclusion

The Project demonstrates potential for the development of an underground copper-silver mining operation, with silver credits; however, the results of this preliminary economic assessment are subject to the limitations and uncertainties inherent to this level of study.”

The Mineral Resource is classified as Measured, Indicated and Inferred, as summarized in Table 14.20. The estimate was prepared in accordance with the CIM Best Practice Guidelines (2019) and reported in accordance with the CIM Definition Standards (2014), as incorporated by reference into NI 43-101.

In the QP’s opinion, the Mineral Resources reported demonstrate RPEEE, based on the selected cut-off grade and the application of appropriate mining and processing assumptions. Mining envelopes used to identify spatially continuous mineralization within potentially mineable shapes using reasonable assumptions based on neighbouring operations and long-term price trends to calculate a pay limit Cu grade. After assessing the optimal cut-off grade, a target head grade was selected at 0.95% Cu to ensure maximum mineable material was included into the mining envelopes. The mining envelopes were used to constrain the block model and identify material that meets RPEEE and were considered for inclusion in the Mineral Resource Statement. The significant Ag content in mineral resource constitutes an improvement opportunity if considered in the mining envelopes cut-off computation that should be investigated at a later stage of the Project.

A conceptual mine design and LoM schedule have been developed for two independent Shaft Complexes, namely the North and South Shaft Complexes, utilising conventional Room and Pillar mining methods targeting an annual production of 11.5 Mt/a, which offer optionality to be developed simultaneously or in phases.

The preliminary flowsheet developed from the testwork program is considered conventional and comprises primary grinding to a P80 size of 60 μm , rougher flotation, rougher concentrate regrinding to approximately 11 μm and two stages of cleaning. The results from the preliminary metallurgical development testwork program suggested copper and silver recoveries into a concentrate grading 26.2% Cu are 88.8% and 86.6%, respectively.

Both Shaft Complexes provide attractive economic returns, though the NPV₇, IRR and payback period, but for the North all exceed the corresponding parameters for the South. The combined pre-tax NPV₇ for both Shaft Complexes is US\$8.33 billion with an IRR of 20.5% and a payback period of 3.2 years.

The PEA demonstrates that the Project has potential to support further technical evaluation and advancement to a PFS. However, the results are preliminary in nature and subject to significant technical, economic, and regulatory uncertainties that will require additional work to address.

1.25.2 Risks and Uncertainties

The Project exhibits typical PEA-stage risks, in some cases elevated given the deep underground mining environment. The most material exposures relate to:

- Geotechnical conditions (high stress & seismicity)
- Backfill system performance at extreme depth
- Salt-related shaft construction risks
- Fiscal exposure (Polish MET)
- Conceptual-level cost estimates

The reported MRE represents a high-level technical evaluation and does not constitute a Mineral Reserve. As such, it is subject to the inherent risks associated with mineral exploration and resource development. These include geological uncertainty, the potential for variability in mineral continuity, fluctuations in commodity prices, and changes in permitting, environmental, regulatory, or socio-political frameworks.

The Inferred classification within the MRE and production schedule is based on limited geological evidence and sampling, where geological and grade continuity are implied but not verified. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to higher confidence categories with continued exploration; however, there is no certainty that such upgrading will occur.

Risks associated with metallurgy, environment, hydrogeology and infrastructure are considered moderate.

Most technical risks are reducible through planned drilling in Q2 2026, metallurgical testwork, geotechnical modelling, and Pre-Feasibility engineering. However, fiscal and commodity price risks remain external.

The reader is cautioned that while the Project has demonstrated technical advancement, further work will be required to assess Project feasibility and advance the Mineral Resources towards potential development.

1.26 RECOMMENDATIONS

1. It is recommended that the Project advance a focused program of technical, environmental and engineering studies to support progression towards a PFS. Key priorities include refining the MRE through further investigation of high-grade copper populations and advanced multivariate modelling and optimising the mine planning to better align development timing with production requirements.
2. Investigate the potential for additional silver production from less copper-rich areas of the deposit based on a copper-equivalent cut-off grade.
3. Further work is recommended to optimize the underground mine shaft design, particularly with respect to ventilation, cooling and shaft configurations. This includes advancing ventilation and refrigeration modelling, incorporating additional contingency into development and costing assumptions, and assessing alternative shaft sinking, sequencing and lining concepts that may improve schedule efficiency and reduce time to first production. Comparative evaluation of alternative shaft layouts should be undertaken, considering capital cost, operating cost, schedule and geotechnical risk.
4. Backfill and dry stack tailings management should be progressed through detailed tailings characterisation using drill core from the 2026 drilling campaign. Metallurgical testwork should be increased to support the next study phase, including comminution, flotation optimization, locked-cycle testing, dewatering, and evaluation of potential lead and zinc by-product recovery, as well as alternative processing technologies.
5. Infrastructure opportunities, particularly the potential reinstatement of nearby rail access, should be actively monitored and pursued through early engagement with relevant authorities to assess feasibility and integration into the Project's logistics strategy.
6. Environmental and social studies should be advanced through comprehensive baseline data collection, hydrology and hydrogeology investigations, and continued stakeholder and community engagement, with the aim of informing future permitting, design and impact assessments.

1.26.1 Future Work and Budget

The Project has produced a promising maiden MRE, indicating the potential for economically viable extraction. The QPs recommends advancing to the next stage of work, which includes initiating a PFS and conducting additional investigative activities. This would involve further drilling, metallurgical and geotechnical testing, hydrogeological studies, and establishing environmental and social baselines. The PFS should explore key trade-offs such as mine life options, throughput rates, potential for backfilling, and ventilation and cooling solutions.

To support regulatory compliance and enhance technical understanding, Lumina plans to drill five (5) diamond drill holes starting in Q2 2026. This initiative is endorsed by the QPs as part of the recommended next phase. Alongside this, environmental and social baseline studies should commence immediately to inform the Environmental and Social Impact Assessment (ESIA) and facilitate the environmental permitting process. These studies, essential for securing a Mining Concession from the MCE and adhering to international best practices, must span all seasons over a 12-month period based on regulatory requirements.

The estimated costs for the recommended future work totals approximately US\$32,000,000. This includes estimated costs of drilling geological, geotechnical, hydrogeological and metallurgical drill holes of US\$25,000,00 with an additional cost of US\$7,000,000 budgeted for the PFS.

The proposed work program and budget are based on the current level of Project knowledge and are consistent with industry practice for advancing a project to PFS stage. The QP considers the proposed work to be reasonable and appropriate to further evaluate the technical and economic potential of the Project.

2 INTRODUCTION

2.1 ISSUER

The Project (or Property) is owned by Lumina Metals Corp. (Lumina). Lumina is a private company, incorporated on November 24, 2010, in British Columbia, Canada, under the corporate number BC0896163. Initially operating as Miedzi Copper Corporation, the company rebranded to Lumina on December 16, 2024. Through its subsidiary, Zielona Góra Copper Sp. z o.o., Lumina holds 100% of the Project, located approximately 320 km west of Warsaw, the capital city of Poland.

In July 2025, Lumina commissioned Micon International Co Limited (Micon) to conduct a Preliminary Economic Assessment (PEA) of the Project prepared in accordance with National Instrument 43-101 (NI 43-101). This Report summarizes historical and current exploration data, other relevant technical information, outlines proposed mining and processing methods, and follows the NI 43-101 disclosure standards. The latest Mineral Resource Estimate (MRE), prepared by Micon in January 2026, is included in this Report, however Mineral Reserves are excluded. The Report serves as a comprehensive technical assessment of the Project.

Micon's QPs assert that the technical data is reliable and that the QPs have no conflicts of interest. The QPs meet the professional standards required under NI 43-101, ensuring an objective and accurate assessment of the Project.

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

2.2 TERMS OF REFERENCE

On July 29, 2025, Lumina commissioned Micon to prepare an independent NI 43-101 Technical Report (the Report) for the Project.

The Report provides a PEA of the Project and incorporates the updated MRE prepared by Micon in January 2026. It aims to support the Project's ongoing development. This Report does not include any Mineral Reserve estimates.

The primary purpose of this Report is to align with the disclosure requirements of National Instrument 43-101 - *Standards of Disclosure for Mineral Projects* (NI 43-101) as set out by the Canadian Securities Administrators (CSA).

The Qualified Persons (QPs) responsible for the preparation of this Report are Liz de Klerk, Pr.Sci.Nat., QMR (Micon Managing Director); Dr. Ryan Langdon, Ph.D, CGeol; Richard Gowans P.Eng; Garth Matti Liukko, P.Eng.; Ben Cottrell P.Eng; Alex Zaitchenko P.Eng; Peter Stevens CGeol; Justin Taylor, P.Eng.; Becky Humphrey CEnv, MIMMM; and Christopher Jacobs CEng, MIMMM.

Lumina has undertaken a comprehensive review of this Report, focusing primarily on verifying factual information. Any modifications implemented due to this review process did not alter the fundamental interpretations or conclusions presented by the QPs. All statements and opinions expressed within this document are made in good faith and are believed to be accurate and free from any misleading content as of the effective date of this Report.

Influences beyond the control of Micon's QPs, geological conditions, the capabilities of management and personnel, the availability of funding for proper operation and capitalization, fluctuations in cost elements and market conditions, efficient development and operation of the Project, and unforeseen changes in legislation and new developments in the industry renders the

statements and estimates subject to unknown risks. Any of these factors may significantly impact the performance of any exploration or mining operation.

2.3 PRINCIPAL SOURCES OF INFORMATION

Micon has reviewed the Property based on information provided by Lumina, technical reports by contractors and associates, and other relevant published and unpublished data. Micon has independently reviewed, analyzed, and interpreted the data provided to arrive at the required conclusions and recommendations. In cases where data was not provided, historical reports by previous owners were used as a reference if they contained technical data and information for the Property.

Technical data and information were provided to Micon as follows:

- **Mr. Leo Hathaway:** provided Micon with license details, original exploration data, and original reports pertaining to the Property.
- **Mr. Adrian Karolko:** provided additional data updates, including maps and photographs.
- **Mr. Krzysztof Napierała:** provided additional data updates.

Additional geological references relevant to the Project have been studied. This data is considered public domain, and the specific papers and documents consulted are listed in Section 27. Lumina acquired additional technical data and further information through discussions with Leo Hathaway, Adrian Karolko, and Krzysztof Napierała.

The district and regional geological data were obtained from public domain sources and have been appropriately referenced. Micon has recreated or modified figures and tables where applicable and cited the original source data accordingly.

The historical MREs documented in Section 6.3 relied on the reports/documents listed below:

1. “Scoping Study and Economic Analysis of Development Options on the Nowa Sól Copper Project, Southwestern Poland” by Bohnet Global Mining Engineering Consultants, LLC (BGMEC) for Miedzi Copper Corp, dated June 6, 2020.
2. Expansion and Addendum No. 1: In January 13, 2023, following additional drilling and geological studies, the resource classification was expanded and upgraded via Addendum No. 1.
3. Upgrading to C₁ Resource Classification: In 2024, the company drilled five additional drill holes (C27, C27st, C29, C31 and C31st) on the Nowa Sól Concession. These efforts aimed to upgrade the deposit’s classification from C₂ to C₁. Addendum No. 2 was as approved by the *Ministerstwo Klimatu i Środowiska’s* (Ministry of Climate and Environment) (MCE) resource commission on April 25, 2025.

The QPs and Lumina have been unable to conduct sufficient work to verify or classify these historical estimates as current mineral resources in accordance with NI 43-101. Lumina does not treat the historical estimate as current Mineral Resources.

The QPs have made every effort to verify the validity, precision, and entirety of the technical information on which this Report is founded by conducting thorough investigations.

2.4 QUALIFIED PERSON SITE VISIT AND PERSONAL INSPECTION

The Project has been visited on several occasions by various members of the Micon team as detailed.

Liz de Klerk, an Author and QP of this Report, visited the Property between February 12, 2025, and February 14, 2025, accompanied by Matthew Ball (Micon Resource Geologist) and Andrzej Zibrow and Tomasz Bieńko, Lumina personnel.

While at site the following were inspected / visited:

- The Nowa Sól and Mozów deposit areas.
- The Sulmierzyce Północ deposit area.
- Polish State Geological Survey Archives.
- Michałów core storage facility at Michałowice.
- Lumina's Core storage facility at Kraków.

The following Micon staff, including QPs and authors of this Report, visited the Property between July 1, 2025, and July 3, 2025, accompanied by Lumina personnel: Liz de Klerk, Dr. Ryan Langdon, Peter Szkilnyk, Richard Gowans and Becky Humphrey.

While at site the following were inspected / visited:

- North, central and south zones within the Nowa Sól exploration concession.
- Nearby villages, waterbodies and protected areas for context.
- Odra River and Nowa Sól Port.
- Lumina's Core storage facility at Kraków.

Meetings were held at Lumina's company headquarters in Warsaw with management personnel during both the February 2025 and July 2025 visits.

In addition, Liz de Klerk visited the University of Wrocław laboratory between September 27, 2025 and September 28, 2025 to oversee the sample dispatch of the metallurgical samples sent to XPS laboratory in Canada.

Ben Cottrell and [David Walker] visited Lumina's core logging facility on December 12, 2025 accompanied by Lumina personnel. While on-site they carried out a review of the geotechnical and hydrogeological characteristics of drill core from several Nowa Sól boreholes and reviewed the 2026 drill hole planning. A memo with recommendations for the 2026 drill hole program was sent to Lumina on December 15, 2025.

2.5 QUALIFIED PERSONS

This Report was prepared by the QPs detailed in Section 2.2 and summarized in Table 2.1, with their respective contributions and responsibilities outlined in Table 2.2.

The Report's authors are QPs, as defined by NI 43-101, possessing the requisite experience, education, and professional standing to take responsibility for the sections of the Report under their purview.

Micon has conducted an internal review to confirm the absence of any conflicts of interest concerning its engagement in this Project or with Lumina. Furthermore, Micon has ascertained that no circumstances could compromise the objectivity or professional judgment of the QPs involved in preparing this Report.

Table 2.1: Qualified Persons Names and Details

Name	Initials	Position	Employer	Professional Designation
Mrs. Liz de Klerk	LdK	Managing Director	Micon International Co Limited	Pr.Sci.Nat., QMR
Dr. Ryan Langdon	RL	Principal Resource Geologist	Micon International Co Limited	Ph.D., CGeol
Richard Gowans	RG	Principal Metallurgist	Micon International Co Limited	P.Eng.
Garth Matti Liukko	GL	Principal Mining Engineer	Micon International Limited	Pr.Eng.
Ben Cottrell	BC	Geotechnical Engineer	Global Mine Design	P.Eng.
Alex Zaitchenko	AZ	Civil Engineer	Micon International Limited	P.Eng.
Peter Stevens	PS	Director and Principal Engineering Geologist	CQA International Limited	CGeol
Becky Humphrey	BH	Environmental Specialist	Micon Associate	CEnv., MIMMM
Christopher Jacobs	CJ	Principal Mineral Economist	Micon International Limited	CEng., MIMMM
Justin Taylor	JT	Director, Principal Mechanical Engineer	Micon International Co Limited	P.Eng.

Source: Micon, 2026.

Table 2.2: Qualified Persons and Report Responsibilities

Name	Initials	Section Responsibility
Mrs. Liz de Klerk	LdK	Sections 1.1 to 1.10, 1.12, 1.14, 1.15, 1.19.1, 1.23, 1.25, 1.26, 2, 3 4 (except 4.2.5), 5 6, 7, 8, 9, 10, 12 (except 12.2), 14.11 (except 14.11.3), 15,19.1. 23, 25.1, 25.2.1, 25.2.3, 26.2, 26.9 and 27
Dr. Ryan Langdon	RL	Sections 1.11, 1.12, 11, 12.2, 14.1 to 14.10 and 14.11.3
Richard Gowans	RG	Sections 1.13, 1.17, 13, 17, 25.2.2 and 26.7
Garth Matti Liukko	GL	Sections 1.16.1, 1.16.3, 16.5 and 26.3
Ben Cottrell	BC	Sections 1.16.2, 1.24, 16.1, 16.2, 16.6, 24, 25.2.4, 25.2.5, 25.2.6, 26.1, 26.4, 26.5 and 26.6
Alex Zaitchenko	AZ	Section 16.3
Peter Stevens	PS	Sections 16.4 and 25.2.9
Becky Humphrey	BH	Sections 1.20, 4.2.5, 20, 25.2.8 and 26.8
Christopher Jacobs	CJ	Sections 1.21, 1.22, 1.19.2, 19.2, 21, 22 and 25.2.10
Justin Taylor	JT	Sections 1.18, 18 and 25.2.7

Source: Micon, 2026.

2.6 UNITS OF MEASURE

All units of measurement used in this Report are metric unless otherwise stated. Tonnages are reported as metric tonnes (t), precious metal values (gold and silver) in grams per tonne (g/t) or parts per million (ppm) and base metal values (tin, copper, lead, and zinc) are reported in weight percent (%) or ppm. Other references to geochemical analysis are in ppm or parts per billion (ppb) as reported by the originating laboratories. Unless otherwise stated, all currency amounts and commodity prices are stated in United States dollars (US\$) or Canadian dollars CD\$. The coordinate system utilized in this Report is WGS84 Zone 33N projection (EPSG:32633), unless otherwise specified. A summary of the units of measure is provided in Table 2.3.

Table 2.3: Units of Measure

Unit	Abbreviation	Unit	Abbreviation
Ampere	A	Microns	µm
Annum (year)	a	Metre	m
Billion	B	Millions of years ago	Ma
Bond Ball Mill Work Index	BBWi	metres above sea level	masl
Bond Rod Mill Work Index	BRWi	Metres per day	m/d
Cubic feet per minute	CFM	Metres Height	mH
Centimetre	cm	Milligram	mg
Cubic centimetre	cm ³	Milligrams per litre	mg/L
Cubic metre	m ³	Millilitre	mL
Day	d	Millimetre	mm
Days per year (annum)	d/a	Metres Wide	mW
Dry metric tonne	dmt	Megawatts	MW
Degree	°	Million	M
Degrees Celsius	°C	Million cubic metres	Mm ³
Diameter	Ø	Million tonnes	Mt
Dollar (American)	US\$ or USD	Million tonnes per annum	Mt/a
Dollar (Canadian)	CD\$	Million tonnes per day	Mt/d
Foot	ft	Minute (time)	min
Gram	g	Ounce	oz
Grams per litre	g/L	Pascal	Pa
Grams per cubic metre	g/m ³	Megapascal	MPa
Grams per tonne	g/t	Gigapascal	GPa
Greater than	>	potential hydrogen	pH
Hour	h	Gigawatt hours per annum	GWh/a
Hours per day	h/d	Parts per billion	ppb
Hours per year	h/a	Parts per million	ppm
Hertz	Hz	Percent and pct	%
Kilo (thousand)	k	Second (time)	sec
Kilogram	kg	Specific gravity	SG
Kilograms per hour	kg/h	Thousand tonnes	kt
Kilograms per cubic metre	kg/m ³	Three dimensional	3D
Kilograms per square metre	kg/m ²	Metric tonne (1,000 kg)	t
Kilometre	km	Tonnes per day	t/d
Square kilometre	km ²	Tonnes per hour	t/h
Kilometres per hour	km/h	Tonnes per year (annum)	t/a
Kilotonne	kt	Tonnes per cubic metre	t/m ³
Kilovolt	kV	Tonnes per metre squared per hour	t/m ² .h
Kilowatt	kW	United States dollars per troy ounce	US\$/oz
Kilowatt hours	kWh	Volt	V
Kilowatt hours per tonne	kWh/t	wet metric tonne	wmt
Less than	<	Weight percent	Wt%
Litre	L	Years	Yrs
Percentage mass	%m		

Source: Micon, 2026.

2.7 ABBREVIATIONS, ACRONYMS AND TRANSLATED TERMINOLOGY

Abbreviations and acronyms used in this Report are listed in Table 2.4. It should be noted that where appropriate throughout the report text, (e.g. Sections 4 and 20) the Polish language equivalent for key legal terminology is provided in brackets and italicised text after the English wording for reference. In keeping with standard scientific writing methods, Section 20.3.3 also contains italicised Latin names for wildlife species, with the English equivalent in brackets afterwards.

Table 2.4: Abbreviations and Acronyms

Unit	Abbreviation / Acronym	Unit	Abbreviation / Acronym
Atomic Absorption Spectroscopy	AAS	metres above sea level	masl
Acme Analytical Laboratories Ltd	Acme Labs	Maximum	Max.
Silver	Ag	Ministry of Climate and Environment / <i>Ministerstwo Klimatu i Środowiska</i>	MCE
All-In Sustaining Costs	AISC	Mineral Extraction Tax	MET
Aluminum	Al	Magnesium	Mg
Acid Production Potential	AP	Methyl Isobutyl Carbinol	MIBC
Arsenic	As	Micon International Co Limited	Micon
Gold	Au	Minimum	Min.
Bohnet Global Mining Engineering Consultants, LLC	BGMEC	Molybdenum	Mo
Blair Multi-Rope	BMR	Manganese	Mn
Carbon	C	Local Spatial Development Plan / <i>Miejscowy Plan Zagospodarowania Przestrzennego</i>	MPZP
Calcium	Ca	Maximum Potential Acidity	MPA
Calcium Carbonate	CaCO ₃	Mineral Resource Estimate	MRE
Compound Annual Growth Rate	CAGR	Mineable Shape Optimiser	MSO
CDN Resource Laboratories	CDN	Sodium	Na
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM	Nitrogen	N
Chlorine	Cl	Nickel	Ni
Cobalt	Co	National Instrument 43-101	NI 43-101
Carbon Monoxide	CO	<i>Narodowy Instytut Dziedzictwa</i>	NID
Carbonate	CO ₃	Nearest Neighbour	NN
Chromium	Cr	Neutralizing Potential	NP
Certified Reference Material	CRM	Net Neutralizing Potential	NNP
Certified Reference System	CRS	Neutralization Potential Ratio	NPR
Committee For Mineral Reserves International Reporting Standards	CRIRSCO	Net Present Value	NPV
Canadian Securities Administrators	CSA	Non-Sulphide Gangue	NSG
Copper	Cu	Net Smelter Return	NSR
Copper / Iron	CuFe	Nowa Sól Substation	NWS
Copper Equivalent	Cu _{eq}	Oxygen	O
Copper / Sulphide	Cu-S	Ordinary Kriging	OK
Coefficient of Variation	CV	<i>Ocena oddziaływania na środowisko</i>	OOŚ
Delivered-at-Place	DAP	Potassium Amyl Xanthate	PAX
Discounted Cash Flow	DCF	Lead	Pb
Diamond Drill Hole	DDH	Palladium	Pd
Differential Global Positioning System	DGPS	Preliminary Economic Assessment	PEA
DMC Mining Services Ltd	DMC	Pre-Feasibility Study	PFS
Dry Stack Tailings Facility	DSTF	<i>Polskie Górnictwo Naftowe i Gazownictwo</i>	PGNiG
		<i>Państwowy Instytut Geologiczny (Polish Geological Institute)</i>	PIG

Exploratory Data Analysis	EDA
Energy Dispersive X-ray Spectroscopy	EDS
Environmental Impact Assessment	EIA
Electrical Instrumentation and Control	EI&C
Environmental Impact Report	EIR
End of Hole	EOH
Equator Principles	EP
Engineering, Procurement, and Construction Management.	EPCM
Electron Probe Microanalysis	EPMA
European Petroleum Survey Group	EPSG
Environmental, Social and Governance	ESG
Environmental and Social Impact Assessment	ESIA
EXALO Drilling S.A.	Exalo
European Union	EU
Iron	Fe
Iron / Chromium	FeCr
General and Administrative	G&A
Geographic Coordinate System	GCS
Good International Industry Practice	GIIP
Global Industry Standard for Tailings Management	GISTM
Głogów Głęboki Przemysłowy	GGP
Gosudarstvennaya Komissia po Zapasam / State Reserves Committee	GKZ
Global Mine Design	GMD
Global Positioning System	GPS
Geological Strength Index	GSI
Water	H ₂ O
Hydrochloric acid	HCl
Perchloric acid	HClO ₄
High-Density Polyethylene	HDPE
Hydrofluoric acid	HF
Nitric acid	HNO ₃
High voltage	HV
Heating, Ventilation, and Air Conditioning	HVAC
Important Bird Area	IBA
International Council on Mining and Metals	ICMM
Inductively Coupled Plasma Optical Emission Spectroscopy	ICP-OES
Inductively Coupled Plasma Mass Spectrometry	ICP-MS
International Copper Study Group	ICSG
Inverse Distance	ID
Babimost Airport	IEG

Polish Geological Institute – National Research Institute	PIG-PIB
Platinum	Pt
Polish złoty	PLN
General Municipal Plan / <i>Plan ogólny gminy</i>	POG
Poznań Airport	POZ
Personal Protective Equipment	PPE
Projection Pursuit Multivariate Transform	PPMT
Principles for Responsible Investment	PRI
<i>Polskie Sieci Elektroenergetyczne S.A.</i> (Polish Transmission System Operator)	PSE
Portable X-Ray Fluorescence	pXRF
Photovoltaic	PV
Deposit Development Plan / <i>plan zagospodarowania złoża</i>	PZZ
Quality Assurance/Quality Control	QA/QC
Qualified for Minerals Reporting	QMR
Qualified Person	QP
<i>Regionalna Dyrekcja Lasów Państwowych a</i> (Regional Directorate of State Forest Service)	RDLP
<i>Regionalny Dyrektor Ochrony Środowiska</i> (Regional Directorate for Environmental Protection)	RDOŚ
Rhenium	Re
Rare Earth Elements	REE
Reasonable Prospects for Eventual Economic Extraction	RPEEE
Rougher	Ro
Sulphur	S
Special Area of Conservation	SAC
Semi-Autogenous Grinding	SAG
Standard Deviation	SD
Selenium	Se
Silicon	Si
Sulphate	SO ₄
Secondary Oxidized System	S.O.S
Special Protected Area for Birds	SPA
Strontium	Sr
Standard	Std.
Total Direct Costs	TDC
Titanium	Ti
Total Plant Mechanical Costs	TPMC
Unconfined Compressive Strength	UCS
Unconfined Compressive Strength rock mass	UCS _{rm}
United Nations Environment Program	UNEP
United Nations Educational, Scientific and Cultural Organization	UNESCO
Uninterrupted Power Supply	UPS

International Finance Corporation Environmental and Social Performance Standards	IFC PS
Internal Rate of Return	IRR
International Organization for Standardization	ISO
Information Technology	IT
Potassium	K
<i>Komisja Dokumentacji Geologicznych</i> Geological Documentation Commission	KDG
KGHM Polska Miedź S.A.	KGHM
Life of Mine	LoM
Polskie Linie Lotnicze LOT S.A.	LOT
Process Design Criteria	PDC
Load Haul and Dump	LHD
<i>Zagłębie Miedziowe</i> Legnica-Głogów Mining District	LGOM
London Metal Exchange	LME
Liquefied Natural Gas	LNG
Lumina Metals Corp.	Lumina

Source: Micon, 2026.

Universal Transverse Mercator	UTM
Vanadium	V
Variation	Var.
Weighted Average Cost of Capital	WACC
World Bank Environmental, Health and Safety Guidelines	WB EHS
World Bank Group	WBG
World Geodetic System	WGS
Wrocław Airport	WRO
Wrocław University of Science and Technology	Wrocław Tech
XPS Industry Relevant Solutions	XPS
X-Ray Difference	XRD
X-Ray Fluorescence	XRF
Portable X-Ray Fluorescence	pXRF
Zinc	Zn

3 RELIANCE ON OTHER EXPERTS

The QPs have relied on information provided by Lumina concerning legal matters relevant to the Report including permitting and concession status. In connection with these matters, the QPs have relied upon official concession details for two contiguous exploration concessions, namely Nowa Sól II (9/2025/p) and Konotop (12/2025/p), obtained from Ministry of Climate and Environment (MCE) dated October 28, 2025 and this information is used in Section 20.1.4. of this Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION AND AREA OF THE PROPERTY

Nowa Sól is an advanced-stage copper and silver exploration project in the Republic of Poland. It is located in the western Lubuskie Province ((*Województwo Lubuskie / Voivodeship*)), approximately 320 km west of the Warsaw capital, 100 km northwest from the city of Wrocław, 85 km east of the border with Germany and 120 km north of the border with Czechia (Figure 4.1).

The Nowa Sól Concession encompasses a surface area of approximately 120 km² and is 17.1 km at its longest extent from north to south, and 10 km at its widest extent from east to west. The topography is relatively flat, with gently undulating lowlands that form part of the larger Polish Plain and elevations ranging from 60 m to 110 metres above sea level (masl). The Odra River flows to the west of the concession boundary and is 1 km away at its closest point.

The concession spans five administrative municipalities (communes / *gmina*) within three larger administrative districts (counties / *powiat*). There are several small residential settlements within the concession, and the nearest urban areas are the town of Nowa Sól, approximately 6 km southwest of the concession boundary, and the city of Zielona Góra, approximately 25 km to the west. The surrounding area is characterized by rivers, lakes and forests.

Figure 4.1 illustrates the location of the Nowa Sól II Concession and other nearby explorations areas currently owned by Lumina.

4.2 MINERAL TENURE

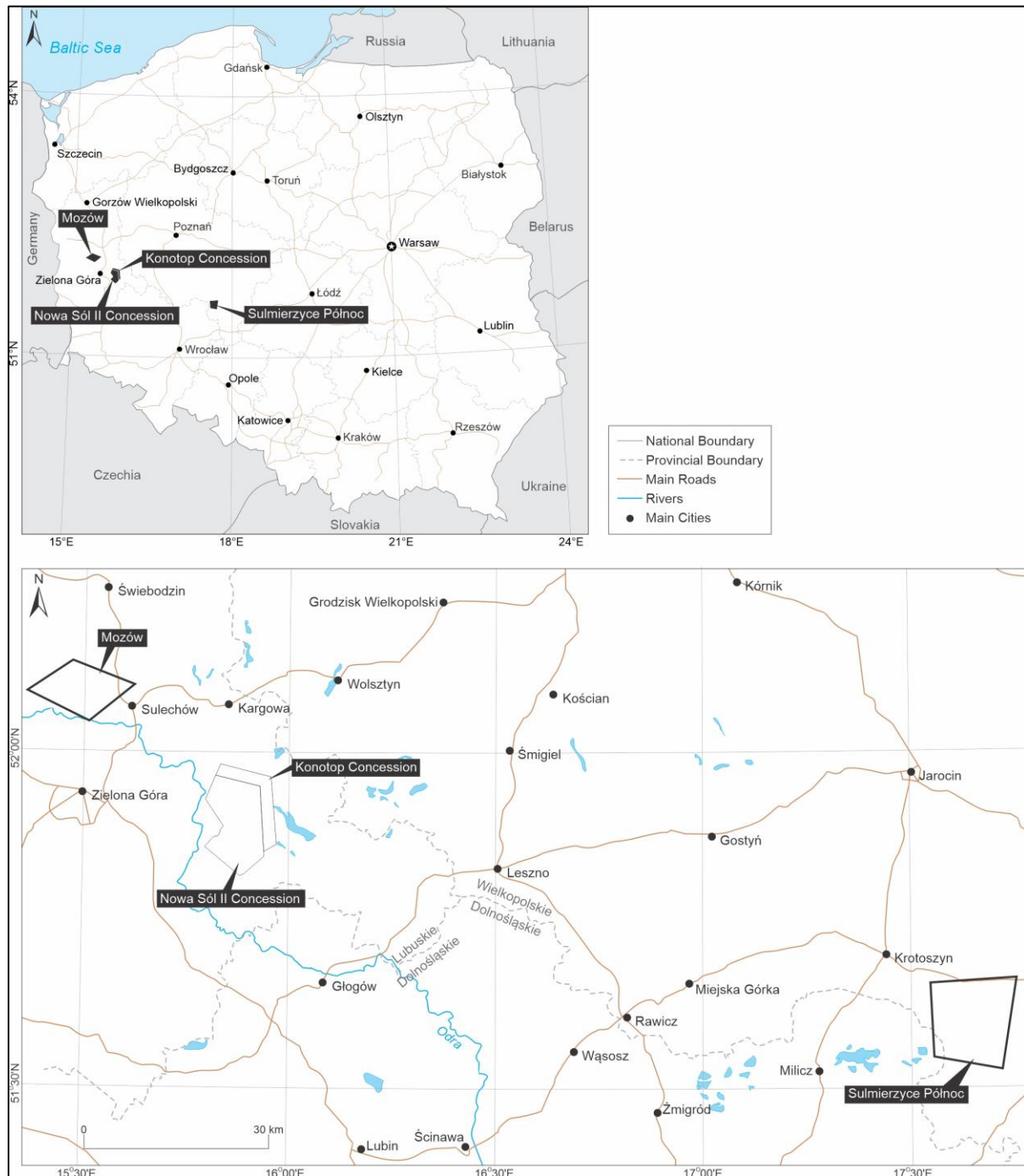
Mineral exploration and mining activities in Poland are overseen by the MCE and administered at local level by the relevant District Mining Office (*Okręgowy urząd górniczy*). An overview of Polish Mining legislation is provided in Section 20.1.

Under Polish law, there are five resource classification categories which can be applied to a solid mineral deposit. A resource classification of C₁ or higher is required before a mining concession can be applied for. The Project was granted a C₁ designation by MCE on April 28, 2025.

The C₁ resource category is approved by way of an administrative decision issued by the competent geological authority following a formal review of the geological documentation of the deposit. For stratiform copper-silver deposits and other projects of national importance, the documentation is typically evaluated by the Geological Documentation Commission (*Komisja Dokumentacji Geologicznych* – KDG), an expert advisory body to the Ministry of Climate and Environment (MCE). The review is substantive and results in detailed comments addressing, among other things, legal compliance, correctness of the applied exploration and estimation methodology, resource calculations, application of balance (economic) criteria, and the definition of deposit boundaries and geometry. Where deficiencies are identified, the authority may require supplementation of the documentation, additional geological work, or, in the case of material non-compliance, refuse approval.

Approval of C₁ resources constitutes a key prerequisite for applying for an extraction (mining) concession and confirms a high degree of geological confidence in the continuity, thickness and grade of the mineralization. In accordance with the Polish classification system, the C₁ category corresponds to an estimated error of the mean values of the principal deposit parameters not exceeding ±30%. Upon approval, the documented resources are entered into the National Mineral Resources Balance maintained by the Polish Geological Institute – National Research Institute (PIG-PIB), and the deposit becomes formally recognized within the Polish legal and planning framework.

Figure 4.1: Regional Location of the Nowa Sól II and Konotop, Mozów and Sulmierzyce Północ Concessions in Poland



Source: Micon, 2026.

The approval of the geological documentation grants Lumina an exclusive right to use the geological information for a period of five years and a priority right to apply for a mining usufruct (*użytkowanie górnicze*) and an extraction concession also for a period of five years. In the case of the Nowa Sól Project, this exclusivity period, which expires on April 24, 2030, secures Lumina's priority position to advance the Project through the preparation of technical studies, environmental permitting, and the concession application process. Further information on the resource classification and permitting process can be found in Section 20.1.

Lumina currently holds exploration concessions (permits) for Nowa Sól and the adjacent Konotop Concession through its wholly-owned Polish subsidiary, Zielona Góra Copper Sp. z o.o. The Nowa Sól II Concession (No. 9/2025/p) was granted on October 28, 2025, for a period of 3.5 years and covers an area of 119.151 km² (Figure 4.1). Upon expiry, Lumina can apply for a new concession covering the same area or amending the boundaries, as needed and warranted by the results of the exploration program conducted. The above-mentioned priority and exclusivity rights remain in effect, independently of the expiry of the exploration concession permit, and they elapse upon the expiration of the five-year period.

The Nowa Sól II license coordinates are provided in Table 4.1. Lumina also holds exploration concessions for the nearby Mozów and Sulmierzyce Północ deposits (Figure 4.1) both hosting historic copper-silver resources with similar style mineralization documented as Polish resource C₂+D categories.

The Konotop Concession (No. 12/2025/p) was granted on December 9, 2025 and is valid for a period of 6 years and covers an area of 39.684 km² (Figure 4.1). This area deposit does not yet have a C₁ resource classification but, subject to further successful exploration activity, it is anticipated that it could eventually become part of the proposed Project and host similar style mineralization. The license coordinates are provided in Table 4.2.

Table 4.1: Nowa Sól II Exploration Concession Coordinates

Point	X	Y	No.	X	Y
1	561605.92	5740316.56	19	557396.95	5747049.11
2	560916.99	5739978.32	20	557444.15	5747117.59
3	555022.46	5744933.06	21	557489.10	5747194.25
4	556487.20	5746553.92	22	557503.35	5747220.97
5	556522.92	5746556.19	23	557548.87	5747323.60
6	556622.21	5746571.26	24	557577.34	5747410.27
7	556624.78	5746571.90	25	557582.21	5747425.56
8	556721.16	5746592.82	26	557605.57	5747526.97
9	556819.62	5746622.97	27	557618.95	5747627.84
10	556917.53	5746663.39	28	557623.04	5747728.19
11	556969.28	5746690.97	29	557619.41	5747806.89
12	557014.91	5746713.30	30	558167.02	5748412.94
13	557111.70	5746773.90	31	555202.95	5752939.80
14	557146.47	5746800.84	32	556857.97	5756915.60
15	557207.79	5746847.47	33	564306.05	5755061.38
16	557272.35	5746907.89	34	565074.18	5744116.56
17	557302.96	5746937.51	35	565025.40	5743346.27
18	557368.78	5747013.32	-	-	-

Note: Coordinates have been converted into UTM WGS84, Zone 33 North from exploration concession documents.

Table 4.2: Konotop Extension Concession Coordinates

Point	X	Y
1	564306.05	5755061.38
2	556858.00	5756915.60
3	557723.20	5758830.77
4	566198.98	5756650.79
5	567085.94	5745284.02
6	565074.18	5744116.56

Note: Coordinates have been converted into UTM WGS84, Zone 33 North from exploration concession documents.

4.2.1 Surface Rights and Legal Access

Lumina previously secured temporary surface rights for exploration by leasing land from private, corporate, and public owners. After completing exploration activities in 2024, all leased properties were restored to their original condition, in compliance with legal requirements. Additional exploration activities conducted under the new Nowa Sól II Concession will require similar agreements. For the purposes of conducting proposed production activities, it would be necessary to obtain legal title to real estate (land) where the mining infrastructure is to be located.

Each documented mineral deposit is registered in the Balance of Mineral Resources in Poland (*Bilans zasobów złoż kopalin Polski*). Lumina has engaged in preliminary discussions with various authorities in the Lubuskie Province (*Województwo Lubuskie / Voivodeship*) to implement respective zoning measures ensuring that the Nowa Sól deposit is duly recognized. Currently the deposit, as registered in the Balance of Mineral Resources following the C₁ category recognition, is observed for the purposes of local zoning measures. Lumina has introduced provisions for the general zoning plans in the respective municipalities informing of the existence of the deposit. Due to the early stage of Project development, and no certainty around the precise location of the surface infrastructure of the project, local detailed zoning plans will be amended at a later date.

4.2.2 Water Rights

Lumina does not yet hold any water rights for the Project. The Project intends to use water recovered from dewatering of the underground mining area, subject to further technical studies. These future studies will include a detailed water balance and evaluation of different water supply options. A water permit for use and potential discharge of any surplus water will be required, as will various other supporting permits, and would be applied for once the mining concession is granted.

4.2.3 Royalties

There are no royalties applied currently on the Property. Under the Polish regulations all operating mines must pay an exploitation fee to the local communities (60%) and the National Environmental Fund (40%), which is calculated based on the tonnage of ore removed from the deposit. The standard fees are published by the Polish government every year, for 2024 it is 4.61 PLN/t of ore. All copper and silver producers must also pay the Mineral Exploitation Tax (MET) as part of the general Polish taxation system (refer to Section 22.3.3).

4.2.4 Material Contracts

Lumina currently has a usufruct agreement for exploration of the Nowa Sól deposit. This mining usufruct agreement (*umowa o ustanowieniu użytkowanie górnicze*) is a civil-law contract between the State Treasury, represented by the MCE, and the concession holder (Lumina) that grants the exclusive right to use the defined mining space for the purpose of conducting exploration for a specified mineral. The agreement precisely delineates the usufruct area (the same as the exploration concession area), is aligned in duration with the term of the exploration concession, and sets out the remuneration payable to the State, an area-based annual fee, together with payment terms. It confirms the exclusivity of the holder's rights within the concession area, establishes the obligation to carry out activities in accordance with the concession and applicable law, and regulates access to and use of geological information.

4.2.5 Environmental Liabilities and Permitting

Micon is not aware of any environmental liabilities or permitting restrictions at this stage of the Project, however it should be noted that the Environmental Impact Assessment (EIA) has not yet been undertaken.

There are no Protected Areas directly within the proposed footprint of the Project. The exploration concession is situated in between the Sławskie Lake District to the east and the Odra River valley to the west, both of which are designated Protected Areas under the EU Natura 2000 network. Further information is provided in Section 20.3.3 and Figure 20.4.

In 2024, Lumina completed the required remediation and clean-up at drilling platforms which were disturbed by previous exploration activities within the Nowa Sól Concession. These areas are in various stages of revegetation, in collaboration with the Polish State Forestry Service (*Las Państwowe*). Currently, Lumina has no outstanding environmental liabilities.

Lumina has not yet begun the environmental permitting process for the Project. An overview of the legislative process is described in Section 20.1.3 of this Report.

Other than as discussed, the QPs are not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Project is located approximately 320 km west of the Warsaw capital, 100 km northwest from the city of Wrocław, 85 km east of the border with Germany and 120 km north of the border with Czechia (refer to Figure 4.1). There are good national and international transport links by road, rail and air.

The western boundary of the Nowa Sól Concession is located approximately 25 km east from the city of Zielona Góra, the primary transportation and service hub of the region. The Project area is accessible all year-round via a well-maintained network of public roads. Two paved roads (No. 315 and 318) traverse the deposit area, connecting the small villages of Konotop, Bojadła, and Lubięcín with the city of Nowa Sól and the S3 major expressway (Figure 5.1).

The S3 expressway that crosses the county to the west of the project, links the region to key seaports in Szczecin and Świnoujście in the north and to Czechia through Legnica in the south. The Nowa Sól river port is situated at Nowa Sól on the Odra River, approximately 6 km to the southwest, and provides links to the Baltic Sea via Szczecin and to the North Sea via Elbe-Odra canal and Elbe River. It can support barges of around 100 t capacity and is linked to the rail network but would require upgrading to handle heavier loads.

Rail links provide both passenger and freight connections to Zielona Góra and other regional hubs in Poland and Europe. A decommissioned railroad (Line No. 371 Wolsztyn-Żagań) crosses the northern portion of the Property (Figure 5.1).

The nearest international airport, Babimost (IEG), also known as Zielona Góra Airport, is situated 20 km from the Property and is primarily serviced by the Polish national carrier Polskie Linie Lotnicze LOT S.A. (LOT) (Figure 5.1) offering direct connections to the capital city of Warsaw. However, the airports in Wrocław (WRO), Berlin (BER) and Poznań (POZ) provide better connectivity (Figure 4.1).

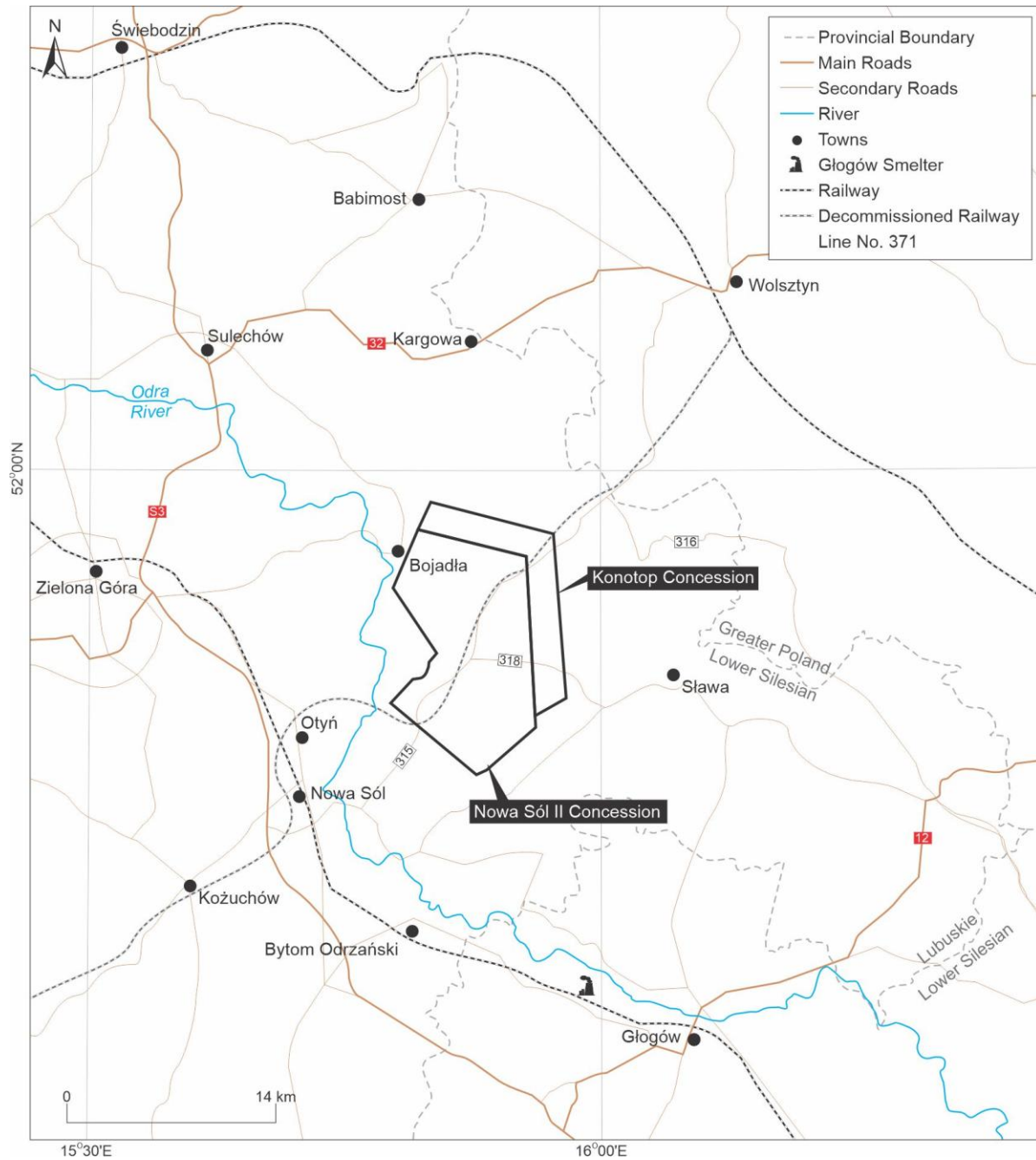
The nearest smelter is located in the city of Głogów, 20 km south of the Project concession area boundary and is owned by KGHM.

5.2 CLIMATE, PHYSIOGRAPHY AND DRAINAGE

The climate in the Lubuskie Province is classified as temperate oceanic, with distinct seasons and average temperatures ranging from around -3°C in winter (January) to 25°C in summer (July). Rainfall occurs year-round and peaks in summer months, with average annual totals of around 600 mm to 650 mm (Figure 5.2). Conditions are relatively mild compared to other areas of Poland, and Project operations can be conducted year-round.

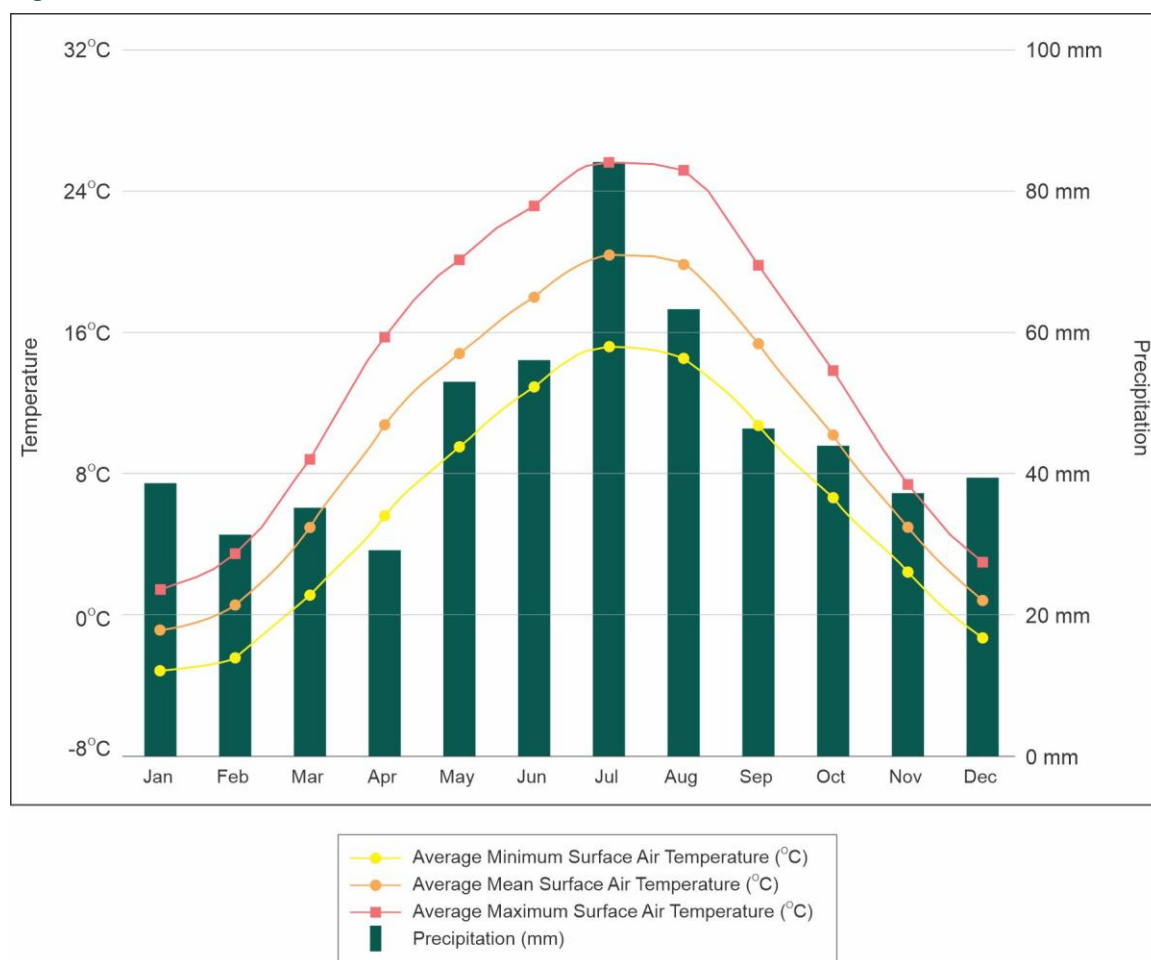
Figure 5.2 shows climate trends in the Lubuskie Province for 1991 to 2020. A dedicated weather monitoring station within the Nowa Sól Concession is recommended as the Project advances.

Figure 5.1: Regional Accessibility



Source: Micon, 2026.

Figure 5.2: Climate Trends in the Lubuskie Province



Source: World Bank Climate Change Knowledge Portal modified Micon, 2025.

The area surrounding the Project is characterized by rivers, lakes and forests. The Project is located in the middle reaches of the Odra River basin and the river is 1 km from the western concession boundary at its closest point. The topography is relatively flat, with gently undulating lowlands that form part of the larger Polish Plain and elevations ranging from 60 masl to 110 masl.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The Lubuskie Province hosts steadily improving infrastructure, supporting agriculture, light industry, and services. However, like many rural regions, it faces challenges such as limited high-tech industries and the need for further modernization of its infrastructure. The Project's strategic location along key transport routes enhances its economic potential. No major industrial centers are located on the Property.

5.3.1 Sources of Power

Towns and villages within the Nowa Sól Concession are integrated into Poland's national energy grid. The 400 kV and 220 kV network is operated by the state-owned Polish Transmission System operator (*Polskie Sieci Elektroenergetyczne S.A./ PSE*) and the 110/30 kV network is operated by Enea Operator, part of the Enea Group. Two major power lines run north-to-south near the Project: a 220 kV line to the west, plus a 110 kV line to the east. The nearest 220/110 kV substations are located in Żukowice (around 20 km from the deposit boundary) and Leśniów Wielki (some 40 km away). Subject to suitable improvements to the substations, these lines may have sufficient capacity for

the future mines. However, the planned 400 kV north/south Baczyna-Plewiska line is reported to include a substation in the vicinity (Grodziec Mały) and given the potential consumption for the mine, it would be preferable to feed the mine sites from such a 400 kV supply.

There are no gas pipelines located on the Property.

5.3.2 Water

It is intended that water supply for the Project will be obtained from dewatering of the underground mining areas and kept in circulation for re-use on-site. Further technical studies, including a better understanding of hydrogeological conditions, water balance and water quality are needed to confirm this and advance the Project.

5.3.3 Local Infrastructure and Mining Personnel

Poland has a well-developed mining industry and therefore supporting infrastructure, services and skilled personnel exist within the wider area. There is a network of schools and vocational training centers focusing on technical and agricultural skills, particularly in Nowa Sól. Additionally, major technical universities in Zielona Góra, Wrocław, and Poznań are within close proximity.

The Polish Copper Mining District (*Zagłębie Miedziowe*) is located to the southeast of the Project. This area hosts KGHM Polska Miedź S.A. (KGHM) a global copper mining and refining company that operates three mines, two smelters, and extensive supporting infrastructure. The closest operational mine is Rudna copper mine, approximately 40 km southeast of the Nowa Sól II Concession.

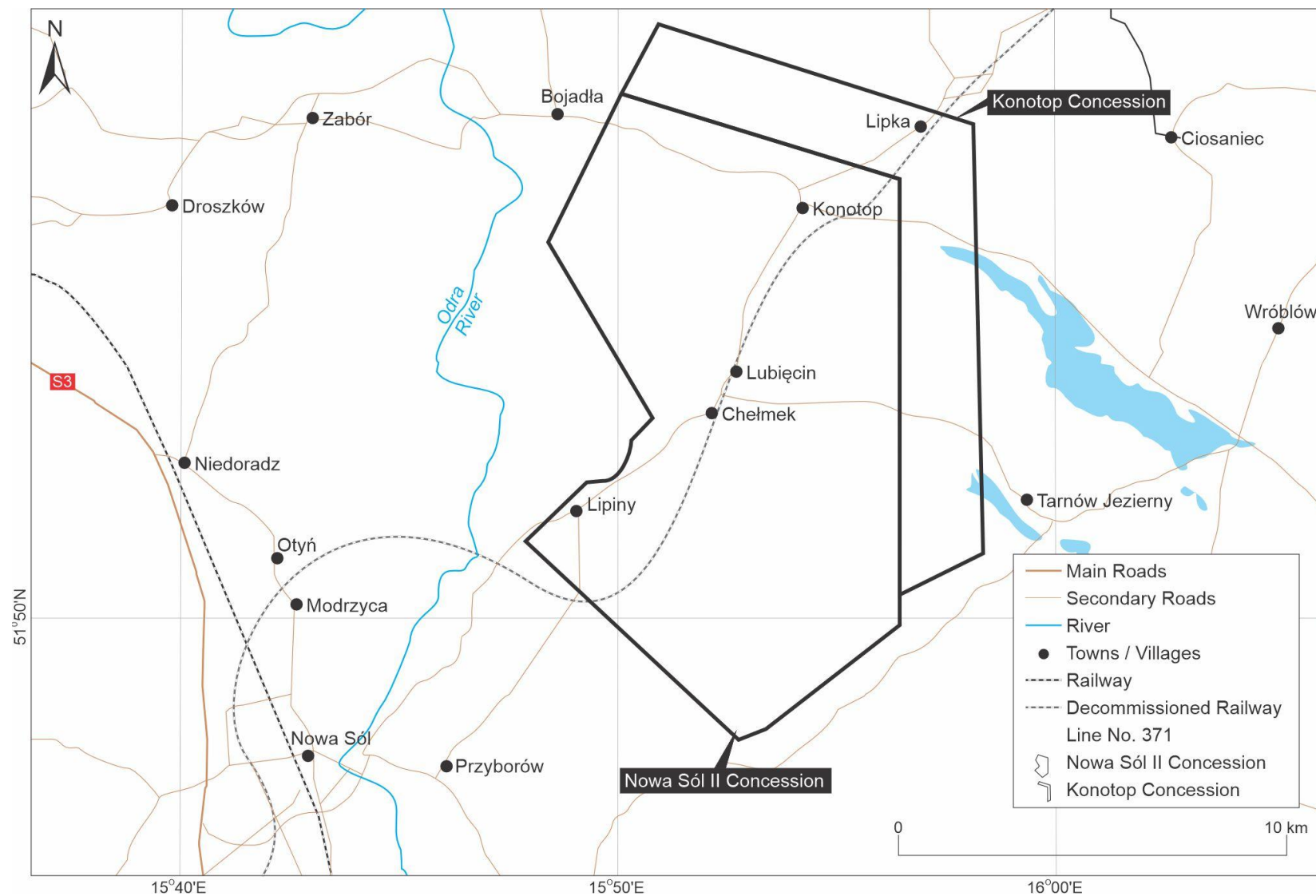
The Lubuskie Province is also part of the Kostrzyn-Słubice Special Economic Zone, offering preferential regulations and corporate income tax exemptions to investors. This zone hosts small and medium-sized enterprises in manufacturing, food processing, and logistics. The region also benefits from EU funding for regional development, which supports projects in infrastructure, education, and innovation.

5.3.4 Existing Infrastructure

The Project is at an advanced exploration stage, and there is no mining or processing infrastructure present within the concession (Figure 5.3).

Photographs of nearby infrastructure taken during the July 2025 site visit are shown in Figure 5.4.

Figure 5.3: Nowa Sól II and Konotop Concession Areas and Local Infrastructure of the Project



Source: Micon, 2026.

Figure 5.4: Photographs of the Infrastructure near the Project Area



Source: Micon, 2025.

6 HISTORY

6.1 INTRODUCTION, HISTORICAL WORK COMPLETED AND RESULTS

The Project area is located within the Kupferschiefer ore series, situated between the Lower Permian (Rotliegendes) terrestrial redbeds/volcanics and the Upper Permian (Zechstein) marine sequence within the Late Palaeozoic Era. The Kupferschiefer has a significant history of copper mining in Poland that has documented historical mining from 1,120 AD, but recent investigations suggest that prehistoric mining of rich oxidized copper ores possibly dates back to the Bronze Age.

Mineralization within the Nowa Sól deposit occurs at depths starting from approximately 1,700 m. Consequently, no surface rock or soil sampling, trenching, pit sampling, or prospecting has been undertaken. All exploration has been conducted through drilling.

6.2 HISTORICAL DRILLING

Lumina have access to an historical drill hole database comprising 411 historic drill holes (Section 9). The historical drill hole database is a regional data set and included exploration results from outside of the Nowa Sól licence area. The use of the historical information as an input to the MRE presented in this Report is discussed in Sections 10, 14 and 15.

Several of these historical drill holes are located within the current outline of the Nowa Sól deposit; however, the majority were shallow and did not intersect the target Kupferschiefer horizon. Only one historical drill hole, Bojadła 1, reached sufficient depth to intersect the mineralized horizon. This hole was drilled in 1986 by Zielonogórski Zakład Górnictwa Nafty i Gazu, Zielona Góra (now PGNiG), to a total depth of 2,205 m as part of an oil and gas exploration program (Figure 6.1).

Drilling at Bojadła 1 was primarily completed using tri-cone rotary drilling, with diamond core drilling employed across key stratigraphic intervals, including the Upper/Lower Permian contact where Kupferschiefer-style copper-silver mineralization is typically hosted. Historical drill logs for Bojadła 1 were retrieved from the digital archive of the Polish Geological Institute (PIG). These logs were limited in detail, comprising only basic “From-To” lithological contacts, lithostratigraphic units, and brief lithological descriptions.

Drill core from Bojadła 1, including the mineralized interval, is stored at Magazyn NAG, the national core repository located in Leszcze, Poland. Lumina personnel conducted a review of the core in October 2014. The core was found to be largely intact, except for the T1 horizon, where limited core material remained due to prior sampling and analysis by previous parties. Lumina relogged the available core and carried out portable X-ray fluorescence (XRF) analysis over all accessible intervals to obtain semi-quantitative data on copper, silver, lead, and zinc concentrations. Figure 6.2 presents the Bojadła 1 drill core with corresponding XRF sample locations and analytical results.

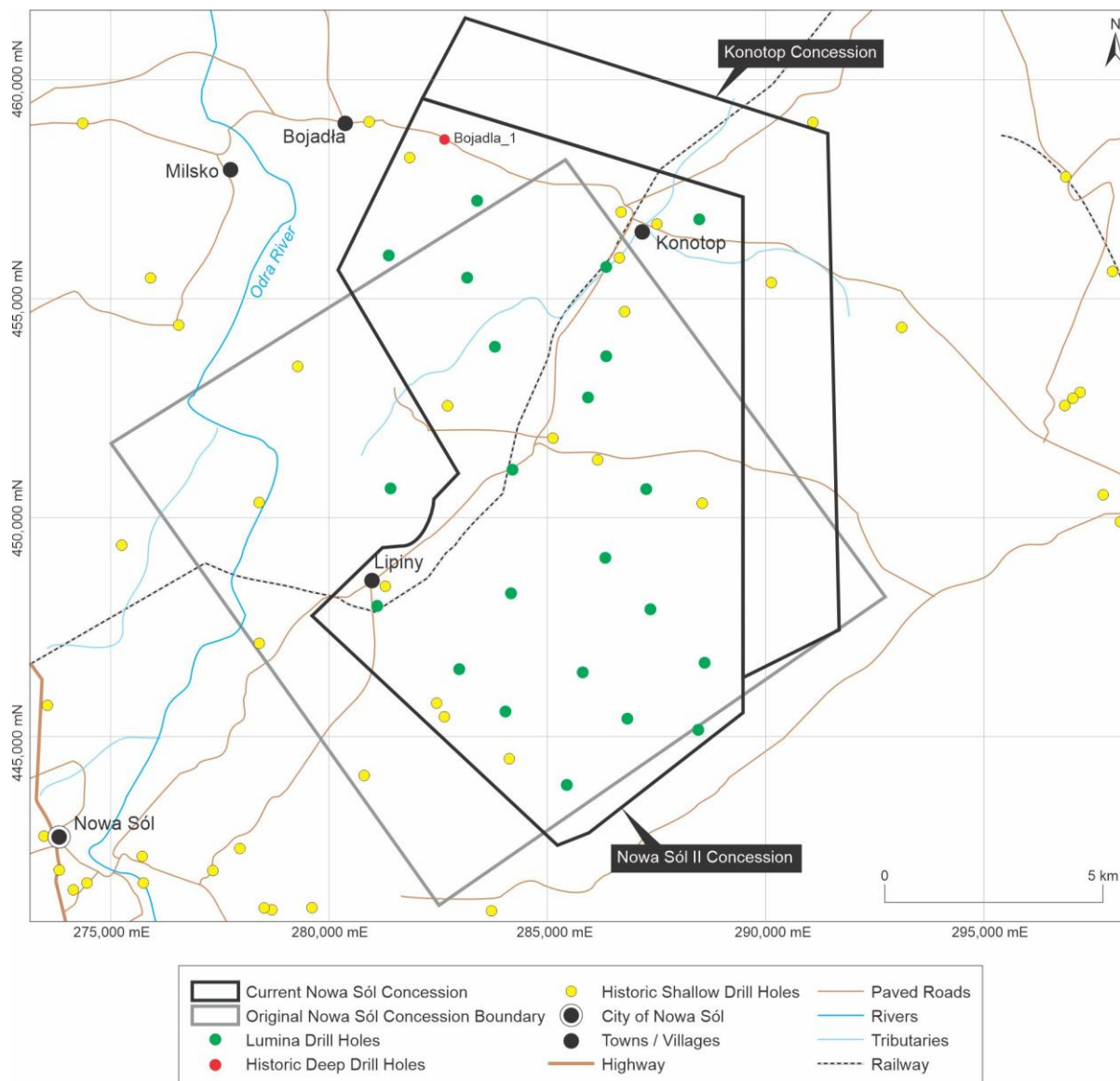
Following encouraging results from the core review and XRF analysis, Lumina acquired the original analytical data from PIG in March 2015, based on agreement with State Treasure. These results are presented in Table 6.1. However, no information is available regarding the sampling procedures, sample preparation, Quality Assurance/Quality Control (QA/QC), or analytical methods used in the historical analysis. Consequently, the reliability of the historical assay results cannot be confirmed by the QP.

6.3 HISTORICAL SAMPLING

The historical sampling strategy was designed to reflect observed mineralization styles and associated oxidation features. Sampled intervals encompassed the stratigraphic boundary between the Rotliegend and Zechstein formations, including the uppermost Rotliegend,

Weissliegend, basal limestone, copper-bearing shale, Zechstein limestone, and the base of the Lower Anhydrite.

Figure 6.1: Plan Map Showing Historical Shallow and Deep Drill Holes, and Lumina Drill Holes

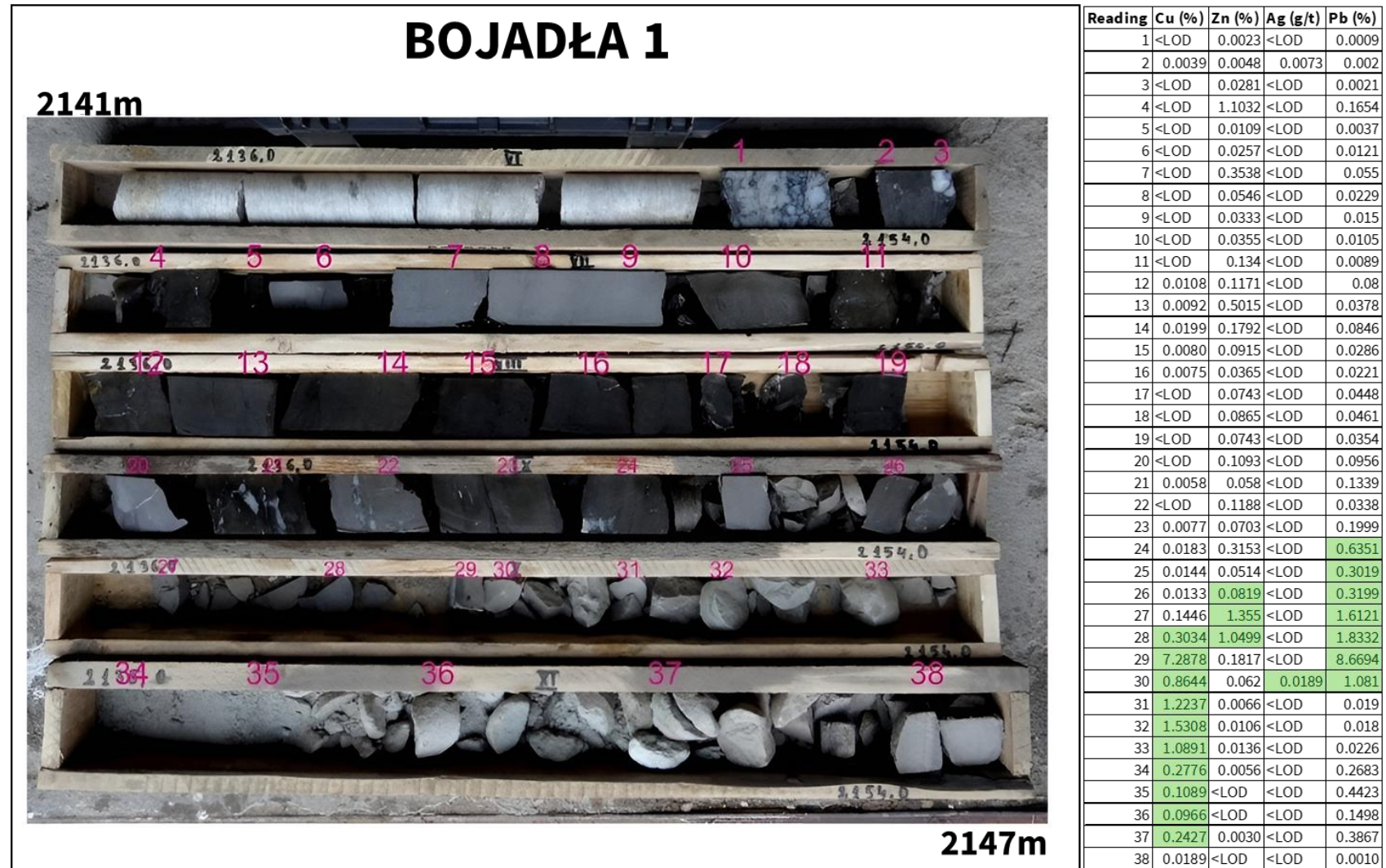


Source: Micon modified after Lumina, 2025.

Sampling methodology was adapted to the condition of the drill core and the intensity of mineralization. In intervals exhibiting moderate to strong mineralization, continuous samples were taken from one-quarter of the core, cut longitudinally along the core axis, with due consideration given to lithological boundaries. In intervals characterized by low-grade mineralization or where core recovery and preservation were poor, representative point samples were collected.

Individual sample lengths typically ranged from 5 cm to 20 cm, depending on lithology, anticipated mineralization grade, and mineralogical composition. In justified cases, such as thin lithological interbeds exhibiting distinct mineralogical or grade contrasts, shorter sample intervals of 1 cm to 5 cm were collected to preserve geological resolution. Where no visible mineralization was observed, continuous samples of 20 cm to 50 cm length were taken. Alternatively, composite samples consisting of multiple point samples collected at regular intervals across a defined section were prepared (MCE, n.d.).

Figure 6.2: Bojadla 1 Historical Drill Core and XRF Results



Source: Lumina, 2025.

Table 6.1: Historical Assay Results for the Bojadła 1 Drill Hole (Wilcze Area)

Sample No.	Lithology	Depth From (m)	Depth To (m)	Interval (m)	Geochemical Zone	Cu (ppm)	Zn (ppm)	Pb (ppm)	Ag (ppm)
1	Ca1	2,141.75	2,142.8	1	Reduced	200	1,100	900	2
2	Ca1	2,142.75	2,143.8	1	Reduced	100	700	1,000	2
3	Ca1	2,143.75	2,144.3	0.5	Reduced	300	1,700	6,800	3
4	Ca1	2,144.25	2,144.8	0.5	Reduced	1,300	6,900	6,800	6
5	T1	2,144.75	2,144.9	0.15	Reduced	3,000	21,500	21,600	29
6	T1	2,144.9	2,145	0.1	Reduced	2,400	18,200	20,100	41
7	Bs	2,145	2,145.2	0.2	Reduced	41,800	1,300	69,000	110
8	Bs	2,145.2	2,145.4	0.15	Reduced	10,000	100	600	60
9	Bs	2,145.35	2,145.8	0.4	Reduced	16,500	100	1,600	779
10	Bs	2,145.75	2,146.1	0.3	Reduced	2,100	0	5,500	11
11	Bs	2,146.05	2,146.3	0.2	Reduced	1,500	0	4,900	4
12	Bs	2,146.25	2,146.4	0.1	Oxidized	200	0	100	2
13	Bs	2,146.35	2,146.8	0.4	Oxidized	100	0	0	2

Source: PIG, 2015.

Note: The QP is not able to verify these results.

Sampling procedures were designed to ensure that each sample met the minimum analytical mass requirements for the planned chemical and petrographic analyses. A minimum analytical weight of 50 g was targeted for routine analyses. Samples designated for gold and platinum-group element (PGE) determinations were selected based on lithology and the presence of hematitization or other relevant alteration features, with a minimum analytical mass of 35 g.

All sample preparation including crushing, pulverizing, and splitting, and analytical testing, were conducted at the Central Chemical Laboratory of the Polish Geological Institute, National Research Institute (PIG-PIB) in Warsaw. The laboratory operates under accredited procedures (Accreditation Certificate AB 283) in accordance with PN-EN ISO/IEC 17025:2005 standards for chemical and physicochemical testing of geological and environmental materials (field/object identification codes C/3; C/9; N/9).

6.4 PREVIOUS MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

6.4.1 Lumina MRE (Non-Compliant), 2020 - 2024

Mineral resources estimates for the Nowa Sól Copper Deposit were prepared by Lumina in accordance with the Polish Resource Classification System, as prescribed by the “Regulation of the Minister of the Environment of the Geological Documentation of Mineral Deposits, July 01, 2015” issued by the MCE. These estimates were reviewed and approved by the Polish geological authorities and were prepared for statutory and regulatory purposes within Poland.

The mineral resource estimates required by the Polish authorities for statutory and regulatory purposes are not compliant with Canadian CIM standards and Definitions and will not be discussed further in this Technical Report.

All prior mineral resource estimates have been superseded by the current mineral resource estimates contained in Section 14 of this Technical Report with summarized

6.4.2 Historical Mineral Reserve Estimates

No historical Mineral Reserve Estimates have been declared on the Property.

6.5 HISTORICAL PRODUCTION

No historical production has taken place on the Property.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The deposits of the Zechstein Group were formed within the Zechstein Sea, a partly barred large basin with relict sea characteristics that covers a vast area of northern Europe (Figure 7.1). The barriers were totally breached on at least four occasions allowing mass influxes of oceanic water followed by regression and prolonged sequential desiccation. Initially the deposits at the base of each sequence were limestones, now dolomitized, marls and shales all supratidal to relatively deep-water origins. These grade vertically and laterally into sulphate and chloride beds. Evaporite sub-cycles show a gradual vertical change from the least soluble calcium salts at the base (calcite, gypsum and anhydrite) followed by halite and polyhalite and lastly potassium and magnesium salts at the top (sylvite, carnallite, kainite and kieserite).

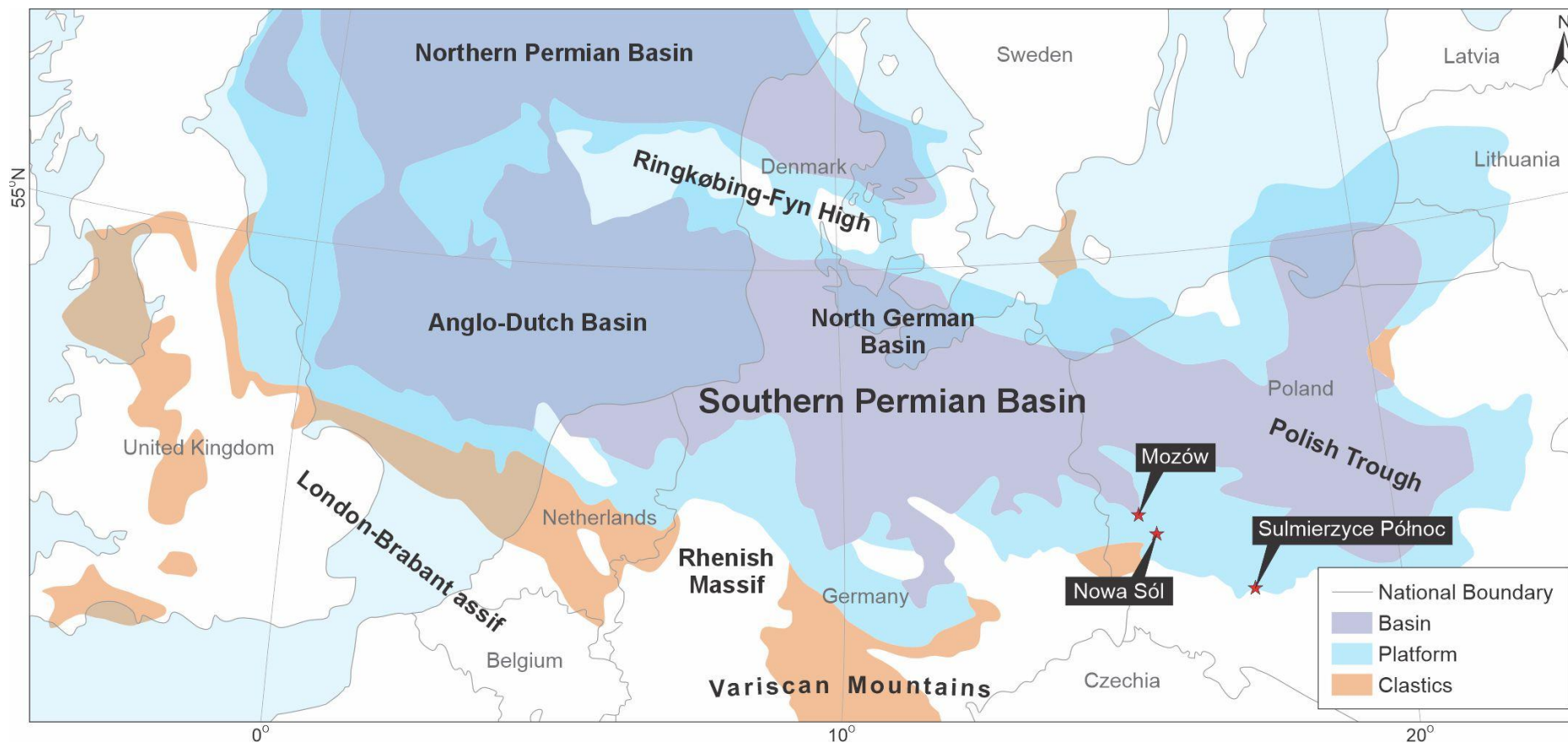
The Polish Zechstein basin is a part of the Southern Permian basin within the large Central European Basin that formed in the late Carboniferous (Kotarba et.al., 2006) (Figure 7.1). The basin stretches roughly 1,500 km from northeast England, across the North Sea, through northern Germany and Poland: an estimated area of 600,000 km² (Wedepohl, 1971).

The Polish sub-basin is bounded by three vast continents: the Scandinavian continent to the north, the vast peneplenized East European continent in the east and the morphologically diversified Caledonian and Variscan Massifs to the south (Wagner and Peryt, 1997).

The Central European Basin was created during the Late Carboniferous to Early Permian by multiple events of strong extensional tectonism and shear faulting resulting in bimodal magmatism from lithospheric thinning and upper mantle decompression (Eckardt, 1979). A subsequent long period of siliciclastic sedimentation occurred in an arid to semi-arid desert environment (Jowett, 1986) throughout the entire Lower Permian and is characterized by alluvial fan, braided river, meandering stream, aeolian, and saline lake siliciclastic deposits. Due to continued thermal subsidence and extensional tectonics during the Lower Permian, sedimentary rock sequences can reach thicknesses of greater than 2,000 m (Borg, et.al. 2012) within the Central European Basin.

Further infilling of the Central European Basin continued into the Upper Permian after a rapid marine transgression resulting in the formation of the Zechstein Sea. The Upper Permian Zechstein marine sedimentation occurred within a shallow vast epicontinental marine basin extending over Central Europe (Wagner, Peryt, 1997) and is characterized by a sequence of four major cyclothems which are dominated by alternating sequences of carbonate and evaporites. The lowermost cyclothem, named the Werra Cyclothem, hosts all known mineralization throughout the Basin and comprises of a Kupferschiefer shale, carbonate, anhydrite, and salt sequence.

Figure 7.1: Structural Geology of the Zechstein Basin with the Nowa Sól, Mozów, and Sulmierzyce Północ Deposit Locations



Source: Micon modified from García-Veigas, J et al, (2011).

7.2 POLISH ZECHSTEIN BASIN GEOLOGY

The regional geology of the Polish Zechstein Basin is described in detail in this section. Figure 7.2 presents a schematic geological plan map, while Figure 7.3 illustrates a cross-section along the A–B line indicated in Figure 7.2.

7.2.1 Stratigraphic Geology

The sedimentary group of rocks which host all mineralization within the Polish Basin consist of the uppermost Lower Permian Rotliegende sandstone, and the lowermost Zechstein marine sediment package consisting of a Boundary Dolomite (+/-), Kupferschiefer, and Zechstein Limestone: all of which total less than 10 m in thickness in the Nowa Sól area.

A stratigraphic column of the Permian in Germany and Poland and the typical zonation of the Kupferschiefer style mineralization is presented in Figure 7.4.

The geological stratigraphic sequence is listed below and described in this section:

1. Lower Permian

- i. Lower Rotliegende
- ii. Upper Rotliegende

2. Upper Permian

- i. Basal Limestone or Boundary Dolomite
- ii. Kupferschiefer
- iii. Zechstein Limestone
- iv. Lower Anhydrite (A1d), Oldest Halite (Na1), Upper Anhydrite (A1g)

7.2.1.1 Lower Permian

The Variscan orogeny of the Late Carboniferous to Early Permian created a series of isolated, elongate and fault-controlled basins that ultimately merged into northwest trending, partially interconnected basins by the deposition of up to 1,000 m of Rotliegende sedimentary rocks in a semi-arid to arid desert environment (Kucha and Piestrzyński, 1995). The end of Lower Permian deposition is marked by the top of the Weisslied (Bs/P1) sandstones. Subsequent ingress of the Zechstein Sea resulted in deposition of marine sedimentary rocks, the basal portion of which host the stratabound deposits.

Lower Rotliegende

The deposition of Rotliegende sedimentary rocks during the Lower Permian can be divided into the Lower Rotliegende and Upper Rotliegende. The Lower Rotliegende contains bimodal volcanic rocks, which include tholeiitic basalt, trachy-basalt, rhyodacite, and rhyolitic tuffs and lavas, that are intercalated with the clastic sedimentary rocks (Kucha and Piestrzyński, 1995). The Lower Rotliegende sedimentary rocks consist of a lateral and upward changing sequence of alluvial fan, braided river, meandering stream, and aeolian and saline lake deposits (Jowett, 1986). Sedimentary rocks mainly consist of red sandstones and conglomerates, they are texturally and mineralogically immature, and contain granite, schist, limestone, and volcanic debris that were derived from the Variscan Mountains (Vaughan, 1989). The Lower Rotliegende clastic rocks are generally <200 m in thickness and are coarser grained than the overlying Upper Rotliegende. Drilling depths in the Nowa

Sól area were never deep enough to intercept the Lower Rotliegende, although they are certain to exist at depth.

Upper Rotliegende

The Upper Rotliegende consists predominately of sandstones with some conglomerates. The sandstones are red in colour, fine- to medium-grained, moderately-well sorted, subangular, and contain 80% quartz and 20% feldspars with a matrix of carbonate, gypsum, and clay (Vaughan, 1989). The conglomerates are not as prevalent as in the Lower Rotliegende, yet they are mineralogically similar containing volcanic clasts derived from the underlying bimodal volcanic rocks. The Upper Rotliegende was deposited in similar environments to the Lower Rotliegende (Jowett, 1986), and the deposition of the Upper Rotliegende considerably levelled the basin floor and connected many of the small, isolated basins (Oszczepalski, 1986).

The uppermost portion of the Upper Rotliegende is comprised of bleached and chemically reduced sandstone referred to as the Weissliiegend and Grauliegend. The boundary between the underlying Rotliegende and the Weissliiegend/Grauliegend is gradational and often difficult to identify.

The Grauliegendes were originally regular red-bed Rotliegende sedimentary rocks that have undergone terrestrial oxidation and hematitic reddening, but were subsequently bleached due to the flooding and reaction with the overlying Zechstein Sea (Borg et.al., 2012).

The Weissliiegendes are considered the last pulse of Lower Permian siliciclastic sedimentation prior to the ingress of the Zechstein Sea. These rocks generally occur in thicknesses of several metres, but have been reported locally up to 65 m (Oszczepalski, 1989). The Weissliiegendes are comprised of aeolian, fluvial and marine sandstones which were deposited and remained white due to insufficient time for oxidation and hematitic reddening (Ehling, et al., 2008). Lithologically, the Weissliiegendes contain quartz, feldspar, and lithic fragments similar in proportion to the underlying Upper Rotliegende red sandstones, are texturally immature, and are cemented by clay, carbonate, or anhydrite. The Weissliiegendes lack the goethite and hematite that are present in the underlying red bed Rotliegende sedimentary rocks (Borg et.al., 2012). The uppermost section of the Weissliiegendes was partially reworked during the transgression of the Zechstein Sea. These reworked Weissliiegendes are more carbonaceous quartz arenites and conglomerates with carbonate and/or anhydrite cement (Borg et.al., 2012).

The upper surface of the Lower Permian Rotliegende/Weissliiegendes forms a series of highs or ridges which have been interpreted by Jowett, (1991) to represent sand dunes and as sandbars by Borg et.al., (2012): both of which are equally plausible. These paleo-topographic highs played a controlling role in the distribution of subsequent Zechstein sedimentation, primarily during Kupferschiefer deposition.

7.2.1.2 Upper Permian

At the end of the Lower Permian, the Central European Basin was estimated to be 200 m to 300 m below sea level. The subsequent flooding of the Central European Basin by the Zechstein Sea led to the creation of a vast epicontinental sea. Most of the Polish Zechstein sedimentary rocks were formed in a span of between 5 to 7 million years ago (Ma) (Wagner and Peryt, 1997) and are categorized into four cyclothems. Cyclothems PZ1 to PZ3 represent transgressive-regressive marine carbonate-evaporitic cycles, while cyclothem (PZ4) occurred during a terrigenous-evaporitic cycle.

The Werra cyclothem (PZ1) is comprised of a Basal Limestone (Ca0), Kupferschiefer (T1), Zechstein Limestone (Ca1), Lower Anhydrite (A1d), Oldest Halite (Na1), and Upper Anhydrite (A1g) units. PZ1

is the main unit which hosts the mineralization at the Nowa Sól deposit, along with the sandstones of the uppermost Upper Rotliegende (Figure 7.4).

Basal Limestone or Boundary Dolomite

The Basal Limestone (Ca0) is composed of bioclastic mudstone, organic bearing biomicrites, sandy micrites and oncolites and locally as packstone and grainstone, rich in foraminifera, brachiopods, and ostracod skeletons (Borg et al., 2012). The Basal Limestone ranges in thickness from 0 cm to 30 cm and is restricted to basin margins and pinches out towards the basin center (Oszczepalski, 1989). In the eastern portion of the Fore-Sudetic Monocline, the unit occurs as a continuous and solid layer, while in the western portion only occurs as lenses (Borg et. al., 2012). Ca0 has been observed in isolated areas within the Nowa Sól deposit, but never more than 30 cm thick.

Kupferschiefer

The Kupferschiefer (T1) or 'copper slate' is a stratigraphic term given to a dark grey to black bituminous calcareous or dolomitic shale which is the dominant host of the copper mineralization within the Central European Basin (Vaughan, et. al., 1989). In the Fore-Sudetic Monocline, the Kupferschiefer varies in thickness depending on paleogeographic locations: along basin highs the Kupferschiefer is absent, but overall, the Kupferschiefer ranges in thickness from 20 cm to 60 cm (Borg et. al., 2012). It is composed of alternating organic rich clays containing fine planar laminae of clays and planar- and wavy-laminated dolomitic-calcareous clays (Kotarba et. al., 2006). The organic matter is preferentially composed of degraded algae and holds an average carbon content of up to 7.34% (Borg et. al., 2012). It is therefore interpreted to have been deposited under a deeper anaerobic marine setting (Sun, et. al., 1995). Within the Kupferschiefer, pyrite occurs primarily as framboidal pyrite originating from bacterial sulphate reduction. However, this form of pyrite is only present in non-mineralized zones within the Kupferschiefer. In mineralized zones, early pyrite is partially or completely replaced by copper sulphides (Sun, et. al., 1995) (Figure 7.4).

Zechstein Limestone

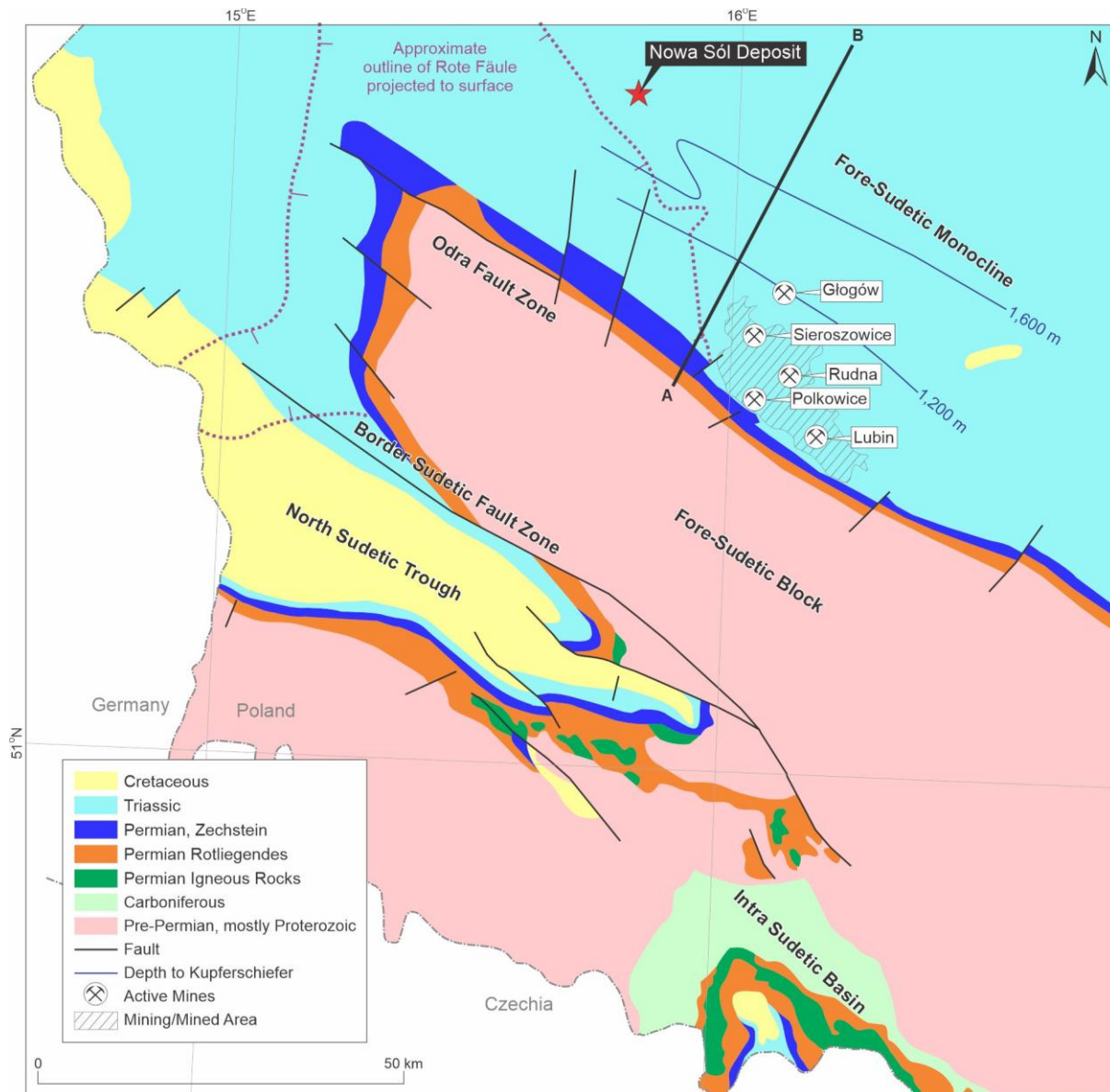
The Zechstein Limestone (Ca1) conformably overlies the Kupferschiefer and the transition between the two units is generally sharp. The Zechstein Limestone was precipitated from seawater of normal, average salinity (Wagner and Peryt, 1997) and is separated into three lithotypes: a clayey dolomite, dolomite containing clay-rich laminae, and a calcareous dolomite (Borg et. al., 2012), all of which exhibit mudstone to wackestone textures (Kotarba et. al., 2006). The thickness of the Zechstein Limestone varies greatly throughout the Basin, with thicknesses up to 110 m; however, the thickness encountered in the Nowa Sól area ranges from 1 m to 7 m. Towards the costal margins of structural highs (Wolsztyn High), the Zechstein Limestone is condensed into very thin sequences with bioclastic accumulations (mostly grainstone and packstone textures) which are regarded as reef bodies (Kotarba et. al., 2006).

Lower Anhydrite (A1d), Oldest Halite (Na1), Upper Anhydrite (A1g)

Overlying the Zechstein Limestone is a series of evaporites which have been subdivided into three units: the Lower Anhydrite (A1d) which ranges from 30 m to 70 m in thickness, the Oldest Halite (Na1) which ranges from 0 m to 250 m in thickness and the Upper Anhydrite (A1g) which can range from 20 m to 100 m, however in the Nowa Sól area the Na1 and A1d package together seldom exceeds a combined thickness of 300 m. The Lower and Upper Anhydrite can occur in many forms including massive, laminated, nodular, and aggregated. The Oldest Halite unit may be completely

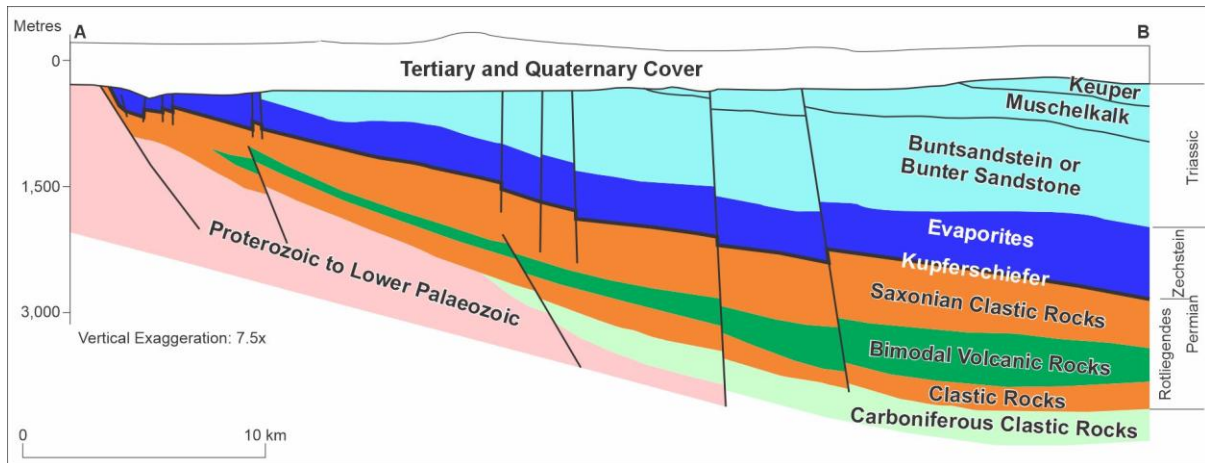
absent due to its proximity away from the basin margins; its thickness may also be amplified greatly due to the formation of salt diapirs. Kluska et al. (2013) concluded that the entire sequence of evaporites were deposited in a hypersaline basin under anaerobic conditions and the depositional environment and salinity were the determining factors on the formation of evaporites.

Figure 7.2: Schematic Geological Plan Map of the Polish Zechstein Basin



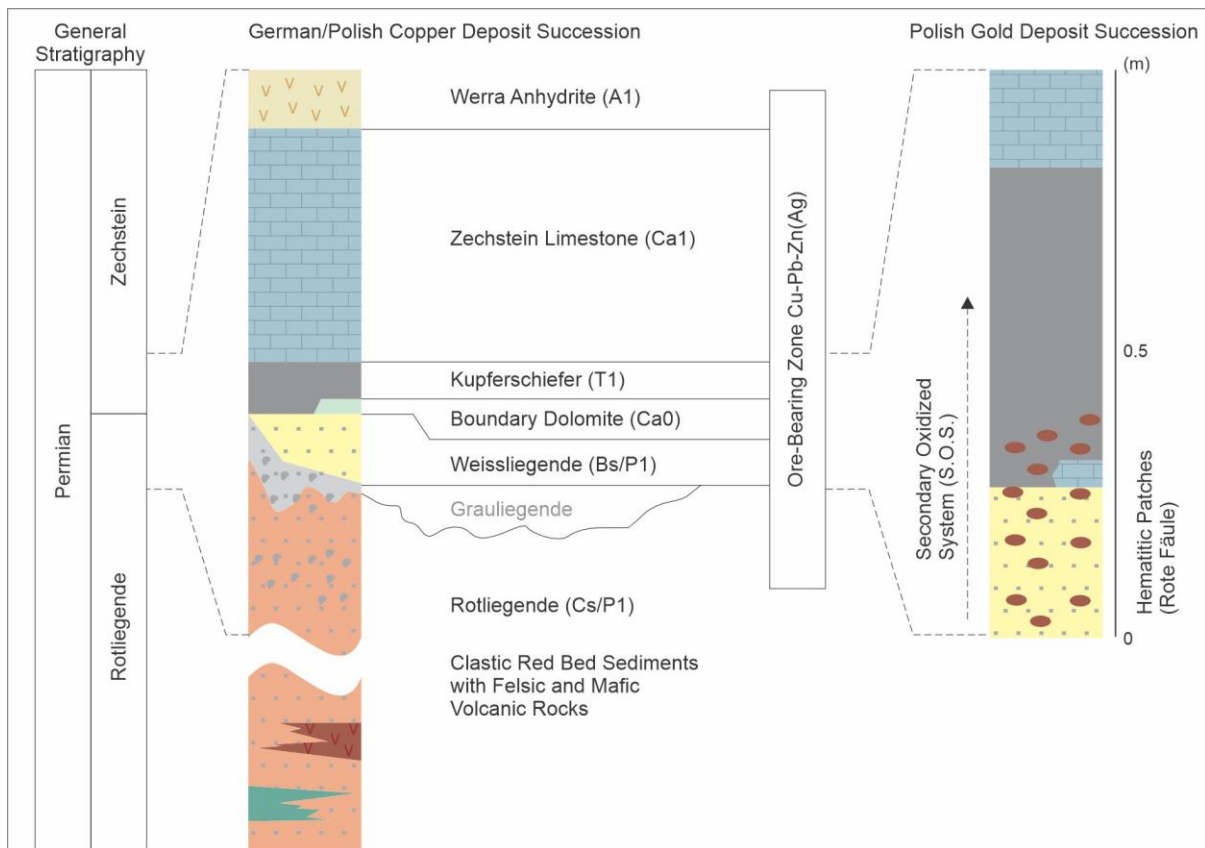
Source: Micon modified from Oszczepalski (1986), Borg et al. (2012) and Lumina (2025).

Figure 7.3: Schematic Geological Cross Section of Polish Zechstein Basin (A – B on Figure 7.2)



Source: Micon modified from Micon Technical Report (February 2013) and Lumina, 2025.

Figure 7.4: Stratigraphic Column of the Lower/Upper Permian in the Zechstein Basin



Source: Micon modified from Borg et al. (2012) and Lumina (2025).

7.3 PROSPECT AND LOCAL GEOLOGY

The Nowa Sól deposit is a part of the newly discovered Northern Copper Belt and is situated around 30 km northwest of the (Legnica-Głogów Mining District (LGOM), also called the New Copper District *Nowe Zagłębie Miedziowe*) in west Poland (Figure 7.5). The ore horizon spans across the upper part of the Lower Permian (Rotliegende) terrestrial redbeds and the lower part of the Upper Permian (Zechstein) marine rocks and comprises three lithotypes: sandstone, shale and carbonate (Speczik, et al, 2022).

The Nowa Sól deposit high-grade shale ore has polymetallic characteristics and is a crucial host for by-product metals such as silver, cobalt, and nickel, but also molybdenum, vanadium, and rhenium. The ore sulphides in the mineralized shale include chalcocite, djurleite, bornite, chalcopyrite, accompanied by digenite, covellite, tennantite, galena, sphalerite, and pyrite (Speczik, et al, 2022).

7.4 DEPOSITIONAL ENVIRONMENT

At the end of the Variscan Orogeny during Late Carboniferous times, much of the European landmass subsided and formed the Central European Permian Basin stretching from England in the northwest to Poland in the southeast. The Central European Basin consisted of multiple isolated-to partly connected-sub basins including the Norwegian-Danish Basin, English Basin, NW German Basin, Hessian Depression, Polish Basin, and Lithuanian Basin (Borg et.al., 2012). The Polish Zechstein Basin was flanked by three vast continents: the Scandinavian continent to the north, the vast penneplained East European continent in the east and the morphologically diversified Caledonian and Variscan Massifs in the south (Wagner and Peryt, 1997).

The deposition of Lower Permian terrestrial clastic sedimentary rocks and bimodal volcanics occurred in an arid to semi-arid desert environment within the vast intracratonic basin. Strong extensional tectonism contributed to continental flood basalts and rhyolitic volcanism during early Permian times, while the entire Lower Permian clastic basin was filled with sedimentary rocks largely derived from the Variscan Mountains and consisted of alluvial fan, braided river, meandering stream, aeolian, and saline lake deposits (Jowett, 1986; Wagner and Peryt, 1997).

Zechstein deposition commenced with the flooding of the Rotliegende desert basin by the Zechstein Sea as a result of subsidence caused by rifting combined with sea level rise (Wagner and Peryt, 1997). Prior to the ingress of the Zechstein Sea, the basin floor of the Upper Rotliegende was estimated to be roughly 200 to 300 m below sea level (Borg et.al. 2012). The transgression was very rapid and catastrophic and a period of only 10 years is estimated before the Zechstein Sea's greatest extension was reached (Glennie and Buller, 1983). The Zechstein transgression led to the reworking of the uppermost portion of the Upper Rotliegende siliciclastic sediments prior to marine Zechstein deposition. The climate then permitted the deposition of the shales, carbonates, and multiple successions of thick evaporitic units largely under anaerobic to dysaerobic environments within a shallow and vast epicontinental marine basin extending over Central Europe (Wagner and Peryt, 1997).

During the Kupferschiefer shale deposition, two palaeogeographical zones have been distinguished: deep shelf (basin facies) and shallow shelf (platform facies). In the deep shelf, shales are characterized by small, but consistent thicknesses (20 cm to 60 cm) composed of alternating organic rich shale containing fine planar laminae of clay and planar- and wavy-laminated dolomitic-calcareous shale. Deposition of the deep shelf was mostly within a low energy anaerobic environment distal to the basin margins more consistent with shales found in the Fore-Sudetic Monocline (Kotarba, et.al., 2006). The shallow shelf facies is located along the margins of the basin

and is characterized by 1 cm to 50 cm thick intervals of planar- to wavy-laminated, dolomitic-to calcareous-claystone.

As the sea level dropped due to evaporation and reduced water input from the Zechstein Sea, carbonates and eventually evaporites formed throughout the majority of the basin in platform to slope depositional environments, depending on their proximity to the basin margins.

7.5 MINERALIZATION

The Nowa Sól deposit and all major Polish copper deposits and existing mines are all located in the Fore-Sudetic Monocline, north of the uplifted Fore-Sudetic Block (Figure 7.2). While the North-Sudetic Trough also hosts mineralization, it occurs in lower grades, thicknesses, and volumes. The styles of mineralization are very similar in both regions, however copper grades within the Fore-Sudetic Monocline are notably higher. Mineralization in the LGOM (Fore-Sudetic Monocline) is entirely subsurface, dips northeasterly at an average of 2° to 3° towards the center of the basin and covers a measured area of 1,100 km² with depths ranging from 400 m to 1,600 m (Micon Technical Report (February 2013)), however, mineralization has been traced to depths greater than 2,100 m.

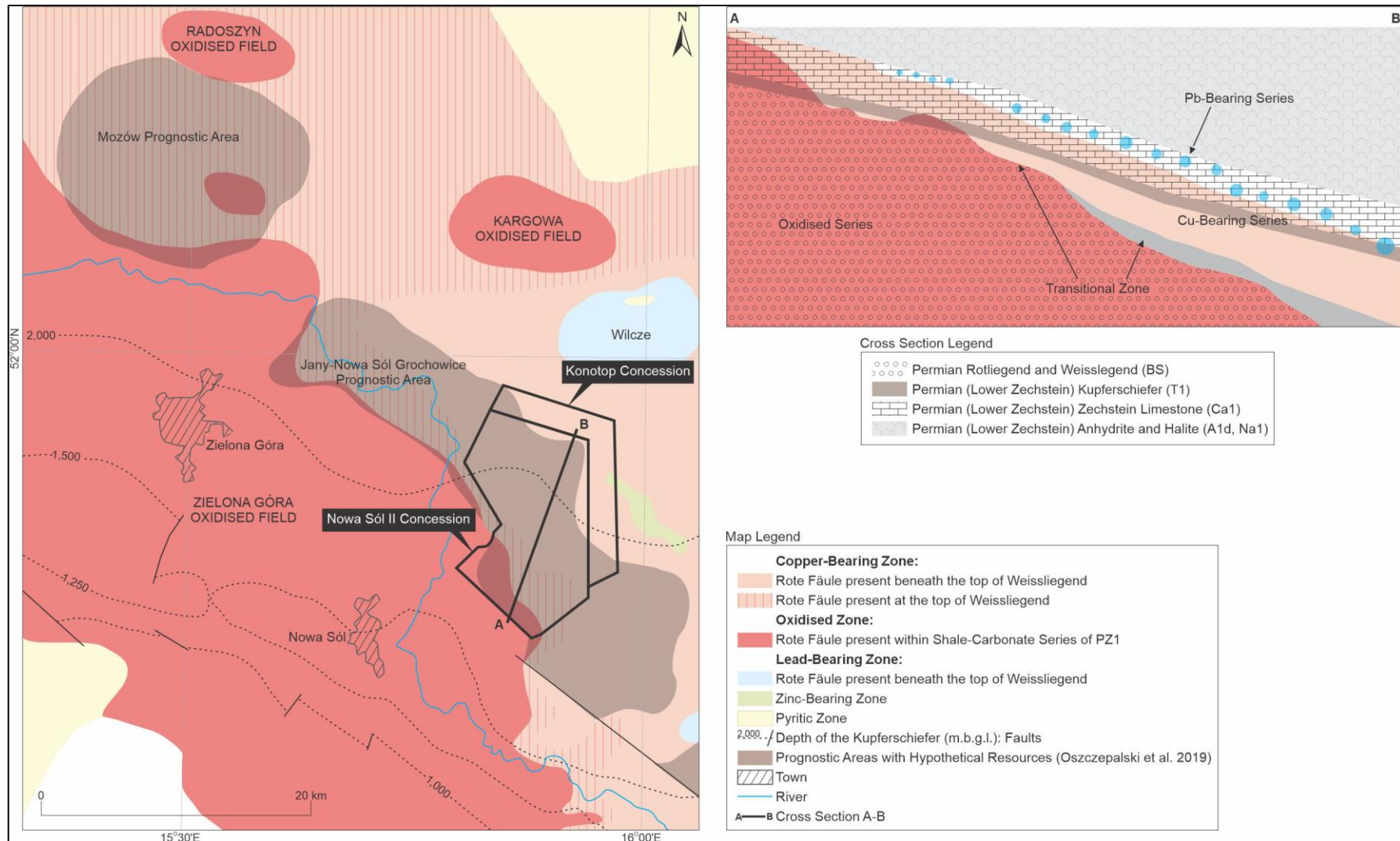
The Kupferschiefer copper-silver deposits have long been considered as stratiform-style with mineralization occurring within a confined sedimentary horizon bordering the Lower- and Upper-Permian contact. The copper mineralization is divided into three main ore types determined by their host lithology: sandstone-ore, hosted by the Weissliegende and Grauliegende sandstone and conglomerates of the Upper Rotliegende; Shale-ore, hosted by the Kupferschiefer black shales; and carbonate ore, hosted by the limestones and dolomites of the Zechstein Limestone.

Within all the ore types, the principal copper minerals are chalcocite in addition to bornite, chalcopyrite, and covellite (Micon Technical Report (February 2013)). Copper sulphides occur most commonly as replacements of lithic fragments or organic material, but are also observed as replacements of pyrite, hydrocarbons, and diagenetic calcite cement. The copper mineral forms occur as disseminated, nodules, bedding parallel lenses, semi massive or bleb-like (from hydrocarbon replacement). Late remobilization of sulphides also appears as short, narrow, discontinuous vertical veinlets up to 2 mm wide. Accessory minerals include tenorite, azurite, cuprite, native copper, enargite, galena, sphalerite, smithsonite, loellingite, arsenopyrite, cobaltite, nickeline, native silver, and stromeyerite (Micon Technical Report (February 2013)).

Examples of copper mineralization within the Bs/P1 sandstone, T1 Kupferschiefer, and Ca1 carbonate are shown in Figure 7.6 to Figure 7.9, respectively.

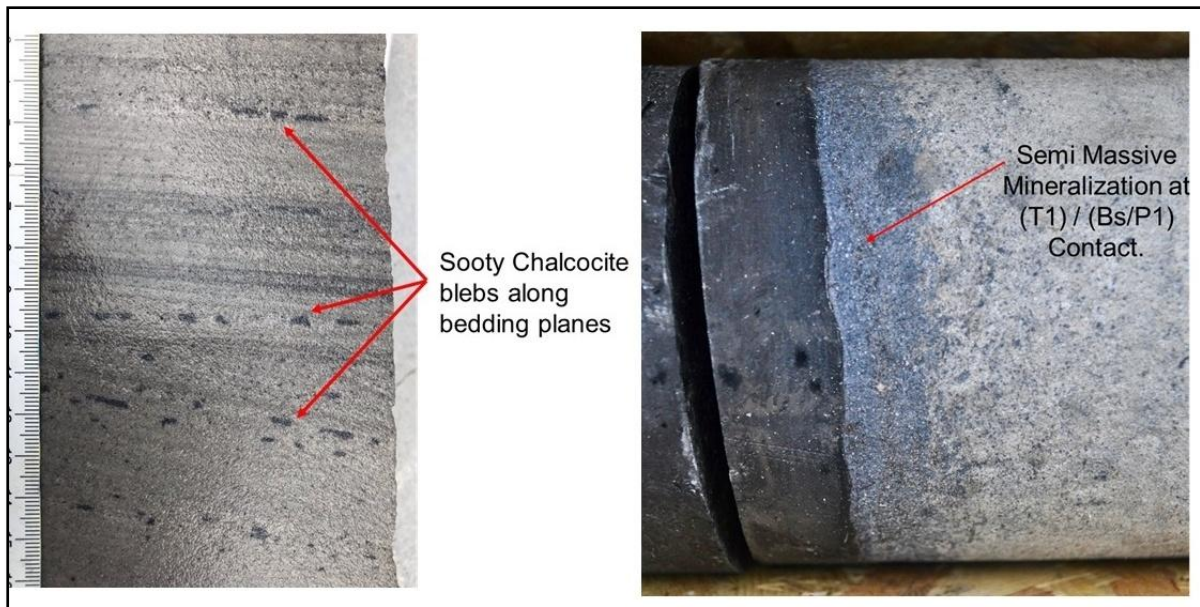
The mineralization is systematically zoned into mineral horizons that are developed both vertically and horizontally within the sedimentary horizon. Starting with the lowermost/outermost horizon, the mineralizing system consistently includes a strongly oxidized redox front (termed Rote Fäule) with a characteristic red hematitic alteration zone (Borg et al., 2012). The mineralization horizon grades through a copper zone, followed by a lead zone and zinc zone and finally into a pre-ore, pyrite zone. The highest copper grades typically occur along the reducing margins of the redox front, while minor gold and platinum group elements occur on the oxidized margin (Borg et. al., 2012).

Figure 7.5: Location of the Nowa Sól Deposit and Adjacent Prospective Areas



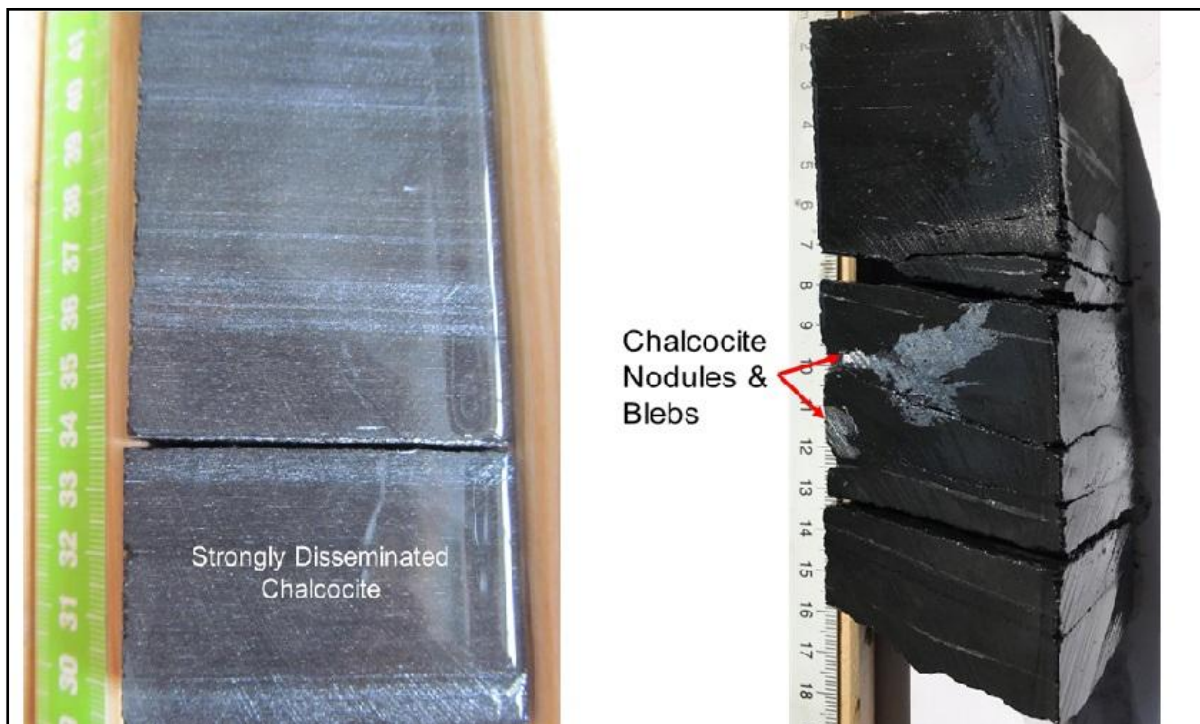
Source: Micon modified after Bienko, T, and Pietrzela, A., (2022).

Figure 7.6: Sandstone (Bs/P1) Mineralization



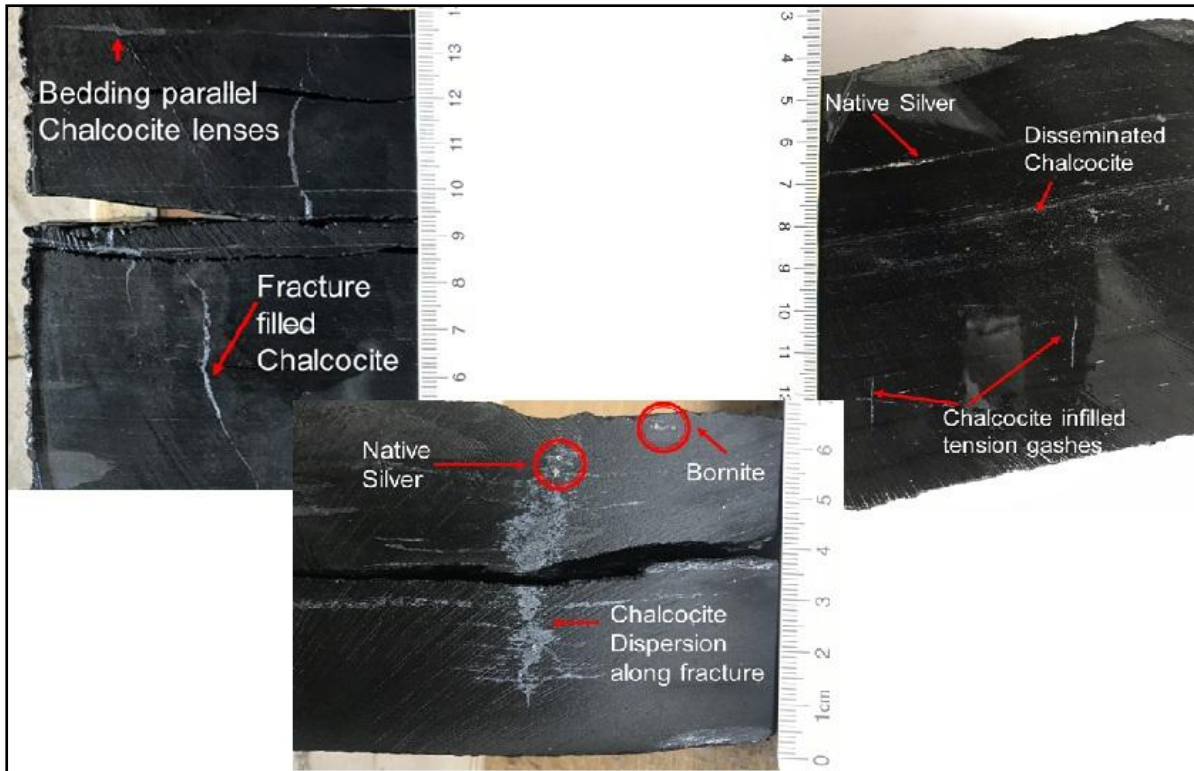
Source: Lumina, 2025.

Figure 7.7: Kupferschiefer (T1) Mineralization



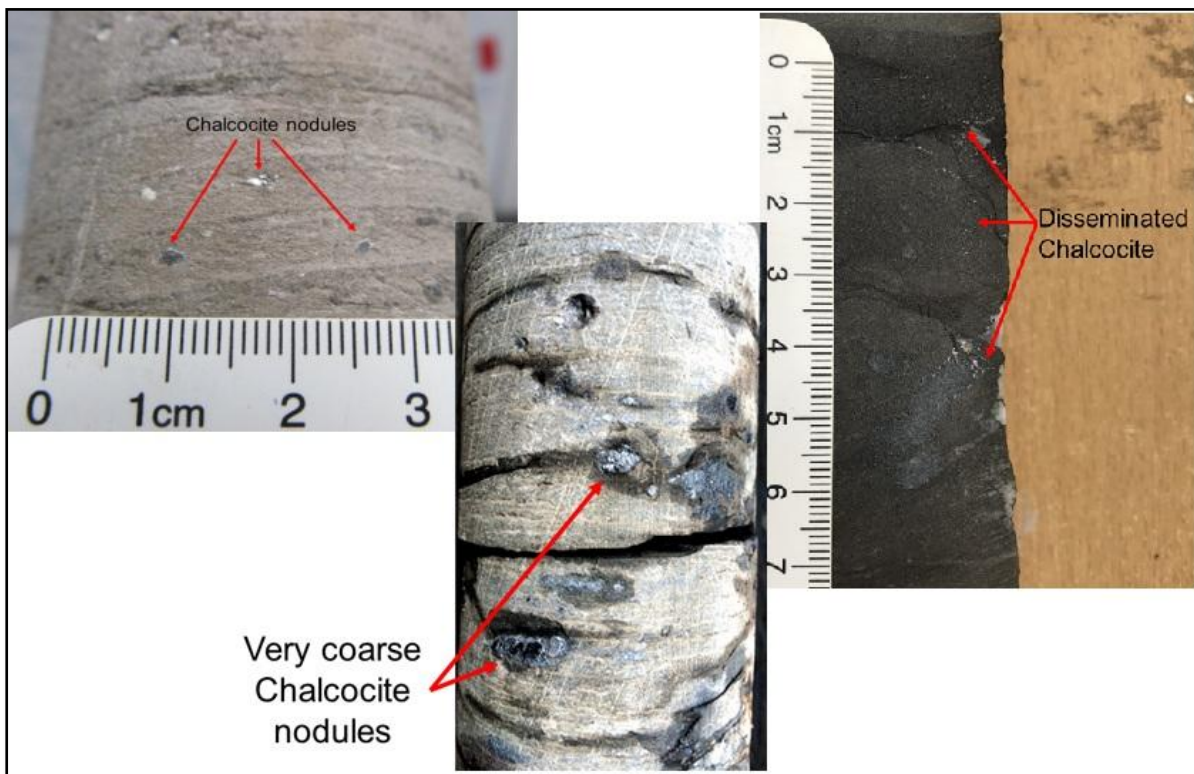
Source: Lumina, 2025.

Figure 7.8: Kupferschiefer (T1) Mineralization (continued)



Source: Lumina, 2025.

Figure 7.9: Carbonate (Ca1) Mineralization



Source: Lumina, 2025.

7.6 PARAGENETIC SEQUENCE

The following paragenetic sequence of the Kupferschiefer copper-silver mineralization in Poland is summarized by Borg et al., (2012).

During the Late Triassic, a major central European transpressional tectonic event reactivated major NW-SE and NNE-SSW-trending fault systems and created regional crustal tilting (Figure 7.10). These faults were deeply seated and dissected both basement rocks of the Mid-European Crystalline High, the Lower Rotliegende basalts, basaltic andesites, and rhyolites, and the overlying basinal volcano-sedimentary succession of the Upper Rotliegende. These source rocks had long acted as a reservoir for moderately heated, metal charged fluids. Upon reactivation of these fault structures, the fluids became mobilized and transported over large vertical and lateral distances, spreading slowly laterally in the Rotliegende clastic rocks to a suitable, structurally controlled trap site.

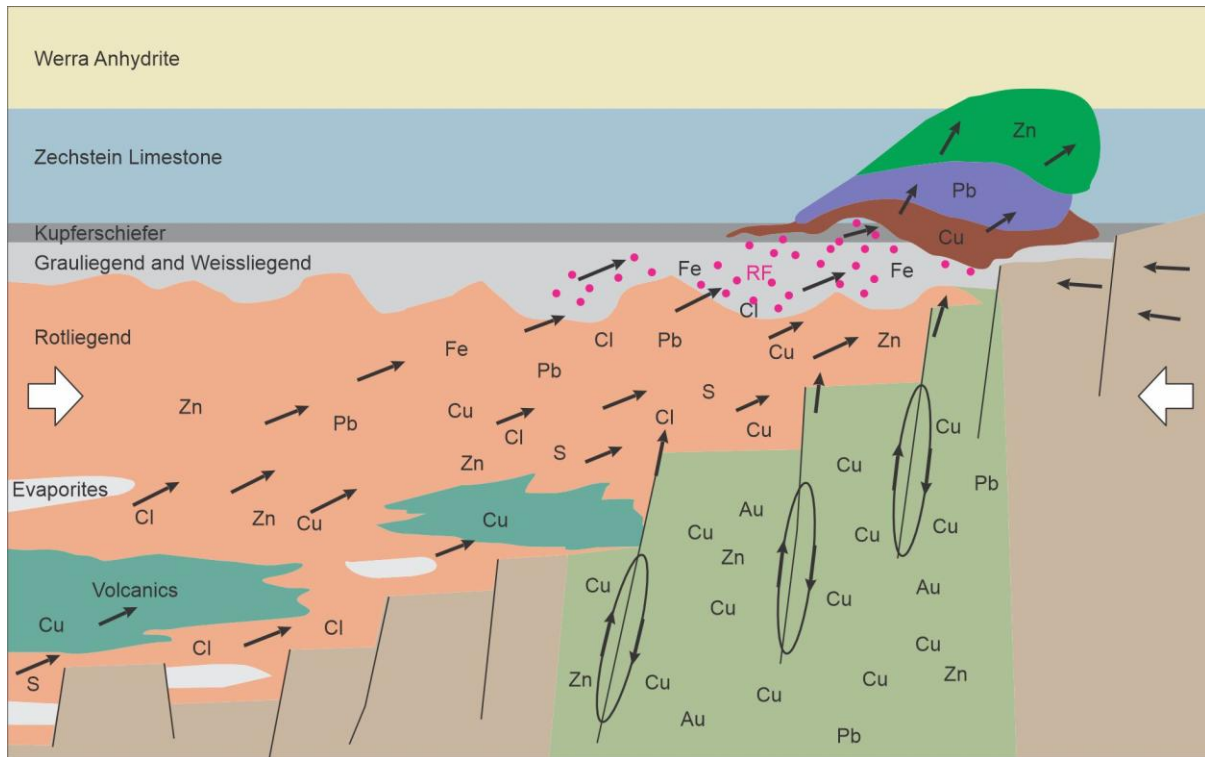
The Kupferschiefer black shale acted as a severe hydrodynamic and geochemical barrier to the migrating fluids, slowing down or stopping the ascending fault-propagated fluid flow and caused “ponding” of the metalliferous fluids below the black shale aquiclude. The mineralizing fluids thus have had a long residence time within the uppermost, partly sulphate-cemented Weissliegende and Grauliegende clastic sedimentary rocks and with the more impermeable, but also more reactive, organic rich pyritic black shale. Gradual fluid release through structurally controlled permeability pathways through the black shale, such as slowly de-cemented sand dunes and fault and shear zones, would have released the metal-bearing fluids into the overlying, reactive carbonate (Figure 7.10).

A late metallogenic pulse occurred during the Palaeogene, approximately 53 Ma, when subcontinental-scale tectonic structures underwent another phase of reactivation and hot-spot magmatic activity provided additional sources of metals and local heat (Figure 7.11). This stage is responsible for localized high-grade massive vein- and breccia-type ores that have overprinted the earlier types of ore.

7.7 AGE DATING

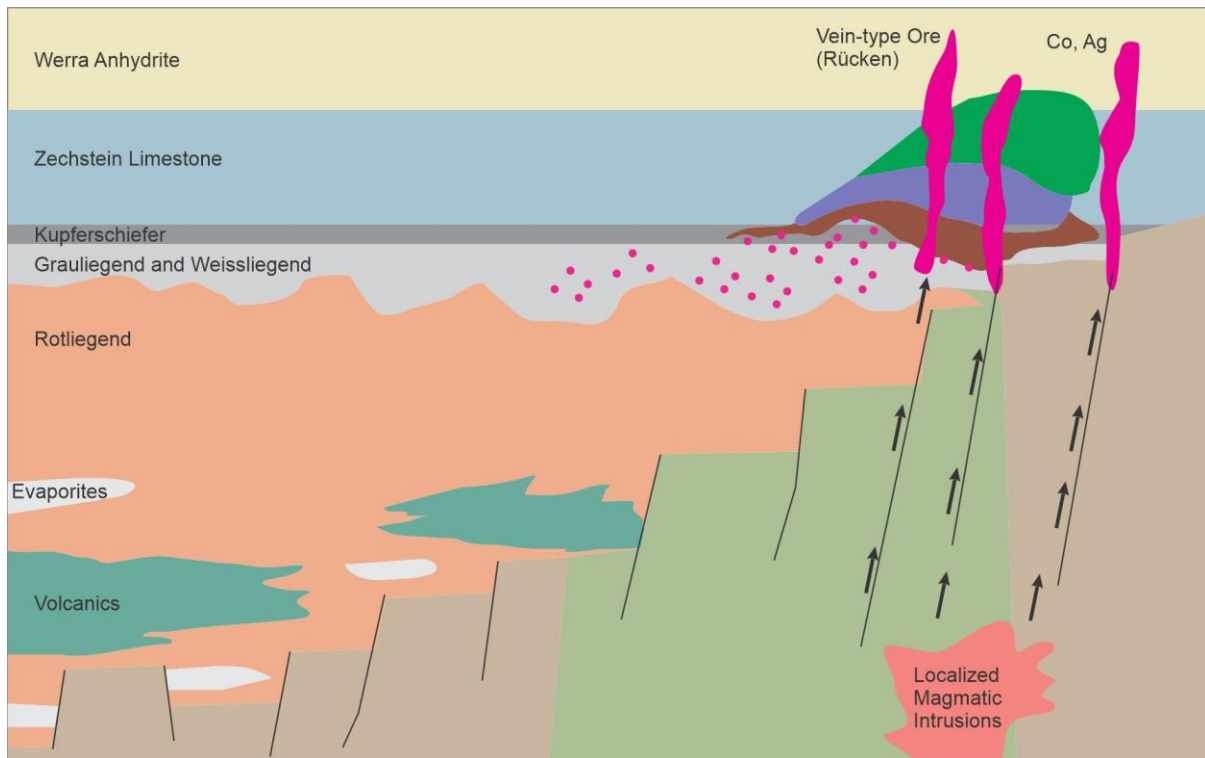
Age dating of host lithologies by various authors have consistently dated the Kupferschiefer strata within a small time frame ranging from 260.4 Ma $\pm$ 0.4 Ma (Słowakiewicz, et al., 2009) to 247 Ma $\pm$ 20 Ma (Pasava, et al., 2010). Until recently, paleomagnetic age dating by Jowett et al., (1987) placed the mineralization at Mid-Triassic and was associated with the continental rifting associated with the opening of the Tethys Ocean. However, Symons et al., (2010) conducted an extensive paleomagnetic and rock magnetic survey and concluded a Late Jurassic age of 149 Ma $\pm$ 3 Ma and a secondary date of 53 Ma $\pm$ 3 Ma revealing a significantly younger age of ore formation.

Figure 7.10: Metallogenic Model of the Kupferschiefer Mineralization – Late Triassic Phase



Source: Micon modified Borg, et al., (2012).

Figure 7.11: Metallogenic Model of the Kupferschiefer Mineralization – Palaeogene Phase



Source: Micon modified Borg, et al., (2012).

8 DEPOSIT TYPES

8.1 SEDIMENT-HOSTED STRATIFORM COPPER DEPOSITS

The Nowa Sól deposit is considered a Kupferschiefer-type deposit: a unique subset of Sediment-Hosted Stratiform Copper Deposits. The Kupferschiefer-type mineralization in Poland is located at the southwestern margin of the Polish Zechstein Basin, which represents a small portion of the expansive Central European Basin.

The siliciclastic and marine sediments which host the Kupferschiefer-type deposits were deposited in the Central European Basin, which is recognized as an intracratonic basin created after the Late Carboniferous Variscan Orogeny by a combination of crustal extension, magmatic underplating, crustal heating, and basin subsidence due to weight of sediment infill (Bachmann and Hoffmann, 1997). In general, the Kupferschiefer mining districts in Poland and Germany occur exclusively within an arcuate belt that is situated above the Mid-European Crystalline High comprised of basement rocks of a magmatic arc origin (Borg et al., 2012).

Genetic models of Sediment-Hosted Stratiform Copper Deposits, such as represented by the classical Kupferschiefer deposits in Poland and/or the Earth's largest sediment-hosted copper ores of the Central African Copper Belt, have evolved from a single-stage, disseminated copper-sulphide mineralization to a multi-stage ore mineralization (Chiaradia et al., 2025).

Recent copper isotope studies on the compositions of sulphides have confirmed that the range in isotopic composition is clearly related to changes in redox conditions at the depositional site, with important transport of the copper as chloride and/or sulphide complexes for the classic Kupferschiefer deposits and in other stratiform sedimentary copper deposits (Asael et al., 2009, 2012).

The first mineralization stage is followed by re-mineralization during early and late diagenesis and subsequently by further hydrothermal redistribution of sulphides under high-temperature tectonic settings (e.g., Dewaele et al., 2006; El Desouky et al., 2009, 2010; Hitzman et al., 2010). Evidence for a multistage origin of the mineralization and accompanying changes in the fluids also comes from detailed petrographic work, fluid inclusion analyzes, as well as C, O, and Sr isotope compositions of different carbonate phases within these deposits (e.g., El Desouky et al., 2010; Muchez et al., 2010; Shanks III, 2014).

The fluid flow in such mineralizing systems is characterized both by unidirectional gravity-driven fluid flow of basinal brines from the near surface to deeper levels, as well as by the establishment of convective cells at deeper levels of the sedimentary basin before the fluid ascends to variably shallower levels where the high-grade copper deposits are formed.

Figure 8.1 illustrates a generalized depositional model with superimposed zonation of mineralization.

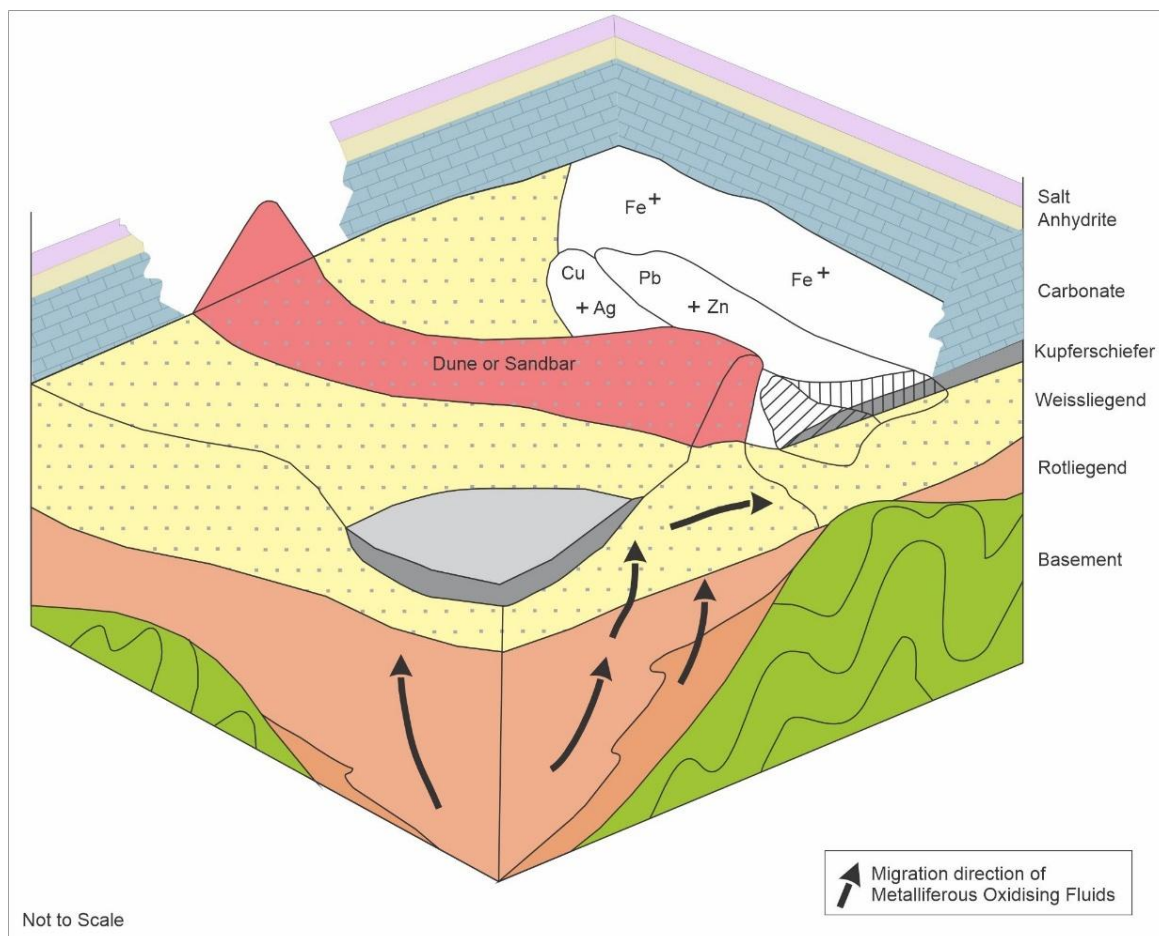
Other comparable Kupferschiefer-type deposits in the region include the historical Mansfeld District and the Spremberg/Weisswasser area in Germany, and most notably KGHM's active Lubin, Rudna, and Polkowice-Sieroszowice underground mines located in southwestern Poland (Bachmann et al., 2012; KGHM, n.d.-b). These Polish mines are situated within the Fore-Sudetic Monocline, immediately southeast of Lumina's Nowa Sól deposit, and form part of the same regional Kupferschiefer metallogenic system extending from Germany into Poland (Bachmann et al., 2012; Oszczepalski et al., 2015).

All of these mines and deposits share the following similarities:

- Kupferschiefer-style mineralization resulted from large-scale migration of oxidizing, metalliferous chloride brines, and display a distinct metal zonation of proximal copper/silver, outward to lead, zinc and finally distal iron (Oszczepalski et al., 2015; Oszczepalski, 2000).
- The metal zonation is observed away from a redox front (termed Rote Fäule) with highest grades of copper and silver mineralization typically found at the unoxidized edge of the Rote Fäule; gold and platinum group elements (PGE) enrichment can also occur immediately within the Rote Fäule margin (Schmidt et al., 2001; Oszczepalski, 2000).
- Mineralization always straddles the Lower/Upper Permian contact with mineralization occurring within footwall Lower Permian white sandstones (Bs/P1, Weissliegende-Rotliegende) and Upper Permian Kupferschiefer shale (T1) and overlying carbonates (Ca1). Mineralization is capped by the overlying impermeable Lower Anhydrite (A1d) (Bachmann et al., 2012; Oszczepalski et al., 2015).

The information on the other deposits and mines is sourced from publicly available information, as referenced. The QPs and Lumina have been unable to verify this information, which is not necessarily indicative of the mineralization on the Property and should not be disclosed out of context.

Figure 8.1: Generalized Depositional Environment and Mineralization of Kupferschiefer-Style Deposits



Source: Micon modified after Rentzsch, (1974); Schmidt, (1987); and Borg et al., (2012).

9 EXPLORATION

9.1 INTRODUCTION

Since its inception in Poland, Lumina has focused on developing large-scale copper and silver mining projects. The company embarked on a country-wide exploration program targeting mainly the Fore-Sudetic Monocline - a region with potential to host stratabound copper and silver deposits.

The program included developing a large-scale model of Zechstein mineralization, determining the expansion of mineralized and oxidized zones and using this data as a basis to narrow down to the most prospective areas for discovering new copper and silver deposits.

The initial phase of the program included:

- Analysis of 411 historic drill holes using publicly available geological data from oil and gas exploration conducted by the Polish state and various other private companies.
- Seismic reprocessing of approximately 1,693 line-km and review of historical data.
- Completion of 27 line-km of magneto-telluric surveying.
- Reprocessing of gravity data, encompassing 23,754 stations.
- Acquisition of a 3D geophysical survey.
- Development of a comprehensive GIS-based database.
- Drilling of 19 initial drill holes across nine concession areas (refer to Section 10).

The exploration program enabled Lumina to prioritize eight high-potential target areas for follow-up drilling. The drilling campaigns confirmed the presence of mineralization described in Section 10. This work, together with the drilling, resulted in delineating the Northern Copper Belt and the identification of three greenfield, sediment-hosted copper-silver deposits: Nowa Sól, Mozów, and Sulmierzyce Północ.

Copper-silver mineralization within the Nowa Sól deposit occurs at depths ranging from approximately 1,700 m in the southern area of the concession to approximately 2,200 m in the north. As the target stratigraphy was known to exist at depth, no surface exploration has been conducted by Lumina on the Property.

9.2 DESKTOP STUDIES

Lumina spent two years conducting an extensive desktop study and reviewing historical drill core (see Section 10 for details of the work conducted on the historical drill core).

Desktop studies included the acquisition of information from the PIG and from scientific publications. The data acquired included historical drill hole information, zones of mineralization from surrounding areas, and geological, geophysical, and topographic maps. All data was then digitized and imported into MapInfo GIS software for further review.

9.3 GEOPHYSICS

Lumina purchased gravity and seismic data from PIG to help aid in determining target areas best suitable for thicker Kupferschiefer-style mineralization. Seismic data helped determine depths to target horizons along with any potential, fluid-controlling deep structures. Rosenbach and Bouguer gravity data was processed into depth slices to help identify sedimentary environments which would be more suitable for thicker Kupferschiefer shale and carbonate accumulation than might be expected.

9.3.1 Seismics

In March 2015, a review and analysis of all seismic data within and surrounding the Nowa Sól area was conducted in a collaborative effort with San Leon Resources. The initial goal was to create maps illustrating the Uppermost of the Rotliegende, (essentially the Kupferschiefer T1 contact), Top of Zechstein, identify basement structures to help distinguish possible deeper seeded fluid conduits, and to identify where thicker accumulations of salt (Na1) and anhydrite (A1d) were expected. The exploration model implied that thicker Na1 accumulations were thought to mark basinal areas likely to host thicker accumulations of the carbonaceous Kupferschiefer T1 shale. The opposite was also true; thicker anhydrite delineated basement “highs” and poor areas for deposition of the Kupferschiefer.

Over 4,200 line-km of 2D and 3D seismic data, along with approximately 90 historical drill holes, were integrated and interpreted using KINGDOM ADVANCED software. Key stratigraphic horizons were identified, including the base of the Palaeogene, an intra-Triassic marker (Tpp2 – Middle Bunter Sandstone), Top Zechstein, Basal Anhydrite (A2, overlying the Main Dolomite), and the Uppermost Rotliegende. Figure 9.1 presents the isopach map of the Uppermost Rotliegende for the Nowa Sól area and its surroundings, illustrating a gently north-easterly sloping basin geometry.

The seismic data was then levelled to the Middle Triassic Tpp2 horizon, which removed the effects of monocline tilting and corrected for tectonic deformation predating Tpp2 in the Fore-Sudetic Monocline. This levelling enabled the construction of a palaeo-topographic model representing the surface of the Uppermost Lower Permian Rotliegende at the time of the initial transgression of the Permian Sea.

The palaeo-topographic model was subsequently used to predict palaeo-depositional environments and to identify areas with the highest potential for thicker accumulations of Kupferschiefer shale (T1) and carbonate (Ca1). This model served as a key exploration tool and is shown in Figure 9.2.

9.4 LOGGING AND SAMPLING OF HISTORICAL DRILL CORE

Of the available 411 historical drill holes, a total of 308 covering Kupferschiefer-style mineralization were relogged and photographed by Lumina, and analysed using an INNOV-X handheld XRF device. Where permitted by the relevant authorities, selected intervals were reassayed, and all results were incorporated into Lumina’s database.

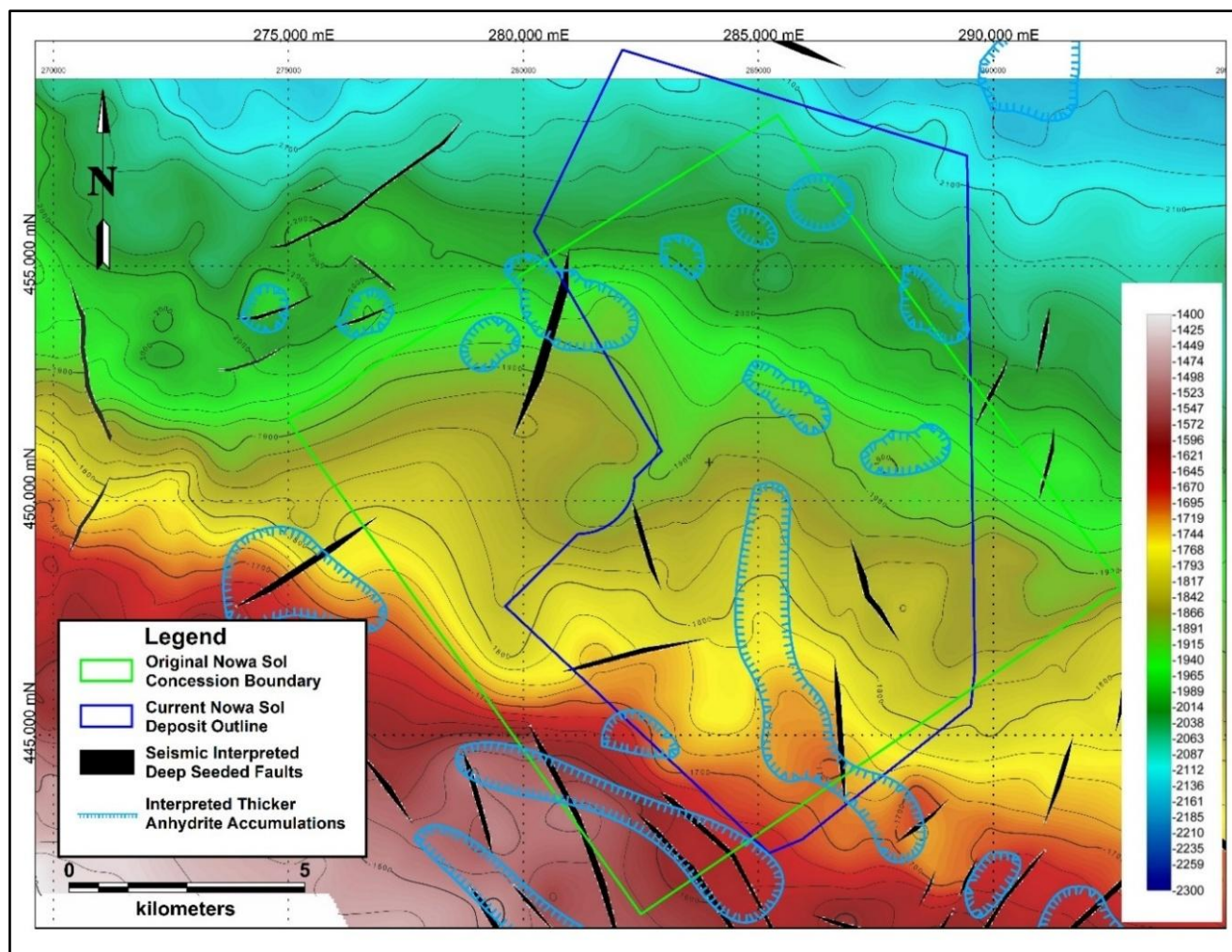
Of the 308 historical drill holes reviewed, permission was granted to sample 241 holes. The remaining 67 holes were limited to non-destructive testing including relogging of core and XRF analysis.

9.5 QPS OPINION ON THE EXPLORATION RESULTS

In the opinion of the QP, the exploration activities undertaken by Lumina have been appropriate and relevant to the geological setting and depth of the target mineralization. Given that the copper-silver mineralization occurs at depths exceeding 1,700 m, conventional surface exploration methods such as geochemical sampling or trenching are not applicable.

As such, Lumina’s approach, centered on the integration and reinterpretation of historical geological and geophysical data, including seismic and gravity surveys, has been technically justified. The development of a palaeo-topographic model to predict favourable depositional environments provided a sound basis for prioritizing mineralized zones and delineating subsurface drill targets. The QP considers the exploration work to be consistent with industry best practices and suitable for advancing the Project.

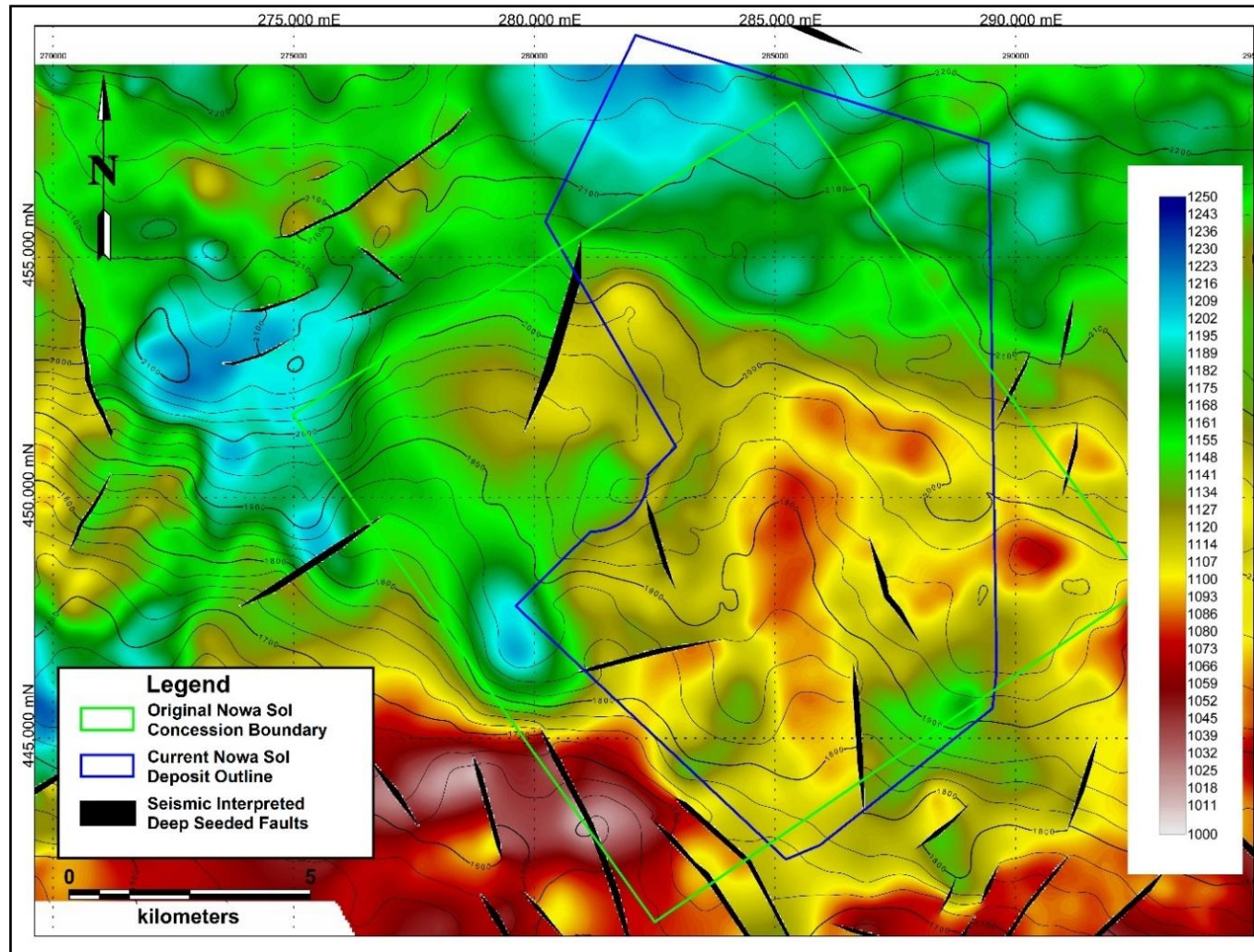
Figure 9.1: Uppermost of Rotliegende Isopac Map with Interpreted Thicker Accumulations of Anhydrite



Source: Lumina, 2025.

Note: Warmer colours indicate shallower depth to T1 horizon, cooler colours indicate deeper depth to T1 horizon. Depths are noted in metres and are apparent depths referenced from the mid Triassic Tp2 marker horizon.

Figure 9.2: Paleotopographic Model



Source: Lumina, 2025.

Note: Warm colours indicate paleotopographic highs such as dunes and sandbars, cool colours indicate paleotopographic lows such as basinal slopes and deep basins..

10 DRILLING

10.1 LUMINA DRILLING (2014 - 2024)

10.1.1 Overview and Summary of Drill Holes

Since 2014, Lumina has completed a total of 26 drill holes on the Project, comprising 51,413.27 m of drilling. The most recent drilling was concluded in August 2024.

All drill holes completed within the Nowa Sól deposit are listed in Table 10.1 and presented in Figure 10.3. Drill hole NS_C17B originally fell within the Nowa Sól concession area, but after the subsequent exploration concession application in 2025 the hole now falls slightly outside the revised permit area.

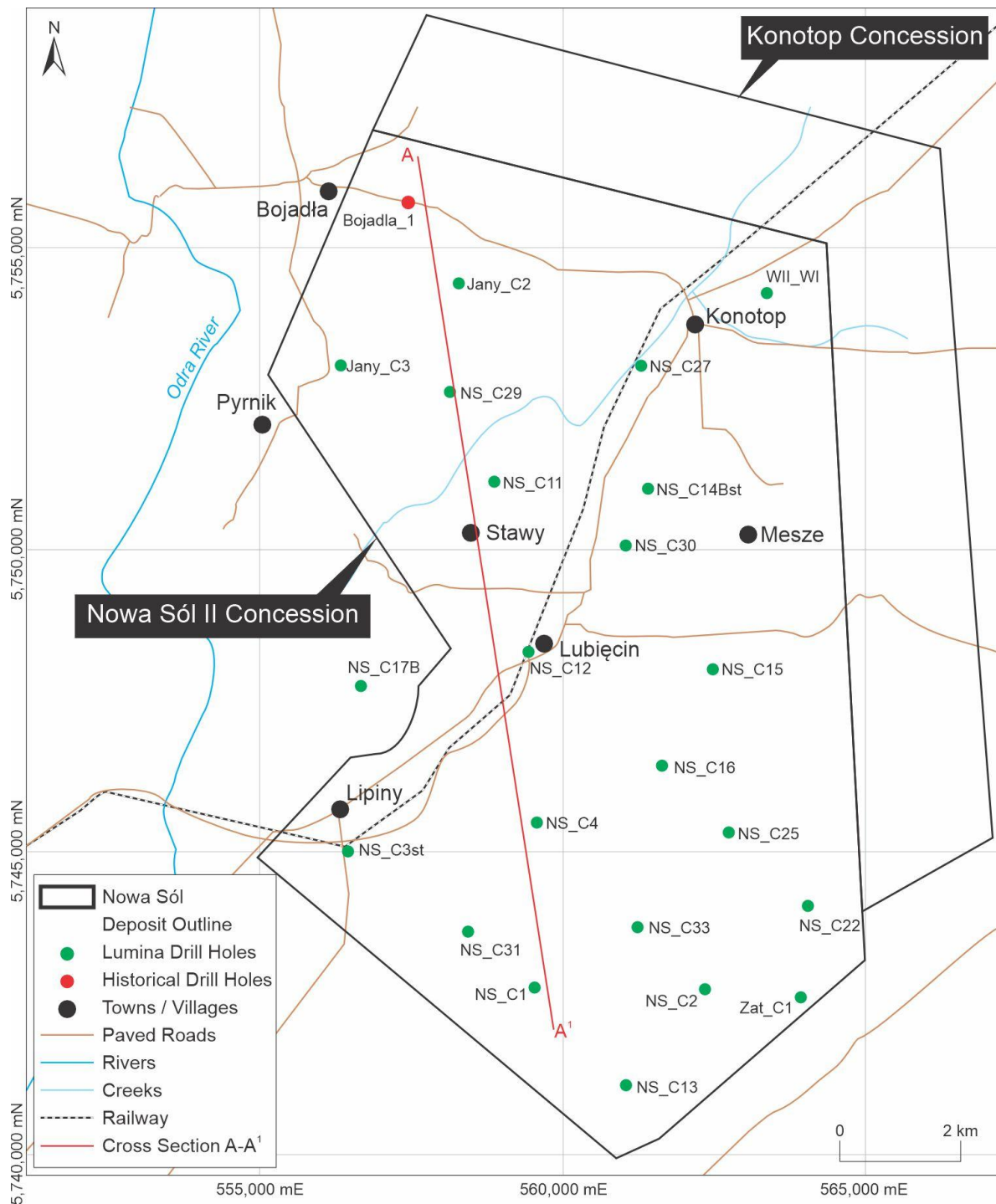
Table 10.1: Nowa Sól Deposit Lumina Drill Hole Data

Hole ID	Year	Easting	Northing	Elevation	Azimuth	Dip	EOH (m)
NS_C1	2014	559,531	5,742,783	74	360	-90	1,849
NS_C3	2014	556,460	5,745,031	81	360	-90	1,899
NS_C3st	2014	556,460	5,745,031	81	360	-90	1,895
NS_C2	2015	562,355	5,742,778	125	360	-90	1,882
NS_C4	2015	559,561	5,745,495	84	360	-90	1,913
NS_C12	2015	559,431	5,748,320	77	360	-90	1,960
NS_C13	2015	561,062	5,741,163	75	360	-90	1,790
NS_C16	2015	561,662	5,746,413	82	360	-90	1,943
NS_C22	2015	564,068	5,744,120	92	360	-90	1,932
Wil_W1	2015	563,362	5,754,259	60	360	-90	2,167
NS_C11	2016	558,850	5,751,085	59	360	-90	2,054
NS_C17B	2016	556,663	5,747,730	78	360	-90	1,968
Jany_C2	2017	558,273	5,754,394	59	360	-90	2,126
Jany_C3	2017	556,333	5,753,045	59	360	-90	2,112
Zat_C1	2017	563,976	5,742,598	85	360	-90	1,895
NS_C14B	2018	561,419	5,751,036	64	360	-90	2,075
NS_C14Bst	2018	561,419	5,751,036	64	360	-90	2,070
NS_C25	2018	562,752	5,745,325	93	360	-90	1,950
NS_C33	2018	561,287	5,743,784	82	360	-90	1,880
NS_C30	2019	561,045	5,750,073	69	360	-90	2,032
NS_C15	2020	562,486	5,748,019	81	360	-90	1,993
NS_C27	2023	561,294	5,753,058	63	360	-90	2,119
NS_C27st	2023	561,294	5,753,058	63	360	-90	2,121
NS_C29	2023	558,132	5,752,624	59	360	-90	2,100
NS_C31	2024	558,437	5,743,681	84	360	-90	1,848
NS_C31st	2024	558,437	5,743,681	84	360	-90	1,842
Total Lumina 2014 - 2024							51,413
<i>Bojadla_1</i>	<i>1986</i>	<i>557,453</i>	<i>5,755,738</i>	<i>57</i>	<i>360</i>	<i>-90</i>	<i>2,205</i>

Source: Lumina, 2025.

Note: CRS: UTM WGS 84, Zone 33 North; the hole in italics, Bojadla_1 is an historical drill hole.

Figure 10.1: Lumina Drill Hole Locations



Source: Micon modified Lumina, 2025.

Note: Cross-Section A-B is presented in Figure 10.6.

10.1.2 Drilling Method and Recoveries

Drilling at the Nowa Sól deposit has been carried out on a relatively continuous basis since 2014, with a temporary suspension during 2021 and 2022. Over the course of the program, three independent drilling contractors, each unaffiliated with Lumina, have been engaged.

The majority of the work, comprising 26 drill holes, was completed by EXALO Drilling S.A. (Exalo), a Polish contractor, using three rigs: the Service King SK-575, IRI 750, and Skytop TR-800. GEOPS Deep Drilling (Bulgaria) completed three drill holes using a Schramm TXD rig, while G-Drilling S.A. (Poland) completed one drill hole with an MU 160-37 rig.

All rigs employed were capable of drilling to depths of at least 2,500 m and were comparable to conventional oil and gas rigs. The rigs were equipped with mud separation systems, blowout preventers, and downhole telemetry for real-time monitoring of drilling parameters. Drilling operations were conducted on a 24-hour basis in two 12-hour shifts, with each hole requiring approximately 30 to 45 days to complete. Drill pads were remediated upon completion of drilling, cementing, and surveying. Pads located in forested areas were replanted as part of site rehabilitation. Figure 10.4 shows a typical example of the drilling setup used.

Geological data collected during drilling comprised rock chips and diamond drill core. Given the target depths, which at times exceeded 2,100 m, most drilling was initially conducted using mud rotary methods—standard practice in deep exploration and the oil and gas sector. Upon nearing the target interval, drilling transitioned to diamond coring.

Diamond core drilling typically commenced from the lower portion of the Lower Anhydrite (A1d) and continued into the Lower Permian Sandstones (P1) to capture the full mineralized sequence. Core was predominantly recovered in PQ size; however, due to geological conditions requiring casing cementation, some intervals were cored in HQ.

During diamond drilling, if core recovery within the mineralized interval was less than 90%, the drill hole was re-entered and re-drilled to ensure complete recovery of the mineralized section. The QP has reviewed the recovery data for the diamond drilling and confirms the average recovery of all sampled drill core was 98%. The standard procedure involved withdrawing the drill rods to a competent geological unit; typically either the Upper Anhydrite (A1g) or Lower Anhydrite (A1d). The hole was then cemented to a depth approximately 10 to 30 m above the intended re-entry point and allowed to cure. Subsequently, a 9 m long steel wedge was installed to enable directional deviation of the borehole. Re-drilled holes are referred to as "sidetracks" and are denoted by a "_st" suffix in the hole identifier.

During the coring phase, drilling was conducted in 9 m intervals using a fiberglass core barrel. The core was retrieved to surface by tripping the drill rods, with the final rod set down on a ground-level working platform. The core was then hydraulically extracted from the barrel onto a steel trough, where it was pressure-washed to remove drilling mud. Core sections were aligned, measured, and compared against drilling telemetry data to calculate core recovery. Lithological contacts were logged at this stage. Subsequently, the core was placed into wooden boxes, each 1 m in length, labelled with Hole ID, Box Number, and 'From-To' intervals marked in black. All measurements were recorded to an accuracy of ± 1 cm.

Figure 10.2: Example of Drill Rig Size and Setup Used by Lumina (2014–2024)



Source: Lumina, 2025.

10.1.3 Logging and Photography

Upon retrieval, the drill core was placed into core boxes, each clearly labelled with the corresponding Hole ID, box number, sample intervals, and XRF sample locations. Initial whole-core photographs were taken at the drill site under natural lighting conditions to provide a preliminary visual record prior to transport.

Preliminary logging of core was conducted on-site by company geologists and details on lithology, mineralization, alteration, structures, and core loss were recorded.

Upon completion of drilling, all core was transported by Lumina personnel to the Company's secure core storage and logging facility located in Kraków, Poland. At the facility, the core was cleaned, systematically logged in detail, and final sampling intervals were selected based on lithological, structural, and mineralogical criteria. Whole-core photography was subsequently conducted under controlled lighting conditions.

Logging was completed using a Microsoft Excel logging template designed by the Company's geology team. The drill log includes:

- Core Box No., From, To and Interval Length in metres.
- Core Recovery.
- Lithology.
- Description / Observations of the Lithology.
- Contacts.
- Weathering and Alteration.
- Depth of Structure.
- Alpha and Beta Angle of the Structure.
- The recorded Portable X-Ray Fluorescence (pXRF) Cu value where applicable.
- General Comments / Observations.
- Stratigraphy.

- Geological Group / Formation.
- Temperature.
- Sampling intervals.

10.1.4 Sampling

During the mud rotary drilling, rock chips were collected at 2 m intervals from surface to the target depth within the Lower Anhydrite (A1d). Collected chips were washed, dried, and placed into clear, labelled plastic vials indicating the hole ID and depth interval. Two chip sets were retained: one submitted to the PIG and the other stored at Lumina's core storage facility in Kraków, Poland.

Portable XRF analysis was conducted along the surface of the core to support geological logging and assist in identifying mineral zonation within the target stratigraphy. Sample spacing and density for XRF measurements varied according to lithology and visual indications of mineralization, with increased sampling density approaching the Kupferschiefer shale (T1) horizon. XRF results were strictly used for internal geological interpretation and selection of assay intervals; they were not used as a substitute for laboratory analyses and are excluded from any mineral resource or mineral reserve estimation.

Core sample intervals ranged from 0.04 m to 1.36 m, selected to respect lithological boundaries and reflect observed variations in mineralization tenor. Sample numbers were written on the core boxes in blue marker and two sample tags were also stapled onto the inside of the boxes at the beginning of each sample interval: one for Lumina records and the other to be used by the laboratory during sampling. Following core cutting, quarter-core samples were bagged and one of the sample tags was removed from the core box and put into the sample bag.

The quarter core samples were submitted to Acme Analytical Laboratories Ltd (Acme Labs) for analysis, and the remaining core was returned to Kraków for permanent storage following detailed photography. Selected samples were also subjected to specific gravity determinations and petrographic analysis, both performed by qualified Lumina personnel.

Polish law mandates that half the core be submitted to the PIG Central Geological Archives located in Leszcze, Poland for record keeping.

10.1.5 Collar Surveys

Initial drill collar locations were established using a handheld Garmin GPS unit. Upon completion of each drill hole, precise collar surveying was conducted by ART-GEO sp. z o.o., a surveying contractor based in Zielona Góra, Poland. Final collar coordinates were measured using a Leica RX1250 TC controller paired with an ATX GG antenna, achieving a horizontal accuracy of ± 0.005 m and a vertical accuracy of ± 0.010 m.

Two coordinate systems are commonly used in Poland: UTM ETRS89 / CS92 (EPSG:2180) and ETRS89 Poland CS2000, Zone 5 (EPSG:2176). Both systems are readily convertible to the WGS84 Zone 33N projection (EPSG:32633), which is the coordinate system adopted in this Report.

10.1.6 Downhole Surveys, Wireline Logging

All drill holes were initially collared vertically. However, downhole surveys were conducted at selected intervals during drilling to monitor and ensure minimal deviation from the planned drill trajectory. Upon completion of each hole, an independent contractor performed downhole wireline logging, recording parameters including azimuth, inclination, temperature, hole diameter, and gamma ray response. Logging data were captured at intervals ranging from 0.1 m to 1.0 m, providing detailed information for geospatial control and geological interpretation.

10.1.7 Data Capture and Verification

Drilling data within the cored interval of the drill hole is captured directly onto Lumina's drill log template in an Excel spreadsheet. Drilling data from surface to immediately above the mineralized zone is collected by Lumina's on-site Well Site Geologist and recorded onto a large continuous strip log and stored in PDF format (Figure 10.5). Downhole geophysical data collected post drilling is also provided by the contractor to Lumina in Excel format and incorporated into Lumina's drill log template. All drilling data is stored on the company's internal dataroom.

10.1.8 Drilling Results

Copper and silver mineralization was intersected in 23 of the 26 drill holes, at depths ranging from 1,790 m to 2,167 m, with mineralized intervals varying in thickness from 0.71 m to 4.97 m. Mineralized intercepts for all holes are detailed in Table 10.3.

Figure 10.6 presents a vertical cross-section illustrating the typically low-angle dip of mineralized lithologies within the Polish Zechstein Basin. All holes were drilled vertically; therefore, all reported mineralized intervals represent true thicknesses.

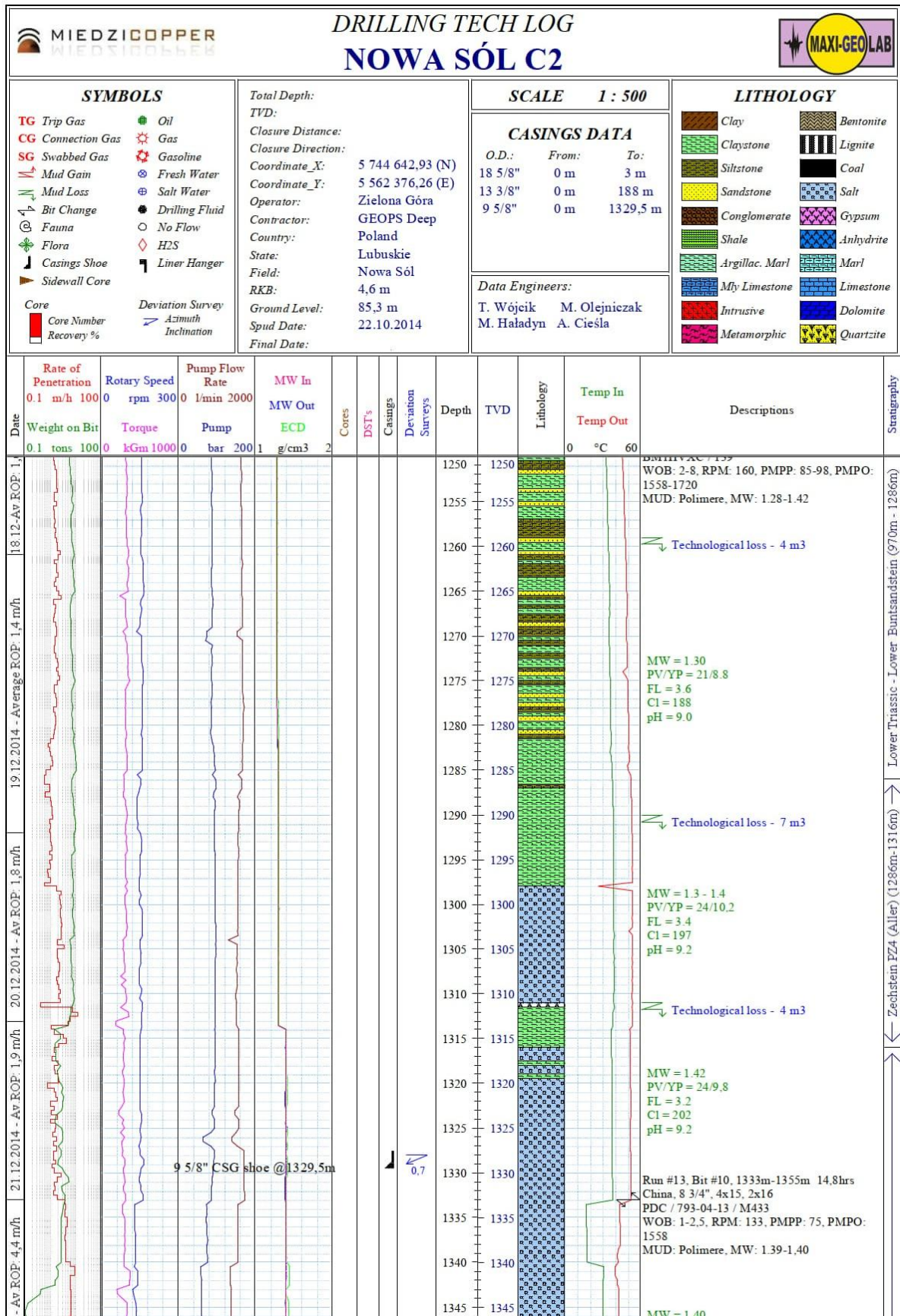
This drilling campaign culminated in the greenfield discovery of the Nowa Sól copper-silver deposit.

10.2 QPS OPINION ON THE DRILLING CAMPAIGNS AND RESULTS

The protocols used for logging and sampling are per industry standards, and the QPs see no reason why the drilling methods and sampling could materially impact the accuracy and reliability of the results.

The QPs are not aware of any drilling, recovery, or sampling factors that could materially impact the accuracy and reliability of the drilling results up to the effective date of this Report. The drilling has been designed to intersect the mineralized zones at a high angle, ideally approaching perpendicular. The downhole sample lengths reported are interpreted to be a reasonable approximation of the true thickness. No significant discrepancies between sample length and true thickness are apparent in these areas based on the current geological understanding and drilling orientation.

Figure 10.3: Nowa Sól C2 Drilling Tech Log Extract



Source: Lumina, 2025.

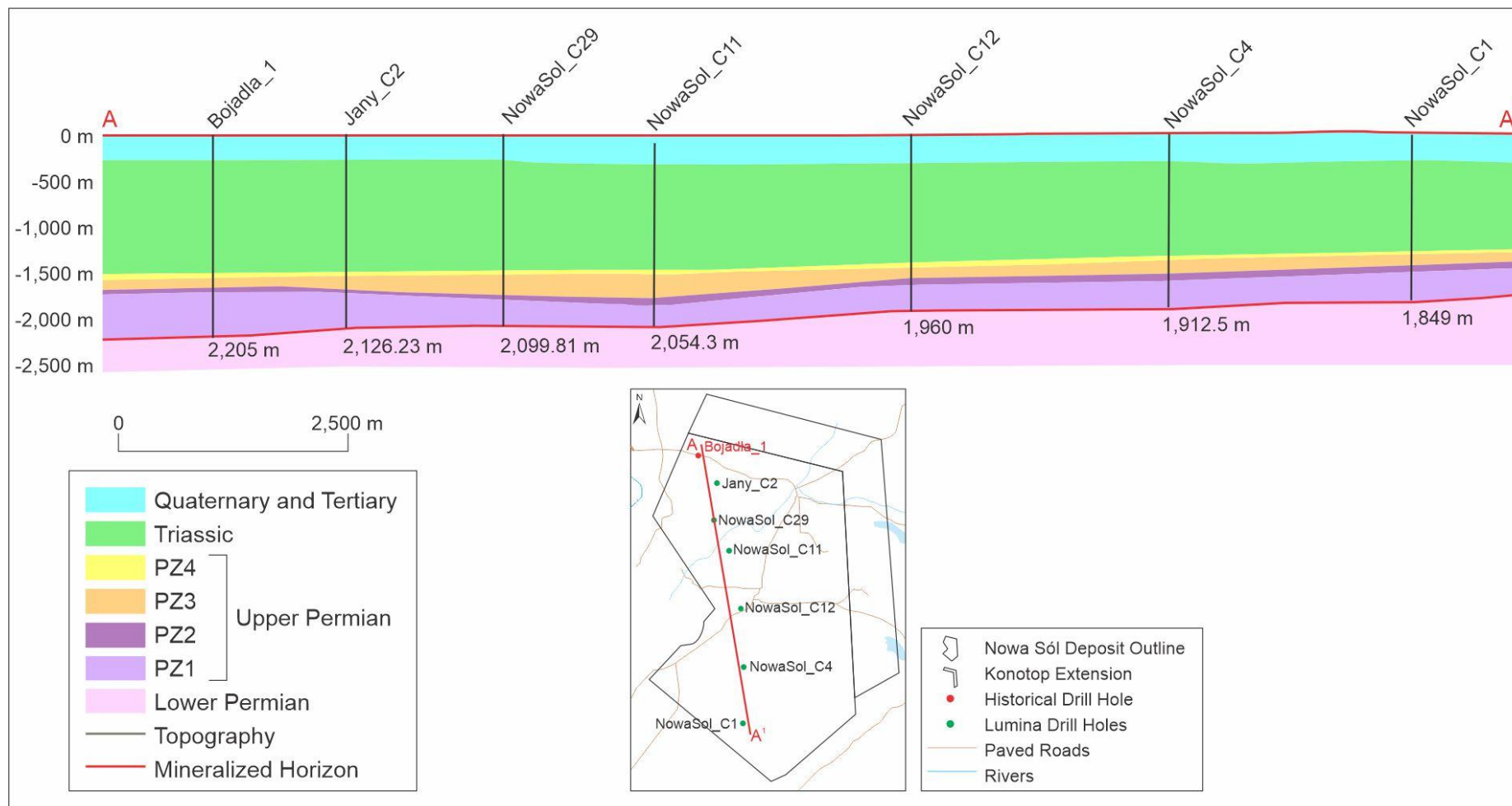
Table 10.2: Nowa Sól Deposit Mineralized Intercepts

Hole ID	Mineralization						Comments
	From (m)	To (m)	Interval (m)	Cu (%)	Ag (g/t Ag)	Cu _{eq} (%)	
NS_C1	1,840.41	1,844.96	4.55	2.04	25.8	2.27	-
NS_C3	1,888.97	1,891.02	2.05	1.06	17	1.21	Incomplete coring of mineralized zones
NS_C3st	1,883.10	1,884.85	1.75	0.96	14	1.09	Substitute for NS_C3
NS_C2	1,873.54	1,875.97	2.43	2.54	91.2	3.36	-
NS_C4	1,892.32	1,896.47	4.15	1.41	23.6	1.62	-
NS_C12	1,942.00	1,946.16	4.16	1.06	3.8	1.09	-
NS_C13	1,779.86	1,780.94	1.08	1.15	10.9	1.25	-
NS_C16	1,924.07	1,929.04	4.97	0.61	14.9	0.75	-
NS_C22	1,927.57	1,928.28	0.71	2.17	221.4	4.17	-
Wil_W1	2,155.89	2,160.53	4.64	0.53	19.4	0.7	-
NS_C11	2,041.56	2,043.52	1.96	2.39	39.8	2.75	-
NS_C17B	-	-	-	-	-	-	Completely oxidized. Non mineralized
Jany_C2	2,110.18	2,113.41	3.23	0.51	25.5	0.74	-
Jany_C3	2,107.45	2,107.91	0.46	5.97	142.2	7.25	-
Zat_C1	1,883.95	1,885.70	1.75	1.37	93.1	2.21	-
NS_C14B	2,064.44	2,066.66	2.22	-	-	-	Core retained for metallurgy sampling
NS_C14Bst	2,064.73	2,066.63	1.9	1.59	99.2	2.49	Substitute for NS_C14B
NS_C25	1,940.06	1,942.20	2.14	1.6	70.6	2.24	-
NS_C33	1,867.15	1,869.93	2.78	0.43	16.6	0.58	-
NS_C30	2,026.29	2,029.69	3.4	1.49	81.7	2.23	-
NS_C15	1,984.04	1,985.87	1.83	2	79.6	2.72	-
NS_C27	2,117.34	2,118.80	1.46	0.83	97.3	1.71	Incomplete coring of mineralized zone
NS_C27st	2,116.99	2,119.58	2.59	0.86	60	1.4	Substitute for NS_C27
NS_C29	2,092.79	2,095.78	2.99	1.54	43.2	1.93	-
NS_C31	-	-	-	-	-	-	Incomplete coring of mineralized zone
NS_C31st	1,837.42	1,838.68	1.26	0.66	5.6	0.71	Substitute for NS_C31
<i>Bojadla_1</i>	<i>2,144.75</i>	<i>2,145.75</i>	<i>1</i>	<i>1.72</i>	<i>351.1</i>	<i>4.88</i>	<i>Historic drill hole</i>

Source: Lumina, 2025.

Notes: Cu_{eq} = Cu (%) + ((Ag (g/t)*13)/1439.9862981)). The hole in italics, Bojadla_1 is an historical drill hole.

Figure 10.4: Cross Section of the Nowa Sól Deposit



Source: Micon modified after Lumina, 2025.

Note: Cross section shows the major stratigraphic units and the mineralized horizon looking East.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 LUMINA SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1.1 Sampling of Historical Drill Core

Between 2011 and 2012 sampling of the historical core was completed by Lumina staff and sent to the independent laboratory Acme Labs in Vancouver, British Columbia. (Acme Labs) for analysis. Lumina routinely inserted in-house standards, blanks, and duplicates into the sample stream for in house QA/QC. Acme Labs also conducted its QA/QC protocol. A total of 3,724 samples were tested including 210 duplicates, 54 blanks and 165 standards that were inserted by Lumina staff.

Two analytical methods were conducted on samples: A lead-collection fire-assay fusion for total sample decomposition and Inductively Coupled Plasma Optical Emission Spectroscopy (ICP-OES) finish (Acme Code 3B) was conducted for gold, platinum, and palladium, while a suite of 23 elements including copper, silver, lead, zinc, and molybdenum was completed using a Hot 4-Acid digestion with Inductively Coupled Plasma Optical Emission Spectroscopy (ICP-OES) finish (Acme Code 7TD).

Within the Nowa Sól licence, Bojadła 1 was the only historical drill hole which was incorporated into the resource estimate. Historical drill core from Bojadła 1 is stored at Magazyn NAG, the Polish Core Depository located in Leszcze, Poland. Viewing access and non-destructive analysis (XRF analysis) were allowed for Bojadła 1, however no sampling for analytical work was allowed.

Lumina subsequently purchased the official analytical results from the State via PIG in March, 2015. Historical sample methodology is discussed in Section 6.3.

11.1.2 Sample Security and Chain of Custody

Throughout the drilling campaigns, Lumina geologists were present on-site to monitor operations and log all rock chips recovered during the mud rotary drilling phase. During diamond drilling, only qualified Lumina geologists were responsible for handling the core to ensure adherence to QA/QC protocols and to maintain full chain of custody and core security.

At the drill site, core boxes were stored in locked steel sea containers, with access restricted to authorized Lumina personnel. Upon completion of drilling, core was transported by Lumina staff directly to the Company's secure core storage facility located in Kraków, Poland.

Drill core requiring analytical testing was hand-delivered by Lumina staff to Acme Labs in Kraków. Core cutting was performed under supervision by qualified Lumina geologists. Quarter-core samples were submitted for assay, while the remaining core was returned to the Kraków core facility for final photography and permanent storage.

Samples were routinely selected for specific gravity determinations and petrographic analysis, both of which were conducted internally by Lumina personnel.

All drill core, rotary chips, assay rejects, and pulps from the Lumina drilling programs are stored in a dry, secure warehouse in Kraków. To support future metallurgical studies, significant sulphide-bearing intervals and associated coarse assay rejects have been preserved in freezers at the Kraków facility to minimise oxidation and preserve sample integrity.

11.1.3 Sample Preparation and Analysis

All preparation and analytical work was completed by Acme Labs. Sample preparation was conducted in Kraków, Poland and analytical work was completed in Vancouver, Canada. Bureau Veritas purchased Acme Labs in February 2012, however the preparation laboratory operated under the Acme Labs name until November 2016.

Drill core was aligned and marked with a wax crayon for reference, then cut in half longitudinally using an electric powered diamond saw. After drill core was cut and placed into stainless steel trays under the supervision of Lumina staff, the samples were placed onto a hanging rack and dried in an oven, afterwards the entire sample was crushed to $\geq 70\%$ passing through 2 mm sieve, homogenized, split into a 250 g sample and pulverized so that $\geq 85\%$ material passed through a 75 μm sieve (Procedure PRP70-250). Samples were then packaged in paper bags, labelled and shipped to Bureau Veritas laboratory in Vancouver, Canada via Courier for analysis. Bureau Veritas in Vancouver is a fully accredited laboratory with current International Organization for Standardization (ISO) 9001:2015, ISO14001:2015 and ISO 45001:2018 accreditations.

In Vancouver, all samples were analyzed using multiple analytical techniques: AQ201 (Later substituted for AQ251-EXT-REE), MA370, and FA330.

The AQ201 and later AQ251-EXT-REE procedure utilizes A 15 g sample of pulp that is digested with a modified Aqua Regia solution of equal parts concentrated HCl, HNO₃ and deionized H₂O for one hour in a heating block or hot water bath. The sample is made up to volume with dilute HCl and the solution is then analyzed by Inductively Coupled Plasma Mass Spectrometry (ICP-MS). The detection limits for AQ201 are 0.1 ppm for silver, copper, lead, and molybdenum, and 1 ppm for zinc.

Around June 2017, Lumina substituted procedure AQ201 for procedure AQ251-EXT-REE to include results for rhenium. The procedure is the same as AQ201, however AQ251-EXT-REE returned a total of 54 elements including rhenium compared to 36 elements from AQ201. The detection limits of AQ251-EXT-REE differed slightly with 2 ppm for silver and gold, 0.1 ppm for copper, lead, molybdenum, and 1 ppb for rhenium. Aqua Regia digestion is considered a partial digestion compared to a four-acid digestion in which a near total digestion of minerals is obtained.

If copper, molybdenum, lead or zinc returned values $>1,000$ ppm, or the silver is >10 ppm, or the gold is >100 ppb during the AQ201 or AQ251-EXT-REE procedure, samples were automatically reassayed using Multi Acid ICP-OES (Procedure MA370) and Fire Assay methods (Procedure FA330) to obtain near-total values for copper, silver, molybdenum, zinc, lead, and gold.

Procedure MA370 is a near total acid digestion with Multi Acid ICP-OES analysis, whereby a 30 g sample was dissolved in a four acid digest consisting of HNO₃, HF, HClO₄, and HCl. The sample is first digested with HNO₃ and HF, taken to dryness, then digested again with HClO₄ and HCl and taken to dryness again. A last digestion of HCl is completed and the solution is bulked to 10 ml using distilled water. This results in a near total digestion of the sample which is then analyzed by ICP-OES. Upper detection limits are 1,000 ppm for silver, 10% for copper, 4% for lead, and 20% for zinc.

Acme Labs also insert their own sequence of blanks, standards and duplicates to ensure their in-house QA/QC is maintained.

Remaining reject and pulp material from Lumina drill core samples were returned to Lumina and is stored in the dry and secure Core Storage facility in Kraków, Poland. Mineralized pulps and selected cores were vacuum packed and stored in a freezer to prevent oxidation.

11.1.3.1 Density Measurements

Lumina staff conducted in-house bulk density measurements on samples from all holes drilled by the company. The measurements employed the Waxed Air/Water method. Selected samples were first logged with depth, lithology, and physical dimensions recorded. Samples were then oven-dried at Acme Labs Preparatory Laboratory in Kraków to remove moisture. Following drying, samples were weighed in air without wax, subsequently coated thoroughly with wax and allowed to dry. The waxed samples were then weighed in air and submerged in water to obtain the necessary weight.

All data was compiled in a spreadsheet, and average densities were calculated for each hole based on lithological units. These average densities were applied in the mineral resource estimations described in Section 14.

11.1.3.2 QA/QC

A comprehensive QA/QC program should incorporate Certified Reference Materials (CRMs), blanks, and duplicates (field, pulp, and umpire) to ensure the reliability and accuracy of exploration data that can be used in Mineral Resources Estimations. This process must be proactive, continuously monitored, and periodically reviewed to allow for timely corrective actions in drilling, sampling, or laboratory procedures (Coombes, 2008).

Micon was not supplied a database of the laboratory's internally implemented QA/QC data. As a result, the Micon QPs have assessed only the QA/QC samples and procedures independently implemented by Lumina. Given that the two Acme (now Bureau Veritas) laboratories utilized are well-established, independent and certified, it is reasonable to assume that they adhered to a rigorous and comprehensive QA/QC program in line with industry standards.

The Micon QPs established an independent QA/QC database by consolidating the various databases and laboratory certificates provided. This database was utilized to generate the charts and data necessary for the analysis. A summary of the QA/QC samples used during the drilling period is presented in the corresponding Table 11.1.

A review of the QA/QC protocols was conducted prior to drilling and formalized in a detailed QA/QC manual developed by Lumina. The QA/QC program was conducted in accordance with industry best practice and no major quality control issues were discovered throughout the drilling period from 2014 to 2024.

The Lumina sampling QA/QC protocols require blanks, duplicates, and standards to be routinely inserted every tenth sample within the sampling sequence. In addition, it was common practice to insert a coarse blank after intervals where high grade mineralization was expected to verify that the laboratory was adequately cleaning their crushing and pulverizing machines between samples.

From 2014 to 2024 a total of 26 drill holes were completed by Lumina totalling 1,249 samples analyzed comprising: 1,090 drill core samples, 50 coarse blanks, 71 CDN (CDN Resource Laboratories) Standards and 38 fine duplicates. After each batch of analytical results was released by the laboratory, the QA/QC samples were reviewed by a Lumina geologist.

During the drilling period the QA/QC sample insertion rate totalled 14.6% of the 1,090 drilling samples with individual insertion rates for blanks, standards and field duplicates covering 4.6%, 6.5% and 3.5%, respectively. These rates align with industry standards, ensuring comprehensive monitoring for accuracy, contamination, and precision.

The charts analysing the results of the blanks, standards, and field duplicates are presented in Figure 11.1 to Figure 11.8.

Table 11.1: Summary of QA/QC Sample Insertion Rates

QA/QC Sample Type	No. Samples	Insertion Rate (%)
Regular Drilling Samples	1,090	-
Blank	50	4.6
Standard	71	6.5
Duplicate	38	3.5
TOTAL QA/QC SAMPLES	1,249	14.6

Source: Micon, 2025.

Blanks

50 coarse blanks were analyzed for copper, silver, lead and zinc. The blank material selected by Lumina was quartzite, from a quarry located in Bukowa Góra, Poland, which was acquired from Acme Labs.

In line with industry best practices, Micon typically deems acceptable individual blank results to be those which fall at or below a threshold of three times the detection limit for the analytical method used and expects a suite of blanks to exhibit no more than 10% of samples above this threshold. A summary table of the blank results is presented in Table 11.2. Based on these thresholds, copper, lead and zinc would all fail as 100%, 100% and 44% of their blanks fall above the threshold, respectively. Silver is considered a pass as only 8% of samples exceed the threshold.

The quarry at Bukowa Góra, while not located in a region known for base metal mineralization, appears to contain trace to locally elevated concentrations of copper, lead, and zinc. This is supported by the consistent presence of low to moderate-level results in the blank assays, irrespective of their position relative to mineralized samples. These results suggest that the blank material itself is not geochemically inert with respect to the target elements. As such, Micon considers the elevated values to be inherent to the blank material rather than the result of cross-contamination or sample preparation error.

Table 11.2: Summary of Blank Sample Results

Element	Number of Samples	Detection Limit (ppm)	Number of Samples Above 3 x Detection Limit	Percentage of Samples Above 3 x Detection Limit
Copper	50	0.1	50	100%
Silver	50	0.1	4	8%
Lead	50	0.1	50	100%
Zinc	50	1	22	44%

Source: Micon, 2025.

Plots for the blank results for copper, silver, lead and zinc can be seen in Figure 11.1, Figure 11.3, Figure 11.5 and Figure 11.7, respectively.

Although Micon would typically consider the blank results for copper, lead and zinc to be failures based on a threshold of three times detection limit, Micon feels that in this instance the blank results do not prevent the dataset from being acceptable for use in a Resource Estimate. This decision is based on the incredibly low detection limits for the analyses methods used, and the fact that for copper, lead and zinc all of the blank results fall below the cut-off used to produce the economic composites which define the orebody extents for Micon's estimate (Section 14), and for zinc only the most extreme outliers fall above the cut-off. A comparison of the detection limit, maximum blank assay value returned cut-off used for economic composites for each element can be seen in Table 11.3.

Table 11.3: Comparison Table of Blank Assay Outliers to Economic Composite Cut-Off Grades

Element	Detection Limit (ppm)	Maximum Blank Value (ppm)	Economic Composite Cut-Off (ppm)
Copper	0.1	389.5	500 (0.05%)
Silver	0.1	2.2	1.0 ppm
Lead	0.1	169.1	1000 (0.1%)
Zinc	1	36	500 (0.05%)

Source: Micon, 2025.

It is Micon's opinion that the blank material chosen was inappropriate, but this does not invalidate the overall QA/QC dataset or the ability to use the drilling data in a compliant Mineral Resource, due to the mitigating circumstances outlined above.

Even in the event that the blank failures were caused by sample contamination rather than poor material selection, the absolute values reported remain well below the economic composite cut-offs used to define the mineralized domains and would therefore be considered immaterial to the resource outcomes. As such, Micon does not consider these failures to compromise the validity of the drilling dataset for use in a Mineral Resource Estimate.

To further investigate the cause of the elevated results in the blank samples, Micon recommends that approximately 10% of the blanks be resubmitted to one or more other accredited laboratories for confirmatory analysis. Comparison of results will help determine whether the source of the elevated values lies in the blank material itself or in possible contamination introduced during sample preparation or analysis.

Micon further recommends that an alternative blank material be sourced for future programs. Ideally, this material should be certified or independently tested and confirmed to contain negligible levels of the elements of interest.

Standards

For all Lumina drilling, standards were randomly inserted into the sample stream by Lumina geologists prior to assaying. Three CDN standards (Table 11.4) were selected that adequately cover the low, medium and high-grade values of the expected ranges.

Table 11.4: Summary Table of CDN CRMs

CRM	Certified Cu Mean (%)	Certified Cu SD (%)	Certified Ag Mean (g/t)	Certified Ag SD (g/t)	Certified Pb Mean (%)	Certified Pb SD (%)	Certified Zn Mean (%)	Certified Zn SD (%)
CDN-ME-6	0.613	0.17	101	3.55	1.02	0.04	0.517	0.020
CDN-ME-11	2.44	0.055	79.3	3	0.86	0.05	0.96	0.03
CDN-ME-17	1.36	0.05	38.2	1.65	0.676	0.027	7.34	0.185

Source: Micon, 2025.

The selected CRMs presented demonstrate that all assay results, for all four elements analyzed, fall within the acceptable limits defined by the ± 2 standard deviations (SD) and ± 3 SD bands around the certified mean values. A summary table of the CRM performance is provided in Table 11.5.

Table 11.5: Summary Table of CRM Performance

CRM	Element	Number of Samples	Number of Samples Outside 2SD	Percentage of Samples Outside 2SD	Number of Samples Outside 3SD	Percentage of Samples Outside 3SD
CDN-ME-6	Copper	17	0	0%	0	0%
	Silver	16	1	6%	0	0%
	Lead	19	0	0%	0	0%
	Zinc	19	2	11%	0	0%
CDN-ME-11	Copper	30	0	0%	0	0%
	Silver	29	0	0%	0	0%
	Lead	30	0	0%	0	0%
	Zinc	30	0	0%	0	0%
CDN-ME-17	Copper	22	0	0%	0	0%
	Silver	19	1	5%	0	0%
	Lead	22	0	0%	0	0%
	Zinc	22	3	14%	0	0%

Source: Micon, 2025.

Plots for the CDN Standards for copper, silver, lead and zinc can be seen in Figure 11.1, Figure 11.3, Figure 11.5 and Figure 11.7, respectively.

Duplicates

The analysis of the 38 fine duplicates demonstrates reliable precision and reproducibility for all elements analyzed. The scatter plot shows a good correlation between original and duplicate samples with all R^2 values exceeding 0.99, indicating consistent assay performance. All data points, except for one single silver outlier, fall within the $\pm 20\%$ tolerance range.

The relative difference plot highlights that most paired differences fall within $\pm 20\%$, but more significant variability is observed at lower concentrations. This is likely due to the fact that analytical error becomes proportionally larger at lower grades and that at low grades the samples are more heterogeneous and exhibit a nugget effect. The precision plot further supports this, showing improved precision at higher grades, with most samples within $\pm 10\%$ and $\pm 20\%$ error margins. However, at lower grades, several points exceed the 20% threshold due to the nugget effect.

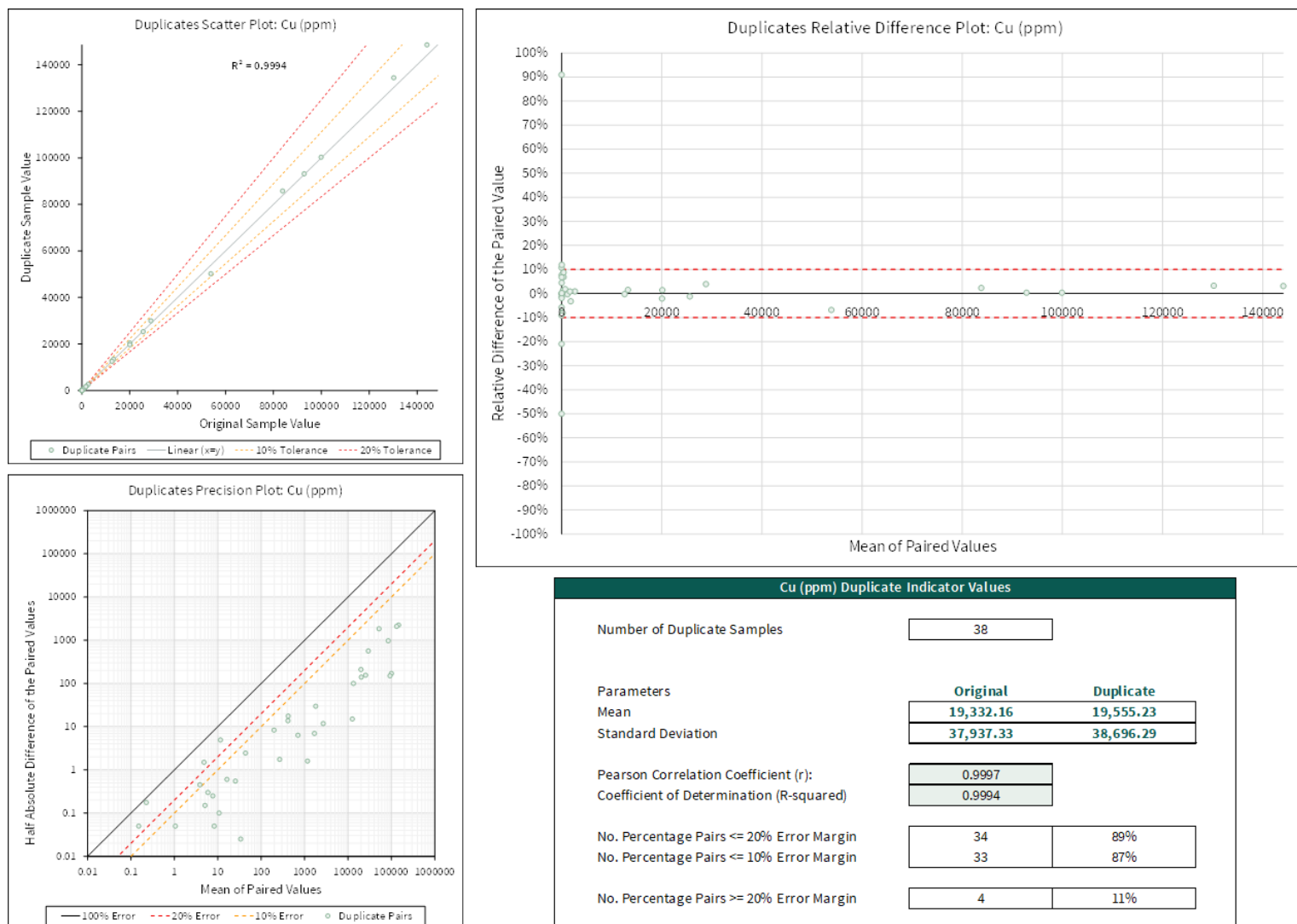
Plots for the duplicate results for copper, silver, lead and zinc can be seen in Figure 11.2, Figure 11.4, Figure 11.6 and Figure 11.8, respectively.

Figure 11.1: Copper Blanks and CRM QA/QC Results



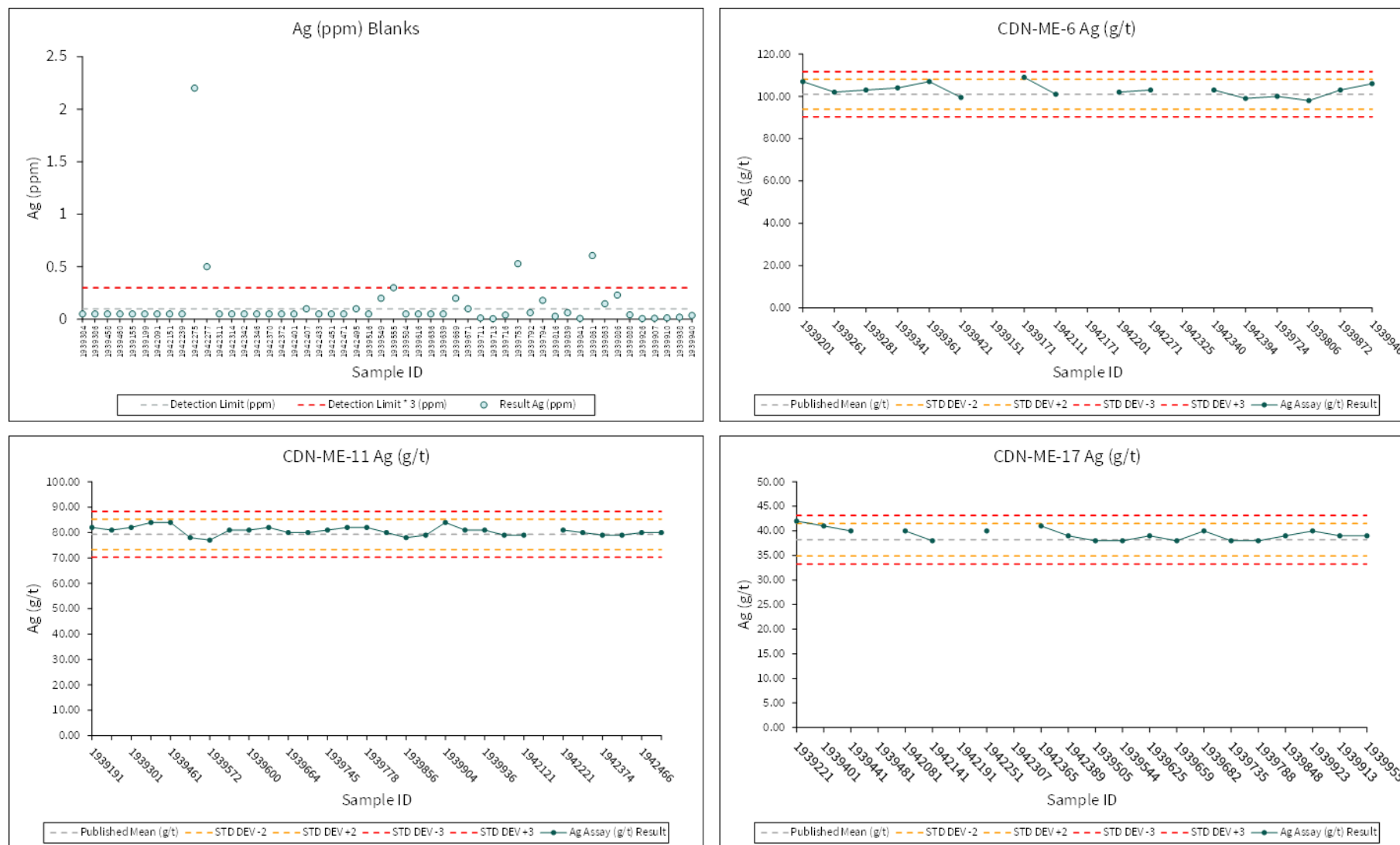
Source: Micon, 2025.

Figure 11.2: Copper Duplicate QA/QC Results



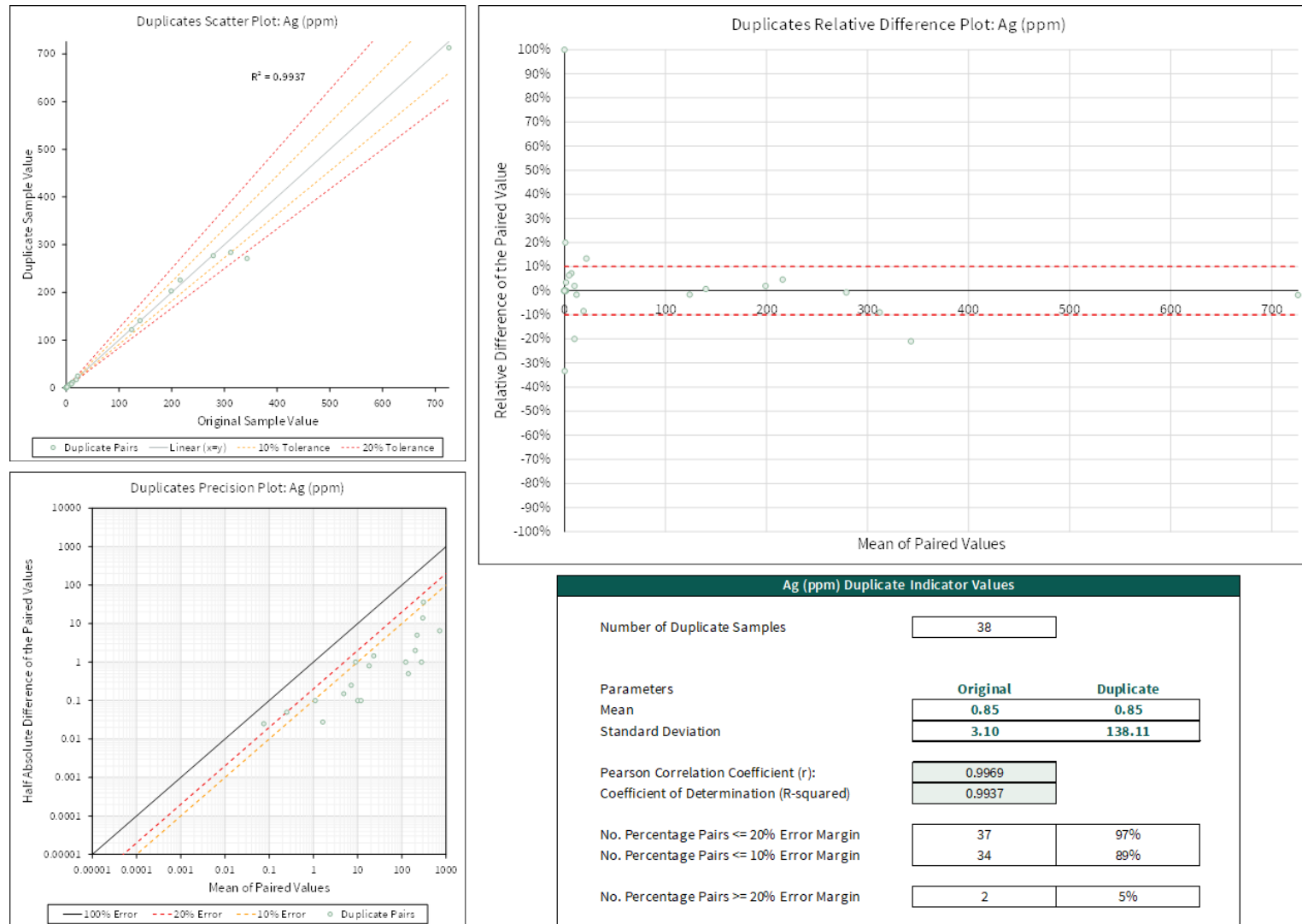
Source: Micon, 2025.

Figure 11.3: Silver Blanks and CRM QA/QC Results



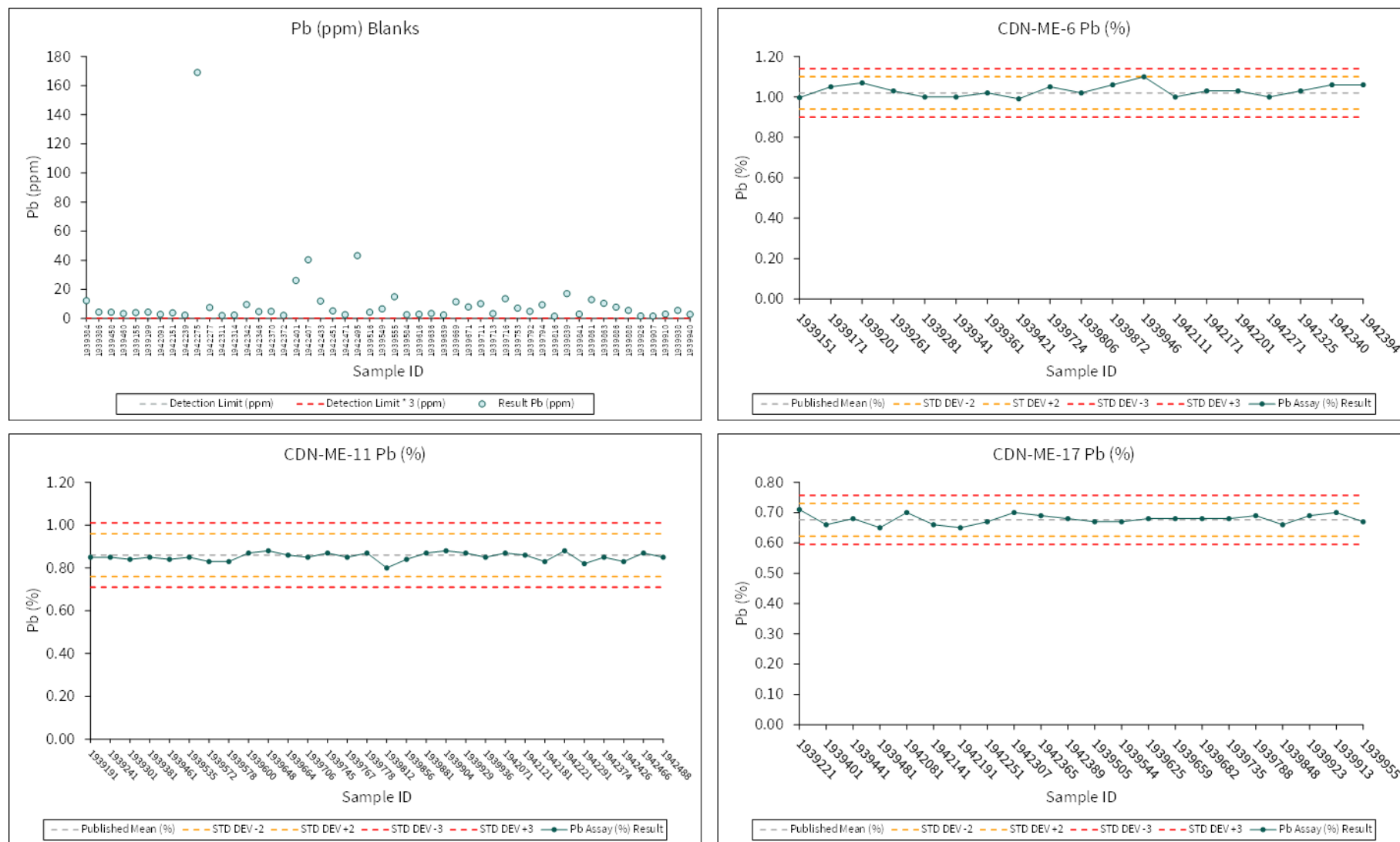
Source: Micon, 2025.

Figure 11.4: Silver Duplicate QA/QC Results



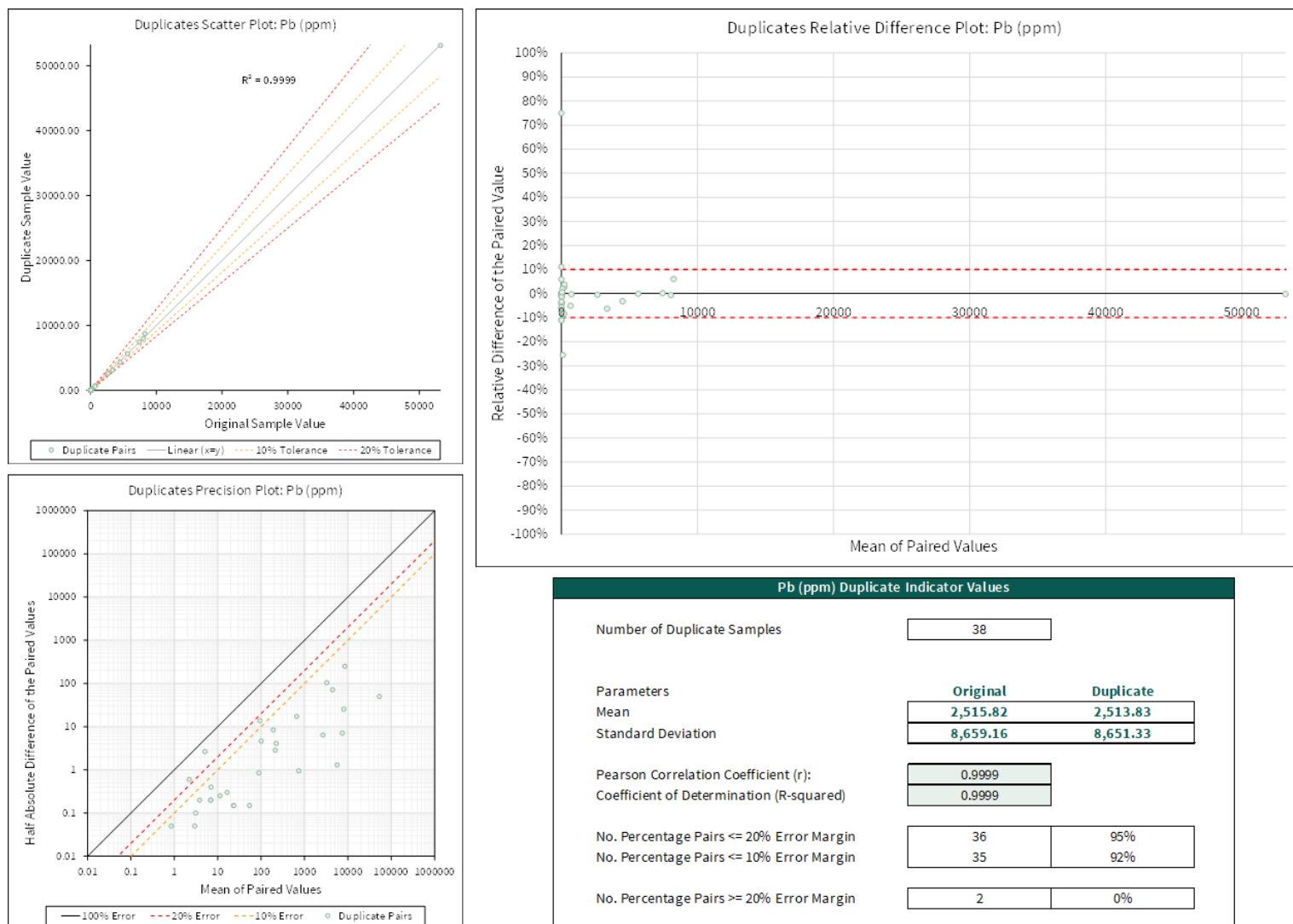
Source: Micon, 2025.

Figure 11.5: Lead Blank and CRM QA/QC Results



Source: Micon, 2025.

Figure 11.6: Lead Duplicate QA/QC Results



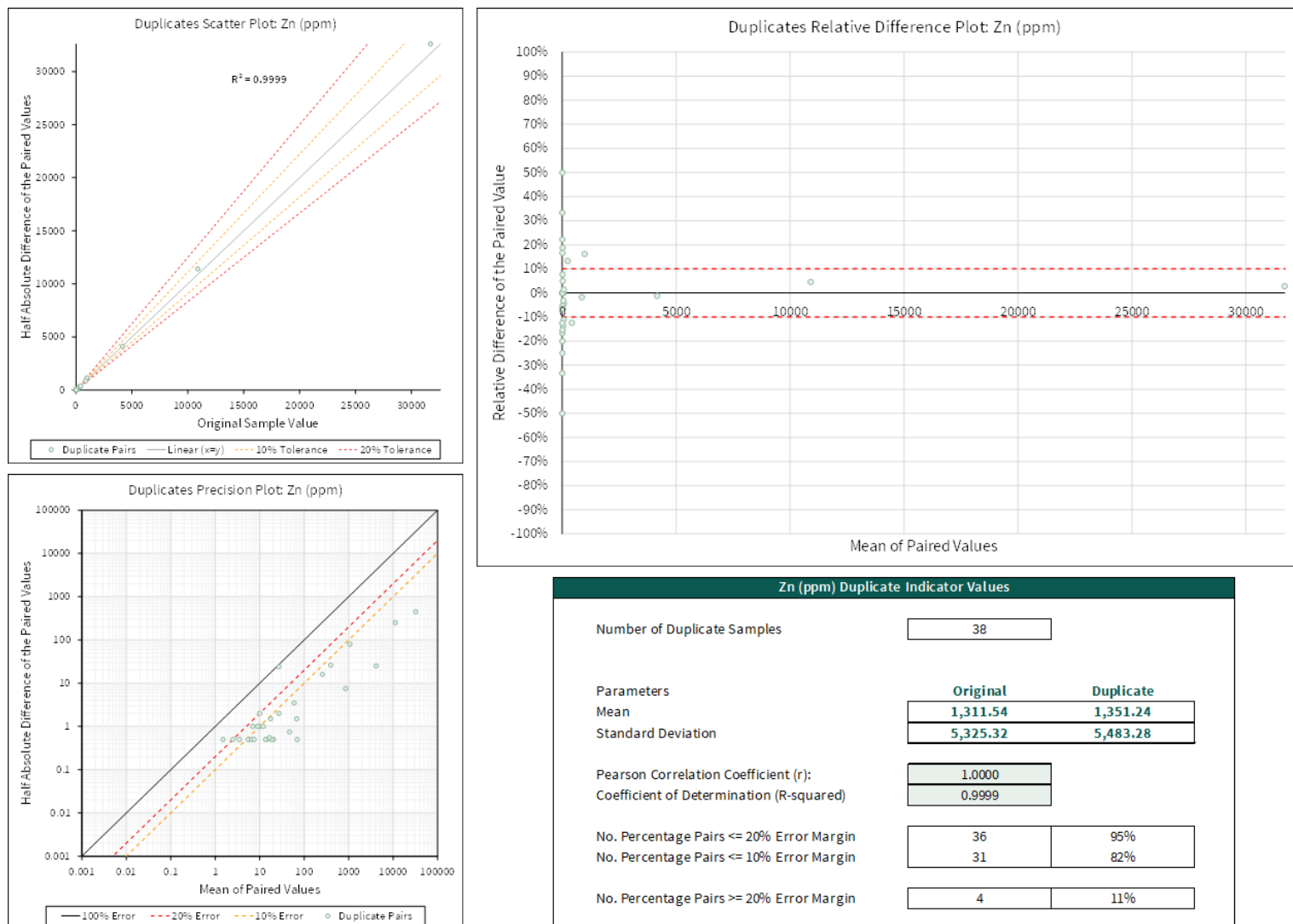
Source: Micon, 2025.

Figure 11.7: Zinc Blank and CRM QA/QC Results



Source: Micon, 2025.

Figure 11.8: Zinc Duplicate QA/QC Results



Source: Micon, 2025.

11.1.4 QPs Opinion of Lumina's Sample Preparation, Analyses and Security

In the QPs' opinion, the security, sampling, and analytical procedures are appropriate and consistent with common industry practice.

Lumina has always used Acme Labs (now Bureau Veritas) for all its preparation and analytical work. Both laboratories are recognized as accredited commercial assayers which are independent from Lumina. Lumina samples were prepared by Acme Labs located in Kraków, Poland and analytical work was completed by Bureau Veritas located in Vancouver, British Columbia, which currently holds an ISO/IEC 17025:2017 accreditation (ALS, 2025). Prior to the purchase of Acme Labs by Bureau Veritas, Acme Labs maintained an ISO/IEC 17025 accreditation.

The sampling of core was entirely conducted by the Lumina staff and in a manner that meets or exceeds common industry standards. Samples were properly identified and transported in a secure manner from site to the company's core storage facility and from the facility to the laboratory.

The author has reviewed the results of the standards, blanks and duplicates and has found no major discrepancies in the results.

It is recommended that a more sufficient coarse blank material is obtained to replace the previously used quartzite material. This would eliminate potential erratic copper values in blanks and eliminate false failures.

It is the opinion of the QP that sample preparation, sample security, and analytical procedures are adequate to support the estimation of mineral resources.

12 DATA VERIFICATION

12.1 SITE VISIT

Liz de Klerk, an Author and QP of this Report, visited the Property between February 12, 2025, and February 14, 2025, accompanied by Matthew Ball (Micon Resource Geologist) and Andrzej Zibrow and Tomasz Bieńko, Lumina personnel.

While at site the following were visited:

- The Nowa Sól deposit areas.
- Polish State Geological Survey Archives.
- Michałów core storage facility at Michałowice.
- Core storage facility at Kraków.
- Bureau Veritas sample preparation laboratory in Kraków.

The following checks were performed:

- A selection of historical and new drill hole collars checked with GPS.
- The historical and modern drill core was reviewed particularly mineralized sections.
- Logging and sampling intervals cross-checked.
- The on-site team ran through the protocols (sampling).

The following Micon staff, including QPs and authors of this Report, visited the Property between July 1, 2025, and July 3, 2025, accompanied by Lumina personnel: Liz de Klerk, Dr. Ryan Langdon, Peter Szkilnyk, Richard Gowans and Becky Humphrey.

While at site the following were inspected / visited:

- North, central and south zones within the Nowa Sól exploration concession.
- Nearby villages, waterbodies and protected areas for context.
- Odra River and Nowa Sól Port.
- Lumina's Core storage facility at Kraków (Dr. Ryan Langdon and Peter Szkilnyk).

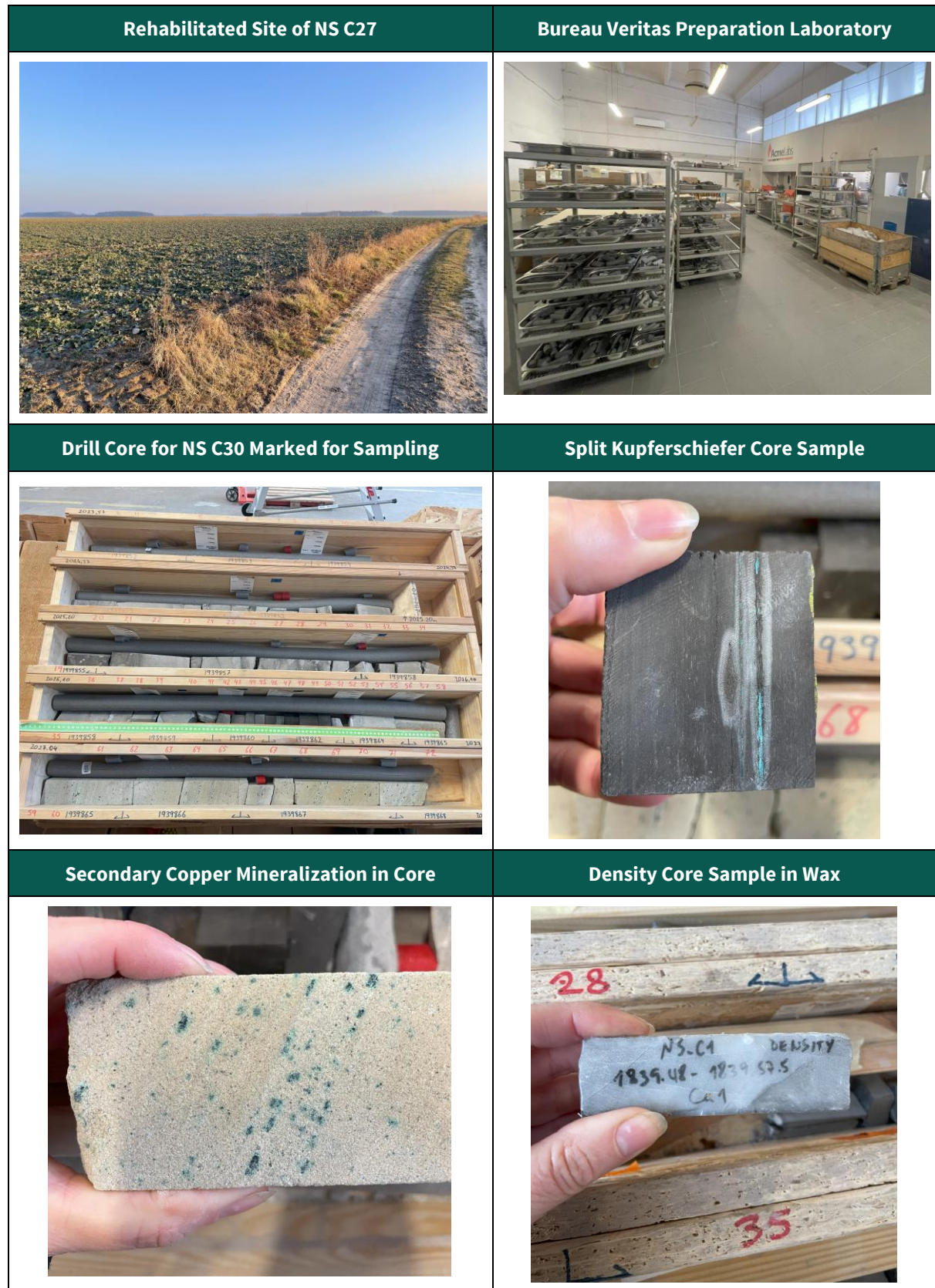
Meetings were held at Lumina's company headquarters in Warsaw with management personnel during both the February 2025 and July 2025 visits. A total of six days have been spent on site by Micon QPs.

A selection of verification photographs taken during the February 2025 site visit are shown in Figure 12.1.

12.2 DATABASE

Drill hole data was provided to Micon by Lumina. Micon has not conducted any drilling, collection of samples, or independent assaying of material from the Project. Micon has reviewed the methods used for logging, sampling, and assaying. This data has been checked and scrutinized by Micon both in person at Lumina's Core storage facility at Kraków to view the 2014 to 2024 core, at the Michałów core storage facility in Michałowice to review the historical core as well as digitally.

Figure 12.1: A Selection of Verification Photographs from the February 2025 Site Visit



Source: Micon, 2025.

The Project drill hole database is maintained on-site in a Microsoft Excel and csv files. Micon received the drill hole data from Lumina in this format. The data was collated into a single Microsoft Excel spreadsheet by the QPs and the database was validated and inspected to ensure that the data is acceptable and in a suitable condition for use in modelling and resource evaluation.

The QP used three validation methods: (1) spreadsheet checks, using both original hard copy and digital soft copy, (2) importation and visualisation in Leapfrog, and (3) plotting of graphs and generation of summary statistics with Snowden Supervisor or R software.

Spreadsheet checks included checking that all intervals recorded in the database tables have a corresponding entry in the collar table and that maximum depths are consistent between the tables. Leapfrog was primarily used to visualise the data in 3D and to check that the collar location and shape of the drill hole traces is logical. Leapfrog also has a database validation tool when importing the drill hole database that highlights any problems such as overlapping intervals or depth errors. Supervisor was used to produce histograms and statistics of sample lengths and recovery values.

Overall, Micon identified few problems with the database, and it was considered to be clean. Minor corrections were generally related to human typing errors or duplication of data and Micon was able to correct the data after inspecting the cause of the error. A clean version of the database tables, alongside the corrections, was supplied to Lumina to update their records on-site.

12.3 COMPARISON OF HISTORICAL AND LUMINA DATA

Only one historical drill hole within the Nowa Sól license, Bojadła 1, reached sufficient depth to intersect the mineralized interval of the Kupferschiefer horizon. Lumina drill hole Jany_C2 was drilled 1.5 km southeast of Bojadła 1 and allows the best comparison of historical and Lumina data. Downhole plots of the two drill holes are shown in Figure 12.2. The plots show comparable grades and grade profiles with mineralization in Jany_C2 being wider than Bojadła 1. This observation is consistent with mineralization thickening to the southeast. Overall, the drill holes show consistent results considering the drill hole spacing.

12.4 QP OPINION ON THE ADEQUACY OF THE DATA

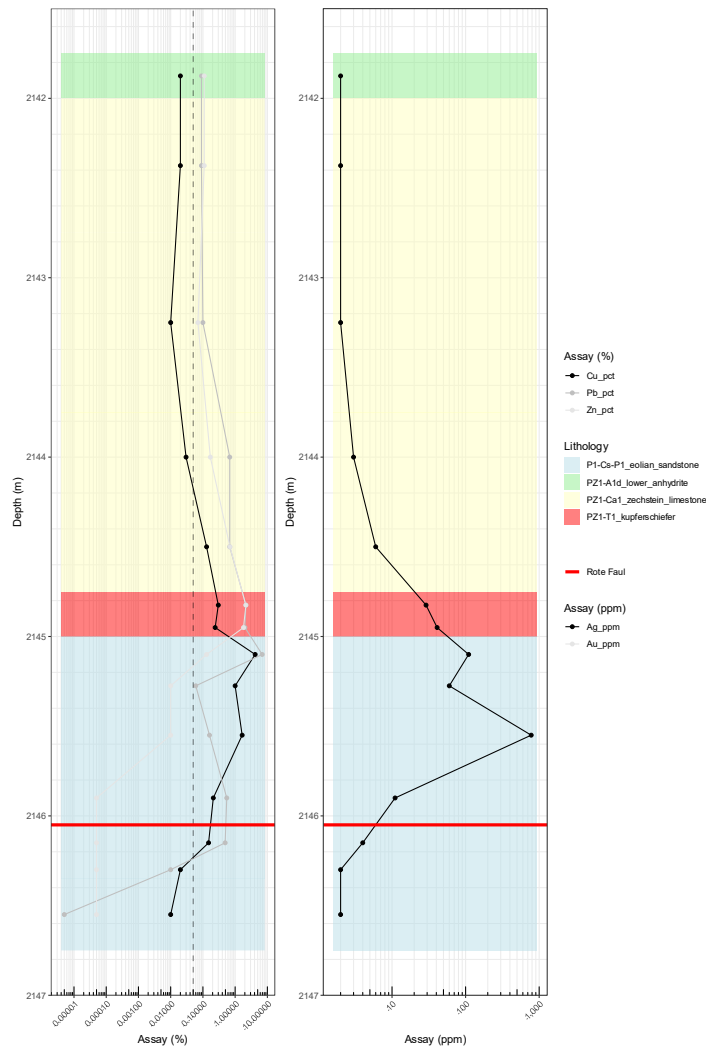
In conclusion, the QPs have reviewed the sample collection, analytical methodologies, QA/QC program, and drill hole database and they are to current industry standards and permit a meaningful investigation of the mineralization at Nowa Sól for the purpose of the Mineral Resource estimate under the CIM Guidelines (CIM, 2019). The database is also sufficiently reliable to be used as the basis for the conclusions and recommendations reached in this Report.

Independent sampling was not performed by Micon as there is very little original core remaining and pulps were used for metallurgical testwork, as described in Section 13. Polish law requires ¼ of all drill core must be submitted to the state for archiving plus additional samples of core had been sampled for other testwork such as mineralogy by Lumina's team and the University of Wrocław.

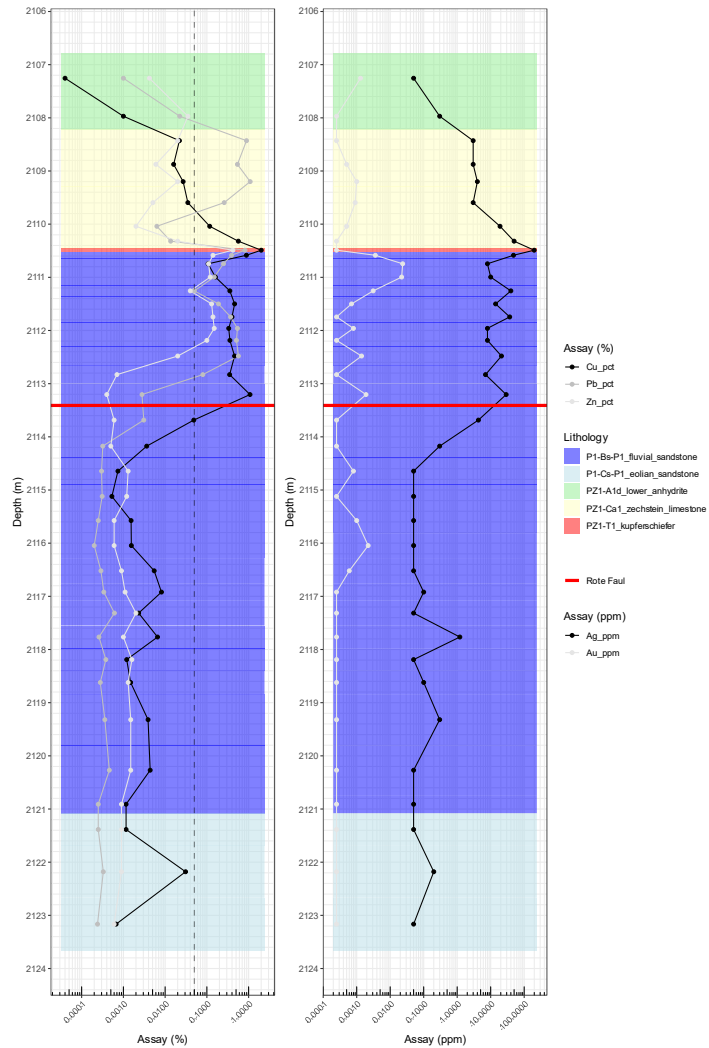
The QP is confident that the extremely well-preserved historical core, historical core logs, new drill core and assay data from independent laboratories is sufficient to confirm adequacy of the data.

Figure 12.2: Downhole Plots for Drill Holes Bojadła 1(Boj_1) and Jany_C2 located 1.5 km Apart

Hole ID. - Boj_1



Hole ID. - Jany_C2



Source: Micon, 2025.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

No historical metallurgical testwork has been completed using mineralized samples from the Nowa Sól Copper-Silver Project. However, KGHM has been mining and processing similar copper mineralization continuously since 1963.

In August 2025 XPS Industry Relevant Solutions (XPS) were engaged by Lumina to undertake a preliminary metallurgical testwork program using mineralized samples from Nowa Sól. The objective of the XPS testwork program was to develop a conceptual process flowsheet and provide preliminary process design criterion to support a PEA. The test program comprised mineralogical and chemical characterisation of the composite feed sample, standard Bond Ball Mill Work Index (BBWi) determination, preliminary flotation circuit development and characterisation studies of final tailings and concentrate samples generated from the concluding confirmatory tests.

In addition to the XPS testwork, Lumina employed Wrocław University of Science and Technology (Wrocław Tech) in Poland to undertake a similar program of work which included mineralogical characterisation studies and preliminary flowsheet development testwork. Wrocław Tech is considered an expert in the beneficiation of copper mineralization originating from the Legnica-Głogów Copper District, Poland, and has considerable experience with the metallurgical development and optimization of Nowa Sól type deposits.

The QP for this section of the Report is Richard Gowans P.Eng., the Principal Metallurgist of Micon International Limited. The QP reviewed the selection of the metallurgical samples and managed of work undertaken by XPS. The QP also reviewed the testwork results and report prepared by Wrocław Tech.

13.2 METALLURGICAL SAMPLES

Three metallurgical composite samples were selected by Lumina geologists and prepared by technical staff from Wrocław Tech. The three composites received by XPS comprised approximately 5.7 kg of mineralized crushed assay rejects, 10.1 kg of mineralized split core, and 6.3 kg of low-grade mineralized core for grindability testing. Approximately 50% of the original composites were retained by Wrocław Tech for its independent parallel testwork program.

The individual samples selected to make-up the composites were inspected by experienced Lumina geologists to ensure minimal weathering with tainted samples being rejected. The intention was to prepare composites from samples produced from the most recent drilling campaigns that represented typical mineralization from the Nowa Sól deposit. The lists of individual samples selected for the three composites used for the XPS metallurgical testing are included in Table 13.1, Table 13.2 and Table 13.3.

The crushed assay rejects composite sample was kept separate from the split core composite and was to be used for initial flowsheet scoping tests. The split core composite was to be used for final development and confirmatory tests. The grindability sample was to be used to obtain a BBWi.

Table 13.1: Metallurgical Composite 1 – Crushed Assay Rejects

Drill Hole	Lithology	Code	From (m)	To (m)	Cu (%)	Ag (g/t Ag)
NS C25	Ca1	1939838	1,940.9	1,941.2	0.82	123
	T1	1939840	1,941.2	1,941.4	9.02	455
	Bs/P1	1939842	1,941.4	1,941.6	3.86	56
	Bs/P1	1939843	1,941.6	1,941.8	1.20	2
NS C15	Ca1	1939884	1,985.2	1,985.5	0.85	201
	Ca1	1939885	1,985.5	1,985.7	3.64	143
	T1	1939887	1,985.7	1,985.9	15.82	238
NS C30	Ca1	1939859	2,026.3	2,026.5	5.37	211
	T1	1939862	2,026.6	2,026.8	2.33	328
	Bs/P1	1939864	2,026.8	2,027.0	0.85	10
	Bs/P1	1939865	2,027.0	2,027.2	0.88	8
	Bs/P1	1939866	2,027.2	2,027.5	0.83	10
	Bs/P1	1939867	2,027.5	2,027.8	1.21	10
	Bs/P1	1939868	2,027.8	2,028.3	1.43	14
	Bs/P1	1939869	2,028.3	2,028.8	0.79	6
NS C29	Ca1	1939937	2,092.8	2,093.3	1.01	77
	T1	1939939	2,093.3	2,093.4	16.52	439
	Bs/P1	1939941	2,093.4	2,093.7	3.02	23
	Bs/P1	1939942	2,093.7	2,094.4	0.78	57
	Bs/P1	1939943	2,094.4	2,095.0	0.81	2
NS C27	Bs	1939928	2,117.5	2,117.8	0.97	95
	Bs	1939931	2,118.1	2,118.8	1.22	44
NS C27 ST	T1	1939909	2,117.0	2,117.1	2.63	107
	Bs	1939911	2,117.1	2,117.5	0.95	267
	Bs	1939914	2,118.0	2,118.3	1.12	20
	Bs	1939915	2,118.3	2,118.7	0.75	14
	Bs	1939916	2,118.7	2,119.0	0.89	15
	Bs	1939917	2,119.0	2,119.6	0.82	16
Lithology		From (m)	To (m)	Proportion (%)	Cu (%)	Ag (g/t Ag)
Zechstein Limestone	Ca1	2,058.7	2,059.1	21%	1.99	143
Eolian Sandstone	Bs/P1	2,029.9	2,030.3	49%	1.22	19
Fluvial Sandstone	Bs	2,118.1	2,118.5	22%	1.03	84
Kupferschiefer Shale	T1	2,032.8	2,032.9	8%	8.78	327
Average				100%	1.91	83

Table 13.2: Metallurgical Composite 2 – Split Mineralized Core

Drill Hole	Lithology	Code	From (m)	To (m)	Cu (%)	Ag (g/t Ag)
NS C25	Ca1	1939838	1,940.9	1,941.2	0.82	123
	T1	1939840	1,941.2	1,941.4	9.02	455
	Bs/P1	1939842	1,941.4	1,941.6	3.86	56
	Bs/P1	1939843	1,941.6	1,941.8	1.20	2
NS C27 ST	T1	1939909	2,117.0	2,117.1	2.63	107
	Bs	1939911	2,117.1	2,117.5	0.95	267
	Bs	1939914	2,118.0	2,118.3	1.12	20
	Bs	1939915	2,118.3	2,118.7	0.75	14
	Bs	1939916	2,118.7	2,119.0	0.89	15
NS C27	Bs	1939928	2,117.5	2,117.8	0.97	95
	Bs	1939931	2,118.1	2,118.8	1.22	44
NS C15	Ca1	1939884	1,985.2	1,985.5	0.85	201
	Ca1	1939885	1,985.5	1,985.7	3.64	143
	T1	1939887	1,985.7	1,985.9	15.82	238
NS C29	Ca1	1939937	2,092.8	2,093.3	1.01	77
	T1	1939939	2,093.3	2,093.4	16.52	439
	Bs/P1	1939941	2,093.4	2,093.7	3.02	23
	Bs/P1	1939942	2,093.7	2,094.4	0.78	57
	Bs/P1	1939943	2,094.4	2,095.0	0.81	2
	Bs/P1	1939944	2,095.0	2,095.8	0.61	2
NS C30	Ca1	1939859	2,026.3	2,026.5	5.37	211
	T1	1939860	2,026.5	2,026.6	5.09	393
	T1	1939862	2,026.6	2,026.8	2.33	328
	Bs/P1	1939864	2,026.8	2,027.0	0.85	10
	Bs/P1	1939865	2,027.0	2,027.2	0.88	8
	Bs/P1	1939866	2,027.2	2,027.5	0.83	10
	Bs/P1	1939867	2,027.5	2,027.8	1.21	10
	Bs/P1	1939868	2,027.8	2,028.3	1.43	14
	Bs/P1	1939869	2,028.3	2,028.8	0.79	6
Lithology		From (m)	To (m)	Proportion (%)	Cu (%)	Ag (g/t Ag)
Zechstein Limestone	Ca1	2,070.4	2,070.7	18%	1.85	139
Eolian Sandstone	Bs/P1	2,077.7	2,078.1	56%	1.17	16
Fluvial Sandstone	Bs	2,043.5	2,043.8	17%	1.04	66
Kupferschiefer Shale	T1	2,057.7	2,058.0	10%	8.33	355
Average				100%	1.95	79

Table 13.3: Metallurgical Composite 3 – Grindability Test Sample

Drill Hole	Lithology	Code	From (m)	To (m)	Cu (%)
NS C15	Bs/P1	1939889	1,985.9	1,986.2	0.07
NS C27	Ca1	1939922 / 1939924	2,116.2	2,117.1	0.00
	Bs	1939930	2,117.9	2,118.1	0.12
NS C29	Ca1	1939935	2,092.3	2,092.8	0.13
	Bs/P1	1939945	2,095.8	2,096.2	0.23
	Bs/P1	1939947	2,096.2	2,096.5	0.00
NS C30	Ca1	1939858	2,025.9	2,026.3	0.09
Lithology		From (m)	To (m)	Proportion (%)	Cu (%)
Zechstein Limestone	Ca1	2,078.1	2,078.7	32%	0.10
Eolian Sandstone	Bs/P1	2,059.3	2,059.6	30%	0.12
Fluvial Sandstone	Bs	2,117.9	2,118.1	38%	0.12
Kupferschiefer Shale	T1	-	-	0%	-
Average				100%	0.11

13.2.1 Sample Characterization

The crushed assay rejects samples and split core samples received by XPS were prepared separately. They were crushed to 6 mesh (-3.36 mm), homogenized, assayed in triplicate and prepared into 1 kg test charges. The Bond Work Index sample was shipped to SGS-Lakefield for BBWi testing. The average triplicate analyses for the two mineralized composites are presented in Table 13.4. Since the two composite samples were mainly prepared from the same original drill hole sample intervals, the comparative analyses were similar.

Table 13.4: Metallurgical Composite Chemical Analyses

Analyte Symbol	Unit	Detection Limit	Composite 2 Split Core	Composite 1 Assay Rejects
Cu	%	0.005	1.96	1.86
Ag	ppm	0.2	92.2	84.7
Au	ppb	2	8.33	<30
Pd	ppb	5	< 5	-
Pt	ppb	5	< 5	-
C-Total	%	0.01	3.38	3.46
C-Organic	%	0.02	1.02	0.9
Al	%	0.01	2.59	2.57
As	%	0.01	0.01	<0.01
Ca	%	0.01	5.72	5.85
Fe	%	0.05	1.11	1.09
Pb	%	0.01	0.32	0.35
S	%	0.01	1.10	1.07
Si	%	0.01	30.0	30.8
Zn	%	0.01	0.04	0.04

As part of the metallurgical program, XPS completed a mineralogical characterisation study of the split core sample at a target grind of 80% passing (P_{80}) 100 μm (actual 99 μm). The program included the following considerations and methodologies:

- Quantitative mineralogical analysis, Energy Dispersive X-ray Spectroscopy (EDS) - TIMA/QEMSCAN (UNSIZED).
- Modal mineralogy and particle imaging including all sulphides and gangue.
- Detailed Cu deportment by mineral phase carrier (with Electron Probe Microanalysis (EPMA) on feed).
- Assessment of Ag mineralogy and discrete Ag phases present as measured.
- Deleterious element assessment and deportment (As) by phase (with EPMA).
- Particle sizes of phases of interest (average (avg) grain size and size distributions).
- Assessment of textures, liberation, locking and associations on an 'unsized' basis.
- Powder XRD - supporting phase identification of crystalline phases and identification of potential carbonaceous phases and other gangue minerals.
- EPMA - Phase chemistry data. ppm level quantification of elements by mineral phase in order to detail full deportment of Cu and Ag (plus deleterious elements where possible).

13.2.1.1 Modal Mineralogy

The modal mineralogy of the sample is presented in Table 13.5. As expected, the sample comprises mainly of quartz and dolomite corresponding to the two main lithologies, sandstone and carbonate. The main sulphide minerals identified are chalcocite (1.99%), galena (0.43%), pyrite (0.37%), covellite (0.31%), chalcopyrite (0.27%), sphalerite (0.10%) and bornite (0.09%).

Table 13.5: Metallurgical Composite Modal Mineralogy

Modal Mineralogy	Mass (%)
Quartz	61.96
Dolomite	12.63
Mica	7.88
Calcite	4.65
Orthoclase	4.09
Albite	2.96
Chalcocite	1.99
Kaolinite	0.80
Anhydrite	0.66
Galena	0.43
Pyrite	0.37
Covellite	0.31
Chalcopyrite	0.27
Sphalerite	0.10
Bornite	0.09
FeCr Oxides	0.08
Ag-Rich Phase	0.02
Tennantite-Tetrahedrite	0.01
Other	0.68
Total	100.00
<i>Total Cu Sulphides</i>	<i>2.66</i>
<i>Total Other Sulphides</i>	<i>0.92</i>
<i>Total Non-Sulphide Gangue (NSG)</i>	<i>96.40</i>

The modal distribution of copper minerals is shown in Table 13.6 which suggests that copper is mainly found as chalcocite (79.6%) and covellite (10.7%).

Table 13.6: Metallurgical Composite Copper Mineral Department

Copper Mineral	Proportion Weight (%)
Chalcocite	79.57
Covellite	10.71
Chalcopyrite	5.03
Bornite	2.81
Other Sulphides	0.59
Silicates	0.74
Carbonates	0.57
Total	100.00

The elemental composition of the significant sulphides found in the sample was estimated using EPMA. The results are summarized in Table 13.7. The measurements of bismuth and mercury in the minerals analyses were negligible.

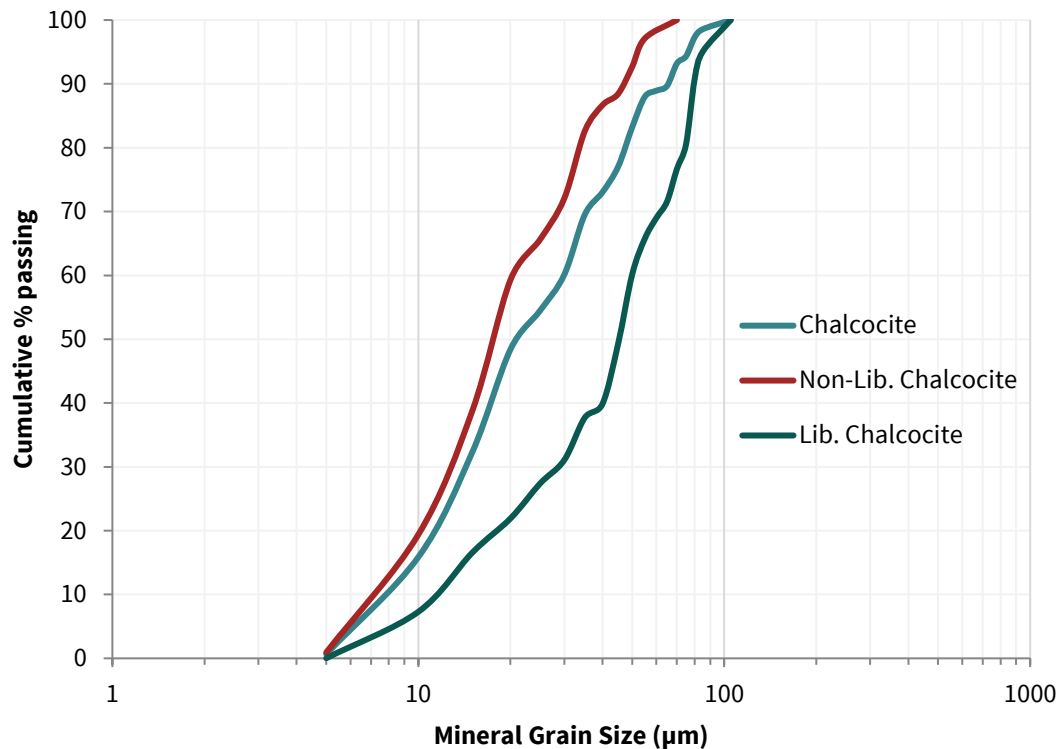
Table 13.7: Preliminary Electron Probe Microanalysis of Sulphides

Mineral	S (%)	Fe (%)	Cu (%)	Zn (%)	As (%)	Ag (%)	Pb (%)	Total
Chalcocite (no Ag)	21.76	0.03	77.94	0.06	0.00	0.02	0.00	99.76
Chalcocite (Ag)	20.87	0.05	78.01	0.08	0.01	0.17	0.00	99.15
Pyrite	52.54	44.98	1.77	0.00	0.19	0.04	0.00	99.46
Covellite	30.78	0.05	68.25	0.06	0.02	0.06	0.00	99.16
CuFe-Sulphide	37.70	22.63	40.51	0.04	0.14	0.04	0.00	101.02
Sphalerite	32.72	0.06	0.20	66.43	0.00	0.00	0.00	99.35
Galena	13.40	0.04	0.10	-0.01	0.00	0.00	84.34	97.79

13.2.1.2 Grain Size Distributions

The estimated grain size distributions for chalcocite (all grains, liberated (Lib.) grains and non-liberated grains) are illustrated in Figure 13.1. The measured P_{80} size for the chalcocite in the feed is 47 μm , liberated grains P_{80} 74 μm and non-liberated grains P_{80} 34 μm .

Figure 13.1: Estimated Grain Size Distribution - Chalcocite



Source: XPS Final Report, January 2026.

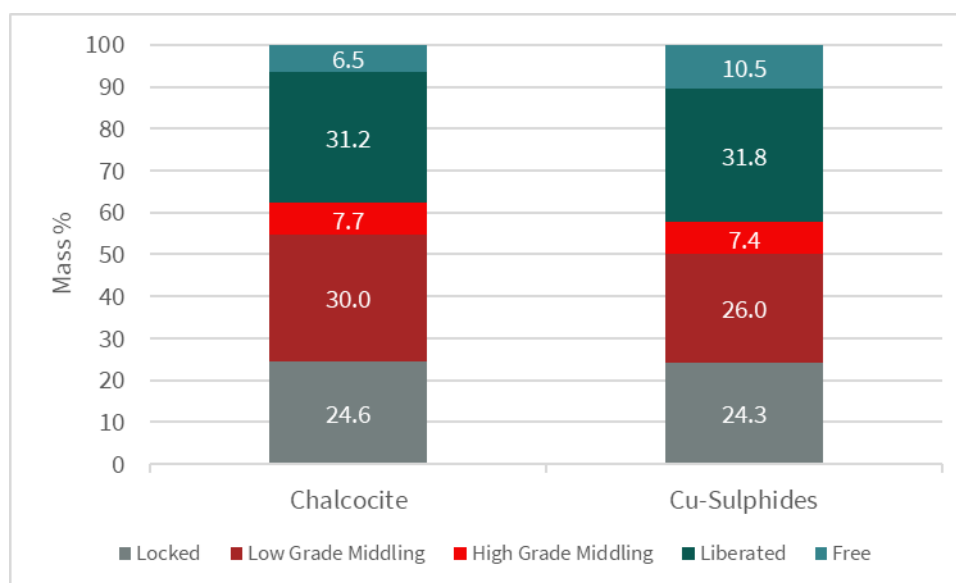
13.2.1.3 Liberation Characteristics

The liberation characteristics of all the copper minerals and the chalcocite mineral alone are illustrated in Figure 13.2. The liberation criteria are defined by XPS in Table 13.8.

Table 13.8: Liberation Criteria Defined by XPS

Liberation Category	Criteria
Free	100% of particle by area is mineral of interest
Liberated	>90% of particle by area is mineral of interest
High Grade Middling	80% - 90% of particle by area is mineral of interest
Low Grade Middling	30% - 80% of particle by area is mineral of interest
Locked	<30% of particle by area is mineral of interest

Figure 13.2: Copper Minerals Liberation



Source: XPS Final Report, January 2026.

The liberation data suggests that up to approximately 75% of the copper mineralization is recoverable at a P₈₀ grind size of 99 µm.

The estimated copper sulphide associations and corresponding average grain size are presented in Table 13.9.

Table 13.9: Copper Sulphide and Chalcocite Mineral Associations

Association	Copper Sulphides		Chalcocite	
	Weight (%)	Grain Size (mm)	Weight (%)	Grain Size (mm)
Liberated	42.36	20	37.71	25
Binary - Cu-S	0.00	0	5.27	13
Binary - Ag-Rich Phase	0.02	5	0.01	8
Binary - Other Sulphides	2.82	13	0.07	8
Binary - Silicates	32.40	12	30.41	13
Binary - Carbonates	4.26	12	3.63	15
Binary - Other NSG	0.07	18	0.03	11
Complex	18.07	10	22.86	11
Total	100.00	-	100.00	-

The preliminary mineralogical study on the composite sample of Nowa Sól mineralization suggests the following:

- The primary Cu-bearing mineral is chalcocite, with an abundance of 1.99%, which accounts for 80% of the contained copper.
- The next prevalent copper minerals are covellite and chalcopyrite that contain 11% and 5% of the copper, respectively.
- Chalcocite is 38% liberated, 5% is associated with Cu sulphides, 30% with silicates, and 23% with two or more minerals forming complex particles.
- The estimated average grain size of non-liberated chalcocite ranges from 21 µm to 9 µm.

13.2.2 QP Comments

The composite samples prepared for the metallurgical testwork program are considered representative of the typical mineralization found at Nowa Sól. Although the samples were not freshly drilled there was no obvious signs of weathering and despite the average copper grade of the composites being significantly higher than the average mineral resource grade, the QP believes that the samples were suitable to be used for a preliminary testwork program that could support a PEA level of techno-economic study.

13.3 METALLURGICAL TESTWORK

Composite 3, comprising of low-grade mineralization, was used to generate a BBWi.

Composite 1, which was prepared from individual crushed assay reject samples, was used for the initial scoping tests while Composite 2, made from split drill core samples, was used for the final development and confirmatory tests. The tailings and final concentrates from the confirmatory tests were used for the final characterisation studies.

13.3.1 Comminution Tests

A single standard BBWi test using a 150 mesh of grind was performed by SGS at the Lakefield facility. The BBWi derived was 15.6 kWh/t with a F_{80} of 2,477 μm (100% passing 6 mesh, 3,360 μm) and a P_{80} of 81 μm (100% passing 150 mesh, 106 μm).

13.3.2 Flotation Tests

The metallurgical testwork undertaken by XPS comprised a series of flowsheet development tests based on conventional single product flotation. Only open circuit bench scale tests were completed by XPS, there was insufficient sample available to undertake locked cycle tests.

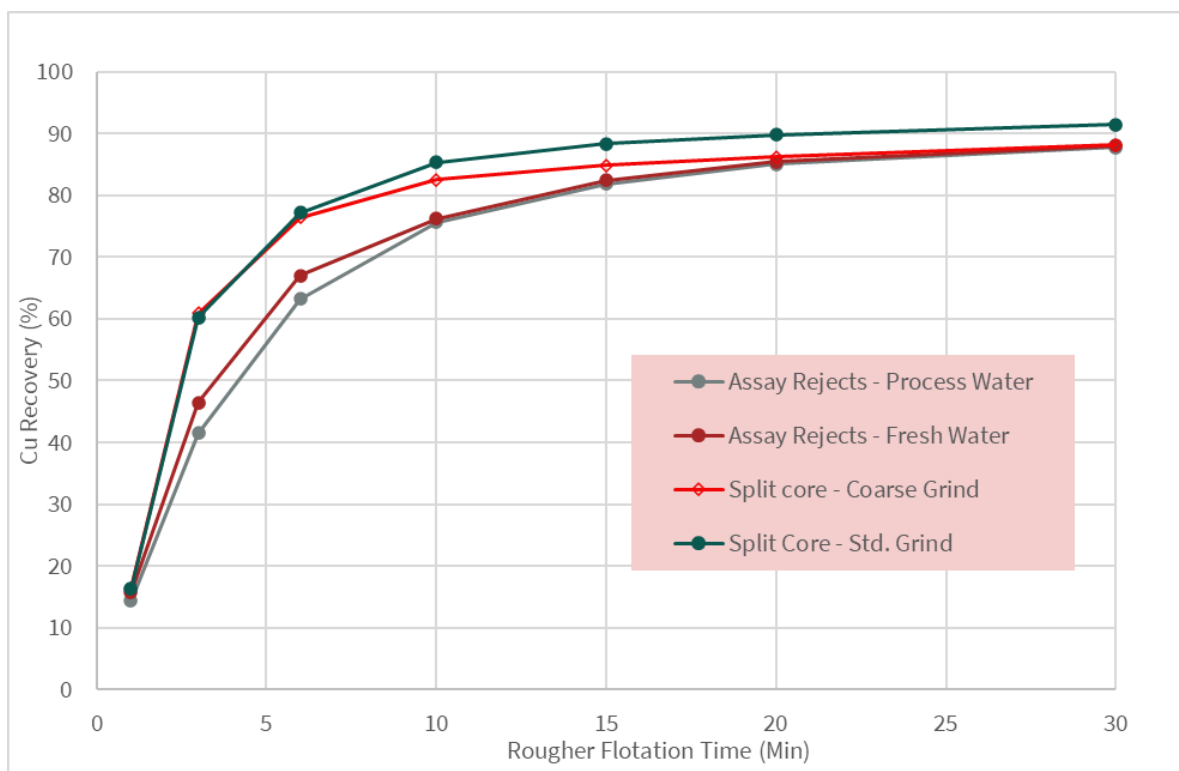
The flotation testing program comprises a series of development rougher kinetic tests followed by flowsheet optimization tests which considered concentrate regrinding and cleaning. Once the flowsheet was selected a series of four confirmatory duplicate tests were completed. The products from the confirmatory tests were submitted for detailed chemical and mineralogical analyses and the overall metallurgical balance from these tests was used as a basis to estimate metallurgical recoveries.

13.3.2.1 Flotation Development Testing

The initial rougher development tests comprised five kinetic rougher tests using Composite 1 and Composite 2 that investigated collector dosage, flotation residence time, primary grind, pH and the effect of synthetic process water verses tap water. The synthetic saline process water was prepared by XPS containing approximately 130 mg/L Mg, 570 mg/L Ca, 2,670 mg/L Na, 960 mg/L SO_4 and 4,620 mg/L Cl.

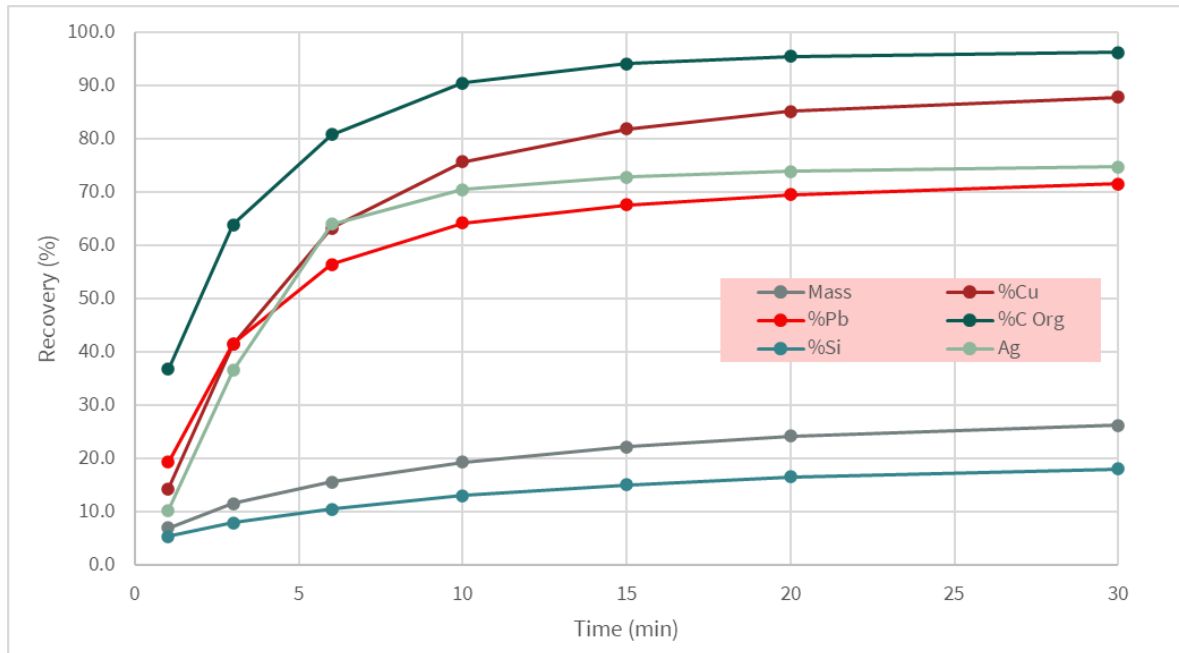
The results from these preliminary tests showing the copper rougher recovery kinetics are presented in Figure 13.3. Figure 13.4 shows the mass and metal recoveries for flotation tests F002 which used Composite 1 and process water.

Figure 13.3: Copper Rougher Flotation Kinetics



Source: XPS Final Testwork Report, January 2026.

Figure 13.4: Flotation Kinetic Test Results (F002)



Source: XPS Final Testwork Report, January 2026.

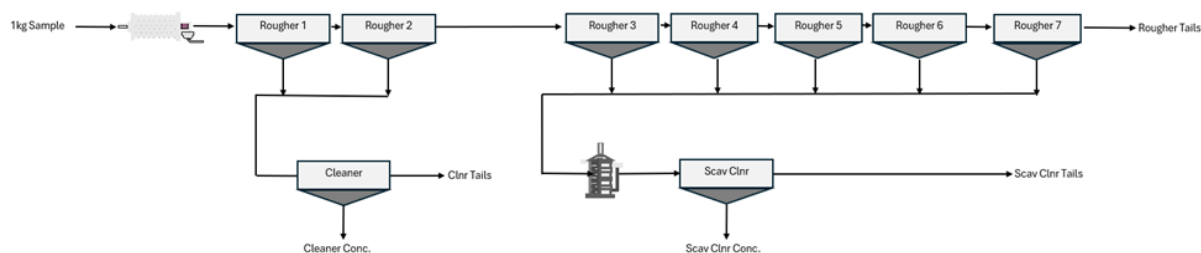
These initial scoping tests suggested the following:

- Fresh water and process water tests provided similar results.
- Composite 2, the spit core sample, provided faster Cu kinetics than Composite 1, crushed assay reject sample.
- The D80 60 µm grind using Composite 2 provided the lowest final tails assay and highest total recovery for Cu and Ag.
- There was very little selectivity between Cu and Pb in all tests.
- Organic carbon appeared to have the fastest kinetics, followed by Cu, Pb and Ag.
- pH adjustment to pH 9.0 using lime appeared to improve the froth stability.

13.3.2.2 Preliminary Flotation Flowsheet Optimization

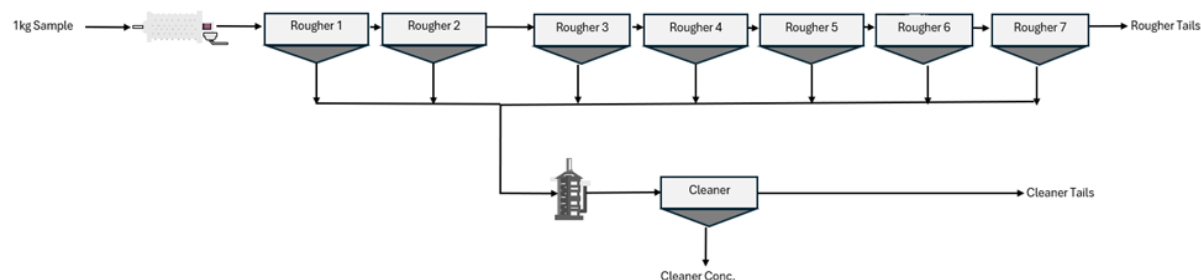
Two conceptual comparative tests were conducted using Composite 1, primary grind of P₈₀ 80 µm, regrind P₈₀ size of 20 µm, process water, pH 9.0 and single cleaner to test rougher and scavenger concentrate regrinding. The two flowsheets tested are illustrated in Figure 13.5 and Figure 13.6 (XPS, 2026).

Figure 13.5: Flotation Flowsheet (Test F005) – Scavenger Cleaner Regrind



Source: XPS Final Testwork Report, January 2026.

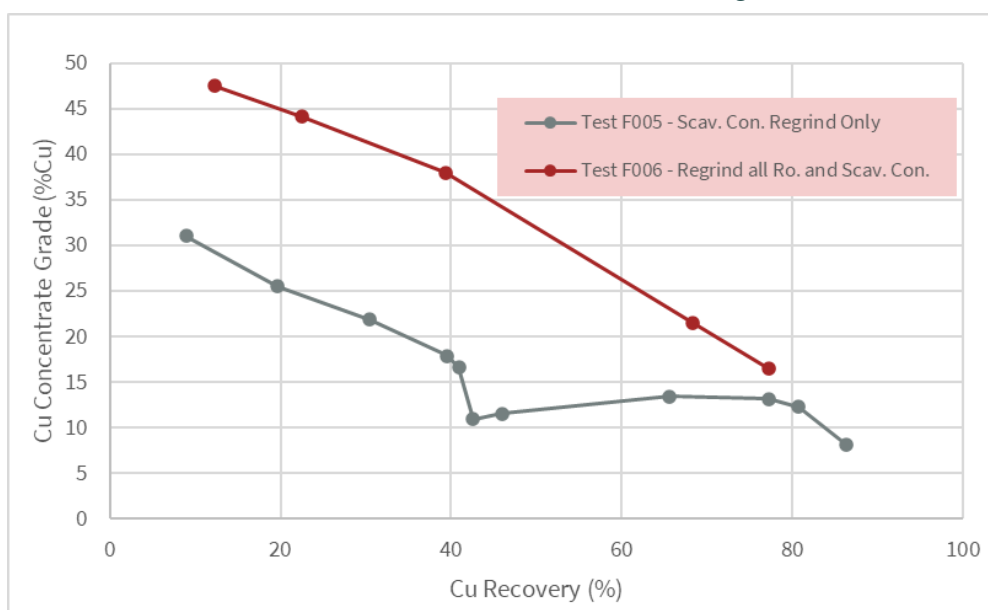
Figure 13.6: Flotation Flowsheet (Test F006) – Rougher Cleaner Regrind



Source: XPS Final Testwork Report, January 2026.

The results from these two tests suggest that regrinding the whole rougher concentrate (Test 006) is a better option than by-passing the initial rougher concentrate (Test 005). The results for these two tests are presented in Figure 13.7.

Figure 13.7: Copper Concentrate Grade versus Recovery for the Two Regrind Flowsheets Tests

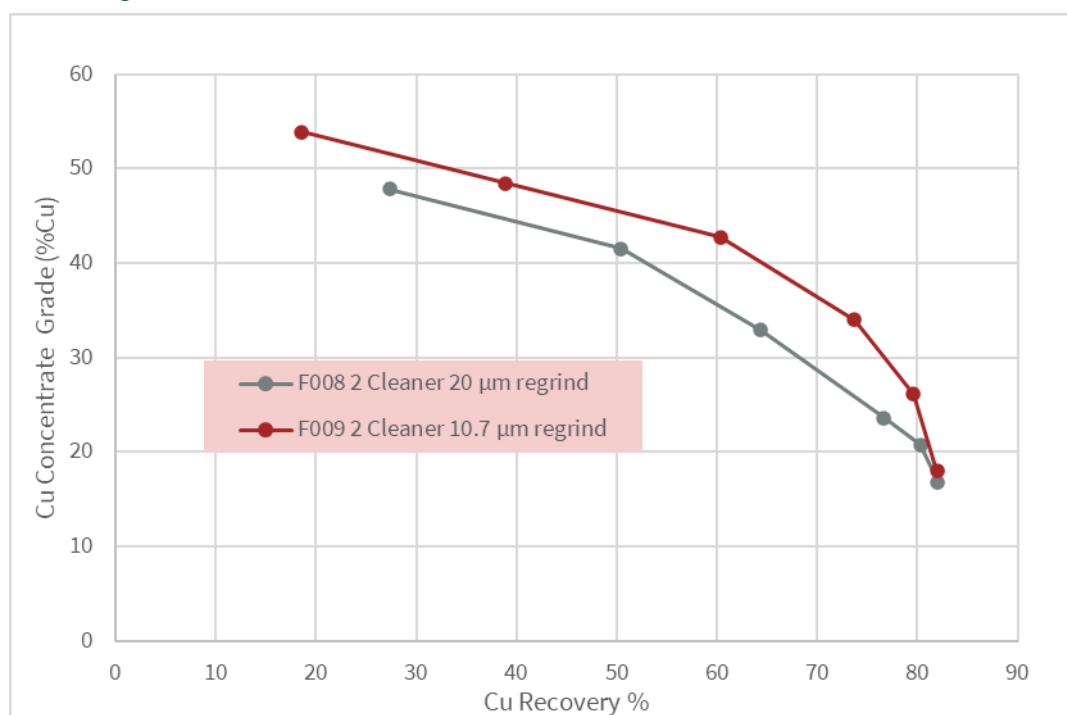


Source: XPS Final Testwork Report, January 2026.

Note: Ro = Rougher; Scav. = Scavenger; Con. = Concentrate.

Regrind target size was tested using Composite 2 and the flowsheet presented in Figure 13.6, but with a finer primary P_{80} grind of 60 μm and two stages of cleaning. The results are shown in Figure 13.8 which suggest that a lower regrind size of around 10 μm produced higher initial concentrate grades and an improved grade recovery curve.

Figure 13.8: Regrind Size - Cleaner 2 Concentrate Copper Grade versus Recovery



Source: XPS Final Testwork Report, January 2026.

The best flotation conditions demonstrated during the preliminary development and optimization testing program completed by XPS comprise a primary P_{80} grind of 60 μm in process water, rougher

regrind flowsheet with a regrind P_{80} of 10.7 μm and 2 stages of cleaning (Test F009). This test provided a Cleaner 2 Concentrate copper grade of 26.1% Cu with 79.6% and 87.2% Cu and Ag recoveries, respectively.

13.3.2.3 Confirmation Flotation Tests

Four identical floatation tests were conducted duplicating the best conditions generated during the development phase of the preliminary program (Test F009). For these four tests the Cleaner 2 Concentrate increment masses were measured separately and were then combined into a single sample for all assay purposes. All products from the four tests were sub-sampled for assay. Individual samples from the Cleaner 2 Concentrate, Cleaner 1 Tailings, and Rougher Tailings were combined into composite samples and submitted for more detailed characterisation studies.

The quadruplicate float test results were very consistent for mass and Cu recovery. Cu recovery to Cleaner 2 Concentrate varied from 78.9% to 81.1% while the concentrate grade varied from 25.3% Cu to 26.3% Cu. The silver results were more variable with the Ag recovery to Cleaner 2 Concentrate ranging from 75.6% to 92.0% across the four tests.

A mass balance for the confirmation work was constructed using the sum of product masses from the four tests, assays from three composite samples (Cleaner 2 Concentrate, Cleaner 1 Tailings, and Rougher Tailings), and a calculated average assay for the fourth product, Cleaner 2 Tailings. The final metal balance is summarized in Table 13.10.

Table 13.10: Combined Confirmation Flotation Tests - Metallurgical Balance

Product	Mass	Wt. (%)	Grade			Recovery		
			Cu (%)	Pb (%)	Ag (g/t Ag)	Cu (%)	Pb (%)	Ag (%)
Cleaner 2 Concentrate	230.8	5.9	26.2	3.4	1,330	79.5	67.3	82.5
Cleaner 1 Tailings	82.7	2.1	1.70	0.40	150	1.8	2.8	3.3
Cleaner 2 Tailings	603.6	15.4	0.92	0.28	63.0	7.3	14.4	10.2
Total Cleaner Tailings	686.3	17.5	1.01	0.29	73.5	9.1	17.2	13.5
Rougher Tailings	3,010.1	76.6	0.29	0.06	4.8	11.4	15.3	3.9
Calculated Heads	3,927.2	100.0	1.94	0.30	95.0	100.0	100.0	100.0
Combined Products								
Cleaner 2 Concentrate	230.8	5.9	26.2	3.4	1,330	79.5	67.3	82.3
Cleaner 1 Concentrate	313.5	8.0	19.7	2.6	1,019	81.3	70.3	85.9
Rougher Concentrate	917.1	23.4	7.3	1.1	390	88.6	84.7	96.1

Source: XPS Final Testwork Report, January 2026.

Confirmation open circuit testing results showed a 79.5% copper recovery and 82.5% silver recovery into a concentrate containing 26.2% Cu. The copper concentrate also contained 3.4% lead, 11.3% sulphur, 7.6% silicon, 18.3% total carbon and 17.6% organic carbon.

13.3.3 Flotation Product Characterization Studies

The Cleaner 2 Concentrate, Cleaner 1 Tailings, and rougher tailings from the confirmation flotation tests were combined into composite samples and submitted for more detailed characterisation studies.

13.3.3.1 Rougher Tailings

Modal mineralogy of the Rougher Tailings is presented in Figure 13.9 and is similar to the feed. The main components are quartz (69%), dolomite (11%), orthoclase (5%) and calcite(5%).

The measured P_{80} size of the Rougher Tailings sample was 78 μm .

Among copper-bearing minerals that make up the 0.29% copper in the Rougher Tailings, chalcocite at 0.26% of the sample mass carries 80% of the copper, followed by chalcopyrite at 0.07% (10% of the copper) and bornite 0.01% (1.4% of the copper).

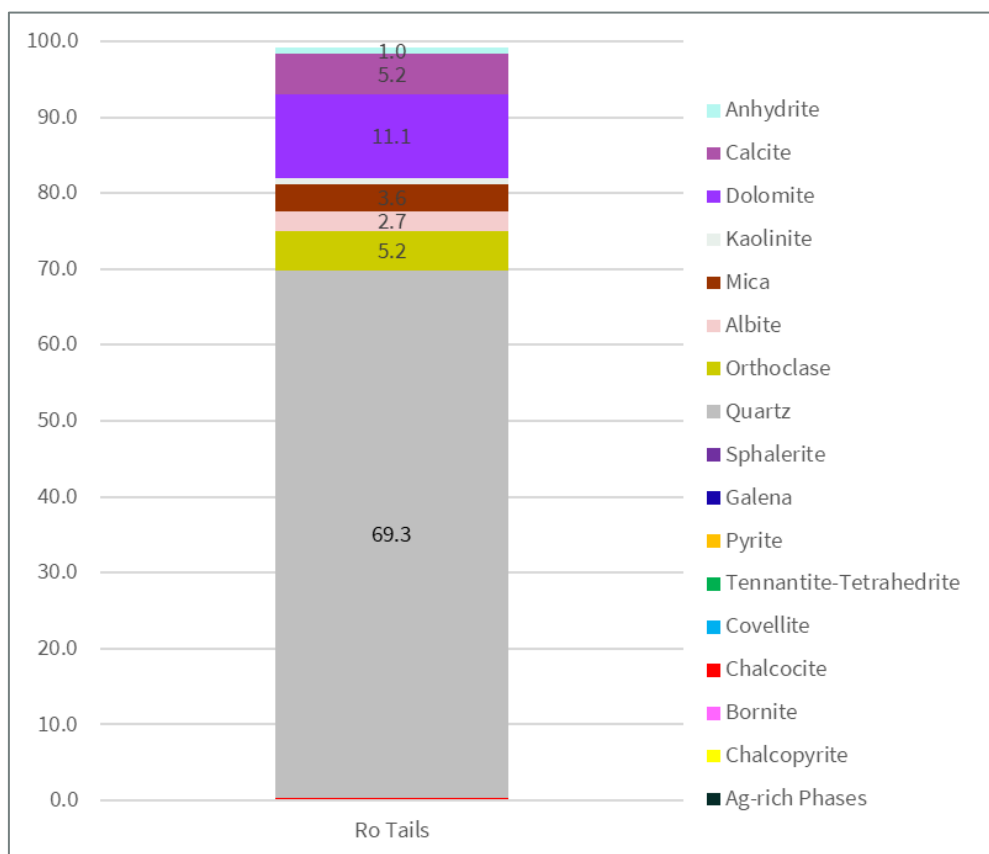
The liberation characteristics of the copper sulphide minerals and chalcocite in the Rougher Tailings is shown in Table 13.11. This table suggests that only approximately 8% of the copper in the tailings could be recoverable. The measured P_{80} size of the copper sulphide minerals and chalcocite are 14 μm and 13 μm , respectively.

Table 13.11: Liberation of Copper Sulphide Minerals and Chalcocite in Rougher Tailings

Liberation	Copper Sulphides		Chalcocite	
	Distribution (Wt.%)	Grain Size (μm)	Distribution (Wt.%)	Grain Size (μm)
Locked	73.22	6	80.58	6
Low Grade Middling	18.79	9	11.33	8
High Grade Middling	1.03	7	1.09	7
Liberated	0.00	0	0.00	0
Free	6.95	10	7.01	10
Total	100.00	-	100.00	-

Source: XPS Final Testwork Report, January 2026.

Figure 13.9: Rougher Tailings Modal Mineralogy (% by Mass)



Source: XPS Final Testwork Report, January 2026.

The acid base accounting analysis was undertaken on the Rougher Tailings composite sample using an enhanced Sobek package. The Acid Production Potential (AP) was measured as 11.3 kg CaCO_3/t while the Neutralizing Potential (NP) was measured as 299 kg CaCO_3/t , hence the net NP was 287 kg CaCO_3/t . These results are summarized in Table 13.12.

Table 13.12: Acid Base Accounting of Rougher Tailings Composite Sample

Fizz Rating	Paste pH	AP-Tot.S (kg CO ₃ /t)	NP (kg CO ₃ /t)	NNP (kg CO ₃ /t)	MPA (kg H ₂ SO ₄ /t)	NPR Ratio	Total S (S%)	Sulphide S (S%)
Moderate	8.11	11.3	299	287	11.1	26.4	0.36	0.03

Source: XPS Final Testwork Report, January 2026.

Notes: Neutralizing Potential = NP; Net Neutralizing Potential = NNP; Neutralization Potential Ratio.

13.3.3.2 Cleaner 2 Concentrate and Cleaner 1 Tailings

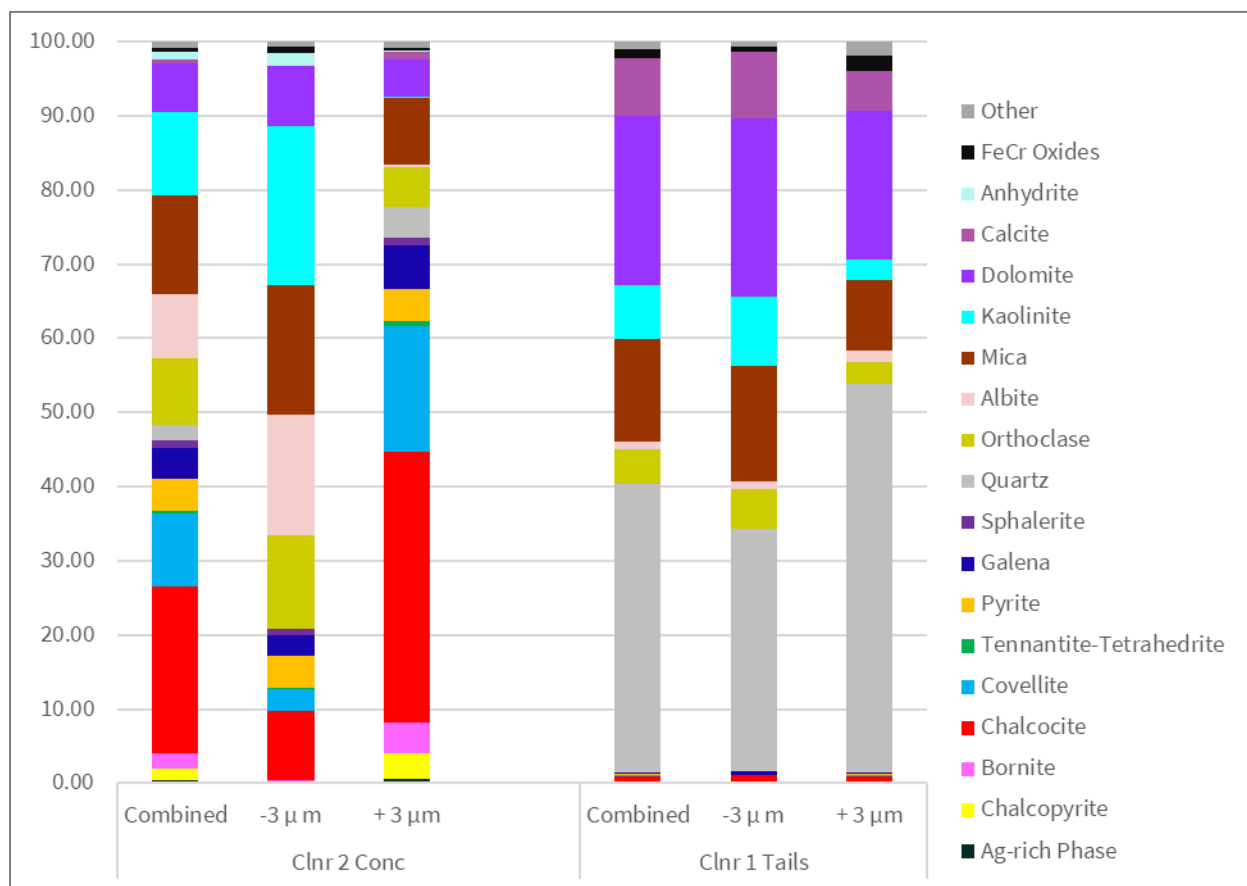
The cleaner samples submitted for mineralogy were de-slimed to r = NPRemove -3 µm particles with the +3 µm submitted for QEMSCAN analyses. Both the +3 µm and -3 µm fractions were assayed for copper for quality verification. The mass distributions and copper analyses are presented in Table 13.13. The -3 µm fraction QEMSCAN analysis is based on XRD results.

Table 13.13: Mass Distributions and Copper Analyses of the Cleaner Product Samples

Description	Cleaner 2 Concentrate			Cleaner 1 Tailings		
	Combined	-3 µm	+3 µm	Combined	-3 µm	+3 µm
Mass Distribution (%)	100.0	51.8	48.2	100.0	68.9	31.1
QEMSCAN/XRD Analysis (%Cu)	26.15	9.57	43.98	0.87	0.92	0.77
Chemical Analysis (%Cu)	24.35	9.52	40.30	0.88	0.93	0.76

Source: XPS Final Testwork Report, January 2026.

Figure 13.10: Cleaner Concentrate and Tailings Modal Mineralogy (% by Mass)



Source: XPS Final Testwork Report, January 2026.

The prominent minerals in the Cleaner 1 Tailings sample +3 μm fraction are quartz (53%), dolomite (20%), mica (10%) and calcite (5%). The total copper sulphides in the Cleaner 1 Tailings sample amount to only 1.02% in the +3 μm fraction and 1.16% in the -3 μm fraction. The proportion of copper minerals in the -3 μm fraction of the Cleaner 1 Tailings sample is 73% with only 27% in the +3 μm fraction. The copper deportment in the Cleaner 1 Tailings sample comprises mainly chalcocite (77%), covellite (9%) and bornite (8%).

The liberation characteristics of copper sulphide minerals and chalcocite in the Cleaner 1 Tailings is shown in Table 13.14. This liberation data suggests that approximately 69% of the copper in the Cleaner 1 Tailings could be recoverable, although the liberation grain size is very small (3 μm to 5 μm).

Table 13.14: Liberation of Copper Sulphide Minerals and Chalcocite in Cleaner 1 Tailings

Liberation	Copper Sulphides		Chalcocite	
	Distribution (Wt.%)	Grain Size (μm)	Distribution (Wt.%)	Grain Size (μm)
Locked	9.42	2	12.59	2
Low Grade Middling	17.68	3	29.72	3
High Grade Middling	13.80	5	28.18	5
Liberated	5.52	7	4.99	7
Free	53.58	4	24.52	3
Total	100.00	-	100.00	-

Source: XPS Final Testwork Report, January 2026.

The precious metal content of the three samples is presented in Table 13.15. These analyses show that only silver appears to present in economic quantities. Gold, platinum, and palladium were measured by fire assay in the three composite samples while silver was measured by fire assay in the Cleaner 2 Concentrate and Cleaner 1 Tailings composite sample, and by aqua regia ICP-OES in the Rougher Tailings sample.

Table 13.15: Composite Sample Precious Metal Analyses

Analyte	Cleaner 2 Concentrate	Cleaner 1 Tailings	Rougher Tailings
Silver (g/t Ag)	1,330	63	5
Gold (Au ppb)	34	6	3
Palladium (Pd ppb)	12	<5	<5
Platinum (Pt ppb)	9	<5	<5

Source: XPS Final Testwork Report, January 2026.

All three composite samples were analysed for trace elements and had specific gravity determinations. These significant results are summarized in Table 13.16. Potential deleterious elements in the Cleaner 2 Concentrate include organic carbon (17.6%), Pb (3.4%), As (768 ppm) and Zn (6,470 ppm).

Table 13.16: Flotation Product Composite Chemical Analyses

Analyte	Unit	Detection Limit	Cleaner 2 Concentrate	Cleaner 1 Tailings	Rougher Tailings
Specific Gravity	-	-	3.21	2.86	2.73
Cu	%	-	26.2	1.70	0.29
Pb	%	-	3.4	0.4	0.06
C-Total	%	-	18.3	4.4	2.34
C-Organic	%	-	17.6	1.1	0.04
Al	%	0.01%	3.13	4.7	2.04
Ca	%	0.01%	1.5	7.28	5.53
Fe	%	0.05%	3.82	2.16	0.59
K	%	0.1%	1.6	2	1
Mg	%	0.01%	0.89	2.82	1.68
S	%	0.01%	11.5	0.31	0.34
Si	%	0.01%	7.6	23.7	35.9
Ti	%	0.01%	0.19	0.22	0.09
As	ppm	5 ppm	768	59	30
Co	ppm	0.2 ppm	649	47	13
Cr	ppm	30 ppm	530	1,860	270
Mn	ppm	3 ppm	409	1,460	1,050
Ni	ppm	10 ppm	810	780	130
Se	ppm	8 ppm	93	15	16
Zn	ppm	30 ppm	6,470	90	30

Source: XPS Final Testwork Report, January 2026.

13.3.4 Wrocław Tech Testwork Program

The testwork program undertaken by Wrocław Tech largely mirrored the work completed by XPS. The program comprised multi-element chemical analyses and mineralogical characterisation of the mineralized composites, BBWi determination and preliminary bench scale open circuit flotation tests.

As expected, the chemical analyses and mineralogical composition analytical results were almost identical to the XPS study. The BBWi derived was 14.8 kWh/t, which was similar to the XPS result.

The standard flotation flowsheet adopted for the testwork comprised a rougher and two cleaning stages, without regrinding. Over 40 flotation tests were completed using the composite samples. These tests investigated flotation kinetics, primary grind size, process water chemistry and reagent selection and dosages.

13.4 RESULTS INTERPRETATION AND CONCLUSIONS

A preliminary metallurgical testwork program has been completed by XPS using samples representing typical Nowa Sól mineralization. Although the average copper grade of the composites used to develop a preliminary flowsheet for the PEA was significantly higher than the average MRE grade (1.9% Cu vs. 1.29% Cu), the sample is considered suitable to support a PEA level of study.

Mineralogical characterisation of the metallurgical sample showed that the primary Cu-bearing mineral is chalcocite, which accounted for 80% of the contained copper. The next prevalent copper minerals were covellite and chalcopyrite that contain 11% and 5% of the copper, respectively. The measured P_{80} size for the chalcocite in the feed was 47 μm , liberated grains P_{80} 74 μm and non-liberated grains P_{80} 34 μm .

The BBWi for the XPS composite sample was 15.6 kWh/t and the Wrocław Tech sample measured 14.8 kWh/t.

The preliminary flowsheet developed from the XPS testwork program is considered conventional and comprises primary grinding to around P₈₀ size of 60 µm, rougher flotation, rougher concentrate regrinding to around 11 µm and two stages of cleaning.

13.4.1 Copper and Silver Metallurgical Recoveries

The average final XPS confirmatory test rougher copper recovery was 88.6% into a rougher concentrate containing 7.3% Cu while the copper recovery into the Cleaner 2 Concentrate grading 26.2% Cu was 79.4% (Table 13.17). Since these results are for an open circuit test (there was insufficient sample to undertake a locked cycle test) a factor is typically applied to allow for additional recovery within a closed circuit cleaner circuit. The mineralogical liberation characterisation of the Cleaner 1 Tailings showed that around 69% of the copper minerals in this process stream were classed as free, liberated or high-grade middling Table 13.14. Therefore, a cleaner recovery factor of approximately 70% is considered reasonable to estimate metallurgical recoveries for a closed circuit system.

Table 13.17: XPS Flotation Product Recoveries

Product	Grade (Cu%)	Open Circuit Test Recovery		
		(Cu%)	(Pb%)	(Ag%)
Cleaner 2 Concentrate	26.2	79.4	66.6	82.3
Cleaner 1 Concentrate	19.7	81.3	70.3	85.9
Rougher Concentrate	7.3	88.6	84.7	96.1

The results from the metallurgical test program conducted by Wrocław Tech were similar to the XPS test results. The rougher stage copper and silver recoveries tended to be higher than XPS (92-94% for copper and 94 to 98% for silver) albeit at higher weight recoveries and lower grades, but recoveries to the Cleaner 2 concentrate were generally lower and lower grade, due to the absence of a rougher concentrate regrind step.

In interpreting the metallurgical open circuit test results, Wrocław Tech observed that the mineralization/chemical characteristics and rougher flotation efficiency parameters for the Nowa Sól sample were similar to comparable mineralization from a nearby operating mine that was previously tested. Assuming that the selectivity of copper and silver enrichment from Nowa Sól copper mineralization was at the level of the industrial selectivity of the comparable material, Wrocław Tech estimate metallurgical recoveries of 88.75% for copper and 86.64% for silver.

13.4.2 Concentrate Quality

The copper concentrate produced by XPS contained significant payable silver. Potential deleterious elements include organic carbon, arsenic, lead and zinc. A detailed analysis of the concentrate produced by XPS is included in Table 13.16.

13.5 RECOMMENDATIONS

It is recommended that further metallurgical testing be undertaken to support the next phase of Project development, probably at the Pre-Feasibility Study (PFS) level. The test program should consider the following:

- Select fresh composite samples to cover the mineral resources spatially, range of copper and silver grades, ore-types and lithologies.
- Detailed chemical and mineralogical characterisation of each composite sample.
- Undertake comprehensive comminution testing for each composite sample.
- Complete development and optimization flotation tests on separate composite samples.
- Investigate the potential of Zn and Pb as by-products.
- Undertake a series of locked cycle tests on separate composite samples.
- Complete characterisation studies on locked cycle flotation products.
- Undertake standard dewatering testing on samples of tailings and final concentrate.
- Consider testing of alternative technologies with vendors to improve energy and metallurgical efficiencies.

14 MINERAL RESOURCE ESTIMATES

The MRE was completed independently by Micon with limited input from Lumina geological personnel. Liz de Klerk, Pr.Sci.Nat., QMR and Dr. Ryan Langdon, Ph.D, CGeol, who are independent of Lumina and are QPs within the meaning of NI 43-101, are responsible for the MRE disclosed in this Report.

Mineral Resources are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and were prepared using the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

14.1 DATA

14.1.1 MRE Database

A database was compiled from Microsoft Excel spreadsheets supplied by Lumina for use in the MRE dated June 13, 2025. The database was compiled for the entire region and the MRE database is constrained to the data used to inform the Nowa Sól MRE model only. The database contains a total of 30 drill holes of which 23 are inside the license boundary (Figure 14.1, Table 14.1). Within the license, 22 were drilled by Lumina between 2014 and 2024 and one historical drill (Bojadła 1 or Boj_1) from 1986.

The MRE database includes seven tables, including: collar, survey, assay, lithology, density, Rote Fäule depth, and core recovery. Systematic checks were performed to ensure that the data is acceptable and in a suitable condition for use in modelling and resource evaluation in accordance with the CIM (2019), as outlined in Section 12.

The provided assay dataset contained multiple reported values for copper (Cu), silver (Ag), lead (Pb), and zinc (Zn), resulting from the use of different analytical methods applied according to the sample grade. These results reflect variations in laboratory protocols for low- and high-grade samples. All samples were analysed using a partial Aqua Regia digest and ICP-MS. If Cu, Mo, Pb or Zn returned values >1,000 ppm, or the Ag was >10 ppm, or the Au was >100 ppb during the Aqua Regia digest, samples were automatically reassayed using Multi Acid ICP-OES and Fire Assay methods to obtain near-total values for copper, silver, molybdenum, zinc, lead, and gold. Aqua Regia digestion is considered a partial digestion compared to a four-acid digestion in which a near total digestion of minerals is obtained.

The map displays the Nowa Sól Deposit area, bounded by a thick black outline. The map includes a coordinate grid with UTM coordinates (mE) on the x-axis (558,000 to 570,000) and Northing coordinates (mN) on the y-axis (5,736,000 to 5,754,000). A north arrow is located in the top left corner. A scale bar indicates 0 to 3 km.

Legend:

- Nowa Sól Deposit Outline:** Thick black line.
- Included in MRE Database:** Green dots.
- Excluded from MRE Database:** Blue dots.
- Towns / Villages:** Black dots.
- Paved Roads:** Orange lines.
- Rivers:** Light blue lines.
- Creeks:** Darker blue lines.
- Railway:** Dashed black line.
- Provincial Boundary:** Dashed grey line.

Sampling Locations:

- Included in MRE Database (Green dots):** Bojadla_1, Jany_C2, Jany_C3, NS_C29, NS_C11, NS_C14Bst, NS_C30, NS_C12, NS_C17B, NS_C16, NS_C4, NS_C25, NS_C33, NS_C22, NS_C1, NS_C2, NS_C13, NS_C31, NS_C15, NS_C17B, Slawa_IG1, S-486, Borowiec_2, S-483, S-487, S-482, S-484, Rozanowka_1, S-476, Grochowice_28, Grochowice_35, Grochowice_31, Grochowice_M9, Grochowice_42, Grochowice_3, Grochowice_27, Grochowice_33.
- Excluded from MRE Database (Blue dots):** Krzepi.

Towns / Villages (Black dots): Bojadla, Pyrniki, Stawy, Konotop, Lubięcin, Mesze, Lipiny, Zat_C1.

Infrastructure:

- Paved Roads:** Orange lines.
- Rivers:** Odra River (light blue).
- Creeks:** Various smaller blue lines.
- Railway:** Dashed black line.
- Provincial Boundary:** Dashed grey line.

Concessions:

- Konotop Concession:** Indicated by a black arrow pointing to the eastern part of the deposit.
- Nowa Sól II Concession:** Indicated by a black arrow pointing to the southern part of the deposit.

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Table 14.1: Drill Holes Included in the MRE Database

Hole ID	Company	Year	Easting	Northing	Elevation	EOH	Sampled (m)
Inside License Boundary							
Jany_C2	Lumina	2017	558,273	5,754,394	59	2,126	16.9
Jany_C3	Lumina	2017	556,333	5,753,045	59	2,112	9.2
NS_C1	Lumina	2014	559,531	5,742,783	74	1,849	19.4
NS_C2	Lumina	2014	562,355	5,742,778	125	1,882	20.1
NS_C3st	Lumina	2014	556,460	5,745,031	80	1,895	16.4
NS_C4	Lumina	2015	559,561	5,745,495	84	1,913	22.2
NS_C11	Lumina	2016	558,850	5,751,085	59	2,054	13.5
NS_C12	Lumina	2015	559,431	5,748,320	77	1,960	14.0
NS_C13	Lumina	2015	561,062	5,741,163	75	1,790	7.8
NS_C14Bst	Lumina	2018	561,419	5,751,036	66	2,070	8.8
NS_C15	Lumina	2020	562,486	5,748,019	81	1,993	8.1
NS_C16	Lumina	2015	561,662	5,746,413	82	1,943	22.8
NS_C17B	Lumina	2016	556,663	5,747,730	78	1,968	8.7
NS_C22	Lumina	2015	564,068	5,744,120	92	1,932	9.8
NS_C25	Lumina	2018	562,752	5,745,325	93	1,950	9.9
NS_C27st	Lumina	2023	561,294	5,753,058	63	2,121	8.3
NS_C29	Lumina	2023	558,132	5,752,624	59	2,100	9.2
NS_C30	Lumina	2019	561,045	5,750,073	69	2,032	10.1
NS_C31st	Lumina	2024	558,437	5,743,681	84	1,842	5.3
NS_C33	Lumina	2018	561,287	5,743,784	82	1,889	7.8
Wil_W1	Lumina	2015	563,362	5,754,259	60	2,167	16.5
Zat_C1	Lumina	2017	563,976	5,742,598	85	1,895	15.1
Boj_1	Historical	1986	557,453	5,755,738	57	2,205	5.0
Outside License Boundary (refer to Section 23)							
Borowiec_2	Historical	1988	559,190	5,737,991	67	1,702	2.6
Grochowice_28	Historical	1990	567,520	5,736,953	72	1,650	8.0
S-482	Historical	1981	564,503	5,735,289	69	1,747	7.6
S-483	Historical	1983	561,918	5,737,040	68	1,683	6.2
S-486	Historical	1985	559,467	5,738,426	67	1,656	8.0
S-487	Historical	1984	557,619	5,736,217	65	1,603	13.4
Slawa_IG1	Historical	1974	570,677	5,744,104	62	2,273	1.8

Source: Micon, 2025.

Note: CRS: UTM WGS 84, Zone 33 North; all drill holes drilled at an azimuth of 360° and dip of -90°.

For the purposes of the MRE, the assay values were selected on the completeness of the digestion method. Where multiple values were available, priority was given in the following order: Fire Assay, Multi-Acid digest, and Aqua Regia digest. This approach ensured the selection of the most representative data for inclusion in the MRE database.

Simplified stratigraphic columns were developed by Micon to support geological modelling of the Nowa Sól deposit. This was based on the stratigraphic and lithological data provided by Lumina. The simplification was necessary to address inconsistencies in lithological naming conventions in the dataset. The resulting stratigraphic columns, referred to as 'litho_model_micon' and 'litho_model_simp_micon,' are consistent with the established stratigraphy of the Zechstein Basin (see Figure 7.4), and are summarized in Table 14.2.

Table 14.2: Simplified Stratigraphic Columns

Simplified Stratigraphic Columns	
litho_model_micon	litho_model_simp_micon
Paleogene and Neogene	Paleogene and Neogene
Triassic	Triassic
PZ4	PZ4
PZ3	PZ3
PZ2	PZ2
PZ1-A1g_upper_anhydrite	PZ1_above_T1_kupferschiefer
PZ1-Na1_oldest_salt	
PZ1-A1d_lower_anhydrite	
PZ1-Ca1_zechstein_limestone	
PZ1-T1_kupferschiefer	PZ1-T1_kupferschiefer
PZ1-Ca0_boundary_limestone	PZ1_below_T1_kupferschiefer
P1-Bs-P1_fluvial_sandstone	
P1-Cs-P1_eolian_sandstone	

Source: Micon, 2025.

14.1.2 Assaying and Detection Limits

All Lumina drill core intersecting the mineralized horizon was sampled and assayed, as was the majority of historical drill core. For historical intervals where assay results were reported as below detection limit, recorded as zero, or were not assayed, values were assigned equal to half the detection limit of the most recent drilling and sampling programs. These substituted values are: 0.000005% Cu, 0.05 ppm Ag, 0.000005% Pb, and 0.00005% Zn.

14.2 GEOLOGICAL MODELLING

All geological modelling was performed in Leapfrog 2024.1.2.

14.2.1 Grade Domains

Grade domains were modelled individually for Cu, Ag, Pb and Zn. Downhole assay plots of the elements of interest show that mineralization approximates a normal distribution, with the highest grades generally correlating with the T1 Kupferschiefer unit (Figure 14.2 to Figure 14.5). The depth of mineralization with respect to the known stratigraphy is controlled by the depth of the Rote Fäule (the strongly oxidized redox front) and its proximity to the T1 Kupferschiefer unit. Due to the fact that mineralization is not constrained to a single lithological unit and the grades show a normal shaped distribution either side of the T1 Kupferschiefer hosted in different lithologies a grade shell approach was used for modelling.

Grade shells were modelled for Cu, Ag, Pb and Zn as they are only moderately correlated (Table 14.3). This is in part due to stratigraphic variations in grade, for example Cu and Ag grades are generally show similar grade distributions above and below the T1 Kupferschiefer unit, whereas Pb and Zn show elevated grades above the T1 Kupferschiefer unit compared to below it (Figure 14.6 to Figure 14.9). Additionally, lateral grade variations were observed and are interpreted to be spatially related to proximity to the Rote Fäule. To preserve the complex multivariate relationships observed a detailed multivariate simulation study using Projection Pursuit Multivariate Transform (PPMT) would need to be undertaken, which was beyond the scope. It is recommended that this is considered in future model updates.

Statistical analysis was performed to select appropriate cut-off grades for grade domain modelling. The selected cut-off grades are shown in Table 14.4. Histograms of grade data show clear bi-modal or multi-modal distributions. The cut-off grades were selected based on these observed populations and inflection points in log-probability plots (Figure 14.6 to Figure 14.9). For Ag, a separate high-grade population was modelled as high-grade intervals were of sufficient thickness to model with the selected composite length and block size of 0.6 m (see Section 14.4). A separate high-grade population is visible in the Cu data, but it is too thin to resolve with the selected composite length and block size. Tests were made to reduce the block size, but the processing times for the model increased considerably. It is recommended that further investigations are undertaken to assess whether a high-grade Cu could be modelled when the MRE is updated.

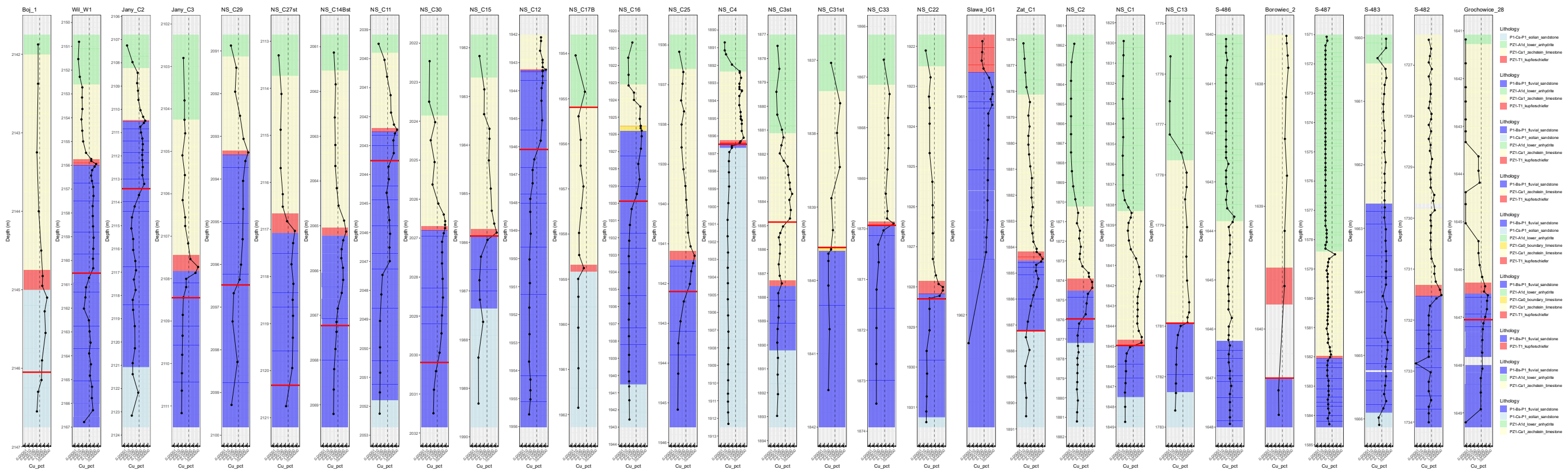
14.2.1.1 Modelling Methodology

The grade domains were modelled in Leapfrog Geo using the following methodology:

- Economic composites were created at the relevant cut-off grades for each variable of interest (Cu, Ag, Pb, Zn). No settings were used to control the minimum ore composite length, maximum included waste, or maximum consecutive waste as statistical analysis showed there was minimal waste material below cut-off within the mineralized horizon.
- The economic composites were merged with assay data and the ore composites were used to guide the manual selection of data inside the ore grade domain. Minor ore intervals above or below the main mineralized horizon were treated as waste. For Cu and Ag a single ore intercept was generated for each drill hole. The Cu grade in Lumina drill hole NS_C17B outside the license boundary was slightly below the cut-off grade (0.02% Cu) but was included to ensure the Cu mineralized wireframe extended across the entire modelled area. For high-grade Ag, Pb and Zn not all drill holes had encountered sufficient grade to generate an ore grade economic composite. Waste was assigned as above or below the ore grade domain where data allowed. A table of statistics for the whole domain width ore grade intercepts within the license boundary is shown in Table 14.5.
- The grade domains were modelled as veins. For Cu and Ag the minimum vein thickness was set to 0.2 m equivalent to the narrowest ore grade intercept. Whereas for high-grade Ag, Pb, and Zn the vein was set to pinch out as the ore domain was not intercepted in all drill holes. In the areas where the vein model pinched out manual points were added to the hangingwall and footwall based on the location of the Cu ore grade domain to ensure consistency in the general orientation on the vein model. For all ore domains the reference surface was set as the Uppermost of the Rotliegende modelled surface (see Section 9.4 and Figure 9.1). The Uppermost surface of Rotliegende paleotopographic surface is equivalent to the basal surface of the T1 Kupferschiefer unit and represents the regional orientation of the mineralization across the Nowa Sól license.

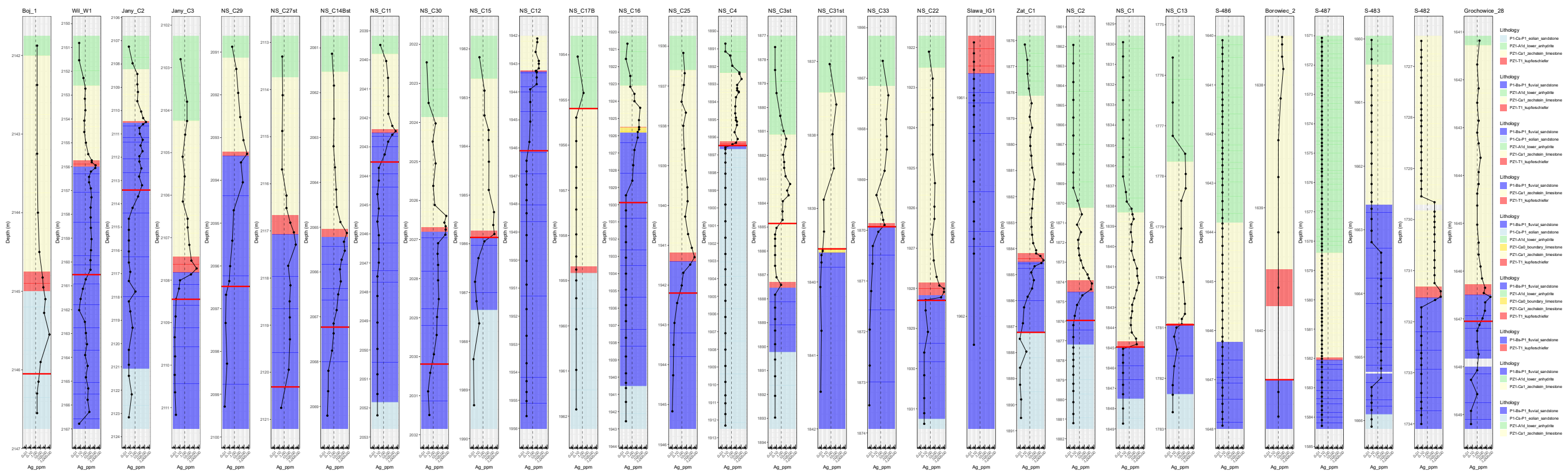
Geological models were generated from the vein models for each variable (Cu, Ag, Pb, Zn) with domains labelled waste_above, ore, and waste_below. For Ag, an additional ore_hg domain was modelled for high-grade Ag. An example of the modelled veins is shown in Figure 14.10.

Figure 14.2: Downhole Cu Assay Plots for the MRE Database Ordered North to South



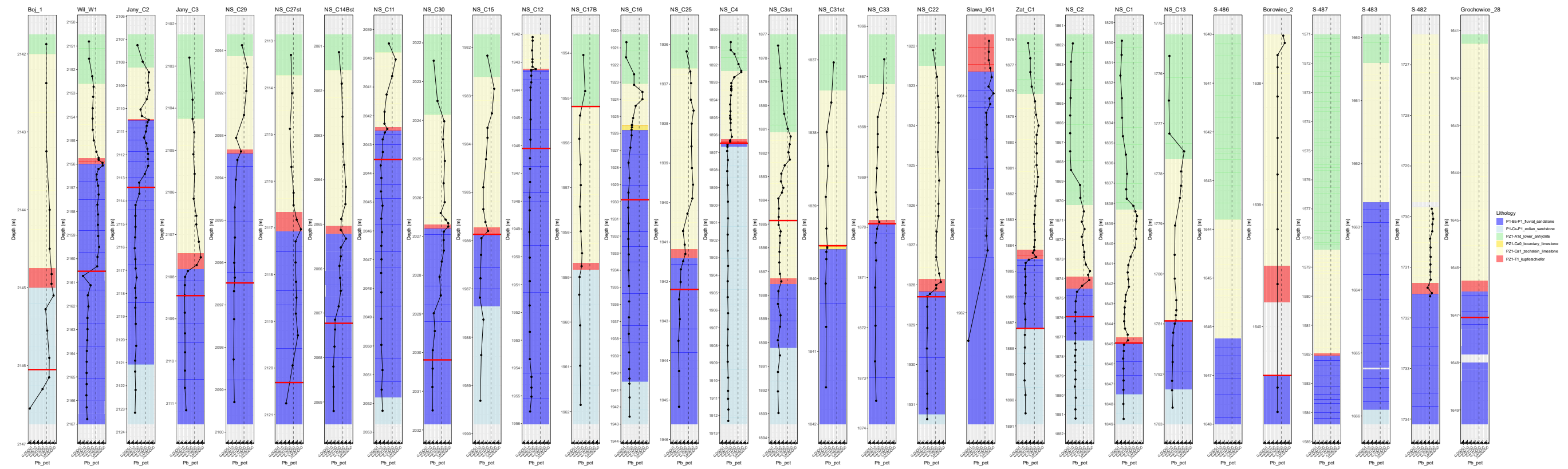
Source: Micon, 2025.

Figure 14.3: Downhole Ag Assay Plots for the MRE Database Ordered North to South



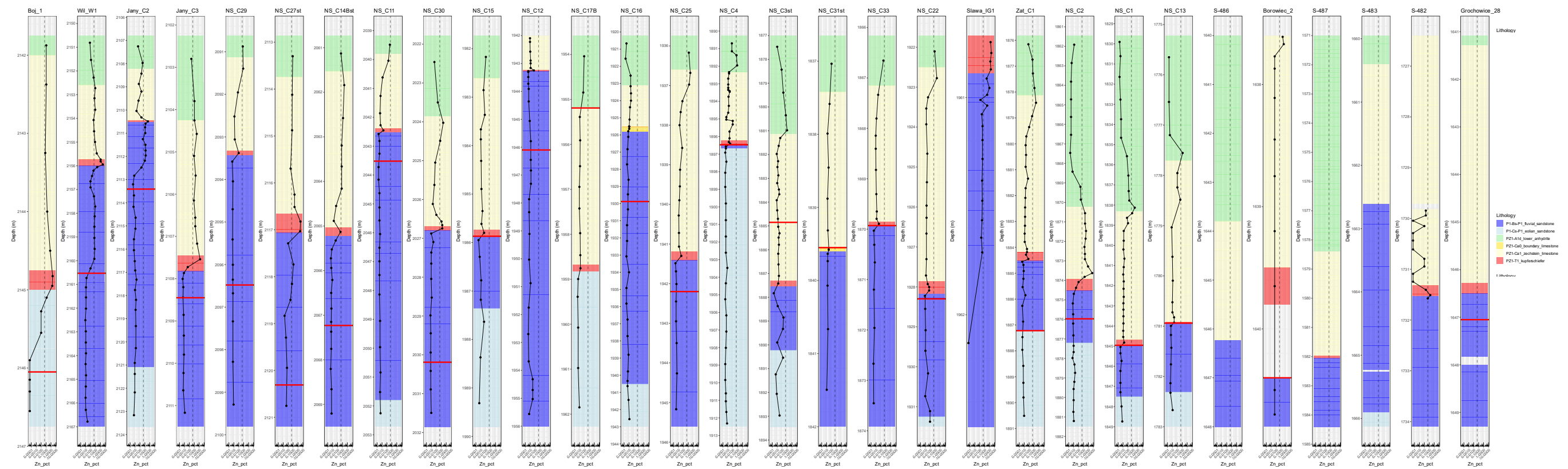
Source: Micon, 2025.

Figure 14.4: Downhole Pb Assay Plots for the MRE Database Ordered North to South



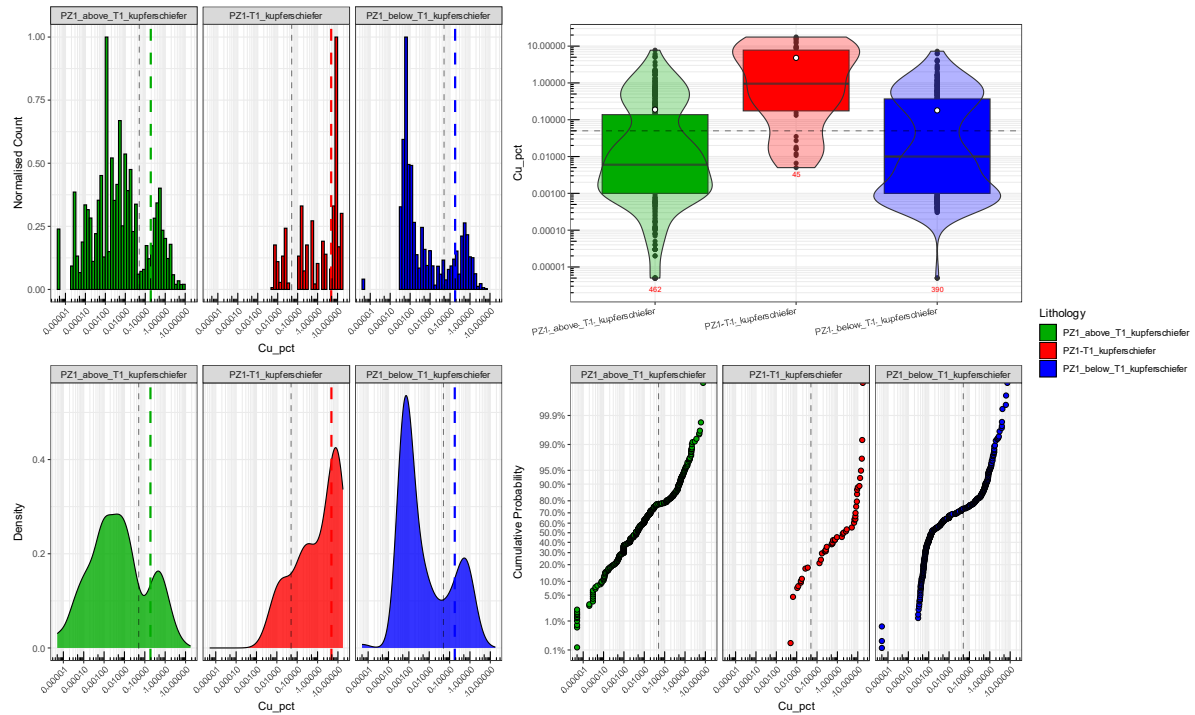
Source: Micon, 2025.

Figure 14.5: Downhole Zn Assay Plots for the MRE Database Ordered North to South



Source: Micon, 2025.

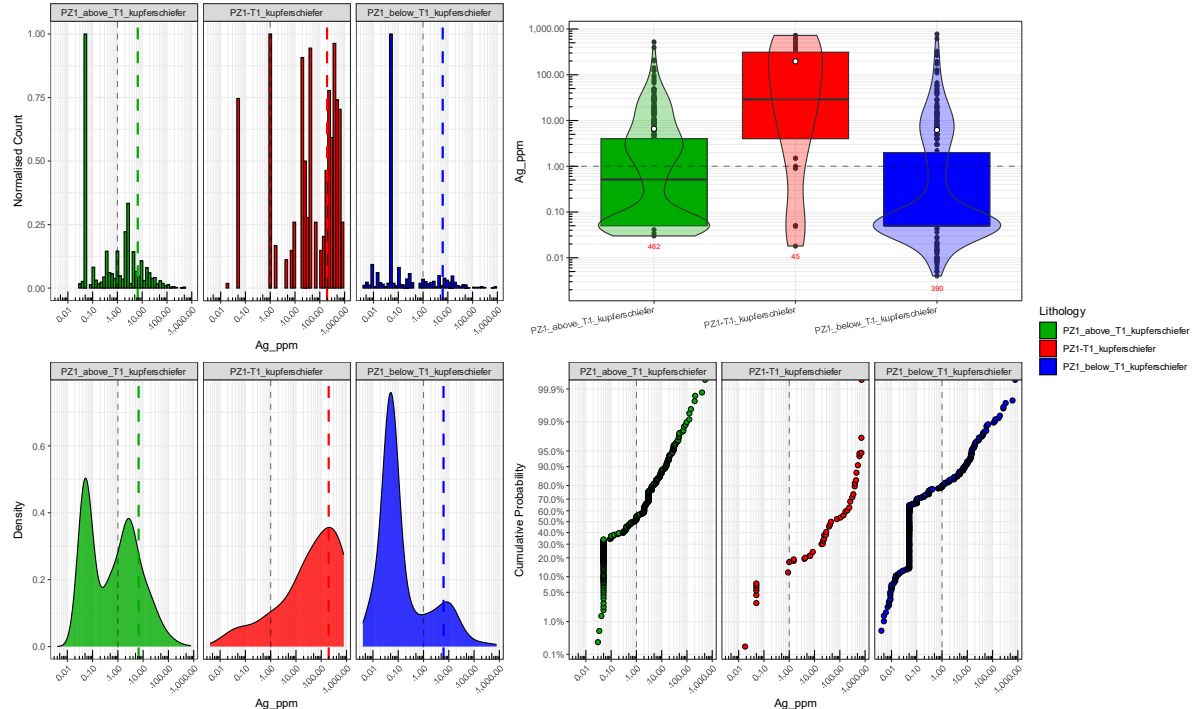
Figure 14.6: Cu Assay Plots by Simplified Stratigraphic Grouping



Source: Micon, 2025.

Note: Cut-off grade for domain modelling shown as dashed line.

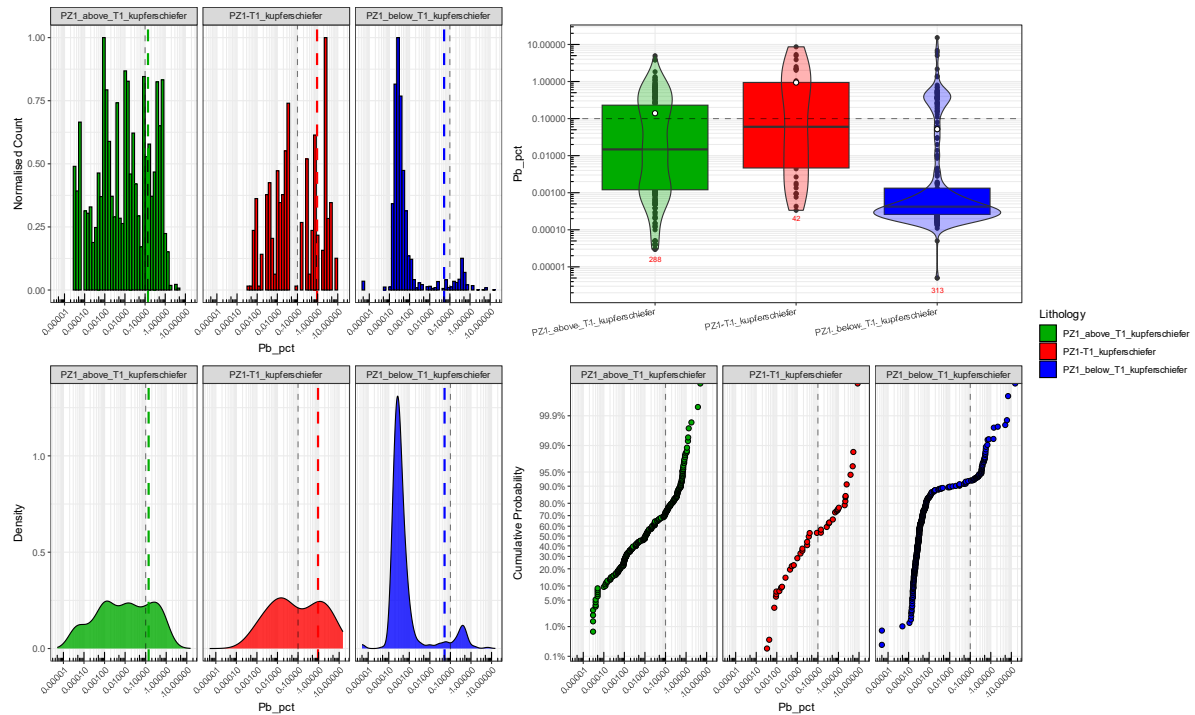
Figure 14.7: Ag Assay Plots by Simplified Stratigraphic Grouping



Source: Micon, 2025

Note: Cut-off grade for domain modelling shown as dashed line.

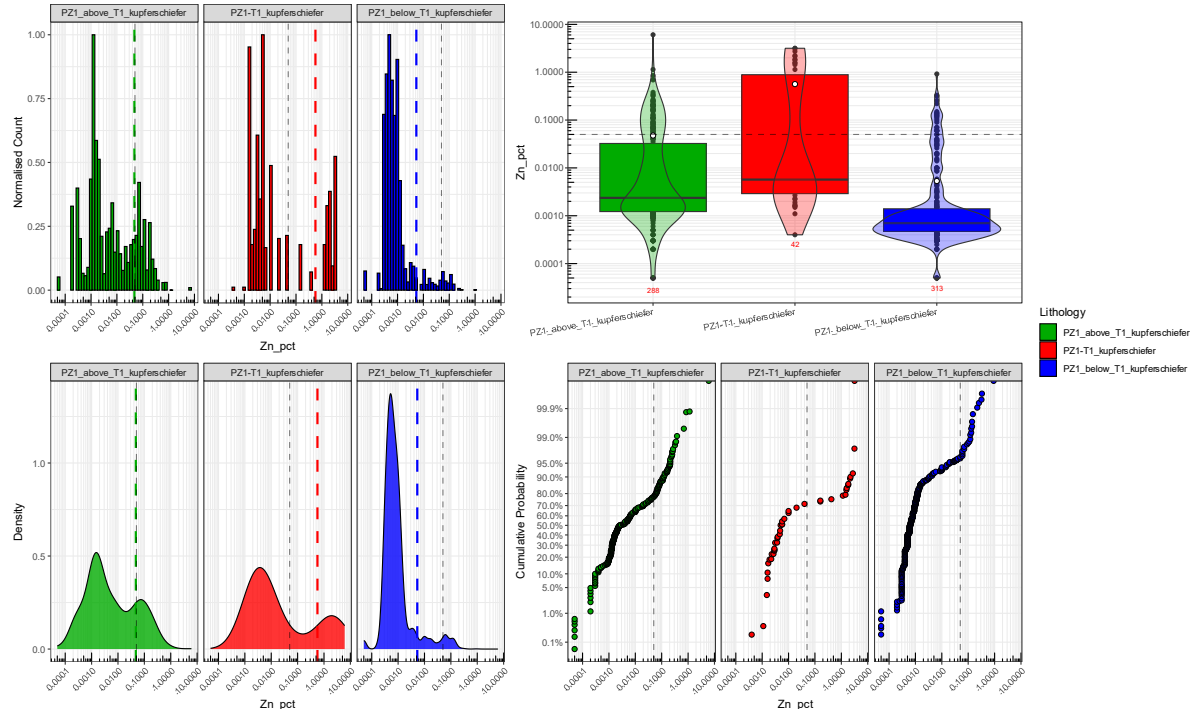
Figure 14.8: Pb Assay Plots by Simplified Stratigraphic Grouping



Source: Micon, 2025

Note: Cut-off grade for domain modelling shown as dashed line.

Figure 14.9: Zn Assay Plots by Simplified Stratigraphic Grouping



Source: Micon, 2025

Note: Cut-off grade for domain modelling shown as dashed line.

Table 14.3: Assay Data Correlation Matrix

	Ag (ppm)	Cu (pct)	Pb (pct)	Zn (pct)
Ag (ppm)	1.00	0.66	0.12	0.04
Cu (pct)	0.66	1.00	0.19	0.03
Pb (pct)	0.12	0.19	1.00	0.43
Zn (pct)	0.04	0.03	0.43	1.00

Source: Micon, 2025.

Table 14.4: Selected Cut-Off Grades for Grade Domain Modelling

Assay	Domain	Cut-Off
Cu (pct)	Ore-Waste	0.05
Ag (ppm)	Ore-Waste	1.00
Ag (ppm)	High Grade	30.00
Pb (pct)	Ore-Waste	0.10
Zn (pct)	Ore-Waste	0.05

Source: Micon, 2025.

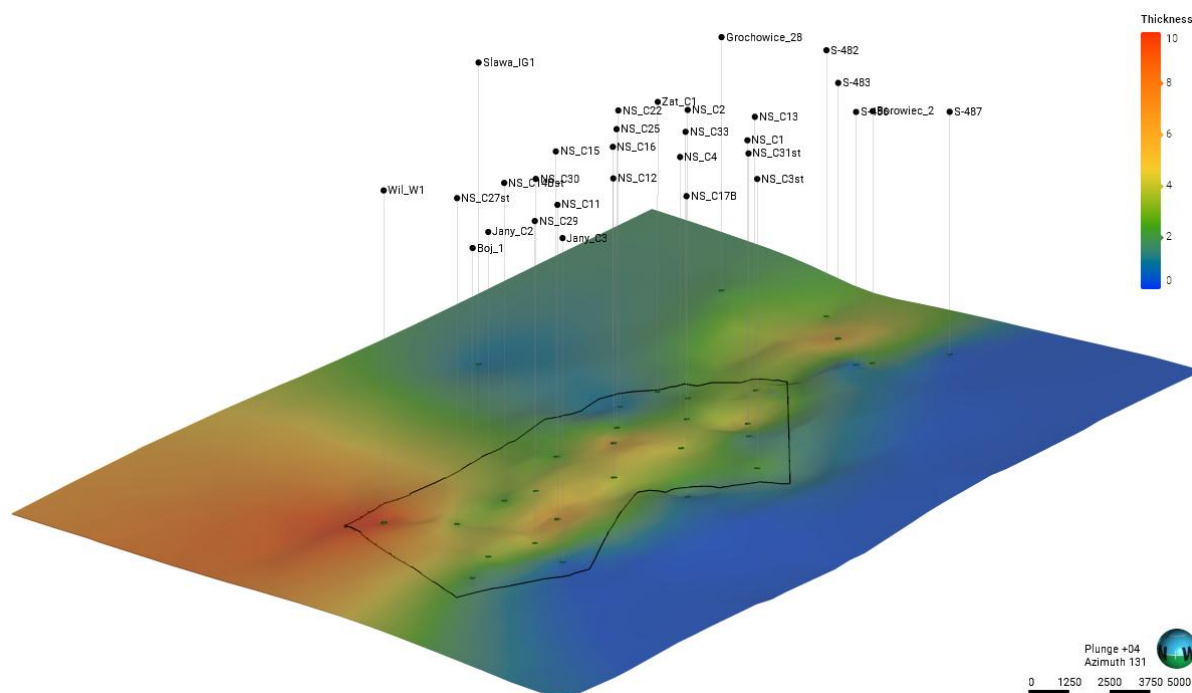
Table 14.5: Length Weighted Whole Domain Width Ore Grade Statistics Within the License Boundary

Domain	Count	Mean	SD	Var.	Min.	Median	Max.	CV
Cu - Ore								
Grade (%)	22	1.02	0.68	0.46	0.29	0.85	5.97	0.66
Length (m)	22	3.42	2.23	4.95	0.46	3.00	10.84	0.65
Ag - Ore (including internal high-grade domain)								
Grade (%)	22	25.89	16.50	272.15	5.64	20.27	72.93	0.64
Length (m)	22	5.05	2.38	5.65	1.26	4.76	13.12	0.47
Ag - High Grade								
Grade (%)	22	135.09	122.35	14968.78	13.00	81.71	407.88	0.91
Length (m)	22	0.78	0.49	0.24	0.19	0.61	1.69	0.63
Pb - Ore								
Grade (%)	19	0.52	0.17	0.03	0.23	0.58	0.86	0.32
Length (m)	19	2.84	1.81	3.27	0.31	2.52	6.39	0.64
Zn - Ore								
Grade (%)	18	0.20	0.14	0.02	0.03	0.14	0.58	0.71
Length (m)	18	2.43	1.92	3.70	0.31	1.58	7.52	0.79

Source: Micon, 2025.

Note: Var. = Variation; Min. = Minimum; Max. = Maximum; CV = Coefficient of variation.

Figure 14.10: Oblique view to the Northeast of the Modelled Cu Ore Domain Coloured by Thickness with the Nowa Sól License Boundary



Source: Micon, 2025.

Note: 5x vertical exaggeration.

14.2.2 Density Domains

A total of 387 dry density measurements were taken on drill core from Lumina drill holes. The density samples were analysed using the Waxed Air/Water method methodology and are well distributed vertically through the different stratigraphic units above, within, and below the mineralized orebody.

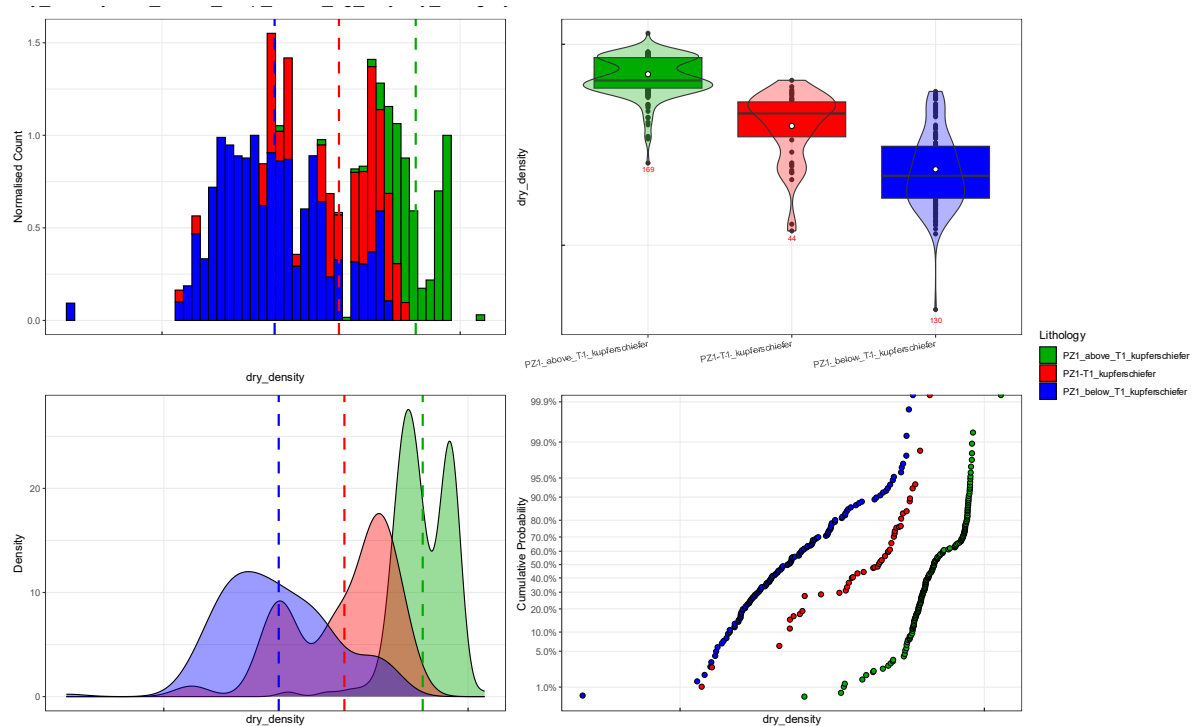
Histograms of the data showed clear multi-modal populations. Furthermore, visually and statistically, there is a clear correlation between decreasing density and stratigraphic depth (Figure 14.11). Density domains were modelled at different density cut-offs and median density values were applied to each domain (see Section 14.8.5). The cut-off grades were selected based on the observed grade populations and inflection points in log-probability plots (Figure 14.11) and are shown in Table 14.6. There is no relationship between grade and density.

Table 14.6: Selected Cut-Offs for Density Domain Modelling

Assay	Domain	Cut-Off
Dry Density (t/m ³)	Low	< 2.4
	Medium	2.4-2.65
	High	2.65-2.85
	Very High	> 2.85

Source: Micon, 2025.

Figure 14.11: Dry Density (t/m^3) Plots by Simplified Stratigraphic Grouping



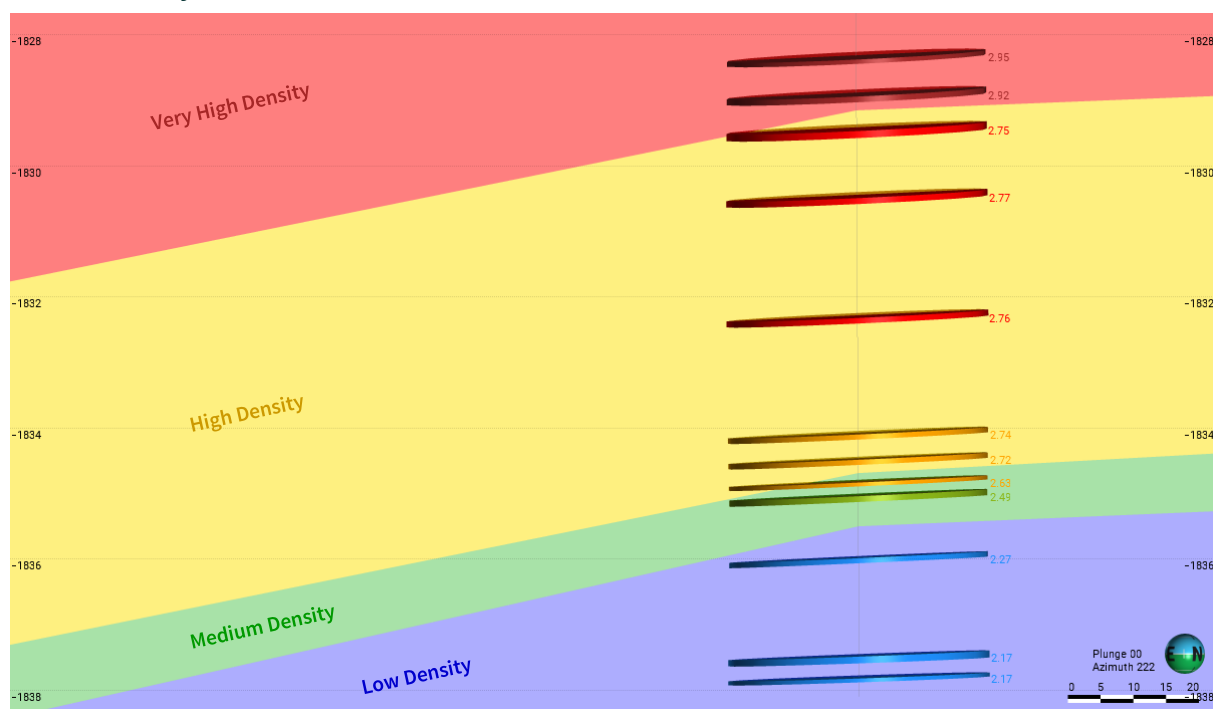
Source: Micon, 2025.

14.2.2.1 Modelling Methodology

The grade domains were modelled in Leapfrog Geo using the following methodology:

- A numerical model was generated based on the selected density cut-off values. Resulting in low-, medium-, high-, and very high-grade wireframes which were utilized exclusively for drill hole data flagging to guide further domain modelling. These preliminary wireframes were not used in the interpolation of the final domain wireframes. An updated evaluation table was created using the numerical model for Lumina drill holes containing density measurements.
- The low- and high-grade domains were interpreted and modelled as veins. For all mineralized domains, the T1 Kupferschiefer unit hangingwall surface was used as the reference surface for structural control.
- A geological model was constructed using the hangingwall and footwall surfaces of the low- and high-grade domain veins. These structural boundaries were then used to generate the final wireframes representing the low-, medium-, high-, and very high-grade density domains. The resultant modelled domains are presented in Figure 14.12.

Figure 14.12: Northwest Southeast Cross-Section of Drill Hole NS_C22 Showing Dry Density (t/m^3) Data and the Density Domain Model Wireframes



Source: Micon, 2025.

Note: 10x vertical exaggeration.

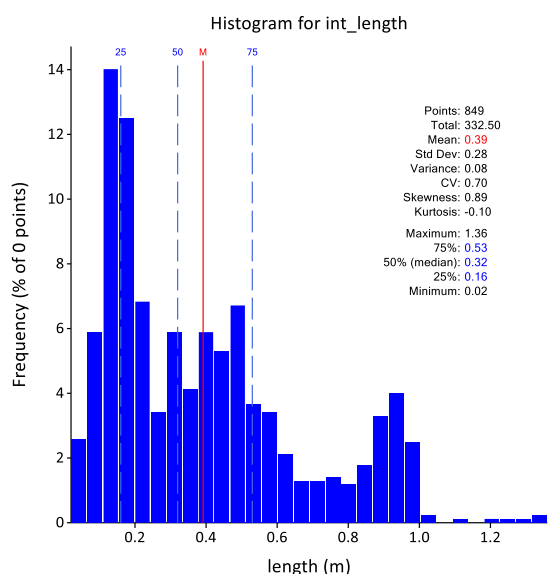
14.3 COMPOSITING

The raw assay sample lengths were reviewed to determine an appropriate composite length (Figure 14.13). The core samples exhibited a wide range of original sample lengths. Considering the minimum mining height of 1.8 m applied at the neighbouring KGHM operations, a block size of 0.6 m was deemed appropriate. This selection reflects a balance between the modelled thicknesses of the ore domains and the computational efficiency required for block model processing. Based on these considerations, a fixed composite length of 0.6 m was applied to the assay dataset for use in the MRE.

Approximately 80% of the data have sample lengths less than 0.6 m, minimising the amount of sample splitting during compositing. Within the ore domains, sample lengths tend to be smaller where grades are higher and 93% of the Cu assay data in the Cu ore domains have sample lengths less than 0.6 m.

Composites were created at 0.6 m equal lengths with a minimum coverage of 0% and residual end lengths less than 0.3 m were added to the previous interval. Compositing was performed within each estimation domain. The minimum coverage was set to 0% as a few intervals outside the Nowa Sól license boundary have sample lengths less than 0.3 m (50% minimum coverage).

Figure 14.13: Histogram of Sample Interval Lengths



Source: Micon, 2025.

14.4 EXPLORATORY DATA ANALYSIS

Exploratory Data Analysis (EDA) was undertaken for assay data to define and validate stationary estimation domains, identify the nature of the contacts between domains, and to inform outlier management. Methods used include contact plots, summary statistics, histogram plots, log probability plots, and swath plots. All EDA was performed on composited assay data unless specified.

14.4.1 Statistics

The composite assay data was not declustered considering that the data is regularly spaced with limited clustering. The length weighted statistics for each domain are shown in Table 14.7.

Table 14.7: Length Weighted Statistics by Grade Domain

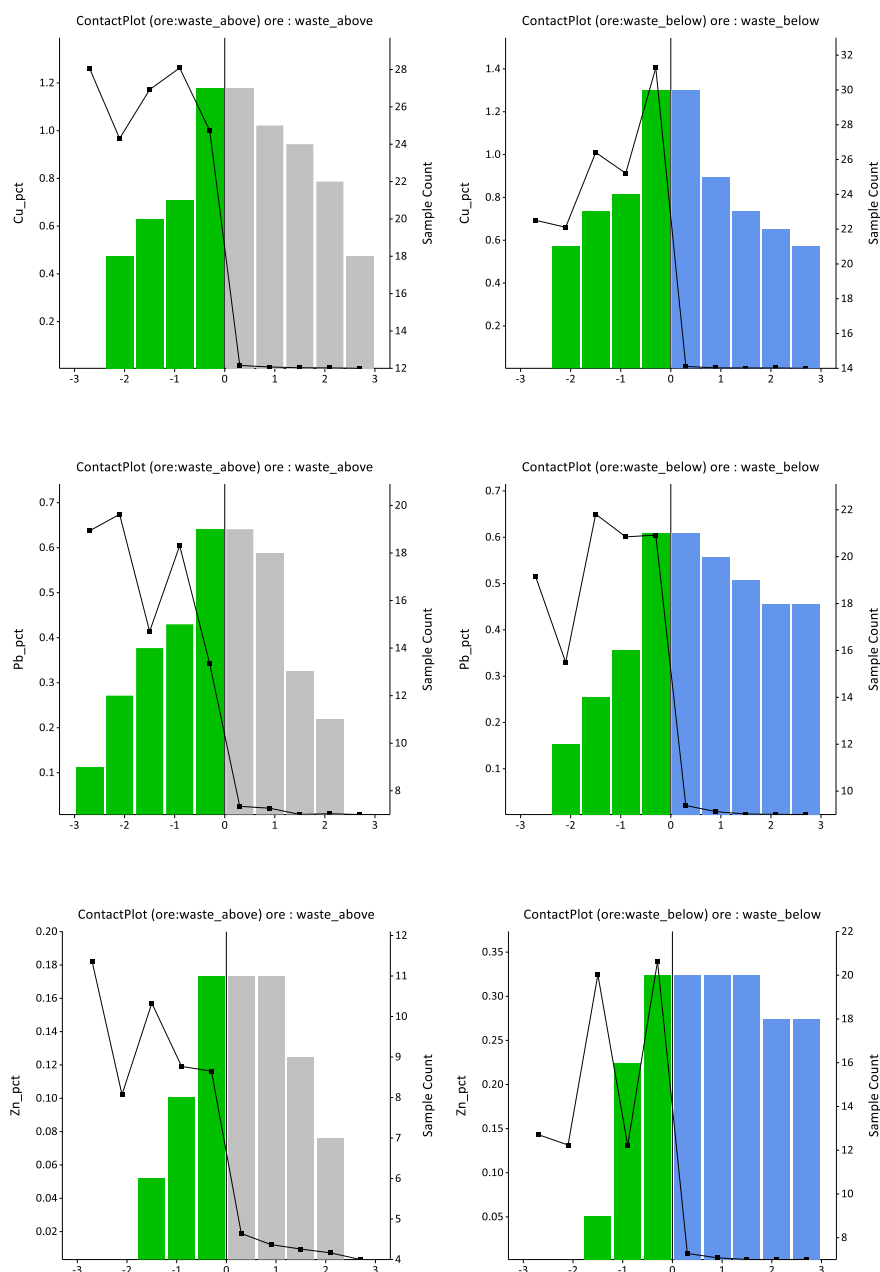
Domain	Count	Mean	SD	Var.	Min.	Median	Max.	CV
Cu %								
Waste Above	196	0.01	0.01	0.00	0.00	0.00	0.04	1.45
Ore	154	0.94	1.20	1.44	0.01	0.56	6.84	1.27
Waste Below	232	0.00	0.01	0.00	0.00	0.00	0.04	1.80
Ag %								
Waste Above	98	0.16	0.18	0.03	0.04	0.05	0.82	1.07
Ore	180	5.61	4.97	24.68	0.07	3.54	23.58	0.89
Ore High Grade	36	129.72	122.24	14,942.24	13.00	58.88	407.88	0.94
Waste Below	226	0.08	0.11	0.01	0.00	0.05	0.86	1.40
Pb %								
Waste Above	121	0.01	0.02	0.00	0.00	0.00	0.09	1.81
Ore	98	0.52	0.50	0.25	0.07	0.39	2.83	0.94
Waste Below	289	0.00	0.01	0.00	0.00	0.00	0.09	3.16
Zn %								
Waste Above	108	0.01	0.01	0.00	0.00	0.00	0.04	1.45
Ore	77	0.21	0.37	0.14	0.00	0.09	2.11	1.77
Waste Below	318	0.00	0.00	0.00	0.00	0.00	0.03	1.95

Source: Micon, 2025.

14.4.2 Contact Plots

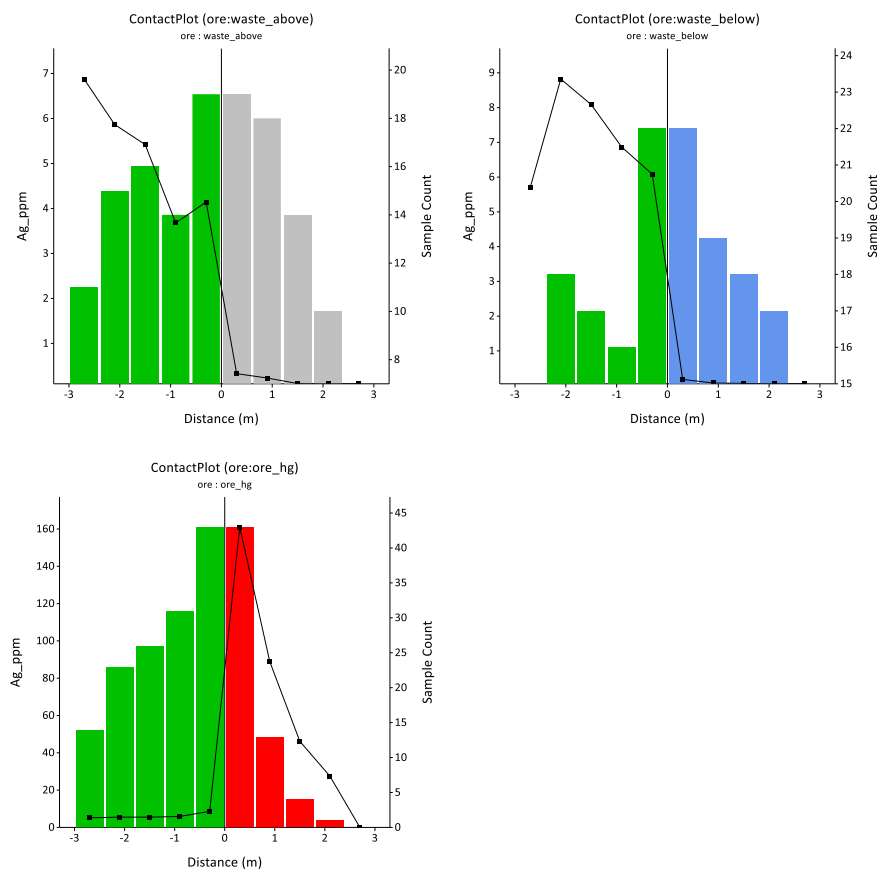
Contact plots were created for the modelled grade domains. All the contact plots demonstrated hard boundaries, indicating that they should be interpolated as separate estimation domains. The contact plots are shown in Figure 14.14 and Figure 14.15.

Figure 14.14: Grade Domain Contact Plots for Cu, Pb, and Zn



Source: Micon, 2025.

Figure 14.15: Grade Domain Contact Plots for Ag



Source: Micon, 2025.

14.4.3 Outliers

Histograms and log probability plots were created to understand where the grade distributions deviated from a normal distribution, indicating the presence of outliers. They are shown in Figure 14.16 for all Cu domains, and in Figure 14.17 for Ag, Pb, and Zn ore domains. The presence of high-grade outlier samples is problematic if they have little spatial connectivity, and they need to be managed appropriately. Outlier grades that required capping before interpolation to limit their influence were identified for the Pb ore domain only. The ore domains contained outlier values that were mitigated by using a restricted search neighbourhood. The outlier treatment cut-off values are detailed in Table 14.8.

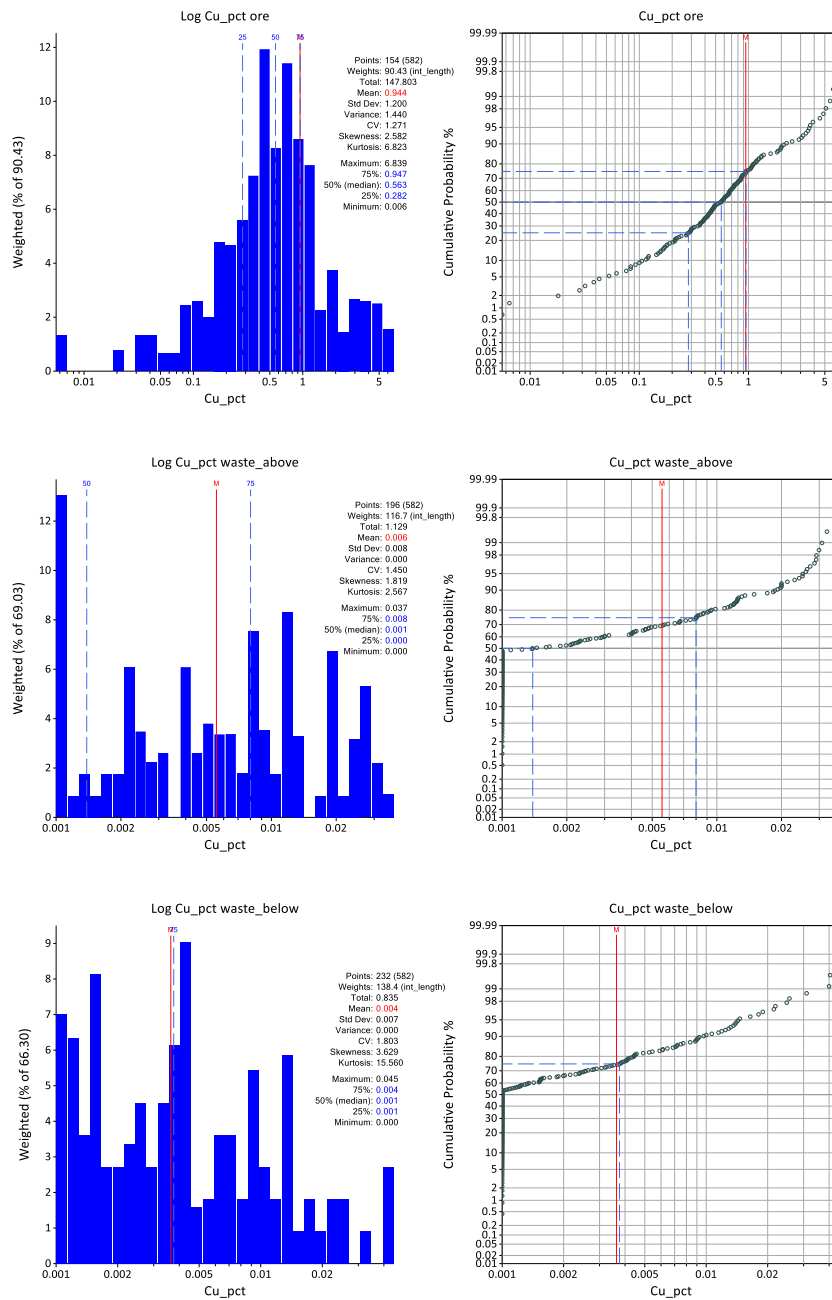
Table 14.8: Outlier Treatment by Grade Domain

Assay	Domain	Capping Grade	Restricted Neighbourhood Grade
Cu %	Ore	-	5.0
Ag ppm	Ore	-	18.0
Pb %	Ore	1.0	0.7
Zn %	Ore	-	0.4

Source: Micon, 2025.

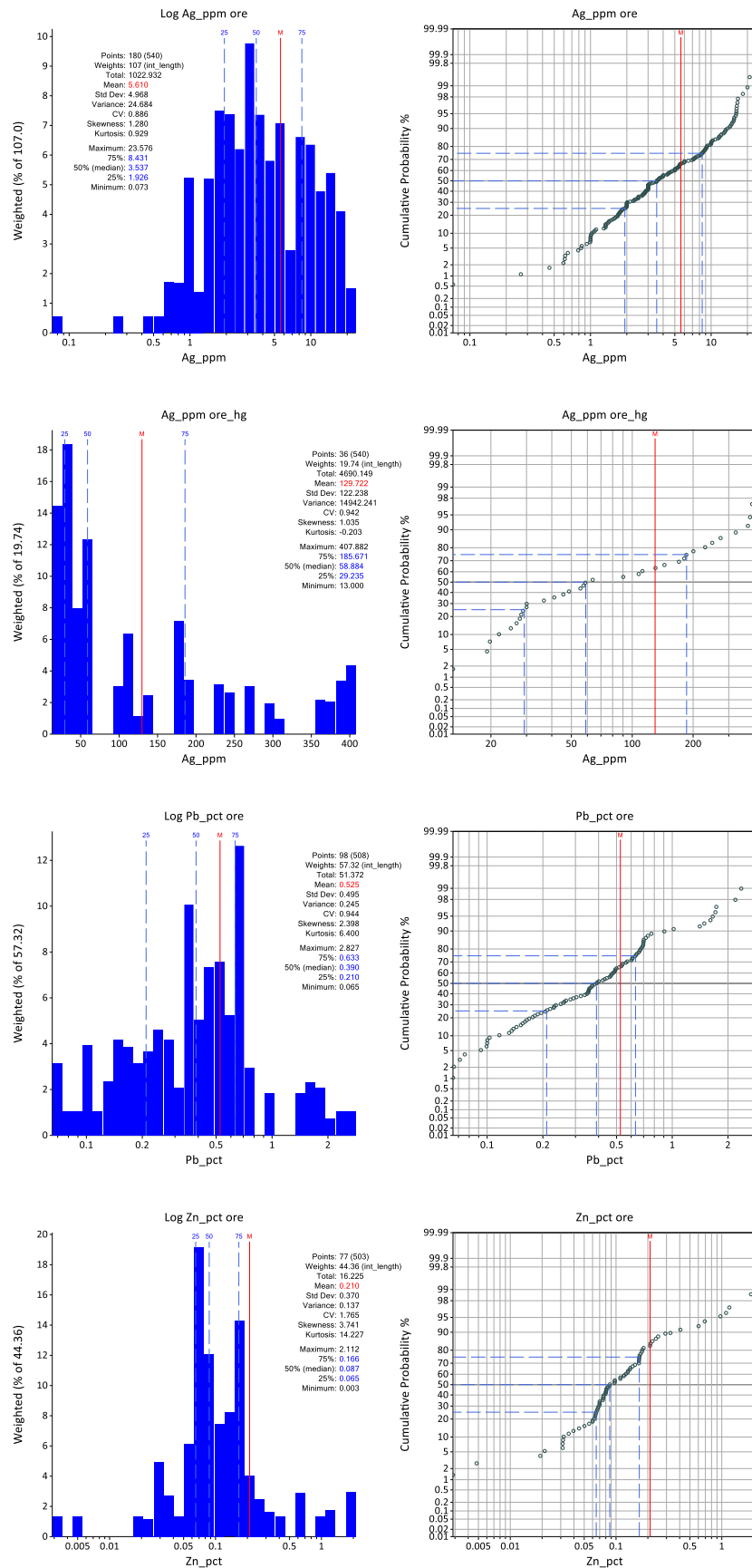
The restricted search neighbourhood was set to 50% of the size of the first pass search ellipse, and assay grades outside of this neighbourhood were capped. As a result, the high grades are only interpolated in the blocks in their vicinity, reducing the risk of smearing any high-grade samples and over estimation.

Figure 14.16: Cu Composite Length Weighted Histograms and Log Probability Plots



Source: Micon, 2025.

Figure 14.17: Ag, Pb, and Zn Composite Length Weighted Histograms and Log Probability Plots



Source: Micon, 2025.

14.5 ESTIMATION DOMAINS

A total of 13 estimation domains with hard boundaries were modelled for the Nowa Sól deposit. These are the ore, waste above and waste below grade domains for Cu, Ag, Pb and Zn, and the additional high-grade Ag grade domain.

14.6 STRATIGRAPHIC COORDINATE TRANSFORMATION

Stratigraphic coordinate transformation involves modifying the spatial coordinates of data points to restore the original continuity of deformed, yet originally flat, geological formations. This process enhances the quantification of spatial continuity and improves the accuracy of geostatistical models by better aligning the coordinate system with geological features (Latifi and Boisvert, 2022).

In structurally complex deposits (where folding and deformation introduce significant non-linearity) the traditional Cartesian coordinate system and Euclidean distance assumptions are inadequate. In such cases, transforming the domain into an alternative, geology-aligned coordinate system facilitates more appropriate use of geostatistical methods such as kriging which rely on variogram-based measures of spatial continuity.

The objective of stratigraphic transformation is to reduce the distortive effects of geological deformation and restore continuity reflective of original depositional settings. Although initially developed for stratified formations, the technique is equally applicable to structurally complex features such as mineralized veins.

The transformation involves modelling bounding surfaces to adjust the spatial coordinates of both sample data and model grids. The original coordinate system (X, Y, Z) is converted to a stratigraphic system (X, Y, Z'), in which geostatistical estimation is carried out. The resulting model is subsequently back-transformed to the original coordinate space.

14.6.1 Forward Transformation

Forward transformation restores original stratigraphic continuity by modifying the elevation of data points using interpreted bounding surfaces. Three transformation types are applied depending on the condition of these surfaces:

Proportional Transformation: Applied when both top and bottom surfaces are intact. Elevations are recalculated based on the relative position of points between these surfaces.

The proportional transformation equation is:

$$Z_{rel} = \frac{Z - Z_{cb}}{Z_{ct} - Z_{cb}} \cdot T$$

Where Z_{rel} is the transformed elevation, Z is the current elevation of the point, and Z_{cb} and Z_{ct} are the elevation of the bottom and top surfaces. T is the average thickness of the layer that can be used to standardise the units of Z_{rel} .

Truncation Transformation: Generally used when the top surface is eroded, but the bottom surface remains intact. Points are flattened relative to the bottom surface and represent vertical distance from this base.

The truncation transformation equation is:

$$Z_{rel} = Z - Z_{cb}$$

Where Z_{rel} is the transformed elevation, Z is the current elevation of the point, and Z_{cb} is the elevation of the bottom surface.

Onlap Transformation: Suitable when the bottom surface is irregular or displaced. Points are flattened relative to the intact top surface and represent distance from it.

The onlap transformation equation is:

$$Z_{rel} = Z - Z_{ct}$$

Where Z_{rel} is the transformed elevation, Z is the current elevation of the point, and Z_{ct} is the elevation of the top surface.

14.6.2 Backward Transformation

Following geostatistical modelling in the stratigraphic coordinate space, data points and model grid cells are transformed back to the original coordinate system. This back-transformation requires continuous bounding surfaces across the domain to ensure the reconstructed geometry aligns with the original spatial configuration between drill holes.

The back transformation formulas are:

Proportional Back Transformation:

$$Z = Z_{cb} + \frac{Z_{rel}}{T} \cdot (Z_{ct} - Z_{cb})$$

Truncation Back Transformation:

$$Z = Z_{rel} + Z_{cb}$$

Onlap Back Transformation:

$$Z = Z_{rel} + Z_{ct}$$

14.6.3 Application to Nowa Sól

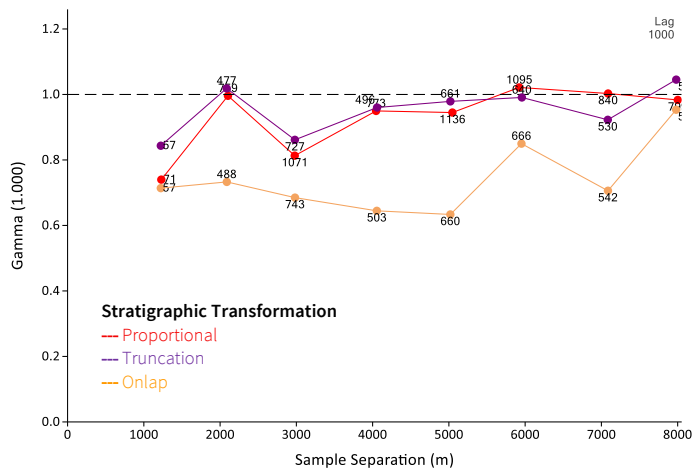
Applying a stratigraphic transformation is an appropriate approach for Nowa Sól considering the laterally continuous mineralization that shows vertical variations in grade. Furthermore, the mineralized host rocks are not flat due to topographic variation at the time of deposition (Top of Rotliegende modelled surface - see Section 9.5) as well as the effects of differential compaction and subsequent structural deformation.

The stratigraphic transformation was applied independently for each grade domain (or layer in this context). For the waste above and waste below domains, the onlap and truncation transformations were applied, respectively. This was because the only bounding surface available were the hangingwall or footwall surface from the ore domain model.

For the ore domains, the appropriate stratigraphic transformation could not be determined from the available geological understanding. As a result, the methodology described by McLennan (2004) was applied. The three transformation types, namely proportional, truncation, and onlap, were each tested. The transformation that resulted in the greatest spatial continuity, as indicated by the most continuous horizontal variogram, was selected as the most appropriate for representing the geometry of the mineralized layer.

An example of the experimental variograms for the copper (Cu) ore grade domain is shown in Figure 14.18. The onlap transformation demonstrated the greatest horizontal spatial continuity and was therefore selected as the most appropriate. This result was consistent across all other ore grade domains. (Table 14.9).

Figure 14.18: Normal Score Experimental Variogram for Cu Domain Stratigraphic Transformations



Source: Micon, 2025.

Table 14.9: Stratigraphic Coordinate Transform Used by Grade Domain

Domain	Transformation
Cu	
Waste Above	Onlap
Ore	Onlap
Waste Below	Truncation
Ag	
Waste Above	Onlap
Ore	Onlap
Ore High Grade	Onlap
Waste Below	Truncation
Pb	
Waste Above	Onlap
Ore	Onlap
Waste Below	Truncation
Zn	
Waste Above	Onlap
Ore	Onlap
Waste Below	Truncation

Source: Micon, 2025.

14.7 VARIOGRAPHY

Variography was performed on transformed data (see Section 14.6). Calculation of the variograms followed the approach for tabular deposits set out in Deutsch (2015). Primarily the dip angle tolerance and vertical bandwidth were restricted for the horizontal variogram to 10° and 2.0 m, preserving the horizontal structures by avoiding crossing stratigraphic layering.

For all domains the data was normal score transformed and the variogram model was fitted on the normal scores data before being back transformed. Variograms were modelled independently for all domains, except for the Ag waste domains which were grouped together due to an insufficient number of samples. Where required the angular tolerance of the semi-major and minor axes was increased to generate a stable experimental variogram as a result of the limited number of sample pairs. The variogram model parameters used in the MRE are shown in Table 14.10 and the experimental variograms and variogram models are shown in Figure 14.19 and Figure 14.20 for the Cu ore grade domain and Figure 14.21 and Figure 14.22 for the Ag ore grade domain.

The modelled variograms all show long range structures in the horizontal plane with short ranges in the vertical plane. This is consistent with the observed down hole grade variability and the lateral continuity of grade and the orebody geometry between drill holes. The ranges of the first structure for the ore grade domains account for 44 to 74% of the variability and range from 2,500 m to 5,500 m; secondary structures extend up to 30,000 m.

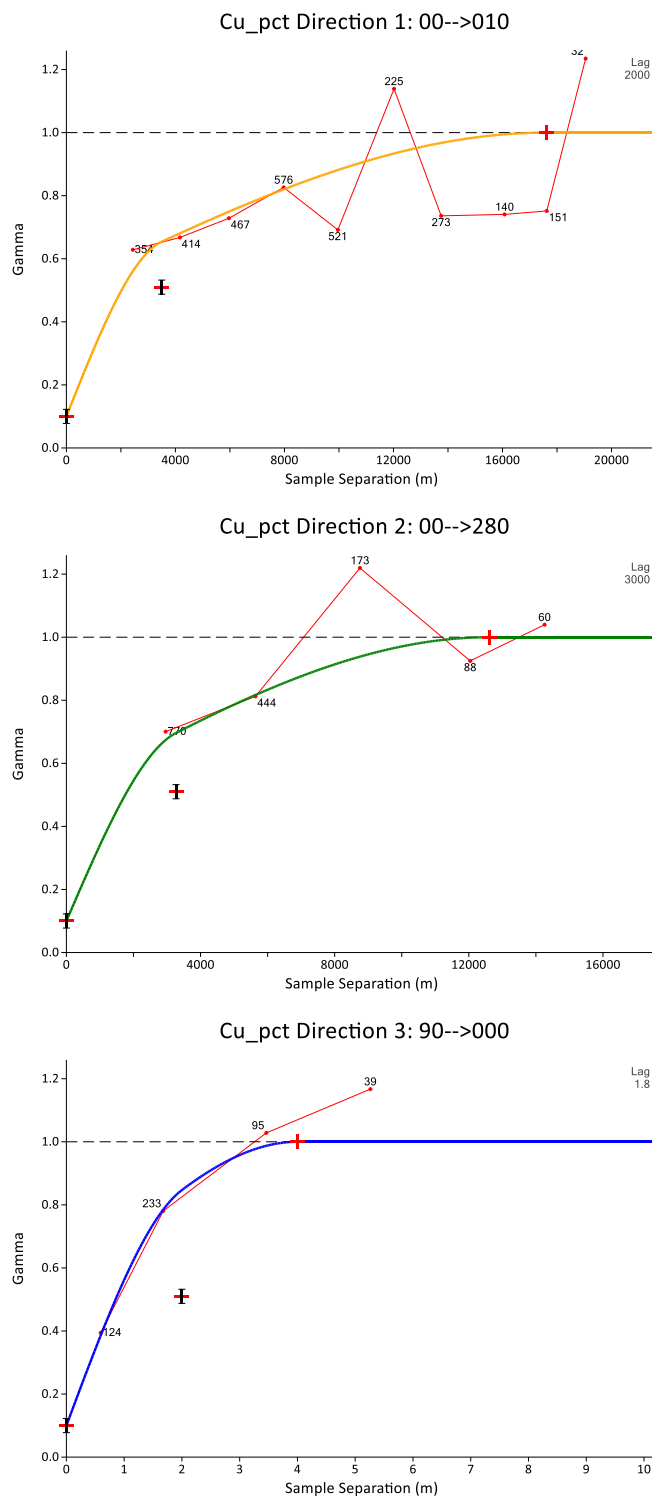
Table 14.10: Variogram Model Parameters

Domain	Directions (°)			Nugget	Structure 1						Structure 2			
	Dip	Dip Azimuth	Pitch		Sill	Type	Range (m)			Sill	Type	Range (m)		
							1	2	3			1	2	3
Cu														
Waste Above	0	0	90	0.07	0.59	Exp	3,500	8,500	7	0.34	Sph	52,300	20,000	8
Ore	0	0	100	0.10	0.53	Sph	3,500	3,270	2	0.37	Sph	17,610	12,610	4
Waste Below	0	0	80	0.08	0.58	Sph	6,800	3,000	2	0.34	Sph	9,470	10,800	25
Ag														
Ore	0	0	60	0.06	0.74	Exp	3,200	2,700	2	0.20	Sph	30,000	35,000	3
Ore High Grade	0	0	130	0.13	0.44	Sph	2,500	2,500	2	0.43	Sph	5,900	5,900	3
Waste Below	0	0	150	0.08	0.59	Sph	2,500	2,500	5	0.33	Exp	7,000	5,500	7
Pb														
Waste Above	0	0	20	0.08	0.58	Sph	4,400	2,600	5	0.34	Sph	15,200	17,000	20
Ore	0	0	70	0.10	0.55	Sph	5,000	4,800	1	0.35	Sph	14,500	11,500	2
Waste Below	0	0	70	0.10	0.56	Sph	3,000	6,500	5	0.34	Sph	14,000	10,000	7
Zn														
Waste Above	0	0	110	0.07	0.59	Exp	6,500	6,400	6	0.34	Sph	15,000	10,300	8
Ore	0	0	140	0.13	0.68	Sph	5,500	5,500	2	0.19	Sph	6,000	6,000	5
Waste Below	0	0	110	0.10	0.66	Exp	4,300	2,000	6	0.24	Sph	20,100	10,300	13

Source: Micon, 2025.

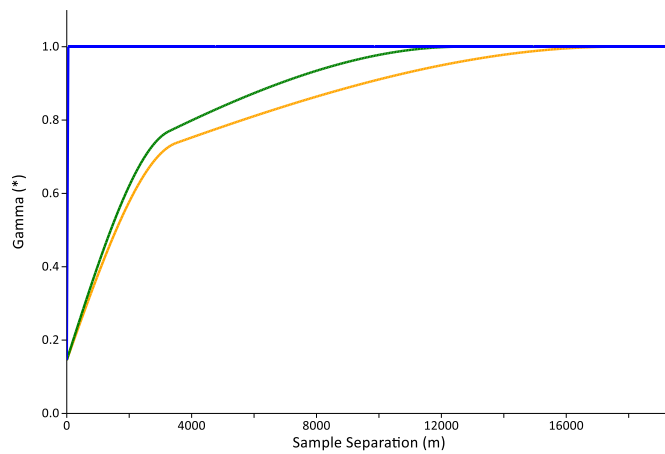
Note: All parameters for back-transformed variogram models. Type: Sph = spherical; Exp = exponential

Figure 14.19: Normal Scores Experimental Variograms and Variogram Models for Cu Ore Grade Domain



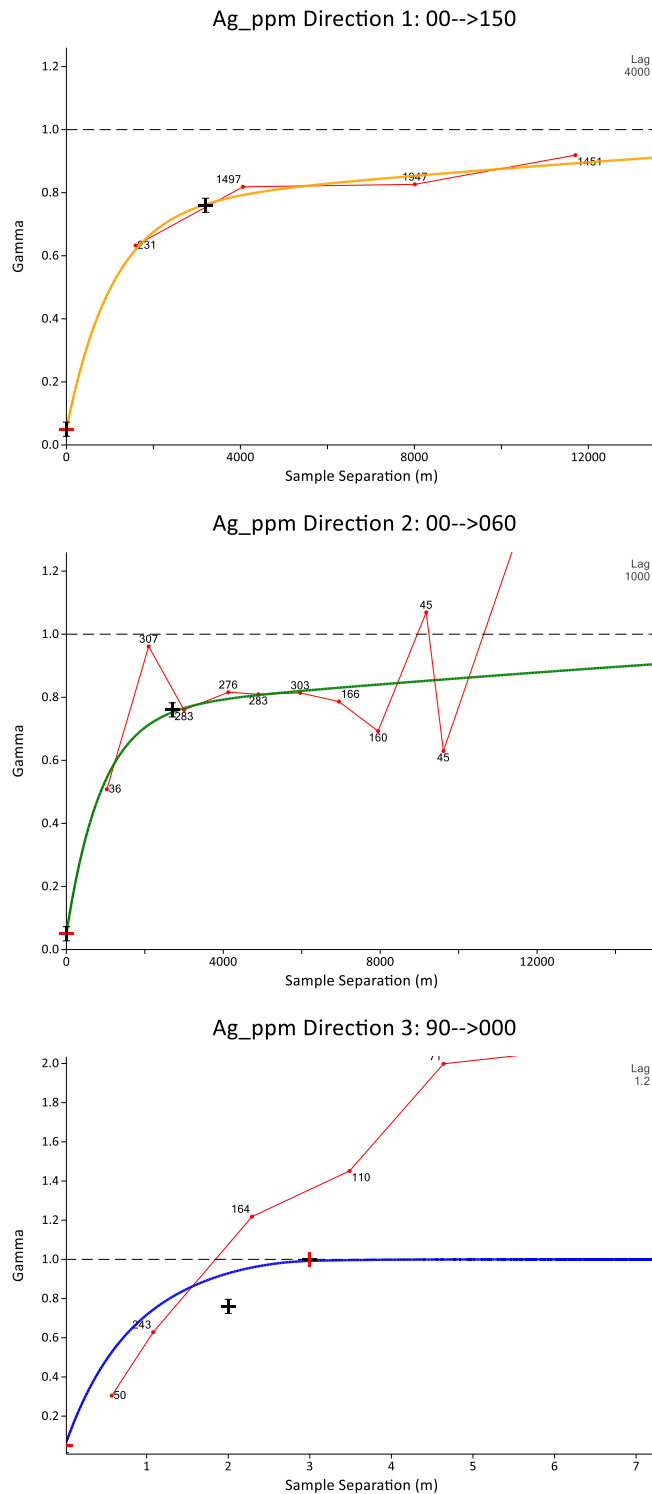
Source: Micon, 2025.

Figure 14.20: Back Transformed Variogram Model for Cu Ore Grade Domain



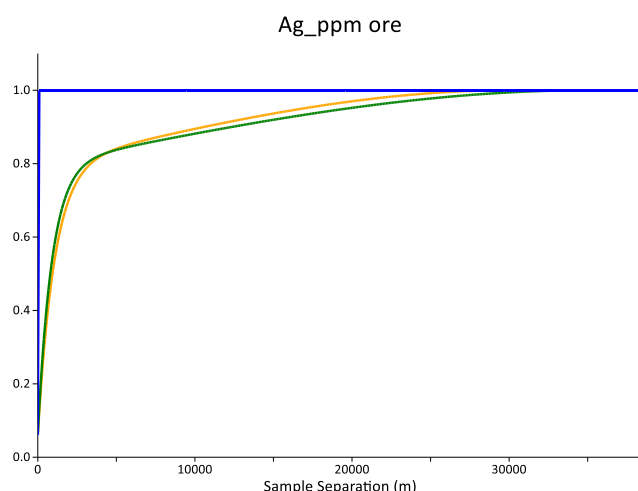
Source: Micon, 2025.

Figure 14.21: Normal Scores Experimental Variograms and Variogram Models for Ag Ore Grade Domain



Source: Micon, 2025.

Figure 14.22: Back Transformed Variogram Model for Ag Ore Grade Domain



Source: Micon, 2025.

14.8 INTERPOLATION

Assay grade data were interpolated using Ordinary Kriging (OK) for all domains. All interpolation was completed with Leapfrog Geo and Leapfrog Edge version 2024.1.0. Inverse distance to the power of two (ID^2) and Nearest Neighbour (NN) interpolation were also performed for comparison.

14.8.1 Stratigraphic Coordinate Transformation

The block grades were interpolated in the stratigraphic transformed space (see Section 14.6) and were subsequently back transformed. The back transformed grades were then assigned to a block model in real-world space using NN interpolation. Figure 14.21 shows the block model and composite data in stratigraphic transformed space and real-world space.

14.8.2 Block Model Description

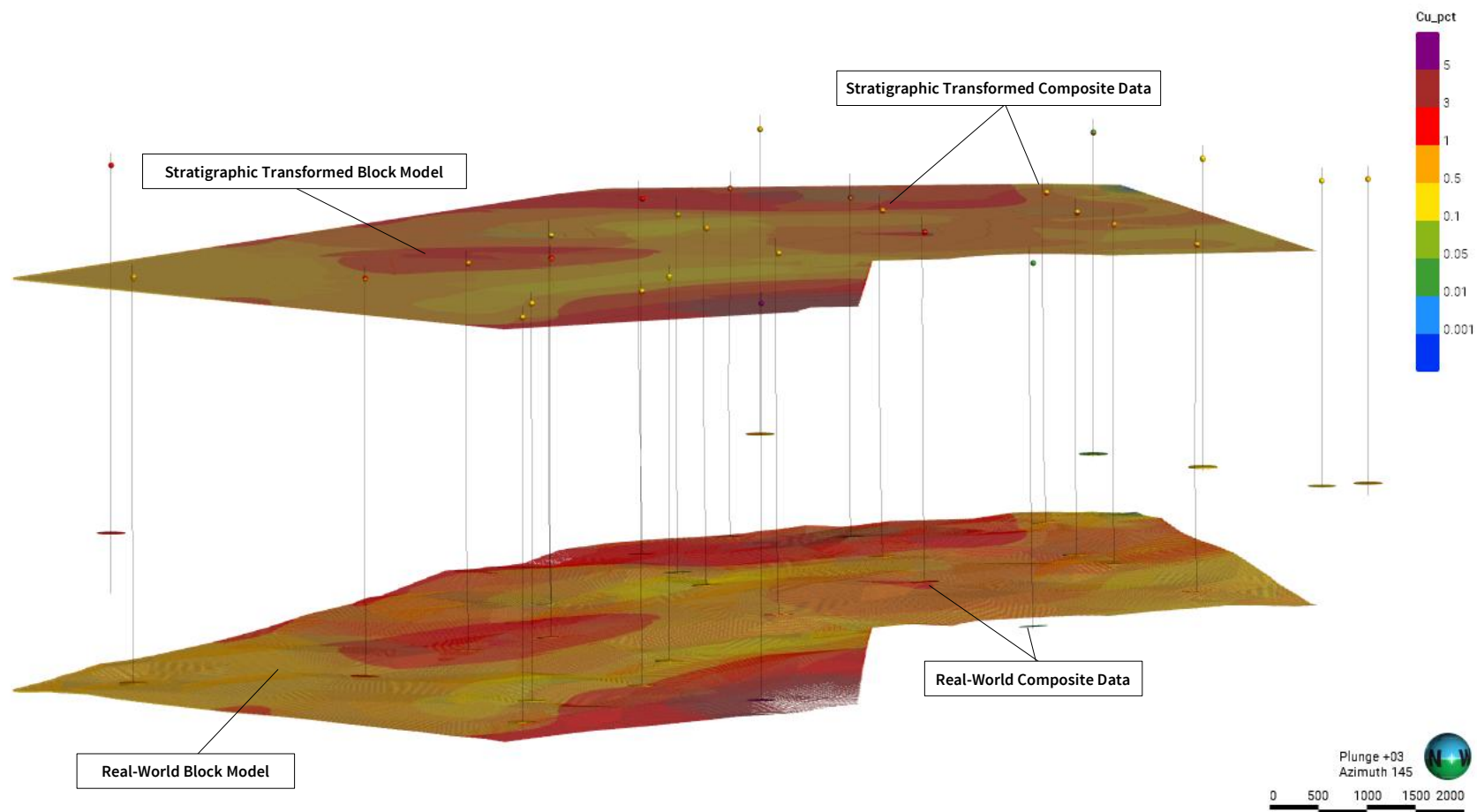
The block model descriptions for the stratigraphic transformed and real-world orientated block models are shown in Table 14.11 and Table 14.12. The block models were constructed with a block size of 20.0 m (X), 20.0 m (Y) and 0.6 m (Z). The block size was chosen to ensure that the geological boundaries of the modelled domains could be accurately represented while ensuring processing times remained reasonable. The vertical block size is also well suited to the operational mining constraints of a minimum mining height of 1.8 m that is utilized at the neighbouring KGHM operation.

Table 14.11: Stratigraphic Transformed Block Model Description

Direction	Parent Block Size (m)	Base Co-Ordinate	Minimum Bounding Co-Ordinate	Number of Parent Blocks
X	20	554,600	554,600	560
Y	20	5,739,400	5,739,400	900
Z	0.6	15	-18.6	56

Source: Micon, 2025.

Figure 14.23: Oblique View to the Northeast of the Stratigraphic Transformed and Real-World Orientated Composites and Block Model



Source: Micon, 2025.

Note: 5x vertical exaggeration.

Table 14.12: Real-World Block Model Description

Direction	Parent Block Size (m)	Base Co-Ordinate	Minimum Bounding Co-Ordinate	Number of Parent Blocks
X	20	554,600	554,600	560
Y	20	5,739,400	5,739,400	900
Z	0.6	-1,502	-2,264	1,270

Source: Micon, 2025.

14.8.2.1 Volume Checks

Validation checks were performed to compare the volumes of the modelled ore grade domain wireframes with the corresponding block model volumes inside the Nowa Sól Concession. This step was undertaken to confirm that the selected block size was appropriate to accurately represent the geometry and extents of the modelled domains. The results are shown in Table 14.13 and illustrate that block model volumes closely match the wireframe volumes.

Table 14.13: Volume Comparison of Ore Grade Domain Wireframes and Block Model

Domain	Wireframe Volume (m ³)	Block Model Volume (m ³)	Percentage Difference (%)
Cu			
Ore	431,380,000	431,397,840	-0.004
Ag			
Ore	516,700,000	516,669,120	0.006
Ore High Grade	82,272,000	82,253,280	0.023
Pb			
Ore	262,820,000	262,656,960	0.062
Zn			
Ore	228,660,000	228,681,600	-0.009

Source: Micon, 2025.

14.8.3 Search Parameters

The domains were estimated with hard boundaries in two passes with increasingly relaxed constraints. For an increasing estimation pass number, the size of the search ellipse was increased, and the minimum number of samples was decreased to ensure all blocks were interpolated. The search ellipse size and anisotropy were related to the modelled variogram for the respective domain. The interpolation search parameters are listed in Table 14.14 and Table 14.15.

Table 14.14: Ordinary Kriging Interpolation Search Parameters for Cu and Ag Domains

Interpolation Pass	Min. Samples	Max. Samples	Samples per Drill Hole	Ellipse Size (m)			Ellipse Orientation (°)			Capping Grade	Restricted Neighbourhood		
				Major	Semi-Major	Minor	Dip	Dip Azimuth	Pitch		Grade Above	% Ellipse Size	Type
Cu Waste Above													
1	24	30	3	5,000	7,000	10	0	0	90	-	-	-	-
2	12	30	3	12,500	17,500	25	0	0	90	-	-	-	-
Cu Ore													
1	24	30	3	7,000	6,500	10	0	0	100	-	5	50	Clamp
2	12	30	3	17,500	16,250	25	0	0	100	-	5	20	Clamp
Cu Waste Below													
1	24	30	3	7,000	5,000	10	0	0	80	-	-	-	-
2	12	30	3	17,500	12,500	25	0	0	80	-	-	-	-
Ag Waste Above													
1	24	30	3	5,000	5,000	10	0	0	150	-	-	-	-
2	12	30	3	12,500	12,500	25	0	0	150	-	-	-	-
Ag Ore													
1	24	30	3	6,000	6,000	10	0	0	60	-	18	50	Clamp
2	12	30	3	15,000	15,000	25	0	0	60	-	18	20	Clamp
Ag Ore High Grade													
1	24	30	3	5,000	5,000	10	0	0	130	-	-	-	-
2	12	30	3	12,500	12,500	25	0	0	130	-	-	-	-
Ag Waste Below													
1	24	30	3	5,000	5,000	10	0	0	150	-	-	-	-
2	12	30	3	12,500	12,500	25	0	0	150	-	-	-	-

Source: Micon, 2025.

Table 14.15: Ordinary Kriging Interpolation Search Parameters for Pb and Zn Domains

Interpolation Pass	Min. Samples	Max. Samples	Samples per Drill Hole	Ellipse Size (m)			Ellipse Orientation (°)			Capping Grade	Restricted Neighbourhood		
				Major	Semi-Major	Minor	Dip	Dip Azimuth	Pitch		Grade Above	% Ellipse Size	Type
Pb Waste Above													
1	24	30	3	7,000	5,000	10	0	0	20	-	-	-	-
2	12	30	3	17,500	12,500	25	0	0	20	-	-	-	-
Pb Ore													
1	24	30	3	5,000	5,000	10	0	0	70	1.0	0.7	50	Clamp
2	12	30	3	12,500	12,500	25	0	0	70	1.0	0.7	20	Clamp
Pb Waste Below													
1	24	30	3	5,000	7,000	10	0	0	70	-	-	-	-
2	12	30	3	12,500	17,500	25	0	0	70	-	-	-	-
Zn Waste Above													
1	24	30	3	6,000	6,000	10	0	0	110	-	-	-	-
2	12	30	3	15,000	15,000	25	0	0	110	-	-	-	-
Zn Ore													
1	24	30	3	5,000	5,000	10	0	0	140	-	0.4	50	Clamp
2	12	30	3	12,500	12,500	25	0	0	140	-	0.4	20	Clamp
Zn Waste Below													
1	24	30	3	6,000	4,000	10	0	0	110	-	-	-	-
2	12	30	3	15,000	10,000	25	0	0	110	-	-	-	-

Source: Micon, 2025.

14.8.4 Interpolation Results

Examples of the interpolation results are shown in Figure 14.22. Cross-sections give a better representation of the grade interpolation due to the strong vertical variation in grade. It is difficult to plot multiple drill holes on a single section due to the difference in scales of the widely spaced drill holes and the flat lying narrow orebody.

14.8.5 Density Assignment

A total of 387 dry density measurements were made on drill core from Lumina drill holes. Histograms of the data showed clear multi-modal populations and a clear correlation between decreasing density and stratigraphic depth. As such density domains were modelled at different density cut-offs as described in Section 14.2.2.

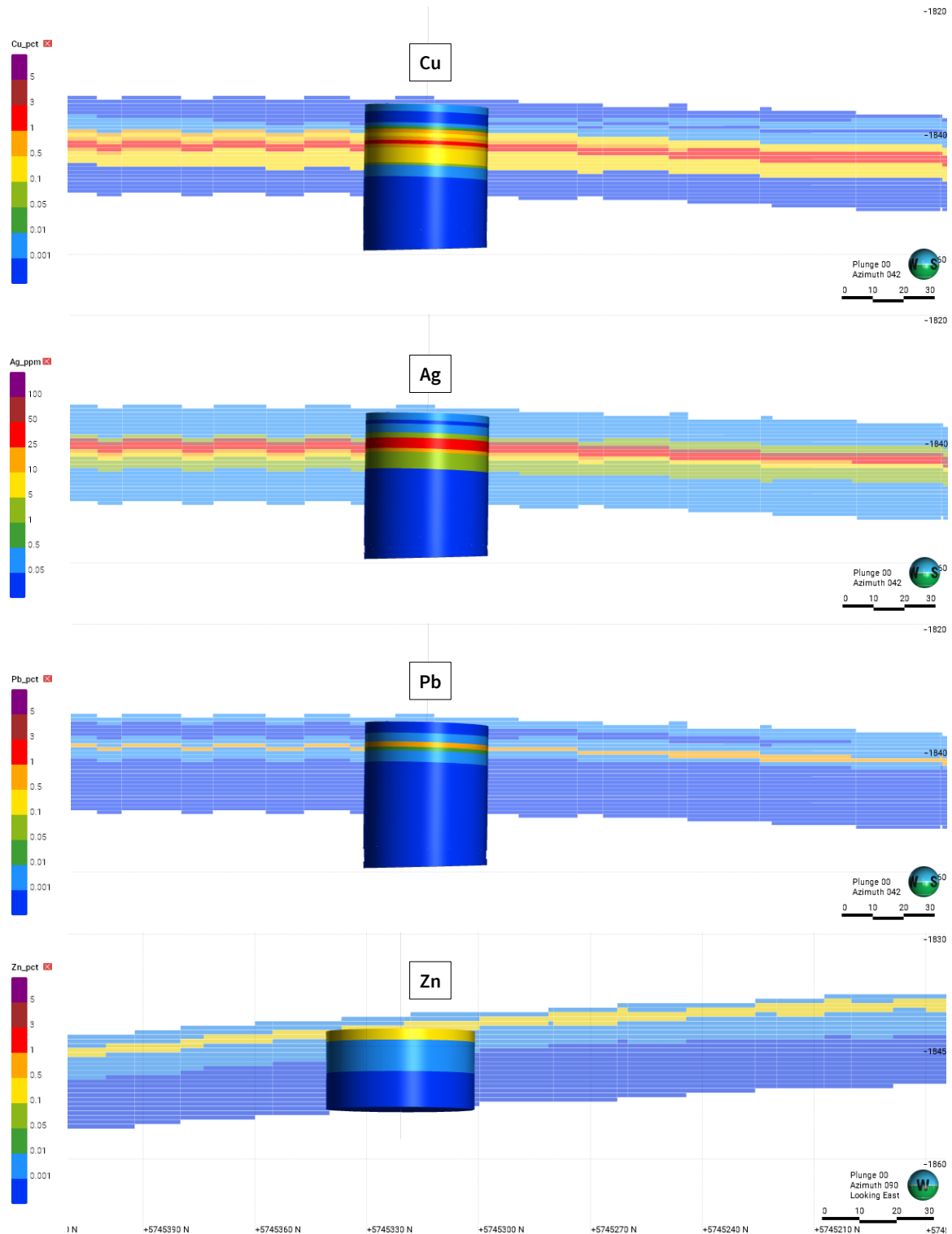
Median density values were calculated for each density domain and were assigned to the block model. The density values are shown in Table 14.16.

Table 14.16: Density Statistics per Density Domain

Domain	Count	Mean	SD	Var.	Min.	Median	Max.	CV
Low	97	2.25	0.11	0.01	1.76	2.25	2.62	0.05
Medium	71	2.53	0.14	0.02	2.06	2.53	2.75	0.06
High	121	2.76	0.07	0.00	2.48	2.76	3.07	0.02
Very High	54	2.93	0.02	0.00	2.86	2.93	2.96	0.01

Source: Micon, 2025.

Figure 14.24: Northwest-Southeast Cross-Sections of NS_C16 Showing the Estimated Grades for Cu, Ag, and Pb, and NS_C25 for Zn



Source: Micon, 2025.

14.9 VALIDATION

To assess the quality of the block model estimate, multiple methods of validation were performed, including visual inspection, statistical comparison, decluster plots, and swath plots. Block model validation was limited to data inside the Nowa Sól license boundary.

14.9.1 Visual Inspection

The different estimation domains were examined in cross-section and the block grade values were compared with the composite assay data. This was performed to check for the misallocation of grade on a local scale. The estimated grades generally match well with the composite assay data as can be seen in Figure 14.22.

14.9.2 Statistical Comparison

Global statistics of the composite assay data and block model grades were compared to verify how well the block model represents the informing composites on a global scale. The statistics are shown in Table 14.17 and examples of histograms of the data are shown in Figure 14.23.

The average block grades compare well with the declustered capped composite grades in the ore grade domains. The shape of the data populations compares well in the histograms with the expected smoothing effect of linear estimation. The results of the statistical comparison indicate that the block model represents the composite data well on a global scale for the ore grade domains. For the waste domains the results are more variable which is in part due to the number of samples close to or at detection limits.

14.9.3 Decluster Plots

The decluster plots compare an Inverse Distance (ID^0) estimate to the power zero of composites within a block on the x-axis versus the kriged block value on the y-axis. An ID^0 interpolant is equivalent to an arithmetic average. Examples of decluster plots are shown in Figure 14.24.

The decluster plots generally show good correlation between the ID^0 estimate and the kriged estimate. Some natural over estimation of the low grades and under estimation of the high grades is evident in the plots. This is to be expected due to the linear method of estimation and the change of support (from point to block) that occurs during grade interpolation.

14.9.4 Swath Plots

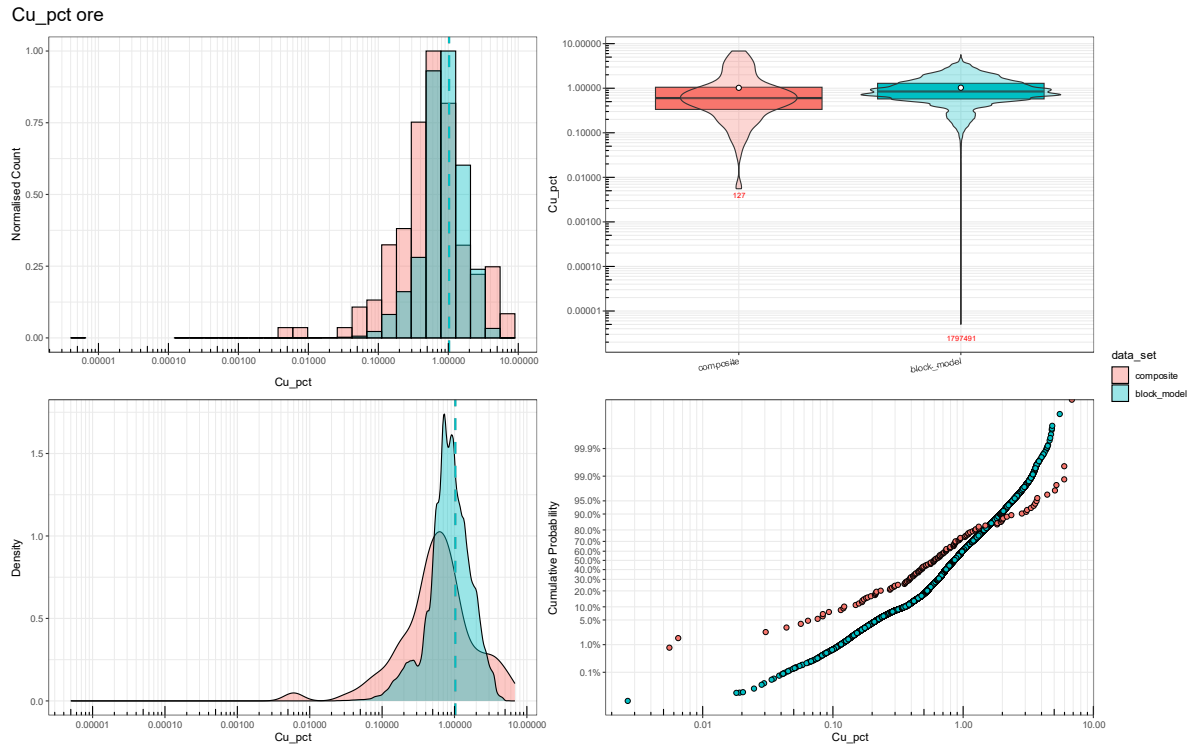
Swath plots of equal dimension slices were created in the X, Y, and Z directions for all the estimation domains to compare the composite data to the block model values. Examples of swath plots are shown in Figure 14.25. The composite data matches well with the OK interpolated block model grades.

Table 14.17: Statistics Comparing Composites and Block Grades

Domain	Count	Mean	SD	Var.	Min.	Median	Max.	CV
Cu % Waste Above								
Composite	130	0.007	0.008	0.000	0.000	0.004	0.037	1.219
Block Model	2,479,342	0.003	0.002	0.000	0.000	0.002	0.023	0.897
Cu % Ore								
Composite	127	1.022	1.254	1.573	0.006	0.600	6.839	1.254
Block Model	1,797,491	1.027	0.675	0.455	0.000	0.855	5.674	0.661
Cu % Waste Below								
Composite	128	0.004	0.007	0.000	0.000	0.001	0.045	1.891
Block Model	2,484,332	0.003	0.002	0.000	0.000	0.002	0.023	0.848
Ag ppm Waste Above								
Composite	53	0.212	0.193	0.037	0.041	0.132	0.815	0.904
Block Model	1,599,595	0.155	0.037	0.001	0.051	0.168	0.342	0.234
Ag ppm Ore								
Composite	157	6.058	5.137	26.393	0.073	3.857	23.576	0.858
Block Model	2,150,475	6.454	2.758	7.605	0.738	6.214	17.226	0.425
Ag ppm Ore High Grade								
Composite	32	135.087	130.343	16989.196	13.000	58.884	407.882	0.952
Block Model	342,722	133.365	70.538	4975.679	26.533	127.058	343.918	0.519
Ag ppm Waste Below								
Composite	144	0.090	0.133	0.018	0.005	0.050	0.855	1.485
Block Model	2,668,244	0.055	0.017	0.000	0.006	0.051	0.260	0.310
Pb % Waste Above								
Composite	83	0.013	0.021	0.000	0.000	0.003	0.090	1.566
Block Model	1,975,389	0.008	0.006	0.000	0.000	0.006	0.045	0.800
Pb % Ore								
Composite	91	0.525	0.493	0.243	0.065	0.395	2.827	0.956
Block Model	1,092,346	0.449	0.147	0.022	0.128	0.439	0.884	0.323
Pb % Waste Below								
Composite	210	0.004	0.010	0.000	0.000	0.000	0.094	2.950
Block Model	3,694,757	0.002	0.004	0.000	0.000	0.001	0.068	1.890
Zn % Waste Above								
Composite	72	0.006	0.008	0.000	0.000	0.002	0.043	1.334
Block Model	1,774,173	0.006	0.003	0.000	0.000	0.006	0.021	0.544
Zn % Ore								
Composite	70	0.207	0.376	0.142	0.003	0.087	2.112	1.816
Block Model	940,463	0.180	0.176	0.031	0.020	0.114	1.405	0.977
Zn % Waste Below								
Composite	238	0.002	0.004	0.000	0.000	0.001	0.034	1.860
Block Model	4,047,792	0.001	0.001	0.000	0.000	0.001	0.017	0.595

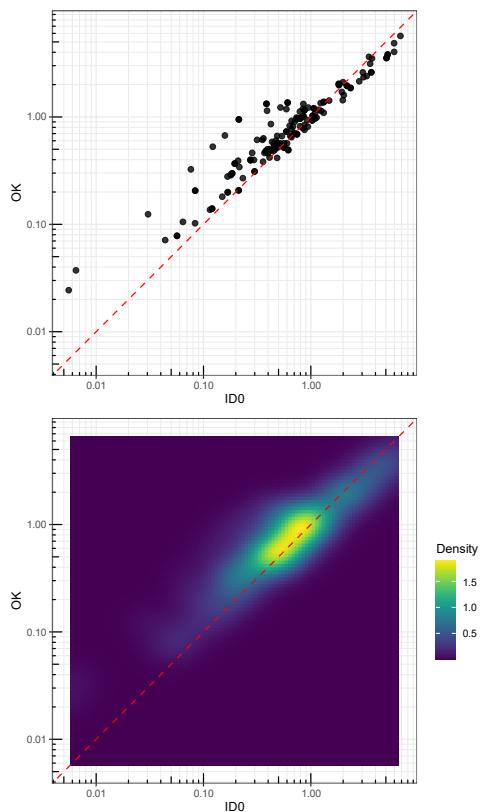
Source: Micon, 2025.

Figure 14.25: Histogram, Box Plot, and Density Plot of Cu Grades in Composites and Blocks for the Ore Grade Domain



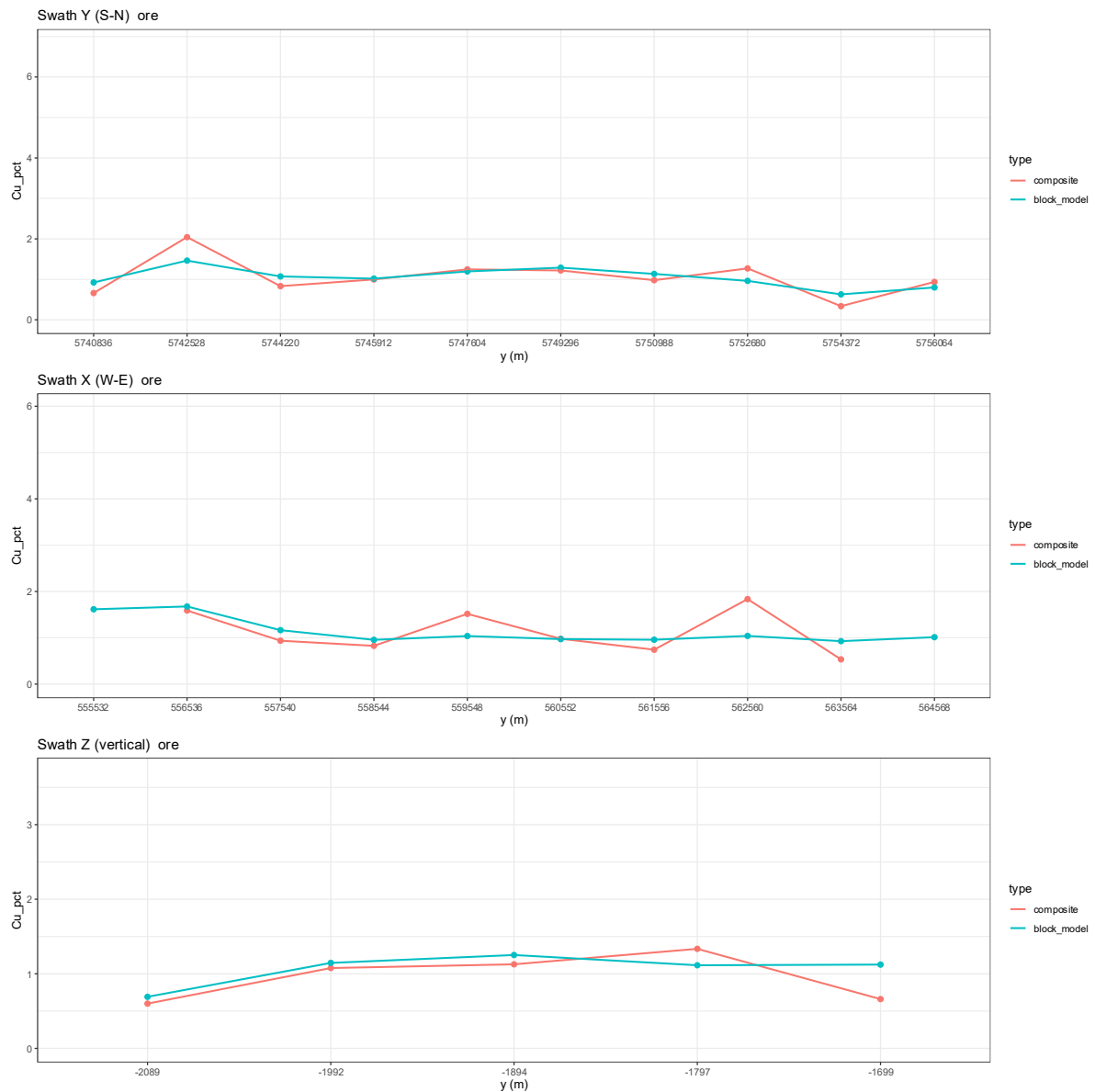
Source: Micon, 2025.

Figure 14.26: Decluster Plot of Cu Grades for the Ore Grade Domain



Source: Micon, 2025.

Figure 14.27: Swath Plots of Cu Grades for the Ore Grade Domain



Source: Micon, 2025.

14.10 MINERAL RESOURCE CLASSIFICATION

The MRE was classified in accordance with National Instrument 43-101 (NI 43-101) Standards of Disclosure for Mineral Projects (CIM, 2014). Mineral Resources were classified as Measured, Indicated, and Inferred.

A measure of data spacing, which is directly linked to the drill hole spacing, was used to inform the Mineral Resource classification. The data spacing thresholds used for Mineral Resource classification were based on the distance to the nearest drill hole. A professional judgement on the appropriate threshold distance was based on numerous factors including the variogram range, confidence in the geological and grade wireframes, the observed geological continuity, and threshold values that gave contiguous areas of blocks with the same Mineral Resource Classification.

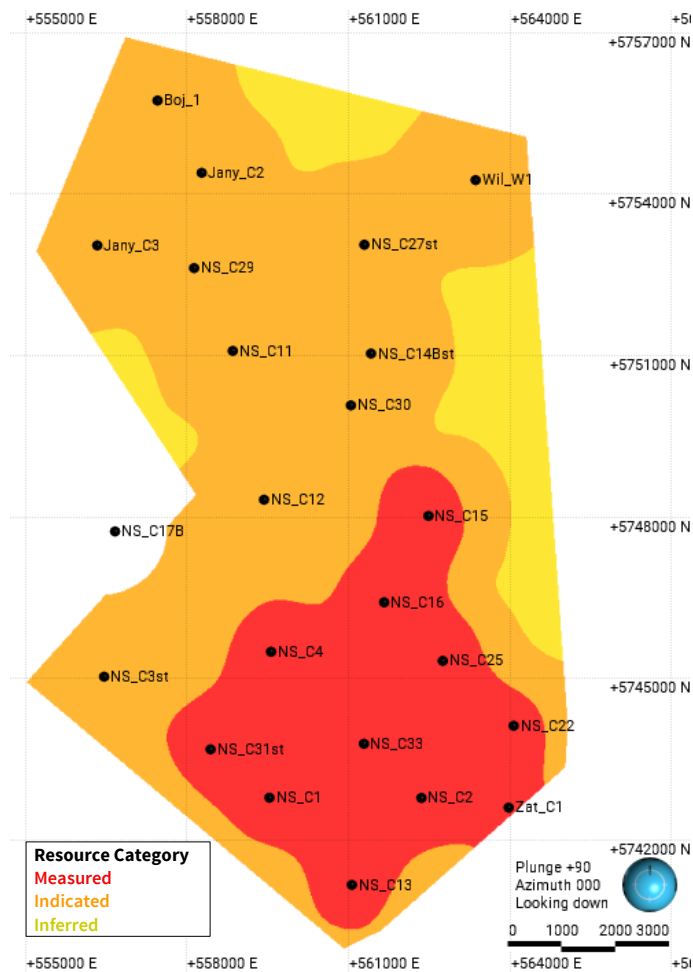
Blocks where the nearest drill hole was > 1,600 m were classified as Inferred, this is limited to areas at the margin of the license where there is the potential for extrapolation of grade data at distance from informing drill hole data. Measured blocks were classified as being < 1,200 m to the nearest drill hole where the drill holes had been drilled on a regular spaced grid, in the south of the license. All other blocks were classified as Indicated.

The classification of Measured Mineral Resources is supported by a high level of geological confidence. The mineralization exhibits strong lateral continuity in both grade and thickness, as demonstrated through detailed lithological logging, consistent assay results, and comprehensive geostatistical analysis. Furthermore, metallurgical testwork and the resultant conceptual process design is based on conventional processing technologies and a proven flowsheet that has been implemented at many other comparable projects, in particular local KGHM mines, which have been successfully operational for 60 years.

Drill hole spacing within the Measured areas ranges from 1,500 m to 2,000 m, which is less than the range of the primary structure in the variogram model, and significantly less the modelled variogram range. This structure accounts for 44% to 74% of the total grade variability within the ore grade domains. The variogram models indicate robust spatial continuity, with ranges that are well supported by the dataset and a relatively low nugget effect.

Interpolation of Measured blocks required a high density of informing samples, including data from a minimum of eight drill holes. The classification is further supported by thorough block model validation using visual checks and statistical comparisons against composited assay data.

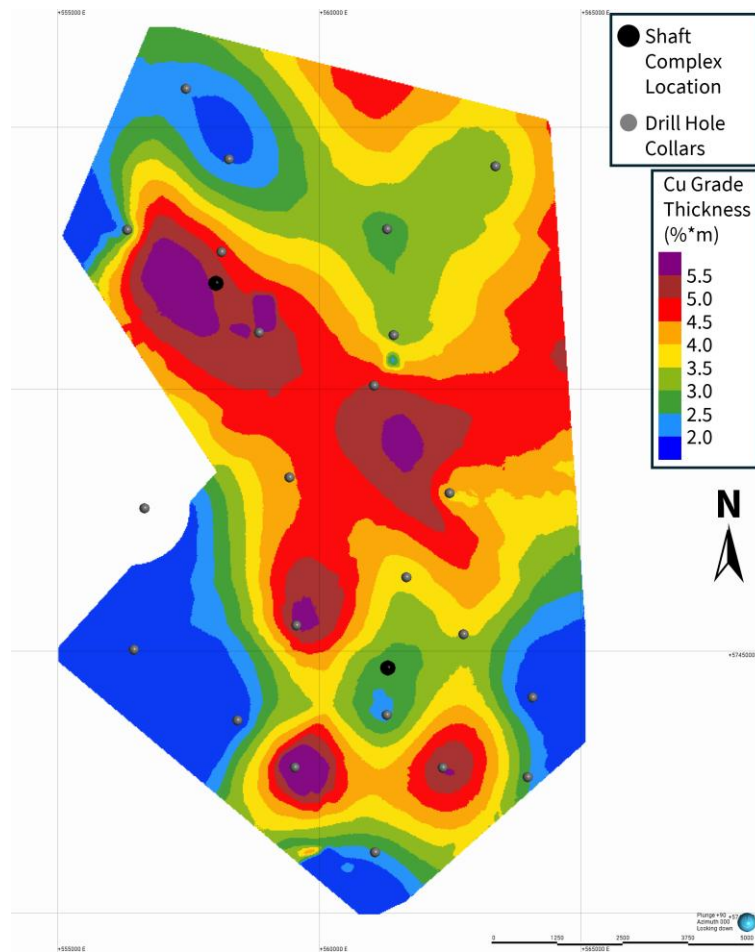
Figure 14.28: Plan View of the Mineral Resource Classification



Source: Micon, 2025.

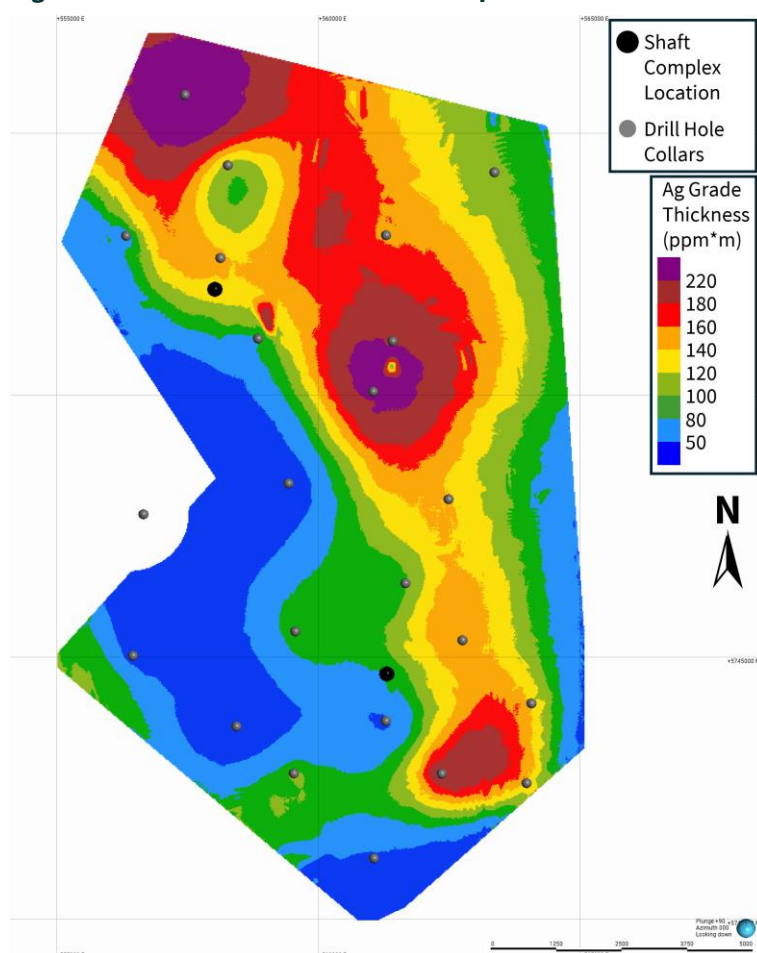
Grade–thickness maps, calculated as grade multiplied by the total thickness of the mineralized interval within the Mineral Resources, are presented for copper and silver in Figure 14.29 and Figure 14.30, respectively.

Figure 14.29: Copper Grade-Thickness Map of the Mineral Resources



Source: Micon, 2026.

Figure 14.30: Silver Grade-Thickness Map of the Mineral Resources



Source: Micon, 2026.

14.11 MINERAL RESOURCE STATEMENT

14.11.1 Reasonable Prospects for Eventual Economic Extraction

By definition, a Mineral Resource has reasonable prospects for eventual economic extraction, or RPEEE (CIM, 2014). Only material that meets RPEEE were considered in the Mineral Resource Statement, assuming inputs derived from metallurgical testwork and the operation, or similar operations.

In accordance with the recommendation of the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines 2019 (CIM, 2019) mining envelopes were defined to identify spatially continuous mineralization within potentially mineable shapes using reasonable assumptions based on neighbouring operations and long-term price trends.

14.11.1.1 Mining Envelope Definition

The objective of the underground envelope definition process is to convert a block model into a practical, mineable shape that reflects both economic and geometric constraints. While block models contain the raw spatial distribution of grade, density, and classification, they do not inherently define what can actually be mined. This process defines a contiguous volume of blocks that meets cut-off criteria, respects minimum mining dimensions, and adheres to geometric rules such as slope, dip, and stope orientation.

Each block in the model is evaluated against defined economic criteria, typically a cut-off grade or net value. Blocks that meet the criterion are the starting point for the algorithm to expand outward grouping adjacent blocks into mining envelopes. The process ensures the resulting shapes reflect realistic mining selectivity, aiming to prevent isolated, un-mineable pockets of ore and enforcing minimum stope sizes. This step alone significantly improves the practical relevance of the resulting shapes, especially in deposits with highly variable grade distributions.

The process utilises multiple optimization algorithms which determine how marginal blocks at the envelope boundary are handled. These modes enable engineers to adjust the aggressiveness of envelope growth, striking a balance between ore recovery and dilution and waste inclusion.

The result is a flexible system that can be adapted to different mining philosophies, from conservative geotechnical designs to aggressive economic scenarios.

The starting point for the process was the definition of a target head grade for the mineable envelope. A pure pay limit calculation for copper results in a 0.70% Cu grade. The calculation is detailed in Table 14.18.

The metal prices were based on consensus price forecasts, and conservative values slightly less than the predicted range were selected (S & P Global, 2025). Process recoveries were taken from the neighbouring KGHM operation as were the smelter terms. The information was taken from publicly available documents (KGHM, 2025a, KGHM, 2025b) and supported by the knowledge of Lumina personnel. The recoveries are comparable with those reported in Section 13.4.1 from metallurgical testwork. The testwork completion postdated the mining envelope definition for RPEEE.

The operational costs are based on the neighbouring KGHM operation and benchmarks from other relevant mines with some adjustments for efficiency savings such as the use of diesel rather than coal power and mining automation to reduce labour costs.

Applying the calculated pay limit to the model will not define what can be mined because the block model defines the raw spatial distribution of grade and would not result in a minable shape. Further work was required to determine the optimal cut-off, which provided guidance for the mining envelope definition. The same economic input parameters were used to calculate the optimal cut-off.

The optimal cut-off calculation determines the Net Present Value (NPV) values for each cut-off value using the grade tonnage curve as the basis of the tonnes and grade contained at each cut-off. The grade tonnage curve showing the tonnes and copper grade is illustrated Figure 14.31 and the optimal cut-off graph in Figure 14.32. The maximum NPV value is achieved at a cut-off of 1.0% Cu. This value is an optimistic, or theoretical value calculated without capital, royalties and taxes. Following design and full Discounted Cash Flow (DCF) analysis, these values will not match. The mineable tonnes in the model at this cut-off are detailed in Table 14.19.

Table 14.18: Pay Limit Calculation

Description	Unit	Value
Production		
Production Rate	t/d	32,000
Production Days per Month	Days	30
Production Rate	t/m	960,000
Production Rate	t/a	11,520,000
Copper Grade	%	1.22%
Silver Grade	g/t Ag	36.82
Lead Grade	%	0.15%
Zinc Grade	%	0.06%
Processing		
Copper Processing Recovery	%	89.0%
Silver Processing Recovery	%	86.0%
Lead Processing Recovery	%	80.0%
Concentrate Grade	% (Cu/Pb)	22.0%
Copper Concentrate Produced	t (Cu Conc)	0.049
Copper in Cu Concentrate	t Cu	0.011
Cost		
Mining Cost	US\$/RoM t	54.50
Overheads	US\$/RoM t	5.50
Processing	US\$/RoM t	8.50
Total Cost	US\$/ RoM t	68.50
Transport to Smelter	US\$/t conc	10.00
Copper Treatment Charge	US\$/t Cu	50.00
Copper Refining Charge	US\$/lb (Cu/Pb)	0.05
Silver Refining Charge	US\$/oz	0.40
Silver Refining Charge	US\$/kg	12.86
Price		
Copper Metal Price	US\$/lb	4.75
Silver Metal Price	US\$/oz	37.50
Lead Metal Price	US\$/lb	1.00
Zinc Metal Price	US\$/lb	1.25
Copper Metal Price	US\$/t	10,472
Silver Metal Price	US\$/kg	1,205.65
Lead Metal Price	US\$/t	2,205
Zinc Metal Price	US\$/t	2,756
Conversion lb per kg	lb	2.2047
Revenue		
Payability	%	96.5%
Copper in Concentrate		
Copper Price	US\$/t (Cu Conc.)	2,303.9
Copper Price (incl. Payability)	US\$/t (Cu Conc.)	2,223.2
Copper Treatment Charge	US\$/t (Cu Conc.)	50.00
Refining Charge	US\$/t (Cu Conc.)	22.73
Copper Price (Paid)	US\$/t (Cu Conc.)	2,150.5
Transport to Smelter	US\$/t (Cu Conc.)	10.0
Copper Price (FOB Mine)	US\$/t (Cu Conc.)	2,140.5
Copper Price (FOB Mine)	US\$/t Cu	9,729.6
Pay Limit		
Cu Pay Limit	Cu% in RoM t	0.70%

Source: Lumina, 2025.

Figure 14.31: Tonnage Grade Curve

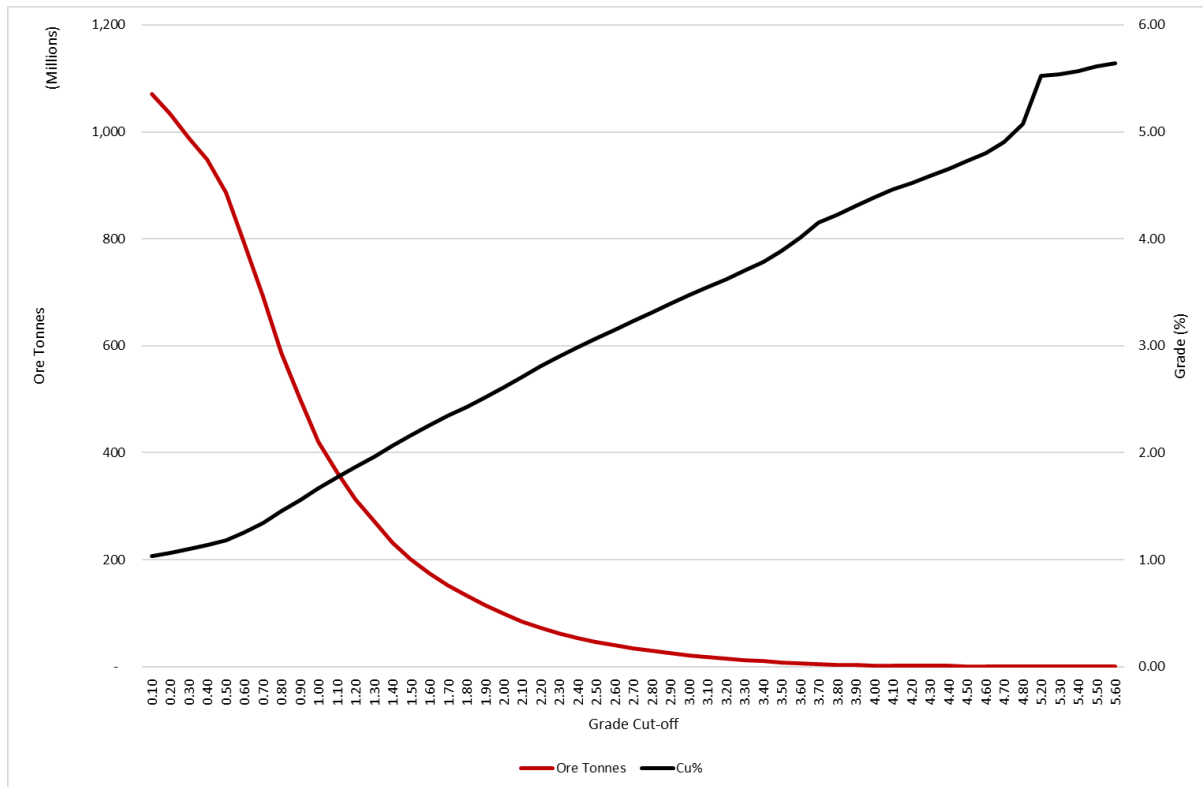


Figure 14.32: Optimal Cut-Off

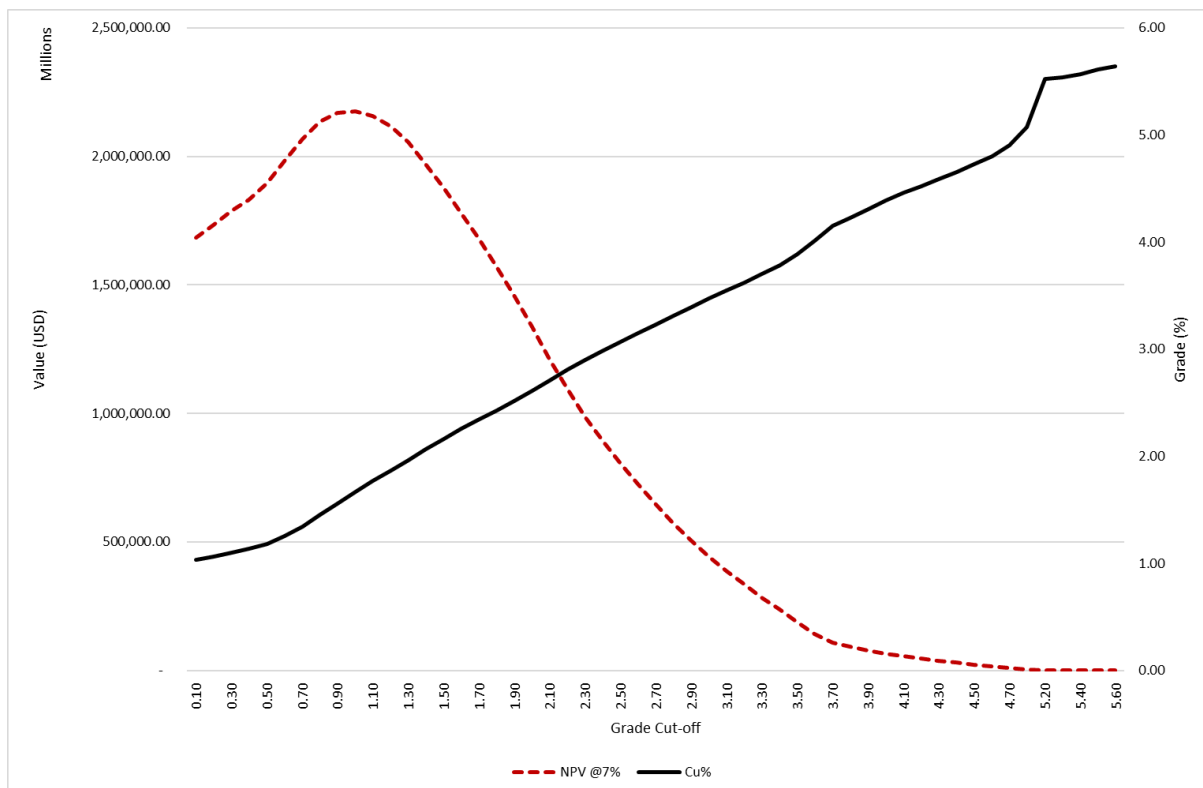


Table 14.19: Optimal Cut-Off Calculation Result (Cut-Off 1.0 % Cu)

Description	Unit	Value
Total Tonnes	t	421,001,518
Mined Tonnes	t	274,071,988
Copper Mined Grade	Cu%	1.669
Silver Mined Grade	g/t Ag	49.768
Lead Mined Grade	Pb%	0.113
Zinc Mined Grade	Zn%	0.030
Mined Grade - NSR/t	US\$/t	162.0
LoM	Yrs	23.8

Source: Micon, 2026.

The optimal value graph shows that selecting a cut-off between 0.1% Cu and 1.0% Cu yields mined grades from 1.03% to 1.67% Cu, which should be targeted when defining the mining envelope.

Understanding that the entire range results in positive values, the target head grade was selected at 0.95% Cu to ensure we included maximum mineable material into the mining envelopes. Also, ensuring the low-grade material included in the envelopes is greater than the pay limit. Blocks classified as Measured, Indicated, and Inferred were considered in the optimization using Datamine Studio OP.

A minimum mining height of 1.8 m was applied with a room width of 6.0 m. No additional dilution from the footwall or hanging wall was applied in the optimization, as it is assumed that any dilution can be effectively contained within the defined minimum mining height. Mining envelopes were generated across the entire license area. A cross-section of the mining envelope is shown in Figure 14.33.

The mining envelopes were used to constrain the block model and identify material that meets RPEEE and were considered for inclusion in the Mineral Resource Statement.

14.11.1.2 Mining Loss Factor

A mining loss factor of 15% was applied to the reported Mineral Resources inside the defined mining envelopes. Applying a percentage loss to an MRE is justified by practical, unavoidable physical constraints during extraction. This adjustment accounts for material that is proven to exist geologically, but cannot be physically recovered in the actual mining operation and therefore does not have RPEEE.

Unavoidable mining losses may include but are not limited to sterilisation of material due to geotechnical barriers around major faults or critical infrastructure and pillars and support that will not be mined and processed. The applied factor reflects intrinsic physical limitations of extraction rather than geological uncertainty or detailed mine design and has been applied uniformly across the Mineral Resource estimate.

14.11.2 Statement

The 2026 MRE was prepared by Micon and reviewed throughout the process and approved by Mrs. Liz de Klerk, Pr.Sci.Nat., QMR and Dr. Ryan Langdon, Ph.D, CGeol, as the QPs. The Mineral Resource statement has an effective date of January 9, 2026. A cut-off grade of 0.95% Cu was used to define mining envelopes and diluted tonnages and grades are reported based on these mineable shapes (Table 14.20). The Mineral Resources are further subdivided and reported by area in Table 14.21.

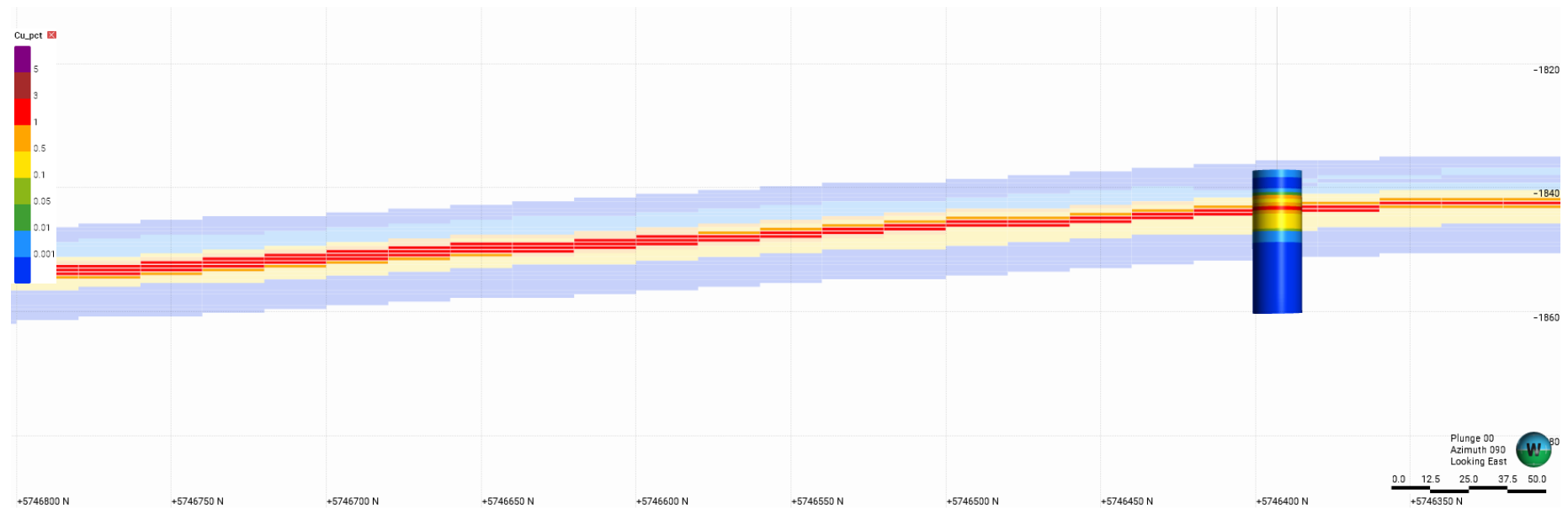
14.11.2.1 Copper Equivalent Calculation

A copper equivalent (Cu_{eq} %) grade was calculated for reporting of the Mineral Resources using the following formula and input parameters:

$$\frac{(Cu_{\%} \times 2204.62 \times Cu_{rec} \times Cu_{price}) + (Ag_{ppm} \times 0.032 \times Ag_{rec} \times Ag_{price})}{2204.62 \times Cu_{rec} \times Cu_{price}}$$

Where $Cu_{\%}$ is the Cu grade in %, Cu_{rec} is the Cu recovery in %, Cu_{price} is the Cu price in US\$/lb, Ag_{ppm} is the Ag grade in ppm, Ag_{rec} is the Ag recovery in %, and Ag_{price} is the Ag price in US\$/oz. The recovery values and prices are listed in Table 14.18.

Figure 14.33: North-South Cross-Section of NS_C16 Showing the Estimated Grades for Cu and the Blocks Included in the Mining Envelope Shaded Darker



Source: Micon, 2025.

Note: 2x vertical exaggeration.

Table 14.20: Nowa Sól Mineral Resource Statement, Effective Date January 9, 2026

Resource Category	Tonnage (kt)	Grade					Contained Metal			
		Cu (%)	Ag (g/t)	Pb (%)	Zn (%)	Cu _{eq} (%)	Cu (kt)	Ag (Moz)	Pb (kt)	Zn (kt)
Measured	183,991	1.37	34.53	0.03	0.00	1.75	2,526.8	204.2	59.4	6.9
Indicated	419,943	1.18	39.93	0.18	0.07	1.62	4,972.1	539.2	760.8	302.4
Measured & Indicated	603,934	1.24	38.29	0.14	0.05	1.67	7,498.9	743.4	820.2	309.4
Inferred	111,855	1.08	28.91	0.25	0.09	1.40	1,206.7	104.0	284.6	99.5

Source: Micon, 2026.

Notes:

- The MRE has been prepared in accordance with NI 43-101 with an effective date of January 9, 2026. Mrs. Liz de Klerk, Pr.Sci.Nat., QMR and Dr. Ryan Langdon, Ph.D, CGeol, of Micon are the QPs responsible for the MRE.
- Cu_{eq} was calculated using the formula $(Cu\% \times 2204.62 \times Cu_{rec} \times Cu_{price}) + (Ag_{ppm} \times 0.032 \times Ag_{rec} \times Ag_{price}) / 2204.62 \times Cu_{rec} \times Cu_{price}$. Where % = Cu grade, ppm = Ag grade, rec = recovery, price = price.
- To RPEEE, mining envelopes used to identify spatially continuous mineralization within potentially mineable shapes using reasonable assumptions based on neighbouring operations and long-term price trends to calculate a pay limit copper grade. These include metal prices of US\$/lb 4.75 for Cu and process recoveries of 89%, payability of 96.5%, transport charges of US\$/wmt 10, treatment charges of US\$/t 50, and refining charges of US\$/t 0.05 for Cu. Operational expenditure costs (OPEX) were determined with reference to comparable operations with some deductions for efficiency savings. These include costs for mining of US\$/t 54.5, processing of US\$/t 8.5, and General and Administrative (G&A) of US\$/t 5.5.
- A target head grade was selected at 0.95% Cu to generate the mining envelopes and was based on the optimal cut-off grade to maximise a theoretical NPV.
- For the mining envelopes a minimum mining height of 1.8 m was used, with a room width of 6.0 m. No additional dilution from the footwall or hanging wall was applied.
- A mining loss factor of 15% was applied to the reported Mineral Resources inside the defined mining envelopes. This adjustment accounts for material that is proven to exist geologically, but cannot be physically recovered in the actual mining operation and therefore does not have RPEEE.
- Diluted tonnages and grades are reported based on the defined mining envelopes. Contained Pb and Zn inside these shapes with RPEEE were included in the Mineral Resources.
- Mineral Resources are not Mineral Reserves and have not demonstrated economic viability. There is no certainty that all or any part of the estimated Mineral Resources will be converted into Mineral Reserves.
- Grade interpolation was by Ordinary Kriging (OK) with a block size of 20 m (X) by 20 m (Y) by 0.6 m (Z). A stratigraphic transform was applied to the data to restore the original continuity, and interpolation was performed in this space before back transforming to real-world space.
- Density domains were modelled, and median density values were assigned: low density 2.25 t/m³, medium density 2.53 t/m³, high density 2.76 t/m³, and very high density 2.93 t/m³.
- The MRE was classified in accordance with NI 43-101 (CIM, 2014). Mineral Resources were classified as Measured, Indicated, and Inferred. Blocks where the nearest drill hole was >1,600 m were classified as Inferred, Measured blocks were classified as being <1,200 m to the nearest drill hole where the drill holes had been drilled on a regular spaced grid, all other blocks were classified as Indicated.
- The totals presented in this table reported from the Mineral Resource models, are subject to rounding, and may not total exactly.

Table 14.21: Nowa Sól Mineral Resource Statement by Area, Effective Date January 9, 2026

Area	Resource Category	Tonnage (kt)	Grade					Contained Metal			
			Cu (%)	Ag (g/t)	Pb (%)	Zn (%)	Cu _{eq} (%)	Cu (kt)	Ag (Moz)	Pb (kt)	Zn (kt)
North	Measured	-	-	-	-	-	-	-	-	-	-
	Indicated	289,010	1.19	47.20	0.25	0.103	1.72	3,446.8	438.6	718.0	297.5
	Measured & Indicated	289,010	1.19	47.20	0.25	0.103	1.72	3,446.8	438.6	718.0	297.5
	Inferred	75,143	0.95	28.18	0.28	0.125	1.26	712.8	68.1	213.6	94.1
South	Measured	183,991	1.37	34.53	0.03	0.004	1.76	2,526.8	204.2	59.4	6.9
	Indicated	130,933	1.16	23.89	0.03	0.004	1.43	1,525.3	100.6	42.8	5.0
	Measured & Indicated	314,924	1.29	30.10	0.03	0.004	1.62	4,052.1	304.8	102.2	11.9
	Inferred	36,712	1.35	30.39	0.19	0.015	1.68	494.0	35.9	71.0	5.4

Source: Micon, 2026.

Notes: See Table 14.20 for Notes related to this table.

14.11.3 Sensitivity Analysis

A sensitivity analysis was conducted by generating grade tonnage data at a variety of cut-offs within the mining envelopes used to constrain the Mineral Resources. As with the Mineral Resources a mining loss factor of 15% was applied to account for material that is proven to exist geologically, but cannot be physically recovered in the actual mining operation.

It was considered appropriate to evaluate varying copper cut-off grades for material constrained by the mining envelopes, as mineralization within the orebody is generally stratigraphically controlled. Higher copper grades occur within the central portion of the orebody, with lower grades developed toward the upper and lower margins. Increasing the copper cut-off grade progressively excludes lower-grade material from these marginal zones. No minimum mining widths have been applied in these cut-off grade calculations.

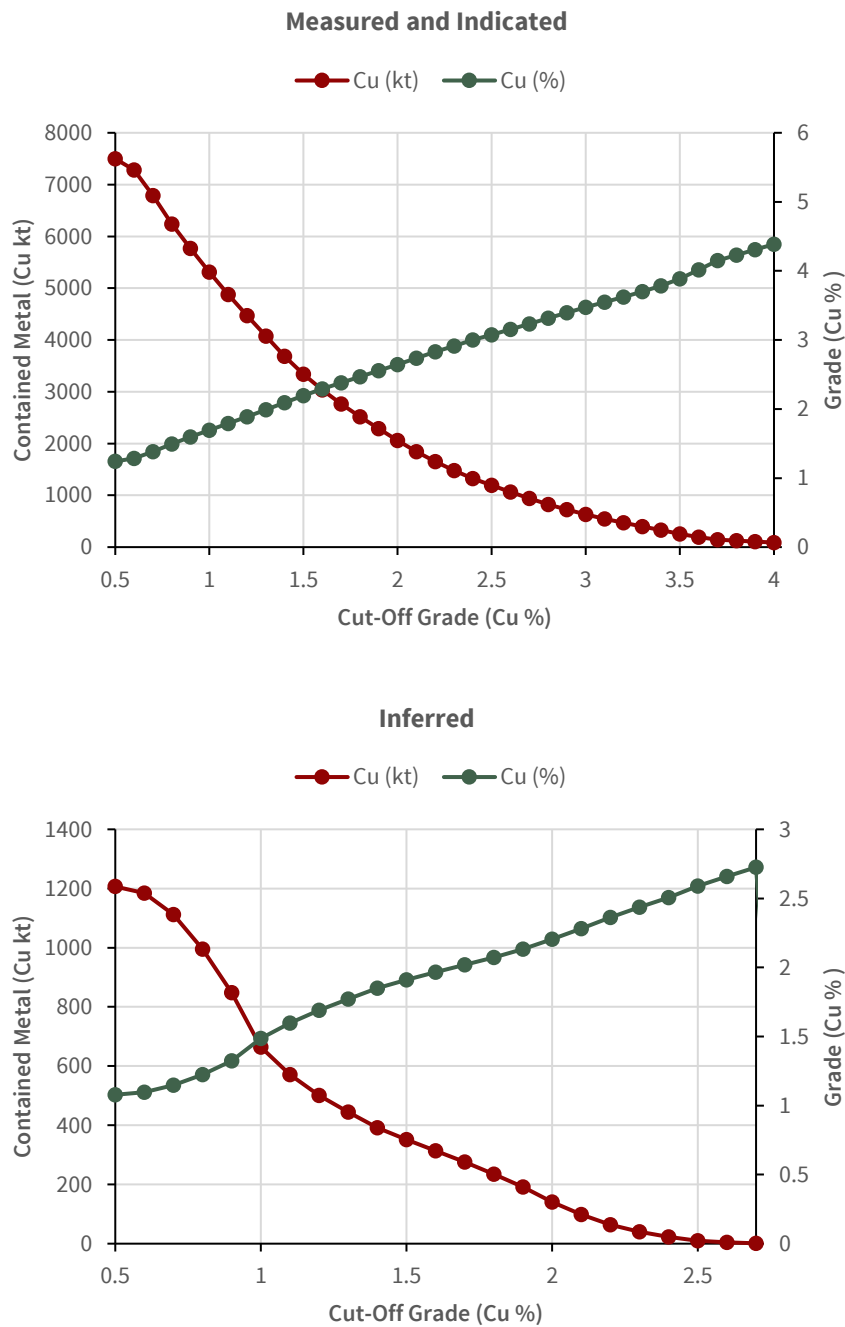
The results of the sensitivity analysis are shown in Table 14.22 and Table 14.23, and graphically in Figure 14.34.

The reported quantities and grade estimates at different cut-off grades are presented for the sole purpose of demonstrating the sensitivity of the Mineral Resource model to varying Cu % cut-off grades. The reader should be cautioned that the results of the sensitivity analysis should not be interpreted as a Mineral Resource statement.

14.11.4 Factors that May Affect the Mineral Resource

As of the Report effective date, the Qualified Person responsible for the MRE, is not aware of any known current environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that might materially affect the MRE that are not discussed in this Report. However, these factors could impact the Mineral Resources and if any such risk materialise the affected areas must be re-evaluated to confirm changes in the Mineral Resources.

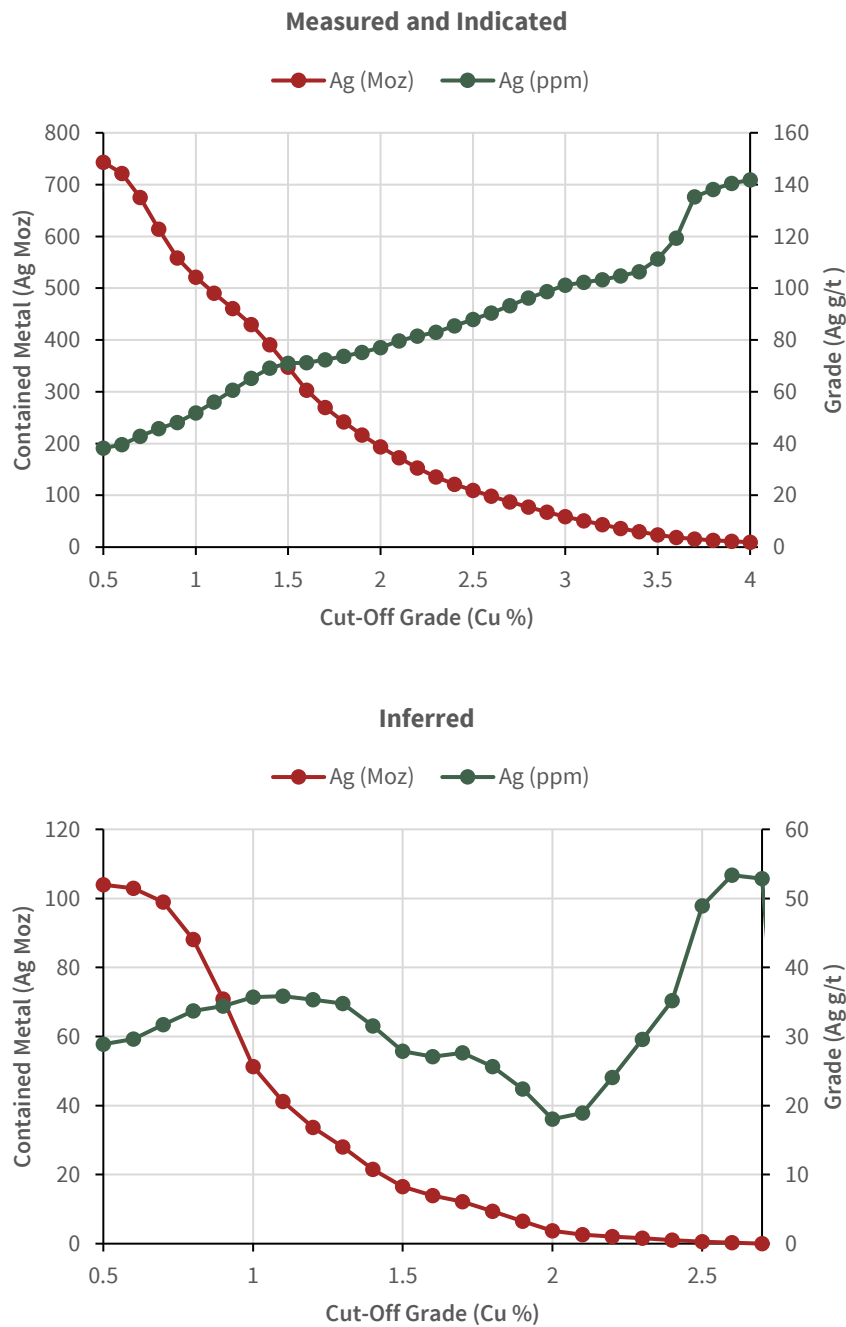
Figure 14.34: Copper Grade Tonnage Graphs for the Sensitivity Analysis



Source: Micon, 2026.

Note: The reader should be cautioned that the results of the sensitivity analysis should not be interpreted as a Mineral Resource statement. The reported quantities and grade estimates at different cut-off grades are presented for the sole purpose of demonstrating the sensitivity of the Mineral Resource model to varying Cu % cut-off grades.

Figure 14.35: Silver Grade Tonnage Graphs for the Sensitivity Analysis



Source: Micon, 2026.

Note: The reader should be cautioned that the results of the sensitivity analysis should not be interpreted as a Mineral Resource statement. The reported quantities and grade estimates at different cut-off grades are presented for the sole purpose of demonstrating the sensitivity of the Mineral Resource model to varying Cu % cut-off grades.

Table 14.22: Nowa Sól Sensitivity Analysis Results for Measured and Indicated Mineral Resources

Cu % Cut-Off	Tonnage (kt)	Measured and Indicated								
		Grade					Contained Metal			
		Cu (%)	Ag (g/t)	Pb (%)	Zn (%)	Cu _{eq} (%)	Cu (kt)	Ag (Moz)	Pb (kt)	Zn (kt)
0.4	603,934	1.24	38.29	0.14	0.05	1.67	7,498.9	743.4	820.2	309.4
0.6	565,911	1.29	39.64	0.13	0.05	1.73	7,280.5	721.3	753.1	294.1
0.8	417,195	1.50	45.77	0.12	0.05	2.00	6,240.4	613.9	480.2	193.9
1.0	313,147	1.70	51.78	0.11	0.03	2.27	5,308.5	521.3	329.9	95.6
1.2	236,516	1.89	60.55	0.10	0.03	2.56	4,468.9	460.5	247.9	61.1
1.4	175,942	2.09	69.08	0.10	0.03	2.86	3,683.0	390.7	183.7	45.0
1.6	132,463	2.29	71.19	0.11	0.02	3.08	3,033.5	303.2	143.9	30.9
1.8	102,094	2.47	73.68	0.12	0.02	3.29	2,518.8	241.8	119.1	24.4
2.0	78,048	2.64	77.08	0.13	0.03	3.50	2,062.7	193.4	98.2	20.1
2.2	58,406	2.83	81.49	0.13	0.03	3.73	1,651.5	153.0	78.3	16.9
2.4	44,196	3.00	85.42	0.14	0.03	3.95	1,325.3	121.4	62.3	14.2

Source: Micon, 2026.

Note: The reader should be cautioned that the results of the sensitivity analysis should not be interpreted as a Mineral Resource statement. The reported quantities and grade estimates at different cut-off grades are presented for the sole purpose of demonstrating the sensitivity of the Mineral Resource model to varying Cu % cut-off grades.

Table 14.23: Nowa Sól Sensitivity Analysis Results for Inferred Mineral Resources

Sensitivity Scenario (%)	Tonnage (kt)	Inferred								
		Grade					Contained Metal			
		Cu (%)	Ag (g/t)	Pb (%)	Zn (%)	Cu _{eq} (%)	Cu (kt)	Ag (Moz)	Pb (kt)	Zn (kt)
0.4	111,855	1.08	28.91	0.25	0.09	1.40	1,206.7	104.0	284.6	99.5
0.6	107,996	1.10	29.63	0.26	0.09	1.43	1,184.5	102.9	283.0	99.4
0.8	81,332	1.22	33.69	0.26	0.10	1.60	995.2	88.1	214.8	82.0
1.0	44,704	1.49	35.68	0.17	0.03	1.88	664.5	51.3	75.4	11.3
1.2	29,658	1.69	35.33	0.08	0.00	2.08	501.2	33.7	24.2	1.1
1.4	21,202	1.85	31.51	0.02	0.00	2.20	392.0	21.5	3.6	0.3
1.6	15,945	1.96	27.08	0.00	0.00	2.27	313.3	13.9	0.6	0.2
1.8	11,320	2.07	25.62	0.00	0.00	2.36	234.7	9.3	0.4	0.2
2.0	6,390	2.21	18.04	0.00	0.00	2.41	140.9	3.7	0.2	0.1
2.2	2,687	2.36	24.07	0.00	0.00	2.63	63.5	2.1	0.0	0.0
2.4	924	2.50	35.18	0.00	0.00	2.90	23.1	1.0	0.0	0.0

Source: Micon, 2026.

Note: The reader should be cautioned that the results of the sensitivity analysis should not be interpreted as a Mineral Resource statement. The reported quantities and grade estimates at different cut-off grades are presented for the sole purpose of demonstrating the sensitivity of the Mineral Resource model to varying Cu % cut-off grades.

15 MINERAL RESERVE ESTIMATES

Lumina has not classified any Mineral Reserve Estimates for the Property.

16 MINING METHODS

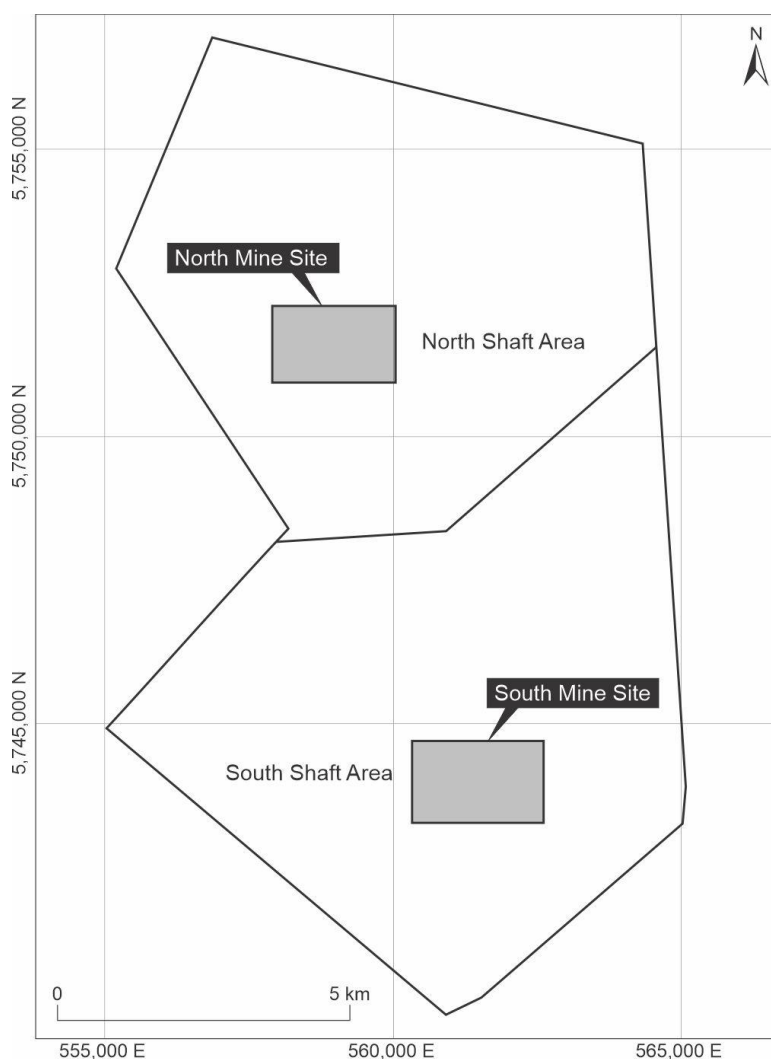
16.1 UNDERGROUND MINING INTRODUCTION

The Nowa Sól underground mine will be accessed by vertical shafts from two separate locations referred to as the North and South Shaft Complexes. These shafts will run as standalone mines with no interdependencies, both shafts targeting production of 11.52 Mt/a. The Project mining areas are illustrated in Figure 16.1.

For the purposes of this PEA a base case Room and Pillar mining method has been selected as this draws on existing knowledge and proven extraction methodology that neighbouring KGHM have been using for six decades to mine the Kupferschiefer. A Drift and Fill mining methodology was also considered at a high level as part of a trade-off study between the two methods and this method is discussed in Section 24.

A geotechnical assessment was carried out to support the selection and preliminary design of underground mining methods for the PEA. The geotechnical work has been completed at a conceptual level appropriate for a PEA and is based on limited geological, geotechnical, and hydrogeological data. A full Geotechnical Analysis was carried out (Global Mine Design, 2025) and a summary of this can be found in Section 16.1.3.

Figure 16.1: Project Area



Source: Micon, 2026.

16.1.1 Mining Envelope Definition

The objective of the underground envelope definition process is to convert a block model into a practical, mineable shape that reflects both economic and geometric constraints. While block models contain the raw spatial distribution of grade, density, and classification, they do not inherently define what can actually be mined. This process defines a contiguous volume of blocks that meets cut-off criteria, respects minimum mining dimensions, and adheres to geometric rules such as slope, dip, and stope orientation.

Each block in the model is evaluated against defined economic criteria, typically a cut-off grade or net value. Blocks that meet the criterion are the starting point for the algorithm to expand outward grouping adjacent blocks into mining envelopes. The process ensures the resulting shapes reflect realistic mining selectivity, aiming to prevent isolated, un-mineable pockets of ore and enforcing minimum stope sizes. This step alone significantly improves the practical relevance of the resulting shapes, especially in deposits with highly variable grade distributions.

The process utilises multiple optimization algorithms which determine how marginal blocks at the envelope boundary are handled. These models enable engineers to adjust the aggressiveness of envelope growth, striking a balance between ore recovery and dilution and waste inclusion.

The result is a flexible system that can be adapted to different mining philosophies, from conservative geotechnical designs to aggressive economic scenarios.

The starting point for the process is the definition of the target head grade for the mineable envelope. A pure pay limit calculation for copper results in a 0.70% Cu grade. The calculation is detailed in Table 16.1, which uses preliminary process recoveries that were benchmarked prior to attaining original metallurgical results for Nowa Sól. The difference between the copper and silver recoveries on the pay limit calculation is negligible and not considered to be material, though should be updated for future studies.

Applying the calculated pay limit to the model will not define what can be mined because the block model defines the raw spatial distribution of grade and would not result in a mineable shape. Further work was required to determine the optimal cut-off, which will provide guidance for the mining envelope definition. The same economic input parameters will be used to calculate the optimal cut-off grade.

The optimal cut-off grade calculation determines the values for each cut-off value using the grade tonnage curve as the basis of the tonnes and grade contained at each cut-off. The grade tonnage curve showing the tonnes and copper grade is illustrated in Figure 16.2.

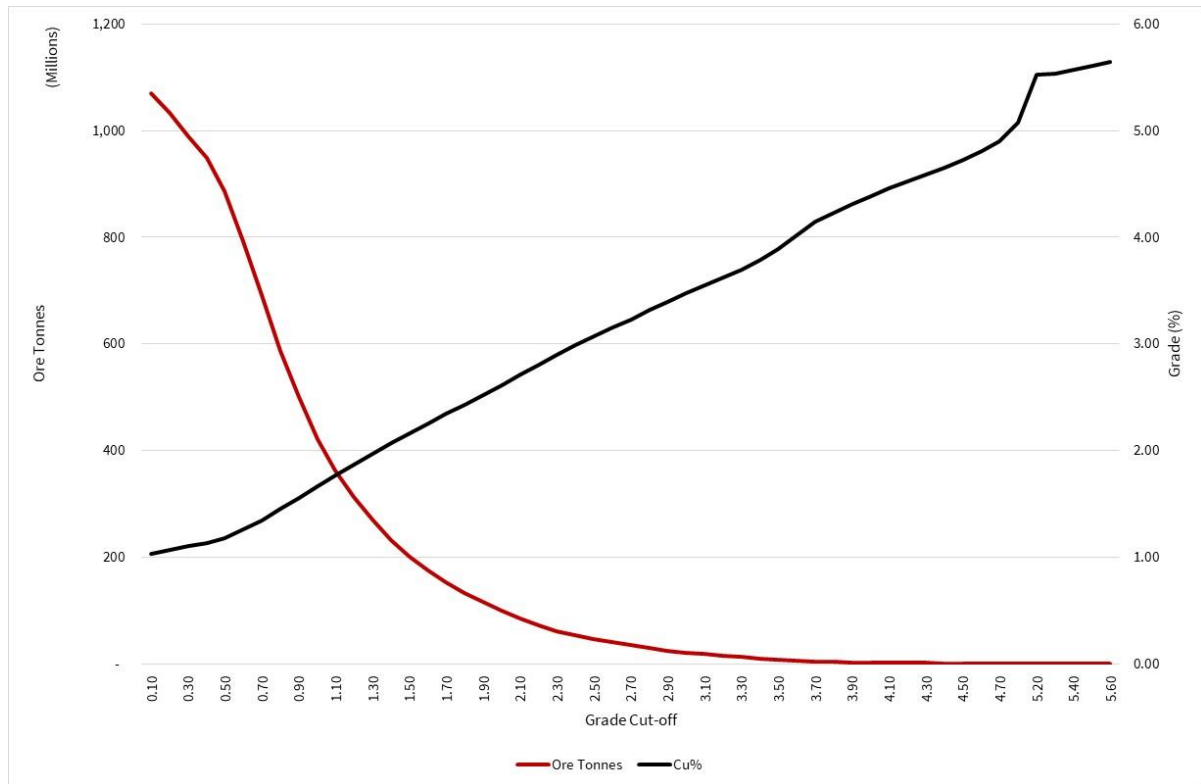
The optimal cut-off graph is detailed in Figure 16.3. The maximum value is achieved at a cut-off of 1.0% Cu. This value is an optimistic, or theoretical value calculated without capital, royalties and taxes. Following design and a full DCF analysis, these values will not match.

Table 16.1: Pay Limit Calculation

Description	Unit	Value
Production		
Production Rate	t/d	32,000
Production Days per Month	Days	30
Production Rate	t/m	960,000
Production Rate	t/a	11,520,000
Copper Grade	%	1.22%
Silver Grade	g/t Ag	36.82
Lead Grade	%	0.15%
Zinc Grade	%	0.06%
Processing		
Copper Processing Recovery	%	89.0%
Silver Processing Recovery	%	86.0%
Lead Processing Recovery	%	80.0%
Concentrate Grade	% (Cu/Pb)	22.0%
Copper Concentrate Produced	t (Cu Conc)	0.049
Copper in Cu Concentrate	t Cu	0.011
Cost		
Mining Cost	US\$/RoM t	54.50
Overheads	US\$/RoM t	5.50
Processing	US\$/RoM t	8.50
Total Cost	US\$/ RoM t	68.50
Transport to Smelter	US\$/t conc	10.00
Copper Treatment Charge	US\$/t Cu	50.00
Copper Refining Charge	US\$/lb (Cu/Pb)	0.05
Silver Refining Charge	US\$/oz	0.40
Silver Refining Charge	US\$/kg	12.86
Price		
Copper Metal Price	US\$/lb	4.75
Silver Metal Price	US\$/oz	37.50
Lead Metal Price	US\$/lb	1.00
Zinc Metal Price	US\$/lb	1.25
Copper Metal Price	US\$/t	10,472
Silver Metal Price	US\$/kg	1,205.65
Lead Metal Price	US\$/t	2,205
Zinc Metal Price	US\$/t	2,756
Conversion lb per kg	lb	2.2047
Revenue		
Payability	%	96.5%
Copper in Concentrate		
Copper Price	US\$/t (Cu Conc.)	2,303.9
Copper Price (incl. Payability)	US\$/t (Cu Conc.)	2,223.2
Copper Treatment Charge	US\$/t (Cu Conc.)	50.00
Refining Charge	US\$/t (Cu Conc.)	22.73
Copper Price (Paid)	US\$/t (Cu Conc.)	2,150.5
Transport to Smelter	US\$/t (Cu Conc.)	10.0
Copper Price (FOB Mine)	US\$/t (Cu Conc.)	2,140.5
Copper Price (FOB Mine)	US\$/t Cu	9,729.6
Pay Limit		
Cu Pay Limit	Cu% in RoM t	0.70%

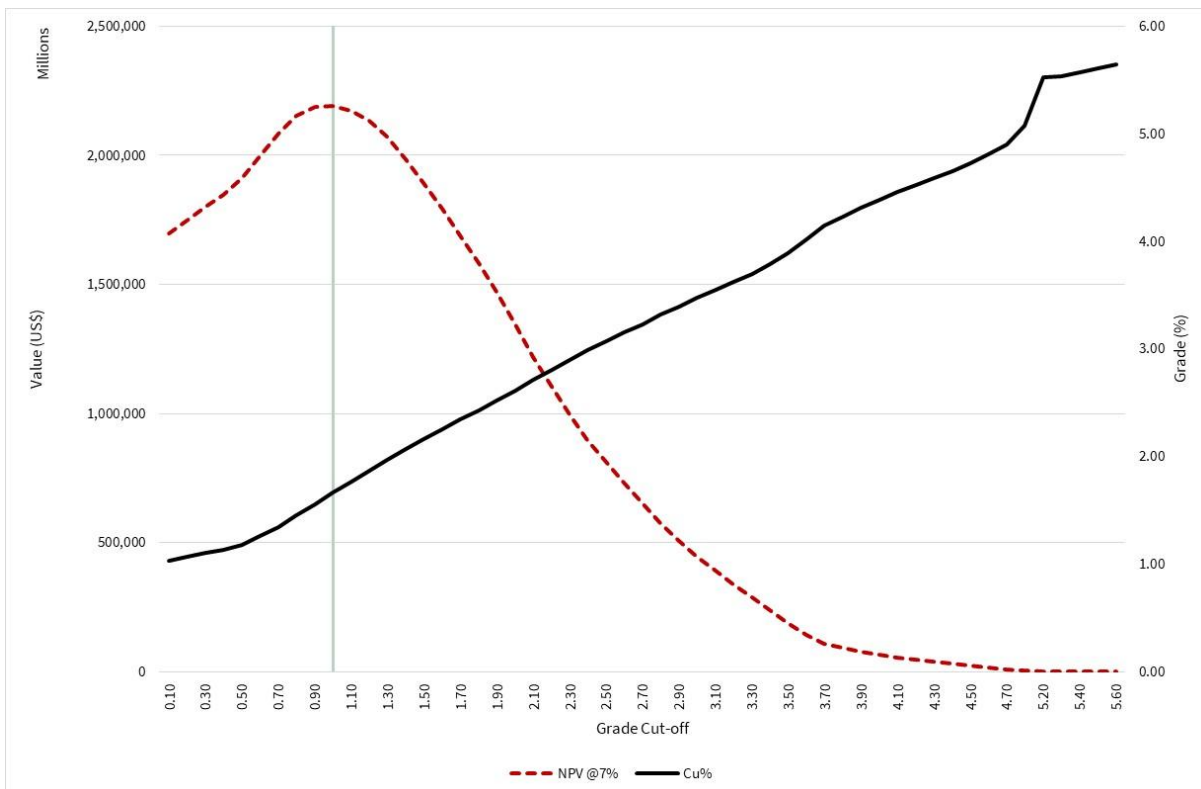
Source: Lumina, 2025.

Figure 16.2: Tonnage Grade Curve



Source: Micon, 2025.

Figure 16.3: Optimal Cut-Off



Source: Micon, 2025.

The mineable tonnes in the model at a cut-off grade of 1% Cu produced from the optimization are detailed in Table 16.2. This result shows the total tonnes, for both mine areas, which is then discounted by the pillar extraction resulting in the target mined tonnes or plant feed tonnes for the total resource. These optimization results are then used as a guide to design the mining envelope.

Table 16.2: Optimal Cut-Off Calculation Results (Cut-Off 1.00 Cu%)

Description	Unit	Value
Total Tonnes	t	421,001,518
Mined Tonnes	t	274,071,988
Copper Mined Grade	Cu%	1.669
Silver Mined Grade	g/t Ag	49.768
Lead Mined Grade	Pb%	0.113
Zinc Mined Grade	Zn%	0.030
Mined Grade - NSR/t	US\$/t	162.0
LoM	Yrs	23.8

Source: Micon, 2026.

The optimal value graph shows that selecting a cut-off between 0.1% Cu and 1.0% Cu yields mined grades from 1.03% to 1.67% Cu, which should be targeted when defining the mining envelope.

Understanding that the entire range results in positive values, the target head was selected at 0.95% Cu to ensure to include maximum mineable material into the mining envelopes. Also, ensuring the low-grade material contained in the envelopes is greater than the pay limit. Including the material below the optimal cut-off will result in much more material in the mining envelope at a lower grade. The high-grade areas will be prioritized in the design and scheduling.

The material contained in the mining envelope defined for further work in the mine design and Life of Mine (LoM) is detailed in Table 16.3.

Table 16.3: Mining Envelope Results for the Total Mineable Resource

Parameter	Unit	High Grade	Medium Grade	Low Grade	Total
Ore Tonnes	t	200,363,342.00	603,946,481.00	37,794,706.00	842,104,529.00
Mined Tonnes	t	130,436,535.64	393,169,159.13	24,604,353.61	548,210,048.38
Copper Mined Grade	%	1.31	1.21	0.78	1.22
Silver Mined Grade	g/t Ag	33.09	39.27	17.45	36.82
Lead Mined Grade	Pb %	0.10	0.18	0.10	0.15
Zinc Mined Grade	Zn %	0.03	0.07	0.04	0.06
LoM	Yrs	11.32	34.13	2.14	47.59

Source: Micon, 2026.

16.1.2 Mine Design Criteria

16.1.2.1 Vertical Shaft Systems

The Nowa Sól production level will be accessed by vertical shafts from two separate locations, the North and South Shaft Complexes. Each Shaft Complex will consist of three shafts; a production shaft for hoisting ore, a service shaft for personnel and equipment access, and a ventilation shaft for mine exhaust (Table 16.4 and Table 16.5)

Table 16.4: South Shaft Parameters

Shaft Parameters	Unit	South Shafts Complex		
		Service Shaft	Production Shaft	Ventilation shaft
Finish Diameter	m	9.5	9.5	9.5
Shaft Depth	m	1,895	1,965	1,850
Ventilation Purpose	-	Air intake	Air intake	Exhaust
Number of Shaft Stations	-	2	2	2
Number of Intersections with Shaft	-	2	2	2
Liner Type				
Concrete	-	+	+	+
Total Concrete Liner Length	m	1,246	1,316	1,201
Composite	-	+	+	+
Total Composite Liner Length	m	350	350	350
Tubbings	-	+	+	+
Total Tubbing Section Length	m	299	299	299
Sinking Quantities				
Total Volume of Excavation	m ³	215,459	222,604	210,866
Total Volume of Concrete for Lining Purpose	m ³	69,551	71,735	68,148
Estimated Tubbings Weight	t	23,741	23,741	23,741
Estimated Steel Weight for Composite Liner Construction	t	10,226	10,226	10,226
Shaft Fitout				
Main Hoist	-	Blair multi rope double drum	Blair multi rope double drum	-
Number of Hoists	-	1	2	-
Auxiliary Hoist	-	+	-	-
Conveyance	-	Large cage with counter weight	4 skips	-

Source: DMC, 2026.

Table 16.5: North Shaft Parameters

Shaft Parameters	Unit	North Shafts Complex		
		Service Shaft	Production Shaft	Ventilation shaft
Finish Diameter	m	9.5	9.5	9.5
Shaft Depth	m	2,120	2,190	2,075
Ventilation Purpose	-	Air intake	Air intake	Exhaust
Number of Shaft Stations	-	2	2	2
Number of Intersections with Shaft	-	2	2	2
Liner Type				
Concrete	-	+	+	+
Total Concrete Liner Length	m	1,451	1,521	1,406
Composite	-	+	+	+
Total Composite Liner Length	m	350	350	350
Tubbings	-	+	+	+
Total Tubbing Section Length	m	319	319	319
Sinking Quantities				
Total Volume of Excavation	m ³	239,641	246,785	235,047
Total Volume of Concrete for Lining Purpose	m ³	77,227	79,410	75,823
Estimated Tubbings Weight	t	25,329	25,329	25,329
Estimated Steel Weight for Composite Liner Construction	t	10,226	10,226	10,226
Shaft Fitout				
Main Hoist	-	Blair multi rope double drum	Blair multi rope double drum	-
Number of Hoists	-	1	2	-
Auxiliary Hoist	-	+	-	-
Conveyance	-	Large cage with counter weight	4 skips	-

The three shafts at each complex all follow the same sequence of development. Following mobilization, the shaft collar will be constructed to a depth of approximately 40 m using the diaphragm wall technique. This initial structure will be constructed in loose surface soils and rock using conventional surface civil construction methodology. This includes the shaft collar and the headframe foundations as well as buried utility conduits, ventilation ducting and subsurface accessways. Also contained in the collar works are starter tubes and manifolds for shaft freezing pipes. Following the construction of the shaft collar is the drilling of freeze pipes, a critical component carried out by specialty drillers who precisely construct a cylinder of pipes. Concurrent to this is the assembly of a temporary freeze plant, which is required to cool brine solution intended to be circulated through the pipes.

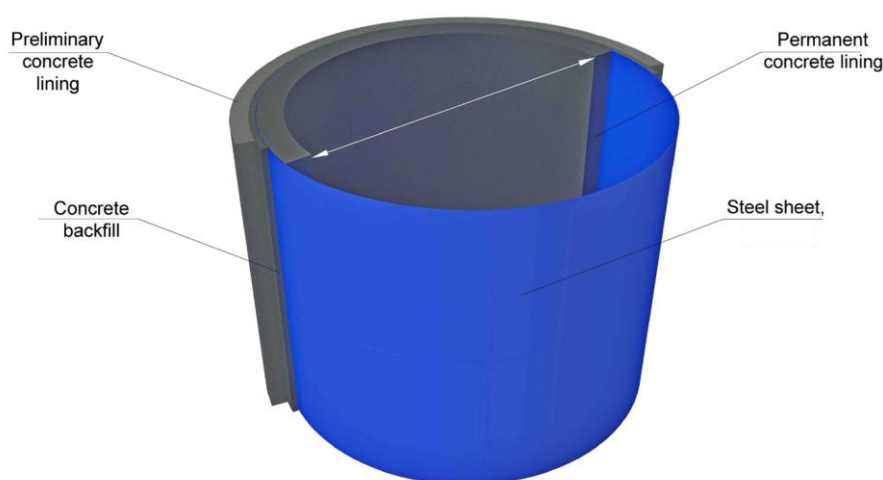
Once the freeze cycle starts a few concurrent activities can commence, including headframe erection, sinking hoist installation and sinking work stage assembly and placement in the collar. Once the shaft cylinder is determined to be adequately frozen, sinking can begin. In addition at this construction the production and service shafts permanent hoisting system will also be installed.

All shafts in this geographical region require ground freezing or grouting of the soft unconsolidated and water bearing layers. Additionally a watertight liner is required to prevent shaft flooding after sinking is completed. Depths of water saturated or soft layers vary from shaft to shaft depending on

the local geology. In the Project location at this stage, it is assumed that ground freezing will be used up to a depth of 350 m which covers Neogene and partially Triassic geological layers.

Sinking in the freeze zone is accomplished using drill and blast, depending on the rock strength and the competency of the freeze wall. The liner for this part of the shaft is assumed to be a composite liner consisting of two concrete layers and welded steel rings installed between them. The sequence of sinking is to excavate and pour the temporary liner in a bottom down direction and in the second step construct the steel rings before pouring the final, second concrete liner. This is completed in a bottom to top direction. At the bottom a chemical seal ring will be constructed which prevents water migration through the shaft liner below the composite liner section. The “all in” advance rate for this type of sinking is estimated at 1.15 m/d. Typical composite liner design are shown in Figure 16.4.

Figure 16.4: Typical Composite Liner



Conventional sinking below the frozen zone requires normally drill and blast excavation with a concrete liner. The expected advance rate of 2.50 m/d for the balance of the shafts below 350 m is assumed.

In both South and North Shaft locations two major and thick halite layers were identified during initial exploration drilling. Halite layers exhibit especially extensive creep and squeezing after excavation requiring heavy duty liner installation in those shaft sections. An allowance for a proven double tubing method in this area has been included in the estimated cost and schedule at an “all in” 0.55 m/d sinking rate. However, other optional shaft lining methods may prove faster and economical with further geotechnical modelling.

The bottom of each shaft is connected to the ancillary facilities (Table 16.6). The construction scope is for a concrete floor on each station, shotcreting, steel brattice and station dewatering.

Production shafts will contain a landing station, loading pockets and overrun allowance. The services shafts have been designed to have a main landing station and an additional sweeping station for ventilation. The ventilation shaft, which is to be completed at a later stage, will have two large sweeping stations.

All shaft sinking will be completed using a contractor’s equipment including hoist, winches and work stages etc. This will enable the permanent hoist to be delivered and installed concurrently with sinking. Additionally, the contractors hoist can be used for initial lateral development while the remaining shafts are excavated and equipped.

Table 16.6: Shaft Station Development Scope

Parameter	Unit	Value	Comment
Vent Station (same scope for all three shafts)			
Number of Stations	-	4	-
Number of Intersections with Shaft	-	2	From both sides of shaft
Station Inset Size	-	9.0 mW x 6.0 mH	-
Station Length	m	15	15 m length extension from shaft edge from both sides (30 m total)
Lateral Development Extension	m	30	Drift size assumed: 6.0 mW x 5.0 mH
Total Excavated Volume per Station	m³	4,285	
Main Station (Main station in Service shaft for material and crew transportation)			
Number of Stations	-	1	-
Number of Intersections with Shaft	-	2	From both side if shaft
Station Inset Size	-	9.0 mW x 6.0 mH	-
Insets Length	m	15	15 m length extension from shaft edge from both sides (30 m total)
Lateral Development Extension	m	100	Drift size assumed: 6.0 mW x 5.0 mH
Total Excavated Volume per Station	m³	6,385	
Loading Station (Loading level station in Production Shaft)			
Number of Stations	-	1	-
Number of Intersections with Shaft	-	2	From both side if shaft
Station Inset Size	-	9.0 mW x 6.0 mH	-
Insets Length	m	15	15 m length extension from shaft edge from both sides (30 m total)
Lateral Development Extension	m	30	Drift size assumed: 6.0 mW x 5.0 mH
Total Excavated Volume per Station	m³	4,285	

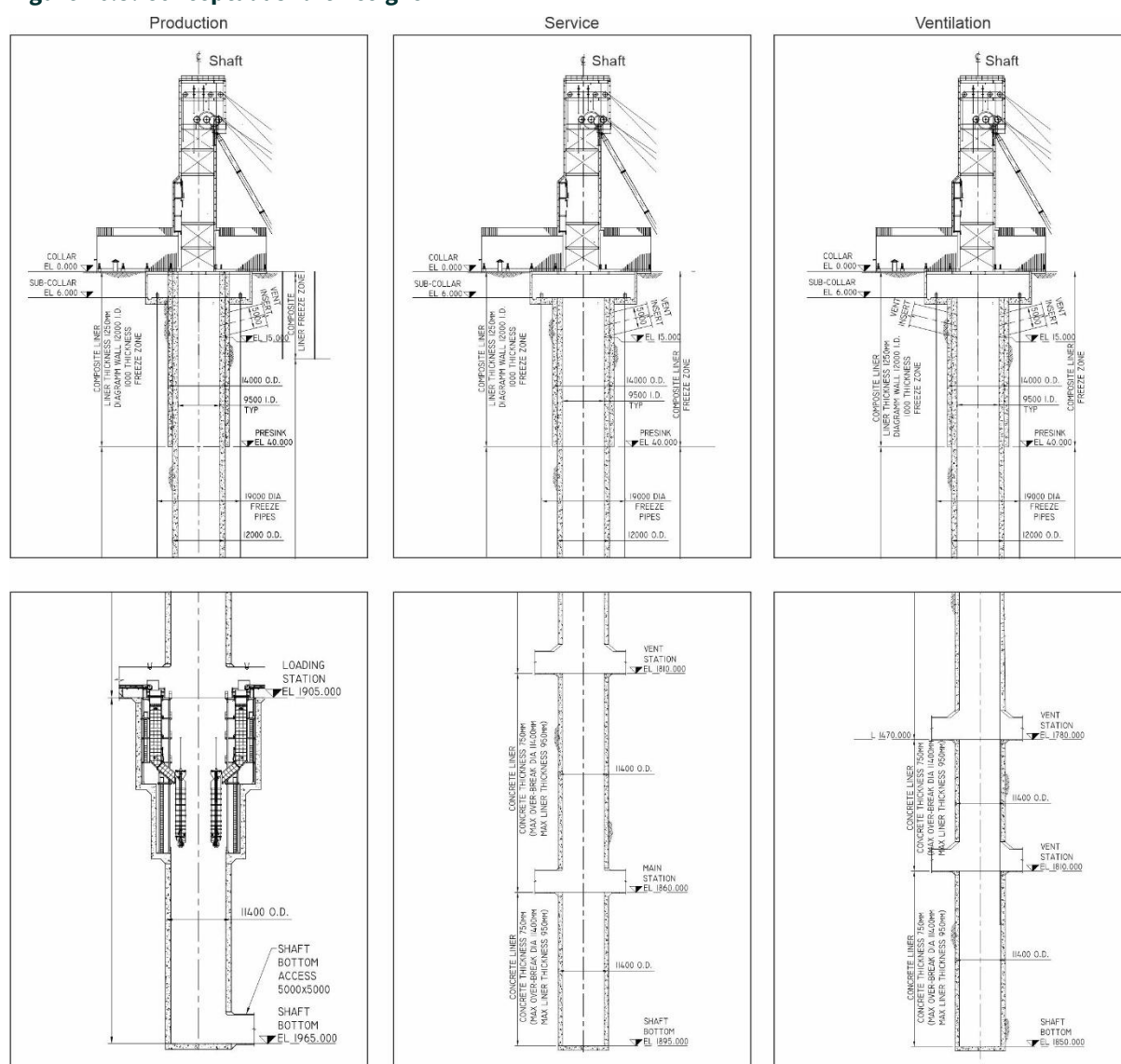
Source: DMC, 2026.

Note: Construction Scope (Concrete floor on station, shotcreting, Steel brattice, Station dewatering).

The total volume of excavated material for each shaft will average 690,000 m³ for each complex (three shafts combined), for which a waste rock dump has been designed close to the shaft collar to store this volume (see Section 18).

The conceptual shaft top and bottom designs are shown in Figure 16.5.

Figure 16.5: Conceptual Shaft Designs



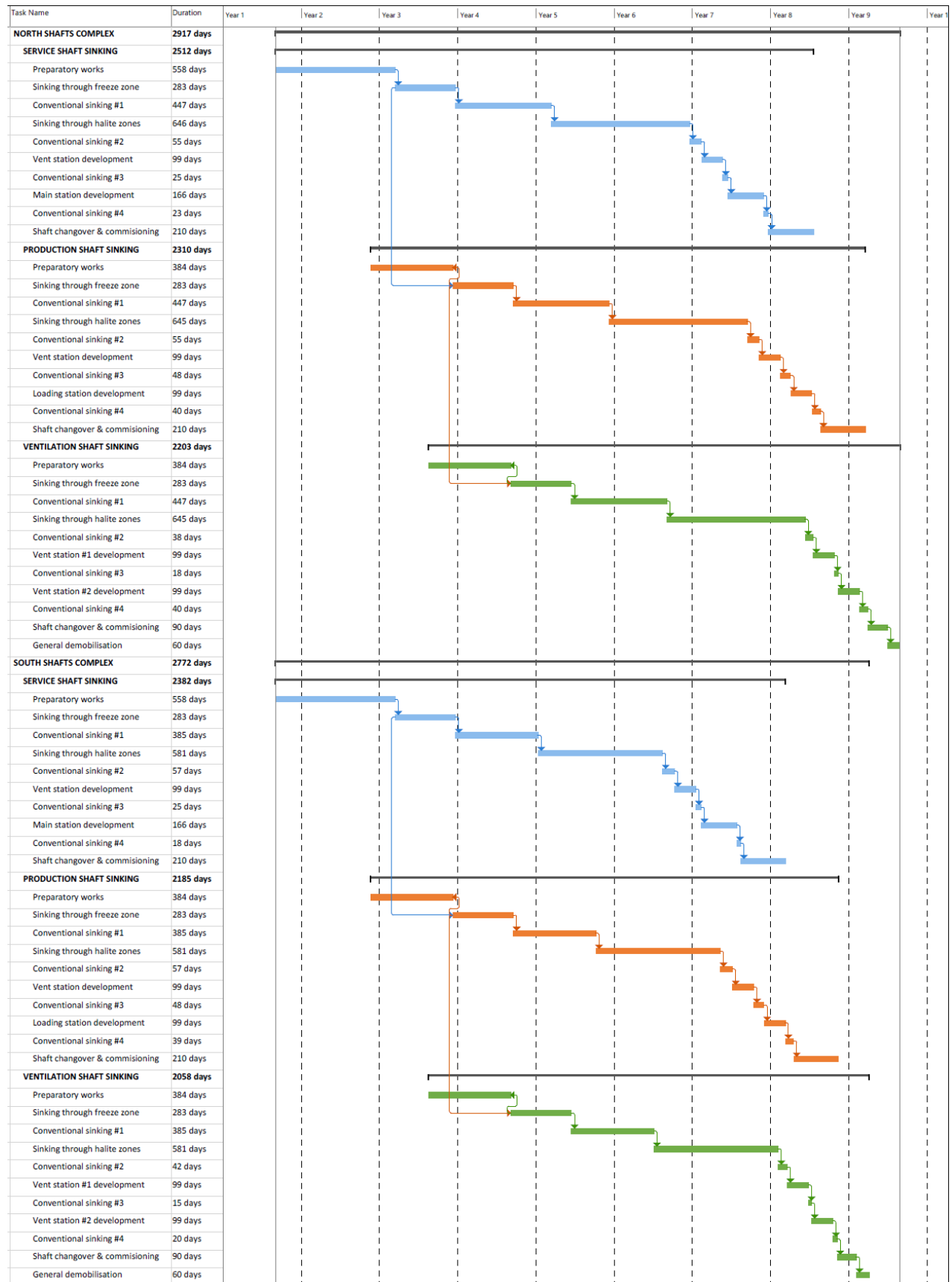
Source: DMC, 2026.

The estimated shaft sinking rate is based on the schedule in Figure 16.6. Variations of shaft sequencing may prove beneficial to improve timing to first production.

KGHM make use of Koepe winders as the primary hoist system. Koepe winders, also known as a friction winder, are a type of mining hoist system that utilize a large, grooved wheel to grip and drive winding ropes via friction, as opposed to winding them onto a drum like traditional winders. This style of winder is a well-established hoist system being used throughout Europe, South Africa and North America, which has proved to improve hoisting from depth, whilst being efficient to operate using less power for large loads.

However, these winders are dependent on friction on the drum to work and as rope weight increases become impractical below 1,700 m depending on payload. In addition, the Koepe hoist requires deeper shafts to accommodate the needed tail ropes. This adds significant capital cost schedule requirements at the end of each shaft development.

Figure 16.6: Shaft Sinking Schedule



Source: DMC, 2026.

Hoisting below 1,700 m is now increasing enabled by Blair Multi-Rope hoist (BMR). A typical Blair Hoist is illustrated in Figure 16.7. Originating in South Africa, BMRs are basically a double drum hoist with two ropes on each counter rotating drum. By using two ropes for each conveyance the weight of the ropes is no longer a critical factor. Moreover, the rope life is longer and rope maintenance cost are lower than for the Koepe Hoist. In addition to configuration, the motor technology and operating software for BMRs has advanced in the last two decades, increasing reliability and reduced operating costs. Hoisting speeds have increased to 18 m/s with up to 40 t loads with BMRs. This hoist technology has greatly improved the practicality of mining at greater depths.

Today the deepest single lift in the world is South Deeps in South Africa where a BMR hoists from almost 3 km. BMR's are now increasing in application in North America and would work well at the Nowa Sól depths.

To achieve 32,000 t/d at each complex two Blair hoists are planned for each production shaft requiring around 14,000 kW motors each. The hoists are sized to operate 19 h/d to achieve the production needed. The service shaft will also have a single Blair hoist with around 10,000 kW motors.

An additional auxiliary hoist will be added to the service shafts and a small emergency escape hoist added to the ventilation shafts.

Figure 16.7: Typical Blair Hoist



Source: DMC, 2026.

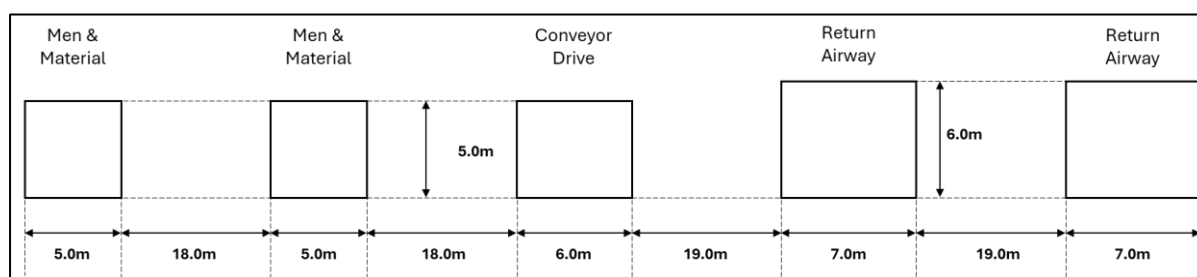
16.1.2.2 Decline Development

Both North and South shafts share the same mine design philosophy, differing only in layout.

Access to the mining horizon is provided through a vertical shaft system. From this vertical shaft, an on-reef decline system extends to allow access to the designated mining sections. The decline system has been designed such that the stoping panels are accessed directly from the main decline, ensuring efficient movement of personnel and materials between the surface and the active mining areas.

The main decline system design criteria is detailed in Figure 16.8.

Figure 16.8: Main Decline System Design Criteria



Source: Micon, 2026.

The decline system will provide access for both men and material, will be equipped with a conveyor belt to remove mined ore from the stoping panels to the shaft and includes the main return airways to the upcast ventilation shafts.

The development design criteria are summarized in Table 16.7.

Table 16.7: Development Design Criteria

Development Description	Width (m)	Height (m)
Conveyor Drive x 1	6.0	5.0
Men and Material x 2	5.0	5.0
Return Airway x 2	7.0	6.0

16.1.2.3 Stopping

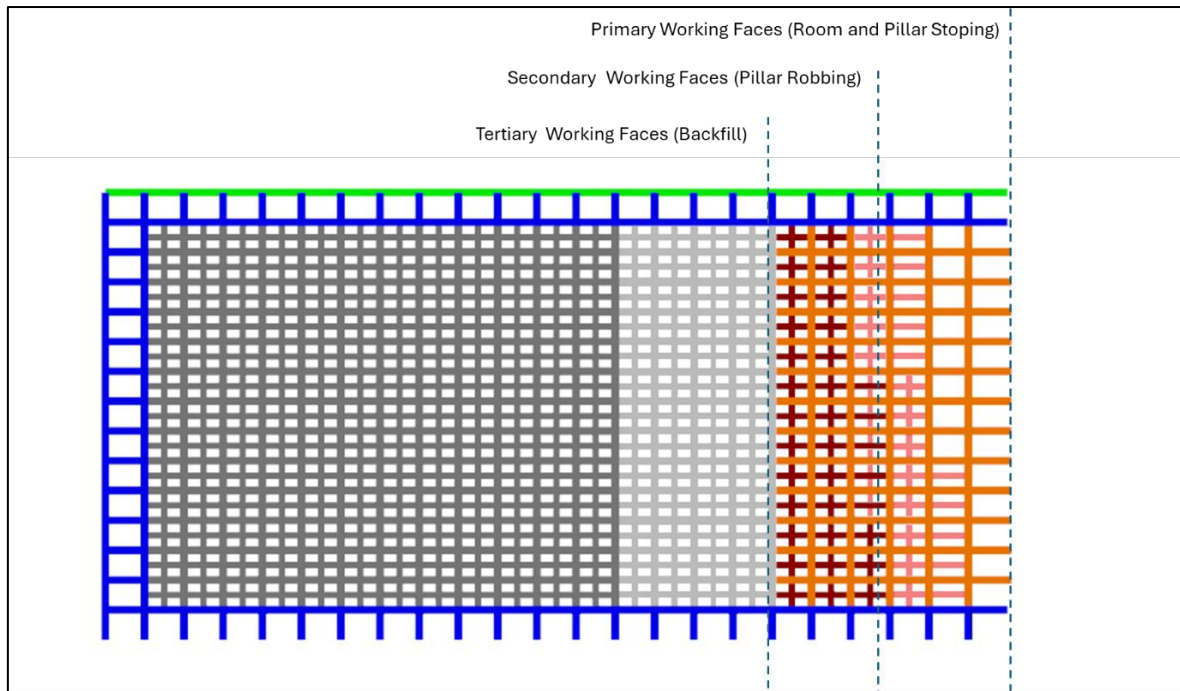
The selected primary mining method is Room and Pillar, the panel being a 15 road panel. This is a selective extraction method used in horizontal or flat-dipping ore bodies. The second phase of stoping will be partial pillar extraction following behind the primary working face.

The mined areas in the panel will be filled with cemented paste fill once the secondary partial pillar extraction is complete., effectively creating a third working face as the panel advances.

The stoping sequence is illustrated in Figure 16.9. This figure shows the 15-road panel, with a total width of 285 m, detailing the access drives which also defines the safety pillars around the stoping panel and the conveyor drive for each panel at the top (green).

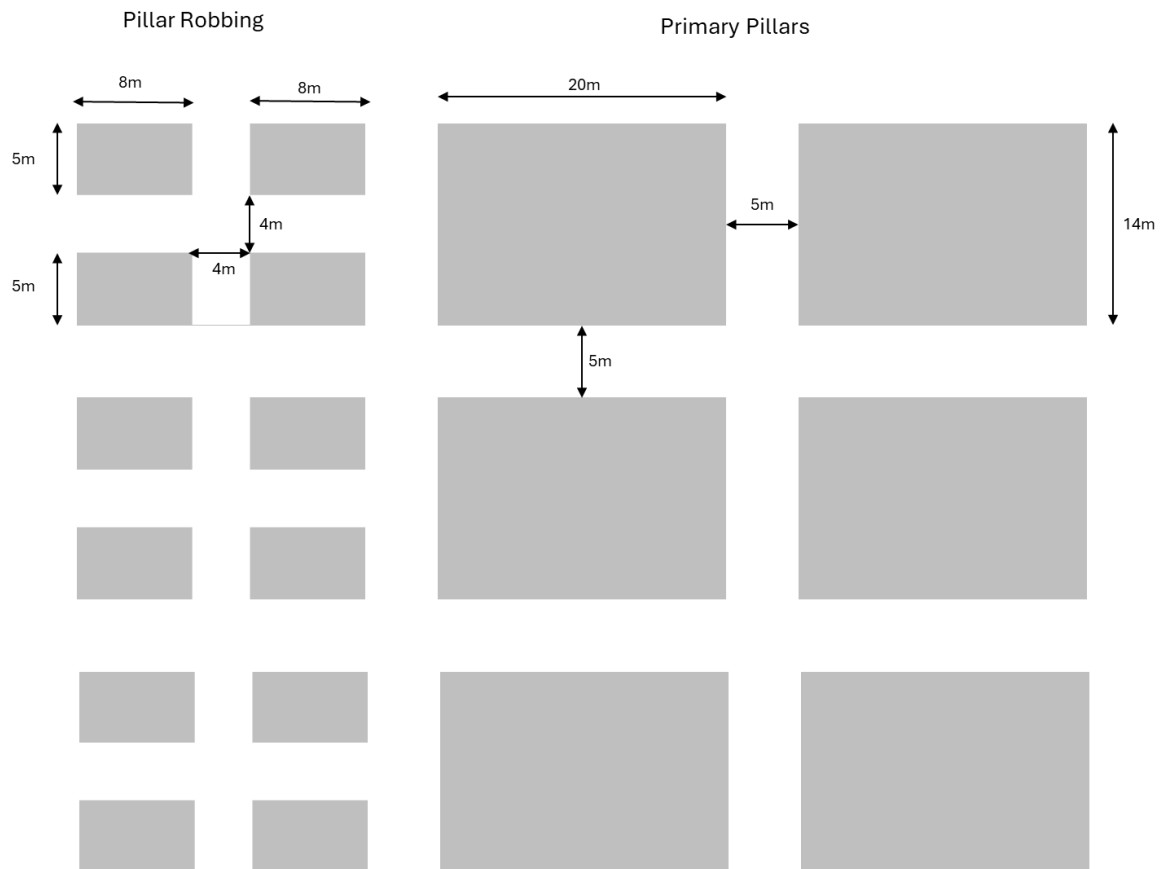
The in-panel Room and Pillar design criteria are detailed in Figure 16.10. The pillar design criteria are supported by the geotechnical design summarized in (Section 16.1.3).

Figure 16.9: Single Panel Stopping Sequence



Source: Micon, 2026.

Figure 16.10: Panel Pillar Design



Source: Micon, 2026.

The planned total panel extraction is detailed in Table 16.8.

Table 16.8: Panel Extraction

Description	Unit	Value
Total Area (One Split)	m ²	7,225
Pillars Area (One Split)	m ²	4,200
Primary Extraction (In Panel)	%	41.9
Remaining Pillar Area after extraction	m ²	2,520
Total Extraction (In Panel)	%	65.1

16.1.3 Geotechnical Studies

16.1.3.1 Introduction

The Nowa Sól Copper-Silver Project is located within the LGOM of western Poland and represents a deep (>1.9 km to 2.1 km below the surface) extension of the Permian hosted copper-silver mineralization currently mined by KGHM. The proposed mining depth significantly exceeds those of the existing KGHM operations, resulting in a materially higher in-situ stress environment and elevated geotechnical and seismic risk.

This section summarises the geotechnical assumptions, analyses, and conclusions used to support the selection and preliminary design of underground mining methods for the PEA. The geotechnical work has been completed at a conceptual level appropriate for a PEA and is based on limited geological, geotechnical, and hydrogeological data.

This section has been summarized from the full Geotechnical Analysis Report (Global Mine Design, 2025).

16.1.3.2 Stratigraphy Relevant to Mining

The Nowa Sól deposit is hosted within the Permian Zechstein stratigraphy and is mineralized along the Kupferschiefer horizon at the contact between underlying Weissliegende sandstones and overlying limestones/dolomites. The deposit occurs beneath a thick sequence of Triassic sediments and Permian evaporites, including anhydrite and halite units.

Mining is anticipated to occur at depths of approximately 1,800 m to 2,100 m below the surface. The mining horizon comprises a thin mineralized interval (generally 0.5 m to 4.0 m thick) requiring a minimum mining height of approximately 1.8 m. The principal geotechnical units relevant to shaft and mine design are outlined in Table 16.9.

Table 16.9: Main Stratigraphic Units and Geotechnical Significance

Stratigraphic Unit	Approx. Depth (m)	Lithology Types	Geotechnical Significance
Quaternary	0 – 300	Sands, gravels, clays.	Weak, water-bearing; shaft freezing required.
Triassic (Keuper–Muschelkalk–Bunter)	300 – 1,300	Claystones, marls, limestones, sandstones.	Variable strength; major aquifers.
Permian Zechstein	1,300 – 2,100	Sandstone, shale, limestone/dolomite, anhydrite, halite.	Hosts orebody; major strength contrasts.
Halite (Na1)	Deep Mine Roof	Rock salt.	Severe squeezing risk.
Lower Anhydrite (A1d)	Immediate Roof	Massive anhydrite.	Strong, brittle.
Zechstein Limestone/Dolomite (Ca1/Ca0)	Mine Roof/Pillars	Strong carbonates.	High stiffness; burst-prone.
Kupferschiefer (Cs)	Mine Ore Marker	Thin shale (~0.2 m).	High grade; geotechnically minor.
Weissliegende Sandstone (Bs)	Mine Floor/Pillars	Quartz sandstone.	Weak, stress-sensitive.

Source: Lumina, 2025.

16.1.3.3 Structural Geology

Major regional fault systems with northeast–southwest and northwest–southeast orientations are interpreted to cross the Project area. Although detailed fault geometries are not yet defined, analogous faults at LGOM are known to be associated with elevated seismicity, groundwater inflows, and excavation instability.

For the purposes of the PEA, it is assumed that major shafts and long-life infrastructure will be located to avoid intersecting major fault zones. Detailed structural characterisation and 3D fault modelling are required at the PFS stage.

16.1.3.4 Hydrogeology Conditions

Hydrogeological data are limited to borehole water level measurements and selected pump test data. Regional experience indicates that:

- Significant groundwater inflows may be encountered within Quaternary sediments, Triassic aquifers, and locally within Permian sandstones, or through faults.
- Thick Zechstein evaporites (anhydrite and halite) form regionally extensive low-permeability barriers that provide hydraulic separation between overlying aquifers and the mining horizon.

Groundwater inflow represents a material risk for shaft construction and underground development and will require robust dewatering and water control measures.

16.1.3.5 In-Situ Stress Regime

No direct in-situ stress measurements have been completed at Nowa Sól. Stress assumptions are inferred from regional KGHM data and published literature. At the anticipated mining depths, vertical stresses are estimated to be approximately 50 MPa. Horizontal stresses are expected to be of similar or greater magnitude, with horizontal-to-vertical stress ratios (K) ranging from approximately 1.0 to 1.5.

Evidence of core discing in exploration drilling indicates potential for high deviatoric stress environment. For PEA level analysis, a range of stress magnitudes and orientations have been adopted to reflect uncertainty.

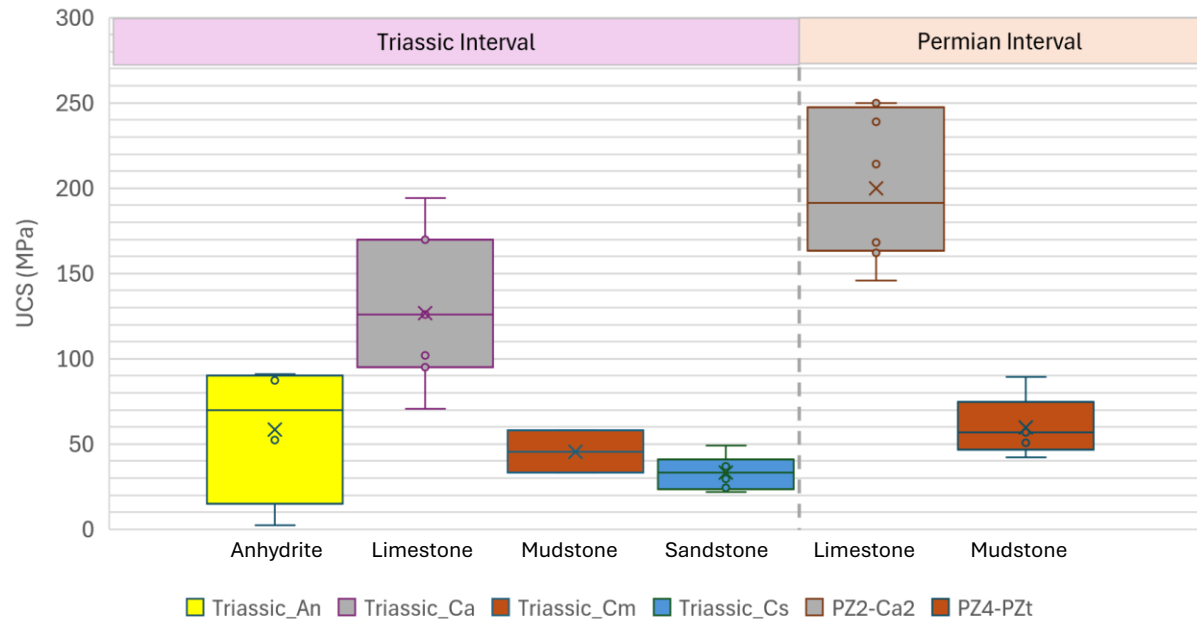
16.1.3.6 Laboratory Testing

Laboratory testing includes Unconfined Compressive Strength (UCS), Young's modulus, tensile strength, and rockburst propensity testing. Key observations include:

- Limestone and dolomite units are very strong (UCS commonly >100 MPa), stiff, and brittle.
- Anhydrites exhibit moderate to high strength.
- Sandstones are weak to moderately strong and show significant strength reduction when saturated.
- Rockburst propensity testing indicates high burst potential in limestones/dolomites, moderate potential in anhydrites, and low potential in sandstones.

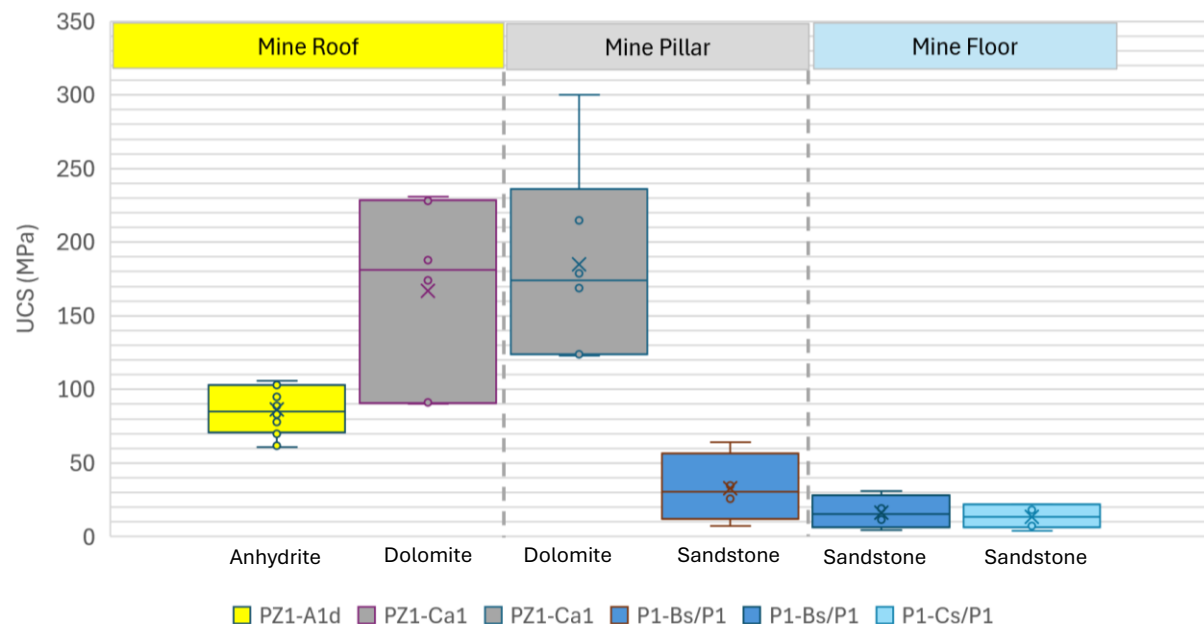
A summary of UCS laboratory testing results by stratigraphy is provided in Figure 16.11 and Figure 16.12. It is noted that no testing is available for the halite rock salt units which is a notable data gap that needs to be addressed in the PFS investigation.

Figure 16.11: UCS Laboratory Testing Summary for Shaft Units



Source: Micon, 2026.

Figure 16.12: UCS Laboratory Testing Summary for Mining Units



Source: Micon, 2026.

16.1.3.7 Rock Mass Classification and Strength

Rock mass properties have been estimated using the Geological Strength Index (GSI) method and the Hoek–Brown failure criterion. Due to limited core logging data, GSI values are based on visual assessment and analogues from available KGHM published references.

Representative GSI values, and rock mass strength and stiffness parameters derived for use in preliminary numerical modelling and stability assessments are summarized in Table 16.10. Hoek-Brown Unconfined Compressive Strength rock mass (UCS_{rm}), rock mass modulus (E_{rm}), rock mass friction angle (ϕ_{rm}), and rock mass cohesion (c_{rm}) are estimated. A disturbance factor (D) of 0.3 has been assumed for drill and blast damage for the mine pillar units of dolomite and sandstone.

Table 16.10: Representative Rock Mass Parameters (Average Values)

Lithology	GSI	UCS_{rm} (MPa)	E_{rm} (GPa)	Φ_{rm} (°)	C_{rm} (MPa)
Mudstone	45 - 65	3 - 9	2 - 8	29 - 34	4 - 5
Halite	85	6 - 7	2	29	4 - 5
Anhydrite	85	35	60	42	10
Dolomite	75	37	32 - 49	45	10
Sandstone	65	2 - 3	3 - 5	23 - 27	3 - 4

16.1.3.8 Room and Pillar Geotechnical Assessment

Room and Pillar mining is a proven method within KGHM's Poland operations and has been evaluated for application at Nowa Sól. However, the greater depth and higher stress environment at Nowa Sól materially increases the geotechnical and seismic risk.

Preliminary analyses indicate that ore recoveries of approximately 80% to 90% (as achieved at shallower KGHM operations) are not realistic at Nowa Sól under acceptable safety margins. For PEA mine planning, Room and Pillar panel recoveries in the range of approximately 50% to 70% are considered more defensible when accounting for high in-situ stresses and increased seismic hazards. The mine pillars and roof will be in a continuous state of rock mass yield which needs to be closely monitored during Room and Pillar advancement.

Mine pillar stress analyses were carried out using 2D numerical modelling and empirical criteria to assess alternative Room and Pillar geometries. Pillar factors of safety were assessed for the Nowa Sól 2 km mining depth to correlate to the KGHM pillar factors of safety at 1 km mining depth. This resulted in significantly larger pillar dimensions at Nowa Sól in comparison to KGHM. Conceptual Room and Pillar configurations were developed for the PEA Room and Pillar arrangements as summarized in Table 16.11. Variable ore seam thickness and pillar lithologies will require adjustments to pillar dimensions, to be assessed in more detail as part of the PFS mine planning work.

Table 16.11: Room and Pillar Dimensions

Lithology	Dimensions (m)	Configuration
Permanent Safety Pillars	50 x 80	Positioned between mining panels and permanent infrastructure (conveyor drifts, ventilation drifts).
Temporary Safety Pillars	30 x 50	Positioned between mining panels and temporary infrastructure.
Technical Pillars	14 x 20	Panel pillars prior to final rob-cuts.
Residual Pillars	5 x 8	Panel pillars after final rob-cuts.
Maximum Room Widths	4 - 6	Panel room widths after final rob-cuts.

Backfilling of mined-out residual pillar areas is required to control roof deformation and manage stress redistribution. Backfilling can be achieved with paste backfill, hydraulic fill, or waste rock fill.

16.1.3.9 Drift and Fill Geotechnical Assessment

Drift and Fill has been assessed as an alternative mine method with potential advantages in terms of selectivity and mechanization. Drift and Fill is not currently used at KGHM's Poland operations, but it is recognized as a potential mining method option for Nowa Sól which would be more conducive to automation and mechanization. Good quality cemented backfill such as paste backfill is required for the implementation of Drift and Fill.

Conceptual Drift and Fill panel configurations were developed and 2D numerical modelling was used to assess pillar and roof behaviour during sequential excavation of drifts. Geotechnical challenges associated with Drift and Fill include large effective roof spans, rapid stress build-up during final extraction stages, and strong dependence on paste backfill performance. Control of the roof stability is critical for Drift and Fill due to the increasing effective spans. Routine use of cable bolting or timber cribs/posts for roof support would be a consideration with this mining method.

In comparison to Room and Pillar, Drift and Fill mining methods have additional productivity constraints due to reduced working faces and backfill cycle times. Conversely, Drift and Fill extraction ratios are expected to be marginally higher than Room and Pillar by using good quality cemented backfill to support the residual pillars and roof. Panel recoveries in the range of 60% to 80% were assessed, depending on configuration and ground conditions. The Drift and Fill dimensions used in mine planning are outlined in Table 16.12. Drift and Fill pillar dimensions would adjust depending on roof and pillar lithologies, and mining seam height. Detailed Drift and Fill configurations will be further assessed in the next study stages including the use of 3D numerical modelling.

Table 16.12: Drift and Fill Dimensions

Lithology	Dimensions (m)	Configuration
Permanent Safety Pillars	50 x 80	Positioned between mining panels and permanent infrastructure (conveyor drifts, ventilation drifts)
Temporary Safety Pillars	30 x 50	Positioned between mining panels and temporary infrastructure
Drift Dimensions	5 x 40	Panel drift dimensions prior to backfill.
Residual Pillars	5 x 40	Panel pillars remaining after excavation. Pillars are spaced every 15 m, or every 3 drift spans.
Permanent Safety Pillars	50 x 80	Positioned between mining panels and permanent infrastructure (conveyor drifts, ventilation drifts)

Cemented backfill such as paste backfill will be a critical component of Drift and Fill mining. Tight backfilling to the drift roofs will be required to maximise loading bearing of the backfill. Moderate to high strength and stiffness paste will be required to support the roof, and to mitigate paste overbreak from roof deformation. Paste strength and stiffness requirements would be assessed in subsequent design stages, also using the results from paste laboratory testing programs to be carried out during the PFS.

16.1.3.10 Ground Support and Seismic Considerations

Both Room and Pillar and Drift and Fill mining methods will require enhanced ground support systems appropriate for high-stress, seismically active conditions. These may include:

- Systematic rock bolting and cable bolting.
- Dynamic ground support elements such as D-bolts and mesh with shotcrete.
- Supplemental support such as timber cribs and timber posts for supporting the roof.
- Backfill systems to support residual pillars and the roof.

Detailed ground support configurations will be developed in subsequent study stages once the mine arrangements are further developed. Limitations on excavation spans, as well as increased pillar widths, will be required at Nowa Sól due to high stress conditions, and will need to be assessed further. Excavation wall geometry will also need to account for potential overbreak and damage due to high stress conditions.

Seismicity and rock burst risk are considered major operational hazards which will also impact mine productivity. At KGHM's Poland operations, routine destress blasting is used in the advancing faces and backs experiencing high stress and seismicity. Similar destressing methodologies will likely be required at Nowa Sól. A robust seismic monitoring system will need to be implemented for ongoing monitoring, stress management strategies, and adaptive mine planning.

16.1.3.11 Shaft Geotechnical Assessment

A series of 2D numerical modelling stress assessments were carried out to assess shaft rock mass yielding and closure for varying depths and lithologies. Shaft excavation diameters of 8.5 m and 11 m were assessed in the models.

Preliminary shaft stability analyses indicate that:

- Severe to very severe squeezing is expected in halite (rock salt) units, with predicted closures ranging from approximately 0.5 m to greater than 1.5 m.
- Mudstone and sandstone intervals may also exhibit significant yielding, particularly where water-bearing.
- Shaft spacing of at least approximately 150 m and offsets of approximately 500 m from active mining areas are preliminarily recommended.

Permanent shaft protection pillars will be required, with a significant impact on ore sterilization and mine layout. Preliminary recommendations assume a 1 km x 1 km shaft pillar area, which will also incorporate other fixed infrastructure such as shops, pump stations, and shaft loadout areas. The shaft loadout and infrastructure arrangements need to be developed in subsequent design stages, incorporating detailed geotechnical assessment of pillar and ground support requirements for the excavations.

Shaft freezing, liner design, and construction methodologies will be an important focus in subsequent design stages. Shaft freezing will be required in the Quaternary units (upper 300 m depth) as per KGHM practice and there is optionality to extend this freezing to greater depths into the Triassic units. Permanent shaft freeze should be assessed as a trade-off of savings on liner design and schedule. Shaft advancement using mechanical cutting is another trade-off study to be

assessed in subsequent design stages. Additional detailed characterisation of the shaft stratigraphy geotechnical and hydrogeological characteristics is required to support future shaft design work.

16.1.3.12 Geotechnical Risks and Uncertainties

The geotechnical parameters, mine designs, and recoveries presented herein are conceptual and are considered appropriate for a PEA. They are not sufficient to support mineral reserves nor a detailed mine design.

The geotechnical assessment is subject to the following key risks and uncertainties:

- Limited geotechnical and hydrogeological drilling data.
- Limited geotechnical stability and stress analysis.
- Uncertainty in detailed stratigraphy, fault geometry and properties.
- Sensitivity of sandstone strength to in-situ stress levels and water ingress.
- Time-dependent deformation in halite units affecting shafts.
- Significant potential for mine seismicity and stress induced rock mass failures.
- High stress and seismicity significantly impacting mine production economics.

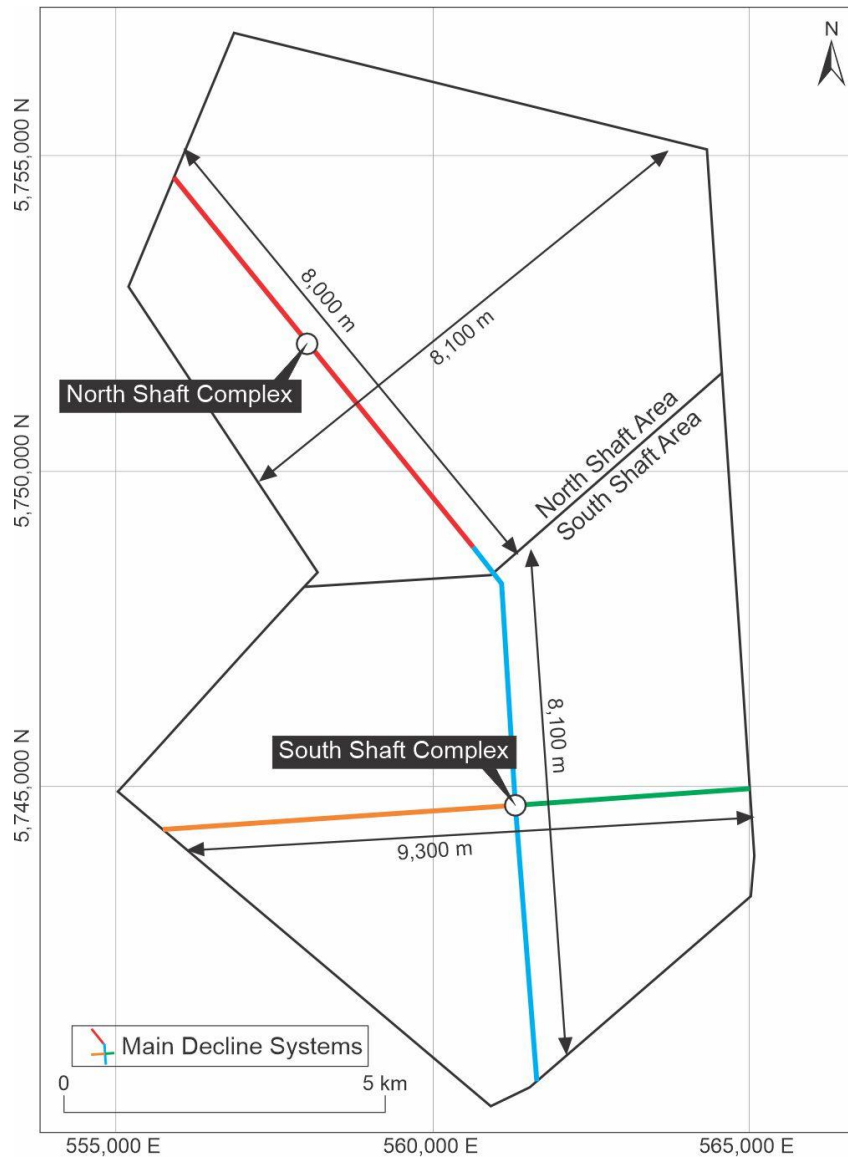
High stress and seismicity could significantly reduce mine productivity for the Nowa Sól deposit. Impacts to the Project economics should be further assessed with geotechnical constraints including mine production delays due to seismicity and stress management factors.

Further geotechnical investigations, laboratory testing, and advanced numerical modelling will be required to support a PFS and Feasibility study (FS). Data collection guidance has been provided to Lumina for the PFS stage.

16.1.4 Mine Design

The vertical shafts give access to the mining horizon from where the main decline systems are developed to access the orebody. The main decline development layout is shown in Figure 16.13.

Figure 16.13: Main Development Layout

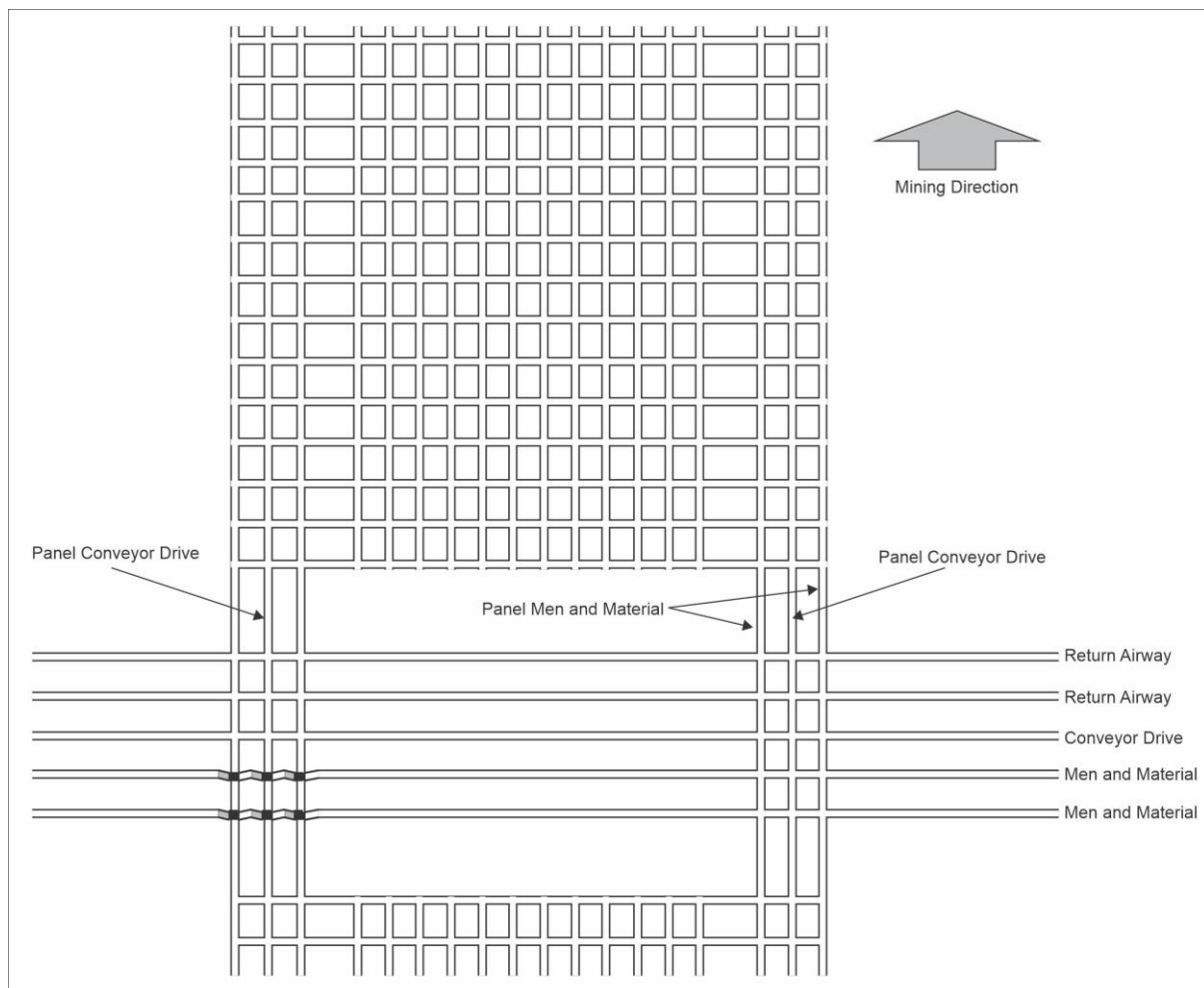


Source: Micon, 2026.

The Room and Pillar stoping panels will be accessed directly from the main declines ensuring efficient movement of personnel and material. Ore will be transported from the working face with a Load Haul and Dump (LHD) to the nearest conveyor belt which will transport the broken ore to the main decline and shaft systems for hoisting.

The detailed panel layout showing the material movement is illustrated in Figure 16.14.

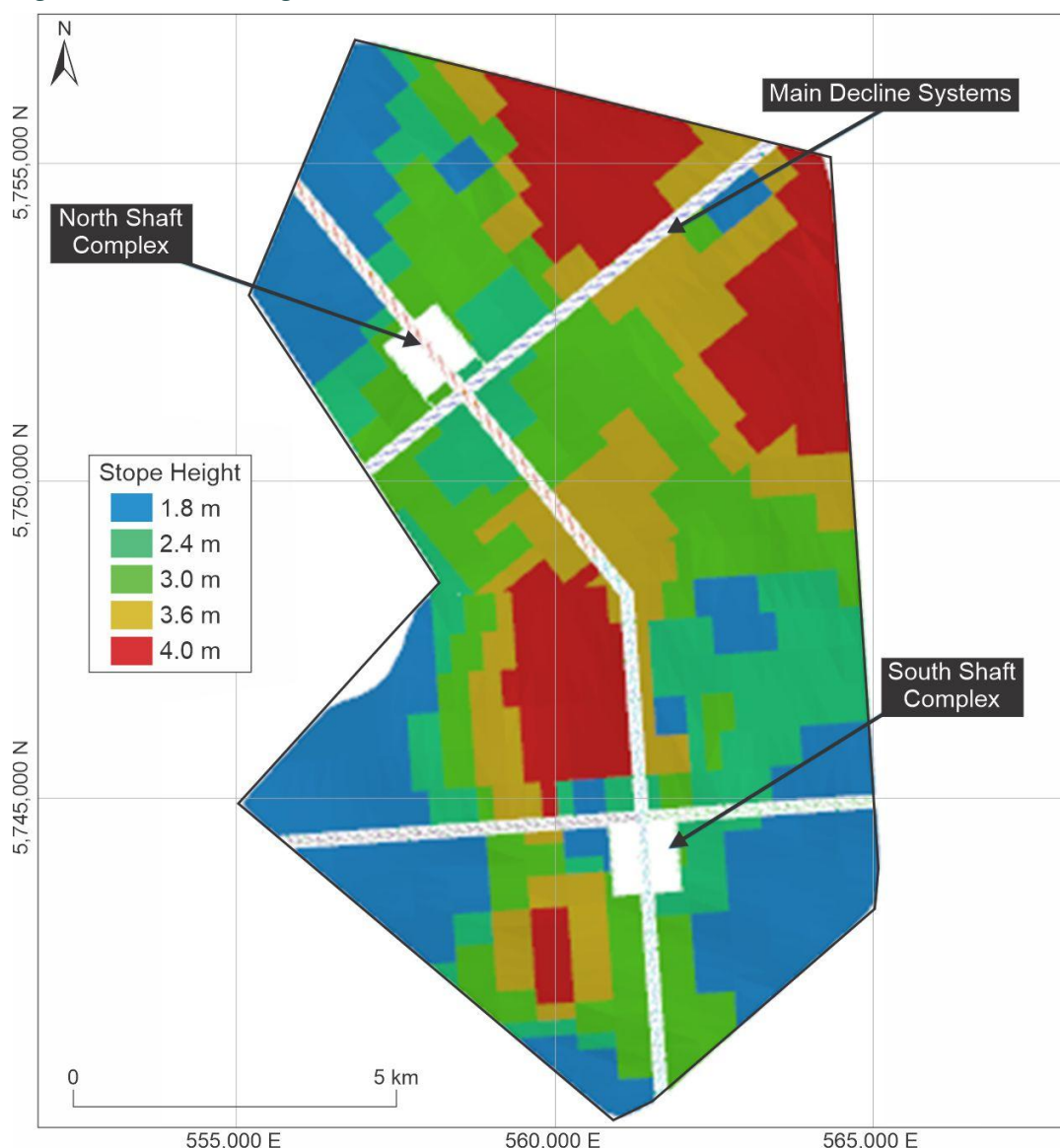
Figure 16.14: Detailed Panel Layout



Source: Micon, 2026.

The full design showing the development and planned panels is illustrated in Figure 16.15.

Figure 16.15: Mine Design (CAD)



Source: Micon, 2026.

16.1.5 Underground Mine Operation, Manpower and Equipment Selection

The mining equipment selected was completed to ensure the fleet would be able to deliver the planned volumes.

The proposed underground mining fleet descriptions are detailed Table 16.13.

The underground fleet requirements were calculated from the mining productivity requirements and machine capacities to ensure effective execution of the LoM. The base case assumption is diesel equipment. Equipment trade-off studies will be completed in future studies.

The fleet requirement of the LoM for both shafts, North and South are detailed in Table 16.14 and Table 16.15, respectively.

Table 16.13: Total Underground Mining Fleet

Underground Mining Fleet - North		Value
Development Fleet		
Drill Rig - Sandvik DD321 (or similar)		14
LHD - Sandvik LH410 (or similar)		20
Roofbolter - Sandvik DS312 (or similar)		26
Charging Vehicle (Normet)		5
Utility Vehicles (Normet - Lifting & Transport)		14
Stoping Fleet		
Drill Rig - Sandvik DD321 (or similar)		73
LHD - Sandvik LH410 (or similar)		73
Roofbolter - Sandvik DS312 (or similar)		73
Charging Vehicle (Normet)		24
Utility Vehicles (Normet - Lifting & Transport)		73
Ancillary Vehicles		
Light Duty Vehicles (LDVs)		20
Fuel & Water Bowser - Bowsers		8
Motor Grader - Graders		3
Underground Mining Fleet - South		Value
Development Fleet		
Drill Rig - Sandvik DD321 (or similar)		16
LHD - Sandvik LH410 (or similar)		23
Roofbolter - Sandvik DS312 (or similar)		30
Charging Vehicle (Normet)		6
Utility Vehicles (Normet - Lifting & Transport)		16
Stoping Fleet		
Drill Rig - Sandvik DD321 (or similar)		73
LHD - Sandvik LH410 (or similar)		73
Roofbolter - Sandvik DS312 (or similar)		73
Charging Vehicle (Normet)		24
Utility Vehicles (Normet - Lifting & Transport)		73
Ancillary Vehicles		
Light Duty Vehicles (LDVs)		26
Fuel & Water Bowser - Bowsers		8
Motor Grader - Graders		3

Source: Micon, 2026.

Table 16.14: Active Fleet Requirements for the North Shaft Underground LoM

North Shaft	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	37	38	39	40	
Development Fleet																																									
Drill Rig - Sandvik DD321 (or similar)	0	0	0	0	0	0	4	14	7	5	4	4	2	0	0	0	0	0	0	0	2	3	3	2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
LHD - Sandvik LH410 (or similar)	0	0	0	0	0	0	5	20	10	7	6	6	3	0	0	0	0	0	0	0	3	3	4	3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Roofbolter - Sandvik DS312 (or similar)	0	0	0	0	0	0	7	26	13	10	8	7	4	0	0	0	0	0	0	0	4	4	4	4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Charging Vehicle (Normet)	0	0	0	0	0	0	2	5	3	2	2	2	1	0	0	0	0	0	0	0	1	1	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Utility Vehicles (Normet - Lifting & Transport)	0	0	0	0	0	0	4	14	7	5	4	4	2	0	0	0	0	0	0	0	2	3	3	2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Stoping Fleet																																									
Drill Rig - Sandvik DD321 (or similar)	0	0	0	0	0	0	2	24	54	54	54	56	56	58	58	58	58	58	58	58	56	56	58	58	58	58	58	58	52	34	16	6	2	0	0	0	0	0	0	0	
LHD - Sandvik LH410 (or similar)	0	0	0	0	0	0	1	12	27	27	27	28	28	29	29	29	29	29	29	29	28	28	29	29	29	29	29	26	17	8	3	1	0	0	0	0	0	0	0	0	
Roofbolter - Sandvik DS312 (or similar)	0	0	0	0	0	0	2	24	54	54	54	56	56	58	58	58	58	58	58	58	56	56	58	58	58	58	58	58	52	34	16	6	2	0	0	0	0	0	0	0	
Charging Vehicle (Normet)	0	0	0	0	0	0	1	12	27	27	27	28	28	29	29	29	29	29	29	29	28	28	29	29	29	29	29	26	17	8	3	1	0	0	0	0	0	0	0	0	
Utility Vehicles (Normet - Lifting & Transport)	0	0	0	0	0	0	2	24	54	54	54	56	56	58	58	58	58	58	58	58	56	56	58	58	58	58	58	58	52	34	16	6	2	0	0	0	0	0	0	0	
Ancillary Vehicles																																									
LDVs	0	0	0	0	0	0	4	18	20	18	17	18	16	15	15	15	15	15	15	15	17	16	16	17	15	15	15	15	13	9	4	2	1	0	0	0	0	0	0	0	
Fuel & Water Bowser - Bowsers	0	0	0	0	0	0	1	3	7	7	7	7	7	8	8	8	8	8	8	8	7	7	8	8	8	8	8	7	5	2	1	1	0	0	0	0	0	0	0		
Motor Grader - Graders	0	0	0	0	0	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	0	0	0	0	0	0	0		

Table 16.15: Active Fleet Requirements for the South Shaft Underground LoM

South Shaft	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	37	38	39	40	
Development Fleet																																									
Drill Rig - Sandvik DD321 (or similar)	0	0	0	0	0	1	7	16	10	5	1	2	0	2	3	0	2	4	4	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
LHD - Sandvik LH410 (or similar)	0	0	0	0	0	1	10	23	14	8	1	2	0	2	4	0	2	5	5	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Roofbolter - Sandvik DS312 (or similar)	0	0	0	0	0	1	13	30	18	10	1	3	0	3	6	0	3	7	7	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Charging Vehicle (Normet)	0	0	0	0	0	1	3	6	4	2	1	1	0	1	1	0	1	2	2	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Utility Vehicles (Normet - Lifting & transport)	0	0	0	0	0	1	7	16	10	5	1	2	0	2	3	0	2	4	4	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Stoping Fleet																																									
Drill Rig - Sandvik DD321 (or similar)	0	0	0	0	0	0	2	46	54	54	56	58	58	58	56	58	58	56	56	56	58	58	58	58	58	58	58	50	28	16	10	2	0	0	0	0	0	0	0	0	0
LHD - Sandvik LH410 (or similar)	0	0	0	0	0	0	1	23	27	27	28	29	29	29	28	29	29	28	28	28	29	29	29	29	29	29	29	25	14	8	5	1	0	0	0	0	0	0	0	0	0
Roofbolter - Sandvik DS312 (or similar)	0	0	0	0	0	0	2	46	54	54	56	58	58	58	56	58	58	56	56	56	58	58	58	58	58	58	50	28	16	10	2	0	0	0	0	0	0	0	0	0	0
Charging Vehicle (Normet)	0	0	0	0	0	0	1	23	27	27	28	29	29	29	28	29	29	28	28	28	29	29	29	29	29	29	29	25	14	8	5	1	0	0	0	0	0	0	0	0	0
Utility Vehicles (Normet - Lifting & transport)	0	0	0	0	0	0	2	46	54	54	56	58	58	58	56	58	58	56	56	56	58	58	58	58	58	58	58	50	28	16	10	2	0	0	0	0	0	0	0	0	0
Ancillary Vehicles																																									
LDVs	0	0	0	0	0	1	7	26	22	18	15	16	15	16	17	15	16	18	17	15	15	15	15	15	15	15	15	13	7	4	3	1	0	0	0	0	0	0	0	0	
Fuel & Water Bowser - Bowsers	0	0	0	0	0	0	1	6	7	7	7	8	8	8	7	8	8	7	7	7	8	8	8	8	8	8	8	7	4	2	2	1	0	0	0	0	0	0	0	0	
Motor Grader - Graders	0	0	0	0	0	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	0	0	0	0	0	0	0	0	0	

16.2 VENTILATION AND COOLING

The ventilation design for the Project is based on an industry-accepted ventilation factor to dilute the fumes generated by diesel equipment, the factor of 0.06 m³/s/kW was applied to determine the ventilation requirement for each Shaft Complex.

The ventilation requirement per panel is defined by the primary diesel equipment in the stope, the ventilation requirement is detailed in Table 16.16.

Table 16.16: Ventilation Requirement Per Panel

Description	No. Machines	kW Rating	Quantity Required (m ³ /s)
LHD	1	185.00	11.10
Roof Bolter	3	74.00	13.32
Face Drill	2	74.00	8.88
Total Quantity Required for Machines in Panel			33.30
Leakage		25%	8.33
Contingency		30%	9.99
Total Quantity Required in Panel			51.62

The production required defines the total active panels and ultimately the total ventilation requirement. The total air quantity required for each Shaft Complex is detailed in Table 16.17. A ventilation study was carried out for alternative mine scenarios as part of the PEA design process (Global Mine Design, 2026). A maximum shaft ventilation capacity was estimated at 1,680 m³/s for a 3-shaft system, assuming an internal diameter of 9.5 m per shaft.

Table 16.17: Total Ventilation Required

Description	Unit	Value
Production Required	Mt/a	11.5
Stoping Rate per Panel	t/m	33,141
No. of Active Panels	No.	29
Total Quantity Required per Panel	m³/s	52
Total Quantity Required	m³/s	1,497

The ventilation will be balanced over the entire mining area to ensure each panel receives the required ventilation. The decline systems will be used as the main intake and return airways to and from the vertical Shaft Complexes.

Based on the limits provided in the ventilation study (Global Mine Design, 2026) completed in support of the Project the maximum velocities in each drive were used to determine if the planned decline system can provide sufficient ventilation. The ventilation study identified that based on the planned active equipment over the LoM, the general ventilation thresholds of 1,680 m³/s will be achieved, however additional ventilation headings will likely be required to reduce ventilation velocities to acceptable levels. The ventilation study also identified that ramp-up periods have additional active equipment for development, and these years are placing strain on the total ventilation capacities.

The calculation in Table 16.18 shows a minimum of four ventilation districts will be required to maintain the velocity limits in the decline system.

Table 16.18: Required Ventilation Districts

Description	Unit	Value
Return Airways		
Return Airway Area (7.0 mW x 6.0 mH)	m ²	42.0
Maximum Velocity in RAW	m/s	5
RAW Quantity (2 x Drives)	m ³ /s	420
Mining District Required	No.	4
Intake Airways		
Intake Airways	m²	80.0
Conveyor Drive (6.0 mW x 5.0 mH)	m ²	30.0
Men and Material (5.0 mW x 5.0 mH) x 2	m ²	50.0
Maximum Velocity in RAW		
Conveyor Drive	m/s	5
Labour and Material	m/s	6
Intake Quantity (3 x drives)	m ³ /s	450
Mining District Required	No.	3

The development layout was done to ensure there is four main mining areas which can be defined as the required ventilation districts, to ensure the ventilation is balanced around the Shaft Complexes. The development layout detailing the planned mining directions and the main intake and return airways is illustrated in Figure 16.16.

The ventilation layout showing some detail on the decline system and stoping panels is illustrated in Figure 16.17.

Ventilation equipment requirements for either the North or South mine operations are summarized in Table 16.19. The main ventilation fans would be positioned on the surface exhaust shaft, pulling the exhaust air through the mine and into exhaust shaft. An equivalent 18 MW of fan power capacity will be required for each of the exhaust shafts. Auxiliary ventilation fans would be positioned throughout the mine panel areas, directing fresh air into the active headings.

Table 16.19: Ventilation Equipment

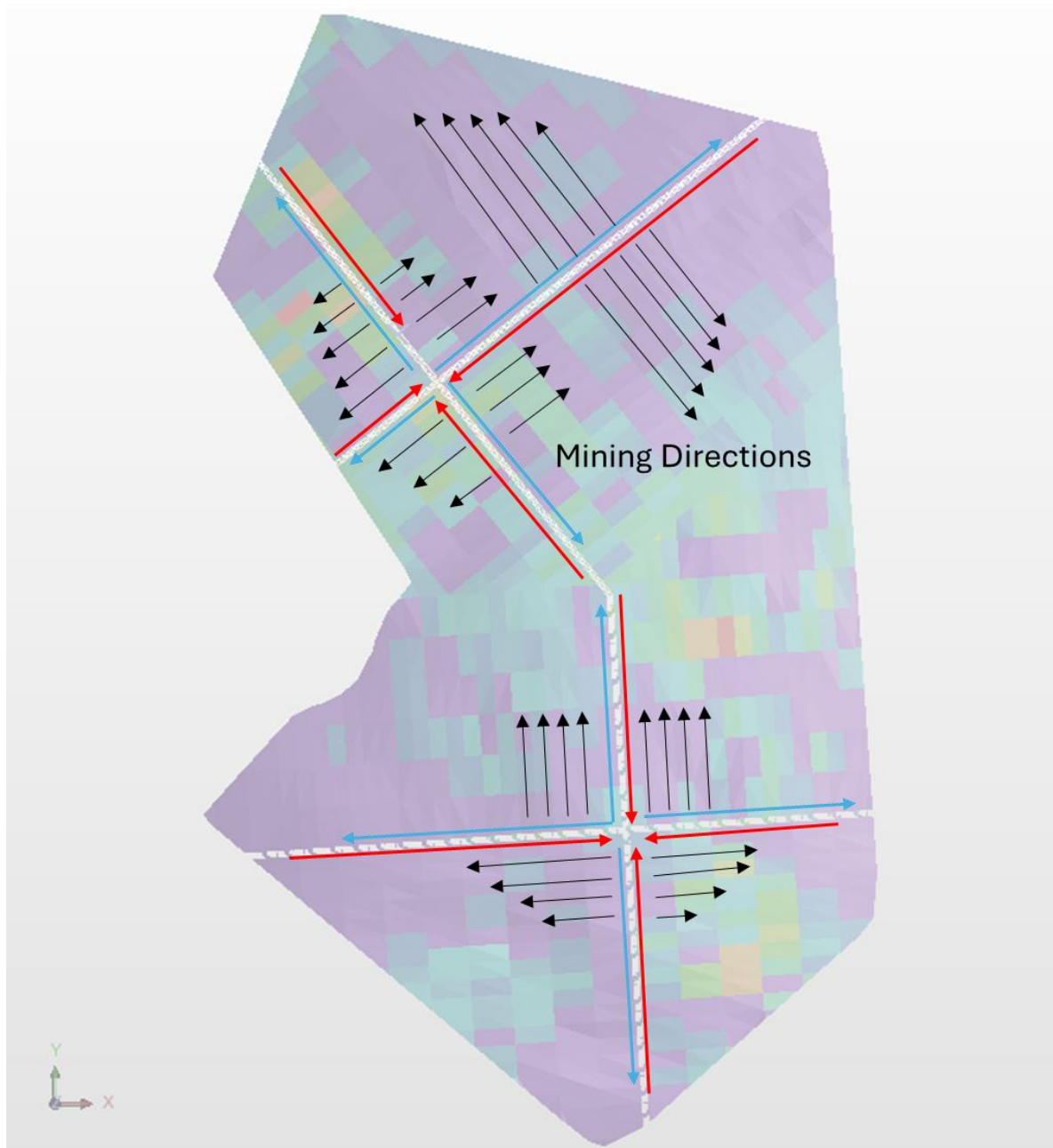
Ventilation Equipment	Quantity	Fan Size
Main Fans (Surface)	4	4.5 MW
Auxiliary Fans (Underground)	96	110 kW

Natural rock temperatures on the ore level are expected to range between 55°C and 70°C. The mine will require a major air-cooling system to maintain working temperatures below regulated requirements. A basic heat model was developed in Ventsim with the parameters outlined in Table 16.20.

Table 16.20: Ventsim Heat Model Assumptions

Ventsim Assumptions	Value (°C)
Geothermal Gradient	2.5 / 100 m
Surface Rock Temperature	20
Surface Dry Bulb Temperature (summer months)	27
Allowable Working Temperature (without restrictions)	28
Allowable Working Temperature (with restrictions)	33

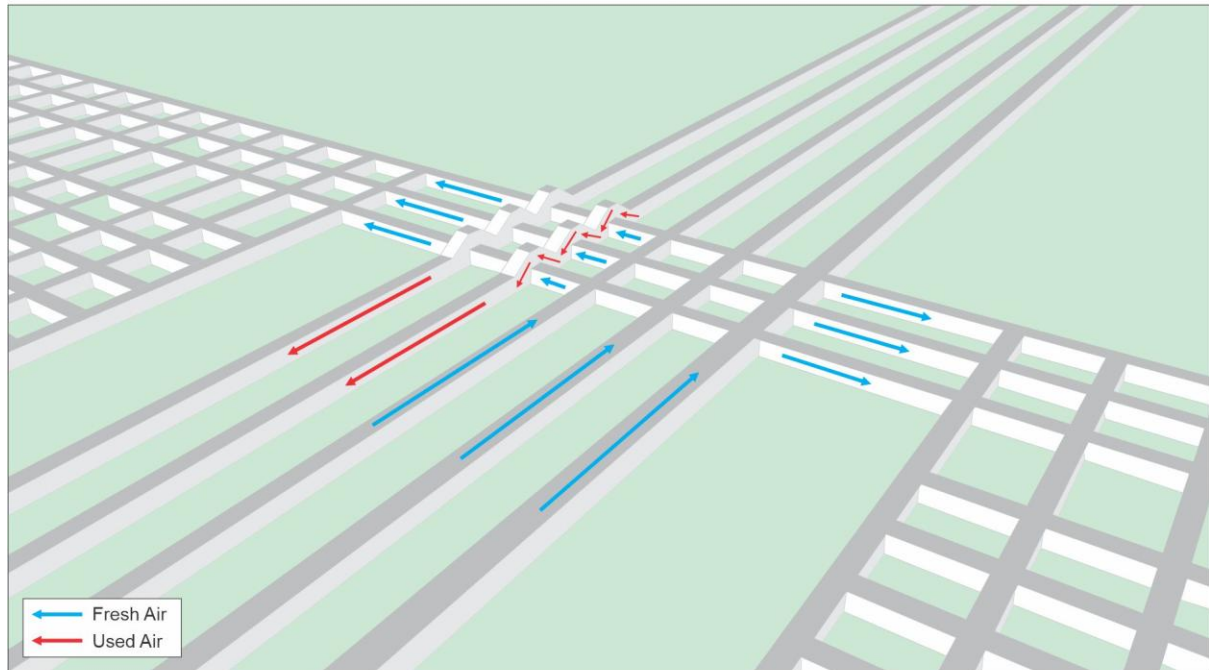
Figure 16.16: Ventilation Layout



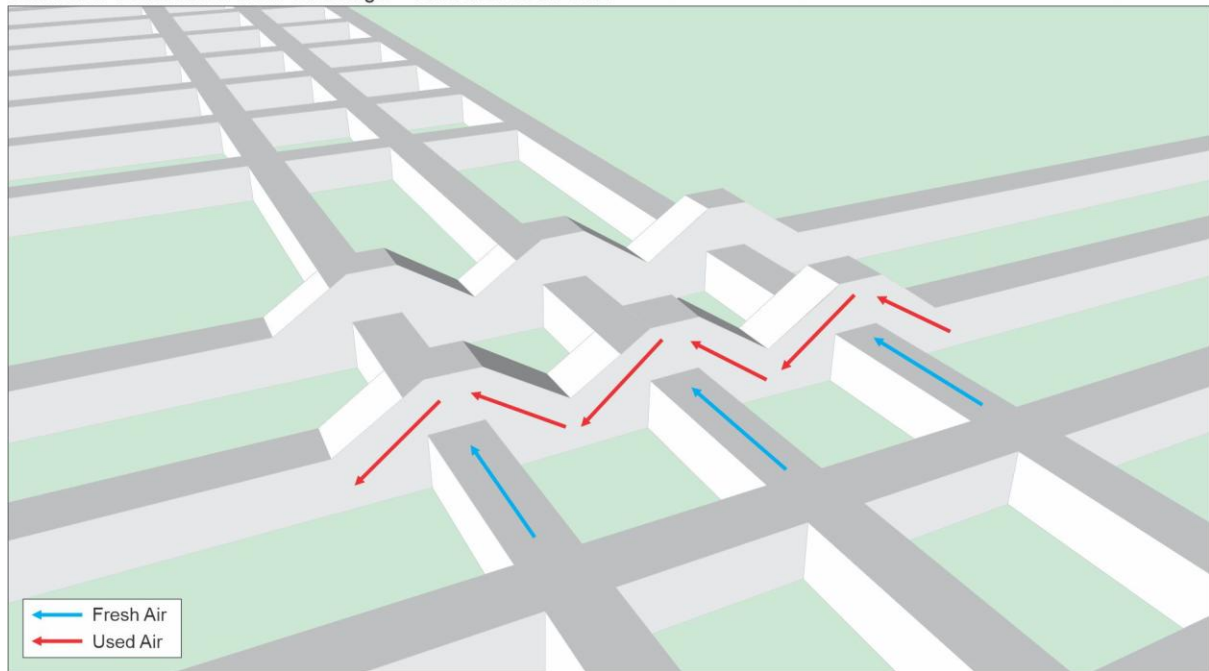
Source: Micon, 2026.

Figure 16.17: Air Crossing Layout

Ventilation Construction of Air-Crossings



Ventilation Construction of Air-Crossings – Ventilation to a Panel



Source: Micon, 2026.

Air cooling requirements equivalent to 40 MW of cooling are estimated for each of the North and South mine areas. The cooling stations would be positioned near the fresh air intake shafts. In winter months, air heating will be required in prolonged periods of sub-zero temperatures to prevent freezing of services and ice formations in the shafts.

Detailed ventilation layouts and modelling have not been carried out for the current arrangement. The ventilation study has outlined potential constraints, alternatives and next steps to further assess the ventilation configurations for the mine design. Alternative configurations could include

return airways positioned within the overlying anhydrite units, or dedicated return airway corridors separate from the fresh air corridors. Battery electric vehicles and 5-shaft versus 3-shaft configurations require more detailed assessment. These alternatives, supported by detailed ventilation modelling, will be assessed in subsequent design stages. Heat modelling should also incorporate heat generated by equipment and groundwater inflows.

16.3 UNDERGROUND INFRASTRUCTURE

Underground infrastructure requirements can be summarized in Table 16.21.

Table 16.21: Underground Infrastructure Requirements

Description	Unit	Value
Ventilation	m ³ /s	1,497
Electrical Power	MW	110
Water Management	m ³ /min	3.0
Compressed Air	m ³ /s	12.2
Rock Breaker Stations	No	24
Belt Conveyors	m	9,200
Ore Bins	No.	4
Explosive Storage	No.	6
Fuel Storage	No.	1
Equipment Maintenance Shop	No.	1
Refuge Stations	No.	12

16.3.1 Underground Ore Sizing and Conveying

Every operational mining panel is equipped with a dedicated rock breaker station, which allows ore handling and sizing to occur close to the source.

After blasting, the loose and fragmented ore will be transported by LHD vehicles, from the mining face directly to the rock breaker station.

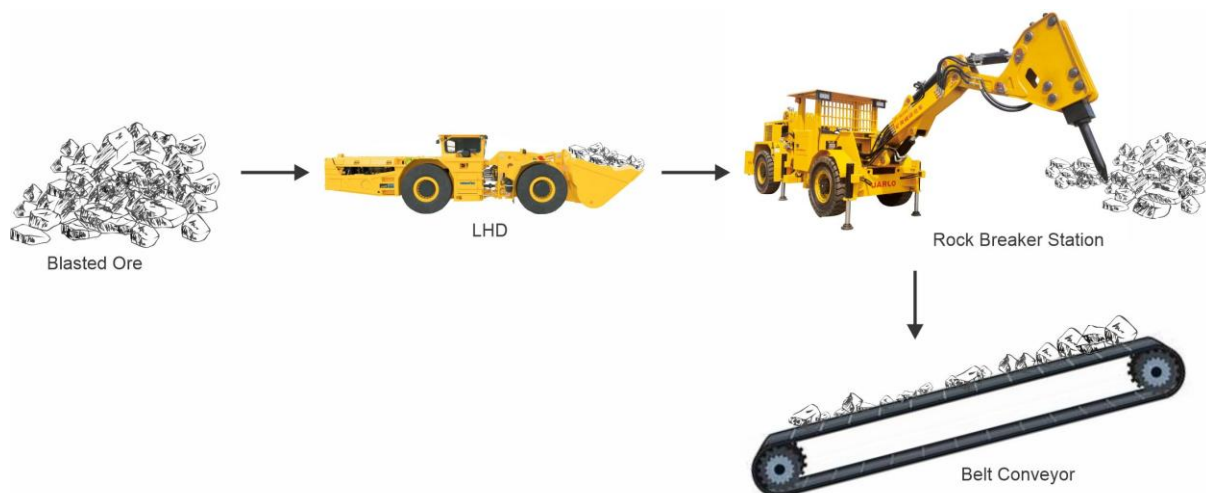
At the rock breaker station, a hydraulic rock breaker is mounted on a pedestal above a grizzly dump. The grizzly dump consists of a grid of heavy bars that permits smaller pieces of ore to pass through, while larger rocks remain on top. The hydraulic rock breaker is then used to break down oversized rocks that cannot pass through the grizzly screen, ensuring that all ore is reduced to a manageable size for further handling.

Once the ore is appropriately sized by the rock breaker, it falls through the grizzly and onto a panel belt conveyor. This conveyor transports the ore to the main haulage conveyor transporting the ore to the ore bin located near the Production shaft.

The main haulage conveyor is a larger, central conveyor system that collects ore from multiple panel conveyors. It is engineered for long-distance transport within the mine extending several kilometres.

At the end of the main haulage conveyor, ore is deposited into an ore bin located adjacent to the production shaft. The ore bin serves as a temporary storage and buffer, regulating the flow of material into the shaft hoisting system for transport to the surface. An illustration of the simplified ore sizing and conveying is provided in Figure 16.18.

Figure 16.18: Simplified Ore Sizing and Conveying



Source: Micon, 2026.

16.3.2 Refuge Stations

Refuge stations will be positioned so that all employees can access a refuge in less than 10 minutes from the moment they leave their workplace or at a maximum of 1,000 m from any workplace. Refuge stations used in the underground mine will be constructed of high fire-resistance rating (e.g., one hour minimum) and be capable of being completely sealed to prevent the entry of toxic gases or smoke, located within a reasonable distance from active working areas. Modular, prefabricated refuge stations may be used to facilitate fast and efficient construction. Modular refuge stations are currently being used in many large-scale underground operations in countries such as Australia, Canada and Mexico.

Refuge stations will be of sufficient size to accommodate the maximum number of workers in the vicinity and provide life support for a 36 hour period whilst maintaining a positive pressure relative to the outside environment to prevent contamination ingress. They will provide an adequate supply of breathable air, either via a dedicated compressed air line from the surface or independent internal oxygen sources (e.g. oxygen cylinders).

Refuge stations will be equipped and supplied with:

- Air scrubbing systems to remove contaminants like carbon dioxide.
- Atmospheric monitoring equipment for O₂, CO₂, and CO levels, with appropriate alarms.
- Sufficient potable water.
- Means of voice communication with the surface and mine rescue teams.
- Backup power supply for lighting, environmental controls, and communication systems.
- First-aid supplies, basic tools, and private toilet facilities.
- Posted emergency procedures, mine maps, and clear signage.

16.3.3 Compressed Air

A surface-mounted compressed air facility will be established adjacent to the headframe complex to supply compressed air for all underground mining operations. This facility will be equipped with multiple rotary screw compressors. Each compressor will be integrated air dryers to ensure the

delivery of moisture-free air. High-capacity air receivers will be installed to buffer fluctuations in demand and maintain stable system pressure.

The compressed air generated at the plant will be conveyed through a dedicated pipeline system. These pipelines will be routed vertically through the Service shaft, utilizing engineered supports and vibration isolation to minimize mechanical stress and ensure long-term integrity. At the shaft station, the pipeline will connect with the underground pneumatic distribution network, which is designed to deliver compressed air to various mining zones, including drilling stations, loading areas, and refuge chambers.

Redundant safety features, such as pressure relief valves and automated shutdown protocols, will be incorporated to mitigate risks associated with over-pressurization or equipment failure.

This integrated approach ensures reliable, high-quality compressed air supply tailored to the operational demands of the mine, supporting both productivity and safety objectives.

The peak compressed air demand is estimated to be 12.2 m³/sec (26,000 CFM).

16.3.4 Electrical Power

The mine and mill complex will be supplied with high voltage (HV) electrical power from two independent sources, specifically at 110 kV and 50 Hz, delivered via connection to the main utility power lines. This configuration aligns with industry standards for HV supply, where voltages equal to or exceeding 110 kV are typical for primary distribution to large-scale mining operations.

Electrical power will be routed through a main substation, which serves as the principal node for voltage transformation and distribution. The main substation will step down the incoming high voltage to a medium voltage level of 15 kV, suitable for underground transmission. Cables originating from a local substation will be installed, providing reliable power delivery to the underground mine infrastructure. A dual feed servicing different shafts is recommended as well as a ring main setup in the shaft bottom and infrastructure area.

Within the underground environment, a series of step-down transformers will be strategically located to further reduce the voltage to levels appropriate for the low-voltage distribution system. This multi-tiered approach ensures that electrical equipment and operational areas receive power at safe and efficient voltage levels, minimizing transmission losses and enhancing operational safety.

During the initial development phase, mobile substations will be deployed at intervals of approximately 500 m. These substations are designed to be relocated as the mine expands, ensuring that new development zones and operational areas are continuously supplied with adequate electrical power. This practice is consistent with best practices observed in technical feasibility studies, where mobile substations are initially used to support development and subsequently remain in place to supply permanent infrastructure.

Areas characterized by elevated power demand, such as conveyor drive stations, rock breaker stations and maintenance shops, will be prioritized for mobile substation deployment. These substations will provide the necessary capacity to support high-load equipment and will remain in-situ to facilitate ongoing operations.

In summary, the electrical power supply system for the mine and mill complex will be designed to deliver robust, scalable, and reliable power through a hierarchy of substations and transformers, with provisions for mobile infrastructure during development and permanent installations for ongoing operations. This system supports both routine and high-demand activities, ensuring operational continuity and compliance with industry regulations.

At this stage of Project planning, the total connected electrical load for each Shaft Complex is estimated to be approximately 110 MW. This figure encompasses all major operational demands, including hoisting systems, ventilation and cooling infrastructure, dewatering pumps, material handling equipment, and supporting surface and underground facilities. As engineering progresses and equipment specifications are refined, these power requirements will be validated and updated to ensure accurate capacity planning and grid integration.

16.3.5 Fuel Storage

Fuel will be stored in large, engineered tanks on the surface, designed to meet both operational needs and regulatory requirements. These tanks will be double-walled and placed within bunded containment areas to prevent leaks and environmental contamination.

A dedicated fuel line (routed through a service shaft) delivers fuel underground. This line is engineered for safety, with leak detection systems, corrosion-resistant materials. The use of high-integrity pumps and flowmeters ensures accurate and efficient fuel transfer, minimizing downtime and operational risks.

Underground, specialized fuel and lube bays will be constructed to store and distribute fuel and lubricants to mining equipment. These bays are strategically located to maximize accessibility for machinery and minimize travel time. Tanks in these bays are also double-walled and installed within impermeable containment berms, meeting the highest safety and environmental standards.

To service equipment that cannot efficiently access the bays, mobile fuel and lube trucks are deployed. These trucks are equipped with multiple tanks for different fluids (diesel, oil, grease, coolant) and advanced dispensing systems. They enable on-site refuelling and maintenance, reducing equipment downtime and improving operational efficiency.

16.3.6 Explosive Storage

All explosives required for underground work will be supplied by an approved explosives contractor in full compliance with regulatory and site specific safety standards. Upon arrival, the materials will be unloaded at surface and transported underground via Service Shaft. They will then be delivered to designated underground magazines using a service truck designed for handling explosive products. From these storage facilities, explosives will be issued as needed to support drilling, blasting, and ongoing production activities. The underground explosives magazines will be ventilated with an independent fresh air current.

A secure explosives magazine building will be constructed on surface within a designated compound to respect safety distances. It will be designed in accordance with regulatory requirements, and for the interim storage of material prior to storage underground.

16.3.7 Equipment Maintenance Shop

Major mechanical maintenance, such as overhauls and complex repairs, is primarily conducted at the surface workshop. This facility is equipped for extensive servicing, leveraging specialized tools and resources not easily transported underground.

For operational efficiency, a dedicated underground garage will be constructed and fully equipped to handle both major repairs and all minor maintenance tasks. This setup allows rapid response to equipment issues, minimizing downtime and avoiding the logistical challenges of moving machinery to the surface for every intervention.

The underground garage will be designed as a multifunctional hub, supporting a wide range of operational and maintenance activities. The garage facilities will include:

- **Parking Area:** Secure space for equipment awaiting service or deployment.
- **Washing Bay:** Enables thorough cleaning of machinery, essential for maintenance and inspection.
- **Five Maintenance Bays:** Dedicated areas for simultaneous servicing of multiple units, improving workflow and reducing bottlenecks.
- **Supervisor's Office:** On-site management and coordination of maintenance activities.
- **Tool Crib:** Centralized storage for tools and spare parts, ensuring quick access for technicians.
- **Tire Storage Bay:** Organized space for tire inventory and repairs.
- **Welding Bay:** Specialized area for structural repairs and fabrication.
- **Electrical Workshop:** Supports diagnostics and repairs of electrical systems.
- **Sump for Water Management:** Manages runoff and wastewater, maintaining a safe and compliant work environment.

16.4 WATER MANAGEMENT

The water will be collected in a holding tank at the base of the shaft, from where it will be pumped to the surface. The total lift is expected to be between 1,500 m and 1,800 m, depending on the location of the shaft. It may be possible to utilise a single pump with a very high working pressure, although it is more likely that the system will operate at a maximum head of 500 m (5,000 kPa). This would require 3 to 4 booster pump stations to be located in the shaft.

The most likely type of pump will be positive displacement, capable of handling water with abrasive suspended solids. It is assumed that there will be duty and standby items for key elements of the system.

Localised sumps near working areas will store excess water with gathering pumps used to deliver dirty water to the main sump and the shaft bottom.

16.5 PRODUCTION SCHEDULE

16.5.1 Production Schedule Strategy

The production schedule is based on 32,000 t/d (11.5 Mt/a) run of mine (RoM) feed to the processing plant for each Shaft Complex.

The aim is to mine at a constant volume as far as possible; this will reduce the need to adjust the mining rates and associated mining fleet provided to produce the required tonnes. This will optimise equipment and resources and reduce fluctuations in production costs per tonne mined.

16.5.2 Scheduling Input Parameters

The production schedule assumes 6 working days in a 7-day cycle, with a maximum of 48 hours per week resulting in three 8-hour shifts per day. The working time is detailed in Table 16.22.

Table 16.22: Working Time

Description	Unit	Value
Working Days in a Month	Days	26
Working Days in Cycle	Days	6
Cycle Time	Days	7
Effective Working Days in Week	Days	6
Maximum Hours per Week	hr	48
Shift Time	hr:min	8:0
Production Efficiency	%	85%
Shift per Day	No.	3
Working Hours per Day	hr	20

16.5.3 Underground Development Rates

The development rates are calculated as per Table 16.23. The development rates considered multiple blast scenarios per shift.

Multiple blasts are required to reach the development targets in the decline system.

Table 16.23: Advance Rates Per Month

Description	Unit	Value
Drill Length	m	4.5
Blast Efficiency	%	90%
Advance per Blast	m	4.05
Lost Blast	%	85%
Advance Rate per Month	m/month	90
Advance Rate per Day	m/d	3.44
Used in the Production Schedule	m/d	3.5

16.5.4 Underground Stopping Rate

Table 16.24 shows the details of the planned stopping rate for a panel.

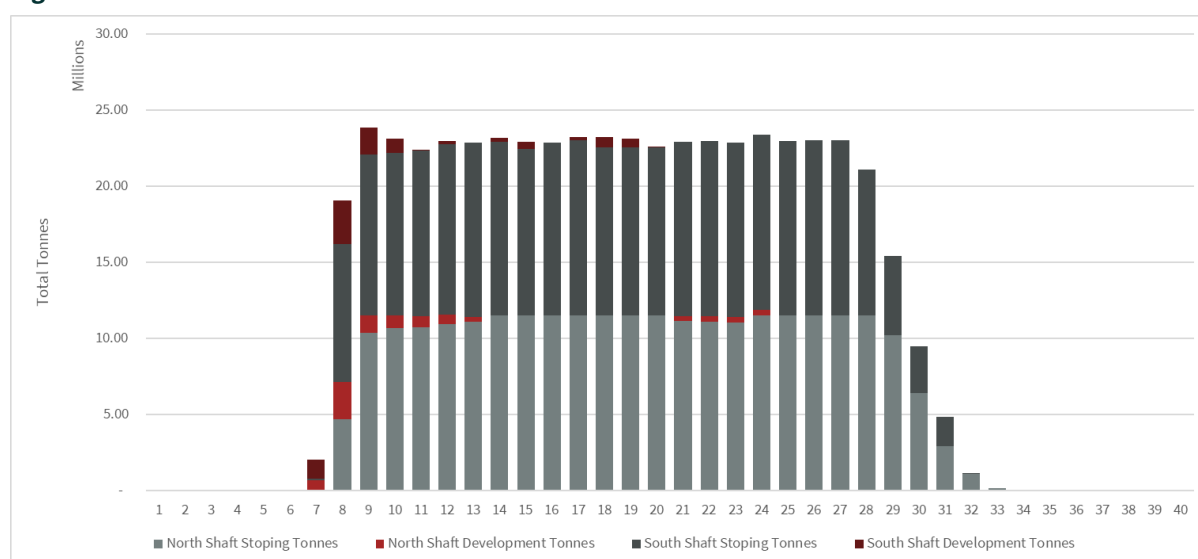
Table 16.24: Stopping Rate

Description	Unit	Value
Stopping Width	m	5
Stopping Height	m	2
Density	kg/m ³	2.53
Tonnes Required	t/d	32,000
Blasting		
Drill Length	m	4.5
Blast Efficiency	%	90%
Advance per Blast	m	4.05
Lost Blast	%	80%
Tonnes per Blast	t	102
Single Heading Advance	m/month	85
Single Heading Advance	t/month	2,138
Drilling		
Time to Drill Face (Benchmarked)	hr	2
No. Faces per Drill Rig	No.	10
Available Faces (16 faces in advance and 15 bords)	No.	31
Face Availability (Assumption)	%	50%
Faces Blasted per Day in the Panel	No.	16
Stopping Rate per Panel	t/m	33,141
Stopping Rate per Panel	t/d	1,271
Used in the Production Schedule	t/d	1,300

16.5.5 Production Results

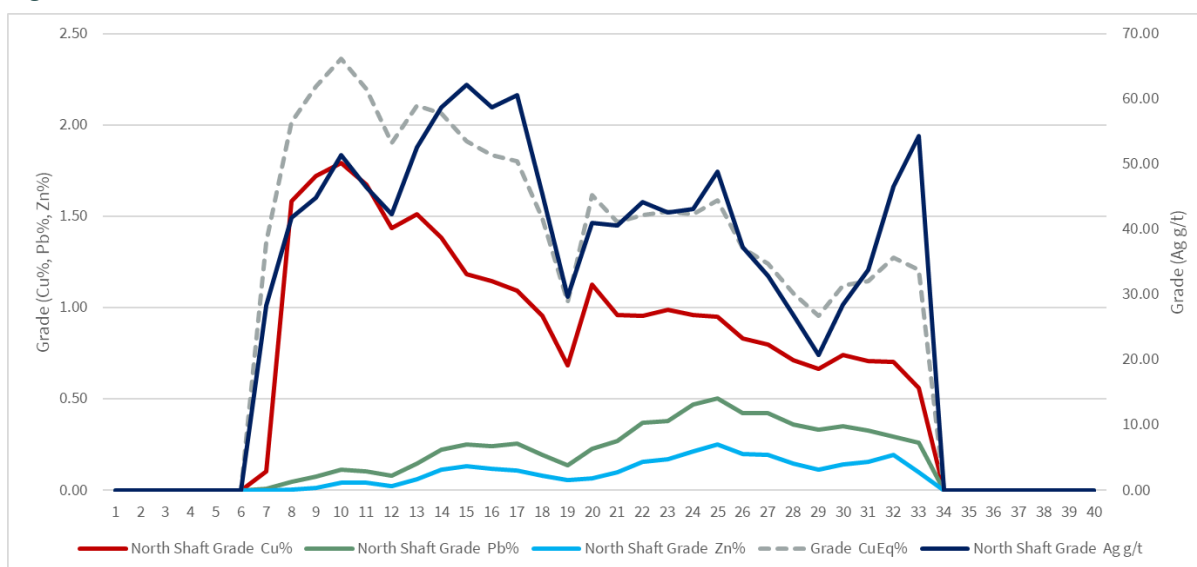
The production profile showing the total tonnes mined and mined grade are illustrated in Figure 16.19. All tonnes mined report to the processing plant for treatment.

Figure 16.19: Production Schedule – Total Tonnes



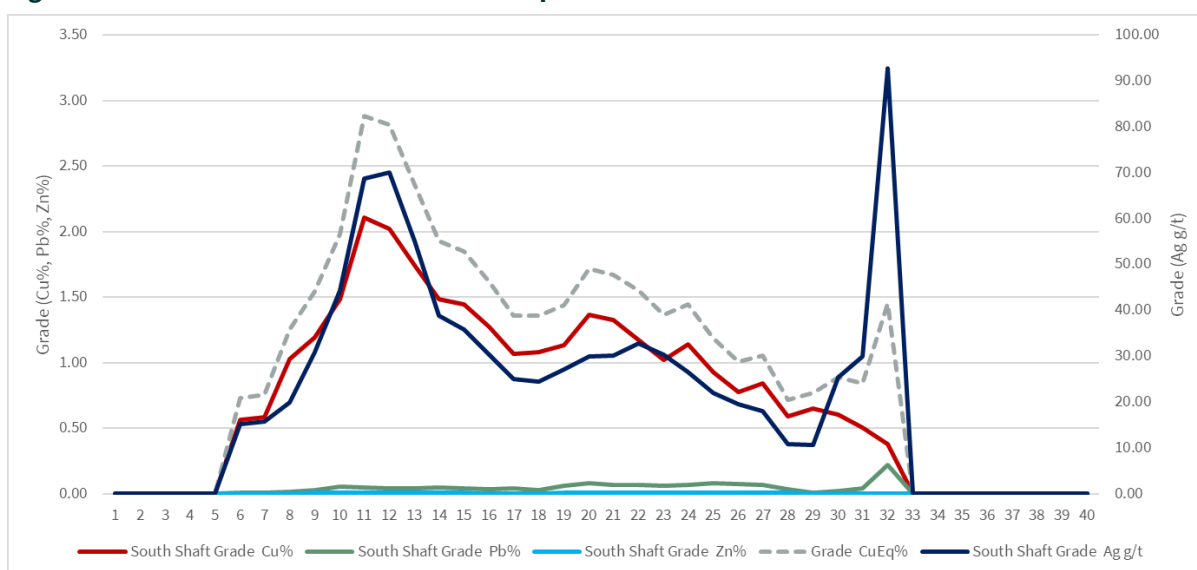
The grade profile for the North shaft is illustrated in Figure 16.20.

Figure 16.20: Grade Profile - North Shaft Complex

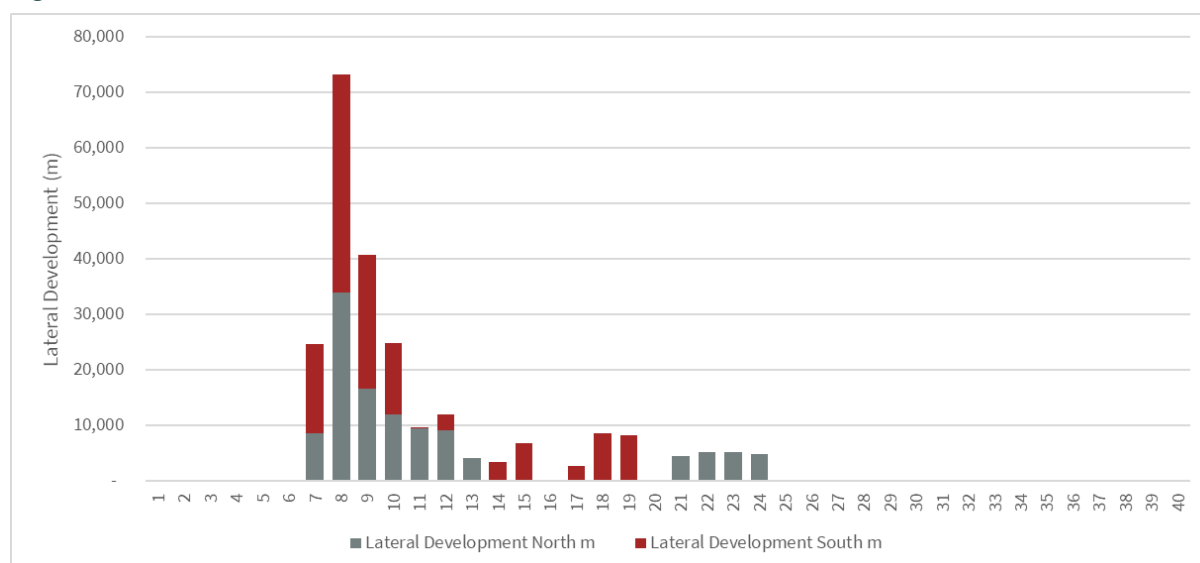


The grade profile for the South shaft is illustrated in Figure 16.21.

Figure 16.21: Grade Profile - South Shaft Complex



The underground development metres profile is illustrated in Figure 16.22.

Figure 16.22: Development Profile


16.5.6 Production Summary

The total production is summarized in Table 16.25.

Table 16.25: Production Summary - Mined

Parameter	Tonnage (kt)	Grade				Contained Metal			
		Cu (%)	Ag (ppm)	Pb (%)	Zn (%)	Cu (kt)	Ag (kt)	Pb (kt)	Zn (kt)
North Shaft									
Stoping	250,680	1.13	44.52	0.27	0.12	2,832.3	11.2	672.3	289.3
Development	8,191	0.71	23.09	0.09	0.04	58.1	0.2	7.4	3.7
Total	258,871	1.12	43.84	0.26	0.11	2,890.4	11.3	679.7	293.0
South Shaft									
Stoping	242,338	1.25	32.81	0.05	0.01	3,021.2	8.0	121.8	12.6
Development	9,304	0.54	16.74	0.01	0.00	50.4	0.2	1.4	0.2
Total	251,642	1.22	32.22	0.05	0.01	3,071.5	8.1	123.1	12.8
Combined									
Stoping	493,018	1.19	38.77	0.16	0.06	5,853.5	19.1	794.1	301.9
Development	17,495	0.62	19.72	0.05	0.02	108.5	0.3	8.8	3.9
Total	510,513	1.17	38.11	0.16	0.06	5,961.9	19.5	802.9	305.7

16.5.7 Mineral Resources used in the Mine Plan

The mineral resources used in the mine plan presented in Table 16.26 are derived from NI 43-101 compliant Mineral Resources and are used solely to demonstrate potential extractable grades and tonnes for the purposes of mine planning, production scheduling, and DCF analysis within this PEA. The mineral resources used in the mine plan are a sub-set of the Measured, Indicated and Inferred Mineral Resources declared in Section 14.11 and do not constitute NI 43-101 compliant Mineral Reserves.

Table 16.26: Mineral Resources used in the Mine Plan

Parameter	Tonnage (kt)	Grade				Contained Metal			
		Cu (%)	Ag (ppm)	Pb (%)	Zn (%)	Cu (kt)	Ag (kt)	Pb (kt)	Zn (kt)
North Shaft									
Measured	0	0.00	0.00	0.00	0.00	0.0	0.0	0.0	0.0
Indicated	209,336	1.11	46.74	0.24	0.10	2,324.7	9.8	507.3	212.3
Inferred	49,535	1.14	31.65	0.35	0.16	565.6	1.6	172.2	79.9
Total	258,871	1.12	43.85	0.26	0.11	2,890.3	11.4	679.5	292.2
South Shaft									
Measured	131,821	1.26	36.07	0.03	0.00	1,658.2	4.8	39.0	4.5
Indicated	93,311	1.12	26.26	0.03	0.00	1,047.5	2.5	28.2	3.6
Inferred	26,510	1.38	33.85	0.21	0.02	365.1	0.9	55.6	4.6
Total	251,642	1.22	32.20	0.05	0.01	3,070.9	8.1	122.7	12.7
Combined									
Measured	131,821	1.26	36.07	0.03	0.00	1,658.2	4.8	39.0	4.5
Indicated	302,647	1.11	40.43	0.18	0.07	3,372.1	12.2	535.5	215.9
Inferred	76,046	1.22	32.42	0.30	0.11	930.8	2.5	227.8	84.5
Total	510,513	1.17	38.11	0.16	0.06	5,961.2	19.5	802.2	304.9

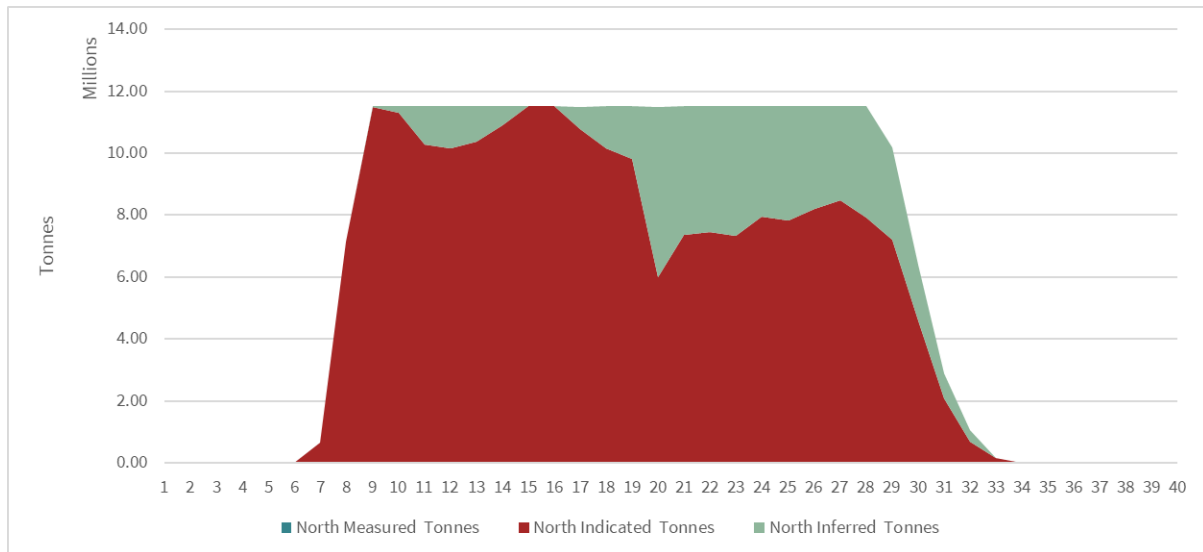
Notes:

- The MRE has been prepared in accordance with NI 43-101 with an effective date of January 9, 2026. Mrs. Liz de Klerk, Pr.Sci.Nat., QMR and Dr. Ryan Langdon, Ph.D, CGeol, of Micon are the QPs responsible for the MRE.
- Cu_{eq} was calculated using the formula $(Cu\% \times 2204.62 \times Cu_{rec} \times Cu_{price}) + (Ag_{ppm} \times 0.032 \times Ag_{rec} \times Ag_{price}) / 2204.62 \times Cu_{rec} \times Cu_{price}$. Where % = Cu grade, ppm = Ag grade, rec = recovery, price = price.
- To RPEEE, mining envelopes used to identify spatially continuous mineralization within potentially mineable shapes using reasonable assumptions based on neighbouring operations and long-term price trends to calculate a pay limit copper grade. These include metal prices of US\$/lb 4.75 for Cu and process recoveries of 89%, payability of 96.5%, transport charges of US\$/wmt 10, treatment charges of US\$/t 50, and refining charges of US\$/t 0.05 for Cu. Operational expenditure costs (OPEX) were determined with reference to comparable operations with some deductions for efficiency savings. These include costs for mining of US\$/t 54.5, processing of US\$/t 8.5, and General and Administrative (G&A) of US\$/t 5.5.
- A target head grade was selected at 0.95% Cu to generate the mining envelopes and was based on the optimal cut-off grade to maximise a theoretical NPV.
- For the mining envelopes a minimum mining height of 1.8 m was used, with a room width of 6.0 m. No additional dilution from the footwall or hanging wall was applied.
- A mining loss factor of 15% was applied to the reported Mineral Resources inside the defined mining envelopes. This adjustment accounts for material that is proven to exist geologically, but cannot be physically recovered in the actual mining operation and therefore does not have RPEEE.
- Diluted tonnages and grades are reported based on the defined mining envelopes. Contained Pb and Zn inside these shapes with RPEEE were included in the Mineral Resources.
- Mineral Resources are not Mineral Reserves and have not demonstrated economic viability. There is no certainty that all or any part of the estimated Mineral Resources will be converted into Mineral Reserves.
- Grade interpolation was by Ordinary Kriging (OK) with a block size of 20 m (X) by 20 m (Y) by 0.6 m (Z). A stratigraphic transform was applied to the data to restore the original continuity, and interpolation was performed in this space before back transforming to real-world space.
- Density domains were modelled, and median density values were assigned: low density 2.25 t/m³, medium density 2.53 t/m³, high density 2.76 t/m³, and very high density 2.93 t/m³.
- The MRE was classified in accordance with NI 43-101 (CIM, 2014). Mineral Resources were classified as Measured, Indicated, and Inferred. Blocks where the nearest drill hole was >1,600 m were classified as Inferred, Measured blocks were classified as being <1,200 m to the nearest drill hole where the drill holes had been drilled on a regular spaced grid, all other blocks were classified as Indicated.

The totals presented in this table reported from the Mineral Resource models, are subject to rounding, and may not total exactly.

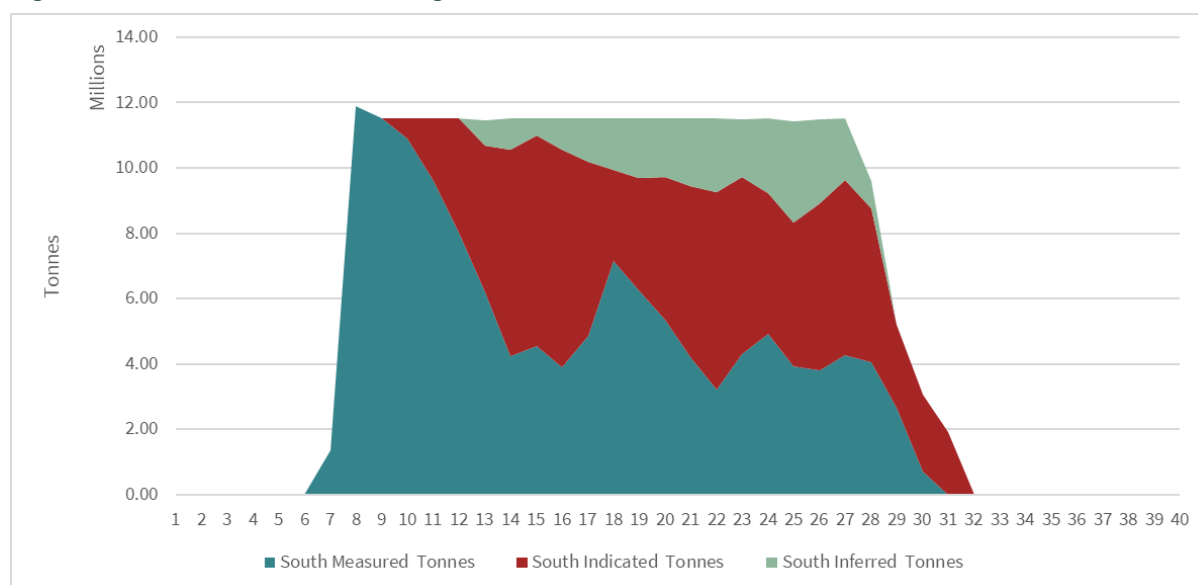
The Mineral Resource classification for the North Shaft Complex production profile is shown in Figure 16.23.

Figure 16.23: Mineral Resource Categories in Production Schedule – North Shaft Complex



The Mineral Resource classification for the South Shaft Complex production profile is shown in Figure 16.24.

Figure 16.24: Mineral Resource Categories in Production Schedule – South Shaft Complex



Inferred Mineral Resources have been included in the production schedule derived from the minerals resources used in the mine plan for the purposes of this PEA; however, there is no certainty that the results will be realized, and the inclusion of Inferred Mineral Resources is subject to the inherent geological uncertainty associated with this level of confidence.

16.6 BACKFILL STUDIES

Backfilling is an integral part of underground mining operations, providing support to excavated voids, reducing surface tailings disposal, controlling ground deformation, improving safety and enabling ore recovery. The selection of an appropriate backfill system depends on mining method, ground conditions, tailings characteristics, environmental constraints and economic considerations.

Several backfill types are used, broadly categorized as:

- Rock-based fills, including waste rockfill and cemented rockfill, which can provide high strength but require a reliable supply of suitable waste rock and typically return limited volumes of tailings underground.
- Hydraulic fill, consisting of classified tailings placed as a low-density slurry, historically used in some deep mining operations but reliant on drainage, decanting, and fine tailings management.
- Paste backfill, produced from dewatered tailings mixed with water and binder to form a high-density, non-segregating material that can be pumped and placed without the need for drainage.

Backfilling is a fundamental element of the mining strategy rather than solely a tailings management measure. It enables extraction sequences that would otherwise be constrained by stability, pillar loading and access, while also improving underground conditions by reducing open void volumes and limiting water ingress. For the Project, Lumina has confirmed that any tailings reporting to surface must be managed as dry stack and that conventional wet tailings storage facility (TSF) will not be permitted; this is a key design driver for both tailings management and backfill selection.

This Report provides a conceptual assessment of backfill requirements for the proposed Room and Pillar and Drift and Fill mining methods proposed at the Project. In the absence of site-specific tailings testwork, the assessment is based on reasonable engineering assumptions informed by industry practice, comparable operations and professional judgement appropriate for PEA level study.

At PEA level, backfill demand has been derived from annual void availability rather than detailed stope-level sequencing. Short-term operational constraints may influence the timing of backfill placement locally but are not expected to materially affect LoM backfill capacity or plant sizing. These aspects will be addressed during further studies at PFS level. The mine has been divided into two areas (North and South), each with an independent material balance and backfill requirements.

16.6.1 Backfill Methods

At the PEA stage, the purpose of this section is not to provide a comprehensive review of all backfill technologies but to identify those methods that are technically credible for this study and to justify the selection of the preferred backfill system based on the site-specific constraints.

16.6.1.1 Overview of Backfill Options

Waste rockfill and cemented rock fill are widely used where large volumes of competent waste rock are available. While these methods can provide high strength, their application is limited where waste rock supply is constrained and where the project objective includes maximising underground tailings placement. As such, rock-based fills are not considered suitable as a primary backfill solution for this Project.

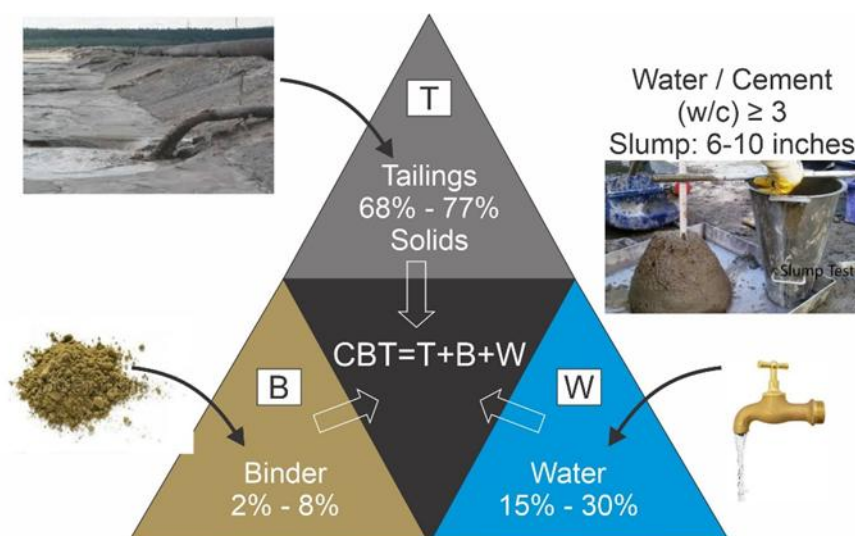
Hydraulic fill was screened as an option but is not compatible with the Project requirement for surface dry stacking with no wet TSF. The basis for this exclusion is provided in Section 16.6.2.1 Hydraulic Fill Assessment.

16.6.1.2 Selected Backfill Method – Paste Backfill

Paste backfill is produced from dewatered tailings, typically at solids concentrations of approximately 70% to 85% by weight, mixed with water and a proportion of binder to achieve the required strength and rheological properties.

Figure 16.25 illustrates the conceptual composition of cemented paste backfill. Tailings form the bulk of the mixture (around 68% to 77% solids by mass), binder (around 2% to 8%) governs the strength development, and water (around 15% to 30%) is adjusted to achieve a pumpable, non-segregating paste. Workability is commonly characterized by a slump of approximately 150 mm to 250 mm (6 inches to 10 inches).

Figure 16.25: Conceptual Cemented Paste Backfill Composition



The resulting paste exhibits low permeability, uniform behaviour and can be pumped over long distances and placed underground without the need for drainage or decant systems. Based on this screening assessment, paste backfill has been selected as the preferred backfill method for both mining scenarios considered in this PEA and forms the basis of the conceptual backfill design.

16.6.2 Backfill Design Overview

The mine design comprises two distinct mining method scenarios, one being Room and Pillar mining and the second Drift and Fill, each with different operational and geotechnical backfill requirements. As no tailings characterisation is yet available, this assessment presents a conceptual evaluation based on reasonable engineering assumptions appropriate for a PEA level study.

Room and Pillar Mining

Backfilling in this method is primarily driven by the objective of maximising the return of tailings underground. Strength requirements are primarily driven by confinement and deformation control; at depths of approximately 2 km, the backfill must exhibit predictable stiffness and confinement behaviour to manage high in-situ stresses.

Drift and Fill Mining

This method requires a competent, low permeability backfill capable of supporting sequential development, providing a stable working platform and controlling dilution. Strength development and uniformity are key design drivers.

16.6.2.1 Hydraulic Fill Assessment

Hydraulic fill was initially considered for the Room and Pillar mining scenario due to its historical application in deep underground mines and its ability to place large volumes of tailings underground. However, hydraulic fill systems rely on cyclone classification, producing a coarse underflow for underground placement and a fine overflow stream that typically requires management in a wet TSF.

Lumina has confirmed that all tailings reporting to surface must be managed via a Dry Stack Tailings Facility (DSTF) and that a conventional wet TSF will not be permitted. The fine tailings fraction generated by hydraulic fill does not lend itself to dry stack deposition due to poor filterability and

would therefore require wet tailings management, which is incompatible with the Project constraints.

In addition, hydraulic fill presents technical limitations for deep mining applications, including low strength and stiffness under high in-situ stress, higher permeability with increased groundwater and geochemical risk, sensitivity to drainage performance, and the potential for segregation and variable in-situ behaviour.

On this basis, hydraulic fill and cemented hydraulic fill have been excluded from further consideration in this study.

16.6.2.2 Paste Backfill Assessment

Following the assessment of hydraulic fill and its incompatibility with the Project's environmental and operational requirements, paste backfill has been selected as the preferred backfilling method for both the Room and Pillar and Drift and Fill mining scenarios.

Paste backfill offers several advantages that align directly with the Project constraints and geotechnical needs:

- Low permeability, which limits groundwater flow through the fill mass and reduces potential geochemical interaction and leaching risks.
- Predictable and controllable strength development, even at relatively low cement dosages, allowing the backfill to meet the strength demands of Room and Pillar mining and the higher structural requirements of Drift and Fill.
- Uniform, non-segregating behaviour, ensuring consistent in-situ performance in both large open rooms and narrow drift geometries.
- Compatibility with surface dry stack deposition, as the same filtration process used to prepare the paste also produces the filtered tailings required for the DSTF, eliminating the need for a conventional wet TSF.

Given these attributes, paste backfill satisfies the operational, geotechnical and environmental criteria of the Project. It enables the safe and efficient extraction of ore, maximises the volume of tailings returned underground, and integrates seamlessly with the dry stack surface storage strategy. Paste backfill therefore forms the basis of the conceptual backfill design.

16.6.3 Backfill Strength Requirements

The required strength of the paste backfill depends on the mining method, depth, excavation span, and sequencing. Strength requirements therefore differ between the Room and Pillar and Drift and Fill mining scenarios. The strength assumptions will be validated by future UCS testing on representative paste mixes and final strength criteria will be confirmed based on geotechnical design checks.

16.6.3.1 Room and Pillar Mining

In Room and Pillar operations backfill is primarily used to provide confinement, control deformation at depth, and enable the return of process tailings underground. Generally, the backfill is not required to provide major structural support or bearing capacity. Typical UCS ranges are:

- Low range (confinement and stability, not bearing): 0.2 MPa to 0.5 MPa.
- Moderate (support during pillar recovery or extraction): 1 MPa to 3 MPa.
- High (bearing and long-term structural support): 3 MPa to 5 MPa+.

For this study, Room and Pillar backfill is assumed to require 0.2 MPa to 0.5 MPa UCS, achievable with a 2% binder addition. Design strength is typically achieved within 28 days, although early strength gain (7-day UCS) is often monitored for operational scheduling.

16.6.3.2 Deep-Mine Rockburst Considerations

At depths approaching 2 km, mining operations are subject to elevated in-situ stresses and potential rockburst conditions associated with stress redistribution and excavation geometry. Management of the rockburst hazard is primarily addressed through mine layout, sequencing, ground support and monitoring practices.

For the purposes of this PEA, the Room and Pillar backfill strength of 0.2 to 0.5 MPa UCS at 28 days, achievable with an assumed 2% binder addition, is adopted as a base-case. The potential requirement for higher-strength or modified backfill in localised high-stress conditions is recognized and will be evaluated during the PFS through site-specific geomechanical modelling, convergence and seismic monitoring, and paste backfill strength testing to confirm final design criteria.

16.6.3.3 Drift and Fill Mining

In Drift and Fill mining, the backfill functions as a load bearing structural element. It must support the overlying rock mass, act as a working platform for subsequent drift development, and maintain stability during sequential extraction. This results in higher strength requirements than Room and Pillar operations. Typical UCS ranges are:

- Early re-entry strength (7-day UCS): 0.4 MPa to 0.7 MPa.
- Standard design strength (28-day UCS): 1.5 MPa to 2.0 MPa.
- High-stress or deep level conditions: 2.0 MPa to 3.0 MPa+.

For this study, Drift and Fill backfill is assumed to require 1.5 MPa to 2.0 MPa UCS, achieved with an assumed 5% binder addition.

16.6.4 Design Criteria

The basis of design for this study has been established using a combination of available Project data, indicative operating parameters, and reasonable engineering assumptions. These criteria underpin the conceptual backfill and surface tailings designs and will be refined during the PFS as laboratory testwork, geotechnical data, and detailed mine scheduling become available. The data used for design criteria is provided in Table 16.27.

Table 16.27: Design Criteria

Parameter	Unit	Value		Basis
		Room and Pillar	Drift and Fill	
Mining				
LoM	Yrs	28		Mine Plan
Process Plant				
Feed to Tailings	%	95		Assumption
Tailings Production	LoM Mt	North – 247.3 South – 238.3	-	-
Thickener Underflow Mass Concentration	%m	60		Assumption
Material Characteristics				
Tailings Solids Density	t/m³	2.5	2.5	Assumed
Binder Density	t/m³	3.15		Assumed
Backfill Characteristics				
Binder Content	%	2	5	-
Backfill Mass Concentration	%m	75	75	-
Backfill Slurry Density	t/m³	1.83	-	Assumed
Consolidated Backfill Density	t/m³	1.98	-	Assumed
Production – Backfill				
Backfill Plant Operating Utilisation	%	72		Assumption
Stope Fill Factor	%	90	97	Assumption
Backfill Operating Rate	t/h	North – 1,050 South – 1,003	-	Calculation
Filtration				
Filter Cake Mass Concentration	%m	85		Assumed

16.6.5 Paste Backfilling Approach

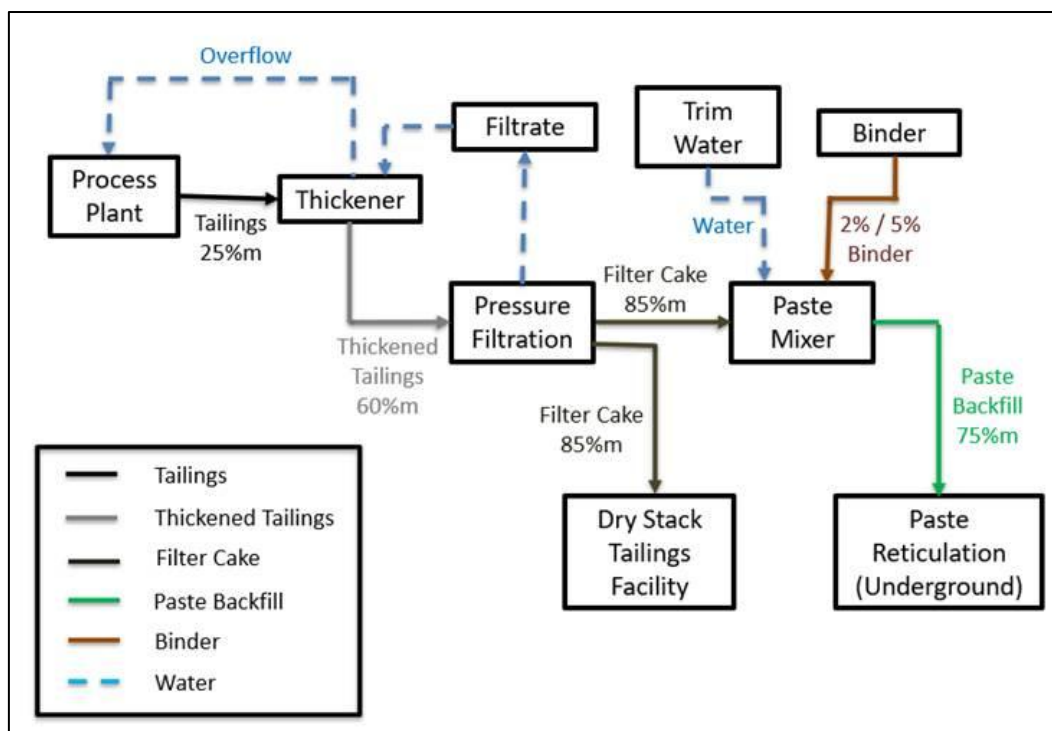
For the Project, paste backfill allows maximisation of underground tailings placement while remaining fully compatible with the requirement for dry stack tailings disposal on the surface.

16.6.5.1 Preparation and Paste Production

All tailings are thickened and filtered at surface to achieve a solids content around 85% by weight. The filtered tailings are transferred to the paste backfill plant, where water and binder are added to achieve the required rheology and strength properties Figure 16.26. Binder dosage varies according to the mining method, approximately 2% for Room and Pillar, as backfill is primarily used for confinement and void replacement; and 5% for Drift and Fill to achieve the higher strengths required for supporting sequential mining.

The paste production process includes controlled solids handling, water addition, binder dosing, homogenisation, and continuous mixing to produce a consistent, pumpable material. The paste is then delivered underground via a dedicated high-pressure pipeline reticulation network.

Figure 16.26: Tailings, Paste Production and Backfill Flowchart

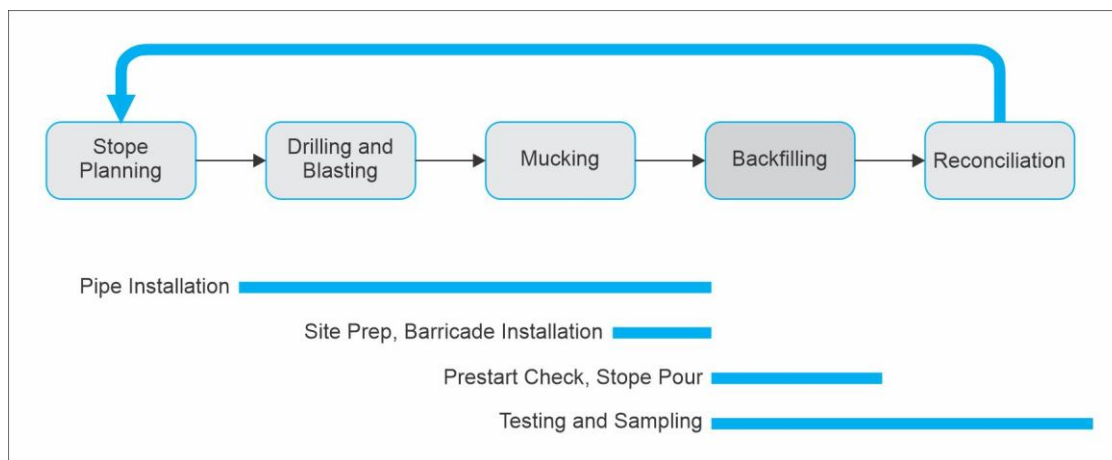


16.6.5.2 General Backfill Cycle and Application

Paste backfilling for both Room and Pillar and Drift and Fill operations is undertaken as a repeatable campaign process. Each campaign involves isolating the target excavation area, establishing paste containment and service connections, delivering paste at a managed rate to control barricade loading, and allowing the placed fill to cure until minimum performance criteria are met for subsequent mining activities. On completion of each filling campaign, the reticulation system is flushed and prepared for changeover to the next placement location.

A conceptual backfill cycle (Figure 16.27) illustrates the interaction between mining, preparation, paste placement and verification activities. The sequence reflects a controlled campaign-based backfilling approach applicable to both Room and Pillar and Drift and Fill mining methods.

Figure 16.27: Typical Stope Cycle Activities

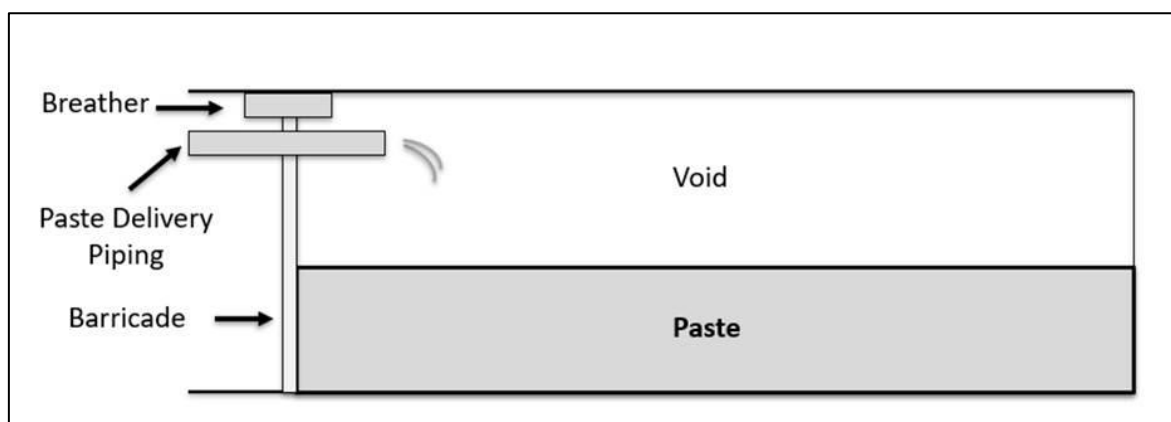


The paste fill cycle progresses along the following steps (Figure 16.27):

1. Upon completion of the mucking activities a paste fill barricade is erected for sealing at the room/stope entrance. Typically, the barricade will be constructed of sprayed concrete as a means of paste containment. Load cells may be installed to the barricade wall for monitoring hydrostatic pressure build up.
2. Paste fill reticulation piping is installed into the stope through the barricade wall. The piping connects to the wider mine reticulation system linking to the backfill preparation plant. A breather pipe is installed through the barricade to prevent the void from pressurising.
3. The backfill is prepared at the backfill plant and then delivered to the required destination through the reticulation system. Once filled the reticulation piping beyond the barricade will generally be lost.
4. The paste will be left to cure until sufficient strength requirements are confirmed via QA/QC testing.
5. Delivery lines will be flushed with water to prevent paste hardening inside the reticulation system.
6. Filling progresses to the next excavation area in the mining sequence.

A schematic cross section of a typical paste backfill placement is provided in (Figure 16.28). This illustrates the relationship between the paste delivery piping, barricade, breather arrangement and the filled excavation during a standard filling cycle.

Figure 16.28: Stope Filling Cross Section



16.6.6 Method Specific Backfilling Requirements

16.6.6.1 Room and Pillar Mining

In Room and Pillar mining, paste backfill is used primarily to maximise underground tailings placement while providing confinement to pillars and limiting deformation at depth. Backfill is not generally required to act as a load bearing structure element.

For this study, a 28-day UCS of 0.2 MPa to 0.5 MPa is assumed, achievable with approximately 2% binder addition. An effective fill factor of approximately 90% is adopted, recognising that full roof contact is not always required and that, in deep high stress environments, convergence during the period between excavation and backfill placement may reduce the effective fillable volume. The allowance accounts for convergence-related and operational losses. Refinement of the effective fill factor will be undertaken at the PFS stage based on-site specific geomechanical assessment and convergence monitoring.

Paste rise rates of 0.2 m/h to 0.5 m/h are adopted to control barricade loading and manage filling pressures. Pumping conditions are more predictable, reflecting more uniform excavation geometry, shorter distribution distances and reduced pressure variability relative to Drift and Fill operations. Flow velocities are maintained in the range of 1.5 m/s to 2.0 m/s to prevent deposition and ensure stable transport. While either centrifugal or positive displacement pumps may be technically feasible depending on confirmed paste rheology, positive displacement pumps have been adopted for this study to provide conservative pressure capability and consistent flow control.

16.6.6.2 Backfilling in Drift and Fill Mining

Drift and Fill mining imposes more stringent backfill requirements because each filled drift must act as a competent working platform for subsequent mining. Consequently, paste must achieve both tight filling to the roof and higher strengths, typically 1.5 MPa to 2.0 MPa at 28 days, with early strength targets of 0.4 MPa to 0.7 MPa at 7 days to allow timely re-entry.

A fill factor of approximately 97% is assumed to ensure full roof contact. Higher binder doses of 5% increase paste viscosity, requiring more precise flow control and generating higher pumping pressures. Paste rise rates are reduced to 0.1 m/h to 0.3 m/h to manage barricade loading and ensure controlled placement.

Due to these requirements, positive displacement pumps are preferred. Reticulation systems are subject to greater pressure variability due to narrow geometries and elevation changes, requiring enhanced pressure monitoring, protection and more frequent flushing.

16.6.6.3 Barricades

Barricades provide structural confinement to the paste backfill during placement and early curing and are a critical control element for safe backfilling operations. Barricades are designed to resist hydrostatic and paste-induced loads while maintaining integrity until sufficient backfill strength is achieved.

For this study, barricades are assumed to be constructed from shotcrete or reinforced concrete and installed at the entrance to rooms or drifts prior to filling. Each barricade incorporates a cast-in delivery pipe for paste introduction, a breather line to allow air displacement during filling, provision for pressure monitoring and ports for temperature and strength monitoring, where required.

In Room and Pillar mining, barricade demands are moderate due to lower paste stiffness, reduced filling heights and lower rise rates. In Drift and Fill mining, barricades must accommodate higher paste viscosity, tighter filling geometry and increased sensitivity to pressure build-up. As a result, barricade design and monitoring requirements are more stringent for Drift and Fill operations.

Final barricade geometry, reinforcement requirements and monitoring arrangements will be refined during further advanced studies based on confirmed paste rheology, filling heights and geotechnical conditions.

16.6.6.4 Pumping, Distribution and Control Systems

Paste backfill for both Room and Pillar and Drift and Fill mining methods is transported from the surface paste preparation plant to the underground workings through a dedicated reticulation system. The system is designed to deliver paste reliably and safely to multiple underground locations while maintaining controlled flow, predictable barricade loading and operational flexibility.

The distribution network comprises a main delivery pipeline extending from the surface plant to underground, followed by a series of distribution headers, branch lines and valve stations that direct paste to individual rooms or drifts in accordance with the mining sequence.

Positive displacement pumps are adopted as the basis of design at PEA level due to their ability to handle high-density, high viscosity paste and to deliver stable volumetric flow largely independent of discharge pressure.

Flow, pressure and density instrumentation is installed at key locations throughout the system to enable real-time monitoring and control. Following completion of each filling cycle, delivery lines are flushed with water to prevent paste hardening within the reticulation system.

Detailed hydraulic sizing, pressure analysis, pipe diameter selection and pressure management strategies are addressed separately in the Hydraulic Design Section (Section 16.6.13) of this Report.

16.6.6.5 Differentiation Summary

The differences between the backfill operation between the two mining scenarios are detailed in Table 16.28.

Table 16.28: Backfill Operation Differences between Room and Pillar versus Drift and Fill

Parameter	Room and Pillar	Drift and Fill
Backfill Purpose	Void replacement, confinement	Structural support enabling sequential mining
28-day UCS	0.2 MPa – 0.5 MPa	1.5 MPa – 2.0 MPa
7-day UCS	0.2 MPa – 0.4 MPa	0.4 MPa – 0.7 MPa
Binder Dosage	2%	5%
Fill Factor	90%	97%
Rise Rate	0.2 m/h – 0.5 m/h	0.1 m/h – 0.3 m/h
Pump Selection	Centrifugal or Positive Displacement	Positive Displacement
Backfill Role	Stability and tailings disposal	Load bearing and sequencing platform

16.6.7 Scenario 1 Design - Paste Fill in Room and Pillar Mining

This section outlines the design basis and operating principles for paste backfill within the Room and Pillar mining scenario.

16.6.7.1 Mine Design and Material Balance

Following the completion of the mine schedule, a LoM material balance was developed to quantify backfill tonnages, underground void availability and surplus tailings requiring surface disposal. The material balance was built utilising inputs derived from the mine schedule and the key assumptions are presented in Table 16.29.

Table 16.29: Input Assumptions

Assumption	Unit	Value
Amount of Feed to Tailings	%	95
Tailings Density	t/m ³	2.5
Binder Content	%	2
Backfill Mass Concentration	%m	75
Backfill Slurry Density	t/m ³	1.83
Stope Fill Factor	%	90
Annual Production Days	Days	365
Daily Production Hours	Hours	24
Backfill Plant Operating Utilisation	%	72
Annual Production Hours	hrs	6,307
Backfill Consolidated Density	t/m ³	1.98

16.6.7.2 Mass Balance and Material Balance – North Shaft Complex

Based on the total LoM void volume for the North Shaft Complex, an average backfill placement rate was derived and expressed as an equivalent steady-state production rate. This resulted in a design backfill throughput of approximately 1,050 t/h (dry solids), which has been adopted as the basis for the process mass balance and backfill system sizing.

A steady-state mass balance (Table 16.30) was developed around the design rate, covering tailings production, thickening, filtration, paste preparation and final placement. The balance reflects the full tailings stream reporting to the thickening, with a split between filtered cake directed to paste backfill and filtered cake directed to the DSTF.

Table 16.30: Mass Balance and Material Balance - North Shaft Complex

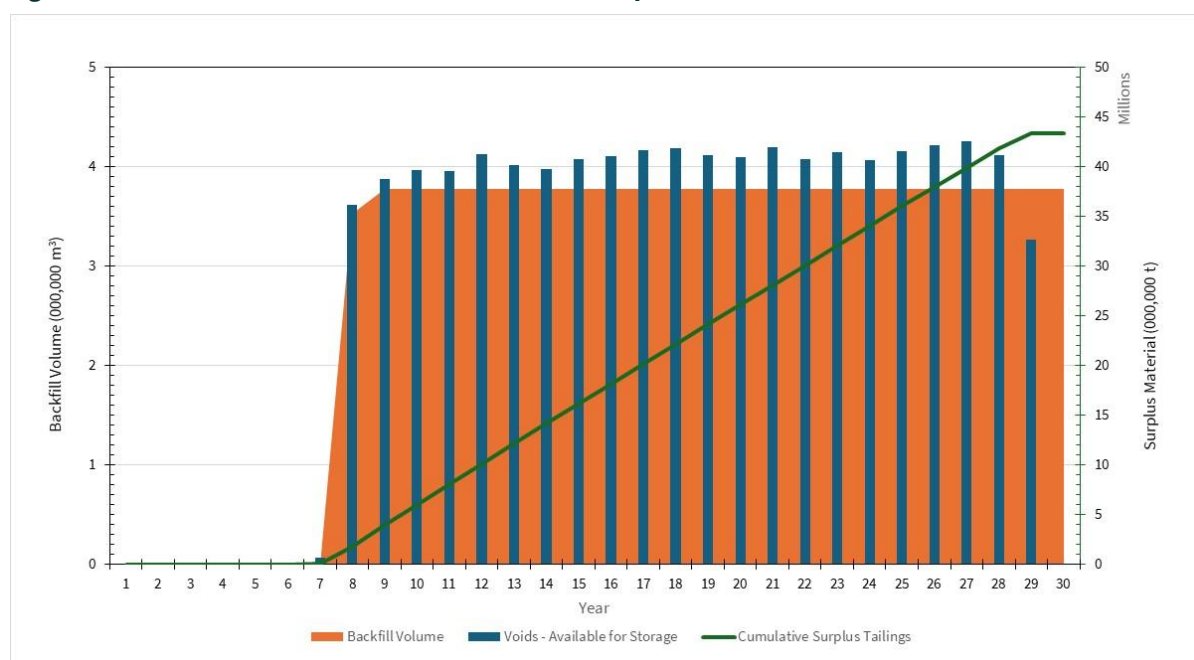
North	Solids (t/h)	Liquid (t/h)	Slurry (t/h)	Slurry (%m)	Solids (m ³ /h)	Liquid (m ³ /h)	Slurry (m ³ /h)	Slurry (%v)	Slurry SG (t/m ³)	Solids Density	Water Density
Tailings	1,633	4,898	6,531	25	648	4,898	5,546	12	1.18	2.5	1
Paste Thickener Feed	1,633	4,898	6,531	25	648	4,898	5,546	12	1.18	2.5	1
Thickener Underflow	1,633	1,089	2,721	60	648	1,089	1,736	37	1.57	2.5	1
Thickener Overflow	-	3,810	3,810	-	-	3,810	3,810	-	1	2.5	1
Thickener Underflow for Backfill	1,633	288	1,921	85	648	288	936	69	2.05	2.5	1
Filter Cake	-	800	800	-	-	800	800	-	1	2.5	1
Filtrate	1,050	185	1,235	85	417	185	602	69	2.05	2.5	1
Cake for Backfill	21	0	21	100	7	0	7	-	3.15	3.15	1
Binder	-	165	165	-	-	165	165	-	-	-	-
Trim Water	1,050	350	1,400	75	417	350	767	54	1.83	2.5	1
Backfill	1,050	230	1,281	82	420	230	651	65	1.98	2.5	1
Consolidated Backfill	583	103	686	85	231	103	334	69	2.05	2.5	1
Cake for DSTF	1,633	4,898	6,531	25	648	4,898	5,546	12	1.18	2.5	1

16.6.7.3 LoM Backfill and Void Balance - North Shaft Complex

While the process plant and backfill system are designed to operate at a broadly steady design throughput, the rate of underground void creation varies over the LoM in response to the mining schedule. To assess the interaction between void availability and backfill placement, a LoM material balance was developed reconciling annual void creation with the selected backfill production rate.

As shown in Figure 16.29 backfill demand increases rapidly during the early years of mining as production areas are established. By approximately Year 9, annual void creation reaches a relatively stable level reflecting sustained Room and Pillar production. From this point onward, void availability and backfill placement remain broadly consistent for the majority of the remaining mine life.

Figure 16.29: LoM Material Balance - North Shaft Complex



Source: Micon, 2026.

The selected paste backfill production rate is sufficient to meet the majority of void filling requirements during the steady-state production phase. Annual backfill placement closely tracks available void volumes, with limited accumulation of unfilled void space. This indicates that the backfill system is well matched to the mining sequence and does not rely on significant deferral of backfilling to the latter stages of the LoM.

Throughout the LoM, total tailings production exceeds the quantity required for paste backfill. Surplus filtered tailings are therefore directed to the surface dry stack storage, with the cumulative surplus tailings increasing progressively over time, as shown by the cumulative surplus curve in Figure 16.29.

16.6.8 Mass Balance and Material Balance – South Shaft Complex

For the South Shaft Complex, the LoM void volume was converted to an equivalent steady-state paste backfill placement rate of approximately 1,003 t/h (dry solids) and forms the basis of the South mine mass balance.

The steady-state mass balance (Table 16.31) is broadly consistent with that developed for the North Shaft Complex, with the same thickening, filtration and paste production approach applied.

Table 16.31: Mass Balance and Material Balance - South Shaft Complex

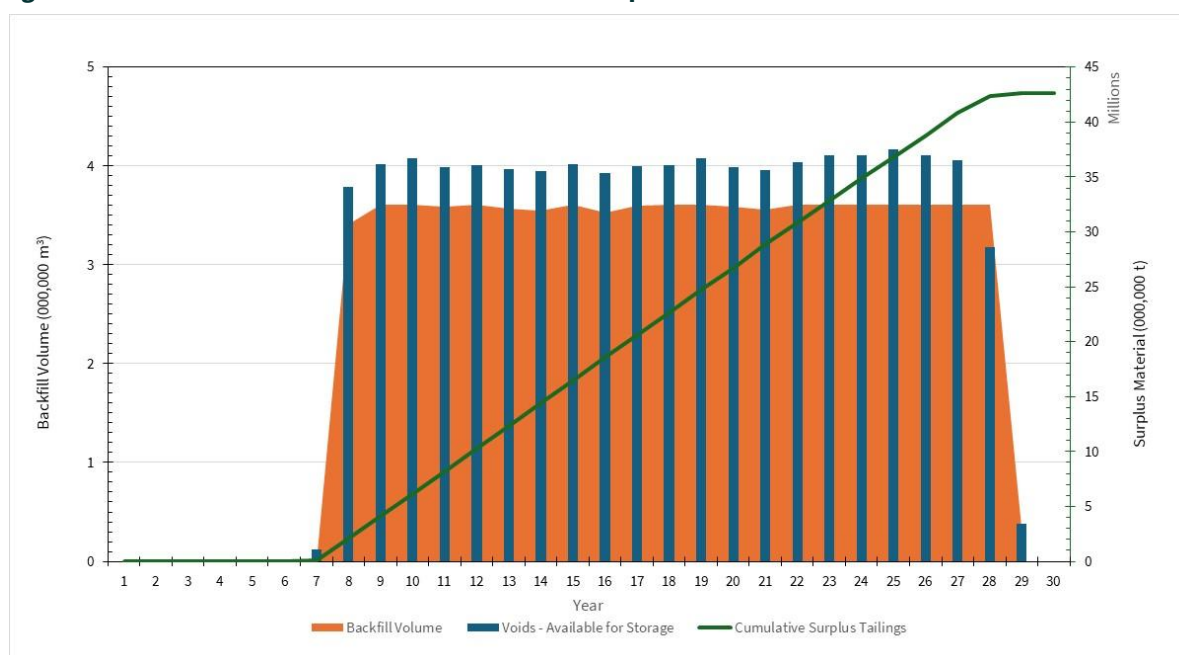
South	Solids (t/h)	Liquid (t/h)	Slurry (t/h)	Slurry (%m)	Solids (m ³ /h)	Liquid (m ³ /h)	Slurry (m ³ /h)	Slurry (%v)	Slurry SG (t/m ³)	Solids Density	Water Density
Tailings	1,624	4,871	6,495	25	644	4,871	5,515	12	1.18	2.5	1.0
Paste Thickener Feed	1,624	4,871	6,495	25	644	4,871	5,515	12	1.18	2.5	1.0
Thickener Underflow	1,624	1,082	2,706	60	644	1,082	1,727	37	1.57	2.5	1.0
Thickener Overflow	-	3,789	3,789	-	-	3,789	3,789	-	1.00	2.5	1.0
Thickener Underflow for Backfill	1,624	287	1,910	85	644	287	931	69	2.05	2.5	1.0
Filter Cake	-	796	796	-	-	796	796	-	1.00	2.5	1.0
Filtrate	1,003	177	1,180	85	398	177	575	69	2.05	2.5	1.0
Cake for Backfill	20	-	20	100	6	0	6	-	3.15	3.15	1.0
Binder	-	157	157	-	-	157	157	-	-	-	-
Trim Water	1,003	334	1,337	75	398	334	732	54	1.83	2.5	1.0
Backfill	1,003	220	1,223	82	398	220	618	64	1.98	2.5	1.0
Consolidated Backfill	621	110	731	85	246	110	356	69	2.05	2.5	1.0
Cake for DSTF	1,624	4,871	6,495	25	644	4,871	5,515	12	1.18	2.5	1.0

16.6.9 LoM Backfill and Void Balance – South Shaft Complex

The LoM material balance for the South Shaft Complex reflects a void creation profile that ramps up rapidly during the early years of production, reaches a sustained plateau through the mid-life of the mine, and then declines more sharply in the latter years as mining activity winds down.

As shown in Figure 16.30 backfill demand increases rapidly during the ramp-up phase as production panels are established. From approximately Year 9 onwards, void creation stabilises and remains broadly consistent through the main production period. During this phase, the selected paste backfill production rate is well matched to void availability.

Figure 16.30: LoM Material Balance - South Shaft Complex



Source: Micon, 2026.

In the latter years of the LoM, the rate of new void creation decreases. During this period the backfill system continues to operate at a broadly consistent throughput, resulting in progressive completion of backfilling within the South Shaft Complex (Figure 16.30). No significant long-term inventory of unfilled voids is carried into mine closure under the current schedule.

Surplus filtered tailings are directed to the surface dry stack storage throughout the LoM, with the cumulative surplus increasing during the main production years and flattening towards the end of the mine life.

Overall, the South LoM material balance demonstrates that a design backfill production rate of approximately 1,003 t/h (dry solids) is technically appropriate and consistent with the planned mining schedule.

16.6.9.1 Design Implications

This approach provides several practical and operational advantages:

- Avoids the need to oversize backfill infrastructure to accommodate short-term fluctuations in void creation, as the backfill system is sized to meet steady-state void demand that dominates the majority of the LoM.

- Enables stable and continuous operation of the paste plant and the underground distribution system, with backfill production governed by plant operability rather than short-term variations in annual void availability.
- Reflects common industry practice for paste backfill systems, where production rates are governed by plant operability rather than instantaneous void availability.

The selected steady-state backfill production rate represents the average LoM requirement, provision for short-term make-up capacity to recover downtime or meet peak placement demand will be evaluated during the PFS through availability analysis and surge capacity assessment.

The LoM material balance therefore demonstrates that the selected backfill production rate of approximately 1,050 t/h (dry solids) is technically appropriate and well aligned with the mining sequence. Surplus filtered tailings are managed via dry stack storage as an integral component of the overall tailings management strategy.

16.6.10 Backfill Ramp-Up Methodology

Paste backfill placement over the LoM is controlled primarily by the rate at which underground void space becomes available, rather than by tailings production capacity. Void creation increases during mine development and ramp-up, stabilises during steady-state production, and declines towards mine closure. Consequently, tailings utilisation for backfill varies through time and cannot be assumed to be uniform across the LoM.

To reflect this behaviour, a staged ramp-up methodology has been adopted. The approach links backfill placement directly to the mine schedule, applies realistic placement efficiencies and reconciles backfill demand against annual tailings production. This ensures that backfill plant sizing is based on achievable steady-state operating conditions, while capturing cumulative utilisation over the LoM.

Mining method specific placement characteristics are accounted for using practical fill factors. A fill factor of approximately 90% is applied for Room and Pillar mining, while a higher fill factor of approximately 97% is assumed for Drift and Fill mining.

Under the Room and Pillar scenario, this approach results in an average LoM tailings utilisation of approximately 50% with the balance of filtered tailings directed to the DSTF. Utilisation for the Drift and Fill scenario is expected to be marginally higher due to improved placement efficiency.

16.6.10.1 Step by Step Ramp Up

The staged ramp-up of paste backfill placement has been implemented using the following steps:

Step 1 – Annual void creation Annual mined void volumes are taken directly from the mine schedule and represent the maximum theoretical space available for backfilling in each year.

Step 2 – Application of practical fill factors Scheduled void volumes are adjusted using mining method specific fill factors of approximately 90% for Room and Pillar mining and 97% for Drift and Fill mining to reflect realistic placement constraints.

Step 3 – Conversion of void volume to backfill demand Effective void columns are converted to annual paste backfill demand using assumed consolidated backfill density and a solids mass concentration of 75%.

Step 4 – Comparison with tailings production Annual backfill demand is compared with annual tailings production to determine the underground placement versus dry-stack disposal.

Step 5 – LoM aggregation Annual backfill placement is aggregated over the LoM to define the overall utilisation profile and cumulative backfill placement.

This staged approach avoids oversizing backfill infrastructure and supports stable plant operation throughout the LoM. The resulting LoM utilisation profile for the Room and Pillar scenario is summarized in Table 16.32.

Table 16.32: Backfill Utilisation

Mine Phase	Backfill Utilisation (%)	Explanation
Years 1 - 7	0	No production voids available.
Years 7 - 8	20 - 50	Initial production voids become available, commissioning/ramp-up.
Years 9 - 28	60 - 65	Steady state production.
Years 28	40 - 60	Void creation declines.
Final Years	0 - 20	Limited remaining void availability.

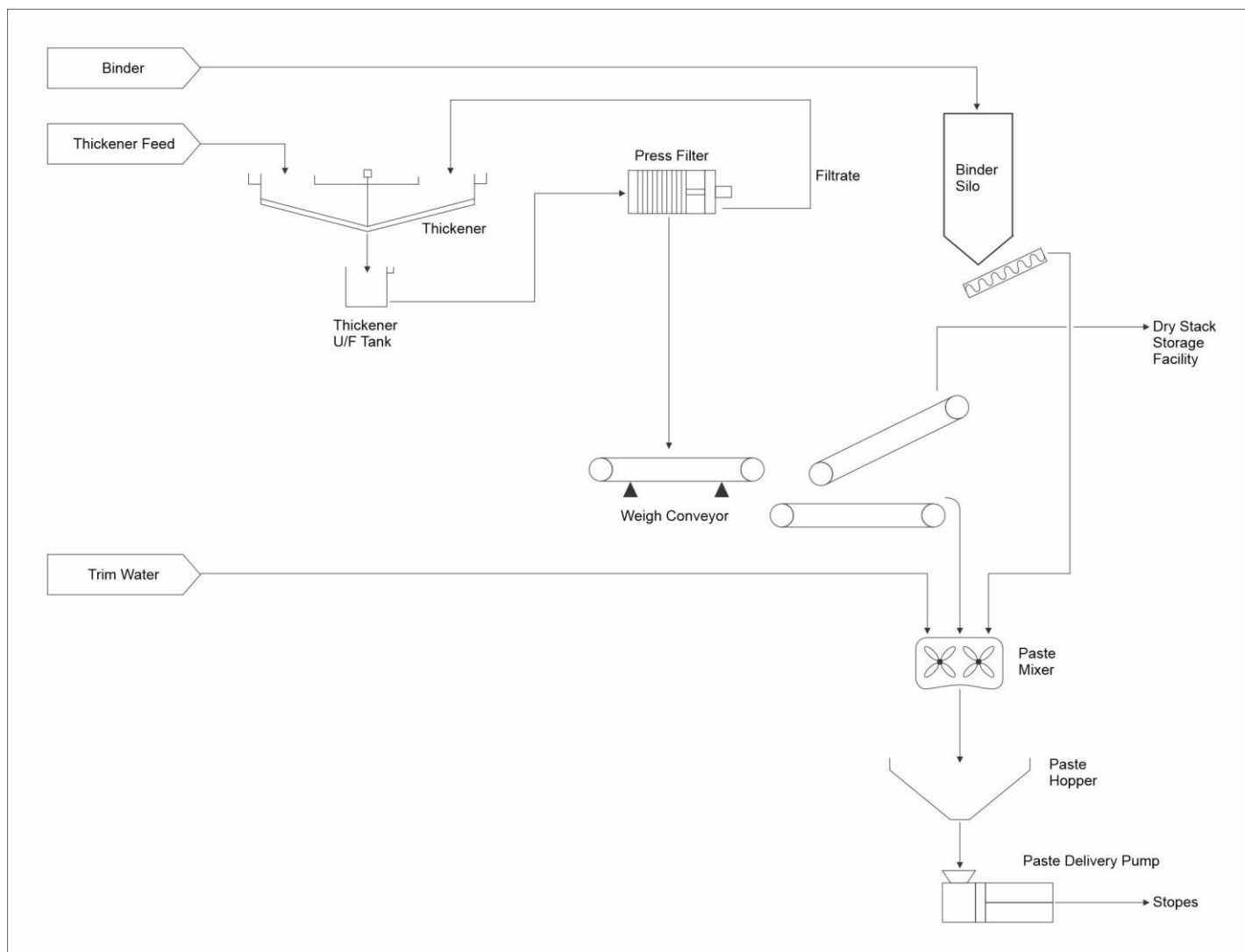
16.6.11 Backfill Plant – Room and Pillar Mining

For this conceptual study, the North and South Shaft Complex areas are treated as independent operational areas, each supported by a dedicated paste backfill and filtration plant operating in parallel. Costs and capacities are presented on a per plant basis.

16.6.11.1 Process Description

Paste backfill for the Project will be produced using the full tailings stream from the processing plant. The backfill system is designed to generate a high density, pumpable paste suitable for pipeline transport and underground placement within both the Room and Pillar and Drift and Fill mining scenarios. The overall process configuration, from tailings thickening and pressure filtration through paste mixing and delivery to underground stopes is shown schematically in the process flow schematic in Figure 16.31.

Figure 16.31: Process Flow Schematic



Source: Micon, 2026.

The overall process consists of tailings thickening, filtration, binder dosing, paste mixing and finally pumping through a dedicated reticulation network to the underground workings.

Tailings Thickening

All tailings slurry from the process plant will be directed to a high-rate or high-density thickener, which provides the first stage of dewatering. The thickener increases the solids concentration of the slurry to a target underflow density of approximately 60% solids by mass. Thickener underflow is pumped or gravity-fed to the agitated Thickener Underflow Tank from which it will be pumped to the filtration plant.

Tailings Filtration

The thickened underflow is dewatered using pressure filtration, producing a filter cake with an assumed solids content of 85% by mass. Filtrate is recovered and returned to the process water or thickener feed system.

The filtration circuit is common to both paste backfill preparation and surface dry stack tailings deposition.

Filtered Tailings Handling

Filtered tailings cake is conveyed to a weigh conveyor, where the required fraction is directed to the paste mixing circuit. Surplus filtered tailings are conveyed to the surface DSTF.

Binder Storage and Dosing

Binder (cement) is stored in dedicated silos and metered into the mixing circuit using screw feeders. Binder addition is assumed at 2% by mass of dry solids for the Room and Pillar mining scenario, and 5% by mass of dry solids for the Drift and Fill mining scenario.

Paste Mixing

Filtered tailings, binder and controlled quantities of water are blended in a continuous twin-shaft mixer to produce a homogeneous paste with controlled rheology, density and strength development characteristics. Trim water addition is used to achieve the target paste consistency required for pumping and underground placement.

Pumping and Reticulation

The paste is discharged to a positive displacement pump for transfer to the underground reticulation network. Positive displacement pumps are selected to provide pressure capacity and stable flow control across a range of paste viscosities.

Paste is delivered to underground rooms or drifts via a dedicated steel or High-Density Polyethylene (HDPE) pipeline system. Following completion of each cycle, delivery lines are flushed with water to prevent paste hardening and blockage within the pipework.

16.6.12 Major Equipment

The major equipment for the conceptual paste backfill plant has been defined based on the selected flowsheet and design criteria. Equipment sizing is indicative and will be refined during the PFS following tailings testwork. The major equipment envisaged for the Nowa Sól backfill plants are as below:

- Tailings thickener (high rate or high density), including rake mechanism and drive.
- Pressure filters.
- Filter feed pumps (duty / standby)
- Filter cake handling system (conveyors/screw conveyors).
- Cement Binder System, consisting of:
 - Cement storage hopper.
 - Cement storage silos.
 - Cement dosing system.
- Paste mixer.
- Positive displacement paste pumps.

Further equipment required for the backfill system includes:

- Paste delivery pipeline (surface).
- Paste delivery pipeline (underground).
- Thickener underflow tank.
- Filtrate return system.
- Flushing water system.
- Instrumentation and control system.

16.6.12.1 Thickener

A single thickener will be installed in each of the North and South areas to provide the initial stage of the tailings dewatering prior to pressure filtration. Based on the mass balance, the thickener sizing has been developed on the basis of a steady-state tailings throughput for each mining area. Indicative thickener sizing and capital costs are summarized in Table 16.33.

Table 16.33: Thickener Summary

Parameter	North	South
Design Throughput	1,633 t/h @25% <i>m</i>	1,624 t/h @25% <i>m</i>
Number of Units	1	1
Estimated Diameter	65	65
Underflow Density	60% <i>m</i>	60% <i>m</i>
Estimated Installed Cost	US\$7 million	US\$7 million

16.6.12.2 Pressure Filters

Pressure filtration provides the final stage of tailings dewatering prior to paste production and dry stack disposal. The circuit has been sized using an indicative duty filtration rate of approximately 1.0 t (dry)/m².h, giving a required duty area of approximately 1,633 m² (North) and 1,624 m² (South).

Each plant is assumed to comprise of ten pressure filters (nine duty, one standby) at around 200 m² per unit, providing 2,000 m² installed and 1,800 m² duty operating area. This provides an operating and maintenance margin in the absence of site-specific filtration testwork. Indicative filtration capacity and capital costs are summarized in Table 16.34.

Table 16.34: Pressure Filter Summary

Parameter	North	South
Design Throughput	1,633 t/h	1,624 t/h
Design Solids Concentration	60% _m	60% _m
Assumed Filtration Rate (Duty)	1.0 t/m ² .h	1.0 t/m ² .h
Calculated Duty Area	1,633 m ²	1,624 m ²
Filter Area per Unit	200 m ²	200 m ²
Number of Units	10 (9 operating, 1 standby)	10 (9 operating, 1 standby)
Installed Filter Area	2,000 m ²	2,000 m ²
Duty Operating Area (9 Filters)	1,800 m ²	1,800 m ²
Assumed Cake Solids Concentration	85% _m	85% _m
Estimated Unit Installed Cost	US\$1.7 million	US\$1.7 million
Total Cost	US\$17 million	US\$17 million

16.6.12.3 Filter Feed Pumps

Filter feed pumps are provided to transfer thickened tailings from the thickener underflow pumping system to the pressure filtration plant. Centrifugal pumps are assumed for this purpose, feeding the thickener underflow to the pressure filters Table 16.35.

Table 16.35: Filter Feed Pump Summary

Parameter	North	South
Design Throughput	1,633 t/h	1,624 t/h
Design Solids Concentration	60% _m	60% _m
Total Flow Rate	1,736 m ³ /h	1,727 m ³ /h
Number of Units	3 (2 operating, 1 standby)	3 (2 operating, 1 standby)
Pump Type	Heavy -duty centrifugal slurry pump	
Cost per Unit	US\$0.25 million	US\$0.25 million
Total Cost	US\$0.75 million	US\$0.75 million

16.6.12.4 Filter Cake Conveyors

Filtered tailings cake discharged from the pressure filtration plant is conveyed to a central distribution point where the flow is measured and split between the paste backfill plant and the DSTF. The conveying system is designed to handle the full filtered cake production rate.

The system is assumed to comprise a weigh conveyor equipped with an integrated belt scale, and two downstream belt conveyors directing cake either to the paste backfill plant or to the DSTF. Indicative conveyor sizing and capital costs are shown in Table 16.36.

Table 16.36: Filter Cake Conveyors

Parameter	Function	No. Units (per Plant)	Estimated Sizing	Installed Cost (US\$)
Primary Filter Discharge Weigh Conveyor	Collects filter cake from pressure filter and measures total cake flow for backfill	1	Belt width 1,200 mm to 1,400 mm	0.4
Paste Feed Conveyor	Transfers weighed cake to paste mixer	1	Belt width 1,000 mm	0.2
Dry Stack Conveyor	Transfers surplus cake to DSTF	1	Belt width 1,200 mm	0.5

16.6.12.5 Paste Mixer

The paste mixer is used to combine filtered tailings, binder and trim water to produce a homogeneous pumpable paste (Table 16.37). The mixer has been sized to match the steady-state backfill production rates for the North and South Shaft Complex areas, based on the mass balance. A twin-shaft paddle mixer configuration is assumed with two mixers in parallel per mining area.

Table 16.37: Paste Mixer

Parameter	North	South
Design Throughput (dry)	1,050 t/h	1,003 t/h
Paste Production (wet)	1,400 t/h	1,337 t/h
Design Solids Concentration	75% _m	75% _m
Number of Units	2	2
Mixer Type	Twin-shaft paddle mixer	
Cost per Unit	US\$0.7 million	US\$0.7 million
Total Cost	US\$1.4 million	US\$1.4 million

16.6.12.6 Binder System

The binder system provides controlled storage and dosing of cement for paste backfill production. Binder is delivered in bulk and stored in vertical steel silos located adjacent to the paste backfill plant. Storage capacity is sized to provide approximately 2 to 5 days of continuous operation. The binder system capacity and capital costs are summarized in Table 16.38.

Table 16.38: Binder System

Parameter	North	South
Max Binder Demand	21 t/h	20 t/h
Number of Silos	1	1
Storage Capacity	2 – 5 days	2 – 5 days
Binder Discharge System	Variable speed screw feeder / rotary valve	
Binder Conveyance	Enclosed screw conveyor to mixer	
Total Cost	US\$0.9 million	US\$0.9 million

16.6.12.7 Positive Displacement Pumps

Positive displacement pumps are used to deliver paste backfill from the surface plant to the underground reticulation system.

The pumps are sized to meet the steady-state paste production rates for each mining area. Variable speed drives are provided to allow adjustment of flow rate in response to operational requirements and to maintain controlled paste rise rates during filling. Pump sizing and capital costs are shown in Table 16.39.

Table 16.39: Positive Displacement Pumps

Parameter	North	South
Paste Flow Rate	767 m ³ /h	732 m ³ /h
Number of Pumps	3 (2 duty, 1 standby)	3 (2 duty, 1 standby)
Type of Pump	Positive Displacement Paste Pump	
Typical Discharge Pressure per Pump	10 MPa – 15 MPa	10 MPa – 15 MPa
Combined System Pressure Capability	>25 MPa	>25 MPa
Drive Type	Electric motor, VSD controlled	
Cost per Unit	US\$1.4 million	US\$1.4 million
Total Cost	US\$4.2 million	US\$4.2 million

16.6.12.8 Surface and Underground Paste Reticulation Piping

A dedicated paste reticulation system conveys paste from the surface plant to underground mining areas. The system comprises a surface delivery pipeline, a main underground pressure column and branching distribution lines supplying individual rooms or drifts.

Steel piping is assumed for high-pressure sections, with pipe diameters selected to maintain transport velocities above deposition thresholds while managing friction losses. Final pipe routing, diameters, pressure ratings and wear allowances will be developed during detailed hydraulic analysis at PFS level.

16.6.13 Hydraulic Design

The paste reticulation system has been developed at a conceptual level using assumed paste properties, indicative production rates and an anticipated mining depth of approximately 2,000 m. The design intent is to provide reliable delivery of paste to underground excavations while maintaining controlled flow conditions, predictable barricade loading and manageable operating pressures. Detailed hydraulic modelling and design verification (including paste rheology testing, final pipeline routing, stress analysis and pressure protection) will be completed during further advanced studies once laboratory data and detailed mine geometry are available.

16.6.13.1 General Hydraulic Considerations

Paste backfill is transported as a high-density, non-segregating material at an assumed solids concentration of approximately 75% by mass. Pipeline sizes are selected to maintain transport velocities in the range of approximately 1.5 m/s to 2.0 m/s, balancing deposition risk against friction losses and wear. Total system pressure is governed primarily by static head due to depth, with additional contributions from friction losses, local losses at fittings/valves and the operational margin.

16.6.13.2 Hydraulic Design – Room and Pillar Mining

For the Room and Pillar mining scenario, paste backfill is produced using filtered tailings with an assumed binder addition of approximately 2% (by mass of dry solids), providing the required strength development while maintaining workable rheology for pipeline transport. Steady state paste delivery rates are assumed to be approximately 770 m³/h for the North mining area and 730 m³/h for the South mining area, corresponding to 1,050 t/h and 1,003 t/h dry solids, respectively.

The mine extends to depths of approximately 2,000 m below surface, giving a static head of approximately 20 MPa (water equivalent). Allowing for friction losses, local losses and operational margin, the total peak required discharge pressure at the surface paste pumps is expected to be approximately 22 MPa, subject to confirmation during more advanced hydraulic modelling and paste rheology testing. The 22 MPa value is treated as a peak design basis for equipment selection, with average operating pressures expected to be lower depending on the operating point, pipeline condition and the final confirmed paste rheology.

The reticulation system is conceptually configured as a surface delivery pipeline from the paste plant to the shaft collar, followed by a main underground vertical pressure column conveying paste to depth, with stepped distribution into local headers and branch lines. The main surface and vertical delivery pipelines are assumed to be DN400 steel maintaining steady-state paste velocities within the target operating range. Underground distribution is assumed to reduce progressively from DN350 primary headers to DN250 secondary headers, with DN200 branch lines to individual rooms. Branch lines are expected to be short, frequently flushed and may be sacrificial beyond the barricade depending on the installation approach.

A conceptual representation of the paste backfill reticulation system, illustrating the hierarchy of surface, vertical and underground distribution pipelines, is shown in Figure 16.32. The schematic is illustrative only and is intended to demonstrate the conceptual arrangement and functional separation of the major pipeline elements rather than define detailed routing or final design.

Figure 16.32: Conceptual Paste Backfill Reticulation System



Source: Micon 2025.

Note: Illustrative schematic, not to scale.

The final pipeline diameter, material and wall thicknesses will be confirmed during the PFS. At this stage steel pipe is assumed for all high-pressure sections (surface delivery line and main underground pressure column) given the operating pressures and the abrasive nature of paste backfill.

Paste delivery is based on positive displacement pumps arranged in a duty/standby configuration for each mining area to provide redundancy and high availability. Paste rise rates for Room and Pillar mining are assumed to be approximately 0.2 m/h to 0.5 m/h to maintain controlled filling and barricade stability. Delivery lines are flushed following each filling cycle to prevent paste hardening. Pressure, flow and density instrumentation is provided at key locations to support safe operation and real time monitoring. Key hydraulic design outputs for the Room and Pillar backfill system are summarized in Table 16.40.

Table 16.40: Key Hydraulic Design Outputs for the Room and Pillar Backfill

Parameter	Unit	Value	Comment
Main Surface Pipeline	-	DN400 Steel	High pressure section sized to keep full flow velocity within target range.
Main Vertical Pressure Column	-	DN400 Steel	Static head dominated.
Primary Underground Headers	-	DN350 Steel	Flow split control.
Secondary Headers	-	DN250 Steel	Progressive reduction.
Branch Lines to Rooms	-	DN200	Short, frequently flushed.
Operating Velocity	m/s	1.6 – 1.9	Within target range.
Estimated Discharge Pressure	MPa	22	Static + friction + margin.
Paste Rise Rate	m/h	0.2 – 0.5	Barricade loading control.

16.6.13.3 Pressure Management and System Flexibility

While the conceptual design assumes surface-based pumping to full depth, it is recognized that operating pressures over 20 MPa place significant demands on pipeline integrity and pump selection. During the PFS, the following pressure management options will be evaluated if required:

- Booster pump stations installed at intermediate underground levels to reduce maximum surface discharge pressure.
- Pressure break or choke stations, subject to confirmation that paste rheology and segregation risks can be managed.
- Underground paste preparation or pumping facilities, which would substantially reduce static head, but increase underground infrastructure and operational complexity.

At the PEA level, these options are not incorporated into the base case, but are identified as credible design refinements should hydraulic modelling or operational risk assessments indicate that pressure reduction is desirable.

16.6.14 Cost Estimation

16.6.14.1 Basis of Estimate

A desktop level cost estimation has been prepared using analogous backfill projects, indicative equipment pricing and factored allowances. The estimate is prepared to an accuracy of $\pm 50\%$, consistent with PEA requirements. Costs are presented per plant (North or South). All costs are quoted in United States dollars (US\$).

Operating costs have been developed on a per plant basis and include both fixed and variable components associated with operation of the filtration, paste backfill and reticulation systems.

The following costs are excluded:

- Supply of power reticulation and fibre optic communication to the backfill plant.
- Clean and process water supply costs.
- Any environmental or regulatory permitting costs.
- Decommissioning and rehabilitation costs.
- Underground ventilation, fire and dust suppression systems.

16.6.15 Capital Costs – Room and Pillar Scenario

The capital costs comprise mechanical equipment costs, direct and indirect costs associated with plant construction as summarized from Table 16.41 to Table 16.44. Costs are presented on a per plant basis (i.e. one plant serving either the North or the South Shaft Complex).

16.6.15.1 Direct Capital Costs

Table 16.41: Mechanical Equipment Costs Summary

Item	No.	Unit Cost (US\$ 000)	Total (US\$ 000)
Binder System	1	900	900
Thickener	1	7,000	7,000
Flocculant System	1	500	500
Thickener Underflow Tank and Agitator	1	600	600
Thickener Overflow Pumps	3	120	360
Filter Feed Pumps	3	250	750
Tailings Pressure Filter	10	1,700	17,000
Filtrate Pumps	2	60	120
Paste Mixer	2	700	1,400
Raw Water Tank	1	90	90
Raw Water Pump – Trim and Gland Water	2	30	60
Emergency Flush Pump	2	60	120
Transfer Conveyor	1	200	200
Weigh Conveyor	1	400	400
Tailings Conveyor	1	500	500
Positive Displacement Pump	3	1,400	4,200
Overhead Crane	1	95	95
Total Plant Mechanical Cost (TPMC)			34,295

16.6.15.2 Additional Direct Costs

The additional direct costs are shown in Table 16.42.

Table 16.42: Additional Direct Costs Summary

Item	Factor of TPMC (%) or No.	Unit Cost (US\$ 000)	Total (US\$ 000)
Reticulation Piping – Overground	1,000	1.5	1,500
Reticulation Piping - Underground	4,000	1.8	7,200
Civils and Earth Works	12%	4,115	4,115
Structural Steel and Platework	10%	3,430	3,430
Plant Equipment, Piping and Control	8%	2,744	2,744
Electricals, Instruments and Control	25%	8,574	8,574
Total			27,563
Total Direct Costs (TDC) (TPMC + Direct Costs)			61,858

16.6.15.3 Indirect Capital Costs

The indirect capital costs and total capital costs per plant are shown in Table 16.43 and Table 16.44, respectively.

Table 16.43: Indirect Capital Costs Summary

Item	Factor (%)	Total (US\$ 000)
EPCM / Engineering	6% Factor of TDC	3,711
Contractor Preliminary and Generals	7% Factor of TDC	4,330
Commissioning and First Fill	3% Factor of TPMC	1,029
Owners Costs	3% Factor of TDC	1,856
Freight and Packaging	2% Factor of TPMC	586
Total Indirect Costs		11,612

Table 16.44: Total Capital Costs per Plant

Cost Type	Total (US\$ 000)
TPMC	34,300
Direct Costs	27,600
Indirect Costs	11,600
Total Capital Costs	73,500
Contingency 25%	18,400
Total Capital Costs with Contingency	91,900

16.6.16 Operating Costs – Room and Pillar Scenario

The operating cost estimate has been developed for the paste backfill and filtration plant on a per plant basis, with costs applicable to either the North or South Shaft Complex areas. The estimate includes both fixed and variable operating costs associated with the operation of a single backfill plant and its associated reticulation system. The costs of which are summarized in Table 16.45.

Table 16.45: Operating Cost Summary

Item	Quantity	Unit Cost (US\$)	Total (US\$ 000)
Variable			
Power	49,431,321 kWh	0.12 US\$/kWh	5,932
Trim Water / Process	1,038,872 m ³	0.10 US\$/m ³ of water	104
Gland Water	-	-	250
Consumable Wear Parts	-	-	3,000
Flocculant	-	-	150
Binder	132,456 t	150 US\$/t	19,868
Reticulation Piping – Placement Lines Consumable	6,622,812	0.6 US\$/t	3,974
Reticulation Piping – Distribution Lines – move/repair	6,622,812	0.3 US\$/t	1,987
Barricades	200	2,500	500
Total Variable OPEX			35,765
Fixed			
Labour	90 people	72,000	6,480
Maintenance	20% of TPMC		6,859
Paste and Slurry Wear Allowance	5% of TPMC		1,715
Reticulation Piping Expansion – Shaft piping maintenance	20% Shaft Piping CAPEX		1,440
Reticulation Piping Expansion – Main Line Expansion	2,000	900 US\$/m	1,800
Reticulation Piping Expansion – Distribution Piping	6,622,812	0.3 US\$/t	1,987
Total Fixed OPEX			20,281
Total Annual OPEX			56,046
Total Monthly Average OPEX			4,670
Total Plant OPEX per Backfill Placed (US\$/m³)			US\$11.59
Total Plant OPEX per Dry Tonne of Filtered Tailings (US\$/t)¹			US\$5.44

Note: ¹ Does not include dry stack tailings costs – just filtration and backfill plant costs.

16.6.17 Binder Sensitivity

Paste binder consumption is a key operating cost driver in the backfill system. The base case assumes a nominal 2% binder addition for routine Room and Pillar backfill. Higher strength backfill maybe required in localized high-stress areas. The extent and distribution of any higher-strength backfill will be confirmed at the PFS stage through geomechanical modelling and paste testing and is expected to be applied selectively rather than mine-wide.

To illustrate the potential cost impact, a sensitivity analysis has been undertaken assuming uniform increases in binder dosing across total backfill production. The resulting effect on annual backfill operating costs is summarized in Table 16.46.

Table 16.46: Binder Sensitivity

Binder (%)	Binder Annual Cost (US\$000)	Total Annual Operating Cost (US\$000)
2%	19,868	56,046
3%	29,802	65,980
4%	39,737	75,914
5%	49,671	85,848

The figures assume uniform application of binder percentage to all backfilled tonnes; selective application would proportionally reduce cost impact.

16.6.18 Surface Disposal

All tailings not utilized for underground paste backfill will be disposed of as dry stack tailings. Lumina has confirmed that no conventional wet TSF will be considered. Accordingly, surplus tailings from the processing plant will be directed to a surface DSTF.

Based on the backfill utilisation assumptions developed for this study, the DSTF is expected to store approximately a half of LoM tailings for the Room and Pillar scenario.

16.6.18.1 Tailings Processing and Preparation

All process tailings are filtered as part of the paste backfill production circuit. Filtered tailings not required for backfilling are diverted to the surface DSTF and discharged as a consolidated, trafficable material suitable for mechanical placement and compaction.

This integrated approach allows a single filtration circuit to support both underground backfilling and surface tailings disposal, eliminating the need for a wet TSF.

16.6.18.2 Dry Stack Placement Methodology

The DSTF will be constructed progressively over the LoM using conventional earthworks equipment such as dozers, loaders and haul trucks. Filtered tailings will be transported by conveyor or truck to active placement areas and deposited in controlled lifts, typically 0.3 m to 0.6 m thick.

Each lift will be spread and compacted, with routine monitoring of moisture content and density to confirm placement performance. This controlled, layered construction methodology promotes uniform consolidation, good trafficability for placement equipment, and long-term geotechnical stability of the stack.

16.6.18.3 Stack Development and Height Progression

The DSTF is conceptually designed to reach a final height of approximately 35 m over the LoM. Stack development will follow a phased sequence, maintaining stable outer slopes typically of the order of 3:1, subject to detailed geotechnical design.

Internal drainage layers or finger drains may be incorporated where required to manage pore pressures. Progressive rehabilitation will be undertaken on completed areas to manage erosion and surface runoff.

16.6.18.4 Drainage and Water Management

Effective drainage is critical to dry stack performance. The DSTF will include perimeter diversion channels and stormwater controls to prevent runoff entering active stacking areas. Foundation level drainage may also be incorporated to limit pore pressure development.

16.6.18.5 PEA Level Design Basis

For PEA purposes, the DSTFs have been sized using surplus LoM tailings volumes of approximately 45.2 Mm³ (North) and 44.4 Mm³ (South). Independent facilities would require footprints of approximately 1.55 km² and 1.52 km², respectively, based on an assumed operating stack height of 35 m and allowance for slope formation and infrastructure.

Final stack geometry, drainage layout and stability analysis will need to be refined during further phases of the study following site-specific geotechnical, hydrological and material characterisation testwork.

16.6.19 Split Dry Stack Facility Configuration

16.6.19.1 Design Basis and Rationale

Following review of the initial dry stack tailings concept, Lumina requested that the surface DSTFs for both the North and South Shaft Complex areas be configured as two discrete dry stack cells per area, developed sequentially rather than as a single full-footprint facility. The objective is to reduce initial footprint capital expenditure by deferring a portion of earthworks, drainage and foundation preparation while maintaining total LoM storage capacity.

16.6.19.2 Conceptual Configuration and Sequencing

Under the split-cell concept, each mining area is served by two adjacent dry stack cells (Cell 1 and Cell 2), designed to identical long-term stability criteria and final elevation. Cell 1 would support initial and mid-life operations, with Cell 2 constructed and commissioned as Cell 1 approaches capacity.

Construction of Cell 2 would be staged to align with mine development and supporting infrastructure, including haul road extensions, drainage tie-ins and progressive foundation preparation. The layout assumes that ongoing Cell 1 operations are not disrupted during Cell 2 construction.

16.6.19.3 Indicative Sizing

The split-cell configuration does not change total LoM tailings volume or assumed final stack height of approximately 35 m. Indicative sizing of the DSTF is summarized in Table 16.47.

Table 16.47: DSTF Conceptual Sizing

Mining Area	Configuration	Filtered Tailings Volume (Mm ³)	Approx. Footprint (km ²)	Development Timing
North	Single Stack (LoM)	45.2	1.55	Full LoM capacity
North	Cell 1	22.6	0.78	Initial operations
North	Cell 2	22.6	0.77	Deferred
South	Single Stack (LoM)	44.4	1.52	Full LoM capacity
South	Cell 1	22.2	0.76	Initial operations
South	Cell 2	22.2	0.76	Deferred

Only Cell 1 in each mining area is required for initial operations. Cell 2 construction can be deferred until later in the mine life.

16.6.19.4 Capital Cost Implications

While Cell 1 represents approximately 50% of the ultimate footprint, dry stack capital costs do not scale linearly with area. Enabling infrastructure such as access roads, stormwater diversion, access roads and initial foundation works must largely be installed upfront.

Allowing for these non-linear cost components, approximately 35% to 45% of total dry stack capital expenditure is expected to be deferred under the split-cell strategy. Total LoM capital expenditure remains broadly unchanged; however, the initial capital requirements is materially reduced. Indicative CAPEX comparisons are summarized in Table 16.48.

Table 16.48: DSTF Costings

Area	Configuration	Initial CAPEX (US\$ million)	Deferred CAPEX (US\$ million)	Total LoM (US\$ million)
North	Single Stack	27.7	0	27.7
North	Split Cells	17	10.7	27.7
South	Single Stack	26.1	0	26.1
South	Split Cells	16	10.1	26.1

Upfront costs include perimeter diversion/primary drainage, initial access roads, starter pad and earthworks to establish Cell 1 operations.

In addition, the dry-stack tailings compound is assumed to incorporate a basal low-permeability insulating layer to limit seepage and reduce the potential for heavy-metal leaching to underlying groundwater. For the purposes of this PEA, the insulating layer is assumed to comprise a compacted clay/silt layer, placed across the active footprint of the facility. An order-of-magnitude unit cost of approximately US\$12/m² has been adopted, consistent with industry benchmarks for locally sourced, compacted low-permeability materials.

On this basis, the estimated liner cost is approximately US\$18.6 million for a single stack facility, equivalent to around US\$9.3 million per stack under the split-cell configuration. Including the insulating layer allowance, the total dry-stack compound capital cost is estimated at approximately US\$46.6 million for the North facility and US\$41.5 million for the South facility over the LoM.

Final liner configuration, material specifications and any requirement for composite liner systems or seepage collection will be confirmed during the PFS and EIA, informed by tailings geochemical characterisation, hydrogeological conditions and regulatory requirements.

16.6.20 Backfill Conclusion

Backfilling forms a critical component of the Nowa Sól mining strategy, supporting underground stability, enabling extraction sequencing and reducing the volume of tailings requiring surface storage. A fundamental Project constraint is Lumina's requirement that all tailings reporting to surface must be managed as dry stack, with no conventional wet TSF permitted. This requirement drives a filtration-led tailings management approach and strongly influences both the selected backfill method and the associated infrastructure requirements at PEA level.

A screening assessment was completed across credible backfill options. Rock-based fills were not considered suitable as a primary solution due to their limited capacity to return process tailings underground. Hydraulic fill was assessed and excluded due to its reliance on wet tailings management for fine fractions and additional technical limitations under deep, high-stress conditions. On this basis, paste backfill has been selected as the preferred backfill method for both Room and Pillar and Drift and Fill mining scenarios, as it is compatible with filtered tailings production, enables high underground tailings placement, and supports low-permeability fill performance consistent with groundwater and geochemical risk reduction objectives.

Conceptual strength criteria were established based on mining method requirements. For Room and Pillar mining, a 28-day UCS of 0.2 MPa to 0.5 MPa has been adopted, achievable with an assumed 2% binder dosage, with backfill primarily serving confinement and deformation control functions. For Drift and Fill mining, higher structural performance requirements have been assumed, with a 28-day UCS of 1.5 MPa to 2.0 MPa and an assumed 5% binder dosage to support sequential extraction and platform stability. These strength assumptions are appropriate for PEA level and will require confirmation through laboratory testwork and geotechnical design verification during the PFS.

LoM material balances were developed for the North and South Shaft Complex areas using annual void availability derived from the mine schedule. The mine has been treated as two operational areas supported by independent paste backfill and filtration plants sized on a per plant basis. For the Room and Pillar scenario, the selected steady-state paste backfill production rates of approximately 1,050 t/h (dry solids) for the North and 1,003 t/h (dry solids) for the South are well aligned with the void creation profiles. The analysis indicates that backfill placement can track void creation closely through the steady-state production period without reliance on significant deferral of backfilling into late mine life. Surplus filtered tailings that are not utilized for underground placement are directed to surface DSTFs.

A conceptual surface-to-underground reticulation system has been developed for a mine depth of approximately 2,000 m, giving a static head of approximately 20 MPa (water equivalent). Allowing for friction losses, local losses and operating margin, a peak design discharge pressure of approximately 22 MPa has been adopted for preliminary equipment selection. Positive displacement pumping has been adopted as the conservative PEA level basis of design to provide the required pressure capability and stable flow control. Pressure management options (including intermediate booster stations or alternative pressure control strategies) have been identified for evaluation during the PFS should detailed hydraulic modelling indicate that pressure reduction is beneficial for operability, integrity management or cost optimization.

16.6.20.1 Cost Summary (PEA level, per Plant)

A PEA level cost estimate ($\pm 50\%$) has been prepared for a single paste backfill and filtration plant serving either the North or South Shaft Complex areas. The estimate includes thickening, pressure filtration, paste mixing, positive displacement pumping, and associated surface and underground reticulation infrastructure, consistent with a flowsheet in which all tailings are filtered and then split between paste preparation and dry stack deposition.

On this basis, the total capital cost (including 25% contingency) is estimated at approximately US\$91.9 million per plant, with an annual operating cost of approximately US\$56.0 million per plant. This equates to an average operating cost of approximately US\$11.59/m³ of backfill placed and US\$5.44 per dry tonne of filtered tailings processed. As the Project comprises two operational areas (North and South), total project costs would scale to two parallel plants, subject to confirmation of shared infrastructure opportunities and economies of scale during the PFS. On a two-plant basis, the indicative combined capital cost would be approximately US\$184 million, with combined annual operating costs of approximately US\$112 million, prior to any shared infrastructure allowances or optimization.

The dominant cost drivers for both capital and operating expenditure are:

- Pressure filtration capacity and installed filter area.
- Positive displacement pumping under high static head conditions.
- Binder consumption to achieve required backfill strength.
- Electrical power demand associated with filtration, paste mixing and paste transport.

These elements represent the primary opportunities for cost optimization and design refinement during the PFS through detailed tailings testwork, hydraulic modelling and vendor engagement.

Overall, the PEA level assessment demonstrates that paste backfill is technically credible and well aligned with Project constraints, providing an integrated filtration-based system that supports both underground backfilling and surface dry stack tailings management while avoiding the need for a wet TSF. The conceptual design basis and cost estimate provide a suitable platform for progression to the PFS, where assumptions, equipment selections and operating parameters will be confirmed and refined.

17 RECOVERY METHODS

17.1 INTRODUCTION

The proposed PEA processing facility has been designed to process approximately 32,000 t/d of mineralized material over most of the Project's mine life. The conceptual process flowsheet and design is based on metallurgical testwork results described in Section 13 of this Report. The proposed process flowsheet comprises primary crushing, Semi-Autogenous Grinding (SAG) and ball milling, pebble crushing and flotation to produce a single copper concentrate containing silver. Tailings handling consists of dewatering and either fed to the tailings backfill plant or to a DSTF.

The PEA assumes that two 32,000 t/d processing facilities (North and South) will be operating once full mine production is achieved (64,000 t/d total). At this preliminary stage of Project development, Micon has used two identical facilities. The descriptions and data presented in the sections below only refer to one processing plant.

17.2 PROCESS DESIGN BASIS

The process design basis is summarized in Table 17.1. The Project design of each processing facility is based on a design throughput of 32,000 t/d and operating for 24 hours per day for 365 days per year. The operating utilization factors shown have been used to develop the mass balance and size the unit operations included in the processing facility.

17.3 PROCESS DESIGN CRITERIA

The detailed Process Design Criteria (PDC) developed for the PEA are based on the metallurgical testwork results presented in Section 13 of this Report, the process design basis and typical industry design factors. A summary of the PDC is included in Table 17.2.

Table 17.1: Process Design Basis - Operating Criteria

Description	Unit	Value	Comments
Mineralized Rock Characteristics			
LoM Average Grade	Cu%	1.19	Economic model - Rev09
LoM Average Grade	Ag ppm	36.9	Economic model - Rev09
LoM Average Grade	Zn%	-	Not included in model
LoM Average Grade	Pb%	0.15	Economic model - Rev04
Process Design Grade	Cu%	1.47	Use annual LoM 80%ile grade
Process Design Grade	Ag ppm	49.6	Use annual LoM 80%ile grade
Process Design Grade	Zn%	0.07	Based on average MRE grade
Process Design Grade	Pb%	0.29	Use annual LoM 80%ile grade
Moisture in Ore	%	4.0	Typical value
Ore Types (Preliminary Description)			
Carbonate	% of feed	65.0	Range 50% - 70%
Carbonate - Gangue Mineralization	Mainly carbonate minerals, minor clay and sulphate minerals		
Carbonate - Copper Mineralization	Mainly chalcocite with minor digenite, bornite and covellite		
Shale	% of feed	7.0	Range 5% - 20%
Shale - Gangue Mineralization	Mainly clay and carbonate minerals, minor organics and quartz		
Shale - Copper Mineralization	Mainly bornite and chalcocite with minor chalcopyrite		
Sandstone	% of feed	28.0	Range 20% - 40%
Sandstone - Gangue Mineralization	Mainly quartz, minor clay and carbonate minerals		
Sandstone - Copper Mineralization	Mainly bornite, chalcopyrite, chalcocite and digenite		
Hardness - Comminution Work Indices			

Description	Unit	Value	Comments
Bond Ball Mill Work Index (BBWi)	kWh/t	15.60	XPS Testwork result
Crusher Work Index (CWI)	kWh/t	10.00	Estimate, typical for Cu, Zn, Pb mineralization
Bond Rod Mill Work Index (BRWI)	kWh/t	16.00	Estimate, typical for Cu, Zn, Pb mineralization
Abrasion Index	g	0.200	Estimate, typical for Cu, Zn, Pb mineralization
RoM Ore Densities and Stockpile Characteristics			
Average Specific Gravity (SG)	-	2.70	Place Holder, to be verified
Ore Bulk Density (for volume calculations)	t/m ³	1.50	Factored from SG
Ore Bulk Density (for weight calculations)	t/m ³	1.69	Factored from SG
Angle of Repose	degree	37	Typical value
Drawdown Angle	degree	57 - 70	Typical value
PROCESS OPERATING CRITERIA			
Operating Days per Year	d/a	365	-
Selected Ore Processing Rate (Nominal)	t/d	32,000	Metric Tonnes
Annual Processing Rate (Nominal)	kt/a	11,680	Calculated
Number of Concentrators at Property	-	2	Assumes 1 North and 1 South plant
Primary Crushing			
Operating Days per Week	d	7.00	-
Total Crusher Operating Utilization	%	65.0	24 h per day and 65% utilization
Average Operating Hours per Operating Day	h	15.6	-
Average Throughput Rate	t/h	2,051	-
Grinding and Flotation Circuit			
Operating Days per Week	d	7.00	-
Total Operating Utilization	%	92.0%	Includes thickening of tails and concentrate
Average Operating Hours per Operating Day	h	22.1	-
Average Throughput Rate	t/h	1,449	-
Tailings Filtration			
Operating Days per Week	d	7.00	-
Total Operating Utilization	%	90.0%	Average for vacuum filter
Average Operating Hours per Operating Day	h	21.6	-
Average Throughput Rate (dry t)	t/h	1,411	-
Concentrate Filtration			
Operating Days per Week	d	7.00	-
Total Operating Utilization	%	85.0	Average for pressure filter
Average Operating Hours per Operating Day	h	20.4	-
Average Throughput Rate (dry t)	t/h	74.7	-
PROCESS UNIT OPERATION DESIGN OPERATING FACTORS			
Primary Crushing	%	30	Throughput
Conveyors and Feeders	%	30	Throughput
Grinding Equipment	%	10	Motor Sizing
Flotation Equipment	%	15	Volume Calculations
Process Pumps	%	20	Flows and Motor Sizing
Concentrate Dewatering	%	30	Cater for High Grade Zone
Tailings Dewatering	%	15	Equipment Sizing

Table 17.2: Process Design Criteria

Parameter	Units	Design	Comments
Mined Material Characteristics			
Total LoM tonnage	kt	509.2	Economic model - Rev09 (North + South)
Average LoM Mined Grade	Cu %	1.19	Economic model - Rev09 (North + South)
Average LoM Mined Grade	g/t Ag	36.8	Economic model - Rev09 (North + South)
Total Mined Copper Inventory	Cu kt	6,045	Economic model - Rev09 (North + South)
Total Mined Silver Inventory	Ag Moz	604	Economic model - Rev09 (North + South)
Primary Crushing - Fresh Mineralization			
Nominal Throughput	t/d	32,000	Design basis
Nominal Throughput	t/h	2,051	Design basis
Crusher Stockpile Total Live Capacity	t	96,000	-
Final Product Size - Passing (P ₈₀)	mm	150	-
Crushed Material Live Capacity	t	64,000	-
Grinding Circuit			
Final product size - passing F ₈₀)	µm	60	Preliminary XPS testwork
Feed size - Passing (P ₈₀)	mm	150	-
Average Throughput Rate	t/d	32,000	Design basis
Average Throughput Rate	t/h	1,449	Design basis
Grinding Circuit - SAG Mill			
Feed Size - Passing (F ₈₀)	mm	150	-
Product Size - Passing (P ₈₀)	µm	850	-
Proportion Circulating Load	%	25	-
Ball Charge -Duty	%vol	12	-
Ball Charge Design (maximum)	%vol	20	-
Estimated Power Draw	kW	15,000	-
SAG Mill Pebble Crusher			
Feed to Pebble Crusher	t/d	8,000	Calculated
Feed to Pebble Crusher	t/h	362	-
Feed Size - Passing (F ₈₀)	mm	75	-
Product Size - Passing (P ₈₀)	mm	14	-
Grinding Circuit - Ball Mill			
Feed size - passing (F ₈₀)	µm	850	-
Product Size - Passing (P ₈₀)	µm	60	-
Proportion Circulating Load	%	300	-
Estimated Power Draw	kW	20,500	-
Flotation Circuit			
Feed Solids Density	%w/w	32.8	Mass balance
Solids Feed Rate to Flotation Circuit	t/h	1,449	Design basis
Feed Rate	m ³ /h	3,512	Calculated
Design Grade	Cu%	26.0	Design basis
Flotation Concentrate Solids SG	-	3.70	Design basis
Number of Cleaner Stages	-	3	Preliminary testwork suggests 2
Rougher Flotation Retention Time	mins	23	Testwork plus scale-up factor of 2.5
Scavenger Flotation Retention Time	mins	30	Testwork plus scale-up factor of 2.5
Cleaner 1 Flotation Retention Time	mins	30	Testwork plus scale-up factor of 2.5
Cleaner 2 Flotation Retention Time	mins	20	Testwork plus scale-up factor of 2.5
Cleaner 3 Flotation Retention Time	mins	18	Testwork plus scale-up factor of 2.5
Grinding Circuit - Cu Concentrate Regrind Mill			
Average Feed Rate	t/d	8,640	Mass balance

Parameter	Units	Design	Comments
Average Feed Rate	t/h	391	Mass balance
Feed Size - Passing (P ₈₀)	µm	60	Preliminary testwork
Product Size - Passing (P ₈₀)	µm	10	Preliminary testwork
Metal Recoveries			
Average Copper Recovery	%	88.75	Wrocław Tech recommendation
Average Silver Recovery	%	86.64	Wrocław Tech recommendation
Average Lead Recovery	%	79.3	Preliminary XPS Testwork results
Copper Concentrate Quality			
Copper Concentrate Grade (Average)	Cu%	26.0	Economic model REV01 (prelim.)
Copper Concentrate Silver Grade (Average)	g/t Ag	854	Calculated
Copper Concentrate Lead Grade (Average)	Pb%	4.53	Calculated
Concentrate Solids SG	-	3.70	Estimated based on 36% Cu minerals in conc.
Copper Concentrate Production			
Copper Concentrate Production (Nominal)	dry t/d	1,761	Calculated
Copper Concentrate Production (Nominal)	kt/a	643	Calculated
Average Concentration Ratio (CR Nominal)	-	19.9	Calculated
Average Concentrate Weight Recovery	wt%	5.03	Calculated
Concentrate Bulk Density (Unconsolidated)	t/m ³	2.00	For volume calculations
Concentrate Bulk Density (Consolidated)	t/m ³	3.00	For weight calculations
Concentrate Dewatering			
Average Feed Rate	t/d	1,609	Design basis
Average Feed Rate	t/h	73	-
Thickener u/f Density	%w/w	60	Place holder, needs testwork
Filter Product Moisture	wt. %	8.5	Place holder, needs testwork
Tailings Dewatering			
Average Feed Rate	t/d	30,239	Design basis
Average Feed Rate	t/h	1,376	Design basis
Average Yield	% wt rec.	94.07	Calculation
Thickener u/f Density	%w/w	60.0	Place holder, needs testwork
Filter Product Moisture	%w/w	15	Place holder, needs testwork

17.4 PROCESS DESCRIPTION

The conceptual process design is based on conventional processing technologies and a proven flowsheet that has been implemented at many other comparable projects, in particular local KGHM mines, which have been successfully operational for 60 years. A simplified process flowsheet is presented in Figure 17.1.

Fresh material will be crushed to approximately -150 mm and fed to a stockpile from which it will be conveyed at a controlled rate to the grinding circuit.

The grinding circuit design consists of a single SAG and ball mill with a pebble crusher. The discharge slurry from both mills will be pumped to a cyclone cluster with the overflow feeding the flotation circuit and cyclone underflow feeding the ball mill. The target cyclone overflow P₈₀ size is 60 µm.

The diagram illustrates the process flow of a copper concentrator plant, starting from ROM Ore and ending with Copper Concentrate Stockpile and Tailings Dry Stack.

Primary Crushing and Stockpiling: ROM Ore is fed into a Primary Crusher, which discharges onto a Primary Crusher Discharge Conveyor. This conveyor feeds into a Stockpile. An Apron Feeder also feeds into the Stockpile. Underground Crushed Ore is also fed into the Stockpile. The Stockpile is fed by a Stockpile Feed Conveyor.

Secondary Crushing and Grinding: The Stockpile Feed Conveyor feeds into a Belt Feeder, which feeds into a Pebble Crusher. The Pebble Crusher discharges into a Pebble Crusher Surge Bin. The Surge Bin feeds into a SAG Mill. The SAG Mill discharges into a Discharge Screen. The Discharge Screen feeds into a Ball Mill. The Ball Mill discharges into a Cyclone. The Cyclone overflow flows to Flotation. The Cyclone underflow feeds into a Pumpbox.

Flotation and Conditioning: The Pumpbox feeds into a Copper Conditioning Tank No. 1. The output of this tank feeds into a series of Copper Rougher Flotation Cells. The output of the Rougher cells feeds into a series of Copper Scavenger Flotation Cells. The output of the Scavenger cells feeds into a series of Copper Reagent Cyclones. The output of the Cyclones feeds into a Copper Reagent Mill. The output of the Reagent Mill feeds into a Copper Reagent Pumpbox. The output of the Pumpbox feeds into a Copper Cleaner Conditioning Tank.

Cleaning and Thickening: The Copper Cleaner Conditioning Tank feeds into a series of Stage 1 Copper Cleaner Cells. The output of Stage 1 feeds into Stage 2 Copper Cleaner Cells. The output of Stage 2 feeds into Stage 3 Copper Cleaner Cells. The output of Stage 3 feeds into a Copper Concentrate Thickener. The output of the Thickener feeds into a Copper Filter Feed Tank. The output of the Filter Feed Tank feeds into a Copper Concentrate Pressure Filter. The output of the Pressure Filter feeds into a Copper Concentrate Stockpile.

Tailings Processing: The output of the Copper Concentrate Thickener feeds into a Tailings Thickener. The output of the Thickener feeds into a Tailings Filter Feed Tank. The output of the Filter Feed Tank feeds into a series of Tailings Filters. The output of the Filters feeds into a Tailings Filter Filtrate Tank. The output of the Filtrate Tank feeds into a Binder Storage tank. The output of the Binder Storage tank feeds into Tanks with agitators. The output of the Tanks with agitators feeds into Positive Displacement Pumps. The output of the Pumps feeds into Underground. The output of the Tailings Filters feeds into a Dry Tailings Primary Stacking System. The output of the Stacking System feeds into a Tailings Dry Stack.

Overflow and Discharge: The output of the Copper Concentrate Thickener feeds into an Overflow to Process Water. The output of the Tailings Thickener feeds into an Overflow to Process Water. The output of the Tailings Filter Filtrate Tank feeds into an Overflow to Process Water. The output of the Dry Tailings Primary Stacking System feeds into an Overflow to Process Water. The output of the Tailings Dry Stack feeds into an Overflow to Process Water.

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The conceptual design of the copper flotation circuit comprises rougher and scavenger flotation stages with regrinding of the combined rougher and scavenger concentrate, and three stages of cleaning. The preliminary testwork suggests that only two stages will be sufficient to produce a 26% copper concentrate but an additional stage has been included to provide an element of conservatism.

The final flotation concentrate will be dewatered using a thickener and pressure filter, then stored before bulk loading and transportation to market.

Tailings from the flotation circuit will be thickened, filtered and either transported to the DSTF or fed to the tailings backfill plant where it will be mixed with cement and water and then pumped underground.

17.4.1 Crushing

The PEA assumes surface crushing using a single conventional crushing unit. Alternative primary crushing options, including crushing underground and alternative technologies will be studied during the next phase of project development.

The primary gyratory crusher circuit is designed for 32,000 Mt/d) and a nominal 2,051 t/h with a 65% utilization.

The gyratory crusher discharge will feed an apron feeder under the crusher which will transfer the crushed ore onto a stacker conveyor that discharges onto the crushed ore stockpile. The total live capacity of the crushed ore stockpile will be 64,000 t. A dust collection system will be utilized to minimize dust emissions.

17.4.2 Grinding Circuit

Mineralized material will be withdrawn from the stockpile using three variable speed apron feeders located in a reclaim tunnel, onto the SAG mill feed conveyor. Crushed pebbles from the SAG mill circuit will also be recycled onto the SAG mill feed conveyor. The SAG mill feed conveyor system will include a magnet, metal detector, and weigh scale.

Process water, along with the ore, will be added to the SAG mill feed to achieve the desired pulp density. The preliminary size of the selected SAG mill is 10.4 m diameter by 5.5 m long (34' diameter by 18' long) with a 12,000 kW (16,000 HP) motor.

The +13 mm oversize material that is screened out by the SAG mill discharge screen will be conveyed to a pebble crusher, for crushing, before being returned to SAG mill feed via the SAG mill feed conveyor.

The SAG mill screen undersize will discharge into a common mill sump, which also receives the ball mill product. The mill sump material will be pumped to a cluster of cyclones with the cyclone underflow reporting to a ball mill and the overflow gravitating to the flotation circuit feed conditioners.

The preliminary ball mill selection for this circuit comprises a 9.2 m diameter by 12.2 m long (30 ft diameter by 40 ft long) equipped with a 22,000 kW (29,500 HP) motor. The target 80% passing product size (P_{80}) from the cyclone overflow is 60 μm .

Space has been allocated within the grinding circuit area for a flash flotation unit which could be fed by a portion of cyclone feed material. The reason for this consideration is to capture coarse copper minerals easily liberated from the friable mineralization. This option will be confirmed during the next Project development phase.

17.4.3 Flotation Circuit

The cyclone overflow from the grinding circuit will feed the flotation rougher conditioners, where flotation reagents will be added. The mixed slurry will flow by gravity to the copper rougher flotation bank at a nominal density of around 33% solids by weight.

The PEA design assumes conventional forced air tank cells as copper rougher flotation cells, although alternative technologies should be investigated during the next phase of Project development. The Rougher Tailings will report to the scavenger circuit while the concentrate collected from the roughers shall be combined with the scavenger concentrate and feed the copper regrind circuit.

The copper regrind circuit will consist of a hydro-cyclone cluster and a stirred vertical mill operating in open circuit. Slurry from the surge tank will be pumped to the cyclone to densify the feed and target an 80% passing size of 10 µm in the overflow, that will feed the copper cleaner flotation circuit. The cyclone underflow will be pumped through the bottom of the vertical mill and discharge from the top and routed to the cleaner flotation circuit.

The copper cleaner circuit consist of three sequential stages of cleaner flotation. The flotation concentrates flow from the first stage downstream until it reaches the third stage cleaner, the concentrate from which will be pumped to the copper concentrate thickener. The tailings from the cleaner cells flow counter-currently to the concentrate movement. The tailings from the first stage of cleaning will be directed to the tailings thickener.

17.4.4 Flotation Concentrate Dewatering and Handling

The concentrate handling circuit consists of a thickener, pressure filter and filter cake handling equipment.

The final concentrate from the flotation circuit will be fed to a dedicated thickener, where flocculant will be dosed to facilitate the settling of solids in the slurry and to reach an underflow density of approximately 60% solids by weight. The thickener overflows will be recycled to the process water system to be re-used within the process plant. The thickener underflow will be pumped to an agitated filter feed tank which will be able to hold 12 hours equivalent of slurry.

The thickened copper concentrate held within the filter feed tank will be pumped to a dedicated concentrate filter press. The filters will be fed according to the required cycle time and will produce a filter cake containing approximately 8.5% moisture by weight. The filter cake will discharge into a concentrate stockpile located below the filter press.

To prepare the concentrate for shipment, a front-end loader will recover the filter cake material from the desired indoor stockpile and load a hopper and horizontal conveyor system. This system will deliver the filtered concentrate to a lined shipping container or trailer that will be used to transport the material off-site. Dust collection systems shall also be installed to manage dust levels within the concentrate handling area.

17.4.5 Tailings Dewatering and Handling

Tailings from the flotation circuits will report to a tailings thickener, where flocculant will be added to enable settling of the solids. The thickener overflow will feed the process water system for re-use in the process plant. The underflow will reach a target density of around 60% solids by weight and will be pumped to an agitated filter feed tank. The filter feed tank will have a residence time of 30 minutes to 90 minutes and will feed a horizontal plate and frame filter press. The filter press will produce a filtered tailings cake containing approximately 15% moisture, this cake will discharge onto a reversible conveyor which can direct the filtered material to either a stockpile from which it

will be loaded onto trucks for long-term dry stack surface tailings storage or feed the paste plant to be used as backfill for the underground mine.

Any water runoff from the DSTF area will be collected and pumped back to the processing plant to be used as process water.

17.4.6 Site Water Balance

Considerable quantitative data would be required to produce a comprehensive water balance for the Nowa Sól mine and the host system of natural aquifers and nearby surface water. This work will be conducted as part of the PFS study and subsequent to drilling hydrogeological boreholes on the Property.

A conceptual site water balance has been prepared using assumed values for hydrogeological parameters and preliminary estimates for mine process water usage (Table 17.3).

Table 17.3: Preliminary Site Water Balance

Category	Item	Unit	Quantity
Water Supply	Dewatering of the Mine	m ³ /a	1,530,428
	Water Released from Tailings Thickening	m ³ /a	16,291,822
	Water Released from Tailings Filtration	m ³ /a	13,191,529
Total Water Supplied			31,031,779
Water Usage	Crushing and Grinding	m ³ /a	333,714
	Flotation	m ³ /a	33,371,429
	Reagent Solutions, Gland Water	m ³ /a	667,429
	Other Uses	m ³ /a	166,857
	Water to Make Paste	m ³ /a	2,183,320
Total Water Used		m³/a	36,722,749
Water Balance	Water Deficit	m³/a	5,708,969
Equivalent Requirement		m³/s	0.18

Source: CQA, Preliminary Hydrogeological Assessment, January 2026.

17.4.7 Reagents and Consumables, Handling and Storage

Reagent preparation and storage systems will be located within dedicated containment areas. Storage tanks will be equipped with level indicators, instrumentation, and alarms to reduce the risk of spills during normal operation. Appropriate ventilation, fire and safety protection, safety shower stations and Safety Data Sheet stations will be located throughout the facility.

Reagents consumed within the process plant will be prepared on-site and distributed via various reagent handling and makeup systems. The preliminary list of reagents includes:

- Potassium Amyl Xanthate (PAX), a flotation collector.
- Methyl Isobutyl Carbinol (MIBC), a flotation frother.
- Lime, a pH modifier.
- Flocculants, used for dewatering of concentrate and tailings.

The reagents will be mixed, stored, and then pumped at a controlled rates to the flotation circuit and dewatering areas. Dosages will be controlled by flow meters and control valves. The capacity of the storage tanks are sized to handle one day of production.

The major consumables for the comminution circuit will be the grinding media (steel balls) and liners.

17.4.8 Plant Services

Compressed air will supply the necessary air for the operation of filter presses, actuation of instruments and maintenance tools. Low pressure blowers will be used to supply air to the flotation cells.

Process water will be recycled from the collection of thickener overflows and de-watering filtrates. Process water make-up will primarily be sourced from mine water and collected tailings storage area runoff.

Fresh water will be used to feed the potable water system, gland water service, fire-water system and reagent preparation.

18 PROJECT INFRASTRUCTURE

18.1 INTRODUCTION

The Project is made up of two separate mining areas within the Nowa Sól II license area, separated approximately 8 km apart, as the crow flies (Figure 18.1 and Figure 18.2). The North and South Shaft Complexes are located within generally wooded, heath and farmed areas as described in Section 20.3.1 and are effectively considered greenfield developments requiring the provision of services and infrastructure to support the mining and beneficiation activities.

However, being relatively close to urban areas, public utilities and other infrastructure, it is anticipated that all requirements will substantially be available from either the existing or planned local provisions.

The general arrangement of each of the shaft/ process sites are shown in Figure 18.1 and Figure 18.2.

18.2 SITE PREPARATION

The sites are located on land that is currently forested and heath areas will need clearing and preparation for the construction of large-scale mining and processing infrastructure.

Site preparation will include the controlled clearing of trees and vegetation, removal and stockpiling of organic topsoil and bulk earthworks to form stable platforms for heavy industrial infrastructure such as shafts, processing plants, workshops, power infrastructure and logistic areas. Some of the plant and machinery will require special foundations and it is expected that further localized excavation and ground preparation will be required once ground geotechnical surveys are carried out.

The long-term stability of the shafts and the close surrounding infrastructure (e.g. main fans) will be additionally protected by a dedicated shaft protection zone. Within this zone, mining operations will be limited to laterals and essential functional facilities only, such as pit bottom infrastructure, while ore extractions will be excluded. This approach is consistent with Polish mining regulations and good international practice, ensuring the integrity of shafts, headframes, surface buildings, and critical services throughout the LoM.

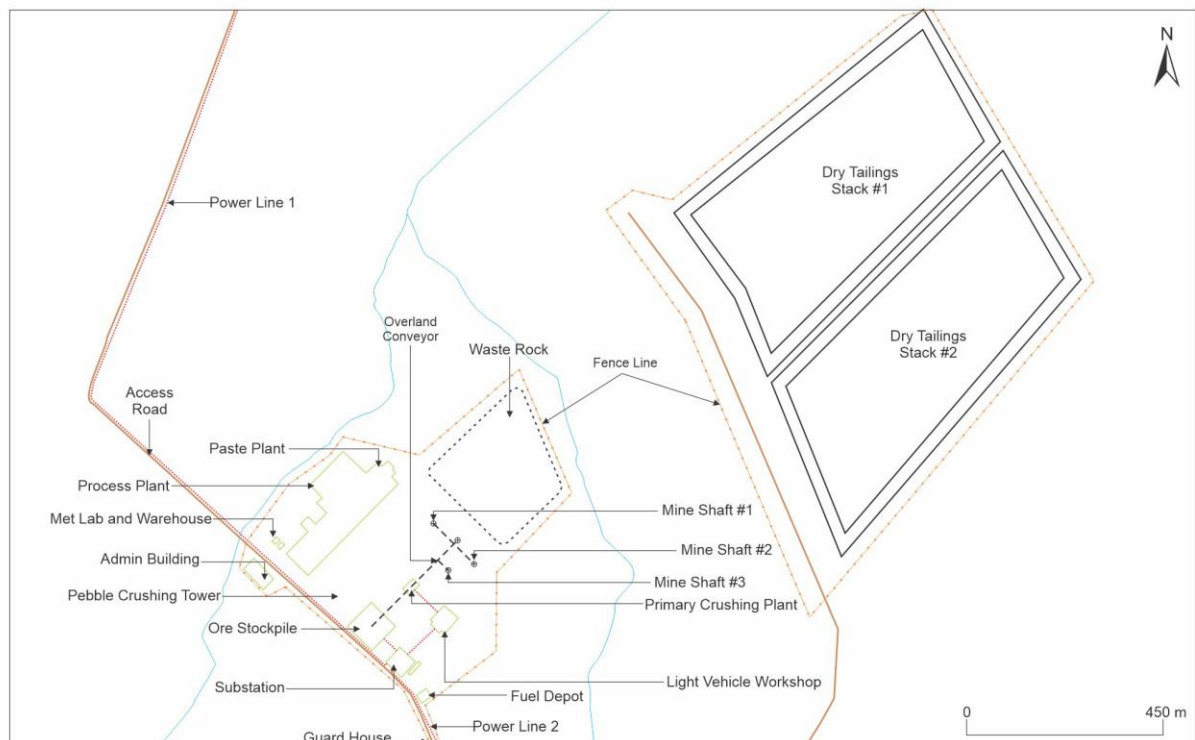
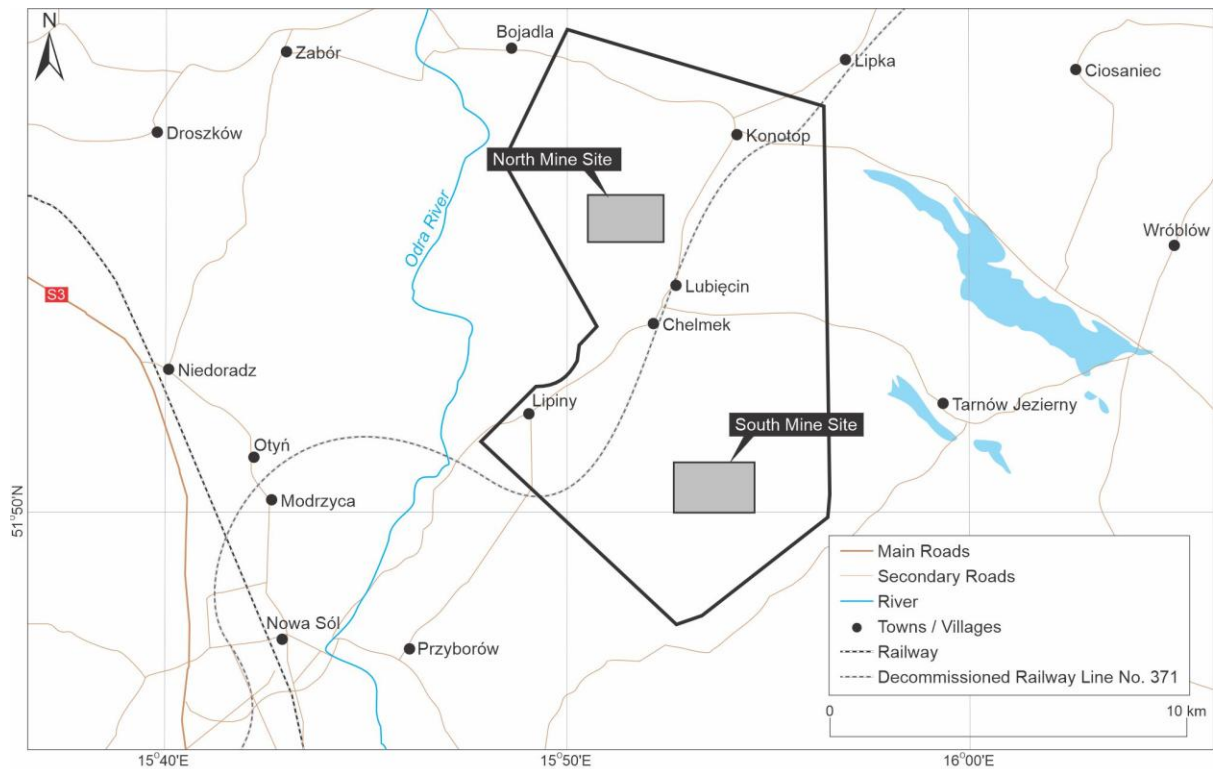
The total area requiring preparation at each site is expected to exceed 100 ha for the mine and surface infrastructure dry stack storage area for tailings. All site preparation activities will be carried out in accordance with Polish environmental and construction regulations, in particular with the Environmental Protection Law, Nature Conservation Act, Construction Law, Geological and Mining Law and other relevant acts. Tree removal, land transformation and foundation works will require specific permits and approvals under some of these acts.

At the PEA level, the exact footprint, scope of site preparation works and the extent of ground works necessary are not yet defined. It is therefore recommended that the next stage of design development be supported by a program of shallow on-site geological and geotechnical boreholes to enable the correct development of foundation design, earthworks quantities and ground treatment requirements for major facilities, shafts and the DSTF.

Any tree clearance should be carried out by specialist contractors in accordance with State Forest requirements.

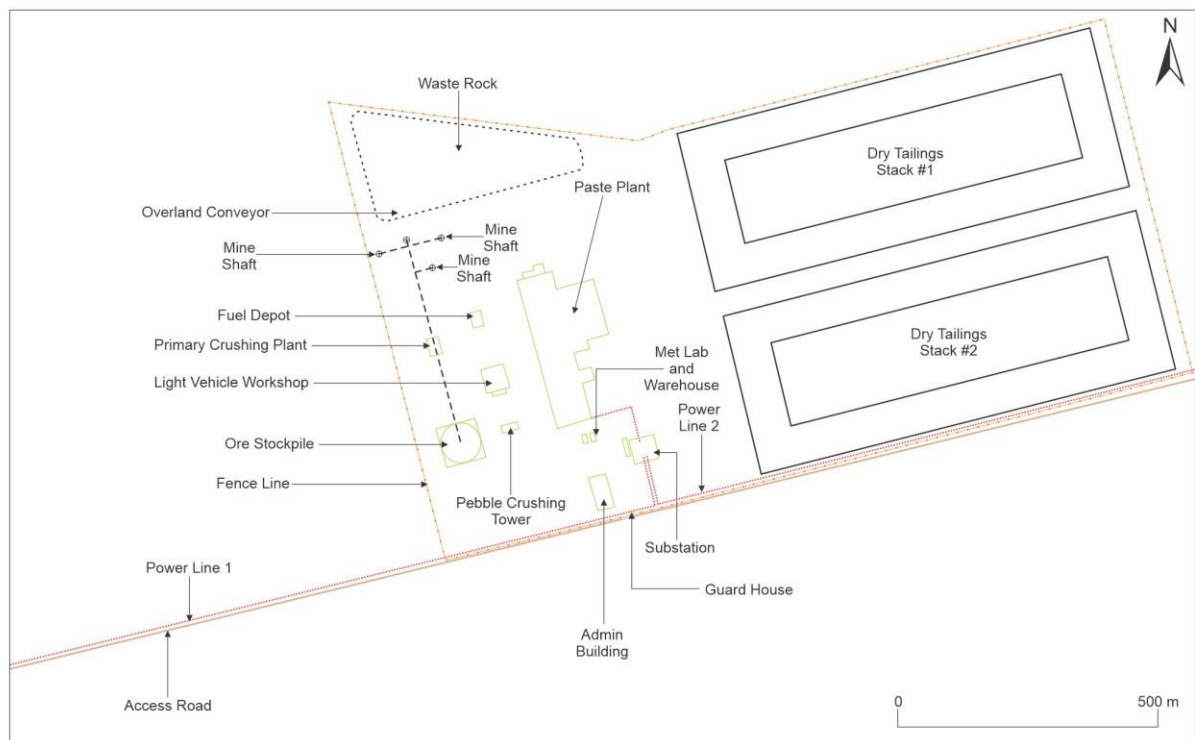
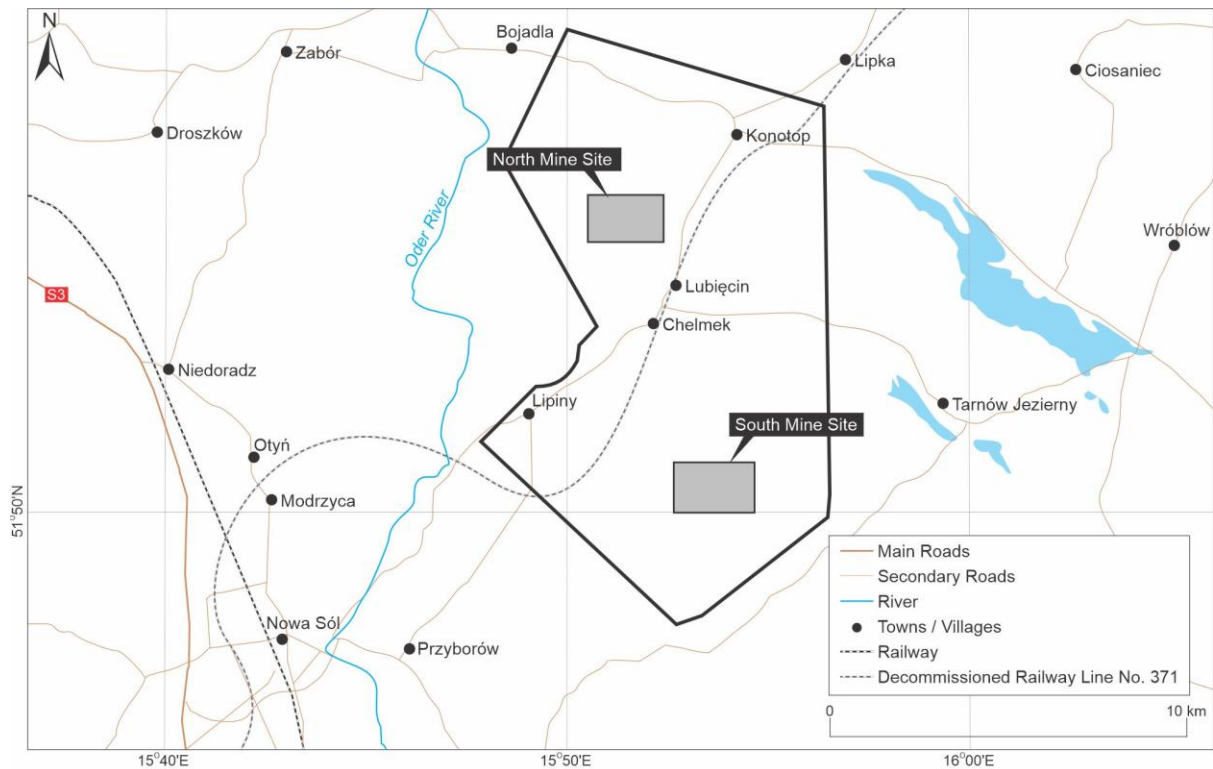
A provisional sum for planting/ environmental compensation is included in the CAPEX estimate.

Figure 18.1: Surface Infrastructure - North Shaft Complex



Source: Micon, 2026.

Figure 18.2: Surface Infrastructure - South Shaft Complex



Source: Micon, 2026.

18.3 ROADS AND OTHER ACCESS

The mine sites are located a few kilometres either side of regional paved road DW 315 which is a single carriageway highway giving access to the S3/ E65 motorway at Nowa Sól around 25 km to the southwest.

A detailed traffic and logistics study has not yet been carried out and this PEA describes access arrangements based on high level assumptions only.

It is assumed that the site will be accessed via a combination of existing public roads and new or upgraded access roads connecting the Project area to the local grid and regional road network. New two-way all-weather access roads from the local highway network to the boundary of the Project sites will be constructed. These will need to be available as part of the enabling works to allow access for the shaft sinking plant and equipment as well as personnel access and will later become the permanent access to the sites to avoid duplication of cost and disruption.

Given the transit requirements necessary for construction and to facilitate operations, these roads will be designed and constructed in accordance with Polish regulations for single carriageway roads suitable for standard, articulated heavy goods vehicles, without streetlighting, and will be designed to be adopted by the local authority later if practical. The carriageways will be 3 m wide which will mean that the total road corridor, plus verge, will amount to around 7.5 m. It may be necessary to fence the road corridor for safety. Depending on the final length and route, and although the speed limit on the site access road will likely be limited to 50 km/h the access roads may need to be provided with laybys, or additional passing places, to allow light vehicular traffic to pass heavy, slow moving loads. Signage and road markings will be in accordance with Polish road traffic regulations.

The site topography is relatively flat, and it is therefore expected that all new roads will have minimal gradients without any requirement for extensive earthworks, embankments or bridge works. Culverts will be provided where existing watercourses are evident.

On-site roads will be provided to connect the areas of hardstanding and facilitate movement between all the areas of the sites. The on-site roads, within the process facility boundaries, will generally be to a similar all-weather standard, which will enable easy access to all active parts of the facilities and will also help reduce any dust generated by frequent traffic activity.

A perimeter zone within the site boundary fence will be established to enable periodic inspection of the perimeter fencing and allow easy repairs and maintenance. This zone will include a track which will be used for intermittent access by site maintenance personnel in light vehicles, and therefore it is proposed that it is formed from a graded and compacted, drained gravel surface.

The main off-site access roads will be provided with drainage via soak aways, whereas runoff from all hard areas within the site perimeters will be collected and passed through an oil separator and treated before discharge.

The results of the detailed traffic study will be used to analyse the connections and routing of Project traffic to ensure that existing local infrastructure can support the additional traffic. Local roads, and particularly junctions and signage, will likely require upgrading to accommodate the additional traffic during construction and operations.

A significant opportunity for the Project is linked to the planned reopening and reconstruction of the currently decommissioned Wolsztyn–Nowa Sól–Żagań railway line, which is located close to the Project area. This creates the potential to use rail transport for heavy and oversized cargo during both the construction phase and later during mine operations, including delivery of equipment, consumables and potentially product transport.

It is strongly recommended that this opportunity is actively monitored and pursued, with the aim of integrating rail access into the Project's logistics and operations strategy through constructing a single-track branching from this line to the projected concentrator plants. Early engagement with the railway authorities and Project proponents should be undertaken to understand timelines, technical requirements and interface points. If feasible, rail access should become an integral part of the Project design and operations concept, as it could significantly reduce road traffic, lower logistics costs and minimize impacts on local communities.

Dispatch of concentrate to a local smelter could be via the railway line or pumped via a concentrate pipeline. A provision for concentrate transport has been made in the cash flow model and requires further investigation at the next phase of study.

18.4 PORT

The village of Nowa Sól contains a small inland port on the Odra River. The old Odra riverbed was closed off, shaping the port basin, which was later engineered to include a port canal.

There could be an opportunity to use river barges (current maximum barge tonnage is limited to 100 t, however plans are in place to increase this capacity) to transport equipment via river transport which provides a direct connection to the Baltic Sea at Szczecin and to the North Sea via the Odra-Elbe canal. The existing small-scale port at Nowa Sól is connected to the national railroad network and could be extended to provide dedicated lay down space, covered storage and loading facilities to accommodate loading product into river barges.

18.5 ELECTRICAL POWER SUPPLY

This section of the PEA covers the design and construction of the HV power supply to the mine site. Each of the mine sites will require two independent power lines to secure mining operations as required by Polish legislation. This is common practice in most countries to ensure continuity of supply and maintenance of all safety critical systems. The design and construction of the substations and their connection to the HV distribution network are essential for mining operations and are based on the following assumptions:

- Each of the mine sites must be equipped with at least two independent power supply systems. One system is required to meet the mine's full demand, while the secondary one must ensure the minimum power necessary to safeguard personnel and the environment, in accordance with the Regulation of the Minister of Energy of November 23, 2016, on detailed requirements for the operation of underground mining plants.
- Although the legislation specifies only a minimum capacity for the secondary power source, it is proposed to consider the assumption that the secondary power line is capable of supporting full mining operations. Both power lines are to be designed as single-circuit lines and positioned at safe distance from one another, as required by the Regulation.
- The total power demand for each of the two Shaft Complexes (North and South) is estimated at the PEA production stage to have a connected load of approximately 215 MW, Maximum Demand approximately 180 MW, annual consumption 1,500 GWh/a.

Two major power lines run north-to-south near the Project: a 220 kV line to the west and a 110 kV line to the east. The nearest 220 kV/110 kV substations are located in Żukowice (around 20 km from the deposit boundary) and Leśniów Wielki (some 40 km away). In addition, it is understood that the planned Baczyna-Plewiska 400/110 kV line will be provided with a substation in the vicinity (Nowa Sól Substation (NWS)) and it would be advantageous if this could be used to provide this connection as the large loads in question would be better suited to the higher voltage network.

Each Shaft Complex will be provided with two separate and independent overhead HV power supplies at 110 kV extended from the national grid to a dedicated private substation close to and within each site boundary. This is a requirement of Polish mining law.

Given that there is unlikely to be another major consumer beyond each site warranting a public supply, the overhead lines will almost certainly be dedicated to the Project and therefore funded by the Project developer.

Nevertheless, the overhead lines, and the private substations will be designed and installed to Polish public and mining supply regulations. This would also facilitate transfer to utility operation and maintenance at a later date.

The HV transmission network is regarded as critical national infrastructure and any connection to a grid substation will require an extensive lead time for planning and execution. As the initial supply will be required early for shaft sinking operations, Lumina have already commenced engagement with the national network operator PSE to ensure that the future requirements are included in strategic development plans. This engagement has resulted in inclusion of a dedicated substation for the Project into development plans of the national state grid operator PSE on a newly projected HV line, the NWS, directly adjacent to the Project site.

The on-site substations will be provided with suitable transformers and switchgear to enable on-site distribution at 15/20 kV for all process activities, such as comminution, thickening, flotation, backfilling, tailings dewatering together with all ancillary requirements and for mining activities, ore extraction and hoisting, ventilation, conveying, dewatering and other underground electrical equipment.

Other on-site distribution will be both at medium and low voltages to suit the load requirements.

In addition to the duplicate overhead supplies used for day-to-day operations, the sites will be provided with emergency, diesel engined, standby generators for essential loads such as mine ventilation and emergency egress, thickeners and similar loads. A full electrical load analysis will be carried out at a later date to optimise the generator configuration, and any load shedding required. Generators will be provided with dedicated, tanked and banded fuel oil supplies.

18.6 GAS SUPPLY

Natural gas is not required for the Project.

18.7 OTHER FUEL SUPPLY

The sites will be provided with bulk diesel oil storage tank farms for mining equipment, surface loaders and other, on-site, vehicular transport, and Liquefied Natural Gas (LNG) or propane storage tanks for the indirect gas fired heaters for the tempering of the mine ventilation supply air.

Dispensing will be provided at surface for surface vehicles and at the underground storage tanks for mining equipment and underground vehicles.

All diesel storage tanks will be in non permeable containment bunds satisfying the biggest of the following conditions: 110% of the capacity of the biggest storage tank, or 100% of the biggest tank and 10% of the capacity of all the other tanks within the same containment area.

The tanks will be serviced by road tankers which will be included in the traffic analysis for the access roads and local highways planning.

18.8 WATER SUPPLY AND WASTE WATER MANAGEMENT

A preliminary water balance is included in Section 17.4.6.

At this stage of the Project, and pending further technical studies, it is assumed that mine dewatering, combined with water recovered from the process plant and tailings, will be sufficient for providing the majority of water supply to the process plant and other Project needs. Water on-site will be kept in circulation and stored, treated and re-used as necessary. Additional studies during the PFS will include a detailed water balance and evaluation of different water supply options, including a supplementary supply.

Detailed water supply and sanitation requirements, including water treatment specifications, will be refined as the Project advances and undertaken in compliance with all applicable regulations.

18.9 BUILDINGS AND FACILITIES

18.9.1 On-Site Buildings

Each Shaft Complex will include a series of buildings as indicated in Figure 18.1 and Figure 18.2, as described in the following sections.

18.9.1.1 Main Process Plant Complex

The main process plant complex will be formed of several connected buildings to create a weatherproof enclosure to allow year-round operations.

The main process building will have a footprint of some 11,000 m² and will be a steel, portal framed structure with insulated cladding to provide both thermal and acoustic protection and concrete floor and intermediate floor slabs.

The main structure will likely include piled foundations to suit the prevailing ground conditions and will incorporate plinths, pits/ sumps and foundations for smaller equipment.

Subject to the proposed site geotechnical survey, it is likely that heavier and or rotating/ vibrating process equipment will need to have dedicated additional ground preparation and foundations.

Buildings will be constructed in accordance with Polish Building Codes and provided with all internal ancillary services such as local heating to internal offices, general lighting, a small power network for maintenance purposes, compressed air system, overhead cranes in the main hall, fire detection and alarm and fire protection, and IT/ comms necessary for the process equipment, for operation and maintenance purposes, for welfare purposes and for health and safety.

An associated steel framed and clad building will be constructed to provide reagent and consumables storage and for maintenance spares.

The main process plant buildings will receive crushed ore from the primary crushing plant via the crushed ore silo by conveyor.

Tailings slurry will be conditioned in the adjacent paste backfill plant before being directed underground as described in Section 16.6. Adjacent to this building will be silos and stockpiles of binder, sand and aggregate to service the paste plant.

Excess tailings will be dewatered before being transported to the DSTF as described in Section 16.6.18.

Product concentrate will be packaged in bulk bags in and dry loaded into containers, ready for trucking off-site.

18.9.1.2 Ancillary Buildings

Ancillary buildings will be necessary for other site purposes as follows:

- **Administration Offices** – a prefabricated modular building on prepared pad foundations to provide office accommodation, changing and welfare facilities for both mine and process plant operatives, canteen and break room, training rooms.
- **Emergency Rescue Facility** - self-contained building including storage and workshop for emergency and rescue vehicles, rescue equipment and rescue crews, first aid and medical treatment facilities and facilities for first aid staff.
- **Gate House/ Security Office** – a prefabricated modular building located alongside the main access road to control access to the site. This building will be suitable for 24-hour occupation and will be self-contained with lockers, toilets and a break room.
- **Metallurgical Laboratory** – modular construction, climate controlled, for on-site and production analysis, testing and assaying of both mine and processing samples. It will provide accommodation for a Supervisor plus shift and day technicians, laboratory equipment and sample storage.
- **Vehicle Workshop** – a steel framed and insulated panel clad building for the maintenance and repair of light vehicles, including vehicle lifting equipment and other garage equipment. The building will be sized for four maintenance bays plus space for spare parts and consumables storage, Supervisor's office and welfare (toilets, changing and lockers and break room).
- **Explosives Magazine** – a secure building within a designated compound to respect safety distances, designed in accordance with regulatory requirements, and for the interim storage of material prior to storage underground.

18.9.1.3 Other Facilities

The site will be bounded by a continuous perimeter security fence. This will not be monitored or provided with security lighting but will be provided with an internal track for access.

A paved car park will be provided in the vicinity of the administration building. This will be sized to provide parking for a full shift complement plus additional space for visitors and temporary staff. The final size of the car park will be influenced by the traffic study.

The car park will be within the secure perimeter and accessed through the main gate.

On-site roads will be generally paved and drained and provided with security lighting.

Drained hard stand areas will be provided for logistics staging, large part storage and vehicle manoeuvring.

A modular water treatment plant with associated holding tanks will be provided for potable water treatment and grey and black water management.

The fuel depot and gas storage tanks will be located close to the site entrance to limit on-site delivery tanker movements and to respect safety distances. The facility will include automatic level and leak monitoring.

18.10 SOLID AND GENERAL COMMERCIAL WASTE DISPOSAL

Waste material produced during the enabling works will mostly comprise waste rock and salt. As mine development is in ore material, the largest component to waste rock will be from the construction of the six shafts at the North and South Shaft Complexes, which will produce approximately 1.2 Mt of waste rock per complex.

These materials will initially be stored on-site for re-use or later disposal. Waste rock, predominantly sandstone, has the possibility of being crushed for use on-site during construction as fill and road bedding. Rock salt may be suitable for road de-icing. Further chemical analysis of the salt will be required to determine whether it will be suitable for sale to an off-taker who will crush, grade and clean the salt prior to sale to local authorities.

In addition to mining wastes (tailings, waste rock, etc.), the mine will produce a range of waste materials during construction and operations. These will largely be commercial and industrial solid wastes such as packaging from reagents and other materials, replaced parts from equipment, off-cuts and off-spec materials, rags and spoiled Personal Protective Equipment (PPE) and office wastes. There will also be domestic wastes from canteen services.

The site waste management plan will ensure that waste is reduced and recycled as much as possible and contracts will be established with licensed waste management operators for the removal of wastes, including any hazardous wastes, to appropriate facilities. Disposal of residues to suitable landfill or other facilities will only take place when there is no alternative. The mine will establish a waste collection area, with containers for the temporary storage of wastes, pending collection. Containers will be weather-proof and colour coded and will be designed to deter vermin. They may be located in a fenced compound, if necessary. Waste will be segregated in coordination with the waste management contractors, with separate storage for, as example, metals, plastics, wood, card and paper, electrical equipment, and rags. Containers will be labelled, and potentially contaminated and hazardous materials will be identified with warning signs. Wood waste will be reused, where possible. Uncontaminated construction wastes, such as surplus or demolished concrete or aggregates may be used as temporary pavements in the DSTF, especially during wet weather.

It is envisaged that liquid wastes, such as waste oils and hydraulic fluid, will be taken off-site and appropriately managed by the vehicle maintenance contractors. Storage tanks or drums would be located in the waste accumulation area for any ad-hoc liquid wastes that arise.

18.11 COMMUNICATIONS AND IT

A hybrid site wide telecoms and Information Technology (IT) network will be provided.

This will include both hard-wired (fibre and copper), wireless (Wi-Fi or cellular) and radio technology and will include connections to, and repeaters, base stations and antennae underground.

Process plant control and instrumentation is included within the process plant package but will be integrated within the site telecommunications strategy.

Hard wired networks comprising fibre optic and multi-core copper will be provided as part of the fit out of the various facilities to allow all systems to be integrated.

Critical system controls will be provided with a battery Uninterrupted Power Supply (UPS) for emergency power supplies.

It is assumed that the external fibre connection and reinforcement of the public cellular network will be provided by third party providers and are not specifically allowed for with the cost estimate.

18.12 TAILINGS MANAGEMENT FACILITY

The storage of tailings on surface is intended to be kept to an absolute minimum, and preferably only as a temporary measure, with the application of underground paste backfill and the DSTF for surplus filtered tailings that are not utilized for underground placement. The backfill study is discussed in Section 16.6.

19 MARKET STUDIES AND CONTRACTS

19.1 MARKET STUDIES

19.1.1 Commodity Overview and Product Context

Copper is a widely used industrial metal and is essential to modern infrastructure due to its electrical conductivity, corrosion resistance and workability. Demand for copper is supported by its use across construction, power transmission, transport systems, defense industry and digital technologies.

The Nowa Sól Copper-Silver Project is located directly north of the LGOM of southwestern Poland in the Polish Northern Copper Belt. The LGOM district has a strong history of copper and silver mining, spanning back to the late 1960s, and currently contains multiple underground mines, processing facilities and smelters with available capacity operated by KGHM Polska Miedź S.A. (KGHM, n.d.-a).

For the purposes of this PEA, it is assumed that any future development of the Project would target the same general product types as those produced by KGHM's nearby Polish operations. These operations, which produce over 600 kt of refined copper and over 40 Moz of silver per year, are considered an appropriate regional analogue for product type, product quality and market access.

Based on publicly available disclosures, KGHM's Polish operations produce copper concentrate at on-site concentrators, with typical concentrate grades of approximately 23% Cu. These concentrates are processed at smelting and refining facilities in Głogów and Legnica to produce refined copper cathodes with a minimum purity of 99.99% Cu. The cathodes are registered as Grade A deliverable on the London Metal Exchange (LME). In addition to cathode production, KGHM manufactures downstream copper products, including copper wire rod, and recovers several by-products during metallurgical processing, including refined lead, sulphuric acid, copper sulphate, nickel sulphate, selenium, and rhenium (KGHM, n.d.-b; KGHM, n.d.-c; KGHM, n.d.-d; KGHM, n.d.-e; KGHM, 2022).

Public disclosures further indicate that silver is recovered at the Precious Metals Plant at the Głogów Copper Smelter and Refinery. KGHM reported production of approximately 45 Moz of metallic silver in 2023, with total group silver production typically exceeding 40 Moz per annum (KGHM, n.d.-c; KGHM, n.d.-f). Silver is reportedly sold in the form of bars and granules.

On the basis of regional operating precedent and publicly available information, the Project is assumed in this PEA to produce a copper concentrate of comparable quality containing payable copper and silver. This assumption is conceptual in nature and will require confirmation through future metallurgical testwork, process design studies, and engineering evaluations.

19.1.2 Global Supply

Global copper supply is dominated by primary mine production. Secondary supply from recycling is material and has grown over time, with current estimates suggesting that recycled copper contributes approximately 30% of annual global demand (International Copper Association, 2023).

Silver supply is similarly derived from a combination of primary mine production and secondary sources. However, unlike copper, a significant proportion of global silver production is generated as a by-product of mining for other metals, including copper, lead, zinc and gold. Recycling also represents an important component of silver supply, contributing meaningfully to overall market balance (Silver Institute and Metals Focus, 2024).

In 2024, total global copper mine production was approximately 23 Mt. Chile remained the largest producing country, with output of around 5.3 Mt, followed by the Democratic Republic of Congo (approximately 3.3 Mt), Peru (approximately 2.6 Mt), China (approximately 1.8 Mt) and the United States (approximately 1.1 Mt) (USGS, 2025).

Global silver mine production in 2024 was approximately 820 Moz, reflecting a recovery from pandemic-related disruptions but remaining constrained relative to consumption. Major silver producing countries include Mexico, China, Peru and Chile, with production trends influenced by the performance of both primary silver mines and base-metal operations from which silver is recovered as a by-product (Silver Institute and Metals Focus, 2024).

A number of supply-side constraints are evident in both the copper and silver markets. Many operating mines are mature and are experiencing declining grades. At the same time, the pipeline of new projects appears smaller than in previous development cycles. This applies to both brownfield expansions and greenfield developments. Higher capital intensity, declining grades, longer permitting timelines and increased environmental and stakeholder requirements are expected to affect future project development across both commodities (BHP, 2024; S&P Global, 2025).

Supply disruptions have also affected both markets in recent years. Reduced output, delays in project ramp-ups and operational issues at several major operations have contributed to periodic supply deficits. In the silver market, supply growth is further constrained by its reliance on base-metal production, limiting the responsiveness of silver supply to price signals. Several independent outlooks reviewed for this study indicate that more persistent supply shortfalls could emerge towards the latter part of the 2020s for a number of critical minerals, including copper, with silver supply also expected to remain relatively tight under current market conditions (USGS, 2024; IEA, 2025). In response, investment in new and expanded copper mining capacity is ongoing, while silver supply growth is expected to remain modest and closely linked to broader base-metal production trends (UNCTAD, 2025; Silver Institute and Metals Focus, 2024).

19.1.3 Global Demand

Demand for copper and silver is supported by a combination of long-term structural trends and cyclical factors. Copper demand continues to be driven primarily by electrification, urbanisation, infrastructure development and the transition toward lower-carbon energy systems, as well as increased defence and infrastructure spending in several regions. Growth in the digital economy and defense industry is also contributing to copper demand, particularly through increased requirements for power transmission, data centres and artificial intelligence-related infrastructure (UNCTAD, 2025; BHP, 2025; IEA, 2025).

The global energy transition is increasing the amount of copper required per unit of output in several end-use sectors, including renewable power generation, electric vehicles and grid infrastructure. As a result, copper intensity is expected to increase across a range of applications relative to historical norms.

Silver demand is driven by a combination of industrial applications and investment demand. Industrial uses represent a significant and growing share of global silver consumption, reflecting silver's high electrical and thermal conductivity. Key industrial applications include electronics, PV solar panels and chemical processes (Silver Institute and Metals Focus, 2024; Silver Institute and Oxford Economics, 2025). The expansion of global solar PV capacity has become an increasingly important driver of silver demand, with PV applications accounting for a growing proportion of total industrial silver consumption in recent years and expected to continue supporting demand over the medium to long term (Silver Institute and Oxford Economics, 2025).

In addition to industrial demand, investment demand for silver, including bullion, coins and exchange-traded products, contributes materially to overall demand and can influence market conditions, particularly during periods of economic uncertainty or heightened inflationary pressures (Metals Focus, 2024; Silver Institute, 2025).

Long-term demand projections reviewed for this assessment indicate that global copper demand could increase by approximately 70% to exceed 50 Mt/a by 2050. Copper demand growth is expected to accelerate relative to recent historical trends, with an estimated compound annual growth rate of approximately 2.6% through to 2035, compared with growth of approximately 1.9% per annum over the preceding 15 years (BHP, 2024; S&P Global, 2025). While long-term silver demand forecasts vary, industrial demand associated with energy transition technologies, particularly solar photovoltaics and electrification, is expected to remain a key structural driver of silver consumption (Silver Institute and Oxford Economics, 2025).

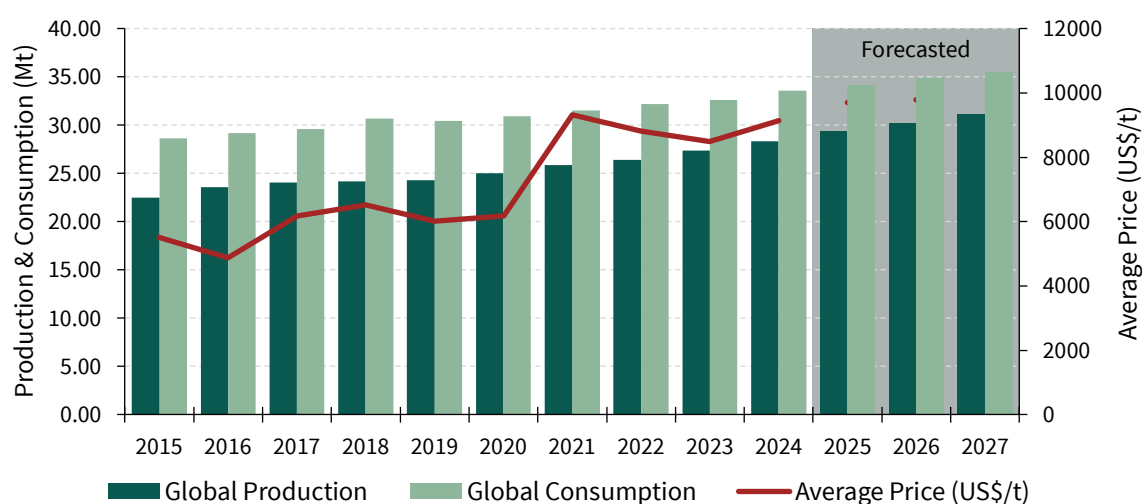
19.1.4 Pricing, Production and Consumption Trends

Copper and silver prices are influenced by a combination of demand conditions, supply availability, inventory levels and broader macroeconomic factors. Copper prices are primarily established on the LME, which provides the global pricing benchmark for physical and financial copper transactions.

Silver prices are determined in highly liquid global markets and are influenced by both industrial demand and investment activity. Benchmark pricing for silver is provided through established bullion markets and exchange-based trading, with prices typically quoted in United States dollars per troy ounce (US\$/oz). As a result, silver prices may exhibit higher short-term volatility than copper, reflecting sensitivity to investment flows as well as underlying industrial fundamentals.

Figure 19.1 shows global copper mine production, global copper consumption (represented by world refined copper usage) and average annual copper prices for the period 2015 to 2027.

Figure 19.1: Overview of Copper Supply, Demand and Price 2015 to 2027



Source: Micon, compiled from International Copper Study Group (ICSG) World Copper Factbook 2025 and World Bank Commodities Price Data (The Pink Sheet) (various issues).

Note: Some series were extracted from published tables/figures; minor rounding differences may occur.

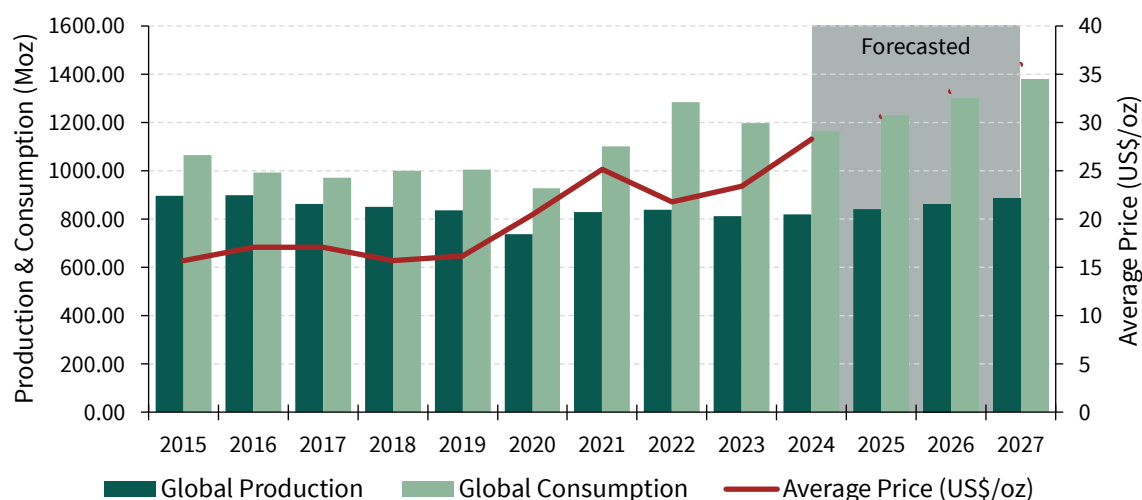
Between 2015 and 2024, both global copper mine production and refined copper usage increased steadily. Copper prices over the same period were volatile but showed an overall upward trend.

Copper prices rose during January and February 2025. This increase was influenced in part by tariff-related concerns and broader market uncertainty (S&P Global, 2025a). Treatment and refining charges fell to historically low levels in early 2025 and are expected to remain low, reflecting tight concentrate supply. Sustained low treatment charges may place pressure on smelter margins and could result in production curtailments at higher-cost operations (S&P Global, 2025a). A copper concentrate deficit anticipated in 2025 is expected to provide additional price support (Mining.com, 2023).

Consumption values shown for 2025 to 2027 are illustrative projections. They are based on extrapolation of recent historical growth in refined copper usage, using ICSG-reported data through 2024. These values are provided for market context only and are not used for economic analysis.

Figure 19.2 shows global silver mine production, global silver consumption and average annual silver prices for the period 2015 to 2027.

Figure 19.2: Overview of Silver Supply, Demand and Price 2015 to 2027



Source: Micon, compiled from the Silver Institute World Silver Survey and World Bank Commodities Price Data (The Pink Sheet) (various issues).

Silver prices strengthened over the period from 2021 to 2024, reflecting a combination of recovering industrial demand following the COVID-19 disruption, increased investment interest, and tightening market conditions. Average annual silver prices increased from approximately US\$20.5/oz in 2020 to US\$28.3/oz in 2024. This upward trend is consistent with a period of sustained growth in global silver consumption, which increased materially over the same period.

Global silver production recovered from pandemic-related disruptions in 2020, but has remained constrained relative to consumption. Between 2020 and 2024, silver mine production increased from approximately 739 Moz to approximately 820 Moz, while global silver consumption increased from approximately 929 Moz to approximately 1,164 Moz. This divergence reflects ongoing supply constraints and strengthening industrial demand, particularly from electronics and PV applications.

Silver consumption values shown for 2025 to 2027 are illustrative projections. They are based on extrapolation of recent post-2020 growth trends in global silver usage, reflecting continued industrial demand growth and energy-transition-related applications. Production values for the

same period assume modest growth, consistent with recent stabilisation in mine supply. Forecast silver price values for 2025 to 2027 reflect continuation of recent price trends and market tightening but remain subject to volatility. These forward-looking values are provided for market context only and are not used for economic analysis.

19.1.5 Market Implications for the Project

Established global pricing mechanisms, active concentrate and refined metal markets, and continued long-term demand growth indicate that viable markets are likely to exist for copper and silver products from the Project, should it advance to development. Copper pricing is benchmarked through established exchanges such as the LME, while silver pricing is supported by transparent and liquid global bullion and industrial markets.

Silver is expected to be recovered as a payable by-product, consistent with other copper-silver operations within the LGOM. The presence of established markets for both industrial and investment silver products supports the potential for silver to contribute positively to overall Project revenues, subject to future metallurgical performance and commercial terms.

The Project's location within an established copper and silver mining district provides access to existing processing, refining and logistics infrastructure. This is expected to allow a range of potential marketing options for copper-silver concentrate and refined products, including treatment at regional smelting and refining facilities, subject to future commercial arrangements.

All market projections discussed in this section are forward-looking and subject to uncertainty. No economic analysis is presented. The QP has reviewed the market information presented and considers it appropriate for a PEA.

19.2 CONTRACTS

As the Project remains in the advanced exploration stage, no formal contracts have been finalized for land use agreements, infrastructure development, mining, offtake arrangements, smelting, refining, transportation, sales, marketing, or other commercial agreements.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section of the report summarizes the legislative requirements and current status of permitting and environmental studies for the Project. It outlines the environmental and social context of the local area and identifies preliminary risks and impacts at a strategic level, along with conceptual approaches for management and closure planning. Information is based on secondary data, technical discussions with the wider Project team, and a site visit undertaken by the QP in July 2025.

20.1 REGULATORY FRAMEWORK AND PROJECT PERMITTING

The Project is subject to the national laws and regulations of the Republic of Poland and the European Union (EU). Lumina also plans to implement the project in accordance with good international industry practice (GIIIP), including potential requirements of international investors and public offering.

20.1.1 Summary of Key Mining Legislation

Mineral rights in Poland are solely owned by the State. Exploration and mining activities are overseen by the MCE (*Ministerstwo Klimatu i Środowiska*) and administered at local level by the relevant District Mining Office (*Okręgowy urząd górnicze*).

The Geological and Mining Law (Journal of Laws of 2011, Law No.163, Item 981 of June 9, 2011, as amended 2018) is the principal legislation which defines the administrative procedures and requirements for obtaining exploration and mining concessions. An exploration or mining concession granted by the MCE represents a license / permit that entitles its holder to conduct a particular activity on a specific deposit in a defined area and within a defined timeframe. The concession document sets out the holder's rights and obligations, as well as the terms and conditions for conducting the planned activity (such as exploration drilling or mining).

During the prospecting and exploration stage of a project, the developer conducts work which leads to the acquisition of geological information. Information acquired through an exploration program is formally documented in a technical report known as the 'geological documentation' and is submitted to the MCE. The MCE approves the geological documentation by issuing an administrative decision, which is based on a review / opinion from the National Resource Commission (*Komisja Zasobów Kopalin*).

The geological documentation contains technical information about the mineralization and, if applicable, the resource classification of a deposit. Under Polish law, there are five resource classification categories which can be applied to a solid mineral deposit: D, C₂, C₁, B, A. These categories are aligned with the former Soviet (Russian) GKZ system (*Gosudarstvennaya Komissia po Zapasam* / State Reserves Committee). The A category is the highest resource classification category with the most precise information about the deposit, whereas the D category is the lowest. Any changes to the geological information, including the mineral classification, are made by preparing an addendum to the documentation when additional work is conducted, and must be approved by the MCE as a separate administrative decision.

A resource classification of C₁ or higher is required before a mining concession can be applied for. The ownership of all geological information lies with the State, however, upon achieving a C₁ resource category or higher (granted by the Polish Resource Commission), the project developer is entitled to an 'exclusivity right' to use the geological information. The developer is also entitled to a 'priority right' to establish a contract with the State to undertake mining operations (known as a 'mining usufruct', *użytkowanie górnicze*) and to subsequently apply for a mining concession.

The priority right and the exclusivity right are both valid for five years. These rights are instruments which ensure the transition from the exploration phase to the mineral production phase of a project. They protect the Project developer's position against competitors and ensure that there is sufficient time to prepare a comprehensive application for a mining concession. No other entity can obtain an exploration or mining concession for the same deposit during this five-year period.

In addition to the Mining Law, various supporting regulations and acts are in place, which provide more detailed requirements on the practical application of the Mining Law. Relevant regulations include, but are not limited to:

- Regulation on detailed requirements for projects of geological works, including works for which concessions are required for their execution (Item 1696 of December 20, 2011, as amended by Item 964 of July 9, 2015).
- Regulation on detailed requirements of mineral deposit development plans (Item 511 of April 24, 2012).
- Regulation on detailed requirements for the operation of underground mining plants (Item 1118, of November 23, 2016).
- Act on mining waste, which requires operators to prepare and implement a comprehensive extractive waste management plan (Journal of Laws of 2008, Item 865 of July 10, 2008).

20.1.2 Summary of Key Environmental Legislation

The Polish Mining Law and associated Regulations incorporate environmental protection as a fundamental component, with mining concessions requiring an environmental permit (environmental decision, *decyzja o uwarunkowaniach środowiskowych inwestycji*). Unlike many jurisdictions, exploration and mining concessions are granted by the MCE, emphasising the importance that the Polish legal system places on environmental compliance.

Specific environmental legislation of relevance to the Project includes, but is not limited to:

- **The Environmental Protection Act** (Journal of Laws 2017, Law No. 62, Item 627 of April 27, 2001, as amended in 2025). This overarching law establishes the legal basis for environmental protection, including general principles for environmental management and public participation. It defines responsibilities for government departments, businesses and citizens and requires that environmental considerations are integrated into development planning.
- **The Act on provision of information on the environment and its protection, public participation in environmental protection and environmental impact assessments** (Journal of Laws 2008, Law No. 199, Item 127 of October 3, 2008). This law mandates that projects with the potential for significant environmental impacts (including mining projects) must be subject to an EIA before approval. The EIA process is outlined, including the right of public participation.
- **The Act on Nature Protection** (Journal of Laws 2004, Law No. 92, Item 880 of April 16, 2004). This law establishes the legal framework for the creation and management of protected areas, including national parks, nature reserves, and EU Natura 2000 sites. It encourages biodiversity conservation, provides protection for endangered species, and regulates hunting activities.
- **The Water Act** (Journal of Laws 2017, Law No. 960, Item 1566 of July 20, 2017). This law is aligned with the EU Water Framework Directive and regulates the use and protection of water resources in Poland, including the management of rivers, lakes, and groundwater. It

establishes the procedure for obtaining permits for water use and discharge, and requires River Basin Management Plans to be developed.

- **The Act on the protection of atmospheric air against pollution** (Journal of Laws 1966, Law No. 14, Item 87 of April 21, 1966). This law is aligned with the EU Ambient Air Quality Directive and establishes monitoring systems and standards for air quality. Poland also has a **Greenhouse Gas Emissions Trading Act** which affects private companies.
- **The Waste Act** (Journal of Laws 2012, Law No. 2013, Item 21 of December 14, 2012). This law establishes the general framework for waste management in Poland, requiring compliance with environmental protection principles throughout the lifecycle of waste. It imposes obligations on entities to ensure proper collection, storage, and disposal of waste, minimizing environmental impacts and ensuring sustainable practices.
- **The Spatial Planning and Development Act** (Law No. 80, Item 717 of March 27, 2003, as amended in 2025) sets out requirements for local governments to develop and implement General Plans and Local Spatial Development Plans, which are updated every five years.

A detailed list of all applicable mining, environmental and supporting legislation will be prepared as the Project develops.

20.1.3 Project Permitting Process

The requirements for obtaining exploration and mining concessions in Poland follow a similar general process, with mining concessions requiring significantly more detail in the supporting documentation and additional consultation. The permitting process is considered to be complex, with separate but interrelated evaluation and administrative decision making needed by the competent regulatory authorities on the geological potential of the deposit and the environmental risks of the proposed operations.

The application process relevant to the Project is that of applying for a mining concession on the basis of exclusivity and priority rights associated with C₁ classification of the state-owned copper deposit at Nowa Sól.

Four key stages, with associated approval decisions, must be followed prior to making the formal application for a mining concession. These stages are summarized below:

1. **Geological Decision** – For all mining concession applications, the ‘geological documentation’ must be prepared and formally approved by MCE. A Deposit Development Plan (*plan zagospodarowania złoża / PZZ*) must also be prepared. The PZZ is based on the geological documentation, but also incorporates elements of the EIA as well as technical and economic assumptions about the Project. The PZZ is not approved by a separate administrative decision, but forms part of the mandatory supporting documents for the mining concession application and is therefore reviewed during the overall application against the legislated requirements.
2. **Environmental Decision** – An EIA (*Ocena oddziaływania na środowisko / OOS*) must be undertaken for the Project and an Environmental Impact Report (EIR) prepared on the findings. Preparation of the EIR requires data from baseline environmental studies and information from the PZZ (the EIR and PZZ are prepared at the same time). The report format is governed by legislative requirements and includes typical content such as a detailed project description, analysis of alternative options, status of natural resources in the area (water, biodiversity etc.), impacts and risks, mitigation and monitoring requirements and identification of potential social conflicts. Particular emphasis is placed

on understanding emissions associated with air, noise, water and waste products. The EIR must be formally submitted for technical review and public consultation, and a positive environmental decision on the Project obtained.

3. **Spatial Planning Decision** – For mining concession applications, the local and regional land use and zoning plans must be reviewed and updated to ensure there is no conflict of interest with the proposed mining operations. Amendment of local spatial planning designations may be required and therefore early communication between the Project developer and representatives of the local commune / municipality is essential. A separate environmental report must be developed specifically for the local authorities based on key elements of the PZZ and EIA, to explain the precise location of the Project and its context in terms of the relevant land use plan and spatial zoning plan. Evidence of support for the Project and intention to incorporate it into the relevant General Municipal Plan (*Plan ogólny gminy* / POG) and Local Spatial Development Plan (*Miejskowy Plan Zagospodarowania Przestrzennego* / MPZP) must be obtained, with formal incorporation into the plans to be completed within 2 years of the concession being granted.
4. **Mining Usufruct** – The terms and conditions for the ‘mining usufruct’ contract between the Project developer and the State must be agreed. The usufruct provides the legal right to exclusively manage and exploit specified mineral deposits or geological resources within a given area, and is distinct from surface land use rights.

The formal application for a mining concession should include evidence and supporting documentation concerning the four decisions outlined above, as well as essential information about the Project developer, a detailed Project description, and financial guarantees. Specific hydrogeological and engineering-geological documentation are also required, and must be approved respectively by the Marshall (*Marszałek Województwa*) of the relevant region and the Chief National Geologist within the MCE. MCE may also request additional supporting information to be provided after the concession application has been made.

The overall mine permitting process is estimated to take at least 48 months (four years) and therefore careful scheduling will be necessary to ensure the process is completed within the five-year exclusivity and priority period. The usufruct agreement will be legally concluded and become effective once the concession is granted, and will last for the duration of the concession, which is requested by the Project developer and can range from 3 to 50 years. The concession boundary will be aligned to the PZZ and a fee is payable within fourteen days of award of the concession.

Once a mining concession has been awarded, the necessary supporting permits can be obtained. A full list of permits will be identified as the Project advances, but will include surface land use permits, building permits for supporting infrastructure, a water permit and a blasting permit. The operator must also prepare, and obtain approval of, a Mining Plant Operation Plan (*Plan Ruchu Zakładu Górniczego*) which is a comprehensive document governing the safe and lawful conduct of mining operations including technical, environmental, and safety measures.

20.1.4 Project Permitting Status

Lumina previously held four contiguous exploration concessions (permits) in the Project area, under their Polish subsidiary Zielona Góra Copper sp. z o.o. These concessions covered areas named Nowa Sól, Zatonie, Jany and Wilcze; the latter three of which expired in May 2022. To support the exploration activities covered by these permits, Lumina secured temporary surface rights by leasing land from private, corporate, and public owners. After completing exploration activities, all

leased land areas were restored to their original use, in compliance with legal requirements, and additional tree planting has been undertaken.

The Nowa Sól exploration concession (17/2011/p) was valid until December 31, 2024. Prior to its expiration, an application was made to upgrade the deposit's resource category to C₁ via Addendum No. 2. Approval for the C₁ designation was granted by MCE on April 25, 2025, thus giving Lumina a five-year exclusivity period to prepare the necessary technical documentation, undertake the environmental permitting process, and apply for a mining concession.

In the interim period, applications for two new contiguous exploration concessions were submitted by Zielona Góra Copper sp. z o.o and subsequently granted by the MCE:

- Nowa Sól II (9/2025/p) – This concession was awarded on October 28, 2025, and covers the same 120 km² surface area (119.151 km²) as the former Nowa Sól Concession, upon which the C₁ resource category was documented. The concession is valid for 3.5 years.
- Konotop (12/2025/p) – This concession was awarded on December 9, 2025, and covers a 40 km² surface area (39.684 km²) tangent to the north-eastern boundary of the Nowa Sól II Concession. The concession is valid for six years.

Refer to Figure 4.1 in Section 4 for the location of both exploration concessions.

20.1.5 Good International Industry Practice

In addition to compliance with all applicable legal requirements, Lumina intends to develop the Project in alignment with GIIP. Lumina publicly state on their website that *“We are committed to responsible and safe resource development that benefits communities and protects the environment”*.

Lumina are in the process of developing their Environmental, Social and Governance (ESG) strategy. Examples of relevant and widely accepted international ESG guidelines are as follows:

- **International Finance Corporation Environmental and Social Performance Standards (IFC PS)** – these are part of the IFC's Sustainability Framework. The IFC PS provide a baseline of environmental and social good practice and form an important assessment reference.
- **Equator Principles (EP)** – these form a risk management framework, adopted by international financial institutions for determining, assessing and managing environmental and social risk in projects. The EP framework is based on the IFC PS and on the World Bank Group (WBG) Environmental, Health and Safety (EHS) Guidelines on environmental and social sustainability.
- **World Bank Environmental, Health and Safety Guidelines (WB EHS)** – these provide a source of technical information during project appraisal. They are widely accepted as technical reference documents presenting general and industry specific examples of GIIP.
- **International Council on Mining and Metals (ICMM) Mining Principles** – these have been developed in response to evolving societal expectations of the mining industry, and include a comprehensive set of Performance Expectations, Position Statements, and Good Practice Guides, including widely recognized guidelines for integrated mine closure.
- **Global Industry Standard for Tailings Management (GISTM)** – the Standard was developed by an independent review process in response to a number of tailings dam failures. It was initiated by the International Council on Mining and Metals (ICMM), United Nations Environment Program (UNEP) and Principles for Responsible Investment (PRI) and

provides a global benchmark to achieve strong social, environmental and technical outcomes in tailings management.

20.2 STATUS OF ENVIRONMENTAL AND SOCIAL STUDIES

Environmental and social baseline studies for the Project have not yet commenced. Lumina has engaged local consultants to undertake these studies and the subsequent EIA. Local consultants will also be contracted to prepare supporting technical documentation such as the Deposit Development Plan (PZZ). It is anticipated that environmental fieldwork will commence in Spring 2026. The permitting process for the Project and associated mining concession application will be scheduled for completion within the five-year exclusivity and priority period associated with C₁ classification of the Nowa Sól deposit.

20.3 ENVIRONMENTAL AND SOCIAL CONTEXT

Environmental and social baseline studies for the Project have not yet commenced. This sub-section is based on publicly available secondary data and observations made by the QP during the July 2025 site visit.

20.3.1 Overview

The Project is located in a rural area of the Lubuskie Province (*Województwo Lubuskie /Lubuskie Vovoideship*) in western Poland, approximately 85 km east of the border with Germany and 120 km north of the border with Czechia (refer to Figure 4.1). The site is approximately 320 km west of the capital city of Warsaw and approximately 100 km north of the city of Wrocław. There are good transport links by road, rail and air and the Project has been incorporated into the Lubuskie regional development strategy (adopted by the Assembly of the Lubuskie Province on February 15, 2021).

The current exploration concession covers a surface area of approximately 120 km² and is 17.1 km at its longest extent from north to south, and 10 km at its widest extent from east to west. The topography is relatively flat, with gently undulating lowlands that form part of the larger Polish Plain and elevations ranging from 60 masl to 110 masl. The climate in the Lubuskie Province is temperate, with distinct seasons and average temperatures ranging from around -3°C in winter (January) to 25°C in summer (July). Rainfall occurs year-round and peaks in summer months, with average annual totals of around 600 mm to 650 mm.

The northern and southern zones of the exploration concession, where mining and processing operations are proposed, comprise significant forest cover which is managed by the Polish State Forestry Service (*Lasy Państwowe*) primarily for commercial timber production, and to a lesser extent nature conservation, hunting and recreational use (Figure 20.1). The central zone of the concession is a mix of agricultural and forestry use, and includes some residential areas.

The surrounding area is characterized by rivers, lakes and forests. Existing baseline conditions will be influenced by a combination of agricultural activities, commercial forestry, domestic village life, and visitors for recreation and tourism. Baseline air quality is anticipated to be relatively good, and background levels of noise and artificial lighting are expected to be low. Traffic volumes are likely to be associated with local road users, since although the area is well connected, the main national and international transport routes do not pass through the concession.

Figure 20.1: Forested Area in South Zone of Nowa Sól II Concession

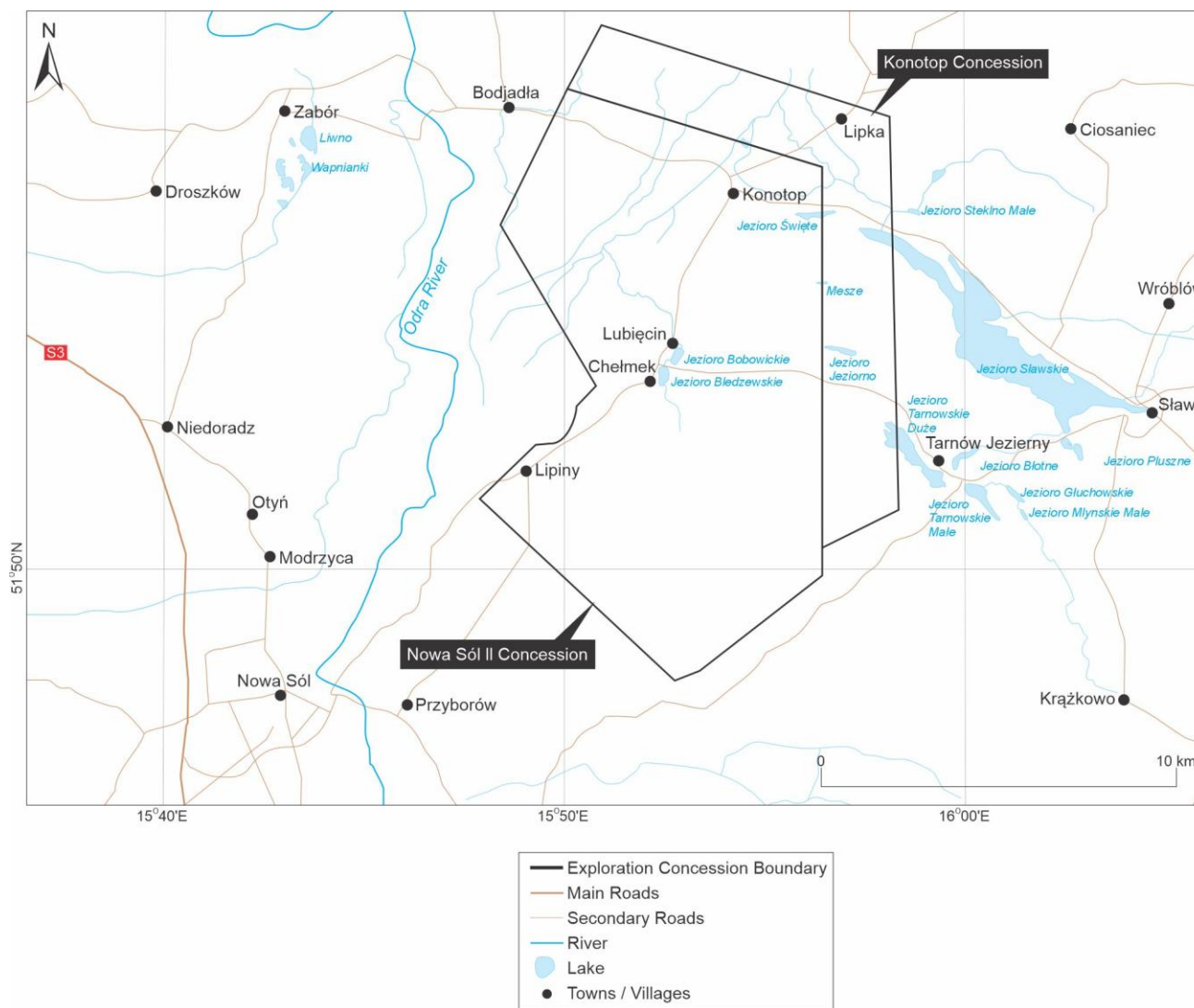


Source: Micon, 2025.

20.3.2 Water Resources

The Project is located in the middle reaches of the Odra River basin, which spans a total length of >800 km from Czechia in the south and into Poland, forming the Polish border with Germany and then flowing north along the border towards the Szczecin Lagoon near the Baltic Sea. The river flows approximately parallel to the west of the Nowa Sól II exploration concession, and is 1 km from the concession boundary at its closest point (Figure 20.2). Water quality sampling for the Project has not yet been undertaken, but the Odra River catchment is known to be influenced by agricultural and mining activity. A significant pollution incident occurred in 2022 which resulted in a large fish kill; various sources of information suggest this was caused by a release of highly saline water from upstream industrial operations.

Figure 20.2: Water Features in the Project Area



Source: Micon, 2026.

The area surrounding the Project hosts numerous freshwater lakes. There are several seasonal streams in the northern zone of the exploration concession, and two freshwater lakes (Figure 20.3). Lake Bledzewskie (*Jezioro Bledzewskie*) and Lake Bobowickie (*Jezioro Bobowickie*) are connected and act as focal points for the villages of Lubięcín and Chelmek in the central zone of the concession. They provide bird and fish habitat and are popular for recreational water use including boating and fishing (Figure 20.3).

Figure 20.3: Lake Bledzewskie, Lubięcín



Source: Micon, 2025.

Hydrogeological studies for the Project area have not yet been undertaken; however a review of secondary data (CQA, 2026) has identified a range of distinct groundwater systems separated by low permeability clay or evaporite layers. Shallower aquifers are comprised of Quaternary and Neogene deposits of glaciofluvial sands and gravels that are close to the surface and likely connected to surface water systems. Intermediate depth aquifers are formed in Triassic sandstones and marls, and deeper aquifers are found in the Zechstein dolomites and limestones, and deeper sandstones.

The underground mining area will require dewatering. The order of magnitude for potential water volumes cannot yet be accurately estimated as detailed hydrogeological studies have not yet commenced. At this stage of the Project, and pending further technical studies, it is assumed that mine dewatering, combined with water recovered from the process plant and tailings, will be sufficient for providing the majority of the water supply to the process plant and other Project needs. Water on-site will be kept in circulation and stored, treated and re-used as necessary.

Some discharge into the river of surplus treated water (subject to permitting) may be required. A conceptual water balance has been prepared and is presented in Section 17.4.6.

20.3.3 Biodiversity and Protected Areas

The Lubuskie Province, and in particular the area around Zielona Góra, has a rich variety of flora and fauna due to the extensive forest cover and network of rivers and lakes. These habitats support mammals such as deer and wild boar, as well as extensive birdlife, including migratory birds.

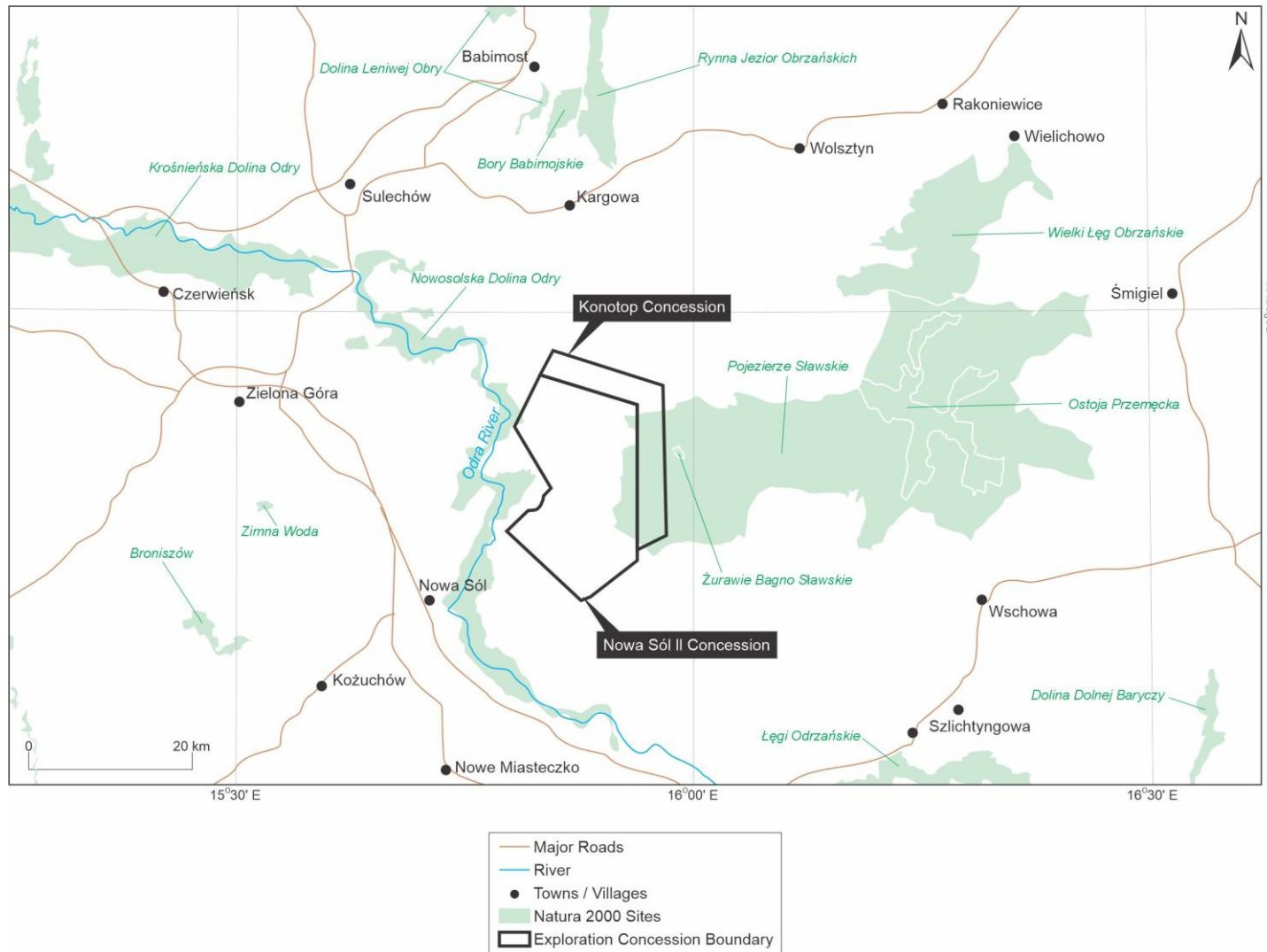
There are no protected areas directly within the proposed surface footprint of the Project. The exploration concession is situated in between the Sławskie Lake District to the east and the Odra River valley to the west, both of which are protected areas under the EU Natura 2000 network.

Protected Areas designated under the EU Natura 2000 Network are shown in Figure 20.4 and include:

- **Nowa Sól Odra Valley** Special Area of Conservation (SAC) – part of the Odra River valley designated under the EU Habitats Directive (PLH080014 *Nowosolska Dolina Odry*). This SAC covers an area of >6,000 ha and hosts nine protected species and ten protected habitats; it is located to the southwest of the Project. The Middle Odra Valley is also designated as an Important Bird Area (IBA) for seven species including *Milvus milvus* (Red Kite), *Crex crex* (Corncrake), and *Cygnus cygnus* (Whooper Swan).
- **Sławskie Lake District** Special Protected Area for Birds (SPA) - designated under the EU Birds Directive (PLB300011 *Pojezierze Sławskie*). This SPA covers an area of >39,000 ha and hosts ten protected species; it is located east of the Project. The borders overlap with the edge of the exploration concession, however no infrastructure is planned in the protected zone. The Sławskie Lake District is also designated as an IBA which supports breeding pairs of *Ixobrychus minutus* (Little Bittern).
- **Ostoja Przemyńska** Special Area of Conservation (SAC) - designated under the EU Habitats Directive (PLH300041). This SAC overlaps part of the Przemyński Landscape Park (*Przemyński Park Krajobrazowy*) within the Sławskie Lake District and covers an area of >4,000 ha; it hosts nine protected species and fifteen protected habitats.
- **Sławskie Crane Swamp** Special Area of Conservation (SAC) - designated under the EU Habitats Directive (PLH 080047 *Żurawie Bagno Sławskie*). This SAC is also within the Sławskie Lake District and covers an area of >41ha; it hosts one protected species and one protected habitat.
- **The Great Obrzański Meadow** Special Protected Area for Birds (SPA) - designated under the EU Birds Directive (PLB300004 *Wielki Łęg Obrzański*). This SPA covers an area of >7,500 ha and hosts two protected species; it is located northeast of the Project, adjacent to the Sławskie Lake District.
- **The Obra Lakes Channel** Special Area of Conservation (SAC) – designated under the EU Habitats Directive (PLH080002 *Rynna Jezior Obrzańskich*). This SAC covers an area of >15,000 ha and hosts six protected species and nine protected habitats; it is located north of the Project.
- **Babimost Forest** Special Area of Conservation (SAC) - designated under the EU Habitats Directive (PLH080063 *Bory Babimojskie*). This SAC covers an area of >600 ha and hosts one protected habitat; it is located north of the Project.
- **The Leniwa Obra Valley** Special Area of Conservation (SAC) – designated under the EU Habitats Directive (PLH080001 *Dolina Leniwej Obry*). This SAC covers an area of >7,000 ha

and hosts six protected species and eleven protected habitats; it is located north of the Project.

Figure 20.4: EU Natura 2000 Sites Surrounding the Project



Source: Micon, 2026.

In addition to Natura 2000 sites, there are a number of other protected areas close to the Project, as designated by Polish legislation. These sites include:

- **Nature Reserves:** Lake Święte and Lake Mesze Nature Reserves are located on the eastern boundary of the exploration concession and are within the Sławskie Lake District SPA.
- **Protected Landscape Areas:** Pojezierze Sławsko – Przemęckie passes through the northern zone of the concession and Rynny Obrzycko-Obrzańskie passes along the eastern concession boundary.
- **Nature and Landscape Complexes:** The Gaj Wandy forest site, located in the southeast of the concession, close to the Sławskie Lake District SPA.
- **Ecological Land:** The Uroczysko Zacisze area is located near Konotop in the northern zone of the concession and Pośród Sosen is located on the northern concession boundary.
- **Natural Monuments:** Several natural monuments have been designated within the concession, mostly protected trees located in residential areas and near watercourses.

The Warta National Park (*Park Narodowy Ujście Warty*), one of Poland's 23 national parks, is located in the Lubuskie Province, approximately 100 km northwest of the Project. This is a wetland area at the confluence of the Warta and Odra Rivers by the border with Germany (85 km away).

A detailed inventory of local, regional and internationally designated protected areas and sites of conservation interest will be included in the future baseline studies as part of the EIA. Designated protected sites such as nature reserves will be avoided as the Project design is refined.

20.3.4 Socio-Economic Setting

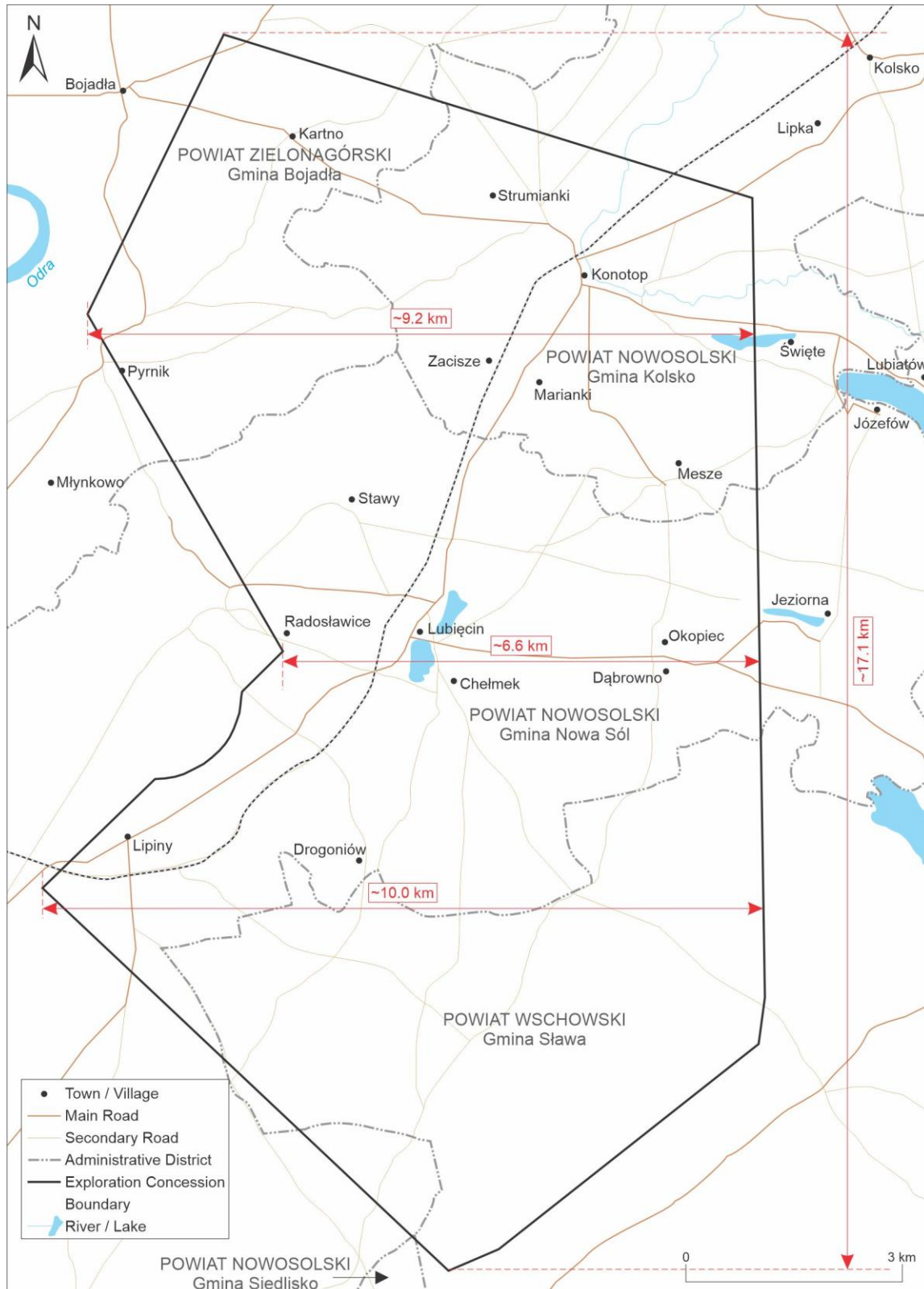
The Nowa Sól II exploration concession spans five administrative municipalities (communes / *gmina*) within three larger administrative districts (counties/*powiat*): Nowa Sól, Kolsko and Siedlisko municipalities, all within Nowa Sól county (*Powiat Nowosolski*), Bojadła municipality within Zielona Góra county (*Powiat Zielonagórski*) and Sława municipality within Wschowa county (*Powiat Wschowski*). Figure 20.5 illustrates these administrative boundaries.

The concession is centered around the village of Lubięcín and includes the villages of Konotop to the north and Lipiny to the south. Smaller settlements within the concession include Chełmek, Radosławice, Stawy, Marianki, Mesze, Zacisze, Okopiec, Pyrnik and Dąbrowno (Figure 20.5).

The nearest urban areas to the Project are the town of Nowa Sól, approximately 6 km southwest of the concession boundary, and the city of Zielóna Góra, approximately 25 km to the west.

The closest copper mine site to the Project is Rudna copper mine, operated by KGHM and located approximately 40 km to the southeast of the Nowa Sól concession. The Głógów copper smelter, also operated by KGHM, is approximately 25 km southeast. Industrial activities closer (<10 km) to the Project site include Bojadła Quarry (northwest of the concession) and Grochowice Gas Field (southeast of the concession).

Figure 20.5: Administrative Boundaries Within the Nowa Sól II Concession



Source: Micon, 2026.

20.3.5 Cultural Heritage

Archaeological surveys have not yet been undertaken at the Project site. The Polish National Institute of Cultural Heritage (*Narodowy Instytut Dziedzictwa/NID*) lists several buildings and monuments associated with villages in the exploration concession. These include, but are not limited to:

- Konotop: Church of St Anne and adjacent cemetery.
- Lubięcin: A traditional windmill dating from 1705 and the wooden church of the Sacred Heart of Jesus.
- Chełmek: Fifteenth century rural layout of the village and manor house complex.
- Lipiny: Railway station, cemetery and thirteenth century rural layout of the village.

There are no United Nations Educational, Scientific and Cultural Organization (UNESCO) World Heritage Sites in the vicinity of the Project; the closest designated site is the landscaped gardens of Muskauer Park (*Park Mużakowski*) on the German border (85 km away). A full review of archaeology and cultural heritage will be undertaken as part of the EIA process.

20.4 STAKEHOLDER ENGAGEMENT

Lumina has a proactive approach to stakeholder engagement and has undertaken preliminary stakeholder mapping for the Project. They have engaged a local community relations officer and the management team have established a good rapport with local government representatives and technical institutions. Numerous informal discussions have also been held with local residents.

As the Project advances, it will be important for Lumina to continue to maintain records of all stakeholder interactions, including with local residents and members of the public. It will also be important to manage community expectations and ensure that the concept of exploration activity is understood and consistent messaging is given, especially during informal/unplanned discussions.

20.5 ENVIRONMENTAL AND SOCIAL RISKS AND IMPACTS

The Project will be designed to minimize environmental impacts as far as possible and enhance socio-economic opportunities. The Nowa Sól II Concession is located on forestry land that is largely undisturbed, however some exploration activity (drilling) has been undertaken and commercial forestry and agricultural enterprises exist in the immediate surrounding area. The wider region supports other mining projects and therefore consideration of cumulative impacts will be important, particularly on water resources and biodiversity. It should be noted that the surface footprints for the proposed north and south mine sites occupy a much smaller area than the current exploration concession.

A full review of the potential environmental and social impacts will be undertaken as part of the future EIA process. Based on the current conceptual Project design, geographical location, and an understanding of metal mining operations in similar environments, the potential environmental and social risks have been identified as follows:

- **Disturbance from Air Quality, Noise, Vibration and Artificial Lighting** – The Project will generate greenhouse gas emissions, dust, noise, vibration and artificial light from routine operational activities. This has the potential to cause disturbance to local residents and wildlife and will need to be monitored. Underground mining will help to minimise this disturbance. A combination of engineering controls, operating procedures and management practices will be used to mitigate these impacts.

- **Habitat and Wildlife Loss** – The Project will directly impact flora and fauna due to the need for habitat removal for Project construction (land clearance). Consideration will be given to the bird nesting season when planning groundworks/construction and wildlife monitoring will be implemented. Bird deterrents will be installed for higher risk infrastructure, such as overhead power lines and process water ponds.
- **Changes to Water Quality, River Dynamics and Groundwater Conditions** – Project operations have the potential to impact groundwater levels in surrounding aquifer zones through pumping of water from the mining area. Further hydrogeology studies, water quality testing and review of potential water treatment options will be necessary to advance the Project, particularly given the anticipated high salt content. Surface and groundwater quality may be impacted by sediment from stormwater runoff/drainage, potential seepage from waste material and accidental spills/leaks. A combination of engineering controls, standard operating procedures and management plans will be implemented to mitigate risks to water resources.
- **Traffic Management** – The Project will generate increased traffic through villages within the concession. Trucks and other heavy equipment will need to pass through/near residential areas and this poses potential health and safety risks to the local community. Options for directing project-related traffic away from residential areas will be examined as well as the potential for rail and river transport. A traffic management plan will be developed and an emergency response procedure implemented, with provision included for accidents and environmental incidents e.g. spillages of reagents/concentrate along all transport routes associated with the Project.
- **Permitting Risks**– The Polish permitting process is complex and this is the first time a foreign company is proposing the development of a completely new, standalone mining and processing project in this region of Poland. There are restrictions associated with the permitting timeline due to the five-year priority and exclusivity rights. Careful schedule management will therefore be needed. It will also be important to ensure that a large enough area of impact/influence is considered for the environmental and social impact assessment process, to take account of cumulative impacts associated with other mining operations and risks to downstream/downgradient water quality and nearby Protected Areas.
- **Socio-Economic Benefits** – The Project is anticipated to have a positive impact on the local and regional economy, through creation of direct and indirect jobs and associated training opportunities. There is no need for community resettlement.

At this stage of the Project, most of the potential environmental and social issues are considered typical of similar mining operations. The most significant risks are likely to be the complexity of the Polish permitting process, and the currently limited understanding of groundwater movement in relation to the deep mine shaft sinking that is proposed, especially given the known salt horizons in the geology.

Any negative impacts resulting from the Project can be managed appropriately, provided that:

- Sufficient technical studies are undertaken to understand hydrological and hydrogeological conditions.
- The environmental and social impact assessment, and associated baseline studies, are undertaken over an appropriately sized area of potential influence.
- The Project Design incorporates sufficient environmental protection measures.

20.6 PROJECT CLOSURE PLANNING

A preliminary rehabilitation and closure plan for the Project has not yet been developed, and is not yet required.

Polish Mining Law states that where a mining operation is decommissioned, in whole or in part, the owner shall:

- Secure or abandon mine workings, and secure or decommission equipment, installations, and facilities of the processing plant.
- Undertake necessary measures to protect any adjacent mineral deposits.
- Take requisite measures to protect the workings of neighbouring mining operations.
- Take requisite measures to protect the environment and reclaim land after concluding mining activities.
- Promptly after the decommissioning of a mining operation, the owner will provide the President of the State Mining Authority with a survey and geological study.

Polish Mining Law also requires the establishment of a decommissioning fund for financial assurance of closure costs. The fund must be held in a separate bank account as cash or alternative guarantee such as treasury bills, bonds issued or guaranteed by the State Treasury, or appropriate insurance cover. The fund must be added to on an annual basis, and for underground mining must cover at least 3% of the depreciation value of fixed assets associated with the Project.

A preliminary total estimate (North and South sites) of US\$283 million has been budgeted for rehabilitation and closure, including the accumulated annual payments to the decommissioning fund (refer to Section 21).

Lumina intends to undertake progressive rehabilitation where possible and will follow appropriate national and international guidelines.

20.7 RECOMMENDATIONS

The EIA process for the Project has not yet commenced, and therefore specific recommendations will arise as a result of this process and the associated public consultation, in addition to any terms and conditions outlined by the regulatory authorities.

Recommendations that are considered important for ongoing development of the Project into the PFS stage and beyond include the following:

- **Baseline Data Collection** - Undertake baseline environmental and social surveys within the Project's potential area of impact/influence. Surveys should include primary and secondary data collection over a minimum 12-month period to account for seasonal changes and to establish existing conditions. Particular attention should be given to biodiversity and water quality, given the known water bodies, proximity to designated conservation areas and the influence of other mining operations in the region.
- **Hydrology and Hydrogeology Studies** - Undertake detailed technical studies to understand surface water and groundwater conditions, including water balance and water quality, in and around the proposed Project site. As well as helping to inform the engineering design for the Project, these studies will assist with evaluation of water supply options and potential environmental impacts.

- **Stakeholder Engagement** – Continue to maintain a register of all formal and informal communications about the Project, with particular attention on the local community/general public.
- **Community Relations** - Establish a non-technical Project factsheet for general reference and use during formal and informal meetings, including community discussions. This will ensure that consistent messaging is provided to local residents and business owners, and should help manage community expectations, particularly regarding potential future employment opportunities.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL EXPENDITURE

21.1.1 Initial Capital Expenditure

Table 21.1 presents a summary of the estimated initial capital expenditures required to bring the Project into production. The estimate has been compiled by the QP from information provided by other authors (QP's) involved in this PEA. The bases of the estimate for the principal components of the estimate are given in the following sections. The estimate is expressed in United States dollars (US\$) as of January 2026.

Table 21.1: Initial Capital Cost Summary

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Mining	2,148.3	2,159.7	4,308.0
Surface Infrastructure	83.0	82.4	165.4
Process Plant and Infrastructure	248.9	248.9	497.7
Backfill/Paste Plant	64.4	64.4	128.8
Owner's Costs	15.0	15.0	30.0
Contingency	636.4	639.1	1,275.5
Grand Total	3,195.9	3,209.5	6,405.4

21.1.2 Mining Capital Cost

Table 21.2 summarises the mining capital cost estimate. The estimate includes the cost of sinking three (production, ventilation and service) shafts at each of the North and South Complexes, mining-related surface and underground static equipment and haulage infrastructure, plus the upfront driving of development haulage and access ways.

Information prepared by DMC Mining Services Ltd (DMC) for the estimate cost of shaft sinking, refrigeration and ventilation, has been reviewed by Micon and included with Micon's estimated costs for lateral development, mining equipment and underground infrastructure. The balance of the estimate comprises provisional amounts estimated by Micon based on benchmarking against similar works on other projects.

Table 21.2 Initial Capital Cost – Mining

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Shafts - Surface Facilities	217.6	218.0	435.5
Shafts - Production Shaft	544.9	529.3	1,074.2
Shafts - Service Shaft	625.7	610.3	1,235.9
Shafts - Ventilation Shaft	366.4	380.5	747.0
Lateral Development Mined	18.2	41.7	59.8
Conveyors	4.9	9.3	14.2
U/G Equipment	170.4	170.4	340.8
U/G Infrastructure	133.6	133.6	267.2
U/G Ventilation & Other	66.7	66.7	133.3
Grand Total	2,148.3	2,159.7	4,308.0

21.1.3 Surface Infrastructure Costs

The Infrastructure costs are split into five main areas, each for the North and South sites:

- Civils / Structural / Building Works - which covers Site preparation and clearance, the provision of buildings and facilities (i.e., car parking, admin, security, changehouse, repair facilities, process and backfill, etc.), plus the installation of roads, tracks, fencing and landscaping
- Provision of Utilities - connection and supply of HV to both North and South sites, emergency generators, substations, water, IT/comms and security requirements.
- Offsite Works - such as local road improvements, connections to national roads and barge loading facilities at Nowa Sól docks.
- Construction Preliminaries - including health & safety and quality control.
- Sundries - generally costs associated with site vehicles.
- Backfilling and Tailings costs - generally costs associated with the provision of DSTFs for the portion of tailings that cannot be accommodated underground as paste backfill.

Table 21.3: Initial Capital Cost – Surface Infrastructure

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Civils / Structural / Building Works	41.0	41.7	82.7
Provision of Utilities	20.2	20.7	40.9
Off Site Works	8.2	7.2	15.4
Construction Preliminaries	1.9	2.0	3.9
Sundries	0.4	0.4	0.7
DSTF	11.2	10.5	21.8
Total Infrastructure Capital Cost	83.0	82.4	165.4

The costs have been developed based upon current costs and the Micon database for such items.

21.1.4 Process Plant

The Initial Capital Costs for the Process Plant are summarized in Table 21.4. Main mechanical item requirements have been developed from the PDC document. Thereafter the equipment supply costs have been estimated based on current costs and the Micon database.

Support costs were then based upon different proportions of the total Mechanical cost.

Construction costs for mechanical installation, Electrical Instrumentation and Control (EI&C), piping and Heating, Ventilation, and Air Conditioning (HVAC) have been estimated as a proportion of the plant mechanical costs.

The costs for temporary facilities, such as labour camp, site facilities and all related supervision required to manage the works are not included above as they are included in the Infrastructure cost.

Indirect costs: Engineering service cost (i.e. Engineering, Procurement, and Construction Management (EPCM)) has been estimated based upon a proportion of the total direct cost (i.e. mechanical item costs + construction cost). It includes: Engineering, design and drafting services; Provision for travel costs; Construction engineering and execution support; Studies and other documents; Site investigation and other surveys (geotechnical, topographic, etc.); Official Project offprints, etc.

Table 21.4: Initial Capital Cost – Processing Plants

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Crushing and Screening	22.3	22.3	44.5
Milling	44.4	44.4	88.8
Flotation	30.5	30.5	61.0
Conc. Handling	7.3	7.3	14.5
Services and Facilities	13.0	13.0	26.0
Installation	42.3	42.3	84.5
Electrical & Instrumentation	24.6	24.6	49.3
Piping	9.4	9.4	18.8
EPCM	24.2	24.2	48.4
Other Indirects	29.1	29.1	58.1
First Fills	1.9	1.9	3.9
Total Process Plant Capital Costs	248.9	248.9	497.7

Note: Civils/structural costs have been included in the Infrastructure estimate (previous section) and are therefore not included within the process estimate.

21.1.5 Paste Backfill Plant

Micon have provided an estimate for construction (and operation) of a paste preparation and backfill process plant and distribution system for the underground mine as part of this PEA project.

The cost estimate is based on Q4 2025 costs.

Total capital cost has been derived from:

- Obtaining budget quotations from equipment manufacturers and vendors.
- Percentage factors for some direct and indirect costs.
- Micon cost database.

Based on the total plant mechanical costs (TPMC), allowances have been made as a percentage of TPMC for installation, EI&C, plus Piping and HVAC. Again, civils/structural costs have not been included within the backfill estimate, as they are part of the Infrastructure cost. It should also be noted that the infrastructure cost estimate includes CAPEX for setting up dry store tailings.

Indirect costs were estimated as a percentage of total direct costs.

Table 21.5 summarizes the estimate for the Paste Plant and Backfill Distribution System.

Table 21.5: Backfill Plant Cost Estimate

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Tailings Thickener	8.0	8.0	15.9
Reagent Addition	1.4	1.4	2.8
Filtration	17.9	17.9	35.7
Paste Mixer	1.4	1.4	2.8
Water Tanks, Pumps	4.5	4.5	8.9
Conveyors, etc.	1.2	1.2	2.4
Piping, Valves	13.0	13.0	26.1
Electrical & Instrumentation	5.5	5.5	11.0
EPCM	6.6	6.6	13.2
Other Indirects	5.0	5.0	10.0
Total Backfill Plant Capital Costs	64.4	64.4	128.8

21.1.6 Owner's Costs

Micon has estimated the Owner's Costs based on benchmarking against similar projects. The total estimated Owner's Cost is US\$30 million, split evenly between the two Shaft Complexes.

Table 21.6 summarizes the breakdown of the provision for Owner's Costs.

Table 21.6: Initial Capital Cost – Owner's Costs

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Labour	8.1	8.1	16.2
Insurance	2.4	2.4	4.8
Professional Services	1.2	1.2	2.4
Legal, Permitting	1.5	1.5	3.0
Property Tax	0.8	0.8	1.6
Misc. Other	1.0	1.0	2.0
Total Owner's Capital Costs	15.0	15.0	30.0

21.1.7 Contingency

Table 21.7 shows the contingency amounts that are included in each area of the Project, as described above. Total contingency amounts to approximately 25% of the estimated total construction cost.

Table 21.7: Initial Capital Cost – Contingency

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Mining	537.1	539.9	1,077.0
Infrastructure	20.7	20.6	41.3
Process Plant	62.2	62.2	124.4
Backfill/Paste Plant	16.1	16.1	32.2
Owner's Costs	0.3	0.3	0.5
Total Contingency	636.4	639.1	1,275.5

21.1.8 Sustaining Capital

Sustaining Capital costs incurred during the operating period of the Project include lateral development of the underground mine at each Shaft Complex, as well as expansion of the DSTF and routine rebuilding or replacement of equipment in the mining, processing and backfill areas.

Table 21.8 shows the provision by area for each Shaft Complex.

Table 21.8: Sustaining Capital Cost

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Mining	827.2	795.1	1,622.3
Infrastructure	90.8	88.3	179.1
Process Plant	246.4	239.2	485.6
Backfill/Paste Plant	63.8	61.9	125.7
Tailings	11.2	10.5	21.8
Closure Cost Net of Salvage	-94.9	-94.1	-189.0
Contingency	309.9	298.8	608.6
Total Sustaining Capital	1,454.4	1,399.7	2,854.1

21.2 OPERATING COSTS

21.2.1 North Shaft Complex

Table 21.9 summarizes the LoM operating costs for the North Shaft Complex, taking into account the by-product (silver) revenues.

Table 21.9: Operating Cost – LoM Total - North Shaft Complex

Area	Total (US\$ million)	Unit Cost (US\$/t Treated)	Unit Cost (US\$/lb Cu)
Mining Costs	10,231	39.70	1.88
UG Backfill Costs	614	2.38	0.11
Ore Processing Costs	4,382	17.00	0.80
G&A	1,547	6.00	0.28
Cash Operating Costs	16,774	65.08	3.07
Product Transport, etc.	987	3.83	0.18
Royalties	-	-	-
LESS By-Product Credits	(10,920)	(42.37)	(2.00)
Total Cash Costs	6,840	26.54	1.25

21.2.2 South Shaft Complex

Table 21.10 summarizes the LoM operating costs for the South Shaft Complex, taking into account the by-product (silver) revenues.

Table 21.10: Operating Cost – LoM Total - South Shaft Complex

Area	Total (US\$ million)	Unit Cost (US\$/t Treated)	Unit Cost (US\$/lb Cu)
Mining Costs	9,932	39.50	1.68
UG Backfill Costs	581	2.31	0.10
Ore Processing Costs	4,275	17.00	0.72
G&A	1,509	6.00	0.25
Cash Operating Costs	16,297	64.81	2.75
Product Transport, etc.	1,025	4.08	0.17
Royalties	-	-	-
LESS By-Product Credits	(7,538)	(29.98)	(1.27)
Total Cash Costs	9,784	38.91	1.65

21.2.3 Mining Costs

Underground mine operating costs are based on standard rates for drilling, blasting loading and hauling in each type (size) of development and from stopes, based on a benchmarked cost for similar operations of US\$41.00/t stoped. However, total milled includes development tonnage, for which the mining cost is capitalized, hence the average cost per tonne treated is slightly less than US\$41.00/t.

Provision is made for the owner's supervision and technical services manpower and associated costs for mine planning, grade control, and survey measure, etc. in G&A costs.

21.2.4 Processing Costs

Processing costs were estimated by Micon, comprising consumables, labour, spares and maintenance, and waste management.

Table 21.11 gives a breakdown of the annual and unit costs for processing at each Shaft Complex.

Table 21.11: Operating Cost – Processing

Area	Annual Total (US\$'000)	Treated (US\$/t)
Process Management and Administrative	454	0.04
Plant Operations	2,268	0.20
Plant Maintenance	2,803	0.24
Chemical Laboratory	540	0.05
Operating Supplies	40,221	3.49
Surface Tailings Management	7,225	0.63
Maintenance Supplies	5,300	0.46
Electrical Power	56,537	4.91
Contingency	80,493	6.99
Process Operating Costs	195,840	17.00

21.2.5 General and Administrative Costs

G&A costs were estimated by Micon, with a forecast annual total of US\$69.1 million comprising labour (US\$22.3 million), and US\$46.8 million per year of fixed costs (Table 21.12).

Table 21.12: Operating Cost – G&A

Area	Annual Total (US\$'000)	Treated (US\$/t)
G&A Costs (Labour)	22,279	1.93
G&A Costs (Fixed)	46,854	4.07
G&A Operating Costs	69,133	6.00

22 ECONOMIC ANALYSIS

22.1 CAUTIONARY STATEMENT

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Information that is forward-looking includes:

- Mineral Resource and Mineral Reserve estimates.
- Assumed commodity prices and exchange rates.
- The proposed mine production plan.
- Projected mining and process recovery rates.
- Assumptions as to mining dilution.
- Capital and operating cost estimates and working capital requirements.
- Assumptions as to closure costs and closure requirements.
- Assumptions as to environmental, permitting and social considerations and risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed.
- Unrecognized environmental risks.
- Unanticipated reclamation expenses.
- Unexpected variations in quantity of mineralized material, grade or recovery rates.
- Geotechnical or hydrogeological considerations differing from what was assumed.
- Failure of mining methods to operate as anticipated.
- Failure of plant, equipment or processes to operate as anticipated.
- Changes in assumptions as to the availability and cost of electrical power, fuel, and process reagents.
- Ability to maintain the social license to operate.
- Accidents, labour disputes and other risks of the mining industry.
- Changes to interest rates.
- Changes to tax rates and availability of allowances for depreciation and amortization.

Furthermore, the preliminary economic assessment is preliminary in nature; it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

22.2 BASIS OF EVALUATION

Micon has prepared its assessment of the Project on the basis of a DCF model, from which pre- and after-tax NPV, Internal Rate of Return (IRR) and payback period can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to determine the economic viability of the Project. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of NPV, IRR and Payback to be made. The sensitivity of NPV to changes in the base case assumptions for price, operating costs and capital expenditure was then examined, as well as the sensitivity of NPV to the discount rate.

22.3 MACRO-ECONOMIC ASSUMPTIONS

22.3.1 Exchange Rate and Inflation

All results are expressed in United States dollars (US\$) except where stated otherwise. Cost estimates and other inputs to the cash flow model for the project have been prepared using constant, Q1 2026 money terms, i.e., without provision for escalation or inflation.

22.3.2 Weighted Average Cost of Capital

In order to find the NPV of the annual cash flows forecast for the Project, an appropriate discount factor must be applied which represents the Weighted Average Cost of Capital (WACC) imposed on the Project by the capital markets.

Determination of an optimal capital structure for the Project is a separate matter from establishing its fundamental technical and economic viability, which is best demonstrated by evaluation on an all-equity basis. That being the case, WACC is equal to the market cost of equity.

In this case, Micon has selected an annual discount rate of 7.0% in real terms for its base case, consistent with a risk-free rate of approximately 2.0% in real terms and a risk premium for equity of 5.0%, assuming a beta of 1.0. Micon has tested the sensitivity of the Project to changes in this rate.

22.3.3 Royalty and Taxation Regime

Polish corporate taxation has been calculated at a rate of 19% of profits before tax, after allowing for depreciation of capital assets over 10 years on a straight-line basis.

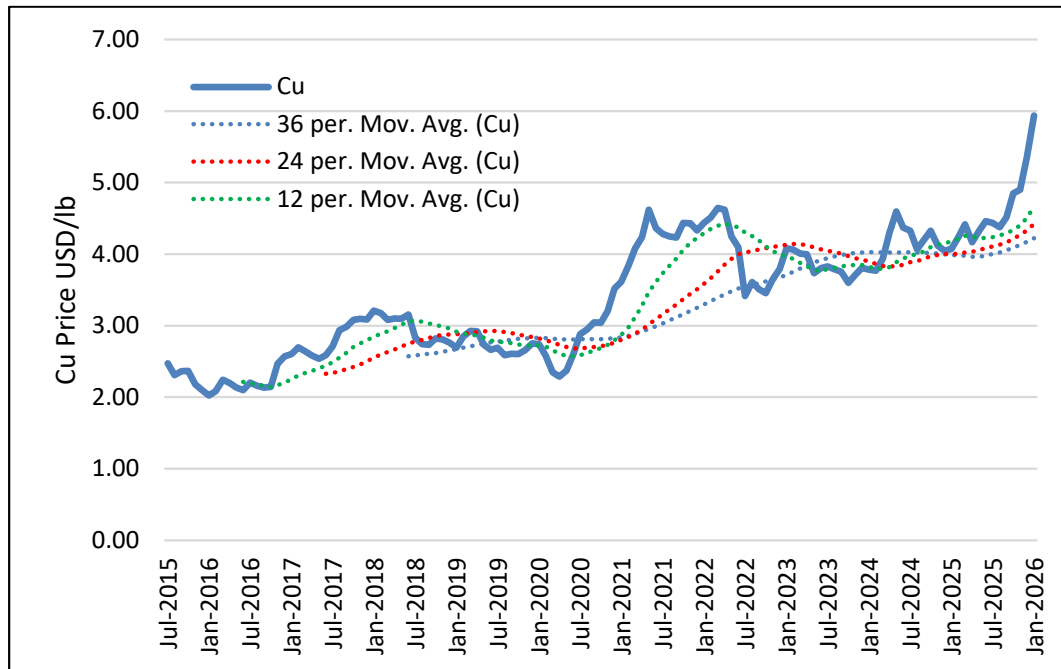
The impact of Polish Mineral Extraction Tax (MET) has been shown separately as it is anticipated that an exemption or amendment to the current legislation would be required for the Project to proceed. Nevertheless, if MET were applied as currently legislated, the Project would generate positive, though considerably less attractive, economic returns.

22.3.4 Expected Metal Prices

Project revenues will be generated from the sale of a copper concentrate with significant silver credits. Although some lead is present in the ore, it is not expected to be a payable metal in the final concentrate.

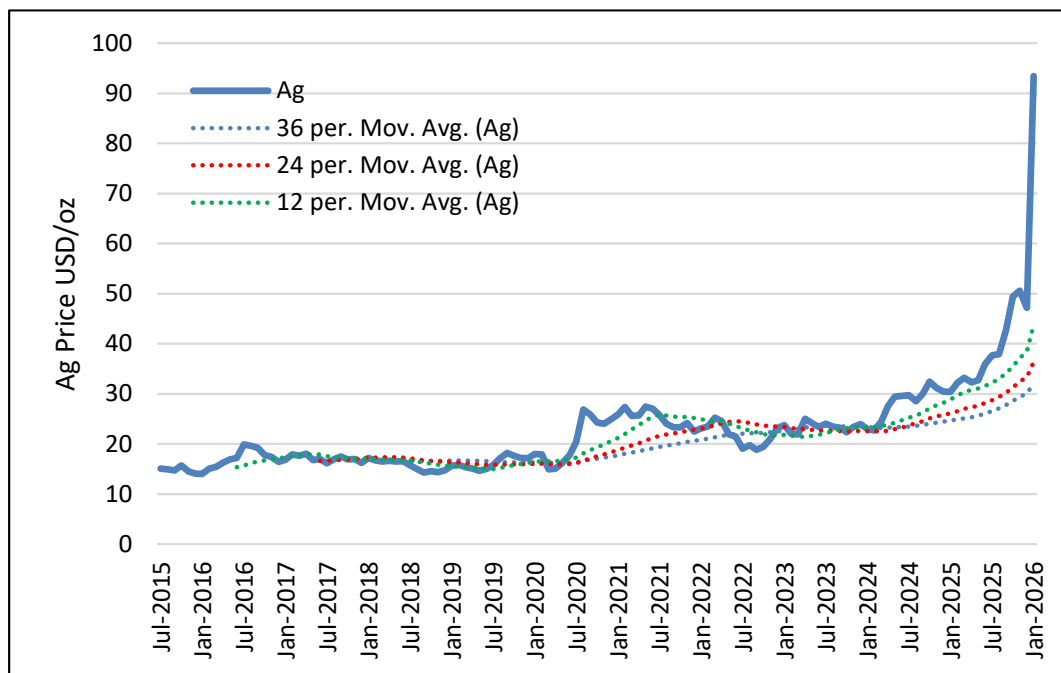
The spot market price of copper and silver have shown considerable growth over the past year, as shown in Figure 22.1 and Figure 22.2.

Figure 22.1: Copper Spot Price History



Source: LME.com, 2026.

Figure 22.2: Silver Spot Price History



Sources: Kitco.com and Umicore.com, 2026.

In January 2026, spot prices averaged around US\$5.94/lb Cu and over US\$90/oz Ag. At that time, the trailing 12-month averages were close to US\$4.66/lb Cu and US\$41.60/oz Ag, respectively.

22.4 TECHNICAL ASSUMPTIONS

The technical parameters, production forecasts and estimates described earlier in this Report are reflected in the base case cash flow model. These inputs to the model are summarized in the following sections.

22.4.1 Capital Costs

Initial capital costs are summarized in Table 22.1. The estimates for each area were prepared by the relevant QPs. Further detail is provided in Section 21 of this Report.

Table 22.1: Initial Capital Cost Summary

Area	North Shaft Complex (US\$ million)	South Shaft Complex (US\$ million)	Total (US\$ million)
Mining	2,148.3	2,159.7	4,308.0
Surface Infrastructure	83.0	82.4	165.4
Process Plant & Infrastructure	248.9	248.9	497.7
Backfill/Paste Plant	64.4	64.4	128.8
Owner's Costs	15.0	15.0	30.0
Contingency	636.4	639.1	1,275.5
Grand Total	3,195.9	3,209.5	6,405.4

22.4.2 Production Schedule

Figure 22.3 shows the annual tonnages of material mined from the North Shaft Complex, and the average annual grade of plant feed from each source, while Figure 22.4 shows the same for the South Shaft Complex.

Figure 22.3: Annual Tonnage Mined – North Shaft Complex

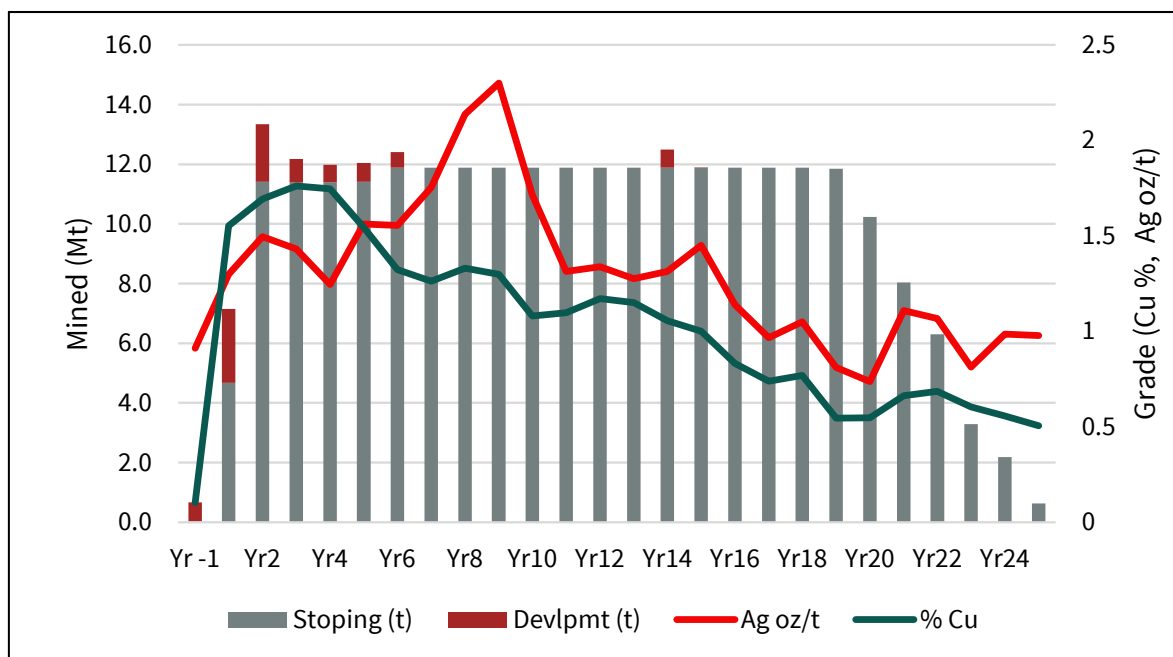
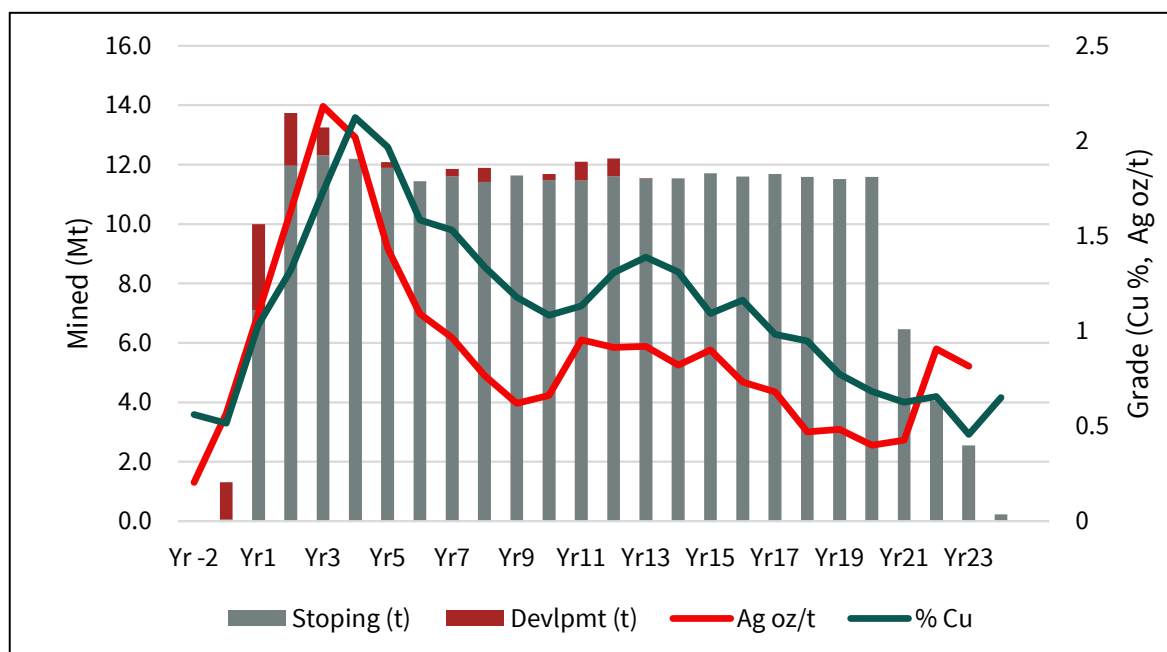


Figure 22.4: Annual Tonnage Mined – South Shaft Complex



At each plant, copper and silver recoveries of 85.8% and 92.0% are forecast for copper and silver, respectively, producing concentrates with a grade of 26% Cu with a moisture content of 8% by mass.

22.4.3 Sales Revenue

It is assumed that all concentrates are sold Delivered-at-Place (DAP) to a local smelter. Concentrate sales assume the parameters for transport, treatment and refining charges shown in Table 22.2.

Table 22.2: Transport, Treatment and Refining Charges

Area	Unit	Value
Minimum Deduction - Copper	%	1.0
Payability - Copper	%	96.5
Payability - Silver	%	96.0
Payability - Lead	%	nil
Concentrate Transport	US\$/wmt conc.	10.00
Concentrate Treatment	US\$/dmt conc.	50.00
Refining Cost - Copper	US\$/lb Cu	0.050
Refining Cost - Silver	US\$/oz Ag	0.40

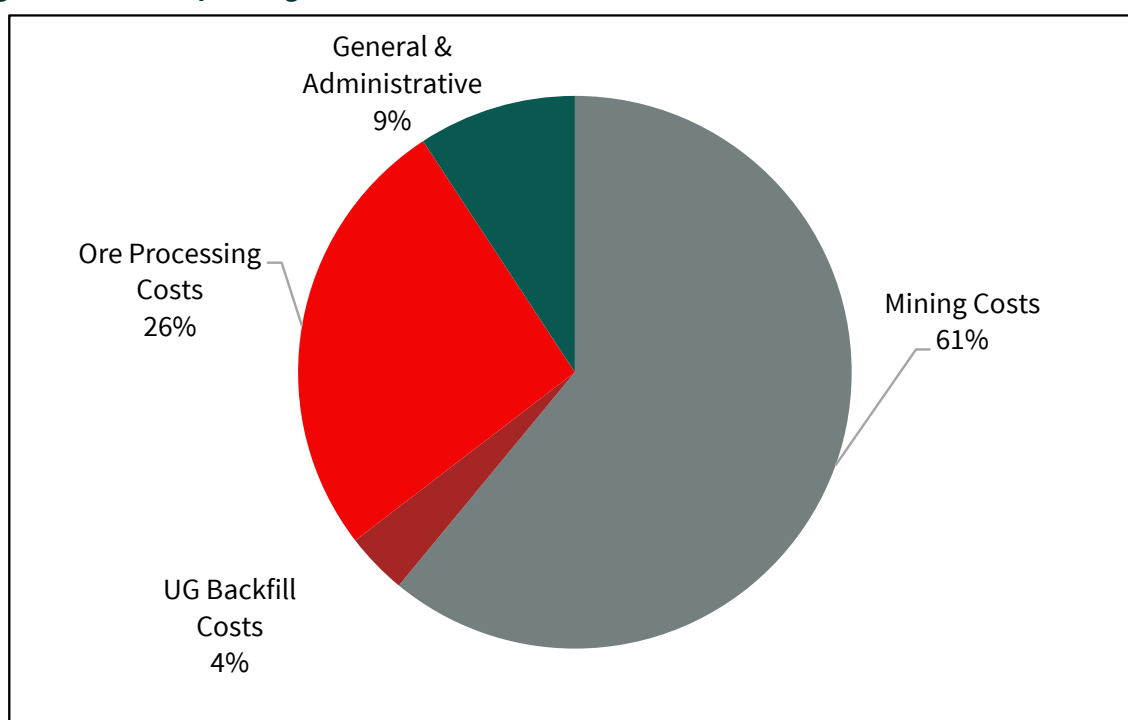
22.4.4 Operating Costs

Table 22.3 presents a summary of the LoM operating costs. Further details of these estimates are provided in Section 21 of this Report.

Table 22.3: Operating Cost Summary

Area	North Shaft (US\$ million)	South Shaft (US\$ million)	Combined (US\$ million)
Mining Costs	10,231	9,932	20,163
UG Backfill Costs	614	581	1,196
Ore Processing Costs	4,382	4,275	8,656
G&A	1,547	1,509	3,056
Cash Operating Costs	16,774	16,297	33,071
Product Transport, etc.	987	1,025	2,012
Royalties	-	-	-
LESS By-Product Credits	-10,920	-7,538	-18,458
Total Cash Costs	6,840	9,784	16,625

Figure 22.5 shows the split of on-site cash operating costs for the combined (North and South) operations. The breakdown for each Shaft Complex is approximately the same before by-product credits are applied.

Figure 22.5: Cash Operating Cost Breakdown


22.5 BASE CASE EVALUATION

Figure 22.6 presents a summary of the annual cash flows for the North Shaft Complex and Figure 22.7 presents the same information for the South Shaft Complex.

Figure 22.6: Annual Cash Flows – North Shaft Complex

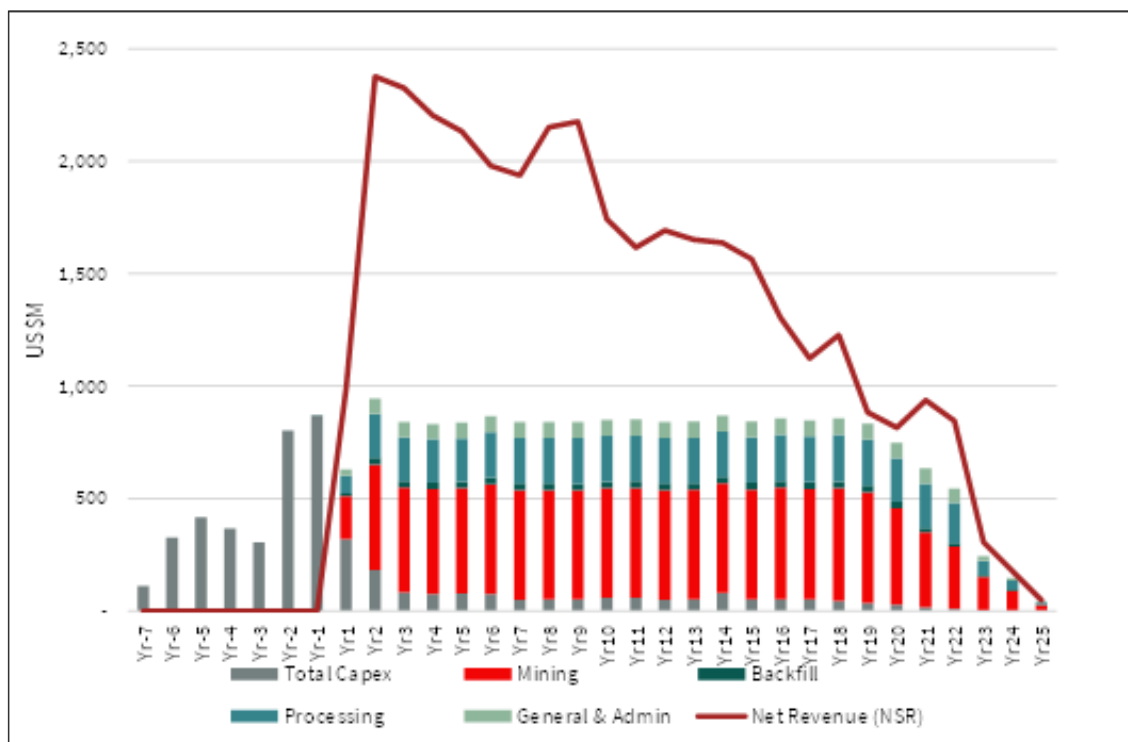


Figure 22.7: Annual Cash Flows – South Shaft Complex

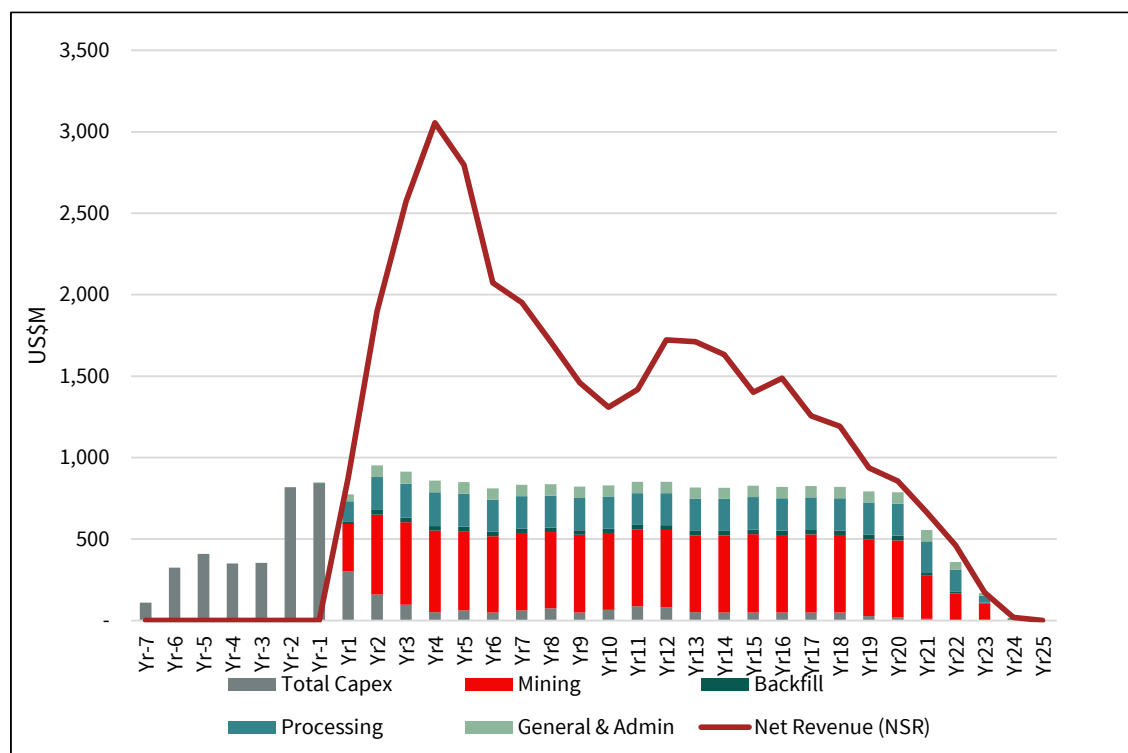


Figure 22.8 presents a summary of the annual net cash flow, the cumulative cash flow and the cumulative DCF for the North Shaft Complex and Figure 22.9 presents the same information for the South Shaft Complex.

Figure 22.8: Cumulative and Discounted Cash Flows – North Shaft Complex

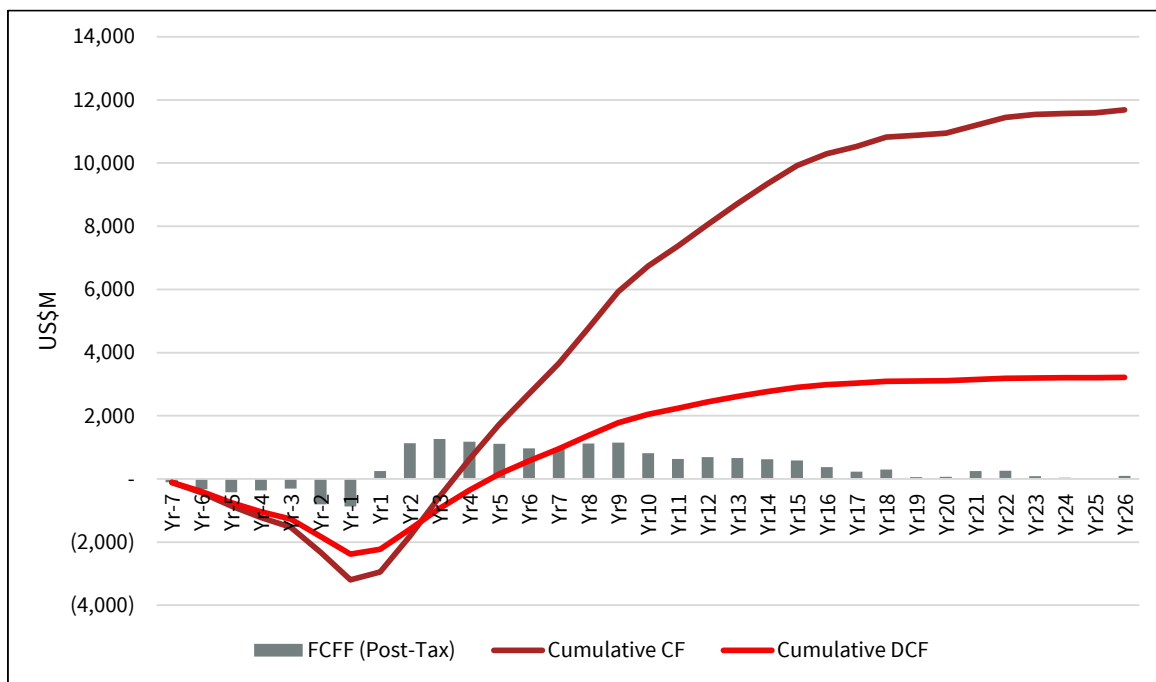


Figure 22.9: Cumulative and Discounted Cash Flows – South Shaft Complex

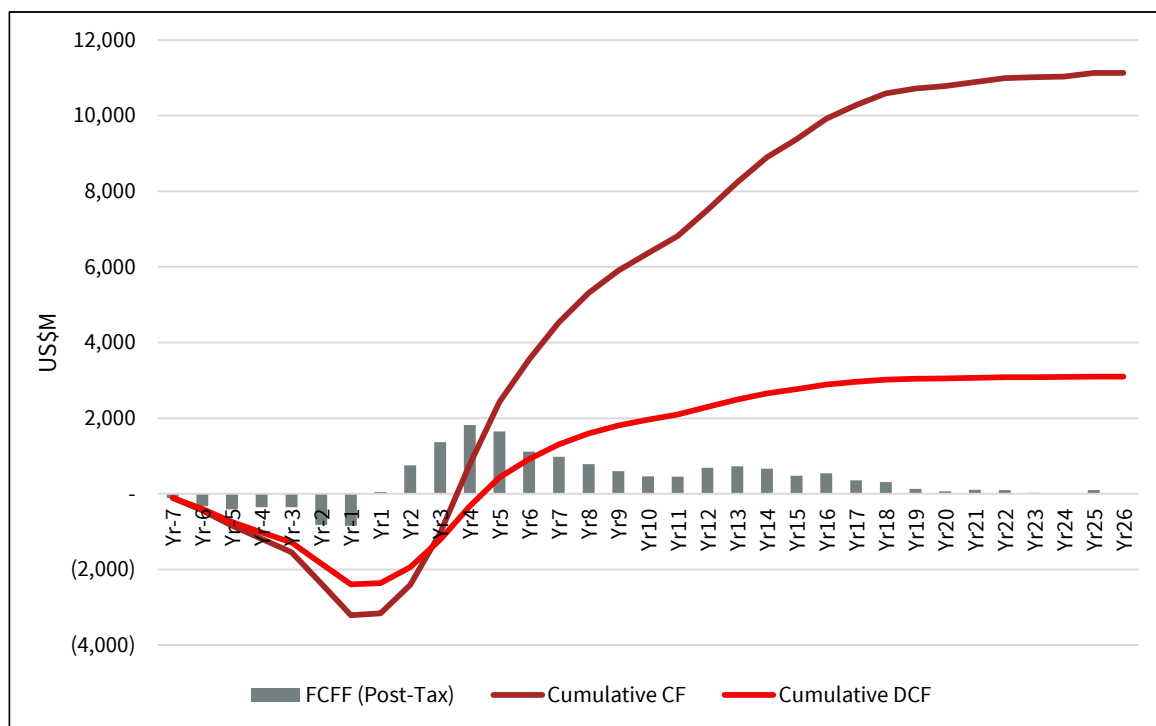


Table 22.4 summarizes the LoM production and cash flows and unit costs for the Project. This includes the average payable Cu, Ag and Cu_{eq} for the first 10 years.

It will be seen that both Shaft Complexes provide attractive economic returns, though the NPV₇, IRR and payback period for the North all exceed the corresponding parameters for the South.

Under the base case assumptions, on a pre-tax basis the consolidated NPV at a discount rate of 7.0% (NPV₇) is estimated at approximately US\$8.33 billion, with a pre-tax IRR of approximately 20.5%. Both the North Shaft Complex and the South Shaft Complex generate positive economic returns in the base case, with the North Shaft Complex demonstrating marginally higher NPVs, IRRs, and shorter payback periods relative to the South Shaft Complex.

After deducting Polish corporate income tax, applied at a rate of 19% after allowing for straight-line depreciation of capital assets over ten years, but excluding the Polish MET, the consolidated Project generates a post-corporate tax NPV₇ of approximately US\$6.32 billion, with a post-tax IRR of approximately 18.0% and an estimated payback period of approximately 3.5 years from the commencement of production. Excluding MET, the economic break-even copper price is \$2.82/lb.

The impact of Polish MET has been shown separately as it is anticipated that an exemption or amendment to the current legislation would be required for the Project to proceed. If MET were applied as currently legislated, the Project would nevertheless generate an NPV₇ of US\$1,60 billion and IRR of 10.8% which, whilst significantly reduced from the pre-MET results, still reflect a positive result under the base case assumptions of this PEA.

The economic results presented herein are forward-looking and are subject to a number of known and unknown risks and uncertainties, including but not limited to changes in commodity prices, capital and operating cost estimates, metallurgical performance, geotechnical and hydrogeological conditions, permitting outcomes, environmental and social considerations, taxation and royalty regimes, and the ability to obtain financing. These factors could cause actual results to differ materially from those presented in this assessment.

Table 22.5 presents the detailed annual cash flows for the North Shaft Complex and Table 22.6 presents the same information for the South Shaft Complex.

Table 22.4: LoM Production and Cashflow Summary

Parameter	Unit	North	South	Consolidation
Tonnes Milled	kt	257,739	251,449	509,188
Copper Grade	Cu %	1.12%	1.25%	1.19%
Silver Grade	g/t Ag	43.25	30.39	36.90
Payable Copper Production	Cu kt	2,474	2,686	5,160
Payable Silver Production	Ag koz	291,209	201,009	492,218
Payable Copper Equivalent Production	Cu _{eq} kt	3,517	3,406	6,923
Net Revenue	US\$ million	35,845	34,638	70,483
Operating Costs	US\$ million	16,774	16,297	33,071
EBITDA	US\$ million	19,071	18,341	37,412
Total Pre-Production Capex	US\$ million	3,196	3,210	6,405
Total Sustaining Capex)	US\$ million	1,454	1,400	2,854
Total Capex	US\$ million	4,650	4,609	9,260
Corporate Taxation*	US\$ million	2,732	2,602	5,334
Net Cash Flow before MET	US\$ million	11,689	11,130	22,819
MET	US\$ million	7,001	6,908	13,909
Net Cash Flow after MET	US\$ million	4,688	4,222	8,910
NPV (Pre-Tax) at 7% Disc.	US\$ million	4,233	4,093	8,326
IRR (Pre-Tax)	%	20.6%	20.4%	20.5%
Payback (pre-tax, years after production startup)	Yrs	3.0	3.3	3.2
NPV (After Corporate Income Tax*) at 7% Disc.	US\$ million	3,214	3,101	6,315
IRR (After Corporate Income Tax*)	%	18.1%	17.9%	18.0%
Payback* Years after Production Startup)	Yrs	3.5	3.6	3.5
NPV After Corporate Tax and MET at 7% Disc.	US\$ million	866	738	1,604
IRR After Corporate Tax and MET	%	11.1%	10.6%	10.8%
Payback (years after production startup)	Yrs	5.1	4.6	4.8
C₁ Cost Net of By-Product Credits	US\$/lb Cu	1.25	1.65	1.46
AISC Net of By-Product Credits	US\$/lb Cu	1.54	1.90	1.73
Break-Even Copper Price (NPV _T =0)	US\$/lb Cu	2.70	2.93	2.82
Average payable copper production (First 10 years)	kt Cu/a	140	150	290
Average payable silver production (First 10 years)	koz Ag/a	15,711	12,210	27,921
Avg. copper equivalent production (First 10 years)	kt Cu _{eq} /a	196	194	390
AISC net of by-product credits (First 10 years)	US\$/lb Cu	0.97	1.35	1.17
Average Annual EBITDA (First 10 years)	US\$ million/a	1,272	1,220	2,492
Capital intensity (US\$ per Cu _{eq} t/a) (First 10 years)	US\$/t/a	16,268	16,566	16,416
Net Cash flow before MET (First 10 years)	US\$ million/a	994	958	1,952
Net Cash flow after MET (First 10 years)	US\$ million/a	645	609	1,254
Avg. payable copper production (First 20 Years)	Cu kt/a	116	129	245
Average Payable Silver Production (20 Year)	Ag koz/a	13,635	9,767	23,402
Average Copper Equivalent Production (20 Year)	Cu _{eq} kt/a	165	164	328
Capital Intensity (US\$ per Cu _{eq} t/a) (20 Year)	US\$/t/a	19,427	19,593	19,509
NPV _T /Initial Capex (after corporate tax*)	US\$/US\$	1.0	1.0	1.0

* Excluding Copper and Silver Mineral Extraction Tax (MET). Note: AISC = All-In Sustaining Costs.

Table 22.5: Annual Production and Cash Flow Schedule – North Shaft Complex

[illegible]

Table 22.6: Annual Production and Cash Flow Schedule – South Shaft Complex

Mining, Processing & Production			SOUTH		Yr-7	Yr-6	Yr-5	Yr-4	Yr-3	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17	Yr18	Yr19	Yr20	Yr21	Yr22	Yr23	Yr24	Yr25	Yr26
Milling & Processing																																					
Plant Ore Feed Tonnes	kt		251,449	-	-	-	-	-	-	-	-	7,177	11,961	12,317	12,183	11,888	11,520	11,609	11,520	11,631	11,520	11,520	11,616	11,543	11,536	11,709	11,600	11,689	11,593	11,520	11,584	11,520	7,920	2,548	226	-	-
Copper Grade	%		1.25%	-	-	-	-	-	-	-	-	1.21%	1.42%	1.81%	2.12%	1.99%	1.58%	1.55%	1.37%	1.18%	1.09%	1.18%	1.36%	1.39%	1.31%	1.09%	1.16%	0.98%	0.95%	0.77%	0.68%	0.59%	0.60%	0.46%	0.65%	-	-
Silver Grade	g/t Ag		30.39	-	-	-	-	-	-	-	-	18.89	36.32	53.26	67.89	63.51	44.38	34.38	30.78	23.83	19.50	21.46	30.74	28.44	28.58	25.56	27.98	22.73	21.19	14.61	15.05	12.59	13.04	28.21	25.36	-	-
Lead Grade	%		0.05%	-	-	-	-	-	-	-	-	0.02%	0.04%	0.05%	0.06%	0.05%	0.05%	0.05%	0.04%	0.03%	0.04%	0.08%	0.06%	0.06%	0.06%	0.05%	0.06%	0.06%	0.08%	0.05%	0.06%	0.02%	0.01%	0.00%	0.00%	-	-
Payable Metal																																					
Copper	kt	28,126	2,686	-	-	-	-	-	-	-	-	74	145	190	221	202	155	154	135	117	107	116	135	137	129	109	115	98	94	76	67	58	40	10	1	-	-
Silver	koz		201,009	-	-	-	-	-	-	-	-	3,591	11,616	17,543	22,119	20,189	13,579	10,672	9,396	7,412	5,986	6,577	9,549	8,780	8,815	8,003	8,681	7,106	6,570	4,499	4,662	2,175	1,416	1,922	153	-	-
Lead	kt		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Copper Equivalent	kt CuEq		3,406	-	-	-	-	-	-	-	-	87	187	253	300	275	204	192	168	144	129	139	169	168	161	138	146	123	117	92	84	65	45	17	2	-	-
Financials					Yr-7	Yr-6	Yr-5	Yr-4	Yr-3	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17	Yr18	Yr19	Yr20	Yr21	Yr22	Yr23	Yr24	Yr25	Yr26
Revenue			5/tonne milled																																		
Copper	US\$M	79%	28,126	-	-	-	-	-	-	-	-	774	1,518	1,991	2,313	2,119	1,624	1,611	1,410	1,225	1,125	1,214	1,415	1,434	1,351	1,143	1,206	1,026	982	796	707	603	422	104	13	-	-
Silver	US\$M	21%	7,538	-	-	-	-	-	-	-	-	135	436	658	829	757	509	400	352	278	224	247	358	329	331	300	326	266	246	169	175	82	53	72	6	-	-
Lead	US\$M	0.00%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Gross Revenue	US\$M		141.83	35,664	-	-	-	-	-	-	-	908	1,954	2,649	3,142	2,876	2,133	2,011	1,762	1,503	1,350	1,461	1,773	1,763	1,681	1,443	1,531	1,293	1,229	965	882	684	475	176	19	-	-
Total Realisation and Freight Costs			4.08	1,025	-	-	-	-	-	-	-	27	56	74	87	79	60	58	51	44	40	43	51	52	49	42	44	37	36	29	26	21	15	4	1	-	-
Net Revenue (NSR)	US\$M		137.76	34,638	-	-	-	-	-	-	-	881	1,898	2,575	3,056	2,796	2,073	1,953	1,711	1,459	1,310	1,417	1,722	1,711	1,633	1,401	1,487	1,256	1,193	936	856	663	461	172	18	-	-
Operating Costs																																					
Mining	US\$M	39.50	9,932	-	-	-	-	-	-	3	292	490	505	500	487	469	476	468	477	471	470	476	473	473	473	480	476	479	475	472	475	265	166	104	9	-	-
Backfill	US\$M	2.31	581	-	-	-	-	-	-	0	17	29	28	28	28	28	28	27	28	28	28	28	28	28	28	28	28	28	29	29	15	9	5	0	-	-	
Processing	US\$M	17.00	4,275	-	-	-	-	-	-	-	122	203	209	207	202	196	197	196	198	196	196	197	196	196	199	197	199	197	196	197	196	135	43	4	-	-	
General & Admin	US\$M	6.00	1,509	-	-	-	-	-	-	-	43	72	74	73	71	69	70	69	70	69	70	69	69	70	69	69	70	70	70	69	70	69	48	15	1	-	-
Total Operating Costs	US\$M	64.81	16,297	-	-	-	-	-	-	3	474	794	816	808	789	762	770	761	773	764	763	771	766	766	778	770	776	770	766	770	545	358	168	15	-	-	
EBITDA	US\$M	72.94	18,341	-	-	-	-	-	-	-3	407	1,104	1,759	2,248	2,007	1,312	1,182	950	687	546	654	950	945	866	624	717	479	423	171	86	118	103	4	4	-	-	
Capital Costs & Annual Depreciation																																					
Total Pre-Production Capex	US\$M		3,210	110	325	409	351	354	819	841	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Sustaining Capex	US\$M		1,494	-	-	-	-	-	-	-	301	159	98	52	61	49	62	75	49	66	88	80	50	49	49	49	49	49	49	27	17	11	1	-	-	-	-
Total Capex	US\$M		4,703	110	325	409	351	354	819	841	301	159	98	52	61	49	62	75	49	66	88	80	50	49	49	49	49	49	27	17	11	1	-	-	-	-	
Annual depreciation			4,647	-	-	-	-	-	-	-	351	367	377	382	388	393	399	407	411	418	76	68	63	63	62	62	61	58	56	51	43	35	30	25	-	-	
EBIT (Operating Profit)			13,694	-	-	-	-	-	-	-3	56	737	1,382	1,866	1,619	919	783	543	275	128	578	882	882	803	562	655	418	365	115	35	75	67	-26	-22	-	-	
Corporate income tax	US\$M		2,602	-	-	-	-	-	-	-1	11	140	263	355	308	175	149	103	52	24	110	168	167	153	107	124	80	69	22	7	14	13	-5	-4	-	-	
Net Income	US\$M		11,092	-	-	-	-	-	-	-2	45	597	1,119	1,511	1,312	744	635	440	223	103	468	715	714	651	455	530	339	295	93	28	61	55	-21	-18	-	-	
Total Change in Working Capital	US\$M		BALANCED	-	-	-	-	-	-	0	48	50	28	20	(11)	(30)	(5)	(10)	(10)	(6)	4	13	(1)	(3)	(9)	3	(9)	(3)	(11)	(3)	(13)	(13)	(17)	(10)	(1)	-	
DCF Valuation					Yr-7	Yr-6	Yr-5	Yr-4	Yr-3	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17	Yr18	Yr19	Yr20	Yr21	Yr22	Yr23	Yr24	Yr25	Yr26
Free Cash Flow to the Firm (FCFF)																																					
EBIT		11,130				-	-	-	-	-3	56	737	1,382	1,866	1,619	919	783	543	275	128	578	882	882	803	562	655	418	365	115	35	75	67	-26	-22	-	-	
Adjustments:																																					
Plus: Depreciation			4,647	-	-	-	-	-	-	-	351	367	377	382	388	393	399	407	411	418	76	68	63	63	62	62	61	58	56	51	43	35	30	25	-	-	
Less: Capex			-4,703	(110)	(325)	(409)	(351)	(354)	(819)	(841)	(301)	(159)	(98)	(52)	(61)	(49)	(62)	(75)	(49)	(66)	(88)	(80)	(50)	(49)	(49)	(49)	(49)	(49)	(27)	(							

22.6 SENSITIVITY STUDY

22.6.1 North Shaft Complex

The sensitivity of the Project's NPV₇ and IRR to changes in prices, operating costs and capital costs were tested over a range of 40% above and below base case values. The results for the North Shaft Complex are presented in Figure 22.10 and Figure 22.11, respectively.

Figure 22.10: NPV Sensitivity - North Shaft Complex

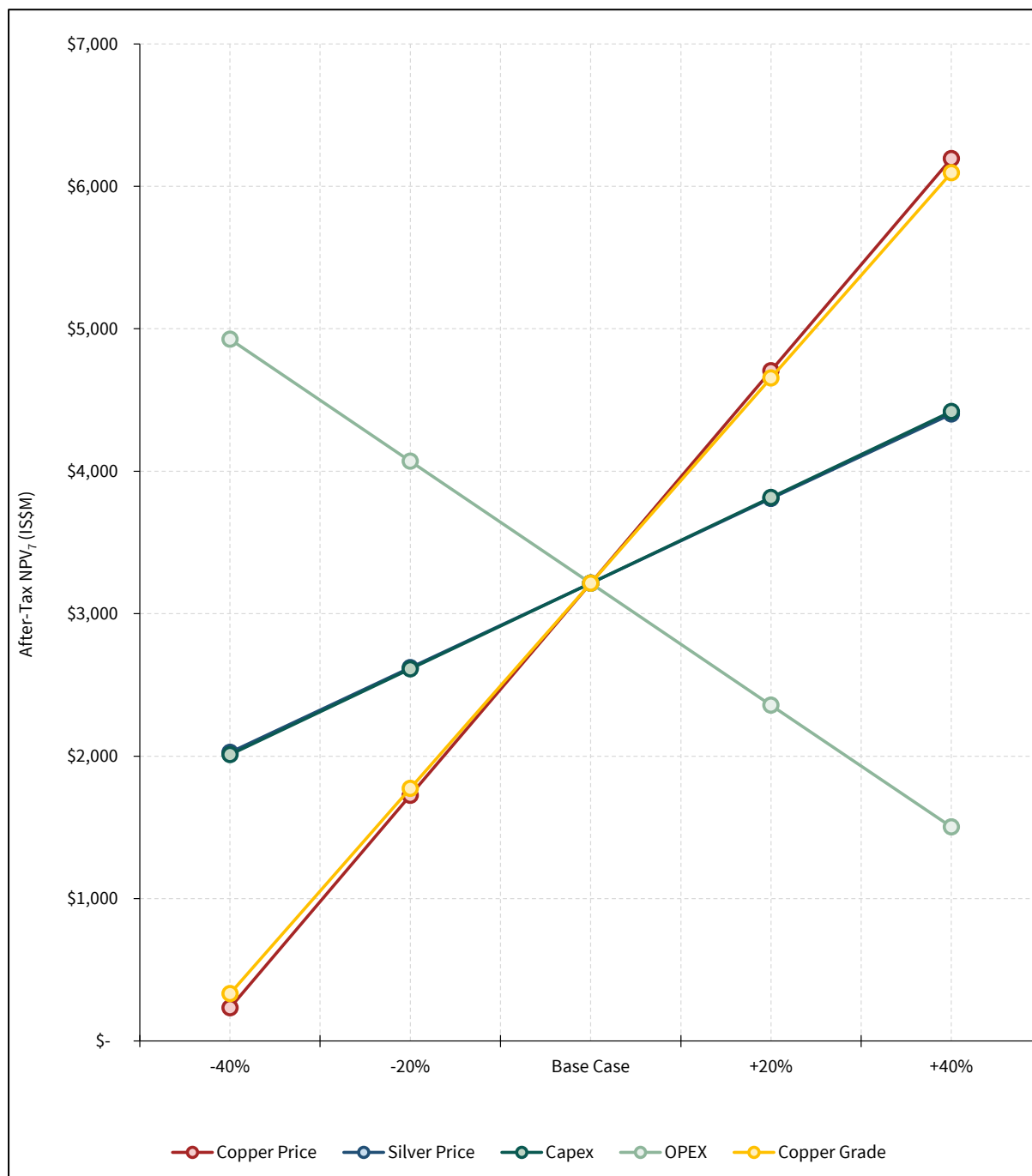
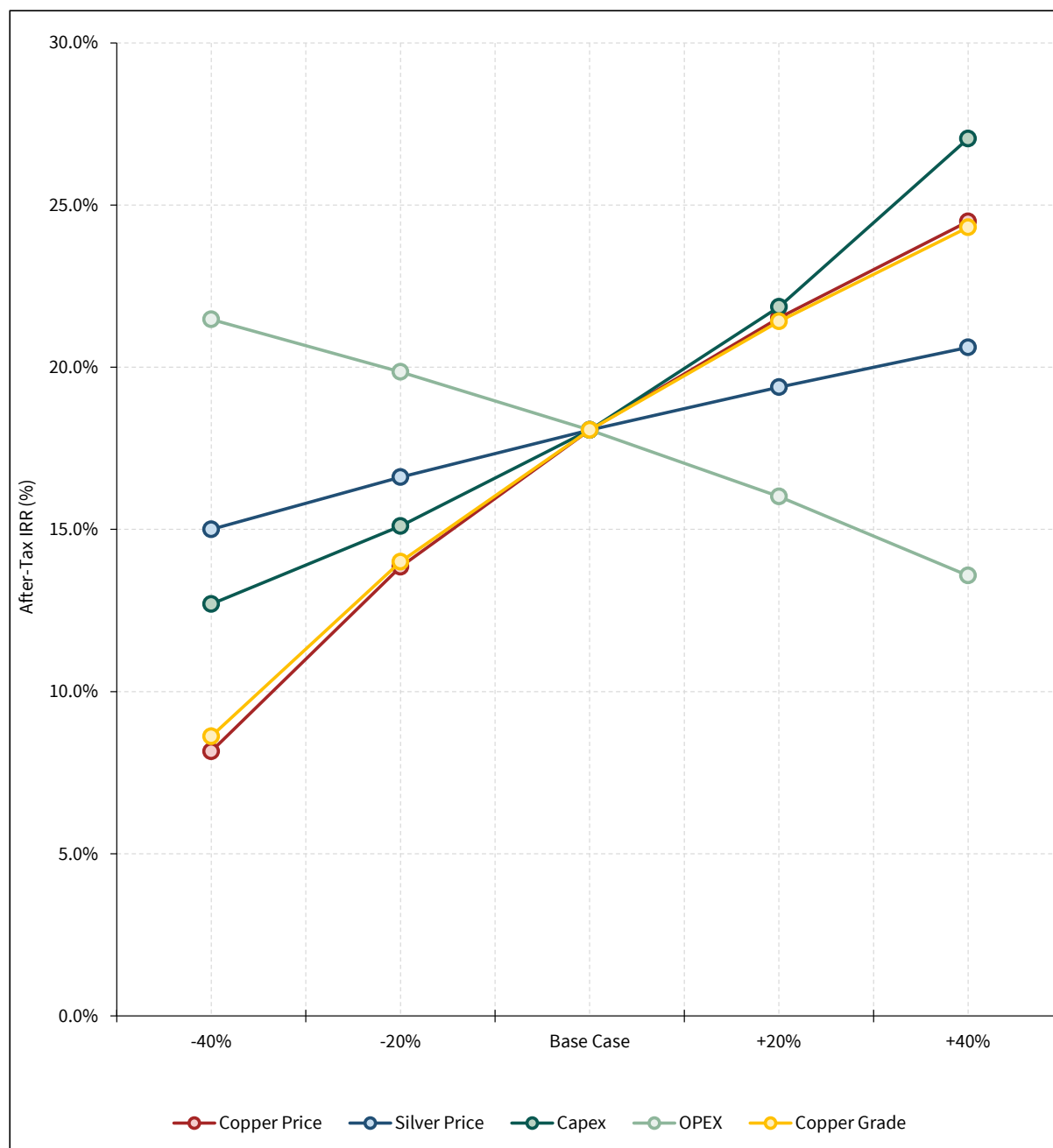


Figure 22.11: IRR Sensitivity - North Shaft Complex



As is typical, both NPV₇ and IRR are most sensitive to copper prices and grade or recovery. The silver price exhibits moderate sensitivity, and the Project is least sensitive to changes in OPEX and CAPEX. Thus, the Project demonstrates reasonable economics within the ranges tested.

22.6.2 South Shaft Complex

The sensitivity of the Project's NPV₇ and IRR to changes in prices, operating costs and capital costs were tested over a range of 40% above and below base case values. The results for the South Shaft Complex are presented in Figure 22.12 and Figure 22.13, respectively.

Figure 22.12: NPV Sensitivity - South Shaft Complex)

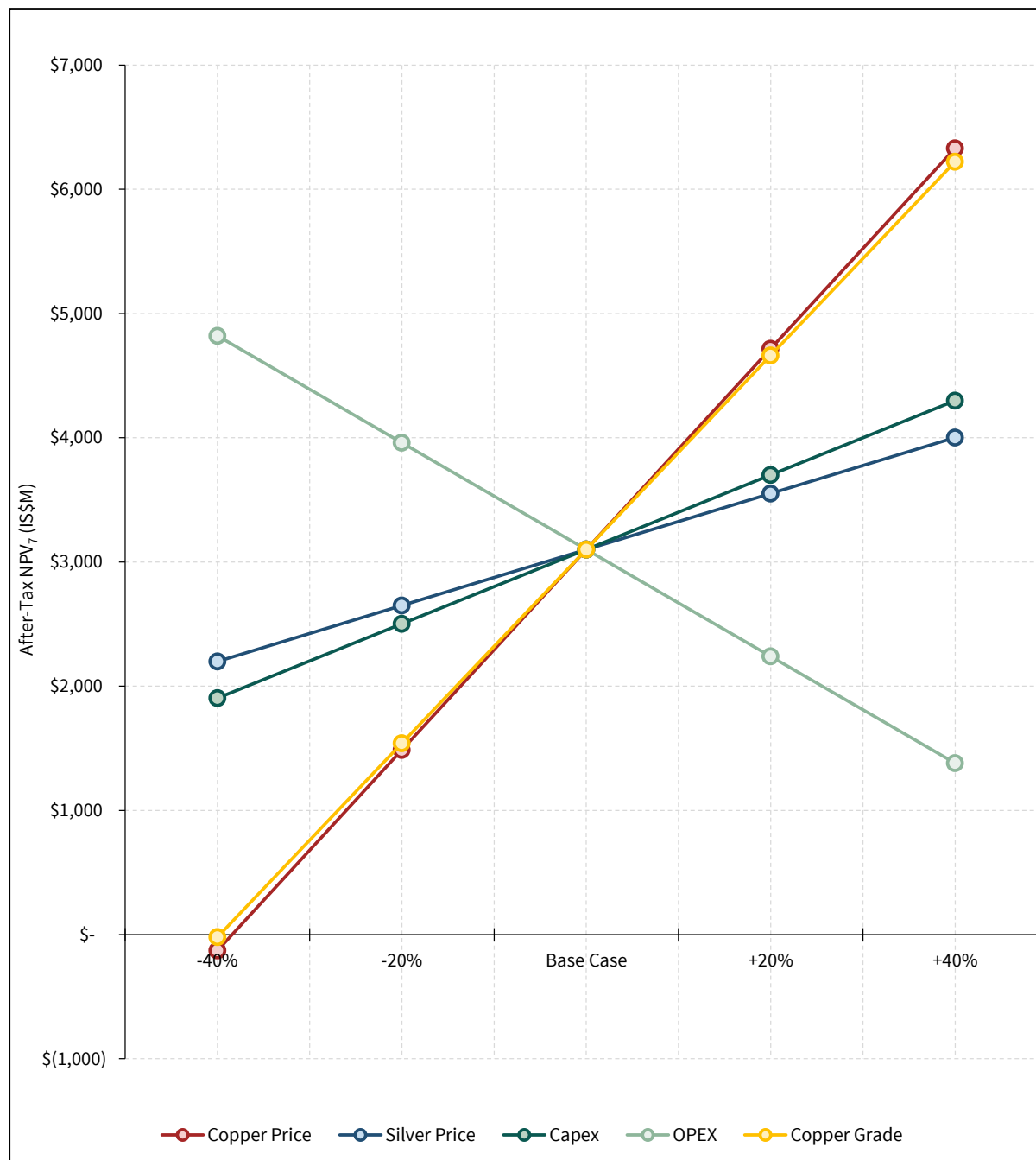
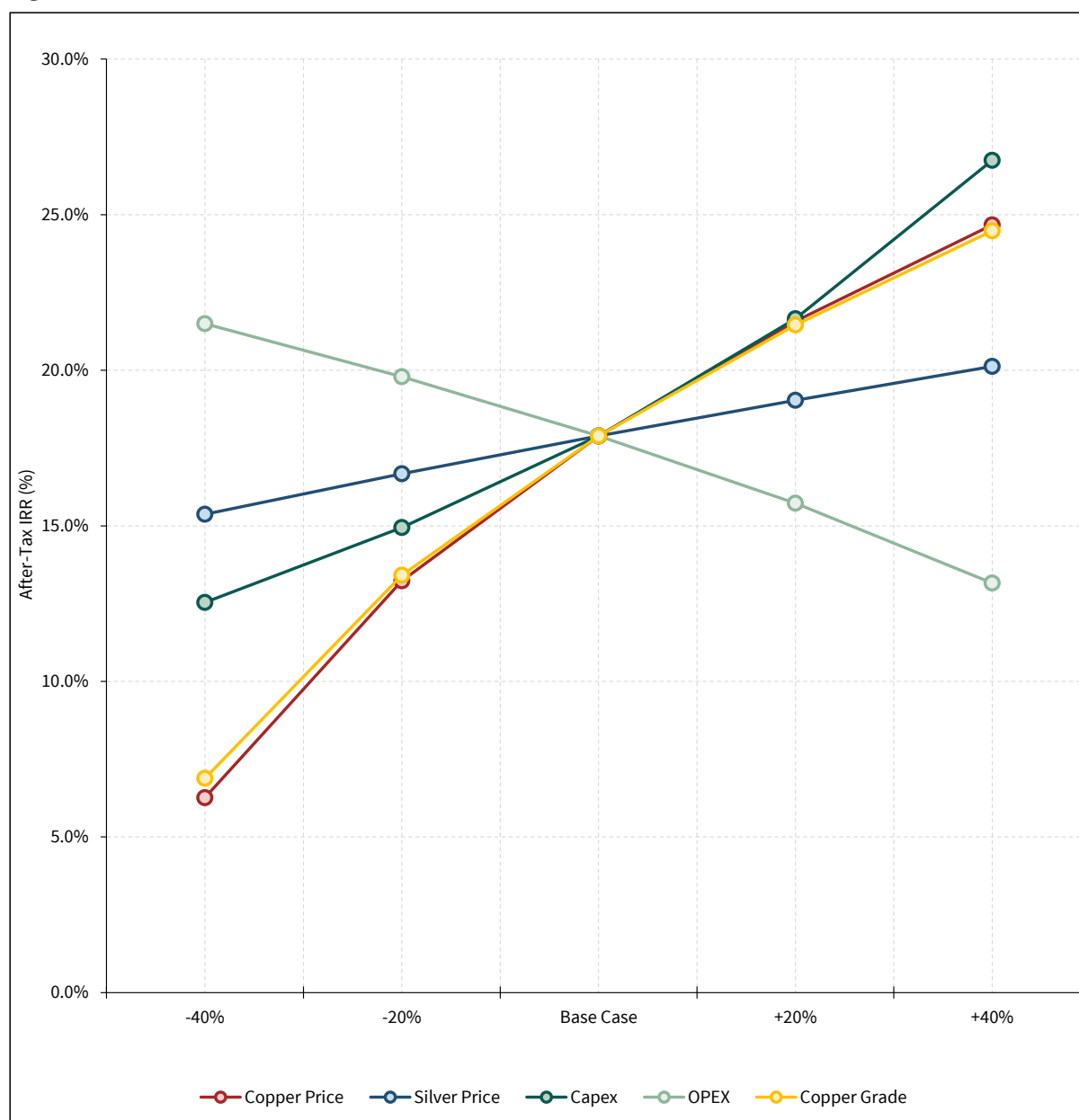


Figure 22.13: IRR Sensitivity - South Shaft Complex



As is typical, both NPV₇ and IRR are most sensitive to copper prices and grade or recovery. Only at a 40% reduction in the copper price (or grade) the South Shaft NPV₇ is reduced to marginally below zero, while the North Shaft NPV₇ remains positive across the range tested. The silver price exhibits moderate sensitivity, and the Project is least sensitive to changes in OPEX and CAPEX.

Thus, the Project base case demonstrates robust economics within the ranges tested.

22.6.3 Discount Rate Sensitivity

Sensitivity of NPV and IRR to changes in copper price at discount rates ranging from 5% to 10% have been tested for each of the Shaft Complexes as well as for the consolidated cash flows. The results for each of these are presented in Figure 22.14, Figure 22.15 and Figure 22.16, respectively.

Figure 22.14: Sensitivity to Copper Price and Discount Rate - North Shaft Complex

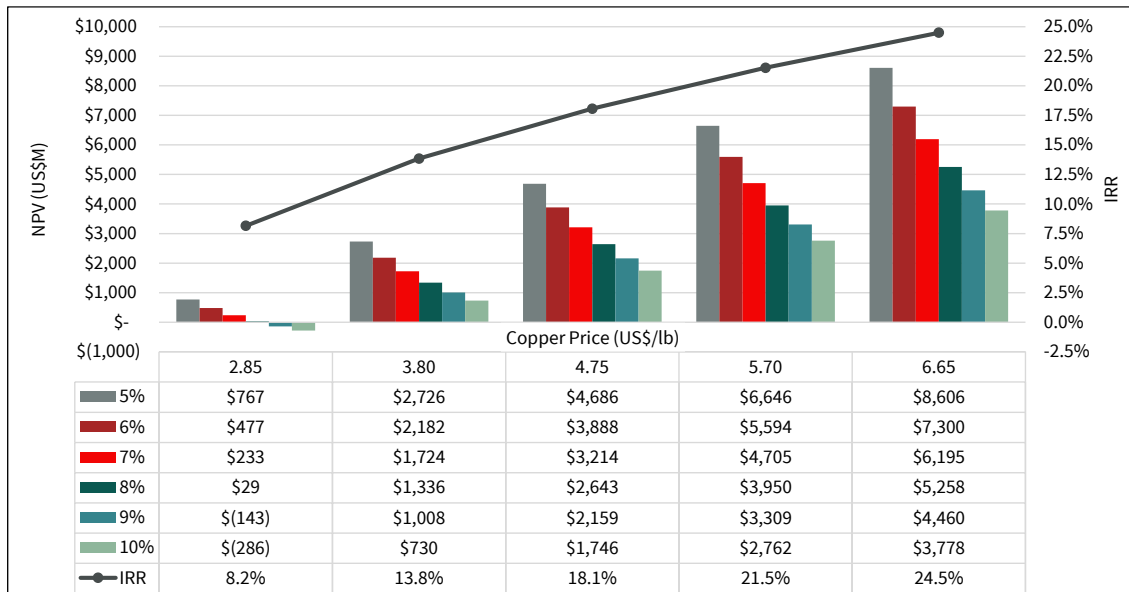


Figure 22.15: Sensitivity to Copper Price and Discount Rate - South Shaft Complex

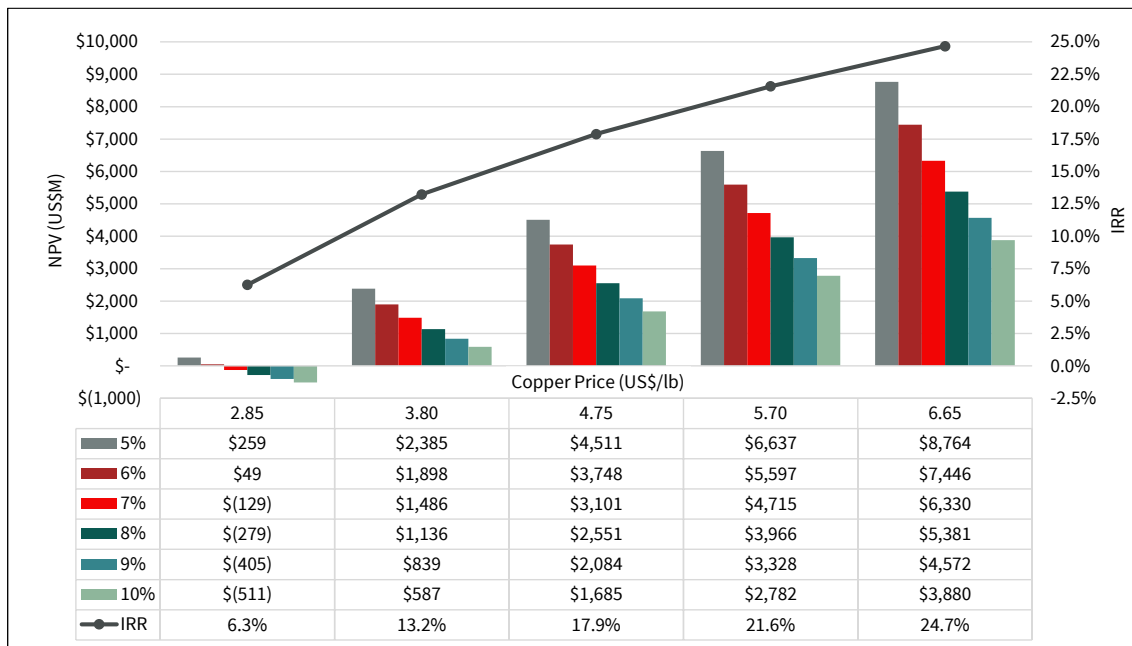
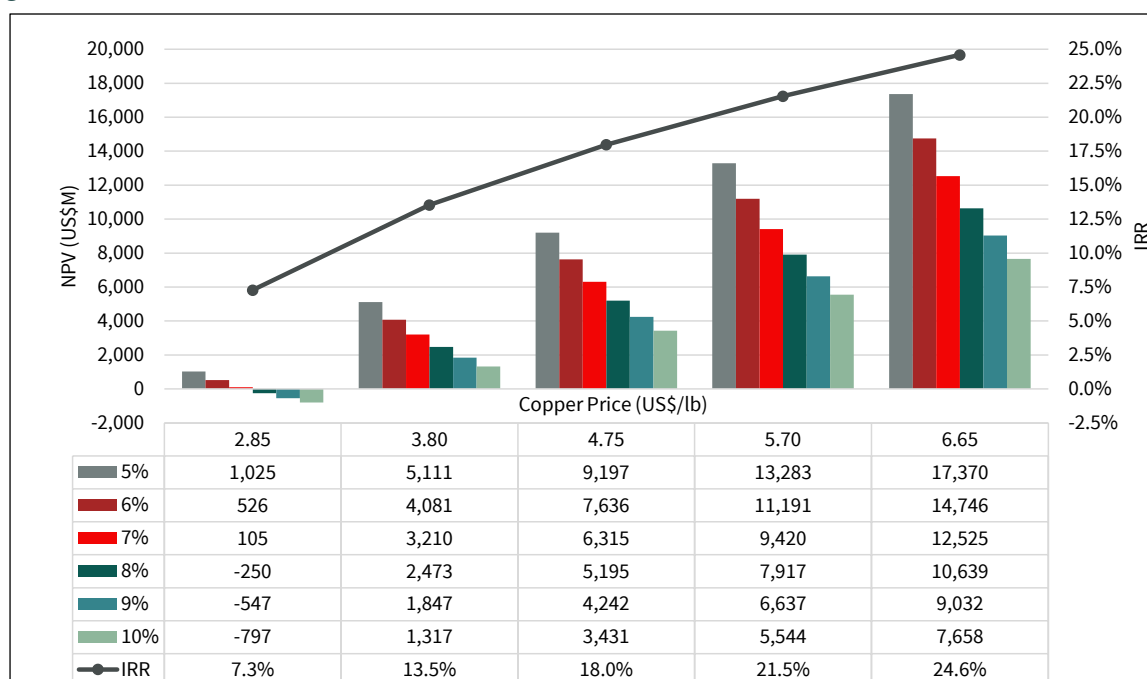


Figure 22.16: Sensitivity to Copper Price and Discount Rate (Consolidation)


22.6.4 Upside Metal Price Case Sensitivity

Table 22.7 show the results for an upside case using metal prices of US\$6.00/lb Cu and US\$90/oz Ag. In this case, NPV₇ and IRR after corporate tax are highly positive at US\$17.7 billion and 29% respectively, while after-MET NPV₇ and IRR remain strongly positive at US\$9.3 billion and 22%, respectively.

Table 22.7: Upside Case with US\$6.00/lb Cu and US\$90/oz Ag

Parameter	Unit	North	South	Consolidation
Net Revenue	US\$ million	57,952	52,593	110,545
Operating Costs	US\$ million	16,774	16,297	33,071
EBITDA	US\$ million	41,178	36,296	77,474
Total Pre-Production Capex	US\$ million	3,196	3,210	6,405
Total Sustaining Capex)	US\$ million	1,454	1,400	2,854
Total Capex	US\$ million	4,650	4,609	9,260
Corporate Taxation*	US\$ million	6,932	6,013	12,945
Net Cash Flow before MET	US\$ million	29,596	25,673	55,269
MET	US\$ million	12,755	12,144	24,899
Net Cash Flow after MET	US\$ million	16,841	13,529	30,370
NPV (Pre-Tax) at 7% Disc.	US\$ million	11,791	10,613	22,404
IRR (Pre-Tax)	%	32.2%	31.6%	31.9%
Payback (pre-tax, years after production startup)	Yrs	1.9	2.3	2.1
NPV (After Corporate Income Tax*) at 7% Disc.	US\$ million	9,332	8,379	17,711
IRR (After Corporate Income Tax*)	%	28.9%	28.3%	28.6%
Payback* Years after Production Startup)	Yrs	2.2	2.5	2.4
NPV After Corporate Tax and MET at 7% Disc.	US\$ million	5,079	4,219	9,299
IRR After Corporate Tax and MET	%	22.3%	21.3%	21.8%
Payback (years after production startup)	Yrs	2.8	3.1	3.0

* Excluding Copper and Silver Mineral Extraction Tax (MET).

23 ADJACENT PROPERTIES

23.1 SOUTHWEST POLAND MINING DISTRICT

Nowa Sól is situated adjacent to a well-established mining district in southwestern Poland and lies approximately 40 km northwest of the Deep Głogów (*Głogów Głęboki Przemysłowy*, GGP) an underground copper-silver mining area; the northernmost extension of the operations in the LGOM owned and operated by KGHM. The GGP mining area is a major contributor of ore produced by the Rudna and Polkowice-Sieroszowice mines, within the LGOM, extracting Kupferschiefer-type mineralization at depths near 1,385 m. KGHM's total throughput capacity is 30 Mt/a with over 400 kt/a of copper mined production, and 600 kt/a refined copper production. In addition, KGHM's operated smelters in Głogów and Legnica import significant quantities of copper concentrates from overseas (Micon Technical Report (February 2013)).

Operational data from the GGP mining area, such as underground mining methods, capital and operating cost parameters, and ventilation and infrastructure design, may provide relevant analogues for conceptual or economic studies on the Property. However, variations in deposit geometry, depth, mining conditions, geotechnical characteristics, hydrology, and metallurgical response between properties may result in materially different engineering designs, operating strategies, and economic outcomes. Additionally, KGHM's integrated downstream infrastructure, including smelters at Głogów and Legnica, offers established regional capacity for copper-silver processing. If the Property advances to the stage of concentrate production, these facilities may present a potential opportunity for toll processing, subject to commercial considerations. Any potential toll processing scenario would be dependent on third-party capacity availability, concentrate quality specifications, required commercial agreements, and applicable permitting and regulatory requirements. Publicly reported metallurgical performance from regional Kupferschiefer operations may provide additional context for potential flotation processing outcomes, noting that reported copper recoveries for certain KGHM operations are approximately 89%. This performance may reflect site-specific factors including mineralogy, liberation characteristics, grind size, flotation residence time, reagent regimes, concentrate grade targets, and operational constraints that can influence achieved recovery in commercial operation. In comparison, Lumina has assumed a copper recovery of 88.8% for the Project based on metallurgical testwork and the process design basis applied in this PEA; however, recoveries derived from laboratory testwork may not be directly comparable to results from operating mines, and actual recoveries achieved by the Project may differ as engineering design and metallurgical testwork are advanced.

Lumina holds exclusive exploration rights to the adjacent Mozów project, located in close proximity to the Nowa Sól deposit within the Fore-Sudetic Monocline of southwestern Poland. The Mozów project targets stratiform sediment-hosted copper-silver mineralization within Permian red-bed sequences. The Mozów project provides additional regional exploration context within the Fore-Sudetic Monocline, demonstrating continued industry activity in stratiform sediment-hosted copper-silver systems associated with Permian stratigraphy in southwestern Poland.

In addition to the Deep Głogów and Mozów properties, the broader Lubin–Głogów mining district contains multiple historical and active copper-silver mining and processing operations developed within the Kupferschiefer-bearing stratigraphic sequence, including KGHM's Lubin, Rudna, and Polkowice–Sieroszowice underground mines, together with regional concentrator plants and downstream smelting capacity at Głogów and Legnica (KGHM, n.d.-b). These operations demonstrate that stratiform sediment-hosted copper-silver mineralization in the region can support large-scale underground mining, and they provide a regional reference point for the types

of mining methods, processing approaches, and infrastructure that may be applicable to future development scenarios at the Property.

The Kupferschiefer copper-silver mineralization style in southwestern Poland is typically hosted within a relatively predictable stratigraphic package that may include Permian red-bed sandstones, the Kupferschiefer horizon, and overlying carbonate units. Mineralization within this package can vary significantly between deposits in terms of thickness, grade distribution, metal ratios (Cu and Ag), sulphide mineral assemblages, and the presence of deleterious elements, all of which may influence resource potential, mine design, and metallurgical performance. As a result, while adjacent or nearby projects are relevant for regional context, their geological and technical characteristics should not be assumed to be directly applicable to the Property.

The existence of established mining and processing activity in the region also implies that supporting infrastructure such as power distribution, skilled labour availability, transport networks, and industrial service providers may be present within practical distance of the Property. These factors may be beneficial in the context of future development; however, the applicability of any such infrastructure to the Project would depend on-site-specific requirements, available capacity, and commercial arrangements.

Publicly reported production and metallurgical performance from nearby operations may provide an indicative range of potential outcomes for stratiform copper-silver mineralization in the region. However, metallurgical response can vary materially based on ore mineralogy, degree of sulphide liberation, grind size and comminution characteristics, flotation kinetics, and concentrate specifications. Accordingly, metallurgical testwork completed to date for the Project and the recovery assumptions adopted in this PEA should be considered preliminary, and additional variability testing and flowsheet optimization work are expected to be required in order to further support future studies (<https://kghm.com.en.our-business>).

It should be noted that the QP has not been able to verify the information presented regarding the KGHM Deep Głogów mining area. The information has been sourced from publicly available information as disclosed in Section 27 and is provided for general informational purposes only and is not necessarily indicative of the mineralization or economic potential of the Property.

23.2 HISTORICAL DRILL HOLES IN THE MRE

In addition to the drill holes inside the Nowa Sól license, six historical drill holes to the south and one to the east of the license were included in the MRE database (Table 23.1). These drill holes were drilled between 1974 and 1990. Historical drill holes exterior to the license were included in the MRE database to limit the extrapolation of grades at the margins of the license. A number of additional historical drill holes were drilled to the northwest and northeast but were not included in the MRE database. Original uncomposited assay results and corresponding sample lengths were not available for review, and as such, these data are considered incomplete and have been excluded from the MRE.

For historical drill holes Boj_1 (Bojadla 1), Borowiec_2, Grochowice_28, S_482, S_483, S_486, S_487, and Slawa_IG1 no downhole survey data was available, and they were treated as vertical with no downhole deviation. The QP considers that this approach reasonable as all the drill holes except Boj_1 are outside the license area and the historical drill holes with survey data show limited deviation from vertical.

Table 23.1: Historical Drill Holes Outside of the Nowa Sól Licence used in the MRE

Hole ID	Company	Year	Easting	Northing	Elevation	EOH	Sampled (m)
Borowiec_2	Historical	1988	559,190	5,737,991	67	1,702	2.6
Grochowice_28	Historical	1990	567,520	5,736,953	72	1,650	8.0
S-482	Historical	1981	564,503	5,735,289	69	1,747	7.6
S-483	Historical	1983	561,918	5,737,040	68	1,683	6.2
S-486	Historical	1985	559,467	5,738,426	67	1,656	8.0
S-487	Historical	1984	557,619	5,736,217	65	1,603	13.4
Slawa_IG1	Historical	1974	570,677	5,744,104	62	2,273	1.8

24 OTHER RELEVANT DATA AND INFORMATION

The Qualified Person is not aware of any additional relevant data or information that would materially affect the conclusions of this Technical Report and that has not been disclosed herein.

25 INTERPRETATION AND CONCLUSIONS

25.1 CONCLUSIONS

The Project demonstrates potential for the development of an underground copper-silver mining operation, with silver credits; however, the results of this preliminary economic assessment are subject to the limitations and uncertainties inherent to this level of study.” The preliminary economic assessment is preliminary in nature, and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized.

The Mineral Resource is classified as Measured, Indicated and Inferred, as summarized in Table 14.20. The estimate was prepared in accordance with the CIM Best Practice Guidelines (2019) and reported in accordance with the CIM Definition Standards (2014), as incorporated by reference into NI 43-101.

In the QP’s opinion, the Mineral Resources reported demonstrate RPEEE, based on the selected cut-off grade and the application of appropriate mining and processing assumptions. Mining envelopes used to identify spatially continuous mineralization within potentially mineable shapes using reasonable assumptions based on neighbouring operations and long-term price trends to calculate a pay limit Cu grade. After assessing the optimal cut-off grade, a target head grade was selected at 0.95% Cu to ensure maximum mineable material was included into the mining envelopes. The mining envelopes were used to constrain the block model and identify material that meets RPEEE and were considered for inclusion in the Mineral Resource Statement (Table 14.20).

A conceptual mine design and LoM schedule have been developed for two Shaft Complexes, namely the North and South Shaft Complexes, utilising conventional Room and Pillar mining methods targeting an annual production of 11.5 Mt/a.

The preliminary flowsheet developed from the testwork program is considered conventional and comprises primary grinding to a P_{80} size of 60 μm , rougher flotation, rougher concentrate regrinding to approximately 11 μm and two stages of cleaning. The results from the preliminary metallurgical development testwork program suggested copper and silver recoveries into a concentrate grading 26.2% Cu are 88.8% and 86.6%, respectively.

Both Shaft Complexes provide attractive economic returns, though the NPV₇, IRR and payback period, but for the North all exceed the corresponding parameters for the South. The combined pre-tax NPV₇ for both Shaft Complexes is US\$8,326 million with an IRR of 20.5 and payback of 3.2 years.

The PEA demonstrates that the Project has potential to support further technical evaluation and advancement to a PFS. However, the results are preliminary in nature and subject to significant technical, economic, and regulatory uncertainties that will require additional work to address.

25.2 RISKS AND UNCERTAINTIES

25.2.1 Mineral Resource Risk

The completion of a maiden NI 43-101-compliant Mineral Resource Estimate marks a key technical milestone for the Project. Notwithstanding this, mineral exploration is subject to inherent uncertainties. These include, but are not limited to, the possibility that further exploration may not result in the identification of additional economically viable mineralization, variability in geological conditions, changes in commodity prices, and potential regulatory or permitting constraints.

Although exploration and drilling activities have been, and are planned to be, conducted in accordance with accepted industry standards, such standards do not eliminate the fundamental uncertainties associated with mineral exploration and resource development.

The reader is cautioned that the assessment of economic potential applied in the reporting of Measured, Indicated, and Inferred Mineral Resources constitutes a high-level evaluation and is intended solely to support the reporting of Mineral Resources. It does not represent an estimate of Mineral Reserves. The Mineral Resource may be materially affected by future infill drilling, which could result in either an increase or decrease in the quantity and classification of the Mineral Resource.

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. Additionally, the Mineral Resource may be impacted by future assessments relating to mining, processing, environmental compliance, permitting, taxation, socio-economic considerations, and other relevant modifying factors.

25.2.2 Metallurgy and Processing Risk

Only a limited amount of metallurgical testwork has been completed to-date on samples from the Nowa Sól deposit. Although the process design and metallurgical performance appears reasonable and in-line with near-by operations that mine and process similar mineralization, additional testwork needs to be undertaken to optimize the design and process equipment selection.

25.2.3 Salt Risk

An unknown amount of salt will be encountered during shaft sinking and assumptions have been made about salt volume using the drill hole logs available. This presents potential engineering and environmental risks, as outlined below.

The shaft design and stability could be at risk if the hoisting and handling system sizing is not appropriate, which could result in an increased risk of operational bottle necks causing shaft sinking delays.

Salt formations are known to have high creep rates and water-bearing strata above or below. An underestimation in salt volume may result in a higher risk of creep closure and higher-than-expected brine generation.

As salt is classified as mining waste the disposal and storage of salt poses a potential environmental risk that could result in ground and surface water contamination plus salinization of soil.

The risks associated with salt will be addressed and mitigated once the planned 2026 drilling program has been completed, which will provide lithological, geotechnical and hydrogeological information for the Project.

25.2.4 Geotechnical Risk

High stress and seismicity will be an appreciable challenge for the Nowa Sól deposit and will be one of the highest risks assumed for the Project. Current mine productivity assumptions have not considered geotechnical constraints from high stress and seismicity. Revised productivity assumptions considering geotechnical constraints could have major implications on Project economics. Detailed geotechnical analysis supported by good data collection is required in subsequent design phases.

High stresses will also impact shaft cost and schedule. The stability analysis work carried out during the PEA has identified very high shaft squeezing and instability risks for some lithologies including halites (salt). Significant shaft deformations and water pressures will need to be considered in shaft liner design and constructability.

25.2.5 Ventilation Risk

Detailed ventilation modelling has not been carried out for the PEA. Ventilation and cooling may require more infrastructure and energy than currently assumed. Ventilation design arrangements should be revisited early in the next stages of study, including detailed modelling. Outcomes could have moderate to major impacts to infrastructure arrangements such as shaft sizing, quantity of shafts, and quantity of ventilation infrastructure.

25.2.6 Backfill Risk

At the PEA stage the backfill design and associated infrastructure are based on conceptual assumptions and limited site-specific data. As a result, a number of key risks and uncertainties remain that may influence detailed design, operability and cost as the Project advances.

The principal backfill related risks and uncertainties identified at this stage include:

- Tailings variability and dewatering response: Limited characterisation of particle size distribution, mineralogy, fines content and rheology introduce uncertainty in thickener performance, achievable underflow density, filtration behaviour and expected cake moisture variability.
- Filtration performance and availability: In the absence of representative filtration testwork, there is uncertainty in achievable filtration flux, cycle times, cloth performance and operability under variable feed conditions. This may influence the required operating margin, maintenance strategy and effective plant availability.
- Paste rheology and pipeline operability: Uncertainty in paste yield stress/viscosity affects friction losses, operating pressure, deposition risk, and wear rates. Rheology is also sensitive to binder type/dose, water quality and fines content, which may require tighter control strategies than assumed at PEA level.
- High static head and pressure management: Mining depths approaching 2,000 m imply static heads of around 20 MPa, increasing exposure to pipeline integrity risk, transient pressure events and the consequence of upset conditions. Detailed hydraulic modelling and a robust pressure protection philosophy will be required to confirm pipe ratings, operating envelopes and pressure management options.
- Binder demand uncertainty in high-stress zones: Localized requirements for higher binder dosages (e.g. in higher-stress or seismically active areas) could materially impact operating cost, cement logistics and paste handling characteristics, and should be evaluated alongside geomechanical modelling outputs.
- Operational Integration with mining sequence: The integrated flowsheet links filtration availability to both backfill placement and dry stack production. Insufficient surge capacity, downtime recovery planning or redundancy in critical equipment could constrain backfill placement and potentially impact mining productivity if not addressed in the next study phase.

These uncertainties will be reduced during the PFS through targeted tailings testwork, paste mix and strength testing, detailed geomechanical and hydraulic modelling and structured vendor engagement to confirm equipment performance and operating envelopes.

25.2.7 Infrastructure Risk

A large power supply is required for the operation, and this requires availability of a high voltage electricity supply. There is potentially limited supply capacity on the existing network but early engagement with the utility supplier is important to mitigate this risk.

25.2.8 Environmental and Social Risk

Environmental and social risks associated with the Project are considered typical of similar mining operations at this stage of development. Important consideration will need to be given to water management, in terms of proximity to the Odra River and the anticipated complexity of groundwater conditions. Careful planning will also be needed to avoid areas of biodiversity importance, including national and internationally designated conservation sites. These risks can be managed to an acceptable level, provided that appropriate mitigation measures and management systems are implemented, and sufficient financial resources and technical staff are provided.

25.2.9 Hydrogeology Risk

The hydrogeological study is based on desktop information only and is therefore conceptual in nature, which is suitable for a PEA. However, the current water balance demonstrated a water deficit which could have potential operation implications.

Hydrogeology testing from the planned 2026 drilling program and a more advanced water balance will inform mitigation.

25.2.10 Economics Risk

The biggest risk to project economics is the Polish MET. Currently there is no exception to this tax which renders the Project less economically attractive.

Metal prices are a high risk but also represent an opportunity. This risk is accepted and borne by the Project's investors.

At the PEA level, capital and operating cost inputs are inherently conceptual and are based on limited engineering definition and assumptions, creating a material risk that actual costs may differ significantly as the Project advances to more detailed studies.

Finding a market for the concentrate is not considered to be a major risk, unless further metallurgical testwork shows a problem with concentrate grades or deleterious elements.

26 RECOMMENDATIONS

The QPs and Micon recommend the following actions:

26.1 METHODOLOGY AND PROTOCOL ENHANCEMENTS

It is recommended that future exploration programs replace the current blank material with certified blank material sourced from reputable CRM providers. Although the trace levels of copper present in the existing blanks are negligible and have not impacted the MRE, the use of certified blanks will ensure full alignment with industry-standard QA/QC protocols and support the integrity of future analytical results.

26.2 MINERAL RESOURCE ESTIMATES

The following improvements could be considered in future updates of the MRE Estimate:

- For Ag, a separate high-grade population was modelled as high-grade intervals were of sufficient thickness to model with the selected composite length and block size of 0.6 m. A separate high-grade population is visible in the Cu data, but it is too thin to resolve with the selected composite length and block size. Tests were made to reduce the block size but the processing times for the model increased considerably. It is recommended that further investigations are undertaken to assess whether a high-grade Cu could be modelled when the MRE is updated.
- To preserve the complex multivariate relationships observed a detailed multivariate simulation study using Projection Pursuit Multivariate Transform could be performed.

26.3 MINE PLANNING

Reduce gap in years between lateral development and actual production so that unnecessary capital is not spent upfront.

Assess high mining stress impacts to production and development planning criteria. Through additional geotechnical field investigation develop 3D numerical stress models for assessing mine seismicity and stress impacts.

Optimising mine planning to better align development timing with production requirements and investigate the potential for additional silver production from less copper-rich areas of the deposit based on a copper-equivalent cut-off grade.

26.4 VENTILATION AND COOLING

To further develop the mine plan configurations and concepts for maintaining the ventilation development on the ore-level are required in a subsequent study. Also details regarding shaft connections and shaft area infrastructure into the mine plan need to be incorporated. Once a mine plan configuration has been developed and scheduled, this configuration can be modelled in Ventsim to assess air flow requirements for the LoM. In addition, incorporation of trade-off assessments for 5-shaft versus 3-shaft configurations will be required and an assessment of ventilation trade-offs for battery electric equipment.

The next phases of study will undertake a detailed analysis of refrigeration and winter heating requirements. The refrigeration system type and cooler locations must be finalized; however, this should be undertaken only once the complete equipment list and associated heat loads are confirmed. Factors such as groundwater inflows should also be assessed in future heat modelling.

26.5 SHAFTS

The sinking concepts presented in Section 16.1.2.1 are proven methods and are based on previous experience in Poland. Shaft sinking is the largest component in the initial cost and schedule for Nowa Sól as with most new deep mining projects. Improving the schedule to reach and establish the deep orebody will provide a good opportunity to shorten the time to full production. Going forward the following concepts require further engineering and analysis to investigate potential optimization.

Mechanical Shaft Excavation

Mechanically cutting the rock with autonomous loading has been successfully used on recent world class shaft projects. It has been marginally successful in faster excavation of shafts and is only restricted by how fast the shaft liner can follow. The machines can be sized to meet the needs of Nowa Sól's 9.5 m shaft diameters. The high-front-end capital needed may be off set with an improved schedule to first production.

Shaft Sequencing

The base schedule presented in this Report involves shaft sinking in sequence and completed to fit the base schedule of production. The ventilation shaft has been delayed until needed to defer the associated capital expenditure. However, the ventilation shaft is the simplest surface set up, therefore sinking could commence earlier. Moreover, the final liner in the salt layer can be delayed, thus giving the ventilation shaft early access to the ore level. By using the sinking hoist, shaft pillar development could start from the ventilation shaft concurrently with the sinking and equipping of the production and service shafts. This concept could net a substantial reduction in the critical path to first ore production. A detailed cost benefit analysis may prove this a viable option.

Permanently Freeze Shaft

Freezing the water bearing ground is necessary to sink through zones of saturated lithology. A full hydrostatic liner must be installed prior to turning off the freeze plant. The estimated cost of US\$306,900 includes the price and schedule for the tubing required to complete these zones. If the freeze plant remains in operation for the LoM the tubing would not be required as well as the subsequent grouting. The result would be an improved sinking schedule off set by the increased operating cost of maintaining the freeze.

Conduct a more detailed comparison of the 5-shaft (Scenario 1) and 3-shaft (Scenario 2) layouts. From a ventilation standpoint, the 5-shaft layout may offer greater flexibility in managing airflows, pressures, and surface-fan operating points. Other factors need to be considered in this trade-off including capital costs, operating costs, schedule, and risks (such as geotechnical risk with larger shafts).

26.6 BACKFILL

Progression of the backfill and tailings management design to PFS level should focus on the following key workstreams, aimed at validating assumptions, reducing technical risk and refining capital and operating cost estimates:

- Tailings characterisation and dewatering testwork, including particle size distribution, mineralogy and clay content, thickening performance, achievable filtration rates, cake moisture targets and variability under representative operating conditions.
- Paste backfill mix design testing, including optimization of binder dosage, UCS development at 7-day and 28-day curing, workability and slump, bleed behaviour,

permeability and long-term durability. This program will validate the assumed binder dosages and confirm strength criteria.

- Detailed hydraulic modelling of the paste reticulation system, to confirm final pipeline diameters, pressure ratings and pumping arrangements.
- Refinement of the DSTF design, incorporating site-specific geotechnical parameters, foundation conditions, drainage and seepage control measures, water balance and storm event management, and confirmation of the staged (split-cell) development strategy.
- Cost optimization and vendor engagement, including budgetary quotations and technical compliance reviews for major equipment. Vendor input will be used to refine equipment selection, installed capacities, power demand and capital cost estimates.

26.7 METALLURGICAL DEVELOPMENT STUDIES

It is recommended that further metallurgical testing be undertaken to support the next phase of Project development, probably a pre-feasibility level of study. The test program should consider the following:

- Select fresh composite samples to cover the mineral resources spatially, range of copper and silver grades, ore-types and lithologies.
- Detailed chemical and mineralogical characterisation of each composite sample.
- Undertake comprehensive comminution testing for each composite sample.
- Complete development and optimization flotation tests on separate composite samples.
- Investigate the potential for lead and zinc as by-products.
- Undertake a series of locked cycle tests on separate composite samples.
- Complete characterisation studies on locked cycle flotation products.
- Undertake standard dewatering testing on samples of tailings and final concentrate.
- Consider testing of alternative technologies with vendors to improve energy and metallurgical efficiencies.

Infrastructure

A significant opportunity for the Project is linked to the planned reopening and reconstruction of the currently decommissioned Wolsztyn–Nowa Sól-Żagań railway line, which is located close to the Project area. This creates the potential to use rail transport for heavy and oversized cargo during both the construction phase and later during mine operations, including delivery of equipment, consumables and potentially product transport.

It is strongly recommended that this opportunity is actively monitored and pursued, with the aim of integrating rail access into the Project's logistics and operations strategy through constructing a single-track branching from this line to the projected concentrator plants. Early engagement with the railway authorities and Project proponents should be undertaken to understand timelines, technical requirements and interface points. If feasible, rail access should become an integral part of the Project design and operations concept, as it could significantly reduce road traffic, lower logistics costs and minimize impacts on local communities.

26.8 ENVIRONMENTAL STUDIES

Baseline Data Collection

Undertake baseline environmental and social surveys within the Project's potential area of impact/influence. Surveys should include primary and secondary data collection over a minimum 12-month period to account for seasonal changes and to establish existing conditions. Particular attention should be given to biodiversity and water quality, given the known water bodies, proximity to designated conservation areas and the influence of other mining operations in the region.

Hydrology and Hydrogeology Studies

Undertake detailed technical studies to understand surface water and groundwater conditions, including water balance and water quality, in and around the proposed Project site. As well as helping to inform engineering design for the Project, these studies will assist with evaluation of water supply options and potential environmental impacts.

Stakeholder Engagement

Continue to maintain a register of all formal and informal communications about the Project, with particular attention on the local community/general public.

Community Relations

Establish a non-technical Project factsheet for general reference and use during formal and informal meetings, including community discussions. This will ensure that consistent messaging is provided to local residents and business owners, and should help manage community expectations, particularly regarding potential future employment opportunities.

26.9 FUTURE WORK AND BUDGET

Nowa Sól demonstrates a robust exploration project with potential for economic extraction and the QP recommends that Lumina proceeds with the next phase of work to further evaluate and de-risk the Project. This work should include the commencement of a PFS as well as additional drilling, metallurgical testwork, geotechnical testwork, hydrogeological studies and environmental and social baseline studies.

The PFS should involve various trade-offs including:

- LoM Options, such as:
 - Project Phasing (North versus South Shaft).
 - Throughput Rates.
- Backfill Possibilities.
- Ventilation and Cooling Options.

To obtain further metallurgical, geotechnical and hydrogeological data, whilst also complying to Polish regulatory requirements, Lumina has indicated that five (5) diamond drill holes are planned, commencing Q2 2026. Based on the current data, the QP supports this initiative and recommends it as part of the next phase of work.

Environmental and social baseline studies should be commenced as soon as possible, to support an Environmental and Social Impact Assessment (EISA) and environmental permitting of the Project. The duration of baseline studies is typically over a period of 12 months to include coverage across all seasons.

These studies are directly linked to the application process for a mining extraction concession, which is granted by the Polish MCE, and are also required as international good practice. The studies should be undertaken by a suitably qualified team of environmental consultants, with experience of mine permitting in Poland, and contracted directly by Lumina. Micon can provide advice and support.

The shaft will be sunk through the complex sediments described in Section 7.3. An exploration hole is planned near the vicinity of the new shafts (50 m to 100 m) to validate the geotechnical conditions expected during sinking and complete detailed shaft design prior to construction.

The estimated costs for the recommended future work totals approximately US\$32,000,000. The estimated costs of drilling geological, geotechnical, hydrogeological and metallurgical drill holes includes US\$25,000,00 of that cost with US\$7,000,000 budgeted for the PFS. Table 26.1 summarizes the breakdown of costs.

The proposed work program and budget are based on the current level of project knowledge and are consistent with industry practice for advancing a project to PFS stage. The QP considers the proposed work to be reasonable and appropriate to further evaluate the technical and economic potential of the Project.

Table 26.1: Summary Budget for the Nowa Sól Copper-Silver Project

Description	Item	Cost per Item US\$	Inclusion Costs per Hole US\$	Total US\$	Notes
PFS	Completion of a PFS	US\$ 7,000,000		US\$ 7,000,000	Full scope of contractor costs for the study and PFS NI 43-101 report.
Drilling	5 holes	US\$ 4,000,000		US\$ 20,000,000	
	Side Tracks for each hole		US\$ 300,000		Included in total hole cost.
	Bore hole geophysics and pumping tests for each hole		US\$ 300,000		Included in total hole cost.
Hydrogeological	Studies and modelling			US\$ 1,200,000	
Metallurgical	Further studies			US\$ 1,800,000	
Environmental and Social	Baseline studies			US\$ 2,000,000	
Total:				US\$ 32,000,000	

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28 CERTIFICATES**CERTIFICATE OF AUTHOR****Liz de Klerk, M.Sc., Pr.Sci.Nat., FIMMM**

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Liz de Klerk, do hereby certify that:

1. I am employed as Managing Director and Senior Geologist by, and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom. tel. 0044 (1603) 501 501, e-mail ldeklerk@micon-international.co.uk
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:
 - B.Sc. Geology, University of Leicester, United Kingdom, 2000;
 - M.Sc. Exploration Geology, University of Rhodes, Grahamstown, South Africa, 2002;
4. I am a registered Professional Natural Scientist (Pr.Sci.Nat. 400090/08) and a Fellow of the Institute of Materials Minerals and Mining (FIMMM) (registration number 682617) and a Fellow of the Geological Society of Africa.
5. I have worked as a geologist in the minerals industry for over 18 years.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. I am a geologist with over 20 years of experience in the minerals industry since 2003, with a focus on independent consulting, exploration, mineral resource estimation, and international code-compliant reporting. My experience includes responsibility for the compilation, estimation, and reporting of Mineral Resources and the preparation of technical and techno-economic studies in accordance with international reporting standards, including NI 43-101 and other CRIRSCO-based codes. I have worked on projects in Africa, Europe, Russia, Canada, and the United Kingdom, across a range of commodities including potash, copper, coal, chromite, manganese, iron ore, gold, and platinum group elements. I have extensive experience working on sedimentary and stratified mineral deposits, including basin-scale and layered systems. My experience includes 3D geological modelling, resource estimation, exploration program support, and technical reporting, as well as contributing to feasibility studies and due diligence assignments. I have held senior leadership and consulting roles, including managing exploration programs and leading multidisciplinary technical teams, and have extensive experience in code compliance and reporting for major international stock exchanges.
7. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I visited the Nowa Sól Copper-Silver Project site between February 12, 2025, and February 14, 2025 and July 1, 2025, and July 3, 2025;

9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.
10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.
11. I am responsible for the preparation of Sections 1.1 to 1.10, 1.12, 1.14, 1.15, 1.19.1, 1.23, 1.25, 1.26, 2, 3 4 (except 4.2.5), 5 6, 7, 8, 9, 10, 12 (except 12.2), 14.11 (except 14.11.3), 15, 19.1. 23, 25.1, 25.2.1, 25.2.3, 26.2, 26.9 and 27 of the Technical Report;
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.



Liz de Klerk, M.Sc., Pr.Sci.Nat., FIMMM
Managing Director
Micon International Co Limited

CERTIFICATE OF AUTHOR**Ryan Langdon, Ph.D., MCSM, MEarthSci, CGeol, FGS**

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Ryan Langdon, do hereby certify that:

1. I am employed as a Principal Resource Geologist by, and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom. tel. 0044 (1603) 501 501, e-mail rlangdon@micon-international.co.uk
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:
 - MEarthSci: Earth Sciences, University of Oxford, United Kingdom, 2010.
 - Ph.D Geology,, Camborne School of Mines, University of Exeter, 2015.
 - Applied Geostatistics Citation, University of Alberta, 2023.
4. I am a Chartered Geologist (CGeol) and Fellow registered with the Geological Society of London (registration number 1022491)
5. I have worked as a geologist in the minerals industry for over 13 years.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. I am a chartered resource geologist with over 14 years of experience in the minerals industry since 2011, with an international background in gold and coltan exploration, industrial minerals mining, and mineral resource estimation. My experience includes responsibility for mineral resource estimation, geological modelling, and geostatistical analysis, as well as the design and implementation of greenfield and brownfield exploration programs. I have worked on projects in Africa, Asia, Europe, and the United Kingdom, including the development of mineralisation models for orogenic gold deposits in West Africa and the preparation of code-compliant Mineral Resource Estimates for base and precious metal deposits, including Cu-Mo and Zn-Pb-Ag-Au systems. I have experience across a range of commodities including gold, base metals, and industrial minerals, and have applied industry-standard software for geological modelling and resource estimation. My work has included exploration program design and execution, geological and structural mapping, geochemical sampling and QA/QC, database management, and resource modelling, supporting projects from exploration through resource definition and technical studies.
7. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I visited the Nowa Sól Copper-Silver Project site between July 1, 2025, and July 3, 2025;
9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.
10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.

11. I am responsible for the preparation of Sections 1.11, 1.12, 11, 12.2, 14.1 to 14.10 and 14.11.3 of the Technical Report.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.



Ryan Langdon, Ph.D., MCSM, MEarthSci, CGeol, FGS
Principal Resource Geologist,
Micon International Co Limited

CERTIFICATE OF AUTHOR
Richard Gowans, B.Sc. (Hons), P.Eng.

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Richard Gowans, do hereby certify that:

1. I am employed as Principal Metallurgist by Micon International Limited, and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom, tel. 0044 (1603) 501 501, e-mail rgowans@micon-international.com.
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:
B.Sc. (Hons) Minerals Engineering, University of Birmingham, UK, 1980.
4. I am a member of the Association of Professional Engineers of Ontario (PEng) (registration number 90529389) and a member of the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM).
5. I have worked in the minerals industry for over 40 years.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. I am a metallurgist with over 40 years of experience in the minerals industry, having gained operational experience in platinum mining in South Africa since 1981 and working in consulting roles since 1987. My experience includes the operation, design, and commissioning of mineral processing and hydrometallurgical plants for a wide range of commodities, including precious metals, base metals, and specialty minerals. I have been involved in operations, management, detailed engineering, and commissioning of processing facilities, as well as feasibility studies, technical audits, project management, and due diligence assignments for projects in Europe, North America, South America, Africa, Asia, and Oceania. My experience includes extensive work in base metals and specialty minerals, including the processing and recovery of copper from multiple deposits, as well as broader experience in mineral processing and hydrometallurgical plant design across multiple jurisdictions.
7. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I have not visited the Nowa Sól Copper-Silver Project site.
9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.
10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.
11. I am responsible for the preparation of Sections 1.13, 1.17, 13, 17, 25.2.2 and 26.7 of the Technical Report;

12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.

A handwritten signature in black ink, appearing to read 'RG', with a long horizontal stroke extending to the right.

Richard Gowans, B.Sc. (Hons), P.Eng.
Principal Metallurgist,
Micon International Limited

CERTIFICATE OF AUTHOR
Garth Matti Liukko, P.Eng

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Garth Liukko, do hereby certify that:

1. I am employed as a Principal Mining Engineer by Micon International Co Limited, and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom.
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:
B.Eng. Mining Engineering, Laurentian University, Sudbury Ontario, Canada, 1991.
4. I am a member in good standing of the Professional Engineers of Ontario (registration number 90533399).
5. I have worked as a mining engineer for over 33 years since graduation.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. I am a professional mining engineer with over 30 years of experience in the minerals industry since 1991, specializing in underground mine design, planning, and economic evaluation across a range of base metals, precious metals, and critical minerals projects. My experience includes responsibility for underground mine planning and design, mineral reserve estimation, life-of-mine scheduling, cost estimation, and financial modelling, as well as the design and optimization of mine infrastructure systems including ventilation, backfill, dewatering, materials handling, and mobile equipment. I have acted as Qualified Person and lead author for NI 43-101 and other code-compliant technical reports, including preliminary economic assessments, prefeasibility and feasibility studies, and have supported projects through all stages of the mining lifecycle from scoping and evaluation through operations support and mine closure. I have worked on projects globally and have extensive experience in technical due diligence, independent technical reviews, and regulatory and ESG audits, supporting investment decisions and project development across multiple jurisdictions.
7. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I have not visited the Nowa Sól Copper-Silver Project site.
9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.
10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.
11. I am responsible for the preparation or supervision of Sections 1.16.1, 1.16.3, 16.5 and 26.3 summarized therefrom of the Technical Report;

12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.



Garth Liukko, P.Eng.
Principal Mining Engineer
Micon International Co Limited

CERTIFICATE OF AUTHOR
Ben Cottrell, BASc., M.Eng., MBA, CIM, P.Eng.

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Ben Cottrell, do hereby certify that:

1. I am employed as a Principal Geotechnical Engineer by Global Mine Design, Tremough Innovation Centre, Penryn, Cornwall, TR10 9TA, United Kingdom, and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom. tel. 0044 (1603) 501 501, e-mail Ben.Cottrell@globalminedesign.com
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:
 - BASc. Geological Engineering, University of Waterloo, Canada, 2004;
 - M.Eng. Mineral Engineering, University of Toronto, Canada, 2008;
 - MBA Mining Management, Schulich School of Business, Canada, 2017;
4. I am a member of the Professional Engineers of Ontario (P.Eng.), and a Member of the Canadian Institute of Mining and Metallurgy (CIM) (registration number 793854).
5. I have worked in the minerals industry for over 20 years.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. I am a principal geotechnical engineer and mine planner with over 20 years of experience in the minerals industry since 2004, encompassing both operations and consulting roles. My experience includes geotechnical investigations, ground characterization, numerical modelling, and the development of geotechnical design parameters for open pit and underground mining operations, as well as mine planning and design optimization with a focus on mining method selection and sequencing. I have worked on over two dozen mining projects globally, including projects in North America, South America, Africa, and Europe, supporting studies from preliminary economic assessment through feasibility and into execution. Since 2011 I have held senior technical and project management roles, including acting as Owners Team technical lead for geotechnical programs and mine design optimization, and serving as project manager for multidisciplinary study teams.
7. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I visited the Lumina Metals Corp. core logging facility on December 12, 2025;
9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.
10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.
11. I am responsible for the preparation of Sections 1.16.2, 1.24, 16.1, 16.2, 16.6, 24, 25.2.4, 25.2.5, 25.2.6, 26.1, 26.4, 26.5 and 26.6 of the Technical Report;

12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.



Ben Cottrell, B.A.Sc., M.Eng., MBA, CIM, P.Eng.
Principal Geotechnical Engineer
Global Mine Design

CERTIFICATE OF AUTHOR
Alex Zaitchenko, P.Eng., PE

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Alex Zaitchenko, do hereby certify that:

1. I am employed as a Civil Engineer by, and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom, tel. 0044 (1603) 501 501, e-mail: azaitchenko@micon-international.com
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:

Bridge Engineer Diploma, University of Architecture and Civil Engineering, St Petersburg, Russia, 1992;

This degree was evaluated by Professional Engineers of Ontario (Canada) and NCEES (USA) and deemed to be equivalent of Bachelors Degree in Civil Engineering for the purpose of obtaining professional engineering license, granted on September 9, 2004.

Civil Engineering discipline is directly related to Mining as Civil Engineers are responsible for design and construction of most of surface and underground mining infrastructure.
4. I am a member of the Professional Engineers of Ontario (P.Eng) (No. 100057759), and a Licenced Civil Engineer (PE) in the State of Nevada (No. 032465);
5. I have worked in the minerals industry for more than 18 years.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101.
7. I am a civil engineer with over 30 years of experience, including more than 18 years in the minerals industry since at least 2008, involving the design and delivery of surface and underground mining infrastructure for projects in North America, the United Kingdom, Russia, and Australia. I have been involved in mining projects from concept and feasibility through detailed design, construction, and commissioning, including both greenfield and brownfield developments. My experience includes the design and delivery of mine infrastructure such as hoisting systems, shaft sinking facilities, ore and waste handling systems, and associated surface and underground facilities, as well as the layout and design of underground access systems, shafts, and tunnels, and supporting infrastructure including power distribution systems, ventilation systems, mine dewatering infrastructure, maintenance facilities, refuge stations, and concrete and shotcrete delivery systems. Since at least 2011 I have led multidisciplinary engineering teams and external consultants in the execution of large-scale EPC projects, with responsibility for planning, organizing, and directing engineering activities, including the management of in-house engineering teams and third-party consultants to deliver engineering scopes in accordance with project schedules and client requirements. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I have not visited the Nowa Sól Copper-Silver Project site.

9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.
10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.
11. I am responsible for the preparation or supervision of Section 16.3 from of the Technical Report.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.



Alex Zaitchenko, P.Eng., PE
Civil Engineer
Micon International Limited

CERTIFICATE OF AUTHOR
Peter Stevens, B.Sc., M.Sc., CGeol

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Peter Stevens, do hereby certify that:

1. I am employed as a Director and Principal Engineering Geologist by CQA International Limited, The Keele Centre, Keele, Three Mile Lane, Newcastle ST5 5HH United Kingdom, and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom. tel. 0044(1603) 501 501, e-mail: peter.stevens@cqainternational.co.uk
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:
 - B.Sc. (Hons) in Geology, Imperial College, London University, UK, 1981.
 - M.Sc. Engineering Geology, Imperial College, London University, UK, 1985.
4. I am a Chartered Geologist (CGeol) registered with the Geological Society of London (registration number 14778).
5. I have worked in the natural resource industries for more than 30 years.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. I am a chartered geologist with over 38 years of experience in the natural resource industries since 1986, with expertise in engineering geology, hydrogeology, waste management, and geotechnical engineering. My experience includes geotechnical and environmental site characterisation, hydrogeological assessment, and the conceptual and detailed design of mining and waste management infrastructure, as well as modelling of ground and groundwater responses, permitting, and construction supervision. I have worked on projects in Europe, Central and Southern Asia, the Middle East, and North Africa, and have extensive international experience across more than 20 countries. My experience includes the design, specification, construction, and quality assurance of containment systems for mining and waste management facilities, including tailings storage facilities, lagoons, and associated infrastructure, as well as broader involvement in water resources and civil engineering projects.
7. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I have not visited the Nowa Sól Copper-Silver Project site.
9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.
10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.

11. I am responsible for the preparation or supervision of Sections 16.4 and 25.2.9 of the Technical Report.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.



Peter Stevens, B.Sc., M.Sc. CGeol
Director and Principal Engineering Geologist
CQA International Limited

CERTIFICATE OF AUTHOR

Justin Taylor, P.Eng.

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Justin Taylor, do hereby certify that:

1. I am the President and Director of Micon International Co Limited and a Principal Mechanical Engineer, and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom tel. 0044 (1603) 501 501, e-mail: justint@halyard.ca
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:
B.Eng. (Hons) Mechanical Engineering, University of Pretoria, 1999.
Diploma in Business Administration from The Gordon Institute of Business Science in Johannesburg 2005.
4. I am a member of the Professional Engineers of Ontario (P.Eng.).
5. I have worked in the minerals and processing industry for more than 25 years.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. I am a mechanical engineer and project manager with more than 25 years of experience in the minerals and chemical processing industries since 2001, involving large-scale, multidisciplinary capital engineering projects across North America, South America, Greenland, Africa, Russia, and the Middle East. My experience includes responsibility for the design, development, and execution of mineral processing and materials handling facilities, as well as the management of projects from concept and feasibility through detailed engineering, procurement, construction, and commissioning. I have been involved in international projects since 2004, including projects in remote and logistically challenging environments, and have held senior project management roles delivering studies, turnkey plant supply, and EPC and EPCM project execution. My experience includes leading multidisciplinary engineering teams, coordinating contractors and suppliers, and delivering engineering solutions for mineral processing operations and associated infrastructure, including ore handling and processing systems.
7. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I have not visited the Nowa Sól Copper-Silver Project site.
9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.
10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.
11. I am responsible for the preparation or supervision of Sections 1.18, 18 and 25.2.7 of the Technical Report;

12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.

A handwritten signature in black ink, appearing to read 'Justin Taylor', with a large, sweeping flourish extending to the right.

Justin Taylor, P.Eng.
President, Director and Principal Mechanical Engineer
Micon International Co Limited

CERTIFICATE OF AUTHOR**Becky Humphrey, B.Sc., M.Sc, CEnv, MISEP, MIMMM**

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Becky Humphrey, do hereby certify that:

1. I am employed as Associate Environmental Consultant by and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom. tel. 0044 (1603) 501 501, e-mail: bhumphrey@micon-international.co.uk
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:
 - B.Sc. (Hons) Geography, University of Wales, Aberystwyth 2001.
 - M.Sc. Water Resources, University of Wales, Bangor, 2002.
4. I am a Chartered Environmentalist (CEnv) registered with the Society for the Environment (registration number 10664) and a Full Member of the Institute of Sustainability and Environmental Professionals (MISEP) (registration number 0060192); as well, I am a member in good standing of:
 - The Institute of Materials, Minerals and Mining
5. I have worked as an environmental professional for more than 23 years.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. I am an environmental consultant with over 20 years of experience working in environmental management, predominantly within the minerals industry, including both consultancy and senior operational roles on mine sites. My experience includes environmental and social impact assessments, environmental permitting, and compliance for mining projects, as well as the development and implementation of environmental management systems, monitoring programs, and compliance reporting. I have managed and provided technical input to environmental and social impact assessments, including terms of reference, scoping studies, baseline studies, stakeholder engagement, and environmental and social impact statements, and have undertaken due diligence and gap analysis assignments in support of prefeasibility and feasibility studies. I have worked on projects in Europe, Africa, and the Americas, including operational mine sites, and have experience managing multidisciplinary teams of consultants and contractors and integrating environmental and social considerations into all stages of mining projects.
7. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I visited the Nowa Sól Copper-Silver Project site between July 1, 2025, and July 3, 2025.
9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.

10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.
11. I am responsible for the preparation of Sections 1.20, 4.2.5, 20, 25.2.8 and 26.8 of the Technical Report;
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.



Becky Humphrey, B.Sc., M.Sc, CEnv, MISEP, MIMMM
Associate Environmental Consultant
Micon International Co Limited

CERTIFICATE OF AUTHOR
Christopher Jacobs, CEng., MIMMM, MBA

As the co-author of this Report entitled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026, I, Christopher Jacobs, do hereby certify that:

1. I am employed as a Mineral Economist by Micon International Limited and carried out this assignment for, Micon International Co Limited, Suite 10, Keswick Hall, Norwich, United Kingdom tel. 0044 (1603) 501 501, e-mail: cjacobs@micon-international.com
2. This certificate applies to the technical report prepared for Lumina Metals Corp. titled “NI 43-101 Preliminary Economic Assessment (PEA) Technical Report on the Nowa Sól Copper-Silver Project, Nowa Sól County, Lubuskie Province, Poland” dated April 8, 2026, with an effective date of January 9, 2026.
3. I hold the following academic qualifications:
 - B.Sc. (Hons) Geochemistry, University of Reading, 1980.
 - M.B.A., Gordon Institute of Business Science, University of Pretoria, 2004.
4. I am a Chartered Engineer (CEng) registered with the Engineering Council of the U.K. (registration number 369178); as well, I am a member in good standing of:
 - The institute of Materials, Minerals and Mining
 - The Canadian Institute of Mining, Metallurgy and Petroleum
5. I have worked in the minerals industry for more than 45 years.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. I am a chartered engineer and mineral economist with over 40 years of experience in the minerals industry since 1980, involving the management of mineral exploration, project development, and mining operations, as well as the technical and economic evaluation of mineral properties. My experience includes economic assessment, due diligence, feasibility studies, and independent technical reporting for open pit and underground mining projects across a wide range of commodities, including gold, silver, platinum group elements, nickel, copper, cobalt, iron ore, lead, zinc, tin, chromite, lithium, rare earth elements, and vanadium. Since 2004, I have undertaken numerous mineral property evaluations and NI 43-101 technical reporting assignments and previously held responsibility for the technical and economic evaluation of large-scale mining operations and projects. I have worked on projects in North and South America, Southern and West Africa, Europe, Asia, and the United Kingdom, and have extensive experience supporting investment decisions and project development through economic analysis and valuation of mineral assets.
7. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.
8. I have not visited the Nowa Sól Copper-Silver Project site.
9. This is the first Technical Report I have co-authored for the Nowa Sól Copper-Silver Project that is the subject of this Technical Report, and I have not been previously involved with the Project that is the subject of this report.
10. I am independent of Lumina Metals Corp. and its subsidiaries according to the definition described in NI 43-101 and the Companion Policy 43-101 CP.

11. I am responsible for the preparation or supervision of Sections 1.21, 1.22, 1.19.2, 19.2, 21, 22 and 25.2.10 of the Technical Report;
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Technical Report Dated this 8th day of April, 2026 with an effective date of January 9, 2026.



Christopher Jacobs, CEng, MIMMM
Principal Mineral Economist
Micon International Limited