Mineral Resources and Ore Reserves Report

Sasa Mine, North Macedonia

Central Asia Metals plc

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1.0 Summary

Central Asia Metals plc (CAML) is a metal mining company that produces lead, zinc, and copper with projects and operations in North Macedonia (the Sasa Mine) and Kazakhstan (Kounrad in situ dump leach operation). CAML also owns an 80% interest in CAML Exploration, a subsidiary formed to progress early-stage exploration opportunities in Kazakhstan, and a 32.6% interest in Aberdeen Minerals Ltd, a privately-owned United Kingdom (UK) company focused on the exploration and development of base metals opportunities in northeast Scotland. CAML is a UK public company, and its common shares are listed on the Alternative Investment Market (AIM) of the London Stock Exchange (LSE).

CAML commissioned SLR Consulting (UK) Ltd (SLR) to prepare this Mineral Resources and Ore Reserves Report in accordance with the guidelines of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves developed by the Joint Ore Reserves Committee (the JORC Code 2012) to disclose information about CAML's wholly owned Sasa Mine in North Macedonia (Sasa Mine or the Property). This report includes Mineral Resource and Ore Reserve estimates with an effective date of December 31, 2025.

The Sasa Mine is an underground polymetallic mine located in the northeast of North Macedonia and is operated by Rudnik SASA DOOEL, a wholly owned subsidiary of CAML.

The Sasa Mine comprises the Svinja Reka and Golema Reka lead-zinc-silver deposits which occur as stratiform deposits hosted by quartz-graphite schist and marbles of Lower Palaeozoic age at Svinja Reka and by gneisses at Golema Reka. The Kozja Reka occurrence was historically mined but is not part of the current operation or mine plan. The lenses of mineralisation dip at approximately 35 degrees to the southwest, range in thickness from 2 to 30 m and occur as stacked sheets (typically two or three bodies namely: the hanging wall, central and footwall orebodies), separated by interburden of 1 to 10 m thickness.

Mineralisation is related to intrusions of tertiary volcanics resulting in metasomatic skarn-hydrothermal lead-zinc-silver deposits. Mineralisation includes sphalerite, galena, pyrite and pyrrhotite and varies from coarse to fine grained. Sliver is generally associated with galena.

Mineral Resources are reported for Svinja Reka and Golema Reka, while Ore Reserves are reported for Svinja Reka only. The Kozja Reka occurrence currently has insufficient exploration to declare Mineral Resources.

The Sasa Mine has been in continuous operation since 2006 and produces approximately 800 ktpa of ore with average grades of 3.4% Pb and 2.6% Zn from mining at Svinja Reka.

The deposit is accessed by a recently completed Central Decline, extending 3,735 m from the portal (located in the proximity of the process plant/surface infrastructure) to the 750 mRL. Commissioned in early 2025, the Central Decline was built to provide a more direct route to the orebody at depth, mitigate any risk from hoisting, and provide an additional means of emergency egress. It is now used for all transportation of equipment and personnel to and from the mine and transports 80% of the ore to the surface by truck. The base of the Central Decline currently marks the lowest point of the mine development.

Alternative mine access is provided by the Golema Reka shaft which is used for hoisting the remainder of the ore from the underground.

The Sasa Mine has historically operated using a sub-level cave (SLC) mining method. However, as the orebody narrows with increasing depth, long-hole stoping (LHS) has recently been adopted as the main mining method for the Life of Mine (LOM). Following construction of a paste backfill plant and underground reticulation system in 2023, cut and fill (CF) mining methods are also used in specific areas where additional selectivity is required. Both historical voids and new excavations are backfilled with paste material containing cement tailings.

In addition to the cut and fill project, a dry stack tailings (DST) project has recently been completed, and 30% of the tailings produced by the mine is stored as dry stack, while 40% is used for paste backfill. The remaining 30% is stored as wet tailings in the existing Tailings Storage Facility (TSF).

Mineral processing is undertaken at the processing plant which was commissioned in 2006 and originally designed to process 650 ktpa of ore. The plant was upgraded in 2007 with additional Metso flotation cells to a nameplate capacity of 850 ktpa. The ore is crushed in three stages, and the milling circuit involves two parallel sets of rod mills, spiral classifiers and ball mills to ensure the extracted material is the appropriate size for liberation by flotation at approximately 74 microns.

The processing plant operates both lead and zinc flotation, producing separate concentrates that are filter pressed to generate products containing 5 to 9% moisture. The concentrates are stored in two separate bays before loading into trucks for transportation.

Average process recoveries of around 94% for lead and 86% for zinc are attained by the plant. The lead concentrate contains on average 71% Pb and the zinc concentrate contains on average 50% Zn. The lead concentrate includes a credit for silver.

The concentrates are sold via an annual rolling hybrid offtake contract with Traxys Europe S.A. (Traxys). Silver in the concentrate is subject to a streaming agreement with OR Resources Inc whereby CAML receives US$6.665 per ounce for silver.

The LOM plan runs for 9 years, and the mining rate is planned to increase to 830 ktpa for the next 8 years before declining in year 9 to around 700 ktpa. Average grades in the LOM plan are 3.5% Pb and 2.5% Zn.

This report was prepared under the supervision of Competent Persons (CPs) named herein. The conclusions, recommendations, and forward-looking statements made by the CPs are based on reasonable assumptions and results interpretations. Forward-looking statements cannot be relied upon to guarantee the Property’s performance or outcomes and naturally include inherent risks and risks relating to the mining industry.

1.1 Property Description and Location

The Property is located in the northeast of North Macedonia, near the border with Bulgaria, and approximately 150 km east of the capital city of Skopje. The nearest town is Makedonska Kamenica, approximately 10 km to the south, and provides a highway conduit between the mine and the region for staff and supplies coming in, and export of concentrate products. The village of Sasa is located 4 km south of the operation with the nearest residential buildings 0.5 km from the operation.

The Sasa Mine is centred on the geographic coordinates of latitude 42°6’44” North and longitude 22°31’10” East.

The Property has an Exploitation Concession (24-5550/1) that was granted to Rudnik SASA DOOEL on November 13, 2014, and is valid until September 28, 2030. The Exploitation Concession was expanded in February 2026 from an area of 4.22 km^2 to approximately 6.87 km^2. Rudnik SASA DOOEL can apply to extend the concession beyond the current expiry date.

1.2 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Sasa Mine is accessed by the A4 and A3 highways from Skopje to Makedonska Kamenica, a distance of 130 km, and then by a minor road (paved) for a distance of 10 km. The A3 highway continues eastwards from Makedonska Kamenica into Bulgaria via the border crossing at Stanke Lisichkovo before continuing northeast to Sofia, a total distance of 160 km. There are international airports at Skopje and Sofia and the closest railhead is 29 km to the southwest at the town of Kočani and is accessed by the A3 highway.

The climate is subject to continental and Mediterranean influences, with hot dry summers and cold winters. The majority of rainfall is during the spring and autumn and occasionally there are heavy snowfalls on the mountains, rarely in the valleys, from December to February.

The Sasa Mine is located at approximately 1,000 masl in a narrow V-shaped valley of the Kamenicka River with associated mine access portals, processing plant, maintenance workshop, office and TSF. Underground workings are to the north of the processing plant predominantly below Svinja Reka. The topography within the concession area ranges from 800 to 2,030 masl.

The Sasa Mine operates year-round, 24 hours a day, 7 days a week with 3 x 8 hour working shifts in both the underground mine and process facilities. Most of the mine staff are from the local area and reside in the town of Makedonska Kamenica with a population of approximately 4,500 and the surrounding region. Total staff numbers are approximately 700.

1.3 History

Initial exploration comprising geological surveys of the Osogovo ore-bearing massif and the Sasa locality date from 1954. Exploration continued between 1954 and 1960 with construction of the mine between 1960 and 1965. In November 1965, the underground mine was opened for trial processing with a production capacity of 0.3 Mtpa of lead-zinc ore. The Sasa Mine started commercial operations in 1966 as a state-owned entity. During the 1990s, ore production at Sasa was approximately 0.5 Mtpa. In 2002, the mine closed due to lack of operating capital.

In 2005, Solway Investments Group (Solway) acquired the mine, and operations were restarted in 2006 following investment and improvements, particularly within the plant which was re-equipped with a modern flotation circuit allowing production of separate zinc and lead concentrates instead of a bulk concentrate.

On November 3, 2015, Solway announced that it had sold the Sasa Mine to Orion Mine Finance Group (Orion) partnered with Fusion Capital AG through its wholly owned subsidiary Lynx Europe SPLLC Skopje (Lynx).

In November 2017, CAML acquired the Sasa Mine from Lynx and took 100% ownership.

A summary of the production history of the Sasa Mine from 2008 to 2025 is shown in Table 1-1.

Table 1-1: Sasa Production History from 2008 to 2025

Item	Unit	2008	2009	2010	2011	2012	2013	2014	2015	2016
Processing Plant										
Ore Processed	kt	854	865	811	758	754	774	780	777	779
Pb Grade	% Pb	4.68	4.43	4.05	3.83	3.93	4.13	4.16	4.04	3.95
Zn Grade	% Zn	4.08	4.14	3.81	3.43	3.35	3.47	3.48	3.52	3.41
Lead Concentrate										
Pb Concentrate	kt	49.1	47.6	41.3	37.1	38.0	41.0	41.6	40.2	39.5
Pb Recovery	%	91.6	94.2	94.4	95.1	94.4	94.4	94.5	94.1	94.1
Pb Grade	% Pb	74.5	75.7	75.2	74.3	73.6	73.6	73.7	73.5	73.3
Pb in Concentrate	kt	36.6	36.1	31.0	27.6	28.0	30.2	30.7	29.5	29.0
Zinc Concentrate										
Zn Concentrate	kt	58.0	61.0	52.8	44.6	43.1	46.2	46.9	47.2	45.5
Zn Recovery	%	82.3	85.5	86.0	86.6	86.2	86.3	86.5	85.8	84.6
Zn Grade	% Zn	49.5	50.2	50.4	50.6	50.5	50.1	50.1	49.8	49.4
Zn in Concentrate	kt	28.7	30.6	26.6	22.5	21.8	23.3	23.5	23.5	22.5
Item	**Unit**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**
Processing Plant										
Ore Processed	kt	793	805	821	820	831	807	806	761	799
Pb Grade	% Pb	3.98	3.90	3.77	3.85	3.52	3.63	3.70	3.71	3.35
Zn Grade	% Zn	3.18	3.31	3.29	3.37	3.14	3.15	2.97	2.87	2.61
Lead Concentrate										
Pb Concentrate	kt	40.8	40.3	40.4	41.3	37.9	38.4	39.1	37.6	35.6
Pb Recovery	%	94.6	93.6	94.5	94.3	93.1	93.4	93.1	94.4	94.1
Pb Grade	% Pb	72.9	73.6	72.3	72.0	71.8	71.2	71.0	70.8	70.6
Pb in Concentrate	kt	29.9	29.4	29.2	29.7	27.2	27.4	27.8	26.6	25.2
Zinc Concentrate										
Zn Concentrate	kt	43.7	46.1	47.1	47.6	44.4	42.8	40.2	37.0	35.6
Zn Recovery	%	85.5	84.6	86.5	86.1	84.9	84.6	85.0	85.2	85.7
Zn Grade	% Zn	49.4	48.9	49.6	50.0	49.9	50.1	50.6	50.2	50.2
Zn in Concentrate	kt	21.6	22.5	23.4	23.8	22.2	21.5	20.3	18.6	17.9

1.4 Geology and Mineralization

The Sasa deposits are located within the Serbo-Macedonian Massif, a region that hosts numerous lead and zinc deposits, stretching across Serbia, North Macedonia, Bulgaria, eastern Greece and into Turkey.

Mineralisation is interpreted as a structurally influenced replacement-style lead-zinc-silver skarn-hydrothermal system controlled by a combination of favourable host lithologies, bedding-parallel replacement zones, and later structural modification. The mineralised lenses are therefore not interpreted as purely stratiform bodies, but as lithologically favourable and structurally modified replacement zones affected by local faulting, shearing and competency contrasts between schist, marble and gneissic host rocks. The deposits consist of well-defined lenses of lead-zinc-silver mineralisation, dipping at approximately 35° and ranging in thickness from 2 to 30 m.

Mineralisation is hosted mainly within Lower Palaeozoic quartz-graphite schists and marbles at Svinja Reka, while at Golema Reka it is primarily developed within gneissic host rocks. Three principal stages of mineralisation have been identified. The first stage comprised of a metasomatic skarn where mineralisation developed mainly through infiltration of mineralising fluids into favourable host rocks. The second stage involved retrograde hydrothermal alteration, during which earlier skarn minerals were progressively altered and overprinted, accompanied by the main phase of sulphide deposition. This is interpreted to reflect mixing between magmatic and meteoric fluids within the evolving hydrothermal system. The final stage involved emplacement of post-ore gangue minerals, dominated largely by carbonate phases precipitated from cooler and relatively diluted hydrothermal fluids during the waning stages of the mineralising event.

1.5 Exploration and Drilling

The main period of discovery exploration in the Osogovo Mountains, including the area comprising the Sasa Mine, was between 1954 and 1960 during which geophysics, mapping, trenching, and drilling from both surface and initial underground excavations were undertaken.

Subsequently, as the underground workings were extended, ongoing exploration comprised drilling from underground, complimented with underground channel sampling. Surface drilling continued to a lesser extent to support the underground exploration.

All drilling completed at the Sasa Mine is by diamond coring methods. From 1954 to 2002, a total of 274 drillholes for a total of 20,597 m (262 from underground and 12 from surface) were completed at Svinja Reka plus a further 82 drillholes (58 from underground and 24 from surface) were completed at Golema Reka.

From 2006 to 2015 (Solway), a total of 1,080 drillholes for 80,737 m were completed at Svinja Reka (1,048 from underground and 32 from surface), and 22 underground drillholes for 1,503 m were completed at Golema Reka.

From 2015 to 2017, under ownership of Lynx, a total of 228 drillholes were completed at Svinja Reka.

Since taking ownership of the Sasa Mine in November 2017, and similar to the previous operators, CAML has continued to explore the Svinja Reka, Golema Reka, and to a lesser extent Kozja Reka zones. To date, CAML has completed a total of 1,497 drillholes for (101,393 m) at Svinja Reka and 16 drillholes at Golema Reka (6,702 m). The majority of this drilling is from underground.

CAML has a 3-year (2026 – 2028) drill exploration programme of approximately 9,000 m per year. Most of the drilling will be from underground but a small number of deep (circa 800 m) surface drillholes are also planned.

1.6 Mineral Processing and Metallurgical Testing

The ore metallurgy at Svinja Reka has been consistent since 2008, despite declining head grades in recent years, with lead and zinc recoveries of around 94% and 86%, respectively. Concentrate grades are approximately 71% Pb and 50% Zn and silver recovery to the lead concentrate is 75%.

The ore is getting harder as the mine deepens. Maintaining the current grind size at increasing ore hardness presents the main bottleneck to increasing plant throughput, however, is preferable to using a coarser grind size which may affect recoveries.

Further optimisation of performance is possible, by balancing throughput with ore hardness, grind size and metals recovery and concentrate grades to achieve the best overall economics.

Initial pre-concentration testwork using ore sorting indicates encouraging results and, on completion of the full testwork programme, should be reviewed and an options study conducted to determine the best overall strategy for potential pre-concentration.

1.7 Mineral Resource Estimate

The Mineral Resource estimate for the Sasa Mine is classified in accordance with the JORC Code (2012).

Mineral Resources are reported for Svinja Reka and Golema Reka deposits. The Kozja Reka occurrence was historically mined; however, insufficient exploration has been undertaken to declare Mineral Resources.

A summary of the Mineral Resource estimate as of December 31, 2025 is shown in Table 1-2.

The stated Mineral Resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, to the best knowledge of the authors. There are no known mining, metallurgical, infrastructure, or other factors that materially affect this Mineral Resource estimate, at this time.

Table 1-2: Audited Mineral Resource Statement for the Sasa Mine (Effective Date of December 31, 2025)

Deposit	Classification	Tonnes (Mt)	Grades			Contained Metal		
			Pb (%)	Zn (%)	Ag (g/t)	Pb (kt)	Zn (kt)	Ag (koz)
Svinja Reka	Measured	-	-	-	-	-	-	-
	Indicated	9.6	4.2	3.0	32.8	404	290	10,100
	Inferred	2.3	2.9	2.4	35.5	68	56	2,662
Golema Reka	Measured	-	-	-	-	-	-	-
	Indicated	1.8	4.1	1.3	13.8	75	24	810
	Inferred	6.8	3.9	1.2	13.2	263	82	2,880
Total	**Measured**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Indicated	**11.4**	**4.2**	**2.8**	**29.8**	**479**	**314**	**10,910**
	Inferred	**9.1**	**3.6**	**1.5**	**18.9**	**331**	**138**	**5,541**

Notes:

1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. The Competent Person is Jason Ché Osmond, an independent Competent Person as defined by JORC.
3. Mineral Resources are estimated using NSR cut-off values of US$53 per tonne for sub-level caving, US$65 per tonne for cut and fill stoping and $60 per tonne for long-hole stoping.
4. Mineral Resources are estimated using metal prices of US$3,041 per tonne for zinc, US$2,506 per tonne for lead and US$31 per ounce for silver.
5. Mineral Resources are estimated using metallurgical recoveries of 94% for lead, 82% for zinc and 75% for silver.
6. Mineral Resources are estimated using average dry densities ranging from 2.7 to 4.0 t/m^3.
7. A minimum thickness of 1.5 m was used to model the mineralised zones.
8. Contained metal refers to estimated contained metal in the ground not adjusted for metallurgical recovery.
9. Mineral Resources are inclusive of Ore Reserves.
10. Mineral Resources that are not Ore Reserves do not have demonstrated economic viability based on a pre-feasibility study or feasibility study.
11. Totals may not represent the sum of the parts due to rounding.

1.8 Ore Reserve Estimate

The Ore Reserve estimate for the Sasa Mine is classified in accordance with the JORC Code (2012). Ore Reserves are derived from Measured or Indicated Mineral Resources after applying economic parameters to calculate that portion of the Measured and Indicated Mineral Resources that can be mined economically. Ore Reserves for the Sasa Mine were classified using the following criteria:

- No Proved Ore Reserves were classified because no Measured Mineral Resources were estimated.
- Probable Ore Reserves are those Indicated Mineral Resources where development work for mining and information on processing/metallurgy and other relevant factors demonstrate that economic extraction is achievable.

Ore Reserves are inclusive of diluting material that will be mined in conjunction with the economically mineralised rock and delivered to the processing plant or equivalent facility. The term 'Ore Reserve' need not necessarily signify that extraction facilities are in place or operative, or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Ore Reserves are reported for the Svinja Reka deposit only. No Ore Reserves are reported for Golema Reka because no Pre-Feasibility Study or Feasibility Study has been undertaken.

The audited Ore Reserve statement for the Sasa Mine is shown in Table 1-3.

The stated Ore Reserves are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, to the best knowledge of the authors. There are no known mining, metallurgical, infrastructure, or other factors that materially affect this Ore Reserve estimate, at this time.

Table 1-3: Audited Ore Reserve Statement for the Sasa Mine (Effective Date of December 31, 2025)

Deposit	Classification	Tonnes (Mt)	Grades			Contained Metal		
			Pb (%)	Zn (%)	Ag (g/t)	Pb (kt)	Zn (kt)	Ag (koz)
Svinja Reka	Proved	-	-	-	-	-	-	-
	Probable	6.9	3.5	2.5	26.1	244	170	5,782
Golema Reka	Proved	-	-	-	-	-	-	-
	Probable	-	-	-	-	-	-	-
Total	**Proved**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Probable	**6.9**	**3.5**	**2.5**	**26.1**	**244**	**170**	**5,782**

Notes:

1. Ore Reserves are reported in accordance with the guidelines of the JORC Code (2012).
2. The Competent Person is Colin Davies, an independent Competent Person as defined by JORC.
3. Ore Reserves are estimated using NSR cut-off values of US$53 per tonne for SLC, US$65 per tonne for CF and $60 per tonne for LHS.
4. Ore Reserves are estimated using metal prices of US$2,644 per tonne for zinc, US$2,179 per tonne for lead and US$27 per ounce for silver.
5. Ore Reserves are estimated using metallurgical recoveries of 94% for lead, 82% for zinc and 75% for silver.
6. A minimum mining width of 1.8 m, including internal but excluding external dilution, has been used.
7. Contained metal refers to estimated contained metal in the ground not adjusted for metallurgical recovery.
8. Totals may not represent the sum of the parts due to rounding.

1.9 Mining Methods

Current production is located within the Svinja Reka deposit, which produces approximately 800 ktpa of ore, with average grades of 3.4% Pb and 2.6% Zn.

The main access to the underground workings is through the Central Decline, which extends a distance of 3,735 m from a portal located adjacent to the process plant to a depth of 750 mRL. Secondary access is provided by the Golema Reka shaft and the Level 14b portal. As of H1 2026, approximately 75% of the ore is hauled by truck and 25% via the shaft, with shaft haulage scheduled to be phased out.

Historically, the mine operated using sub-level caving with 7.0 m sublevel spacing. As the orebody narrows with depth, the suitability of this method has declined. Since 2023, a transition to long hole stoping (LHS) with backfill has been ongoing as the primary mining method. Cut and fill mining (CF) is also used in areas requiring greater selectivity or with difficult ground conditions. Both historical voids and newly mined stopes are backfilled using cemented tailings.

The Sasa Mine uses conventional trackless underground mining equipment. Ventilation is based on a mechanical overpressure system that supplies fresh air to active mining areas and exhausts return air through a network of raises and airways. Two principal fans are installed, and fresh air is supplied primarily via the decline portals.

The current LOM plan extends over approximately nine years. Production is expected to increase to around 830 ktpa over the next three years, before reducing towards the final years. Average planned grades over this period are 3.5% Pb and 2.5% Zn. Mining is split between long hole stoping (67%), cut and fill (17%) and sub-level caving (15%).

The LOM schedule for the Sasa Mine is shown in Table 1-4.

Table 1-4: Sasa Life of Mine Schedule

Item	Unit	Total LOM	2026	2027	2028	2029	2030	2031	2032	2033	2034
Ore Reserves											
Proved Ore Tonnes	kt	**-**	-	-	-	-	-	-	-	-	-
Pb Grade	% Pb	**-**	-	-	-	-	-	-	-	-	-
Zn Grade	% Zn	**-**	-	-	-	-	-	-	-	-	-
Ag Grade	g/t Ag	**-**	-	-	-	-	-	-	-	-	-
Probable Ore Tonnes	kt	**6,882**	815.2	799.5	797.0	780.5	767.8	821.7	768.3	698.3	633.1
Pb Grade	% Pb	**3.55**	3.65	3.77	3.81	3.63	3.42	3.81	3.85	3.43	2.30
Zn Grade	% Zn	**2.47**	2.85	2.57	2.55	2.52	2.99	2.39	2.34	2.45	1.35
Ag Grade	g/t Ag	**26**	21	25	26	28	26	27	29	28	26
Total Ore Tonnes	kt	**6,882**	815.2	799.5	797.0	780.5	767.8	821.7	768.3	698.3	633.1
Pb Grade	% Pb	**3.55**	3.65	3.77	3.81	3.63	3.42	3.81	3.85	3.43	2.30
Zn Grade	% Zn	**2.47**	2.85	2.57	2.55	2.52	2.99	2.39	2.34	2.45	1.35
Ag Grade	g/t Ag	**26**	21	25	26	28	26	27	29	28	26
Inferred Mineral Resources											
Inferred Resources	kt	**445.8**	0.4	30.7	33.1	49.8	62.8	8.4	61.7	127.0	72.0
Pb Grade	% Pb	**2.46**	2.31	2.26	2.25	3.03	3.88	2.56	1.97	2.11	2.04
Zn Grade	% Zn	**2.87**	2.39	2.74	2.77	3.73	5.06	2.69	2.07	2.54	1.77
Ag Grade	g/t Ag	**22**	22	16	17	20	19	23	22	22	28

Notes:

1. Ore tonnes are Probable Ore Reserves as presented in Section 14 of this report.
2. Mining losses and mining dilution applied as detailed in Section 14 of this report.
3. Totals may not represent the sum of the parts due to rounding.

1.10 Recovery Methods

The Sasa processing plant was commissioned in 2006 and was originally designed to process 650 ktpa of ore. The plant was upgraded in 2007 with additional Metso flotation cells to a nameplate capacity of 850 ktpa of ore. However, to match the mine production rate, the plant operates at throughputs less than this and, in recent years, has processed approximately 800 ktpa.

The process plant comprises a three-stage crushing circuit and conventional grinding, classification and selective flotation circuits to produce lead and zinc concentrates that are thickened and filtered for transportation by truck. A paste backfill plant and dry stack plant were constructed and commissioned in 2023 and 2025, respectively. For 2026, it is planned that circa 31% of the tailings will be dry stacked, 36% will be used for paste backfill and 33% stored in the existing TSF-4. Paste backfill is the priority method for tailings disposal.

Design lead and zinc recoveries are excellent at 94% and 86% respectively. Silver is recovered to both the lead and zinc concentrates but is payable only in the lead concentrate. Silver recovery to the lead concentrate is 75% to a grade of 254 g/t Ag. Current and historical metallurgical performance is very consistent and aligns well with design values, despite the trend of lower head grades in recent years. These lower head grades are reflected in the LOM plan.

Metallurgical recoveries used in the LOM plan from 2026 to 2034 are 94% for lead, 86% for zinc and 75% for silver. The total LOM tonnes treated are 7.327 Mt at 3.48% Pb, 2.50% Zn and 26 g/t Ag. Average throughput is 830 ktpa for the period 2027 to 2033, with 815 ktpa in 2026 and 705 ktpa in 2034.

1.11 Project Infrastructure

Infrastructure associated with the Sasa Mine includes:

- Svinja Reka underground mine (operating).
- Golema Reka underground mine (non-operational).
- Processing plant, dry stack tailings plant and paste backfill plant.
- Tailings storage facility and dry stack storage facility (landform).
- Site offices, staff facilities, laboratory, stores, and maintenance workshops.

High voltage (HV) power from the state grid to the underground mine is fed at 35 kV to three transformers where the voltage is stepped down to 6 kV with a capacity of 4 MVA (Megavolt-Ampere).

The process water intake is situated upstream of the confluence of the Kamenica River with the Kozje River and is used for the flotation circuit. Mine water and return dry stack tailings (DST) water are used for the grinding circuit.

The Kamenica River is captured in a concrete diversion tunnel, constructed beneath the TSF, and exits through the western abutment of TSF-4. A diversion channel also exists on the eastern side for the Petrova River.

The Sasa TSFs have successively developed downstream from TSF-1, TSF-2, TSF-3.1, TSF-3.2 to TSF-4 in the steep-sided valley of the Kamenica River. Except for TSF-4, all the other TSFs are inactive with TSF-1 and TSF-3.1 rehabilitated with soil cover and vegetation, TSF-2 and part of TSF-1 are used for construction of the DST, and TSF 3.2 is in the closure phase and is being capped and remediated. TSF-4 was designed by the Faculty of Engineering, Skopje, in March 2015 to international standards, and external reviews are carried out annually by Knight Piésold Ltd (Knight Piésold) to Global Industry Standard on Tailings Management (GISTM) standards.

The paste backfill plant and DST plant were commissioned in 2023 and 2025, respectively.

All major infrastructure required by the operation is currently in place and no significant additional infrastructure is planned.

1.12 Market Studies and Contracts

Both lead and zinc concentrates, are sold via an annual rolling hybrid offtake contract with Traxys Europe S.A. (Traxys), to the regional and international well diversified customer base. The commercial terms of the contracts with final customers are negotiated on an annual basis based on the prevailing market conditions with the off taker assistance.

The payable silver contained in the lead concentrate is subject to a life of mine streaming agreement with OR Resources Inc. With the assistance of the broker, on monthly basis CAML buys and delivers the physical silver to OR Resources Inc., which is the equivalent of the payable silver sold during the calendar month to final customers. CAML receives US$6.665 per ounce for silver.

1.13 Environmental Studies, Permitting and Social or Community Impact

The Sasa Mine is well managed with regards to the environmental and social aspects of its operations. There is an ISO 9001 accredited Integrated Management System in place, including an ISO14001 accredited environmental management system, and an ISO 18001 accredited health and safety management system.

The minimum environmental monitoring and reporting requirements for the site are described in the IPPC (A) permit for the mine. Rudnik SASA DOOEL undertakes additional monitoring above these requirements, and all monitoring data is recorded on a database.

Rudnik SASA DOOEL's community engagement and support activities are well governed, with clear controls and criteria for investment requests. The dependency of the local community on the mine is clearly acknowledged, and projects are underway to support diversification of the economy away from mining.

A Mine Closure Plan was developed by Golder Associates (Golder) in 2022.

1.14 Capital and Operating Costs

1.14.1 Capital Costs

A summary of the capital costs for the Sasa Mine is presented in Table 1-5. Total capital costs are estimated at US$75.7 million. A cost of US$61 million is also estimated for closure.

Table 1-5: Sasa Mine Capital Costs (US$M)

Item		2026	2027	2028	2029	2030	2031	2032	2033	2034
Mining	Underground Development	4.66	3.73	4.06	2.92	1.16	0.60	0.66	0.36	0.09
	Raise Boring	1.43	0.58	-	-	-	-	-	-	-
	Underground Mine	4.61	4.14	4.22	4.92	3.52	4.44	3.28	3.23	0.66
	Other Mining	0.18	0.10	-	-	-	-	-	-	-
Process Plant		1.64	1.77	1.56	1.52	1.16	1.24	1.14	1.08	1.16
Surface Infrastructure		0.13	0.16	0.10	0.10	0.16	0.16	0.17	0.16	0.10
Environment & Corporate, Social Responsibility		0.36	0.51	0.21	0.21	0.26	0.16	0.31	0.21	0.21
Other		2.02	3.57	0.57	-	-		-	-	-
Total Capital Costs		**15.04**	**14.56**	**10.73**	**9.67**	**6.26**	**6.60**	**5.56**	**5.04**	**2.23**

1.14.2 Operating Costs

A summary of the annual operating costs for the Sasa Mine is provided in Table 1-6. The total operating cost for the LOM is estimated at US$419.3 million.

Table 1-6: Sasa Mine Operating Costs (US$M)

Item	2026	2027	2028	2029	2030	2031	2032	2033	2034
Mining	28.68	27.26	26.92	26.40	24.34	24.77	24.42	24.63	19.77
Backfill Plant	3.09	3.83	3.29	3.59	4.20	3.56	3.56	3.29	3.67
Backfill Underground Reticulation	0.60	0.73	0.75	0.89	0.94	0.91	0.88	0.80	0.96
Dry Stack Tailings Plant	0.79	0.44	0.50	0.46	0.37	0.39	0.39	0.46	0.36
Dry Stack Tailings Landform	1.24	0.56	0.66	0.62	0.52	0.48	0.48	0.57	0.52
TSF	0.60	0.24	0.24	0.22	0.16	0.34	0.32	0.45	0.19
Processing	12.49	11.93	11.34	11.29	11.29	11.18	11.23	11.16	9.56
G&A	4.62	4.56	4.50	4.55	4.46	4.38	4.40	4.39	2.64
Total Operating Costs	**52.10**	**49.55**	**48.21**	**48.02**	**46.27**	**46.01**	**45.68**	**45.75**	**37.68**

1.15 Economic Analysis

The financial model for the LOM was reviewed by the authors and showed the Ore Reserves to be economic based on the assumptions used and metal prices of US$2,644 per tonne for zinc, US$2,179 per tonne for lead, and US$27 per ounce for silver.

A silver price of US$6.665 per ounce was also used in the financial model to reflect the value attributable to CAML based on the silver streaming agreement with OR Resources Inc. The Ore Reserves were also shown to be economic at this price.

1.16 Interpretation and Conclusions

The CPs have reviewed the licensing, geology, exploration, Mineral Resources and Ore Reserve estimation methods, mining, mineral processing, infrastructure requirements, environmental, permitting, social considerations and financial information.

The CPs consider the Mineral Resources and Ore Reserves for the Sasa Mine have been prepared to industry best practice and are reported in accordance with the guidelines of the JORC Code (2012).

1.17 Recommendations

The CPs make the following recommendations for the respective study areas.

1.17.1 Geology, Exploration and Mineral Resource Estimates

- Although the majority of the Mineral Resource is informed by recent drilling, some historical data still exists within the database that has less confidence in location accuracy and is less well supported by QA/QC protocols. Consideration should be given to removal of historical data in areas where more recent drilling is sufficient to support the estimate.
- Increase the collection of density measurements at all deposits to obtain a better understanding of the behaviour of density across lithologies and mineralized domains. As the mine development deepens it is understood that the ore is becoming harder and may therefore vary in density.
- The QA/QC processes would benefit from additional blank samples (fine and coarse), as well as additional external check analysis at an independent laboratory, to increase

support for the data being provided by the Sasa laboratory. In addition, it is recommended that the laboratory pursues accreditation for AA analysis used for exploration samples.

- The Sasa laboratory reports a backlog of samples that, even with prioritisation, may lead to samples not being processed in sufficient time. This is particularly critical to production samples that are required to inform mining activities and measures should be considered to alleviate this situation. Laboratory turnaround performance should continue to be monitored through operational KPIs, and external laboratory support may be used during periods of increased sample volume, maintenance downtime or backlog risk.
- The current Mineral Resource is based on evaluation using a NSR cut-off value. SLR considers that the estimate should also apply the use of Mineable Shape Optimisation (MSO) to define the Mineral Resources.

1.17.2 Mining and Ore Reserve Estimates

- Grade control should be improved and better aligned with production scheduling with increased availability of drill rigs to reduce conflicts with development activities, along with shortened assay turnaround times for more effective grade control and short-term planning.
- Geological reconciliation should continue to compare Mineral Resource model predictions against stope production, grade control samples, mined tonnes and plant performance. Reconciliation practices would benefit from increased resolution (current reporting at a 50 m by 50 m panel scale) to capture local grade distribution and provide better defined stope level detail.
- Enhance ongoing analysis of hanging wall stability to better determine achievable unsupported spans.
- Short term sequencing is being adapted to maintain production by targeting additional CF stopes where access to LHS areas is not progressing as planned though this may complicate the overall LOM sequencing. However, as the mine improves its LHS operation alignment with long term sequencing it is expected become more consistent.
- The planned increase in production from approximately 800 ktpa to 830 ktpa is reliant on the effectiveness of LHS stoping. Operational performance in LHS areas is improving as experience with the method increases.

1.17.3 Mineral Processing and Recovery Methods

- Continued metallurgical testwork is recommended to monitor and balance throughput, grind size, metals recovery and concentrate grades for optimal economics. In addition, regular testing and monitoring of ore hardness is recommended as mining progresses at depth.
- The initial pre-concentration testwork programme using ore sorting indicates encouraging results and, on completion of the full testwork programme, should be reviewed and an options study conducted to determine the best overall strategy for potential pre-concentration.

1.17.4 Infrastructure

- Testing should continue to investigate the potential for recycling TSF-4 supernatant water back to the process plant and reduce the fresh (river) water make-up required for the flotation circuit.

- Current monitoring of the TSFs and water discharge streams should continue, aligned with on-going support from Knight Piésold, and implementing any further recommendations that may ensue. The beach length and level of supernatant water in TSF-4 should be monitored to maintain the minimum freeboard and beach length, to ensure no overspill can occur.

1.17.5 Environmental Studies, Permitting and Social or Community Impact

- Consider Good International Industry Practice (GIIP) with regards to Biodiversity Net Gain (BNG) during expansion works for the DST and any other surface developments where vegetation is cleared.
- Continue ongoing monitoring programmes of the TSF and external reviews by Knight Piésold. Implement any further recommendations to mitigate risks to the environment and local communities.
- Continue with the current environmental monitoring regime, updating in line with IPPC Permit revisions, and international best practice.

2.0 Introduction

Central Asia Metals plc (CAML) is a metal mining company that produces lead, zinc, and copper with projects and operations in North Macedonia (the Sasa Mine) and Kazakhstan (Kounrad in-situ dump leach operation). CAML also owns an 80% interest in CAML Exploration, a subsidiary formed to progress early-stage exploration opportunities in Kazakhstan, and a 32.6% interest in Aberdeen Minerals Ltd, a privately-owned United Kingdom (UK) company focused on the exploration and development of base metals opportunities in northeast Scotland. CAML is a UK public company, and its common shares are listed on the Alternative Investment Market (AIM) of the London Stock Exchange (LSE). CAML owns a 100% interest in the mineral rights for the Sasa Mine through Rudnik SASA DOOEL, a wholly owned subsidiary. CAML acquired the Property from Lynx Resources Ltd in 2017.

The Property is an underground lead-zinc-silver mining operation. The Sasa Mine has been in continuous operation since 2006 and uses a combination of sub-level caving (SLC), cut and fill (CF) and long-hole stoping (LHS) mining methods to extract ore. Mineral processing is undertaken on the surface at a processing plant where the ore is processed by conventional flotation into separate zinc and lead concentrates. Approximately 800 ktpa of ore are processed and produces around 40 ktpa of lead concentrate and 45 ktpa of zinc concentrate.

2.1 Terms of Reference

The scope of work included a review of the technical information used to derive the Mineral Resources and Ore Reserves estimates for the Sasa Mine with an effective date of December 31, 2025. The scope included a site visit to the Property.

The objective of the review was to provide an independent opinion on the Mineral Resources and Ore Reserves of the Sasa Mine and prepare a Mineral Resources and Ore Reserves report in accordance with the guidelines of the JORC Code (2012).

2.2 Competent Persons and Site Visits

The following Competent Persons (CPs) from SLR have reviewed the Mineral Resource and Ore Reserve estimates and supervised the production of this report:

- Jason Ché Osmond, BSc, MSc, MCSM, FGS, CGeol, EurGeol, Technical Director (Geology and Mineral Resources).
- Colin Davies, BEng, MSc, CEng, ACSM, MIMMM, QMR, Associate Director (Mining Engineering).
- James Turner, BSc, MSc, CEng, ACSM, MCSM, MIMMM, Technical Director (Mineral Processing).
- Alessandra (Alex) Pheiffer, BSc, MSc, PrSciNat, MIMMM, Technical Director (Environmental and Social).

These consultants by virtue of their education, experience, and professional association, are considered to be independent CPs according to the JORC Code (2012) and are members in good standing of appropriate professional institutions.

The CP responsible for the Mineral Resource estimate is Ché Osmond.

The CP responsible for the Ore Reserve estimate is Colin Davies.

A site visit to the Sasa Mine was undertaken by Ché Osmond, Colin Davies and James Turner from May 13 to 14, 2026.

2.3 Declaration

SLR has provided the mineral industry with specialised geological, mining engineering, mineral processing, infrastructure, environmental and social, and project economics expertise since 1994. SLR's experience is worldwide and has been developed in the industrial minerals and metalliferous mining sectors.

SLR is not an insider, associate or affiliate of CAML. SLR neither has nor holds:

- Any rights to subscribe for shares in CAML either now or in the future;
- Any vested interests in any mining or exploration concessions (licences) held by CAML;
- Any rights to subscribe to any interests in any of the licences held by CAML either now or in the future;
- Any vested interests in either any licences held by CAML or any adjacent licences; or
- Any right to subscribe to any interests or licences adjacent to those held by CAML, either now or in the future.

The results of the technical review by SLR are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understanding concerning any future business dealings.

2.4 Basis of Report

The information, conclusions, opinions, and estimates contained in this report are based on:

- Data, reports, and other information provided by CAML.
- A site visit conducted on May 13 to 14, 2026.
- Discussions with CAML personnel.

The Competent Persons have reviewed the information and used all means necessary in their professional judgement to verify it and have no reasons to doubt its reliability and have determined it adequate for the purposes of this report. The authors do not disclaim any responsibility for the information contained herein.

2.5 Forward Looking Information

This report contains "forward-looking information" and "forward-looking statements" within the meaning of applicable law or the Australian Securities Exchange Listing Rules which involve a number of risks and uncertainties. Forward-looking information and forward-looking statements include, but are not limited to, statements with respect to the future prices of zinc, lead and silver, the estimation of mineral resources and reserves, the realization of mineral estimates, the timing and amount of estimated future production, costs of production, capital expenditures, costs (including capital costs, operating costs and other costs) and timing of the LOM, rates of production, annual revenues, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

Often, but not always, forward-looking statements can be identified by the use of words such as "plans", "expects", or "does not expect", "is expected", "budget", "scheduled", "estimates", "forecasts", "intends", "anticipates", or "does not anticipate", or "believes", or variations of such words and phrases or state that certain actions, events or results "may", "could", "would", "might" or "will" be taken, occur or be achieved.

Forward-looking statements are based on the opinions, estimates and assumptions of contributors to this report. Certain key assumptions are discussed in more detail. Forward looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements of CAML to be materially different from any other future results, performance or achievements expressed or implied by the forward-looking statements.

Such factors include, among others: the actual results of current development activities; conclusions of economic evaluations; changes in project parameters as plans continue to be refined; future prices of zinc, lead and silver; possible variations in ore grade or recovery rates; failure of plant, equipment or processes to operate as anticipated; accidents, labour disputes and other risks of the mining industry delays in obtaining governmental approvals or financing or in the completion of development or construction activities; shortages of labour and materials, the impact on the supply chain and other complications associated with pandemics, as well as those risk factors discussed or referred to in this report and in CAML's documents filed from time to time with the securities regulatory authorities in Australia.

There may be other factors than those identified that could cause actual actions, events or results to differ materially from those described in forward-looking statements, there may be other factors that cause actions, events or results not to be anticipated, estimated or intended. There can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. Accordingly, readers are cautioned not to place undue reliance on forward-looking statements. Unless required by securities laws, the authors undertake no obligation to update the forward-looking statements if circumstances or opinions should change.

2.6 Units and Abbreviations

All units of measurement are reported in the Système Internationale d'Unités (SI), as utilised by international mining industries, including: metric tonnes (tonnes, t), million metric tonnes (Mt), kilograms (kg) and grams (g) for weight; kilometres (km), metres (m), centimetres (cm) or millimetres (mm) for distance; cubic metres (m^3), litres (l), millilitres (ml) or cubic centimetres (cm^3) for volume, square metres (m^2), acres, square kilometres (km^2) or hectares (ha) for area, and tonnes per cubic metre (t/m^3) for density. Elevations are given in metres above sea level (masl).

Unless stated otherwise, all currency amounts are stated in United States dollars (US$). The units of measure presented in this report are metric units. Grades of the main elements (Zn and Pb) are reported in percentage (%) while Ag is reported in grammes per tonne. Tonnage is reported as metric tonnes (t), unless otherwise specified.

Acronyms and abbreviations used in this report are summarised in Table 2-1.

Table 2-1: Acronyms and Abbreviations

Acronym / Abbreviation	Definition	Acronym / Abbreviation	Definition
°C	Degrees Celsius	JORC	Joint Ore Reserves Committee
2D	Two-dimensional	IPPC	International Plant Protection Convention
3D	Three-dimensional	KAX	Potassium Amyl Xanthate
AA	Atomic Absorption	KEX	Potassium Ethyl Xanthate
ABCD	Alpine-Balkan-Carpathian-Dinaride	KPI	Key Performance Indicator
BNG	Biodiversity Net Gain	kV	Kilovolt
BWi	Bond Work Index	kW	Kilowatt
CANMET	Canada Centre for Mineral and Energy Technology	kWh	Kilowatt hour
CF	Cut and Fill	kWh/t	Kilowatt hour per tonne
CMS	Cavity Monitoring System	LHD	Load Haul Dump
CO_2	Carbon dioxide	LHS	Long-Hole Stoping

Acronym / Abbreviation	Definition	Acronym / Abbreviation	Definition
COG	Cut-off Grade	LIMS	Laboratory Information Management System
COV	Cut-off-Value	LOM	Life of Mine
CP	Competent Person	M	Million(s)
CRM	Certified Reference Material	Ma	Million years ago
CSR	Corporate Social Responsibility	masl	Metres above sea level
Datamine	3D geological modelling, mine design and planning software	MPa	Megapascal
DST	Dry Stack Tailings	mRL	metres Reduced Level
EA	Environmental Assessment	MRMR	Mining Rock Mass Rating
ELOS	Equivalent Linear Overbreak/Slough	MSO	Mineable Shape Optimizer
EHS&S	Environment, Health, Safety and Sustainability	Mtpa	Million tonnes per annum
EIA	Environmental Impact Assessment	MVA	Megavolt-Ampere
EIS	Environmental Impact Statement	NQ	47.6 mm diameter drill core
EMS	Environmental Management System	PFS	Prefeasibility Study
EPR	Environmental Permitting Regulations	PPE	Personal Protective Equipment
EPRP	Emergency Preparedness and Response Plan	ppm	parts per million
ESG	Environmental and Social, Governance	QA/QC	Quality Assurance and Quality Control
ESIA	Environmental and Social Impact Assessment	RMR	Rock Mass Rating
FEL	Front End Loader	ROM	Run of Mine
FS	Feasibility Study	SLC	Sub-Level Cave/Caving
GCL	Geosynthetic Clay Liner	SLR	SLR Consulting Limited
GIIP	Good International Industry Practice	SMD	Stirred Media Detritor
GIS	Geographical Information System	SRK	SRK Consulting (UK) Ltd
GISTM	Global Industry Standard on Tailings Management	t	Tonne metric unit of mass (1,000kg or 2,204.6 lb)
GPS	Global Positioning System	t/a or tpa	Tonnes per annum
GSI	Geological Strength Index	t/d or tpd	Tonnes per day
H&S	Health and Safety	t/h or tph	Tonnes per hour
Ha	Hectare (10,000m^2)	TARP	Triggered Action Response Plan
HDPE	High-Density Polyethylene	TCR	Total Core Recovery
HQ	63.5 mm diameter drill core	TSF	Tailings Storage Facility
Hr	Hour/s	UTM	Universal Transverse Mercator
HV	High Voltage	WAI	Wardell Armstrong International
ID	Identification (number or reference)	XRD	X-ray powder Diffraction
IPJ	Inline Pressure Jig	XRF	X-ray powder Fluorescence

3.0 Property Description and Location

The Sasa Mine is located in the northeast of North Macedonia, in a land-locked mountainous region, near the Bulgarian border. The mine site is 10 km north of the town of Makedonska Kamenica, 150 km east from Skopje, the country's capital city and 29 km north from Kočani, the largest community in the region. The village of Sasa is located 4 km south of the operation with the nearest residential buildings 0.5 km from the operation.

The Sasa Mine is centred on the geographic coordinates of latitude 42°6'44" North and longitude 22°31'10" East. The location of the Property is shown in Figure 3-1.

Figure 3-1: Location of the Sasa Mine

Source: United Nations (August 2020)

3.1 Mineral Rights

Mining is regulated by the Law on Mineral Resources ("Official Gazette of the Republic of North Macedonia" Nos. 136/12, 25/13, 93/13, 44/14, 160/14, 129/15, 192/15, 39/16, 53/16, 120/16, 189/16, and Official Gazette of the Republic of North Macedonia No. 7/19, 99/22, 129/23, and 253/23, referred to as the "Law on Mineral Resources").

Under the Law on Mineral Resources, all mineral deposits are state-owned, regardless of surface land ownership, and private entities can only explore or exploit these resources by acquiring an administrative concession from the national government. Concessions are awarded on the basis of the surface area while the depth of exploration and exploitation activities is not specified.

The right to conduct the exploitation of Mineral Resources is acquired by receiving a concession for exploitation by the Government. Any legal entity, including foreign legal entities with branch offices registered within the Central Register, is entitled to such right on obtaining an exploitation concession, under the terms prescribed by the Law.

3.2 Mineral Tenure

The Property has an Exploitation Concession (24-5550/1) that was granted to Rudnik SASA DOOEL on November 13, 2014, and is valid until September 28, 2030. The Exploitation Concession was expanded in February 2026 from an area of 4.22 km^2 to approximately 6.87 km^2. Rudnik SASA DOOEL can apply to extend the concession beyond the current expiry date by submitting an application no earlier than one year and no later than six months before the expiry of the existing concession. As part of the application, a study on the justification for the concession extension and a report detailing the geological exploration and the Ore Reserves of the concession should be submitted, together with a completed expert assessment (audit) of the area to which the existing concession has been granted.

The Exploitation Concession comprises four zones and the boundary coordinates are defined in the Gauss-Krüger system (Hermannskogel datum) and are shown in Table 3-1, Table 3-2, Table 3-3, and Table 3-4, respectively. The extent of the concession (inclusive of the four zones) is shown in Figure 3-2.

The authors have relied on information provided by CAML, regarding the legal status of the rights pertaining to the Sasa Mine and have not independently verified the legality of surface land ownership, mineral tenure, legal status or ownership of the properties or any agreements that pertain to the licence areas.

The authors did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties but have relied on information provided by CAML for land title issues.

Table 3-1: Sasa Mine Exploitation Concession 24-5550/1 – Zone 1

Point	Easting (UTM)	Northing (UTM)	Point	Easting (UTM)	Northing (UTM)
T-1	7,628,152	4,661,525	T-54	7,626,954	4,663,641
T-2	7,628,141	4,661,538	T-55	7,626,923	4,663,654
T-3	7,628,095	4,661,561	T-56	7,626,938	4,663,678
T-4	7,628,074	4,661,579	T-57	7,626,248	4,664,647
T-5	7,628,062	4,661,596	T-58	7,625,838	4,665,023
T-6	7,628,056	4,661,622	T-59	7,625,750	4,665,811
T-7	7,628,073	4,661,656	T-60	7,625,131	4,666,857
T-8	7,628,081	4,661,686	T-61	7,624,786	4,666,967
T-9	7,628,074	4,661,720	T-62	7,624,339	4,667,778
T-10	7,628,055	4,661,788	T-63	7,624,200	4,667,700
T-11	7,628,050	4,661,826	T-64	7,623,836	4,667,490
T-12	7,628,049	4,661,853	T-65	7,624,027	4,666,706
T-13	7,628,055	4,661,890	T-66	7,623,602	4,666,461
T-14	7,628,073	4,661,960	T-67	7,623,888	4,666,020
T-15	7,628,074	4,661,975	T-68	7,624,058	4,665,567
T-16	7,628,072	4,662,028	T-69	7,624,526	4,665,291
T-17	7,628,065	4,662,049	T-70	7,624,820	4,665,460
T-18	7,627,851	4,662,241	T-71	7,625,167	4,664,627
T-19	7,627,811	4,662,345	T-72	7,625,123	4,664,616

Point	Easting (UTM)	Northing (UTM)	Point	Easting (UTM)	Northing (UTM)
T-20	7,627,813	4,662,415	T-73	7,625,148	4,664,588
T-21	7,627,861	4,662,528	T-74	7,625,169	4,664,577
T-22	7,627,822	4,662,592	T-75	7,625,206	4,664,577
T-23	7,627,794	4,662,687	T-76	7,625,289	4,664,588
T-24	7,627,805	4,662,697	T-77	7,625,360	4,664,588
T-25	7,627,780	4,662,720	T-78	7,625,431	4,664,576
T-26	7,627,795	4,662,734	T-79	7,625,593	4,664,591
T-27	7,627,776	4,662,756	T-80	7,625,821	4,664,573
T-28	7,627,784	4,662,784	T-81	7,625,835	4,664,553
T-29	7,627,802	4,662,777	T-82	7,625,822	4,664,553
T-30	7,627,828	4,662,842	T-83	7,625,746	4,664,522
T-31	7,627,848	4,662,863	T-84	7,625,698	4,664,526
T-32	7,627,814	4,662,938	T-85	7,625,661	4,664,543
T-33	7,627,770	4,662,995	T-86	7,625,625	4,664,545
T-34	7,627,764	4,663,022	T-87	7,625,556	4,664,537
T-35	7,627,779	4,663,074	T-88	7,625,556	4,664,552
T-36	7,627,759	4,663,091	T-89	7,625,554	4,664,553
T-37	7,627,716	4,663,068	T-90	7,625,537	4,664,535
T-38	7,627,633	4,663,072	T-91	7,625,470	4,664,529
T-39	7,627,577	4,663,120	T-92	7,625,474	4,664,540
T-40	7,627,506	4,663,157	T-93	7,625,460	4,664,543
T-41	7,627,437	4,663,202	T-94	7,625,453	4,664,528
T-42	7,627,321	4,663,236	T-95	7,625,429	4,664,526
T-43	7,627,291	4,663,277	T-96	7,625,355	4,664,538
T-44	7,627,298	4,663,368	T-97	7,625,292	4,664,538
T-45	7,627,299	4,663,384	T-98	7,625,209	4,664,527
T-46	7,627,252	4,663,455	T-99	7,625,157	4,664,527
T-47	7,627,221	4,663,647	T-100	7,625,117	4,664,548
T-48	7,627,201	4,663,674	T-101	7,625,067	4,664,604
T-49	7,627,135	4,663,730	T-102	7,624,820	4,664,546
T-50	7,627,112	4,663,735	T-103	7,626,352	4,663,765
T-51	7,627,088	4,663,722	T-104	7,626,600	4,663,377
T-52	7,627,012	4,663,655	T-105	7,627,611	4,661,971
T-53	7,626,988	4,663,643	T-106	7,628,065	4,661,526

Table 3-2: Sasa Mine Exploitation Concession 24-5550/1 – Zone 2

Point	Easting (UTM)	Northing (UTM)	Point	Easting (UTM)	Northing (UTM)
T-1	7,627,190	4,663,760	T-13	7,627,586	4,663,133
T-2	7,627,139	4,663,743	T-14	7,627,640	4,663,086
T-3	7,627,141	4,663,742	T-15	7,627,712	4,663,083
T-4	7,627,208	4,663,689	T-16	7,627,745	4,663,104
T-5	7,627,236	4,663,656	T-17	7,627,738	4,663,110
T-6	7,627,268	4,663,462	T-18	7,627,617	4,663,123
T-7	7,627,321	4,663,389	T-19	7,627,628	4,663,206
T-8	7,627,322	4,663,365	T-20	7,627,474	4,663,317

Point	Easting (UTM)	Northing (UTM)	Point	Easting (UTM)	Northing (UTM)
T-9	7,627,313	4,663,283	T-21	7,627,368	4,663,431
T-10	7,627,332	4,663,251	T-22	7,627,274	4,663,641
T-11	7,627,446	4,663,216	T-23	7,627,236	4,663,659
T-12	7,627,514	4,663,169			

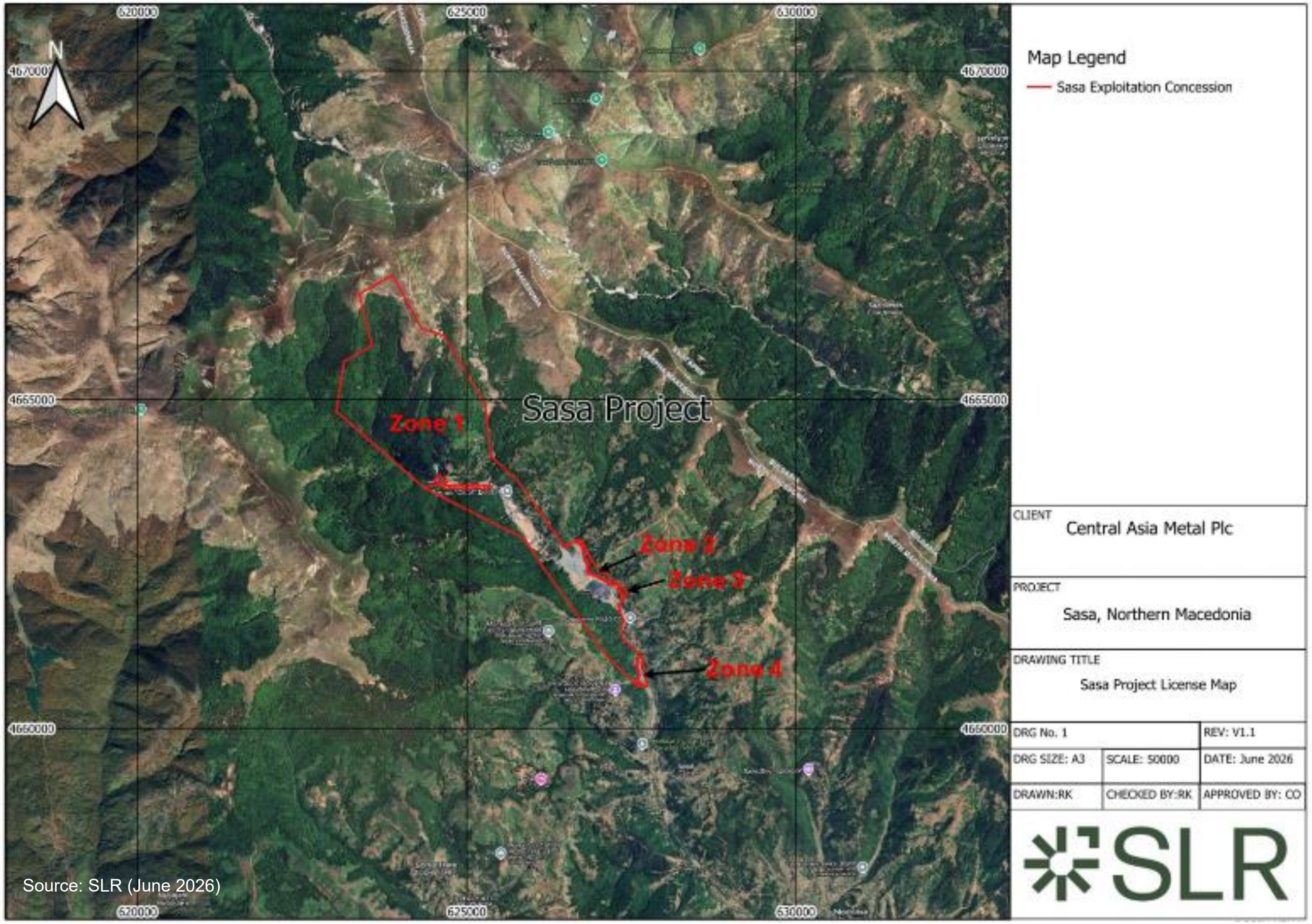
Table 3-3: Sasa Mine Exploitation Concession 24-5550/1 – Zone 3

Point	Easting (UTM)	Northing (UTM)
T-1	7,627,890	4,662,978
T-2	7,627,884	4,662,972
T-3	7,627,908	4,662,927
T-4	7,627,861	4,662,877
T-5	7,627,826	4,662,945
T-6	7,627,785	4,663,002
T-7	7,627,780	4,663,022
T-8	7,627,796	4,663,060
T-9	7,627,890	4,662,978

Table 3-4: Sasa Mine Exploitation Concession 24-5550/1 – Zone 4

Point	Easting (UTM)	Northing (UTM)	Point	Easting (UTM)	Northing (UTM)
T-1	7,628,168	4,661,525	T-16	7,628,086	4,661,985
T-2	7,628,148	4,661,548	T-17	7,628,088	4,661,979
T-3	7,628,102	4,661,571	T-18	7,628,139	4,661,954
T-4	7,628,083	4,661,588	T-19	7,628,142	4,661,931
T-5	7,628,074	4,661,602	T-20	7,628,140	4,661,919
T-6	7,628,069	4,661,621	T-21	7,628,139	4,661,910
T-7	7,628,084	4,661,651	T-22	7,628,137	4,661,894
T-8	7,628,094	4,661,685	T-23	7,628,130	4,661,880
T-9	7,628,086	4,661,723	T-24	7,628,125	4,661,868
T-10	7,628,066	4,661,791	T-25	7,628,124	4,661,855
T-11	7,628,062	4,661,827	T-26	7,628,128	4,661,850
T-12	7,628,061	4,661,853	T-27	7,628,134	4,661,850
T-13	7,628,067	4,661,888	T-28	7,628,155	4,661,847
T-14	7,628,085	4,661,958	T-29	7,628,207	4,661,525
T-15	7,628,086	4,661,975			

Figure 3-2: Sasa Mine Concession



Source: SLR (June 2026)

The current LOM at Sasa runs until 2034 (inclusive), the authors consider there are reasonable grounds to expect the concession to be extended beyond the September 28, 2030, expiry.

Rudnik SASA DOOEL has all other required permits to conduct the proposed work on the Property and to continue production as planned. SLR is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work on the Property.

3.3 Surface Ownership and Access Rights

It is noted that Mineral Resources are owned by the Republic of Macedonia, irrespective of the ownership of the land they are located on.

Rudnik SASA DOOEL owns the land on which the surface facilities are located.

3.4 Royalties and Other Payments

There is a payment relating to the compensation for exploitation of the minerals (Concession Fee) based on the market value of the metals contained in the concentrates as follows:

- 4% of the market value of the metal of lead per tonne in each tonne of lead concentrate produced.
- 4% of the market value of the metal of zinc per tonne in each tonne of zinc concentrate produced.
- 4% of the market value of the metal of silver per tonne in each tonne of lead concentrate produced.

Metal prices are determined using the average price of the metals in the month it was produced. This Concession Fee is paid quarterly to the Ministry of Economy of the Republic of North Macedonia.

The silver contained in the concentrates is subject to a streaming agreement signed in 2017 with OR Resources Inc and is invoiced separately when the silver content reaches payable levels. CAML receives US$6.665 per ounce for its silver production for the life of the Sasa Mine.

3.5 Environmental and Social Commitments

The concession holder is obliged to implement at its own expense measures for environmental protection and protection of nature and cultural heritage, as well as measures for re-cultivation of the land.

The concession holder is also obliged to compensate in whole the damage caused to third persons during the exploitation of minerals and the damage caused to the environment.

Upon completion of the exploitation and the mining works, the concession holder is obliged to undertake restoration of the area and re-cultivation of the land. If restoration is not undertaken then the state will perform the restoration at the expense of the concession holder.

3.6 Environmental Liabilities and Permitting Requirements

On August 30, 2003, approximately 150,000 m^3 of tailings leaked from TSF3-1 via a diversion tunnel and into the Kamenica River during a period when the mine was not operational and prior to CAMLs ownership. The cause of the failure was a defective cap within the roof of the ancillary pipe carrying TSF 3.1 drainage into the underlying river diversion structure (i.e. not a direct failure of the river diversion structure). A clean up of the tailings that were released into the river and a full repair of the diversion tunnel was performed according to a design approved by the regulatory authorities. All activities were conducted by independent contractors under the supervision of the regulatory authorities.

A legal review was carried out by Georgi Dimitrov Attorneys (April 2016) assessing the risk and potential obligations arising from historic contamination. It was concluded in this review that under Macedonian law there is no liability sitting with the current owner to clean up downstream of the site any residual contamination with respect of the 2003 incident. In addition, it was further concluded that "Rudnik SASA DOOEL can only be liable for the environmental damage occurred following the moment the company began to operate the mine".

On September 14, 2020, approximately 3,000 m^3 of tailings, supernatant water and suspended sediment, spilled from TSF4 via a breach in the dam embankment and flowed for approximately 11 km along the Kamenica River to the Kalimanci reservoir. The initial leakage of water from the crest of the dam became progressively worse, eroding part of the embankment and allowing the upper layer of tailings to subside through an opening with supernatant water draining afterwards. The leakage subsided after approximately two hours, once a portion of the ponded supernatant water had drained out and the remaining tailings recompacted and reconstituted. Rudnik SASA DOOEL commissioned specialist consultants to support with the remediation actions, which were undertaken in line with State Inspectorate and Global Industry Standard on Tailings Management (GISTM) requirements. An independent technical review of the remediation process was commissioned in January 2021 and concluded that the remediation was executed in a thorough and diligent manner.

The authors are not aware of any other environmental liabilities on the Property. Environmental Permits obtained by Rudnik SASA DOOEL are detailed in Section 19 (Environmental Studies, Permitting, and Social or Community Impact).

3.7 Adjacent Properties

The Sasa Mine is located in a mineralised region of North Macedonia with the Toranica Mine to the north and Kaltun Exploration Project to the east.

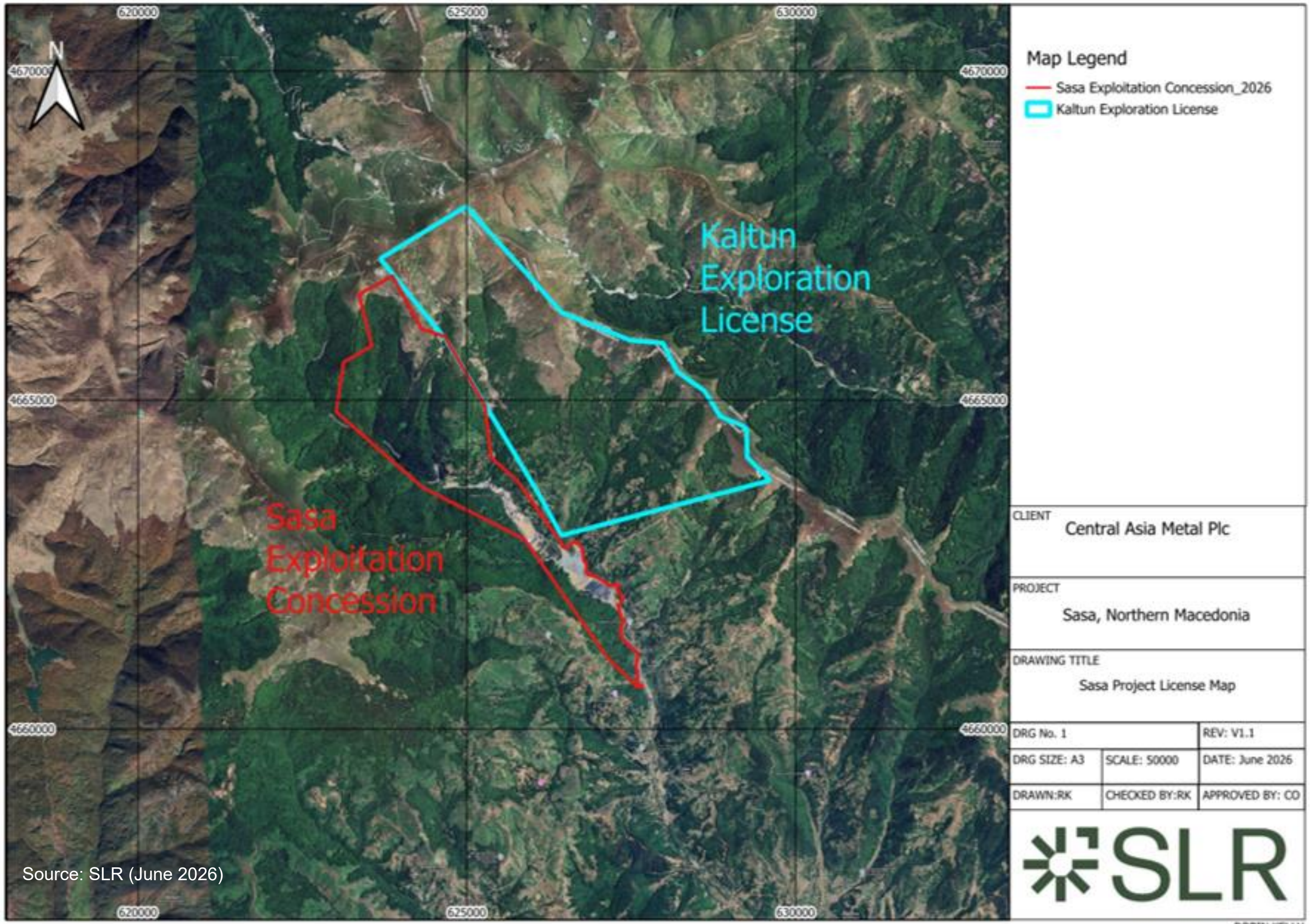
The Kaltun Exploration Project (Baltašnica-Svinja Reka-Sokol and Petrova Reka zones) is under the ownership of Kaltun Maddenjilik and the boundary of the property is adjacent to the Sasa Exploitation Concession as shown in Figure 3-3.

The Kaltun licence covers an area of approximately 12.58 km^2 and is typically accessed via the road passing through the Sasa Mine and during the dry months access is permissible via the Toranica Mine to the north. The geology of the Kaltun property is considered an extension of the same geology and mineralisation found within the Sasa Mine. The mineralisation at Baltašnica-Svinja Reka-Sokol is the up-dip extension of Svinja Reka, above the historical mine workings. The Petrova Reka mineralisation is in the footwall of the Baltašnica-Svinja Reka-Sokol zone. Exploration was undertaken on the property between July 2016 and February 2019 and included 134 diamond drillholes.

The Toranica Mine is owned and operated by Minstroy Holding AD and is located some 5 km to the north (outside the area shown in Figure 3-3) and comprises a mineralised extension of the Sasa deposits. Toranica is an underground operation producing some 320,000 tpa of ore and annual production of lead and zinc concentrates is around 30,000 t. The Toranica licence is not contiguous with the Sasa licence with an approximate separation of 2.0 km.

The authors have not relied upon any information from these adjacent claims in the preparation of this report.

Figure 3-3: Location and Relation of the Kaltun Licence Area to the Sasa Licence



Source: SLR (June 2026)

4.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

4.1 Accessibility

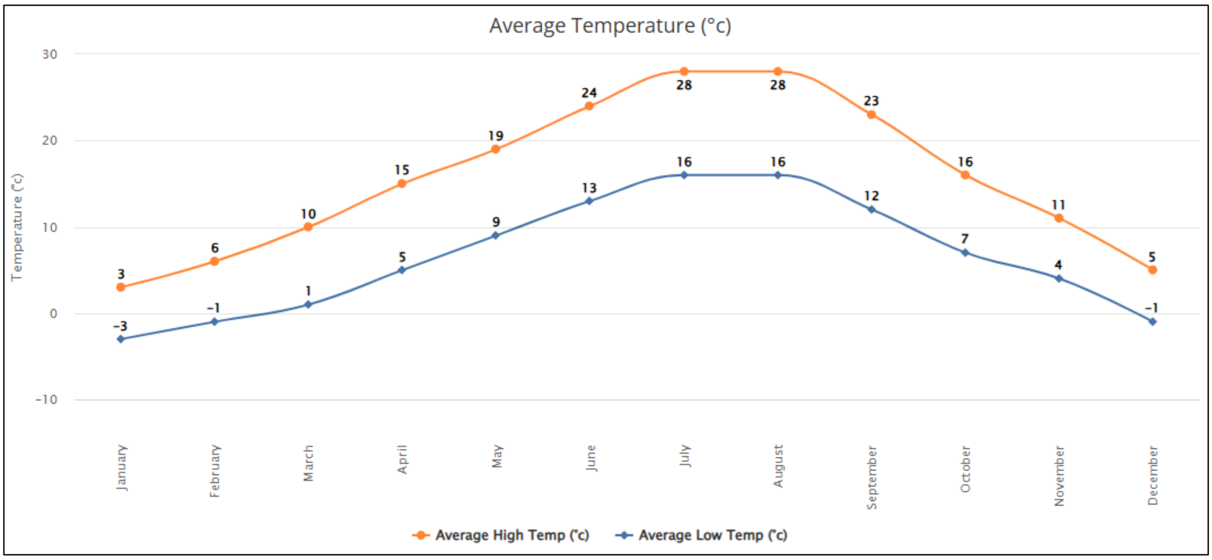
The Sasa Mine is accessed by the A4 and A3 highways from Skopje to Makedonska Kamenica, a distance of 130 km, and then by the R1210 minor road (paved) for a distance of 10 km. The A3 highway continues eastwards from Makedonska Kamenica into Bulgaria via the border crossing at Stanke Lisichkovo before continuing northeast to Sofia, a total distance of 160 km. There are international airports at Skopje and Sofia and the closest railhead is 29 km to the southwest at the town of Kočani and is accessed by the A3 highway.

4.2 Climate

The climate is subject to continental and Mediterranean influences, with hot dry summers and cold winters. The majority of rainfall occurs during the spring and autumn and occasionally there are heavy snowfalls on the mountains, rarely in the valleys, from December to February. Average precipitation is 600 mm per year at the Sasa Mine.

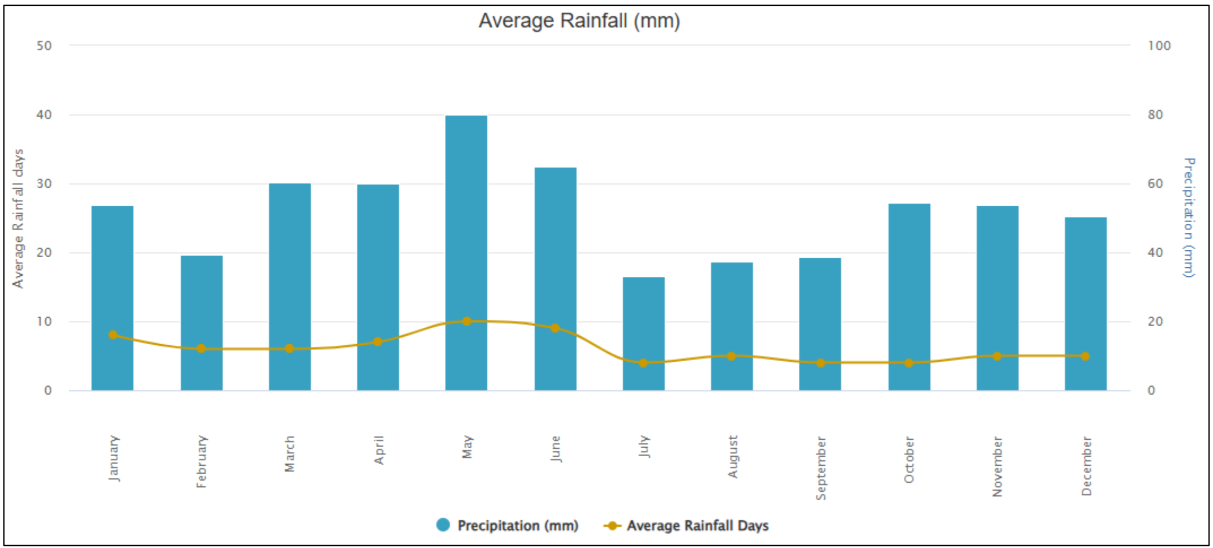
The average monthly temperature for Makedonska Kamenica is shown in Figure 4-1 and the average monthly rainfall in Figure 4-2.

Figure 4-1: Average Temperature (°C) for Makedonska Kamenica



Source: worldweatheronline.com (June 2026)

Figure 4-2: Average Rainfall (days / mm) for Makedonska Kamenica



Source: worldweatheronline.com (June 2026)

4.3 Local Resources

Makedonska Kamenica has approximately 4,500 inhabitants. The Sasa Mine is the largest direct employer in the region with approximately 700 employees. Most employees live near the mine. The mine has an established network of mining suppliers and contractors.

There are existing transport links, telecommunications facilities, electricity and water supply.

4.4 Infrastructure

A summary of the infrastructure associated with the Sasa Mine is provided below and is further detailed in Section 17 (Project Infrastructure):

- Svinja Reka underground mine (operating).
- Golema Reka underground mine (non-operational).
- Processing plant, dry stack tailings plant and paste backfill plant.
- Tailings storage facility and dry stack storage facility (landform).
- Site offices, staff facilities, laboratory, stores, and maintenance workshops.

Electrical power is supplied by the national grid, and water is recycled by the operation and supplemented by permitted abstraction from the Kozje, Saska and Petrova Rivers.

All infrastructure required by the operation is currently in place. The authors are of the opinion that there is sufficient land, water, and power for the planned mining and processing operations.

4.5 Physiography

The Sasa Mine is located at approximately 1,000 masl in a narrow V-shaped valley of the Kamenicka River with associated mine access portals, processing plant, maintenance workshop, office and TSF (Figure 4-3). Underground workings are to the north of the processing plant predominantly below Svinja Reka. The topography within the concession area ranges from 800 to 2,030 masl.

Vegetation consists of secondary growth beech, birch, and pine forest. Much of the pine forest is associated with reforestation and is interspersed with scattered pasture giving subsistence level agriculture which includes livestock and bee keeping on sub alpine meadows. Upland areas above 1,700 masl typically consist of high alpine grasses.

Figure 4-3: View Looking Northwest of Sasa Mine Surface Infrastructure

Source: Rudnik SASA DOOEL (June 2026)

5.0 History

5.1 Ownership, Exploration and Development History

The Sasa deposits were discovered during exploration from 1954 to 1965. Under state ownership trial mining commenced in 1965. Commercial production started in 1966 at a mining rate of 0.3 Mtpa and involved the production of a bulk saleable concentrate.

In the 1990s, production reached 0.5 Mtpa.

In 2002, the mine closed and entered bankruptcy when government funding ended.

Approximately 15 Mt at average grades of 5.2% Pb and 4.6% Zn was produced by the historical operation from the upper levels of the deposits with access provided by tracked adits.

From 1954 to 2002, a total of 274 drillholes (12 from surface and 262 from underground) were completed at Svinja Reka for a total of 20,597 m) and 82 drillholes (24 from surface and 58 from underground) were completed at Golema Reka.

In 2005, Solway Investment Group Ltd (Solway) acquired the mine from a group of creditors. The mine was restored within six months after acquisition, with the engagement of Metso to assist in design of the processing plant. The plant was re-equipped with a modern flotation circuit allowing production of separate zinc and lead concentrates instead of a bulk concentrate.

Solway purchased new mining equipment from Atlas Copco, and Paus and invested in environmental protection, including construction of a new TSF, rehabilitation of the old TSF, achievement of ISO 14001 Certification, and an Integrated Environmental Permit.

In 2006, the Sasa Mine restarted at a production rate of 0.78 Mtpa.

From 2006 to 2015, a total of 1,080 drillholes for 80,737 m were completed at Svinja Reka (32 from surface and 1,048 from underground), and 22 underground drillholes for 1,503 m were completed at Golema Reka.

On November 3, 2015, Solway announced that it had sold the Sasa Mine to Orion Mine Finance Group (Orion) partnered with Fusion Capital AG (Fusion) through its wholly owned subsidiary Lynx Europe SPLLC Skopje (Lynx).

From 2015 to 2017, under ownership by Lynx a total of 228 drillholes were completed at Svinja Reka.

In November 2017, CAML acquired the Sasa Mine from Lynx and took 100% ownership.

In 2023 and 2024, CAML started transitioning the mining methods at Svinja Reka from the historical top-down SLC mining to bottom-up mining of LHS and CF. A new Central Decline was constructed to improve underground access and haulage. A new paste backfill plant and associated underground reticulation and a dry stack tailings plant and dry stack landform were constructed resulting in a reduced requirement for tailings storage within the existing TSF.

5.2 Production History

A summary of ore processed by the plant and the resulting concentrate production from 2008 to 2025 is shown in Table 5-1.

Table 5-1: Sasa Production History from 2008 to 2025

Item	Unit	2008	2009	2010	2011	2012	2013	2014	2015	2016
Processing Plant										
Ore Processed	kt	854	865	811	758	754	774	780	777	779
Pb Grade	% Pb	4.68	4.43	4.05	3.83	3.93	4.13	4.16	4.04	3.95
Zn Grade	% Zn	4.08	4.14	3.81	3.43	3.35	3.47	3.48	3.52	3.41
Lead Concentrate										
Pb Concentrate	kt	49.1	47.6	41.3	37.1	38.0	41.0	41.6	40.2	39.5
Pb Recovery	%	91.6	94.2	94.4	95.1	94.4	94.4	94.5	94.1	94.1
Pb Grade	% Pb	74.5	75.7	75.2	74.3	73.6	73.6	73.7	73.5	73.3
Pb in Concentrate	kt	36.6	36.1	31.0	27.6	28.0	30.2	30.7	29.5	29.0
Zinc Concentrate										
Zn Concentrate	kt	58.0	61.0	52.8	44.6	43.1	46.2	46.9	47.2	45.5
Zn Recovery	%	82.3	85.5	86.0	86.6	86.2	86.3	86.5	85.8	84.6
Zn Grade	% Zn	49.5	50.2	50.4	50.6	50.5	50.1	50.1	49.8	49.4
Zn in Concentrate	kt	28.7	30.6	26.6	22.5	21.8	23.3	23.5	23.5	22.5
Item	**Unit**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**
Processing Plant										
Ore Processed	kt	793	805	821	820	831	807	806	761	799
Pb Grade	% Pb	3.98	3.90	3.77	3.85	3.52	3.63	3.70	3.71	3.35
Zn Grade	% Zn	3.18	3.31	3.29	3.37	3.14	3.15	2.97	2.87	2.61
Lead Concentrate										
Pb Concentrate	kt	40.8	40.3	40.4	41.3	37.9	38.4	39.1	37.6	35.6
Pb Recovery	%	94.6	93.6	94.5	94.3	93.1	93.4	93.1	94.4	94.1
Pb Grade	% Pb	72.9	73.6	72.3	72.0	71.8	71.2	71.0	70.8	70.6
Pb in Concentrate	kt	29.9	29.4	29.2	29.7	27.2	27.4	27.8	26.6	25.2
Zinc Concentrate										
Zn Concentrate	kt	43.7	46.1	47.1	47.6	44.4	42.8	40.2	37.0	35.6
Zn Recovery	%	85.5	84.6	86.5	86.1	84.9	84.6	85.0	85.2	85.7
Zn Grade	% Zn	49.4	48.9	49.6	50.0	49.9	50.1	50.6	50.2	50.2
Zn in Concentrate	kt	21.6	22.5	23.4	23.8	22.2	21.5	20.3	18.6	17.9

6.0 Geological Setting and Mineralization

6.1 Regional Geology

The Sasa Mine is located in the Dinaride collisional mountain belt, part of the Alpine-Balkan-Carpathian-Dinaride (ABCD) metallogenic and geodynamic province within the Alpine-Himalayan orogenic system which resulted from convergence of the African, Arabian and Indian plates and their collision with Eurasia, from the Cretaceous to the present.

The ABCD formed during the closure of the Tethys ocean and represents the long, deformed margin of the Adria microplate, preserved along the eastern Adriatic coast and inland through Croatia, Bosnia and Herzegovina, Montenegro, Serbia and Albania. Structurally and tectonically, it links north-westward into the Alps and south-eastward into the Hellenides, forming part of the Mediterranean-European Alpine belt.

The Dinarides preserves almost the full transition from passive continental margin through oceanic basin development to subduction, obduction, collision, and post-collisional deformation.

Prior to collision, the region was dominated by a broad warm-water carbonate platform on the margin of Adria. During the Triassic and Jurassic, extensive shallow marine limestones and dolomites accumulated across a stable passive margin setting. These significant carbonate successions reached several kilometres thick in some places, producing what is now known as the Adriatic Carbonate Platform. Much of the modern landscape of the External Dinarides reflects these rocks. The spectacular karst terrain of the region developed in these massive Mesozoic limestones.

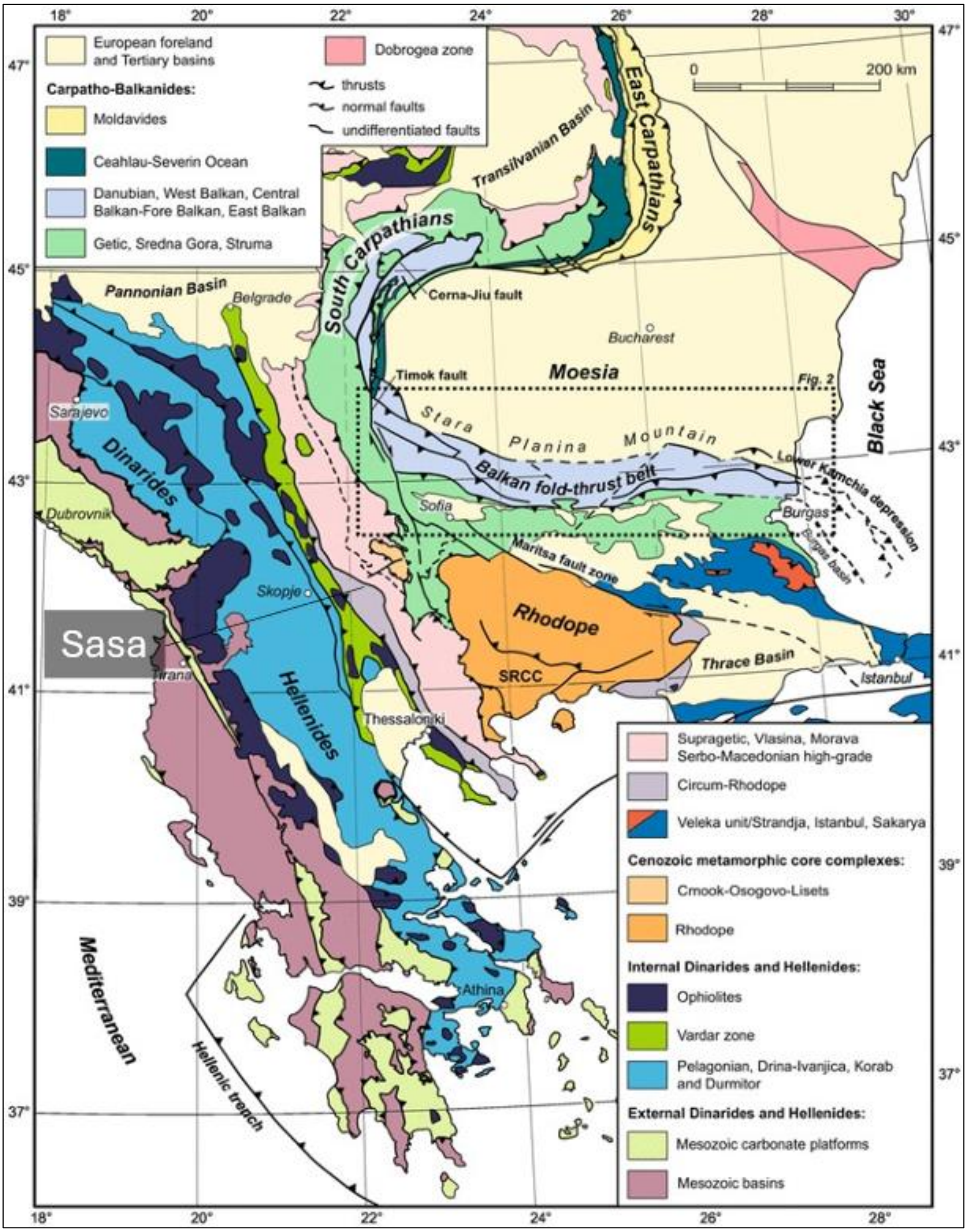
As Neotethyan oceanic basins opened during Jurassic rifting and seafloor spreading, the passive margin evolved into a more complex ocean-continent system. Eventually convergence began, and oceanic lithosphere started to subduct beneath Eurasian-affiliated terranes. The Dinarides contains large ophiolite belts, especially within the Internal Dinarides. In the Balkans these ophiolite belts are regionally extensive and are central to reconstruction of Neotethyan palaeogeography.

The complex arcuate geometry of the collision interface, and the presence of several micro-plates within the orogenic collage, resulted in a variety of collision products. Some segments are characterised by extensive regional metamorphism, whereas others by calc-alkaline igneous activity. The structural complexity and present-day geometry of the region reflect large-scale oroclinal bending during post-collision tectonics throughout the Tertiary, including major transcurrent fault systems with overall dextral displacements exceeding 100 km.

Orogenic segmentation resulted in a discontinuous distribution of mineral deposits within the province and limited the lateral extents of the various metallogenic belts along the trace of the orogen. Whereas magmatic activity is evident from the Cretaceous to the present, two overlapping arcuate segments represent the bulk of magmatism, one located to the south (in which the majority of the intrusions and volcanic rocks are of Oligocene and Miocene age), and a second to the north (where Cretaceous magmatic activity is characteristic).

The regional geology of Southeast Europe is shown in Figure 6-1.

Figure 6-1: Regional Geology of Southeast Europe



Source: Kounov et al., (2017)

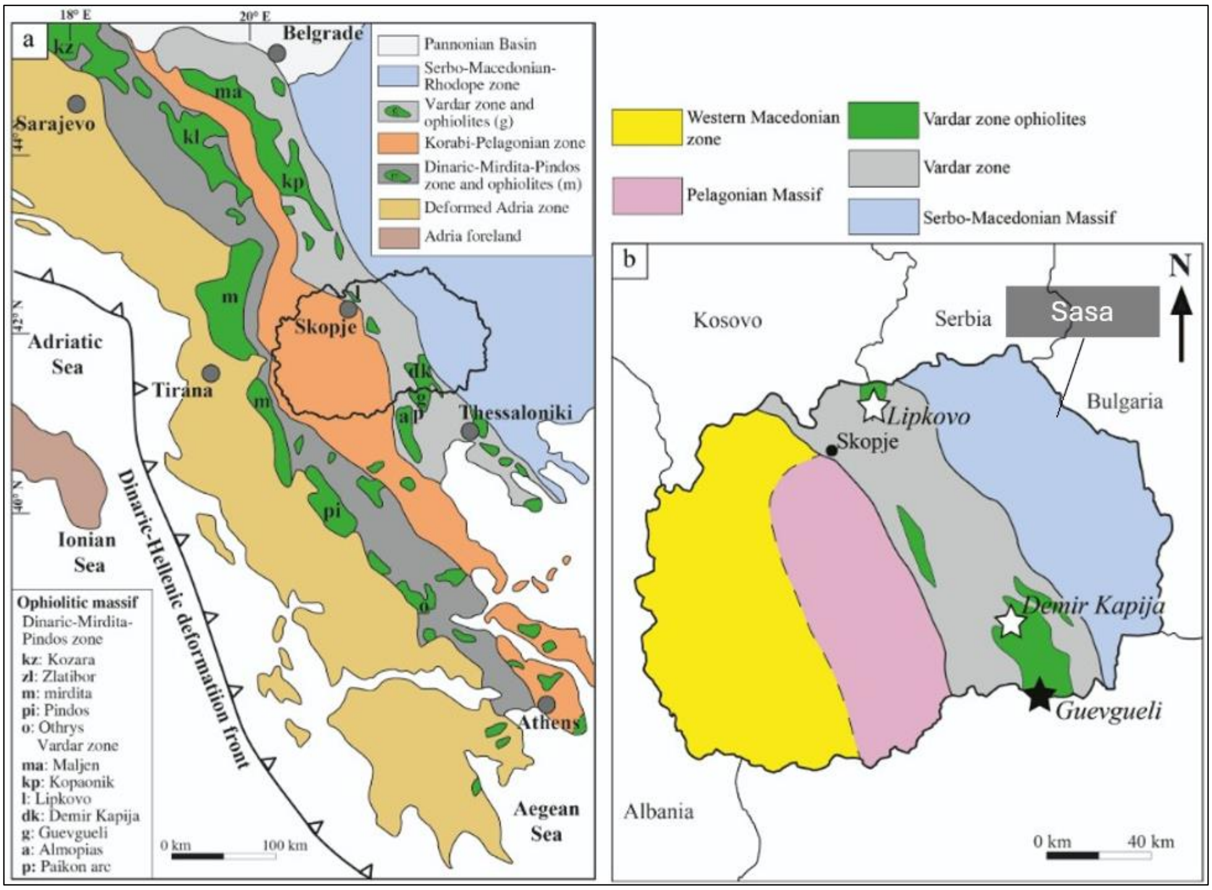
North Macedonia is situated within the broader Cenozoic Southern Balkan extensional regime, a tectonic setting shaped by post-collisional processes that followed the Alpine orogeny. This extensional overprint reflects the collapse and reorganisation of earlier compressional structures linked to the closure of Tethyan oceanic domains and the convergence between the African-Adriatic and Eurasian plates.

The country's geology is organised into a series of elongated structural belts that predominantly trend north-northwest, mirroring the inherited grain of the Alpine orogenic system. These belts record a complex evolution involving Mesozoic subduction and collision, followed by Cenozoic extension, strike-slip deformation, and basin formation. As a result, the present-day architecture is a superposition of older compressional structures that have been reactivated and dissected by younger extensional tectonics.

Within North Macedonia, three major structural zones are typically distinguished. The first is generally associated with more external, relatively less metamorphosed units dominated by sedimentary sequences and fold-and-thrust structures related to earlier compressional phases. The second zone represents more internal or intermediate domains, where Mesozoic oceanic remnants, mélange complexes, and higher degrees of deformation are more common, reflecting former suture-related processes. The third zone corresponds to the most internally derived units, which are strongly influenced by Cenozoic extension, basin development, and localised magmatism, often linked to broader back-arc and extensional dynamics in the southern Balkan region.

Three major structural zones recognised in North Macedonia are shown in Figure 6-2 and are further described below.

Figure 6-2: Structural Zones of Geology of North Macedonia Tectonic Units



Source: Brombin (2022)

6.1.1 Serbo-Macedonian Massif

Sasa is located in the Serbo-Macedonian Massif, a structural zone composed of Precambrian to Palaeozoic metamorphic rocks that record a long and polyphase tectonometamorphic history. It extends southwards from Serbia through Kosovo, North Macedonia, and Bulgaria to the Chalkidiki Peninsula in northern Greece, and hosts numerous economically significant Cu, Au, Pb, and Zn deposits, including Bor and Majdanpek in Serbia, Toranica, Sasa, and Bucim in North Macedonia, Osogovo in Bulgaria, and Skouries in Greece.

The Serbo-Macedonian Massif is an accretionary prism folded at the base by metamorphic rocks. Within this basement, two main metamorphic assemblages are typically distinguished, reflecting different crustal levels and metamorphic conditions during regional orogenic events.

The lower complex represents the deeper crustal section and is characterised by medium- to high-grade metamorphism under amphibolite facies conditions. It is mainly composed of mica gneisses, mica schists, amphibolites, quartzites, marbles, and locally migmatitic rocks, indicating partial melting in some domains. These lithologies reflect intense deformation and recrystallisation at elevated temperatures and pressures, consistent with deep burial within a thickened orogenic crust during earlier tectonic cycles.

In contrast, the upper complex represents a structurally higher level, formed under lower-grade metamorphic conditions. It consists predominantly of volcano-sedimentary sequences that have been metamorphosed to greenschist facies. Typical rock types include chlorite-bearing shales, chlorite-amphibolite assemblages, chlorite-sericite schists, and quartz-rich schists and metasiltstones. These rocks preserve more of their original sedimentary and volcanic textures, reflecting relatively shallower burial and lower temperatures compared to the lower complex.

Both metamorphic complexes were subsequently affected by multiple phases of granitoid intrusion. These intrusions occurred episodically throughout different geological periods and were strongly controlled by pre-existing regional structures

6.1.2 Vardar Zone

The Vardar Zone is one of the most metallogenically significant tectonic domains in the southern Balkans (Volkov et al., 2019), and its evolution has been widely studied. It is situated between the Serbo-Macedonian Massif to the east and the Dinaride-Pelagonian system to the west, although the nature of its western boundary is less clearly defined than the sharply expressed eastern contact. The eastern margin is commonly interpreted as a major longitudinal dislocation zone along which parts of the Serbo-Macedonian Massif were thrust over or structurally juxtaposed against the Vardar domain, whereas the western transition into the Dinarides is more gradational and structurally complex.

The Vardar Zone represents a composite tectonic belt made up of strongly deformed and juxtaposed crustal blocks derived from Mesozoic oceanic and continental environments. Its fundamental components include Jurassic ophiolitic assemblages such as serpentinised peridotites, gabbros, and diabase complexes, which are widely interpreted as remnants of Neotethyan oceanic lithosphere. These are locally associated with volcano-sedimentary sequences and are structurally mixed with fragments of older continental-derived rocks.

In addition to the ophiolitic basement, the zone contains marine Triassic sedimentary units, as well as Jurassic and Cretaceous granitoid intrusions and calc-alkaline volcanic rocks, reflecting repeated episodes of subduction-related and post-collisional magmatism. The upper structural levels are commonly characterised by widespread Upper Cretaceous flysch deposits, which record foreland basin sedimentation during progressive orogenic loading and closure of oceanic basins.

A defining feature of the Vardar Zone is its tectonic mélange character. It consists of a chaotic mixture of rock fragments embedded in a fine-grained, schistose to shale-clay matrix. These mélanges incorporate a wide range of lithologies, including greywackes, ophiolitic fragments, quartzites, marbles, and other metamorphic and sedimentary rocks, many of which are of Late Cretaceous to Paleocene age. This assemblage is interpreted as representing disrupted pieces of rift-related and oceanic crustal complexes formed during the evolution and eventual closure of the Neotethys Ocean more than 150 ma.

6.1.3 Pelagonian Massif

This Precambrian massif is composed of a lower basement dominated by gneisses and mica schists, which is overlain by a heterogeneous metasedimentary sequence including gneiss, schist, quartzite, marble, and dolomite. In several areas, these units are intruded by granitoid bodies and pegmatitic veins of uncertain age, reflecting multiple phases of crustal reworking.

6.1.4 Tectonic Setting and Regional Seismicity

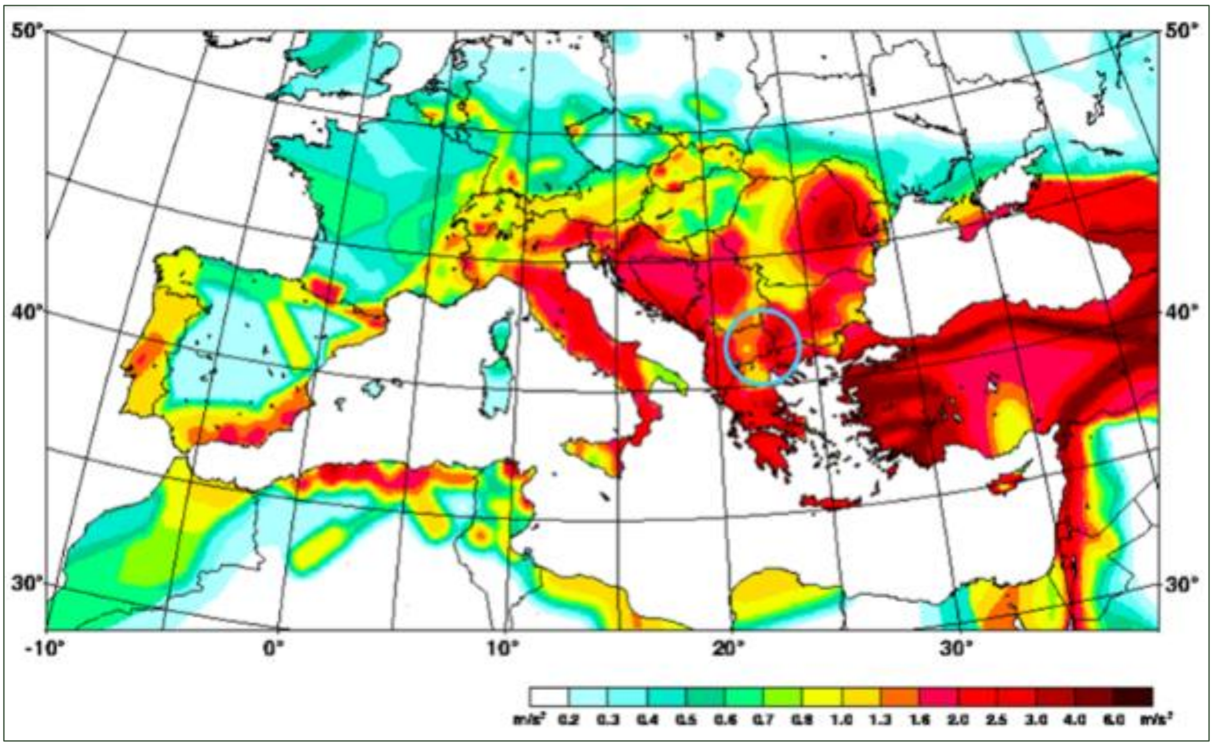
The Mediterranean seismic belt is as an area of high seismicity. The seismicity of Macedonia is related to destructive tectonic processes associated primarily with vertical movement of tectonic blocks. Two regions of specific neotectonic features are well distinguished:

- West Macedonia, characterised by longitudinal (northeast trending) structures, and
- Central and East Macedonia with transverse (east trending) stretching of principal tectonic morphostructures.

Earthquakes of magnitudes between 6.0 to 7.8 on the Richter Scale have been experienced in the country. The strongest recent earthquake occurred in Skopje (1963; Magnitude 6.1 at a depth of 6.0 km). Prior to this, other notable earthquakes include Pehcevo-Kresna (1904; Magnitude =7.8) and Valandovo-Dojran (1931; Magnitude = 6.7).

The Sasa area (Figure 6-3) is considered of 'moderate to high' seismicity.

Figure 6-3: Seismic Hazard Assessment Map



Source: SRK (2017)

6.2 Local Geology

The Sasa Mine is situated within the Serbo-Macedonian Massif, a geological terrane recognised as a highly prospective environment for lead-zinc-silver mineralisation. Occupying much of the eastern third of North Macedonia, the massif comprises a diverse assemblage of metamorphic, sedimentary, and igneous rock units that reflect a long and complex tectonometamorphic history.

6.2.1 Precambrian Metamorphic Rocks

The Precambrian rocks constitute the basement of the Sasa area and are dominated by gneisses and mica-schists, locally interbedded with quartzite layers and containing lenses of amphibolite and metabasalts. The gneissic units are particularly well exposed in the Kamenicka River valley, where they occur in tectonic contact with mica-schists on one side and Riphean-Cambrian rock sequences on the other.

They are generally very competent, hard, and massive rocks, occasionally exhibiting a lenticular internal fabric. In fresh exposure, they typically display a light coloration ranging from yellow to whitish-yellow tones, reflecting their mineralogical composition and degree of metamorphic recrystallisation.

6.2.2 Riphean-Cambrian Metamorphic Rocks

This unit is composed predominantly of albite-epidote-chlorite schists, amphibolites, and metadolerites. The albite-epidote-chlorite schists form a characteristic greenschist-facies assemblage, interpreted as part of the geosynclinal basement sequence. Within this package, lenses and bands of amphibolite, epidote-bearing schist, metadolerite, and locally metagabbro are common.

Amphibolites and metadolerites typically occur in sharp tectonic contact with surrounding lithologies, indicating a strong structural control. The amphibolites are generally green in colour and display either schistose or massive textures, while the metadolerites are commonly preserved as elongated, lens-shaped bodies with a more massive, less foliated fabric.

6.2.3 Lower Palaeozoic Rocks

The Lower Palaeozoic succession in the Sasa area varies considerably in thickness, ranging from a few metres to several hundred metres, and generally dips toward the southwest at approximately 35°. It is dominated by quartz-graphite schists and marbles.

The quartz-graphite schists occur within a narrow, northwest-trending structural zone extending for roughly 25 km. In places, this sequence appears as a transgressive cover over gneiss-mica-schist and greenschist basement units, although most observed contacts are tectonic in nature.

The unit also includes chloritic and sericitic schists, within which marble horizons occur as discrete lenses or bands. These marbles can reach several hundred metres in strike length and up to 30 m in thickness, forming laterally continuous carbonate bodies (bands and lenses).

6.2.4 Upper Palaeozoic Igneous Rocks

The Upper Palaeozoic igneous suite is represented by granitoid intrusions that cut both Precambrian and Riphean-Cambrian metamorphic rocks. These granitoids frequently contain xenoliths of older schists as well as fragments of gabbro-dolerite material.

At their margins, the granitoids commonly exhibit strong deformation features, including schistosity and mylonitisation, reflecting syn-tectonic intrusion and subsequent dynamic metamorphism. Two main intrusive phases are recognised: an earlier coarse-grained granitic phase and a later, more fine-grained phase.

6.2.5 Tertiary Igneous Rocks

At Sasa, the Tertiary magmatic suite is represented by rhyolites, quartz-latites, andesite-trachyandesites, occurring predominantly as dykes and sills that intrude Precambrian, Riphean-Cambrian, and Palaeozoic metamorphic basement rocks. These intrusions formed at hypabyssal to subvolcanic levels and are generally of intermediate to acidic composition.

Porphyritic quartz-latites are the most widespread intrusive lithology in the area, commonly forming thick dyke swarms. Hydrothermal alteration is well developed and includes argillitic, propylitic, chloritic, and epidote alteration assemblages. Polymetallic mineralisation is typically associated with contact zones between these intrusions and surrounding metasedimentary rocks.

Porphyritic light-grey andesite-trachyandesites are also widespread, particularly along the western margin of the Sasa-Toranica volcanic zone, where they form part of a broader Neogene volcanic-subvolcanic system.

6.3 Property Geology

6.3.1 Stratigraphy

The Sasa area is characterised by a multi-phase Tertiary intrusive system dominated by porphyritic stocks ranging in composition from quartz monzonite to granodiorite, accompanied by extensive development of intrusion breccias. These intrusions represent successive pulses of magmatic activity, reflecting a long-lived subvolcanic system in which repeated emplacement, fracturing, and hydrothermal fluid circulation created a highly fractured and chemically reactive host environment.

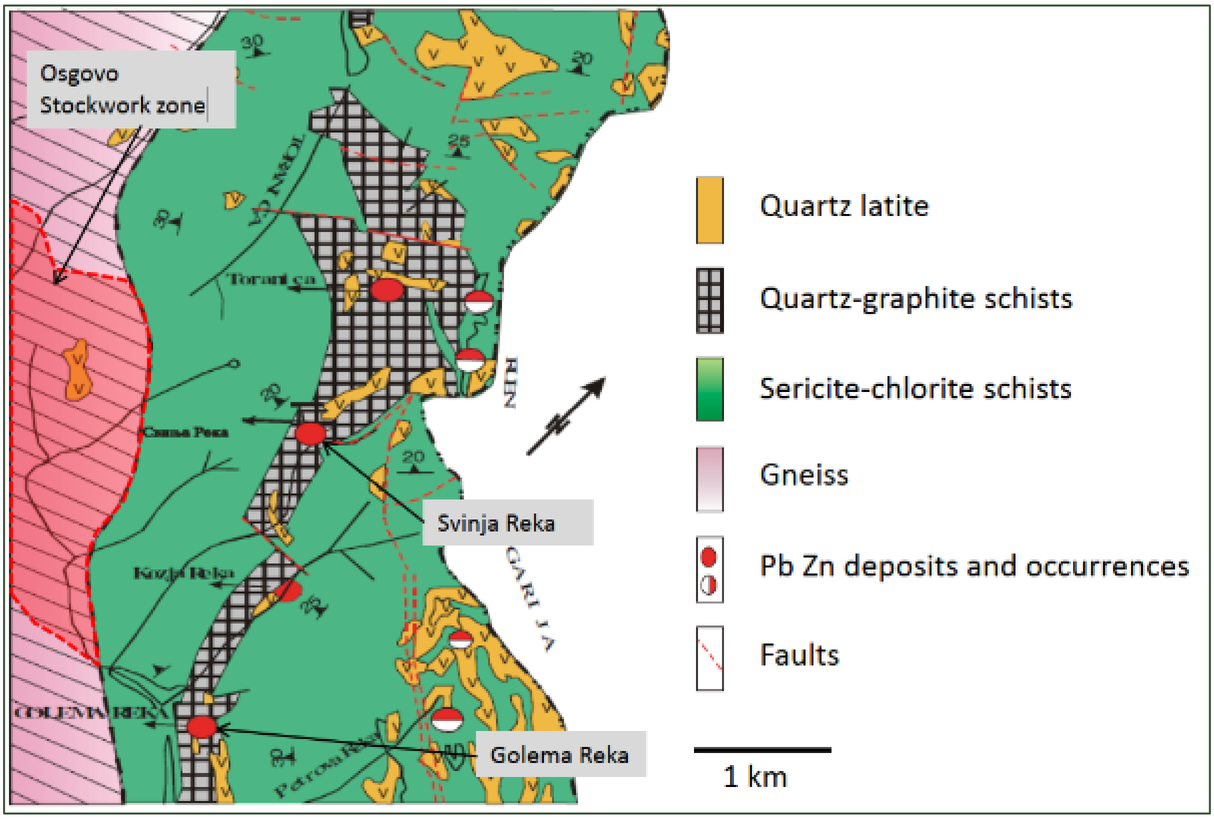
A significant feature of the system is the presence of high-sulphidation hydrothermal alteration, developed both within the porphyritic stockworks and along the margins of associated hydrothermal breccias.

The mineralisation is spatially associated with the eastern flank of a larger intermediate intrusive complex related to the Osogovo porphyry Cu-Mo system (Figure 6-4), which is interpreted to be of late Oligocene to early Miocene age. This timing places the mineralising event within a broader phase of post-collisional to extensional magmatism in the southern Balkan region, when calc-alkaline magmatism and crustal-scale faulting facilitated the ascent of metal-rich magmas and hydrothermal fluids.

At Osogovo, a prominent northwest-trending stockwork alteration zone is developed, with an estimated surface footprint of approximately 4 km². This zone comprises intensely altered quartz latite to quartz monzonite intrusions, gneissic intrusion breccias, and fine-grained aphyric igneous phases.

Mineralisation is predominantly hosted within the contact aureoles of calc-alkaline intrusive stocks, where thermal and chemical gradients promoted extensive hydrothermal replacement and vein formation. Ore deposition is commonly concentrated along lithological boundaries, particularly at the interfaces between gneisses and quartz-graphitic schists, where contrasting mechanical properties enhanced fracturing and fluid focusing. However, significant ore accumulations are also present within both the quartz-graphitic schists and the surrounding gneissic basement, demonstrating that mineralisation is not restricted to intrusive contacts but also exploits pre-existing structural and lithological weaknesses within the country rock.

Figure 6-4: Osogovo Region Geological Map



Source: SRK (2018)

6.3.2 Structural Geology

The structural geology of the Sasa Mine reflects the complex tectonic evolution of the Serbo-Macedonian Massif within the broader ABCD orogenic system. The region has undergone multiple phases of compression, metamorphism, magmatism, and later extensional reactivation associated with closure of the Tethys Ocean and subsequent Alpine collision. Structurally, the area is characterised by north-northwest-trending tectonic belts inherited from the Alpine orogen.

At the deposit scale, the geology is dominated by strongly deformed Precambrian to Palaeozoic metamorphic rocks, including gneisses, mica schists, quartz-graphite schists, marbles, and amphibolites. Many lithological contacts are tectonic in nature, and deformation has produced pronounced foliation, schistosity, local mylonitisation, and tight folding. The Lower Palaeozoic quartz-graphite schist sequence forms a northwest-trending structural corridor extending for approximately 25 km.

Mineralisation is strongly structurally controlled and occurs as bedding-concordant, stratabound lenses that dip approximately 35° to the southwest. Ore zones are localised along lithological contacts and bedding-parallel fault zones, particularly between quartz–graphite schists, marbles, and gneissic rocks. Repeated deformation and fracture reactivation created favourable pathways for hydrothermal fluid flow, resulting in metasomatic replacement, skarn development, and stockwork veining associated with Tertiary calc-alkaline intrusive activity.

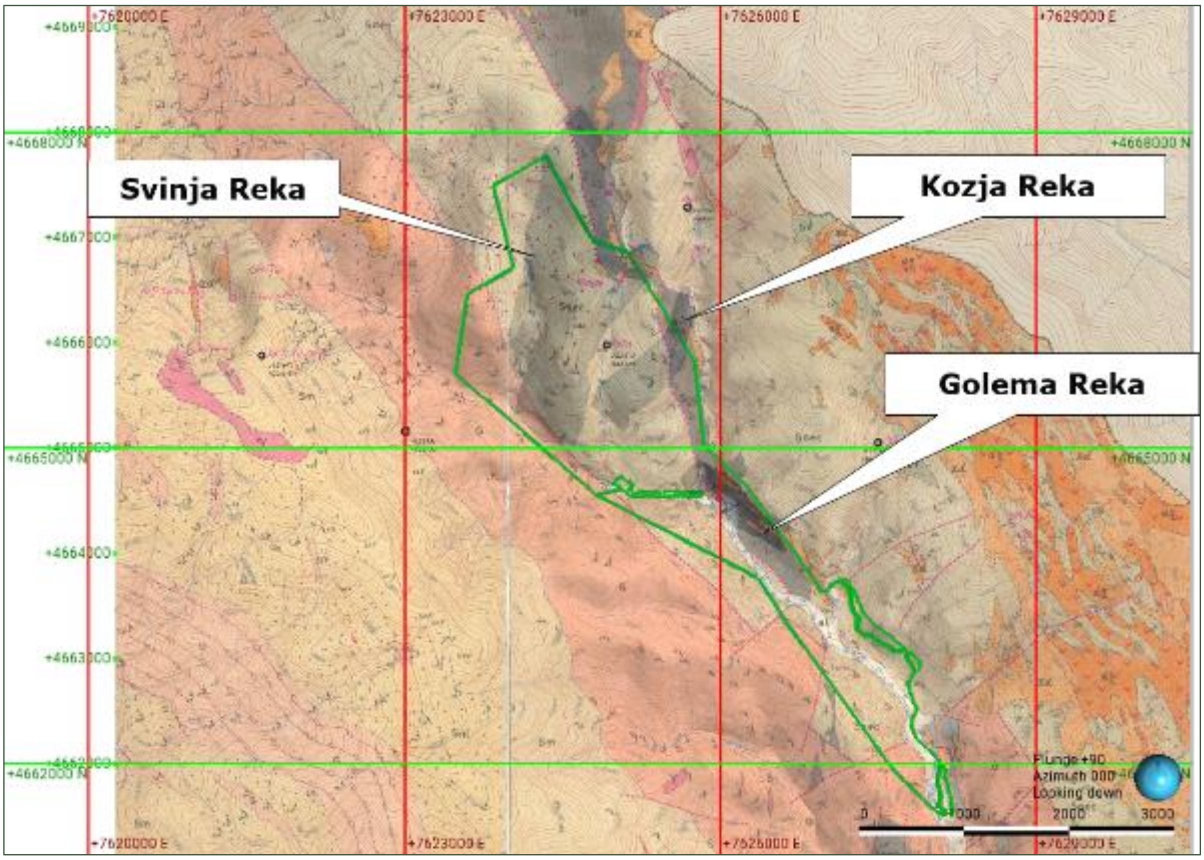
Structural reactivation and local deformation are interpreted to have influenced orebody geometry, continuity, thickness distribution and local grade variability within both Svinja Reka and Golema Reka. Underground mapping and drilling indicate that local faulting, shearing and competency contrasts can segment mineralised horizons and locally disrupt continuity. These structural controls are important and are considered by Rudnik SASA DOOEL during geological domaining, short-term grade control and Mineral Resource classification.

6.3.3 Mineralization

Lead-zinc-silver mineralisation at Sasa occurs as bedding-concordant ore bodies hosted mainly within Lower Palaeozoic quartz-graphite schists and marbles at Svinja Reka, while at Golema Reka it is primarily developed within gneissic host rocks.

The mineralisation is genetically linked to the emplacement of Tertiary volcanic and subvolcanic intrusions dated at approximately 27 - 24 Ma, which cut Precambrian gneisses and Palaeozoic schists and marbles of the Serbo-Macedonian Massif. Heat and fluid flow associated with these intrusions, combined with deformation along bedding-parallel faults, drove intense hydrothermal alteration and metasomatism of Mn-bearing sedimentary protoliths. This process led to the development of skarn-like calc-silicate assemblages, including calcite, actinolite, chlorite, and epidote, alongside the precipitation of base metal sulphides (Serafimovski et al. 2006).

Figure 6-5: Relationship and Locations of the Svinja Reka, Kozja Reka, and Golema Reka Zones



Source: Rudnik SASA DOOEL (June 2026)

Mineralisation at Sasa is interpreted as a structurally influenced replacement-style lead-zinc-silver skarn-hydrothermal system. Mineralisation continuity is controlled by a combination of favourable host lithologies, bedding-parallel replacement zones and later structural modification. The mineralised lenses are therefore not interpreted as purely stratiform bodies, but as lithologically favourable and structurally modified replacement zones affected by local faulting, shearing and competency contrasts between schist, marble and gneissic host rocks.

Mineralisation preferentially occurs within favourable reactive lithologies, and structurally prepared zones associated with hydrothermal fluid pathways. Sulphide assemblages are dominated by sphalerite, galena, pyrite and pyrrhotite, with local textural and mineralogical variation interpreted to reflect multiple hydrothermal stages and variable fluid-rock interaction conditions. Alteration assemblages, gangue relationships, replacement textures and local brecciation provide important support for geological interpretation and domaining.

The ore bodies occur as well-defined, partly mined lenses that dip gently to the southwest at around 35°. Individual mineralised zones typically vary from 2 to 30 m in true thickness and are arranged as subparallel sheets, commonly two to three ore horizons, separated by relatively unmineralised interburden ranging from 1 to 10 m. Along strike and down-dip, the ore lenses display characteristic pinch-and-swell geometry, with a general tendency towards a southerly plunge, reflecting structural control during deformation and fluid focusing. At Svinja Reka the thickness of the mineralised lenses reduces with increasing depth of the orebody.

Hydrothermal and metasomatic processes operated through multiple stages, particularly within quartz-muscovite-graphite schists and associated marble horizons (locally termed cipolines). Mineralisation occurs in two main structural styles: massive, lens-shaped metasomatic replacement bodies developed parallel to foliation (Figure 6-6), and more irregular elongated stockwork zones with variable vein density and intensity. The geometry and distribution of stockwork mineralisation are highly variable, reflecting fluctuating permeability and repeated reactivation of fracture networks.

Despite local disruption by tight folding, including kink structures and rapid lateral changes in thickness, the mineralised horizons exhibit notable continuity along strike. Thicker zones are commonly associated with fold hinges, particularly near fold noses, while barren marble lenses tend to occur along the margins of ore bodies and within zones of maximum dilation or structural transition.

Figure 6-6: 754 Level Middle Orebody Block 2

Source: Rudnik SASA DOOEL (June 2026)

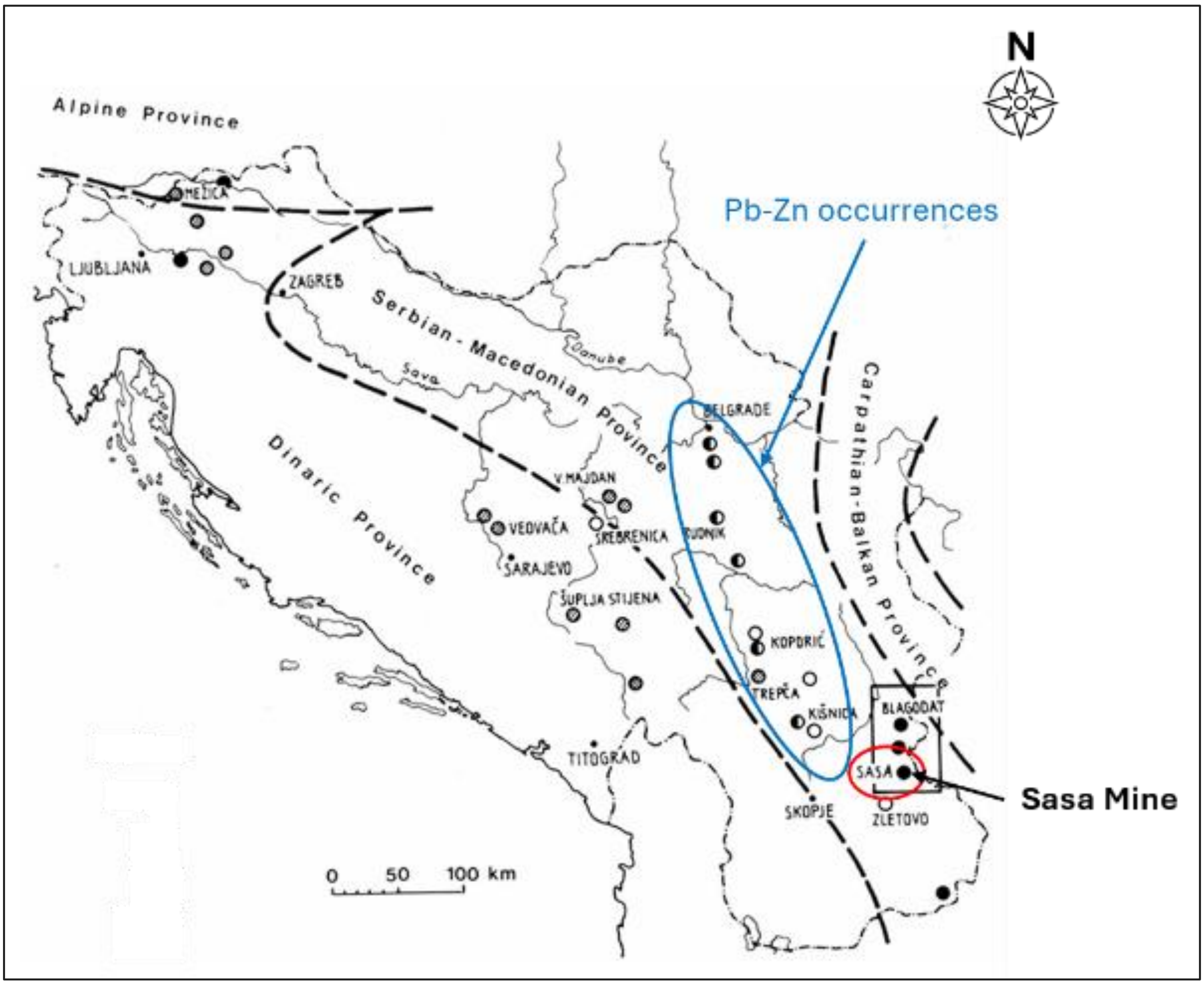
7.0 Deposit Types

The Sasa deposits (Svinja Reka and Golema Reka) are located within the Serbian Macedonian Province which extends through Serbia, North Macedonia, Bulgaria, and eastern Greece into Turkey.

The province hosts numerous lead-zinc deposits (Figure 7-1) which have largely been interpreted as epigenetic-hydrothermal formations and, as such, are related to the Tertiary magmatism which is predominantly present as occurrences of granodiorite and quartz latite.

The deposits are typically stratabound or stratiform and have been affected by regional metamorphism. As a result, they show mineral transformations, redeposition, re-mobilisation and recrystallization of their primary mineral constituents (Tufar and Strucl, 1984).

Figure 7-1: Serbian Macedonian Province



Source: Modified from Tufar and Strucl (1984)

The Sasa deposits are considered as polymetallic lead-zinc-silver skarn-hydrothermal replacement systems developed within favourable metamorphic host rocks and influenced by later structural reactivation.

While mineralisation is broadly stratabound and commonly follows favourable lithological horizons, local geometry, thickness and grade distribution are also influenced by faulting, shearing, replacement intensity and competency contrasts. This interpretation provides the basis for ongoing exploration targeting and geological domaining.

Metasomatic skarn-hydrothermal systems develop where magmatic fluids interact chemically with reactive carbonate-rich host rocks, adjacent to intrusive bodies. During emplacement and cooling of an intrusion, high-temperature, metal-bearing fluids migrate outward and drive intense metasomatism, replacing the original carbonate assemblage with calc-silicate minerals. These skarn zones commonly evolve through prograde and retrograde alteration stages, with early high-temperature mineral formation followed by later hydration and sulphide deposition as fluid temperatures decline. The hydrothermal component is critical for transporting and precipitating economically significant metals, including Fe, Cu, Zn, Pb, W, Mo, Au, and Sn. Fluid flow pathways are typically controlled by structural permeability, lithological contrasts, and intrusive contacts, resulting in spatially complex mineralisation patterns.

At Sasa, skarn development occurs primarily through the replacement of marble horizons, while the associated hydrothermal lead-zinc-silver mineralisation is present both as replacement bodies and as open-space infill within fractures and structurally prepared zones.

The skarn assemblages display a well-developed zonal mineralogical pattern and are composed predominantly of calc-silicate minerals, including Fe-Mn pyroxenes, Fe-Mn pyroxenoids, garnet, ilvaite, and epidote, together with magnetite, pyrite, and pyrrhotite. Superimposed upon these skarn assemblages is a later hydrothermal sulphide stage containing argentiferous galena, sphalerite, pyrite, and subordinate chalcopyrite.

Three principal stages of mineralisation have been identified at Sasa (Palinkas *et al.* 2013).

The earliest stage involved metasomatic skarn formation during a prograde phase, driven by high-temperature, saline fluids that were most likely derived from a distal magmatic source. The lack of direct mineralised intrusive contacts, together with the manganese-rich mineral assemblage, suggests that mineralisation developed mainly through infiltration of mineralising fluids into favourable host rocks, rather than through classic diffusion-driven contact metasomatism adjacent to intrusions.

The second stage corresponds to retrograde hydrothermal alteration, during which earlier skarn minerals were progressively altered and overprinted by assemblages including ilvaite, chlorite, magnetite, pyrrhotite, carbonates, and quartz. This stage was accompanied by the main phase of sulphide deposition and is interpreted to reflect mixing between magmatic and meteoric fluids within the evolving hydrothermal system.

The final stage involved the emplacement of post-ore gangue minerals, dominated largely by carbonate phases precipitated from cooler and relatively diluted hydrothermal fluids during the waning stages of the mineralising event.

8.0 Exploration

Exploration work conducted by previous owners is considered historical. Activities conducted prior to CAML are summarized in Section 5, while the most recent exploration work completed by previous owner Lynx Europe SPLLC Skopje (Lynx) is summarized in Section 8.1.

Exploration completed by CAML is summarized in Section 8.2.

8.1 Exploration by Lynx (November 2015 to November 2017)

Exploration work completed by Lynx comprised the following:

- Underground geological mapping.
- Surface and underground diamond drilling at Svinja Reka. Underground drilling focussed on mineralization at the 990, 910, and 830 level horizons of the mine, while surface drilling targeted potential extensions to mineralization below 830 level.
- An IP geophysical survey was completed during early 2016 to explore for extensions to the mineralisation towards the south and down-dip from the currently drilled areas. Multiple test profiles through known mineralisation were completed to calibrate the survey. The survey could not accurately delineate the mineralisation in these profiles, so this technique has not been used to guide future exploration drilling programmes.

8.2 Exploration by CAML (2017 - current)

Exploration work completed by CAML, from November 2017 to date, comprised the following:

- Underground geological mapping. Sasa Mine geologists regularly conduct geological mapping in which major rock types and mineralised contacts are recorded at 1 m drift height. The mapping is compiled on 1:250 scale master paper plans kept in the geological office.
- Underground diamond drilling at Svinja Reka focussed on increasing confidence in mineralization below the 830-level horizon of the mine.
- Surface diamond drilling at Golema Reka focussed on increasing confidence and confirming historic drill intersections in mineralization below the 830-level horizon of the mine.

Details of all drilling activities are described in Section 9.

9.0 Drilling

The primary data source on which the Mineral Resource estimate is based is a mix of surface and underground exploration drilling (Table 9-1). Exploration at Sasa began in 1954 and exploration drilling continued under different ownerships as summarised below:

- Government Ownership (1954 - 2002): 274 core holes (12 surface and 262 underground) at Svinja Reka (20,597 m) and 82 core holes (24 from surface and 58 underground) at Golema Reka (11,942 m);
- Solway (2006 - 2015): 1,080 core holes (32 surface and 1,048 underground) at Svinja Reka (80,737 m) and 22 underground drillholes at Golema Reka (1,503 m);
- Lynx (2015 - 2017): 228 drillholes at Svinja Reka with the majority of drilling being underground fan drillholes to improve confidence in active mining areas; and
- CAML (2017 onwards): A total of 1,513 drillholes have been completed by CAML (108,095 m).

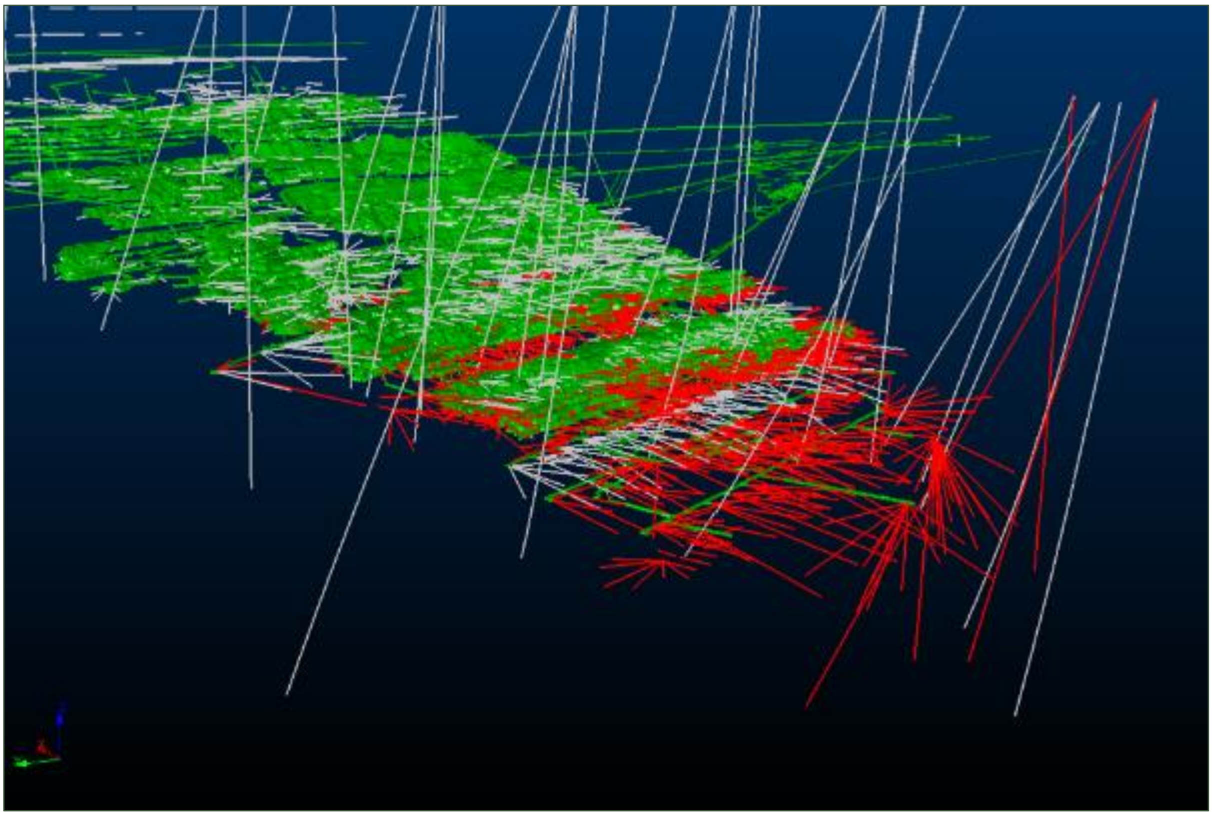
Table 9-1: Summary of Drilling at Sasa (1954 – Current)

Period	Company	Deposit	№ of Drillholes		Meterage		Total №	Total Metres
			Surface	Underground	Surface	Underground		
1954 - 2002	Government	Svinja Reka	12	262	-	-	274	20,597
		Golema Reka	24	58	-	-	82	11,942
2006 - 2015	Solway	Svinja Reka	32	1,048	-	-	1,080	80,737
		Golema Reka	-	22	-	-	22	1,503
2015 - 2017	Lynx	Svinja Reka	-	228	-	-	228	-
		Golema Reka	-	-	-	-	-	-
Since 2017	CAML	Svinja Reka	3	1,494,	2,351	99,042	1,497	101,393
		Golema Reka	16	-	6,702	-	16	6,702

Drilling orientations are designed to intersect the dominant mineralised horizons at favourable angles to maximise geological confidence and improve true thickness estimation. Underground drilling remains an important source of geological continuity information for both Mineral Resource evaluation and short-term production planning. Where intersections are oblique or structurally complex, interpretation is supported by adjacent drilling, underground mapping and geological continuity assessment.

The extent of the drilling at Svinja Reka is shown in Figure 9-1 (mined out areas (green), drillholes prior to 2017 (grey) and drillholes after 2017 (red)).

Figure 9-1: Isometric View of Svinja Reka Looking Southeast



Source: SLR (June 2026)

9.1 Drilling by Previous Operators

Historical drilling and sampling datasets have been reviewed as part of the resource evaluation process. Confidence has been assigned based on available collar survey control, downhole survey information, logging quality, assay support and consistency with modern drilling. Where appropriate, modern drilling and underground mapping have been used to validate historical geological interpretations. Lower-confidence historical information is treated conservatively during geological interpretation and Mineral Resource classification.

9.1.1 Government Ownership (1954 – 2002)

Between 1954 and 2002, a total of 274 core drillholes were completed at the Svinja Reka deposit of which 12 were drilled from surface and 262 from underground for a cumulative length of 20,597 m.

Most of the surface holes were 76 mm diameter core (changing to HQ (68 mm) after 200 m and NQ (47.6 mm) beyond 500 m) and oriented towards northeast, providing one or two mineralised intercepts at Svinja Reka, situated on section lines approximately 200 m apart.

Underground drilling was oriented towards the southwest and focused on the upper six mining levels down to 1130 masl, and at a drillhole spacing of 15 - 25 m.

At Golema Reka, some 82 core drillholes were completed for a cumulative length of 11,942 m, of which 24 were drilled from surface and 58 from underground.

Surface drilling was completed on section lines approximately 100 m apart.

Underground drilling was oriented towards the southwest on the upper two levels of the mine, and at a drillhole spacing of 20 - 50 m.

9.1.2 Solway (2006 – 2015)

Between 2006 and 2015, a total of 1,080 core drillholes were drilled at the Svinja Reka deposit for a cumulative length of 80,737 m, of which 32 were drilled from surface and 1,048 from underground.

Surface drilling used 76 mm diameter core and was typically completed as fans collared approximately 200 m apart and orientated towards the northeast. Drillhole spacings were typically 50 to 150 m.

Underground fan drilling was undertaken to a maximum depth of 840 masl at drillhole spacings of 10 - 30 m and up to 80 m between mining levels.

At Golema Reka, some 22 underground drillholes were completed for a cumulative length of 1,503 m. Drilling was oriented towards the southwest at spacings of 10 - 50 m.

9.1.3 Lynx (2015 – 2017)

Between 2015 and 2017, Lynx completed 228 underground and surface drillholes at the Svinja Reka deposit.

Most of the drilling was underground fan drilling to improve confidence in active mining areas and at drillhole spacings of 10 – 30 m (up to 80 m between mining levels) and to a maximum depth of the 840 level.

9.2 Drilling CAML (2017 onwards)

All drilling completed by CAML is by diamond core and includes underground and a lesser amount of surface drilling. To date, CAML has completed a total of 1,513 drillholes for 108,095 m. Drilling is performed by both inhouse and contractor (GEOPS, Bulgaria) rigs and managed by the Rudnik SASA DOOEL geology department.

9.2.1 Drillhole Spacing and Orientation

Surface drillholes at Svinja Reka are plotted on sections orientated northeast-southwest across the deposit and are spaced approximately 200 m apart. Drillhole lengths range from 650 to 849 m and intersection angles with the mineralisation are broadly perpendicular.

Surface drillholes at Golema Reka have intersected the mineralisation, down-dip of the mined workings, on 50 to 100 m centres.

Underground drillholes are typically drilled as fan holes at varying inclinations that intersect the mineralisation at angles ranging from perpendicular to 45° dip. Whilst non-perpendicular angles are not ideal in gauging true thickness of mineralisation, the close spaced drilling from underground removes much of this uncertainty and underground mapping from nearby development allows refinement of the model prior to mining. The spacing of the underground holes can be as close as 20 m.

9.2.2 Surveys

Collar surveys of surface drillholes are completed by Rudnik SASA DOOEL surveyors using a Precision GPS.

Underground drillhole collars are located based on total station surveys by the mine survey department and are translated to the Macedonian grid (Gauss-Krüger coordinate system, Hermannskogel datum) for storage in the master database.

Downhole survey information for the start and the end of each surface hole is completed by Rudnik SASA DOOEL surveyors. Intermediate readings at approximately every 50 m are taken using a reflex gyro probe by Geoma (Bulgaria) survey contractors.

For short underground drilling (<70m), only the start inclination of the drillhole is recorded.

9.2.3 Core Diameter

All surface drillholes are completed using 76 mm diameter, HQ or NQ diamond core. Underground drillholes are completed using 76 mm (NQ) diameter, 36 mm diameter (BQ) or AX diamond core in up-holes.

Surface drilling uses double tube core barrels and primarily produces core of 76 mm diameter.

Underground drilling produces core of 76 mm and 36 mm diameter.

9.2.4 Core Recovery

Core recovery is not recorded in the database for drillholes completed before 2017 or holes drilled from 2018 - 2020. After this date, coverage is better but not complete. Core recovery information from earlier programmes was recorded but was not transferred to the drillhole database.

Where core recovery is routinely recorded for holes drilled from 2022 onwards, total core recovery averages 86% rising to 90.5% for sample intervals with a Pb + Zn grade of >1%, and 91.3% for sample intervals with a Pb + Zn grade of >4%.

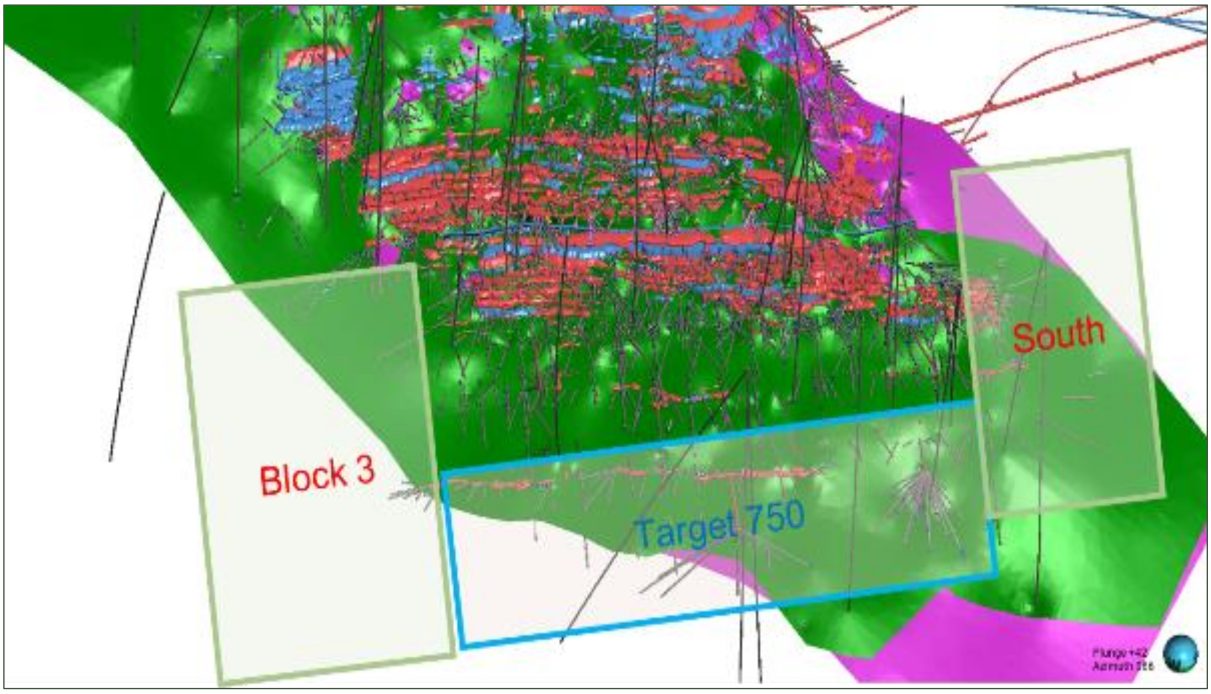
9.2.5 Core Handling

Core is typically produced in 3 m length core runs and then placed in core trays at the drill site. The trays are labelled with hole ID and downhole interval information. Marker blocks are marked up to record downhole depth intervals within the core trays. The core is transported to the Sasa Mine core logging facility at the end of each drilling shift.

9.2.6 Exploration and Infill Drill Plan (2026 – 2028)

In 2026, CAML instigated a 3-year drill programme at Svinja Reka and focused on three key areas (Block 3, Target 750, and South) as shown in Figure 9-2.

Figure 9-2: Isometric View of CAML Exploration Drill Target Areas at Svinja Reka



Source: Rudnik SASA DOOEL (June 2026)

Block 3 covers profiles 1100 to 1500 and extends on the 750, 990, and 910 levels north. Target 750 comprises profiles 300 to 1400 and mineralisation targets below 750 to the 650 level with the aim to convert Inferred resources to Indicated. South covers profiles south of 300 and extend development southwards into the hanging wall.

CAML also intends to complete a small number of deep (circa. 800 m) surface drillholes.

9.3 CP Opinion

Whilst little information exists as to the methodology and quality of the historical drilling, much of the historical drilling targeted near surface mineralisation which has mostly been depleted by mining. Notably, most of the drilling that informs the Mineral Resource estimate has been completed by CAML.

It is the CP’s opinion that the drilling and core sample collection at Sasa are an appropriate type of drilling and orientation to assess the Project and provide suitable level and quality of information required to interpret the underlying geology and mineralization and to form the basis for a Mineral Resource estimate. The samples are representative of the mineralisation and there is no evidence that the drilling or sample collection process has resulted in a bias that could materially impact the accuracy and reliability of the results.

10.0 Sample Preparation, Analyses, and Security

Samples from drill core are collected by Rudnik SASA DOOEL geological staff. Logging, sampling and sample cutting are undertaken at the on-site core logging facility at the Sasa Mine. Sample preparation and analysis is undertaken at the Sasa Mine analytical laboratory. The laboratory is not accredited. A systematic Quality Assurance / Quality Control (QA/QC) system is used to monitor the accuracy and precision of assaying. Mineralised half core is retained for archive at the core storage facility located at the mine. The procedures used in the preparation and analysis of the drill core samples are summarised in the following subsections.

10.1 Core Logging

Core is logged for lithology and numeric lithological codes and qualitative descriptions are recorded. Logging codes and descriptions match the accepted geological model and provide enough detail for modelling of the extent of mineralised domains.

Core is photographed wet and dry, and photographs are stored digitally.

10.2 Core Sampling

Sample intervals are marked out with respect to logged observations around lithological and mineralogical changes. Sample cutting is carried out using industry standard methodology with core cut lengthways using a diamond core saw in a dedicated area within the core shed. Samples are bagged in a suitable manner with individual sample number labels recorded to give proper chain of custody.

10.3 Sample Preparation

Sample preparation takes place at the on-site Sasa Mine laboratory. The laboratory processes drill core samples, plant concentrate samples and other samples as required (for example tailings monitoring). Drill core samples follow an industry standard sample preparation route with drying, crushing and pulverising to produce a -0.74 µm 50 g pulp sample for analysis.

10.4 Sample Analysis

Sample analysis is conducted at the Sasa Mine analytical laboratory located at the mine site. Analysis for Zn, Pb and Ag is by atomic absorption (AA) using three acid digestion.

The laboratory is not accredited for AA analysis used for drill core samples. However, the laboratory takes part in round robin analysis programmes and is accredited for concentrate analysis but not the analytical route used for drill core analysis.

10.5 Density Measurements

Specific gravity measurements are routinely collected from representative mineralised and waste material samples across the deposit. Density values are reviewed by lithology, mineralisation style and geological domain to ensure appropriate tonnage estimation support within the Mineral Resource model.

Measurements are undertaken on whole core samples at a facility located within the on-site core shed. Measurements are taken after logging of core whereby the geologist selects visually representative 10 cm pieces of core for analysis using the water immersion (Archimedes) method. Density standards are employed in the density determination process to monitor reliability of the results.

A total of 2,290 density measurements has been taken for Svinja Reka, and 81 density measurements have been taken for Golema Reka.

Whilst the methodology is industry standard, not all sample intervals are tested. The CP recommends that core from every sample should be measured for density to potentially improve local density estimates.

10.6 Sample Security and Chain of Custody

Sample collection and transportation of drill core is undertaken by Rudnik SASA DOOEL geology department staff. Exploration core boxes are transported to the core logging facilities located at the Sasa Mine site. Once logging and sampling have been completed, mineralised half core is retained for between three to five years. The core is transferred to storage facilities located at the mine site. The drill core boxes are covered, and the storage facilities are dry with internal lighting and metal shelving. Pulp duplicate material is stored in the same facility.

The authors consider the sample security and chain of custody procedures to be of a high standard.

10.7 Quality Assurance and Quality Control Programmes

The implementation of a quality assurance and quality control programme (QA/QC) programme is industry best practice and involves establishing appropriate procedures and the routine insertion of reference material, blanks and duplicates to monitor the sampling, sample preparation and analytical process. Analysis of QC data is made to assess the reliability of sample assay data and the confidence in the data used for the estimation.

Sample analysis QA/QC consists of the insertion of control samples (Certified Reference Material) and reject pulp samples as check duplicates. It is reported on-site that, historically, blank samples were used in this process but are not currently. QA/QC samples are inserted at a rate of roughly one CRM and two duplicate samples per batch of 30 submitted to the on-site laboratory. Results for a total of 16 different CRM used over recent years are available but only two are currently in use. CRM are sourced from Geostats Pty Ltd (Australia), and OREAS (Australia) and results are checked against the published target and standard deviation limits. Duplicate results are reviewed regularly against percentage tolerances from the primary sample result and any results outside acceptable tolerance limits are investigated, with re-assay or further validation completed where required. This process supports analytical reliability for exploration, grade control and Mineral Resource estimation.

10.7.1 Certified Reference Material

Results of the regular submission of Certified Reference Material (CRM, or *standards*) are used to identify issues with specific sample batches, and biases associated with the non-certified internal laboratory (Sasa Laboratory).

CRM samples from Geostats and OREAS are incorporated into the assay process by a mine geologist at a frequency of two per batch of 30 samples, regardless of the sample type. The mine geologist records the identification numbers of the CRM samples introduced into the assay stream and assesses them as either a pass or fail upon receiving laboratory results. Batches with failed CRM results undergo re-analysis. CRM results are digitally documented using predefined pass-fail criteria in MX Deposit, enabling the tracking of pass and fail rates for each reference material. The compiled results are included in a monthly report and annual report, which is shared with relevant laboratories involved in the assay process.

A total of sixteen different CRMs were inserted between 2022 and 2025, totalling 1,071 individual samples analysed for Zn, Pb and Ag. A summary of these CRMs analysis at the Sasa Mine laboratory are presented in Table 10-1, Table 10-2 and Table 10-3 showing Zn, Pb and Ag results respectively. In analysis of the CRM results, SLR prepared control charts and analysed temporal and grade trends, reviewed the data for low and high biases, and the failure rate of each CRM against a failure criteria of ±3SD.

Table 10-1: Expected Values and Ranges of CRM – Zn Results

CRM	Period Range	№ Samples	Std Dev	Mean (Zn %)	Expected Value (Zn%)	№ Outliers	Bias (%)	Outliers (%)
GBM910-11	2022-2024	137	0.25	3.75	3.93	18	-4.7	13.1
GBM910-12	2022-2025	131	0.55	4.33	4.49	2	-3.6	1.5
GBM913-11	2022-2023	80	0.7	8.06	7.9	2	2.1	2.5
GBM913-12	2022-2024	76	0.3	7.62	7.26	8	5	10.5
GBM916-9	2022-2025	169	0.23	3.05	2.97	11	2.7	6.5
GBM310-13	2022-2025	84	1.37	10.75	10.87	3	-1.1	3.6
GBM916-11	2022-2025	95	0.22	6.15	6.09	0	1	0
GBM915-11	2022-2023	10	0.14	0.3	0.34			
GBM919-13	2022-2025	95	0.7	6.58	6.59	2	-0.1	2.1
GBM920-7	2022-2025	52	0.12	4.19	4.1	1	2	1.9
GBM311-11	2022-2025	53	0.49	3.35	3.11	4	7.7	7.5
GBM923-14	2022-2025	9	0.08	4.18	4.06	0	2.9	0
GBM319-11	2022-2025	8	0.01	1.48	1.58	0	-6.38	0
OREAS 313	2022-2025	5	0.04	5	3.44	0	-3.1	0
OREAS 314	2022-2025	9	0.05	4.6	4.75	0	-3.2	0
OREAS 135b	2022-2025	4	0.05	2.65	2.73	0	-3.1	0

Table 10-2: Expected Values and Ranges of CRM – Pb Results

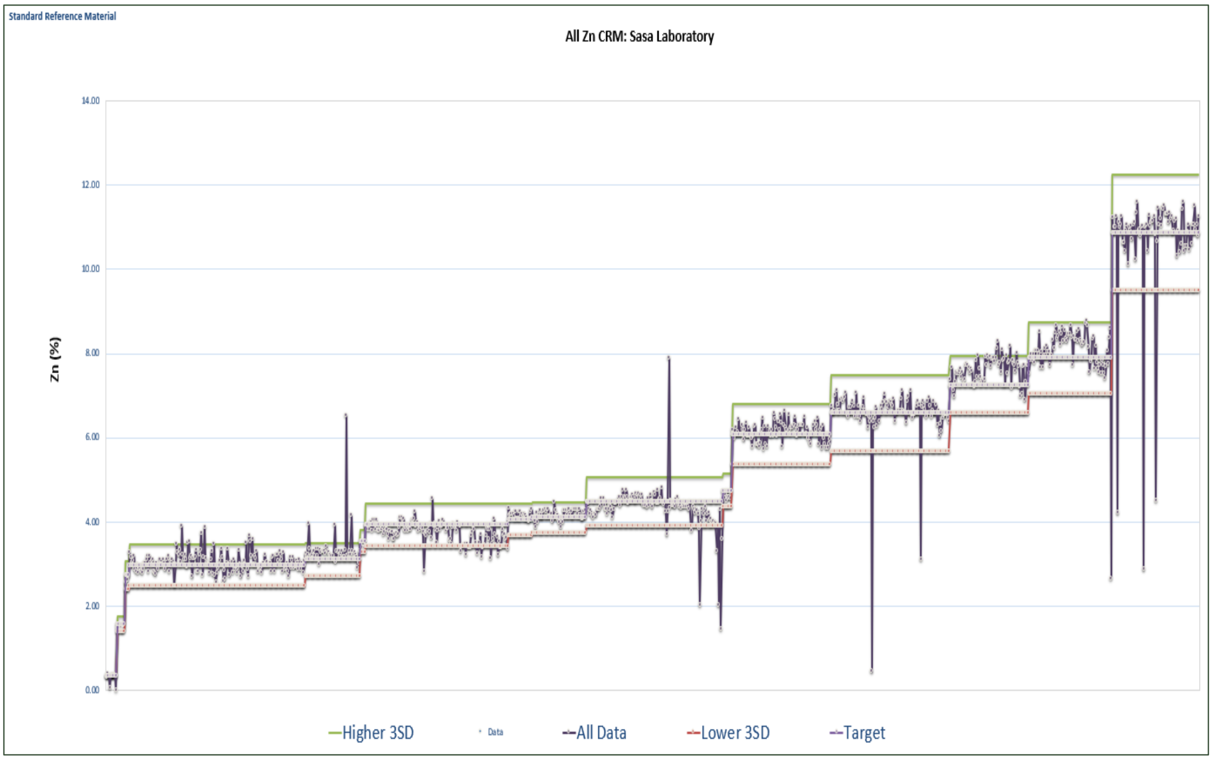
CRM	Period Range	№ Samples	Std Dev	Mean (Pb %)	Expected Value (Pb%)	№ Outliers	Bias (%)	Outliers (%)
GBM910-11	2022-2024	137	0.17	1.34	1.34	11	0.29	8
GBM910-12	2022-2025	131	0.24	2.18	2.16	3	0.89	2.3
GBM913-11	2022-2023	80	0.33	3.6	3.58	5	0.43	6.3
GBM913-12	2022-2024	76	0.22	2.42	2.3	5	5.44	6.6
GBM916-9	2022-2025	169	0.09	0.29	0.27	28	5.7	16.6
GBM310-13	2022-2025	84	0.3	2.18	2.14	2	1.88	2.4
GBM916-11	2022-2025	95	0.01	0.27	0.28	0	-4.18	0
GBM915-11	2022-2023	10	0.19	0.1	0.02			
GBM919-13	2022-2025	114	0.07	0.38	0.4	2	-3.92	1.8
GBM920-7	2022-2025	52	0.08	3.15	3.08	0	2.32	0
GBM311-11	2022-2025	53	0.29	1.15	1.07	4	6.93	7.5
GBM923-14	2022-2025	23	8	3.17	3.09	0	2.65	0
GBM319-11	2022-2025	8	0.03	1.16	1.14	0	1.93	0
OREAS 313	2022-2025	5	0.12	3.89	3.9	0	-0.31	0
OREAS 314	2022-2025	9	0.09	3.75	3.9	0	1.26	0
OREAS 135b	2022-2025	4	5	1.75	1.71	0	2.34	0
GBM910-11	2022-2024	137	0.17	1.34	1.34	11	0.29	8

Table 10-3: Expected Values and Ranges of CRM – Ag Results

CRM	Period Range	№ Samples	Std Dev	Mean (Ag g/t)	Expected Value (Ag g/t)	№ Outliers	Bias (%)	Outliers (%)
GBM910-11	2022-2024	138	2.59	18.77	19.2	15	-2.23	10.9
GBM910-12	2022-2025	131	2.58	22.98	23.5	11	-2.23	8.4
GBM913-11	2022-2023	80	3.41	32.71	32.5	6	0.66	7.5
GBM913-12	2022-2024	77	2.56	22.03	21.8	8	1.04	10.4
GBM916-9	2022-2025	169	5.49	39.14	40.6	16	-3.59	9.5
GBM310-13	2022-2025	84	2.78	30.65	30.7	3	-0.15	3.6
GBM916-11	2022-2025	95	3.3	66.36	68	0	-2.41	0
GBM915-11	2022-2023	10	6.1	4.8	5.3	5	11.63	50
GBM919-13	2022-2025	114	1.62	17.46	17.3	2	0.95	1.8
GBM920-7	2022-2025	52	11.125	127.73	133.9	1	-4.61	1.9
GBM311-11	2022-2025	53	15.2	20.75	19.6	2	5.89	3.8
GBM923-14	2022-2025	23	7.9	132.7	133.3	0	-0.45	0
GBM319-11	2022-2025	8	2.86	107.25	108	0	-0.69	0
OREAS 313	2022-2025	5	2.71	73.8	75.6	0	-2.38	0
OREAS 314	2022-2025	9	1.76	53.67	54.8	0	-2.07	0
OREAS 135b	2022-2025	4	3	53	52.9	0	0.19	0
GBM910-11	2022-2024	138	2.59	18.77	19.2	15	-2.23	10.9

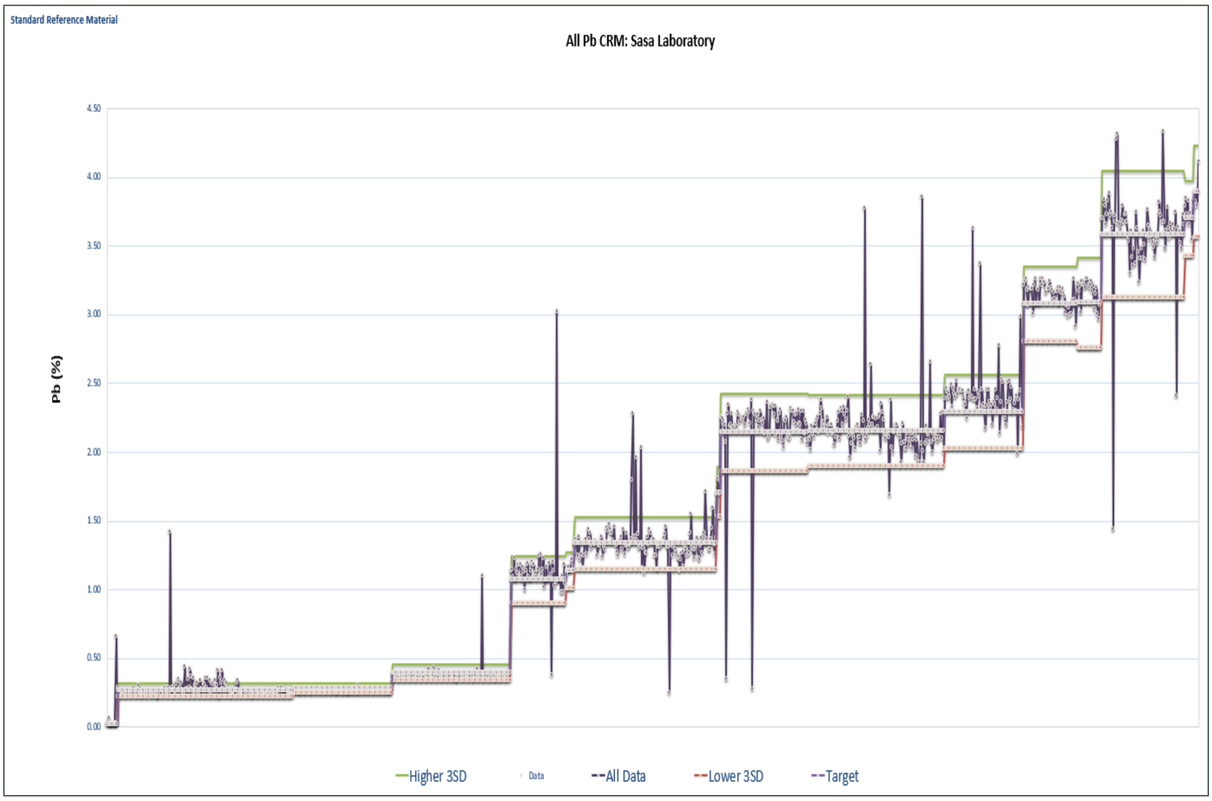
Between 2022 and 2025, the Sasa laboratory carried out analysis of 16 different CRM. Whilst a number of failures were observed, the majority of these are ascribed to CRM sample transcription errors, as the assay results of the failures tended to fall within the expected results bracket of a different CRM as shown in Figure 10-1 to Figure 10-3. No systematic bias was seen in the CRM results as seen in Figure 10-4 to Figure 10-6.

Figure 10-1: Compiled CRM Results for Zn (%) - Sasa Laboratory 2022-2025



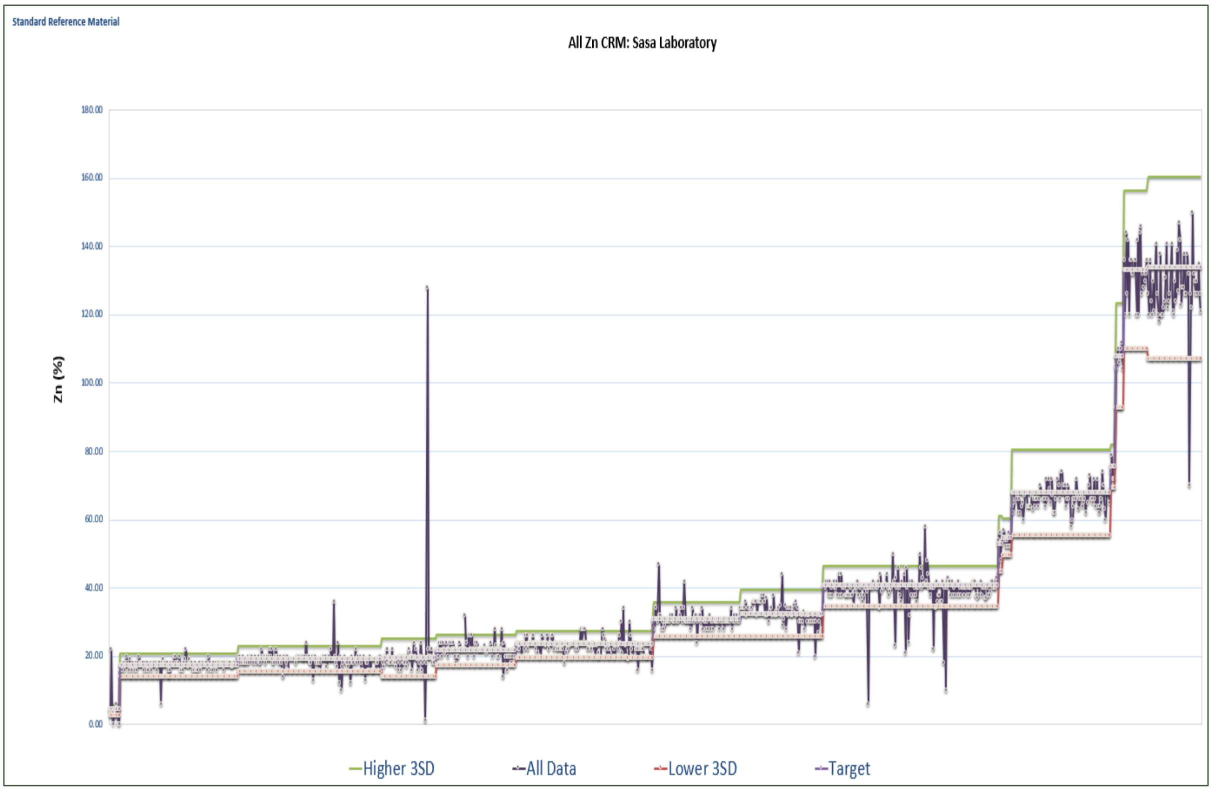
Source: SLR (June 2026)

Figure 10-2: Compiled CRM Results for Pb (%) - Sasa Laboratory 2022-2025



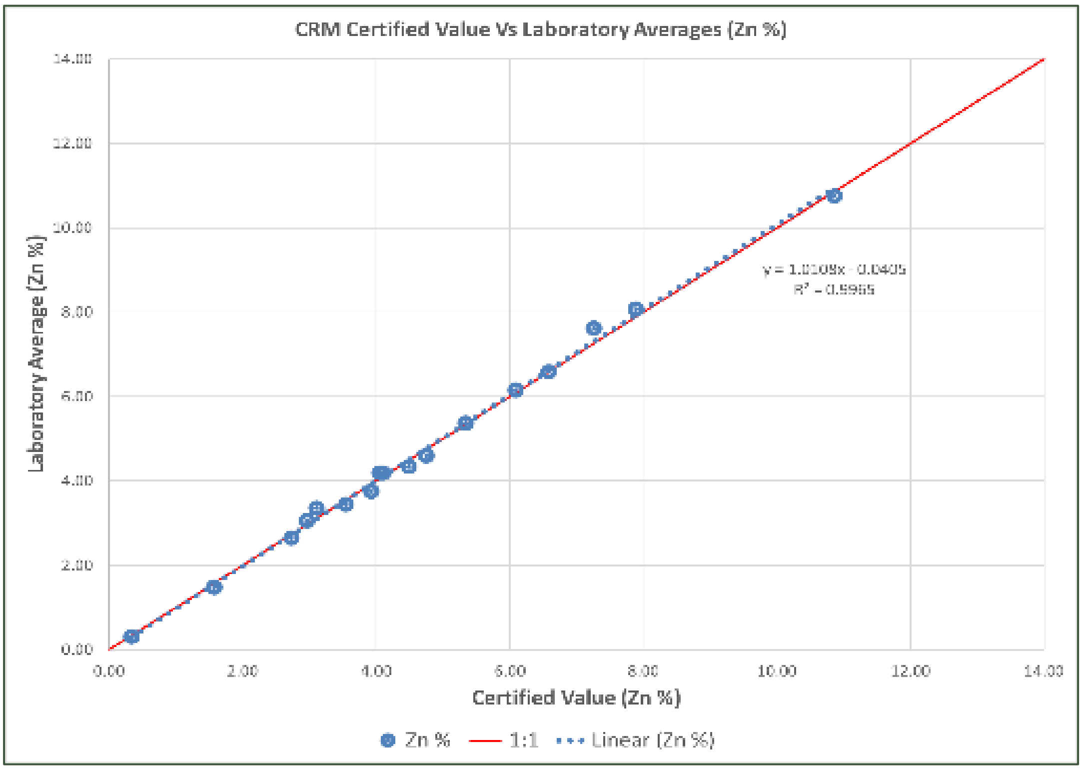
Source: SLR (June 2026)

Figure 10-3: Compiled CRM Results for Ag (g/t) - Sasa Laboratory 2022-2025



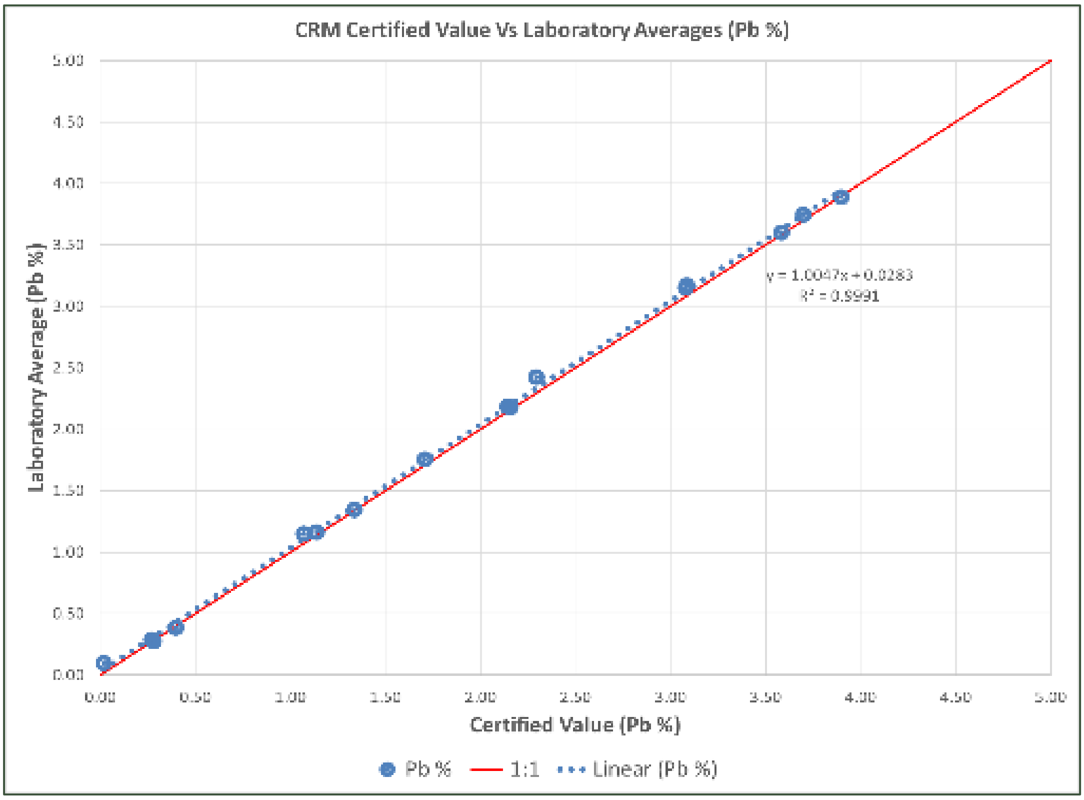
Source: SLR (June 2026)

Figure 10-4: Mean CRM Results vs Expected Results - Sasa Laboratory 2022-2025 for Zn (%)



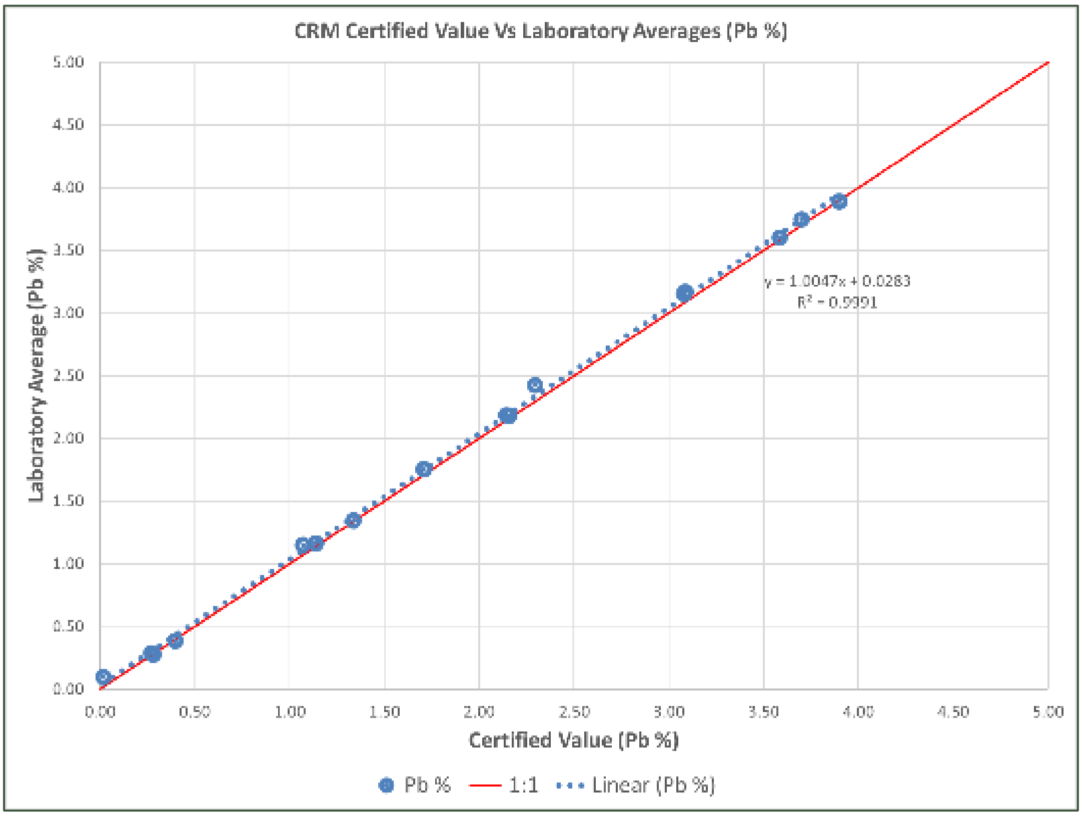
Source: SLR (June 2026)

Figure 10-5: Mean CRM Results vs Expected Results - Sasa Laboratory 2022-2025 for Pb (%)



Source: SLR (June 2026)

Figure 10-6: Mean CRM Results Versus Expected Results Sasa Laboratory 2022-2025 for Ag (g/t)



Source: SLR (June 2026)

10.7.2 Duplicates

The precision of sampling and analytical results can be measured by re-analysing a portion of the same sample using the same methodology. The variance between the results is a measure of their precision. Precision is affected by mineralogical factors such as grain size, distribution and inconsistencies in the sample preparation and analysis processes.

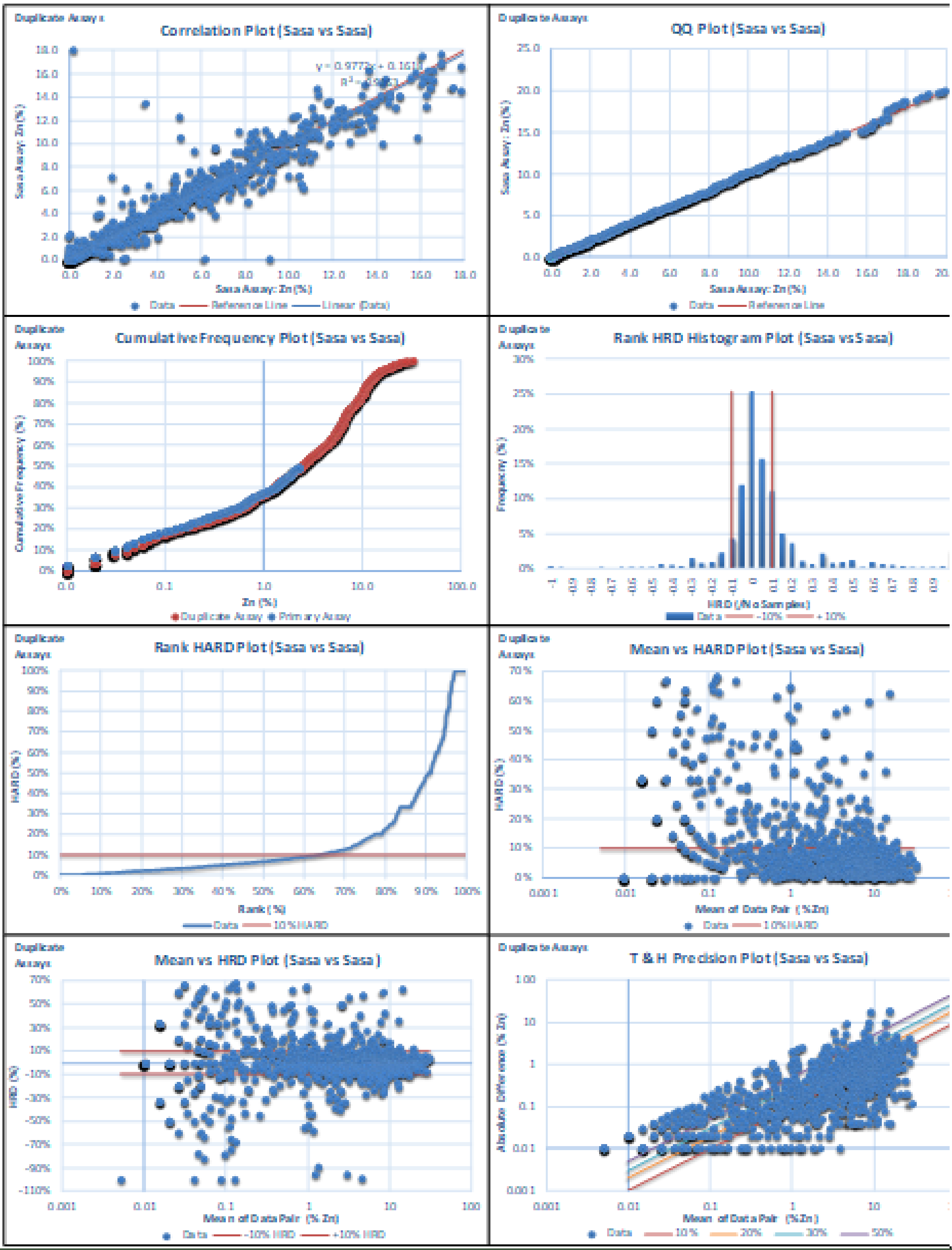
10.7.2.1 Internal Duplicates

SLR were provided with results for pulp duplicate assay results for samples analysed at the Sasa mine laboratory between 2022 and 2025.

A summary of the review undertaken by SLR is shown in Figure 10-7, Figure 10-8 and Figure 10-9 for Zn, Pb and Ag respectively. Sample pairs where assay values were 0% or 0 g/t were omitted.

Figure 10-7: Duplicate Assay Results for Zn (%) - Sasa Laboratory 2022-2025

SASA Project | All Campaign | Sasa vs Sasa | Duplicate Assays | Zinc						
Primary Laboratory:	Sasa		Primary	Duplicate		Primary vs Duplicate
Assay Method 1:	AA	Minimum:	0	0	Correlation Coefficient:	0.97
Duplicate Laboratory:	Sasa	Maximum:	30	32.4	Mean HARD:	15.88%
Assay Method 2:	AA	Mean:	4.59	4.65	Median HARD:	6.56%
Number of Pairs:	2027	Median:	2.45	2.42	Mean HRD:	4.80%
Element:	Zn	Standard Deviation:	5.48	5.53	Median HRD:	0.00%
Unit:	%	Coefficient of Variation:	1.19	1.19	% of Pairs <10% HARD:	62.8%



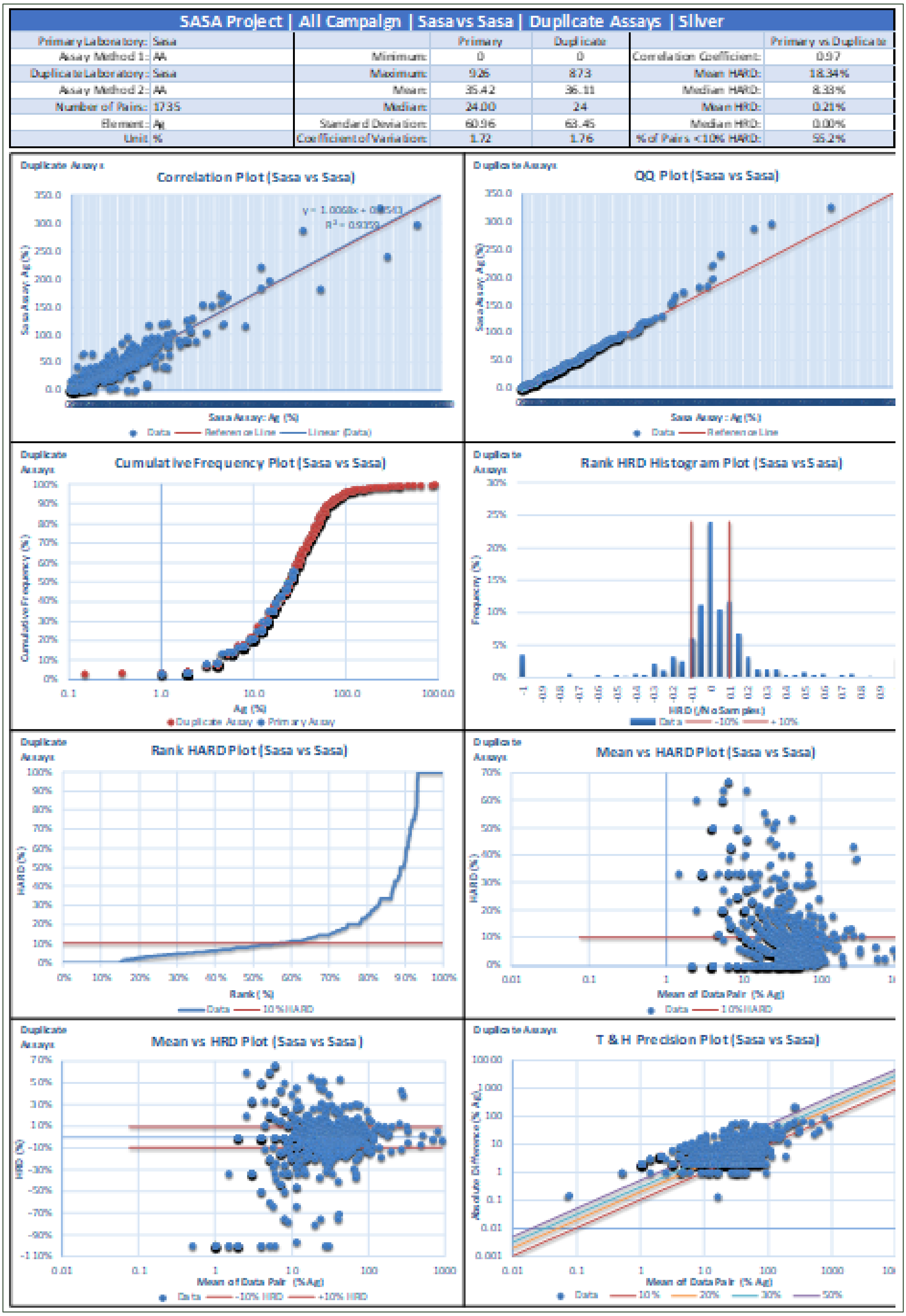
Source: SLR (June 2026)

Figure 10-8: Duplicate Assay Results for Pb (%) - Sasa Laboratory 2022-2025

SASA Project | All Campaign | Sasa vs Sasa | Duplicate Assays | Lead						
Primary laboratory:	Sasa		Primary	Duplicate		Primary vs. Duplicate
Assay Method 1:	AA	Minimum:	0	0	Correlation Coefficient:	0.97
Duplicate laboratory:	Sasa	Maximum:	65.74	65	Mean HARD:	12.59%
Assay Method 2:	AA	Mean:	5.64	5.84	Median HARD:	5.57%
Number of Pairs:	2036	Median:	4.18	4.41	Mean HRD:	4.59%
Element:	Pb	Standard Deviation:	6.08	6.29	Median HRD:	0.81%
Unit:	%	Coefficient of Variation:	1.08	1.08	% of Pairs <10% HARD:	67.1%

Source: SLR (June 2026)

Figure 10-9: Duplicate Assay Results for Ag (g/t) - Sasa Laboratory 2022-2025



Source: SLR (June 2026)

10.7.2.2 External Duplicates

In 2021 and 2024, two batches of duplicate pulp samples to Eurotest-Control EAD in Sofia, Bulgaria, for external checks. The laboratory is accredited. In 2021, 40 samples were sent and an additional 60 samples were sent in 2024. The results were reviewed by SLR and shown in Table 10-4.

Table 10-4: External Check Duplicates

Duplicate	Number of Pairs	Mean Primary Samples (%)	Mean Duplicate Samples (%)	CV Primary Samples	CV Duplicate Samples	Correlation Coefficient	% of Pairs <10% HARD
2021							
Ag (g/t)	40	48.07	51.28	1.84	1.90	0.98	71.0
Pb (%)	40	5.71	5.69	1.65	1.53	0.99	85.0
Zn (%)	40	3.67	3.68	1.45	1.58	0.99	73.0
2024							
Ag (g/t)	60	23.18	22.19	0.87	0.84	0.96	78.3
Pb (%)	60	4.79	4.55	0.89	0.88	0.95	95.0
Zn (%)	60	5.77	5.65	1.13	1.14	0.96	86.0

10.7.3 CP Opinion

In the CP's opinion, the sample preparation, analysis, and security procedures at Sasa are adequate for use in the estimation of Mineral Resources. There are no factors associated with sampling or sample preparation that would significantly affect the accuracy or reliability of the samples or assay results. In the CP's opinion, the QA/QC programme as designed and implemented by CAML is adequate and the assay results within the database are suitable for use in a Mineral Resource estimate. The results of the QA/QC procedures demonstrate that the assay results fall within acceptable ranges of accuracy and precision, affirming the adequacy of the resulting database to underpin the estimation of Mineral Resources.

CRM samples currently used for monitoring accuracy are from a reputable source and are appropriate for the type and tenor of mineralisation being analysed. CRM samples are included in the sample stream at a reasonable rate. CRM results are monitored on site with suitable warning and failure limits recognised although no systematic check for overall bias is applied so long as results fall within the expected standard deviation limits. Failure rate is generally low, and a number of these failures appear to be the result of sample mislabelling (results are similar to other CRM target grades). Notwithstanding, failures are investigated and resolved prior to approval.

Duplicate pulp samples currently used for review of analytical precision are submitted at an appropriate rate and results are regularly monitored within defined tolerances.

A SLR review of duplicate analysis results indicated a slightly higher than expected rate of failure for the type of mineralisation and type of duplicate sample. However, the failure rates are higher in lower grade samples (<1% Zn or Pb, below domaining cut-off grade) with higher grade samples showing better correlation.

Currently no blank samples are used for monitoring potential contamination or sample result switching. SLR recommends that appropriate blank material be sourced and used for this purpose and that two types of blank samples be considered for completeness. Firstly, a coarse blank sample for monitoring of potential contamination and sample switching from sample preparation through to sample analysis; and secondly, pulp blank samples to monitor contamination at the analytical stage. SLR suggests one of each type of blank be submitted within each batch of samples with warning limits being established at 5 x detection level of any of Pb, Zn and Ag and failures at 10 x detection limit.

A number of CRMs, whilst exhibiting low overall failure rates, show a slight positive bias. For Zn these are GBM913-12 and GBM311-11, and 923-14 and 920-7 for Pb. These are some of the higher-grade CRM being analysed. Whilst not necessarily a cause for concern, SLR recommends that selected higher grade Zn and Pb reject pulp samples are sent for external analysis as a check exercise to determine if this bias is consistent across all higher-grade samples, or if the issue is with the CRM batch. It is possible the bias may be a result of contamination during certain periods, but without blank sample results, this is impossible to rule out.

External pulp duplicates were submitted in 2021 and 2024. The Pb check assays perform well and improve from 2021 to 2024. The performance of the Zn check assays also increases from 2021 to 2024, whilst the performance of the Ag check assays is 71 - 79% of check samples being <10% Half Absolute Relative Difference (HARD). Assay pairs with the highest variance are predominantly associated with lower grades (i.e., less than 10 ppm Ag and less than 1% Zn). The overall performance of the check assays for both periods is considered acceptable. SLR recommends that internal analysis should be continually verified using external checks at an independent accredited laboratory to identify any drift in analytical quality in a timely manner.

Whilst the evaluation of density is regularly undertaken based on a conventional site-based method it would be beneficial if the sample rate is increased and that confirmatory laboratory testwork is undertaken to validate the results. It is understood that the ore is becoming harder with depth and thus accurate density determination may become more significant to the Mineral Resource estimate.

Additional supporting duplicate QA/QC samples for the current drilling programme could be implemented to review the whole cycle of the process of sample selection through to analysis. For full QA/QC coverage SLR recommends the additional analysis of second halves of core as field duplicates to review representativeness of the sampling methodology, and coarse reject material (after the primary crushing stage) to review the sample representativeness attained prior to pulverisation. However, the addition of these sample types is seen as less of a priority than carrying out regular external duplicate checks using pulp rejects and the inclusion of blank samples, and additional sample numbers may further impact on the ability of the laboratory to process samples in a timely manner.

11.0 Data Verification

11.1 Verifications by Rudnik SASA DOOEL

Data entry, validation, storage and database maintenance is carried out by Rudnik SASA DOOEL staff using established procedures. The diamond core drilling data used for the Mineral Resource estimates are stored in a central MX Deposit database located at the Sasa Mine offices. The database has a series of automated validation tools during import and export for error identification. The quality of the assay data contained in the database is monitored by Rudnik SASA DOOEL staff using established QA/QC procedures.

11.2 Verifications by SLR

11.2.1 Site Visits

A site visit to the Sasa Mine was undertaken by SLR on May 13 to 14, 2026 and included the following inspections:

- Extent of exploration work completed to date;
- Review of drill core logging, sampling, sample preparation, analysis and QA/QC procedures;
- Inspection of the core logging, sampling and storage facilities;
- Inspection of selected drill core to confirm the nature of the mineralisation and the geological descriptions;
- Inspection of selected drill core to confirm the down hole depths of geological contacts compared with the drillhole logging data;
- Inspection of geology and mineralisation in underground workings at Svinja Reka (development/production drive, 754 level “Middle Orebody” Block 2 [No.1]); and
- Inspection of underground drilling procedures (drillhole B258-2026 on the 750 level, section 1150).

On site data verification includes reviews of geological logging consistency, collar and downhole survey information, assay database integrity, QA/QC performance and comparison of historical interpretation against recent drilling and underground exposure. These checks are used to identify data limitations and assign appropriate confidence during geological modelling and Mineral Resource classification.

The CPs found the data collection methods used by Rudnik SASA DOOEL to be appropriate.

11.2.2 Review of Drillhole Database

The databases were received by the authors in Microsoft® Excel format. A review of the databases was carried out by the authors and included the following:

- Review of the geological and geographical setting of the Svinja Reka and Golema Reka deposits;
- Review of extent of the exploration work completed to date;
- Inspection of drill core to assess the nature of the mineralisation and to confirm geological descriptions;
- Inspection of geology and mineralisation in underground exposures;
- Review of drilling, logging, sampling and analysis procedures;
- An evaluation of minimum and maximum grade values and sample lengths;

- Assessing for inconsistencies in spelling or coding (typographic or case sensitive errors);
- Ensuring full data entry for each drillhole and that a specific data type (collar, survey, lithology and assay) is not missing;
- Assessing for sample gaps and overlaps;
- A review of assay detection limits;
- Identification of any problematic assay records;
- A spatial on-screen review of the grade and lithology distributions of the drillholes was undertaken to identify any additional data reliability issues; and
- A review of collar locations for underground or surface drilling relative to underground workings or the topographical surface.

Minor validation errors were discovered in terms of duplicate and overlapping intervals; however, the CP does not consider these to be significant. Overall, no significant issues in terms of data collection, data entry or data storage were identified by the authors in a review of the electronic databases.

11.2.3 Limitations

The authors have not undertaken any independent check analysis of samples nor conducted any twin hole drilling to confirm the assays contained in the electronic database. The authors do not consider referee samples necessary given:

- The procedures used by Rudnik SASA DOOEL for sampling, logging, sample preparation, analysis, sample security and data storage are considered to be robust;
- Routine monitoring of QA/QC data demonstrates an acceptable level of accuracy and precision; and
- The Sasa Mine is a mature operation with a significant production history. On-going reconciliation studies (detailed in Section 13) are undertaken by Rudnik SASA DOOEL and show that the Mineral Resource model compares sufficiently with monthly mine production data.

11.2.4 CP Opinion

Overall, no significant issues were identified during the data verification review. The data verification procedures confirmed the integrity of the data contained in the drillhole database and the CP is of the opinion that the database is suitable for use in Mineral Resource estimation. No material sample bias was identified during the review of the drill data and assays. Observation of the drill core during the site visit and inspection and validation of the data collected indicate that the drill data is adequate for the estimation of Mineral Resources. However, it is recommended that the Sasa Mine laboratory works towards accreditation for analysis of exploration drill samples and routinely completes round robin checks (further to the external check analysis completed in 2021 and 2024) with independent (accredited) laboratories. Further, it is recommended that the assay database be enhanced to encompass specific details such as analysis dates, batch identifiers, and pertinent laboratory information. The introduction of a LIMS will aid in achieving this improved standard.

12.0 Mineral Processing and Metallurgical Testing

As an established operating mine, metallurgical testwork is generally focussed with on-going plant optimisation projects using the on-site metallurgical testwork laboratory.

The ore metallurgy has been very consistent since 2008 with lead and zinc recoveries of circa 94% and 86% respectively. Concentrate grades are circa 71% Pb and 50% Zn. Silver recovery to the lead concentrate is 75%.

The main minerals present are sphalerite, galena, pyrite and pyrrhotite and the mineralisation varies from coarse to fine grained depending on the ore source. Silver is present and is generally associated with the galena.

Production data (Table 5-1) shows gradually decreasing lead and zinc head grades, although historical data also indicates that metal recoveries and concentrate grades have not been negatively affected by the lower grades.

Rudnik SASA DOOEL reports that the ore is increasing in hardness as the mine deepens with the Bond Work index (BWi) increasing from circa 13-14 kWh/t to circa 18 kWh/t. Maintaining the current grind size at increasing ore hardness presents the main bottleneck to increasing plant throughput, however, is preferable to using a coarser grind size which may affect recoveries.

12.1 Ore Sorting & Pre-Concentration

There is an on-going project to investigate the potential for ore sorting. If successful, both technically and economically, the rejection of waste material prior to further processing would reduce the throughput bottleneck and potentially give higher flotation recoveries from an upgraded feed. There is also the potential for lower grade ore to be mined and upgraded to allow economical extraction.

Initial testwork results, using both ore sorting and Gekko Inline Pressure Jig (IPJ) technologies, have highlighted that ore sorting of the –60+25 mm fraction within the crushing circuit (tertiary crusher feed) using a single ore sorter can recover circa 98-99% of the metals values to the accepts fraction while rejecting 30% of the ore sorter feed as waste.

As minus 10 mm is not ore sorted, the total rejection rate in terms of ROM ore is circa 15%.

Further but more complex opportunities could be to ore sort both the –60+25 mm and –25+6.3 mm fractions, with Gekko jigging of the –6.3+1.0 mm fraction for maximum rejection of waste material. For the –25+6.3 mm fraction, metals recovery was slightly lower at circa 94-95% for a 30% rejection to waste.

Gekko IPJ results were lower with circa 88-90% metals recovery for a 39% reject rate.

Further testwork, including flotation and mineralogy tests on ore sorter product, is on-going and options studies are planned to be conducted to define an overall optimum strategy for potential pre-concentration. It should be noted that space within the existing process plant for new equipment is limited and may influence the optimal pre-concentration strategy.

12.2 CP Opinion

The ore metallurgy is well understood with consistent performance since 2008, despite declining head grades in recent years. Further optimisation of performance is possible, by balancing throughput with ore hardness, grind size and metals recovery to achieve the best overall economics.

The initial pre-concentration testwork programme using ore sorting indicates encouraging results and, on completion of the full testwork programme, should be reviewed and an options study conducted to determine the best overall strategy for potential pre-concentration.

13.0 Mineral Resource Estimates

13.1 Summary

The Mineral Resource estimate for the Sasa Mine is classified in accordance with the JORC Code (2012). The Mineral Resource models were produced by CAML and reviewed by the CP. The review confirmed the models were completed to an acceptable standard and were in accordance with JORC guidelines.

Mineral Resources are reported for Svinja Reka and Golema Reka deposits. The Kozja Reka occurrence was historically mined; however, insufficient exploration has been undertaken to declare Mineral Resources. The Mineral Resource estimate is based on data derived from diamond drilling completed from surface and underground.

Geological and mineralisation modelling was undertaken using Leapfrog Geo software (Leapfrog), while statistical and geostatistical analysis, three-dimensional (3D) block modelling, Mineral Resource classification, and reporting were completed using Leapfrog Edge. Mineralised domains were constrained using three-dimensional wireframe models developed from lithological interpretations, underground mapping, and a combined lead and zinc (Pb + Zn) grade of 2.0%.

Geological domaining was developed using an integrated interpretation approach incorporating lithology, structural controls, mineralisation continuity, underground mapping observations and drillhole logging. Mineralised domains are constrained by geological interpretation supported by assay continuity and local structural relationships. The interpretation remains iterative and is refined as additional drilling, mapping and mining exposure become available.

Statistical analysis and grade continuity assessments were undertaken to characterise the spatial distribution of the mineralisation and to support the development of appropriate grade estimation parameters. Ordinary Kriging was used as the primary interpolation method to estimate zinc, lead and silver grades within the defined mineralised domains and the adjacent host rock domains of the 3D block model. Nearest Neighbour estimation was used to validate the Ordinary Kriging results. Bulk density was estimated using a combination of Nearest Neighbour interpolation and regression methods based on the combined lead and zinc block grades.

Mineral Resources were classified in accordance with JORC (2012) guidelines, taking into account data quality, geological continuity, and estimation confidence. Mineral Resources were classified as Indicated and Inferred categories and were constrained using Net Smelter Return cut-off values based on suitable technical and economic parameters and assumptions of mining method.

The Mineral Resource estimate for the Svinja Reka and Golema Reka deposits reflects the CPs assessment that the deposit demonstrates reasonable prospects for eventual economic extraction, based on the technical and economic assumptions used.

The consolidated Mineral Resource statement for the Sasa Mine is shown in Table 13-1. The CP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimates.

Table 13-1: Audited Mineral Resource Statement for the Sasa Mine (Effective Date of December 31, 2025)

Deposit	Classification	Tonnes (Mt)	Grades			Contained Metal		
			Pb (%)	Zn (%)	Ag (g/t)	Pb (kt)	Zn (kt)	Ag (koz)
Svinja Reka	Measured	-	-	-	-	-	-	-
	Indicated	9.6	4.2	3.0	32.8	404	290	10,100
	Inferred	2.3	2.9	2.4	35.5	68	56	2,662
Golema Reka	Measured	-	-	-	-	-	-	-
	Indicated	1.8	4.1	1.3	13.8	75	24	810
	Inferred	6.8	3.9	1.2	13.2	263	82	2,880
Total	**Measured**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Indicated	**11.4**	**4.2**	**2.8**	**29.8**	**479**	**314**	**10,910**
	Inferred	**9.1**	**3.6**	**1.5**	**18.9**	**331**	**138**	**5,541**

Notes:

1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. The Competent Person is Jason Ché Osmond, an independent Competent Person as defined by JORC.
3. Mineral Resources are estimated using NSR cut-off values of US$53 per tonne for sub-level caving, US$65 per tonne for cut and fill stoping and $60 per tonne for long-hole stoping.
4. Mineral Resources are estimated using metal prices of US$3,041 per tonne for zinc, US$2,506 per tonne for lead and US$31 per ounce for silver.
5. Mineral Resources are estimated using metallurgical recoveries of 94% for lead, 82% for zinc and 75% for silver.
6. Mineral Resources are estimated using average dry densities ranging from 2.7 to 4.0 t/m^3.
7. A minimum thickness of 1.5 m was used to model the mineralised zones.
8. Mineral Resources are inclusive of Ore Reserves.
9. Contained metal refers to estimated contained metal in the ground not adjusted for metallurgical recovery.
10. Mineral Resources that are not Ore Reserves do not have demonstrated economic viability based on a pre-feasibility study or feasibility study.
11. Totals may not represent the sum of the parts due to rounding.

13.2 Resource Database

The drillhole database for the Sasa Mine, including Svinja Reka and Golema Reka deposits, was provided by Rudnik SASA DOOEL and reviewed by SLR.

All drillhole information, including lithological and mineralisation logging, collar coordinates, downhole survey data, bulk density measurements, core recovery information, and assays, is stored in the MX Deposit database at the Sasa Mine. The database was interfaced directly with Leapfrog Geo through an ODBC (Open Database Connectivity) connection, enabling the import of validated datasets via structured queries of the master tables.

The database for Svinja Reka consists of 209,848 m of diamond drilling completed in 3,035 drillholes. The database includes a total of 72,703 assay records and 2,298 bulk density determinations. Most of the drillholes are collared from underground drill stations located in the footwall at approximately 50 m intervals and are generally oriented sub-perpendicular to the strike of the mineralisation. Owing to the shallow dip of the mineralised horizons, drill intersections are typically oblique rather than perpendicular to the mineralisation.

Drill spacing within active mining areas generally ranges from 10 to 30 m along strike and down-dip, increasing to up to 90 m in areas between active mining zones. As the majority of drillholes are collared from footwall positions, the footwall (Podina) mineralised lenses are generally defined at a closer spacing than the hanging wall (Krovina) lenses. However, in areas where drillholes have been collared in developed or mined-out footwall workings, local coverage of the footwall domain is less.

The database for Golema Reka consists of 21,510 m of diamond drilling completed in 241 drillholes. The database contains a total of 2,562 assay records.

The majority of drillholes are collared from underground drill stations located in the footwall, typically spaced at approximately 50 m intervals. Similar to Svinja Reka, due to the shallow dip of the orebody underground drillholes are commonly oblique rather than perpendicular to the mineralisation.

Drill spacing within the mineralised zones typically ranges from approximately 10 to 30 m along strike and down-dip, although spacing locally increases to up to 200 m in less densely drilled areas.

Core recovery is consistently high for both deposits, averaging between 95% and 98%, and is considered appropriate for Mineral Resource estimation. No assay data were excluded from the estimation process based on poor core recovery.

Any absent values in the database were replaced with nominal low grades (0.01% Pb and 0.01% Zn).

A summary of the drillhole database used for the Mineral Resource estimate is shown in Table 13-2.

Table 13-2: Summary of Mineral Resource Database

Deposit	Database Cut-Off Date	Number of Drillholes	Length (m)	Number of Assays
Svinja Reka	December 31, 2025	3,035	209,848	72,703
Golema Reka	December 31, 2025	241	21,510	2,562

13.3 Svinja Reka

13.3.1 Geological Interpretation

The Svinja Reka deposit comprises well-defined, partially exploited lenses of lead-zinc-silver mineralisation that dip at approximately 35° to the southeast. Individual lenses typically range in true thickness from approximately 2 to 30 m and occur as sub-parallel bodies, commonly consisting of two or three discrete lenses separated by interburden zones of 1 to 10 m in thickness. The mineralised lenses exhibit pinch and swell features along strike and down-dip, with an apparent dominant plunge to the south. The principal lenses of economic interest are identified as Podina (footwall), Sredisno (central), and Krovina (hanging wall), all of which are characterised by relatively sharp geological contacts.

Mineralisation is interpreted to remain open down-dip towards the southeast; however, it may locally terminate or be truncated against the southeast-bounding fault that separates the Svinja Reka deposit from the Kozja Reka area.

Three-dimensional wireframe models representing the mineralised domains were developed by CAML, using Leapfrog Geo geological modelling software. These domains were constructed as vein-type geometries based on integrated interpretation of lithological data, underground mapping, a Pb + Zn cut-off grade of 2.0% and a minimum orebody thickness of 1.5 m. Where necessary, additional polylines were incorporated along the reference, hanging wall, and footwall surfaces to improve the geological continuity of the wireframes.

A categorical variable was derived from the Pb + Zn assay values, which was subsequently used to define interval selection criteria and assign domain codes to the drillhole dataset. A total of 14 discrete domains were interpreted and modelled, with an additional domain created representing the surrounding waste. These zones are listed in Table 13-3 along with their percentage of the total mineralised tonnes (after depletion for mining). Approximately 92% of the final Mineral Resources are contained in three domains: KRO_0, POD_0 and SRE_3.

Table 13-3: Modelled Domains at Svinja Reka

Type	Domain Code	Description	Percentage of Total Mineralised Tonnes (%)
Mineralised	POD_0	Podina Main Zone - Footwall	40
	POD_1	Podina Footwall 1	4
	POD_2	Podina Footwall 2	1
	POD_3	Podina Footwall 3	1
	SRE_2	Sredisno 2 Middle	0
	SRE_3	Sredisno 3	39
	KRO_0	Krovina Main Zone Hangingwall	13
	KRO_1	Krovina Hanging wall 1	0
	KRO_2	Krovina Hanging wall 2	1
	KRO_3	Krovina Hanging wall 3	1
	KRO_4	Krovina Hanging wall 4	0
Internal Waste	POD_0_W	Podina Internal Waste	-
	SRE_3_W	Sredisno Internal Waste	-
	KRO_0_W	Krovina Internal Waste	-
External Waste	EXT_W	Waste halo (Schist)	-

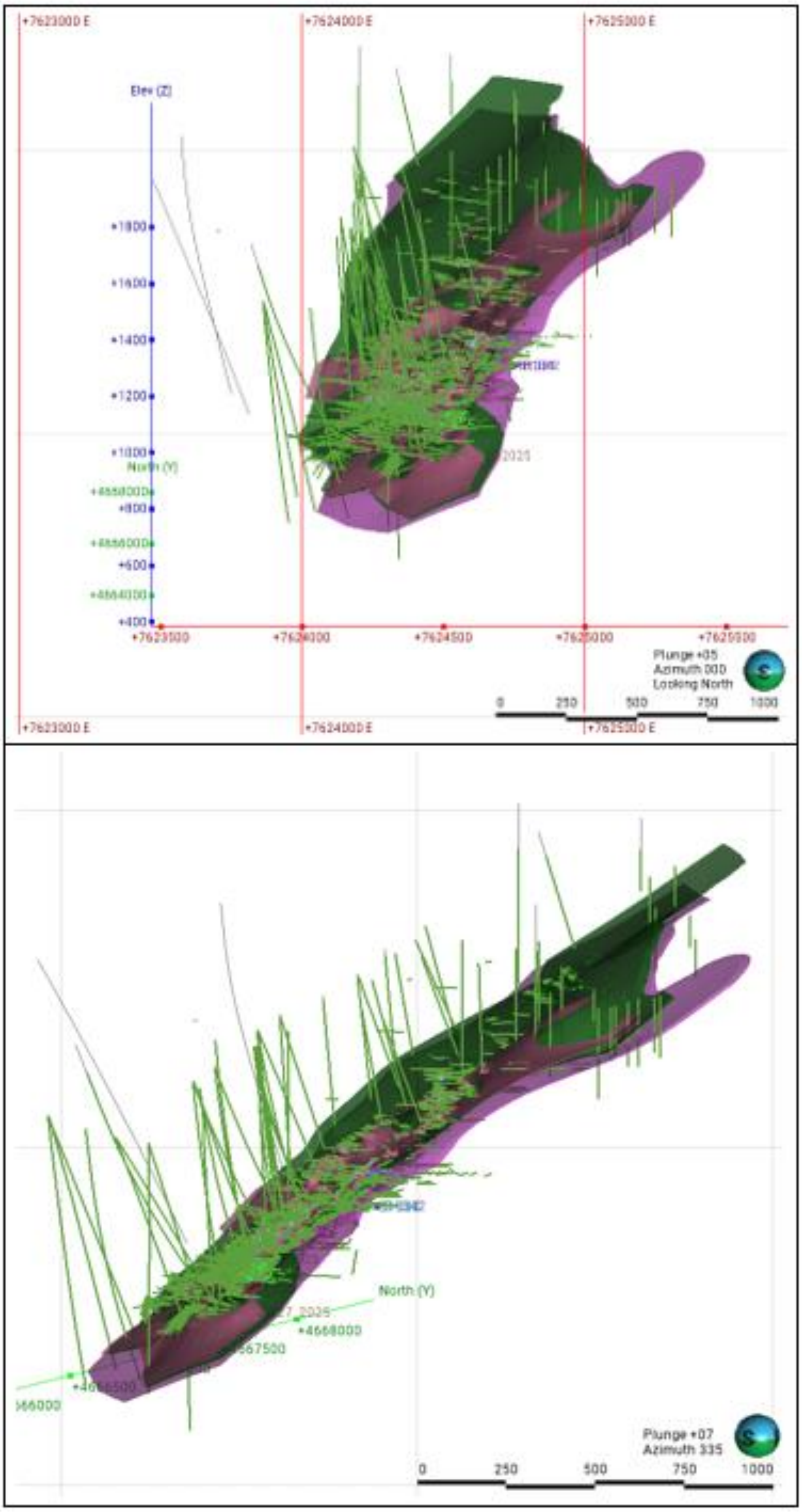
Internal waste zones (defined where Pb + Zn is <2.0%) are commonly present within the Podina, Sredisno, and Krovina mineralised lenses. These internal waste zones were interpreted and modelled as separate domains where sufficient geological continuity was demonstrated through multiple drillhole intersections or supported by underground mapping. However, narrow, isolated low-grade intervals lacking demonstrated continuity were not explicitly modelled.

The domain wireframes were constructed using an integrated approach that incorporated underground level mapping and drillhole data. These separate wireframes were subsequently combined into a single geological model within the surrounding country rock, classified as schist. Gneiss units were not explicitly modelled.

Isolated mineralised intersections occurring outside the defined wireframe domains were identified but were considered discontinuous and not geologically significant. These intersections were excluded from the domain interpretation.

The resulting wireframe models (Figure 13-1) were used to constrain and code the drillhole assay intervals for subsequent compositing, statistical analysis, and Mineral Resource estimation.

Figure 13-1: Isometric Views of Svinja Reka Domains and Drillholes, Looking North (top) and Northwest (bottom) (Scale in Meters)



Source: CAML (June 2026)

13.3.2 Resource Assays

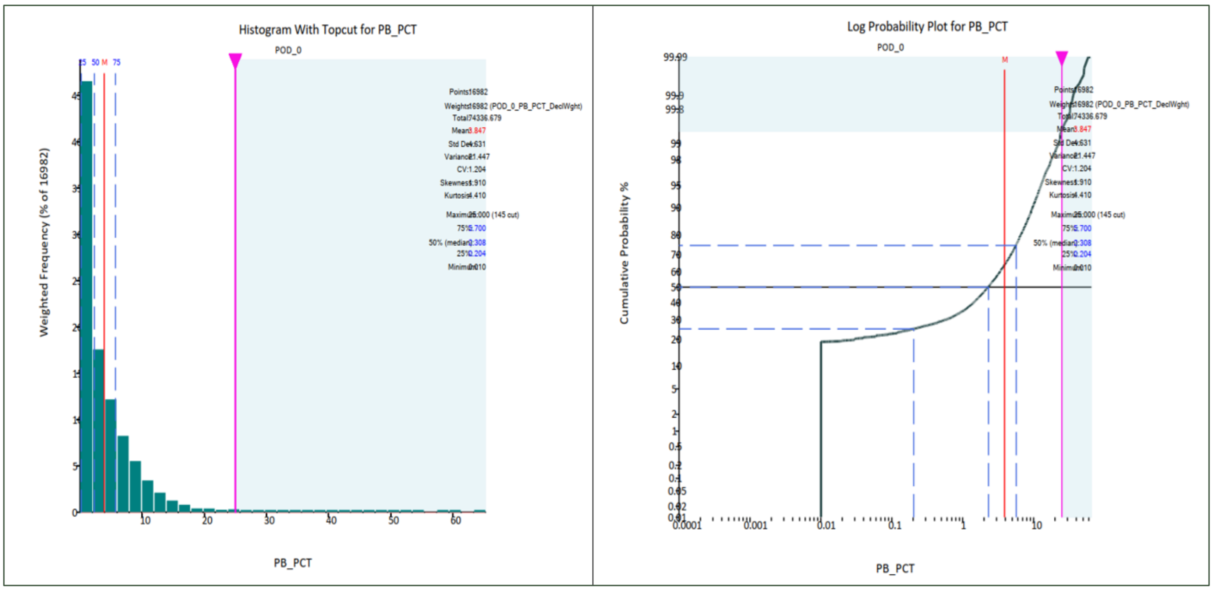
13.3.2.1 Compositing

Sample compositing was undertaken prior to grade capping, using a composite length of 1.0 m, which reflects the dominant sample interval. Residual intervals of less than 0.5 m were proportionally distributed across adjacent composites, resulting in slight variations in the final composite lengths. Consequently, the effective composite intervals can locally exceed the nominal 1.0 m composite length.

13.3.2.2 Treatment of High-Grade Assays

Grade estimation may be influenced by the presence of high-grade assays, which have the potential to introduce bias into block grade estimates. Accordingly, an assessment of high-grade outliers was undertaken using the Snowden Supervisor Global Top-Cut Analysis tool. This analysis included evaluation of grade distributions through histograms, log-probability plots, mean and variance analyses, and cumulative metal plots for composited data within each domain. Examples of the top-cut analysis plots are shown in Figure 13-2.

Figure 13-2: Top-Cut Analysis for Pb in POD_0 Domain



Source: CAML (June 2026)

Grade capping thresholds were determined based on identified breaks in population distributions from the log-probability plots, with additional consideration given to reducing the coefficient of variation to below 1.5 and limiting metal loss to less than 5%. The resulting top-cut values were applied to the composite data as summarised in Table 13-4 for the major domains at Svinja Reka.

No high-grade distance restrictions were applied during grade estimation, except for the POD_0 domain, where an additional constraint was applied for grade estimation of Pb on the third search pass. Composite values exceeding a threshold of 5.0% Pb were subject to a distance restriction, whereby their influence was limited to 20% of the search radius.

Table 13-4: Svinja Reka Major Domains Grade Capping Levels

Domain	Pb% Cap	Zn% Cap	Ag g/t Cap
POD_0	25	25	175
POD_0 Internal Waste	3	1	50
POD_1	25	14	210
POD_2	25	15	100
SRE_2	-	15	-
SRE_3	20	20	180
SRE_3 Internal Waste	2	1	-
KRO_0	18	20	260
KRO_2	15	20	80
KRO_0 Internal Waste	1	1	-
External Waste	8	8	150

13.3.2.3 Final Statistics

Statistics of the capped composites for the major domains at Svinja Reka are shown in Table 13-5.

Table 13-5: Statistics of Capped Composites for Major Domains at Svinja Reka

Variable	Domain	Number of Composites	Minimum	Maximum	Mean	SD	CV
Zn (%)	KRO_0	6,432	0.01	20	3.19	3.61	1.13
	POD_0	16,983	0.01	25	3.67	4.71	1.28
	SRE_3	13,567	0.01	20	3.07	4.03	1.31
Pb (%)	KRO_0	6,432	0.01	18	4.01	3.61	0.90
	POD_0	16,982	0.01	25	4.35	4.86	1.12
	SRE_3	13,567	0.01	20	3.67	4.18	1.14
Ag (g/t)	KRO_0	2,076	0.42	260	28.30	33.65	1.19
	POD_0	3,959	0.26	175	29.82	33.53	1.12
	SRE_3	4,608	0.08	180	27.57	30.65	1.11

13.3.3 Variography

Spatial continuity of the Zn, Pb, and Ag grades was assessed using geostatistical methods applied to composites in Leapfrog Edge software. The analysis was undertaken to characterise the grade continuity in three-dimensional space and to define the anisotropy of the mineralisation and used variogram fans and evaluation of the resulting anisotropy ellipsoids within the interpreted geological domains.

Leapfrog Edge was used to model the variograms using normal score-transformation of the composites. Downhole variograms were generated to quantify the nugget effect, while directional variograms were modelled along the principal axes of continuity to define the structures. The final variogram models for the major domains at Svinja Reka are shown in Table 13-6. These analyses provided input for the selection of search parameters and estimation strategies used for grade estimation.

Table 13-6: Variogram Model Parameters for the Main Domains at Svinja Reka

Domain	Element	Rotation[1]	Nugget	Variance[2]	Structure 1 Type	Structure 2 Type	Structure 1 Range (m)	Structure 2 Range (m)
POD_0	Zn	32, 239, 5	0.23	0.39, 0.38	Spherical	Spherical	4, 13, 7	40, 27, 11
POD_0	Pb	32, 239, 5	0.25	0.37, 0.38	Spherical	Spherical	4, 10, 2	28, 36, 8
POD_0	Ag	32, 239, 5	0.11	0.59, 0.30	Spherical	Spherical	9, 11, 7	40, 50, 9
SRE_3	Zn	34, 244, 45	0.32	0.39, 0.30	Spherical	Spherical	5, 5, 1	24, 9, 2
SRE_3	Pb	34, 244, 45	0.34	0.39, 0.27	Spherical	Spherical	6, 26, 5	51, 31, 7
SRE_3	Ag	34, 244, 45	0.34	0.39, 0.28	Spherical	Spherical	6, 26, 5	51, 31, 7
KRO_0	Zn	38, 245, 161	0.42	0.26, 032	Spherical	Spherical	22, 10, 3	36, 19, 6
KRO_0	Pb	38, 245, 161	0.32	0.39, 0.30	Spherical	Spherical	8, 15, 5	45, 27, 7
KRO_0	Ag	38, 245, 161	0.39	0.01, 0.60	Spherical	Spherical	37, 12, 2	47, 33, 5

Notes:

1. Leapfrog rotation (Dip, Dip Azimuth, Pitch)
2. Variance for structures 1 and 2 (C1, C2)
3. Ranges in Major, Semi-Major and Minor directions

13.3.4 Search Strategy and Grade Interpolation Parameters

Grade estimation was carried out in Leapfrog Edge software using Ordinary Kriging. Boundaries between domains were considered as hard boundaries. Grade estimation was undertaken in parent cells with all sub-cells assigned the grade of the parent. Block discretisation was set to 5 x 5 x 2. The orientations of the search radii were variable based on the structural trend of the geological interpretation by using dynamic anisotropy.

The reference surfaces derived from the geological modelling for each domain were utilised to define the local structural trends of the mineralisation. These structural trends were incorporated into the estimation process to orientate and refine the search ellipsoids, thereby improving the alignment of the interpolation with the geometry of the mineralised domains.

Selection of the search radii and the orientation of search ellipsoids was guided by the results of the grade continuity analysis and the interpreted geometry of the mineralisation. Due to the variable orientation of the mineralised lenses, a locally varying anisotropic search approach was adopted, whereby each block was assigned a locally optimised search orientation that conforms to the prevailing structural trend.

Grade estimation was performed using multi-passes, with up to three search passes of progressively increasing search radii. Blocks that did not meet the criteria for estimation in earlier passes were considered in subsequent passes.

The first pass used a relatively restricted search ellipsoid, designed to reflect the higher confidence areas defined by close-spaced infill drilling, and based on modelled variogram ranges. The second pass used an expanded search ellipsoid, with dimensions increased by up to 100% relative to the first pass, to estimate grades in areas supported by underground exploration drilling and in zones between established mining levels. Where required, a final third pass was applied to estimate any remaining blocks, typically located at the margins of the model and informed by more widely spaced surface drilling.

A summary of the estimation and search parameters is shown in Table 13-7.

Table 13-7: Search and Estimation Parameters for the Main Domains at Svinja Reka

General		Ellipsoid Ranges (m)			Ellipsoid Orientation	Number of Samples		Max Samples per Drillhole
Estimator Name	Element	Max	Int	Min		Min	Max	
KRO_0 Search 1	Ag	28.32	34.7	6.62	Variable	8	26	4
KRO_0 Search 2	Ag	200	150	18	Variable	6	18	4
KRO_0 Search 3	Ag	300	250	24	Variable	4	10	3
SRE_3 Search 1	Ag	57.74	39.39	7.42	Variable	8	26	4
SRE_3 Search 2	Ag	200	150	18	Variable	6	18	4
SRE_3 Search 3	Ag	300	250	24	Variable	4	10	3
POD_0 Search 1	Ag	42.33	22.39	7.63	Variable	8	26	4
POD_0 Search 2	Ag	50	70	14	Variable	6	18	4
POD_0 Search 3	Ag	160	280	56	Variable	4	10	3
KRO_0 Search 1	Pb	20.2	18.56	7.81	Variable	8	20	4
KRO_0 Search 2	Pb	200	150	18	Variable	6	18	4
KRO_0 Search 3	Pb	350	350	50	Variable	2	10	2
SRE_3 Search 1	Pb	48.79	30.74	9.59	Variable	8	20	4
SRE_3 Search 2	Pb	200	150	18	Variable	6	18	4
SRE_3 Search 3	Pb	300	300	30	Variable	2	10	2
POD_0 Search 1	Pb	48.79	30.74	9.59	Variable	8	20	4
POD_0 Search 2	Pb	80	80	30	Variable	6	18	4
POD_0 Search 3	Pb	450	450	200	Variable	2	10	2
KRO_0 Search 1	Zn	48.48	47.41	6.54	Variable	8	20	4
KRO_0 Search 2	Zn	200	150	18	Variable	6	18	4
KRO_0 Search 3	Zn	350	350	50	Variable	2	10	2
SRE_3 Search 1	Zn	45.7	27.96	10.63	Variable	8	20	4
SRE_3 Search 2	Zn	200	150	18	Variable	6	18	4
SRE_3 Search 3	Zn	300	300	30	Variable	2	10	2
POD_0 Search 1	Zn	45.7	27.96	10.63	Variable	8	20	4
POD_0 Search 2	Zn	100	60	44	Variable	6	18	4
POD_0 Search 3	Zn	450	450	200	Variable	2	10	2

13.3.5 Density

Density measurements are conducted on core samples of approximately 10 cm length using the Archimedes method.

Density values for the mineralisation and internal waste domains were assigned in the Mineral Resource model based on a two-pass nearest neighbour interpolation approach. In the first pass, a nearest neighbour estimation was applied using a search ellipsoid of 50 x 50 x 12 m. For any unestimated blocks, density values were assigned based on empirical relationships between the combined Pb and Zn grades and the measured densities. Regression equations were developed from a total of 2,290 core measurements and are shown in Table 13-8.

Within the schist domain, a hybrid approach was adopted. All blocks with an estimated Pb + Zn grade of less than 2.0% were assigned a density of 2.75 t/m³. Blocks with grades equal to or greater than 2.0% were assigned density values calculated using the regression equation.

Table 13-8: Density Regression Formula for Svinja Reka

<table>
<tr><th colspan="2">Domain</th><th>Regression or Assigned Value</th></tr>
<tr><td colspan="2">Schist</td><td>Pb + Zn >= 2.0%: Density = Pb + Zn * 0.027 + 2.975
Pb + Zn < 2.0%: Density = 2.75 t/m³</td></tr>
<tr><td rowspan="3">Internal Waste</td><td>KRO_0_W</td><td>Density = Pb + Zn * 0.0299 + 3.0273</td></tr>
<tr><td>POD_0_W</td><td>Density = Pb + Zn * 0.025 + 3.02</td></tr>
<tr><td>SRE_3_W</td><td>Density = Pb + Zn * 1.044 + 2.683</td></tr>
<tr><td colspan="2">All mineralised domains</td><td>Density = Pb + Zn * 0.017 + 3.181</td></tr>
</table>

13.3.6 Block Models

Block model setup and interpolation was completed in Seequent's Leapfrog Edge software. The block model parameters for Svinja Reka are shown in Table 13-9. The model was not rotated. The CP considers the block model sizes appropriate for the deposit geometry and proposed mining methods.

Table 13-9: Svinja Reka Block Model Extents and Dimensions

Type	X Direction	Y Direction	Z Direction
Base Point (m)	7,623,850	4,664,500	370
Boundary Size (m)	1,081.5	1,904	546
Number of Blocks	479	309	78
Parent Block Size (m)	3.5	14	7
Min. Sub-block Size (m)	0.21875	0.875	0.4375

13.3.7 Resource Classification

Definitions for resource categories used in this report are consistent with those defined by JORC (2012).

Classification of the Svinja Reka Mineral Resources considered the quality and reliability of the drillhole data, the confidence in the geological interpretation, the spatial distribution of data, variogram continuity, and the robustness of the grade estimation. Classification considers geological continuity, drilling density, data quality, QA/QC performance and estimation confidence. Areas supported by closer spaced drilling, consistent geological interpretation and stronger continuity evidence are classified as Indicated Resources. Areas with more limited drilling support, increased geological complexity or lower confidence continuity remain classified as Inferred Resources pending further drilling and geological confirmation.

Based on these factors, the CP considers that the quality and spatial distribution of the available data, together with the demonstrated geological continuity of the mineralisation and the integrity of the block model estimates, are sufficient to support the classification of portions of the deposit as Indicated and Inferred Mineral Resources.

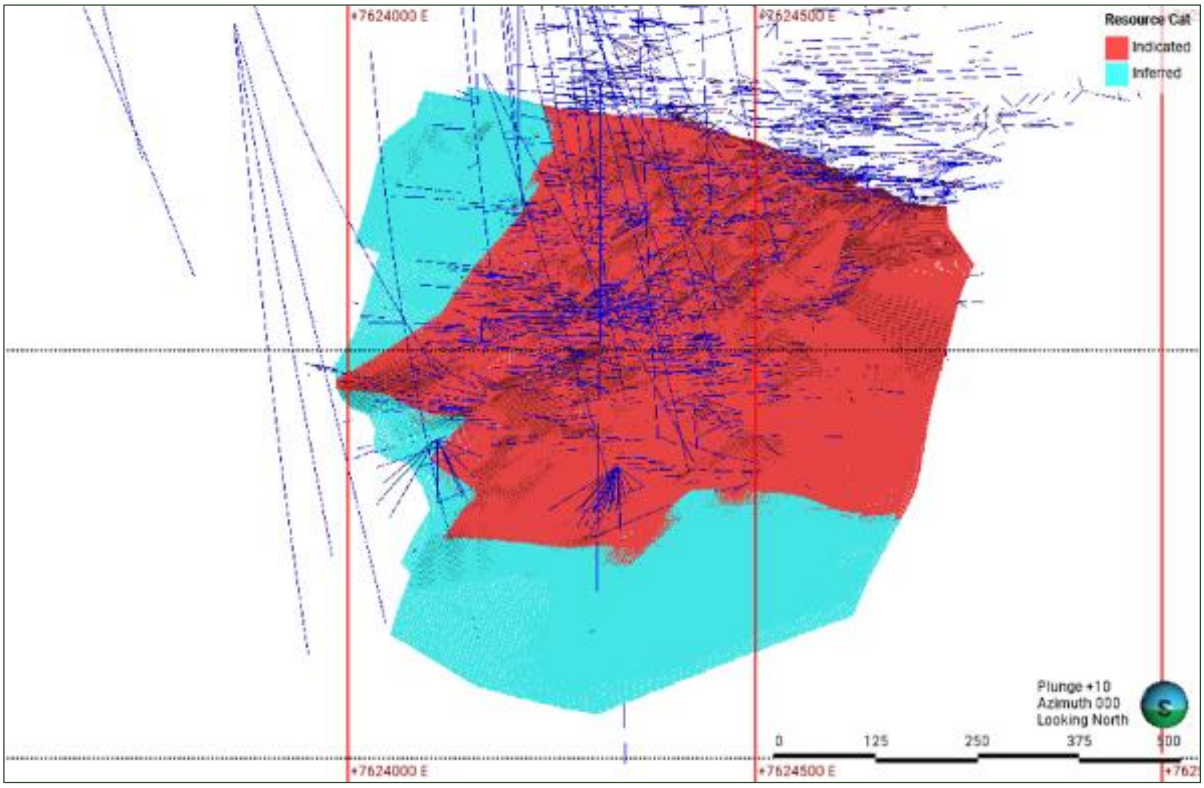
Owing to the spatial complexity and local variability of the mineralisation, both along strike and between the 7.0 m sublevels, the short-range grade and structural variability is considered high and does not support classification of Measured Mineral Resources based on the current drillhole spacing.

Indicated Mineral Resources were assigned based on a minimum of four drillholes within a maximum distance of approximately 50 m, while Inferred Mineral Resources were defined in areas informed by at least two drillholes within a maximum distance of approximately 150 m.

Classification was applied in the Mineral Resource model using distance-based wireframe shells generated in Leapfrog Geo with the distance function at thresholds of 50 m and 150 m from the drillhole composites. These wireframes were used to assign Indicated and Inferred Mineral Resource categories in the block model, respectively.

The Mineral Resource classification for Svinja Reka is shown in Figure 13-3.

Figure 13-3: Isometric View of Svinja Reka Resource Classification



Source: CAML (June 2026)

13.3.8 Block Model Validation

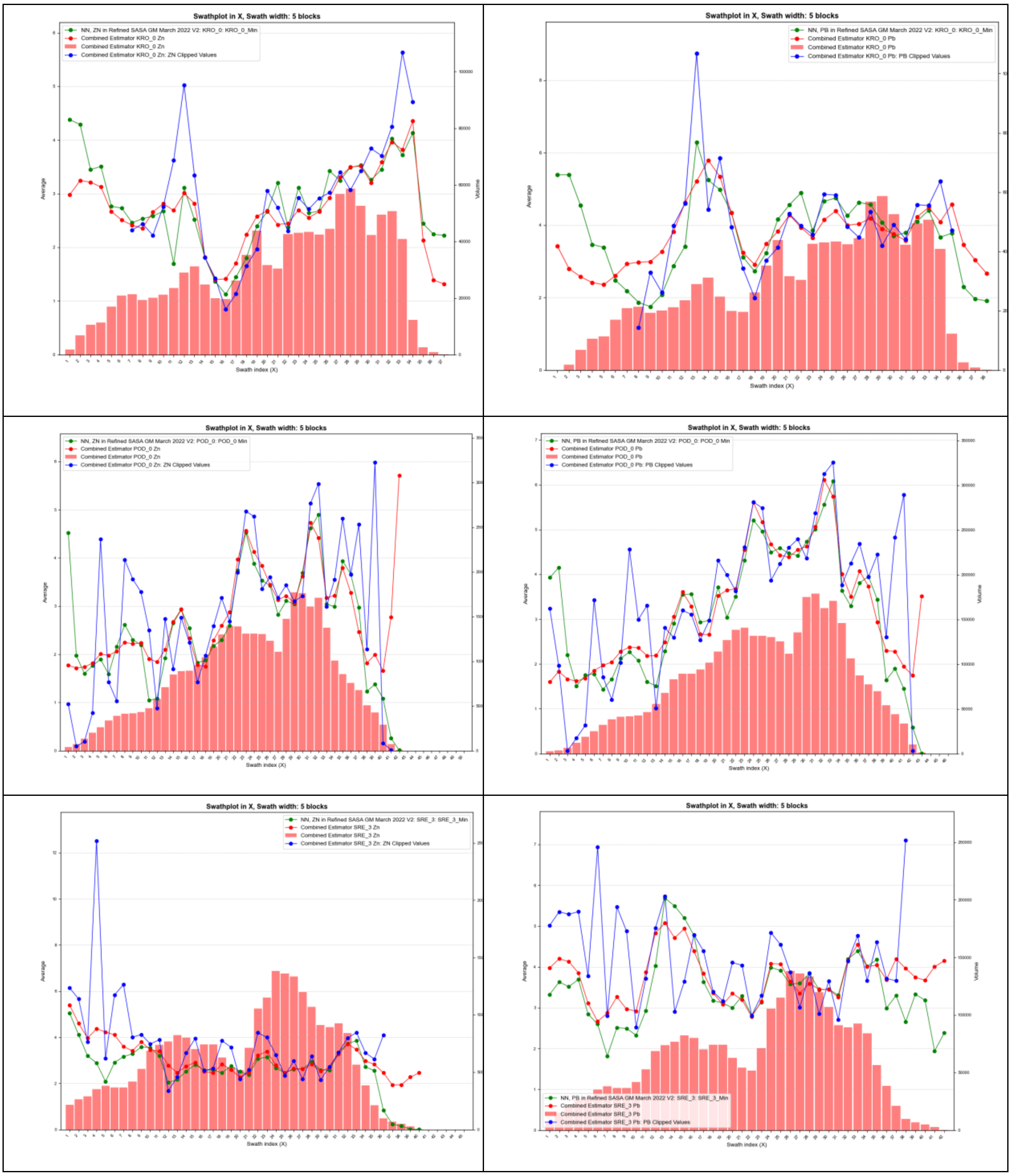
A visual and statistical validation of the block model was undertaken by the CP and included the following checks:

- Visual inspection comparing composite and block grades.
- Comparison of swath plots for ordinary kriging and nearest neighbour methods.
- A review of the reconciliation of the block model with actual production.

13.3.8.1 Swath Plots

Example swath plots for Svinja Reka are shown in Figure 13-4. The CP considers the swath analysis shows the estimated block grades to have a generally acceptable correlation with the spatial trends observed in the input composite grades.

Figure 13-4: Svinja Reka Swath Plots for KRO_0 (top), POD_0 (middle) and SRE_3 (bottom) in X Direction for Zn % (left) and Pb % (right)



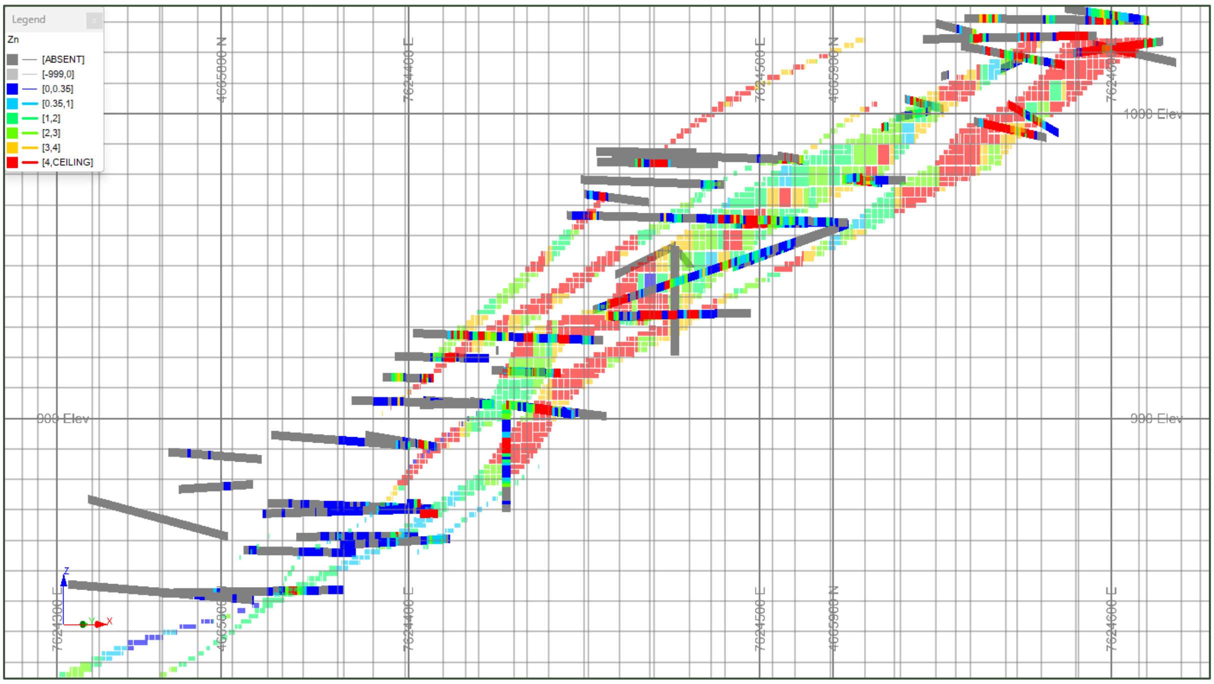
Source: CAML (June 2026)

13.3.8.2 Visual Comparison of Composites and Block Grades

The estimated grades for Zn, Pb and Ag in the block model were validated through visual inspection, using a series of cross-sections and plan views compared against the drillhole composite grades as shown in Figure 13-5 for Zn (%).

A reasonable correlation was observed between the input composites and the estimated block grades and reflected the spatial trends of the mineralisation with no material discrepancies or inconsistencies identified in the CPs review.

Figure 13-5: Example Vertical Section of Svinja Reka Block Grades and Composites for Zn (%)



Source: SLR (June 2026)

13.3.8.3 Reconciliation

Reconciliation comparing planned production from the resource model against actual performance of the processing plant is undertaken monthly. The results of the reconciliation are accumulated annually and are shown in Table 13-10.

On an annual basis, the reconciliations show a generally good agreement between the resource model and actual production, and report less than 5% difference in contained Pb and Zn metal. Overall, the results of the reconciliations are within acceptable tolerances and support the methodology used for resource modelling.

Table 13-10: Svinja Reka Reconciliation for 2022 to 2025

Time Period	Comparison of Mill Figures to Planned Production (Pb)			Comparison of Mill Figures to Planned Production (Zn)		
	Tonnes	Grade	Contained Metal	Tonnes	Grade	Contained Metal
2022	101%	95%	96%	101%	100%	101%
2023	101%	97%	97%	101%	104%	105%
2024	95%	97%	93%	95%	97%	93%
2025	99%	98%	97%	99%	90%	83%
Total	**99%**	**97%**	**96%**	**99%**	**98%**	**96%**

13.3.9 Cut-Off Values for Reporting Mineral Resources

Due to the polymetallic nature of the mineralisation, a NSR calculation was used to determine the value of each individual block in the Mineral Resource model. The NSR was calculated based on metal prices, payable metals considering lead, zinc and silver grades, metallurgical recoveries, and realisation costs.

Metal prices used in the calculation of NSR for Mineral Resources were US$3,041 per tonne for zinc, US$2,506 per tonne for lead, and US$31 per ounce for silver.

Metallurgical recoveries used in the calculation of NSR were 94% for lead, 82% for zinc and 75% for silver. The zinc recovery is conservative and less than the 86% achieved by the processing plant. Royalties of 2% were used and are in the process of being updated to 4% based on the updated Concession Fee in 2025.

The NSR calculation also considered the planned underground mining methods based on orebody geometry for the following mining levels:

- Sub-level caving between levels 810 and 962 mRL, and above 990 mRL.
- Cut-and-fill: between levels 750 and 760 mRL, and between 800 and 810 mRL.
- Long-hole stoping: below level 750 mRL, between levels 760 and 800 mRL, and between levels 962 and 990 mRL.

NSR cut-off values (COVs) of US$53 per tonne for SLC, US$65 per tonne for CF and US$60 per tonne for LHS were used to estimate the Mineral Resources.

13.3.10 Mining Depletion

Underground surveys of the mine development are undertaken using total station surveying. Surveying procedures include measurement of lateral advance, acquisition of development extents at approximately 1 m above the sill (floor) to generate two-dimensional (2D) as-built plans, and estimation of three-dimensional (3D) drift volumes. The 3D volumes are derived from surveyed centrelines of lateral advances combined with a standardised drift profile.

Due to the nature of the sublevel caving method, physical access to caved stopes is not possible, and consequently, direct surveys of these voids cannot be completed. As a result, no robust 3D representation of the actual caved volumes is possible. To approximate these voids, volumetric wireframes are constructed between level development to provide an estimate of the caved material removed by mining.

For cut-and-fill mining areas, once sufficient advance has occurred, 3D volumetric models are generated on a monthly basis. Although the derived 3D drift volumes do not precisely represent locally mined geometries, they are considered a reasonable approximation when evaluated over larger volumes and are used in the reconciliation process.

Wireframes representing the mined-out volumes are constructed and used to code mined-out areas in the block model.

13.3.11 Mineral Resource Reporting

A summary of the Mineral Resource estimate for Svinja Reka is shown in Table 13-11.

The stated Mineral Resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, to the best knowledge of the authors. There are no known mining, metallurgical, infrastructure, or other factors that materially affect this Mineral Resource estimate, at this time.

Table 13-11: Audited Svinja Reka Mineral Resource Statement (Effective Date of December 31, 2025)

Deposit	Classification	Tonnes (Mt)	Grades			Contained Metal		
			Pb (%)	Zn (%)	Ag (g/t)	Pb (kt)	Zn (kt)	Ag (koz)
Svinja Reka	Measured	-	-	-	-	-	-	-
	Indicated	9.6	4.2	3.0	32.8	404	290	10,100
	Inferred	2.3	2.9	2.4	35.5	68	56	2,662

Notes:

1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. The Competent Person is Jason Ché Osmond, an independent Competent Person as defined by JORC.
3. Mineral Resources are estimated using NSR cut-off values of US$53 per tonne for sub-level caving, US$65 per tonne for cut-and-fill stoping and $60 per tonne for long-hole stoping.
4. Mineral Resources are estimated using metal prices of US$3,041 per tonne for zinc, US$2,506 per tonne for lead and US$31 per ounce for silver.
5. Mineral Resources are estimated using metallurgical recoveries of 94% for lead, 82% for zinc and 75% for silver.
6. Mineral Resources are estimated using dry densities ranging from 2.7 to 4.0 t/m^3.
7. A minimum thickness of 1.5m was used to model the mineralised zones.
8. Contained metal refers to estimated contained metal in the ground not adjusted for metallurgical recovery.
9. Mineral Resources are inclusive of Ore Reserves.
10. Mineral Resources that are not Ore Reserves do not have demonstrated economic viability based on a pre-feasibility study or feasibility study.
11. Totals may not represent the sum of the parts due to rounding.

13.4 Golema Reka

13.4.1 Geological Interpretation

Lead-zinc-silver mineralisation at the Golema Reka deposit is hosted within granitoid gneiss, striking approximately 135° and dipping moderately (approximately 30 to 45°) to the southeast. Mineralisation occurs as stacked, massive to semi-massive sulphide lenses with thicknesses ranging from approximately 0.5 to 10 m along strike and down dip. A total of seven mineralised lenses have been identified and have been partly exploited by historical mining. Individual lenses display pinch-and-swell geometries both along strike and down dip, with a dominant plunge to the south. The lenses are separated by intervals of weakly mineralised to barren gneiss.

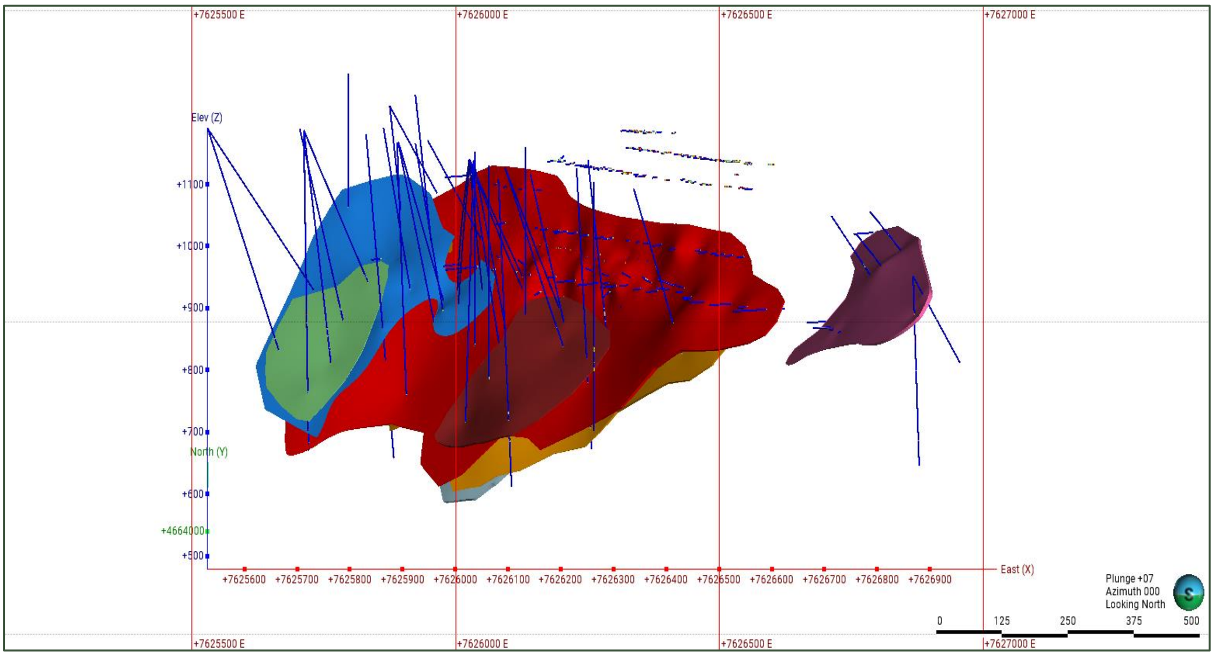
Mineralisation remains open down dip towards the southeast and along strike towards the northwest, although it may locally terminate against a northwestern fault that separates the Golema Reka deposit from the Kozja Reka occurrence.

Notwithstanding, geological continuity at Golema Reka remains less certain than at Svinja Reka due to lower drilling density and increased geological complexity. Further drilling, mapping and structural interpretation are required to improve confidence in continuity, geometry and classification. This uncertainty is reflected in the current classification balance and should remain a focus of future geological work.

Three-dimensional wireframe models of the mineralised domains were developed by CAML geologists, using Leapfrog Geo software. The domains were constructed as vein-type solids based on lithological constraints, a nominal combined Pb + Zn grade cut-off of 2.0% and a minimum orebody thickness of 1.5 m. Where necessary, additional polylines were incorporated to refine the geometry of the hanging wall, footwall, and reference surfaces. In addition to the seven mineralised domains (ORE 1 to 7) a surrounding waste rock domain was also defined. Internal low-grade zones (<2.0% Pb + Zn) were not explicitly modelled due to their narrow and discontinuous nature and are not considered amenable to selective mining.

The resulting wireframe models as shown in Figure 13-6 were used to code and select assay intervals from the drillhole database for subsequent compositing and statistical analysis.

Figure 13-6: Isometric View of Golema Reka Domains and Drillholes Looking North (Scale in Meters)



Source: CAML (June 2026)

13.4.2 Resource Assays

13.4.2.1 Compositing

Sample compositing was undertaken prior to grade capping, using a composite length of 1.0 m, which reflects the dominant sample interval. Residual intervals of less than 0.5 m were proportionally distributed across adjacent composites, resulting in slight variations in the final composite lengths. Consequently, the effective composite intervals can locally exceed the nominal 1.0 m composite length.

13.4.2.2 Treatment of High-Grade Assays

No capping was applied to Pb or Ag grades. An outlier restriction was placed on Zn by clamping values greater than 1.5% Zn to 15% of the search distance during grade estimation.

13.4.2.3 Final Statistics

Statistics of the capped, composites for the major domains at Golema Reka are show in Table 13-12. Statistics for domain ORE 4 have been excluded. This zone is not included in the Mineral Resource estimate due to a limited number of samples and the wide drillhole spacing in this area.

Table 13-12: Statistics of Capped Composites for Major Domains at Golema Reka

Variable	Domain	Number of Composites	Minimum	Maximum	Mean	SD	CV
Zn (%)	ORE 1	1,182	0.01	27.03	1.95	2.53	1.30
	ORE 2	501	0.01	27.13	1.35	2.21	1.63
	ORE 3	97	0.01	10.00	1.16	1.59	1.37
	ORE 5	58	0.01	3.85	0.64	0.78	1.22
	ORE6	35	0.01	3.28	1.20	0.93	0.78
	ORE 7	11	0.01	3.03	0.76	1.12	1.48
Pb (%)	ORE 1	1,182	0.01	48.50	3.66	4.98	1.36
	ORE 2	501	0.01	27.41	2.90	4.00	1.38
	ORE 3	97	0.01	16.75	2.66	3.39	1.27
	ORE 5	58	0.01	38.48	5.22	6.47	1.24
	ORE6	35	0.01	7.71	1.58	1.64	1.03
	ORE 7	11	0.01	16.25	4.53	5.76	1.27
Ag (g/t)	ORE 1	1,182	0.01	425.37	1.42	15.13	10.63
	ORE 2	501	0.01	557.67	10.51	45.85	4.36
	ORE 3	97	0.01	97.11	6.57	17.52	2.67
	ORE 5	58	0.01	404.00	12.69	60.64	4.78
	ORE6	35	0.01	18.00	2.43	4.72	1.95
	ORE 7	11	0.01	18.00	3.15	5.98	1.90

13.4.3 Variography

Spatial continuity of the Zn, Pb, and Ag grades was assessed using geostatistical methods applied to composites in Leapfrog Edge software. The analysis was undertaken to characterise the grade continuity in three-dimensional space and to define the anisotropy of the mineralisation and used variogram fans and evaluation of the resulting anisotropy ellipsoids within the interpreted geological domains.

Leapfrog Edge was used to model the variograms using normal score-transformation of the composites. Downhole variograms were generated to quantify the nugget effect, while directional variograms were modelled along the principal axes of continuity to define the structures. Robust variogram models were only attained for Ore 1 and Ore 2 domains. The final variogram models for these domains are shown in Table 13-13. These analyses provided input for the selection of search parameters and estimation strategies used for grade estimation.

Table 13-13: Variogram Model Parameters for Golema Reka

Domain	Element	Rotation[1]	Nugget	Variance[2]	Structure 1 Type	Structure 2 Type	Structure 1 Range (m)	Structure 2 Range (m)
ORE 1	Zn	47, 225, 0.1	0.15	0.40, 0.46	Spherical	Spherical	9, 21, 6	70, 31, 8
ORE 1	Pb	47, 225, 0.1	0.15	0.64, 0.22	Spherical	Spherical	5, 23, 11	51, 30, 14
ORE 1	Ag	47, 225, 0.1	0.54	0.24, 0.22	Spherical	Spherical	5, 23, 11	51, 30, 14
ORE 2	Zn	58, 226, 179	0.16	0.60, 0.24	Spherical	Spherical	4, 15, 5	17, 50, 6
ORE 2	Pb	58, 226, 179	0.18	0.44, 0.38	Spherical	Spherical	11, 23, 9	22, 30, 12
ORE 2	Ag	58, 226, 179	0.46	0.15, 0.38	Spherical	Spherical	11, 23, 9	22, 30, 12

Notes:

1. Leapfrog rotation (Dip, Dip Azimuth, Pitch)
2. Variance for structures 1 and 2 (C1, C2)
3. Ranges in Major, Semi-Major and Minor directions

13.4.4 Search Strategy and Grade Interpolation Parameters

Grade estimation was carried out in Leapfrog Edge software using Ordinary Kriging for domains ORE 1 and ORE 2 where robust variogram models were attained. For the other domains a Radial Base Function (RBF) estimator was used. Boundaries between domains were considered as hard boundaries for grade estimation.

Grade estimation was undertaken in parent cells with all sub-cells assigned the grade of the parent. Block discretisation was set to 5 x 5 x 2. The orientations of the search radii were variable based on the structural trend of the geological interpretation by using dynamic anisotropy.

The reference surfaces derived from the geological modelling for each domain were utilised to define the local structural trends of the mineralisation. These structural trends were incorporated into the estimation process to orientate and refine the search ellipsoids, thereby improving the alignment of the interpolation with the geometry of the mineralised domains.

Selection of the search radii and the orientation of search ellipsoids was guided by the results of the grade continuity analysis and the interpreted geometry of the mineralisation. Due to the variable orientation of the mineralised lenses, a locally varying anisotropic search approach was adopted, whereby each block was assigned a locally optimised search orientation that conforms to the prevailing structural trend.

Grade estimation was performed using multi-passes, with up to three search passes of progressively increasing search radii. Blocks that did not meet the criteria for estimation in earlier passes were considered in subsequent passes.

The first pass used a relatively restricted search ellipsoid, designed to reflect the higher confidence areas defined by close-spaced infill drilling. The second pass used an expanded search ellipsoid with dimensions increased by up to 100% larger relative to the first pass. This pass was designed to estimate blocks in areas constrained by more widely spaced underground exploration drilling. Where required, a final third pass was applied to estimate any remaining blocks, typically located at the margins of the model and are informed by more widely spaced surface drilling.

A summary of the estimation and search parameters for the Main Domains is shown in Table 13-14.

Table 13-14: Search and Estimation Parameters for the Main Domains at Golema Reka

General		Ellipsoid Ranges (m)			Ellipsoid Orientation	Number of Samples		Max Samples per Drillhole
Estimator Name	Element	Max	Int	Min		Min	Max	
ORE 1 Search 1	Ag	50.53	30.27	13.75	Variable	6	16	3
ORE 1 Search 2	Ag	100	60	18	Variable	6	16	3
ORE 1 Search 3	Ag	300	300	90	Variable	4	16	2
ORE 2 Search 1	Ag	29.99	30.27	12.19	Variable	6	16	3
ORE 2 Search 2	Ag	60	60	24	Variable	6	16	3
ORE 2 Search 3	Ag	300	300	125	Variable	6	16	3
ORE 1 Search 1	Pb	50.53	30.27	13.75	Variable	6	16	3
ORE 1 Search 2	Pb	100	60	18	Variable	6	16	3
ORE 1 Search 3	Pb	300	300	90	Variable	4	16	2
ORE 2 Search 1	Pb	29.99	30.27	12.19	Variable	6	16	3
ORE 2 Search 2	Pb	60	60	24	Variable	6	16	3
ORE 2 Search 3	Pb	300	300	125	Variable	6	16	3
ORE 1 Search 1	Zn	70	31.03	7.91	Variable	6	16	3
ORE 1 Search 2	Zn	140	60	15	Variable	6	16	3
ORE 1 Search 3	Zn	300	300	125	Variable	6	16	3
ORE 2 Search 1	Zn	29.99	30.27	12.19	Variable	6	16	3
ORE 2 Search 2	Zn	60	60	24	Variable	6	16	3
ORE 2 Search 3	Zn	300	300	125	Variable	6	16	3

13.4.5 Density

A regression formula based on the relationship between Pb and Zn grades and density was used to assign density in the block model and was applied post-grade estimation. The regression formula used was Density = ((Pb + Zn) * 0.0354) + 2.7058 and was derived from testwork based on core samples of 10 cm length using the Archimedes method. For blocks with no estimated Pb or Zn grades, a density of 2.7 t/m^3 was used.

13.4.6 Block Model

Block model setup and interpolation was completed in Seequent's Leapfrog Edge software. The block model parameters for Golema Reka are shown in Table 13-15. The model was not rotated. The CP considers the block model sizes appropriate for the deposit geometry and proposed mining methods.

Table 13-15: Golema Reka Block Model Extents and Dimensions

Type	X Direction	Y Direction	Z Direction
Base Point (m)	7,625,500	4,663,300	1,160
Boundary Size (m)	1,690.5	1,680	686
Number of Blocks	483	120	98
Parent Block Size (m)	3.5	14	7
Min. Sub-block Size (m)	0.4375	1.75	0.875

13.4.7 Resource Classification

Definitions for resource categories used in this report are consistent with those defined by JORC (2012).

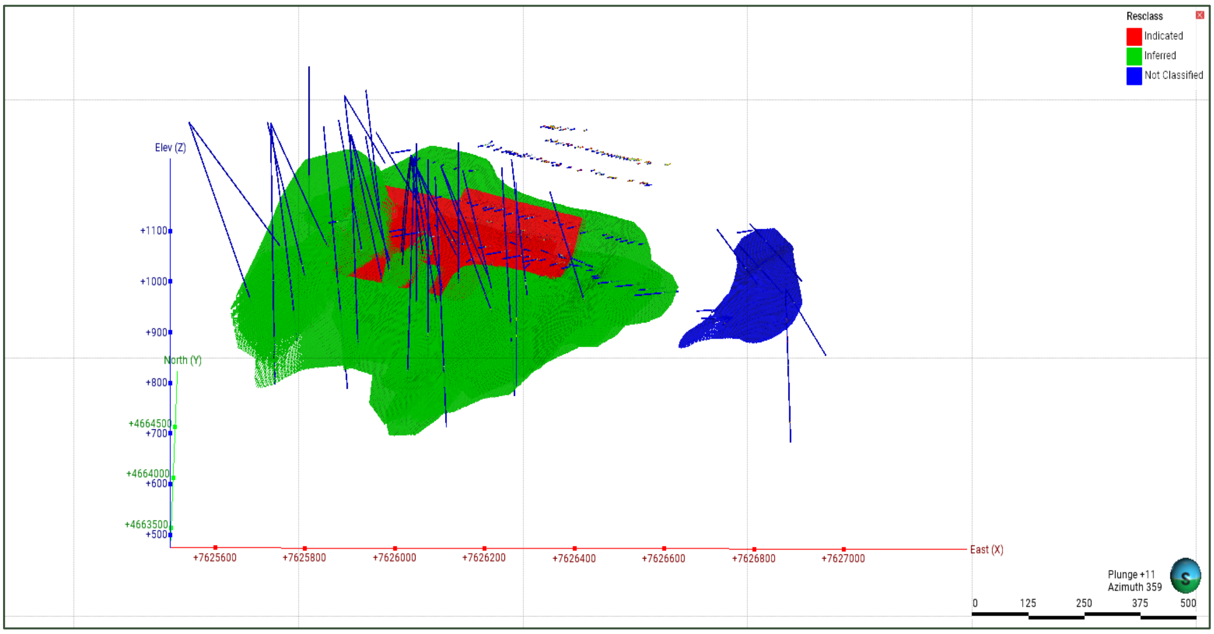
Classification of the Golema Reka Mineral Resources considered the quality and reliability of the drillhole data, the confidence in the geological interpretation, the spatial distribution of data, variogram continuity, and the robustness of the grade estimation. Classification considers geological continuity, drilling density, data quality, QA/QC performance and estimation confidence. Areas supported by closer spaced drilling, consistent geological interpretation and stronger continuity evidence are classified as Indicated Resources. Areas with more limited drilling support, increased geological complexity or lower confidence continuity remain classified as Inferred Resources pending further drilling and geological confirmation.

Based on these factors, the CP considers that the quality and spatial distribution of the available data, together with the demonstrated geological continuity of the mineralisation and the integrity of the block model estimates, are sufficient to support the classification of portions of the deposit as Indicated and Inferred Mineral Resources.

For domains ORE 1 and 2, Indicated Mineral Resources were assigned based on a minimum of four drillholes within a maximum distance of approximately 50 m. Inferred Mineral Resources were defined in areas informed by at least two drillholes within a maximum distance of approximately 150 m. Inferred Mineral Resources were also assigned to any areas below and elevation of 700 masl. For the other domains a classification of Inferred Mineral Resources was assigned except for domain ORE 4 which was unclassified due to the limited number of samples and the wide drillhole spacing.

The Mineral Resource classification for Golema Reka is shown in Figure 13-7.

Figure 13-7: Isometric View of Golema Reka Resource Classification (Note: Domain ORE 4 is unclassified)



Source: CAML (June 2026)

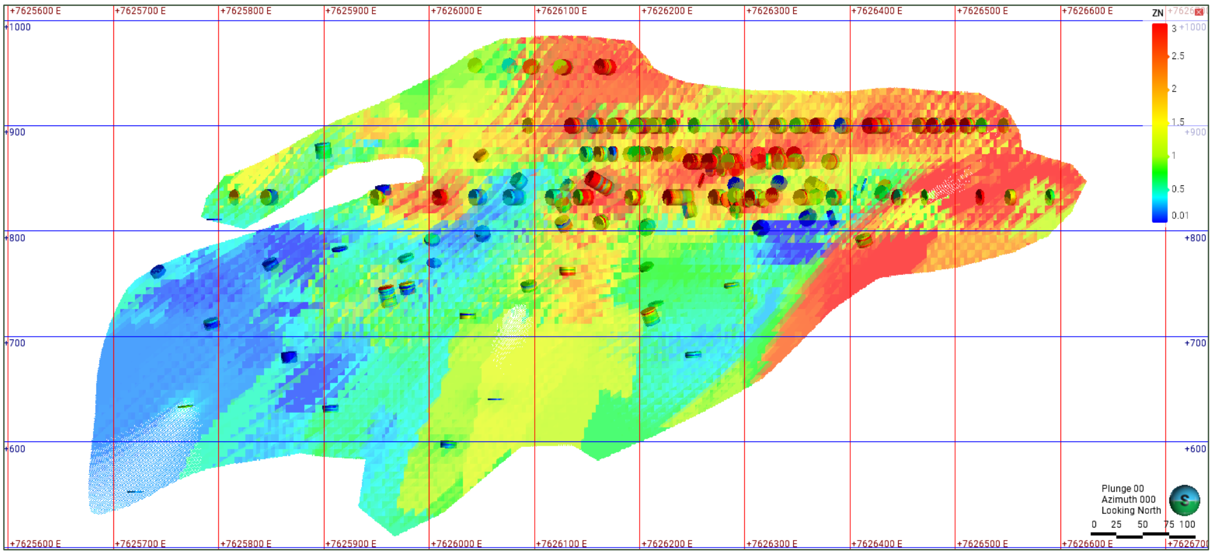
13.4.8 Block Model Validation

A visual and statistical validation of the block model was undertaken by the CP and included the following checks:

- Visual inspection comparing composite and block grades (Figure 13-8 and Figure 13-9).
- Comparison of swath plots for ordinary kriging and nearest neighbour methods (Figure 13-10).

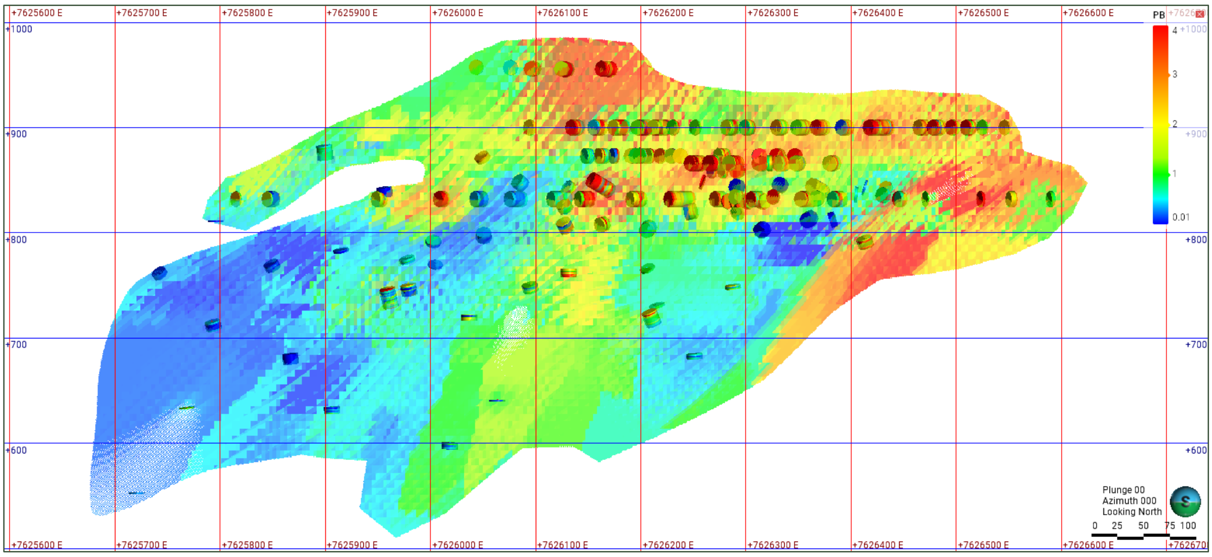
Overall, a reasonable correlation was observed between the input composites and the estimated block grades and reflected the spatial trends of the mineralisation with no material discrepancies or inconsistencies identified in the CPs review.

Figure 13-8: Isometric View of Domain ORE 1 at Golema Reka Showing Block Grades and Composites for Zn for Domain ORE 1 (looking north, scale in meters)



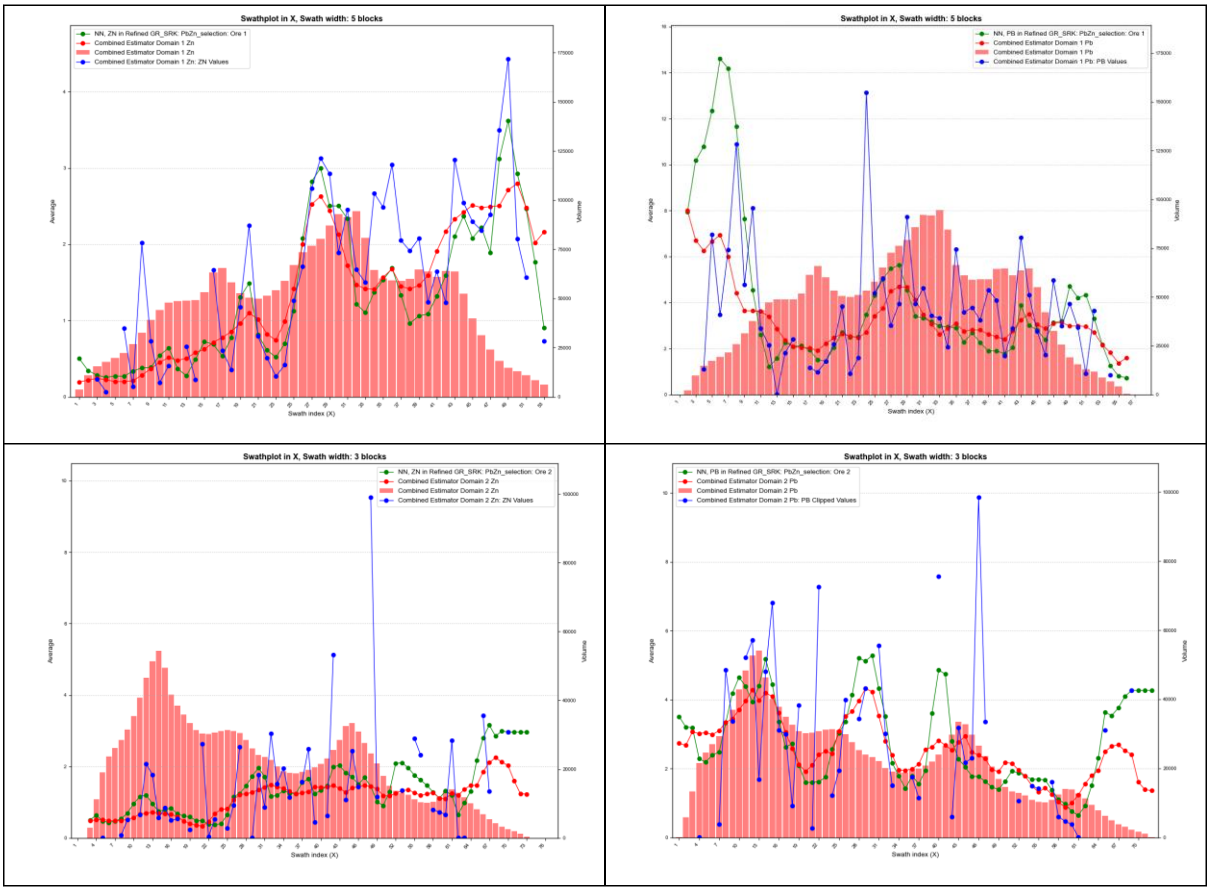
Source: SLR (June 2026)

Figure 13-9: Isometric View of Domain ORE 1 at Golema Reka Showing Block Grades and Composites for Pb for Domain ORE 1 (looking north, scale in meters)



Source: SLR (June 2026)

Figure 13-10: Example Swath Plots for Domain ORE 1 (top) and ORE 2 (bottom) in X Direction for Zn (left) and Pb (right) at Golema Reka



Source: CAML (June 2026)

13.4.9 Cut-Off Values for Reporting Mineral Resources

NSR was used to determine the value of each individual block in the Mineral Resource model and was based on the same parameters as for Svinja Reka (detailed in Section 13.3.9). Based on the geometry of the orebody at Golema Reka it was assumed that mining would use LHS mining only. Based on this, a NSR cut-off value of US$60 per tonne was used to estimate the Mineral Resources.

13.4.10 Mining Depletion

The Golema Reka deposit was historically mined between levels 1,080 and 820 mRL. No accurate depletion wireframes of the mining activity are available for depletion, therefore wireframes defining the full extent of the volumes contained in these levels were created and used to deplete the Mineral Resource model.

13.4.11 Mineral Resource Reporting

A summary of the Mineral Resource estimate for Golema Reka is shown in Table 13-16.

The stated Mineral Resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, to the best knowledge of the authors. There are no known mining, metallurgical, infrastructure, or other factors that materially affect this Mineral Resource estimate, at this time.

Table 13-16: Audited Golema Reka Mineral Resource Statement (Effective Date of December 31, 2025)

Deposit	Classification	Tonnes (Mt)	Grades			Contained Metal		
			Pb (%)	Zn (%)	Ag (g/t)	Pb (kt)	Zn (kt)	Ag (koz)
Golema Reka	Measured	-	-	-	-	-	-	-
	Indicated	1.8	4.1	1.3	13.8	75	24	810
	Inferred	6.8	3.9	1.2	13.2	263	82	2,880

Notes:

1. Mineral Resources are reported in accordance with the guidelines of the JORC Code (2012).
2. The Competent Person is Jason Ché Osmond, an independent Competent Person as defined by JORC.
3. Mineral Resources are estimated using NSR a cut-off value of $60 per tonne for long-hole stoping.
4. Mineral Resources are estimated using metal prices of US$3,041 per tonne for zinc, US$2,506 per tonne for lead and US$31 per ounce for silver.
5. Mineral Resources are estimated using metallurgical recoveries of 94% for lead, 82% for zinc and 75% for silver.
6. Mineral Resources are estimated using dry densities ranging from 2.7 to 4.0 t/m^3.
7. A minimum thickness of 1.5m was used to model the mineralised zones.
8. Mineral Resources are inclusive of Ore Reserves.
9. Contained metal refers to estimated contained metal in the ground not adjusted for metallurgical recovery.
10. Mineral Resources that are not Ore Reserves do not have demonstrated economic viability based on a pre-feasibility study or feasibility study.
11. Totals may not represent the sum of the parts due to rounding.

13.5 Mineral Resource Uncertainty

Mineral Resources are not Ore Reserves and do not have demonstrated economic viability, nor is there certainty that all or any part of the Mineral Resource estimated here will be converted to Ore Reserves through further study. Sources of uncertainty that may affect the reporting of Mineral Resources include sampling or drilling methods, data processing and handling, geologic modelling, and estimation. There are sources of uncertainty in the Mineral Resource estimate at the Property which depend on the classification assigned.

The CP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

13.6 CP Opinion

The CP reviewed the assumptions, parameters, and methods used to prepare the Mineral Resources Statement and is of the opinion that the Mineral Resources are estimated and prepared in accordance with the guidelines of the JORC Code (2012).

The CP considers that the knowledge of the deposit setting, lithologies, structural controls on mineralization, and the mineralization style and setting, is sufficient to support the Mineral Resource estimate to the level of classification assigned.

The CP considers the resource cut-off grade, and underground reporting shapes identify those portions of the Mineral Resource estimate that meet the requirement of the reasonable prospects for economic extraction to be appropriate for this style of deposit and mineralization, but that future updates consider the application of MSO to better define the Mineral Resources.

The level of uncertainty has been adequately reflected in the classification of Mineral Resources for the Property. The Mineral Resource estimate presented may be materially impacted by any future changes in the break-even cut-off grade, which may result from changes in mining method selection, mining costs, processing recoveries and costs, metal price fluctuations, or significant changes in geological knowledge.

14.0 Ore Reserve Estimates

The Ore Reserve estimates detailed in this report were prepared in accordance with JORC (2012) guidelines. The authors have reviewed the Ore Reserve estimation methodology, including mine design, operational factors, life of mine production scheduling, and confirmation of a positive financial analysis.

Ore Reserves are reported for the Svinja Reka deposit only. No Ore Reserves are reported for the Golema Reka deposit because no Pre-Feasibility Study or Feasibility Study has been undertaken.

Mining operations have been continuous at Svinja Reka since 2006, and mining methods have been developed and upgraded over time as understanding of the orebody geometry and geotechnical properties has increased.

The mining methods used at Svinja Reka are Sub-Level Caving (SLC), Long Hole Stoping with backfill (LHS), and Cut and Fill (CF). Historically, SLC was the main method, however, as the orebody narrows with increasing depth, LHS has recently been adopted as the main mining method for the LOM. Following construction of a paste backfill plant and underground reticulation system in 2023, CF mining methods are also used in areas where additional selectivity is required. Approximately 15% of the Ore Reserves will be mined by SLC, 17% by CF and 67% by LHS.

Mining methods are described further in Section 15.

14.1 Cut-Off of Values for Reporting Ore Reserves

NSR was calculated to determine the value of each individual stope and used metal prices, payable metals considering lead, zinc and silver grades, metallurgical recoveries, and realisation costs.

Metal prices used in the calculation of NSR for Ore Reserves were US$2,644 per tonne for zinc, US$2,179 per tonne for lead, and US$27 per ounce for silver.

Metallurgical recoveries used in the calculation of NSR were 94% for lead, 82% for zinc and 75% for silver. The zinc recovery is conservative and less than the 86% achieved by the processing plant. Royalties of 2% were used and are in the process of being updated to 4% based on the updated Concession Fee in 2025.

The NSR calculation also considered the planned underground mining methods based on orebody geometry for the following mining levels:

- SLC between levels 810 m and 962 m, and above 990 m.
- CF between levels 750 m and 760 m, and between 800 m and 810 m.
- LHS below level 750 m, between levels 760 m and 800 m, and between levels 962 m and 990 m.

NSR cut-off values (COVs) of US$53 per tonne for SLC, US$65 per tonne for CF and US$60 per tonne for LHS were used to estimate the Ore Reserves.

14.2 Dilution and Mining Recovery

Mining dilution was applied to excavations based on mining method, development profile and stope dimensions. Total stope dilution includes overbreak (including from backfill), Inferred Mineral Resources and material below COV contained within the stope designs. Inferred Mineral Resources contained within the stope designs were considered as having zero grade.

Mining recovery was applied based on operational experience of the different mining methods.

A summary of the dilution and mining recovery factors is shown in Table 14-1.

Table 14-1: Dilution and Mining Recovery Factors

Mining Method	Dilution (%)	Mining Recovery (%)
Sub-Level Caving	30	82
Cut and Fill	10	98
Long-Hole Stoping	18	92

14.3 Design Process

Mine design and scheduling works are carried out using the Deswik Software suite and follow a defined process which conforms to industry best practice for the estimation of Ore Reserves.

Stope locations are identified using stope optimisation, with final stope designs augmented by manual checks and verifications. Development headings are designed manually according to mining method and location.

Stopes are designed based on average block NSR values which are incorporated into the Mineral Resource block model prior to commencement of the mine design.

Stopes are designed using a combination of optimisation and manual stope/development design. The design basis is driven by the NSR COVs, in conjunction with the geotechnical parameters defining maximum stope dimensions, selected mining method and block value.

Stope optimisation creates a stope shape that maximises the recovered Mineral Resource value above the COV while also allowing for practical mining parameters including minimum and maximum mining width, anticipated wall dilutions, minimum and maximum wall angles, minimum separation distances between parallel and/or sub-parallel stopes, minimum and maximum stope heights and widths. Mineable shapes are evaluated using the Mineral Resource block model, based on the NSR COVs and the Mineral Resource classification.

Dilution factors, including consideration of dilution from backfill and dilution by Inferred Mineral Resources, are applied to the mineable shapes with a diluted stope grade calculated for each stope within the design. Stopes which have an average NSR value below the appropriate COV are excluded from the design and scheduling process.

Stopes which achieve the appropriate NSR are then processed in Deswik to calculate design quantities, including application of mining recovery factors, to define the Ore Reserves.

14.4 Ore Reserve Statement

The Ore Reserve estimate for the Sasa Mine is classified in accordance with the JORC Code (2012). Ore Reserves are derived from Measured or Indicated Mineral Resources after applying economic parameters to calculate that portion of the Measured and Indicated Mineral Resources that can be mined economically. Ore Reserves for the Sasa Mine were classified using the following criteria:

- No Proved Ore Reserves were classified because no Measured Mineral Resources were estimated.
- Probable Ore Reserves are those Indicated Mineral Resources where development work for mining and information on processing/metallurgy and other relevant factors demonstrate that economic extraction is achievable.

Ore Reserves are inclusive of diluting material that will be mined in conjunction with the economically mineralised rock and delivered to the processing plant or equivalent facility. The term ‘Ore Reserve’ need not necessarily signify that extraction facilities are in place or operative, or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

The audited Ore Reserve statement for the Sasa Mine is shown in Table 14-2.

The stated Ore Reserves are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, to the best knowledge of the authors. There are no known mining, metallurgical, infrastructure, or other factors that materially affect this Ore Reserve estimate, at this time.

Table 14-2: Audited Ore Reserve Statement for the Sasa Mine (Effective Date of December 31, 2025)

Deposit	Classification	Tonnes (Mt)	Grades			Contained Metal		
			Pb (%)	Zn (%)	Ag (g/t)	Pb (kt)	Zn (kt)	Ag (koz)
Svinja Reka	Proved	-	-	-	-	-	-	-
	Probable	6.9	3.5	2.5	26.1	244	170	5,782
Golema Reka	Proved	-	-	-	-	-	-	-
	Probable	-	-	-	-	-	-	-
Total	**Proved**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Probable	**6.9**	**3.5**	**2.5**	**26.1**	**244**	**170**	**5,782**

Notes:

1. Ore Reserves are reported in accordance with the guidelines of the JORC Code (2012).
2. The Competent Person is Colin Davies, an independent Competent Person as defined by JORC.
3. Ore Reserves are estimated using NSR cut-off values of US$53 per tonne for sub-level caving, US$65 per tonne for cut and fill stoping and $60 per tonne for long-hole stoping.
4. Ore Reserves are estimated using metal prices of US$2,644 per tonne for zinc, US$2,179 per tonne for lead and US$27 per ounce for silver.
5. Ore Reserves are estimated using metallurgical recoveries of 94% for lead, 82% for zinc and 75% for silver.
6. A minimum mining width of 1.8 m, including internal but excluding external dilution, has been used.
7. Contained metal refers to estimated contained metal in the ground not adjusted for metallurgical recovery.
8. Totals may not represent the sum of the parts due to rounding.

15.0 Mining Methods

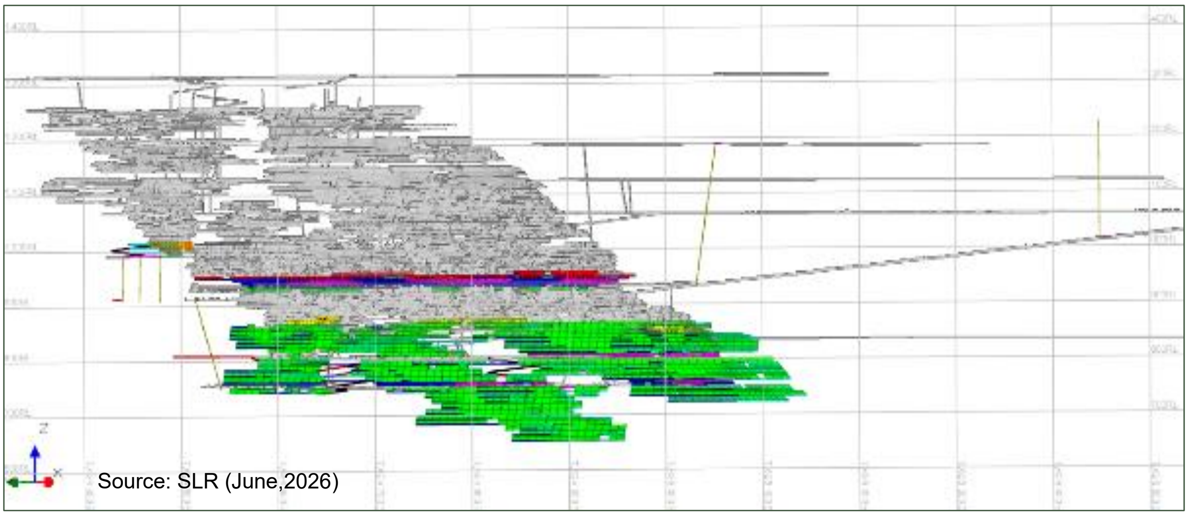
15.1 Mine Overview

The Sasa Mine has been in continuous operation since 2006, and historically from the 1960s to 2002. Current production is located within the Svinja Reka deposit, which produces approximately 800 ktpa of ore, with average grades of 3.4% Pb and 2.6% Zn.

An overview of the mine is shown below. Figure 15-1 shows an isometric view of the existing and planned development looking perpendicular to the strike of the orebody. Figure 15-2 is looking along strike and shows the dip of the lower orebody.

Figure 15-1: Isometric Mine Design Overview looking Northeast

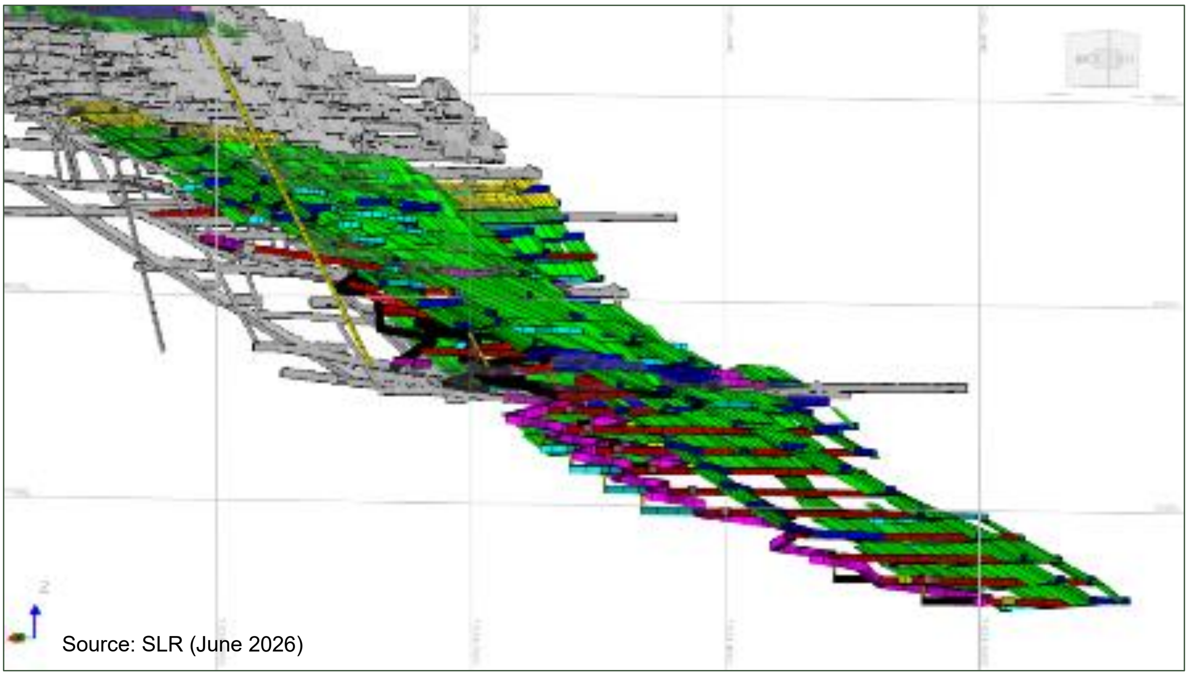
Showing Existing Mined Areas (grey), LHS (green), CF (pink), SLC (red), Planned Development (blue)



Source: SLR (June,2026)

Figure 15-2: Isometric Mine Design Overview looking Southeast

Showing Existing Mined Areas (grey), LHS (green), CF (pink), SLC (red), Planned Development (blue)



Source: SLR (June 2026)

The main access to the underground workings is through the Central Decline, which extends 3,735 m from a portal located adjacent to the process plant to a depth of 750 mRL. The decline connects to the 910, 830 and 800 levels and was commissioned in early 2025. The Central Decline is now used for all personnel and equipment movement and accounts for approximately 80% of ore haulage, transported to surface by truck. The base of the decline represents the current lowest extent of mine development.

Secondary access is provided by the Golema Reka shaft, which remains in use hoisting the remaining ore, and the Level 14b portal, which connects to the legacy service ramp and is no longer used for ore transport. Shaft haulage is scheduled to be phased out in 2026.

The underground layout comprises a series of production levels, sublevels and strike drives developed within, or adjacent to, the orebody. These provide access for drilling, blasting and mucking activities, and have been designed to support long hole open stoping with paste backfill as the principal mining method, supplemented by localised cut-and-fill. The layout also accounts for interaction with historical caved areas from previous mining.

Three principal production horizons are active, centred around the 990, 910 and 750 – 800 levels, with the 830-level functioning as a rail haulage level linking to the Golema Reka shaft system. Prior to commissioning of the Central Decline, this level formed the main ore handling route.

Ore is mined from stopes and hauled to ore passes by Load Haul Dump machines (LHDs). Material is then hauled to surface by truck or, until recently, via the shaft system. Current operations typically maintain approximately 10 to 12 active stopes per month.

Haulage within the legacy mine development is undertaken using 20 t trucks operating in headings approximately 3.5 m by 3.5 m in profile. Newer development associated with the decline has been constructed to larger dimensions, 4.0 m by 4.0 m, to accommodate a transition to 32 t haulage trucks. Selected existing headings are being widened to match the larger truck requirements principally through slashing of arched profiles to more rectangular sections.

Historically, the mine operated using sub-level caving with 7.0 m sublevel spacing. As the orebody narrows with depth, the suitability of this method has declined. Since 2023, a transition to long hole stoping with backfill has been ongoing as the primary mining method. Cut and fill mining is also used in areas requiring greater selectivity or with difficult ground conditions. Both historical voids and newly mined stopes are backfilled using cemented tailings.

The current LOM plan extends over approximately nine years. Production is expected to increase to around 830 ktpa over the next three years, before reducing towards the final years. Average planned grades over this period are 3.5% Pb and 2.5% Zn. Mining is split between long hole stoping (67%), cut and fill (17%) and sub-level caving (15%).

15.2 Stoping Methods

15.2.1 Sub-Level Caving

Sub-level caving (SLC) is the historic bulk mining method at Sasa. Mining is undertaken from sublevels developed at approximately 7.0 m vertical spacing, with longhole ring drilling and recovery of blasted material via drawpoints. Drawpoints induce controlled caving of the overlying rock mass, allowing steady production with relatively limited development.

SLC is well suited to the broader, more continuous mineralised zones encountered at shallower levels. However, the method is associated with elevated dilution due to the ingress of caved waste. Dilution of approximately 30 % and ore loss of 18 % are estimated based on historical performance and reconciliation.

Narrowing ore widths and greater geological complexity as the mine extends deeper has resulted in SLC becoming less suitable, with difficult draw control, higher dilution and reduced selectivity. SLC is not considered appropriate for areas of the orebody where mineralisation is narrow or discontinuous.

SLC continues to be used in a limited number of locations. These include parts of the 948 – 955 levels in the North zone, where existing planning pre-dates the transition and where ore geometry, together with proximity to established cave zones, allows relatively efficient extraction with reduced development. It is also used at the 868 level to complete the final panel and within the 854 – 846 levels in the South zone, which constitutes a standalone mining area. In these cases, SLC provides a practical means of achieving short-term production at relatively low cost.

15.2.2 Long-Hole Stoping

Long hole open stoping (LHS) with paste backfill is the principal mining method under the current mine plan. Mining is carried out by opening a slot between two levels, and longhole drilling between them to blast rings. Broken ore is recovered at the base of the stope using LHD loaders and transported to ore passes or direct to haul trucks.

Stopes are 10.0 m in height (between sublevels), 2.0 – 8.0 m in width, and 15 - 20 m along strike constrained primarily by hanging wall stability and allowable unsupported spans. The minimum mining width is 1.8 m, including a dilution skin applied during design, with an additional allowance for overbreak. Most stopes are wider than 2.0 m and the limited number of very narrow stopes are located on the deeper levels such as the 750 m horizon.

LHS is better suited than SLC to the narrower and more variable mineralisation encountered at depth and should result in lower dilution. For modelling and reporting purposes, dilution of approximately 18% and ore loss of 8% have been applied. However, the method is more sensitive to drilling accuracy, stope preparation, and sequencing, and requires all development to be completed prior to stoping. As a result, the rate at which new production areas can be brought online is dependent on development performance and stope readiness.

The shallow dip of the orebody, typically around 35° and in places as low as 24°, presents additional challenges. Slot raise development at these angles has historically been problematic with failures delaying production. Modified practices, including steepening slot orientation to position the foot of the slot outside the orebody, or firing shallow rings against paste fill, have improved performance and reduced slot failure rates to 5-10%. Ore movement along shallow footwalls is challenging and is reported to be assisted by flushing with water from upper levels.

Development drives may be located in either the footwall or hanging wall depending on ground conditions and drilling requirements. Access strategy varies according to orebody geometry. In thicker zones exceeding 8.0 m, transverse stoping is used, with crosscuts developed into the orebody rather than longitudinal retreat drives. Retreating full length along an >8.0 m open span often provokes instability.

Following extraction, stopes are filled with cemented paste backfill, providing both ground support and void elimination. Paste is produced from filtered tailings with binder and delivered underground via a reticulation system. A curing period of 10 days is required before adjacent mining, although limited access above stopes for preparatory work may occur after around five days. Paste availability and curing time are critical constraints on the production sequencing.

15.2.3 Cut and Fill Stoping

Cut and Fill (CF) stoping is used in areas where ore geometry or ground conditions are not conducive to LHS. This includes zones of reduced rock mass competence, variable thickness, or increased structural complexity, where greater control of excavation is required.

Mining is undertaken using a modified CF approach. A central drive is developed within the orebody and subsequently widened through controlled slashing, typically 2.0 m per side. The excavation is then filled with waste rock and paste fill prior to advancing to the next slice. Slice heights are approximately 4.0 m, with each slice drilled, blasted and mucked before fill placement.

CF mining methods are typically applied in poorer ground, where unsupported spans associated with LHS cannot be sustained. An unsupported span of approximately 8.0 m is generally considered acceptable under current conditions, although local support, including shotcrete, bolts and mesh, is installed where required. Fibre-reinforced shotcrete is commonly applied in backs to maintain stability. Access configuration is adapted to local conditions. Where ore widths exceed approximately 8.0 m, transverse access is used to improve excavation control and stability. Dilution of approximately 5% and ore loss of 2% are estimated.

CF is used in numerous areas in the mine. In the 924 - 932 sill pillar zone it is used around the historic caved area, and it is used as part of a hybrid production method in the 750 - 800 levels with LHS continuing above the initial lift. CF is reportedly being used more extensively than planned, such as in the central Southern zone, due to poor rock quality. Additional use of CF is reasonable to provide improved control and reduce dilution; however, it is more labour intensive with lower productivity compared to LHS, and unplanned use of CF has knock-on effects on the overall mine plan.

15.3 Reconciliation and Grade Control

Reconciliation at Sasa is undertaken by comparing mined production against the resource block model, with reporting carried out monthly. Calculations are based on 50 m by 50 m grid panels, incorporating both development and stoping material, which provides a high-level indication of tonnes and grade but does not allow detailed assessment at individual stope scale, or give clear data on specific variances.

Reconciliation performance is variable across the operation. In some areas, local reconciliation variance has been reported in the order of 60–70%. These variances reflect geological variability, limitations in block model resolution, and constraints in grade control practices. Misalignment between the resource model and the actual mineralisation has been identified locally, although this is improving as additional data are incorporated. Over the past year, a cavity monitoring system (CMS) scanning of open LHS stopes has been introduced to support reconciliation. The resulting survey data are incorporated into grade control models and are contributing to improved confidence in reconciliation on a local scale.

Dilution is a primary contributor to reconciliation variance and differs by mining method and ground conditions. SLC has the highest dilution due to the nature of caving and the difficulty of controlling waste ingress. LHS has lower dilution, particularly in narrower ore zones, but this is sensitive to ground conditions and drilling accuracy. For planning, dilution estimates of 30% have been applied for SLC and 18% for LHS, although LHS reconciliation data is limited. Development dilution is assumed at approximately 10%, with reported values generally between 10% and 14%.

The ore contact is visible in the face for experienced operators, which aids control of extraction boundaries. Dilution does remain difficult to manage consistently, influenced by ground conditions, blasting performance, and particularly the alignment of stopes with the orebody. In

LHS areas, there are significant discrepancies between stope design and actual geometry arising from differences between the resource model and the geology.

Grade control is implemented but constrained by operational factors. Grade control drilling is planned prior to stope opening, however, delays in drill availability and competing development priorities limit coverage. Assay turnaround times are typically two weeks or longer, limiting the timely availability of data, so grade control results are often not incorporated into final stope designs. As a result, stopes are often designed using the longer-term resource model rather than the more detailed short-term model.

Technical Services report that grade control data for more than 50% of stopes is subject to delays of up to one month due to grade control drilling and assay turnaround, which results in stopes entering production without incorporating updated data. Drill access limitations, such as the need for footwall development to position drilling equipment, is also restrictive. In areas of high geological variability, the current level of grade control should be increased to better identify local geological variability.

15.4 Mining Rates

Development and production rates at Sasa are based on current equipment capabilities and operating experience. Development advance using jumbo drills is typically in the order of 2.3 m to 2.8 m per round, depending on the equipment in use.

Production rates vary by mining method. LHS design extraction rates are 750 t per day per stope, and SLC design rates are 500 t per day per stope (24-hour basis). Individual stopes typically contain between 2,500 t and 3,500 t of material, with an average tonnage of 160 t per metre of stope length, depending on width.

Development rates are 100 m per month in the decline and main drives, and 95 m per month in ventilation and ore pass infrastructure. CF heading advance is 60 m per month which considers the additional requirements for backfilling.

Backfill placement rates are 1,200 m^3 per day. The backfilling cycle includes four days for barricade installation, followed by a curing period of up to 10 days prior to adjacent mining.

The operation typically maintains 10 to 12 active stopes per month.

While design LHS production rates are higher than historical methods, actual performance is currently reported at approximately 50% of planned rates. This reflects the ongoing transition to LHS and operational learning. Performance is improving as experience increases and with mining of thicker, more continuous zones better suited to the method.

15.5 Geotechnics

15.5.1 Geotechnical Setting and Rock Mass Conditions

The Svinja Reka deposit is hosted within a sequence of quartz-graphite schists with mineralisation occurring as stratabound lenses. The orebody strikes broadly northwest–southeast and dips to the southwest. Thickness is variable and geological complexity increases at depth. Recent mining indicates narrower ore zones and greater structural complexity in deeper areas requiring tighter control of excavation geometry and sequencing.

The structural setting is dominated by a complex fault network. Four principal fault systems have been identified, generally steeply dipping from southeast to west. The rock mass has been significantly affected by tectonic deformation, resulting in intersecting fault sets and a major structure sub-parallel to the orebody.

Geotechnical characterisation has improved significantly in recent years. Systematic geotechnical logging of drill core began in 2022, and this data has been built into a geotechnical block model based primarily on Q-prime classification, supported by various additional parameters including total core recovery (TCR) to identify voids. Logging is completed for all drillholes in areas of interest with historical core and geological logs retained for future reference.

A site-wide geotechnical and structural model was further developed following a visit by TECT Geological Consulting in April 2024. This work incorporated detailed drill core logging and scanline mapping, improving structural interpretation and supporting refinement of the geotechnical domains and design inputs.

Laboratory and geotechnical testwork were undertaken in 1978, 2017 and 2019, and is still considered representative, although current observations suggest that the skarn and schist units being mined may be marginally stronger than those encountered historically.

The mine is not considered high stress. Stress redistribution does lead to localised unloading effects near historical voids which influence ground response.

15.5.2 Geotechnical Domains

The Svinja Reka deposit has been split into four principal geotechnical domains based on lithology, structure and observed behaviour. These domains capture variations in rock mass strength, discontinuity characteristics and the presence of localised weak ground.

Each domain is characterised using standard rock mass classification systems, as summarised in Table 15-1. These classifications are derived from drill core logging, mapping and testwork, and provide the basis of design and support selection.

Table 15-1: Rock Mass Rating Values

Lithology	RMR	MRMR	Q	GSI
Schist	58-75 (72) (Good)	48-67 (62) (Good)	9-28 (20) (Good)	53-70 (67)
Sheared Schist	26-46 (41) (Fair)	13-29 (22) (Poor)	0.20-1.11 (0.67) (Very Poor)	21-41 (36)
Skarn	64-80 (77) (Good)	55-72 (70) (Good)	30-60 (57) (Very Good)	59-75 (72)
Gneiss	(76) (Good)	(63) (Good)	(28) (Good)	(71)

15.5.3 Ground Support Practices

Ground support is installed in development and production areas in accordance with defined mine standards based on the Q system and supported by local site correlation. Typical support comprises rock bolts, mesh and shotcrete. Fibre-reinforced shotcrete has replaced conventional shotcrete, improving support capacity and durability. An on-site batch plant provides flexibility in shotcrete manufacturing.

Additional support is applied in weaker ground. Cable bolting has recently been introduced in the hanging wall of long hole stopes in areas with lower Q-prime values. This has been implemented over the past five months to improve stability in locally poor ground. Installation is selective due to the impact on production cycles as it requires use of longhole drilling equipment.

Historically, the operation experienced fatalities and reportable injuries related to falls of ground during the 2009–2016 period. These incidents were primarily attributed to working under unsupported ground and have informed current support standards and practices.

15.5.4 Stope Stability and Span Design

Stope stability is assessed through a combination of empirical and numerical methods. Stable spans are derived from Mathews' Stability Graph, supported by site-specific empirical data. An empirical assessment by SRK in 2023 formed the basis of this, and the site-specific database is being constantly updated.

Numerical analyses are carried out using RS2 to evaluate stress conditions and excavation response. EX3 boundary element modelling is used for mine-scale assessment, particularly for paste fill performance. Kinematic analyses using UnWedge are applied to assess structurally controlled failures.

Maximum unsupported spans are typically limited to 15 – 20 m, particularly within the weaker hanging wall. Stability is sensitive to span length. Reducing strike length improves stability but increases cycle time due to additional barricade construction and paste curing requirements. Current design adjustments are incremental, with LHS stopes reportedly being shortened from 20 m to nearer 17 m in strike.

15.5.5 Backfill and Ground Control

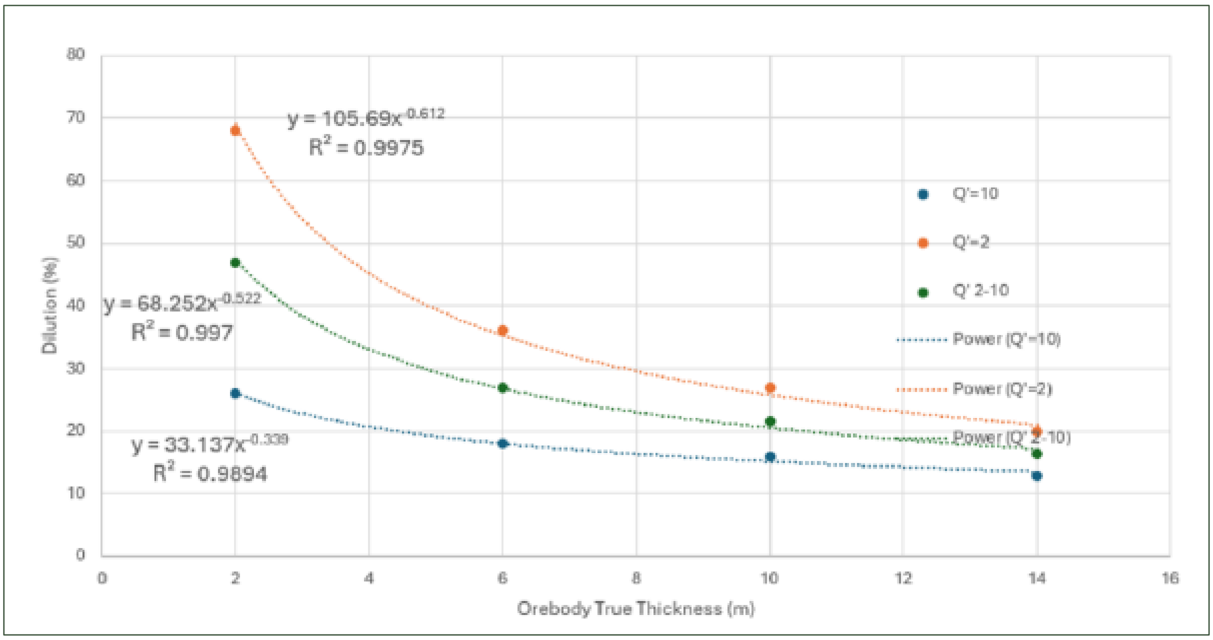
Paste backfill provides local and regional ground support. It is used to fill mined voids and provide confinement for adjacent and overlying stopes. In some areas, such as alongside the sill pillar and existing caved areas on the 924 - 932 level, paste is designed to higher strengths to act as structural support.

Design parameters include a target unconfined compressive strength of approximately 1.5 MPa for underhand mining and 0.5 MPa for adjacent exposures. A minimum curing period of 10 days is required before adjacent mining.

15.5.6 Dilution and Hanging Wall Stability

Hanging wall overbreak is a major contributor to dilution. Current estimates indicate a global overbreak of approximately 18%. A site-specific regression model has been developed to relate overbreak to geotechnical parameters, supported by empirical stability and Equivalent Linear Overbreak/Slough (ELOS) analysis. The summary regression is shown in Figure 15-3.

Figure 15-3: Dilution vs Orebody Thickness



Source: Rudnik SASA DOOEL (June 2026)

Overbreak is influenced by both ground conditions and drill and blast practices. Deviations of drillholes into the hanging wall are a recognised cause of excess dilution in LHS stopes. Improvements in blasting design and interdepartmental feedback over the past six months have been implemented to address this.

Early results suggest that targeted cable bolting in weaker hanging walls may reduce overbreak to approximately 15%. This practice began towards the end of 2025, and reconciliation is still being validated.

Production face mapping, scanlines and infill drill core logging is carried out routinely and incorporated into Leapfrog models. These allow fast visual updates to geological and geotechnical interpretations to support ongoing design improvement.

15.6 Mine Water

The Sasa Mine is dewatered primarily by gravity drainage, supported by localised pumping where required. Mine water originates from groundwater inflow, surface water ingress through historical cave zones and openings, and operational sources such as drilling. Inflows are reported to be stable, with no significant increases expected over the LOM.

Mine water is managed in two principal drainage levels, the 1060 and 830 levels. Most of the water from higher workings is drained through the 1060 level to an upper adit which discharges to a surface settlement facility. The lower drainage system on the 830 level drains through a lower 5 km long adit. Contact water is pumped back to the tailings storage facility. Non-contact water flows via sedimentation ponds prior to discharge to the Kamenica River. In addition, mine water from other active levels is collected and directed to the tailings facility.

The main pumping installation comprises two multi-stage Wilco pumps located at the 830- level. These are estimated to manage approximately 860 m³/day (10 L/s) for local water transfer within the system rather than primary dewatering. The 1065-level is linked to the 830-level system via the decline, providing additional connectivity within the drainage network. While some localised pumping is required, overall dewatering demand is low due to favourable mine geometry and drainage conditions.

Based on site observations and previous technical reviews, mine water management is not considered a high operational risk. Significant water accumulation has not been observed underground, and existing infrastructure is considered adequate. Inflow from caved areas in the upper levels remain sensitive to seasonal rainfall and require ongoing monitoring, and continued maintenance of drainage and pumping systems is required to continue to operate efficiently.

15.7 Ventilation

The Sasa Mine is ventilated using a mechanical overpressure system that supplies fresh air to active mining areas and exhausts return air through a network of raises and airways.

The system currently incorporates two principal fans: a 450 kW Zitron ZVN unit at the Level 16 portal and a 200 kW Zitron ZVN unit installed underground to drive airflow through a raise system at approximately 990 mRL. Fresh air is supplied primarily via the Central Decline portal (approximately 67 m³/s) and the Level 14b portal (approximately 45 m³/s), with additional intake from the Golema Reka shaft (approximately 19 m³/s), giving a total airflow of approximately 131 m³/s.

The ventilation system has been designed and reviewed by external specialists at the local university working alongside an on-site ventilation engineer. Ventilation at the mine is understood to be modelled in Ventsim software. A high-level assessment by WAI in 2025 indicated that installed airflow meets CANMET (Canada Centre for Mineral and Energy Technology) requirements for the existing diesel fleet (although CANMET does not apply to

North Macedonia it was deemed a representative analysis). Total airflow demand from LHS and haul trucks was estimated at approximately 129 m³/s suggesting that overall ventilation capacity is broadly aligned with operational requirements.

Underground observations during the site visit indicated that airflow distribution is not uniform. Localised areas of reduced ventilation were noted, particularly in development headings where auxiliary fans are required. The Technical Services department reported that elevated temperatures have led to the temporary suspension of activity in certain stoping areas until ventilation conditions improve. Although this is reported to be infrequent, it highlights the sensitivity to ventilation and the limited flexibility in capacity. Additional ventilation infrastructure is being developed, including raise boring in the northern areas to improve connectivity within the circuit as mining advances to greater depth. The ventilation improvement plan includes an additional primary fan on 14b level which will provide an additional 45 m³/s flow in conjunction with the additional raise bore vent shafts.

15.8 Backfill Systems

Backfilling at the Sasa Mine is undertaken using a cemented paste backfill system. The system has been developed in conjunction with the transition to LHS and CF mining methods.

Paste backfill is produced at a surface plant, where tailings from the flotation process are dewatered using filtration and combined with binder and water to form a paste of suitable consistency. The paste is then piped underground for placement in stopes. This approach allows the significant proportion of process tailings to be returned underground as engineered fill rather than being deposited in surface storage facilities.

Key design parameters for the paste backfill system include target unconfined compressive strength of 1.5 MPa for underhand mining and 0.5 MPa for adjacent exposures, and a curing period of approximately 10 days prior to adjacent mining. Backfill delivery is modelled at 1,200 m^3/d, and barricade construction takes four days.

The availability and performance of paste backfill is critical to production. Mining sequences are dependent on fill placement and curing cycles, and any interruption in backfill supply has the potential to impact stope turnaround times and overall output. No issues with the paste system or cycle were reported on site.

15.9 Waste Dumps

Waste rock generated at Sasa arises from underground development and stope preparation. A proportion of this material is utilised underground where practicable, including for construction purposes and as part of the backfilling to support mining activities.

Waste material not retained underground is placed in controlled surface disposal areas alongside the historic tailings and newer dry stack facilities. Waste rock placement is managed in accordance with standard mining practice, and appropriate consideration is given to stability, drainage and environmental controls.

15.10 Mining Equipment

The Sasa Mine uses conventional underground mining equipment. LHDs are used for mucking ore from the stope draw points and development headings. Blasted ore is mucked into 20 t and 32 t trucks which transport the ore to the surface via the Central Decline or to the Golema Reka shaft for hoisting. Development drilling is carried out using single-boom jumbo rigs, while production drilling in stopes is undertaken using dedicated long hole drill rigs.

The primary mining equipment used at the Sasa Mine is summarised in Table 15-2. Underground mining equipment is owned and operated by Rudnik SASA DOOEL.

Table 15-2: Summary of Underground Mining Equipment

Equipment Category	Equipment Type	Total Units
Primary Production Equipment		
Development Drills	Boomer Epiroc S1D + S2D	8
Long Hole Drills	Epiroc Simba S7	2
LHDs	Epiroc Scooptram ST7	8
Haul Trucks	Epiroc MT2010 + MT2200 + MT436B	8
Ancillary Equipment		
Ground Support & Backfill Equipment	Bolter, Shotcrete Sprayers, Mixers, Scalers	12
Underground Support & Services	Rail (locos/trolleys), grader, utility equipment, light vehicles, personnel carriers, fuel and explosives vehicles	34
Drilling & Rock Breakage Support	Exploration drills and rock breakers	4
Surface Support Equipment	Loaders, forklifts, telehandler, light vehicles, medical and site infrastructure	11

Development drilling capacity comprises seven Epiroc S1D units and one S2D unit. The haulage fleet includes Epiroc MT2010, MT2200, and MT436B trucks. Ground support and backfill equipment covers bolting units, shotcrete sprayers, mixers and scaling equipment. Underground support services include mobile auxiliary equipment and legacy rail haulage components. Surface equipment supports logistics, maintenance and emergency response functions.

The fleet is largely consistent over the LOM. Minor changes are planned to include the addition of one development drill in 2028 and one long hole drill in 2031. The authors consider the equipment configuration is appropriate for the mining methods, production requirements and mine layout.

15.11 Personnel

The current mine workforce at the Sasa Mine totals 319, including capital development, transport, and underground paste backfill.

Most are employed by Rudnik SASA DOOEL with contractor support used for exploration drilling and special projects.

15.12 Life of Mine Schedule

The LOM schedule for the Sasa Mine is shown in Table 15-3 and runs from 2026 to 2034 (inclusive). The Ore Reserve estimate is based on the LOM schedule.

Inferred Mineral Resources are those that will be mined during the LOM, these total 6%, the majority of which are concentrated towards the end of the schedule. Inferred Mineral Resources are not included in the Ore Reserves, and no value is assigned to them in the economic analysis.

Table 15-3: Sasa Life of Mine Schedule

Item	Unit	Total LOM	2026	2027	2028	2029	2030	2031	2032	2033	2034
Ore Reserves											
Proved Ore Tonnes	kt	**-**	-	-	-	-	-	-	-	-	-
Pb Grade	% Pb	**-**	-	-	-	-	-	-	-	-	-
Zn Grade	% Zn	**-**	-	-	-	-	-	-	-	-	-
Ag Grade	g/t Ag	**-**	-	-	-	-	-	-	-	-	-
Probable Ore Tonnes	kt	**6,882**	815.2	799.5	797.0	780.5	767.8	821.7	768.3	698.3	633.1
Pb Grade	% Pb	**3.55**	3.65	3.77	3.81	3.63	3.42	3.81	3.85	3.43	2.30
Zn Grade	% Zn	**2.47**	2.85	2.57	2.55	2.52	2.99	2.39	2.34	2.45	1.35
Ag Grade	g/t Ag	**26**	21	25	26	28	26	27	29	28	26
Total Ore Tonnes	kt	**6,882**	815.2	799.5	797.0	780.5	767.8	821.7	768.3	698.3	633.1
Pb Grade	% Pb	**3.55**	3.65	3.77	3.81	3.63	3.42	3.81	3.85	3.43	2.30
Zn Grade	% Zn	**2.47**	2.85	2.57	2.55	2.52	2.99	2.39	2.34	2.45	1.35
Ag Grade	g/t Ag	**26**	21	25	26	28	26	27	29	28	26
Inferred Mineral Resources											
Inferred Resources	kt	**445.8**	0.4	30.7	33.1	49.8	62.8	8.4	61.7	127.0	72.0
Pb Grade	% Pb	**2.46**	2.31	2.26	2.25	3.03	3.88	2.56	1.97	2.11	2.04
Zn Grade	% Zn	**2.87**	2.39	2.74	2.77	3.73	5.06	2.69	2.07	2.54	1.77
Ag Grade	g/t Ag	**22**	22	16	17	20	19	23	22	22	28

Notes:

1. Ore tonnes are Probable Ore Reserves as presented in Section 14 of this report.
2. Mining losses and mining dilution applied as detailed in Section 14 of this report.
3. Totals may not represent the sum of the parts due to rounding.

16.0 Recovery Methods

The Sasa processing plant was commissioned in 2006 and was originally designed to process 650 ktpa of ore. The plant was upgraded in 2007 with additional Metso flotation cells to a nameplate capacity of 850 ktpa of ore. However, to match the mine production rate, the plant operates at throughputs less than this and, in recent years, has processed approximately 800 ktpa.

16.1 Process Description

The process plant comprises a three-stage crushing circuit and conventional grinding, classification and selective flotation circuits to produce lead and zinc concentrates that are thickened and filtered for shipment by truck. A paste backfill plant and dry stack plant were constructed and commissioned in 2023 and 2025, respectively. For 2026, it is planned that circa 31% of the tailings will be dry stacked, 36% will be used for paste backfill and 33% stored in the existing TSF-4. Paste backfill is the priority method for tailings disposal.

The TSF, dry stack tailings and paste backfill plant are further described in Sections 17.4, 17.5 and 17.6, respectively.

The post-crushing flowsheet for the Sasa processing plant is shown in Figure 16-1 and is described in the following sections. Note that this flowsheet does not show the additional Stirred Media Detritor (SMD) regrind mill that was installed in the zinc circuit in 2017.

16.1.1 Crushing

The crushing circuit operates 24 hours per day, 7 days per week on a three-shift basis. The three-stage crushing circuit was installed in 2006 using new Metso crushers and screens and incorporates refurbished feeders and conveyors from the original Russian-designed plant. The crushing plant throughput is 150 tph at a design utilisation of 67%. A C106 jaw crusher and GP220 secondary cone crusher operate in open circuit with the HP200 tertiary crusher operating in closed circuit with a vibrating screen to produce crushed minus 20 mm feed for the grinding circuit.

16.1.1.1 Primary Crushing

Run of mine (ROM) ore at nominally minus 350 mm is fed via feed conveyor to a vibrating screen ahead of the Nordberg C106 jaw crusher. Screen undersize and jaw crusher product (-100 mm) are conveyed to the secondary crushing circuit. Tramp iron is removed from the secondary crusher feed by an overhead magnet with secondary protection from a metal detector.

16.1.1.2 Secondary Crushing

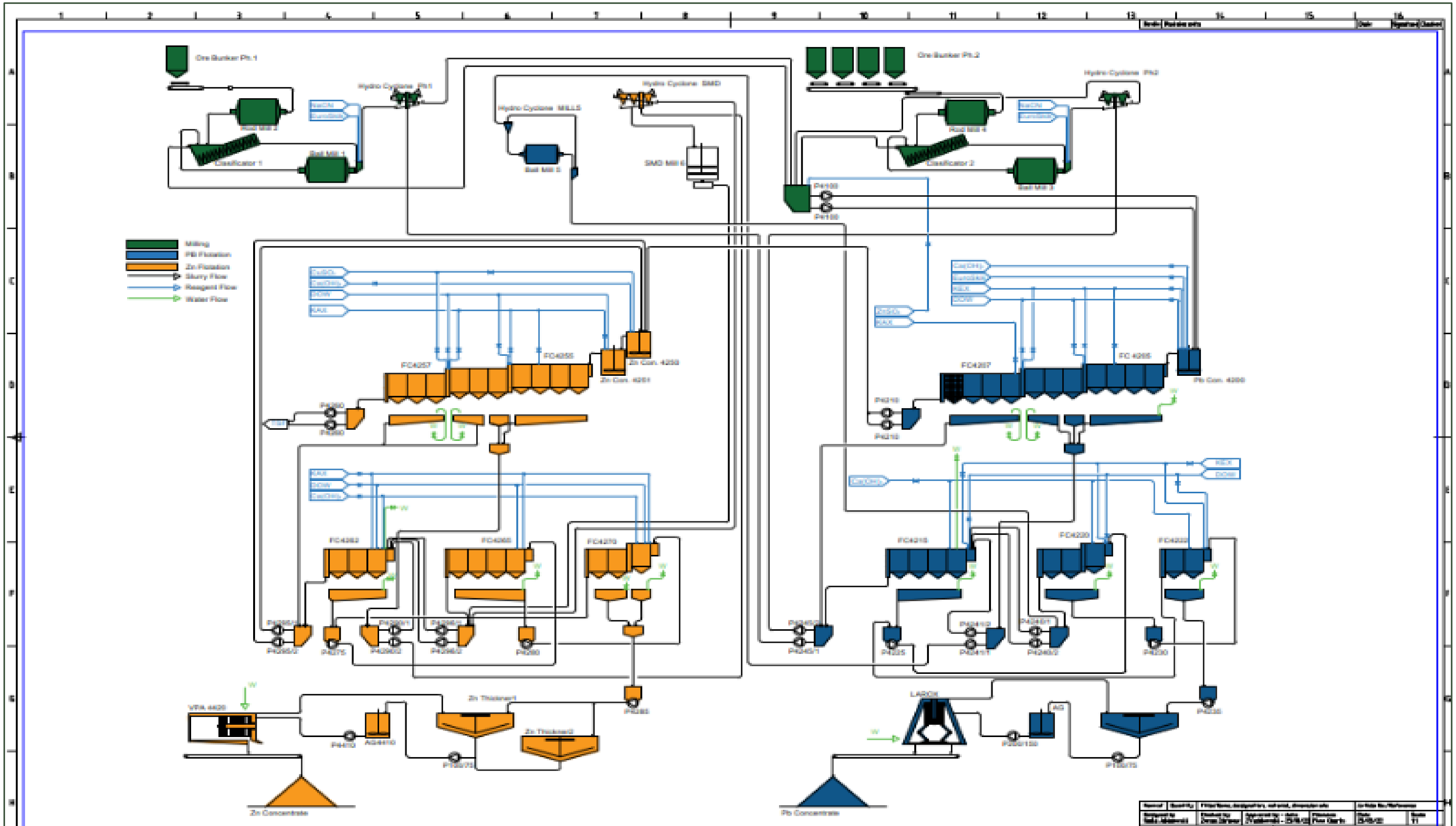
Secondary crushing uses a Nordberg GP220 cone crusher operating in closed circuit with a vibrating screen. Primary crushed ore is screened at 30 mm and the +30 mm oversize crushed in the Nordberg GP220 cone crusher operating in open circuit. Screened and secondary crushed ore is conveyed to the tertiary crushing circuit.

16.1.1.3 Tertiary Crushing

Secondary crushed ore is conveyed to the tertiary vibrating screen where final crushed ore at minus 20 mm reports to the grinding circuit feed bins. Screen oversize is crushed in the HP200 tertiary crusher operating in closed circuit with the screen.

A bifurcated chute with a manual diverter flap routes the final crushed product to either the 1,500 t bin (stream 1 for 1 phase milling) or the 4 x 500 t bins (stream 2 for 2 phase milling). A tripper conveyor is used to distribute material to the 4 x 500 t bins.

Figure 16-1: Flowsheet for the Sasa Processing Plant (Sasa Mine)



Source: Rudnik SASA DOOEL (2016)

16.1.2 Grinding

The grinding circuit comprises two parallel lines. Line 1 includes a single 1,500 t capacity crushed ore feed bin, while Line 2 includes four 500 t capacity feed bins fed via a tripper conveyor. Each grinding line operates independently and incorporates an open circuit rod mill, a twin spiral classifier and a ball mill operating in closed circuit with hydrocyclone classifiers. Both rod mills and both ball mills each have 380kW motors installed and all the mills are of original ex-Soviet manufacture. All four mills are of the same size, namely 2.7 m diameter x 3.6 m long. Steel liners are used for the rod mills and rubber liners for the ball mills.

The capacity of the grinding circuit is 109 tph at a design utilisation of 92%. Historically, utilisations of between 94 and 96% have been achieved. The final grind size feeding the flotation circuit is circa 70-75% minus 75 microns.

The -20 mm crushed ore is fed to the rod mill via variable speed feeders and a mill feed conveyor, with the feed tonnage measured by a weightometer. Rod mill discharge gravitates to the spiral classifier where fines are removed via the classifier overflow weir and report via a transfer pump to the lead flotation circuit conditioner. The coarse sands are fed via the classifier screw to the ball mill feed. Ball mill discharge gravitates back to the spiral classifier.

The rod mills use 85 mm diameter steel rods, and the ball mills use 60 mm diameter steel balls.

Sodium cyanide (NaCN), used as a zinc and pyrite depressant, and EuroSkik (a specific lead collector), are added to the ball mill feed. Water addition to the grinding circuit is automatically controlled.

16.1.3 Flotation

16.1.3.1 Lead Flotation Circuit

Lead conditioning, after zinc sulphate (used as a zinc depressant) is added, is achieved in a single agitated conditioning tank. Further reagents added to the conditioner include lime for pH control, EuroSkik and Potassium Ethyl Xanthate (KEX) collectors and Dowfroth frother. The lead flotation circuit comprises lead rougher and scavenger flotation and three stages of lead concentrate cleaning.

Initial flotation is undertaken in 7 off (4 + 3) Metso RCS20 Rougher flotation cells, each of 20m3 capacity, for a residence time of 15 minutes. Additional KEX and Dowfroth reagents are added. The rougher tails are treated in 4 off (3 + 1) Metso RCS20 Rougher Scavenger cells, also of 20 m^3 capacity, with a residence time of 15 minutes. Potassium Ethyl Xanthate (KEX) is added as collector to the Rougher Scavengers. The Rougher Scavenger tails are pumped to the first of two zinc conditioning tanks. The pH is adjusted in the first tank, and reagents are added to the second tank. The Rougher Scavenger concentrate is pumped back to the grinding circuit (cyclone feed).

The (4 + 3) rougher concentrate is pumped to the regrind circuit, consisting of a 1.8 m diameter x 2.4 m long ball mill (110 kW), operating in closed circuit with hydrocyclones. The cyclone overflow reports to a sump and is pumped to the first set of Cleaners with 5 off Metso RCS5 cells (4 + 1), each of 5m^3 capacity, for a residence time of 20 minutes. Cleaner 1 concentrate is cleaned in the second set of cleaners, consisting of 3 off (1 + 2) Metso RCS5 cells, each of 5m3 capacity for a residence time of 20 minutes. Cleaner 2 concentrate is further cleaned in the final and third set of cleaners, consisting of 2 off Metso RCS5 cells, each of 5m^3 capacity for a residence time of 20 minutes.

The final Cleaner 3 lead concentrate is sampled prior to thickening (one thickener) and filtration with an automated Larox filter. The final moisture content of the lead concentrate is typically 6 to 7% w/w. Lead concentrate thickener overflow is recirculated to the lead flotation circuit.

The Cleaner 1 tails are pumped back to the grinding circuit with the Rougher Scavenger concentrate. Cleaner 2 tails are pumped back to the Cleaner 1 feed. Cleaner 3 tails are pumped to the Cleaner 2 feed.

Additional lime is added to all three cleaner stages.

16.1.3.2 Zinc Flotation Circuit

Zinc conditioning is achieved in two conditioning tanks. Copper sulphate (as the zinc activator) and lime (for pH control) are added to the first tank, with KAX and Dowfroth added to the second tank. The zinc flotation circuit comprises zinc rougher and scavenger flotation and three stages of zinc concentrate cleaning.

Initial flotation is undertaken in 7 off (4 + 3) Metso RCS20 Rougher flotation cells, each of 20 m^3 capacity, for a residence time of 15 minutes. Additional KAX and Dowfroth reagents are added. The rougher tails are treated in 3 off Metso RCS20 Rougher Scavenger cells, also of 20 m^3 capacity, with a residence time of 15 minutes. Additional copper sulphate is added to the Rougher Scavengers. The Rougher Scavenger tails are pumped as final tails to each of the three main tailings disposal circuits as required (paste backfill, dry stacking and TSF-4). The Rougher Scavenger concentrate (plus the last two rougher cells concentrate) is pumped back to the first conditioning tank.

The rougher concentrate (first five cells) is pumped to the regrind circuit, consisting of an SMD mill operating in closed circuit with hydrocyclones. The cyclone overflow (circa 85% minus 38 microns) reports to a sump and is pumped to the first set of Cleaners with 3 off Metso RCS10 cells, each of 10 m^3 capacity, for a residence time of 20 minutes. Cleaner 1 concentrate is cleaned in the second set of cleaners, consisting of 4 off Metso RCS5 cells, each of 5m3 capacity for a residence time of 20 minutes. Cleaner 2 concentrate is further cleaned in the final and third set of cleaners, consisting of 3 off (2 + 1) Metso RCS5 cells, each of 5 m^3 capacity for a residence time of 20 minutes.

The final Cleaner 3 zinc concentrate is sampled prior to thickening in two parallel thickeners and subsequent filtration with an automated Metso VPA filter. The final moisture content of the zinc concentrate is typically 9 to 10% w/w. Zinc concentrate thickener overflow is recirculated to the zinc flotation circuit.

The Cleaner 1 tails are pumped back to the first conditioning tank, together with the Rougher Scavenger concentrate. Cleaner 2 tails are pumped back to the Cleaner 1 feed. Cleaner 3 tails are pumped to the Cleaner 2 feed.

Additional lime is added to the first two cleaner stages.

16.2 Reagents and Consumables

Lime is used for pH control. Sodium cyanide is used as a zinc depressant in the lead circuit and as an overall pyrite depressant. Potassium ethyl xanthate and EuroSKIK, a lead specific collector, are used as the lead collectors. Potassium amyl xanthate is used as the zinc collector. Zinc sulphate is also used as a zinc depressant in the lead circuit and copper sulphate used as the zinc activator in the zinc circuit. Nasfroth 240 is used as the frother throughout. No flocculants are used in the concentrate dewatering circuits.

Reagents are stored and mixed in a reagent building annexed to the concentrator. Each reagent system comprises a mix tank, a storage tank and a recirculating pump and head tank for distribution. The reagent mixing tanks are ventilated.

Sodium cyanide is stored, mixed and held as a solution in a separate locked area adjacent to the main reagent mixing area.

Lime for pH control is received as bulk hydrated lime powder and is stored in a silo and mixed to a slurry in an agitated mixing tank.

16.3 Historical Performance

The historical performance of the processing plant since 2008 is shown in Table 16-1.

Table 16-1: Historical Plant Performance Data Since 2008

Item	Unit	2008	2009	2010	2011	2012	2013	2014	2015	2016
Processing Plant										
Ore Processed	kt	854	865	811	758	754	774	780	777	779
Pb Grade	% Pb	4.68	4.43	4.05	3.83	3.93	4.13	4.16	4.04	3.95
Zn Grade	% Zn	4.08	4.14	3.81	3.43	3.35	3.47	3.48	3.52	3.41
Lead Concentrate										
Pb Concentrate	kt	49.1	47.6	41.3	37.1	38.0	41.0	41.6	40.2	39.5
Pb Recovery	%	91.6	94.2	94.4	95.1	94.4	94.4	94.5	94.1	94.1
Pb Grade	% Pb	74.5	75.7	75.2	74.3	73.6	73.6	73.7	73.5	73.3
Pb in Concentrate	kt	36.6	36.1	31.0	27.6	28.0	30.2	30.7	29.5	29.0
Zinc Concentrate										
Zn Concentrate	kt	58.0	61.0	52.8	44.6	43.1	46.2	46.9	47.2	45.5
Zn Recovery	%	82.3	85.5	86.0	86.6	86.2	86.3	86.5	85.8	84.6
Zn Grade	% Zn	49.5	50.2	50.4	50.6	50.5	50.1	50.1	49.8	49.4
Zn in Concentrate	kt	28.7	30.6	26.6	22.5	21.8	23.3	23.5	23.5	22.5
Item	**Unit**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**
Processing Plant										
Ore Processed	kt	793	805	821	820	831	807	806	761	799
Pb Grade	% Pb	3.98	3.90	3.77	3.85	3.52	3.63	3.70	3.71	3.35
Zn Grade	% Zn	3.18	3.31	3.29	3.37	3.14	3.15	2.97	2.87	2.61
Lead Concentrate										
Pb Concentrate	kt	40.8	40.3	40.4	41.3	37.9	38.4	39.1	37.6	35.6
Pb Recovery	%	94.6	93.6	94.5	94.3	93.1	93.4	93.1	94.4	94.1
Pb Grade	% Pb	72.9	73.6	72.3	72.0	71.8	71.2	71.0	70.8	70.6
Pb in Concentrate	kt	29.9	29.4	29.2	29.7	27.2	27.4	27.8	26.6	25.2
Zinc Concentrate										
Zn Concentrate	kt	43.7	46.1	47.1	47.6	44.4	42.8	40.2	37.0	35.6
Zn Recovery	%	85.5	84.6	86.5	86.1	84.9	84.6	85.0	85.2	85.7
Zn Grade	% Zn	49.4	48.9	49.6	50.0	49.9	50.1	50.6	50.2	50.2
Zn in Concentrate	kt	21.6	22.5	23.4	23.8	22.2	21.5	20.3	18.6	17.9

For the period 2008 to 2016, the average throughput was 794.8 ktpa. The average lead grade was 4.13% Pb and recovery was 94.1% to a concentrate grading 74.16% Pb. The average zinc grade was 3.63% Zn, and recovery was 85.5% to a concentrate grading 50.1% Zn.

In recent years, there has been a decline in both lead and zinc head grades. However, despite this, metals recoveries and concentrate grades have remained at design levels and are very consistent, indicating excellent and consistent metallurgy as the mine deepens.

In 2008 and 2009, the plant operated at its nameplate capacity of 850 ktpa, but from 2010 to 2016 the feed tonnage was reduced to match the mine output. In 2017 to 2021, feed tonnes increased from circa 800 ktpa to 830 ktpa but, in recent years, has reduced again to circa 800 ktpa. The LOM plan is based on 830ktpa.

The silver recovery to the lead concentrate for the LOM plan is based on 75% silver recovery to the lead concentrate at a grade of 254 g/t Ag.

16.4 Forecast Performance

Metallurgical recoveries used in the LOM plan from 2026 to 2034 are 94% for lead, 86% for zinc and 75% for silver. The total LOM tonnes treated are 7.327 Mt at 3.48% Pb, 2.50% Zn and 26 g/t Ag. Average throughput is 830 ktpa for the period 2027 to 2033, with 815 ktpa in 2026 and 705 ktpa in 2034, respectively.

16.5 Personnel

The plant personnel in the processing department includes 18 in crushing, 41 in the main grinding and flotation plant and 13 in tailings, for a total complement of 72. The plant operates an 8-hour shift pattern. Maintenance is included under a separate department that services the whole mine site.

16.6 CP Opinion

The authors consider the Sasa Mine has sufficient supply of energy and water for the forecast throughputs for the processing plant. The sourcing of process consumables is well established with recognised suppliers and logistics.

17.0 Project Infrastructure

Infrastructure associated with the Sasa Mine includes the Svinja Reka underground mine, processing plant and associated infrastructure, paste backfill plant, dry stack plant, dry stack storage facility, and tailings storage facility.

There is a well-maintained network of paved highways, telecommunications facilities, national grid electricity, and sufficient water supply.

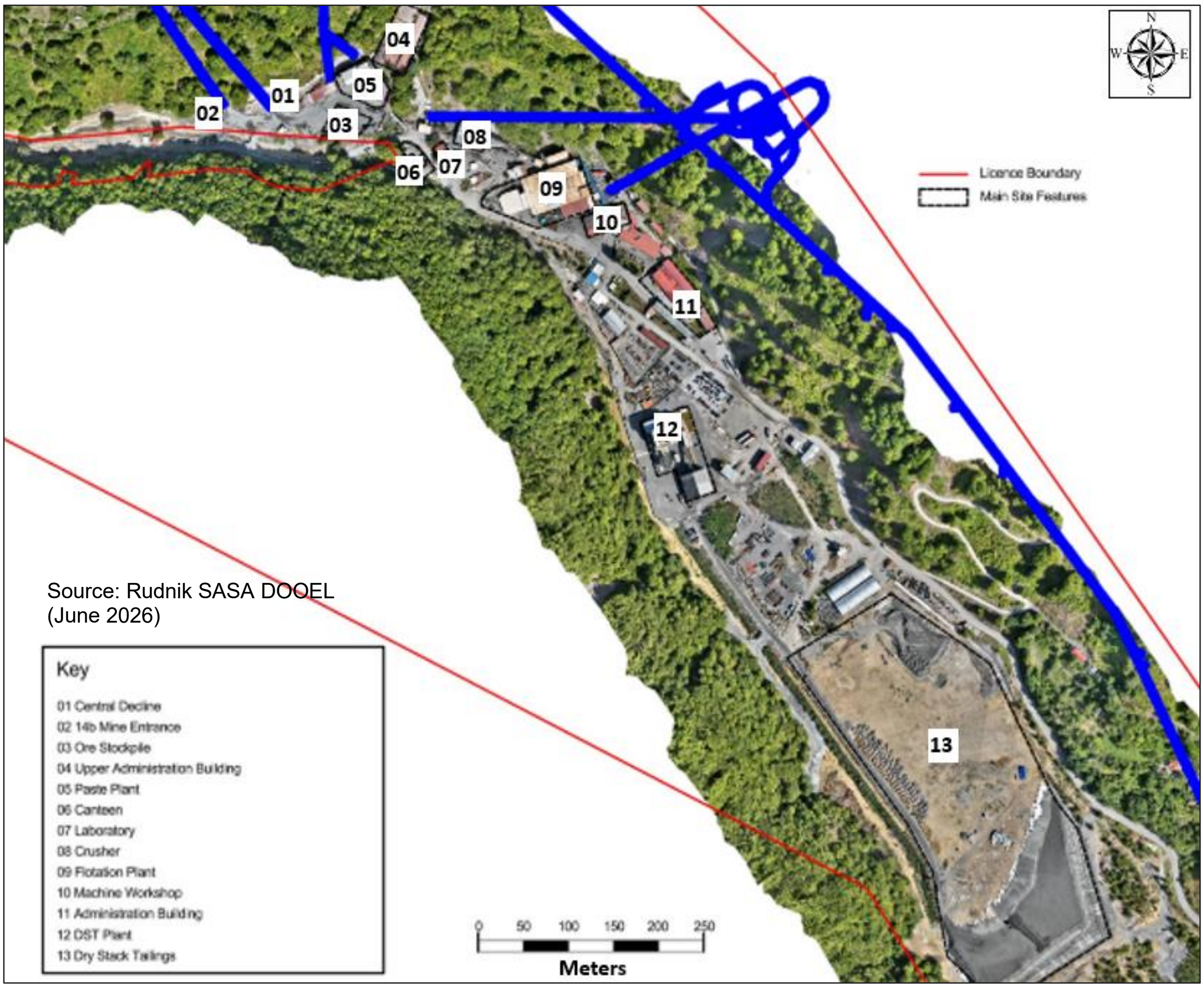
17.1 Site Layout

The site layout is shown in Figure 17-1 and summarised in Table 17-1. The numbers assigned to infrastructure items in Table 17-1 correspond to those shown in Figure 17-1.

Table 17-1: Sasa Mine Key Infrastructure

Infrastructure Item		
Central Decline (01)	Canteen (06)	Administration building (11)
14b mine portal and decline (02)	Laboratory (07)	DST plant (12)
Ore stockpiles (03)	Crusher (08)	Dry stack tailings facility (13)
Upper administration building (04)	Flotation plant (09)	Tailings Storage Facility (located south of Item 13)
Paste plant (05)	Workshop (10)	

Figure 17-1: Layout of the Sasa Mine



Source: Rudnik SASA DOOEL (June 2026)

17.2 Power

The high voltage (HV) power from the state grid to the underground mine is fed at 35 kV to three transformers where the voltage is stepped down to 6 kV with a capacity of 4 MVA. Extensions to the HV circuit are typically required every vertical 50 m in the mine where 250 kVA transformers step down to a working low voltage (LV) of 380 V in the underground mine. HV cables are typically routed down through sub-vertical ventilation and escapeway accesses to reduce cable lengths (reducing costs and voltage drop).

The main requirement for underground power in the mine is for electric drills, ventilation fans (primary and auxiliary) and the rail system on 830 mRL.

17.3 Water

The Sasa Mine operations are situated within the Kamenica River watershed. The Kamenica River runs from northwest to southeast. Two smaller creeks connect to the Kamenica River upstream of the current mine operations, the Svinja and Crvena. Both contain legacy mine workings. The Kozje River connects to the Kamenica River at the area of the mining facilities.

The process water intake structure is situated upstream and sourced mainly from the Crvena and Svinja creeks and the Kozje River when required and consists of a concrete storage tank which fills by gravity from an overflow along the Kamenica River. The intake pipeline collects water from the storage tank and flows by gravity to the process plant, generating sufficient pressure required for the flotation plant. Mine water (and potentially return DST water) are used for the grinding plant.

Outflows from the mine are collected in a collection ditch and tunnel which can discharge to the TSF pond or to the process plant as required.

Downstream of the confluences of the Kozje and Svinja Rivers, the Kamenica River is captured in a concrete diversion tunnel, constructed beneath TSF-1, TSF-2, TSF-3.1 and exits through the western abutment of TSF-4.

A diversion channel also exists on the eastern side for the Petrova River which discharges to the Kamenica River downstream of the western diversion tunnel outlet located at the toe of TSF-4.

Discharge of the supernatant solution in TSF-4 is achieved by pumping to the settling ponds downstream of TSF-4. Sprinklers are used on the downstream dam wall face to control dust. Supernatant water is not currently recycled back to the plant.

Seepage collection systems for TSF-4, and any remaining seepage from TSF-3.1 and TSF-3.2, discharge downstream of TSF-4 into a collection tank which discharges to the Kamenica River downstream of the diversion tunnel outlet.

Mine Adit 830 is located downstream of the diversion tunnel outlet and discharges to the Kamenica River.

Storm run-off from the plant site is discharged into the Kamenica River upstream of the river diversion tunnel.

Continuous water quality, flow rate measurement and monitoring of discharges are undertaken as required by permitting. Groundwater monitoring began in 2016.

Drinking water is abstracted from Toplici well (Zanofito Spring), located at Mount Ruen. Water is stored in a 37 m^3 water storage tank where it is chlorinated. The Sasa Mine is dewatered primarily by gravity drainage to two levels; Level 1,065 mRL and Level 830 mRL. Pumps are installed to move water locally within the adits as required, but significant pumping capacity is not required. In general, mine dewatering is not considered a high-risk item for the mine operation, although pumping infrastructure must be maintained to keep the underground workings dewatered.

A water balance has been generated and is continuously updated. There is a focus on recycling water, particularly the supernatant water, although this is not currently re-used in the plant, which uses fresh (river) water for the flotation circuit and mine water (with potentially DST return water) for the grinding circuit. There is a general strategy to minimise the use of fresh water where possible and recycle as much as possible. The water balance plan continuously evaluates the monthly and annual variability of inflows and outflows, specifically related to rainfall and runoff contributions.

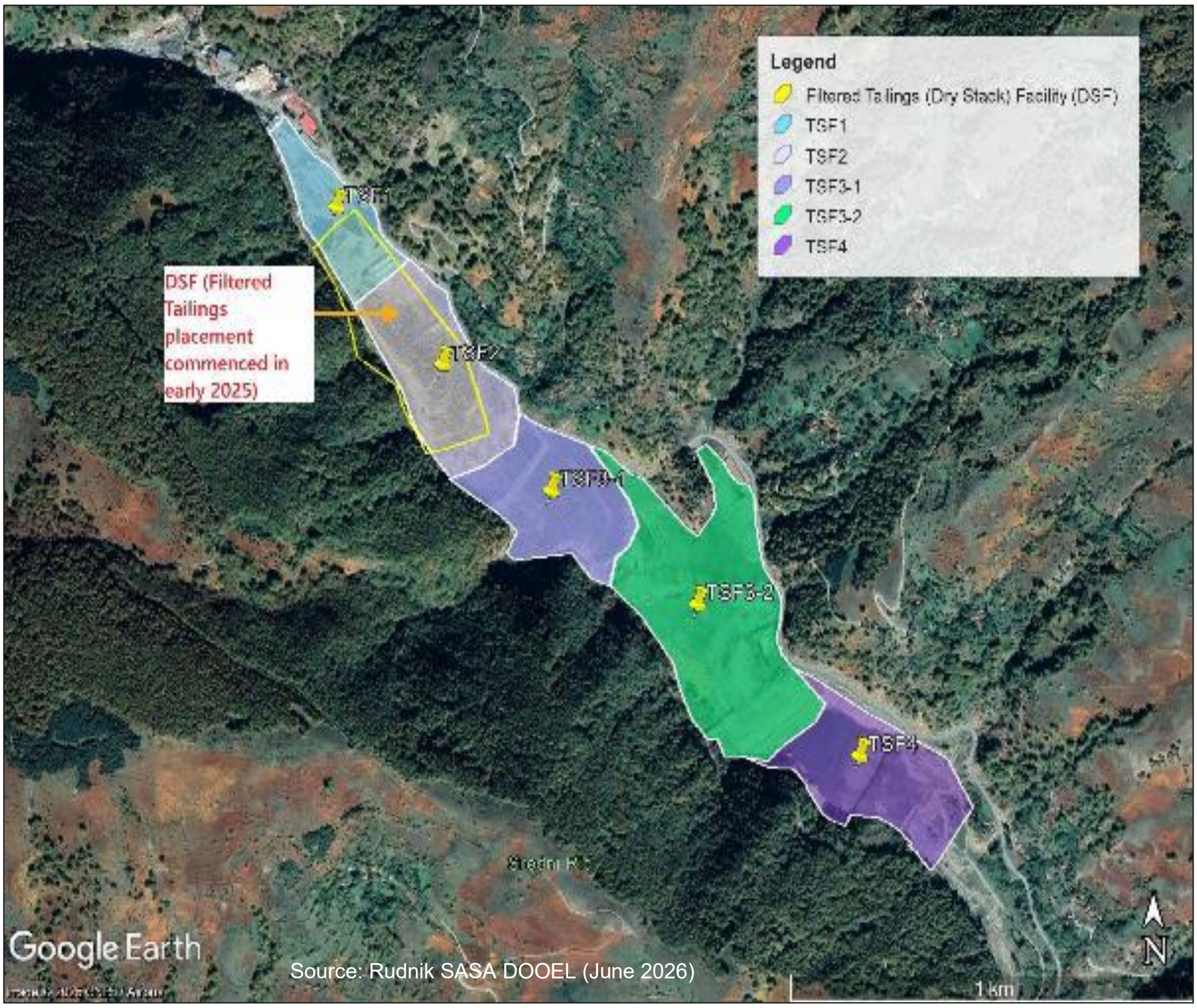
17.4 Tailings Storage Facility

The Sasa TSFs have been operational since the 1960s, with successive TSF development progressing downstream from TSF-1, TSF-2, TSF-3.1, TSF-3.2 to TSF-4 in the steep-sided valley of the Kamenica River, as shown in Figure 17-2.

The Dry Stack Tailings (DST) plant was commissioned in 2025. Filtered tailings from the plant are stored on top of TSF-2, and partially TSF-1.

TSF-4 is the only TSF required by the operation for storing wet tailings for the life of mine. TSF-1 and TSF-3.1 have been rehabilitated with soil cover and vegetation, and TSF 3.2 is in the closure phase and has been capped with waste rock.

Figure 17-2: Sasa Mine TSF Outline



Source: Rudnik SASA DOOEL (June 2026)

The Kamenica River is carried below the TSFs within an engineered river diversion structure that exits through the western rock abutment of TSF-4. The entrance portal is located at the northern end of TSF-1. There is also a channel diversion of the Petrova stream along the eastern side of the valley. Catchment surface water is also directed to the diversion structure as well as any remaining seepage solution from the previous TSFs.

The river diversion structure consists of approximately 40% within culverts routed under the TSFs and the remaining 60% as tunnel sections within the in-situ rock.

A Tunnel Integrity Assessment on the entire length of the diversion tunnel was carried out in 2017. Remediation works to parts of the tunnel floor were carried out in 2019. In 2025, a further assessment of the tunnel was undertaken by the Faculty of Civil Engineering of Skopje University and included recommendations for additional remediation works.

TSF-4 was constructed with an HDPE liner whereas the previous TSFs are unlined. The TSF was designed by the Faculty of Engineering, Skopje, in March 2015 to international standards.

Progressive development of TSF-4 (after construction of the initial granular rock fill toe buttress) comprises downstream raising using tailings fed to cyclones (three are currently located on the embankment), with the coarse underflow forming the embankment and downstream construction, and the fine overflow deposited within the dam, where the supernatant solution separates out from the sand which forms a beach from the embankment.

Although not currently required by the operation, an overflow concrete collector pipe is constructed within the dam on the western side and connects with the river diversion structure.

Water from the DST is directed to TSF-4.

Supernatant water is pumped via a pontoon equipped with two 37 kW pumps (one operating, one standby) to the temporary sedimentation ponds downstream of TSF-4 and then discharged to the natural reservoir of the river. Supernatant water is not currently re-used in the process plant.

A minimum freeboard of 2.0 m is maintained from the crest of the embankment. Each raise of the embankment is 1.25 m in height.

The crest is at elevation 944.3 masl with a final design crest elevation of 952 masl to encompass the LOM plan. The supernatant pond is at elevation 942.3 masl.

In 2025, a total of 272,787 t of tailings was deposited in TSF-4 representing 37.1% of the total tailings (by tonnage) produced by the operation.

Tailings management follows the guidelines of the Global Industry Standard on Tailings Management (GISTM). External reviews are carried out annually by Knight Piésold Ltd (Knight Piésold). The latest report (Knight Piésold, October 2, 2025) including risk assessment, concludes that the Sasa Mine TSFs are generally compliant with the minimum stability criteria adopted for GISTM requirements but list several actions to be undertaken, most critically, a raise of the waste rock toe berm and installation of additional piezometers at TSF4. Knight Piésold state that, subject to completion of the actions, TSF4 can continue to operate without an unacceptable risk to public safety or the environment.

Rudnik SASA DOOEL has since initiated a 6.5 m raise of the waste rock berm from 894.5 masl to 901.0 masl. In addition, three further standpipe piezometers have been installed at TSF-4 to monitor phreatic surface conditions beneath and downstream of the TSF-4 embankment.

17.5 Dry Stack Tailings Plant

The DST plant was commissioned in April 2025, and filtered tailings from the plant are stored at the DST facility on top of TSF-2 and part of TSF-1. The plant equipment is entirely sourced from Metso. A geotextile membrane is included as a liner on the boundary between the old tailings and the stacked tailings of the first lift.

The process plant tails are thickened in a 14.0 m diameter High-Rate Thickener, the underflow is filtered in a fully automatic Metso VPA filter press operating at circa 8-10 bar. The filtered solids at circa 11-12% moisture are stored in a building and loaded via a FEL into 35 t trucks for transportation over the short distance to the DST facility.

The tailings are stacked in 5.0 m lifts, with the third lift currently being stacked. The tailings are spread out with a dozer in sub-layers of 25-30 cm and compacted, with geotechnical testing carried out on each sub-layer, as well as additional tests for each lift. In general, the optimum moisture content of circa 11.5% is achieved after filtration but water is added when required if the tailings become overly dry.

Trucking, loading, dozing and compacting are all carried out by contractor.

The water from the thickener overflow is directed to TSF-4, although studies are being undertaken to evaluate recycling it back to the process plant to minimise the use of fresh water.

During wet weather, dry stacking is stopped and more tailings report to TSF-4, in addition to the paste backfill requirement.

A batching plant used to produce shotcrete is also located at the DST.

In 2025, 260,549t of solids were dry stacked, representing 35.4% of the total tails.

The DST LOM design is for 3.3 Mt of tailings.

17.6 Paste Backfill Plant

The paste backfill plant project was started in September 2022, and the first paste was delivered underground in August 2023. The maximum distance pumped to-date is circa 2.8 km down to the 750 level. The plant was designed by Patterson & Cooke, and construction was undertaken by Aktiva Engineering, a Macedonian company. The plant equipment is sourced from various international suppliers.

Circa 36% of the process plant tailings is targeted for paste backfill in 2026. After thickening in a dedicated high-rate thickener, the thickener underflow at circa 60% solids density is filtered in a Bokela Vacuum Disk Filter. The filter cake reports to a mechanical breaker to which cement and water is added in a mixer to produce the required paste. This is transferred via a centrifugal pump to the main ABEL diaphragm pump which pumps the paste underground via a thermally traced pipeline.

Cement addition is generally in the range of 3.0 - 5.4%. Cement is stored in two silos, each of circa 380 t capacity.

The same operator crew (2-3 per shift) also operates the Dry Stack Facility.

In 2025, 202,496 t of solid tailings as paste was delivered underground, representing 27.5% of the total plant tailings.

Paste backfill remains the priority method for tailings disposal.

17.7 Transportation of Concentrates

Lead and zinc concentrates are stored in separate areas and loaded into road trucks by FEL. Each truck load is sampled before shipping. The trucks are weighed empty and loaded for accounting.

18.0 Market Studies and Contracts

Both lead and zinc concentrates, are sold via an annual rolling hybrid offtake contract with Traxys Europe S.A. (Traxys), to the regional and international well diversified customer base.

The commercial terms of the contracts with final customers are negotiated on an annual basis based on the prevailing market conditions with the off taker assistance.

The payable silver contained in the lead concentrate is subject to a life of mine streaming agreement with OR Resources Inc. With the assistance of the broker, on a monthly basis CAML buys and delivers the physical silver to OR Resources Inc., which is the equivalent of the payable silver sold during the calendar month to final customers. CAML receives US$6.665 per ounce for silver.

CAML undertakes a marketing process on its concentrates together with Traxys on an annual basis, which guarantees market terms received.

19.0 Environmental Studies, Permitting, and Social or Community Impact

19.1 Permitting

North Macedonia is not an EU Member State but has fully transposed the EU IPPC Directive (2008/1/EC) into its national legal framework, including the differentiation between "A" (large/mid-size) and "B" (smaller) installations. Sasa Mine is permitted as an "A" installation due to its scale and operation in the mining sector. This permit includes provisions for the monitoring of any fugitive or planned emissions to the environment, including to water, air or soils. Any changes to the mining method or process that may have an impact on human health, the environment or property must be approved by the government prior to the changes being enacted and must be accompanied by relevant updates to the IPPC permit.

In 2024, the IPPC permit was amended to cater for a change to dry stack tailings and paste backfill. To support this application an ESIA was produced by in-country consultants in accordance with the legal requirements, including additional baseline studies.

The new IPPC permit was issued in February 2024 and is valid for seven years.

Water abstraction is covered by a number of permissions. Rudnik SASA DOOEL also holds permissions to divert the Petrova River away from TSF 3.2 and divert the Kamenica River through the diversion channel.

Table 19-1: Sasa Mine Environmental and Special Water use Permits

Permit type	Validity dates	Permit number
IPPC Environmental Permit	2024 - 2031	UP 1-11/3-2549/2022
Water abstraction permit for the Crvena River	2021 - 2027	UP 1-11/5-1401/2021
Water abstraction permit from Kozje River and Zanofito Spring	2024 - 2034	UP 1-11/5-1324/2023

The site also holds the agreements and permits related to the physical exploitation of the metal resources, namely:

- Mining Operations Permit/Approval of Mining Project, which approves the underground mining method, production plans and safety systems; and
- Concession Agreement, which sets the concession area, the duration of the permission, and any royalties or financial obligations.

Further details on the concession are provided in Section 3.

19.2 Environmental Liabilities

On August 30, 2003, approximately 150,000m^3 of tailings leaked from TSF3-1 via a diversion tunnel and into the Kamenica River during a period when the mine was not operational and prior to CAMLs ownership. The cause of the failure was a defective cap within the roof of the ancillary pipe carrying TSF 3.1 drainage into the underlying river diversion structure (i.e. not a direct failure of the river diversion structure). A clean up of the tailings that were released into the river and a full repair of the diversion tunnel was performed according to a design approved by the regulatory authorities. All activities were conducted by independent contractors under the supervision of the regulatory authorities.

A legal review was carried out by Georgi Dimitrov Attorneys (April 2016) assessing the risk and potential obligations arising from historic contamination. It was concluded in this review that under Macedonian law there is no liability sitting with the current owner to clean up downstream of the site any residual contamination with respect of the 2003 incident. In addition, it was further concluded that “Rudnik SASA DOOEL can only be liable for the environmental damage occurred following the moment the company began to operate the mine”.

On September 14, 2020, approximately 3,000 m^3 of tailings, supernatant water and suspended sediment, spilled from TSF4 via a breach in the dam embankment and flowed for approximately 11 km along the Kamenica River to the Kalimanci reservoir. The initial leakage of water from the crest of the dam became progressively worse, eroding part of the embankment and allowing the upper layer of tailings (‘slimes’) to subside through an opening with supernatant water draining afterwards. The leakage subsided after approximately two hours, once a portion of the ponded supernatant water had drained out and the remaining tailings recompacted and reconstituted. Sasa commissioned specialist consultants to support with the remediation actions, which were undertaken in line with State Inspectorate and Global Industry Standard on Tailings Management (GISTM) requirements. An independent technical review of the remediation process was commissioned in January 2021 and concluded that the remediation was executed in a thorough and diligent manner.

The authors are not aware of any other environmental liabilities on the Property.

19.3 EHS Management

19.3.1 Policies and Practice

CAML has a series of policies, applicable to all group companies, available on its website. These relate to Anti-bribery, Whistleblowing, Code of Conduct, Modern Slavery Act, Supplier Code of Conduct, Human Rights, Tailings, People, Health & Safety, Environment, Community and Climate Change. These documents are dated and state that the policies have been approved by CAML’s Board of Directors.

Since 2007, the mine has had an ISO14001 certified Environmental Management System (EMS) as part of its Integrated Management System (ISO9001). Rudnik SASA DOOEL has a team of three environmental specialists, including the Environmental Manager, and two environmental engineers who are responsible for completing, recording and reporting on all environmental aspects associated with the mine. This includes monitoring requirements related to the IPPC permit, abstraction and discharge licences, as well as internal monitoring requirements. The Environmental Manager is responsible for ensuring that all monitoring is completed and accurately reported to state authorities in accordance with legal and permit requirements, and ensuring that accurate records are kept, management systems are updated and maintained in line with ISO requirements and providing information or evidence for ISO audits of the Sasa Mine.

There is a suite of management plans in place for various environmental aspects of site operations, which are coordinated under the EMS to ensure that regular updates are made to keep the documents current.

19.3.2 Security

The site is fenced and gated to prevent unauthorised access. Access to the site is via a public road from Makedonska Kamenica, which continues after the site, as an unmade track, into the mountains above. Public access through the site is permitted but users are limited to the residents of the few houses located above the mine. The hazardous waste storage compound on site is fenced and locked.

19.3.3 Emergency Preparedness

An Emergency Preparedness and Response Plan (EPRP) is in place, accessible to staff via QR codes across the site. The document includes a Triggered Action Response Plan (TARP) and contact details for key people and organisations to inform of an incident. Radios are in place in areas where incident coordination rooms would be established.

The EPRP is regularly updated, and at the time of SLRs site visit, the emergency contact list was being updated to ensure that all contact numbers were valid and correct for the relevant roles of emergency response processes.

Theoretical training is provided to all staff, as well as desktop and practical exercises being run for those in supervisory or managerial roles.

19.3.4 Staff Training and Supervision

All staff are required to undergo induction training prior to starting work on the site. This includes aspects of environmental and social importance, as well as health and safety, and wider role-specific topics.

Regular refresher training is provided to all staff, and those with responsibilities for operations which could have adverse impacts on people or the environment, are provided with role or task specific training to control risks as far as possible.

19.4 Internal and External Stakeholder Dialogue

19.4.1 Internal Employee Consultation Practice

Employees are considered integral stakeholders to the Sasa Mine, and as such a dedicated internal communications plan has been in place since 2018, which explicitly links to the external communications plan, ensuring that consistent information is presented to employees and the community. Approximately 80% of the employees are based in the local communities. The purpose of the internal communications plan is to ensure employees are informed on upcoming changes to the mine, new initiatives (both external and internal), and have the opportunity to ask questions. There is active encouragement for employees to contact their line manager with any concerns, but also an anonymous grievance mechanism to provide anonymity should this be required.

19.4.2 External Stakeholder Dialogue

In 2018, Rudnik SASA DOOEL developed a Stakeholder Engagement Plan and structured community engagement programme, including establishing the Sasa Foundation, to ensure that appropriate governance is in place for administering financial support to local organisations.

A formal grievance mechanism is in place, enabling communities to raise questions or concerns. These are recorded internally, and responses documented and fed back to the enquirer.

19.5 Environmental Monitoring

In 2022, CAML introduced a software-based system for storing monitoring data. Since the introduction, all monitoring data has been manually uploaded into the system, and historic records also added to allow trend analysis to be undertaken. The software contains monitoring data for water, air, dust, weather, waste tonnages, and TSF levels and discharge volumes.

The monitoring required by the IPPC permit includes a programme of weekly, monthly and quarterly sampling for water, as well as noise monitoring at the site boundary and in communities. Rudnik SASA DOOEL undertakes additional monitoring, above what is required by the IPPC permit, which includes meteorological conditions, ambient air quality and PM_{10}. There are point source emissions to water specified in the IPPC permit, but no point source emissions to air are included. There are however legal requirements relating to ambient dust limits and both legal and IPPC Permit requirements for mitigation measures.

Weekly testing of discharge water quality is required under the permit. Parameters include physio-chemical aspects such as pH, suspended solids, as well as organic pollution indicators such as biochemical oxygen demand and chemical oxygen demand. Nutrients such as nitrogen, nitrate, and phosphorus are included in the analytical suite, along with hazardous substances such as heavy metals and cyanides. Water samples are tested at an external accredited laboratory, and do not show any exceedances of permit parameters, with exception of increased manganese in the TSF drainages.

There are no point source emissions to air covered under the IPPC permit, however nuisance dust monitoring is required, and is in place, in addition to proactive management actions. Sprinklers are active on the TSF wall to control dust, as the wall is constructed from tailings material. Generally, there are few complaints relating to dust, and these only occur during periods of extreme weather. Regular observations of nuisance dust are undertaken at the site boundary and nearest residential receptors during periods of dry weather to ensure that management actions are effective.

Noise monitoring results have been recorded online via a government system since the start of 2026, prior to which paper copy reports were issued to the government instead. There are no reported non-compliances or exceedances relating to noise.

Nationally there are no laws relating to soils in North Macedonia, and the IPPC permit only requires soil sampling every 10 years. Rudnik SASA DOOEL undertakes quarterly soil monitoring and has observed no changes from baseline conditions in recent data. It is noted that soils within the mine area itself have been subject to historical industrial (mining) use and therefore are not a reflection on baseline conditions in the wider area.

The movement of project-related vehicles through the local area is monitored by Rudnik SASA DOOEL, by information requested from the contractors. Information such as driver details, routes and driving speed records are requested and reviewed by Rudnik SASA DOOEL on a regular basis.

19.6 Raw Materials, Products and Waste Streams

19.6.1 Raw Materials

The flotation process uses standard reagents for lead and zinc flotation which are stored and mixed in a reagent building annexed to the processing plant. The reagent building is appropriately bunded with contained drainage.

Sodium cyanide is stored, mixed and held as a solution in a separate locked area adjacent to the main reagent mixing area. This area is also bunded with contained drainage to ensure the containment of cyanide in the event of an incident.

19.6.2 Explosives

Explosives are brought to the site under police escort and stored in a secure compound.

19.6.3 Energy

The site purchases electricity from renewable sources and has “Guarantees of Origin” certificates for this. The mine has had, since 2018, a Greenhouse Gas Inventory in place, for scope 1 and 2 emissions, which is updated annually and provided to CAML for publication.

19.6.4 Effluent

A biological sewage treatment plant is in place at the mine, which treats all site sewage prior to discharge of treated effluent to the river. All discharge points are monitored in line with the IPPC permit for the site and no exceedances have been reported.

19.6.5 Air Emissions

There are no point sources of emission covered by the permit; however, the TSF and dry stack are the main potential sources of fugitive emissions, or nuisance dust. Other sources such as site traffic or site roadways could also cause issues, but to a lesser extent. The proactive management measures in place keep the dust emissions under control during normal weather conditions however in periods of prolonged dry weather complaints from residents near the TSF are sometimes received regarding dust.

19.6.6 Noise

Mine activities do not significantly contribute to the low ambient levels, due to mining operations being underground, and receptors being located at a distance from noise-generating surface activities such as crushing. Workers are provided with appropriate personal protective equipment (PPE).

19.6.7 Waste

Non-mining waste tonnages and final disposal destinations are reported annually to the government via an online portal. National legislation requires waste transport and disposal companies to be appropriately licenced, and all contractors hold the relevant permits for the work carried out. Domestic waste on site is well segregated, with recycling facilities available throughout. The hazardous waste storage compound is fenced and locked, with a concrete base and contained drainage. There is a three-year waste improvement programme in place, through which waste reduction and recycling improvements are being driven. Reverse vending machines for plastic bottles have been installed on site and in Makedonska Kamenica to promote recycling.

19.7 Biodiversity Management

The site is located within the Osogovo Mountains, areas of which are designated as a National Protected Landscape, Important Bird Area, Important Plant Area and part of the Emerald Network, however it should be noted that the Sasa Mine is located outside of any designated areas.

There is a Biodiversity Management Plan in place to ensure that the site is managed and rehabilitated with consideration to the surrounding area.

19.8 Water Management

The Kamenica River flows through the site and passes under the TSF via a specifically designed culvert. There is a secondary river, the Petrova River, which joins the Kamenica River at the toe of TSF3.2 and is separated from the Project by a diversion channel.

The Kamenica River is the main source of non-potable water for the mine, supplemented by a secondary river, both of which begin upstream of the mine. The water is settled in pools then piped to the mine buildings. A small spring is used to supplement water into the thickener during periods of dry weather, however in general there is sufficient recirculation of process water to prevent the need for additional water inputs.

Spring water is collected and stored in 2 tanks prior to passing through a chlorination unit, which then supplies potable water to the site. Approximately 6% of potable water produced is piped into the underground workings for use by the workforce in the mine.

A water management plan is in place, including flow diagrams to show water recirculation routes, and is updated annually. The ESIA undertaken in 2022 as part of the move to paste backfill, included a study on the effects of the paste backfill on the groundwater in the area, both during operations and post-closure.

The relevant permits and licences for water extraction and use are in place, which detail the permitted water take from each extraction point.

The underground workings are dewatered by two main pumping stations located at the 830 level with the total capacity of 70 l/s, and water is pumped from there to a settling pool on the surface where 45 - 50% of the water is used in the processing activities. Additionally, the water from 990 level and horizon XIVb is dewatered by two additional pumps with a total capacity of 30 l/s into a settling pool at the entrance of XIVb, and from there it is connected to a pipeline to the tailings pond.

The road forms a watershed barrier between the eastern mountain slopes and the site, preventing surface water runoff entering the mine site. Site runoff is collected and directed to the western edge of the site where it is collected and flows in a dedicated channel to TSF 4. A new diversion channel is being created around the western edge of the planned DST extension, which will contain the runoff from the western mountain slopes. This diversion will run along the edge of the site and then discharge into the Kamenica River as it emerges from the toe end of TSF 4.

Flood water management has also been considered in the site design. The processing plant is designed to contain the contact flood water, with surface runoff being diverted via roads. As part of the DST expansion, new stormwater drainage will be constructed on the western side of the site, opposite the processing plant.

19.9 Tailings Management

To mitigate risks to the environment and local communities downstream of the TSF an ongoing monitoring programme is in place. The programme is aligned with GISTM requirements and national requirements for structures classed as dams of ‘significant importance’. External reviews of tailings management is carried out annually by Knight Piésold.

Prior to commencing paste backfill and DST operations, all tailings from the mine were stored as wet tailings in the various phases of the TSF. TSF4 is the only phase currently used for storage of wet tailings, whilst the others are closed or non-operational.

A summary of the status of the TSF phases is shown in Table 19-2.

Table 19-2: TSF Phases

TSF Phase	Status
TSF1	Closed and capped, under new Dry Stack Tailings (DST) Plant.
TSF2	Closed and capped, in use for DST deposition.
TSF3.1	Closed and capped, revegetated with trees.
TSF3.2	Undergoing closure works - closed and capped, not yet revegetated (will be revegetated with low-growing vegetation), expected to conclude in 2029. Forms part of site’s extreme flood management area for the Petrova River.
TSF4	Active phase, designed to have 2m freeboard.

An ongoing programme of slope measurements, seismic monitoring, water quality, evaporation, flow rate, geotechnical stability and piezometric data is in place for the active TSF. The data is regularly reviewed to detect any changes, as well as reported annually to the authorities. The information is also included in the CAML annual report.

Knight Piésold has been engaged to undertake annual reviews in accordance with GISTM standards to review the construction and monitoring of the dam. The latest report (Knight Piésold, October 2, 2025) concludes that the Sasa Mine TSFs are generally compliant with the minimum stability criteria adopted for GISTM requirements but list several actions to be undertaken, including the raising of the waste rock toe berm and installation of additional piezometers at TSF4. Knight Piésold state that, subject to completion of the actions, TSF4 can continue to operate without an unacceptable risk to public safety or the environment.

Rudnik SASA DOOEL has since initiated a 6.5 m raise of the waste rock berm from 894.5 masl to 901.0 masl. In addition, three further standpipe piezometers have been installed at TSF-4 to monitor phreatic surface conditions beneath and downstream of the TSF-4 embankment.

Both the Kamenica River and Petrova River have been diverted through dedicated channels to ensure there is no contact between the river water and any site-related water, including tailings. The under-drainage from the TSF goes to a discharge point on the Kamenica River, which is monitored as part of the IPPC permit.

Rudnik SASA DOOEL has also installed an early warning system, as part of the national crisis warning system. This will allow Rudnik SASA DOOEL, through nationally managed processes, to alert the local communities to any incidents, as well as broadcasting any messages from the national crisis centre.

19.10 Geochemical Characterisation

A hydrogeological and geochemical modelling study was undertaken by SRK in 2022. Geochemical testwork showed the tailings to be potentially acid generating (PAG). However, static leach tests including denoised water leachate tests (DI) and net acid generation (NAG) tests generally indicated no leachable parameters of concern from the tailings. Monolithic leach tests (MLT) and humidity cell tests (HCT) showed the paste backfill to be geochemically less reactive than the tailings. Waste rock geochemical characterisation by Rudnik SASA DOOEL showed both the schist and gneiss waste rock to be PAG.

Water samples from the current operation at Sasa are tested at an external accredited laboratory, and do not show any exceedances of permit parameters, with exception of increased manganese in the TSF drainages. Legacy mining features (including adits) upstream of the current operation impact on water quality with elevated levels of manganese, zinc and cadmium and to a lesser extent lead and iron.

At mine closure, bulkheads are planned to be installed in the main drainage adits and will ensure the paste backfill or exposed footwall and hanging wall rock up to Adit 14b level will be submerged, thus removing the free oxygen required to generate acid rock drainage (ARD). Adit 14b, together with the shallower historical adits, will remain open and drain the mine. Passive water treatment technologies to remove metals from the post-closure water discharge were recommended for further investigation by SRK (2022). Waste rock transported from the underground mine to the surface is currently stockpiled on the TSF and is planned to be incorporated as part of the TSF closure.

19.11 Closure

A Mine Closure Plan, developed by Golder in 2022, is in place for the Project, and an update to move this to a fully integrated closure plan is underway. The IPPC Permit in place for the site describes the closure and restoration obligations for the Sasa Mine, and the financial model shows “Expansionary Capex & Closure” costs included on a monthly basis up to and beyond the current expected life of mine.

The closure status of the various TSF phases is outlined in Section 19.9. Current regulations require TSF capping to achieve impermeability of 10^{-9} m/s prior to revegetation commencing. Local species are used in the revegetation process.

19.12 Health and Safety

An ISO 18001 complaint management system is in place (currently migrating to ISO45001), with regular audits carried out to ensure compliance and recommend improvements, and in 2024 the company was awarded a National Award for Safety Systems, and positive feedback has been received via ISO audits. The company expectation is to go beyond the legal minimum safety requirements and aim to achieve the best available standards.

Visitor induction videos are available and cover both above ground and underground aspects. The videos are updated regularly and include both the dry stack and paste plants. Induction training is accompanied by a hard-copy safety guide, and visitors must complete a short quiz to check their understanding. Nine golden safety rules have been established and are shown in English and Macedonian across the site.

Drager monitors, with alarm systems built in, are in use by underground personnel. Ventilation testing is carried out regularly, and improvement works are ongoing to increase ventilation and reduce temperatures at the deeper levels of the mine.

Noise monitoring is carried out in working areas, and following a recommendation from a recent ISO audit, vibration monitoring has also commenced. Antivibration mats are in place for drillers underground.

Fire suppression systems are installed throughout the site, including underground.

All staff undergo an annual medical examination, and Rudnik SASA DOOEL support further testing or procedures if anomalies are detected. The on-site medical centre always operates 24/7 with a trained medical practitioner available. An accident reporting system is in place, with requirements for a 24hr notification report, followed by a full investigation completed within 5 days. Where longer investigation is required, the 5-day period can be extended.

All staff have the authority to stop unsafe work or practices. There is a described process for closing the area, reporting the incident or occurrence, and remediating the issues to allow safe work to continue. All incidents or issues are recorded, including the date when the issue was resolved.

A culture of continuous improvement is growing across the organisation, and there is an annual survey for all staff to gauge the attitude of the workforce to health and safety responsibilities, actions and engagement. A safety culture plan has been developed for 2026-2028, with ‘visible, felt, leadership’ being embedded. This includes managers undertaking safety walkabouts on a bi-monthly basis, and the introduction of a ‘safety ideas’ scheme.

19.13 Community Development

The small village of Sasa, and larger town of Makedonska Kamenica are the nearest settlements. The village of Sasa is approximately 4 km downstream of TSF 4, however, the nearest residential properties are approximately 0.5 km from the TSF. No resettlement is known to have been required during the mine development, nor for future mine plans.

In 2018, Sasa Mine developed a structured community engagement programme, including establishing the Sasa Foundation, to ensure that appropriate governance is in place for administering financial support to local organisations. Applications are logged electronically and the approvals from the management board are also recorded electronically. Feedback to unsuccessful applicants is provided to ensure clarity on the process is maintained. Clear governance is in place regarding the selection criteria for projects, and the value of request which can be approved by which of the boards.

Rudnik SASA DOOEL actively supports skill development and has initiated an apprenticeship programme for engineering subjects. A dedicated training centre has been established in Makedonska Kamenica, and although the mine cannot employ all the apprentices that the programme supports, the students are successful in securing work in other industries in neighbouring towns and cities. The technical leads for the apprenticeship programme are Rudnik SASA DOOEL employees, who have completed a technical verification process to be approved as trainers.

Reverse vending machines, installed on site and in Makedonska Kamenica, are used to promote recycling, and generate credits which can be used in the Makedonska Kamenica market or donated to the kindergarten. Between March 18 and April 22, 2026, the value of these credits was doubled, and the recycling rate also doubled.

The economy of Makedonska Kamenica is almost entirely reliant on the mine, as has been the case historically, with 80% of the current mine workforce living locally. Generations of families have worked in the mine, and during the period of closure in the early 2000s many of the workforce moved abroad to find work. It is openly acknowledged that the local community is heavily reliant on employment at the mine as there are minimal alternatives in the area.

In this regard, the Sasa Foundation has recently run a Business Accelerator project to support development of non-mining income streams for the town. The business and livelihood ideas were submitted by local residents and reviewed by a panel of external business experts. Of the 25 ideas submitted, 18 were chosen to receive specialist training and support to develop their ideas into detailed business plans. These business plans were reviewed by the expert panel, and 4 have been chosen to receive active support to develop into fully functioning businesses. These include the development of tourist accommodation in the town, expanding the cultivation and sale of locally grown forest fruits, and developing mountain biking trails and facilities in the mountains. Feedback on this initiative has been positive from the community, with several enquiries already received about the potential for another round of selection.

A formal grievance mechanism is in place, enabling communities to raise questions or concerns. These are recorded internally, and responses documented and fed back to the enquirer. There is a community office in Makedonska Kamenica attended every Friday (market day) by Rudnik SASA DOOEL community personnel where members of the community can discuss any issues related to the Sasa Mine.

20.0 Capital and Operating Costs

20.1 Capital Costs

A summary of the capital costs for the Sasa Mine is presented in Table 20-1. Total capital costs are estimated at US$75.7 million. A cost of US$61 million is also estimated for closure.

Table 20-1: Sasa Mine Capital Costs (US$M)

Item		2026	2027	2028	2029	2030	2031	2032	2033	2034
Mining	Underground Development	4.66	3.73	4.06	2.92	1.16	0.60	0.66	0.36	0.09
	Raise Boring	1.43	0.58	-	-	-	-	-	-	-
	Underground Mine	4.61	4.14	4.22	4.92	3.52	4.44	3.28	3.23	0.66
	Other Mining	0.18	0.10	-	-	-	-	-	-	-
Process Plant		1.64	1.77	1.56	1.52	1.16	1.24	1.14	1.08	1.16
Surface Infrastructure		0.13	0.16	0.10	0.10	0.16	0.16	0.17	0.16	0.10
Environment & Corporate, Social Responsibility		0.36	0.51	0.21	0.21	0.26	0.16	0.31	0.21	0.21
Other		2.02	3.57	0.57	-	-		-	-	-
Total Capital Costs		**15.04**	**14.56**	**10.73**	**9.67**	**6.26**	**6.60**	**5.56**	**5.04**	**2.23**

20.2 Operating Costs

A summary of the operating costs for the Sasa Mine is provided in Table 20-2. The total operating cost for the LOM is estimated at US$419.3 million.

Table 20-2: Sasa Mine Operating Costs (US$M)

Item	2026	2027	2028	2029	2030	2031	2032	2033	2034
Mining	28.68	27.26	26.92	26.40	24.34	24.77	24.42	24.63	19.77
Backfill Plant	3.09	3.83	3.29	3.59	4.20	3.56	3.56	3.29	3.67
Backfill Underground Reticulation	0.60	0.73	0.75	0.89	0.94	0.91	0.88	0.80	0.96
Dry Stack Tailings Plant	0.79	0.44	0.50	0.46	0.37	0.39	0.39	0.46	0.36
Dry Stack Tailings Landform	1.24	0.56	0.66	0.62	0.52	0.48	0.48	0.57	0.52
TSF	0.60	0.24	0.24	0.22	0.16	0.34	0.32	0.45	0.19
Processing	12.49	11.93	11.34	11.29	11.29	11.18	11.23	11.16	9.56
G&A	4.62	4.56	4.50	4.55	4.46	4.38	4.40	4.39	2.64
Total Operating Costs	**52.10**	**49.55**	**48.21**	**48.02**	**46.27**	**46.01**	**45.68**	**45.75**	**37.68**

The LOM process operating cost averages US$13.8/t, decreasing progressively from a maximum of US$15.17/t in 2026 to US$13.46/t in 2031. The decrease is due to a planned reduction in the workforce from 707 in 2026 to 470 by 2034, primarily reflecting changes in mine production levels and ore transportation via the 830 level.

Additionally, the long-term power supply unit cost is assumed at US$8/kWh in line with European Energy Exchange (EEX) power exchange futures and analyst consensus. CAML expects power prices to revert to the long-term mean.

Further cost optimisation programmes are also in progress, and recent tendering shows unit price improvements on consumables.

21.0 Economic Analysis

An economic analysis using a Discounted Cash Flow (DCF) method was used to estimate the Projects return based on expected future revenues, costs, and investments. Cash flows were prepared in real terms with no allowance for inflation. A discount rate of 8% was used.

The economic analysis was based on the Probable Ore Reserves in the LOM schedule. Inferred Mineral Resources were excluded from the economic analysis.

The financial model for the LOM was reviewed by the authors and showed the Ore Reserves to be economic based on the assumptions used and metal prices of US$2,644 per tonne for zinc, US$2,179 per tonne for lead, and US$27 per ounce for silver.

A silver price of US$6.665 per ounce was also used in the financial model to reflect the value received by CAML based on the silver streaming agreement with OR Resources Inc. The Ore Reserves were also shown to be economic at this price.

CAML monitors key economic assumptions, operational performance and technical parameters and has undertaken sensitivity analysis on the LOM financial model for the Sasa operation to assess the impact of changes in the principal value drivers. Sensitivities were applied to zinc and lead prices of ±10%, head grades of ±10%, metallurgical recoveries of ±5%, operating costs of ±10%, capital expenditure of ±10%, treatment charges of ±20%, and discount rates ranging from 8% to 12%. The sensitivity analysis on all inputs delivered a positive NPV within the ranges tested.

22.0 Interpretation and Conclusions

The authors make the following interpretations and conclusions in their respective areas of expertise based upon the review of the available data.

22.1 Geology, Exploration and Mineral Resource Estimates

- Mineral Resources for the Svinja Reka and Golema Reka deposits have been prepared to industry best practice and conform to the resource categories defined by the JORC Code (2012).
- The geology and mineralization of the deposits is well understood and includes significant operational experience at Svinja Reka.
- Mineralisation at Sasa is interpreted as a structurally influenced replacement-style Pb-Zn-Ag skarn-hydrothermal system. Mineralisation continuity is controlled by a combination of favourable host lithologies, bedding-parallel replacement zones and later structural modification. The mineralised lenses are therefore not interpreted as purely stratiform bodies, but as lithologically favourable and structurally modified replacement zones affected by local faulting, shearing and competency contrasts between schist, marble and gneissic host rocks.
- Structural reactivation and local deformation are interpreted to have influenced orebody geometry, continuity, thickness distribution and local grade variability within both Svinja Reka and Golema Reka. Underground mapping and drilling indicate that local faulting, shearing and competency contrasts can segment mineralised horizons and locally disrupt continuity. These structural controls are important for geological domaining, short-term grade control and Mineral Resource confidence.
- Kozja Reka does not form part of the current Mineral Resource due to paucity of data but remains a future exploration opportunity area under modern exploration criteria.
- Exploration drilling is continually ongoing at Svinja Reka with approximately 8,850 m completed each year. A further 6,000 m of resource upgrade/infill and 3,600 m infill/grade control drilling is also completed annually at Svinja Reka. No exploration is currently being undertaken at Golema Reka and the last drilling programme was completed in 2023. No exploration data is available for the Kozja Reka occurrence, although this was also historically mined.
- As of December 31, 2025, a total of 3,035 drillholes for 209,848 m have been completed at Svinja Reka. A total of 241 drillholes for 21,510 m have been completed at Golema Reka.
- The sample preparation, analyses, QA/QC procedures, and sample security are acceptable and in line with industry standard practice. Data verification identified no significant issues with the databases used for Mineral Resource estimation.
- Sasa mine laboratory report a backlog of samples waiting to be processed but remains operationally important for both exploration and grade control activities. Sample prioritisation procedures are applied where required to support short-term production decisions.
- The Mineral Resource estimation process follows industry standard methodology to produce 3D block models representing the interpreted zones of mineralisation. Grade estimation is undertaken for Pb, Zn and Ag, again using industry standard techniques. Validation of the block model shows a close match of estimated grades to input composite sample grades.

- No Measured Mineral Resources are classified at either Svinja Reka or Golema Reka due to the short scale spatial complexity and variability of the mineralized zones. Indicated Mineral Resources are classified based on a drill spacing of 50 m. Inferred Mineral Resources include the remaining areas of the modelled mineralization. Classification is appropriate based on the data spacing and ability to predict continuity of mineralisation.
- The Mineral Resource models are appropriately depleted with the mined-out areas.
- Reconciliation on an annual basis shows a reasonable correlation between the mining plan (based on the resource model) and actual processing plant performance.
- Mineral Resources are reported using a NSR cut-off value considering metal prices and other economic and technical parameters.
- Future exploration potential at Sasa remains favourable both along strike and at depth beneath currently defined mineralised zones. Additional opportunities may exist for structurally repeated lenses, down-dip extensions and blind mineralised zones associated with untested structural corridors. There is good exploration potential at Golema Reka to convert Inferred Mineral Resources to Indicated. Exploration at Kozja Reka has the potential to identify new Mineral Resources.
- SLR has reviewed and agrees with CAML's proposed (2026 – 2028) exploration drill programme. The results of this work programme can be used to assess the potential for along-strike and down-dip extension of known mineralization and to potentially identify additional mineralized zones. A total of circa 35,000 m of underground and surface drilling is planned (2026 – 2028) with an anticipated budget of around US$2.9 million.

22.2 Mining and Ore Reserve Estimates

- Ore Reserves for Svinja Reka have been prepared to industry best practice and conform to the reserve categories defined by the JORC Code (2012). No Ore Reserves are reported for Golema Reka because no Pre-Feasibility or Feasibility Studies have been completed.
- Mining operations have been continuous at Svinja Reka since 2006, and mining methods have been developed and upgraded over time as understanding of the orebody geometry and the geotechnical properties has increased.
- Major mining projects have recently been completed by CAML and include the Central Decline, paste backfill system, and the transition from SLC to LHS and CF as the main mining methods. The LHS and CF methods are conventional and have been operationally proven during 2024 and 2025.
- The Central Decline provides efficient access to the mine production areas, improves ventilation and has the potential to accommodate larger electric vehicles or other material transport options in the future.
- The mine design has been created using Deswik mine planning software and has been completed to a level of detail that is appropriate for estimation of Ore Reserves and LOM planning.
- Stope definition is conducted using a combination of Deswik stope optimisation software and manual design and considers block NSR values specific to the technical and economic parameters of the mining method assigned to the block. NSR cut-offs are derived from cost calculations and estimations specific to the respective mining methods employed.

- The LHS mining method was introduced in 2024 as the main mining method and uses a 10 m sub-level spacing to accommodate the shallow dip and narrow width of the orebody. Drill accuracy is key to the success of a narrow LHS operation with hole deviation increasing with drillhole length. Ore drive dimensions are minimised to 3.5 x 3.5 m to limit dilution of development ore however; dimensions are increased to 4.0 x 4.0 m where the width of the stope allows.
- The CF mining method was also introduced in 2024. CF is used alongside LHS in areas requiring greater selectivity or with challenging ground conditions.
- The mine production sequencing is considered logical and is inclusive of critical production cycle activities, namely stope drilling, paste backfill barricade construction, paste backfilling, and paste backfill cure times. The paste backfill plant and reticulation system reportedly operates well.
- The available mining fleet and associated capacities have been applied as direct planning inputs into the scheduling. The schedule is constrained by user defined equipment capacities that are easily adjusted to analyse schedule sensitivity relating to equipment performance.
- The LOM schedule runs for 9 years, and the mining rate is planned to increase to 830 ktpa for the next 8 years before declining in year 9 to around 700 ktpa. Average grades in the LOM schedule are 3.5% Pb and 2.5% Zn.
- Ongoing underground geological mapping, face sampling, channel sampling and definition drilling provide important support for short-term production planning and grade control confidence. Timely assay turnaround is important for effective stope definition and ore-waste control.

22.3 Mineral Processing and Recovery Methods

- The operation has a proven track record of processing the lead-zinc-silver mineralization from Sinja Reka, since 2006. Processing methods are conventional and the metallurgy for lead and zinc is relatively simple.
- Metallurgical recoveries of 94% for lead, 86% for zinc and 75% for silver were used in the LOM plan and are based on actual plant performance.
- The processing plant has a conventional flowsheet for the selective flotation and production of separate lead and zinc concentrates. Concentrate grades are approximately 71% Pb and 50% Zn. Silver is recovered to both the lead and zinc concentrates but is payable only in the lead concentrate. Silver recovery to the lead concentrate is 75% to a grade of 254 g/t Ag.
- No deleterious elements affect the lead concentrate. For the zinc concentrate, no significant deleterious elements result, although it is important to control the silica and iron content, with maximum levels of 2.5% SiO_2 and 11% Fe, respectively.
- One challenge is the variation in head grades in the processing plant, often over a short time frame. This is due to variations in ore grades and the small ROM ore pad, relatively small storage in the mill feed bins and lack of significant blending facilities. The plant has an automatic Metso Courier assay analysis system which provides real time assays that the plant operators use to manually control and vary reagent dosages as the head grades vary. The Pb:Zn head grade ratio is also an important factor to control.
- The use of sodium cyanide as a zinc depressant in conjunction with zinc sulphate is less common now (for environmental reasons). Rudnik SASA DOOEL reports that, if

not using sodium cyanide and only zinc sulphate, metals recovery is negatively affected.

- Continual process optimisation is achieved with use of the on-site metallurgical testwork laboratory.
- In terms of water management, fresh (river) water is used as make-up water for the flotation plant. For the grinding circuit, both mine water and return water from the DST plant are used. TSF-4 return water is currently not used for process water.
- In terms of maintenance and spares, Metso provides sufficient spares on consignment. Spares for the ex-Soviet grinding mills are also readily available and can be manufactured as required based on engineered drawings.
- The commissioning of the paste backfill plant in 2023 and the dry stack facility in 2025 obviates the requirement for any new TSF construction, which is beneficial to the mine in terms of operational and environmental risk management (two new TSFs would otherwise have been required with the necessary extensions of the Kamenica and Petrova River diversion channels).
- All three systems (paste backfill, DST and TSF) are required to operate throughout the LOM with paste backfill taking priority. The dry stack operation is affected by high rainfall and must stop for periods during the rainy season when more tailings is deposited in TSF-4. These variations are incorporated into the LOM plan. Currently, there is an approximate even split of tailings over the three methods.
- The paste backfill and dry stack facilities are modern conventional plants, automated with simple flowsheets and minimal operator requirements, and appear to be operating well to the design parameters.
- The Sasa Mine has sufficient supply of energy and water for the forecast throughputs for the processing plant. The sourcing of process consumables is well established with recognised suppliers and logistics.

22.4 Infrastructure

- Infrastructure projects that have recently been completed include the paste backfill plant and reticulation system and the dry stack tailings plant and landform. The introduction of paste backfill and dry stack tailings has reduced the requirement for tailings storage in the existing TSF-4.
- All major infrastructure required by the operation is in place and no additional significant infrastructure projects are currently planned.

22.5 Environmental Studies, Permitting, and Social or Community Impact

- Environmental and social management systems, permitting processes, monitoring programmes, and closure planning at the Sasa Mine are sufficiently developed to support ongoing operations and the assumptions typically required for Mineral Resource and Reserve estimation and future production, subject to continued compliance performance and active management of key risks.
- Permits held by Rudnik SASA DOOEL for the Property are sufficient to ensure that mining activities are conducted within the regulatory framework.
- There are currently no known environmental, permitting, or social/community risks that could impact the Mineral Resources or Ore Reserves.

22.6 Capital and Operating Costs

- Total capital costs are estimated at US$75.7 million. A cost of US$61 million is also estimated for closure.
- The total operating cost for the LOM is US$419.3 million.

22.7 Economic Analysis

- The financial model for the LOM was reviewed by the authors and showed the Ore Reserves to be economic based on the assumptions and metal prices used.

22.8 Risks and Uncertainties

- The Sasa Mine is a mature operation with a long production history. The authors consider there are no significant risks or uncertainties that could be expected to affect the reliability or confidence of the Mineral Resources, Ore Reserves and projected production based on the assumptions used.

23.0 Recommendations

The authors make the following recommendations in their respective areas of expertise, based upon the review of the available data.

23.1 Geology, Exploration and Mineral Resource Estimates

- Although the majority of the Mineral Resource is informed by recent drilling, some historical data still exists within the database that has less confidence in location accuracy and is less well supported by QA/QC protocols. Consideration should be given to removal of historical data in areas where more recent drilling is sufficient to support the estimate.
- Increase the collection of density measurements at all deposits to obtain a better understanding of the behaviour of density across lithologies and mineralized domains. As the mine development deepens it is understood that the ore is becoming harder and may therefore vary in density.
- The QA/QC processes would benefit from additional blank samples (fine and coarse), as well as additional external check analysis at an independent laboratory, to increase support for the data being provided by the Sasa mine laboratory. In addition, it is recommended that the laboratory pursues accreditation for AA analysis used for exploration samples.
- The mine laboratory reports a backlog of samples that, even with prioritisation, may lead to samples not being processed in sufficient time. This is particularly critical to production samples that are required to inform mining activities and measures should be considered to alleviate this situation. Laboratory turnaround performance should continue to be monitored through operational KPIs, and external laboratory support may be used during periods of increased sample volume, maintenance downtime or backlog risk.
- The current Mineral Resource is based on evaluation using a NSR cut-off value. SLR considers that the estimate should also apply the use of Mineable Shape Optimisation (MSO) to define the Mineral Resources.

23.2 Mining and Ore Reserve Estimates

- Grade control should be improved and better aligned with production scheduling with increased availability of drill rigs to reduce conflicts with development activities, along with shortened assay turnaround times for more effective grade control and short-term planning.
- Geological reconciliation should continue to compare Mineral Resource model predictions against stope production, grade control samples, mined tonnes and plant performance. Reconciliation practices would benefit from increased resolution (current reporting at a 50 m by 50 m panel scale) to capture local grade distribution and provide better defined stope level detail.
- Enhance ongoing analysis of hanging wall stability to better determine achievable unsupported spans.
- Short term sequencing is being adapted to maintain production by targeting additional CF stopes where access to LHS areas is not progressing as planned though this may complicate the overall life of mine sequencing. However, as the mine improves its LHS operation alignment with long term sequencing it is expected become more consistent.

- The planned increase in production from approximately 800 ktpa to 830 ktpa is reliant on the effectiveness of LHS stoping. Operational performance in LHS areas is improving as experience with the method increases.

23.3 Mineral Processing and Recovery Methods

- Continued metallurgical testwork is recommended to monitor and balance throughput, grind size, metals recovery and concentrate grades for optimal economics. In addition, regular testing and monitoring of ore hardness is recommended as mining progresses and the ore gets harder, as maintaining the current grind size at increasing ore hardness presents the main bottleneck to increasing plant throughput.
- A planned ore sorting testwork programme will be used to inform an options study to evaluate the potential for rejecting a significant fraction of waste material prior to further processing which would reduce the throughput bottleneck and potentially give higher flotation recoveries from an upgraded feed. There is also potential for lower grade ore to be mined and upgraded to allow economical extraction.

23.4 Infrastructure

- Testing should continue to investigate the potential for recycling TSF-4 supernatant water back to the process plant and reduce the fresh (river) water make-up required for the flotation circuit.
- Current monitoring of the TSFs and water discharge streams should continue, aligned with on-going support from Knight Piésold, and implementing any further recommendations that may ensue. The beach length and level of supernatant water in TSF-4 should be monitored to maintain the minimum freeboard and beach length, to ensure no overspill can occur.

23.5 Environmental Studies, Permitting, and Social or Community Impact

- Consider Good International Industry Practice (GIIP) with regards to Biodiversity Net Gain (BNG) during expansion works for the DST and any other surface developments where vegetation is cleared.
- Continue ongoing monitoring programmes of the TSF and external reviews by Knight Piésold. Implement any further recommendations to mitigate risks to the environment and local communities.
- Continue with the current environmental monitoring regime, updating in line with IPPC Permit revisions, and international best practice.

24.0 References

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25.0 Date and Signature Page

Competent Person's Consent Form

Pursuant to the requirements of ASX Listing Rules 5.6, 5.22 and 5.24 and Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

Report Description

Mineral Resources and Ore Reserves Report on the Sasa Mine

(*Insert name or heading of Report to be publicly released*) *('Report')*

SLR Consulting Limited

(*Insert name of company releasing the Report)*

Sasa Mine (North Macedonia)

(Insert name of the deposit to which the Report refers)

August 5, 2026

(Date of Report)

Statement

I,

Jason Ché Osmond

(*Insert full name(s)*)

confirm that I am the Competent Person for the Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Chartered Geologist with the Geological Society of London a 'Recognised Professional Organisation' (RPO) included in a list promulgated by ASX from time to time.
- I have reviewed the Report to which this Consent Statement applies.

I am a full-time employee of

SLR Consulting Limited

(Insert company name)

and have been engaged by

Central Asia Metals plc

(Insert company name)

to prepare the documentation for

The Sasa Mine (North Macedonia)

(Insert deposit name)

on which the Report is based, for the period ended

December 31, 2025

(Insert date of Resource/Reserve statement)

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

Consent

I consent to the release of the Report and this Consent Statement by the directors of:

Central Asia Metals plc

(Insert reporting company name)

Signature of Competent Person:	Date:
	August 5, 2026
Professional Membership:	**Membership Number:**
Registered Chartered Geologist (CGeol) with the Geological Society of London (UK)	**Reg No. 1016839**
Signature of Witness:	Print Witness Name and Residence:
	Richard Ellis **SLR Consulting Limited** **Baldhu House** **Wheal Jane Earth Science Park** **Baldhu, Truro, TR3 6EH** **United Kingdom**

Statement

I,

Colin Davies

(*Insert full name(s)*)

confirm that I am the Competent Person for the Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Chartered Engineer with the Institute of Mining, Metallurgy and Materials a 'Recognised Professional Organisation' (RPO) included in a list promulgated by ASX from time to time.
- I have reviewed the Report to which this Consent Statement applies.

I am a full-time employee of

SLR Consulting Limited

(Insert company name)

and have been engaged by

Central Asia Metals plc

(Insert company name)

to prepare the documentation for

The Sasa Mine (North Macedonia)

(Insert deposit name)

on which the Report is based, for the period ended

December 31, 2025

(Insert date of Resource/Reserve statement)

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Ore Reserves.

Consent

I consent to the release of the Report and this Consent Statement by the directors of:

Central Asia Metals plc

(Insert reporting company name)

Signature of Competent Person:

Date:

August 5, 2026

Professional Membership:

Registered Chartered Engineer of the Institute of Mining, Metallurgy and Materials

Membership Number:

Reg No. 449545

Signature of Witness:

Print Witness Name and Residence:

Richard Ellis

SLR Consulting Limited

Baldhu House

Wheal Jane Earth Science Park

Baldhu, Truro, TR3 6EH

United Kingdom

26.0 JORC Code, 2012 Edition - Table 1 Checklist of Assessment and Reporting Criteria

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	• Surface and underground diamond core drilling is used to provide samples as the basis for the Mineral Resource estimates. • Drilling orientations are designed to intersect the mineralised horizons at angles as close to perpendicular as possible and provide representative samples approximating the true thickness of the mineralisation. • Samples from drill core are collected by Rudnik SASA DOOEL geological staff. A sample interval of 0.3 to 1.0 m is used. • Logging, sampling and sample cutting (half core) are undertaken at the on-site core logging facility at the Sasa Mine. • Mineralised half core is retained for archive at the core storage facility located at the mine. • Sample preparation and analysis is undertaken at the Sasa Mine analytical laboratory. A 50 g pulp sample is used for analysis by atomic absorption (AA). The Sasa Mine laboratory is not accredited. • A systematic Quality Assurance / Quality Control (QA/QC) system is used to monitor the accuracy and precision of assaying.
Drilling techniques	• All surface drillholes were completed using 76 mm diameter, HQ or NQ diamond core. • Underground drillholes were completed using 76mm (NQ) diameter ,36 mm diameter (BQ) or AX diamond core in up-holes.
Drill sample recovery	• The sample interval was reconciled to the length of the recovered core to derive sample recovery. • Core recovery information for most drilling programmes has not been transferred to the drillhole database and therefore is not readily available for review; however, based on observations made at drill sites and the core storage facility, core recovery is considered to be good (generally greater than 90%). • Historical core recovery, where recorded, is generally above 80% in surface drillholes. • No relationship between grade and core recovery has been identified.
Logging	• All drill core is logged for geology, core recovery, geotechnical parameters and digital photographs of the core are taken. • Logging is considered qualitative in nature.
Sub-sampling techniques and sample preparation	• Sample lengths for drill core are guided by visually logged geological contacts and typically range between 0.3 and 1 m in length, using half core for analysis. • The remaining half core is stored for three to five years in case external controls require re-assaying. • Samples are submitted for preparation to the Sasa Mine laboratory, where half core is crushed to -3 mm and then dried in an oven at 130°C. • The sample is passed through a riffle splitter to derive a 50% split, which is pulverised using a disc mill to give a -0.74 µm, 50 g pulp sample for analysis. • Sub-sampling and laboratory preparation methods are considered satisfactory for the intended purpose.

Criteria	Commentary
Quality of assay data and laboratory tests	• The Sasa Mine laboratory analyses samples for Pb, Zn and Ag by AA using 3-acid digestion. • The Geology Department submits routine QA/QC samples (certified standards, blanks and coarse reject duplicates along with the core sample batches. Where sample results are out of specification then portions of the batch are re-submitted for re-assay. Generally, the assay results are within acceptable levels of accuracy and precision. • In 2021 and 2024, the Sasa Mine laboratory submitted check samples to the accredited Eurotest-Control EAD laboratory in Sofia, Bulgaria. The results of the check analysis for both periods are considered acceptable and identified no significant issues with the Sasa Mine laboratory analysis.
Verification of sampling and assaying	• Significant intersections are cross-checked with the logged geology and drill core after final assays are received. • Drillhole data is stored in an MX Deposit database which was implemented in June 2022. • No holes have been twinned. • No adjustments have been made to the assay data.
Location of data points	• The position of all surface drillhole collars completed by the Company was completed using a Precision GPS. • Underground drillhole collars and development surveys are located based on total station surveys by mining surveyors and are translated to the Macedonian grid (Gauss-Krüger coordinate system, Hermannskogel datum) for storage in the master database. • Downhole surveys for surface holes, where recorded, were completed using a reflex gyro probe by Geops (Bulgaria) drill and survey contractor, with readings taken at approximately every 50 m. • No downhole surveys were recorded for the underground drillholes. These holes are typically short and range in length between 50 and 70 m.
Data spacing and distribution	• Surface drillholes provide intersections at approximate 50 to 200 m spacing. • Underground drilling is used to infill the surface drilling. Within mining areas these are typically collared at 20 to 30 m spacing with multiple (fan) holes often drilled at a range of inclinations from a single collar, providing sample coverage ranging from 10 to 30 m. • SLR considers the drill spacings to be sufficient to demonstrate spatial and grade continuity to support the definition of Inferred and Indicated Mineral Resources. • Raw sample data has not been composited.
Orientation of data in relation to geological structure	• Surface drillholes are oriented northeast-southwest across the deposit and intersection with the mineralisation is broadly perpendicular. • Underground drilling is typically drilled towards the southwest (from footwall to hanging wall) with multiple (fan) holes drilled at a range of inclinations providing intersections with the mineralisation that typically range from perpendicular to -45°.
Sample security	• The core is transported to the core logging facility at the end of each second drilling shift. • All sampling and analysis are carried out by the Sasa Mine Laboratory. • Industry best practices for Chain of Custody procedures have been used and archived drillcore is stored on site in a secure facility.

Criteria	Commentary
Audits or reviews	• In 2021 and 2024, the Sasa Mine laboratory submitted check samples to the Eurotest-Control EAD laboratory in Sofia, Bulgaria. The results of the check analysis for both periods are considered acceptable and identified no significant issues with the Sasa Mine laboratory analysis. • A review of the Sasa Mine Mineral Resource estimation procedures (including sampling techniques and data) was undertaken by Wardell Armstrong International (WAI) in May 2025. Recommendations for further work were made; however, no fatal flaws were identified in the review.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	• The Property comprises an Exploitation Concession located in northeastern North Macedonia, some 150 km east of the capital, Skopje. • The Exploitation Concession (24-5550/1) was granted to Rudnik SASA DOOEL on November 13, 2014, and is valid until September 28, 2030. The Exploitation Concession was expanded in February 2026 from an area of 4.22 km^2 to 6.866079 km^2. Rudnik SASA DOOEL can apply to extend the concession beyond the current expiry date. • Rudnik SASA DOOEL is a 100% owned subsidiary of CAML.
Exploration done by other parties	• During 1936, 1945, and 1946 the Project area was prospected by Bulgarian geologists, but detailed work only began in 1952 when geological mapping by a team from the Geological Institute of Skopje produced maps over the mineralised occurrences. This was followed by geophysics, trenching, drilling and excavating crosscuts through mineralisation at surface. • Exploration by other parties at the Sasa Mine is summarised below: o The Sasa Mine originally started working in 1965 but stopped in 2001 when government funding of operating capital ended. From 1954 to 2002, a total of 274 drillholes (12 from surface and 262 from underground) were completed at Svinja Reka for a total of 20,597 m and 82 drillholes (24 from surface and 58 from underground) were completed at Golema Reka for a total of 11,942 m. o Solway Investment Group Ltd (Solway) acquired the mine from a group of creditors in 2005. The plant was re-equipped with a state-of-the-art flotation circuit allowing production of separate zinc and lead concentrates. From 2006 to 2015, a total of 1,080 drillholes for 80,737 m were completed at Svinja Reka (32 from surface and 1,048 from underground), and 22 underground drillholes for 1,503 m were completed at Golema Reka. o On November 3, 2015, Solway announced that it had sold the Sasa Mine to the Orion Mine Finance Group (Orion) partnered with Fusion Capital AG (Fusion) through its wholly owned subsidiary, Lynx Europe SPLLC Skopje (Lynx). From 2015 to 2017, under ownership of Lynx a total of 228 drillholes were completed at Svinja Reka. o In November 2017, CAML acquired 100% ownership of the Project. A total of 1,513 drillholes for 108,095 m have been completed by CAML.
Geology	• The Project is located in the Serbo-Macedonian massif which comprises greenschist and amphibolite facies metamorphic rocks, Precambrian to Palaeozoic in age, which have been variably intruded by andesitic to trachytic volcanic rocks during the Tertiary. • Lead-zinc-silver mineralisation at Sasa occurs as bedding concordant deposits hosted predominantly by quartz-graphite schist and marbles of Lower Palaeozoic age at Svinja Reka, while at Golema Reka it is primarily developed within gneissic host rocks. • High-temperature hydrothermal fluids and bedding-parallel faulting (related to the intrusion of Tertiary volcanics) is responsible for metasomatism of the host sediments to develop skarn and base metal mineralisation.

Criteria	Commentary
	• The well-defined, partially exploited lenses of lead-zinc-silver mineralisation dip at approximately 35° to the southwest and typically range in true thickness from between 2 – 30 m.
Drill hole Information	• No exploration results are being reported; therefore, this section is not relevant to this report on Mineral Resources and Ore Reserves.
Data aggregation methods	• No exploration results are being reported; therefore, this section is not relevant to this report on Mineral Resources and Ore Reserves.
Relationship between mineralisation widths and intercept lengths	• No exploration results are being reported; therefore, this section is not relevant to this report on Mineral Resources and Ore Reserves.
Diagrams	• Relevant maps, sections and diagrams are provided in the report.
Balanced reporting	• No exploration results are being reported; therefore, this section is not relevant to this report on Mineral Resources and Ore Reserves.
Other substantive exploration data	• No exploration results are being reported; therefore, this section is not relevant to this report on Mineral Resources and Ore Reserves.
Further work	• A structural study is ongoing and, in the future, will help, highlight areas that may host further lead-zinc-silver mineralisation. • Three deep surface drillholes were drilled in 2022. One proved the southwestern extension of the mineralisation. However, two drillholes did not intersect mineralisation but confirmed the limits of the mineralisation in the northern central area. • Diagrams highlighting areas of possible extensions and future drilling areas are provided in the report (Section 10.2.6).

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	• Data entry, validation, storage and database maintenance is carried out by Rudnik SASA DOOEL staff using established procedures. The diamond core drilling data used for the Mineral Resource estimates are stored in a central MX Deposit database located at the Sasa Mine offices. The database has a series of automated validation tools during import and export for error identification. The quality of the assay data contained in the database is monitored by Rudnik SASA DOOEL staff using established QA/QC procedures. • Data validation conducted by SLR included: ○ A site visit undertaken on May 13 to 14, 2026. ○ Review of the geological and geographical setting of the Svinja Reka and Golema Reka deposits. ○ Review of extent of the exploration work completed to date. ○ Inspection of drill core to assess the nature of the mineralisation and to confirm geological descriptions. ○ Inspection of geology and mineralisation in underground exposures. ○ Review of drilling, logging, sampling and analysis procedures. ○ An evaluation of minimum and maximum grade values and sample lengths. ○ Assessing for inconsistencies in spelling or coding (typographic or case sensitive errors). ○ Ensuring full data entry for each drillhole and that a specific data type (collar, survey, lithology and assay) is not missing. ○ Assessing for sample gaps and overlaps. ○ A review of assay detection limits. ○ Identification of any problematic assay records. ○ A spatial on-screen review of the grade and lithology distributions of the drillholes was undertaken to identify any additional data reliability issues. ○ A review of collar locations for underground or surface drilling relative to underground workings or the topographical surface. • Overall, no significant issues in terms of data collection, data entry or data storage were identified by the authors in a review of the electronic databases.
Site visits	• The Competent Person under the JORC Code (2012) for this Mineral Resource estimate is Mr Jason Ché Osmond, Technical Director for Geology and Mineral Resources, and a full-time employee of SLR. • Mr Osmond visited the Sasa Mine on May 13 to 14, 2026.
Geological interpretation	• The geology of the Svinja Reka and Golema Reka deposits is well known and is summarised below: ○ The Svinja Reka deposit comprises well-defined, partially exploited lenses of lead-zinc-silver mineralisation that dip at approximately 35° to the southeast. Individual lenses typically range in true thickness from approximately 2 to 30 m and occur as sub-parallel bodies, commonly consisting of two or three discrete lenses separated by interburden zones of 1

Criteria	Commentary
	to 10 m in thickness. The mineralised lenses exhibit pinch and swell features along strike and down-dip, with an apparent dominant plunge to the south. The principal lenses of economic interest are identified as Podina (footwall), Sredisno (central), and Krovina (hanging wall), all of which are characterised by relatively sharp geological contacts. The thickness of the mineralised lenses reduces with increasing depth of the orebody. o Lead-zinc-silver mineralisation at the Golema Reka deposit is hosted within granitoid gneiss, striking approximately 135° and dipping moderately (approximately 30 to 45°) to the southeast. Mineralisation occurs as stacked, massive to semi-massive sulphide lenses with thicknesses ranging from approximately 0.5 to 10 m along strike and down dip. A total of seven mineralised lenses has been identified and have been partly exploited by historical mining. Individual lenses display pinch-and-swell geometries both along strike and down dip, with a dominant plunge to the south. The lenses are separated by intervals of weakly mineralised to barren gneiss. Mineralisation remains open down dip towards the southeast and along strike towards the northwest, although it may locally terminate against a northwestern fault that separates the Golema Reka deposit from the Kozja Reka occurrence. Geological continuity at Golema Reka remains less certain than at Svinja Reka due to lower drilling density and increased geological complexity. • Underground mapping combined with diamond drilling information has been used to interpret the mineralised zones using Leapfrog Geo software. • A nominal 2% Pb+Zn cut-off plus lithology was used to constrain the mineralised domains and improve the continuation of the orebodies, although, since the contacts are generally sharp, some weaker mineralisation was included in the model. • Due to the multiple lenses, and pinching and swelling morphology, underground mapping was also used to guide the interpretation of the 3-dimensional domain solids. • Internal waste zones (<2% Pb+Zn) are common within the mined, mapped, and modelled zones for Svinja Reka. In many cases it is not practical to separate these zones.
Dimensions	• At Svinja Reka, the well-defined, partially exploited lenses of mineralisation dip at approximately 35° to the southwest and typically range in true thickness between 2 – 30 m. • The mineralised lenses are present in parallel sheets (typically 2 or 3 bodies), separated by an interburden with thicknesses of 1 - 10 m. • The lenses pinch and swell along strike and down-dip, with an apparently dominant southerly plunge. • The mineralisation is continuous along strike, in some lenses up to 1,000 m.
Estimation and modelling techniques	• The following estimation and modelling techniques were used by CAML to estimate the Mineral Resources for the Svinja Reka deposit (a similar workflow was also adopted for the Golema Reka deposit). Estimation was undertaken using Leapfrog, Supervisor and Datamine software. o Database compilation and review. o Compositing the sample data to 1 m intervals. o Grade capping levels were based on appropriate population breaks indicated in the log probability plots, resulting in a reduction of coefficient of variation to less than 1.5. Grade caps were applied to the composites.

Criteria	Commentary
	○ Geostatistical analyses and variography was undertaken. ○ A block model was generated within the geological model wireframes using parent blocks of dimensions 3.5 mE by 14 mN and 7 mRL. Sub-cell splitting was applied to better represent the natural geometry and distribution of the mineralisation and reflect the ability to discriminate between higher grade and lower grade blocks or waste blocks at a size corresponding to the proposed mining methods. ○ For Pass 1 a minimum of 8 and maximum of 26 composites were used. For Pass 2 a minimum of 6 and maximum of 18 composites were used. For Pass 3 a minimum of 4 and maximum of 10 composites were used. ○ Grades were estimated using Ordinary Kriging for Pb, Zn and Ag. Bulk density values were interpolated using Nearest Neighbour and/or a regression formula based on estimated Pb grades. ○ Hard boundary estimation was applied during the grade estimation. ○ No correlation exists between Pb and Zn. There is a strong correlation between Pb and Ag. ○ Visually and statistically validated the estimated block grades relative to the original sample results.
Moisture	• Tonnages of the Mineral Resources are estimated on a dry weight basis.
Cut-off parameters	• The Mineral Resources are reported using Net Smelter Return cut-off values of $53/t for sub-level stopes, $65/t for cut-and-fill and $60/t for longhole stopes. • Metal price assumptions are $3,041/t for zinc, $2,506/t for lead and $31/oz for silver. The metal prices used for reporting the Mineral Resource are approximately 15% above those used for the Ore Reserves. • Metallurgical recoveries used in the calculation of NSR were 94% for lead, 82% for zinc and 75% for silver. The zinc recovery is conservative and less than the 86% achieved by the processing plant. Royalties of 2% were used and are in the process of being updated to 4% based on the updated Concession Fee in 2025. • The NSR cut-off values are based on projected operating costs for mining, processing and G&A costs provided by the Sasa Mine Financial Department.
Mining factors or assumptions	• Mining of the Svinja Reka deposit is planned to continue using sub-level caving, cut and fill stoping and long-hole stoping. • Mining of the Golema Reka deposit is planned to use long-hole stoping. • Further information on Mining Factors is provided in Section 4 of this Table 1.
Metallurgical factors or assumptions	• The Sasa process plant has been operating since 2006 and comprises a three-stage crushing circuit and conventional grinding, classification and selective flotation circuits to produce lead and zinc concentrates that are thickened and filtered for shipment by truck. • Actual metallurgical recoveries achieved by the processing plant are 94% for lead, 86% for zinc and 75% for silver. • Further information on Metallurgical Factors is provided in Section 4 of this Table 1.
Environmental factors or assumptions	• The Project is considered compliant with local North Macedonian legislation, and a considerable amount of work has been undertaken to bring the Project in line with international best practice. • SLR is not aware of any waste storage, environmental or permitting issues that prevent the reporting of a Mineral Resource

Criteria	Commentary
	estimate for the Sasa Mine. • Further information on Environmental Factors is provided in Section 4 of this Table 1.
Bulk density	• Density measurements are conducted on core samples of approximately 10 cm length using the Archimedes method. • For Svinja Reka density values for the mineralisation and internal waste domains were assigned in the Mineral Resource model based on a two-pass nearest neighbour interpolation approach. In the first pass, a nearest neighbour estimation was applied using a search ellipsoid of 50 x 50 x 12 m. For any unestimated blocks, density values were assigned based on empirical relationships between the combined Pb and Zn grades and the measured densities. Regression equations were developed from a total of 2,290 core measurements. Within the schist domain, a hybrid approach was adopted. All blocks with an estimated Pb + Zn grade of less than 2.0% were assigned a density of 2.75 t/m³. Blocks with grades equal to or greater than 2.0% Pb + Zn were assigned density values calculated using the regression equation. • For Golema Reka a regression formula based on the relationship between Pb and Zn grades and density from 81 samples was used to assign density in the block model and was applied post-grade estimation. For blocks with no estimated Pb or Zn grades, a density of 2.7 t/m³ was used.
Classification	• Mineral Resource classification was made following the guidelines of the JORC Code (2012). • No Measured Mineral Resources were classified due to the spatial complexity and local variability of the mineralisation. • Indicated Mineral Resources were classified based on a minimum of 4 drillholes and a spacing of 50 m or less. • Inferred Mineral Resources were classified based on a minimum of 2 drillholes and a spacing of up to 150 m. • Wireframes depicting contiguous areas were constructed and used to assign the Mineral Resource classification in the block model. • The applied classification is considered appropriate by the Competent Person to reflect the geological interpretation and estimation risk.
Audits or reviews	• A review of the Sasa Mine Mineral Resource estimation procedures was undertaken by Wardell Armstrong International (WAI) in May 2025. Recommendations for further work were made; however, no fatal flaws were identified in the review.
Discussion of relative accuracy/ confidence	• The relative accuracy and confidence in the Mineral Resource estimate is reflected in the resource classification categories applied which were developed in accordance with the guidelines of the JORC Code (2012). • The geological interpretation used to generate the Mineral Resource presented herein is generally considered to be robust and is based on significant exploration and operational experience; however, there are areas of lower geological confidence which may be subject to further revision in the future. • In areas of limited drilling and where an Inferred Resource classification has been applied there is insufficient information to support a robust estimate on which to support the application of Modifying Factors in sufficient detail to support mine planning and the evaluation of the economic viability of the deposit. Geological evidence within these Inferred areas is sufficient to imply but not confirm geological or grade continuity. • Reconciliation comparing planned production from the resource model against actual performance of the processing plant is

Criteria	Commentary
	undertaken monthly. The results of the reconciliation are accumulated annually. • On an annual basis, the reconciliations show a generally good agreement between the resource model and actual production and generally report less than 5% difference in contained Pb and Zn metal. Overall, the results of the reconciliations are considered by the Competent Persons to be within acceptable tolerances. • The statement relates to global estimates of tonnes and grade.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• The Mineral Resource estimate on which this Ore Reserve estimate has been based was prepared by CAML and dated December 31, 2025. The Mineral Resource estimate was audited by SLR. • The Indicated Mineral Resources have been used as the basis of the Ore Reserve estimate. No Measured Mineral Resources have been declared. • The Mineral Resource estimate is inclusive of the Ore Reserve estimate. • Ore Reserves are reported for the Svinja Reka deposit only (currently operational). No Ore Reserves are reported for the Golema Reka deposit because no Pre-Feasibility Study or Feasibility Study has been undertaken.
Site visits	• The Competent Person under the JORC Code (2012) for this Ore Reserve Estimate is Mr Colin Davies, Associate Director for Mining, and a full-time employee of SLR. • Mr Davies visited the Sasa Mine on May 13 to 14, 2026.
Study status	• The Sasa Mine is currently operating. Modifying Factors were based on actual mining, mineral processing, geotechnical, hydrogeological, mine method, infrastructure, environmental and social, market and economic model information. • Material with a geological confidence classification of Inferred has not been included within the Ore Reserve estimate. • The mine plan is considered technically achievable and economically viable. All material Modifying Factors were considered. • The Mineral Resources have been converted to an Ore Reserve estimate by means of a Mineable Shape Optimiser (MSO) based on reasonable technical and economic parameters and a scheduled underground mine design.
Cut-off parameters	• NSR was calculated to determine the value of each individual stope and used metal prices, payable metals considering lead, zinc and silver grades, metallurgical recoveries, and realisation costs. • Metal prices used in the calculation of NSR for Ore Reserves were US$2,644 per tonne for zinc, US$2,179 per tonne for lead, and US$27 per ounce for silver. • Metallurgical recoveries used in the calculation of NSR were 94% for lead, 82% for zinc and 75% for silver. The zinc recovery is conservative and less than the 86% achieved by the processing plant. Royalties of 2% were used and are in the process of being updated to 4% based on the updated Concession Fee in 2025. • The NSR calculation also considered the planned underground mining methods based on orebody geometry for the following mining levels: o SLC between levels 810 m and 962 m, and above 990 m. o CF between levels 750 m and 760 m, and between 800 m and 810 m. o LHS below level 750 m, between levels 760 m and 800 m, and between levels 962 m and 990 m. • NSR cut-off values (COVs) of US$53 per tonne for SLC, US$65 per tonne for CF and US$60 per tonne for LHS were used to estimate the Ore Reserves.

Criteria	Commentary
Mining factors or assumptions	• Svinja Reka is an operating underground mine and produces approximately 800 ktpa of ore, with average grades of 3.4% Pb and 2.6% Zn. • Mining operations have been continuous at Svinja Reka since 2006, and mining methods have been developed and upgraded over time as understanding of the orebody geometry and geotechnical properties has increased. • The main access to the underground workings is through the Central Decline, which extends a distance of 3,735 m from a portal located adjacent to the process plant to a depth of 750 mRL. Secondary access is provided by the Golema Reka shaft and the Level 14b portal. As of H1 2026, approximately 75% of the ore is hauled by truck and 25% via shaft, with shaft haulage scheduled to be phased out in 2026. • Historically, the mine operated using sub-level caving (SLC) with 7.0 m sublevel spacing. As the orebody narrows with depth, the suitability of this method has declined. Since 2023, a transition to long hole stoping (LHS) with backfill has been ongoing as the primary mining method. Cut and fill mining (CF) is also used in areas requiring greater selectivity or with difficult ground conditions. Both historical voids and newly mined stopes are backfilled using cemented tailings. • The Sasa Mine uses conventional trackless underground mining equipment. Ventilation is based on a mechanical overpressure system that supplies fresh air to active mining areas and exhausts return air through a network of raises and airways. Two principal fans are installed, and fresh air is supplied primarily via the decline portals. • The current life of mine plan extends over approximately nine years. Production is expected to increase to around 830 ktpa over the next three years, before reducing towards the final years. Average planned grades over this period are 3.5% Pb and 2.5% Zn. Mining is split between long hole stoping (67%), cut and fill (17%) and sub-level caving (15%). • Mine design and scheduling works are carried out using the Deswik Software suite and follow a defined process which conforms to industry best practice for the estimating Ore Reserves. • Stope locations are identified using stope optimisations, with final stope designs augmented by manual checks and verifications. Development headings are designed manually according to mining method and location. • Stopes are designed based on average block NSR values which are incorporated into the Mineral Resource block model prior to commencement of the mine design. • Stopes are designed using a combination of optimisation and manual stope/development design. The design basis is driven by the NSR COVs, in conjunction with the geotechnical parameters defining maximum stope dimensions, selected mining method and block value. • Stope optimisation creates a stope shape that maximises the recovered Mineral Resource value above the COV while also allowing for practical mining parameters including minimum and maximum mining width, anticipated wall dilutions, minimum and maximum wall angles, minimum separation distances between parallel and/or sub-parallel stopes, minimum and maximum stope heights and widths. Mineable shapes are evaluated using the Mineral Resource block model, based on the NSR COVs and the Mineral Resource classification. • Dilution factors, including consideration of dilution from backfill and dilution by Inferred Mineral Resources, are applied to the mineable shapes with a diluted stope grade calculated for each stope within the design. Stopes which have an average NSR value below the appropriate COV are excluded from the design and scheduling process.

Criteria	Commentary
	• Stopes which achieve the appropriate NSR are then processed in Deswik to calculate design quantities, including application of mining recovery factors, to define the Ore Reserves. • Mining dilution was applied to excavations based on mining method, development profile and stope dimensions. Total stope dilution includes overbreak (including from backfill), Inferred Mineral Resources and material below COV contained within the stope designs. Inferred Mineral Resources contained within the stope designs were considered as having zero grade. • Mining recovery was applied based on operational experience of the different mining methods. • A summary of the dilution and mining recoveries is provided below: o SLC – 30% dilution and 82% mining recovery. o CF – 10% dilution and 98% mining recovery. o LHS – 18% dilution and 92% mining recovery. • Mining recovery was applied based on operational experience of the different mining methods. • A minimum mining width of 1.8 m was used. • Inferred Mineral Resources contained within the stope designs were considered as having zero grade. • Inferred Mineral Resources were excluded from the Ore Reserve estimate.
Metallurgical factors or assumptions	• The Sasa processing plant was commissioned in 2006 and was originally designed to process 650 ktpa of ore. • The plant was upgraded in 2007 with additional Metso flotation cells to a nameplate capacity of 850 ktpa of ore. However, to match the mine production rate, the plant operates at throughputs less than this and, in recent years, has processed approximately 800 ktpa. • The process plant comprises a three-stage crushing circuit and conventional grinding, classification and selective flotation circuits to produce lead and zinc concentrates that are thickened and filtered for shipment by truck. • Metallurgical recoveries used in the LOM plan from 2026 to 2034 are 94% for lead, 86% for zinc and 75% for silver and are consistent with actual recoveries achieved by the plant. • A paste backfill plant and dry stack plant were constructed and commissioned in 2023 and 2025, respectively. • For 2026, it is planned that circa 31% of the tailings will be dry stacked, 36% will be used for paste backfill and 33% stored in the existing TSF-4. Paste backfill is the priority method for tailings disposal.
Environmental	• Permits held by Rudnik SASA DOOEL for the Property are sufficient to ensure that mining activities are conducted within the regulatory framework. • A hydrogeological and geochemical modelling study was undertaken by SRK in 2022. • Geochemical testwork showed the tailings to be potentially acid generating (PAG). However, static leach tests including denoised water leachate tests (DI) and net acid generation (NAG) tests generally indicated no leachable parameters of concern from the tailings. Monolithic leach tests (MLT) and humidity cell tests (HCT) showed the paste backfill to be geochemically less reactive than the tailings. Waste rock geochemical characterisation by Rudnik SASA DOOEL showed both the schist and gneiss waste rock to be PAG.

Criteria	Commentary
	• Water samples from the current operation at Sasa are tested at an external accredited laboratory, and do not show any exceedances of permit parameters, with exception of increased manganese in the TSF drainages. Legacy mining features (including adits) upstream of the current operation impact on water quality with elevated levels of manganese, zinc and cadmium and to a lesser extent lead and iron. • At mine closure, bulkheads are planned to be installed in the main drainage adits and will ensure the paste backfill or exposed footwall and hanging wall rock up to Adit 14b level will be submerged, thus removing the free oxygen required to generate ARD. Adit 14b, together with the shallower historical adits, will remain open and drain the mine. Passive water treatment technologies to remove metals from the post-closure water discharge were recommended for further investigation by SRK (2022). Waste rock transported from the underground mine to the surface is currently stockpiled on the TSF and is planned to be incorporated as part of the TSF closure. • SLR is not aware of any waste storage, environmental or permitting issues that prevent the reporting of an Ore Reserve estimate for the Svinja Reka deposit.
Infrastructure	• The high voltage (HV) power from the state grid to the underground mine is fed at 35 kV to three transformers where the voltage is stepped down to 6 kV with a capacity of 4 MVA (Megavolt-Ampere). • The process water intake structure is situated upstream of the confluence of the Kamenica River with the Kozje River and is used for the flotation circuit. Mine water and return Dry Stack Tailings (DST) water are used for the grinding circuit. • The Kamenica River is captured in a concrete diversion tunnel, constructed beneath the Tailings Storage Facility (TSF), and exits through the western abutment of TSF-4. A diversion channel also exists on the eastern side for the Petrova River. • The Sasa TSFs have successively developed downstream from TSF-1, TSF-2, TSF-3.1, TSF-3.2 to TSF-4 in the steep-sided valley of the Kamenica River. Except for TSF-4, all the other TSFs are inactive with TSF-1 and TSF-3.1 rehabilitated with soil cover and vegetation, TSF-2 and part of TSF-1 are used for construction of the DST, and TSF 3.2 is in the closure phase and is being capped and remediated. TSF-4 was designed by the Faculty of Engineering, Skopje, in March 2015 to international standards, and external reviews are carried out annually by Knight Piésold Ltd (Knight Piésold) to Global Industry Standard on Tailings Management (GISTM) standards. • The paste backfill plant and DST plant were commissioned in 2023 and 2025, respectively. • The town of Makedonska Kamenica has approximately 4,500 inhabitants. The Sasa Mine is the largest direct employer in the region with approximately 700 employees. Most employees live near the mine. • The mine has an established network of mining suppliers and contractors. • There are existing transport links, telecommunications facilities, electricity and water supply. • All major infrastructure required by the operation is in place and no additional significant infrastructure projects are currently planned. • The authors are of the opinion that there is sufficient land, water, and power for the planned mining and processing operations.
Costs	• Capital Costs are based on FY26 Budget costs that were estimated based on actual costs. Total capital costs for the LOM (2026 to 2034) are estimated at US$75.7 million. A cost of US$61 million is also estimated for closure.

Criteria	Commentary

	• Operating Costs are based on FY26 Budget costs that were estimated based on actual costs. Total operating costs for the LOM (2026 to 2034) are estimated at US$419.3 million.
	• No deleterious elements affect the lead concentrate. For the zinc concentrate, no significant deleterious elements result, although it is important to control the silica and iron content, with maximum levels of 2.5% SiO_2 and 11% Fe, respectively.
	• All financial modelling for the Ore Reserve estimate was completed in US dollars (US$).
	• Transportation and refinery treatment charges are based on current agreements.
	• There is a payment relating to the compensation for exploitation of the minerals (Concession Fee) based on the market value of the metals contained in the concentrates as follows:
	  o 4% of the market value of the metal of lead per tonne in each tonne of lead concentrate produced.
	  o 4% of the market value of the metal of zinc per tonne in each tonne of zinc concentrate produced.
	  o 4% of the market value of the metal of silver per tonne in each tonne of lead concentrate produced.
	• Metal prices are determined using the average price of the metals in the month it was produced. This Concession Fee is paid quarterly to the Ministry of Economy of the Republic of North Macedonia.
	• The silver contained in the concentrates is subject to a streaming agreement signed in 2017 with OR Resources Inc. and is invoiced separately when the silver content reaches payable levels. CAML receives US$6.665 per ounce for its silver production for the life of the Sasa Mine.

Revenue factors	• Head grades used to derive revenue were based on the LOM schedule. • Metal prices were provided by CAML and were considered by the Competent Person to be reasonable for estimating Ore Reserves. • All financial modelling for the Ore Reserve estimate was completed in US dollars (US$). • Transport, treatment charges and payabilities of concentrate sold are based on current agreements. • The principal metals are zinc, lead and silver. There are no co-products.
Market assessment	• Both lead and zinc concentrates, are sold via an annual rolling hybrid offtake contract with Traxys Europe S.A. (Traxys), to the regional and international well diversified customer base. • The commercial terms of the contracts with final customers are negotiated on an annual basis based on the prevailing market conditions with the off taker assistance. • The payable silver contained in the lead concentrate is subject to a life of mine streaming agreement with OR Resources Inc. With the assistance of the broker, on a monthly basis CAML buys and delivers the physical silver to OR Resources Inc., which is the equivalent of the payable silver sold during the calendar month to final customers. CAML receives US$6.665 per ounce for silver. • Concentrates have been sold to Traxys for many years and a customer and competitor analysis was not deemed necessary. • Price forecasts were based on metal prices provided by CAML. Volume forecasts were based on the LOM schedule. • The lead and zinc concentrates are transported from the Sasa Mine by truck.

Criteria	Commentary
Economic	• The Sasa Mine has produced at consistent rates for many years and costs and revenue are well understood. • The economic analysis was based on the Probable Ore Reserves in the LOM schedule. Inferred Mineral Resources were excluded from the economic analysis. • The economic analysis used a Discounted Cash Flow (DCF) method to estimate the Projects return based on expected future revenues, costs, and investments. Cash flows were prepared in real terms with no allowance for inflation. A discount rate of 8% was used. • The financial model for the LOM was reviewed by the authors and showed the Ore Reserves to be economic based on the assumptions used and metal prices of US$2,644 per tonne for zinc, US$2,179 per tonne for lead, and US$27 per ounce for silver. • A silver price of US$6.665 per ounce was also used in the financial model to reflect the value received by CAML based on the silver streaming agreement with OR Resources Inc. The Ore Reserves were also shown to be economic at this price. • CAML monitors key economic assumptions, operational performance and technical parameters and has undertaken sensitivity analysis on the LOM financial model for the Sasa operation to assess the impact of changes in the principal value drivers. Sensitivities were applied to zinc and lead prices of ±10%, head grades of ±10%, metallurgical recoveries of ±5%, operating costs of ±10%, capital expenditure of ±10%, treatment charges of ±20%, and discount rates ranging from 8% to 12%. The sensitivity analysis on all inputs delivered a positive NPV within the ranges tested.

Social	• Employees and external stakeholders are considered as follows:
	  ○ Employees are considered integral stakeholders to the Sasa Mine, and as such a dedicated internal communications plan has been in place since 2018, which explicitly links to the external communications plan, ensuring that consistent information is presented to employees and the community. Approximately 80% of the employees are based in the local communities. The purpose of the internal communications plan is to ensure employees are informed on upcoming changes to the mine, new initiatives (both external and internal), and have the opportunity to ask questions. There is active encouragement for employees to contact their line manager with any concerns, but also an anonymous grievance mechanism to provide anonymity should this be required.
	  ○ In 2018, Rudnik SASA DOOEL developed a Stakeholder Engagement Plan and structured community engagement programme, including establishing the Sasa Foundation, to ensure that appropriate governance is in place for administering financial support to local organisations. A formal grievance mechanism is in place, enabling communities to raise questions or concerns. These are recorded internally, and responses documented and fed back to the enquirer
	• There are no known social issues which are expected to materially impact the Ore Reserve estimate.

Other	• No major issues were identified that will materially affect the estimation or classification of the Ore Reserve estimates. • No material risks with the potential to prevent the planned production were identified. • No outstanding legal issues exist that could compromise the Ore Reserve estimate have been identified. • The current LOM at Sasa runs until 2034, the authors consider there are reasonable grounds to expect the concession to be extended beyond the current expiry of September 28, 2030. • Rudnik SASA DOOEL has all other required permits to conduct the proposed work on the Property and to continue

Criteria	Commentary
	production as planned. SLR is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work on the Property.
Classification	• Ore Reserves are classified in accordance with the guidelines of the JORC Code (2012). • No Proved Ore Reserves were classified because no Measured Mineral Resources were estimated. Probable Ore Reserves were classified based on Indicated Mineral Resources. • Inferred Mineral Resources contained within the stope designs were considered as having zero grade during the stope design process. Inferred Mineral Resources comprise around 6% of the total LOM tonnes and are mainly concentrated towards the end of the LOM schedule. Inferred Mineral Resources were excluded from the Ore Reserve estimate, and no value was assigned to them in the economic analysis. • It is the opinion of the Competent Person that the classification adequately represents the degree of confidence in the deposit. • No Probable Ore Reserves have been derived from Measured Mineral Resources.
Audits or reviews	• A review of the Sasa Mine Ore Reserve estimation procedures was undertaken by Wardell Armstrong International (WAI) in May 2025. Recommendations for further work were made; however, no fatal flaws were identified in the review.
Discussion of relative accuracy/ confidence	• It is the Competent Person's view that the quality and accuracy of the Modifying Factors are sufficient for reporting of Ore Reserves in accordance with the JORC Code (2012). • The Modifying Factors are derived from actual operating experience. • The statement relates to global estimates. • Reconciliation comparing planned production from the resource model against actual performance of the processing plant is undertaken monthly. The results of the reconciliation are accumulated annually. • On an annual basis, the reconciliations show a generally good agreement between the resource model and actual production and generally report less than 5% difference in contained Pb and Zn metal. Overall, the results of the reconciliations are considered by the Competent Persons to be within acceptable tolerances.

Making Sustainability Happen