Final

Independent Technical Report on the Mengya'a Lead and Zinc Project

Jiali County, Nagqu City, Xizang Autonomous Region, China
Xizang Zhihui Mining Co., Ltd.

SRK Consulting (Hong Kong) Ltd ▪ TZM001 ▪ 11 December 2025

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Jiali County, Nagqu City, Xizang Autonomous Region, China

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Useful definitions

This list contains definitions of symbols, units, abbreviations, and terminology that may be unfamiliar to the reader.

Abbreviation	Meaning/Definition
°	degrees
°C	degrees Celsius
µm	micrometres, equal to one millionth of a metre
AAS	atomic absorption spectroscopy
AIG	Australian Institute of Geoscientists
Anticline	An anticline is an arch-like fold in rock layers, where the oldest rocks are at the core of the fold, and the layers dip away from the centre.
ARD	acid rock drainage
asl	above sea level
AusIMM	Australasian Institute of Mining and Metallurgy
BGRIMM	Beijing General Research Institute of Mining & Metallurgy
bulk density	A physical property of mineral components, defined by the weight of an object or material divided by its volume, including the volume of its pore spaces
CGCS	China Geodetic Coordinate System
Channel sample	Sample collected by cutting a continuous groove or channel into the rock face using tools such as chisels, saws, or drills. The groove is typically uniform in width and depth to ensure consistency.
cm	centimetres
compressive	The capacity of a material or structure to withstand loads tending to reduce size, measured by plotting.
concentrate	These concentrates are produced by separating zinc-, lead- and copper-bearing minerals from other minerals and impurities found in the ore. The concentration process often involves crushing, grinding, and various separation techniques such as flotation, gravity separation, or leaching.
CRM	Certified Reference Material
drill core	A solid, cylindrical sample of rock produced by an annular drill bit, generally rotatively driven but sometimes cut by percussive methods (drill core is extracted from a drill hole).
drill hole	A hole drilled in the ground by a drill rig, usually for exploratory purposes to obtain geological information and to allow sampling of rock material.
EDTA	ethylene diamine tetraacetic acid; a polyamino carboxylic acid with the formula C10H16N2O8. It is known for its ability to sequester metal ions, forming stable complexes. This property makes it useful in applications where metal ions need to be controlled or removed.
EIA	Environmental impact assessment, a comprehensive analysis of the environmental consequences of a mining project

EPCM	Engineering, Procurement and Construction Management
EqPb (lead equivalent)	Calculated by converting the grades of zinc and silver into an equivalent lead grade, taking into account their prices and recovery rates.
exploration	Activities undertaken to prove the location, volume and quality of a deposit
fault	A fracture or fracture zone in rock along which movement has occurred
fold	A bend or flexure in a rock unit or series of rock units that has been caused by crustal movements
formation	A body of rock having a consistent set of characteristics (lithology) that distinguish it from adjacent bodies of rock
g	grams
g/cm^3	grams per cubic centimetre
g/t	grams per tonne
HDPE	high density polyethylene
HKEx	The Stock Exchange of Hong Kong Limited
IDW	inverse distance weighted
ITR	Independent Technical Report
JORC	Joint Ore Reserves Committee
JORC Code	Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC), December 2012.
JV	joint venture
k	thousand
kg	kilograms
km	kilometres
km^2	square kilometres
kt	thousand tonnes
kV	kilovolts
kW	kilowatts
m	metres
LOM	life-of-mine
M	million
m asl	metres above sea level
m/d	metres per day
m^3	cubic metres

magmatic	Pertaining to, or derived from, magma
metamorphic rock	A rock formed by transformation of existing rocks subject to elevated heat and pressure
mg	milligrams
Mineral Resource	Concentration or occurrence of material of intrinsic economic interest on or inside the Earth's crust in such form, quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a mineral resource are known, estimated or interpreted from specific geological evidence and knowledge.
mm	millimetres
MRE	Mineral Resource Estimate
Mt	million tonnes
mudstone	Mudstone is a fine-grained sedimentary rock composed primarily of clay-sized particles.
OK	Ordinary Kriging
open-pit	Mining of a deposit from a pit open to the surface and usually carried out by stripping of overburden materials (equivalent to a quarry)
Ore	Ore is a naturally formed material containing metals or minerals that can be economically extracted.
Ore Reserve	The economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.
PBX	potassium butyl xanthate
Permian	Time period 299–252 million years ago
phyllite	A type of foliated metamorphic rock created from slate that has fine-grained mica
QA/QC	Quality Assurance and Quality Control
Quaternary	Time period 2.58–0 million years ago
RCE	Registered Civil Engineer in China
Report	Independent Technical Report
RL	relative level
ROM	run-of-mine
RQD	rock quality designation
RTK	real-time kinematics
sedimentary rock	A rock formed from the accumulation and consolidation of sediment, usually in layered deposits and that may consist of rock fragments of various sizes, remains or products of animals or plants, products of chemical action or of evaporation, or mixtures of these
shale	A fine-grained sedimentary rock, formed from mud that is a mix of clay and silt

sill	A tabular sheet intrusion of molten rock (magma) that has intruded between older layers of sedimentary rock; a sill does not cut across the pre-existing formations
slope of regression	a fundamental aspect of linear regression analysis, illustrating the relationship between the independent variable (predictor) and the dependent variable (response). In the context of Kriging estimates, the slope of the regression line serves as an indicator of the quality of the estimation.
specific gravity	The ratio of its mass to the mass of an equal volume of water
SRK	SRK Consulting (Hong Kong) Limited
SRK Group	SRK Global Limited
stratigraphy	The study of sedimentary rock units, including their geographic extent, age, classification, characteristics and formation
strength	applied force against deformation in a testing machine. It is the maximum compressive stress that can be applied to a material, such as a rock, under given conditions, before failure occurs.
strike	Direction of line formed by intersection of a rock surface with a horizontal plane. Strike is always perpendicular to direction of dip.
swath plot	A swath plot is typically created by dividing the study area into parallel slices or swaths along a specific direction (e.g., north – south, east – west, or vertical). For each swath, the average estimated value and the average actual value (from sample data) are calculated and plotted against the swath position.
t	tonnes
t/a	tonnes per annum
t/d	tonnes per day
TSF	tailings storage facility
variogram model	A variogram is a graph that represents the degree of spatial dependence between sample points as a function of distance. It plots the semivariance (half the average squared difference between paired data points) against the distance separating those points.
variography	A fundamental technique in geostatistics used to analyse the spatial variability and correlation of a regionalised variable, such as mineral grades, soil properties, or any other spatially distributed data.
vein	Sheet-like body of minerals formed by fracture filling or replacement of lost rock
waste	The part of an ore deposit that is too low in grade to be of economic value at the time of mining, but which may be stored separately for possible treatment later.
wireframe	A skeletal three-dimensional model in which only lines and vertices are represented, a preliminary stage used in preparing a full three-dimensional model.

Executive summary

SRK Consulting (Hong Kong) Limited (SRK) has been commissioned by Xizang Zhihui Mining Co., Ltd. (Zhihui) to prepare an Independent Technical Report (ITR) on the Mengya'a Lead and Zinc Project in Jiali County, Xizang Autonomous Region, China. The Report has been prepared in accordance with the VALMIN Code (2015), JORC Code (2012) and the Hong Kong Stock Exchange (HKEx) Listing Rules in relation to Zhihui's initial public offering (IPO) and capital raising on HKEx.

Scope of work

The scope of work for this Report includes a review of the following technical aspects:

- Geology and Mineral Resources
- Mining and Ore Reserves
- Mineral processing
- Tailings storage facility
- Capital and operating costs
- Permitting, environmental and social considerations.

A risk assessment is also included.

Work program

SRK's work program included a review of the provided information, site visits by SRK personnel in September and November 2024, estimation of the Mineral Resource and Ore Reserve, and preparation of this Report.

Mengya Lead and Zinc Project

Lead and zinc mineralisation was first identified within the current project area in 2002 during reconnaissance work by the Second Brigade of the Xizang Autonomous Region Geological and Mineral Exploration Bureau (Second Brigade). Initial prospecting concluded in 2003, followed by a multi-phase exploration program (2004–2021) led by the Second Brigade and Xizang Huaxia Mining Co., Ltd.,(Huaxia Mining), a subsidiary of Zhihui. This program included geological mapping, geophysical surveys, trenching, and diamond core drilling. In late 2024, Zhihui conducted an infill and validation drilling campaign to verify historical exploration data.

Operational development began in 2006, when Huaxia Mining secured the initial mining licence. Open-pit mining commenced at the Pb14 deposit in 2007. The 2,000 t/d processing plant was established in October 2007 and put into operation in December 2010. By 2022, the plant's processing nameplate capacity had been upgraded to 3,000 t/d. In 2024, the exploration licence ,originally granted in 2002 was renewed multiple times and remains valid until 24 June 2026. The mining licence was initially granted in 2006 and renewed multiple times. The latest mining licence was granted in 2024 and the licence area was expanded to 4.4544 km² in February 2025, covering the open-pit and the underground development site. This new mining licence is valid until 25 November 2045.

The project's infrastructure includes a processing plant located 10 km north of the mine site, a base camp administration and accommodation complex, and two tailings storage facilities (TSFs). The open-pit mine and underground project are situated 3 km apart, at elevations of 5,000–5,300 m above sea level (asl), while the base camp administration and accommodation complex and processing plant occupy a valley 10 km north at approximately 4,600 m asl. The TSFs are located 5 km west of the camp, spanning elevations of 4,600–5,600 m asl. Construction of the underground mining system completed in May 2025 and production commenced in the second quarter of 2025. The combined output from the open-pit and underground mines is expected to deliver 400,000 t of ore annually to the processing plant, producing lead (Pb), zinc (Zn), and copper (Cu) concentrates.

Geology and mineralisation

The project is located within the Gangdese Metallogenic Belt on the southeastern Xizang Plateau in western China, a region renowned for porphyry, skarn, and epithermal polymetallic deposits. This belt formed through complex geological processes, including the northward subduction of the Neo-Tethys Ocean and subsequent collisional orogeny between the Indian and Eurasian tectonic plates.

The local geological setting is dominated by the Quesang–Songduo stratigraphic group, comprising Late Carboniferous to Jurassic sedimentary sequences. These strata consist of terrigenous clastic and carbonate rocks, which host the project's primary base metal mineralisation.

Skarn-type Pb–Zn–(Ag–Fe–Cu) deposits are widely present within the Northern Gangdese Metallogenic Belt (NGMB). A magmatic–metallogenic model explains their formation, linking mineralisation to the emplacement of sedimentary(S)-type granitic magmas derived from partial melting of ancient crust. These magmas, enriched in Pb and Zn, are critical to the metal endowment of the Mengya'a deposit and similar systems in the NGMB.

Over 30 skarn prospects and deposits have been identified within the project tenures, with known mineralisation are predominantly layered or lenticular in shape, occurring within the Laigu and Luobadui formations at limestone-siltstone contacts. The primary ore minerals include sphalerite (Zn), galena (Pb), and minor pyrrhotite, pyrite and chalcopyrite (Cu). Gangue minerals comprise typical skarn assemblages such as garnet, diopside, actinolite and calcite, reflecting metasomatic alteration processes during mineralisation.

Exploration

Exploration at the project area has progressed systematically since the discovery of base metal mineralisation in 2002. The initial phase of exploration (July 2002–2003) involved reconnaissance activities performed by the Second Brigade. Through surface excavations and geophysical surveys, the Second Brigade identified the Pb4 mineralisation, marking the first significant base metal discovery in the area.

From 2004 to 2021, comprehensive exploration programs were conducted across the project's tenure area. The Second Brigade executed a series of ground-based electromagnetic surveys in 2004, 2008, 2012 and 2017, which led to the discovery of the Pb12 deposit and led to follow-up drilling campaigns. During this period, a total of 253 diamond drill holes , totalling 60,073.68 m were completed, with 4,805 samples assayed for Pb, Zn, Cu and Ag and 428 samples tested for bulk density.

To validate the historical exploration data, Zhihui implemented a validation program in 2024, as recommended by SRK. This program included two diamond twinned holes, three trenches and 19 underground channels. Within the project's exploration licence area, over 30 prospects or deposits have been identified to date. Further exploration is warranted to evaluate the potential of these prospects.

Mineral Resources

The current Mineral Resource for the Pb12 and Pb14 deposits within the Mengya'a Project area were estimated in 2024 using Leapfrog Edge software (version 2024.1). This software was used to create geological and mineralisation domain models, prepare statistical analysis, construct block model, and estimate Pb, Zn, Cu and Ag grades. The database used for estimation purposes comprised 246 historical drill holes (59,275.98 m), three infill trenches (108 m), two twinned holes (164.95 m) and 19 underground channels (57 m), totalling 4,891 assay samples. Wireframe models for these two deposits were developed using Leapfrog's vein selection and domain functions to define contacts between mineralisation and host rock. A total of 37 domains were constructed for the Pb12 and Pb14 deposits. Mineralised intervals were delineated using cut-off grades: Pb ≥0.5% or Zn ≥1.0% for Pb12, and Pb ≥0.3% or Zn ≥0.5% for Pb14. Raw assay data were composited into 1.5 m-wide intervals, with a 39% Pb grade cap applied to mitigate outlier influence. No capping was applied for Zn, Cu or Ag.

Ordinary Kriging (OK) was primarily used for grade estimation purposes. Variogram models from six key domains (e.g. D12103, D14102) were applied to 31 other domains considered to have similar mineralisation trends but insufficient data able to support standalone variograms. For domains with distinct mineralisation trends, the Inverse Distance Weighted (IDW) method was used. The block model was constructed with a parent block size of 20 mE × 20 mN × 4 mZ and a sub-block size of 2 mE × 2 mN × 2 mZ, without rotation.

As at 31 July 2025, the depleted resources from the existing Pb14 deposit open pit have been excluded. For the Pb12 deposit, wireframe modelling and deductions have been carried out for the underground exploration adit areas. Resource classification criteria included geological confidence, data quality, and geostatistical reliability, with the following criteria applied according to classification: Measured (sampling ≤50 m or regression slope >0.85), Indicated (sampling ≤100 m or regression slope >0.6), and Inferred (sampling >100 m or IDW-estimated domains).

Applied cut-off grades were derived from historical operational data. The Mineral Resource estimate within the mining licence was prepared in accordance with the JORC Code (2012) with an effective date of 31 July 2025 as presented in Table ES 1, excluding sterilised and mined-out areas.

Table ES 1: Mineral Resource Statement for the Mengya'a Project as at 31 July 2025

Domain	Cut-off	Category	Tonnage (kt)	Pb Grade (%)	Zn Grade (%)	Cu Grade (%)	Ag (g/t)	Pb Metal (kt)	Zn Metal (kt)	Cu Metal (kt)	Ag Metal (t)
Pb12 deposit	EqPb ≥4.7% (oxide) EqPb ≥2.9% (fresh)	Measured	6,299	2.81	4.73	0.25	42	176.73	298.02	15.82	264
		Indicated	6,027	4.65	4.94	0.26	41	280.30	297.88	15.66	249
		Inferred	2,583	3.23	3.70	0.25	40	83.42	95.43	6.49	104
		Subtotal	14,909	3.63	4.64	0.25	41	540.45	691.33	37.97	617
Pb14 deposit	EqPb ≥1.4%	Measured	427	1.13	4.64	0.11	15	4.84	19.80	0.45	7
		Indicated	1,172	0.52	5.29	0.11	9	6.10	62.00	1.28	10
		Inferred	305	0.77	5.32	0.17	17	2.35	16.24	0.51	5
		Subtotal	1,904	0.70	5.15	0.12	12	13.28	98.04	2.24	22
Total		Measured	6,726	2.70	4.73	0.24	40	181.57	317.82	16.27	270
		Indicated	7,198	3.98	5.00	0.24	36	286.40	359.88	16.94	259
		Inferred	2,888	2.97	3.87	0.24	38	85.77	111.67	7.00	110
		Total	**16,813**	**3.29**	**4.70**	**0.24**	**38**	**553.73**	**789.37**	**40.21**	**639**

Notes:

1 Any differences between totals and sum of components are due to rounding.

2 EqPb 4.7% (oxide) and EqPb 2.9% (fresh) cut-off grades were applied to the resource block models of the Pb12 deposit. An EqPb 1.4% cut-off grade was applied to the resource block models of the Pb14 deposit.

3 Mineral Resources are not Ore Reserves and do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

4 Mineral Resources are reported inclusive of Ore Reserves.

5 The Mineral Resources are effective as at 31 July 2025.

6 Lead equivalent (EqPb) formulas were applied: fresh ore: EqPb = Pb + 1.1457 × Zn + 2.5464 ×Cu + 0.0296 × Ag and oxide ore: EqPb = Pb + 1.3315 × Zn + 2.7501 × Cu + 0.0062 × Ag.. Metal price assumptions included 18,600 RMB/Pb t for lead concentrate, 21,100 RMB/Zn t for zinc concentrate, 81,500 RMB/Cu t for copper concentrate, and 8.10 RMB/g for silver. The recovery assumptions for fresh ore are as follows: zinc recovery is 91.0%, copper recovery is 52.0%, silver recovery in the copper concentrate is 9.2%, and silver recovery in the lead concentrate is 66.4%. For oxide ore, the recovery assumptions are zinc recovery at 66.1%, copper recovery at 35.1%, silver recovery in the copper concentrate at 10.4%, and silver recovery in the lead concentrate at 57.3%.

Competent Person's Statement: The information in this Report that relates to Mineral Resources is based on information compiled by Dr (Tony) Shuangli Tang who is a Member of the Australian Institute of Geoscientists (AIG) and a Member of the Australasian Institute of Mining and Metallurgy (AusIMM). Dr Tang is a full-time employee of SRK Consulting (Hong Kong) Limited and has sufficient experience that is relevant to the style of mineralisation, type of deposit under consideration and to the activity which he under takes to qualify as a Competent Person as defined in the 2012 edition of the *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* (the JORC Code).

Mining

There are two base metal deposits defined within the Mengya'a mine tenures. The Pb14 deposit currently operates as an open-pit mine (known as Pb14 OP), while the Pb12 deposit is planned for underground mining operations (known as Pb12 UG), which commenced production in the second quarter of 2025.

Pb14 Open-Pit

The Pb14 Open-Pit mine operates using a conventional truck and shovel method, with operations scheduled for 200 days per year and mining activities carried out as a contractor operation.

Material is blasted and subsequently loaded into trucks using 2 m^3 hydraulic excavators. Run-of-mine (ROM) ore is transported by 25-t trucks to the processing plant and the distance from the mining area to the processing plant is approximately 11 km. The mineralised waste is hauled by 25-t trucks to the mineralised waste dump, which is located approximately 600 m northeast of the open pit, on the northern side.

The average mining dilution rate is 5% and the ore loss is 5%.

The open-pit life-of-mine (LOM) is scheduled over 8 years, starting in July 2025 with production of 200 kt/a.

Pb12 Underground

The Pb12 Underground operations are designed for a production capacity of 200 kt/a in alignment with the Pb14 Open-Pit operations. Upon the depletion of resources at Pb14 OP, the Pb12 UG is scheduled to ramp up production to 400 kt/a to ensure a consistent feed to the processing plant.

The mining sequence for Pb12 UG, from top to bottom, will consist of two phases: Stage I involves mining above the 5,100 m RL, with 5,180 m RL as the pilot mining level; Stage II involves mining below the 5,100 m RL

The development system for the Pb12 UG will utilise existing exploration adits. A decline system will be constructed to connect levels, where a level adit is not available.

The expected LOM for the underground is 31 years. Room and pillar with delayed backfill method and sublevel open stoping (longitudinal or transverse) are nominated for the Pb12 UG.

Ore Reserves

SRK has estimated the Ore Reserves for the Mengya'a mine in accordance with the guidelines of the JORC Code (2012). The Ore Reserve estimates are supported by technical studies and ongoing operational records, which are considered to be at the level of a pre-feasibility study (PFS).

The Mengya'a mine comprises two distinct deposits: Pb14 OP and Pb12 UG. The economically mineable portions of the Measured and Indicated Mineral Resources within the designed stopes or the designed open-pit, inclusive of diluting materials and allowances for losses, have been classified as Proved and Probable Ore Reserves, respectively. The estimation of feed ore is determined with the reference point at the stockpile at the crusher feed.

As at 31 July 2025, Pb14 OP contains total Ore Reserves of 1,438 kt at an average grades of 0.69% Pb, 4.90% Zn, 0.10% Cu and 10.09 g/t Ag. (Table ES 2). All Ore Reserves are fresh material.

Table ES 2: Ore Reserves Statement for Pb14 OP at Mengya'a mine as at 31 July 2025

Category	Ore Reserve (kt)	Pb Grade (%)	Zn Grade (%)	Cu Grade (%)	Ag Grade (g/t)	Pb Metal (kt)	Zn Metal (kt)	Cu Metal (kt)	Ag Metal (t)
Proved	400	1.14	4.54	0.10	14.51	4.57	18.16	0.41	5.80
Probable	1,038	0.52	5.04	0.10	8.39	5.41	52.27	1.07	8.71
Total	**1,438**	**0.69**	**4.90**	**0.10**	**10.09**	**9.99**	**70.43**	**1.48**	**14.51**

Source: SRK

Notes:

[1] The cut-off grades used to distinguish ore from waste are set at EqPb ≥1.8%.

[2] The mining dilution rate is 5% and the ore loss is 5%.

[3] The Ore Reserves are reported on a dry metric tonne basis.

[4] The reference point for reporting of Ore Reserves is the stockpile at the crusher feed.

[5] The Mineral Resources are effective as at 31 July 2025.

As at 31 July 2025, the Pb12 UG contains total Ore Reserves of 10,623 kt at average grades of 2.99% Pb, 4.14% Zn, 0.21% Cu and 35.00 g/t Ag (Table ES 3). The Ore Reserves are divided into fresh and oxide materials.

Table ES 3: Ore Reserves Statement for Pb12 UG at Mengya'a mine as at 31 July 2025

Type	Category	Ore Reserve (kt)	Pb Grade (%)	Zn Grade (%)	Cu Grade (%)	Ag Grade (g/t)	Pb Metal (kt)	Zn Metal (kt)	Cu Metal (kt)	Ag Metal (t)
Oxide	Proved	32	5.69	3.34	0.31	33.94	1.80	1.06	0.10	1.07
	Probable	427	4.96	2.74	0.18	26.82	21.17	11.71	0.78	11.45
	Subtotal	458	5.01	2.79	0.19	27.31	22.97	12.77	0.88	12.52
Fresh	Proved	4,728	2.91	5.12	0.26	44.30	137.65	241.90	12.34	209.47
	Probable	5,436	2.88	3.41	0.17	27.56	156.82	185.58	9.38	149.82
	Subtotal	10,165	2.90	4.21	0.21	35.35	294.47	427.48	21.72	359.29
Total	Proved	4,760	2.93	5.10	0.26	44.23	139.44	242.95	12.44	210.55
	Probable	5,863	3.04	3.37	0.17	27.51	177.99	197.30	10.16	161.27
	Total	**10,623**	**2.99**	**4.14**	**0.21**	35.00	317.43	440.25	22.60	371.81

Source: SRK

Notes:

[1] The cut-off grades used to distinguish ore from waste are set at EqPb of ≥6.0% for oxide and ≥3.7% for fresh material in the Pb12 UG.

[2] The Ore Reserves are reported on a dry metric tonne basis.

[3] The reference point for reporting of Ore Reserves is the stockpile at the crusher feed.

[4] The Mineral Resources are effective as at 31 July 2025.

Competent Person's Statement: The information in this Report that relates to Ore Reserve is based on information compiled by Mr Falong Hu who is a Fellow of the Australasian Institute of Mining and Metallurgy (AusIMM). Mr Hu is a full-time employee of SRK Consulting (China) Limited and has sufficient experience that is relevant to the style of mineralisation, type of deposit under consideration and to the activity which he undertakes to qualify as a Competent Person as defined in the 2012 edition of the *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* (the JORC Code).

Processing

The Mengya'a concentrator was established in 2010 with an initial processing capacity of 2,000 t/d and 200 effective working days per year. The technical foundation of the processing plan is based on the mineralogy research and metallurgical testing program conducted by the Beijing General Research Institute of Mining and Metallurgy in 2007. In December 2016, the Guangdong Metallurgical and Architectural Design Institute completed a feasibility study to increase the processing capacity to 3,000 t/d. Following several technological upgrades, the processing nameplate capacity reached 3,000 t/d in 2022.

The processing plant employs a traditional three-stage, closed-circuit crushing process followed by a two-stage, closed grinding circuit. The mill feed ore is crushed and ground to the particle size of 70–75% passing approximately 74 µm. The Cu-Pb mixed flotation, separation flotation and Zn flotation process produces saleable copper, lead, and zinc concentrates, with silver being concentrated in the copper and lead concentrates. A mild reagent system is applied to mitigate the impact of recycled water on the flotation process and to ensure full utilisation of recycled water. With regular metallurgical tests, various technological upgrades and operational optimisations, the metallurgical performance of the plant has been continuously improving over the past few years. Table ES 4 presents flotation test indicators, actual production indicators and forecast indicators.

Table ES 4: Results of processing test, historical production and future forecast

Performance parameters		Unit	[1]Test	2022	2023	2024	Forecast
Copper Concentrate Grade	Cu	%	21.95	18.67	19.62	19.78	20.0
	Ag	g/t	355.0	559.0	641.6	725.3	600
Copper Concentrate Recovery	Cu	%	54.97	43.16	53.26	49.89	52.0
	Ag	%	6.48	7.53	10.77	10.89	9.2
Lead Concentrate Grade	Pb	%	60.11	57.44	62.76	60.12	60.0
	Ag	g/t	401.0	594.1	784.5	815.7	700
Lead Concentrate Recovery	Pb	%	87.80	85.71	89.54	88.14	88.0
	Ag	%	58.22	64.68	66.63	75.53	66.4
Zinc Concentrate Grade	Zn	%	48.12	45.75	46.08	44.64	46.0
Zinc Concentrate Recovery	Zn	%	90.54	90.03	91.91	94.81	91.0
Raw Feed Grade	Cu	%	0.20	0.18	0.25	0.09	0.20
	Pb	%	2.72	2.24	2.42	0.92	2.20
	Zn	%	4.92	3.95	4.33	4.29	4.00
	Ag	g/t	27.41	30.75	40.59	15.36	34.0

Source: Huaxia

Note: [1] Processing test results of Guangdong Institute of Comprehensive Utilization of Resources (Guangdong RCUI)

Tailings Storage Facilities

Currently, the processing plant has been in operation since 2010 and the resulting tailings is classified as Category I general industrial solid waste. The tailings stream has a particle size of 65.27% passing -200 mesh (-74 μm). After being concentrated to a slurry weight density of 32.5%, the tailings slurry is pumped to the tailings storage facility (TSF) for disposal.

The currently active TSF lies in a valley 5.1 km southwest of the processing plant and was commissioned in June 2011. An upstream tailings dam was constructed for the disposal of wet tailings. The starter embankment of the tailings dam has a height of 40 m, with a perimeter embankment height of 36 m, resulting in a total embankment height of 76 m. The final design elevation for the tailing dam is 5,036 m asl, with a total storage capacity of 2.97 million m^3. The catchment area spans 4.83 km^2. The base of the dam is lined with a geotechnical membrane liner for seepage control. In addition, flood diversion ditches and an internal drainage system (i.e. inclined drainage channels and culverts) are in place. Tailings water is recycled to the plant via gravity flow through the drainage system.

The existing TSF is expected to reach full capacity in 2026, after which a new facility will be commissioned. The new TSF is scheduled to be completed in 2026 and is located in an adjacent valley. It uses the upstream method for construction. The initial construction phase involves a 53 m-high permeable mixed-material dam, with a later stage 100 m-high tailings dam, reaching a total dam height of 153 m. The final design elevation is 5,210 m asl, with a total storage capacity of 21.2 million m^3. The catchment area covers 3.64 km^2. Key infrastructure includes an upstream flood control dam combined with a spillway tunnel, western interception ditches, an internal drainage system (inclined channels and tunnels), and a geotechnical membrane liner system. The service life of the new TSF is estimated to be 64 years at the current annual production capacity of 400,000t.

Both TSFs are equipped with an online and manual monitoring system to track pond level, beach length, phreatic line, embankment deformation, tailings discharge rates, and precipitation. Monitoring data are integrated into the mine network, with predefined control thresholds and regulatory protocols.

SRK has reviewed both facilities and concluded that the tailings disposal plans and infrastructure configurations are technically reasonable, with identified risks assessed at a low level and controllable.

Permitting, environmental and social considerations

SRK reviewed the licences and permits provided by the Company and concluded that the project has obtained the key environmental and operational permits required for its ongoing operation, including the Safety Production Permit, Water Abstraction Permit and Pollutant Discharge Permit. In addition, the Company provided three Environmental Impact Assessment (EIA) reports, which cover the mining area, processing plant, and TSFs, including both existing operations and proposed projects. These EIA reports were prepared in accordance with Chinese laws and regulations and subsequently approved by the relevant government authorities.

There are no registered scenic spots, nature reserves, forest parks, or other ecologically sensitive areas within the current project's tenures. Although the construction may cause localised loss of the original vegetation, it is not expected to lead to changes in the species composition of the plant communities in the evaluation area, nor will it cause the extinction of any plant species. The water sources for the mine's production and domestic use are a nearby river and mountain springs, respectively. The current wastewater sources for the project include mine dewatering water, leachate from mineralised waste dumps, leachate from raw ore stockpiles, processing wastewater, and domestic wastewater. The EIA reports recommend that all the wastewater be recycled for production use. During the exploration phase of the Pb12 deposit, several adit mineralised waste dumps were formed. In addition, there are other mineralised waste dumps on site to store mineralised waste generated from the previous mining at the Pb14 OP and the construction of the Pb12 UG. According to the EIA reports, the mineralised waste and tailings from the project are classified as Category I general industrial solid waste.

Public participation was conducted during the EIA process. No objections from the public were received during the announcement period.

Capital and operating costs

The Project's capital and operating costs reflect significant investments in infrastructure and operational activities. Historical capital cost from 2022–July 2025 totalled RMB 446.4 million, primarily driven by the construction of the new TSF and Pb12 underground development. Forecast capital cost for August 2025–2035 highlights phased investments, peaking at RMB 68.6 million during the period of August-December 2025, followed by a gradual reduction to RMB 18.9-56.2 million annually, focusing on sustaining capital such as equipment upgrades.

Operating costs between 2022–2024 increased from RMB 224.5 million in 2022 to RMB 249.9 million in 2023, driven by higher mining costs, before declining to RMB 130.9 million in 2024 due to reduced open-pit mining volumes. Forecast operating costs for August 2025–2035 range from RMB 140.3 million to RMB 285.4 million annually, with mining activities dominating expenditures. Unit costs for copper, lead, and zinc concentrates are influenced by variable grades and energy consumption.

1 Introduction

1.1 Background

SRK Consulting (Hong Kong) Limited (SRK) is an associate company of the international group holding company, SRK Global Limited (the SRK Group). SRK was commissioned by Xizang Zhihui Mining Co., Ltd. (Zhihui or the Company) to prepare an Independent Technical Report (ITR or the Report) relating to the Mengya'a Lead and Zinc Project located near Rongdoi (town), Jiali County, Nagqu City, in the Xizang Autonomous Region.

The project comprises an actively producing open-pit mine and an underground project, which is currently under development. Other key assets include a base camp administration and accommodation complex, a processing plant and two tailings storage facilities (TSFs). An aerial view of the site layout is presented in Figure 3.1. The assets are clustered within a 10 km radius at elevations of 4,600–5,600 m asl. The underground project lies 3 km east of the open-pit mine, and both mining sites are located at elevations of 5,000–5,300 m asl. The base camp and processing plant are situated in a valley 10 km north of the mining sites at ~4,600 m asl, while the TSFs occupy a valley 5 km west of the camp and plant, at elevations spanning 4,600–5,600 m asl.

The open-pit operation has been in production since 2007. The construction of the underground mining system completed in May 2025, and underground production commenced in the second quarter of 2025. The combined production from the open-pit and underground operations is estimated to supply a total of 400,000 tonnes of ore annually to the processing plant. The ore is processed to produce lead (Pb), zinc (Zn) and copper (Cu) concentrates. The project comprises a granted exploration licence that surrounds both the open-pit mine and the underground development project.

1.2 Purpose of the Report

This Report has been prepared by SRK for inclusion in the prospectus to be published by the Company in connection with the initial public offering (IPO) of shares in the Company and associated capital raising on the Main Board of the Hong Kong Stock Exchange (HKEx).

1.3 Scope of work

The scope of work for this Report includes a review of the following technical aspects:

- geology and Mineral Resources
- mining and Ore Reserves
- mineral processing
- tailings storage facility
- capital and operating costs
- permitting, environmental and social considerations.

A risk assessment is also included.

1.4 Work program

SRK's work program completed under this commission included:

- review of supplied information
- site visit by SRK consultants and associates
- estimation of Mineral Resources and Ore Reserves in accordance with the JORC Code (2012)
- preparation of this Report.

1.5 Reporting standard

Key authors of this Report are Members or Fellows of either the Australasian Institute of Mining and Metallurgy (AusIMM) and/or the Australian Institute of Geoscientists (AIG) or other international Recognised Professional Organisations. These authors are bound by international mineral reporting codes, namely the VALMIN and JORC codes.

For the avoidance of doubt, this Report has been prepared according to:

- 2015 edition of the *Australasian Code for Public Reporting of Technical Assessments and Valuations of Mineral Assets* (VALMIN Code)
- 2012 edition of the *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* (JORC Code).

In accordance with the stated reporting guidelines, all geological and other relevant factors defining the Company's Exploration Results, Exploration Targets, Mineral Resources and Ore Reserves have been considered in sufficient detail to serve as a guide for future exploration and development activities. Table 1 of the JORC Code has been used as a checklist during the preparation of this Report and any comments are provided on an 'if not, why not' basis to ensure clarity to an investor on whether aspects of the future development program have been considered as they apply to the JORC Code (2012) Table 1.

The criteria of the JORC Code Table 1 reflect the normal systematic approach to exploration and target evaluation. *Relevance* and *Materiality* are overriding principles which determine the information that needs to be publicly reported. This Report has attempted to provide sufficient comment on all matters that might materially affect a reader's understanding or interpretation of the results being reported. The criteria under which the project is being evaluated are consistent with the current understanding of the geological controls on the known mineralisation, but as more knowledge is gained, these criteria could change and be improved over time.

As per the VALMIN Code (2015), a draft of the Report was supplied to Zhihui to check for material error, factual accuracy and omissions before the final version of the Report was issued.

1.6 Effective Date and Report Date

The Effective Date for this Report is 31 July 2025. The Report Date is 11 December 2025.

The Mineral Resource and Ore Reserve statements set out in this Report are reported as at 31 July 2025.

1.7 Units, projection and currency

Throughout this Report, SRK has used the International System of Units. All units used in this Report are defined under Useful definitions.

All monetary values used in this Report are in Chinese Renminbi (RMB).

Unless otherwise specified, all coordinates in this report were in China Geodetic Coordinate System (CGCS) 2000/3-degree Gauss-Kruger zone 31 datum.

1.8 Limitations

SRK's opinion contained herein is based on information provided by Zhihui throughout the course of SRK's investigations, which in turn reflects various technical and economic conditions at the time of writing. SRK has taken the technical information as provided by Zhihui in good faith.

This Report includes technical information, which requires subsequent calculations to derive subtotals, totals, averages and weighted averages. Where such calculations involve a degree of rounding, SRK does not consider such rounding to be material.

The input, handling, computation, and output of the geological data and Mineral Resource and Ore Reserve information has been conducted professionally and accurately and to the high standards commonly expected within the geoscience profession

In conducting this assessment, SRK has assessed and addressed all activities and technical matters that might reasonably be considered to be relevant and material to such an assessment conducted to internationally accepted standards. Based on observations, interviews with appropriate staff and a review of available documentation, SRK is, after reasonable enquiry, satisfied that there are no outstanding relevant material issues other than those indicated in this Report. However, it is impossible to dismiss absolutely the possibility that parts of the site or adjacent properties may give rise to additional issues.

The conclusions presented in this Report are professional opinions based solely on SRK's interpretations of the documentation received, interviews and conversations with personnel knowledgeable about the site, and other available information, as referenced in this Report. These conclusions are intended exclusively for the purposes stated herein.

1.9 Legal matters

SRK has not been engaged to comment on any legal matters. SRK is not qualified to make legal representations as to the ownership and legal standing of the mineral tenements that are the subject of this Report. SRK has not attempted to confirm the legal status of the mineral titles, joint venture (JV) agreements, local heritage or potential environmental or land access restrictions. SRK understands such matters are discussed elsewhere within Zhihui's prospectus.

1.10 Reliance on other experts

SRK has not performed an independent verification of the mining licence and land titles nor the legality of any underlying agreements that may exist concerning the permits, commercial agreements with third parties or sales contracts and instead has relied on information as provided to SRK by Zhihui's independent legal advisers.

The commodity price forecasts used in this Report for economic evaluation purpose is provided by Zhihui's industry expert, Shanghai Metals Market (SMM), an independent market research and consulting company.

1.11 Warranties

Zhihui has represented in writing to SRK that full disclosure has been made of all material information and that, to the best of its knowledge and understanding, such information is complete, accurate and true.

1.12 Indemnities

As recommended by the VALMIN Code (2015), Zhihui has provided SRK with an indemnity under which SRK is to be compensated for any liability and/or any additional work or expenditure resulting from any additional work required:

- which results from SRK's reliance on information provided by Zhihui or Zhihui not providing material information; or
- which relates to any consequential extension workload through queries, questions or public hearings arising from this Report.

1.13 Statement of SRK independence

Neither SRK, nor any of the authors of this Report, has any material present or contingent interest in the outcome of this Report, nor any pecuniary or other interest that could be reasonably regarded as capable of affecting their independence or that of SRK. SRK has no beneficial interest in the outcome of this Report capable of affecting its independence.

1.14 Corporate capability

SRK is an independent, international group providing specialised consultancy services. Among SRK's clients are many of the world's mining companies, exploration companies, financial institutions, engineering, procurement and construction management (EPCM) and construction firms, and government bodies.

Formed in Johannesburg in 1974, the SRK Group now employs some 1,700 staff internationally in over 40 permanent offices in 20 countries on 6 continents. A broad range of internationally recognised associate consultants complements the core staff.

The SRK Group's independence is ensured by the fact that it is strictly a consultancy organisation, with ownership by staff. SRK does not hold equity in any projects or companies. This permits SRK's consultants to provide clients with conflict-free and objective support on crucial issues.

1.14.1 Stock exchange public report

SRK has prepared many public reports for the HKEx. Selected examples are listed in Table 1.1.

Table 1.1: Public reports prepared by SRK for disclosure on the HKEx

Company	Year	Project Name
Anhui Jinyan Kaolin New Materials	2025	Listing on HKEx
Zijin Gold International	2025	Listing on HKEx
Jiaxin International Resources Investment	2025	Listing on HKEx
Xinjiang Xinxin Mining Industry	2025	Major acquisition
Chifeng Jilong Gold Mining	2025	Listing on HKEx
Persistence Resources Group	2024	Listing on HKEx
Huaibei GreenGold	2023	Listing on HKEx
China Graphite	2022	Listing on HKEx
Pizu Group	2020	Major acquisition
Heaven-Sent Gold Group	2019	Listing on HKEx
China Unienergy	2016	Listing on HKEx
China Mining Resources	2016	Major acquisition
Agritrade Resources	2015	Major acquisition
Feishang Non-metals	2015	Listing on HKEx
Future Bright Mining	2014	Listing on HKEx
Hengshi Mining	2013	Listing on HKEx
Jinchuan Group International	2013	Major acquisition
China Daye Non-Ferrous	2012	Very substantial acquisition
MMG	2012	Very substantial acquisition
China Nonferrous Metal Mining	2012	Listing on HKEx
China Hanking Holdings	2011	Listing on HKEx
CNNC International	2010	Major acquisition
Sino Prosper	2010	Major acquisition
United Company RUSAL	2010	Listing on HKEx

Source: SRK

1.15 Project team

This Report has been prepared by a team of SRK consultants and associates from SRK's offices in Hong Kong, Beijing, Almaty and Brisbane. The qualifications and experience of the consultants and associates who carried out the work in this Report are listed in Table 1.2. They have extensive experience in the mining industry and are members in good standing of appropriate professional institutions.

Table 1.2: Details of qualifications and experience of the project team

Specialist	Position/Company	Responsibility	Length and type of experience	Site inspection	Professional designation
(Gavin) Heung Ngai Chan	Principal Consultant/ SRK Hong Kong	Project Management, capital and operating costs and report compilation	20 years – 17 years in consulting specialising in valuation, financial modelling, project evaluation, geological modelling and resource estimation; 3 years in academia	4-7 November 2024	BSc, MPhil, PhD (Earth Sciences), GradDip (AppFin), GradCert (Geostats), FAIG
(Tony) Shuangli Tang	Senior Consultant/ SRK Hong Kong	Geology and Mineral Resource	11 years – 2 years in exploration geology and valuation; 9 years in consulting specialising in geological modelling and resource estimation	2-5 September 2024	BSc, MSc, PhD, MAusIMM, MAIG
Falong Hu	Principal Consultant/ SRK China	Mining and Ore Reserve	17 years – 3 years in mining engineering; 14 years in consulting specialising in mine planning, technical studies and Ore Reserve estimation	4-7 November 2024	MBA, BEng, FAusIMM
Lanliang Niu	Principal Consultant/ SRK China	Mineral processing	39 years – 20 years in academic research and gold and rare earth mineral processing; and 19 years in consulting specialising in mineral processing	4-7 November 2024	Beng, MAusIMM
Nan Xue	Principal Consultant/ SRK China	Environmental and Social	19 years – 19 years in consulting specialising in environmental impact assessment and environmental technical studies	No site visit	BSc, MSc, MBA, MAusIMM
Chunfu Yang	Associate Principal Consultant/ SRK Hong Kong	Tailings storage facilities	50 years – 5 years as a lecturer in university, 31 years in engineering and technical management in mine tailings, 14 years in consulting specialising in tailings engineering	4-7 November 2024	BEng, RCE, CSPM
Robin Simpson	Principal Consultant/ SRK Kazakhstan	Peer review – Geology and Mineral Resource	28 years – 7 years in mine and exploration geology, 21 years in consulting specialising in geological modelling and resource estimation	No site visit	BSc(Hons), MSc, MAIG
Jeames McKibben	Principal Consultant/ SRK Australasia	Overall Report	31 years in consulting specialising in valuation and corporate advisory; 2 years as an analyst; 8 years in exploration and project management roles	No site visit	BSc(Hons), MBA, MRICS, FAusIMM(CP), MAIG

Source: SRK，2025

1.16 Consent

SRK consents to this Report being included, in full, in Zhihui's HKEx listing documents in the form and context in which it is provided and not for any other purpose. SRK provides this consent on the basis that the Technical Assessment expressed in the Executive summary and in the individual sections of this Report is considered with, and not independently of, the information set out in the complete Report.

Practitioner Consent

The Competent Person who has overall responsibility for the preparation of this Report is Dr (Gavin) Heung Ngai Chan. Dr Chan is a Fellow of the AIG and a full-time employee of SRK Consulting (Hong Kong) Limited. Dr Chan has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined under the 2012 edition of the JORC Code.

The Competent Person who has overall responsibility for Mineral Resource is Dr (Tony) Shuangli Tang. Dr Tang is a Member of the AIG, a full-time employee of SRK Consulting (Hong Kong) Limited. Dr Tang has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined under the 2012 edition of the JORC Code. Dr Tang consents to inclusion in the Report of the Mineral Resources in the form and context in which they appear.

The Competent Person who has overall responsibility for Ore Reserves is the Mr Falong Hu. Mr Hu is a Fellow of the AusIMM and a full-time employee of SRK Consulting (China) Limited. Mr Hu has sufficient experience relevant to the style of mineralisation, type of deposit under consideration, and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 edition of the JORC Code. Mr Hu consents to the inclusion in the Report of Ore Reserves in the form and context in which they appear.

HKEx requirements

Dr (Gavin) Heung Ngai Chan meets the requirements of the Competent Person, as set out in Chapter 18 of the Hong Kong Stock Exchange Listing Rules. Dr Chan is a Fellow of good standing of AIG; has more than 5 years' experience relevant to the style of mineralisation and type of deposit under consideration; is independent of the issuer applying all the tests in sections 18.21 and 18.22 of the Listing Rules; does not have any economic or beneficial interest (present or contingent) in any of the reported assets; has not received a fee dependent on the findings of this ITR; is not officer, employee of a proposed officer for the issuer or any group, holding or associated company of the issuer; and takes overall responsibility for the ITR.

2 Mengya'a Lead and Zinc Project

2.1 Location

The project comprises an open-pit mine and an underground project which is currently under development. The construction of the underground project is scheduled to be completed in May 2025, with target production in the second half of 2025. Other key assets include a base camp complex, a processing plant and two TSFs. These assets are clustered within a 10 km radius, at elevations, spanning at spanning 4,600–5,600 m asl.

The project is located near Rongdoi township, Jiali County, Nagqu City, in the Xizang Autonomous Region of China, approximately 120 km northeast of Lhasa (Figure 2.1).

Figure 2.1: Project location map

Source: SRK

Note: CGCS 2000 datum.

2.2 Access, climate and physiography

2.2.1 Accessibility

Access from Lhasa to the project is approximately 165 km traveling via the G4218 Ya'an–Yecheng Expressway to Mozhugongka County and then continuing on National Highway 349 (Figure 2.2).

The nearest international airport is Lhasa Gonggar Airport, located approximately 62 km southwest of the city centre of Lhasa. It offers regular flights to major cities in China and neighbouring countries. The distance from Lhasa Gonggar Airport to the project site is approximately 180 km.

Figure 2.2: Project access map

Sources: SRK, ESRI, 2025.

Note: CGCS 2000 datum.

2.2.2 Climate

The local climate is characterised by a harsh plateau alpine environment, with extreme daily temperature variations and sharply divided seasons. A brief, humid rainy season extends from June to August bringing heavy rain, sleet and hail, while the prolonged dry season, spanning September to May, is cold and windy, and marked by frequent snowfall in autumn and winter. From November to March or April, heavy snowfall and sub-zero temperatures freeze and dry up surface streams. Average annual temperatures range between -3.3°C and -0.9°C, with January being the coldest month (-12.5°C) and July the warmest (9.8°C). Annual rainfall averages 300–500 mm, and relative humidity ranges between 48% and 51%.

The region's high elevation results in low oxygen levels, increasing the risk of altitude sickness, while intense ultraviolet radiation increases sunburn risk. This climate requires that operations be equipped to handle rapid weather shifts, extreme cold, and altitude-related health challenges.

2.2.3 Physiography

The project is located on the southern slope of the Nyainqêntanglha Mountains, upstream of the Lhasa River, within a high-altitude, mountainous region. The area is characterised by expansive, towering mountains with significant erosional features. Elevations in the project area range from 4,800 m to 5,570 m. The western side of the project area is marked by the Gento Peak, the region's highest elevation (5,570 m). In contrast, the northern Mengya'a valley lies at an elevation of 4,800–4,900 m, resulting in a vertical relief of up to 770 m. The erosion base level of the area is approximately 4,806 m, with the mineralisation cropping out at elevations ranging from 4,950 m to 5,400 m.

The area is primarily traversed by rivers flowing in north-south, northwest-southeast, and northeast-southwest directions, forming a dendritic river system in the upper reaches of Mengya'a. These rivers flow from south to north, eventually merging with Rezheng Zangbu, a tributary of the Lhasa River. The project area's water supply is primarily from atmospheric precipitation and snowmelt.

2.3 Power supply

Primary power is sourced from the Tangjia 110 kV substation, located 80 km from the site, via two self-constructed 35 kV double-circuit transmission lines. These lines terminate at a dedicated 35 kV substation within the processing plant, which houses two 6.3 MVA step-down transformers to regulate voltage for downstream distribution.

From the 35 kV substation, power is distributed through a 10 kV prefabricated substation. This substation feeds four outgoing 10 kV circuits to the processing plant, while two additional 10 kV sectionalised distribution lines supply the Pb14 OP operations. The Pb12 UG workings are energised via a T-tapped connection from the Pb14 OP 10 kV line, ensuring coordinated power delivery to both surface and subsurface operations.

Domestic power for the processing plant, living areas, and base camp complex is sourced independently from the Rongdoi township 10 kV rural distribution grid, providing segregated residential and operational power networks.

Surface mine infrastructure incorporates three dedicated substations: the 5100 Adit Central Substation, the 5210 Fan Substation and the 5100 Adit Entrance Power Substation. Underground, the 5100 adit is serviced by a low-voltage substation that delivers 380 V three-phase power for heavy equipment and 220 V single-phase power for lighting across all adit levels.

The system employs a dual-circuit, sectionalised power supply architecture, which does not rely on diesel generators. This configuration ensures redundancy, minimises downtime, and guarantees uninterrupted power availability for both surface and underground operations.

SRK considers the current power supply setup to have sufficient capacity and stability to support both current and proposed operations.

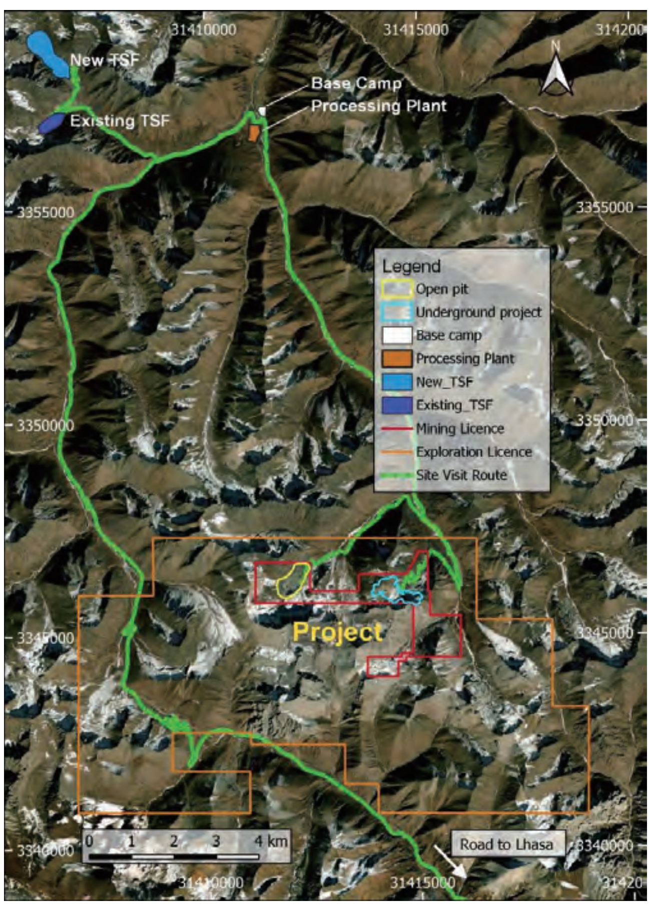
3 Project overview

3.1 Background

The Mengya'a Lead and Zinc Project comprises an open-pit mine, an underground project which is currently under development, a base camp administration and accommodation complex, a processing plant, and two TSFs as shown in Figure 3.1.

The underground project is located approximately 3 km east of the open-pit mine. The mine and the project sites are situated at elevations between 5,000 m and 5,300 m asl. The base camp and processing plant are positioned in a valley approximately 10 km north of the mining sites, at an elevation of around 4,600 m asl. The TSFs are located in a valley 5 km west of the camp and processing plant, with elevations ranging from 4,600 m to 5,600 m asl. The open-pit mine and underground project lie within a granted mining licence which is surrounded by a granted exploration licence.

Figure 3.1: Plan view of the project



Source: SRK (site visit route, September 2024)

The known mineralisation at the project is interpreted to represent a Pb–Zn(–Cu–Ag) skarn deposit. The mineralisation exhibits as layers or lenses hosted within the Carboniferous-Permian sedimentary rocks of the region.

The open-pit mine commenced production in September 2007 and currently yields 400,000 t/a ore for processing. The underground mining system construction completed in May 2025, with production commencing in the second quarter of 2025. Once operational, ore from the open-pit and underground mines will collectively supply 400,000 t/a to the processing plant, which produces lead, zinc and copper concentrates.

3.2 Tenure

The initial mining licence (5400000610026) was granted in September 2006, and was subsequently renewed in 2008, 2011, 2015, 2019 and 2024. The extent of the mining licence (C5400002011043210111466), issued on 25 June 2024 was expanded. The expanded licence, issued on 25 November 2025, is valid until 25 November 2045. The licensed area increased from 3.6061 km² to 4.4544 km², with an approved mining elevation range between 4,964 m and 5,400 m asl. The approved production capacity remains 400,000 t/a (Table 3.1).

Table 3.1: Mining licence coordinates

Corner	X	Y
1	31411114.46	3346720.36
2	31412398.46	3346709.33
3	31412392.49	3346032.35
4	31413544.67	3346025.05
5	31413544.66	3346422.06
6	31414664.24	3346422.06
7	31414914.79	3346828.06
8	31414914.94	3346952.02
9	31415172.44	3346950.86
10	31415172.4	3346009.71
11	31415227.5	3346009.26
12	31415224.51	3345455.3
13	31415945.53	3345450.26
14	31415937.54	3344464.28
15	31414653.52	3344474.33
16	31414653.52	3344382.33
17	31414465.53	3344383.33
18	31414463.54	3344013.34
19	31413740.53	3344019.35
20	31413744.53	3344481.31
21	31414546.53	3344475.3
22	31414547.53	3344567.3

Corner	X	Y
23	31414815.53	3344565.32
24	31414824.51	3345735.28
25	31411106.47	3345765.34

Source: Zhihui 2024

Note: CGCS 2000 datum. SRK has sighted a copy of the licence and checked against the details shown on the National Mineral Exploration and Mining Information Management System. SRK has not conducted any legal due diligence of the status of the licence and is not qualified to do so.

The mining licence is surrounded by an exploration licence (T54000020090330100256 99), which was initially granted in August 2002. The exploration licence was renewed in 2025 and remains valid until 24 June 2026. The licence is registered to Huaxia Mining and covers an area of 58.45 km^2. Under the licence conditions, the next renewal requires the exploration stage to be advanced, with a mandatory reduction of at least 20% of the exploration area. The exploration licence coordinates are shown in Table 3.2. A map showing tenure is presented in Figure 3.2.

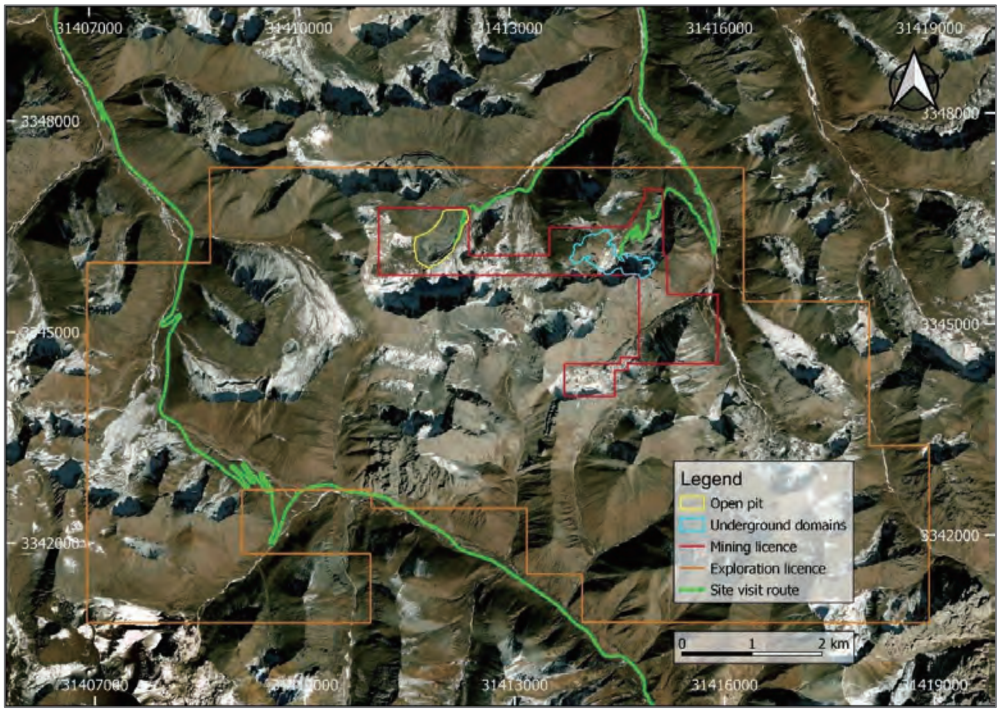
Table 3.2: Exploration licence coordinates

Corner	X	Y
1	31408697.49	3347311.402
2	31416330.41	3347250.412
3	31416315.86	3345346.745
4	31418108.53	3345333.20
5	31418093.20	3343281.51
6	31418949.84	3343281.427
7	31418931.20	3340758.893
8	31413970.97	3340804.60
9	31413976.33	3341489.021
10	31413185.83	3341495.247
11	31413193.25	3342432.128
12	31410970.13	3342449.948
13	31410972.28	3342714.172
14	31409125.07	3342729.322
15	31409117.54	3341820.028
16	31410953.80	3341804.97
17	31410945.81	3340820.97
18	31406891.73	3340854.616
19	31406935.20	3345974.829
20	31408700.66	3345959.973

Source: Zhihui 2024

Note: CGCS 2000 datum. SRK has sighted a copy of the licence and checked against the details shown on the National Mineral Exploration and Mining Information Management System. SRK has not conducted any legal due diligence of the status of the licence and is not qualified to do so.

Figure 3.2: Mining and exploration licences



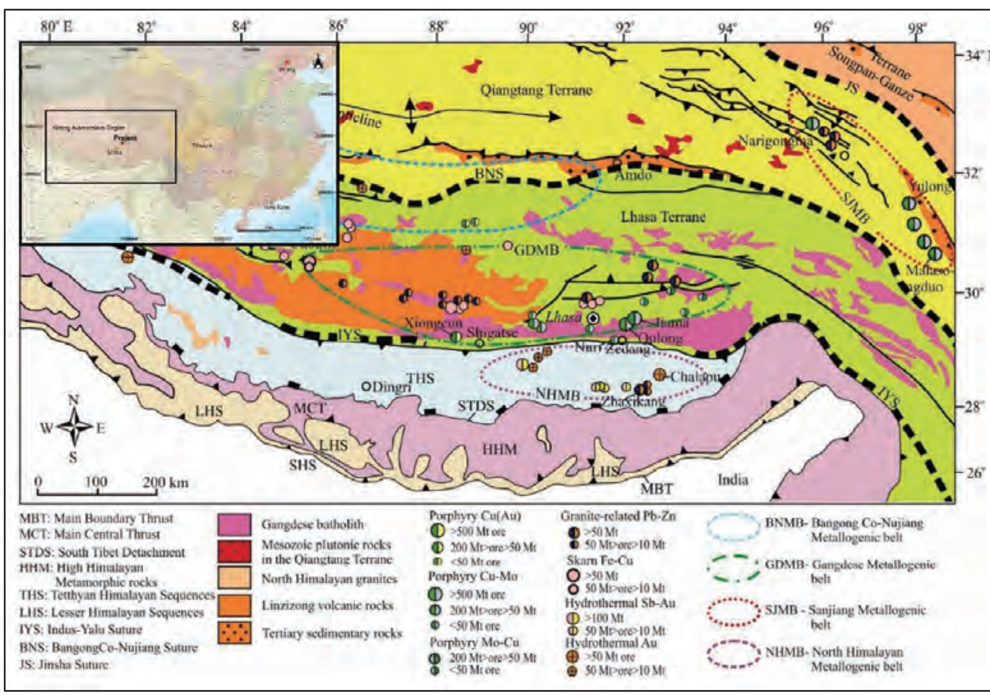
Source: SRK site visit, September 2024

4 Geological setting and mineralisation

4.1 Regional geology

The Xizang Plateau was formed as a result of closure of the Neo-Tethys Ocean and the collision between the Indian and Eurasian tectonic plates. It is characterised by three major suture zones. From south to north, these are the Indus-Yarlung Zangbo Suture (IYS), the Bangonghu–Nujiang Suture (BNS), and the Jinsha–Jiang Suture (JS). These suture zones divide the Xizang Plateau into three distinct tectonostratigraphic terranes: the Lhasa Terrane, the Qiangtang Terrane, and the Songpan–Ganzi Terrane (Chan et al. 2015). The project is located in the eastern part of the Lhasa Terrane (Figure 4.1).

Figure 4.1: Tectonic map of Xizang Plateau



Source: After Lin et al., 2017.

A series of east–west trending thrust belts developed in the Lhasa Terrane due to the combined effects of the southward subduction of the Meso-Tethys Ocean (Bangong-Nujiang Ocean) beneath the Lhasa Terrane during the Late Jurassic to Early Cretaceous, and the northward subduction of the Neo-Tethys Ocean beneath Eurasia during the Cretaceous to Paleogene. These subduction processes generated intense compression, resulting in widespread overturned folds and thrust faults.

During the India–Eurasia collision (beginning ~50 Ma), the Lhasa Terrane experienced a dominant north–south compressive stress regime, which produced extensive east–west trending overthrust faults (e.g. the Gangdese thrust system) as the crust shortened horizontally. During the Miocene (~23 Ma onwards), as part of a post-collisional east–west extensional regime. This extension is attributed to gravitational collapse of the thickened Xizang Plateau and regional tectonic uplift.

4.1.1 Gangdese Metallogenic Belt

The Gangdese Metallogenic Belt (GDMB) is a metallogenic belt in the mid-southern portion of the Lhasa Terrane (outlined by a green dashed line in Figure 4.1) known for the presence of numerous porphyry, skarn, and epithermal polymetallic deposits formed as a result of intense deformation and magmatic activities.

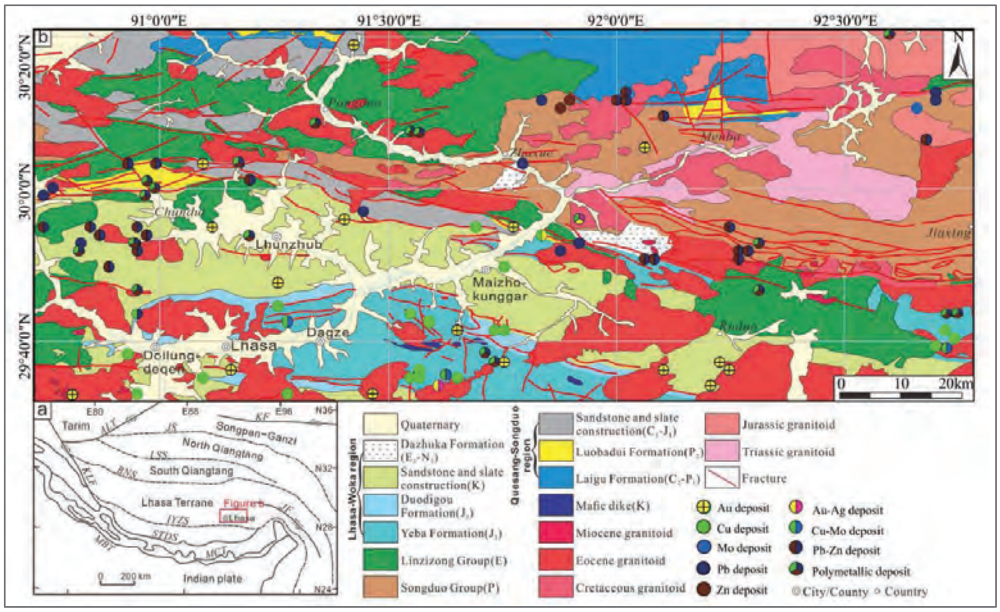
The GDMB trends parallel to the IYS, extending approximately 750 km in length and 200 km in width and comprises rock units ranging from the Palaeozoic to the Cenozoic. The belt has experienced several significant geological processes, including the northward subduction of the Neo-Tethys Ocean, followed by the orogenic events associated with the continental collision between the Indian and Eurasian plates. The northward subduction of the Neo-Tethys Ocean beneath the Lhasa Terrane from the Late Triassic to the Late Cretaceous has led to the emplacement of Gangdese batholith, a series of intermediate to acidic intrusions emplaced between 210 Ma and 60 Ma. The large-scale magmatism and the presence of trans-crustal structures provided favourable conditions for metallogenesis.

The GDMB is known for three primary types of ore deposits: Cu-Au-Mo porphyry, Fe-Cu skarn, and Pb-Zn deposits. The Miocene epoch was a significant metallogenic period for the emplacement of porphyry copper-polymetallic deposits in the mid-eastern GDMB. Numerous large-scale deposits were formed during this epoch, including the Qulong (approximately 16.13 Ma), Jiama (14.66–15.37 Ma), Zhunuo (approximately 13.72 Ma), Lakange (approximately 13.12 Ma), Tinggong (approximately 15.49 Ma), and Chongjiang deposit (approximately 14 Ma). The GDMB is recognised as one of the most well-endowed copper provinces in the Tethyan-Himalayan metallogenic belt, hosting some of China's largest porphyry copper deposits.

4.2 Local geology

The project is located in the northeastern part of the GDMB. The regional geology can be divided into the southern and northern sub-groups. The project is situated in the northern sub-group, which belongs to the Quesang–Songduo stratigraphic group, comprising the Late Carboniferous to Jurassic Luobadui and Laigu Formations (Figure 4.2). The southern sub-group belongs to the Lhasa–Woka stratigraphic group, including the Dazhuka Formation, the Duodigou Formation and the Yeba Formation.

Figure 4.2: Geological map of northeastern Gangdese Metallogenic Belt



Source: After Zhang et al., 2024.

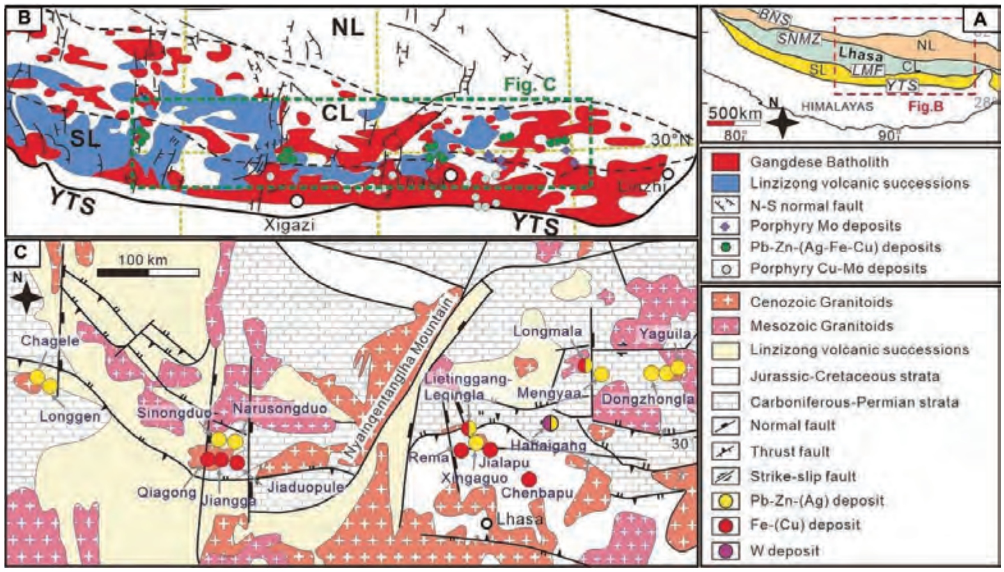
The Carboniferous-Jurassic strata are the primary host to the known mineralisation and are characterised by interbedded terrigenous clastic and carbonate rocks. Porphyry copper deposits are primarily found within the Jurassic Yeba and Duodigou formations. Skarn deposits occur locally along the unconformity between the Early Cretaceous Linzizong Formation sandstone-slate and the Late Jurassic Duodigou Formation limestone, as well as the Late Carboniferous Laigu Formation and the Permian Luobadui Formation.

Volcanic rocks are found in the Yeba Formation of the Lhasa–Woka stratigraphic zone and are extensively distributed across various strata in the Quesang–Songduo stratigraphic area. Metamorphic rocks, such as quartzite, quartz schist, phyllite with gneiss, and marble, are also prevalent in this region.

4.2.1 Pb-Zn, Cu and Fe deposits

Several skarn Pb-Zn-(Ag-Fe-Cu) deposits have been identified and developed in the northern Gangdese Metallogenic Belt (NGMB) (Figure 4.3). It is interpreted that metals in these Pb-Zn-(Ag) deposits are sourced from upper-crust magma, while metals in Fe-(Cu) and Pb-Zn-Fe-Cu deposits are sourced from deeper magma, involving mantle-derived components. Previous research (Zhang et al., 2023) proposed that metals in Pb-Zn-(Ag) deposits were derived from S-type granites (granitic rocks derived from the partial melting of subducted sedimentary materials), whereas those in Pb-Zn-Fe-Cu and Fe-(Cu) deposits are sourced from igneous-type (I-type) granites (granitic rocks derived from the partial melting of igneous materials).

Figure 4.3: Pb-Zn, Cu and Fe deposits in the Gangdese Metallogenic Belt



Source: Zhang et al., 2023.

Notes: BNS–Bangong–Nujiang suture zone; CL–Central Lhasa subterrane; JSS–Jinshajiang suture zone; LMF–Luobadui–Mila mountain fault; NL–Northern Lhasa subterrane; SL–Southern Lhasa subterrane; SNMZ–Shiquan River–Nam Tso ophiolite mélange zone; YTS–Yarlung–Tsangpo suture zone.

Within the NGMB, the granite-related skarn Pb-Zn-(Ag) and Pb-Zn-Fe-Cu polymetallic deposits are among the most economically significant (Figure 4.3C). These deposits are hosted within granite porphyry and quartz-rich granite units. These intrusives are emplaced within a host rock sequence, consisting of carbonate and clastic sedimentary rock, spanning Carboniferous, Permian, Jurassic, or Cretaceous periods (Table 4.1).

Table 4.1: Geological features of major Pb-Zn, Cu and Fe deposits in the NGMB

Name	Type of deposit	Tonnage/ore	Grade	Ore-bearing rock	Host rock	Alteration	Ore minerals	Orebody shape
Yaguila	Skarn Pb-Zn-Ag	52 Mt	Pb: 4.04%	granite porphyry, quartz porphyry	quartz sandstone and limestone, Carboniferous-Permian Laigu Formation	silicification	galena	layered
			Zn: 2.00%			chloritisation	sphalerite	stratabound
			Ag: 77.33 g/t			sericitisation	silver	lenticular
						epidotisation	pyrrhotite	vein
						hornfelsisation	pyrite	
						skarnisation		
Mengya'a	Skarn Pb-Zn-Ag	Pb-Zn: >12.0 Mt	Pb: 10.11%	granite porphyry	clastic rock and limestone, Carboniferous-Permian Laigu Formation; limestone and tuff, Permian Luobadui Formation	carbonatisation	galena	layered
			Zn: 11.88%			silicification	sphalerite	stratabound
			Ag: 100 g/t			chloritisation	pyrrhotite	lenticular
			Cu: 0.32%			sericitisation	chalcopyrite	vein
						epidotisation	pyrite	
						marbleisation		
						hornfelsisation		
						skarnisation		
Dongzhongla	Skarn Pb-Zn-(Ag)	Medium-large	Pb: 2.55%	monzonitic granite, granite, granite porphyry	limestone and tuff, Permian Luobadui Formation	carbonatisation	galena	layered
			Zn: 2.57%			silicification	sphalerite	stratabound
			Ag: 43.7 g/t			chloritisation	silver	vein
						sericitisation	magnetite	
						epidotisation	pyrrhotite	
						skarnisation	chalcopyrite	
							pyrite	
Xingaguo	Skarn Pb-Zn-(Ag)	Medium-large	No.2 orebody:	biotite granite	sandstone, siltstone and limestone, Cretaceous Take Formation	chloritisation	magnetite	layered

Name	Type of deposit	Tonnage/ore	Grade	Ore-bearing rock	Host rock	Alteration	Ore minerals	Orebody shape
			Pb:3.82%			silicification	sphalerite	stratabound
			Zn:7.02%			carbonatisation	galena	lenticular
			No.4 orebody:			marbleisation	pyrite	
			Pb: 9.97%			skarnisation	marcasite	
			Zn: 9.33%				chalcopyrite	
							pyrrhotite	
Longgen	Skarn Pb-Zn	Pb: 6.3Mt Zn: 5.7 Mt Ag:109 t	Pb:3.21%	granite porphyry	sandstone, slatea and limestone, Permian Xiala Formation	carbonatisation	magnetite	stratabound
			Zn: 2.43%			chloritizatio	pyrite	lenticular
			Ag: 46.74 g/t			pyritisation	chalcopyrite	vein
						marbleisation	bornite	
						skarnisation	sphalerite	
							galena	
							pyrrhotite	
Chagele	Skarn Pb-Zn-(Cu-Mo)	no data	no data	granite porphyry	limestone and slate, Permian Xiala Formation	hornfelsisation,	pyrite	stratabound
						silicification	chalcopyrite	vein
						carbonatisation	sphalerite	
						chloritisation	galena	
						epidotisation	molybdenite	
						sericitisation	pyrrhotite	
						marbleisation		
						skarnisation		
Narusongduo	Cryptoexplosive breccia Pb-Zn	> 15 Mt	Pb: 7.63%	granite porphyry	metasandstone, siltstone, slate, limestone, Permian Xiala Formation; crystal tuff, sandstone, andesite, Dianzhong Formation	kaolinisation	sphalerite	angular , oval
			Zn: 2.41%			sericitisation	galena	vein
						silicification	pyrite	

Name	Type of deposit	Tonnage/ore	Grade	Ore-bearing rock	Host rock	Alteration	Ore minerals	Orebody shape
						carbonatisation	chalcopyrite	
						chloritisation	arsenopyrite	
						epidotisation	rhodochrosite	
						marbleisation		
						skarnisation		
Sinongduo	Hydrothermal vein Pb-Zn	Large	Pb+Zn: 11.25%	granite porphyry	limestone, dolostone, slate, quartz sandstone, Carboniferous Angjie Formation	pyritisation	galena	lenticular
						sericitisation	sphalerite	vein
						skarnisation	smithsonite	
						silicification	chalcopyrite	
						carbonatisation	pyrrhotite silver	
						marbleisation		
						chloritisation		
						kaolinitisation		
Longmala	Skarn Pb-Zn-Fe-Cu	Pb-Zn: 3 Mt	Pb: 6.63%	biotite granite	limestone and marble, Permian Luobadui Formation; dolostone, Permian Wululong Formation	carbonatisation	galena	layered
		Cu: 1.4 Mt	Zn: 5.46%			silicification	sphalerite	stratabound
			Cu: 1.99%			chloritisation	magnetite	lenticular
						sericitisation	chalcopyrite	vein
						epidotisation	pyrrhotite	
						marbleisation	pyrite	
						hornfelsisation		
						skarnisation		
Lietinggang-Leqingla	Skarn Pb-Zn-Fe-Cu	Pb-Zn: 0.55 Mt	Pb+Zn: 7.74%	granite porphyry	limestone, tuff, sandstone, Permian Mengla Formation	carbonatisation	galena	layered
		Fe: 8.1 Mt	Fe: 55.27%			silicification	sphalerite	stratabound
		Cu: 0.32 Mt	Cu: 1.07%			chloritisation	magnetite	lenticular

Name	Type of deposit	Tonnage/ore	Grade	Ore-bearing rock	Host rock	Alteration	Ore minerals	Orebody shape
						sericitisation	chalcopyrite	vein
						epidotisation	molybdenite	
						marbleisation	pyrrhotite	
						skarnisation	pyrite	
Jialapu	Skarn Fe-(Cu)	Fe: >8.0 Mt	Fe: 40–65.97%	granodiorite	limestone and marble, Early Triassic Mailonggang Formation	carbonatisation	magnetite	layered
						chloritisation	limonite	stratabound
						silicification	pyrite	lenticular
						marbleisation	pyrrhotite	
						malachilisation	chalcopyrite	
						skarnisation		
Rema	Skarn Fe	0.34 Mt	Fe: 43.96%	granite	sandstone and limestone, Cretaceous Take Formation	carbonatisation	magnetite	lenticular
						chloritisation	pyrrhotite	vein
						epidotisation	pyrite	
						limonitisation	chalcopyrite	
						marbleisation		
						skarnisation		
Jiaduobule	Skarn Fe-(Cu)	no data	no data	biotite monzogranite	limestone and marble, Permian Xiala Formation	carbonatisation	magnetite	layered
						chloritisation	chalcopyrite	vein
						epidotisation	pyrite	
						marbleisation	pyrrhotite	
						skarnisation		
Chengbapu	Skarn Cu	0.63 Mt	Cu:1.3%	diorite, granodiorite porphyry	limestone and marble, Jurassic Duodigou Formation	carbonatisation	chalcopyrite	layered
			Pb:3.9%			epidotisation	molybdenite	stratabound
			Zn: 0.2%			hornfelsisation	pyrrhotite	lenticular
						skarnisation	sphalerite	
Jiangga	Skarn Fe	no data	no data	quartz porphyry	limestone and marble, Permian Xiala	carbonatisation	magnetite	lenticular

Name	Type of deposit	Tonnage/ore	Grade	Ore-bearing rock	Host rock	Alteration	Ore minerals	Orebody shape
					Formation	chloritisation	pyrite	
						limonitisation	pyrrhotite	
						skarnisation		
Qiagong	Skarn Fe	Fe: 90 Mt	Fe: 55%	monzonite granite porphyry	sandstone and limestone, Cretaceous Take Formation	carbonatisation	magnetite	layered
						chloritisation	limonite	stratabound
						epidotisation	pyrite	lenticular
						limonitisation	pyrrhotite	vein
						marbleisation	chalcopyrite	
						skarnisation		

Source: Zhang et al., 2023

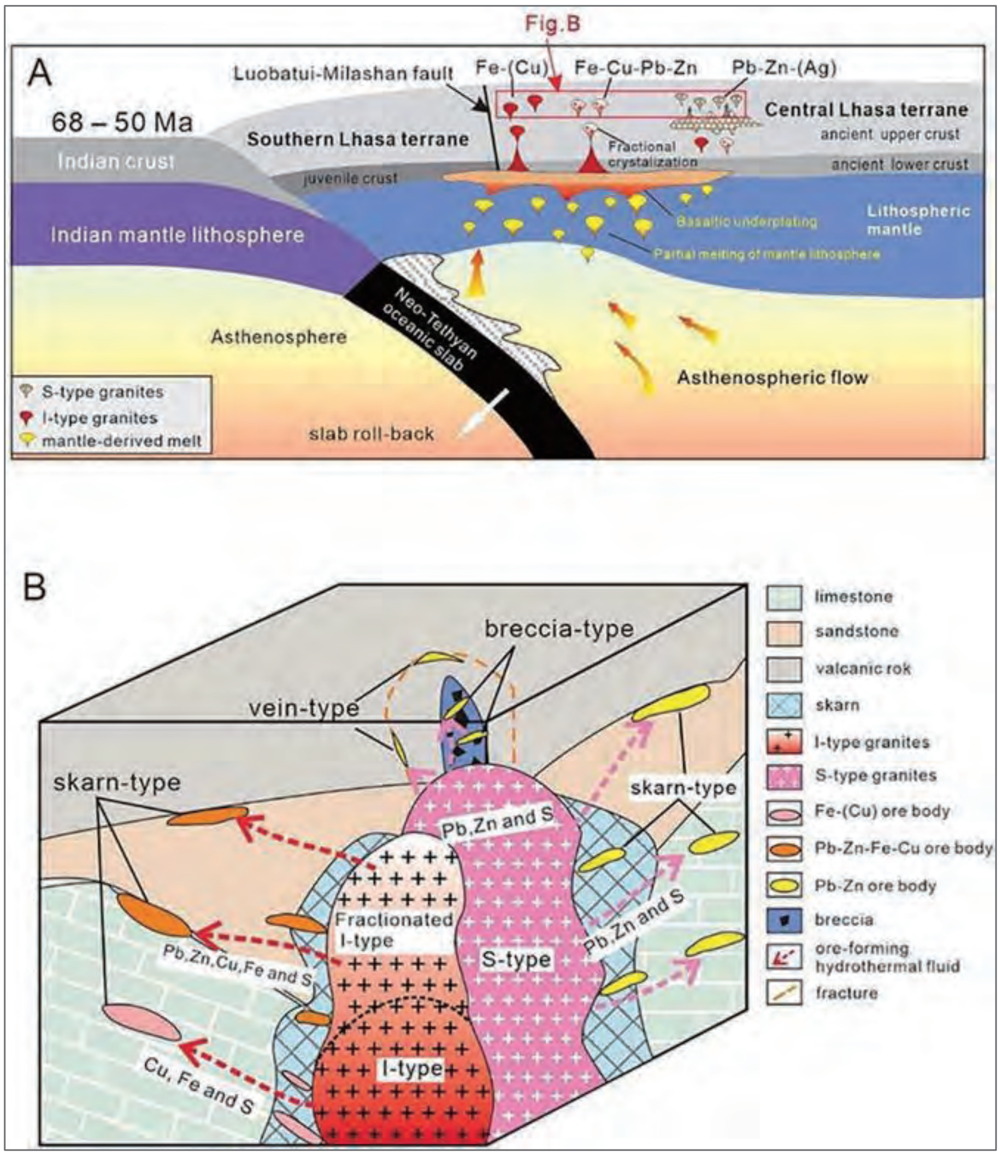
4.2.2 Metallogenic model

Zhang et al. (2023) proposed a magmatic-metallogenic model for skarn Pb-Zn-(Ag) and Fe-(Cu) mineralisation in the NGMB (Figure 4.4). According to this model, the collision between the Indian and Asian continental lithospheres that occurred between 68 Ma and 50 Ma led to the thickening of the Lhasa lithosphere and the rollback of the Neo-Tethyan oceanic slab. The slab rollback facilitated mantle wedge melting and the resulting mantle-derived melts ascended through the crust and assimilated crustal components from the ancient Lhasa basement.

I-type granitic magmas generated from the mixing of mantle-derived melts with the ancient Lhasa basement underwent fractional crystallisation. The subsequent melts became highly differentiated, with the formation of granitic melts that are rich in metals such as Cu, Fe, Ag, Pb and Zn. Hydrothermal fluids rich in these metals and other volatiles were released during the cooling of the granitic magma and interacted with carbonaceous and siliciclastic wallrock to form the skarn deposits. Notable examples include the Rema and Jialapu Fe-(Cu) deposits, as well as the Longmala and Lietinggang-Leqingla Pb-Zn-Fe-Cu deposits.

S-type granitic magmas were formed during the partial melting of the ancient crust and were enriched in Pb and Zn. They played a significant role in the formation of Pb-Zn-(Ag) deposits. Examples of these deposits are Yaguila, Mengya’a and Nurusongduo.

Figure 4.4: Magmatic-metallogenic model of skarn Pb-Zn-(Ag) and Fe-(Cu) mineralisation in the NGMB

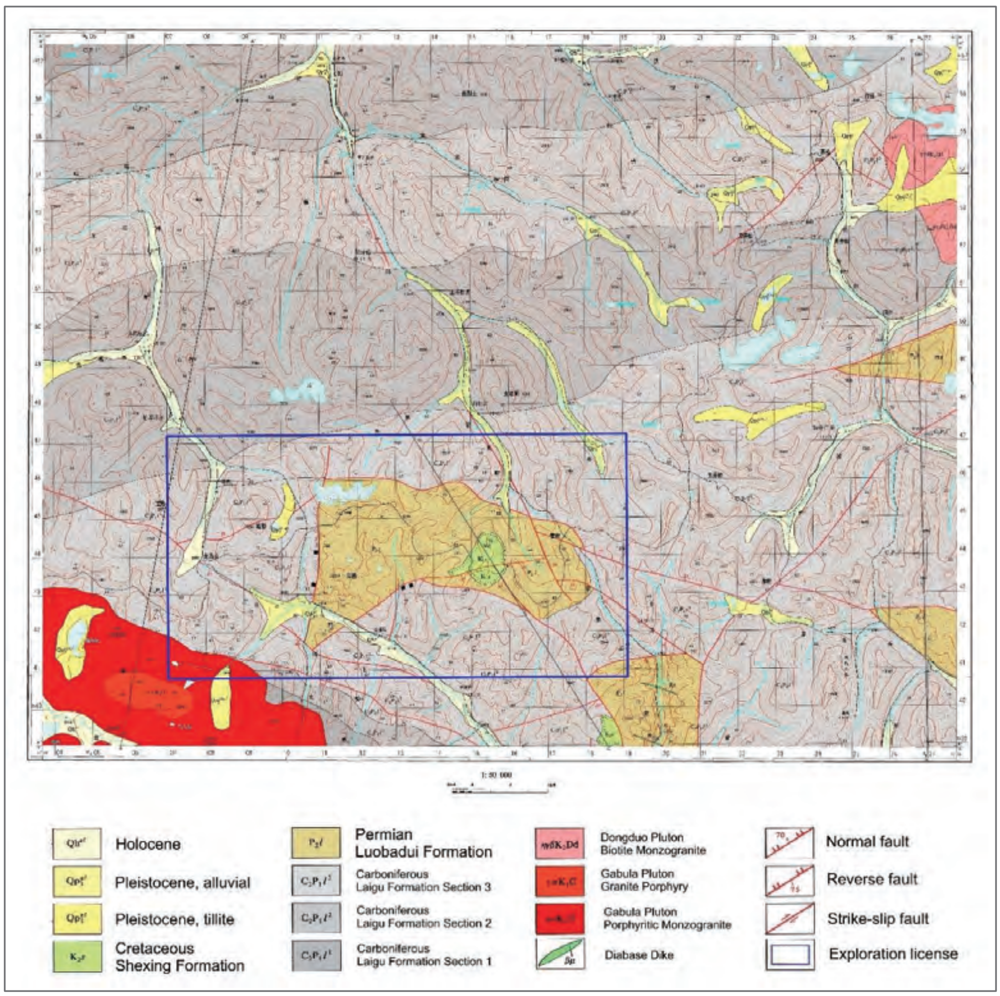


Source: Zhang et al., 2023.

4.3 Project geology

The geological setting of the project area and its surroundings is primarily composed of Carboniferous-Cretaceous clastic rocks and carbonates, together with small-scale intermediate to felsic granitic intrusions. The structural framework is mainly defined by approximately east–west trending faults. A geological map illustrating the mining area and its surroundings is provided in Figure 4.5.

Figure 4.5: Geological map of project and adjacent areas



Source: Zhihui

4.3.1 Stratigraphy

The stratigraphy in the project area includes the Late Carboniferous-Early Permian Laigu Formation, Middle Permian Luobadui Formation, Late Cretaceous Shexing Formation and Quaternary deposits (Table 4.2).

Table 4.2: General stratigraphy

Period	Stratum	Description	Key features
Quaternary		Widely distributed, controlled by topography, and classified into glacial (Qgl), alluvial (Qapl), and colluvial (Qedl) deposits.	Present in valleys and slopes, with glacial deposits near specific orebodies.
Cretaceous	Shexing Formation (K_2s)	Late Cretaceous Formation: Its lower contact is marked by the angular unconformity with the Middle Permian Luobadui Formation, with a slightly uneven erosional base. It is characterised by purple-red and brown-red beds, primarily composed of medium to thick layers, with occasional thin layers. The main rock types include siliceous siltstone, calcareous siltstone, muddy siltstone, silty mudstone, and mudstone. These are interbedded with fine-grained quartz sandstone, microcrystalline limestone, and micritic limestone.	Localised occurrence, angular unconformity overlying the Luobadui Formation or in fault contact, with no overlaying rock unit observed. In the central-southern part of the mining area, there is a small distribution with strata dipping southeast, at an angle of 30° to 38°.
Permian	Luobadui Formation (P_2l)	Middle Permian Formation: It is generally in fault contact with the Laigu Formation, with some areas showing conformable contact. This group mainly consists of carbonate sedimentary rocks, which exhibit significant drag folds due to faulting. Commonly observed are hornfels or recrystallised limestone and marble as thermal contact metamorphic rocks. Near the faults, local folding is observed, and due to the influence of fault-aligned intrusive dykes and hydrothermal activity, contact metasomatic rocks such as skarn are developed. The main rock types include garnet-diopside-tremolite skarn, epidote-diopside skarn, and diopside skarn, with occurrences of skarn-type polymetallic mineralisation. The contact zone with the Laigu Formation is an important host to Pb-Zn orebodies in the Mengya'a mining area, such as the Pb12 ore group and the Pb13 orebody.	Exposed in the central part of the project area, the strata dip southeast and south-southeast, at 19° to 67°.
Carboniferous to Permian	Laigu Formation (C_2P_1l)	Upper Carboniferous to Lower Permian Formation	Widely distributed across the project area and can be divided into three members. In the project area, the Middle and Lower members are present.
	Upper Member	Not present in the project area	
	Middle Member	The main lithology is metamorphosed quartz sandstone, followed by mud-sandy slate and marl, interbedded with quartzite, metamorphosed feldspar quartz sandstone, granulite, quartz schist, and mica schist. Small-scale Pb-Zn orebodies, are found at the interface and on both sides of the mud-sandy slate, tuff, and metamorphosed sandstone layers.	The strata generally dip northwest or southeast, with dip angles ranging from 20° to 70°.
	Lower Member	It consists of grey-black carbonaceous mud-sandy slate interbedded with metamorphosed quartz sandstone, sandy mudstone slate, limestone, and marble lenses. It is generally in fault contact with the overlying Luobadui Formation limestone, with some areas showing conformable contact. This lithological section is an important ore-bearing stratigraphy in the Mengya'a Project area.	The strata generally dip northwest or southeast, with dip angles ranging from 25° to 68°.

Source: Zhihui, 2019

4.3.2 Intrusions

The intrusive rocks in the project are dominated by intermediate to felsic intrusions, which were emplaced into the Upper Carboniferous to Lower Permian Laigu Formation strata. These intrusions are distributed in east–west trending bands as irregular blocky patches. Thermal contact metamorphism halos are developed in the host rock along the contact with these plutons. Magmatic activity in the region is mainly represented by the Early Cretaceous granitoid bodies, with spatial distribution of the Late Cretaceous granites more limited. Key examples include the Early Cretaceous Gabu La biotite monzogranite and the Late Cretaceous Dongduo biotite monzogranite.

Within the project area, the magmatic rocks primarily comprise granite porphyry, diabase and diabase porphyrite. These intrusions are relatively small in scale. Diabase and diabase porphyrite typically occur as dykes, distributed in the central and eastern parts of the project area, forming groups and belts with a nearly east–west orientation. Granite porphyry is found in the central and northern parts of the project area, mostly occurring as dykes that conform to the beds of the Laigu Formation and the Luobadui Formation. Some small stocks intruding into the Luobadui Formation strata near the contact boundaries are observed.

4.3.3 Structure

The structures in the surrounding area are significantly influenced by the north–south compression of the Xizang Plateau, leading to the development of well-defined, nearly east–west oriented, regional faults and fold structures. These regional fault systems are highly dynamic, with faults of varying directions, characteristics, and ages undergoing multiple episodes of superposition and cross-cutting, resulting in a complex and diverse array of structural forms. In contrast, the northeast and north-northeast oriented faults developed later and are predominantly characterised by strike-slip movements. The nearly north–south oriented faults initially displayed extensional and torsional features, but later transitioned to exhibit compressional and torsional characteristics, serving a regulatory function within the east–west structural framework.

Exploration work conducted by Second Brigade on faults is insufficient for a detailed assessment of the impact of these structures on domains and thus no hypothesis or model for mineralisation potential has been developed. SRK recommends further investigation on the correlation between faults and mineralised zones be undertaken.

4.3.4 Alteration

The project area exhibits extensive development of various alteration in host rocks, including sericitisation, chloritisation, epidotisation, skarnification, pyritisation, carbonatisation and silicification.

4.4 Mineralisation

The magmatic rocks exposed in the project area are primarily granite porphyry and diabase. According to Fu et al. (2017), the granite porphyry was formed at 13 Ma, while the diabase intruded into the Luobadui Formation as dykes. Previous studies suggested that these magmatic rocks and mineralisation are genetically unrelated. More recent drill hole data by Zhang et al. (2023) reveals the presence of a granite porphyry at the deeper levels of the Laigu Formation which was dated at 68 Ma. Currently, 31 prospects or deposits have been identified in the project area, most of which are layered or lenticular, occurring in skarns within the Laigu and Luobadui formations and along the contacts between the limestones and siltstones. The principal mineralisation includes vein-hosted to massive sphalerite and galena, with minor amounts of pyrrhotite, pyrite and chalcopyrite. The gangue minerals are comprised of common skarn minerals, such as garnet, diopside, actinolite and calcite.

At the Mengya'a deposit, hydrothermal alteration of the host rocks is characterised by skarn, potassic and sericitic alterations. The skarn alteration assemblage mainly comprises coarse-grained garnet, diopside, actinolite, epidote, muscovite and chlorite. Abundant calcite, chlorite, limonite and malachite formed in later stages and are unrelated to Pb-Zn mineralisation.

5 Exploration

Exploration has been ongoing at the project since the initial discovery of mineralisation in 2002.

From July 2002 to 2003, the Second Brigade carried out a reconnaissance in the area surrounding the project. This phase involved surface digging and geophysical surveys, leading to the discovery of the Pb4 mineralisation. No drilling was performed during this period.

Following the initial reconnaissance, comprehensive exploration programs were implemented from 2004 to 2021. These programs focused on the mineralised deposits (e.g. Pb14 and Pb12) within the project's exploration licence area and included mapping, surface sampling, geophysical surveys, trenching and drilling activities.

Between 2022 and 2024, exploration activities were reduced to focus on validation of geophysical anomalies.

5.1 Geophysical survey

The Second Brigade conducted a series of electrical and magnetic surveys in 2004, 2008, 2012 and 2017.

In 2004, a resistivity profiling survey totalling 25 km was conducted within the exploration licence area. By 2008, an induced polarisation intermediate gradient profiling survey (IP profiling survey) extended over 70 km in the central part of the exploration licence, successfully identifying the Pb12 mineralisation and prompting a subsequent drilling campaign. In 2012, a 5 km^2 induced polarisation intermediate gradient area survey (IP area survey) was executed to encompass the Pb12 deposit.

In 2017, a comprehensive geophysical survey program was launched in the northeastern section of the exploration licence area (Figure 5.1). This program included an IP area survey, induced polarisation electrical sounding profiling survey (IP sounding survey), magnetotelluric sounding survey (MT sounding survey), and a high-accuracy ground magnetic survey (HGM survey).

IP area survey

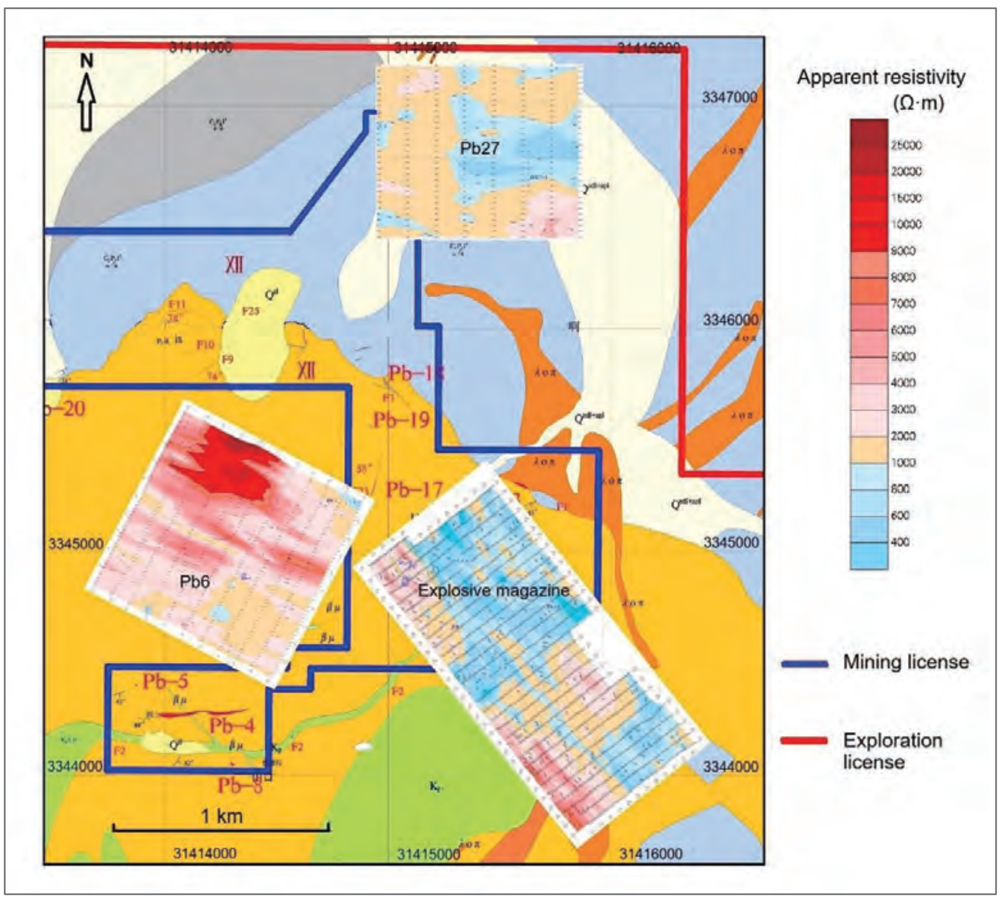
The survey used a grid spacing of 100 m × 20 m for survey points and concentrated on three primary targets: the Pb6 deposit, the Pb27 deposit and the proposed location of an explosives magazine.

- Pb6 deposit: This area covered a square measuring 1,000 m in both the NW-SE and NE-SW directions, with a total area of 1.0 km^2
- Pb27 deposit: This area encompassed a square measuring 700 m in the E-W direction and 600 m in the N-S direction, with a total area of 0.42 km^2
- explosives magazine location: This area spanned a square measuring 1,260 m in the SW-NE direction and 2,800 m in the SE-NW direction, with a total area of 3.408 km^2。

The IP area survey was conducted to assess two critical parameters of geological formations: apparent resistivity (ρs) and apparent chargeability (ηs). This survey used the DZD-6A multi-functional digital DC induced polarisation instrument (manufactured by Chongqing Geological Instrument Factory).

To gain a comprehensive spatial understanding of the anomalies detected in the IP area survey, further investigations were carried out using IP sounding and magnetotelluric (MT) sounding.

Figure 5.1: Apparent resistivity results of IP area survey – 2017



Source: Zhihui, 2020

IP sounding survey

The IP sounding profiles were designed to be strategically placed at the centre of the IP area survey anomalies, taking the terrain and topography into consideration. Three IP sounding profiles were conducted with a station spacing of 20 m:

- the Pb27 deposit area, with a length of 180 m
- the two locations to the east of explosives magazine, with lengths of 460 and 20 m, respectively.

The total length of the IP sounding profiles is 0.66 km.

MT sounding

The MT sounding survey was carried out over five profiles, which are as follows:

- the western side of the Pb6 deposit area, with a length of 270 m
- the Pb27 deposit area, with a length of 210 m
- three profiles in the Pb12 deposit area, each with a length of 120 m.

The total length of the MT sounding profiles is 0.84 km.

HGM survey

The HGM survey covered an area of 37.7 km^2 at a scale of 1:10,000. The data collection for the magnetic survey was conducted using the GSM-19T proton magnetometer (manufactured in Canada).

5.2 Drilling campaign

Drilling programs conducted at major prospects (Pb4, Pb12, Pb13 and Pb14) within the project area can be grouped into two stages: the early-stage exploration in 2004 and the advanced exploration in 2006–2021. All drilling was completed by the Second Brigade.

5.2.1 2004 early-stage exploration

In 2004, the Second Brigade commenced an initial exploration campaign to enhance the understanding and definition of potential mineralisation in the area. Exploration work included geological mapping at a scale of 1:10,000, drilling of four drill holes totalling 304.96 m, excavation of 4,745.2 m^3 of trenches, removal of 9,602.5 m^3 of overburden, and construction of 81.1 m of shallow shafts.

An XY-4 drill rig was employed for diamond core drilling. All drill holes were collared with PQ size (85 mm) to penetrate the topsoil, followed by HQ size (63.5 mm) for deeper drilling. Core recovery ranged from 71% to 100%, with an average recovery of 78%. A total of 189 chemical analysis samples were collected (some of the data were not preserved).

The drill hole core was logged by onsite geologists, with detailed documentation covering lithology, mineral composition, structural features, texture, vein type, oxidation levels, alteration minerals, and rock quality designation (RQD).

The exploration work followed prevailing Chinese standards, with drill hole spacing being approximately 100 m × 50 m. All samples were sent to the Xizang Autonomous Region Geological and Mineral Exploration and Development Bureau Central Laboratory (Xizang Central Laboratory) for testing. The core was not preserved, and the laboratory has disposed of the pulps, making it impossible to verify these assays.

5.2.2 2006–2021 advanced exploration

During exploration between 2006 and 2021, the Second Brigade carried out a series of advanced exploration campaigns aimed at further delineating resources and assessing the potential for additional extents to the known mineralisation to be defined.

The Second Brigade successfully completed 249 drill holes, totalling 59,768.72 m, and collected 4,777 samples which were assayed for Pb, Zn, Cu and Ag, together with 428 samples for bulk density analysis. The drill hole spacing ranged from approximately 50 m × 50 m to 50 m × 100 m.

Diamond core drilling was conducted using XY-2 and XY-4 drill rigs. Each drill hole began with a PQ size (85 mm) collar to penetrate the topsoil, transitioning to HQ size (63.5 mm) for deeper sections.

The drilling activities are detailed by year as follows:

1. In September 2006, Huaxia Mining obtained the initial mining licence for the central region within the project's exploration tenure area. The Second Brigade drilled 43 diamond holes, totalling 5,157.62 m, and excavated 80 m of horizontal adits. A total of 336 samples were collected from these drill holes and 16 samples from the adits for assay analysis.
2. Between 2007 and 2008, the Second Brigade completed 7 holes, totalling 1,599.85 m, and collected 115 samples for assay analysis, together with 18 samples for bulk density testing. Hydrogeological surveys were conducted and geotechnical samples for further testing were collected.
3. From 2009 to 2010, the Second Brigade drilled 5 holes, totalling 350.14 m, and collected 87 samples for assay analysis.
4. In 2011, the Second Brigade drilled 40 diamond holes, totalling 10,472.69 m, and collected 681 samples for assay analysis.
5. In 2012, the Second Brigade scompleted 34 diamond holes, totalling 11,688.78 m, alongside the removal of 4,745.2 m^3 of overburden and the excavation of 644.75 m of horizontal adits. They collected 605 assay samples and 89 bulk density samples. Comprehensive hydrogeological and geotechnical surveys were conducted over an area of 2 km2, complemented by 938.6 m of hydrogeological drilling through reamed boreholes. Additionally, 1,844.93 m of grade control drilling were carried out at the Pb14 deposit to optimise open-pit mining operations.
6. In 2014, the Second Brigade drilled 14 diamond holes, totalling 3,041.6 m, and collected 249 samples for assay analysis and 33 samples for bulk density testing.
7. In 2015, the Second Brigade drilled 15 diamond holes, totalling 4,230.25 m, and collected 472 samples for assay analysis and 42 samples for bulk density testing.

8. In 2017, the Second Brigade drilled 27 diamond holes, totalling 7,472.95 m, and collected 476 samples for assay analysis and 47 samples for bulk density testing.
9. In 2018, the Second Brigade drilled 10 diamond holes, totalling 1,892.07 m, and collected 266 samples for assay analysis and 21 samples for bulk density testing.
10. In 2019, the Second Brigade drilled 15 diamond holes, totalling 4,314.48 m, and collected 385 samples for assay analysis and 40 samples for bulk density testing. Additionally, 144 shallow diamond drill holes using AQ (30 mm) size, each less than 10 m deep, were completed to collect samples as an alternative to trenching for the Pb6 and Pb27 deposits. These shallow drill holes were drilled for mineralisation target identification only, and were therefore not included in the database for resource estimation).
11. In 2020, the Second Brigade drilled 20 diamond holes, totalling 3,960.26 m, and collected 525 samples for assay analysis and 38 samples for bulk density testing.
12. In 2021, the Second Brigade drilled 19 diamond holes, totalling 5,588.03 m, and collected 580 samples for assay analysis and 33 samples for bulk density testing.
13. Core recovery in 2006–2010 ranged from 69.1% to 98.4%, with an average recovery of 85%. Core recovery in 2011–2018 ranged from 80.8% to 94.8%, with an average recovery of 90%. Core recovery in 2019–2021 ranged from 73.8% to 98.7%, with an average recovery of 91%. The improvement of core recovery from 2006 to 2021 explorations was likely due to the upgrade of drilling rigs and the accumulation of experience of the drilling team.
14. The drill core was logged by onsite geologists, with detailed documentation covering lithology, mineral composition, structural features, texture, vein type, oxidation levels, alteration minerals, and RQD. Intervals do not cross lithological or mineralisation boundaries.
15. The core samples collected from holes completed prior to 2010 (70 holes) were not retained on site, and the assaying laboratory has since disposed of the associated pulps samples, rendering it impossible to verify those assay results. These holes were drilled at the Pb4, Pb13, and Pb14 prospects. However, since 2011, all core from holes drilled in the Pb12 deposit have been preserved in a well-maintained core shed at the project's base camp.

All core samples were boxed in accordance with established protocols. Each core box was clearly labelled in red paint with the drill hole number, box number, and the start and end depths. Once boxed at the drill site, the core samples were transported to the core storage facility. This was followed by detailed logging, stratification, sampling, photography, sawing, and sampling. Diamond core samples were collected by half-coring lengthwise, which was considered representative. After these processes, the cores were moved to the core storage facility located at the base camp. These cores were systematically stored, registered according to protocols, and their storage locations were marked on a core distribution map to facilitate easy retrieval in the future.

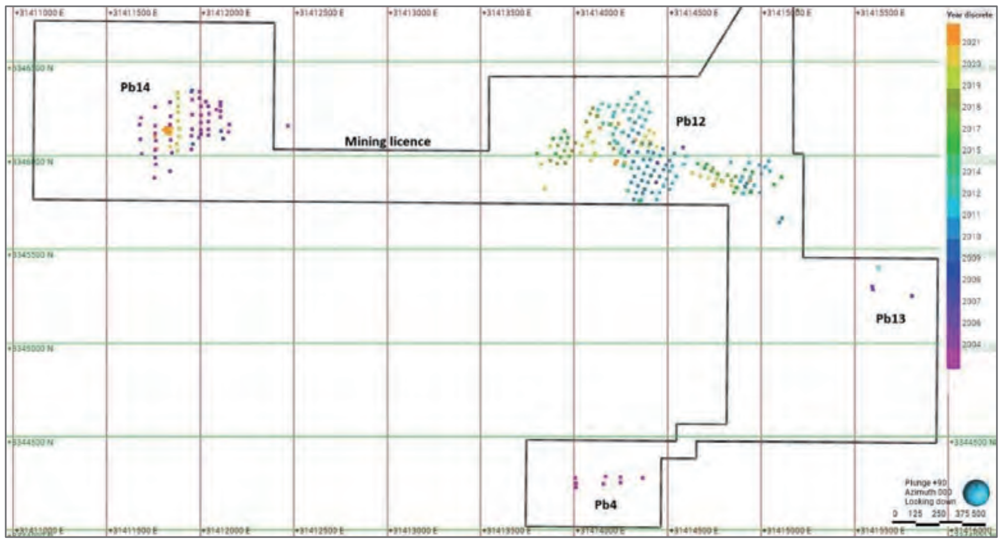
Collected samples were dispatched to various laboratories for analysis. From 2004 to 2010, samples were analysed at Xizang Central. For the periods of 2011–2012 and 2017–2021, samples were analysed by Lhasa Jinzang Rock Testing Co., Ltd. (Jinzang Laboratory). During 2014 and 2015, the samples were tested at the Yanjiao Central Laboratory of the North China Nonferrous Geological Exploration Bureau (Yanjiao Central Laboratory).

5.2.3 Summary

A total of 253 drill holes were completed during historical exploration from 2004 to 2021 (Table 5.1). However, the cores of 70 drill holes drilled prior to 2010 were not preserved.

Of these 253 drill holes, SRK used 246 drill holes completed at the Pb12 and Pb14 deposits for resource estimation purposes. The drill holes of these historical exploration phases are illustrated in the plan view map shown in Figure 5.2.

Figure 5.2: Plan view of historical drill holes and channels



Source: Compiled by SRK, 2025

The information relating to all drill holes and channel samples is summarised in Table 5.1.

Table 5.1: Summary of historical drill holes

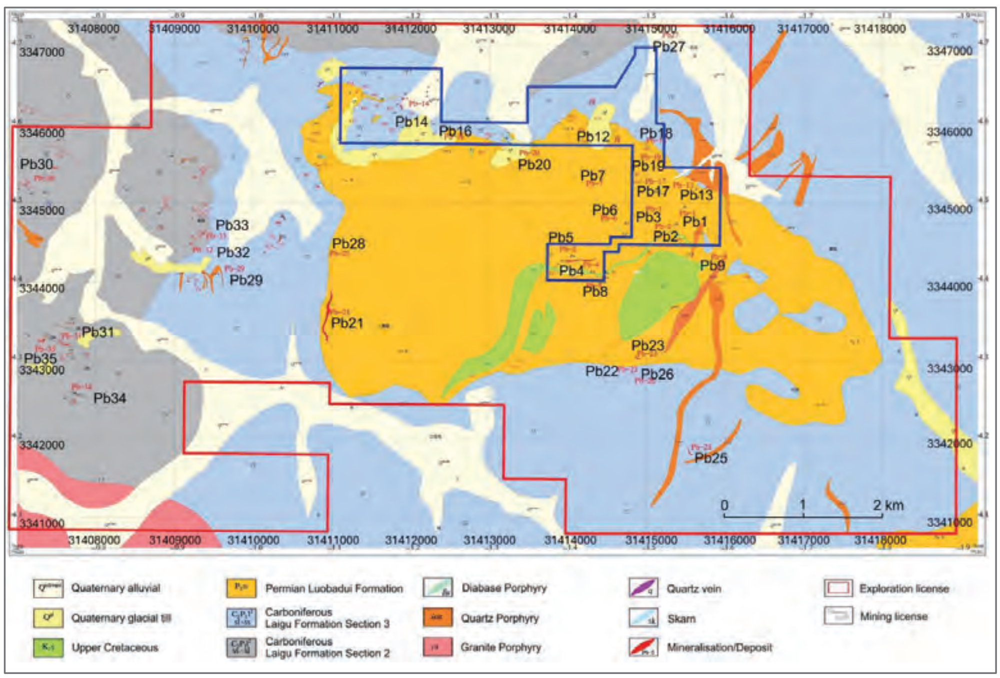
Year	Hole (count)	Length (m)	Assay sample (count)	Bulk density samples (count)
2004	4	304.96	28	0
2006	43	5,157.62	336	67
2007	3	492.74	25	18
2008	4	1,107.11	90	0
2009	3	253.31	50	0
2010	2	96.83	37	0
2011	40	10,472.69	681	0
2012	34	11,688.78	605	89
2014	14	3,041.60	249	33
2015	15	4,230.25	472	42
2017	27	7,472.95	476	47
2018	10	1,892.07	266	21
2019	15	4,314.48	385	40
2020	20	3,960.26	525	38
2021	19	5,588.03	580	33
Total	**253**	**60,073.68**	**4,805**	**428**

Source: SRK, 2025

5.3 Prospects and deposits

A total of 31 prospects and deposits have been identified within the project mining and exploration licences to date (Figure 5.3). These prospects and targets are labelled from Pb1 to Pb35 (noting that Pb10, Pb11, Pb15 and Pb24 are not included).

Figure 5.3: Prospects and deposits



Source: Zhihui, 2020

Among these, the Pb1, Pb2, Pb3, Pb4, Pb5, Pb12, Pb13, Pb14, Pb16, Pb17, Pb18 and Pb19 prospects are located within the current mining licence. The remaining 19 prospects are located within the central and western parts of the exploration licence. Detailed information about these prospects and deposits is presented in Table 5.2 and Section 7.

Table 5.2: Prospects and deposits features

No.	Prospect/Deposit	Description
1	Pb1	■ Dimensions: 50 m long by 3 m wide ■ Ore type: oxide ore ■ Mineralisation: vein-shaped ■ Drilling: 9 shallow drill holes ■ Ore minerals: galena, sphalerite ■ Strike: 111° with a steep dip
2	Pb2	■ Dimensions: 30 m long by 5 m wide ■ Mineralisation: vein-shaped ■ Strike: 103° with a steep dip
3	Pb3	■ Dimensions: 20 m long by 10 m wide ■ Ore minerals: galena, pyrite, limonite ■ Strike: 102°
4	Pb4	■ Dimensions: 450 m long by 11 m wide ■ Ore type: oxide ore ■ Drilling: Controlled by 4 holes and 1 trench, and samples were not preserved
5	Pb5	■ Dimensions: 43 m long by 14 m wide ■ Ore Type: oxide ore ■ Ore minerals: galena, sphalerite ■ Strike: 137°
6	Pb6	■ Size: Small ■ Ore type: oxide ore ■ Ore minerals: galena, sphalerite ■ Drilling: 4 shallow drill holes ■ Strike: 132°, vertical
7	Pb7	■ Dimensions: 15 m long by 2 m wide ■ Mineralisation: vein-shaped ■ Ore minerals: galena, limonite ■ Strike: 280° with a steep dip angle
8	Pb8	■ Dimensions: 15 m long by 1 m wide ■ Mineralisation: None discovered ■ Drilling: 7 shallow drill holes
9	Pb9	■ Features: 4 small veins (0.1 m in width) ■ Drilling: 1 trench ■ Ore minerals: galena ■ Strike: 125° with a moderate dip

No.	Prospect/Deposit	Description
10	Pb12	Described in Table 7.3
11	Pb13	■ Dimensions: 80 m long by 80 m wide ■ Ore minerals: galena, sphalerite ■ Drilling: 2 drill holes and 1 horizontal adit ■ Strike: 280° with a steep dip angle
12	Pb14	Described in Table 7.3
13	Pb16	■ Dimensions: 30 m long by 1.7 m wide ■ Strike: 140°
14	Pb17	■ Dimensions: 120 m long by 0.6 m wide ■ Ore type: oxide ore ■ Mineralisation: Vein-shaped ■ Ore minerals: galena, sphalerite ■ Strike: 133° with a steep dip angle
15	Pb18	■ Dimensions: 20 m long by 0.5 m wide ■ Ore type: oxide ore ■ Drilling: 14 shallow drill holes ■ Strike: 270° with a subvertical dip angle
16	Pb19	■ Dimensions: 135 m long by 2.1 m wide ■ Trenching: 1 trench ■ Mineralisation: vein-shaped ■ Ore minerals: galena, sphalerite, cerussite ■ Strike: 220° with a subvertical dip angle
17	Pb20	■ Dimensions: 193 m long by 9.6 m wide ■ Alteration: Limonitisation ■ Strike: 90°
18	Pb21	Dimensions: 600 m long by 40 m wide
19	Pb22	No information provided
20	Pb23	No information provided
21	Pb25	No information provided
22	Pb26	No information provided

No.	Prospect/Deposit	Description
23	Pb27	▪ Drilling: 19 shallow drill holes, 3 intersected mineralisation ▪ Domains: 3 separate domains
	Pb27-1	▪ 8 m long by 1.5 m wide ▪ Ore minerals: galena, sphalerite ▪ Strike: 175° with a steep dip angle
	Pb27-2	▪ 4 m long by 1 m wide ▪ Ore minerals: galena, sphalerite ▪ Strike: 175° with a steep dip angle
	Pb27-3	▪ 2 m long by 2 m wide ▪ Ore type: oxide ore ▪ Ore minerals: galena, sphalerite ▪ Strike: dips to 110° with a 30° dip angle
24	Pb28	No information provided
25	Pb29	▪ Dimensions: 193 m long by 9.6 m wide ▪ Alteration: Hematitisation
26	Pb30	▪ Location: Adjacent to the Longmala Pb-Zn-Cu Project, west of Mengya'a exploration licence ▪ Mineralisation: Weak hematitisation ▪ Ore minerals: Pyrite
27	Pb31	No information provided
28	Pb32	No information provided
29	Pb33	No information provided
30	Pb34	No information provided
31	Pb35	No information provided

Source: Zhihui, 2020

The mineralisation at the Pb12 deposit has been interpreted to comprise a total of 31 domains, whereas the Pb14 deposit has been interpreted to consist of 6 domains. The dimensions and orientation of these domains are presented in Table 7.3.

Following discussions with site geologists, oxidation has been observed in both the Pb12 and Pb14 deposits. Oxidation logging records are available for the Pb12 deposit but are not available for the Pb14 deposit.

5.4 Historical quantum of mineralisation estimation

Between 2004 and 2021, the Second Brigade, in collaboration with Huaxia Mining, undertook exploration activities and quantification of the likely mineralisation. These estimates were conducted in accordance with prevailing Chinese standards. The results are detailed in Table 5.3 .

Table 5.3: Historical quantum of mineralisation estimates

Year	Team	Cut-off grade (%)	Tonnage (kt)	Pb grade (%)	Pb metal (kt)	Zn grade (%)	Zn metal (kt)
2004	Second Brigade	Open-pit, fresh ore: Pb>0.3% or Zn>0.5% Underground fresh ore: Pb>0.5% or Zn>1.0%	111	21.22	23	1.48	2
2007	Second Brigade		5,349	1.40	75	6.16	330
2012	Second Brigade		8,336	4.67	184	5.33	455
2013	Second Brigade		8,565	2.84	243	5.67	481
2019	Second Brigade		10,669	3.67	392	4.79	511
2020	Huaxia Mining		5,580	4.90	273	6.25	349
2023	Huaxia Mining		12,977	3.31	430	4.65	604

Source: Zhihui

Notes: Estimation of quantum of mineralisation beyond current mining licence; numbers are rounded; the estimations were not prepared in accordance with the JORC Code (2012).

5.5 2024 infill and validation program

From October to November 2024, following SRK's recommendations, Zhihui conducted an infill and validation drilling program to confirm the historical exploration results and upgrade the resource categories. This program included two diamond twinned holes (SRK001 and SRK002) on the pit bench floor, three trenches (TR001 to TR003) on the pit bench face and 19 underground channels (Figure 5.4).

Figure 5.4: Infill and verification drilling and underground channel sampling

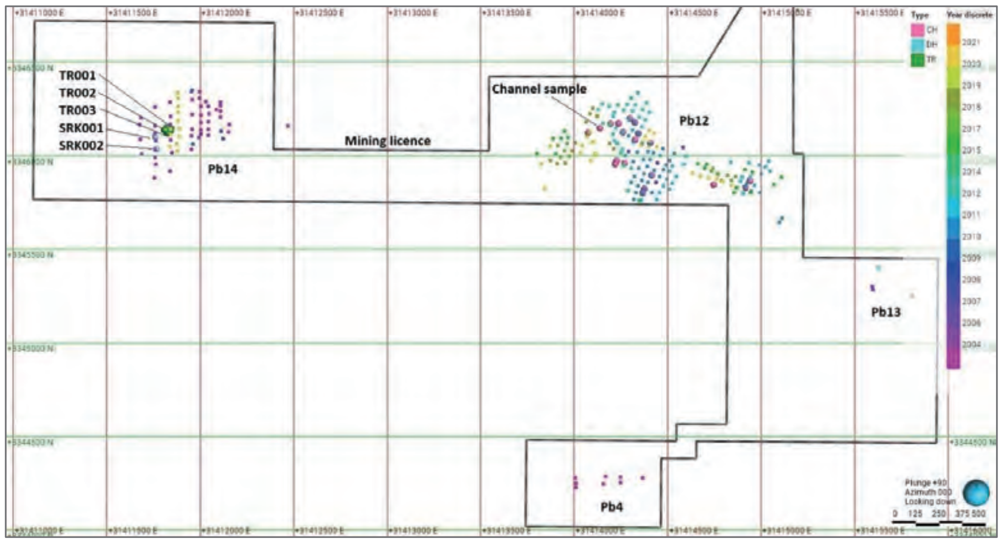
Source: SRK, 2025

Core drilling was conducted using XY-4 drill rigs. Each drill hole began with a PQ size (85 mm) collar to penetrate the topsoil, transitioning to HQ size (63.5 mm) for deeper sections. Core recovery ranged from 95% to 100%, with an average recovery of 98%.

Trench samples collected from the pit bench face and underground channel samples from the drives were all continuous channel intervals of consistent width, depth and length measuring approximately 10 cm × 5 cm × 1.5 m, channelled either by chisels or saws.

The plan view of the location for these drill holes and channels is presented in Figure 5.5. Table 5.4 presents the statistics for the 2024 validation program. A total of 139 samples were taken and sent to SGS Tianjin Laboratory for determination of Pb, Zn, Cu and Ag contents.

Figure 5.5: 2024 infill and validation exploration program



Source: SRK, 2025

Table 5.4: 2024 infill and validation program statistics

Type	Deposit	Purpose	Drill hole (count)	Profiles (m)	Assay records (count)
Twinned hole	Pb14	Validation	2	164.95	47
Trench	Pb14	Infill	3	108	54
Channel sampling	Pb12	Validation	19	57	38
Total			**24**	**329.95**	**139**

Source: SRK, 2025

6 Sample preparation and assaying

6.1 Historical samples

Samples were sent to various laboratories for analysis over different periods. From 2004 to 2010, all samples were analysed at the Central Laboratory of the Xizang Autonomous Region Geological and Mineral Exploration and Development Bureau (Xizang Central Laboratory). For the years, 2011 to 2012 and 2017 to 2021, samples were analysed at Lhasa Jinzang Rock and Mineral Analysis and Testing Co., Ltd., also known as Lhasa Rock and Mineral Analysis and Testing Co., Ltd. (Jinzang Laboratory). In 2014 and 2015, the samples were sent to the Yanjiao Central Laboratory of the North China Nonferrous Geological Exploration Bureau (Yanjiao Laboratory).

6.1.1 Sample preparation

Sample preparation procedures at various laboratories followed the prevailing Chinese standard protocols. The samples were crushed to a size of 2 mm using a jaw crusher, and then pulverised to a fine particle size of 140 mesh (106 μm). Following this, the powdered samples were thoroughly blended and divided into two 500 g aliquots: a primary sample and a duplicate sample. The primary samples were further pulverised to a finer particle size of 200 mesh (74 μm). Finally, pulp samples weighing 20 g each were analysed using atomic absorption spectroscopy (AAS) to determine the concentrations of Pb, Zn, Cu and Ag.

Samples with a combined Pb+Zn grade of greater than 20% were titrated prior to analysis to ensure the accuracy:

1. Lead (Pb): Pulp samples, each weighing 0.30 g, were dissolved using a mixture of nitric acid (HNO_3), sulfuric acid (H_2SO_4) and bromine solution (Br_2). Hydrobromic acid (HBr) was added to the solution to volatilise arsenic, antimony and tin. The precipitated lead sulphate ($PbSO_4$) was separated and dissolved in an ammonium acetate solution (CH_3COONH_4). The lead content was then titrated using an ethylene diamine tetraacetic acid (EDTA) solution, with xylenol orange serving as the indicator.
2. Zinc (Zn): Pulp samples, each weighing 0.20 g, were dissolved using a mixture of hydrochloric acid (HCl), nitric acid (HNO_3) and sulfuric acid (H_2SO_4). Elements such as iron, manganese and lead were precipitated and filtered out. The resulting solution was transferred to an acetic acid-sodium acetate buffer solution at a pH of 5–6. The combined content of zinc and cadmium was titrated with a Na_2-EDTA standard solution, using xylenol orange as the indicator. The zinc content was determined by subtracting the cadmium content from the combined total.

6.1.2 Quality assurance and quality control

The quality assurance and quality control (QA/QC) protocol adopted historically included the insertion of laboratory duplicates and inter-laboratory checks.

2004 to 2010 batches

During the 2004–2010 exploration period, 566 samples were collected by the Second Brigade and sent to Xizang Central Laboratory for assay testing. However, no laboratory duplicates or QA/QC program was documented for these samples.

2011–2012 and 2017–2021 batches

During the 2011–2012 and 2017–2021 exploration phase, a total of 3,518 samples were collected by the Second Brigade and analysed at the Jinzang Laboratory.

Among the 742 samples collected in the 2017–2018 exploration phase, 85 laboratory duplicates and 60 inter-laboratory checks were re-assayed, accounting for 11.46% and 8.09% of all the assays, respectively (Table 6.1 and Table 6.2). SRK considers the proportion of duplicate assays to be reasonable. The inter-laboratory duplicates were analysed at the Hebei Provincial Geological and Mineral Centre Laboratory (Hebei Laboratory). SRK considers the historical QA/QC results to be satisfactory, with no material bias evident.

Table 6.1: 2017–2018 exploration phase laboratory duplicates statistics

Metal	Primary			Duplicates			No. of pairs	Correlation coefficient
	Mean	Standard deviation	Median	Mean	Standard deviation	Median		
Pb%	2.59	1.94	2.20	2.61	1.91	2.19	85	1.000
Zn%	2.49	2.54	1.82	2.50	2.52	1.82	85	1.000
Cu%	0.17	0.20	0.13	0.18	0.20	0.14	85	0.997
Ag (g/t)	26.78	25.56	19.90	27.34	25.31	20.30	85	0.997

Source: SRK, 2025

Table 6.2: 2017–2018 exploration phase inter-laboratory check statistics

Metal	Primary			Duplicates			No. of pairs	Correlation coefficient
	Mean	Standard deviation	Median	Mean	Standard deviation	Median		
Pb%	2.93	2.09	2.53	2.91	2.03	2.51	60	0.996
Zn%	2.59	2.75	1.81	2.55	2.56	1.83	60	0.981
Cu%	0.22	0.26	0.16	0.23	0.26	0.15	60	0.998
Ag (g/t)	36.29	47.74	19.55	35.46	47.67	18.65	60	0.996

Source: SRK, 2025

2014–2015 batches

During the 2014–2015 exploration phase, a total of 721 samples were collected by the Second Brigade and analysed in Yanjiao Central Laboratory. Among them, 76 laboratory duplicates and 53 inter-laboratory checks were re-assayed, accounting for 10.54% and 4.85% of the 721 assays, respectively (Table 6.3 and Table 6.4). SRK considers the proportion of duplicate assays to be reasonable. The inter-laboratory duplicates were analysed at the Hebei Laboratory. SRK considers the historical QA/QC results to be satisfactory, with no material bias evident.

Table 6.3: 2014–2015 exploration phase laboratory duplicates statistics

Metal	Primary			Duplicates			No. of pairs	Correlation coefficient
	Mean	Standard deviation	Median	Mean	Standard deviation	Median		
Pb%	8.15	12.10	4.14	8.14	12.12	4.11	76	1.000
Zn%	4.96	5.99	2.39	4.97	5.99	2.40	76	1.000
Cu%	0.28	0.35	0.18	0.29	0.34	0.19	76	0.999
Ag (g/t)	56.46	61.87	33.10	56.11	61.46	34.05	76	0.999

Source: SRK, 2025

Table 6.4: 2014–2015 exploration phase inter-laboratory check statistics

Metal	Primary			Duplicates			No. of pairs	Correlation coefficient
	Mean	Standard deviation	Median	Mean	Standard deviation	Median		
Pb%	5.82	9.53	35	5.84	9.54	2.11	35	1.000
Zn%	4.04	5.12	35	4.05	5.13	1.85	35	1.000
Cu%	0.20	0.23	35	0.21	0.24	0.17	35	0.993
Ag (g/t)	51.24	48.66	35	51.45	48.98	33.00	35	0.999

Source: SRK, 2025

6.2 2024 infill and validation program

6.2.1 Sample preparation and test

Sample preparation was conducted at the Tianjin SGS laboratory. Initially, samples were dried and crushed to a size of 3 mm, then homogenised. A 1.0 kg portion was extracted and further ground to a particle size of 200 mesh size (74 μm).

For analysis, pulp samples weighing 0.20 g each were placed in a 50 mL PTFE (polytetrafluoroethylene) container. To each sample, 2 mL of nitric acid (HNO_3), hydrochloric acid (HCl), hydrofluoric acid (HF), and perchloric acid ($HClO_4$) were added. The solution was evaporated to dryness, and the remaining residue was cooled. Subsequently, 2 mL of HCl, 1 mL of nitric acid (HNO_3), and 10 mL of water were added, and the mixture was heated to dissolve any precipitate. The resulting solution was then diluted with 12 mL of water, making it ready for inductively coupled plasma optical emission spectroscopy (ICP-OES) testing. This testing method is referred to as ICP40Q in the SGS laboratory.

6.2.2 SRK validation

For the 2004–2010 period, core and pulp samples were not retained. For verification of the results from these campaigns, SRK has relied on information from twinned hole drilling and channel sampling, with associated assays demonstrating a reasonable alignment.

For the 2011–2021 period, 52 pulp samples and 107 core samples were collected and sent to SGS Tianjin Laboratory for validation.

Pulps duplicates

Over one third of the historical pulp samples were not available for re-sampling. A total of 52 pulp duplicates were collected from the historical pulp samples of the Pb12 drill holes, which represented 1.29% of the 4,025 drill hole samples. SRK considers the proportion of duplicate assays to be barely adequate and recommends improving the storage conditions and protocols. The results are shown in Table 6.5. Despite the low count, SRK considers the available results to be satisfactory, with no material bias evident.

Table 6.5: Pulp duplicates statistics

Metal	Primary			Duplicates			No. of pairs	Correlation coefficient
	Mean	Standard deviation	Median	Mean	Standard deviation	Median		
Pb%	7.33	8.55	3.30	7.45	8.66	3.23	52	0.993
Zn%	5.52	7.28	2.55	5.69	7.28	2.72	52	0.987
Cu%	0.38	0.53	0.15	0.40	0.55	0.17	52	0.996
Ag (g/t)	55.88	73.83	26.25	55.27	72.71	32.35	52	0.959

Source: SRK, 2025

Core duplicates

Duplicates were collected from a total of 107 quarter-core historical samples of Pb12 drill holes, which represented 2.66% of the 4,025 drill hole samples. The results are shown in Table 6.6. SRK considers the duplicate ratios to be reasonable.

Table 6.6: Core duplicates statistics

Metal	Primary			Duplicates			No. of pairs	Correlation coefficient
	Mean	Standard deviation	Median	Mean	Standard deviation	Median		
Pb%	5.76	6.73	3.21	6.23	7.29	3.79	107	0.936
Zn%	6.45	6.45	4.68	6.62	6.48	5.42	107	0.870
Cu%	0.36	0.44	0.17	0.39	0.50	0.22	107	0.783
Ag (g/t)	60.61	79.72	28.60	60.01	68.62	38.40	107	0.658

Source: SRK, 2025

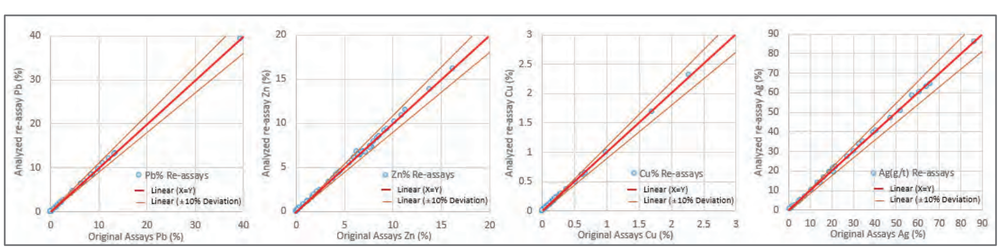
SRK considers the results for Pb, Zn, and Cu to be satisfactory, with no significant bias detected. However, the results revealed a slight bias in Ag duplicates between the SGS Laboratory and previous laboratories. SRK noticed that the assaying methods used by the SGS Laboratory differ from those employed by the previous laboratories. SRK recommends further investigation in future to determine the underlying cause of this bias.

6.2.3 Quality assurance and quality control

Laboratory duplicates

In all, 46 duplicates were re-assayed in SGS Laboratory testing, which represented 15.44% of the 298 field samples. The results are shown in Figure 6.1. SRK considers the results to be satisfactory, with no material bias evident.

Figure 6.1: Laboratory duplicates



Source: SRK, 2025

Blanks

A total of 20 blank tests were inserted in the sample batches at a frequency of one in every 15 samples. The blank is a test conducted with deionised water, but following the same procedures and conditions as normal sample measurement. The results were returned with most of Pb, Zn and Cu values lower than 0.0001%, and Ag values lower than 0.5 g/t, demonstrating there were no contamination issues (Figure 6.2 and Figure 6.3).

Figure 6.2: Blanks – Cu, Pb and Zn

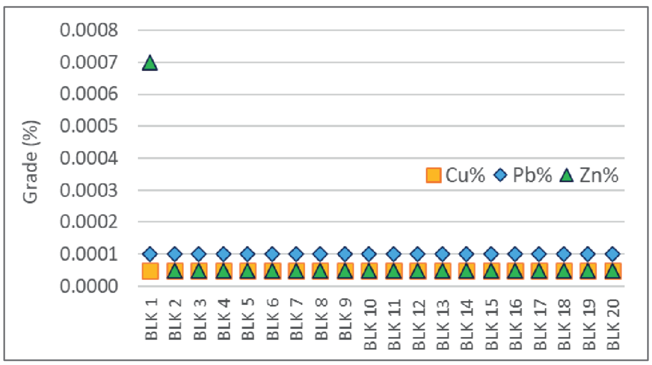
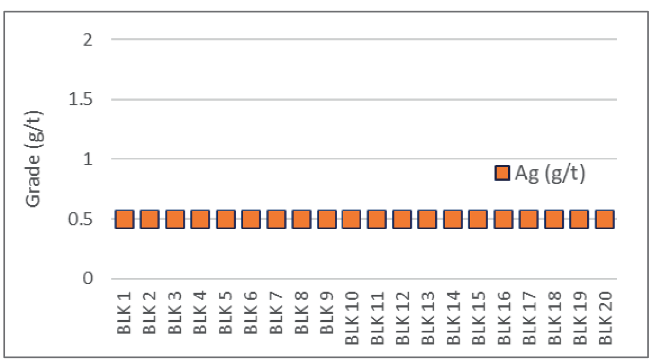


Figure 6.3: Blanks – Ag



Source: SRK, 2025

Standards

A total of 20 certified reference materials (CRMs) were inserted in the sample batches at a frequency of one in every 15 samples.

The CRM results, as well as the expected mean values and their acceptable limits, are presented in Figure 6.4, Figure 6.5, Figure 6.6, Figure 6.7 and Figure 6.8, and listed in Table 6.7.

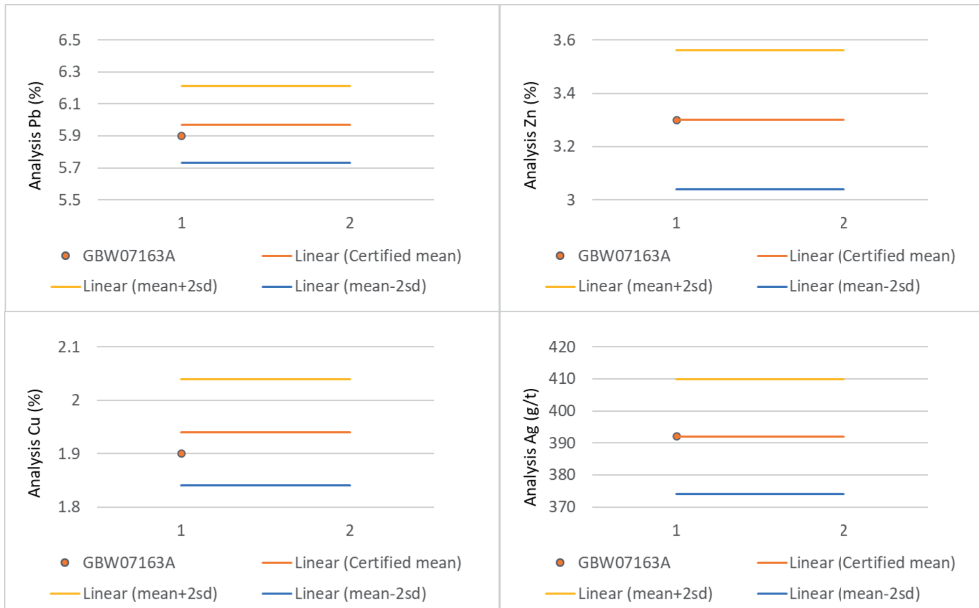
All results are within the expected values, except the Pb and Zn grades of the OREAS624 samples that yielded a value slightly below the expected value (Figure 6.8, upper left and upper right), and one Zn grade of the GBW07709 samples that yielded a value slightly above the expected value (Figure 6.6, upper right). In SRK's opinion, no systemic bias is evident.

Table 6.7: Standards used in the program

Standard	Metal	Certified mean	Acceptable deviation limit (2 SD)	Unit	Number of samples
GBW07163A	Pb	5.97	±0.24	%	1
	Zn	3.3	±0.26	%	1
	Cu	1.94	±0.1	%	1
	Ag	392	±18	g/t	1
GBW07165	Pb	5.13	±0.16	%	7
	Zn	13.9	±0.4	%	7
	Cu	0.096	±0.014	%	7
	Ag	148	±12	g/t	7
GBW07709	Pb	0.1	±0.004	%	2
	Zn	0.1	±0.004	%	2
	Cu	0.1	±0.004	%	2
	Ag	10	±1	g/t	2
OREAS 620	Pb	0.774	±0.044	%	5
	Zn	3.15	±0.194	%	5
	Cu	0.173	±0.008	%	5
	Ag	38.5	±3.08	g/t	5
OREAS 624	Pb	0.624	±0.038	%	5
	Zn	2.4	±0.156	%	5
	Cu	3.1	±0.158	%	5
	Ag	45.3	±2.52	g/t	5

Source: SRK, 2025

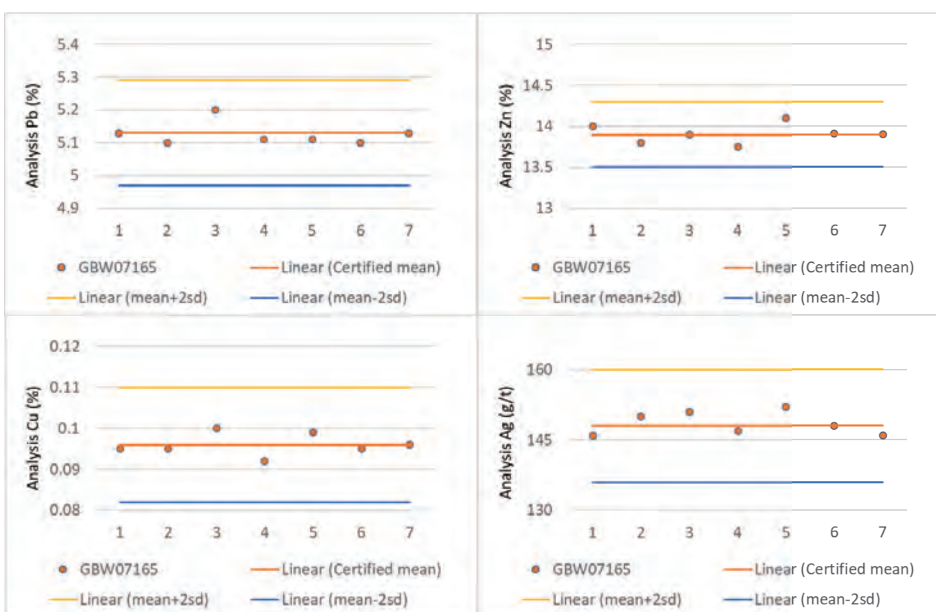
Figure 6.4: CRM results – GBW07163A



Source: SRK, 2025

Notes: Solid orange line represents the certified value while the yellow and blue lines indicate the ±2 SD levels.

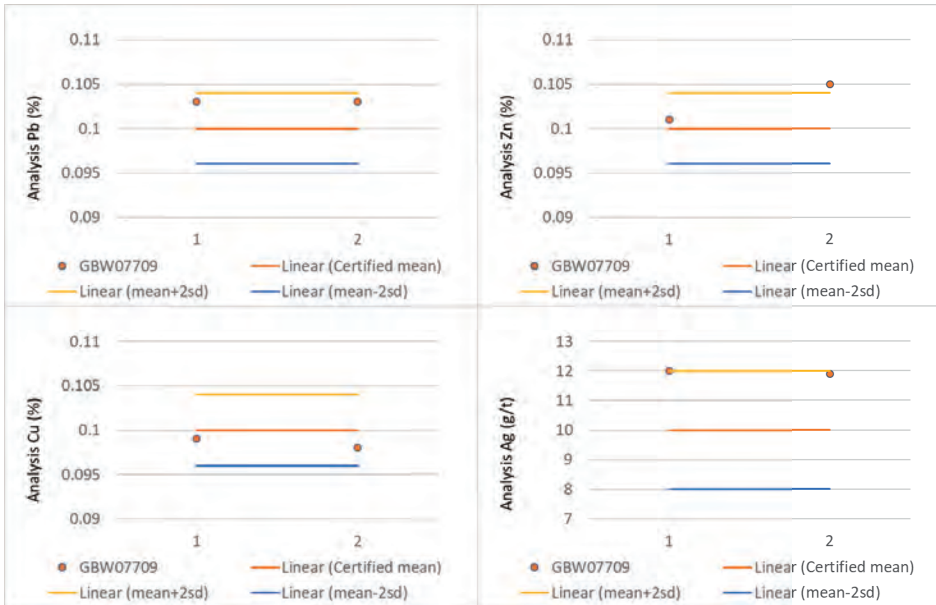
Figure 6.5: CRM results – GBW07165



Source: SRK, 2025

Notes: Solid orange line represents the certified value while the yellow and blue lines indicate the ±2 SD levels.

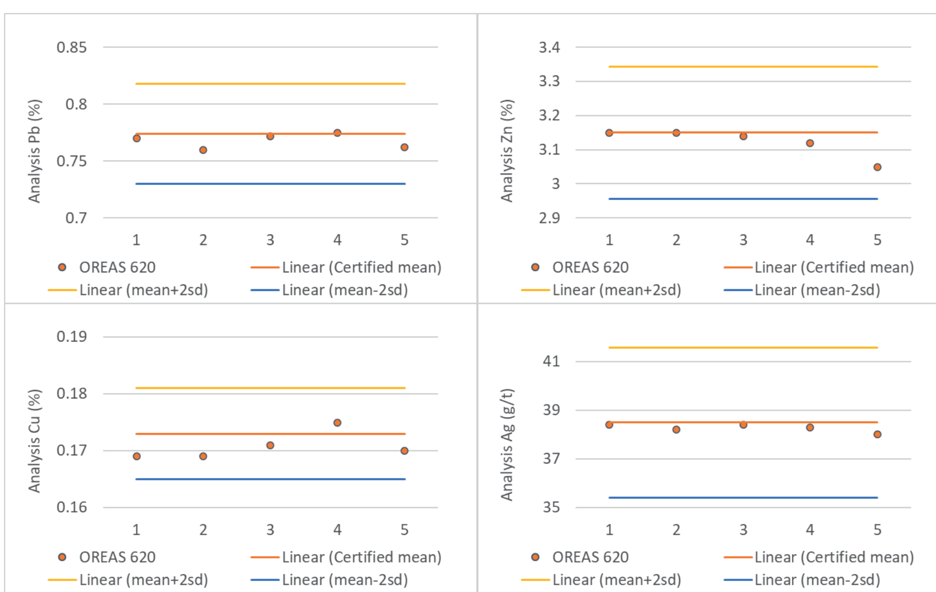
Figure 6.6: CRM results – GBW07709



Source: SRK, 2025

Notes: Solid orange line represents the certified value while the yellow and blue lines indicate the ±2 SD levels.

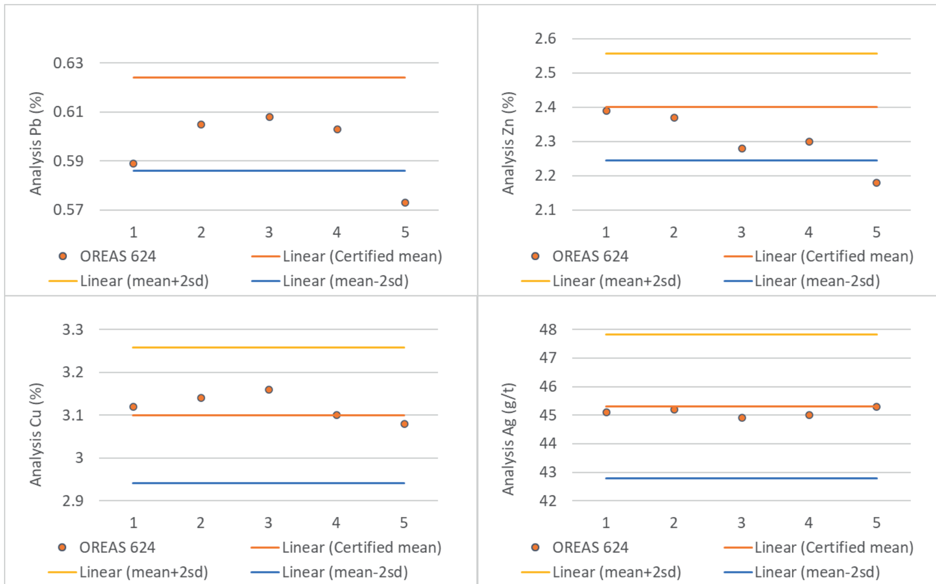
Figure 6.7: CRM results – OREAS 620



Source: SRK, 2025

Notes: Solid orange line represents the certified value while the yellow and blue lines indicate the ±2 SD levels.

Figure 6.8: CRM results – OREAS 624



Source: SRK, 2025

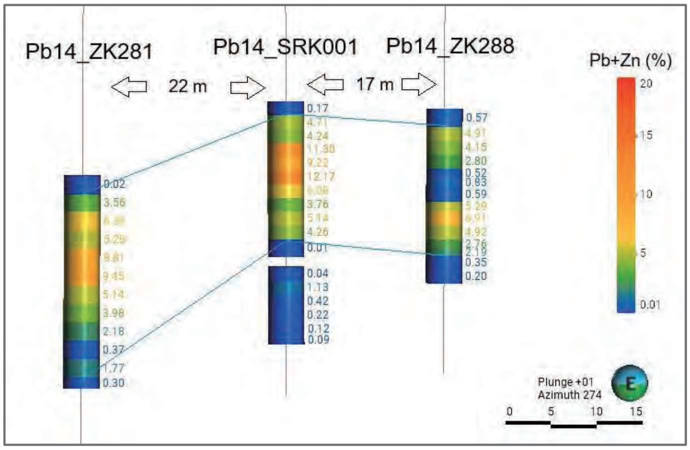
Notes: Solid orange line represents the certified value while the yellow and blue lines indicate the ±2 SD levels.

6.2.4 Twinned holes and channels validation

Twinned holes

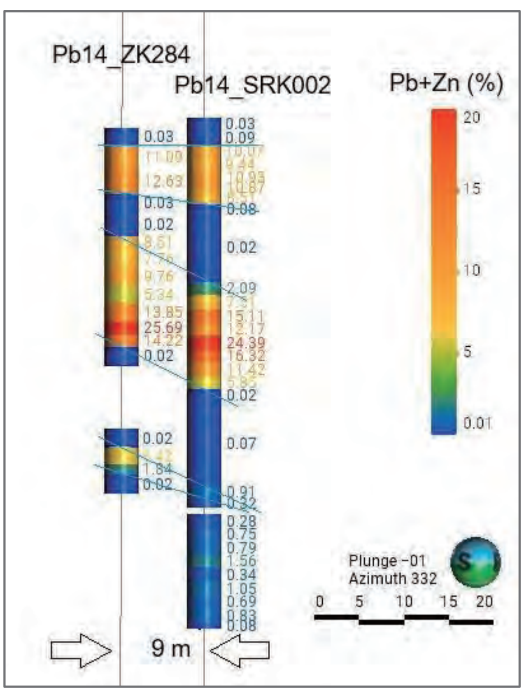
Twinned holes, denoted as SRK001 and SRK002, were completed close to the historical drill holes (Hole Pb14_ZK288 and Hole Pb14_ZK284). These twinned holes were drilled 17 m and 9 m from the original holes, respectively (Figure 5.5). Core samples were collected from intersections of mineralised limestones for assaying (Pb, Zn, Cu and Ag). The assay results of Pb+Zn grade from the twinned holes were compared to those from the original holes (Figure 6.9 and Figure 6.10).

Figure 6.9: Twinned hole Pb14_SRK001 comparison



Source: SRK, 2025

Figure 6.10: Twinned hole Pb14_SRK002 comparison



Source: SRK, 2025

The assay result for twinned hole SRK001 indicates a 14 m-thick mineralised zone, constrained by Pb+Zn grade. It is comparable to the 14 m-thick mineralised intervals in Hole Pb14_ZK288.

In twinned hole SRK002, the result indicates three mineralised zones of 6.75 m, 12 m and 1.5 m, from top to bottom respectively, constrained by combined Pb+Zn grade. They are comparable to three mineralised zones of 5.5 m, 12 m and 3 m mineralised intervals in Hole Pb14_ZK284, from top to bottom respectively.

These averaged grades indicated a reasonable consistency, except for a lower Zn grade in twinned hole SRK002, as shown in Table 6.8.

Table 6.8: Comparison of average grades – twinned holes vs original holes

Metal	SRK001	Pb14_ZK288				
	Zone 1 (14 m thick)					
Pb%	0.01	0.26				
Zn%	6.76	3.00				
Cu%	0.21	0.11				
Ag (g/t)	2.51	10.43				
Metal	**SRK002**	**Pb14_ZK284**				
	Zone 1 (6.75 m thick)		**Zone 2 (12 m thick)**		**Zone 3 (1.5 m thick)**	
Pb%	0.52	0.19	0.73	0.14	0.83	0.63
Zn%	9.44	11.67	11.13	12.02	0.08	3.51
Cu%	0.11	0.26	0.07	0.14	0.00	0.13
Ag (g/t)	29.06	16.50	22.19	8.99	34.70	12.80

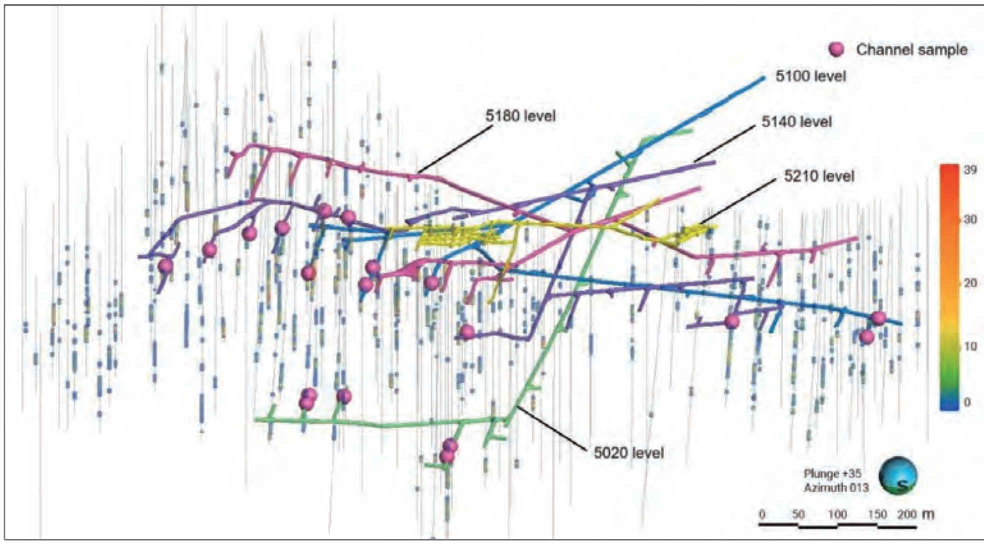
Source: SRK, 2025

The results shown in Figure 6.9 and Figure 6.10 suggest a good consistency in the mineralised zones between twinned and original holes.

Channels

Further to SRK's recommendations, 38 channel samples were collected from 19 locations across three levels (5020, 5100, and 5140 levels) for validation (Figure 6.11).

Figure 6.11: Channel sampling – Pb12 deposit

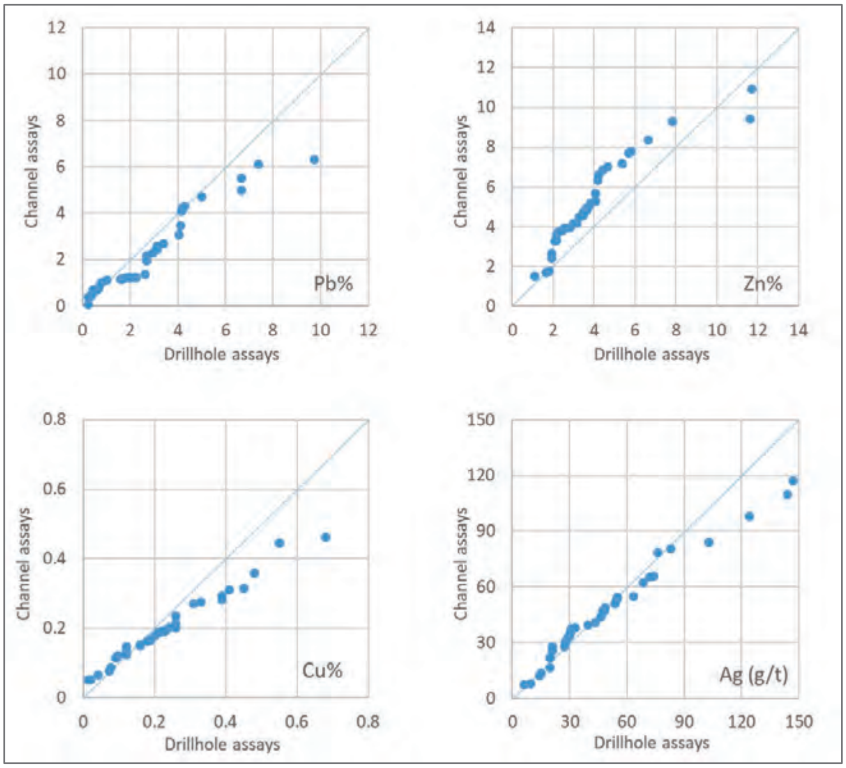


Source: SRK, 2025

The assays from these channel samples were compared with nearby drill holes to assess the representative grade of the known mineralisation. The average distance between the channel samples and the nearest drill holes is 11.6 m.

The assay results for Pb, Zn, Cu and Ag from the channel sampling were plotted against those from the nearby drill holes using Q-Q plots, as illustrated in Figure 6.12.

Figure 6.12: Q-Q plots of channel sampling against nearby holes



Source: SRK, 2025

The results shown in Figure 6.12 are a reasonable reflection of the interpretation of the mineralised zones based on nearby drill holes with an average distance of 11.6 m. However, there is a noted deviation in the Zn grade Q-Q plot (Figure 6.12, upper right), suggesting that further investigation is warranted.

6.3 Bulk density

A total of 428 samples were collected for bulk density measurement. SRK compiled the received information for 316 samples, comprising oxide ore from Pb12 and fresh ore from Pb4, Pb12, Pb13 and Pb14 (Table 6.9). These samples were measured using the water immersion method, whereby the density of the sample is determined by dividing its mass measured in air by the volume of water it displaces when submerged. The results are shown in Table 6.9.

Table 6.9: Bulk density results

Ore type	Deposit	Count	Average density (g/cm^3)	Average Pb+Zn grade (%)
Oxide	Pb12	20	2.48	15.47
	Pb4	12	4.20	23.47
Fresh	Pb12	209	3.60	13.97
	Pb13	8	3.60	16.00
	Pb14	67	3.27	6.79
Total		316	3.49	12.96

Source: SRK, 2025.

6.4 Conclusion

A review of the Xizang Central Laboratory sampling procedures and preparation during the 2004–2010 exploration period suggests that significant issues with the sample preparation methods are unlikely. However, no laboratory duplicates or QA/QC program was documented in association with these samples. To address this, SRK implemented a twinned holes validation process, and the results demonstrated good consistency with historical assay results. Core recoveries for the 2004 drill holes ranged from 71% to 100%, with an average of 78%, which is suboptimal. In contrast, core recoveries for the 2006–2010 drill holes ranged from 69.1% to 98.4%, with an average of 85%, which is considered satisfactory.

In the 2014–2015 exploration period, laboratory duplicates and inter-laboratory checks of samples analysed at the Yanjiao Laboratory suggested good reproducibility. For the samples analysed at the Jinzang Laboratory during the 2011–2012 and 2017–2021 exploration periods, duplicates records are only available for the 2017–2018 period. These records from the Jinzang Laboratory demonstrated strong reproducibility, reflecting the laboratory's analytical reliability. Core recoveries for the 2011–2018 drill holes ranged from 80.8% to 94.8%, with an average of 90%. For the 2019–2021 drill holes, core recoveries ranged from 73.8% to 98.7%, with an average of 91%. Overall, core recoveries from 2011–2021 were considered satisfactory.

SRK considers the intervals collected from the 2004–2021 program, as well as the 2024 exploration program, to be adequate for constraining and modelling the mineralised domains.

SRK recommends conducting further studies on the fresh ore density and the correlation between grade and density to better understand the variance in density.

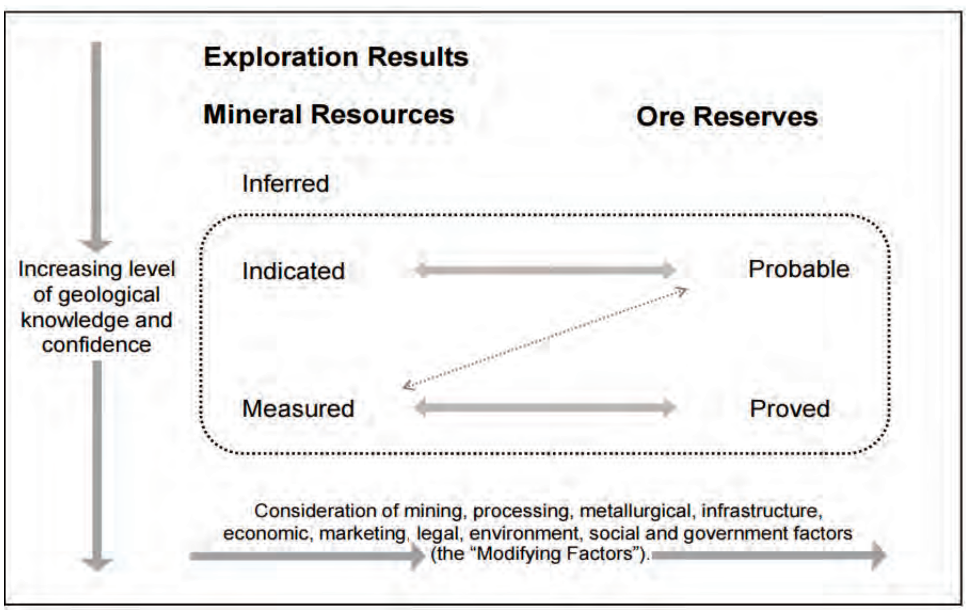
7 Mineral Resource estimation

7.1 Introduction

The quality of the exploration data collected by Zhihui from the period 2004–2024 varies. However, the results from the 2024 validation program and geostatistical analysis indicate that the historical data are generally reasonable. SRK considers that the historical data can be used for modelling the estimation domains to constrain grade estimation. The drilling data – together with trench and underground channel sampling – are considered suitable for estimating block grades in the Mineral Resource block model.

The JORC Code states that '*A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade (or quality), and quantity that there are reasonable prospects for eventual economic extraction*'. Mineral Resources are classified as Measured, Indicated and Inferred according to the degree of geological confidence (Figure 7.1).

Figure 7.1: General relationship between Exploration Results, Mineral Resources and Ore Reserves



Source: JORC Code, 2012.

The following sections summarise the key assumptions, parameters and methods used to estimate the Mineral Resources for the deposit.

7.2 Mineral Resource estimation procedures

Leapfrog software (version 2024.1) was used to generate the geological and ore domain models and used to prepare assay data for statistical/geostatistical analysis, construct the block model, estimate Pb, Zn, Cu and Ag grade and tabulate Mineral Resources.

Three-dimensional (3D) block grade estimation approach was applied for the Project's Mineral Resource estimation, based on kriging or inverse distance weighting (IDW).

The estimation methodology involved the following steps:

- database compilation, verification as well as adjustment
- establishment of geological model based on the logging lithology and oxidation record
- definition of a Resources domain by grade shell
- data conditioning (compositing and capping) exploratory data analysis (compositing) and geostatistical analysis using variography
- block modelling and interpolation of grade and density
- Mineral Resource estimation and validation
- classification of the Mineral Resources
- assessment of 'reasonable prospects for economic extraction' and selection of appropriate cut-off grades
- preparation of the Mineral Resource Statement.

7.3 Database compilation and validation

The Pb4 and Pb13 deposits are in the early stages of exploration; the work completed to date has not identified mineralisation of economic significance. Consequently, the Mineral Resource estimation database was only compiled for the P12 and Pb14 deposits.

Collar, assay, and survey information from previous exploration campaigns were compiled into an MS Excel spreadsheet and validated using Leapfrog software to search for errors such as missing or overlapping intervals, correct hole or channel identifier (ID), azimuths, dips and duplicated samples. The projection of collars and the geological and resource models generated during this study were in China Geodetic Coordinate System (CGCS) 2000/3-degree Gauss-Kruger zone 31 datum.

The exploration database statistics are summarised in Table 7.1.

Table 7.1: Summary of database used for Mineral Resource estimation

Method of sampling	Drill hole (count)	Profiles (m)	Assay records (count)
Historical drilling	246	59,275.98	4,752
2024 SRK trenching	3	108.00	54
2024 SRK drilling	2	164.95	47
2024 SRK channel sampling	19	57.00	38
Total	**270**	**59,605.93**	**4,891**

Source: SRK, 2025

7.4 Wireframe modelling

The wireframe models for the deposits were established using Leapfrog software. A nominal cut-off grade of either Pb ≥0.5% or Zn ≥1.0% was applied to define the mineralised intervals in the Pb12 deposit, while a lower nominal cut-off of either Pb ≥0.3% or Zn ≥0.5% was used to delineate the mineralised intervals in Pb14 deposit (Table 7.2).

Associated Cu and Ag mineralisation, which are of low grade, were constrained within the defined Pb-Zn domains.

Table 7.2: Criteria for determination of mineralised intervals

Deposit	Mining method	Cut-off
Pb12	Underground	Pb ≥0.5% or Zn ≥1.0%
Pb14	Open-pit	Pb ≥0.3% or Zn ≥0.5%

Source: SRK, 2025

Intervals were selected on each drill hole, trench or underground channel. Within each drill hole, the average grades of Pb and Zn weighted by interval length were employed for mineralised intervals determination. The contact points between mineralisation and host rock were generated using the 'vein selection' function, and the mineralised envelopes were built by the 'vein modelling' and 'domain' functions.

A total of 37 domains were defined for the Pb12 and Pb14 deposits. The domain parameters are listed in Table 7.3. The domain extents are shown in Figure 7.2.

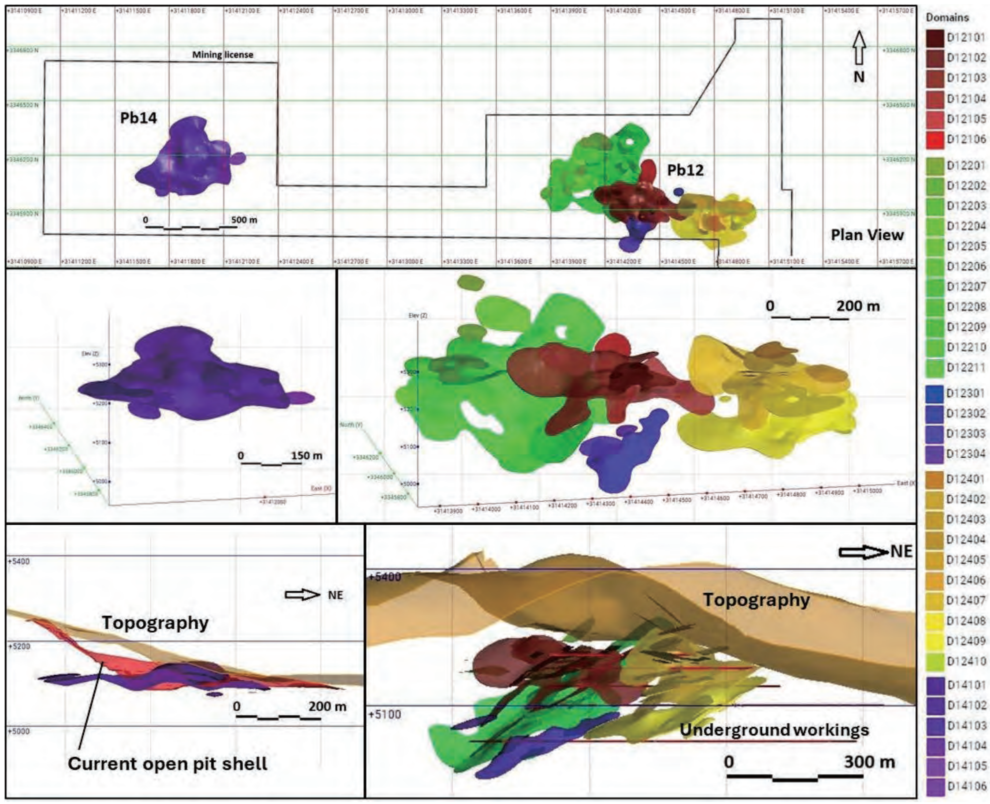
Table 7.3: Interpreted domain parameters – Pb12 and Pb14 deposits

Deposit	Domain	Extension			Dip direction (°)	Dip angle (°)
		Length (m)	Width (m)	Thickness (m)		
Pb12	D12101	35	35	12	240	12.5
	D12102	1430	120	11	221	17
	D12103	446	227	25	201	15
	D12104	282	135	11	180	34
	D12105	170	138	10	228	42
	D12106	373	83	5	193	24
	D12201	78	70	5	161	22.5
	D12202	81	69	4.2	162	22.5
	D12203	173	160	3	210	10
	D12204	150	113	4	184	34
	D12205	88	63	2.7	189	31
	D12206	39	36	7.5	159	22
	D12207	38	34	7	159	22
	D12208	323	276	4.3	203	9
	D12209	211	166	0.58	222	37
	D12210	702	400	14.5	152	30
	D12211	70	61	2.5	192	35
	D12301	58	49	9.5	180	22
	D12302	351	134	43.5	20	32
	D12303	113	96	7	200	21
	D12304	75	51	3	196	9

Deposit	Domain	Extension			Dip direction (°)	Dip angle (°)
		Length (m)	Width (m)	Thickness (m)		
	D12401	120	88	2	211	15
	D12402	102	75	2.5	192	17
	D12403	84	60	3	159	9.5
	D12404	93	74	1.5	78	33
	D12405	173	116	2.5	13	13.5
	D12406	111	88	4	145	17
	D12407	424	265	17	181	35
	D12408	218	183	4.5	182	13
	D12409	475	243	7.5	178	27
	D12410	91	47	2	139	22
Pb14	D14101	190	82	8	203	11
	D14102	530	422	32	145	14
	D14103	160	83	7	127	19
	D14104	44	42	9	153	13
	D14105	48	42	8	138	8
	D14106	90	66	7	152	12

Source: SRK, 2025

Figure 7.2: Interpreted domains



Source: SRK, 2025

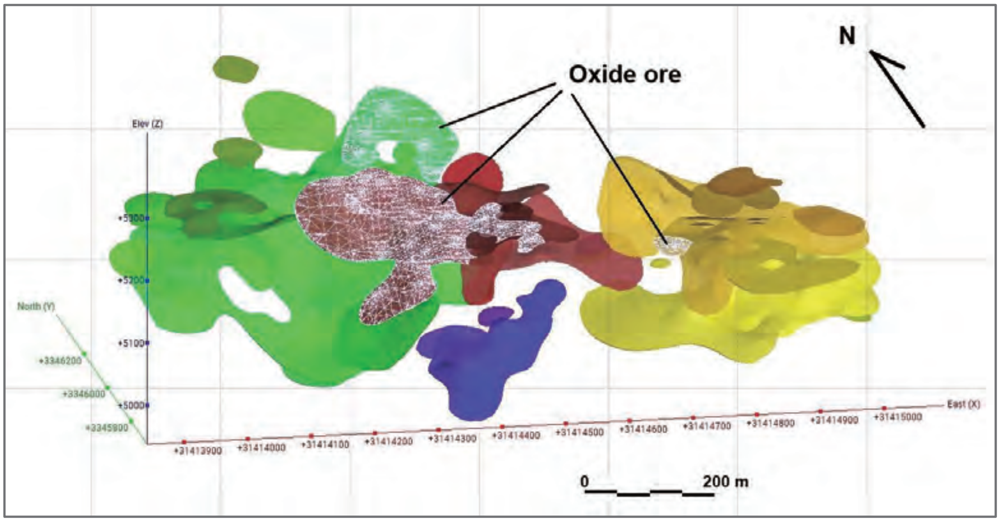
Oxidised mineralisation for the Pb12 deposit was wireframed using the available oxidation logging data. However, there were no oxidation logging records available for the Pb14 deposit to reference.

In the Pb12 deposit, Domain D12105 was found to be completely oxidised, while domains D12101, D12102, D12103, D12210, and D12407 were partially oxidised.

For the Pb14 deposit, all oxide ore has been previously mined out, as the open-pit operation has removed the shallow mineralisation.

The location of the oxidised mineralisation with respect to the other modelled domains is demonstrated in Figure 7.3.

Figure 7.3: Wireframed oxidised mineralisation – overview



Source: SRK, 2025

7.5 Exploratory data analysis

All intervals were used in the development of geological models and the interpretation of mineralised domains. Table 7.4 shows the exploratory data analysis for Pb and Zn raw samples and samples within the interpreted domains of Pb12 and Pb14 deposits.

Table 7.4: Basic statistics for Pb and Zn for all raw samples and samples within the domains

Item	Pb12 raw data (Pb%)	Pb12 within domains (Pb%)	Pb14 raw data (Pb%)	Pb14 within domains (Pb%)
Number of samples	4,063	2,274	828	676
Minimum value	0.0001	0.0009	0.0001	0.0001
Maximum value	60.70	60.70	14.30	14.30
Mean	2.27	3.86	1.04	1.26
Variance	19.50	28.10	2.34	2.59
Standard Deviation	4.42	5.30	1.53	1.61
Coefficient of variation	1.94	1.37	1.47	1.28

Item	Pb12 raw data (Zn%)	Pb12 within domains (Zn%)	Pb14 raw data (Zn%)	Pb14 within domains (Zn%)
Number of samples	4,062	2,274	828	676
Minimum value	0.0010	0.0150	0.0091	0.0180
Maximum value	32.62	32.62	36.12	36.12
Mean	2.48	4.21	4.53	5.51
Variance	16.01	21.00	16.52	14.95
Standard Deviation	4.00	4.58	4.06	3.87
Coefficient of variation	1.61	1.09	0.90	0.70

Source: SRK, 2025

7.5.1 Compositing

Many geostatistical methods of grade estimation assume that the input grade data are on a constant 'support' (mass and shape). Therefore, it is standard practice to composite the samples to a consistent length before conducting interpolation.

SRK conducted a sample composite analysis to determine the most suitable composite length for grade interpolation. This analysis involved examining variations in composite length and the minimum composite lengths for inclusion. The analysis compared the average grade obtained from composites against the length-weighted average grade of the individual raw samples. Additionally, the percentage of total sample length that would be excluded when applying the minimum composite length was assessed.

For the Resources domain (grade shell), the raw samples were composited at intervals of 1.5 m. A minimum coverage of 0.5 m was selected to ensure sufficient representation of the mineralisation was achieved. The basic statistics for each domain are provided in Table 7.5. Histograms are shown in Figure 7.4.

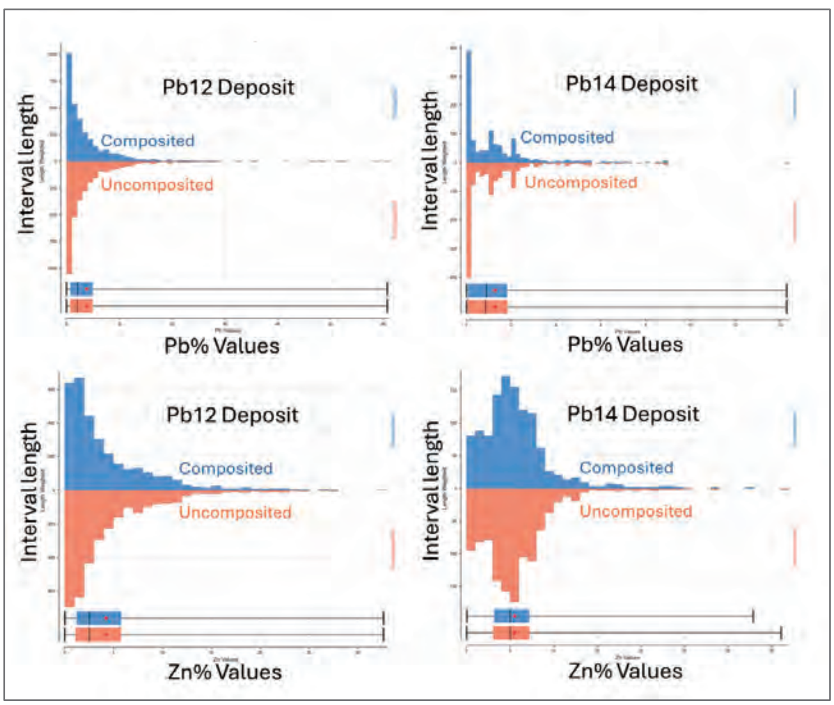
Table 7.5: Basic statistics for Pb and Zn composite values – Resources domain

Item	Pb12 within domains (Pb%)	Pb12 composites (Pb%)	Pb14 within domains (Pb%)	Pb14 composites (Pb%)
Number of samples	2,278	2,188	682	804
Minimum value	0.0009	0.0009	0.0001	0.0001
Maximum value	60.70	39.00	14.30	14.30
Mean	3.90	3.90	1.26	1.26
Variance	28.93	28.14	2.87	2.74
Standard Deviation	5.38	5.31	1.69	1.66
Coefficient of variation	1.38	1.36	1.34	1.31

Item	Pb12 within domains (Zn%)	Pb12 composites (Zn%)	Pb14 within domains (Zn%)	Pb14 composites (Zn%)
Number of samples	2,278	2,188	682	804
Minimum value	0.0150	0.0270	0.0180	0.0500
Maximum value	32.62	32.62	36.12	32.90
Mean	4.25	4.25	5.53	5.53
Variance	21.10	19.98	16.20	14.69
Standard Deviation	4.59	4.47	4.03	3.83
Coefficient of variation	1.08	1.05	0.73	0.69

Source: SRK, 2025

Figure 7.4: Frequency statistics on composites – Resources domain



Source: SRK, 2025

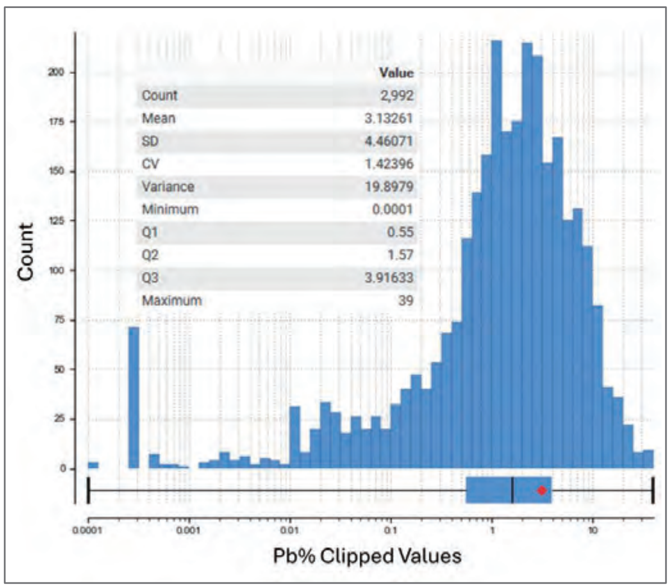
7.5.2 Capping

For some estimates, grade capping may be appropriate to control the influence of the highest-grade samples or composites. After reviewing the composited samples, SRK elected to apply capping to the current estimate. To determine the appropriate capping levels, SRK performed an analysis of the grade distributions using cumulative frequency analysis. The objective of this analysis was to identify the grades at which samples significantly impact the local estimation and exhibit an extreme influence.

Based on the analysis of cumulative frequency for all composites, a grade capping of 39% Pb was used. The statistics and histogram of capped composites are presented in Figure 7.5.

In general, SRK aims to limit the impact of the capping to less than 5% change in the mean value. However, in cases with extreme outliers, the change in the mean can exceed 5%. For this study, 7 composites scattered throughout the deposit are capped, which is equivalent to 0.23% of all composites. The average grade of uncapped composites is 3.20% Pb, while the average grade of capped composites is 3.13% Pb. The difference therefore falls within the acceptable 5% limit on change in mean value.

Figure 7.5: Capped composites frequency – Pb%



Source: SRK, 2025

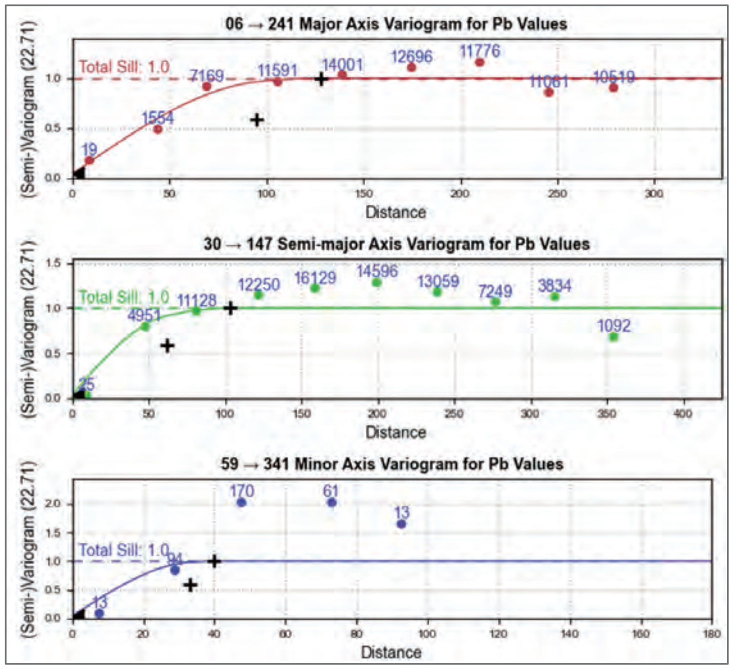
7.6 Variogram modelling

Variogram modelling for the Resources domain was conducted using Leapfrog Edge. The variogram fitting process was completed in the following steps:

- The nugget was determined by the downhole variogram.
- Based on 3D visualisation of grade data, the plane of maximum continuity of mineralisation was interpreted for individual domains.
- Within this plane, the direction of maximum continuity was selected as the major axis of the variogram anisotropy ellipsoid.
- The perpendicular direction within the plane was taken as the semi-major axis of the anisotropy ellipsoid.
- The direction perpendicular to the plane was used as the minor axis of the anisotropy ellipsoid.
- The variogram model was set to fit the three principal directions and checked against other directions.

Figure 7.6 shows an example of the variogram map and fitted variogram model of the Resources domain.

Figure 7.6: Variogram map and fitted model – Domain D12210



Source: SRK, 2025

Due to insufficient samples in 31 domains for fitting a meaningful variogram, the variogram models developed for domains D12103, D12210, D12302, D12407, D12409 and D14102 have been applied to the domains with similar features. However, for domains with a distinct dipping direction against the overall orientation, the squared inverse distance weighting (IDW) method has been used instead of ordinary kriging (OK) method. The methods applied to each domain are detailed are shown in Table 7.6.

Table 7.6: Estimation method used for Resources domains

Method	Variogram	Domains applied to
Ordinary kriging method (OK method)	D12103	D12101 D12102 D12103 D12104 D12106
	D12210	D12201 D12202 D12203 D12204 D12205 D12206 D12207 D12210 D12211
	D12302	D12301 D12302
	D12407	D12401 D12402 D12403 D12406 D12407
	D12409	D12409 D12410
	D14102	D14101 D14102 D14103 D14104 D14105 D14106
Squared inverse distance weighting method (IDW method)		D12105 D12208 D12209 D12303 D12304
		D12404 D12405 D12408

Source: SRK, 2025

The metal content of the domains was estimated using the IDW method and constitutes 2.37% of the total metal content across all domains within the Pb12 and Pb14 deposits.

The parameters of the fitted variogram models are shown in Table 7.7 (directions) and Table 7.8 (nugget, sill and ranges).

Table 7.7: Variogram models – direction

Metal	Domain	Direction		
		Dip (˚)	Dip azimuth (˚)	Pitch (˚)
Pb	D12103	20	193	32
	D12210	31	161	168
	D12302	26	189	148
	D12407	37	170	12
	D12409	28	195	124
	D14102	16	152	143
Zn	D12103	29	194	165
	D12210	34	164	31
	D12302	25	199	165
	D12407	37	169	33
	D12409	23	191	124
	D14102	15	178	78

Source: SRK, 2025

Table 7.8: Variogram models – nugget, sill and ranges

Metal	Domain	Nugget	Structure 1				Structure 2			
			Sill	Major axis range (m)	Semi-major axis range (m)	Minor axis range (m)	Sill	Major axis range (m)	Semi-major axis range (m)	Minor axis range (m)
Pb	D12103	0.02	0.35	65	55	3	0.63	167	68	55
	D12210	0.04	0.55	95	62	33	0.41	128	103	40
	D12302	0.07	0.37	89	91	7	0.56	114	109	20
	D12407	0.37	0.15	120	110	22	0.48	135	124	45
	D12409	0.18	0.23	47	101	76	0.59	112	109	89
	D14102	0.05	0.03	166	86	26	0.92	221	126	38
Zn	D12103	0.18	0.61	69	45	10	0.21	108	103	20
	D12210	0.12	0.27	79	78	10	0.61	109	82	20
	D12302	0.09	0.04	130	111	10	0.87	168	152	20
	D12407	0.22	0.14	86	74	10	0.64	140	91	20
	D12409	0.18	0.35	52	86	5	0.47	185	115	18
	D14102	0.11	0.39	20	63	3	0.5	150	123	24

Source: SRK, 2025

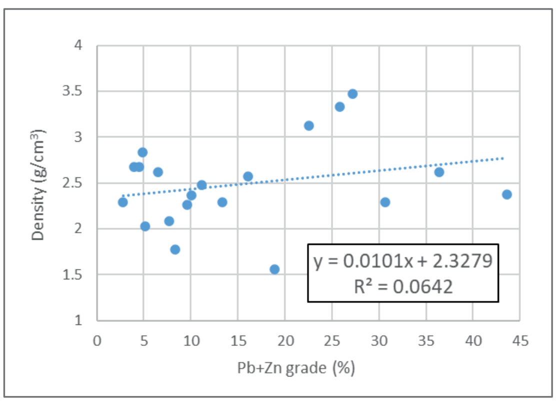
7.7 Density

Exploration at the Pb4 and Pb13 deposits remains at an early stage, and the Pb4 and Pb13 deposits were therefore not included in the Mineral Resource estimate. Consequently, bulk density analyses and modelling were only conducted for the Pb12 and Pb14 deposits.

Out of 316 available bulk density samples, all 219 samples with specific interval positions were collected from the Pb12 deposit. These samples were used to interpolate the density of the domains where they are located, using the squared inverse distance method.

For the Pb12 and Pb14 deposits, correlations between density and the combined Pb+Zn grade were analysed. This analysis was conducted separately for oxide ore and fresh ore in the Pb12 deposit (Figure 7.7 and Figure 7.8) and for fresh ore in the Pb14 deposit (Figure 7.9).

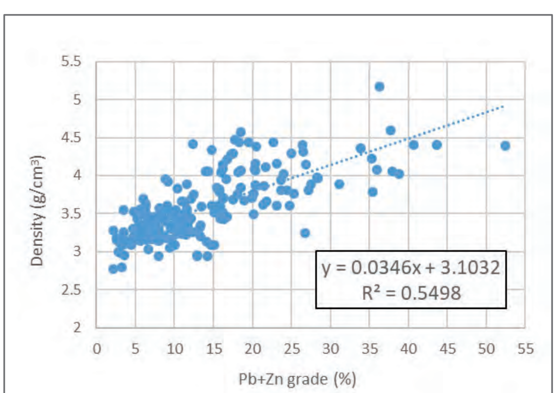
Figure 7.7: Density distribution of oxide ore samples – Pb12



Source: SRK, 2025.

The distribution shown in Figure 7.7 does not suggest a significant correlation. Therefore, an average density of 2.48 g/cm³ has been used for the estimation of oxide ore resources.

Figure 7.8: Density distribution of fresh ore samples – Pb12

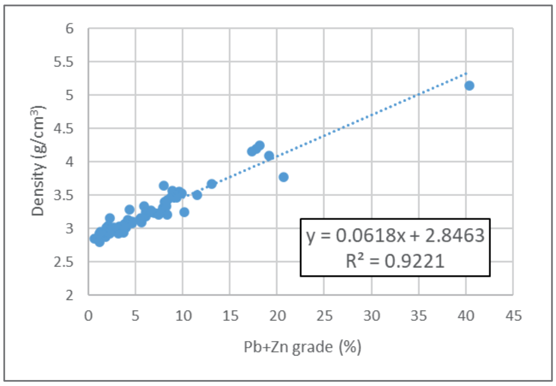


Source: SRK, 2025.

For the fresh ore in the Pb12 deposit, the following formula has been applied:

Density = 0.0346 × (Pb+Zn grade) + 3.1032.

Figure 7.9: Density distribution of fresh ore samples – Pb14



Source: SRK, 2025.

For the fresh ore in the Pb14 deposit, the following formula has been applied:

Density = 0.0618 × (Pb+Zn grade) + 2.8463.

7.8 Block model and grade estimation

7.8.1 Block model parameters

SRK produced the block models for all Resources domains with dimensions of 20 m × 20 m × 4 m (East × North × Elevation) and sub-blocking with dimensions of 2 m × 2 m × 2 m (East × North × Elevation) in Leapfrog Edge. No rotation has been applied. The block model origin and local dimensions of Pb12 and Pb14 deposits are shown in Table 7.9 and Table 7.10, respectively.

Table 7.9: Summary of block model parameters – Pb12 deposit

Dimension	Base point	Block size (m)	Sub-block size (m)	Boundary size (m)
X	31413700	20	2	1,440
Y	3345600	20	2	800
Z	5500	4	2	600

Source: SRK, 2025

Table 7.10: Summary of block model parameters – Pb14 deposit

Dimension	Base point	Block size (m)	Sub-block size (m)	Boundary size (m)
X	31411500	20	2	800
Y	3345900	20	2	600
Z	5300	4	2	400

Source: SRK, 2025

7.8.2 Grade estimation

Block grade values were interpolated using the ordinary kriging (OK) method or squared inverse distance weighted (IDW) method.

The variograms for the Pb% and Zn% were modelled and used for interpolating the grade. Due to insufficient samples in several domains for fitting a meaningful variogram, the variogram models developed for domains D12103, D12210, D12302, D12407, D12409 and D14102 have been applied to the domains with similar features. However, for those domains with a different dipping direction, the IDW method was used instead of the OK method (Table 7.6).

The Cu%, Ag (g/t) and density were interpolated using the IDW method.

The directions of search ellipsoids are locally adjusted based on the domain's occurrence, using Leapfrog's variable orientation module. The parameters used for the search ellipsoid are summarised in Table 7.11.

Table 7.11: Search ellipsoid parameters used for Mineral Resource estimation

Metal	Domain	Major axis (m)	Semi-major axis (m)	Minor axis (m)	Minimum number of composites	Maximum number of composites
Pb	D12101, D12102, D12103, D12104, D12105, D12106	200	150	50	2	40
	D12201, D12202, D12203, D12204, D12205, D12206, D12207, D12208, D12209, D12210, D12211	150	100	40	2	40
	D12301, D12302, D12303, D12304	150	120	40	2	40
	D12401, D12402, D12403, D12404, D12405, D12406, D12407, D12408,	200	150	50	2	40
	D12409, D12410	150	120	90	2	40
	D14101, D14102, D14103, D14104, D14105, D14106	250	120	40	2	40
Zn	D12101, D12102, D12103, D12104, D12105, D12106	120	120	20	2	40
	D12201, D12202, D12203, D12204, D12205, D12206, D12207, D12208, D12209, D12210, D12211	120	100	20	2	40
	D12301, D12302, D12303, D12304	180	160	20	2	40
	D12401, D12402, D12403, D12404, D12405, D12406, D12407, D12408,	160	100	20	2	40
	D12409, D12410	200	120	20	2	40
	D14101, D14102, D14103, D14104, D14105, D14106	140	120	30	2	40
Cu	All domains	160	100	40	2	40

Metal	Domain	Major axis (m)	Semi-major axis (m)	Minor axis (m)	Minimum number of composites	Maximum number of composites
Ag	All domains	160	100	40	2	40
Density	D12103 and D12104	120	120	20	2	40
	D12208	160	100	20	2	40
	D12210	120	100	20	2	40
	D12302 and D12303	180	160	20	2	40
	D12405 and D12407	160	100	20	2	40
	D12409	200	120	20	2	40

Source: SRK, 2025

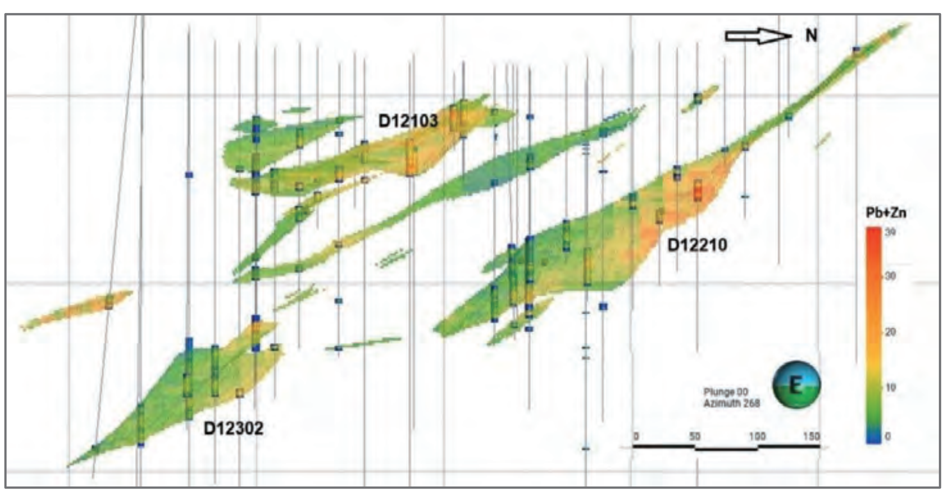
7.9 Model validation

SRK undertook block model validation to confirm the reasonableness of the estimation parameters and estimation results. SRK adopted the following methods for the validation:

- visual validation of block grades against drill hole grades
- trend analysis.

In sectional views, SRK performed visual validation of the sample grades (drill holes and channel sampling) against the intersection composites and estimated block model grades (Figure 7.10). This validation process demonstrated good correlation between local block estimations and nearby samples, without excessive smoothing in the block model.

Figure 7.10: Visual validation of selected cross section (looking west)



Source: SRK, 2025

The arithmetic mean values of the composite and the block were also compared and generally show that the deviations for major domains are within an acceptable level (Table 7.12). The largest deviations were examined further, and can be accounted for, in terms of the influence of clustering.

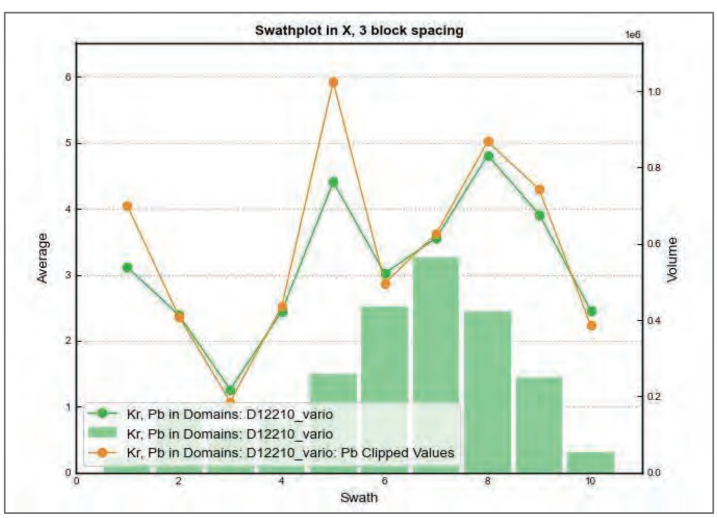
Table 7.12: Composite and block means comparison for major domains

Domain	Pb%				Zn%			
	Composite Mean	Block Mean	Absolute deviation	Relative deviation (%)	Composite Mean	Block Mean	Absolute deviation	Relative deviation (%)
D12103	7.76	6.48	-1.28	-16%	4.22	4.30	0.07	2%
D12210	3.68	3.37	-0.31	-8%	5.28	5.04	-0.24	-5%
D12302	1.86	1.77	-0.08	-5%	6.21	6.60	0.39	6%
D12407	2.97	2.95	-0.01	0%	2.45	2.55	0.10	4%
D12409	3.03	2.83	-0.20	-7%	2.82	2.86	0.04	1%
D14102	1.19	1.10	-0.08	-7%	5.52	5.36	-0.16	-3%

Source: SRK, 2025

Figure 7.11, Figure 7.12 and Figure 7.13 show swath plots of Pb% for Domain 12210 in the east−west, north–south and RL (elevation) directions. Figure 7.14, Figure 7.15 and Figure 7.16 show swath plots of Zn% for Domain 14102 in the east−west, north–south and RL directions. These swath plots show that the capped composites (in yellow) and block grades (in green) generally correlate at an acceptable level.

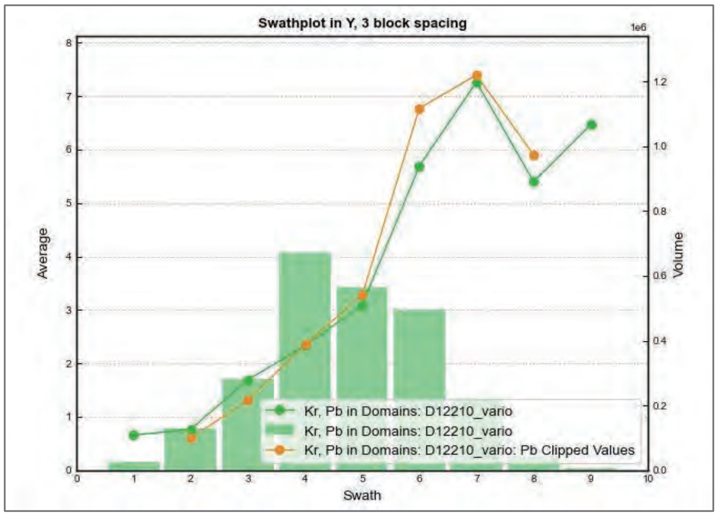
Figure 7.11: Swath plot along east–west direction – Domain 12210



Source: SRK, 2025

Notes: Capped composites are in yellow and block grades are in green.

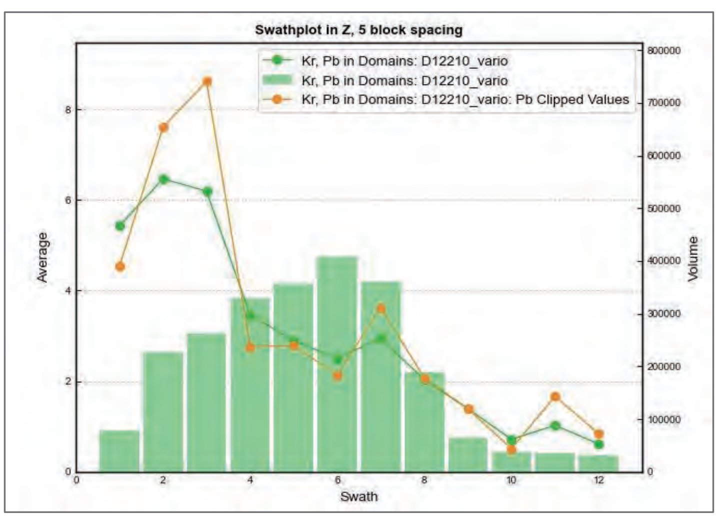
Figure 7.12: Swath plot along north–south direction – Domain 12210



Source: SRK, 2025

Notes: Capped composites are in yellow and block grades are in green.

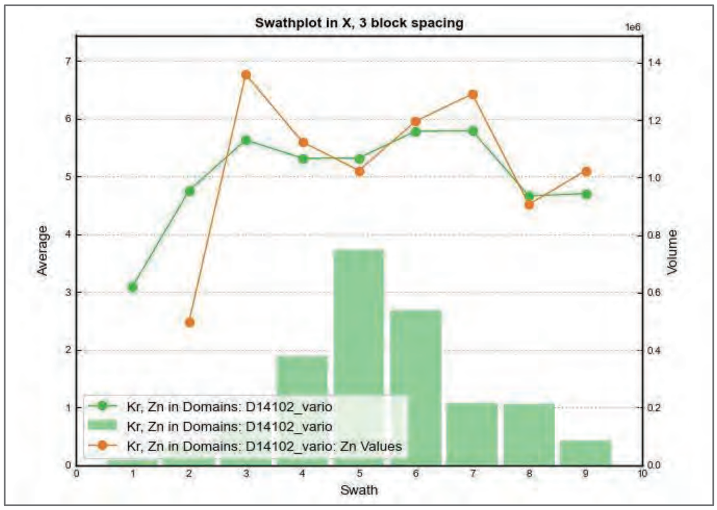
Figure 7.13: Swath plot along RL direction – Domain 12210



Source: SRK, 2025

Notes: Capped composites are in yellow and block grades are in green.

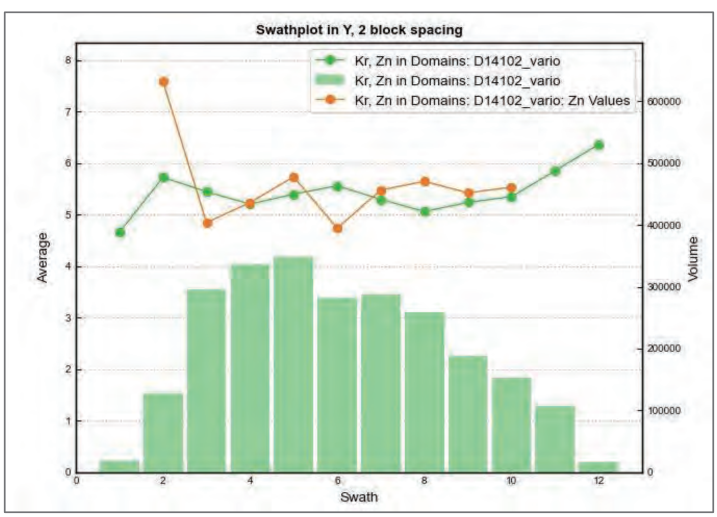
Figure 7.14: Swath plot along east–west direction – Domain D14102



Source: SRK, 2025

Notes: Capped composites are in yellow and block grades are in green.

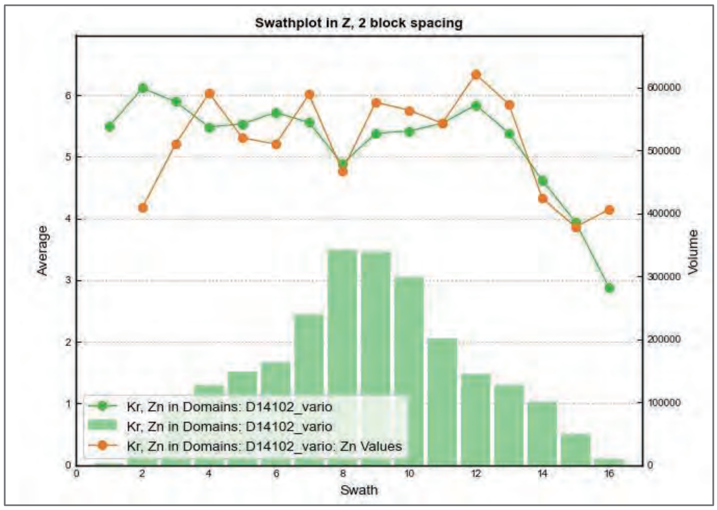
Figure 7.15: Swath plot along north–south direction – Domain D14102



Source: SRK, 2025

Notes: Capped composites are in yellow and block grades are in green.

Figure 7.16: Swath plot along RL direction – Domain D14102



Source: SRK, 2025

Notes: Capped composites are in yellow and block grades are in green.

7.10 Mined-out areas

As at 31 July 2025, the depleted resources from the existing Pb14 deposit open pit and Pb12 deposit underground development and workings have been excluded. The underground production of the Pb 12 deposit commenced on 12 June in the second quarter of 2025, following the completion of its construction.

7.11 Classification

Mineral Resource classification should consider several factors, including the confidence level in the geological continuity of the mineralised structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. The classification criteria should aim to integrate these concepts to delineate consistent areas with similar Mineral Resource classifications.

The following items have been considered during classification of the Mineral Resources:

- geological continuity and reliability of interpretation
- sample support and exploration workings density
- OK attributes (kriging variance, slope of regression, kriging efficiency).

The classification criteria are shown in Table 7.13.

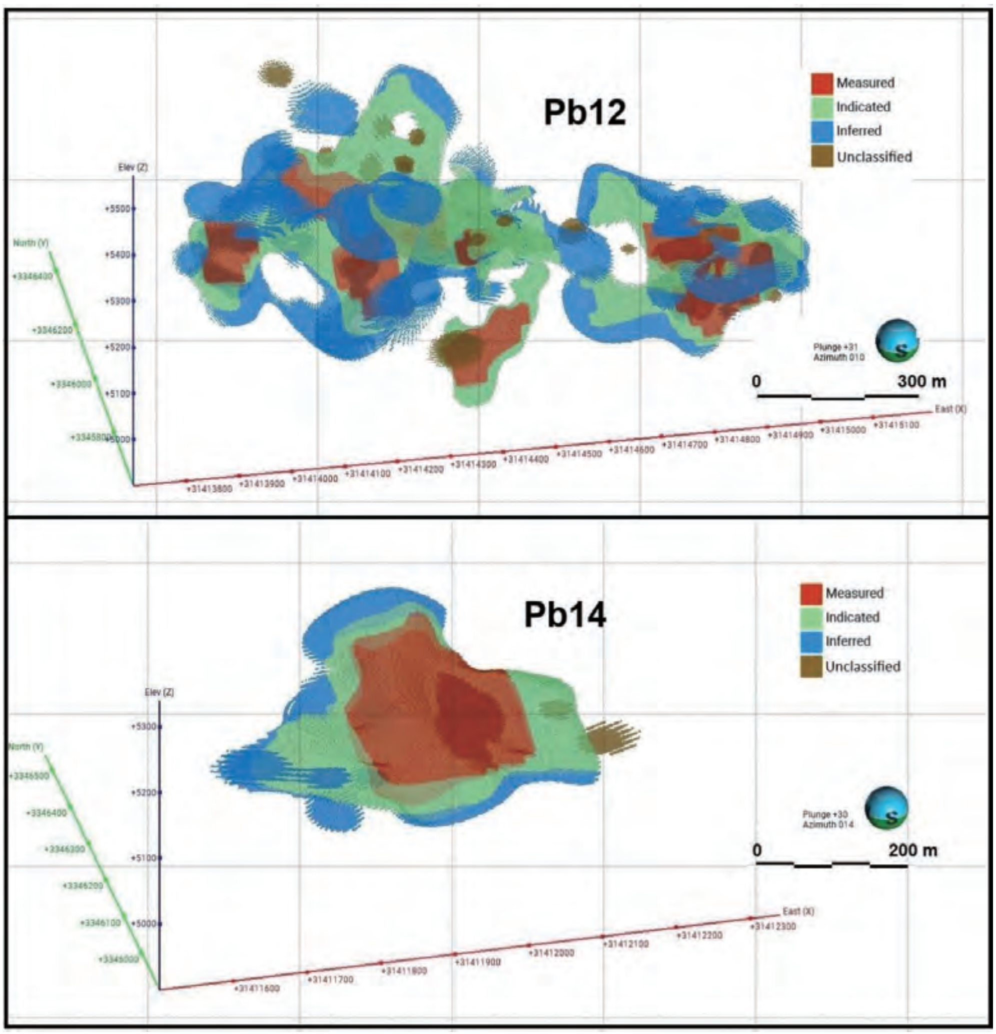
A 3D view of the Mineral Resource classification distribution is shown in Figure 7.17.

Table 7.13: Mineral Resource classification criteria used in estimation

Category	Mineral Resource classification criteria
Measured	Spacing between drill holes or channel sampling is within 50 m, or slope of regression is greater than 0.85.
Indicated	Spacing between drill holes or channel sampling is within 100 m, or slope of regression is greater than 0.6.
Inferred	Spacing between drill holes or channel sampling is more than 100 m, or the domains estimated using Inverse Distance Weighted (IDW) method.

Source: SRK, 2025

Figure 7.17: Mineral Resource classification – 3D view



Source: SRK, 2025

7.12 Mineral Resource Statement

7.12.1 Cut-off assumptions

Zhihui has mined the Mengya'a lead-zinc deposit and processed the ore since 2007. The conceptual economic cut-off grades for blocks are estimated based on the historical operations data and cut-off estimation presented in Table 7.14. In this context, the term 'cut-off' refers to the grade applied to the block model to determine the portion of the model that qualifies as Mineral Resources. The price of the lead concentrate, zinc concentrate, copper concentrate (per metal tonne) and silver metal (per gram) within these concentrates, are assumed to be Chinese Renminbi (RMB) 18,600, RMB21,100, RMB81,500 and RMB8.1, respectively. The recovery assumptions for fresh ore are as follows: zinc recovery is 91.0%, copper recovery is 52.0%, silver recovery in the copper concentrate is 9.2%, and silver recovery in the lead concentrate is 66.4%. For oxide ore, the recovery assumptions are zinc recovery at 66.1%, copper recovery at 35.1%, silver recovery in the copper concentrate at 10.4%, and silver recovery in the lead concentrate at 57.3%.

For fresh ore, an equivalent lead content has been used based on the calculation formula:

$EqPb = Pb + 1.1457 * Zn + 2.5464 * Cu + 0.0296 * Ag$.

For oxide ore, an equivalent lead content has been used based on the calculation formula:

$EqPb = Pb + 1.3315 * Zn + 2.7501 * Cu + 0.0062 * Ag$.

Table 7.14: Estimation parameters for conceptual economic cut-off grades

Item	Open-pit ore	Underground ore
Processing recovery	88%	88%
Mining cost	- Note1	229.5 RMB/t Ore
Processing cost	154.2 RMB/t Ore	154.2 RMB/t Ore
General and Administration (G&A) cost	66.8 RMB/t Ore	66.8 RMB/t Ore
Resource tax	5.9%	5.9%
Lead concentrate price	18,600 RMB/Pb t	18,600 RMB/Pb t
Cut-off grade (EqPb grade)	1.4%	2.9%

Source: SRK, 2025

Note: 1: For the purpose of the conceptual economic cut-off calculation, the mining cost for open-pit ore is considered negligible and is therefore assumed to be zero.

An EqPb cut-off grade of 1.4% is applied to fresh ore from open-pit operations, while an EqPb cut-off grade of 2.9% is used for fresh ore from underground operations. For oxide ore from underground operations, an EqPb cut-off grade of 4.7% is applied.

7.12.2 Mineral Resource Statement

Clause 20 of the JORC Code (2012) requires that all reports relating to Mineral Resource estimates must have '*reasonable prospects for eventual economic extraction*' (RPEEE).

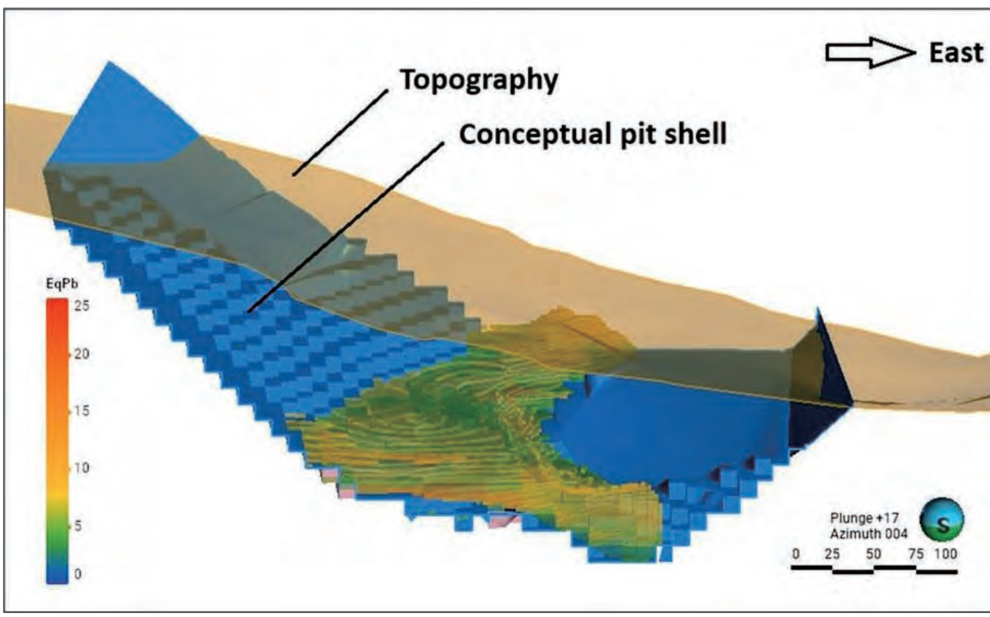
Zhihui has been operating project's Pb14 open-pit extracting Pb, Zn, Cu and Ag ores since 2007. The open-pit mining method and mineral processing flowsheet employed for this deposit has proven to be both appropriate and successful. The underground Pb12 deposit has a higher combined Pb + Zn grade and similar ore features, and a Chinese preliminary design has also been completed by Xinjiang Engineering & Research Institute of Nonferrous Metals Co., Ltd. in August 2024.

To demonstrate satisfaction of the RPEEE criterion, a pit optimisation study using the Lerchs-Grossmann algorithm was undertaken by SRK in GEOVIA Whittle software for Pb14 deposit in March 2025.

The operating parameters for the Pb 14 optimisations and cut-off grade estimates were based on the price, cost and recovery assumptions listed in Section 7.12.1, and a maximum pit slope of 42°. The Mineral Resource estimate is constrained by the pit shell corresponding to a revenue factor of 1. The pit optimisation study considered Measured, Indicated and Inferred Mineral Resources.

The Mineral Resource estimate for the Pb14 deposit constrained by conceptual pit shell as at 31 July 2025 is shown in Figure 7.18.

Figure 7.18: Mineral Resource distribution of Pb14 Deposit within conceptual pit shell



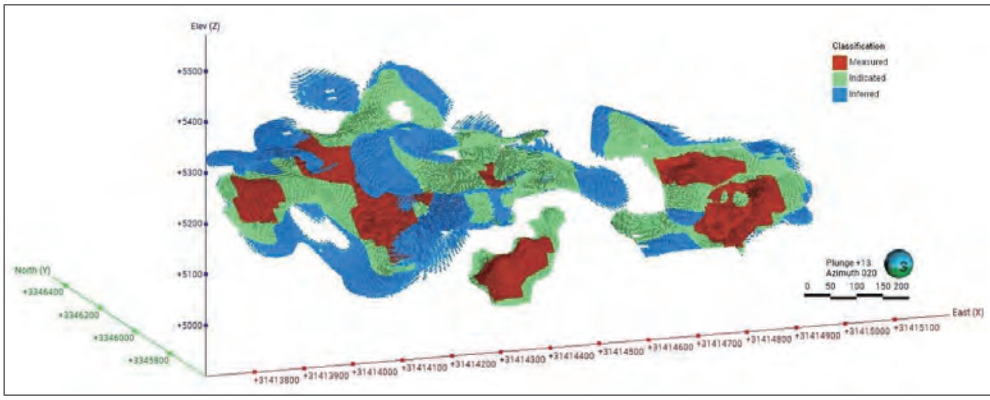
Source: SRK, 2025

A conceptual study of the mineable area was performed in Leapfrog for the Pb12 UG deposit, incorporating the commodity price, operating cost, and recovery as detailed in Section 7.12.1. Furthermore, three additional criteria were applied to assess the RPEEE for Pb12:

- A maximum distance of 50 m horizontally and 10 m vertically delineated the proximity of the domain to the nearest domain.
- A minimum individual domain size of 50,000 tonnes was mandated.
- Continuity of the resource model blocks was validated.

The Mineral Resource estimate for the Pb12 deposit constrained by criteria presented above as at 31 July 2025 is shown in Figure 7.19.

Figure 7.19 Mineral Resource distribution of Pb12 Deposit



Source: SRK, 2025

Mineral Resource Statement for the Mengya'a Project within the mining licence area as at 31 July 2025 is shown in Table 7.15.

Table 7.15: Mineral Resource Statement for the Mengya'a Project as at 31 July 2025

Ore type	Domain	Cut-off	Category	Tonnage (kt)	Pb grade (%)	Zn grade (%)	Cu grade (%)	Ag grade (g/t)	Pb metal (kt)	Zn metal (kt)	Cu metal (kt)	Ag metal (t)
Oxide	Pb12: D12101–D12106	EqPb ≥4.7%	Measured	92	4.83	3.07	0.28	30	4.45	2.83	0.26	3
			Indicated	699	6.75	3.73	0.24	39	47.17	26.09	1.68	27
			Inferred	645	5.26	2.97	0.30	38	33.90	19.13	1.92	25
			Total	1,435	5.96	3.35	0.27	38	85.51	48.04	3.86	55
	Pb12: D12201–D12211		Indicated	141	6.02	5.03	0.23	25	8.51	7.12	0.33	4
			Inferred	43	5.17	2.65	0.31	23	2.20	1.13	0.13	1
			Total	184	5.82	4.48	0.25	25	10.71	8.25	0.46	5
	Subtotal		Measured	92	4.83	3.07	0.28	30	4.45	2.83	0.26	3
			Indicated	840	6.63	3.95	0.24	37	55.68	33.21	2.01	31
			Inferred	687	5.25	2.95	0.30	37	36.10	20.25	2.05	26
			Total	1,619	5.94	3.48	0.27	37	96.23	56.29	4.32	59
Fresh	Pb12: D12101–D12106	EqPb ≥2.9%	Measured	88	5.93	4.96	0.40	36	5.20	4.35	0.35	3
			Indicated	733	4.09	4.36	0.21	29	29.98	31.93	1.55	21
			Inferred	163	3.65	3.21	0.21	41	5.96	5.24	0.35	7
			Total	984	4.18	4.22	0.23	32	41.14	41.52	2.24	31
	Pb12: D12201–D12211		Measured	4,050	2.94	5.09	0.25	42	119.03	206.07	10.06	172
			Indicated	3,482	4.79	5.59	0.29	46	166.91	194.72	9.98	160
			Inferred	1,195	2.02	4.21	0.25	40	24.11	50.33	2.97	48
			Total	8,728	3.55	5.17	0.26	44	310.05	451.12	23.00	380
	Pb12: D12301–D12304		Measured	899	1.45	6.06	0.35	60	13.00	54.47	3.16	54
			Indicated	190	3.10	8.90	0.37	59	5.90	16.94	0.70	11
			Inferred	-	-	-	-	-	-	-	-	-
			Total	1,089	1.73	6.56	0.35	60	18.89	71.41	3.87	65
	Pb12: D12401–D12410		Measured	1,170	3.00	2.59	0.17	27	35.06	30.30	1.99	32
			Indicated	782	2.79	2.70	0.18	33	21.83	21.08	1.42	26
			Inferred	537	3.21	3.65	0.21	44	17.25	19.61	1.12	23
			Total	2,489	2.98	2.85	0.18	33	74.14	70.99	4.53	81
	Subtotal		Measured	6,207	2.78	4.76	0.25	42	172.29	295.20	15.56	261

Ore type	Domain	Cut-off	Category	Tonnage (kt)	Pb grade (%)	Zn grade (%)	Cu grade (%)	Ag grade (g/t)	Pb metal (kt)	Zn metal (kt)	Cu metal (kt)	Ag metal (t)
			Indicated	5,187	4.33	5.10	0.26	42	224.62	264.67	13.64	218
			Inferred	1,896	2.50	3.97	0.23	41	47.32	75.18	4.44	79
			Total	13,289	3.34	4.78	0.25	42	444.23	635.04	33.64	557
	Pb14: D14101–D14106	EqPb ≥1.4%	Measured	427	1.13	4.64	0.11	15	4.84	19.80	0.45	7
			Indicated	1,172	0.52	5.29	0.11	9	6.10	62.00	1.28	10
			Inferred	305	0.77	5.32	0.17	17	2.35	16.24	0.51	5
			Total	1,904	0.70	5.15	0.12	12	13.28	98.04	2.24	22
Total	Pb12 subtotal	Oxide: EqPb ≥4.7% Fresh: EqPb ≥2.9%	Measured	6,299	2.81	4.73	0.25	42	176.73	298.02	15.82	264
			Indicated	6,027	4.65	4.94	0.26	41	280.30	297.88	15.66	249
			Inferred	2,583	3.23	3.70	0.25	40	83.42	95.43	6.49	104
			Total	14,909	3.63	4.64	0.25	41	540.45	691.33	37.97	617
	Pb14 subtotal	EqPb ≥1.4%	Measured	427	1.13	4.64	0.11	15	4.84	19.80	0.45	7
			Indicated	1,172	0.52	5.29	0.11	9	6.10	62.00	1.28	10
			Inferred	305	0.77	5.32	0.17	17	2.35	16.24	0.51	5
			Total	1,904	0.70	5.15	0.12	12	13.28	98.04	2.24	22
	Total		Measured	6,726	2.70	4.73	0.24	40	181.57	317.82	16.27	270
			Indicated	7,198	3.98	5.00	0.24	36	286.40	359.88	16.94	259
			Inferred	2,888	2.97	3.87	0.24	38	85.77	111.67	7.00	110
			Total	16,813	3.29	4.70	0.24	38	553.73	789.37	40.21	639

Notes:

1 Any differences between totals and sum of components are due to rounding.

2 EqPb 4.7% and EqPb 2.9% cut-off grades were applied to the resource block models of Pb12 deposit oxide ore and fresh ore, respectively. An EqPb 1.4% cut-off grade was applied to the resource block models of Pb14 deposit.

3 Mineral Resources that are not Ore Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

4 Mineral Resources are reported inclusive of Ore Reserves.

5 The Mineral Resources are effective as at 31 July 2025.

6 Lead equivalent (EqPb) formulas were applied: fresh ore: EqPb = Pb + 1.1457 × Zn + 2.5464 ×Cu + 0.0296 × Ag and oxide ore: EqPb = Pb + 1.3315 × Zn + 2.7501 × Cu + 0.0062 × Ag.. Metal price assumptions included 18,600 RMB/Pb t for lead concentrate, 21,100 RMB/Zn t for zinc concentrate, 81,500 RMB/Cu t for copper concentrate, and 8.10 RMB/g for silver. The recovery assumptions for fresh ore are as follows: zinc recovery is 91.0%, copper recovery is 52.0%, silver recovery in the copper concentrate is 9.2%, and silver recovery in the lead concentrate is 66.4%. For oxide ore, the recovery assumptions are zinc recovery at 66.1%, copper recovery at 35.1%, silver recovery in the copper concentrate at 10.4%, and silver recovery in the lead concentrate at 57.3%.

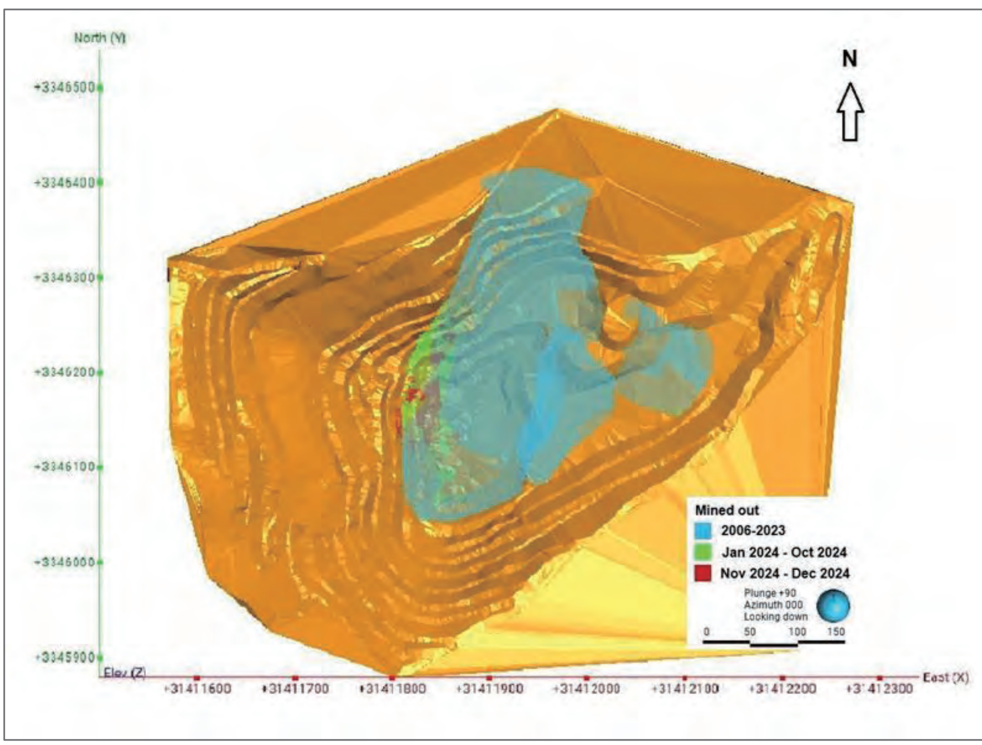
Competent Person's Statement

The information in this Report that relates to Mineral Resources is based on information compiled by Dr (Tony) Shuangli Tang, who is a Member of the AIG and Member of the AusIMM. Dr Tang is a full-time employee of SRK Consulting (Hong Kong) Limited and has sufficient experience that is relevant to the style of mineralisation, type of deposit under consideration and to the activity that he undertakes to qualify as a Competent Person as defined in the 2012 edition of the *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* (the JORC Code).

7.12.3 Reconciliation

SRK reviewed the 2024 and 2025 production records as provided by Zhihui and conducted a reconciliation analysis. This analysis compared the actual produced resources to the model depletion for two periods: January 2024 – December 2024 and January 2025 – July 2025 (Figure 7.20).

Figure 7.20: Model depletion



Source: SRK, 2025

Table 7.16: 2024-2025 reconciliation statistics

Period	Model depletion			Actual production			Difference	
	Tonnes (kt)	Pb (%)	Zn (%)	Tonnes (kt)	Pb (%)	Zn (%)	Tonnes (kt)	(%)
From Jan 2024 to Dec 2024	166	0.43	5.94	197	0.63	5.88	-30	-15%
From Jan 2025 to Jul 2025	190	1.07	4.94	192	1.44	4.96	-2	-1%

Source: SRK, 2025

8 Mining

8.1 Introduction

The Mengya'a Mine comprises two deposits:

- Pb14: operated as an open-pit mine
- Pb12: planned for underground mining.

The Pb14 deposit commenced open-pit (OP) operations in 2007, whereas the Pb12 deposit has existing underground (UG) infrastructure that previously served exploration purposes and can be further utilised for production.

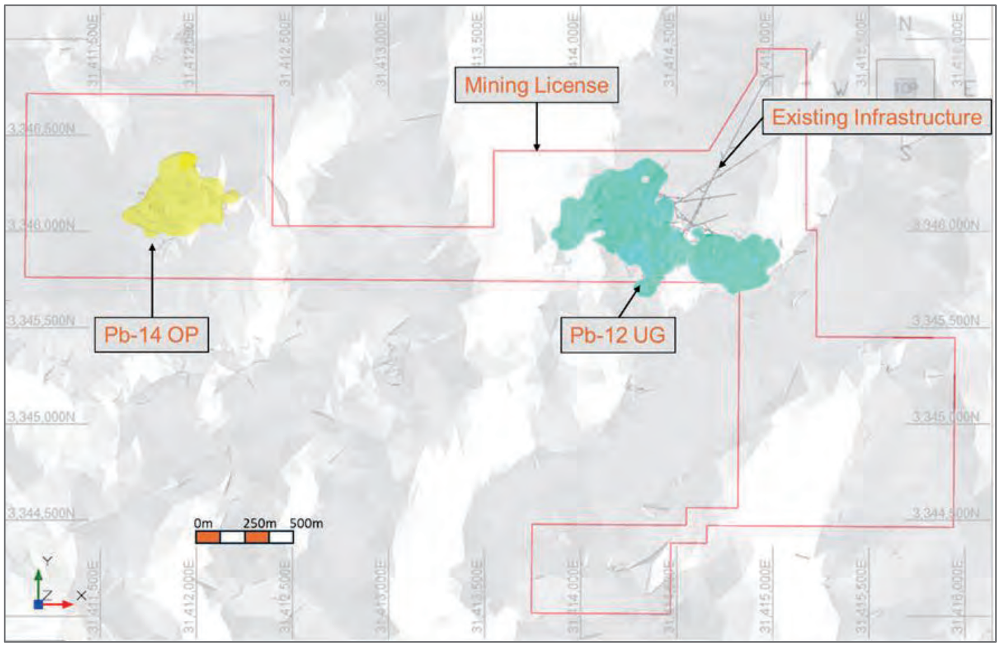
This chapter provides a comprehensive review of the mining methods at the Mengya’a mine, encompassing mining conditions, mining methodology, development systems, and mine service systems. Additionally, this chapter evaluates the overall applicability of the mining plan, its associated equipment, and the Life of Mine (LOM) plan. The objective of this assessment is to provide the basis for the declaration of Ore Reserve estimates in accordance with the JORC Code (2012).

As shown in Figure 8.1, the mining licence currently covers an area of approximately 4.4544 km^2, with a collective permitted mining capacity for both open-pit and underground operations of 400 kt/a. The permitted mining elevation ranges from 4,964 m to 5,400 m above sea level (asl).

Based on references from the technical studies and discussions with Huaxia Mining, the strategic plan for the Mengya’a mine is as follows:

- The Pb14 open-pit (OP) will maintain operations using the open-pit methodology and the current contractor mining equipment fleet.
- The designed production rate for the Pb14 OP is planned at 200 kt/a of run-of-mine (ROM) ore.
- The Pb12 underground (UG) has a designed production capacity of 200 kt/a in alignment with the Pb14 OP operations. Upon the depletion of resources at Pb14 OP, the Pb12 UG will ramp up to 400 kt/a to ensure a consistent feed to the processing plant.
- The mining sequence for Pb12 UG, from top to bottom, will consist of two phases: Stage I involves mining above the 5,100 m RL, with 5,180 m RL as the pilot mining level, and Stage II involves mining below the 5,100 m RL.

Figure 8.1: General location of Mengya'a mining operations



Source: SRK

8.2 Technical studies

SRK's mining review is primarily based on the following information:

- *Feasibility Study of the Mengya'a Lead-Zinc Mine in Jiali County, Nagqu, Xizang*, May 2013, Guangdong Metallurgical & Architectural Design Institute (GDMADI), hereafter referred to as **13 GDMADI FS**.
- Preliminary Design for Underground Mining of the Pb12 Deposit at the Mengya'a Lead-Zinc Mine, Jiali County, Nagqu, Xizang., April 2022, CINF Engineering Corporation Limited (CINF), hereafter referred to as **22 CINF Mining Design**.
- *Preliminary Design for Underground Mining of the Pb12 Deposit at the Mengya'a Lead-Zinc Mine, Jiali County, Nagqu, Xizang*, August 2024, Xinjiang Engineering & Research Institute of Nonferrous Metals Co., Ltd. (XERI), hereafter referred to as **24 XERI Mining Design**.

According to the technical studies as outlined above, and geological definitions, the Pb14 deposit is designed to be excavated through open-pit mining, while the Pb12 deposit will employ underground mining methods.

The Pb14 deposit has been in operation since 2007, while the Pb12 deposit has existing development tunnels that were developed for exploration purposes and can be used for production purposes upon mine commissioning.

As per the 13 GDMADI FS, the design capacity for the Pb14 open-pit mine is 400 kt/a, and for the Pb12 underground mine, it is 200 kt/a (from the beginning). According to the 22 CINF Mining Design and the 24 XERI Mining Design, the technical study is focused on the Pb12 underground project, with a nominated capacity also set at 200 kt/a. The 24 XERI Mining Design is the update from 22 CINF Mining Desing, which includes the following differences:

1. The design scope is different: 24 XERI Mining Design considered the MRE within the previous mining licence, whereas the 22 CNFI Mining Design considered the expanded mining licence limit, as the current one.

2. Mining method: Without altering the sub-level open stoping method (longitudinal or transverse), the 24 XERI Mining Design proposed replacing the room-and-pillar method outlined in the 22 CNFI Mining Design with shrinkage stoping. However, after reviewing the orebody geometry, SRK considers the room-and-pillar method proposed in the 22 CNFI Mining Design to be more appropriate due to the shallow dip angle. The mine states that they will consider the mining method that proposed by 22 CNFI Mining Design for the operation, as well the mine plan scope studied in that.

   For the Pb12 UG, the level of accuracy of the Modifying Factors described in the 22 CINF Mining Design, the 24 XERI Mining Design and the existing underground infrastructure, and for the Pb14 OP, the level of accuracy of the Modifying Factors described in the 13 GDMADI FS and ongoing production, are considered by SRK to be similar to a prefeasibility level study (PFS), prepared in accordance with the JORC Code (2012).

8.3 Pb14 open-pit

8.3.1 Geotechnical and hydrological conditions

Geotechnical conditions

The Pb14 deposit area predominantly consists of clastic and carbonate rocks and contact metasomatic rocks. The rock mass is characterised by a layered structure with anisotropic properties and varying strength. The overall stability of the rock is significantly influenced by the presence of soft interlayers, weak strata, structural fractures, and the degree of weathering. The complexity of the geotechnical conditions is assessed as moderate based on the 13 GDMADI FS.

The geotechnical evaluation conducted in the 13 GDMADI FS relied exclusively on geological borehole data, as in situ stress testing has not yet been performed. SRK has determined the designed overall slope angle to be 42°, adopting a conventional approach that does not have a significant impact on mine stability or operational safety.

Hydrological conditions

The Pb14 deposit area is bordered by valleys oriented in the northwest-southeast and southwest-northeast directions on its southern and northern sides, respectively. The surface elevation in the Pb14 deposit area varies from 5,100 m to 5,220 m. The terrain slopes from southwest to northeast, with a gradient ranging from 5° to 12°, which facilitates the drainage of groundwater.

These geomorphological conditions are not conducive to groundwater retention, but are advantageous for water recharge and drainage. However, the upstream sections of two tributary valleys are situated in a region that is perpetually snow-covered and features gentle slopes. The substantial glacial deposits within these glacial valleys enhance the recharge of groundwater by surface water. Based on 11 LERI Mining Design, the overall hydrogeological conditions of the mining area are classified as moderate.

8.3.2 Pit optimisation

Mining Block Model

The mine design and resource estimation within the pit design were based on the Mineral Resource model (MRM) developed by SRK (Chapter 7), with an effective date of 31 December 2024. The model was provided in Datamine (.dm) format.

Key parameters of the block model are outlined in Table 8.1.

Table 8.1: Mineral Resource block model parameters

Range	Minimum	Maximum
Easting	31,411,400	31,412,300
Northing	3,345,800	3,346,700
Elevation	4,900	5,500
X size	2	2
Y size	2	2
Z size	2	2
Rotation	-	-

Source: SRK

SRK converted the MRM to a Mining Block Model (MBM) in Block Geomodel (.gmdlb) format for mine planning purposes using the Deswik software suite.

Pit optimisation parameters

The first step in transforming a Mineral Resource into a mineable open-pit Ore Reserve involves the process of open-pit optimisation. At this stage, the physical, technical, and economic parameters are applied to the mineralisation area to determine the optimal geometry for the open-pit excavation. If the economic assessment of this optimal pit shell is positive, the resulting pit shell can be used as a reference for the subsequent pit design process.

SRK used the Whittle™ software for open-pit optimisation with the Pseudoflow algorithm. In general terms, Whittle™ adjusts the base input price by a range of revenue factors (RFs) above and below a base value of 1. For each RF, Whittle™ generates a three-dimensional shape pit shell that maximizes the value based on all input parameters and the adjusted price. Lower RFs produce smaller shells, while higher RFs result in larger shells. This approach creates a series of 'nested' shells, with each shell contained within the next larger one.

A summary of the pit optimisation parameters is presented in Table 8.2.

Table 8.2: Pit optimisation parameters

Optimisation parameter	Unit	Inputs	Notes
Revenue and selling costs			
Lead price	RMB/t	14,300	50% Pb concentrate, SMM price excluding VAT
Treatment charge	RMB/t	-250	Grade premium applied due to higher Pb concentrate grade (≥ 60% Pb)
Royalties	%	5.9	22 CINF Mining Design
Mining parameters and costs			
Mining recovery	%	5	13 GDMADI FS, Huaxia Mining
Dilution	%	5	13 GDMADI FS, Huaxia Mining
Mining and stripping cost	RMB/t TMM	21	24 XERI Mining Design
Overall slope angle	°	42	13 GDMADI FS, Huaxia Mining
Processing parameters and costs			
Design throughput capacity	kt/a	200	Huaxia Mining
Process plant recovery	%	88.0	Huaxia Mining
Processing cost	RMB/t ROM	154	Huaxia Mining
G&A cost	RMB/t ROM	66.8	24 XERI Mining Design

Sources: 13 GDMADI FS, 22 CINF Mining Design, 24 XERI Mining Design, Huaxia Mining, and SRK

Pit optimisation results

A series of nested pit shells were generated based on a range of RFs reflecting lead concentrate prices. Preliminary cash flows were estimated using a 10% discount rate and a nominal lead price of RMB14,300/t. The results of the pit optimisation are detailed in Table 8.3.

Three optimisation scenarios were automatically generated by Whittle™ software:

- **Best Case**: Each of the pit shells is mined sequentially, one after the other.
- **Worst Case**: The final pit is mined bench by bench.
- **Specified Case**: This scenario is based on predefined pushback geometries.

In the Specified Case, the highest discounted cash flow was derived using pit shell 33, which was subsequently selected as the basis for pit design as it is closest to the economic maximum. This shell has a total tonnage of 9,641 kt, including 1,650 kt of ROM ore.

A pit-by-pit graph of the Measured and Indicated Mineral Resource is shown in Figure 8.2.

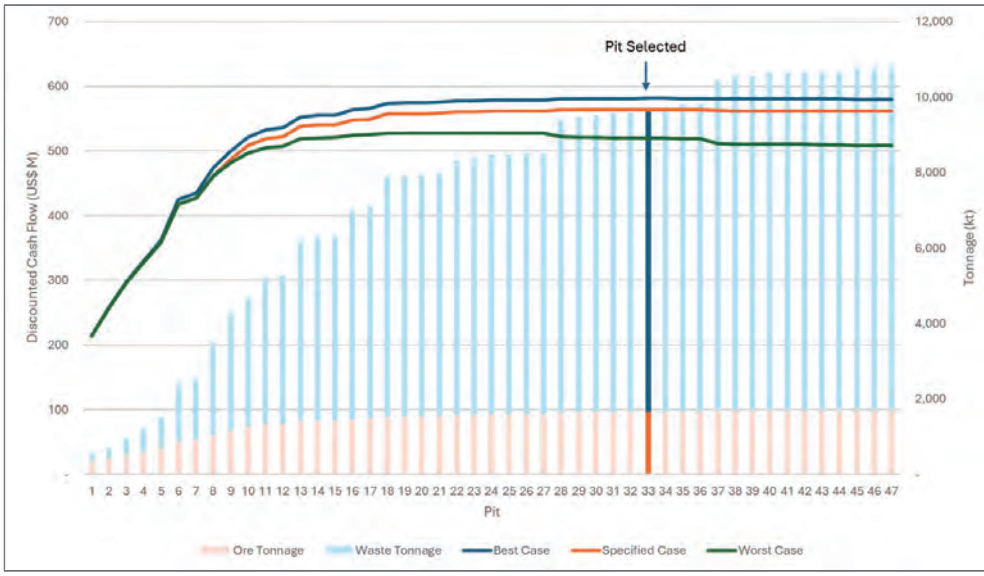
Table 8.3: Whittle™ pit optimisation results

Pit Shell	Revenue Factor	Best Case (RMB million)	Specified Case (RMB million)	Worst Case (RMB million)	Total (kt)	Ore (kt)	Waste (kt)	Stripping Ratio (t: t)
1	0.30	214.27	214.27	214.27	512	329	182	0.55
2	0.32	257.43	257.06	257.06	690	419	271	0.65
3	0.34	298.74	297.62	297.62	947	517	430	0.83
4	0.36	329.90	327.88	327.88	1,206	596	610	1.02
5	0.38	362.70	359.05	359.05	1,516	692	824	1.19
6	0.40	424.78	417.88	417.88	2,385	876	1,509	1.72
7	0.42	434.97	426.79	426.79	2,517	917	1,600	1.75
8	0.44	474.12	461.66	461.66	3,493	1,041	2,452	2.36
9	0.46	499.61	487.07	481.58	4,273	1,138	3,135	2.76
10	0.48	521.04	507.98	496.42	4,685	1,262	3,423	2.71
11	0.50	533.06	519.53	505.47	5,192	1,312	3,880	2.96
12	0.52	535.66	522.08	507.26	5,267	1,328	3,939	2.97
13	0.54	552.60	538.00	519.45	6,171	1,401	4,770	3.41
14	0.56	555.38	540.40	520.61	6,296	1,420	4,875	3.43
15	0.58	555.78	540.74	520.78	6,307	1,424	4,882	3.43
16	0.60	564.20	547.73	524.68	6,990	1,473	5,517	3.74
17	0.62	565.60	549.02	525.06	7,093	1,485	5,608	3.78
18	0.64	573.64	557.00	527.68	7,859	1,537	6,321	4.11
19	0.66	574.37	557.72	527.56	7,903	1,546	6,357	4.11
20	0.68	574.60	557.94	527.46	7,924	1,549	6,375	4.12
21	0.70	575.24	558.57	527.63	8,000	1,555	6,445	4.14
22	0.72	577.48	560.67	527.83	8,318	1,578	6,740	4.27
23	0.74	577.94	561.11	528.04	8,381	1,586	6,796	4.29
24	0.76	578.59	561.70	527.29	8,483	1,597	6,886	4.31
25	0.78	578.62	561.73	527.28	8,489	1,597	6,891	4.31
26	0.80	578.67	561.78	527.23	8,500	1,598	6,902	4.32
27	0.82	578.69	561.80	527.21	8,507	1,599	6,909	4.32

Pit Shell	Revenue Factor	Best Case (RMB million)	Specified Case (RMB million)	Worst Case (RMB million)	Total (kt)	Ore (kt)	Waste (kt)	Stripping Ratio (t: t)
28	0.84	581.12	563.63	521.88	9,409	1,635	7,774	4.75
29	0.88	581.26	563.72	521.40	9,464	1,640	7,824	4.77
30	0.90	581.35	563.75	520.86	9,519	1,644	7,875	4.79
31	0.92	581.40	563.78	520.51	9,567	1,646	7,922	4.81
32	0.94	581.41	563.78	520.46	9,575	1,646	7,928	4.82
33	0.96	581.43	563.78	520.02	9,641	1,650	7,991	4.84
34	0.98	581.43	563.78	520.01	9,644	1,650	7,994	4.84
35	1.00	581.39	563.66	519.14	9,798	1,656	8,142	4.92
36	1.02	581.37	563.62	518.99	9,821	1,658	8,163	4.92
37	1.04	580.84	562.38	511.95	10,438	1,685	8,753	5.20
38	1.06	580.71	562.12	510.69	10,548	1,690	8,858	5.24
39	1.08	580.71	562.12	510.69	10,549	1,690	8,859	5.24
40	1.10	580.62	561.98	510.14	10,620	1,691	8,929	5.28
41	1.12	580.60	561.96	510.09	10,627	1,692	8,935	5.28
42	1.14	580.59	561.94	510.06	10,631	1,692	8,939	5.28
43	1.16	580.52	561.84	509.74	10,666	1,693	8,972	5.30
44	1.22	580.50	561.82	509.66	10,671	1,694	8,977	5.30
45	1.26	580.15	561.32	507.97	10,782	1,699	9,083	5.35
46	1.28	580.11	561.28	507.85	10,793	1,699	9,094	5.35
47	1.30	580.10	561.26	507.83	10,796	1,700	9,096	5.35

Source: SRK

Figure 8.2: Pit-by-pit graph with preliminary cash flow



Source: SRK

8.3.3 Pit design

The optimal pit shell detailed above was used as a guide for final pit designs.

The design ramp parameters adopted by SRK were as follows:

- Ramp grade: 9%
- Ramp width (single lane): 9 m
- Ramp width (double lane): 13 m.

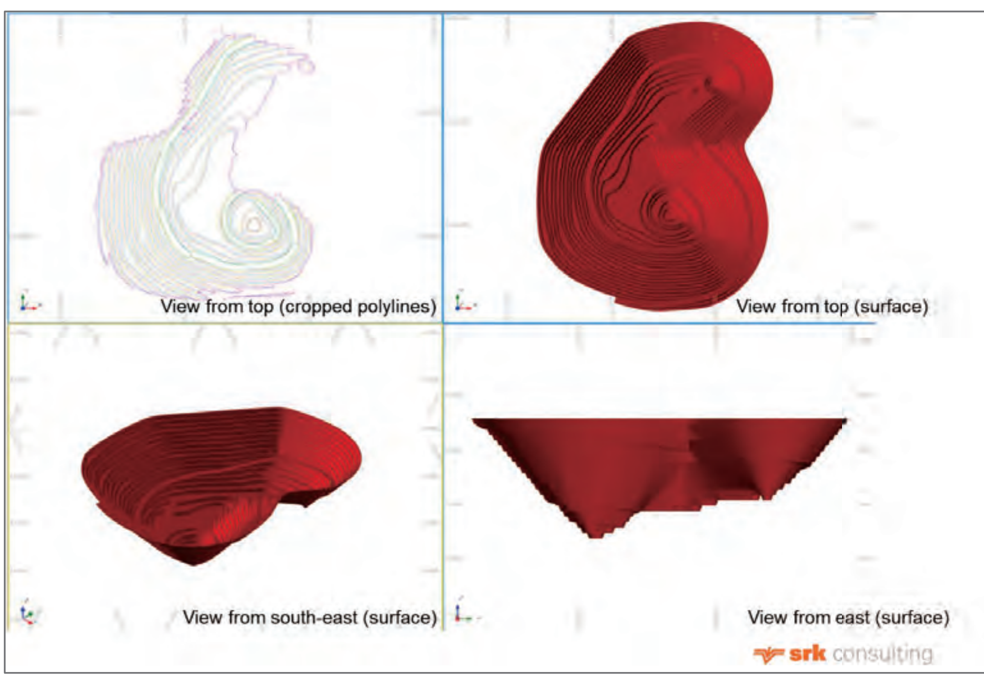
Single-lane ramps are implemented at the pit bottom when the benches begin to narrow, and mining rates are significantly reduced. Table 8.4 provides a summary of pit design parameters. Figure 8.3 shows the final pit design.

Table 8.4: Summary of pit design parameters

Item	Unit	Value
Terrain	m RL	5,260
Bottom of the pit	m RL	5,040
Open-pit length	m	530
Open-pit width	m	430
Bottom length	m	25
Bottom width	m	25
Bench height	m	10
Bench face angle (BFA)	°	68
Berm width	m	5
Overall slope angle	°	42
Closed contour line	RL	5080

Source: SRK

Figure 8.3: Final pit design



Source: SRK

Whittle™ pit shell versus final pit design

The design process transforms the Whittle™ pit shell into a practical pit design by incorporating benches, berms and ramps. This process has resulted in 7.5% less waste material and 6.1% less ROM compared to the Whittle™ pit shell. The ongoing production activities and the terrain of the hillside part of the pit have contributed to the reduction in both waste and ROM in the design.

A comparison of the tonnage and grades between the optimal pit shell and the designed pit is provided in Table 8.5. SRK considers the differences in tonnage and grades to be acceptable.

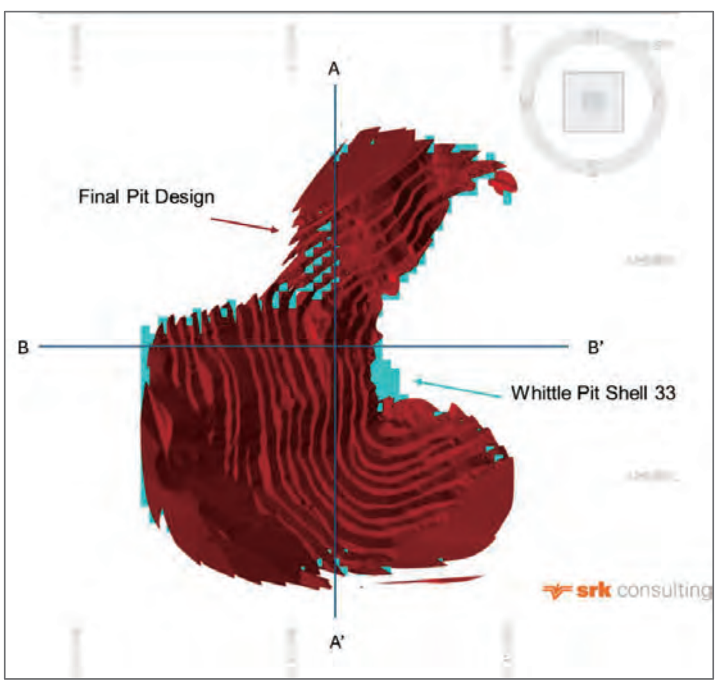
A comparison between the pit shell 33 and the final pit design is shown in Figure 8.4. Cross sections are shown in Figure 8.5 and Figure 8.6. It should be noted that the comparison between the open pit design and the Whittle-generated pit shell is based on the topography and pit shell optimisation results as at 31 December 2024. This ensures that the comparison between the Whittle shell and the pit design is made on a consistent basis, using the same topographic surface and pit shell optimization inputs.

Table 8.5: Comparison between pit shell 33 and final pit design

Pit	ROM tonnes (kt)	Diluted EqPb grade (%)	Waste tonnes (kt)	Stripping ratio (t: t)	Contained metal (kt EqPb)
Pit shell 33	1,648	6.87	9,641	5.85	113.21
Final pit design	1,548	6.81	8,917	5.76	105.40
Difference (%)	-6.1%	-0.9%	-7.5%	-1.5%	-6.9%

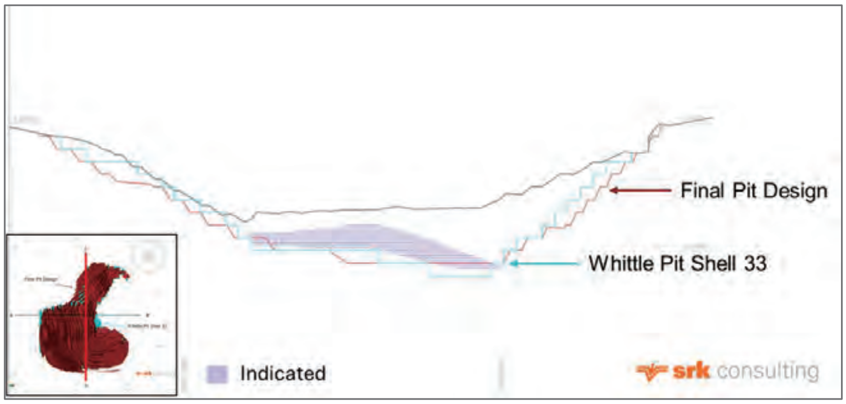
Source: SRK, 2025

Figure 8.4: Comparison between pit shell 33 and final pit design



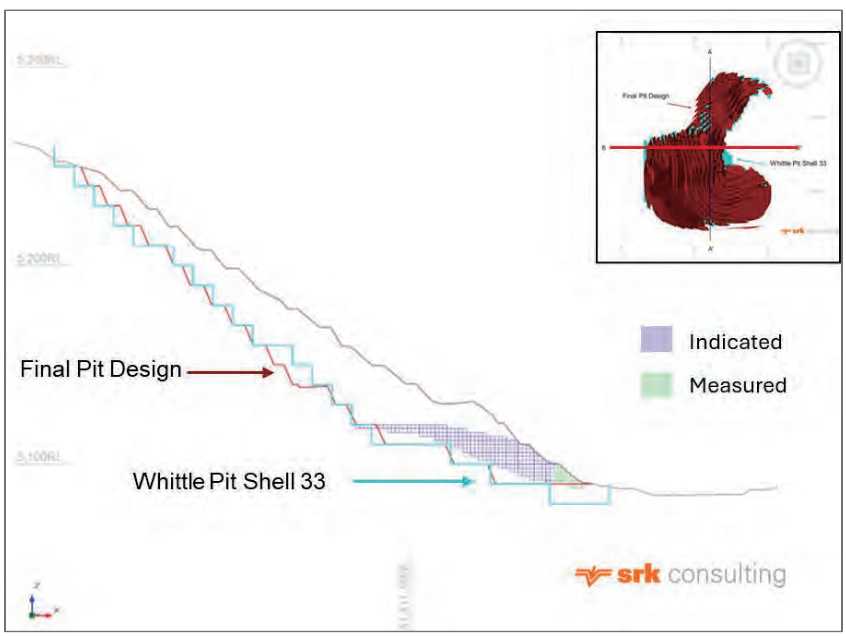
Source: SRK, 2025

Figure 8.5: Final pit design versus pit shell 33 (Section AA')



Source: SRK,2025

Figure 8.6: Final pit design versus pit shell 33 (Section BB')



Source: SRK, 2025

8.3.4 Mining inventory

The material mined in the final pit design (bench-by-bench) is detailed in Table 8.6 and shown in Figure 8.7.

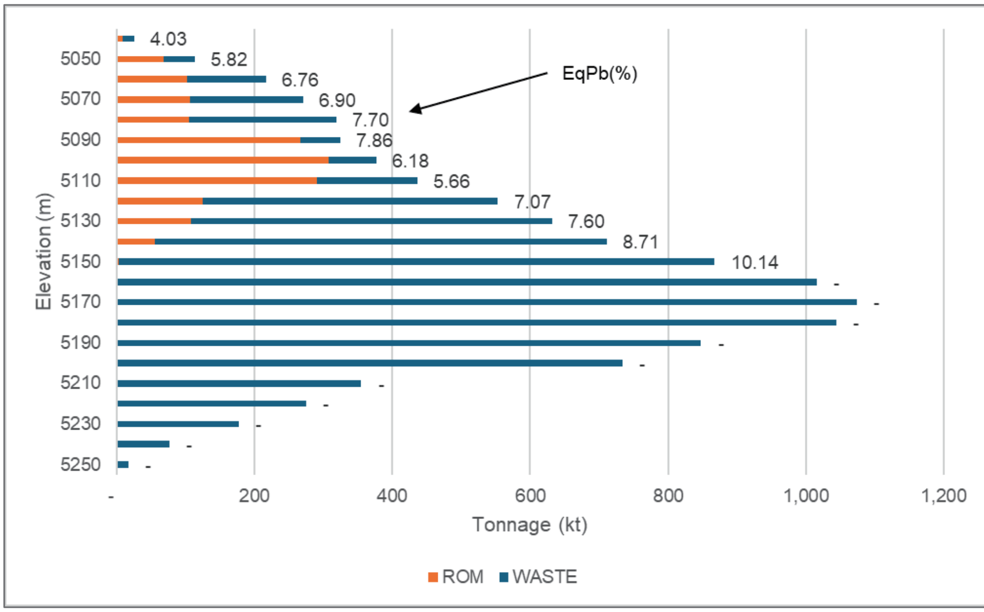
Table 8.6: Summary of bench-by-bench materials – final pit design

Bench #	Toe elevation (m asl)	ROM tonnes (kt)	Waste tonnes (kt)	Pb (%)	Zn (%)	Cu (%)	Ag (g/t)	Pb+Zn (%)	EqPb (%)	Stripping ratio (t: t)
22	5250	-	14	-	-	-	-	-	-	-
21	5240	-	56	-	-	-	-	-	-	-
20	5230	-	183	-	-	-	-	-	-	-
19	5220	-	262	-	-	-	-	-	-	-
18	5210	-	311	-	-	-	-	-	-	-
17	5200	-	318	-	-	-	-	-	-	-
16	5190	-	369	-	-	-	-	-	-	-
15	5180	-	417	-	-	-	-	-	-	-
14	5170	-	516	-	-	-	-	-	-	-
13	5160	-	587	-	-	-	-	-	-	-
12	5150	2	683	0.77	6.84	0.49	22.37	7.61	10.51	452
11	5140	50	807	1.00	5.33	0.25	23.99	6.33	8.47	16
10	5130	108	895	1.11	4.87	0.14	18.87	5.98	7.60	8.32
9	5120	124	894	0.80	4.85	0.12	13.41	5.65	7.07	7.20
8	5110	268	715	0.20	4.43	0.11	7.41	4.62	5.78	2.67
7	5100	250	535	0.18	4.96	0.10	5.05	5.14	6.27	2.14
6	5090	247	464	0.63	5.93	0.09	10.21	6.56	7.96	1.88
5	5080	105	249	1.46	4.88	0.09	14.45	6.34	7.70	2.37
4	5070	106	169	1.18	4.63	0.06	8.93	5.81	6.90	1.59
3	5060	102	75	1.08	4.63	0.07	7.37	5.70	6.76	0.74
2	5050	68	8	1.14	3.74	0.06	8.05	4.89	5.82	0.12
1	5040	9	9	0.97	2.40	0.05	6.28	3.37	4.03	0.97
Total		1,438	8,534	0.69	4.90	0.10	10.09	5.59	6.87	5.94

Source: SRK, 2025

Notes: 5% ore loss and 5% dilution have been applied. The sum of individual amounts may not equal due to rounding.

Figure 8.7: Bench-by-bench materials – final pit design



Source: SRK, 2025

Notes: 5% ore loss and 5% dilution have been applied.

8.3.5 Mining method

Mining operations

The open-pit mine is planned to operate using a conventional truck and shovel method, with operations scheduled for 200 days per year and under a contractor mining scenario.

The drilling operations are assumed to use the Flexi ROC T35 top hammer drill rig, which accommodates hole diameters ranging from 64 mm to 115 mm. The drilling is completed on a drill pattern of 5.3 m × 4.6 m. Delay blasting technique is employed. The blast hole inclination ranges from 70° to 90°. Blasting operations are forecast to be carried out every 2 to 3 days.

The designed bench height is 5 m, with double benching (10 m) employed. To meet the bench height and balance the number of 25-tonne trucks, 2 m³ hydraulic excavators were selected.

Mined materials will be transported along haul roads, which maintain a suitable gradient for efficient movement. Double-lane haul roads have been constructed (13 m wide), while single-lane roads (9 m wide) provide access to the pit bottom. The maximum gradient for these roads is set to 9% to ensure safe and efficient haulage.

Materials transportation

The main access ramp is located to the northeast of the Pb14 deposit area. The Pb14 deposit has been actively mined for many years, with a relatively complete road transportation system in place.

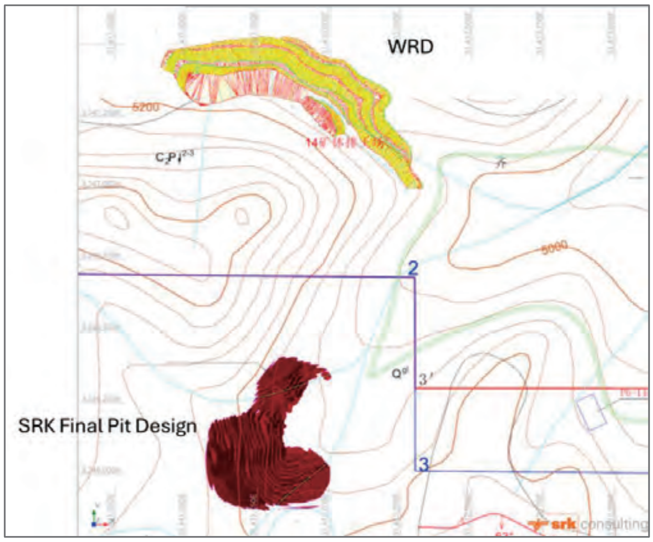
The ROM ore will be transported by truck to the processing plant. The distance from the mining area to the processing plant is approximately 11 km (green line in Figure 8.8).

The mineralised waste is hauled by trucks to the mineralised waste dump located approximately 600 m northeast of the open-pit, on the northern side (Figure 8.9).

Figure 8.8: Transportation route from mining area to processing plant

Source: Huaxia Mining

Figure 8.9: Mineralised waste dump location



Source: SRK, 2025

Equipment

Material will be blasted and subsequently loaded into trucks using 2 m³ hydraulic excavators. These trucks, with a capacity of 25 tonnes, will transport the loaded material to the processing plant or the mineralised waste dump. SRK received the contractor equipment list from Huaxia Mining (as outlined in Table 8.7).

The equipment fleet includes a total of 5 excavators and 15 trucks.

Table 8.7: Primary and ancillary equipment list

Equipment	Model	No. of Units
Hitachi 360 Excavator	ZX360H-5A	1
Komatsu 360 Excavator	PC360-7	1
Sumitomo 300 Excavator	SH360HD-6	1
Sumitomo 350 Excavator	SH350-5	1
Sumitomo 380 Excavator	SH380HD-5	1
Sanhe 385 Breaking hammer	SWE385E	1
Lingong Loader	965F	1
Trucks	NA	15
DTH drill rig	HT600	1
Top hammer drill rig	FLexi ROC T35-11	1

Source: Huaxia Mining, summarised by SRK

Mineralised waste dump design

Material below the designated cut-off grade will be stored in mineralised waste dump located approximately 600 m northeast of the open-pit. The mineralised waste dump is designed with an in-place density assumption of 2.2 t/m^3 and has sufficient capacity to accommodate the 8.9 Mt of waste material projected to be stripped over the LOM.

Dump trucks and bulldozers are expected to be used for mineralised waste disposal. Excavators load the stripped mineralised waste onto trucks, which then transport the material via designated access ramps and mine roads to the mineralised waste dump. At the mineralised waste dump, bulldozers are deployed to spread and level the mineralised waste material, ensuring efficient and orderly site management.

Water supply

For Pb14 OP operation, water is sourced from the Qizu River, approximately 770 m east of the mining area, via a riverside sump and pump system to an elevated tank for production and fire suppression.

Mine dewatering

Dewatering operations primarily address the management of pit inflows and precipitation.

In the hillside section of the pit, mechanical drainage is not required. Instead, strategically placed water channels along the benches efficiently direct runoff towards the main drainage channel, facilitating the discharge of water beyond the pit boundaries.

In the sunken section of the pit, drainage infrastructure includes the following:

- in-pit drainage ditches along the cleaning platform and water channels along pit roads that direct runoff to a sump at the pit bottom
- submersible pumps installed to discharge accumulated water from the pit
- an external diversion ditch to prevent external runoff from entering the pit.

8.3.6 Production schedule

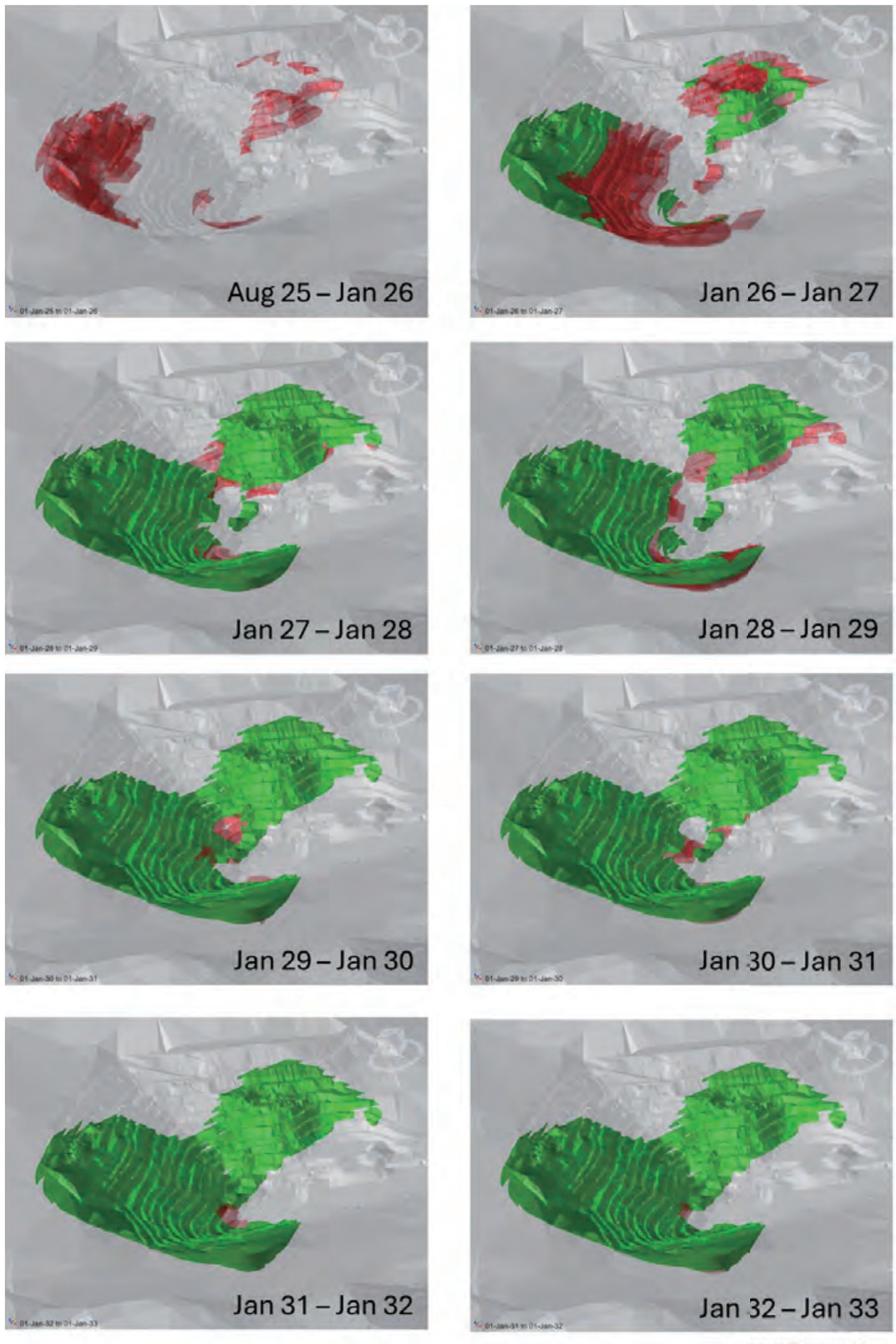
The LOM schedule has been developed using the Deswik suite software. The production schedule is organised on a yearly basis with an output of 200 kt.

The following assumptions were made during the production scheduling:

- Only blocks classified as Measured and Indicated Mineral Resources with an EqPb (lead equivalent) grade above 1.8% were considered and considered as ROM ore.
- Blocks classified as Inferred Mineral Resource, those with an EqPb grade below 1.8% and sterilised material were designated as waste.
- A conceptual starter pit was planned to guarantee supply of ROM ore in 2025.
- The maximum annual movement of waste material is at 4,000 kt/a.
- The vertical lag between mining phases is restricted to a maximum of 9 benches.
- The minimum mining width is 25 m.

The end-of-period mine designs are shown in Figure 8.10. The forecast LOM is 8 years, starting from August 2025, with 200 kt annually. The ROM amount is 1,438 kt with average EqPb grade of 6.87%, and the total EqPb contained metal is 98.74 kt. The waste amount is 8,534 kt, with an average stripping ratio of 5.94 t/t.

Figure 8.10: End-of-period mine designs



Source: SRK, 2025

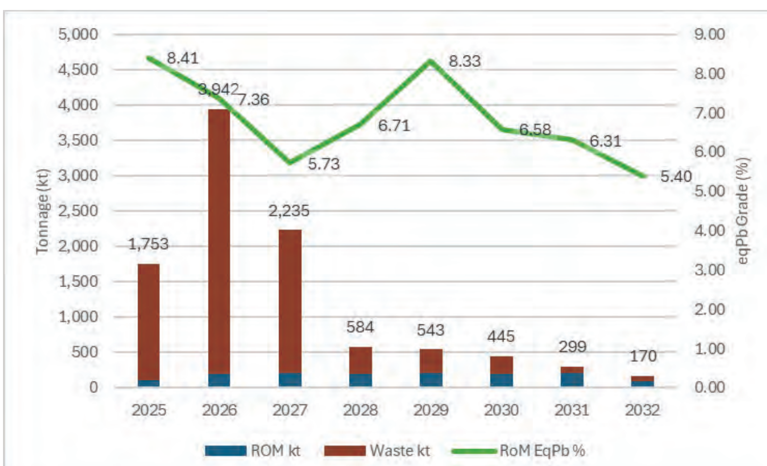
The annual mine production schedule is shown in Table 8.8 and Figure 8.11.

Table 8.8: Annual mine production schedule

Mining Physicals	Unit	LOM	2025	2026	2027	2028	2029	2030	2031	2032
ROM	kt	1,438	113	199	206	200	214	197	208	101
ROM Pb Grade	%	0.69	1.11	1.00	0.30	0.35	0.63	0.70	0.85	0.94
ROM Zn Grade	%	4.90	5.50	4.92	4.20	5.12	6.26	4.76	4.43	3.52
ROM Cu Grade	%	0.10	0.19	0.12	0.12	0.11	0.09	0.09	0.08	0.06
ROM Ag Grade	g/t	10.09	17.63	14.34	10.53	7.68	10.27	6.84	6.80	9.99
ROM EqPb Grade	%	6.87	8.41	7.36	5.73	6.71	8.33	6.58	6.31	5.40
ROM Pb+Zn Grade	%	5.59	6.61	5.92	4.51	5.46	6.89	5.47	5.47	4.45
Waste	kt	8,534	1,640	3,744	2,029	384	329	248	90	70
Total Material Movement	kt	9,972	1,753	3,942	2,235	584	543	445	299	170
Stripping Ratio	t: t	5.94	14.51	18.86	9.85	1.92	1.54	1.26	0.43	0.69

Source: SRK, 2025

Figure 8.11: Annual mine production schedule



Source: SRK, 2025

Notes:

[1] The line represents the average EqPb grade, corresponding to the right axis.

[2] The column represents the ROM tonnes amount, corresponding to the left axis.

8.4 Pb12 underground

8.4.1 Geotechnical and hydrogeology

Hydrological conditions

The Second Brigade has conducted various phases of hydrogeological surveys in the mining area since 2003.

The mine is located on the southern side of the Nyainqentanglha Mountains, within the Yulonglang Valley of the upper Lhasa River, at an elevation ranging from 4,800 m to 5,570 m. The area features steep slopes and a general topography that is higher in the south and lower in the north.

The climate is classified as belonging to a typical temperate plateau semi-arid monsoon type. From 1978 to 2009, the average temperature was 6.1°C; from 1998 to 2010, the average annual rainfall was 591.76 mm, with a range of 413.0 mm to 761.4 mm. The maximum daily rainfall recorded was 47.3 mm on 19 August 2002. The rainy season, from June to September, accounts for 83.51% of the annual rainfall, with the heaviest single rainstorm delivering 33.0 mm over 3.0 to 4.0 hours, causing localised soil erosion.

Surface water

The nearby river flows through the Mengya'a Valley from south to north, passing east of the mining area. It is a seasonal river with a U-shaped valley. The width of the main channel ranges from 10 m to 20 m, with gentler slopes in the middle and lower reaches (3° to 7°) and steeper slopes upstream (10° to 14°), with incision depths of 1.4 m to 4.8 m. The channel is relatively straight. During the summer, the flow rate ranges from 16.7 L/s to 67.0 L/s.

Freezing begins in late October, and thawing occurs in March to April. The channel surface consists of 1.0–3.0 m of Quaternary glacial boulder soil, which is highly permeable, allowing most water to flow as subsurface runoff at the channel bed.

Faults

Regional faults are not well developed in the mining area, but numerous small faults, ranging from 0.6 km to 5.0 km in length, are present.

The Pb12 UG contains nine small faults: F1, F5, F6, F7, F8, F9, F10, F11 and F25.

The Pb14 OP contains six faults: F16, F19, F20, F21, F22 and F23.

According to drilling data, the vertical thickness of the structural fracture zones generally ranges from 2.59 m to 44.51 m.

Groundwater

Groundwater in the mining area is categorised into two main types: above and below the permafrost. The above-permafrost water has a limited distribution and thickness, with poor aquifer properties in the mining area, and its main concentration is in the lower-lying valley areas, posing minimal impact on mineral extraction. Below-permafrost water consists mainly of Bedrock Weathering Fracture

Water and Bedrock Structural Vein Fracture Water.

Bedrock Weathering Fracture Water: In the Pb12 UG, the base of the weathered zone is generally 46.00 m to 283.25 m below the surface, with a thickness of 26.24 m to 279.75 m. The aquifer thickness is 3.4 m to 139.95 m, averaging 46.08 m. In the Pb14 OP, the base of the weathered zone is generally 38.7 m to 124.4 m below the surface, with a thickness of 29.6 m to 149.3 m.

Bedrock Structural Vein Fracture Water: This is found within fault fracture zones, mainly distributed along the watershed, at higher elevations. The recharge area is small, and the surrounding rock has poor aquifer properties, limiting recharge sources.

Prediction of mine water inflow

The Pb12 UG is located over 2.2 km from the Mengya'a riverbed, with the orebody elevation higher than the riverbed elevation, resulting in minimal river water impact on mine water inflow. The average thickness of the weathered zone in Pb12 UG is 132.63 m (31.0 m strongly weathered, 101.0 m slightly weathered), with an average aquifer thickness of 46.08 m, a water level elevation of 5,179.45 m, and a permeability coefficient of 0.153 m/d. The Pb12 UG area has nine small faults, with structural fracture zones having an average vertical thickness of 15.58 m.

As for the Pb14 OP, drilling data from 16 boreholes indicate an average weathered zone thickness of 80.27 m (27.0 m strongly weathered, 53.0 m slightly weathered). On 8 November 2018, the total spring flow from the pit slopes was measured at 20.5 L/s. The Pb14 OP is associated with six small faults, with shallow fracture zones consistent with the weathered fracture zone elevations.

CINF conducted water inflow predictions using two methods: the 'Large Well Method' and the 'Hydrogeological Analogy Method'. The results are summarised in Table 8.9.

Table 8.9: Water inflow prediction

Deposit	Elevation (m)	Large Well Method (m^3/d)	Hydrogeological Analogy Method (m^3/d)
Pb12 UG	4,980	4,679	3,705
Pb12 UG	5,100	3,133	3,012
Pb14 OP	5,088	1,715 (Actual 1,771)	1,771

Source: 22 CINF Mining Design

Eventually, for the Pb12 UG, CINF predicted normal water inflow at the 5,100 m RL to be 3,100 m^3/d with a maximum inflow of 4,700 m^3/d. At the 4,980 m RL, normal inflow is predicted at 4,700 m^3/d, with a maximum inflow of 7,000 m^3/d.

As for the Pb14 OP, there is no recent prediction available; the primary management strategy is based on ongoing operation.

Geotechnical conditions

The surrounding rock, as well as the hanging and footwall rocks, primarily consists of limestone, sandstone and slate. The rock quality designation (RQD) values, categorised by different rock type and degree of weathering, are listed in Table 8.10.

Table 8.10: Rock quality designation in Pb12 UG

Rock type	Degree of weathering	RQD
Sandstone	Highly Weathered	0.4–0.65
	Moderately Weathered	0.6–0.8
	Slightly Weathered	0.75
Slate	Highly Weathered	0.35–0.4
	Moderately Weathered	0.45–0.76
	Slightly Weathered	0.75
Limestone	Highly Weathered	0.85
	Moderately Weathered	0.75
	Slightly Weathered	0.8

Source: 22 CINF Mining Design

The definitive rock strength properties for each rock type are detailed in Table 8.11.

Table 8.11: Rock strength summary in Pb12 UG

Rock Type	Degree of Weathering	Density (g/cm^3)		Water Absorption	Saturation Rate	Softening Coefficient	UCS (MPa)		DM (MPa)	EM (MPa)	SS (MPa)	Poisson's Ratio
		Natural	Saturated				Natural	Saturated				
Sandstone	Fresh to Slightly	2.74–2.78	2.75–2.79	0.12–0.51	0.54–0.84	0.85–0.91	42.5–94.2	36.3–85.4	42,978–70,706	44,438–72,420	6.30–6.85	0.08–0.10
	Highly to Moderately	2.66–2.68	2.67–2.72	0.42–0.44	0.78–0.84	0.58–0.65	31.3–35.3	23.2–28.4		11,000–20,600	2.46–2.72	0.16
Slate	Fresh to Slightly	2.66–2.71	2.71–2.77	0.43–0.82	0.47–1.44	0.62–0.87	40.1–69.0	33.2–61.2	40,655–59,969	1,738–2,044	0.93–2.06	0.14–0.38
	Highly to Moderately	2.64–2.76	2.67–2.77	0.43–0.82	0.75–2.52	0.56–0.61	3.5–5.7	1.8–3.5	1,307–1,526	1,617–1,842	0.73–0.75	0.37–0.41
Limestone	Fresh to Slightly	2.63–2.75	2.64–2.76	0.43–0.82	0.67–3.15	0.74–0.86	62.6–70.9	52.6–63.3	1,587–44,632	1,893–45,880	0.75–7.67	0.15–0.38
	Highly to Moderately	2.58–2.70	2.60–2.71	0.43–0.82	0.95–3.30	0.53–0.65	3.7–21.6	2.1–14.4	1,143–2,578	1,472–2,914	0.72–0.78	0.36–0.40

Source: 22 CINF Mining Design

Notes:

[1] UCS: Uniaxial Compressive Strength
[2] DM: Deformation Modulus
[3] EM: Elastic Modulus
[4] SS: Shear Strength

CINF established a comprehensive rock model, including the surface. The numerical model has dimensions of 1,900 m in the x-direction, 1,000 m in the y-direction, and extends from an elevation of 4,500 m to the surface in the z-direction, comprising 90,230 nodes and 480,668 elements. Table 8.12 summarises the physical and mechanical parameters of the materials used for the analysis.

Table 8.12: Physical and mechanical parameters of the materials

Material Type	Density (t/m^3)	EM/GPa	Poisson's Ratio	IFA (°)	Cohesion (MPa)	TS (MPa)
Glacial till	2.4	0.40	0.29	31.00	0.10	0.05
Surrounding rock	2.7	4.00	0.25	43.00	2.30	1.50
Orebody	3.52	2.00	0.23	38.00	2.00	1.20
Primary backfill	1.82	0.57	0.24	29.00	0.78	0.67
Secondary backfill	1.8	0.06	0.26	19.00	0.17	0.11

Source: 22 CINF Mining Design

Notes:

[1] EM: Elastic Modulus

[2] IFA: Internal Friction Angle

[3] DM: Deformation Modulus

[4] TS: Tensile Strength

CINF made the following assumptions during the analysis of stope stability and surface stability:

- The influence of distant minimal orebodies and structural planes from the main orebody is not considered.
- Each type of rock mass and fill material is regarded as an isotropic continuous medium.
- The mined-out stope is assumed to be completely filled to the roof, with no gaps between the fill material and the rock mass.
- The effects of seismic waves, explosive shock waves, and groundwater on rock mass stability are not considered.
- The physical and mechanical parameters of the rock mass used in the analysis are selected based on geological reports and by referencing benchmarks.

Stope stability geotechnical analysis

CINF used FLAC3D to analyse the stope stability for the Sublevel Open Stoping with Delayed Backfill mining method, considering both transverse and longitudinal arrangements. FLAC3D is a sophisticated numerical modelling software designed for the analysis of geotechnical and rock mechanics in three dimensions. During the simulation, the mining sequence for sublevels is conducted from bottom to top.

For stoping, the mining sequence follows a primary and secondary approach.

Sublevel Open Stoping with Delayed Backfill (Transverse)

Table 8.13 summarises the stope parameters used for the geotechnical analysis of stope stability.

Table 8.13: Stope parameters used for geotechnical analysis – Sublevel Open Stoping with Delayed Backfill (Transverse)

	Level spacing (m)	Stope width (m)	Stope length (m)
Scenario 1	40	12	Orebody thickness
Scenario 2	40	15	Orebody thickness
Scenario 3	40	18	Orebody thickness

Source: 22 CINF Mining Design

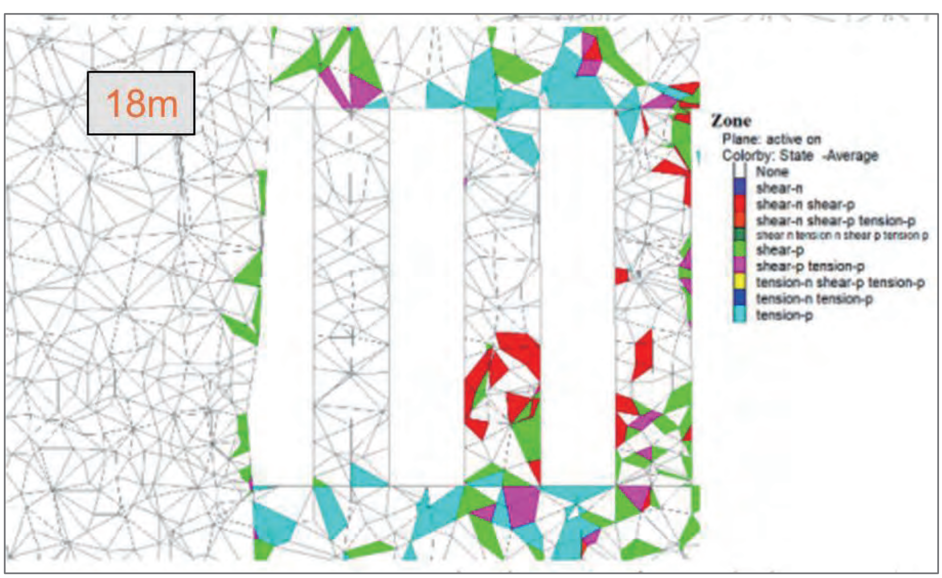
Table 8.14 summarises the analysis results. Figure 8.12 illustrates the results of the plastic zone analysis.

Table 8.14: Plastic zone analysis results – Sublevel Open Stoping with Delayed Backfill (Transverse)

	Displacement (mm)	Maximum principal stress (MPa)	Tensile strength (MPa)	Plastic zone penetration?
Scenario 1	28.37–60.89	6–8	0.15–0.45	No
Scenario 2	30.25–65.03	6–8	0.15–0.45	No
Scenario 3	62.87–118.36	8–11.5	0.68–0.95	Yes

Source: 22 CINF Mining Design

Figure 8.12: Plastic zone analysis results (primary stoping, front view, 18 m)



Source: 22 CINF Mining Design

Based on this analysis, CINF has concluded the following:

- **Displacement analysis**: For both primary and secondary stoping, the simulation of stopes with widths of 12 m, 15 m and 18 m showed that the maximum deformation occurs at the hanging wall and footwall. It was observed that when the stope width increased to 18 m, there was a sharp increase, with a maximum total displacement of 118.36 mm. This indicates that when stope width reaches 18 m, the stope is prone to delamination and collapse, which may cause increased waste dilution.
- **Maximum principal stress analysis**: For both primary and secondary stoping, the maximum principal stress is concentrated at the bottom corners of the sidewall and the hanging wall. Overall, the backfill material does not exhibit excessive compressive stress concentration. However, when the stope width is 18 m, compressive stress concentration may occur in the surrounding rock.
- **Tensile stress analysis**: During primary stoping, the maximum tensile stress in the surrounding rock of stopes with widths of 12 m and 15 m is less than that for the 18 m width. As mining progresses, the tensile stress is primarily concentrated in the surrounding rock of the sidewalls or the hanging wall and footwall. In secondary stoping, the maximum tensile stress for 12 m and 15 m widths is less than the tensile strength of the backfill material (0.67 MPa). However, for an 18 m width stope, the tensile stress exceeds the tensile strength of the backfill material, resulting in damage to the backfill.
- **Plastic zone analysis**: When the stope width reaches 18 m, the extent of plastic zone damage in the surrounding rock is significant, and plastic zone penetration was observed.

Sublevel Open Stoping with Delayed Backfill (Longitudinal)

Table 8.15 summarises the stope parameters used for the geotechnical analysis of stope stability.

Table 8.15: Stope parameters used for geotechnical analysis – Sublevel Open Stoping with Delayed Backfill (Longitudinal)

	Level spacing (m)	Stope width (m)	Stope length (m)
Scenario 4	40	Orebody thickness	30
Scenario 5	40	Orebody thickness	40
Scenario 6	40	Orebody thickness	50

Source: 22 CINF Mining Design

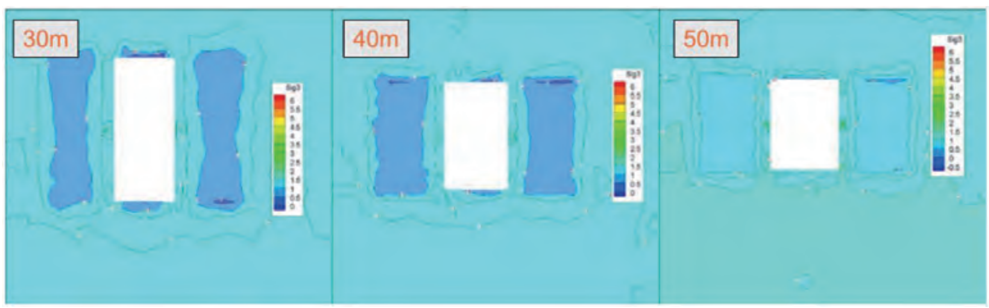
Table 8.16 summarises the analysis results. Figure 8.13 illustrates the results of the tensile stress analysis.

Table 8.16: Tensile stress analysis results – Sublevel Open Stoping with Delayed Backfill (Longitudinal)

	Displacement (mm)	Maximum principal stress (MPa)	Tensile strength (MPa)	Plastic zone penetration?
Scenario 4	18.84~23.59	6~10	0.34~0.37	No
Scenario 5	23.01~27.64	6~10	0.43~0.5	No
Scenario 6	48.39~55.62	8~12	0.63~0.7	No

Source: 22 CINF Mining Design

Figure 8.13: Tensile stress analysis results (secondary, front view)



Source: 22 CINF Mining Design

Based on this analysis, CINF concluded the following:

- **Displacement analysis**: When the stope length is 50 m, the back displacement deformation is approximately twice that of stope lengths of 40 m and 30 m.
- **Maximum principal stress analysis**: For all three mining lengths, the maximum principal stress is concentrated at the top and bottom corners of the stope sidewalls. During the extraction process, an increase in stope length leads to excessive compressive stress concentration in the surrounding rock.
- **Tensile stress analysis**: During the primary stoping, the maximum tensile stress generated in the back is less than the tensile strength of the surrounding rock. In the secondary stoping, the maximum tensile stress is distributed within the backfill. Given that the tensile strength of the backfill is 0.67 MPa, the backfill becomes unstable when the stope length reaches 50 m, potentially leading to backfill failure.
- **Plastic zone analysis**: The distribution of the plastic zone does not vary significantly with different stope lengths, and no plastic zone penetration is observed.

Stope design recommendations

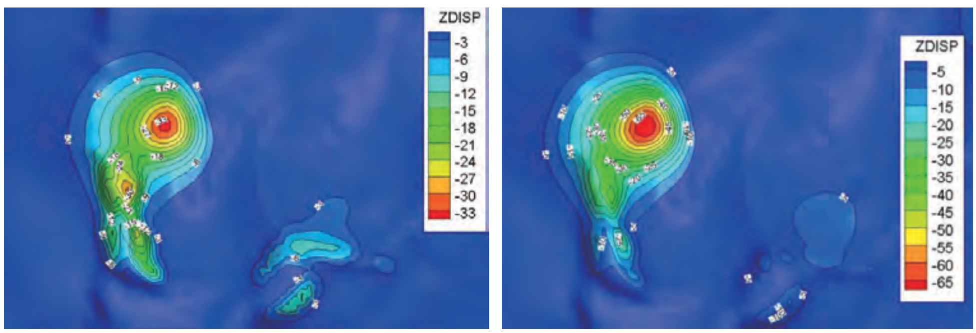
Based on the stope stability analysis, CINF recommended the following:

- For Sublevel Open Stoping with Delayed Backfill (Transverse), the recommended stope width is set at 15 m.
- For Sublevel Open Stoping with Delayed Backfill (Longitudinal), the recommended stope length is set at 40 m.

Surface stability analysis

CINF also conducted a surface stability analysis to determine the impact of surface subsidence deformation under both backfilled and non-backfilled mining conditions. The direction of maximum displacement observed on the surface is vertical. In the backfilled scenario, the maximum vertical displacement was -33.43 mm, whereas in the non-backfilled scenario, the maximum vertical displacement was -68.96 mm.

Figure 8.14: Tensile stress analysis results (secondary, front view)



Backfilled mining **Non-backfilled mining**

Source: 22 CINF Mining Design

8.4.1 Mineable shape optimisation

Mining Block Model

The mine design and resource estimation within the pit design were based on MRM developed by SRK, with an effective date of 31 December 2024. The model was provided in Datamine (.dm) format.

Key parameters of the block model are outlined in Table 8.17.

Table 8.17: Resource block model parameters

Range	Minimum	Maximum
Easting	31,413,700	31,415,140
Northing	3,345,600	3,346,400
Elevation	4,900	5,500
X Size	2	20
Y Size	2	20
Z Size	2	4
Rotation	-	-

Source: SRK

SRK converted the MRM to a MBM in Block Geomodel (.gmdlb) format for mine planning purposes in Deswik.

The Deswik.SO (Stope Optimizer) module was used on the MRM to generate mineable shapes.

These shapes were subsequently used to refine and enhance the proposed mine plan. The input parameters used in the SO module are summarised in Table 8.18.

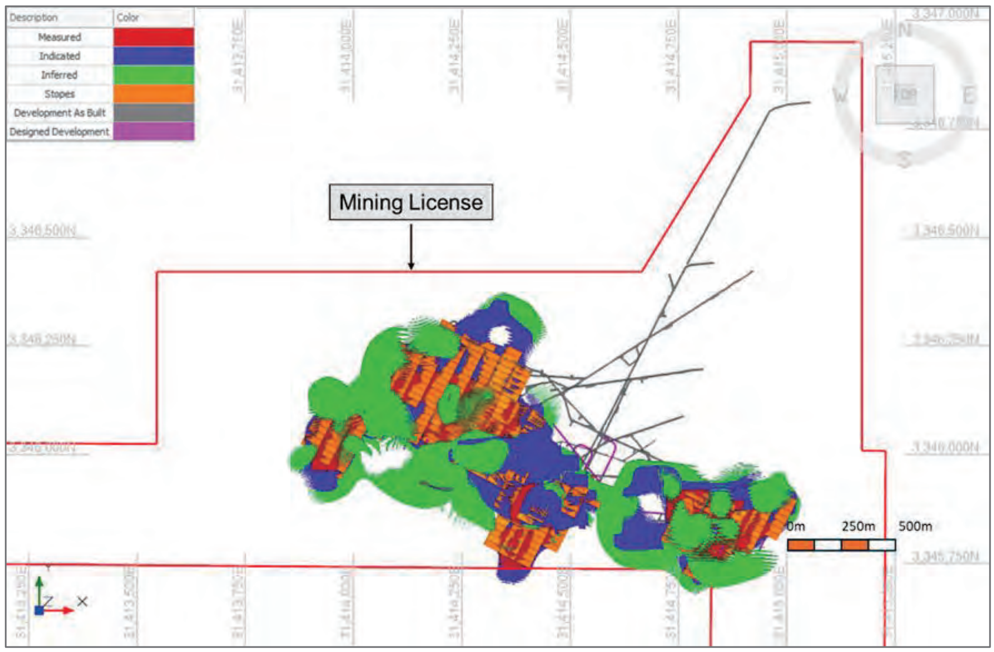
Table 8.18: SO parameters for underground mining

Parameters	Unit	Inputs
Default Density	t/m^3	2.81
Optimisation Field		EqPb
Cut-off Grade	%	2.9 for fresh and 4.6 for oxide
Head Grade	%	3.7 for fresh and 6.0 for oxide
Block Size	m	2×2×2
Stope Orientation Plan		XY
Rotate	°	27 on Z axis
Mining Width	m	15
Mining Length	m	40
Minimum Stope Height	m	3
Minimum Pillar Height	m	5
Roof Dilution	m	0.5
Floor Dilution	m	0.2
Default Strike	°	0
Strike Range	°	-10 to 10
Maximum Strike Change	°	5
Default Dip	°	-22.5
Maximum Dip Change	°	5 for Roof, 1 for Floor
Dip Range	°	-45 to 0 for Roof, -1 to 1 for Floor

Source: SRK

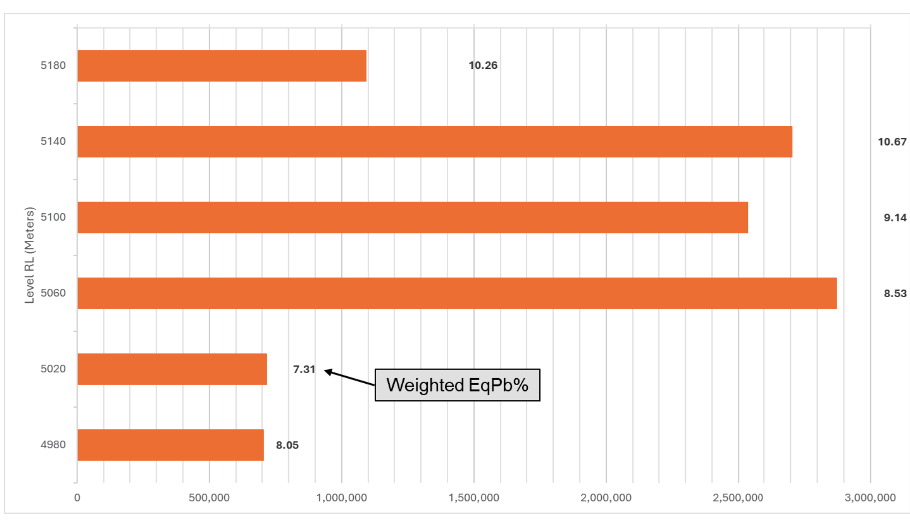
The mine design result generated is illustrated in Figure 8.15. Figure 8.16 presents the tonnes and average EqPb grade within the stopes at each vertical interval.

Figure 8.15: Plan view of Ore Reserve estimate areas – Pb12 UG



Source: SRK

Figure 8.16: Tonnes and average EqPb grade within stopes per vertical interval



Source: SRK, 2025

8.4.2 Mining method

The Pb12 deposit is planned to be mined as an underground operation, with work scheduled to occur over 200 days per year, comprising three shifts of 8 hours each. The annual production capacity is set at 200 kt.

The following section provides a discussion regarding the mining methods, development, and essential mine services required for underground mining exploitation.

Mining method selection

The selection of a mining method involves considering a variety of factors to ensure safety, efficiency, and environmental responsibility. In the 22 CINF Mining Design, four mining methods were nominated for the Mengya'a mine. Table 8.19 summarises the mining methods considered for the Pb12 UG.

Table 8.19: Mining methods – Pb12 UG

Thickness	Dip (°)	Mining Method
<8 m	≤30	Room and Pillar with Delayed Backfill
≥8 m	≤30	Sublevel Open Stope with Delayed Backfill (Gently Dip Orebody & Longitudinal)
<15 m	>30	Sublevel Open Stope with Delayed Backfill (Steep Dip Orebody & Longitudinal)
≥15 m	>30	Sublevel Open Stope with Delayed Backfill (Steep Dip Orebody & Transverse)

Source: 22 CINF Mining Design

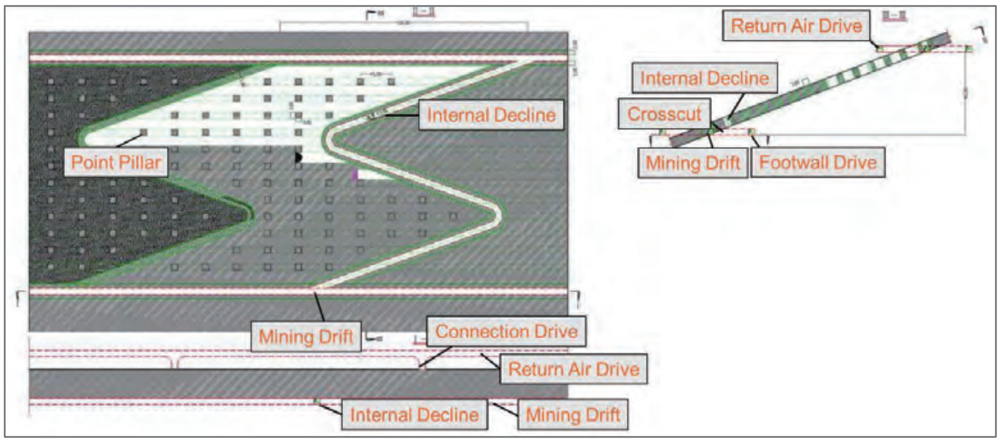
Room and Pillar with Delayed Backfill (RPDB)

- **Stope Configuration**: The stope extends 120 m along the strike, with its width corresponding to the orebody thickness and a level spacing of 40 m. The rib pillar measures 3 m, and the crown and sill pillars are both 2 m. Along the dip, an internal decline, featuring a switchback height of 13–14 m, serves two adjacent stopes, with 2 m pillars left on either side of the internal decline. As mining progresses, 3 m × 3 m square point pillars are placed, spaced 15 m along the strike and 8–9 m along the dip.
- **Stope Preparation**: A crosscut is driven perpendicular to the orebody from the footwall drive. Upon reaching the orebody, a mining drift is developed along the strike. From the mining drift, an internal decline is constructed within the stope, maintaining a gradient below 15% to facilitate access for loaders and drilling jumbos. The internal decline uses switchbacks to manage the gradient and connects to the upper level.
- **Stoping**: Mining is initiated from the top downwards. A Sandvik DD2710 single-boom jumbo drill is used to drill blast holes with diameters ranging from 43 mm to 64 mm and lengths of 2.5–3.5 m. Mining commences from the primary drift, followed by the secondary drift, which are mined in intervals, ultimately leaving 3 m × 3 m square point pillars in the secondary drift. Blasted materials are extracted using an ACY 204 diesel-powered loader with a 4-tonne capacity. These materials are transported via the internal decline to the mining drive, then to the footwall drive, and loaded onto trucks. CINF assumed that the crown and sill pillars could be recovered. Regarding rib pillars, CINF considers that the resource loss is not significant (3 m against 120 m).

- **Ventilation**: Fresh air flows from the mining drift through the internal decline to the stope working face. Exhaust air is channelled through the primary/secondary drift and connection drive into the return air drive, eventually exiting to the surface via a blind return air raise and return air adit.
- **Backfill**: Upon completion of mining, backfill barricades are erected, and backfill is introduced through pipelines from the upper level.

Figure 8.17 illustrates a schematic diagram of the mining method.

Figure 8.17: Schematic of RPDB mining method



Source: 22 CINF Mining Design and modified by SRK

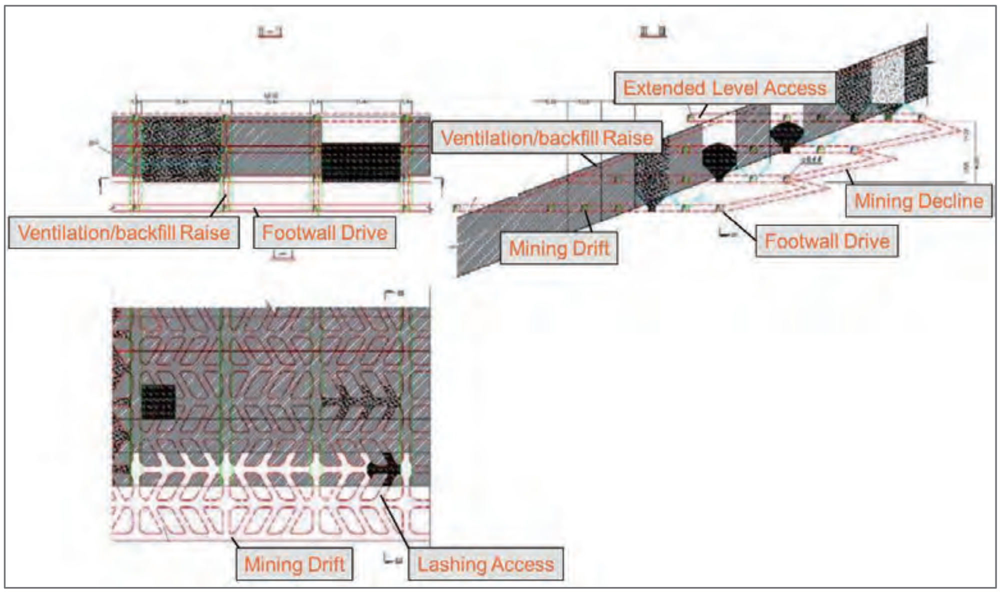
Sublevel Open Stope with Delayed Backfill (Gently Dip Orebody & Longitudinal) (SLOSG)

According to the 22 CINF Mining Design guidelines, SLOSG is typically applied to orebodies with a dip angle of less than 30° and an average thickness normally exceeding 8 m.

- **Stope Configuration:** The stope extends 40 m along the strike, with a rib pillar measuring 5 m, and no crown or sill pillars are left. Along the dip, the stopes are divided into primary and secondary stopes, both with a width of 15 m. The level spacing is 40 m, with sublevel spacing ranging from 13 m to 14 m.
- **Stope Preparation:** From the footwall drive, mining declines are drifted, and from the mining decline, level or sublevel access drifts are created to reach the stopes in a transverse direction. Mining drifts are arranged in the centre of the stopes. Along the strike, inclined loading accesses are drifted to reach the stopes. Wedge-shaped bottom structures will be recovered at a later stage. The level or sublevel accesses are extended to the hanging wall to connect with the ventilation/backfill raise.
- **Stoping:** Mining is initiated from the primary stopes first, then proceeds to the secondary stopes. A Sandvik DL2710 longhole drill is used to drill blast holes in a fan pattern (diameters range from 64 mm to 89 mm). Blasted materials are extracted using an ACY 204 diesel-powered loader with a 4-tonne capacity. For the bottom structures and rib pillars, drift mining is considered for recovery.
- **Ventilation:** Fresh air flows from the footwall drive, mining decline, and level/sublevel access to the mining drift and stope working face. Exhaust air is channelled through the ventilation/backfill raise and upper-level access, eventually exiting to the surface via a blind return air raise and return air adit.
- **Backfill:** Upon completion of mining the stopes, backfill barricades are erected, and backfill is introduced through pipelines from the ventilation/backfill raise. To reduce backfill dilution during mining of the secondary stopes, high-cement backfill should be considered for the primary stopes. For the secondary stopes, the bottom 5 m should be filled with high-cement backfill, while the rest of the void can be filled with low-cement backfill.

Figure 8.18 illustrates a schematic diagram of the SLOSG mining method.

Figure 8.18: Schematic of SLOSG mining method



Source: 22 CINF Mining Design and modified by SRK

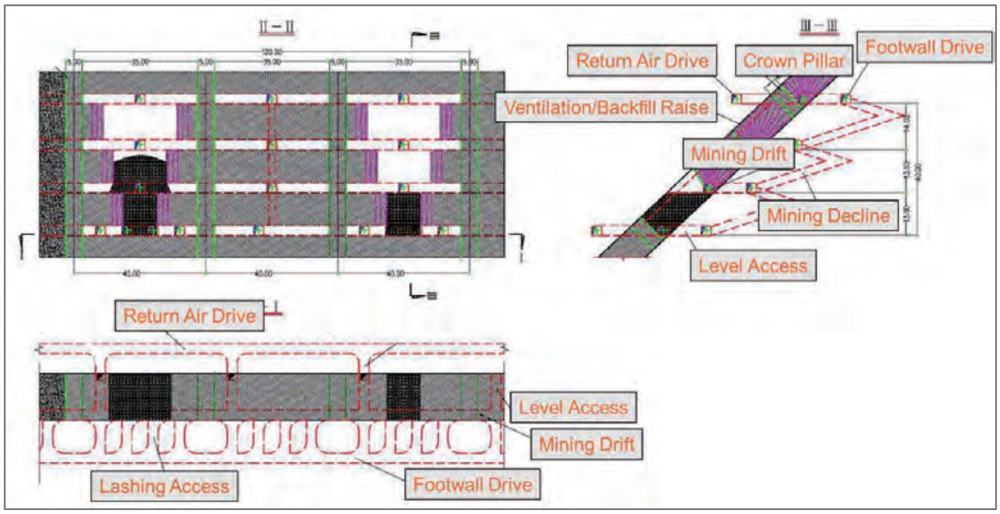
Sublevel Open Stope with Delayed Backfill (Steep Dip Orebody & Longitudinal) (SLOSS)

The fundamental philosophy of SLOSS is similar to that of SLOSG. The primary distinction between these two methods lies in the shape of the orebody, particularly the steeper dip angle associated with SLOSS compared to SLOSG.

- **Stope Configuration:** The stope extends 40 m along the strike, with a rib pillar measuring 5 m, and no sill pillars are left. However, a 4 m crown pillar is left between the levels. The level spacing is 40 m, with sublevel spacing ranging from 13 m to 14 m.
- **Stope Preparation:** Stope preparation is similar to that of SLOSG, with the only difference being that in SLOSS, only level access should be extended to form the ventilation system.
- **Stoping:** Mining is initiated by sublevels, proceeding from top to bottom. The drilling, blasting, and lashing processes are the same as those used in SLOSG. For the bottom structures and rib pillars, drift mining is considered for recovery.
- **Ventilation:** The ventilation philosophy is the same as that of SLOSG.
- **Backfill:** Upon completion of mining the stopes, backfill barricades are erected, and backfill is introduced through pipelines from the ventilation/backfill raise. The bottom 5 m should be filled with high-cement backfill, while the remainder of the void can be filled with low-cement backfill.

Figure 8.19 illustrates a schematic diagram of the SLOSS mining method.

Figure 8.19: Schematic of SLOSS mining method



Source: 22 CINF Mining Design and modified by SRK

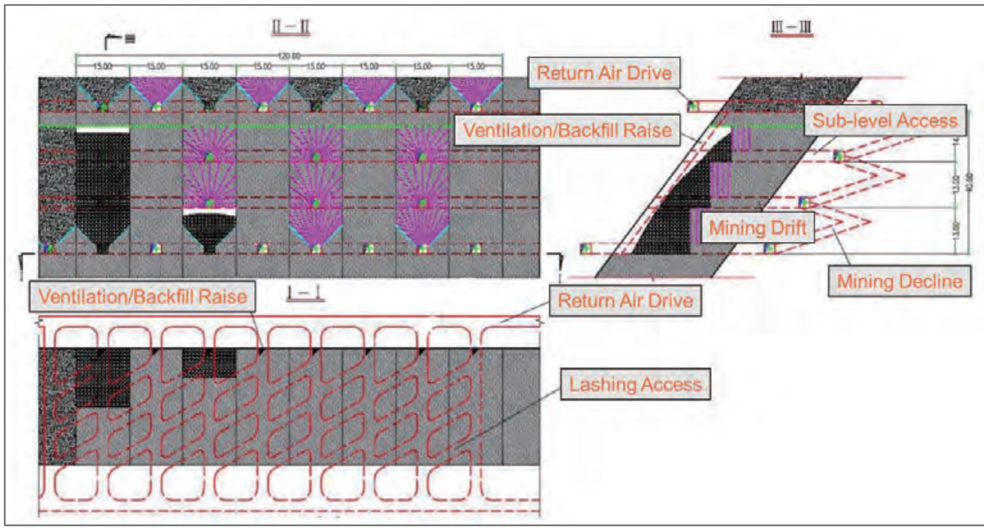
Sublevel Open Stope with Delayed Backfill (Steep Dip Orebody & Transverse) (SLOST)

When applying SLOST, the shape of the orebody is similar to that of SLOSS. However, the horizontal thickness of the orebody is greater, typically exceeding 15 m, which allows mining to commence in a transverse direction towards the orebody.

- **Stope Configuration**: The stope width is 15 m, and the transverse length of the stope corresponds to the horizontal thickness of the orebody. No rib pillars or sill pillars are left; however, a 4 m crown pillar is left between the levels. The level spacing is 40 m, with sublevel spacing ranging from 13 m to 14 m.
- **Stope Preparation**: Stope preparation is similar to that of SLOSS. The main difference with SLOST is that once the level or sublevel access reaches the stope, the mining drift continues in a transverse direction towards the orebody.
- **Stoping**: Mining is initiated by sublevels, proceeding from top to bottom, with each sublevel following a retreat mining sequence. The drilling, blasting, and lashing processes are the same as those used in SLOSS. For the bottom structures and rib pillars, drift mining is considered for recovery.
- **Ventilation**: The ventilation philosophy is the same as that of SLOSS.
- **Backfill**: The backfill process is the same as that of SLOSS.

Figure 8.20 illustrates a schematic diagram of the SLOST mining method.

Figure 8.20: Schematic of SLOST mining method



Source: 22 CINF Mining Design and modified by SRK

Mine design

Level spacing

The main level spacing is set at 40 m, with sublevel spacing ranging from 13 m to 14 m. The main levels are established at elevations of 5,210 m RL, 5,180 m RL, 5,140 m RL, 5,100 m RL, 5,060 m RL, 5,020 m RL and 4,980 m RL.

As at 31 December 2024, the main development system at 5,210 m RL, 5,180 m RL, 5,140 m RL, 5,100 m RL and 5,020 m RL was completed during the previous geological exploration period. These levels are accessed through adits. The planned development at 5,060 m RL involves the construction of a decline from 5,020 m RL. The same approach applies to the 4,980 m RL, which is also connected to the 5,020 m RL by a decline.

Development system

The 22 CINF Mining Design conducted trade-off studies for the development system, focusing on two primary options. The main difference between these options lies in the transportation method for materials above the 5,100 m RL.

Option I involved transporting materials through individual level adits, whereas Option II proposed constructing an ore pass to transfer materials to the 5,100 m RL, subsequently using the adit at this level to haul materials out from underground.

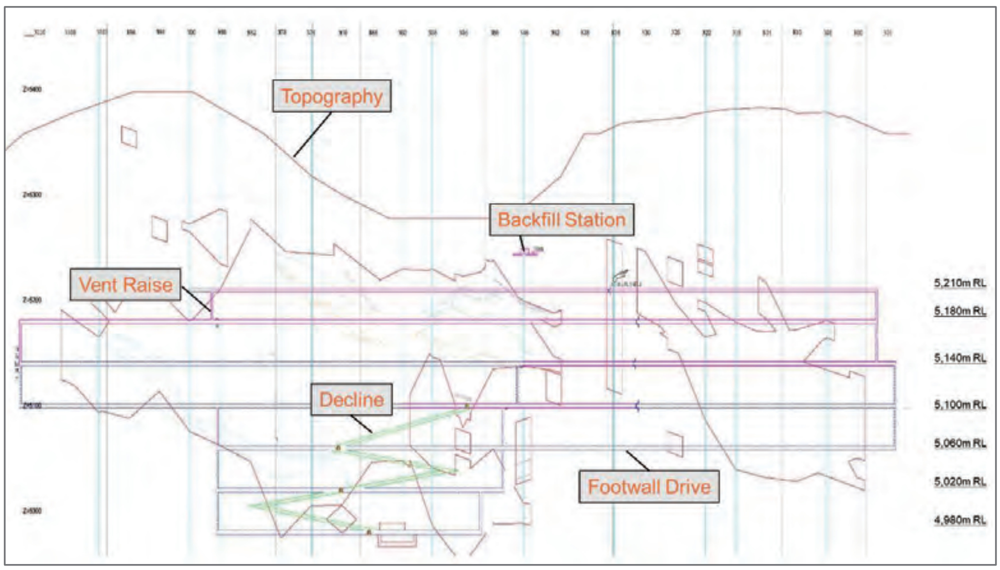
After evaluating capital expenditure, time consumption, and existing surface infrastructure, the 22 CINF Mining Design concluded that Option I was preferable due to its lower capital expenditure and shorter construction time.

Based on Option I, 22 CINF Mining Design proposed two stages for mine exploitation, each with a production capacity of 1,000 t/d.

Overall, the selection of the development strategy is considered reasonable from SRK's perspective.

Figure 8.21 illustrates the sectional view of the development system at the Mengya'a mine.

Figure 8.21: Schematic of development system



Source: 22 CINF Mining Design and modified by SRK

Mine services

The mine service infrastructure is designed to ensure operational reliability, safety, and efficiency across all mining activities. Systems are tailored to high-altitude conditions (4,980–5,210 m RL) and phased production requirements, incorporating redundancy and modularity for long-term adaptability.

The mine services are primarily based on the 24 XERI Mining Design. The following is a discussion of the key aspects.

Ventilation

The ventilation design employs a central intake with a two-wing return configuration.

During Stage I, fresh air is introduced through various adit entrances to the working face. The exhaust air is directed through ventilation/backfill raises within the stopes to the upper-level return air drive and then expelled to the surface via the eastern and western blind return air raises between levels, ultimately exiting through the 5,210 m RL return air adit.

In Stage II, fresh air enters through the 5,100 m RL adit and decline, as well as the 5,020 m RL adit, reaching each individual level and stope. The exhaust air follows the same path as in Stage I.

A K40-6-№21 counter-rotating axial flow fan is installed in the fan room at the 5,210 m RL adit entrance. This fan is equipped with a 200 kW electric motor operating at a voltage of 380 kV. It provides an air pressure range of 339–1,563 Pa and an airflow rate of 53.3–116.1 m³/s.

Dewatering

The estimated underground water inflow is as follows:

- Stage I: At the 5,100 m RL, the normal inflow rate is estimated at 3,100 m³/d, with a maximum inflow rate of 4,700 m³/d, and production wastewater amounts to 747 m³/d.
- Stage II: At the 4,980 m level, the normal inflow rate is estimated at 4,700 m³/d, with a maximum inflow rate of 7,000 m³/d.

The philosophy of dewatering is as follows:

- **Stage I:** Dewatering is managed through gravity flow. The underground water inflow from various levels naturally flows through boreholes and cable raises to the 5,100 m RL adit. It then continues to flow to a sedimentation pond near the adit entrance, where it is treated for use in other facilities.
- **Stage II:** The 5,020 m RL serves as the boundary for drainage. Underground water inflows at and above this level, is directed through drainage holes and boreholes to the 5,020 m RL adit. From there, it flows naturally to a sedimentation pond near the adit. As for the levels below 5,020 m RL, a drainage pump station is located near the decline at the 4,980 m RL. Water is pumped to the 5,020 m adit and then flows to the adit entrance. The 4,980 m pump station is equipped with three D300-65×3 pumps, each with a flow rate of 300 m³/h and a head of 195 m, powered by 250 kW motors. Under normal inflow conditions, one pump operates, while another is on standby, and the third is reserved for maintenance. During maximum inflow conditions, two pumps operate simultaneously.

Compressed air

Based on calculations, the mine's maximum compressed air requirement per shift is 48 m³/min. Considering relevant factors, the maximum air consumption is 90.6 m³/min. A compressor station is designed to be located near the 5,100 m RL adit entrance, equipped with three LGFD-315F-7bar screw compressors. These compressors are air cooled, with two in operation and one on standby. Each compressor has an air discharge capacity of 60 m³/min at a pressure of 0.8 MPa and is powered by a 315 kW motor with a supply voltage of 10 kV.

Water supply

The underground production water requirement is 1,000 t/d. Water for production is sourced from adit and mine seepage. The collected water is first treated in a sedimentation pond before being pumped to a high-level surface reservoir. From the reservoir, water is redistributed through pipelines to the entrances of each adit level. It is then further distributed to the underground working faces via the level adits, supporting both mining production and fire suppression activities.

Backfill

The backfill system employs a mineralised waste cemented backfill process, with mineralised waste sourced from the Pb14 OP. The mineralised waste is crushed to a particle size of less than 10 mm and then transported by truck to the 5,180 m RL aggregate stockpile. The slurry is conveyed through boreholes and pipelines to the underground voids. The backfill station is designed to operate for 150 days per annum, with three shifts per day, each shift lasting 16 hours. Calculations indicate a design slurry flow rate of 28.59 m^3/h, with a daily slurry consumption of 457.39 m^3 at a concentration of 74%. The backfill system uses gravity flow where applicable.

Power supply

All power supply systems are designed to meet high-altitude requirements. The installed capacity of the equipment is 2,834.4 kW, with an operational capacity of 2,357.4 kW. The maximum annual electricity consumption is 6.55 million kWh.

Other infrastructure

The Mengya'a mine is equipped with existing facilities such as a fuelling station, explosives storage, and administration facilities. The mine is accessible via two main routes, one to the south and one to the north, providing generally convenient transportation conditions and adequate external support.

During mining operations, a workshop for the maintenance and repair as well as a materials warehouse, will be established at the 5,100 m RL.

8.4.3 Mining equipment

The primary equipment is mainly categorised into the following groups: development, production, lashing, and hauling.

Table 8.20 summarises the equipment selection results, including the type, model, and quantity requirements.

Table 8.20: Mining equipment configuration

Category	Type	Model	Quantity
Development	Development Drill Rig	DD2710	2
	Loader	ACY204 - 2 m^3	4
	Mine Truck	12t	4
Production	Jack Hammer	YT-28	4
	Long Hole Drill Rig	DL2710	2
Lashing	Loader	ACY204 - 2 m^3	4
Hauling	Mine Truck	12t	4

Source: 22 CINF Mining Design and 24 XERI Mining Design

8.4.4 Mine logistics

In Stage I, which is above the 5,100 m RL, all equipment, materials, and personnel will enter and exit through each level adit. Mine trucks will transport the mineable materials to a surface stockpile located near the adit entrance. Subsequently, the materials will be transferred to larger trucks for transport to the processing plant, which is 11 km away. There will be no coarse crushing machinery underground, and waste materials will be transported to the surface during the construction period, subsequently being used as backfill in mined-out stopes during the production period.

In Stage II, which spans from 4,980 m RL to 5,100 m RL, the mineable materials will be transported through each individual footwall drive, then to the decline, and finally through the 5,100 m RL to the surface stockpile. From there, they will be transferred to larger trucks for transport to the processing plant. As in Stage I, waste materials will be used as backfill in mined-out stopes.

8.4.5 Production schedule

SRK re-evaluated the production schedule. The productivity and scheduling philosophy applied to the mine schedule is summarised as follows:

- Mine area: The mine is divided into distinct areas based on the location of the veins.
- Mining unit: Along the strike direction, the mine is segmented into units every 40 m.
- Level RL: Stopes are defined at various level RLs according to the section of mine design.
- Material type: Stopes are classified as either oxide stopes or fresh stopes based on geological definitions.
- Operational stages: According to the 22 CINF Mining Design, mining operations are organised into two stages, and from upper levels to lower levels.
- Primary/secondary stopes: Primary stopes will be mined first, followed by secondary stopes after the backfill has cured. The plan is to mine the secondary stope until 2027, necessitating the commissioning of the backfill station after 2027.
- Mining capacity: The initial production capacity is set at 100 kt/a. Huaxia Mining planned to gradually ramp up the production capacity as the Pb14 OP mining is ceased. The planned operation hours of the mining fleet would be expanded and/or the backup equipment would be used, which is capable of achieving the mining capacity of 400 kt/a. From 2031 to 2033, the Pb12 UG will undergo a production ramp-up to achieve 400 kt/a, continuing until the end of the mine's life.

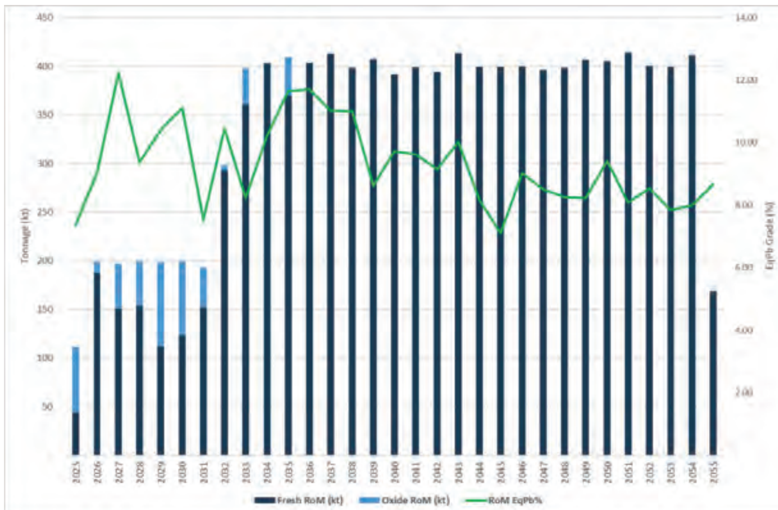
Table 8.21 and Figure 8.22 indicate the production schedule over the LOM for Pb12 UG.

Table 8.21: Production schedule over the LOM – Pb12 UG

Year	ROM tonnes (t)	ROM Pb%	ROM Zn%	ROM Cu%	ROM Ag (g/t)	ROM EqPb%	ROM Pb+Zn%
2025	**110,840**	**3.26**	**2.72**	**0.16**	**18.92**	**7.38**	**5.98**
2026	198,970	3.38	3.86	0.19	27.41	9.06	7.25
2027	197,116	5.46	4.56	0.24	32.19	12.23	10.02
2028	199,433	3.79	3.68	0.23	26.19	9.38	7.47
2029	198,271	4.56	3.90	0.22	28.71	10.40	8.46
2030	199,761	5.35	3.93	0.20	29.12	11.11	9.29
2031	193,288	3.52	2.70	0.14	21.83	7.54	6.23
2032	298,259	4.11	4.23	0.22	31.29	10.45	8.34
2033	397,531	2.97	3.32	0.20	32.32	8.22	6.29
2034	402,951	3.83	4.27	0.22	33.40	10.24	8.10
2035	409,435	4.49	4.69	0.25	38.70	11.65	9.18
2036	**403,223**	**4.69**	**4.71**	**0.24**	**33.70**	**11.70**	**9.40**
2037	413,232	4.29	4.33	0.24	39.90	11.04	8.62
2038	398,263	4.18	4.50	0.23	39.39	11.01	8.68
2039	407,049	2.53	4.13	0.18	33.39	8.62	6.66
2040	391,968	3.59	3.90	0.21	38.18	9.72	7.49
2041	398,783	2.81	4.52	0.21	37.80	9.63	7.33
2042	394,121	2.65	3.92	0.24	47.71	9.15	6.57
2043	413,536	3.08	4.47	0.23	43.04	10.02	7.55
2044	399,374	2.18	3.93	0.20	34.65	8.16	6.10
2045	398,960	1.74	3.72	0.16	24.67	7.10	5.46
2046	399,850	2.45	4.18	0.23	39.82	9.01	6.63
2047	395,993	2.30	4.17	0.20	31.23	8.50	6.47
2048	398,199	2.20	4.10	0.19	32.35	8.26	6.30
2049	406,708	2.03	4.48	0.17	25.89	8.23	6.51
2050	405,111	2.85	4.40	0.20	34.66	9.41	7.25
2051	414,000	2.46	3.69	0.19	32.68	8.08	6.14
2052	400,329	1.99	4.22	0.25	37.29	8.54	6.20
2053	399,013	1.44	4.27	0.21	33.26	7.84	5.71
2054	411,141	0.91	4.49	0.25	43.92	7.99	5.40
2055	168,196	0.94	4.51	0.30	61.10	8.67	5.45

Source: SRK

Figure 8.22: Production schedule over the LOM – Pb12 UG



Source: SRK, 2025

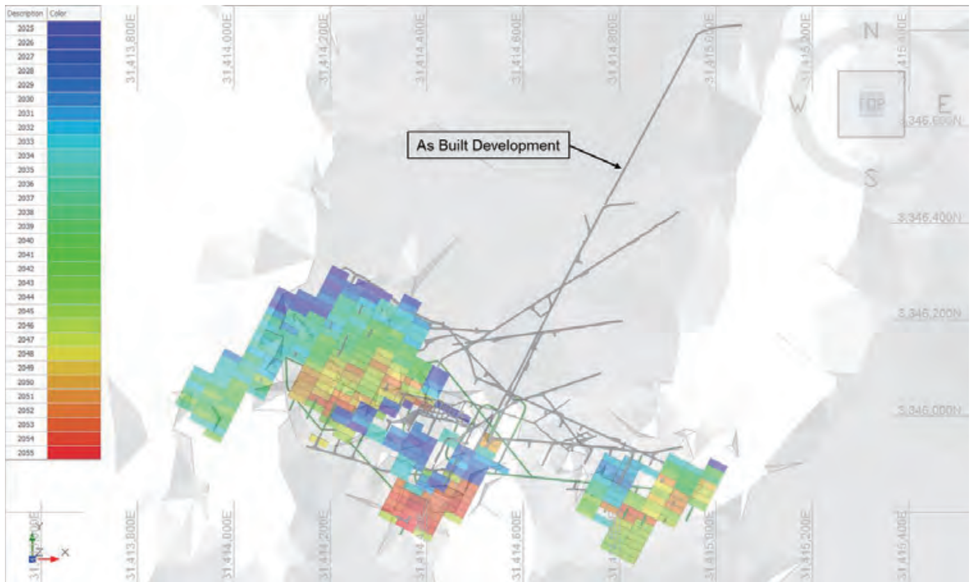
Notes:

[1] The line represents the average EqPb grade, corresponding to the right axis.

[2] The column represents the fresh and oxide ROM tonnes amount, corresponding to the left axis.

Figure 8.23 shows the stopes mined annually.

Figure 8.23: Stopes mined annually over the LOM – Pb12 UG



Source: SRK, 2025

8.5 Combined mine production plan

SRK combined the Pb14 OP and Pb12 UG production plans in Table 8.22 and Figure 8.24. At the current designed production capacity, the LOMs of Pb14 OP and Pb12 UG are estimated at 8 and 31 years, respectively.

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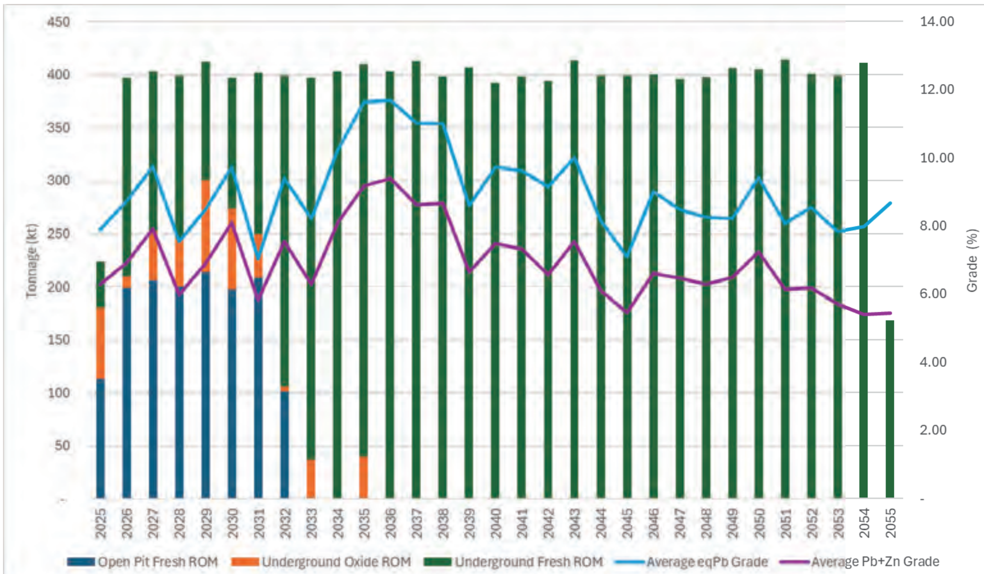
Table 8.22: Combined mine production schedule

	Unit	Year LOM	1 2025	2 2026	3 2027	4 2028	5 2029	6 2030	7 2031	8 2032	9 2033	10 2034	11 2035	12 2036	13 2037	14 2038	15 2039
Open Pit																	
Fresh																	
ROM Tonnes	kt	1,438	113	199	206	200	214	197	208	101							
Pb	%	0.69	1.11	1.11	1.00	0.30	0.35	0.63	0.70	0.85							
Zn	%	4.90	5.50	5.50	4.92	4.20	5.12	6.26	4.76	4.43							
Cu	%	0.10	0.19	0.19	0.12	0.12	0.11	0.09	0.09	0.08							
Ag	g/t	10.09	17.63	17.63	14.34	10.53	7.68	10.27	6.84	6.80							
EqPb	%	6.87	8.41	8.41	7.36	5.73	6.71	8.33	6.58	6.31							
Pb+Zn	%	5.59	6.61	6.61	5.92	4.51	5.46	6.89	5.47	5.27							
Waste	kt	8,534	1,640	3,744	2,029	384	329	248	90	70							
Total Material Movement	kt	9,972	1,753	3,942	2,235	584	543	445	299	170							
Underground																	
Oxide																	
ROM Tonnes	kt	458	67	11	47	46	87	76	42	5	37	-	40	-	-	-	-
Pb	%	5.01	3.89	4.37	6.15	4.85	4.76	6.03	5.65	2.59	3.97	-	5.15	-	-	-	-
Zn	%	2.79	3.09	3.20	2.53	3.35	2.60	2.56	2.48	2.80	2.13	-	3.58	-	-	-	-
Cu	%	0.19	0.19	0.18	0.13	0.21	0.17	0.21	0.20	0.11	0.19	-	0.25	-	-	-	-
Ag	g/t	27.31	23.07	23.64	24.53	23.12	25.90	36.25	35.93	13.29	21.88	-	27.49	-	-	-	-
EqPb	%	9.42	8.68	9.28	10.03	10.04	8.86	10.23	9.71	6.71	7.45	-	10.80	-	-	-	-
Pb+Zn	%	7.80	6.98	7.57	8.67	8.20	7.36	8.59	8.13	5.39	6.10	-	8.74	-	-	-	-
Fresh																	
ROM Tonnes	kt	10,165	44	168	150	154	112	123	152	293	360	403	370	403	413	398	407
Pb	%	2.90	2.29	3.32	5.25	3.47	4.41	4.94	2.94	4.14	2.86	3.83	4.42	4.69	4.29	4.18	2.53
Zn	%	4.21	2.15	3.90	5.19	3.78	4.90	4.79	2.77	4.25	3.45	4.27	4.81	4.71	4.33	4.50	4.13
Cu	%	0.21	0.10	0.19	0.28	0.23	0.26	0.20	0.12	0.22	0.20	0.22	0.25	0.24	0.24	0.23	0.18
Ag	g/t	35.35	12.53	27.84	34.57	27.11	30.89	24.89	17.98	31.63	33.40	33.40	39.91	33.70	39.90	39.39	33.39
EqPb	%	9.28	5.37	9.05	12.92	9.18	11.59	11.66	6.94	10.52	8.30	10.24	11.74	11.70	11.04	11.01	8.62
Pb+Zn	%	7.10	4.44	7.23	10.44	7.25	9.31	9.72	5.70	8.39	8.31	8.10	9.23	9.40	8.62	8.68	6.66
Subtotal Underground																	
ROM Tonnes	kt	10,623	111	199	197	199	198	200	193	298	398	403	409	403	413	398	407
Pb	%	2.99	3.26	3.39	5.46	3.79	4.56	5.35	3.52	4.11	2.97	3.83	4.49	4.69	4.29	4.18	2.53
Zn	%	4.14	2.72	3.86	4.56	3.68	3.90	3.93	2.70	4.23	3.32	4.27	4.69	4.71	4.33	4.50	4.13
Cu	%	0.21	0.16	0.19	0.24	0.23	0.22	0.20	0.14	0.22	0.20	0.22	0.25	0.24	0.24	0.23	0.18
Ag	g/t	35.00	18.92	27.41	32.19	26.19	28.71	29.12	21.83	31.29	32.32	33.40	38.70	33.70	39.90	39.39	33.39
EqPb	%	9.29	7.38	9.06	12.23	9.36	10.40	11.11	7.54	10.45	8.22	10.24	11.65	11.70	11.04	11.01	8.62
Pb+Zn	%	7.13	5.98	7.25	10.02	7.47	8.46	9.29	6.23	8.34	6.29	8.10	9.18	9.40	8.62	8.68	6.66
Total																	
Oxide																	
ROM Tonnes	kt	458	67	11	47	46	87	76	42	5	37	-	40	-	-	-	-
Pb	%	5.01	3.89	4.37	6.15	4.85	4.76	6.03	5.65	2.59	3.97	-	5.15	-	-	-	-
Zn	%	2.79	3.09	3.20	2.53	3.35	2.60	2.56	2.48	2.80	2.13	-	3.58	-	-	-	-
Cu	%	0.19	0.19	0.18	0.13	0.21	0.17	0.21	0.20	0.11	0.19	-	0.25	-	-	-	-
Ag	g/t	27.31	23.07	23.64	24.53	23.12	25.90	36.25	35.93	13.29	21.88	-	27.49	-	-	-	-
EqPb	%	9.42	8.68	9.28	10.03	10.04	8.86	10.23	9.71	6.71	7.45	-	10.80	-	-	-	-
Pb+Zn	%	7.80	6.98	7.57	8.67	8.20	7.36	8.59	8.13	5.39	6.10	-	8.74	-	-	-	-
Fresh																	
ROM Tonnes	kt	11,602	157	386	356	354	326	321	360	394	360	403	370	403	413	398	407
Pb	%	2.62	1.44	2.18	2.79	1.68	1.74	2.29	1.64	3.29	2.86	3.83	4.42	4.69	4.29	4.18	2.53
Zn	%	4.29	4.57	4.73	5.03	4.02	5.04	5.69	3.92	4.30	3.45	4.27	4.81	4.71	4.33	4.50	4.13
Cu	%	0.20	0.18	0.19	0.19	0.17	0.16	0.13	0.10	0.19	0.20	0.22	0.25	0.24	0.24	0.23	0.18
Ag	g/t	32.22	16.21	22.49	22.87	17.73	15.64	15.82	11.52	25.27	33.40	33.40	39.91	33.70	39.90	39.39	33.39
EqPb	%	8.98	7.56	8.72	9.71	7.23	8.38	9.81	6.73	9.44	8.30	10.24	11.74	11.70	11.04	11.01	8.62
Pb+Zn	%	6.92	6.01	6.91	7.82	5.70	6.78	7.98	5.57	7.59	6.31	8.10	9.23	9.40	8.62	8.68	6.66
Subtotal																	
ROM Tonnes	kt	12,061	224	397	403	399	412	397	402	399	398	403	409	403	413	398	407
Pb	%	2.71	2.17	2.25	3.18	2.04	2.37	3.01	2.06	3.29	2.97	3.83	4.49	4.69	4.29	4.18	2.53
Zn	%	4.23	4.13	4.88	4.74	3.94	4.53	5.09	3.77	4.28	3.32	4.27	4.69	4.71	4.33	4.50	4.13
Cu	%	0.20	0.17	0.19	0.18	0.17	0.16	0.15	0.11	0.18	0.20	0.22	0.25	0.24	0.24	0.23	0.18
Ag	g/t	32.03	18.27	22.52	23.06	18.35	17.79	19.76	14.06	25.11	32.32	33.40	38.70	33.70	39.90	39.39	33.39
EqPb	%	9.00	7.90	8.73	9.75	7.55	8.48	9.73	7.04	9.40	8.22	10.24	11.65	11.70	11.04	11.01	8.62
Pb+Zn	%	6.95	6.30	6.93	7.92	5.98	6.90	8.09	5.83	7.56	6.29	8.10	9.18	9.40	8.62	8.68	6.66

	Unit	16 2040	17 2041	18 2042	19 2043	20 2044	21 2045	22 2046	23 2047	24 2048	25 2049	26 2050	27 2051	28 2052	29 2053	30 2054	31 2055
Open Pit																	
Fresh																	
ROM Tonnes	kt																
Pb	%																
Zn	%																
Cu	%																
Ag	g/t																
EqPb	%																
Pb+Zn	%																
Waste	kt																
Total Material Movement	kt																
Underground																	
Oxide																	
ROM Tonnes	kt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pb	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zn	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
EqPb	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pb+Zn	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Fresh																	
ROM Tonnes	kt	392	399	394	414	399	399	400	396	398	407	405	414	400	399	411	168
Pb	%	3.59	2.81	2.65	3.08	2.18	1.74	2.45	2.30	2.20	2.03	2.85	2.46	1.99	1.44	0.91	0.94
Zn	%	3.90	4.52	3.92	4.47	3.93	3.72	4.16	4.17	4.10	4.48	4.40	3.69	4.22	4.27	4.49	4.51
Cu	%	0.21	0.21	0.24	0.23	0.20	0.16	0.23	0.20	0.19	0.17	0.20	0.19	0.25	0.21	0.25	0.30
Ag	g/t	38.18	37.80	47.71	43.04	34.65	24.67	39.82	31.23	32.35	25.89	34.66	32.68	37.29	33.26	43.92	61.10
EqPb	%	9.72	9.63	9.15	10.02	8.16	7.10	9.01	8.50	8.26	8.23	9.41	8.08	8.54	7.84	7.99	8.67
Pb+Zn	%	7.49	7.33	6.57	7.55	6.10	5.46	6.63	6.47	6.30	6.51	7.25	6.14	6.20	5.71	5.40	5.45
Subtotal Underground																	
ROM Tonnes	kt	392	399	394	414	399	399	400	396	398	407	405	414	400	399	411	168
Pb	%	3.59	2.81	2.65	3.08	2.18	1.74	2.45	2.30	2.20	2.03	2.85	2.46	1.99	1.44	0.91	0.94
Zn	%	3.90	4.52	3.92	4.47	3.93	3.72	4.16	4.17	4.10	4.48	4.40	3.69	4.22	4.27	4.49	4.51
Cu	%	0.21	0.21	0.24	0.23	0.20	0.16	0.23	0.20	0.19	0.17	0.20	0.19	0.25	0.21	0.25	0.30
Ag	g/t	38.18	37.80	47.71	43.04	34.65	24.67	39.82	31.23	32.35	25.89	34.66	32.68	37.29	33.26	43.92	61.10
EqPb	%	9.72	9.63	9.15	10.02	8.16	7.10	9.01	8.50	8.26	8.23	9.41	8.08	8.54	7.84	7.99	8.67
Pb+Zn	%	7.49	7.33	6.57	7.55	6.10	5.46	6.63	6.47	6.30	6.51	7.25	6.14	6.20	5.71	5.40	5.45
Total																	
Oxide																	
ROM Tonnes	kt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pb	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zn	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
EqPb	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pb+Zn	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Fresh																	
ROM Tonnes	kt	392	399	394	414	399	399	400	396	398	407	405	414	400	399	411	168
Pb	%	3.59	2.81	2.65	3.08	2.18	1.74	2.45	2.30	2.20	2.03	2.85	2.46	1.99	1.44	0.91	0.94
Zn	%	3.90	4.52	3.92	4.47	3.93	3.72	4.16	4.17	4.10	4.48	4.40	3.69	4.22	4.27	4.49	4.51
Cu	%	0.21	0.21	0.24	0.23	0.20	0.16	0.23	0.20	0.19	0.17	0.20	0.19	0.25	0.21	0.25	0.30
Ag	g/t	38.18	37.80	47.71	43.04	34.65	24.67	39.82	31.23	32.35	25.89	34.66	32.68	37.29	33.26	43.92	61.10
EqPb	%	9.72	9.63	9.15	10.02	8.16	7.10	9.01	8.50	8.26	8.23	9.41	8.08	8.54	7.84	7.99	8.67
Pb+Zn	%	7.49	7.33	6.57	7.55	6.10	5.46	6.63	6.47	6.30	6.51	7.25	6.14	6.20	5.71	5.40	5.45
Subtotal																	
ROM Tonnes	kt	392	399	394	414	399	399	400	396	398	407	405	414	400	399	411	168
Pb	%	3.59	2.81	2.65	3.08	2.18	1.74	2.45	2.30	2.20	2.03	2.85	2.46	1.99	1.44	0.91	0.94
Zn	%	3.90	4.52	3.92	4.47	3.93	3.72	4.16	4.17	4.10	4.48	4.40	3.69	4.22	4.27	4.49	4.51
Cu	%	0.21	0.21	0.24	0.23	0.20	0.16	0.23	0.20	0.19	0.17	0.20	0.19	0.25	0.21	0.25	0.30
Ag	g/t	38.18	37.80	47.71	43.04	34.65	24.67	39.82	31.23	32.35	25.89	34.66	32.68	37.29	33.26	43.92	61.10
EqPb	%	9.72	9.63	9.15	10.02	8.16	7.10	9.01	8.50	8.26	8.23	9.41	8.08	8.54	7.84	7.99	8.67
Pb+Zn	%	7.49	7.33	6.57	7.55	6.10	5.46	6.63	6.47	6.30	6.51	7.25	6.14	6.20	5.71	5.40	5.45

Source: SRK

Figure 8.24: Combined mine production schedule



Source: SRK

Notes:

[1] The line represents the average EqPb grade, corresponding to the right axis.

[2] The column represents the fresh and oxide ROM tonnes amount in Pb14 OP and Pb12 UG, corresponding to the left axis.

8.6 Conclusions and recommendations

SRK has prepared the Ore Reserve estimates for the Mengya'a mine, encompassing both open-pit and underground operations in accordance with the JORC Code (2012). These estimates are supported by technical studies conducted in 2013, 2022, and 2024, as well as operational data. The selected mining methods are consistent with industry standards, technically sound, and have proven their viability through actual production performance.

For the Pb14 OP, SRK recommends revising the pit optimisation and mine plan prior to closure to account for evolving market conditions, such as changes in metal prices and plant efficiency. This review should include an assessment of potential pit expansion opportunities (pushbacks) using updated operational and economic data to extend mine life and maximise resource recovery.

For the Pb12 UG operations, SRK advises ongoing technical reviews during pilot and operational phases to optimise stope blasting designs for improved safety and efficiency, ventilation systems to address high-altitude challenges like reduced oxygen levels, and equipment performance to mitigate derating impacts and temperature extremes. Additionally, production capacity increases should be evaluated by analysing geological continuity, infrastructure scalability (haulage and power), and economic feasibility under current market dynamics. These measures will help mitigate high-altitude operational risks while enhancing efficiency and ensuring long-term resource utilisation.

9 Ore Reserve estimation

9.1 Introduction

The definition of Ore Reserves in accordance with the JORC Code (2012) is as follows:

An 'Ore Reserve' is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

9.2 Ore Reserve estimation procedures

SRK conducted a review of technical studies and operational records for the Mengya'a lead-zinc mine, which included mine planning input parameters and Modifying Factors provided by Huaxia Mining. Using the latest Mineral Resource estimate (Chapter 7) and the associated block model prepared by SRK, along with the verified mining Modifying Factors, SRK modelled the mine design and production schedule according to the strategy proposed by the technical studies. This was completed to report an Ore Reserve in compliance with the guidelines of the JORC Code (2012).

The Ore Reserve estimation involved the following steps:

- Process Mineral Resources Models (MRMs) to meet the requirements for Ore Reserve estimates.
- Conduct site inspections.
- Review the previous studies and designs for the project.
- Define the ore/ waste cut-off.
- Assess, modify and apply mining factors to the Mineral Resource estimate.
- Consider Modifying Factors from the other disciplines (relying on other experts).
- Review technical economic analysis for the project.
- Undertake an internal review of the estimation process/results.
- Prepare the Ore Reserve Statement.

9.3 Technical studies

SRK's mining review is primarily based on 13 GDMADI FS, 22 CINF Mining Design and 24 XERI Mining Design, as follows:

- The Mengya'a mine consists of two main deposits: Pb12 and Pb14.
- Pb14 is designed and operated as an open-pit mine with a ROM mining capacity of 400 kt/a.
- Pb12 is designed as an underground mine with a mining capacity of 200 kt/a.

For the Pb14 deposit, the level of accuracy of the Modifying Factors described in the 13 GDMADI FS and ongoing production, and for the Pb12 deposit, the level of accuracy of the Modifying Factors described in the 22 CINF Mining Design, the 24 XERI Mining Design, and the existing underground infrastructure, are considered by SRK to be equivalent to a pre-feasibility level study (PFS), prepared in accordance with the guidelines of the JORC Code (2012).

To summarise, Ore Reserve estimates for:

- The Pb14 deposit (Pb14 OP) evaluation would be conducted based on the 13 GDMADI FS and ongoing open-pit production.
- The Pb12 deposit (Pb12 UG) evaluation would be conducted primarily based on the 22 CINF Mining Design, with reference to the 24 XERI Mining Design and existing underground infrastructure.

Based on the reviewed results of the technical studies and production records, SRK conducted mine design and production schedule modelling against the updated MRM constructed by SRK.

9.4 Cut-off grade

The Mineral Resources contain lead, zinc, copper and silver elements. The lead equivalent (EqPb) grade was used as a cut-off grade to define 'ore'.

The following formula is applied to estimate the cut-off grade of EqPb for the feed ore from underground extraction:

$$A = \frac{Cm + Cp + Cg}{(P - C_r) * P_r * (1 - R_t)}$$

Parameters that are applied for estimates of the cut-off grade are presented in Table 9.1. The preferred cut-off grade is the estimated value which is round up to the nearest 0.1. SRK is of the view that material above the cut-off grade could be defined as economically extractable under those specific conditions.

Table 9.1: Parameters applied to cut-off grade

Item	Unit	Pb14 OP	Pb12 UG		Description
		Fresh	Oxide	Fresh	
Preferred cut-off grade	%	1.8	6.0	3.7	Round up to nearest 0.1
A	%	1.83	5.98	3.74	Estimated feed cut-off grade of PbEq
Cm	%		229.5	229.5	Mining cash cost
Cp	RMB/t feed	154.2	154.2	154.2	Processing cash cost
Cg	RMB/t feed	66.8	66.8	66.8	Total General & Administration cash cost
P	RMB/t feed	14,300	14,300	14,300	Forecast 50% Pb concentrate price excluding VAT
Cr	RMB/t metal	250.0	250.0	250.0	Grade premium applied due to higher Pb concentrate (≥60% Pb)
Pr	RMB/t metal	88.0	55.0	88.0	Processing recovery for lead in concentrate
R_t	%	5.9	5.9	5.9	Resource tax rate to revenue

Sources: 22 CINF Mining Design, 24 XERI Mining Design, Huaxia Mining, SMM and SRK

The processing plant treats two types of ROM ore: oxide and fresh and yields three marketable concentrates: lead concentrate, zinc concentrate and copper concentrate. Silver is primarily recovered in the lead concentrate, achieving payable grades in both lead and copper concentrates. However, silver grades in the zinc concentrate remain sub-economic and do not meet payable thresholds.

The payability of elements within each concentrate is determined by their grades and contractual terms between Huaxia Mining and its customers, supplemented by pricing benchmarks from SMM. Payability thresholds for silver, lead, and zinc are derived from these agreements, with specific criteria varying by element and concentrate type (Table 9.2).

Table 9.2: Assumptions for payable factors

Item	Unit	Ag in lead concentrate	Pb in lead concentrate	Zn in zinc concentrate	Ag in copper concentrate	Cu in copper concentrate
Long-term forecast price [2], contained metal in concentrates.	RMB/g for Ag, RMB/t for Cu, Pb and Zn	6.2	14,300	16,200	6.233	62,700
Average element grade in concentrate from oxide ore feed	g/t for Ag, % for Cu, Pb and Zn	276	49.1	44.7	593	15.8
Processing recovery rates – oxide	%	57%	55%	66%	10%	35%
Payable factors, net received against LTP	% Factor	0%	NA	NA	76%	NA
Equivalent factors against Pb – oxide	Factor	-[1]	1.0000	1.3315	0.0062	2.7501
Average element grade in concentrate from fresh ore feed	g/t for Ag, % for Cu, Pb and Zn	821	60.0	45.0	701	19.9
Processing recovery rates – fresh	%	66%	88%	91%	9%	52%
Payable factors, net received against LTP	% Factor	81%	1.02	NA	76%	NA
Equivalent factors against Pb – fresh	Factor	0.0262	1.0000	1.1457	0.0034	1.1457

Sources: Huaxia Mining; SMM; SRK

Notes:

[1] The Ag content is approximately 276 g/t according to production performance in 2024, which is below the threshold for payment (300 g/t). Consequently, the payable factor for silver when processing oxide ROM ore is zero.

[2] LTP – long-term price. The long-term price (real) is the metal in standard concentrate price, not the metal price.

The equivalent factor of Zn, Cu and Ag against Pb is estimated is based on prices, recovery rates, and payable factors differences. The estimates are presented in Table 9.3.

Table 9.3: Estimates for equivalent factors of Zn, Cu and Ag to Pb

Material		Ag	Cu	Zn	Pb
Oxide	Equivalent factors against Pb	0.0062	2.7501	1.3315	1.0000
Fresh	Equivalent factors against Pb	0.0296	2.5464	1.1457	1.0000

Sources: Huaxia Mining; SMM; SRK

9.5 Mining Modifying Factors

Typically, the in situ Mineral Resources of Measured and Indicated categories are converted to ROM by applying Modifying Factors. The principal factors applied include mining loss and dilution. Other considerations include the quality of the resource, as well as environmental, legal, or political constraints, and any other factors that could affect the proportion of the in situ resource that will eventually be sold.

9.5.1 Pb14 OP

The following Modifying Factors are used to determine the mining inventory:

- Optimisation: This includes the most economic pit shell relative to the vein domains; only Mineral Resources in the Measured and Indicated categories are used in the optimisation.
- Pit design: The conversion between the optimal pit shell and the practical mine design.
- Dilution: Mining dilution was estimated at 5% by 13 GDMADI FS and stated by operation data.
- Mining recovery: A 95% mining recovery rate was applied by 13 GDMADI FS, as well as the operation data.

9.5.2 Pb12 UG

The following mining factors are applied to the Ore Reserve estimates:

- Mining Design Scope:
  - Only Measured and Indicated Mineral Resources are considered for the estimation.
  - Deswik.SO was used to optimise the mineable shapes and the refinement of these shapes into stopes was conducted as part of the stoping scope.
- Design Loss:
  - Stopes with grades below the cut-off grade will be filtered out as design loss.
  - Resources that are isolated and located far from the main design are difficult to reach.
- Mining Dilution:
  - A dilution skin of 0.5 m on the hanging wall or back, and a dilution skin of 0.2 m on the footwall or floor, were considered to simulate the overbreak during mining. This contributes approximately 6%, which is less than the 22 CINF Mining Design at 12%.

- Mining Loss:
  - A loss rate of 8% is applied to account for the loss incurred during the transportation from the workface handover to the ROM bin, which is consistent with 22 CINF Mining Design. An additional 4% of the planned loss was considered for room and pillar mining, which includes the loss attributed by the pillars.

9.6 Ore Reserve Estimates

9.6.1 Pb14 OP

The estimation process details are provided in Table 9.4. The corresponding waterfall charts are shown in Figure 9.1 and Figure 9.2.

Approximately 76% of tonnes and 72% of metal have been converted from Mineral Resources to Ore Reserves. Key negative factors affecting this conversion include the exclusion of Inferred Mineral Resources.

Table 9.4: Estimate process summary[1] – Pb14 OP

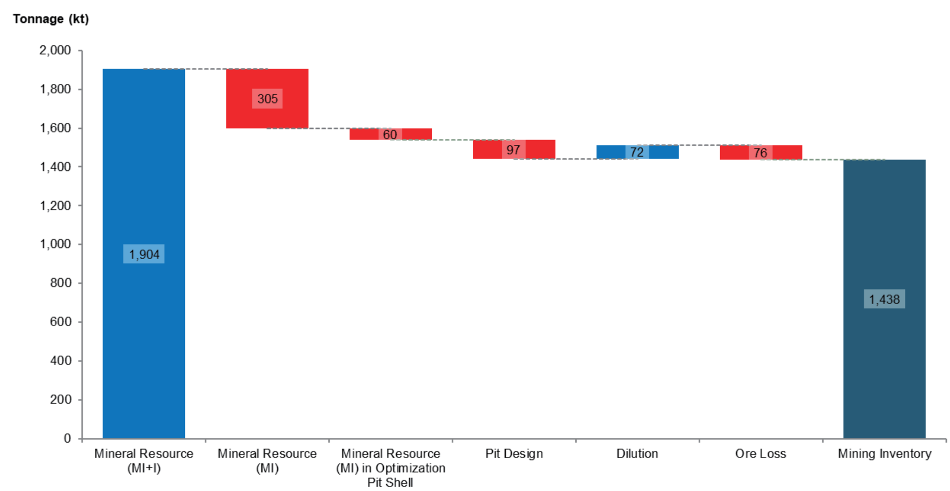
Description	Material Tonnes (kt)	Contained EqPb Metal (kt)
Mineral Resource (Measured, Indicated and Inferred)	1,904	138
Mineral Resource (Measured and Indicated)	305	24
Mineral Resource (Measured and Indicated) in Optimisation Pit Shell	60	2
Pit Design	97	13
Dilution	72	5
Ore Loss	76	5
Mining Inventory as at 31 July 2025	1,438	99

Source: SRK

Notes:

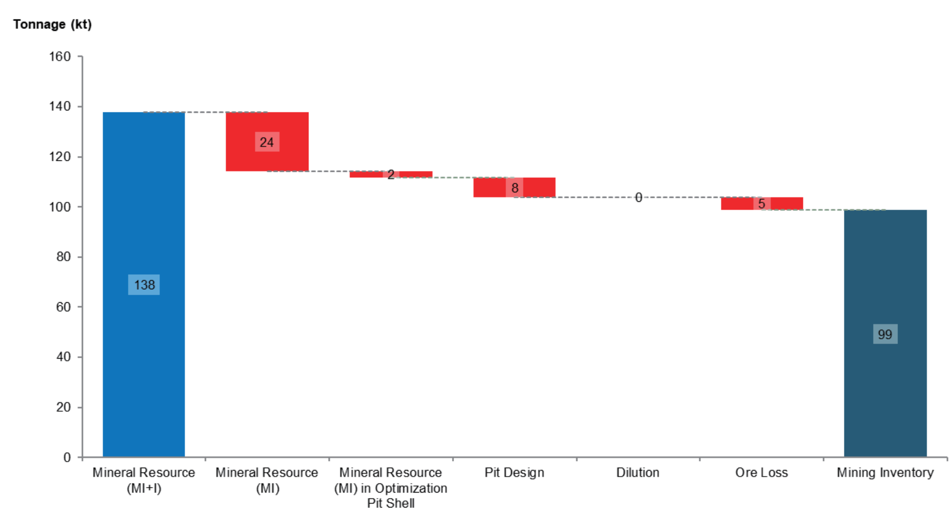
[1] Any differences between totals and sum of components are due to rounding.

Figure 9.1: Estimation process – change in material tonnes – Pb14 OP



Source: SRK

Figure 9.2: Estimation process – change in contained EqPb metal – Pb14 OP



Source: SRK
Note: MI+I = Measured. Indicated and Inferred Mineral Resources, MI = Measured and Indicated Mineral Resources

9.6.2 Pb12 UG

The estimated Mining Inventory based on Mineral Resource estimates and Modifying Factors applied to the material tonnes and contained EqPb metal is summarised in Table 9.5. The corresponding waterfall charts are shown in Figure 9.3 and Figure 9.4.

Table 9.5: Estimation process summary[1] – Pb12 UG

Description	Material Tonnes (kt)	Contained Metal (kt)	EqPb
Mineral Resource (Measured, Indicated and Inferred)	14,909	1,609	
Mineral Resource (Measured and Indicated) in Design Scope	12,326	1,371	
Mineral Resource (Measured and Indicated) within the SO[1] Results	8,943	1,072	
Dilution	2,644	12	
Ore Loss	964	98	
Mining Inventory as at 31 July 2025	10,623	986	

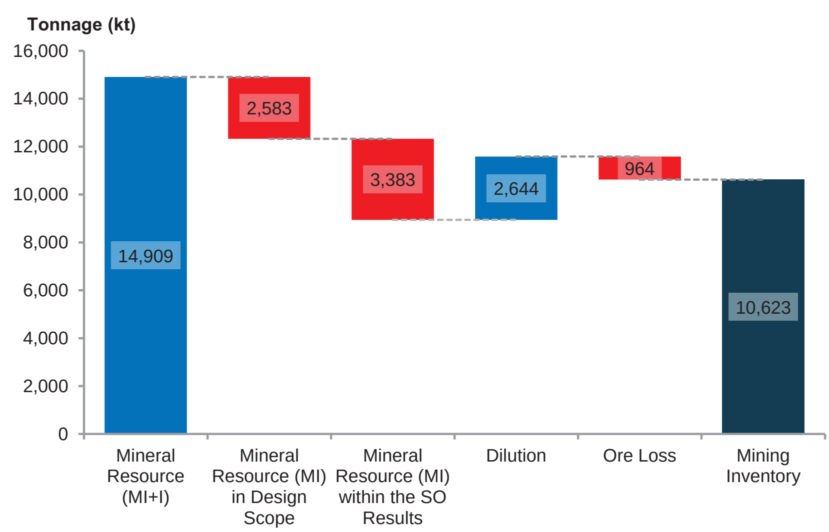
Source: SRK, 2025

Notes:

[1] Any differences between totals and sum of components are due to rounding.

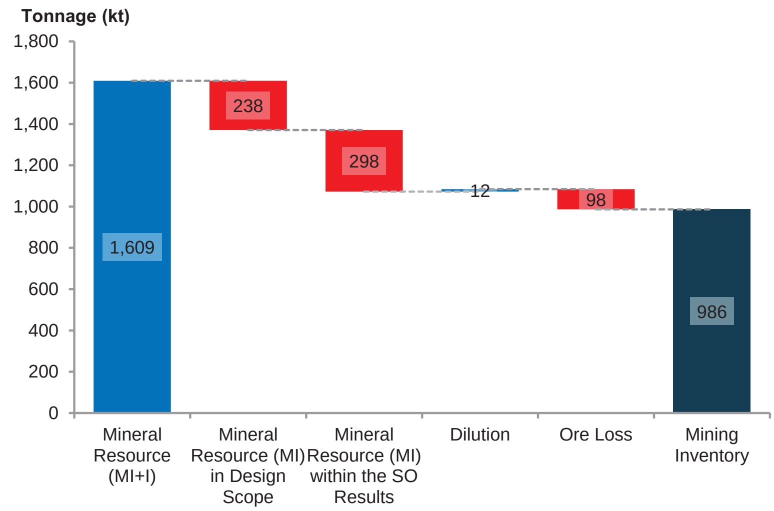
[2] SO: Stope Optimisation

Figure 9.3: Estimation process – change in material tonnes – Pb12 UG



Source: SRK, 2025

Figure 9.4: Estimation process – change in contained EqPb metal – Pb12 UG



Source: SRK, 2025

9.7 Ore Reserve Statement

SRK has estimated the Ore Reserves for the Mengya'a mine in accordance with the JORC Code (2012). These Ore Reserve estimates are based on the technical studies and ongoing operational records, which are considered to be equivalent to the level of a PFS.

The Mengya'a mine comprises two distinct mining operations: Pb14 OP and Pb12 UG. The economically mineable portions of the Measured and Indicated Mineral Resources within the designed stopes or the designed open-pit, inclusive of diluting materials and allowances for losses, have been classified as Proved and Probable Ore Reserves, respectively. The estimation of feed ore is determined with the reference point at the stockpile at the crusher feed.

As at 31 July 2025, Pb14 OP contains total Ore Reserves of 1,438 kt at grades of 0.69% Pb, 4.90% Zn, 0.10% Cu and 10.09 g/t Ag. (Table 9.6). As there are no oxide Mineral Resources left, all stated Ore Reserves are fresh material.

As at 31 July 2025, Pb12 UG contains total Ore Reserves of 10,623 kt at average grades of 2.99% Pb, 4.14% Zn, 0.21% Cu and 35.00 g/t Ag (Table 9.7).

Table 9.6: Ore Reserves Statement for Pb14 OP at Mengya'a mine as at 31 July 2025

Category	Ore Reserve (kt)	Pb Grade (%)	Zn Grade (%)	Cu Grade (%)	Ag Grade (g/t)	Pb Metal (kt)	Zn Metal (kt)	Cu Metal (kt)	Ag Metal (t)
Proved	400	1.14	4.54	0.10	14.51	4.57	18.16	0.41	5.80
Probable	1,038	0.52	5.04	0.10	8.39	5.41	52.27	1.07	8.71
Total	**1,438**	**0.69**	**4.90**	**0.10**	**10.09**	**9.99**	**70.43**	**1.48**	**14.51**

Source: SRK, 2025

Notes:

[1] The cut-off grades used to distinguish ore from waste are set at EqPb ≥1.8%.
[2] The mining dilution rate is 5% and the ore loss is 5%.
[3] The Ore Reserves are reported on a dry metric tonne basis.
[4] The reference point for reporting of Ore Reserves is the stockpile at the crusher feed.
[5] The Mineral Resources are effective as at 31 December 2024.

Table 9.7: Ore Reserves Statement for Pb12 UG at Mengya'a mine as at 31 July 2025

Type	Category	Ore Reserve (kt)	Pb Grade (%)	Zn Grade (%)	Cu Grade (%)	Ag Grade (g/t)	Pb Metal (kt)	Zn Metal (kt)	Cu Metal (kt)	Ag Metal (t)
Oxide	Proved	32	5.69	3.34	0.31	33.94	1.80	1.06	0.10	1.07
	Probable	427	4.96	2.74	0.18	26.82	21.17	11.71	0.78	11.45
	Subtotal	458	5.01	2.79	0.19	27.31	22.97	12.77	0.88	12.52
Fresh	Proved	4,728	2.91	5.12	0.26	44.30	137.65	241.90	12.34	209.47
	Probable	5,436	2.88	3.41	0.17	27.56	156.82	185.58	9.38	149.82
	Subtotal	10,165	2.90	4.21	0.21	35.35	294.47	427.48	21.72	359.29
Total	Proved	4,760	2.93	5.10	0.26	44.23	139.44	242.95	12.44	210.55
	Probable	5,863	3.04	3.37	0.17	27.51	177.99	197.30	10.16	161.27
	Total	**10,623**	**2.99**	**4.14**	**0.21**	**35.00**	**317.43**	**440.25**	**22.60**	**371.81**

Source: SRK, 2025

Notes:

[1] The cut-off grades used to distinguish ore from waste are set at EqPb ≥6.0% for oxide and ≥3.7% for fresh material.
[2] The Ore Reserves are reported on a dry metric tonne basis.
[3] The reference point for reporting of Ore Reserves is the stockpile at the crusher feed.
[4] The Mineral Resources are effective as at 31 December 2024.

Competent Person's Statement: The information in this Report that relates to Ore Reserve is based on information compiled by Mr Falong Hu who is a Fellow of the Australasian Institute of Mining and Metallurgy (AusIMM). Mr Hu is a full-time employee of SRK Consulting (China) Limited and has sufficient experience that is relevant to the style of mineralisation, type of deposit under consideration and to the activity which he undertakes to qualify as a Competent Person as defined in the 2012 edition of the *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* (the JORC Code).

10 Mineral processing

In March 2007, the Beijing General Research Institute of Mining and Metallurgy (BGRIMM) completed a process mineralogy study and processing testing on a composite sample of sulfide ore from the cores of 23 drill holes from Pb14. The results of this test served as the technical foundation for the feasibility study conducted by the Lanzhou Nonferrous Metallurgy Design Institute in March 2007 and the engineering design in March 2011.

The processing plant was established in October 2007 and put into operation in 2010, with a processing capacity of 2000 t/d. The plant uses a Cu-Pb mixed-separation flotation and Zn flotation process to produce Cu, Pb and Zn concentrates. During production, variations in ore oxidation levels and Cu-Pb grades, together with the impact of recycled water, led to instability in processing indicators, sometimes making it difficult to produce qualified Cu concentrates. Consequently, several specialised tests were conducted to optimise the process, reagent regimes, and operational conditions. Based on these test results, the concentrator underwent multiple process and operational optimisations to stabilise the processing indicators and lay the technical foundation for future upgrades and expansions.

Historical processing tests include:

- BGRIMM, Small-Scale Processing Test Report, March 2007
- Xinjiang Nonferrous Metals Research Institute (Xinjiang NMRI), Recycled Water Test Report, November 2015
- Guangdong Institute of Comprehensive Utilization of Resources (Guangdong RCUI), Flotation Process Test Report, May 2017
- Guangdong RCUI, Processing Test Report on Improving Recovery of High-Oxidation Lead-Zinc Ore, July 2018
- Kopper Chemical Industry Co., Ltd (Kopper Chem), Mineral Processing Test Report, November 2021
- Guangdong RCUI, Technological Processing Test Report, August 2019.

10.1 Processing test on sulfide ore – BGRIMM 2007

10.1.1 Test sample

The composite sample was derived from the cores of 23 drill holes from Pb14, with the combinations as shown in Table 10.1. The measured grades of the composite sample are 0.11% Cu, 0.95% Pb, 3.95% Zn, and 20.39 g/t Ag. Huaxia Mining considers this sample to be representative and, based on the spatial distribution of the core samples within the orebody, SRK concurs with Huaxia Mining's assessment.

Table 10.1: Assembly of composite sample for BGRIMM 2007 test

Drill hole ID	Weight (kg)	Grade (%)		
		Cu	Pb	Zn
ZK005, ZK046	13	0.13	3.19	6.99
ZK05K, ZK201, ZK102	30	0.09	0.79	3.00
ZK203	40	0.14	0.05	6.86
ZK284	20	0.023	0.081	1.54
ZK046	37	0.11	0.73	3.36
ZK103	30	0.08	0.19	6.52
ZK361	31	0.096	0.043	1.20
ZK005	26	0.096	2.38	2.88
ZK003	17	0.027	3.29	2.15
ZK285	11	0.02	0.043	8.78
ZK047	24	0.14	0.031	10.76
ZK006	11	0.15	0.074	3.21
ZK043	10	0.34	6.94	5.26
ZK051	21	0.10	0.54	2.40
ZK032	4	0.082	1.58	2.33
ZK102	27	0.07	1.45	1.96
ZK045	26	0.042	0.83	1.18
ZK105	11	0.058	0.069	7.77
ZK281	30	0.10	0.062	5.74
ZK072	16	0.16	2.63	2.00
ZK104	10	0.16	0.088	4.12
ZK071	10	0.074	2.38	1.39
ZK202	16	0.17	0.049	5.79
Calculated grade	471	0.10	0.94	4.20
Assay grade		0.11	0.95	3.95

Source: BGRIMM, Small-Scale Processing Test Report, March 2007

10.1.2 Process mineralogy

Chemical composition and mineral composition

The chemical composition and mineral composition of the composite sample are detailed in Table 10.2 and Table 10.3, respectively. The results of the mineral phase analysis for Zn and Pb are shown in Table 10.4.

Zinc, lead, copper, silver, and sulfur are the target economic elements. The contents of precious metal gold and the harmful element arsenic are both very low, which is not expected to result in a premium or discount for the final product.

The predominant mineral of zinc is sphalerite, with zinc in zinc sulfide accounting for 93.2% of the total zinc content, indicating a low oxidation rate and classifying it as a sulfide ore. Sphalerite has an average zinc content of 53.48% and iron content of 12.19%. It is essentially christophite $Cu_5(PO_4)_2(OH)_4$.

The predominant mineral of lead is galena, with trace amounts of jamesonite and cerussite. Lead in lead sulfide accounts for 91.8% of the total lead content, also indicating a low oxidation rate and classifying it as a sulfide ore.

The primary mineral for copper is chalcopyrite, with occasional occurrences of chalcocite.

The main mineral of sulfur is pyrrhotite, which is the most abundant sulfide mineral in the ore, followed by pyrite, with minor amounts of marcasite and colloidal pyrite.

The target minerals for processing recovery are zinc minerals, lead minerals, and copper minerals, primarily including sphalerite, galena, chalcopyrite, and chalcocite.

Table 10.2: Multi-element chemical analysis results

Component	**Cu**	**Fb**	**Zn**	**S**	**Fe**	**Au**[1]	**Ag**[1]	**As**
Content (%)	0.11	0.95	3.95	8.11	15.31	0.04	20.39	0.0036
Component	MgO	CaO	SiO_2	Al_2O_3	K_2O	Na_2O	P	
Content (%)	1.92	12.79	32.68	12.66	0.10	0.026	0.043	

Source: BGRIMM, Small-Scale Processing Test Report, March 2007.
Note: [1] The unit of measurement for Au and Ag content is g/t.

Table 10.3: Mineral composition analysis results

Metallic Minerals	**Content (%)**	**Non-metallic Minerals**	**Content (%)**
Sphalerite (ZnFe)	6.97	Quartz	24.77
Galena (PbS)	1.04	Calcite	25.7
Stibnite (Sb_2S_3)	Micro	Dolomite	
Cerussite ($PbCO_3$)	Micro	Diopside ($CaFe_2Si_2O_6$)	25.96
Chalcopyrite ($CuFeS_2$)	0.32	Manganocalcite ($MnCaSiO_3$)	
Chalcocite (Cu_2S)	Micro	Chlorite	
Pyrrhotite (Fe1-xS)	11.06	Sericite (Muscovite)	
Pyrite (FeS_2)	2.21	Mica (Muscovite)	
Marcasite (FeS_2)		Clay Minerals	
Goethite (FeO(OH))		Fluorite	0.78
Tin Ore (Cassiterite)	Micro	Apatite	
Cassiterite (SnO_2)	Micro	Rutile (TiO_2)	
Magnetite (Fe_3O_4)	1.19	Zircon ($ZrSiO_4$)	
Hematite (Fe_2O_3)		Ilmenite	
Limonite ($FeO(OH){\cdot}nH_2O$)		Other Minerals	
Limonite (FeO(OH))			
Rhodochrosite ($MnCO_3$)	Micro		

Source: Beijing General Research Institute of Mining & Metallurgy (BGRIMM), Small-Scale Processing Test Report, March 2007.

Table 10.4: Chemical phase analysis of lead and zinc

Zinc Phase	Zinc in Zinc Oxide	Zinc in Zinc Sulfide	Other Zinc	Total
Content (%)	0.24	3.70	0.03	3.97
Percentage (%)	6.04	93.20	0.76	100.00
Lead Phase	Lead in Lead Oxide	Lead in Lead Sulfide	Other Lead	Total
Content (%)	0.07	0.90	0.01	0.98
Percentage (%)	7.14	91.84	1.02	100.00

Source: Beijing General Research Institute of Mining & Metallurgy (BGRIMM), Small-Scale Processing Test Report, March 2007.

Ore texture and structure

Ore consists of various minerals, and its texture and structure describe the geometric characteristics and mutual relationships of these minerals within the orebody. Texture refers to the degree of crystallinity, crystal grain shape, size, and their interrelationships, while structure describes the shape, size, and spatial arrangement of mineral aggregates. The texture and structure of ore are critical factors influencing the ease of mineral liberation during grinding and provide an initial assessment of ore processing potential.

The primary ore structures observed in the sample are massive structure, disseminated structure, vein structure and banded structure.

Ore textures classified by genesis:

- **Textures formed by crystallisation and precipitation**: Euhedral, subhedral, inclusion, interstitial, and zonal textures are commonly observed in the ore and represent the dominant texture types. Sphalerite, galena, pyrrhotite, and chalcopyrite primarily exhibit inequigranular textures. Medium- to fine-grained pyrite commonly forms euhedral and subhedral crystal textures. Pyrrhotite inclusions are frequently observed in galena and sphalerite, forming a penetration texture.
- **Textures formed by metasomatism**: Symplectitic (fine-scale intergrowths of two or more mineral phases) texture, dissolution texture, and rim texture are typical replacement features. Sphalerite and galena exhibit close intergrowth, where replacement processes generate symplectitic and dissolution textures. Marcasite is embedded around pyrrhotite and sphalerite, forming a rim texture.
- **Textures formed by infilling processes**: Vein and network textures are formed by mineral infill processes. Chalcopyrite, galena, and sphalerite are found filling fractures within pyrrhotite, pyrite, or gangue mineral aggregates, forming vein and network structures.
- **Textures formed by stress effects**: Crush textures result from stress-related deformation. Coarse-grained pyrite commonly exhibits crush textures, and chalcopyrite often fills and cements fractured pyrite, forming complex intergrowth relationships.

The ore structure suggests that the useful mineral aggregates are relatively easy to liberate. However, the ore texture reveals complex intergrowth relationships among target minerals, making their separation more challenging.

Embedding characteristics and liberation degree of important minerals

Sphalerite

Sphalerite is irregularly embedded in the ore. It is closely associated with chalcopyrite, where chalcopyrite often appears as droplet-like and worm-like structures within sphalerite due to exsolution, leading to the inclusion of copper in the zinc concentrate during flotation. Sphalerite also exhibits a complex intergrowth relationship with galena, commonly forming tightly associated aggregates where galena appears irregularly or in vein-like patterns within sphalerite. Additionally, sphalerite is closely associated with pyrrhotite and pyrite, often filling their intergranular spaces or acting as a cementing and metasomatic mineral, creating a complex embedding pattern. Fine-grained pyrrhotite and pyrite, for which it is difficult to achieve complete monomeric liberation, tend to enter the zinc concentrate during flotation, affecting its zinc grade. In coarse-grained pyrite, sphalerite inclusions can be observed, with inclusion sizes generally ranging from 0.005 mm to 0.015 mm.

The grain size distribution of sphalerite is relatively coarse (Figure 10.1), generally ranging from 0.15 mm to 1.17 mm, with 80% of grains larger than 0.15 mm and 92% larger than 0.074 mm. Sphalerite has a relatively high iron content, averaging 12.19%, indicating that it is an iron-rich sphalerite, where iron and zinc cannot be effectively separated, thereby affecting the zinc grade of the concentrate.

Galena

Galena primarily exhibits an irregular or star-like distribution in the ore. It is closely associated with sphalerite, forming interwoven textures where galena appears in irregular, star-like, or vein-like patterns within sphalerite. Galena also has a close relationship with pyrrhotite, pyrite and chalcopyrite, frequently filling fractures in pyrrhotite and pyrite, leading to dissolution and symplectitic textures. In coarse-grained pyrite, galena inclusions are commonly observed, with inclusion sizes generally ranging from 0.005 mm to 0.011 mm.

The grain size of galena is finer than that of sphalerite (Figure 10.1), generally ranging from 0.02 mm to 0.83 mm, with 83% of particles falling within this range. Particles larger than 0.074 mm account for 72.6%, while 9.5% are smaller than 0.01 mm. The lead and sulfur content in galena are stable, with an average lead content of 86.57% and sulfur content of 13.43%, close to their theoretical values.

Chalcopyrite

Chalcopyrite is mainly irregularly embedded in the ore. It is closely associated with sphalerite, pyrrhotite, pyrite, galena and stannite. Due to exsolution, chalcopyrite commonly occurs as droplet-like and worm-like structures within sphalerite, making it difficult to separate as a monomer, often entering the zinc concentrate. Chalcopyrite sometimes acts as a cementing mineral for pyrite, forming a complex intergrowth, and frequently coexists with galena, exhibiting dissolution and symplectitic textures. In some cases, cassiterite appears as a network-like embedding within chalcopyrite, forming a rare texture.

The grain size of chalcopyrite is relatively fine (Figure 10.1), generally ranging from 0.02 mm to 0.21 mm, with 73% of grains falling within this range. Particles larger than 0.074 mm account for 56.5%. The chemical composition of chalcopyrite is stable, with an average copper content of 34.52%, sulfur content of 34.94%, and iron content of 30.54%, all of which are close to theoretical values.

Pyrrhotite

Pyrrhotite is the dominant metallic sulfide mineral in the ore, mainly exhibiting irregular embedding patterns. It is closely associated with pyrite, marcasite and sphalerite, often forming tight intergrowths with sphalerite and pyrite. Pyrrhotite sometimes exhibits lattice-like textures within sphalerite due to immiscibility effects. Coarse-grained pyrrhotite frequently displays crush textures, and sphalerite inclusions can be observed within pyrrhotite, with inclusion sizes ranging from 0.005 mm to 0.015 mm. A small portion of pyrrhotite is finely disseminated within gangue minerals, while marcasite, formed by alteration, often appears along the edges and fractures of pyrrhotite.

Pyrrhotite and pyrite often undergo mutual metasomatism, forming dissolution structures, and are mostly embedded within gangue minerals as aggregates. Some pyrrhotite is cemented and metasomatised by sphalerite and galena, forming complex intergrowth relationships. The grain size of pyrrhotite is coarser than that of pyrite, generally ranging from 0.02 mm to 0.59 mm. Single-mineral analysis shows that pyrrhotite contains 0.03% Pb and 0.11% Zn.

Pyrite and marcasite

Pyrite and marcasite are also important minerals in the ore, mainly occurring irregularly. Coarse-grained pyrite often exhibits crush textures, while medium- to fine-grained pyrite sometimes appears as euhedral, subhedral, and zoned crystals. Marcasite commonly occurs along the edges of pyrrhotite, forming rim textures, likely resulting from the alteration of pyrrhotite.

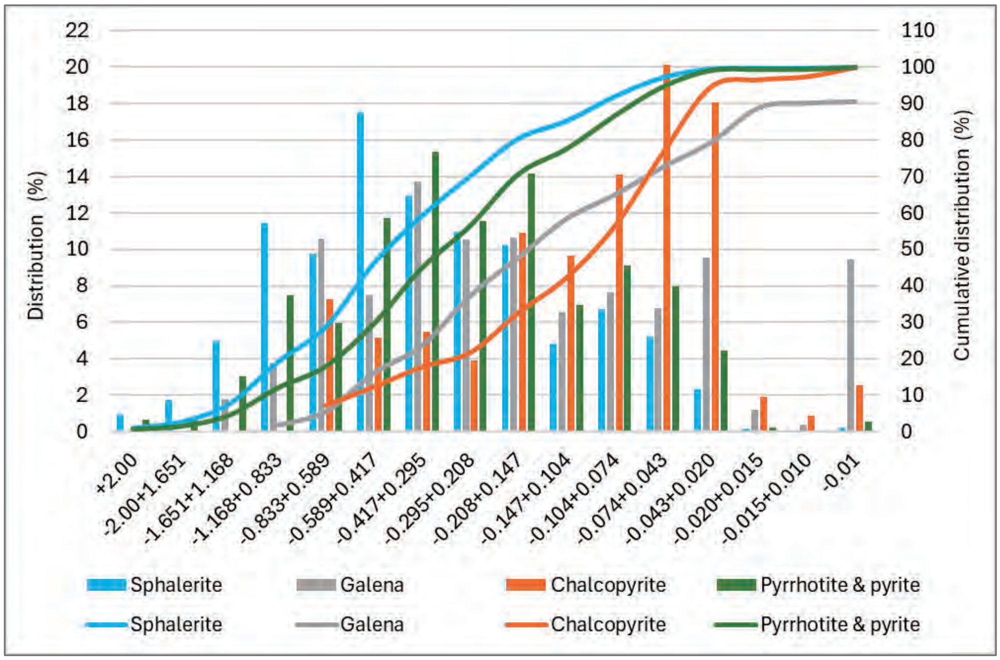
Pyrite is closely associated with sphalerite, galena and pyrrhotite, often irregularly embedded within these minerals or their aggregates. Meanwhile, sphalerite and galena frequently cement and metasomatised pyrite, forming complex intergrowth relationships. In coarse-grained pyrite, sphalerite and galena inclusions are commonly observed, with inclusion sizes ranging from 0.005 mm to 0.012 mm. These ultra-fine inclusions are difficult to achieve monomeric liberation and are often lost to the sulfur concentrate (pyrite concentrate) during Pb-Zn flotation. Pyrite and marcasite commonly coexist with pyrrhotite as aggregates within gangue minerals.

The grain size of pyrite and marcasite is smaller than that of sphalerite. Their chemical composition is very close to theoretical values. Single-mineral analysis shows that pyrite contains 0.085% Pb and 0.13% Zn.

The grain size distribution of pyrrhotite and pyrite is finer than sphalerite but coarser than galena, ranging from 0.043 mm to 1.17 mm, with 90% of particles falling within this range. Particles larger than 0.074 mm account for 86.7%.

Pyrrhotite, pyrite, and marcasite are non-recoverable iron sulfides, collectively referred to as pyritic sulfur minerals in this study.

Figure 10.1: Embedding grain size distribution of key minerals



Source: Based on process mineralogy data from the 2007 Small-Scale Processing Test conducted by BGRIMM, compiled by SRK.

The ore was ground to 70% passing 0.074 mm (70%, -74 µm), and the liberation degree of sphalerite and galena was determined. The results shown in Table 10.5 indicate that the liberation degree of sphalerite is 90.1%, while that of galena is 87.5%.

Table 10.5: Liberation characteristics of sphalerite and galena (grinding fineness: 70%, -0.074 mm)

Liberation Characteristics		Sphalerite (%)	Galena (%)
Monomeric Liberation		90.07	87.54
Intergrowth with:	Galena/Sphalerite	4.28	4.47
	Chalcopyrite and Other Sulfides	3.07	2.23
	Gangue Minerals	2.58	5.76

Source: Small-Scale Processing Test Research Report, BGRIMM, March 2007.

One of the key mineralogical factors influencing processing performance is the uneven grain size distribution of target minerals. Among them, sphalerite has the coarsest grain size, followed by pyrrhotite and pyrite, while galena and chalcopyrite have the finest grain size. To achieve both adequate monomeric liberation and avoid overgrinding, selecting an appropriate grinding process is considered important.

Another critical factor is the complex ore texture and mineral intergrowth relationships. Chalcopyrite occurs as droplet-like and worm-like textures within sphalerite, while some galena is finely disseminated in gangue minerals, and another portion exhibits highly irregular intergrowth with sphalerite.

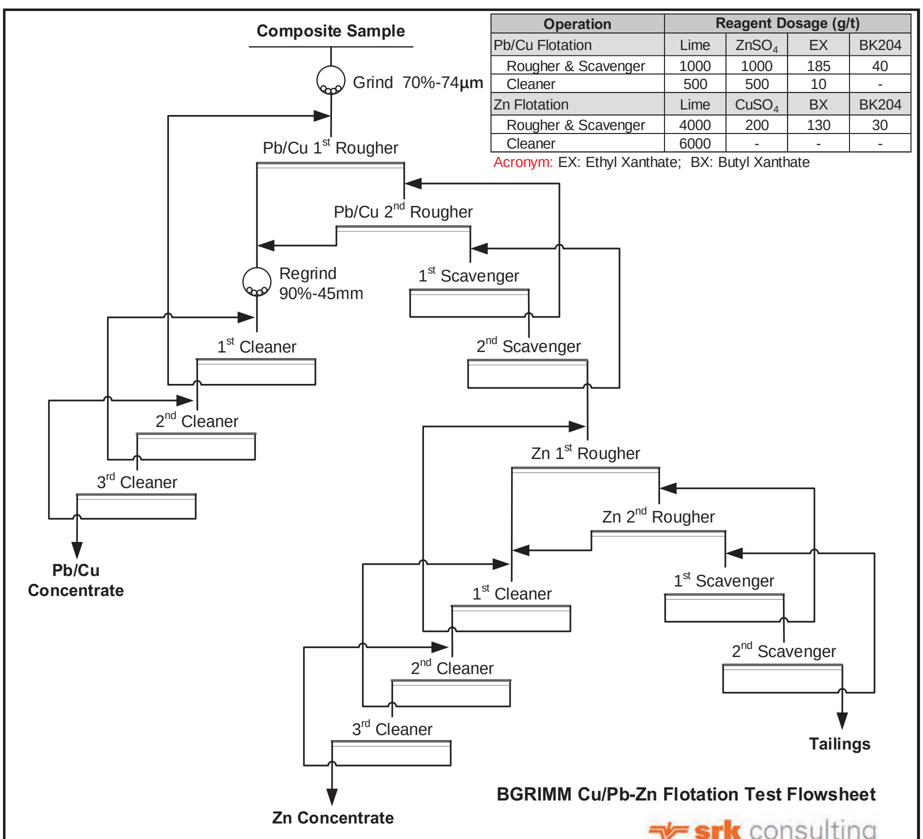
These characteristics directly affect the separation of lead from zinc, and copper from zinc, making the processing process more challenging.

10.1.3 Processing test

Based on mineralogical studies, a Cu-Pb bulk flotation followed by Zn flotation process was proposed.

Given the similar floatability of Cu and Pb, they were first collectively enriched into a Cu-Pb bulk concentrate, while the tailings was subjected to Zn flotation. Exploratory tests were first conducted to determine the process flow and reagent conditions. Based on these results, optimisation tests for Cu-Pb rougher flotation conditions were carried out, including reagent regime tests and grinding fineness tests. Further optimisation of Cu-Pb cleaner flotation conditions was conducted, including regrinding tests of the bulk rough concentrate and reagent dosage tests. The Zn flotation conditions were also optimised, including lime dosage tests and copper sulfate dosage tests. The most favourable conditions were then selected for open-circuit and closed-circuit flotation tests. The closed-circuit flotation process and test conditions are shown in Figure 10.2.

Figure 10.2: Closed-circuit flotation test process – BGRIMM



Operation	Reagent Dosage (g/t)			
Pb/Cu Flotation	Lime	$ZnSO_4$	EX	BK204
Rougher & Scavenger	1000	1000	185	40
Cleaner	500	500	10	-
Zn Flotation	Lime	$CuSO_4$	BX	BK204
Rougher & Scavenger	4000	200	130	30
Cleaner	6000	-	-	-

Acronym: EX: Ethyl Xanthate; BX: Butyl Xanthate

Source: Modified after BGRIMMM

The grinding fineness was set at 70%-74 μm. Lime and zinc sulfate were used as depressants for sphalerite and pyrite, while potassium butyl xanthate (PBX) was used as the collector for Cu-Pb minerals in the bulk flotation stage. In the Zn flotation stage, lime was used to depress pyrite, copper sulfate was used to activate the previously depressed sphalerite, and butyl xanthate was used as the collector for sphalerite flotation. The test results are presented in Table 10.6.

Table 10.6: Closed-circuit flotation test results – BGRIMM

Product	Yield (%)	Grade (%, Ag g/t)				Recovery (%)			
		Cu	Pb	Zn	Ag	Cu	Pb	Zn	Ag
Cu-Pb Concentrate	1.32	3.73	61.65	1.77	949	44.74	85.44	0.59	62.38
Zn Concentrate	7.43	0.55	0.52	48.09	16.76	37.05	4.05	90.4	6.19
Tailings	91.25	0.022	0.11	0.39	6.93	18.21	10.51	9.01	31.43
Raw Ore	100.0	0.11	0.95	3.95	20.12	100.0	100.0	100.0	100.0

Source: Small-Scale Processing Test Research Report, BGRIMM, March 2007.

The flotation tests achieved favourable processing indices. The Cu-Pb concentrate was enriched with significant amounts of copper and silver, leading to further Cu-Pb separation tests. Due to the limited sample quantity, only open-circuit separation tests were conducted. The tests yielded marketable copper concentrate and high-grade lead concentrate. The results are shown in Table 10.7.

Table 10.7: Cu-Pb separation flotation test results

Product	Yield (%)	Grade (%, Ag g/t)			Process Recovery (%)			Recovery to Raw Ore (%)		
		Cu	Pb	Zn	Cu	Pb	Zn	Cu	Pb	Zn
Cu Concentrate	5.34	26.67	1.21	1.16	49.23	0.10	5.00	22.03	0.09	0.03
Middling 1	3.65	19.25	18.83	2.40	22.62	1.07	7.08	10.12	0.91	0.04
Middling 2	7.58	9.34	38.08	2.92	22.79	4.51	17.88	10.20	3.85	0.11
Pb Concentrate	83.43	0.20	72.33	1.04	5.37	94.31	70.05	2.40	80.58	0.41
Cu-Pb Bulk Concentrate	100.0	3.11	63.98	1.24	100.0	100.0	100.0	44.74	85.44	0.59

Source: Small-Scale Processing Test Research Report, BGRIMM, March 2007.

The obtained flotation products were subjected to multi-element chemical analysis, with results summarised in Table 10.8. The copper concentrate from the Cu-Pb separation flotation was not analysed due to its small quantity. The zinc concentrate met commercial standards, and the levels of other beneficial and harmful elements were within acceptable limits, neither qualifying for premium pricing nor incurring penalties. However, the zinc grade was relatively low at only 48%, as the sphalerite contained iron and was classified as iron-rich sphalerite. The Cu-Pb bulk concentrate contained 3.73% Cu, which would exceed impurity limits if sold as lead concentrate. The lead concentrate obtained from the separation flotation of the Cu-Pb bulk concentrate had a high lead grade, with silver content increasing to 979 g/t, and no harmful elements exceeding acceptable limits, making it a high-quality lead concentrate.

Due to the low copper grade in the ore, the copper recovery rate was relatively low. The copper recovery rate in the Cu-Pb bulk concentrate was 44.7%, while the recovery rate in the final copper concentrate after separation was only 22%. However, the copper concentrate grade reached 26.7%, demonstrating that even with low copper content in the ore, a marketable copper concentrate could still be obtained.

Table 10.8: Flotation test product quality

Element	Content (%, g/t for Ag, Au)			
	Zn Concentrate	Cu/Pb Concentrate	Tailings	Pb Concentrate
Pb	0.52	61.65	0.11	72.33
Cu	0.55	3.73	0.022	0.20
Zn	48.09	1.77	0.39	1.04
Fe	15.45	7.16	11.73	2.85
S	31.43	16.02	1.3	12.94
Ag	10.06	949	5.23	979
Au	0.10	0.18	0.09	0.22
As	<0.001	0.013		0.008
MgO		0.13	2.48	0.10
CaO			19.27	
SiO_2	0.84		38.92	
Al_2O_3		0.19	4.95	0.11
Bi		0.23		0.20
Sn	0.017			
Sb	0.033			
Cd	0.27			

Source: Small-Scale Processing Test Research Report, BGRIMM, March 2007.
Note: The blue font indicates the beneficial elements in this concentrate, while the others are impurities.

10.1.4 Recycled water test – Xinjiang Nonferrous Institute, 2015

Test samples

In the 2007 BGRIMM test, PBX, a strong collector, was used as the collector for sphalerite, while a large amount of lime (11.5 kg/t) was applied to depress pyrite, following an aggressive reagent regime characterised as 'strong depression and strong activation'. The residual reagents and metal ions in the recycled water could have an adverse effect on processing performance. However, as no recycled water test was conducted, the impact of recycled water on flotation performance remains to be further studied.

To optimise the use of recycled water, assess its impact on flotation indices, and determine the most suitable reagent regime for its utilisation, the Xinjiang Nonferrous Metals Research Institute conducted a recycled water test on a sulfide ore sample.

The test samples were collected from Pb14 and Pb12, both sulfide ores. The samples were blended by weight according to Table 10.9, with an analysed grade of 0.3% Cu, 3.78% Pb, 6.14% Zn, and 59.11 g/t Ag. Although the sample had a relatively high grade, it was deemed suitable for the recycled water test.

The primary copper-, lead- and zinc-bearing minerals were chalcopyrite, galena and sphalerite, with sulfide phase contents of 99%, 89% and 94%, respectively. The oxidation rate was low, confirming the sample as a typical sulfide ore.

Table 10.9: Sample preparation for recycled water test – Xinjiang Nonferrous Institute

Sample	Grade (%)			Sample Weight	
	Cu	Pb	Zn	Weight (kg)	Proportion (%)
14#High-Grade Ore	0.34	1.72	9.44	30	9.68
14#Low-Grade Ore	0.33	3.15	7.41	90	29.03
14#Primary Ore	0.15	1.46	5.23	60	19.35
12#Primary Ore	0.39	7.14	6.07	120	38.71
12#Country Rock	0.0096	0.043	0.046	10	3.23
Calculated Grade	0.311	4.18	6.41	310	100.00
Measured Grade	0.303	3.78	6.14	310	

Source: Recycled Water Test Report, Xinjiang Nonferrous Institute, November 2015.

Recycled water test

In actual concentrator operations, to achieve zero discharge of wastewater, all recycled water must be fully utilised. The recycled water originates from the overflow of the tailings thickener, return water from the tailings pond, concentrate dewatering, and floor washing water from the concentrator. Since different reagents are used in the separation of copper, lead and zinc, the recycled water contains various metal ions and residual reagents, which affect the separation of copper, lead and zinc, leading to a decline in concentrate grade or metal recovery rates. The water quality of tailings return water from laboratory tests is shown in Table 10.10. The laboratory tailings water contains low concentrations of metal ions, with a pH value of 9–10. After standing for a period, natural degradation can reduce the pH to 7.

Table 10.10: Recycled water quality

Recycled water	Content (mg/L)				pH
	Cu^{2+}	Pb^{2+}	Zn^{2+}	Ca^{2+}	
Fresh Tailings Water	0.089	<0.01	0.12	333	9.5
After 3 hours	0.089	<0.01	0.06	298	9.0
After 6 hours	0.062	0.025	0.042	321	8.0
After 24 hours	0.044	0.01	0.011	276	7.0
Secondary Circulated Water	0.091	0.05	0.16	401	10.0
After 3 hours	0.081	0.04	0.10	376	9.5
After 6 hours	0.06	0.04	0.07	396	9.0
After 24 hours	0.02	0.01	0.04	399	7.0

Source: Recycled Water Test Report, Xinjiang Nonferrous Institute, November 2015.

To mitigate the impact of recycled water, adjustments were made by adding reagents in the mill and modifying the flotation reagent regime. Based on experimental exploration and reagent system analysis, sodium sulfide was added in the mill to reduce the effects of metal ions and residual reagents. Additionally, the pyrite depressant FSD was introduced to decrease lime consumption and lower pulp alkalinity. Zinc sulfate and sodium sulfite were used as sphalerite depressants. A selective reagent combination of SN-9# (ethyl thiocarbamate) and ADD (ammonium dibutyl dithiophosphate) replaced PBX as the collector for Cu-Pb flotation. For Zn flotation, a combination of SN-9# and Z-200 (ethyl thiocarbamate ester) was employed as a selective collector to reduce pyrite flotation during zinc recovery.

Following extensive exploratory and parameter optimisation tests, both freshwater closed-circuit flotation tests and recycled water closed-circuit flotation tests were conducted. The process flow and reagent scheme for the recycled water closed-circuit test are shown in Figure 10.3. The freshwater closed-circuit test followed the same process flow as the recycled water test but omitted the use of sodium sulfide and FSD, while all other reagent conditions remained the same. The test results are presented in Table 10.11. The results indicate that adopting a 'mild reagent regime' while using recycled water led to a slight decrease in copper and zinc recovery rates. This demonstrates that by adjusting the reagent system, recycled water can be used effectively without the need for expensive water treatment processes.

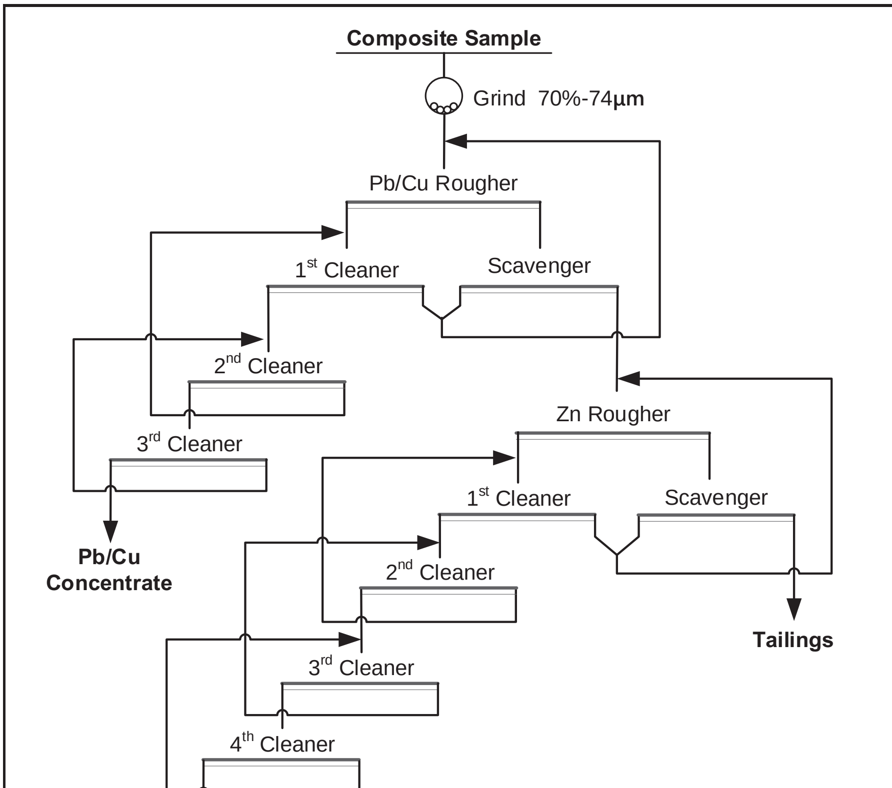
FSD is an organic pyrite depressant that can replace lime. However, its dosage must be carefully controlled, as excessive amounts not only depress pyrite but can also inhibit sphalerite, galena and chalcopyrite, resulting in the loss of valuable metals.

Table 10.11: Recycled water test results

Product	Yield (%)	Grade (%)			Recovery (%)			Water Quality
		Cu	Pb	Zn	Cu	Pb	Zn	
Cu-Pb Bulk Concentrate	7.36	2.73	46.98	6.66	65.33	92.45	8.09	Recycled Water
Zn Concentrate	11.22	0.66	0.92	45.38	24.08	2.76	83.99	
Tailings	81.42	0.04	0.22	0.59	10.59	4.79	7.92	
Raw Ore	100.0	0.31	3.74	6.06	100.0	100.0	100.0	
Cu-Pb Bulk Concentrate	7.10	3.01	47.65	5.89	70.39	92.82	6.87	Freshwater
Zn Concentrate	11.24	0.56	1.02	46.23	20.73	3.15	85.35	
Tailings	81.66	0.03	0.18	0.58	8.88	4.03	7.78	
Raw Ore	100.0	0.30	3.65	6.09	100.0	100.0	100.0	

Source: Recycled Water Test Report, Xinjiang Nonferrous Institute, November 2015.

Figure 10.3: Recycled water test process and conditions



Operation	Reagent Dosage (g/t)				
Grind	Lime	$ZnSO_4$	Na_2SO_3	Na_2S	FS
Ball Mill	1500	1600	200	600	150
Cu/Pb Flotation		$ZnSO_4$	Na_2SO_3	SN-9#	ADD
Rougher & Scavenger	-	400	400	3	2
Cleaner	-	300	200	-	-
Zn Flotation	Lime	$CuSO_4$	FS	SN-9#	Z200
Rougher & Scavenger	-	700	100	2	6
Cleaner	-	-	-	-	-

Acronym: ADD: ammonium dibutyl dithiophosphate

Source: Recycled Water Test Report, Xinjiang Nonferrous Institute, November 2015.

10.1.5 Process optimisation test – Guangdong Institute of Comprehensive Utilization, 2017

To further optimise the process flow and reagent regime, as well as to refine the reagent scheme for recycled water use, the Guangdong Institute of Comprehensive Utilization of Resources conducted a flotation test in 2017 on a sulfide ore sample. The study compared two process flows: Cu-Pb bulk flotation followed by separation and zinc flotation, and sequential preferential flotation of copper, lead, and zinc. Additionally, both process flows were tested using freshwater and recycled water to evaluate their impact.

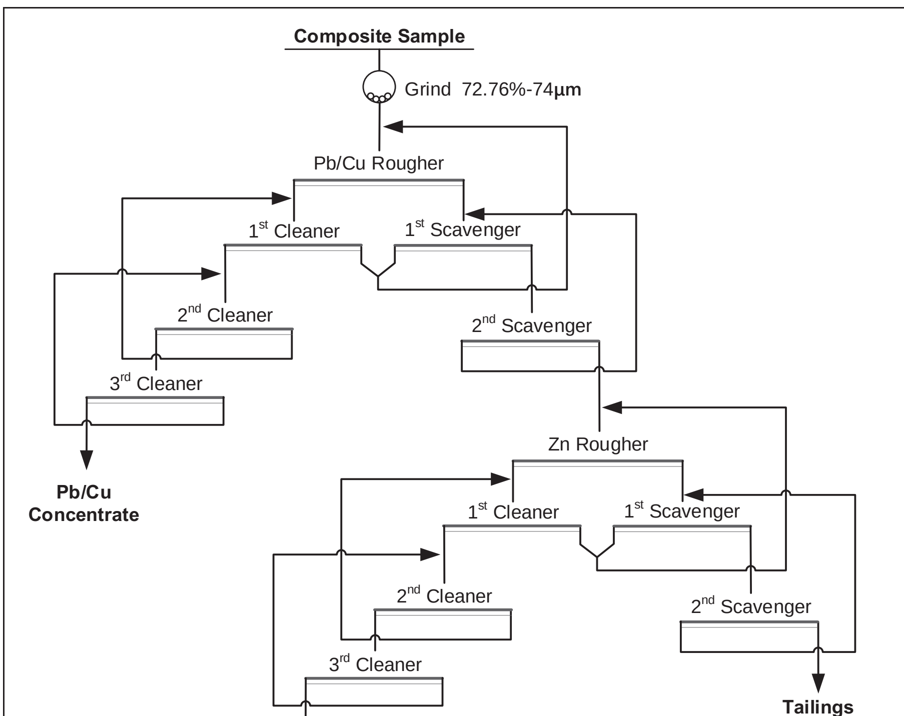
The assay results of the test sample were 0.21% Cu, 2.74% Pb, 4.90% Zn, 30.0 g/t Ag and 10.48% S.

10.1.6 Copper-lead bulk flotation – zinc flotation process test

A series of optimisation tests were conducted for the selected process, followed by a closed-circuit flotation test. The test flowsheets for both freshwater and recycled water were identical, with only slight differences in the reagent regime. In the recycled water test, sodium sulfide and FSD (pyrite depressant) were added in the grinding mill to mitigate the effects of metal ions and residual reagents in the recycled water, whereas in the freshwater test, these reagents were not used. The closed-circuit flowsheet and reagent regime for the recycled water test are shown in Figure 10.4, and the test results are presented in Table 10.12.

When using recycled water, the recovery rates of copper and lead were slightly lower compared to freshwater, while the impact on zinc recovery was minimal. This result is consistent with the conclusions from the tests conducted by the Xinjiang Nonferrous Institute, confirming that adding sodium sulfide and FS in the grinding stage can effectively mitigate the negative effects of using recycled water.

Figure 10.4: Cu-Pb bulk flotation – Zn flotation test process and conditions



Operation	Reagent Dosage (g/t)				
Grind	Na_2S	FS			
Ball Mill	150	200			
Cu/Pb Flotation	$ZnSO_4$	Na_2SO_3	DP25#	BX	MIBC
Rougher & Scavenger	800	800	28	20	10
Cleaner	700	450	5	-	-
Zn Flotation	Lime	$CuSO_4$	BX		MIBC
Rougher & Scavenger	2400	300	44		10
Cleaner	900	-	-	-	-

Acronym: DP25#: Dithiophosphate 25#; BX: Butyl Xanthate

Source: Process Optimization Test, Guangdong Institute of Comprehensive Utilization, May 2017.

Table 10.12: Cu-Pb bulk flotation – Zn flotation process test results

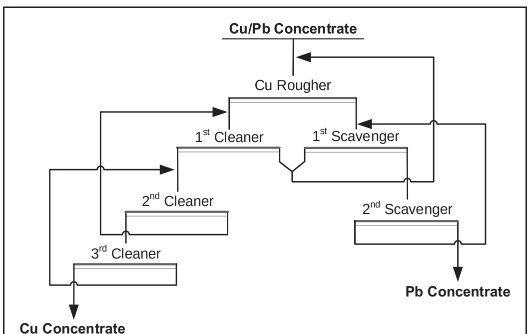
Product	Yield (%)	Grade (%)			Recovery (%)			Water Quality
		Cu	Pb	Zn	Cu	Pb	Zn	
Cu/Pb Concentrate	4.58	2.91	54.95	4.71	66.09	89.63	4.42	Freshwater
Zn Concentrate	9.19	0.48	0.45	48.36	21.64	1.47	90.83	
Tailings	86.23	0.03	0.29	0.27	12.27	8.90	4.76	
Total	100.0	0.20	2.81	4.89	100.0	100.0	100.0	
Cu/Pb Concentrate	4.48	2.87	53.80	4.41	64.34	88.45	4.02	Recycled Water
Zn Concentrate	9.25	0.49	0.51	48.12	22.70	1.73	90.54	
Tailings	86.27	0.03	0.31	0.31	12.96	9.81	5.44	
Total	100.0	0.20	2.72	4.92	100.0	100.0	100.0	

Source: Process Optimization Test, Guangdong Institute of Comprehensive Utilization, May 2017.

10.1.7 Copper-lead separation test

The Cu-Pb bulk concentrates obtained from the freshwater closed-circuit test and recycled water closed-circuit test were subjected to a closed-circuit Cu-Pb separation flotation test using a lead depression and copper flotation process. The test flowsheet is shown in Figure 10.5. The same reagent regime and flotation time were applied to both concentrates, and the results are presented in Table 10.14. Regardless of whether the bulk concentrate was obtained using freshwater or recycled water, the Cu-Pb separation performed effectively, yielding marketable copper, lead and zinc concentrates. Using recycled water, the results were copper concentrate with 21.9% Cu and 55.0% recovery, lead concentrate with 60.1% Pb and 87.8% recovery, and zinc concentrate with 48.1% Zn and 90.5% recovery. The flotation indices for recycled water were slightly lower than those for freshwater, but considering the full utilisation of recycled water, the trade-off was deemed acceptable.

Figure 10.5: Cu-Pb separation flotation process for bulk concentrate



Operation	Reagent Dosage (g/t)						Time
	Na_2S	Active C	FC	$ZnSO_4$	Na_2SO_3	Z200	min
Cu Rougher	100	100	600	200	400	10	2.0
1st Scavenger						5	2.0
2nd Scavenger						5	2.0
1st Cleaner			300		200	5	1.5
2nd Cleaner			225				1.2
3rd Cleaner			150		100		1.0

Source: Process Optimization Test, Guangdong Institute of Comprehensive Utilization, May 2017.

Table 10.13: Cu-Pb separation flotation closed-circuit test results

Selection Index		Unit	Test[1]	Historical Production			Forecast		
				2022	2023	2024	Average	Downside	Upside
Copper Concentrate Grade	Cu	%	21.95	18.67	19.62	19.78	20.0	19.0	20.0
	Ag	g/t	355.0	559.0	641.6	650.0	700.0	650.0	620.0
Copper Concentrate Recovery Rate	Cu	%	54.97	43.16	53.26	49.89	51.0	50.0	52.0
	Ag	%	6.48	7.53	10.77	9.8	10.2	10.3	10.1
Lead Concentrate Grade	Pb	%	60.11	57.44	62.76	60.12	60.0	60.0	62.0
	Ag	g/t	401.0	594.1	784.5	747.3	720.0	750.0	630.0
Lead Concentrate Recovery Rate	Pb	%	87.80	85.71	89.54	88.14	88.0	87.0	88.0
	Ag	%	58.22	64.68	66.63	65.8	66.4	65.3	67.1
Zinc Concentrate Grade	Zn	%	48.12	47.75	46.08	45.76	46.0	46.0	46.0
Zinc Concentrate Recovery Rate	Zn	%	90.54	90.03	91.91	91.89	91.0	90.0	92.0
Copper concentrate yield		%	0.50	0.41	0.68	0.44	0.51	0.47	0.65
Lead Concentrate yield		%	3.98	3.35	3.45	2.58	3.23	2.61	4.26
Zinc Concentrate production		%	9.25	7.77	8.64	7.04	7.91	6.85	10.00
Copper concentrate production		t		2,315	4,028	586	3,060	2,842	3,900
Lead Concentrate production		t		18,707	20,375	3,433	19,360	15,660	25,548
Zinc Concentrate production		t		43,405	51,097	9,358	47,478	41,087	60,000
Ore Feed Amount		kt		558.76	591.06	133.01	600.00	600.00	600.00
Ore Feed Grade	Cu	%	0.20	0.18	0.25	0.1746	0.20	0.18	0.25
	Pb	%	2.72	2.24	2.42	1.76	2.20	1.80	3.00
	Zn	%	4.92	3.95	4.33	3.50	4.00	3.50	5.00
	Ag	%	27.41	30.75	40.59	29.33	35.0	30.0	40.0

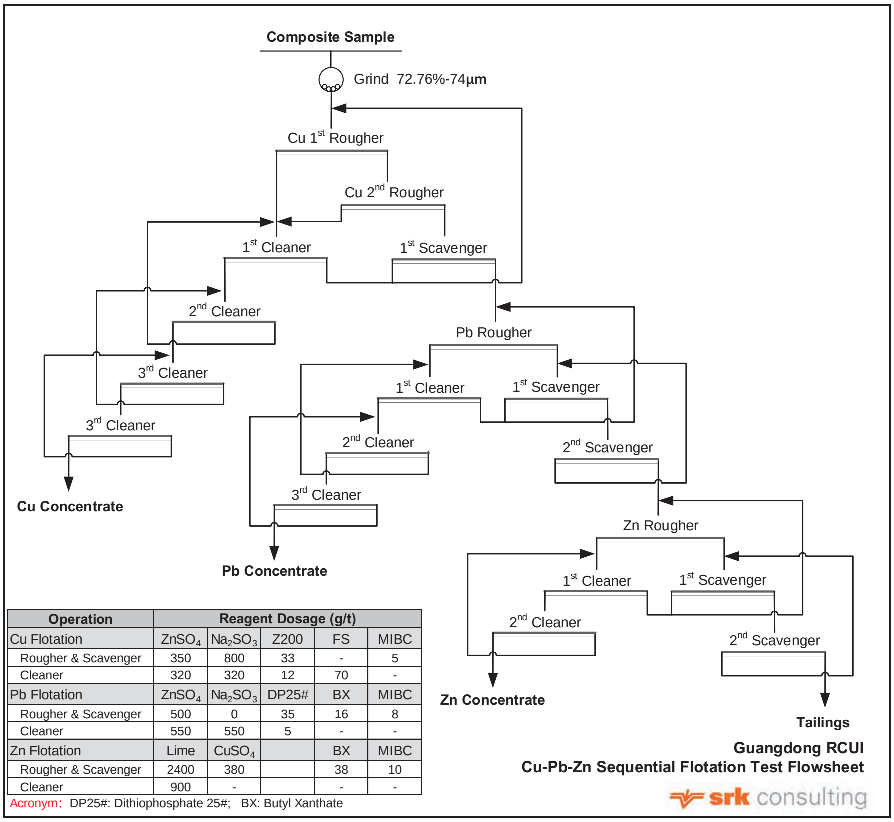
Source: Process Optimization Test, Guangdong Institute of Comprehensive Utilization, May 2017.

10.1.8 Sequential preferential flotation process for copper, lead and zinc

For polymetallic Cu-Pb-Zn ores with high Cu-Pb grades or complex processing characteristics, the sequential preferential flotation process can be used to obtain high-quality concentrates. However, this process is generally considered longer and more costly compared to the Cu-Pb bulk flotation followed by separation and Zn flotation process.

Based on a series of optimisation tests and open-circuit flotation tests, closed-circuit flotation tests were conducted. The test flowsheets for both freshwater and recycled water were identical, and the reagent regimes were also largely consistent, achieving satisfactory results. The test process is illustrated in Figure 10.6, which also shows the reagent scheme. The test results are presented in Table 10.14. Compared to using only freshwater, using recycled water resulted in a slight decrease in lead and zinc concentrate grades and recovery rates.

Figure 10.6: Sequential preferential flotation process and conditions for Cu-Pb-Zn



Source: Process Optimization Test, Guangdong Institute of Comprehensive Utilization, May 2017.

Table 10.14: Closed-circuit test results of sequential preferential flotation for Cu-Pb-Zn

Product	Yield (%)	Grade (%)			Recovery (%)			Water Quality
		Cu	Pb	Zn	Cu	Pb	Zn	
Copper Concentrate	0.62	20.37	3.46	4.04	61.20	0.76	0.51	Freshwater
Lead Concentrate	3.89	0.28	64.39	3.01	5.28	88.43	2.39	
Zinc Concentrate	9.20	0.48	0.42	48.75	21.40	1.36	91.47	
Tailings	86.29	0.03	0.31	0.32	12.13	9.44	5.63	
Total	100.0	0.21	2.83	4.90	100.0	100.0	100.0	
Copper Concentrate	0.64	20.08	3.86	3.92	61.88	0.91	0.51	Recycled water
Lead Concentrate	3.77	0.29	63.08	4.50	5.26	87.46	3.44	
Zinc Concentrate	9.42	0.45	0.43	47.61	20.41	1.49	90.98	
Tailings	86.17	0.03	0.32	0.29	12.45	10.14	5.07	
Total	100.0	0.21	2.72	4.93	100.0	100.0	100.0	

Source: Process Optimization Test, Guangdong Institute of Comprehensive Utilization, May 2017.

10.2 Processing test on oxidised ores

The Guangdong Institute of Comprehensive Utilization completed processing tests on three oxidised ore samples in July 2018 and August 2019. Since oxidised ores are classified as refractory ores, the so-called 'oxidised ore samples' were blended ore samples with a controlled Pb-Zn oxidation rate not exceeding 20%. The mine classifies ores based on the oxidation rate of lead minerals: oxidation rate <10% as sulfide ore; oxidation rate 10–30% as mixed ore; oxidation rate >30% as oxidised ore.

10.2.1 Test samples

The 2018 test sample (Composite Sample 1) was prepared by blending one low-oxidation sample and one high-oxidation sample in a 2:1 weight ratio. The resulting composite test sample had a lead oxidation rate of 17.3%, as detailed in Table 10.15.

Table 10.15: 2018 composite test sample (Composite 1)

Sample	Grade (%)				Oxidation Rate (%)	
	Cu	Pb	Zn	Ag	Pb	Zn
Low-Oxidation Sample		4.34	5.97		12.4	7.4
High-Oxidation Sample		6.17	6.73		27.1	13.8
Composite 1	0.33	4.90	6.20	42.42	17.3	9.5

Source: Processing Test Report on Improving Recovery of High-Oxidation Lead-Zinc Ore, Guangdong Institute of Comprehensive Utilization, July 2018.

Two test samples were used in the 2019 experiments. Composite Sample 2 was prepared by combining one oxidised ore composite sample with one primary ore sample. The oxidised ore composite sample consisted of five subsamples, and their Pb-Zn phase analysis results are shown in Table 10.16.

Table 10.16: Pb-Zn phase composition of five subsamples

Phase	Content (%)					Distribution Rate (%)				
	1#	2#	3#	4#	5#	1#	2#	3#	4#	5#
Sulfide Lead (PbS)	2.2	3.86	1.3	1.24	1.9	38.87	55.62	32.02	40.13	30.45
Cerussite ($PbCO_3$)	0.36	0.79	0.19	0.5	1.47	6.36	11.38	4.68	16.18	23.56
Anglesite ($PbSO_4$)	2.51	1.93	2.09	1	2.31	44.35	27.81	51.48	32.36	37.02
Plumbojarosite	0.59	0.36	0.48	0.35	0.56	10.42	5.19	11.82	11.33	8.97
Total Lead (Pb)	5.66	6.94	4.06	3.09	6.24	100.0	100.0	100.0	100.0	100.0
Sulfide Zinc (ZnS)	3.04	4.96	2.09	2.24	2.15	64.68	64.75	62.39	59.42	61.78
Oxidised Zinc (ZnO)	1.51	2.57	1.11	1.38	1.23	32.13	33.55	33.13	36.60	35.34
Zinc Ferrite ($ZnFe_2O_4$)	0.15	0.13	0.15	0.15	0.1	3.19	1.70	4.48	3.98	2.87
Total Zinc (Zn)	4.7	7.66	3.35	3.77	3.48	100.0	100.0	100.0	100.0	100.0

Source: Processing Test Report on Oxidised Lead-Zinc Ore, Guangdong Institute of Comprehensive Utilization, August 2019.

The combination details are provided in Table 10.17.

Table 10.17: Composition of oxidised ore samples

Subsample	Weight (kg)	Proportion (%)	Grade (%)				Oxidation Rate (%)	
			Cu	Pb	Zn	Ag[1]	Pb	Zn
1#	172.2	25.7		5.66	4.70		61.1	35.3
2#	149.0	22.2		6.94	7.66		44.4	35.2
3#	117.4	17.5		4.06	3.35		68.0	37.6
4#	70.0	10.4		3.09	3.77		59.9	40.6
5#	161.4	24.1		6.24	3.48		69.6	38.2
Calculated	670.0	100.0		5.54	4.73		60.5	37.0
Measured Oxidised Ore Composite Sample			0.32	5.40	4.72	35	59.6	36.5

Source: Processing Test Report on Oxidised Lead-Zinc Ore, Guangdong Institute of Comprehensive Utilization, August 2019.

Composite Sample 2 was composed of 85% primary ore and 15% oxidised ore, as shown in Table 10.18.

Table 10.18: Sample composition of 2019 composite sample (Composite 2)

Sample	Proportion (%)	Grade (%)		Oxidation Rate (%)	
		Pb	Zn	Pb	Zn
Oxidised Ore Composite Sample	15	5.40	4.72	59.6	36.5
Sulfide Ore Sample 1	85	3.35	5.21	7.26	4.43
Composite Sample 2 (Calculated)		3.66	5.14	15.10	9.24
Composite Sample 2 (Measured)		3.64	5.14	18.84	8.86

Source: Processing Test Report on Oxidised Lead-Zinc Ore, Guangdong Institute of Comprehensive Utilization, August 2019.

Composite Sample 3 was prepared by blending one oxidised ore composite sample with one sulfide ore sample in a ratio of 4% to 96%, as detailed in Table 10.19.

Table 10.19: Sample composition of 2019 verification test (Composite 3)

Sample	Proportion (%)	Grade (%)		Oxidation Rate (%)	
		Pb	Zn	Pb	Zn
Oxidised Ore Composite Sample	4	5.40	4.72	59.6	36.5
Sulfide Ore Sample 2	96	2.51	4.72	14.56	8.19
Composite Sample 3 (Calculated)		2.63	4.72	16.40	9.34
Composite Sample 3 (Measured)		2.74	4.72	19.93	8.89

Source: Processing Test Report on Oxidised Lead-Zinc Ore, Guangdong Institute of Comprehensive Utilization, August 2019.

10.2.2 Processing test

Based on a series of optimisation tests, closed-circuit flotation tests were conducted under optimal conditions. The test process followed a Cu-Pb bulk flotation – Zn flotation scheme. The Cu-Pb bulk flotation process was configured as '1 rougher, 3 scavengers, 3 cleaners', while the Zn flotation process followed a '1 rougher, 2 scavengers, 2 cleaners' configuration. For Composite 1 and Composite 2, comparative tests using freshwater and recycled water were conducted, while Composite 3 adopted the reagent regime and conditions from the recycled water test of Composite 2.

Sodium sulfide (Na_2S) was used as an activator for oxidised copper and lead minerals. Zinc sulfate ($ZnSO_4$) and sodium sulfite (Na_2SO_3) were applied as zinc mineral depressants. 25# black reagent and another reagent were used as collectors for Cu-Pb minerals. Lime (CaO) acted as a pyrite depressant, and copper sulfate ($CuSO_4$) was used as a zinc activator. MA was also employed as the collector for Zn flotation. In the recycled water test, FS was used as the recycled water treatment reagent. The reagent schemes for freshwater and recycled water tests of the three composite samples are summarised in Table 10.20. The grinding fineness for Composite 1 was 71.47%, -74 μm, with lead nitrate ($Pb(NO_3)_2$) added as a lead activator. The grinding fineness for Composite 2 and Composite 3 was 72.96%, -74 μm.

Table 10.20: Flotation reagent scheme for oxidised ores

Operation	Reagent Dosage (g/t)										
	Lime	Na_2S	FS	$Pb(NO_3)_2$	$ZnSO_4$	Na_2SO_3	$CuSO_4$	MA	DP25#	MIBC	2# Oil
Composite 1 in fresh water											
Ball mill		300		200							
Cu/Pb flotation	1150	500			1800	400		78	30	10	
Zn flotation	3100						410	95		15	
Composite 1 in returned water											
Ball mill		300	150	150							
Cu/Pb flotation	1000	500			1860	400		83	30	10	
Zn flotation	3600						380	90		20	
Composite 2 in fresh water											
Ball mill	400	700									
Cu/Pb flotation	700				1450	600		30	40	10	
Zn flotation	2900						350	32			10
Composite 2 and Composite 3 in returned water											
Ball mill		700	150								
Cu/Pb flotation	850				1550	600		30	35	10	
Zn flotation	3200						350	32			10

Source: Processing Test Report on Improving Recovery of High-Oxidation Lead-Zinc Ore (Guangdong Institute of Comprehensive Utilization, July 2018), Processing Test Report on Oxidised Lead-Zinc Ore (Guangdong Institute of Comprehensive Utilization, August 2019).

The closed-circuit flotation test results are summarised in Table 10.22. Copper and lead recovery rates were slightly lower with recycled water, but the impact on zinc recovery was minimal. Compared with the results from sulfide ore tests (Table 10.12), the degree of oxidation significantly affected the recovery rates of target metals.

Table 10.21: Flotation test results of oxidised ores

Product	Yield (%)	Grade (%)					Recovery (%)				
		Cu	Pb	Zn	Ag[1]	S	Cu	Pb	Zn	Ag	S
Composite 1 in fresh water											
Cu/Pb Concentrate	7.48	3.14	57.11	3.67			71.68	83.54	4.29		
Zn Concentrate	11.31	0.49	1.41	50.87			16.91	3.12	89.88		
Tailings	81.21	0.05	0.84	0.46			11.40	13.34	5.84		
Feed	100.0	0.33	5.11	6.40			100.0	100.0	100.0		
Composite 1 in recycled water											
Cu/Pb Concentrate	7.59	2.94	55.85	3.89	336	18.65	67.34	83.72	4.72	62.73	13.22
Zn Concentrate	11.13	0.60	1.49	50.09	48.5	30.12	20.15	3.28	89.05	13.28	31.30
Tailings	81.28	0.05	0.81	0.48	12.0	7.31	12.51	13.00	6.23	23.99	55.48
Feed	100.0	0.33	5.06	6.26	40.65	10.71	100.0	100.0	100.0	100.0	100.0
Composite 2 in fresh water											
Cu/Pb Concentrate	5.49	2.85	57.55	4.46	380	19.14	65.42	82.54	4.84	62.24	10.25
Zn Concentrate	9.16	0.53	0.96	48.89	45	31.13	20.30	2.30	88.42	12.30	27.81
Tailings	85.35	0.04	0.68	0.40	10	7.44	14.28	15.16	6.74	25.46	61.94
Feed	100.0	0.24	3.83	5.06	33.52	10.25	100.0	100.0	100.0	100.0	100.0
Composite 2 in recycled water											
Cu/Pb Concentrate	5.44		57.09	4.46				82.69	4.71		
Zn Concentrate	9.35		1.03	48.71				2.56	88.34		
Tailings	85.21		0.65	0.42				14.75	6.95		
Feed	100.0		3.76	5.15				100.0	100.0		
Composite 3 in recycled water											
Cu/Pb Concentrate	4.00		56.84	4.17				81.49	3.52		
Zn Concentrate	8.56		1.13	48.52				3.47	87.63		
Tailings	87.44		0.48	0.48				15.04	8.85		
Feed	100.0		2.79	4.74				100.0	100.0		

Note: The grade unit of Ag is g/t

Source: Processing Test Report on Improving Recovery of High-Oxidation Lead-Zinc Ore (Guangdong Institute of Comprehensive Utilization, July 2018), Processing Test Report on Oxidised Lead-Zinc Ore (Guangdong Institute of Comprehensive Utilization, August 2019).

10.3 Pre-concentration

10.3.1 Pre-concentration test

X-ray ore sorting is an advanced pre-concentration technology that enables the separation of lump ore, achieving waste rejection and pre-enrichment before grinding. This process improves the feed grade, reduces milling costs, and enhances processing efficiency. Huaxia Mining commissioned Beijing Horus Technology Co., Ltd. (Horus) to conduct pre-concentration tests on mineralised waste samples using its intelligent sorting machine. The objective was to discard mineralised waste before processing the ore further.

Two test samples were prepared, each crushed to 100% passing 60 mm, with fines (<10 mm) screened out. The -60+10 mm fraction was processed using the Horus intelligent sorting machine. The test was conducted at different discard rates, evaluating the grade of discarded mineralised waste (tailings) and metal loss rates. The results are summarised in Table 10.22.

For Sample #1, the -60+10 mm fraction yield was 81.6%. At a discard rate of 41%, the metal loss rates were 3.4% Pb and 6.2% Zn, with tailings grades of 0.04% Pb and 0.20% Zn. For Sample #2, the -60+10 mm fraction yield was 87.1%. At a discard rate of 39.4%, the metal loss rates were 3.6% Pb and 5.0% Zn, with tailings grades of 0.10% Pb and 0.34% Zn.

The results demonstrate that pre-concentration using an intelligent sorting machine is feasible. Horus estimates that its intelligent sorter has a processing capacity of 60–80 t/h for the -60+10 mm ore fraction.

Table 10.22: Results of discard test using intelligent sorting machine

Discard Control	Product	Grade (%)		Against raw ore			Against 10~60 mm		
				Yield (%)	Recovery (%)		Yield (%)	Recovery (%)	
		Pb	Zn		Pb	Zn		Pb	Zn
Sample #1									
Scheme 1	Concentrate	0.72	1.82	56.2	78.6	67.1	68.9	97.4	95.0
	Tailings	0.04	0.20	25.4	2.1	3.3	31.1	2.6	4.7
	-60+10 mm	0.51	1.32	81.6	80.7	70.6	100.0	100.0	99.7
	-10 mm	0.54	2.44	18.4	19.3	29.4			
	Raw ore	0.51	1.53	100.0	100.0	100.0			
Scheme 2	Concentrate	0.83	2.10	48.1	78.0	66.2	59.0	96.7	93.9
	Tailings	0.04	0.20	33.5	2.7	4.4	41.0	3.4	6.2
	-60+10 mm	0.51	1.32	81.6	80.7	70.6	100.0	100.1	100.1
	-10 mm	0.54	2.44	18.4	19.3	29.4			
	Raw ore	0.51	1.53	100.0	100.0	100.0			
Scheme 3	Concentrate	0.97	2.40	41.0	77.3	64.5	50.3	95.8	91.4
	Tailings	0.04	0.22	40.6	3.4	5.8	49.7	4.2	8.3
	-60+10 mm	0.51	1.32	81.6	80.7	70.6	100.0	100.0	99.7
	-10 mm	0.54	2.44	18.4	19.3	29.4			
	Raw ore	0.51	1.53	100.0	100.0	100.0			

Discard Control	Product	Grade (%)		Against raw ore			Against 10~60 mm		
				Yield (%)	Recovery (%)		Yield (%)	Recovery (%)	
		Pb	Zn		Pb	Zn		Pb	Zn
Sample #2									
Scheme 1	Concentrate	1.66	4.17	52.8	82.3	80.1	60.7	96.4	95.1
	Tailings	0.10	0.34	34.3	3.1	4.2	39.4	3.6	5.0
	-60+10 mm	1.04	2.66	87.1	85.3	84.2	100.0	100.1	100.1
	-10 mm	1.21	3.37	12.9	14.7	15.8			
	Raw ore	1.06	2.75	100.0	100.0	100.0			
Scheme 2	Concentrate	1.96	4.79	44.0	81.0	76.7	50.6	95.0	91.1
	Tailings	0.11	0.49	43.1	4.3	7.7	49.4	5.1	9.1
	-60+10 mm	1.04	2.66	87.1	85.3	84.2	100.0	100.1	100.2
	-10 mm	1.21	3.37	12.9	14.7	15.8			
	Raw ore	1.06	2.75	100.0	100.0	100.0			
Scheme 3	Concentrate	2.26	5.14	37.8	80.4	70.6	43.4	94.2	83.8
	Tailings	0.11	0.76	49.3	5.0	13.6	56.6	5.8	16.2
	-60+10 mm	1.04	2.66	87.1	85.3	84.2	100.0	100.0	100.0
	-10 mm	1.21	3.37	12.9	14.7	15.8			
	Raw ore	1.06	2.75	100.0	100.0	100.0			

Source: Horus Technology, Intelligent Sorting Test Report for Xizang Huaxia Lead-Zinc Mine, May 2024.

10.3.2 Pre-concentration production

In 2024, Huaxia Mining established a pre-concentration plant at the open-pit mining site that incorporated crushing, screening and intelligent ore sorting. The trial production results from September to December 2024 are summarised in Table 10.23. A total of 72.36 kt of pre-concentration feed was processed, achieving a discard rate of 15.55% (relative to the ROM ore), with a metal loss rate of 1.3% Zn, indicating that intelligent ore sorting is feasible for pre-concentration of mineralised waste.

Since the ore contains low grades of copper, lead and silver, potential measurement errors may be significant. Therefore, zinc grade and recovery rate were primarily used to evaluate the sorting performance. SRK recommends that future pre-concentration operations focus on enhanced sampling, monitoring, and process control. Conducting tests on ores with different grades to obtain comprehensive technical parameters and cost data is recommended, which will guide future pre-concentration production.

Table 10.23: Results of trial production using intelligent sorting machine

Product	Production (t)	Yield (%)	Grade (%)			Metal Recovery (%)		
			Cu	Pb	Zn	Cu	Pb	Zn
Concentrate	42,676	58.98	0.07	0.05	3.60	52.3	62.5	73.8
Tailings	11,253	15.55	0.04	0.03	0.24	8.3	10.7	1.3
Fines	18,431	25.47	0.13	0.05	2.82	39.4	26.9	24.9
Raw ore	72,360	100.00	0.08	0.05	2.88	100.0	100.0	100.0

Source: Estimated by SRK based on data provided by Huaxia Mining.

10.4 Test conclusions

- Sphalerite is the primary zinc-bearing mineral, with a relatively coarse grain size. Galena is the main lead-bearing mineral, with a finer dissemination size, while chalcopyrite, the primary copper-bearing mineral, has an even finer grain size. The dominant iron sulfide is pyrrhotite, followed by pyrite and marcasite. Due to the complex intergrowth relationships between minerals, at a grinding fineness of 70%, -74 µm, the liberation degree of sphalerite reaches 90%, while galena achieves 87.5%.
- The reagent scheme is the primary factor affecting processing performance, particularly when using recycled water. A 'strong reagent regime' (strong depressants and collectors are used), leads to higher impurity ion concentrations, elevated pH values, and residual reagent accumulation in recycled water, negatively impacting flotation efficiency. A 'mild reagent regime' (using selective pyrite depressants and highly selective lead (copper) collectors with reduced lime dosage) results in lower impurity ion concentrations, pH, and residual reagent levels in recycled water. The addition of sodium sulfide and FS in the grinding stage helps mitigate the adverse effects of recycled water, achieving better processing results.
- Both the Cu-Pb bulk flotation – Cu-Pb separation – Zn flotation process and the sequential preferential flotation process (Cu-Pb-Zn) achieved satisfactory processing performance. However, the latter process is longer and more costly. Given the low copper grade in Mengya'a ore, the Cu-Pb bulk flotation – separation – Zn flotation process is recommended.
- Ore oxidation significantly impacts flotation performance. Since no flotation tests were conducted on ores with oxidation rates exceeding 20%, a direct correlation between ore oxidation and recovery rate has not yet been established. However, the quantity of oxidised ore is minimal, and its economic impact is insignificant.
- Pre-concentration using an intelligent sorting machine for mineralised waste is feasible. The laboratory tests and trial production both yielded positive results. SRK recommends enhancing process sampling and monitoring to obtain comprehensive trial production and cost data, providing technical and economic support for expanding the pre-concentration operation.

11 Processing plant

11.1 Overview

The Processing Plant was established in October 2007 and commenced production in December 2010. Designed by Lanzhou Nonferrous Metallurgical Design and Research Institute Co., Ltd., the plant initially had a designed processing capacity of 2,000 t/d, operating 200 days per year, with an annual ore throughput of 400 kt/a. The final products include copper concentrate, lead concentrate and zinc concentrate.

In December 2016, the Guangdong Institute of Metallurgical and Architectural Design completed a feasibility study for an expansion project to increase the processing nameplate capacity to 3,000 t/d. Based on the feasibility study, modifications in the crushing, grinding, flotation, and concentrate dewatering systems were gradually implemented from 2017 onwards. By 2022, the plant's processing nameplate capacity had successfully increased to 3,000 t/d.

The current production process is well structured, including traditional three-stage closed-circuit crushing, two-stage closed-circuit grinding, Cu-Pb bulk flotation followed by separation flotation, Zn flotation, and two-stage concentrate dewatering through thickening and pressure filtration.

The reagent scheme has been optimised by adding sodium sulfide in the grinding mill and using Y2 (an organic pyrite depressant similar to FS), zinc sulfate, and sodium sulfite as pyrite depressants. The benefits of doing so include reducing lime consumption, mitigating the impact of tailings return water on processing performance, improving water recycling efficiency, and achieving a closed-loop water circulation system with zero wastewater discharge.

In 2024, a pre-concentration plant for mineralised waste was constructed at the open-pit mining site, using an intelligent sorting machine to process -60+10 mm ore fraction and pre-discard mineralised waste, with promising trial production results.

Figure 11.1 is an aerial photograph taken by drone of the processing plant. From top to bottom, the key facilities include the freshwater high-level reservoir, recycled water high-level reservoir, ROM stockpile, crushing workshop, grinding workshop, flotation workshop, concentrate thickening workshop, concentrate pressure filtration workshop, concentrate warehouse, tailings pump station, and tailings thickening workshop. On the left side are the 35/10 kV substation and the Langgezhanlong River, the plant's key water source. The lower-left buildings house the laboratory and processing plant office, while the lower-right buildings contain the materials and reagent storage facility.

Figure 11.1: Drone photograph of the Mengya'a processing plant

Source：SRK site visit

11.2 Processing process

The processing flowsheet shown in Figure 11.2 includes crushing, grinding, flotation, and product dewatering. The flotation process consists of Cu-Pb mixed flotation, Zn flotation, and Cu-Pb concentrate separation flotation. All intermediate ores are sequentially returned to the previous operation.

Crushing

The ore transported from the both the open-pit and underground portions of the mine is unloaded into a 120 m³ raw ore bin or stockpiled in the raw ore yard. It is then fed into the primary jaw crusher by a heavy-duty plate feeder at the bottom of the bin. A traditional three-stage closed-circuit crushing process is adopted. The feed size is ≤500 mm, and the product size is less than 12 mm. The crushed product is transported by belt conveyors to two 1,500 t fine ore bins, each corresponding to a grinding-flotation series.

Grinding

The crushed ore in the fine ore bin is fed into the ball mills by multiple vibrating feeders and belt conveyors. The grinding process uses a two-stage closed-circuit system, with the crushed ore ground to a fineness of 70–75% passing -200 mesh (74 μm). Stage 1 consists of a grid-type ball mill and a double-screw classifier forming a closed circuit, while Stage 2 consists of a cyclone group and an overflow ball mill forming a closed circuit. There are two identical grinding circuits, and the subsequent flotation is also divided into two corresponding series.

Flotation

The 'Cu-Pb mixed – separation flotation – Zn flotation' process is used, where sulfur and zinc minerals are first depressed, and the Cu-Pb mixed concentrate is selected preferentially. The Cu-Pb tailings is then subjected to Zn flotation. In the Cu-Pb mixed flotation, flotation columns and flotation machines are employed. The flotation column is used to preferentially recover easily floatable minerals, and then the flotation machine follows a '1 roughing, 3 cleaning, 3 scavenging' process to recover Cu-Pb minerals that are more difficult to float. The concentrates from the flotation column and flotation machine are combined to form the Cu-Pb mixed concentrate. The Cu-Pb mixed concentrates from the two-flotation series are merged for Cu-Pb separation flotation, following a '1 roughing, 3 cleaning, 3 scavenging' process, where lead is depressed to produce both copper concentrate and lead concentrate.

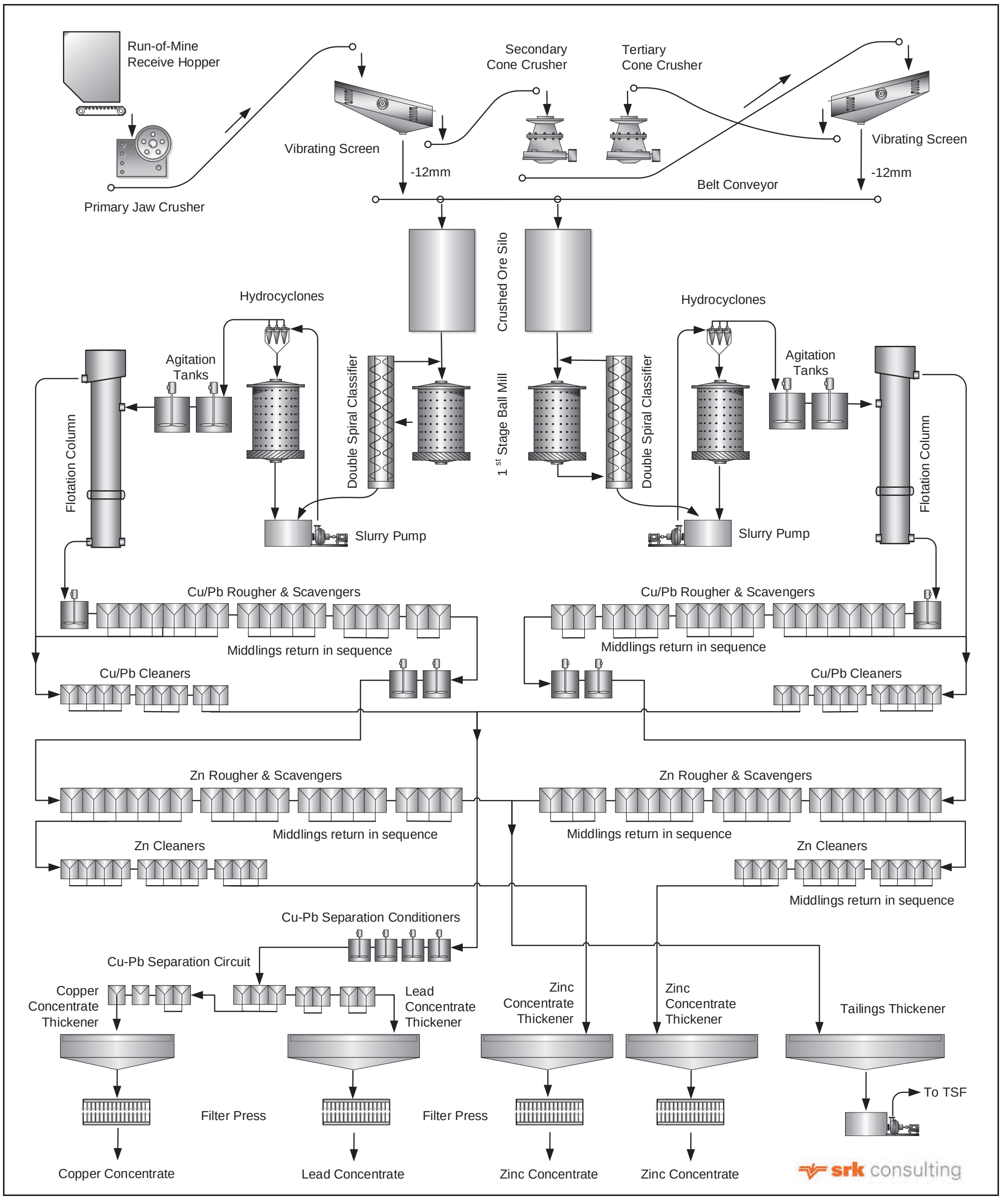
The Cu-Pb flotation tailings undergo the '1 roughing, 3 cleaning, 3 scavenging' process for Zn flotation. All intermediate concentrates from flotation are sequentially returned to the previous process.

Dewatering

Copper concentrate, lead concentrate, and zinc concentrate all undergo a two-stage dewatering process, consisting of thickening and filter pressing, to control the final moisture content of the concentrate to no more than 9%, which is the transportable level. If the moisture content exceeds 9%, further drying is carried out in the concentrate storage. All water extracted from the concentrates is pumped into the processing plant's high-level water recovery tank.

The tailings reports a concentration of 20–25%, which is thickened to 38% in the tailings thickener. The overflow is pumped into the high-level water recovery tank, where it is returned to the production process as process water. The underflow is pumped to the tailings storage facility (TSF). The return water from the TSF is also conveyed through pipelines into the high-level water recovery tank in the processing plant.

Figure 11.2: Processing production process for Mengya'a ore



Source: SRK, 2025, prepared based on site visit and client-supplied data.

Note: Prior to crushing, mineralised waste may be mixed with high-grade ore to optimise the feed ore grade, allowing the mineralised waste to be effectively utilised for concentrate production.

11.3 Expansion and technological renovation of processing equipment

The expansion and renovation of the processing plant commenced in 2017, and by 2022, the processing nameplate capacity had reached 3,000 t/d.

The key milestones in the specific renovation process were as follows:

- **2017**: The crushing system was upgraded by replacing the secondary and tertiary cone crushers and adding a vibrating screen before secondary crushing, which reduced the particle size of the crushed product.
- **2018**: A flotation column was added before the Cu-Pb mixed flotation to extend the flotation time and improve Cu-Pb recovery. Additionally, one more filter press was added for dewatering the lead and zinc concentrates.
- **2020**: The number of flotation cells for Cu-Pb rough flotation and zinc rough flotation was increased from 4 to 6. The scavenger flotation cells were also adjusted to extend the flotation time.
- **2021**: The reagent system underwent a significant adjustment, drastically reducing lime consumption. The pH of the process water was reduced from 13–14 to 7–8, leading to a gradual improvement in processing performance.
- **2022**: The secondary grinding cyclone classifier was replaced from FX500 to FX400, and the grinding fineness increased from 65–70% passing 74 μm to 70–75% passing 74 μm. The Cu-Pb separation process added a rough flotation cell, along with an additional scavenger and cleaner flotation stage, which improved the Cu-Pb separation efficiency.

Following completion of the above technological renovations, the processing nameplate capacity of the processing plant has reached 3,000 t/d. Although these renovations lack detailed engineering design, they have optimised the production process and operational parameters, improving the processing technical indicators.

Table 11.1 lists the current main processing equipment at the Mengya'a processing plant. Equipment for conveying ore via belt conveyors, pumps for transporting slurry and water, various slurry mixing tanks, reagent preparation and dosing equipment, maintenance equipment, and other auxiliary devices is not listed.

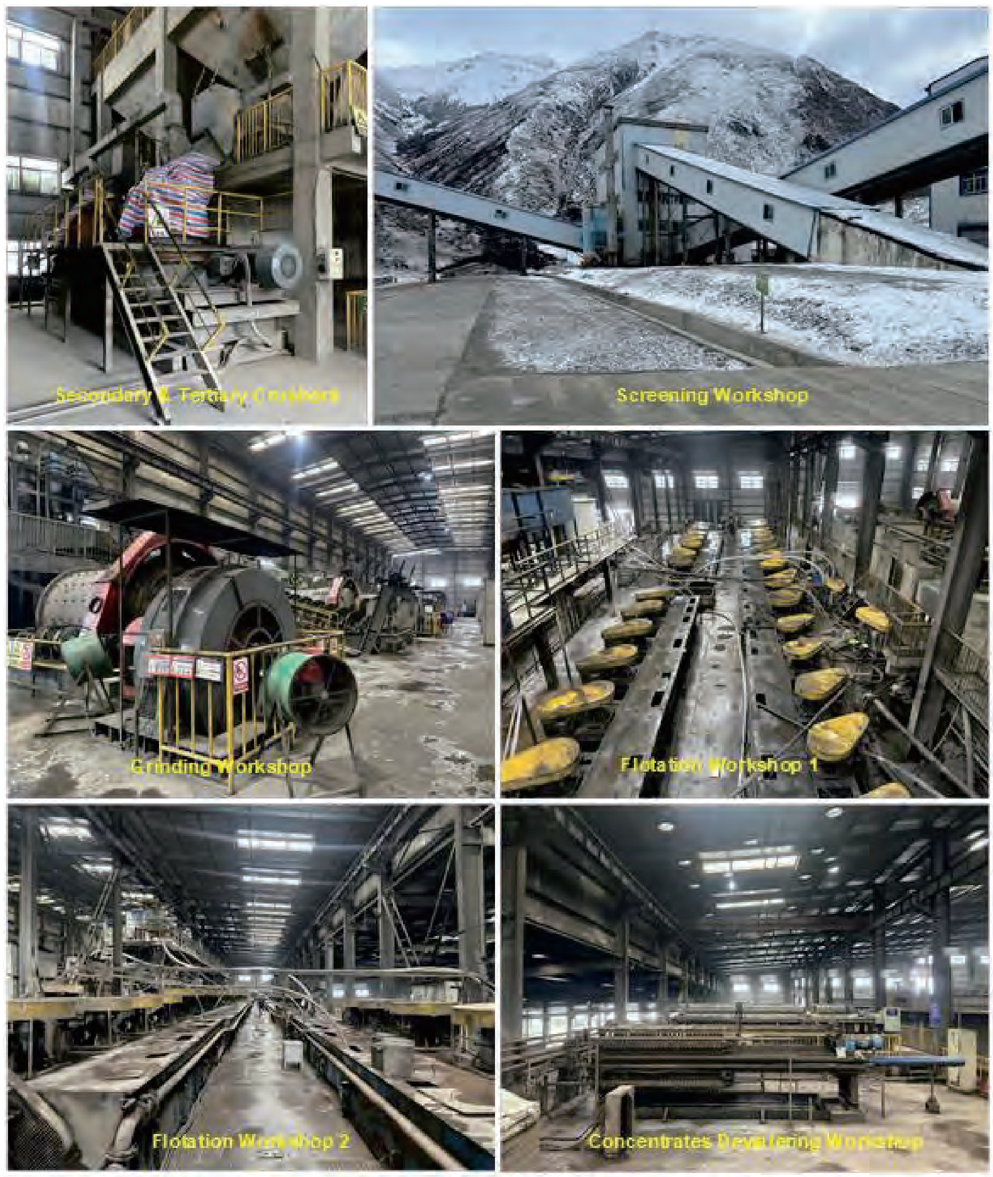
Table 11.1: Main processing equipment

No.	Equipment name	Model/specification	Power (kW)	Quantity
1	Raw Ore Bin	120 m^3	-	1
2	Plate Feeder	BZ1200-6	22	1
3	Jaw Crusher	C100	110	1
4	Cone Crusher	NH400C	220	1
5	Cone Crusher	CH440MF	315	1
6	Vibrating Screen	YSC2148	22	1
7	Vibrating Screen	YA2160	22	1
8	Fine Ore Bin	1500t	-	2
9	Electric Vibrating Feeder	GZ1006	1.1	8
10	Grid Type Ball Mill	MQG2736	400	2
11	Overflow Ball Mill	MQY2736	400	2
12	Double Spiral Classifier	2FG-20	15×2	2
13	Cyclone Group	FX400×4	-	2
14	Cyclone Feed Slurry Pump	150/100E-AH	90	4
15	Stirring Tank	Φ2.5×2.5 m	30	2
16	Stirring Tank	Φ2.5×2.5 m	22	8
17	Stirring Tank	Φ2.5×2.5 m	18.5	1
18	Stirring Tank	Φ2.5×2.5 m	11	3
19	Flotation Column	-	37+75	2
20	Flotation Machine	XCF-8	22	18
21	Flotation Machine	KYF-8	15	46
22	Flotation Machine	XCF-4	15	16
23	Flotation Machine	KYF-4	11	35
24	Thickener	NZS-9	7.5	1
25	Thickener	NT-18	11	3
26	Filter Press	CJZJ-10 m^2	11	1
27	Filter Press	CJZJ-30 m^2	22	3
28	Filter Press	XMZGF220/1250	5.5+15	2
29	Thickener	NZY-30	15	1

Source: Equipment ledger of Mengya'a beneficiation plant
Note: including flotation column installed in 2024.

Figure 11.3 presents a series of photographs from the main workshops at the processing plant.

Figure 11.3: Main workshops at the Mengya'a processing plant



Source: SRK site visit

11.4 Reagent system

For Mengya'a, a mildly oxidised polymetallic ore, to achieve ideal processing indicators and use all recycled water, the reagent system is a key factor.

Two types of reagent systems have been tested. One is the 'strong reagent system', which uses large amounts of lime supplemented with zinc sulfate as depressants for pyrite and sphalerite.

A strong collector, ethyl xanthate, is used for Cu-Pb flotation, and butyl xanthate is used as the collector for zinc. This process can achieve satisfactory processing indicators, but the excessive use of lime results in a high pH of the recycled water, which greatly affects the processing indicators and is detrimental to water recycling.

The second is the 'mild reagent system' such as experiments conducted by the Xinjiang Nonferrous Metal Research Institute and the Guangdong Province Comprehensive Resource Utilization Research Institute. This system uses sodium sulfide, zinc sulfate, sodium bisulfite, and a small amount of lime as depressants for pyrite and sphalerite, with collectors like 25# black reagent, ethyl thiocarbamate, butyl ammonium black reagent, and MA for Cu-Pb and Zn flotation. These collectors are characterised by weak collection ability but good selectivity. This process significantly reduces the amount of lime used, lowers the pH of the recycled water, and by adding sodium sulfide and FS in the mill, the recycled water utilisation problem is easily solved.

The processing plant continuously optimised the reagent system during production. The current reagent system, as shown in Table 11.2, is the 'mild reagent system' and has achieved good technical indicators. Sodium sulfide is added at a dosage of 800 g/t in the mill to mitigate the impact of recycled water on the process. No lime is used in Cu-Pb mixed flotation, and a small amount of pyrite organic depressant Y2 is added during Zn flotation, significantly reducing lime consumption, and achieving 100% utilisation of the recycled water. SRK considers it may be possible to further optimise the current reagent system, which will require ongoing exploration and improvement in future production.

Table 11.2: Current rWeagent system

Operation	Reagent Dosage (g/t)					
Cu-Pb Flotation	Na_2S	$ZnSO_4$	Na_2SO_3	DP25#[1]	MX[1]	2# Oil
	800	1000	350	40	40	
Zn Flotation	Lime	$CuSO_4$	Y2	/	MX[1]	2# Oil
	1700	550	60	/	40	
Cu-Pb Separation	Carbon[1]	CMC	Silicate[1]	$ZnSO_4$	Na_2SO_3	Z200
	100	40	200	100	200	25

Source: Provided by Mengya'a Processing Plant.

Note : [1]DP25#: Dithiophosphate 25#; MX: Mixed Xanthate; Carbon: Active Carbon; Silicate: Soluble Silicate

11.5 Processing production performance

Table 11.3 shows the production performance of the Mengya'a processing plant from 2022 to July 2025. In 2023, the Feed processing volume reached nearly 600 kt. The copper concentrate grade was 19.62% Cu with 641.6 g/t Ag, the copper recovery rate was 53.26%, and the silver recovery rate was 10.77%. The lead concentrate grade was 62.76% Pb with 784.5 g/t Ag, the lead recovery rate was 89.54%, and the silver recovery rate was 66.63%. The zinc concentrate grade was 46.08% Zn, with a zinc recovery rate of 91.91%. In 2024 and 2025, 4,251 t and 58,510 t of externally purchased ores were processed respectively.

In the copper concentrate, the impurity contents were zinc 9–11%, lead 3.5–4.5%, and arsenic less than 0.2%. The higher zinc content affects the copper concentrate quality and leads to zinc loss. In the lead concentrate, the impurity contents were zinc 3.0–4.5%, copper 0.6–1.2%, and arsenic less than 0.3%, all of which were within standard limits. In the zinc concentrate, the impurity contents were copper 0.5–0.75%, lead 0.5–0.75%, which were also within standard limits. No data for arsenic or other impurities in the zinc concentrate are available but, based on the arsenic content and distribution in the ore, it is considered that arsenic in the gold concentrate will not exceed the limits. Gold is not concentrated in any of the concentrates to a payable level. Silver is mainly concentrated in the lead concentrate, reaching payable levels in both the lead and copper concentrates, but its content in the zinc concentrate only reaches 25–40 g/t, which is below the payable threshold.

Table 11.3: Historical operation performance

Indicator Items		Unit	2022	2023	2024		Jan-July 2025
					Oxidised Ore	Sulfide Ore	
Feed Processed		t	558,762	591,188	74,904	258,271	204,603
Average Head Grades	Cu	%	0.18	0.25	0.19	0.10	0.30
	Pb	%	2.24	2.41	4.03	0.96	2.13
	Zn	%	3.95	4.33	3.82	4.31	4.84
	Ag	g/t	30.75	40.58	24.62	16.59	52.00
Contained Metal Quantity	Cu	t	1,002	1,484	145	246	610
	Pb	t	12,538	14,276	3,021	2,469	4,352
	Zn	t	22,055	25,617	2,862	11,136	9,911
	Ag	kg	17,182	23,988	1,844	4,285	10,639
Product Output	Copper concentrate	t	2,315	4,028	322	737	1,732
	Lead concentrate	t	18,707	20,375	3,135	3,656	6,576
	Zinc concentrate	t	43,405	51,097	4,515	23,085	20,797
Product Grades	Copper concentrate	Cu %	18.67	19.56	15.81	18.21	21.20
		Ag g/t	559.0	641.5	633.4	616.6	448.0
	Lead concentrate	Pb %	57.44	62.75	47.37	59.77	59.93
		Ag g/t	594.1	784.5	189.2	814.8	1265.9
	Zinc concentrate	Zn %	45.75	45.90	44.60	44.68	44.63
Contained Metal Quantity in Products	Copper concentrate	Cu t	432	788	51	134	367
		Ag kg	1,294	2,584	204	454	776
	Lead concentrate	Pb t	10,746	12,785	1,485	2,185	3,941
		Ag kg	11,113	15,984	593	2,979	8,324
	Zinc concentrate	Zn t	19,857	23,453	2,014	10,315	9,282
Metal Recovery Rate	Copper concentrate	% (Cu)	43.16	53.10	35.13	54.45	60.19
		% (Ag)	7.53	10.77	11.07	10.60	7.29
	Lead concentrate	% (Pb)	85.71	89.56	49.17	88.50	90.57
		% (Ag)	64.68	66.63	32.16	69.51	78.25
	Zinc concentrate	% (Zn)	90.03	91.55	70.36	92.63	93.65
Product Moisture Content	Copper concentrate	%	7.17	9.46		9.50	9.27
	Lead concentrate	%	8.12	8.27		10.25	8.55
	Zinc concentrate	%	6.69	6.90		6.90	7.38

Source: Huaxia

Note: [1] In 2024 and 2025, 4,251t and 58,510t of externally purchased ores were processed respectively.[2] Feed materials included direct processing plant feed and mineralised waste. The volume of mineralised waste were 204.7 kt in 2022, 217.0 kt in 2023, 11.4 kt in 2024 and nil in 1 January to 31 July 2025, respectively.

Based on test results and historical production performance, SRK forecasts the technical indicators for sulfide ores shown in Table 11.4. Using the predicted concentrate grades, copper, lead and zinc recovery rates, and the head ore grades, it is possible to calculate the concentrate yield and output. For example, lead concentrate yield = lead recovery rate × head ore lead grade ÷ lead concentrate lead grade, and lead concentrate output = head ore quantity × lead concentrate yield.

In 2024, a pre-selection workshop for mineralised wastes was established, employing a crushing-sieving-intelligent sorting process to treat ores with a particle size of -60+10 mm. The pilot production results, shown in Table 10.23, indicated a discard rate of 15.55% (relative to ROM ore) and a zinc metal loss rate of only 1.3%, demonstrating that pre-selection is feasible.

In 2024, due to a decrease in ore supply, the plant processed 75 kt of stockpiled oxidised ore and 4,251 t of externally purchased sulfide ore (included in the 258.27 kt of sulfide ore). The oxidation degree of the oxidised ore is unknown. Due to limited test data on oxidised ores, it is challenging to establish a response relationship between oxidation rate and processing indicators, making it difficult to forecast the processing indicators for oxidised ores. SRK does not recommend mixing oxidised ores with sulfide ores for processing production. Conducting processing tests on ores with different oxidation rates, determining suitable processing conditions for each, and carrying out separate, focused processing of oxidised ores is recommended.

Table 11.4: Test results, historical and forecast performance parameters

Performance parameters		Unit	[1]Test	2022	2023	2024[2]	Jan-Jul 2025[2]	Forecast[3]
Copper Concentrate Grade	Cu	%	21.95	18.67	19.56	19.78	20.67	20.0
	Ag	g/t	355.0	559.0	641.6	725.3	463.8	600
Copper Concentrate Recovery	Cu	%	54.97	43.16	53.10	49.89	58.60	52.0
	Ag	%	6.48	7.53	10.77	10.89	7.59	9.2
Lead Concentrate Grade	Pb	%	60.11	57.44	62.76	60.12	60.43	60.0
	Ag	g/t	401.0	594.1	784.5	815.7	1,202.8	700
Lead Concentrate Recovery	Pb	%	87.80	85.71	89.56	88.14	90.49	88.0
	Ag	%	58.22	64.68	66.63	75.53	75.99	66.4
Zinc Concentrate Grade	Zn	%	48.12	45.75	45.90	44.64	44.51	46.0
Zinc Concentrate Recovery	Zn	%	90.54	90.03	91.55	94.81	94.45	91.0
Raw Feed Grade	Cu	%	0.20	0.18	0.25	0.09	0.24	0.20
	Pb	%	2.72	2.24	2.41	0.92	1.77	2.20
	Zn	%	4.92	3.95	4.33	4.29	4.76	4.00
	Ag	g/t	27.41	30.75	40.58	15.36	41.94	34.0

Note：[1] Test results from Guangdong Province Comprehensive Resource Utilization Research Institute；[2] Third parties ores exclusive, [3] Forecast is based on historical performance

11.6 Conclusion and suggestions

- The Mengya'a flotation plant has a designed production capacity of 2,000 t/d, and after multiple technological upgrades, the processing nameplate capacity has reached 3,000 t/d. The flotation plant uses a traditional three-stage closed-circuit crushing process to crush ore from the mining site to less than 12 mm. To prevent clogging of the secondary cone crushers and improve crushing capacity, a vibrating screen for pre-screening has been added before the secondary crushing stage, which can be activated or bypassed as needed. The plant employs a conventional two-stage closed-circuit grinding process with a grinding fineness control of 70-75% passing through a 200-mesh sieve.
- The plant adopts a 'Cu-Pb mixed flotation – Cu-Pb Separation flotation process' to recover copper and lead, producing copper concentrate and lead concentrate. The Cu-Pb mixed flotation uses a '2-stage roughing, 3-stage scavenging, and 3-stage cleaning process'. The first roughing stage uses a flotation column, which extends the flotation time and nominally improves the Cu-Pb recovery rate. However, it also increases electricity consumption. SRK considers that the configuration of the existing flotation machine is sufficient to meet the flotation time requirements. Its contribution to the flotation performance warrants further investigation.
- The process and equipment configuration for the separation flotation of Cu-Pb mixed concentrate, zinc recovery from the Cu-Pb mixed flotation tailings, and subsequent product dewatering are reasonable.
- The use of a 'gentle reagent regime' has reduced the pH value of the recycled water, enabling 100% utilisation of the recycled water, with continuous improvements in flotation performance. SRK considers it may be possible to further optimise the reagent regime, which should be explored and refined continuously during production.
- From September to December 2024, 72,360 t of pre-concentration feed were processed with a discard rate of 15.55% (relative to ROM ore), and the zinc metal loss rate was only 1.3%, indicating that pre-selection using the intelligent sorting machine is feasible. SRK recommends improving the process monitoring and gathering more pilot production and cost data before researching the feasibility of expanding the pre-selection scale.
- In 2024, 75,000 t of oxidised ore were processed. Compared with sulfide ore, flotation performance (concentrate grade and recovery rate) showed a significant decrease, indicating that oxidised ore is more difficult to process. Due to the lack of data on the oxidation rate of the ore, it is difficult to quantify the effect of oxidation on flotation performance. SRK suggests that the oxidised ore mined be separately processed for flotation.

12 Tailings storage facilities

12.1 Introduction

The project includes one existing TSF and a new TSF. Deposition into the existing TSF began in 2011 and the TSF is expected to reach full capacity by 2026, after which the new TSF will be put into use. The construction of the new Tailings Storage Facility is planned for completion in 2026.

12.1.1 Site conditions

The elevation of the TSFs ranges from 4,600 m to 5,600 m asl, exhibiting the typical characteristics of a plateau climate: thin air, low atmospheric pressure, and low oxygen content; strong solar radiation, long daylight hours, low temperatures, a small annual temperature difference, and large daily temperature fluctuations; a distinct dry season and rainy season, with low precipitation and high evaporation rates; frequent strong winds and a long frost period.

The average annual temperature at the proposed site is -2.2°C, with the lowest temperature occurring in January at an average of -12.5°C, and the highest temperature occurring in July with an average of 9.8°C. The site's average annual rainfall is 429 mm (Table 12.1), with the maximum rainfall occurring in August, reaching 67.7 mm, and the minimum rainfall occurring in December, at 15.9 mm. The average annual evaporation can reach 1,664.5 mm. The locations of the TSFs are characterised by the development of seasonal frozen soil, with a typical depth of 1.20–1.50 m, and a very short freezing and thawing period.

Table 12.1: Monthly average temperature and precipitation – Rongdoi township, Jiali County

Item	1	2	3	4	5	6	7	8	9	10	11	12	Year (avg.)
Temperature (°C)	-12.5	-8.5	-5.8	-4.0	-1.1	-0.6	6.5	9.8	1.5	-1.2	-3.0	-6.8	-2.2
Precipitation (mm)	23.5	23.8	24.9	26.0	27.2	33.9	47.9	67.7	61.9	50.4	25.9	15.9	429

Source: Jiali County Meteorological Bureau

The basic seismic intensity of the proposed site is 8 Degrees, with a peak ground acceleration value of 0.20g. The soil type at the proposed site is medium-hard soil to hard soil, and the design seismic classification is Group II. The site is classified as Category II for building purposes, with a spectral response characteristic period of 0.45 seconds. For the new TSF site, the design peak ground acceleration for a 50-year exceedance probability at the surface with a 5% damping ratio is 185 cm/s^2.

12.1.2 Tailings characteristics

The flotation plant produces lead concentrate, copper concentrate and zinc concentrate. The process flow adopts Cu-Pb preferential flotation, Cu-Pb separation, and Cu-Pb flotation tailings for zinc recovery. The tailings generated from the zinc flotation process constitutes the final tailings of the flotation plant. The plant operates for 200 days each year, with an annual production capacity of 400 kt (2,000 t/d). The tailings yield is 86.0%, with a daily tailings output of 1,720 t. The tailings has a specific gravity of 2.9–3.18 g/cm^3. The particle size distribution is shown in Table 12.2, where -0.074

mm particles account for 65.27% of the tailings mass. Toxic leaching tests indicate that the tailings is classified as Category I general industrial solid waste.

Table 12.2: Particle size analysis test results (new TSF data)

Particle Size (mm)	Content (%)
+0.15(+100mesh）	17.14
+0.097~0.15(+160mesh-100mesh）	9.81
+0.074 mm~0.097 mm (+200mesh-160mesh）	7.79
+0.045 mm~0.074 mm (+325mesh-200mesh）	19.79
-0.045 mm（-325mesh）	45.48
Total	**100.0**

The main components of the tailings are non-metallic minerals rich in SiO_2, Al_2O_3, CaO, $MgCO_3$, and other resources. The elemental composition is shown in Table 12.3.

Table 12.3: Composition analysis of the Mengya'a tailings

Element	Pb (%)	Zn (%)	S (%)	Fe (%)	Au (g/t)	Ag (g/t)	CaF_2 (%)	Cu (%)	CaO (%)	MgO (%)	Cd (%)	Na_2O (%)
Content	0.42	0.31	8.25	13.52	0.06	7.61	0.54	0.03	13.63	3.87	<0.001	0.235
Element	**SiO_2**	**TiO_2**	**Mo**	**Sn**	**Ge**	**Ga**	**Al_2O_3**	**Sr**	**P**	**C**	**Cr**	**K_2O**
Content	46.66	0.85	<0.001	0.025	0.0006	0.001	8.87	0.0013	0.028	0.56	<0.001	0.65
Element	**Ni**	**Rb**	**Li**	**In**	**Ta**	**Nb**	**W_20_3**	**Sb**	**Bi**	**MnO_2**	**Hg**	**As**
Content	<0.001	<0.001	<0.0001	<0.001	<0.001	<0.001	<0.001	<0.002	<0.001	1.32	<0.0001	0.013

12.1.3 Tailings disposal plan

The tailings from the flotation plant is first thickened in a thickener to a weight concentration of 38%, then transported via pipeline to the tailings storage area. The tailings particles settle and are stacked in the tailings storage area. After the tailings water in the decant pond undergoes clarification, evaporation, and biological degradation, all of it, excluding evaporation losses and void retention in the tailings, is returned to the flotation plant for recycling. During the winter shutdown period, seepage from the tailings storage area is returned to the decant pond. The TSF includes a clean water diversion system and the tailings dam is constructed using the upstream method.

12.2 Existing TSF

The existing TSF is located in the downstream left bank of the Xiong Jun Jiong Yong Qu's first-level tributary, Anong Qu, upstream of the valley, 5.1 km southwest of the flotation plant. It is a valley-type TSF with a design final stacking elevation of 5,036 m and a final dam height of 76 m. The initial dam height is 40 m (the difference is a tailings stacking dam height of 36 m). The total storage capacity is 2.97 million m^3. A trial operation commenced in June 2011 together with the flotation plant, and tailings discharge was commenced. In 2014, a seepage control modification was carried out, with geotextile membranes installed at the bottom of the tailings storage area.

The initial dam is a rock-fill dam with average upstream and downstream slopes of 1:2.0. Every 10 m of dam height on the downstream slope has a 2 m-wide access road. The dam crest is 12.5 m wide, and the dam axis length is 261 m. The subsequent rise is constructed by the upstream method using tailings. The external slope has an average slope of 1:5, and the outer slope is covered with weathered sand soil to a thickness of no less than 0.5 m.

As a seepage control measure, a 1.5 mm-thick high density polyethelyene (HDPE) geomembrane is laid below the 5,050.0 m elevation in the tailings storage area, with a drainage layer placed above the membrane. A seepage interception dam is installed approximately 50 m downstream of the initial (starter) dam, with a return water pool placed in front of the interception dam. The seepage water from the geomembrane in the tailings storage area is guided into the downstream return water pool via drainage channels.

The catchment area of the existing tailings storage area is 4.83 km^2, and the land area covers 12.6 ha. The surrounding area of the tailings storage area is equipped with flood diversion ditches, which direct the diverted rainwater to the downstream valley. The tailings storage area also has sloped channels and culverts that direct clarified tailings water and rainwater to the return water pool downstream of the initial dam. The TSF is designed to withstand a 1 in 200-year rainfall event.

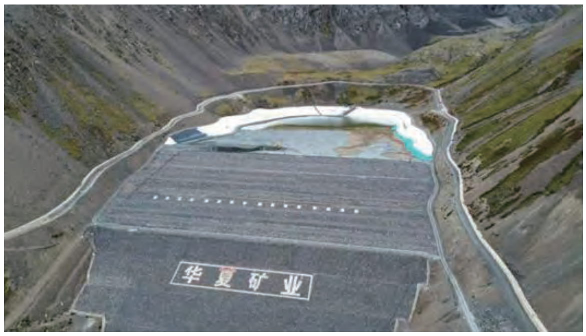
The tailings from the flotation plant is concentrated and dewatered using a φ30 tailings thickener in the plant. The slurry is then pumped by three plunger pumps through two D194x(8+8) high-pressure steel-plastic composite tailings pipelines, which transport it to the top of the tailings storage area. The tailings pipelines are approximately 5.4 km in length, with discharge points at the top of the dam. The clarified water and seepage from the tailings storage area are directed via sloped channels and culverts to the return water pool downstream of the initial dam. The water is then returned to the flotation plant through a D325x12.5 welded steel pipe return water pipeline by gravity for recycling, with a pipeline length of 5.3 km. The return water pipeline runs parallel to the tailings transport pipeline along the same route.

The TSF is equipped with online and manual monitoring and detection facilities for production, safety, and environmental protection. These include rainfall, dam surface displacement, internal horizontal and vertical dam displacement, water level monitoring, high-definition video surveillance, online monitoring centre equipment, and groundwater environmental monitoring wells.

The TSF is also equipped with access, communication, lighting, duty, safety protection and signage facilities.

Since its construction and after the seepage control modification in the pond area, the tailings pond has been in conventional use, with all safety indicators meeting design requirements and environmental protection standards meeting national regulations. The remaining storage capacity is expected to last for another 2 years.

Figure 12.1: Existing TSF



Source: SRK site visit, September 2024

Figure 12.2: Site video surveillance equipment

Source: SRK site visit, November 2024

Figure 12.3: Tailings discharge pipe at the dam crest

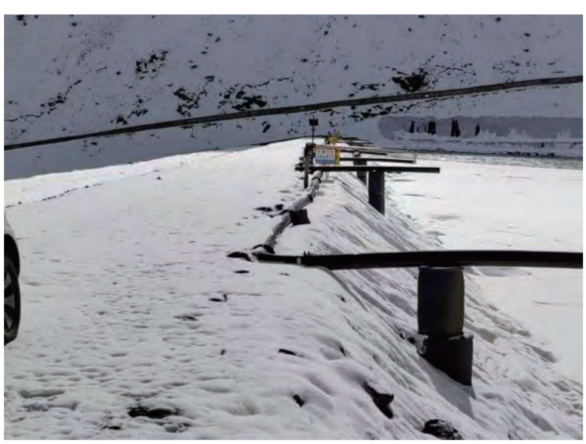


Figure 12.4: Downstream dam slope

Source: SRK site visit, November 2024

12.2.1 New TSF

At the current production capacity, the existing TSF will be full in 2 years' time. To maintain normal mine production and ensure sustainable development, a new TSF was proposed to be constructed in another tributary upstream of the Anong Qu, located northeast of the existing TSF. The construction of the new TSF is planned for completion in 2026 and is expected to commence operations in the same year.

The new TSF is administratively located within the jurisdiction of Rongdoi township, Jiali County, and is adjacent to the existing TSF. The straight-line distance between the initial dams of the two TSFs is approximately 1.05 km, and the distance from the flotation plant is approximately 5.6 km. The new TSF is a valley-type TSF with a design final stacking elevation of 5,210 m. The total dam height is 153 m, with an initial dam height of 53 m and final tailings stacking dam height of 100 m. The total storage capacity is 21.2 million m³, and the effective storage capacity is 13.78 million m³. The total storage capacity of the tailings pond represents the full geometric volume of the impoundment. However, the effective storage capacity reflects the actual usable space available for tailings deposition after considering essential safety and operational requirements. The difference between these capacities is due to design provisions such as flood retention capacity for extreme rainfall events, freeboard requirements to prevent overtopping, space occupied by tailings beach slopes during deposition and the void ratio between tailings particles. At the current flotation plant capacity of 400,000 t/a, the estimated service life is approximately 64 years, which is sufficient to cover the entire LOM of Mengya'a Mine.

The geological conditions of the new TSF site are relatively simple. The overall strata are stable, with no fault zones or other geological hazards such as large landslides, debris flows, subsidence, karst, soil caves, expansive soil, or liquefaction that could affect site stability. The foundation of the proposed dam and the upper part of the TSF comprise Quaternary residual slope debris and weathered material. The lower part consists of the Carboniferous Laigu Formation carbonaceous slate. The geology is simple and the site is stable, making it suitable for TSF construction.

The initial dam is constructed from permeable rock-fill mixed with weathered material, with upstream and downstream average slopes of 1:95 and 1:2.2, respectively. The dam crest width is 4 m, and the dam axis length is 342.6 m. The crest elevation is 5,110 m.

The new TSF dam has been constructed in stages using the upstream method, with external slopes at 1:5. Every 5 m of dam height has a 5.0 m-wide platform, and the slope between platforms is 1:4. The sub-dams are constructed by cyclone classification and bottom flow stacking. The final dam crest is expected to have an axis length of approximately 493.0 m and a crest elevation of 5,210 m. The external slope is covered with 0.3–0.5 m of soil, and cold-resistant grass or low shrubs are planted on the surface. Longitudinal and transverse drainage channel networks were constructed on the external slopes and shoulders of the dam. The dam includes a multi-layer drainage network.

Below the 5,210.0 m elevation in the tailings storage area, a 2.0 mm-thick HDPE geomembrane has been laid for seepage control, with drainage facilities placed above and below the membrane. A seepage interception dam was set approximately 300 m downstream of the initial dam, with a return water pool in front of the dam and a row of pumping wells installed behind the dam. The drainage and seepage water from the geomembrane is directed through the drainage facilities to the return water pool before the interception dam. The water is then directed downstream of the interception dam through the drainage facilities.

The catchment area above the toe of the tailings dam is 3.64 km^2, and the tailings storage area covers 0.58 km^2. A flood interception dam is set at the 5,210 m elevation upstream of the tailings storage area, with a flood diversion ditch on the west side and a vertical overflow spillway in front of the dam. The diverted rainwater is drained to the downstream valley through the main drainage tunnel. Four drainage channels are laid at different elevations in the tailings storage area, and they are connected to upstream drainage tunnels via four supporting sub-tunnels. The main tunnel is equipped with a clean water separation wall, and the clarified tailings water and rainwater are directed to the return water pool downstream of the initial dam. The TSF is designed to withstand a 1 in 500-year to 1 in 1,000-year rainfall event.

Seepage water from the tailings storage area membrane, as well as other external wastewater, is all directed into the return water pool. During production, the water is returned to the flotation plant via the return water system by gravity for recycling. During the shutdown period, it is pumped back into the tailings storage area for storage, with no external discharge allowed. Groundwater below the tailings storage area membrane is directly led to the tailings storage area's seepage interception dam through the drainage system. Groundwater quality is monitored via a group of pumping wells, and the groundwater can normally be discharged. If groundwater contamination is detected, the contaminated groundwater will be pumped back to the tailings pond.

Since the new and existing TSFs are adjacent to each other, the tailings transportation and return water pipelines still use the original pipelines. Only 780 m of new pipeline will be used to connect with the existing pipeline, and the plunger pumps will be replaced with diaphragm pumps.

The new TSF is equipped with online and manual monitoring and detection facilities for production, safety, and environmental protection. These include rainfall, dam surface displacement, internal horizontal and vertical displacement of the dam, pond water level monitoring, high-definition video surveillance, online monitoring centre equipment, and groundwater environmental monitoring wells.

Facilities for access, communication, lighting, duty, safety protection, and safety signage have been designed for the new TSF.

Seepage and stability analyses of the new tailings dam have been carried out under normal working conditions, flood conditions, and seismic conditions (8 Degree earthquake). Structural safety calculations have been made for the drainage facilities. The calculation results meet the relevant standards. Flood safety analysis has been performed for the TSF at different operational stages under a 1 in 500-year to 1 in 1,000-year rainfall event, and the results show sufficient safety margins.

The environmental impact assessment department has conducted a monthly and annual water balance analysis for the early operational period of the new TSF (at the stacking elevation of 5,110 m) under average water year conditions.

Figure 12.5: Downstream slope and return water pool – new TSF

Figure 12.6: Ongoing construction – new TSF

Source: SRK site visit, September 2024

12.2.2 Safety supervision

Design organisations have established clear monitoring indicators and countermeasures for the tailings dam, including water level, sedimentation beach length, slope, depth of the infiltration line, dam deformation, and groundwater quality. Government agencies and enterprises at various levels are equipped with safety and environmental protection supervisory bodies and personnel for tailings dam management. They have formulated specific supervisory requirements and systems, as well as emergency response plans and drill requirements for safety incidents.

12.3 Conclusion and suggestions

- The existing TSF is expected to be closed after 2 more years of operation. All current facilities are intact, and safety operational indicators are normal. Appropriate measures should be taken to ensure appropriate closure of the TSF.
- The construction of the new TSF are nearing completion with these facilities appropriately configured. The design standards, flood and hydrological calculations, structural safety and stability analysis, monitoring indicators, and safety supervision requirements all align with relevant Chinese standards and regulations, and the analysis results are credible.
- The design organisation has carefully considered all potential risks of the new tailings dam and has developed corresponding countermeasures. After implementing these measures in accordance with design specifications, the possible risk levels can be kept at a relatively low level. However, to ensure long-term safety and reliability throughout the entire lifecycle of the tailings facilities, it is essential to maintain construction quality for the initial infrastructure and to place a stronger focus on accident risks that could result in moderate consequences. Such potential incidents include heavy rain, earthquakes, tailings dam collapse, blockage of drainage facilities, changes in tailings characteristics, and changes in disposal methods. The sampling points for tailings characteristics should be representative.
- There is uncertainty regarding the quantity of discharge water from the TSF in various rainfall years. It is recommended the water balance estimation for the tailings dam be supplemented or contingency plans for non-compliant discharge water quality be established.

13 Environmental studies, permitting, and social or community impact

13.1 Objective

The objective of this chapter is to identify and or verify the existing and potential environmental, permitting, and social or community liabilities and risks, and assess any associated proposed remediation measures for the project. The project comprises an open-pit and underground project which is currently under development, processing plant and TSFs.

13.2 Process, scope and standards

The process for verifying the environmental permitting and licensing compliance and operational conformance for the project comprised a review and inspection of the project's environmental management performance against:

- Chinese national environmental regulatory requirements
- World Bank/International Finance Corporation (IFC) environmental standards and guidelines, and internationally recognised environmental management practices.

The methodology applied for this environmental review of the project consisted of a combination of documentation review, site visit and interviews with technical representatives of the Company. The site visit was undertaken on 4–7 November 2024.

13.3 Permitting

According to relevant Chinese laws and regulations, additional environmental-related operating permits are required for project construction and production. These include a Safety Production Permit, Water Extraction Permit, and Pollutant Discharge Permit.

13.3.1 Safety Production Permit

According to the Safety Production Law of the People's Republic of China and the Regulations on Safety Production Permits, the state implements a safety production permit system for mining enterprises. Before a mine can commence production or usage after construction is completed, the construction unit is responsible for organising an acceptance check of the safety facilities. Production and use can only commence after passing the acceptance check. Before starting production, the enterprise must apply to the competent authority for a Safety Production Permit.

The Safety Production Permits for the project are presented in Table 13.1.

Table 13.1: Details of Safety Production Permits

Area	**Mengya'a Lead-Zinc Mine Pb14 Orebody**
Safety Production Permit No.	[2024] 022
Issued To	Xizang Huaxia Mining Co., Ltd.
Issued By	Xizang Autonomous Region Emergency Management Bureau
Licensed Activity	Open-Pit Mining
Issue Date	15 August 2024
Expiry Date	16 August 2027
Area	**Mengya'a Lead-Zinc Mine Pb14 Orebody**
Safety Production Permit No.	[2025] 016
Issued To	Xizang Huaxia Mining Co., Ltd.
Issued By	Xizang Autonomous Region Emergency Management Bureau
Licensed Activity	Underground Mining
Issue Date	10 June 2025
Expiry Date	9 June 2028
Area	**Tanggongma TSF**
Safety Production Permit No.	[2024] 023
Issued To	Xizang Huaxia Mining Co., Ltd.
Issued By	Xizang Autonomous Region Emergency Management Bureau
Licensed Activity	TSF Operation
Issue Date	15 August 2024
Expiry Date	16 August 2027

13.3.2 Water Extraction Permit

As per the Water Law of the People's Republic of China and the Regulations on Water Extraction Permits, any unit directly drawing water resources from rivers, lakes, or underground must apply to the water administration department or the relevant river basin management agency for a Water Extraction Permit and pay water resource fees to obtain the water extraction rights. For construction projects requiring water extraction, the applicant should commission a qualified agency to prepare a Water Resource Assessment Report.

The Water Extraction Permit for the project is presented in Table 13.2.

Table 13.2: Details of Water Extraction Permit

Water Extraction Permit No.	B540621S2022-0034
Issued To	Xizang Huaxia Mining Co., Ltd.
Issued By	Xizang Autonomous Region Water Resources Bureau
Issue Date	6 August 2024
Expiry Date	8 August 2029
Water Supply Source	Surface Water and Groundwater
Water Use Allocation	360,600 m^3/year

13.3.3 Pollutant Discharge Permit

According to the Regulations on the Administration of Pollutant Discharge Permits and the Measures for the Administration of Pollutant Discharge Permits, pollutant discharge units must apply for a Pollutant Discharge Permit from the approval authority before starting production facilities or engaging in actual pollutant discharge activities. Applications for the Pollutant Discharge Permit can be submitted through the National Pollutant Discharge Permit Management Information Platform or by mail. The Pollutant Discharge Permit is valid for 5 years.

The project has applied for a fixed pollution source emission registration with registration number 91542400741912920K001X, valid from 26 November 2023 to 25 November 2028. The permitted emissions include exhaust gases, noise and industrial solid waste.

13.4 Status of Environmental Approvals

The basis of environmental policy in China is contained in the *2018 Constitution of the People's Republic of China*. Pursuant to Article 26 of the Constitution, the state protects and improves the environment in which people live and the ecological environment. It prevents and controls pollution and other public hazards. The state organises and encourages afforestation and the protection of forests.

The following are other Chinese laws that provide environmental legislative support to the *Minerals Resources Law of the People's Republic of China* and the *Environmental Protection Law of the People's Republic of China (2014)*:

- *Environmental Impact Assessment (EIA) Law (2018)*
- *Law on Prevention & Control of Atmospheric Pollution (2018)*
- *Law on Prevention & Control of Noise Pollution (2021)*
- *Law on Prevention & Control of Water Pollution (2017)*
- *Law on Prevention & Control Environmental Pollution by Solid Waste (2020)*
- *Forestry Law (2021)*
- *Water Law (2016)*
- *Land Administration Law (2019)*
- *Protection of Wildlife Law (2023)*
- *Regulations on the Administration of Construction Project Environmental Protection (2017)*.

The Environmental Impact Assessment Law of the People's Republic of China and the Regulations on the Administration of Environmental Protection for Construction Projects require that projects with potentially significant environmental impacts prepare an Environmental Impact Report, which must comprehensively evaluate the pollution generated by that project and its impact on the environment.

Before construction begins, the construction unit must submit the Environmental Impact Report or Environmental Impact Statement to the relevant environmental protection authority for approval. The Company provided SRK with the Environmental Impact Assessment (EIA) reports and approvals for the project, with specific details outlined in Table 13.3.

Table 13.3: Details of EIA reports and approvals

Project	Produced by	Production date	Approved by	Approval date
Mengyaa Lead-Zinc Mine Mining (Renovation and Expansion) Project (0.6Mt/a)	Sichuan Guohuan Environmental Engineering Consulting Co., Ltd.	May 2022	Xizang Autonomous Region Ecology and Environment Bureau	2 September 2022
Mengyaa Lead-Zinc Mine Mining and Processing (Renovation and Expansion) Project - Processing Plant (0.6Mt/a)	Zhongsheng Environmental Technology Development Co., Ltd.	March 2023	Xizang Autonomous Region Ecology and Environment Bureau	24 July 2023
Mengyaa Lead-Zinc Mine Mining and Processing Project - Phase II Tailings Storage Facility	Zhongsheng Environmental Technology Development Co., Ltd.	February 2021	Xizang Autonomous Region Ecology and Environment Bureau	23 February 2021

Under the Soil and Water Conservation Law of the People's Republic of China, construction projects in mountainous areas, hilly areas, desertified areas, and other regions prone to soil erosion, as identified in soil and water conservation planning, must prepare a Water and Soil Conservation Plan (WSCP). The Company provided SRK with the WSCP reports and approvals for the project, with specific details outlined in Table 13.4.

Table 13.4: Details of WSCP reports and approvals

Project	Produced By	Production date	Approved By	Approval date
Mengya'a Lead-Zinc Mine Mining and Processing (Renovation and Expansion) Project - Processing Plant (0.6Mt/a)	Beijing Linfengyuan Ecological and Environmental Planning & Design Institute Co., Ltd.	May 2017	Xizang Autonomous Region Water Bureau	29 June 2017
Mengya'a Lead-Zinc Mine Mining and Processing Project - Phase II Tailings Storage Facility	Xizang Hengfeng Information Engineering Co., Ltd.	April 2021	Nagqu City Water Bureau	16 September 2021

SRK also reviewed a WSCP Report for the Mengya'a Lead-Zinc Mine Mining (Renovation and Expansion) Project (0.6Mt/a) prepared by the Chongqing Water Resources and Electric Power Architectural Survey and Design Institute in October 2014, which had been approved by the relevant government authorities. However, the validity period of this report has expired. The Company informed SRK that a new WSCP is currently being prepared.

SRK reviewed above EIA reports and WSCP reports and concluded that the EIA and WSCP reports basically cover the main production facilities including mine sites, processing plants and TSFs. SRK considers that the EIA reports and WSCP reports have been prepared in accordance with relevant Chinese legal requirements and corresponding government approvals have been obtained.

13.5 Environmental and social aspects

13.5.1 Flora and fauna

The mining area is located on the southern side of the Nyenchen Tanglha Mountains in the upper reaches of the Lhasa River basin. It sits on the left bank of the Yulonglang River, a first-order tributary of the Rezheng Zangbo River. The terrain generally slopes from south to north, with elevations ranging from 4,800 m to 5,570 m. Geomorphologically, the region is predominantly shaped by alpine erosion, with relatively well-developed riverine depositional landforms.

The area is characterised by two main vegetation zones: the alpine desert zone and the alpine meadow zone. The alpine desert zone is found above 5,300 m in elevation and is devoid of soil. The surface is typically covered with weathered gravel and rock boulders, with some areas being snow-covered year-round. Vegetation is scarce in this zone and is limited to sparse grasslands in foothill areas.

By contrast, the alpine meadow zone is located below 5,300 m, primarily in the mountain valleys, where vegetation is more abundant and is estimated to cover around 50% of the region. Field surveys indicate that the vegetation coverage within the project area is approximately 30%. Dominant shrub species in the alpine meadow zone include *Rhododendron nivale* and *Rosa sericea*. In the river valleys, where moisture and soil are abundant, various herbaceous plant species such as *Artemisia* are commonly found.

The fauna in the project area is diverse, and include species such as wolves, Xizang foxes, corsac foxes, hog badgers, Xizang gazelles, bharals, black-lipped pikas, plateau hares, and Himalayan marmots. Notably, the wolf, Xizang gazelle, and bharal are classified as national second-class protected species, although these species are rarely sighted within the evaluation area. The project does not include any scenic spots, nature reserves, forest parks, or other ecologically sensitive areas.

The EIA report for the project suggests that while the construction may cause localised loss of the native vegetation, the disruption to the species composition of the plant communities in the evaluation area was deemed insignificant, and will not lead to the extinction of any plant species. The types of surface vegetation will remain largely unchanged, except the primary impact on *Artemisia*. However, most of the affected vegetation can be restored through human intervention.

The project will also cause disturbances to wildlife in certain areas, leading to changes in the composition, displacement and redistribution of wildlife communities. However, due to the low population of wildlife and the fact that the species are widespread in the region, the overall impact on animals is expected to be minimal.

To mitigate the possible effects, the EIA report also outlines several biodiversity conservation measures, such as selection of native plant species that are well-suited for disturbed soils with strong vitality, and implementation of an ecological environmental monitoring system to aid in vegetation restoration and minimise temporary land occupation during construction.

13.5.2 Water management

The project is located in a plateau with cold-temperate, monsoon-influenced climate. Summers are mild and humid, while winters are cold and dry. The region receives an average annual precipitation of 591.76 mm, with most of the rainfall occurring between June and October. Surface water in the area is replenished primarily by atmospheric precipitation and snowmelt. Most of the precipitation flows as surface runoff, with a small portion infiltrating through soil and bedrock fractures that eventually discharge at the outflow reference points. The hydrological system in the mining area includes the Yunongqu River, Xiongjunjiongyongqu River, Rezhengzangbu River, and the Lhasa River, flowing from upstream to the main stream.

The water used for production in the mining site is sourced from surface water on the eastern side of the mining area. Domestic water is sourced from a hillside spring near the living quarters. For the processing plant, freshwater is drawn from a river to the east of the plant by a fixed pump station. Drinking water is supplied by a spring near the mining headquarters. Other domestic and fire suppression water needs at the processing plant are met using surface water from the nearby river. The project is permitted by the government to withdraw up to 360,600 m^3 of water annually.

The wastewater generated by the project includes mine dewatering water, leachate from mineralised waste dumps, leachate from raw ore stockpiles, processing wastewater, and domestic wastewater. A sedimentation pond has been established downstream of the Pb14 open-pit mining site to collect dewatering water. After sedimentation, the water is re-used for production, industrial site dust suppression, and landscaping irrigation. No wastewater is discharged outside the site.

Similarly, a leachate collection pond was constructed downstream of the Pb14 mineralised waste dump, where the wastewater is re-used for dust suppression at the Pb14 mineralised waste dump. A leachate sedimentation pond, walls lined with impermeable membranes, has been constructed on the north side of the raw ore stockpile to collect and recycle processed leachate for dust suppression.

Processing wastewater mainly includes tailings overflow and reclaimed water from the TSFs, and all of which are recycled within the processing process and not discharged externally.

Domestic sewage is treated in the septic tank and then reused for landscaping or as fertiliser for grasslands within the processing plant area. The EIA report requires that during the mining phase, mine dewatering water from the Pb12 mine and leachate from the mineralised waste dump are strictly prohibited from entering the Mongyaa River and its tributaries.

The Company provided SRK with two environmental monitoring reports for the third quarter of 2024, issued in October 2024 by Xizang Jingbo Environmental Monitoring Technology Co., Ltd. The reports cover the surface water and groundwater monitoring plans of the mining site and the processing plant.

Water samples were collected on 9 September 2024 for environmental monitoring of the mining site and on 7 September 2024 for the processing plant. Sampling locations included areas upstream and downstream of the mining site, raw ore stockpile, and TSFs, as well as rivers near villages downstream. According to the analysis in the monitoring reports, all evaluation indicators for the water samples complied with the Class III standard limits specified in the Environmental Quality Standards for Surface Water (GB 3838-2002) and the Quality Standards for Groundwater (GB/T 14848-2017, Table 1).

The EIA approvals for the project require the establishment of a groundwater pollution monitoring system to monitor regional groundwater quality. If any signs of pollution are detected, the cause must be promptly investigated, and effective measures must be implemented to prevent and control the pollution. The approval specifies the installation of three groundwater monitoring wells in the Pb12 mining area and five groundwater monitoring wells around the new TSF.

Additionally, SRK recommends that anti-seepage measures be implemented in key areas such as the mine dewatering water collection pond, leachate collection pond at the mineralised waste dump, hazardous waste temporary storage facility, and maintenance workshop to reduce the risk of groundwater contamination.

13.5.3 Waste and tailings management

The waste from the initial stripping of the Pb14 orebody has been fully deposited in the construction waste dump within the mining area, with the surface compacted and levelled. During subsequent mining operations, the waste is deposited in the production mineralised waste dump. During the exploration phase of the Pb12 orebody, it is noted that several waste dumps created from adits development did not have accompanying leachate collection ponds. Some of the adit waste dumps at Pb12 will be expanded and converted into the Pb12 waste dump for the mining phase. This will include the construction of diversion ditches, sediment retention dams, and leachate collection ponds.

The waste generated by the existing processing plant primarily consists of tailings discharged from the processing process. These tailings is transported under pressure through pipelines and deposited in the TSF, where the supernatant is recycled for production use. The existing TSF is located to the southwest of the processing plant, with a total capacity of 2.97 million m^3 and is equipped with anti-seepage measures. The new TSF is located in a valley to the northeast of the existing facility, with a total capacity of 21.2 million m^3. The new TSF will feature full-scale anti-seepage treatment. HDPE geomembranes will be installed at the base and side slopes of the facility for seepage prevention. Safety monitoring measures for the new TSF include manual inspections, monitoring of dam body displacement, TSF water levels, dry beach conditions, and precipitation levels.

In September and November 2015, the Sichuan Nuclear Industry Radiation Testing and Protection Institute conducted a solid waste toxicity leaching test on the mineralised waste from the Pb14 orebody and tailings from the processing plant. According to the test results, all indicators from the acid leaching tests were below the limits specified in the Identification Standards for Hazardous Waste—Leaching Toxicity Identification (GB 5085.3-2007), classifying the mineralised waste as general industrial solid waste. Additionally, all indicators from the water leaching tests were below the maximum allowable discharge concentrations specified in the Integrated Wastewater Discharge Standard (GB 8978-1996). Therefore, the mineralised waste and tailings generated by the project are categorised as Type I general industrial solid waste.

13.5.4 Air and noise emissions

The main air pollutant generated by the project is dust produced during processes such as drilling, blasting, excavation, rock crushing, and mining operations, as well as dust from the raw ore stockpile and mineralised waste dump, dust from the crushing and screening processes, dust from the dry beach area of the TSF, and dust from transportation roads. Dust management measures for the mine site, processing plant and TSF proposed in the EIA reports mainly comprise wet drilling, water sprinkling of haul roads and raw ore stockpile, road maintenance, and using dust collectors in the processing plant. In addition, dust control nets will be used to cover the dry beach area of the TSF. Currently, dust collectors are in operation for dust control in the crushing and screening processes. Information regarding greenhouse gas emissions or a reduction plan for the project has not been sighted as part of this review.

The main sources of noise for the project are blasting noise and noise from large machinery and equipment in the open-pit mining area, such as drills, electric shovels, bulldozers, transport vehicles, fans, and other mobile equipment. Additionally, key noise sources include the feeder, vibrating screens, cone crushers, belt conveyors, flotation machines, and concentrate filter presses at the processing plant. The noise mitigation measures proposed in the project's EIA include reasonably scheduling blasting times, enhancing maintenance and upkeep of vehicles and equipment, selecting low-noise equipment, and arranging indoor placement of equipment.

The project uses wet underground operations, including water spraying and other dust suppression measures to control underground dust. Surface dust control measures include enclosing the conveyor belt corridors for ore transport, fully enclosing transfer and discharge points with spray systems, and regularly spraying water on storage yards and roads in the plant area to suppress dust. According to the noise monitoring for the third quarter of 2024 conducted by Xizang Jingbo Environmental Monitoring Technology Co., Ltd., the boundary and ambient noise levels of the project meet the Class 2 standard limits of the Emission Standards for Industrial Enterprises Noise at Boundary (GB 12348-2008) and Environmental Quality Standards for Noise (GB 3096-2008).

13.5.5 Hazardous substances management

Hazardous materials have the characteristics of corrosive, reactive, explosive, toxic, flammable and potentially biologically infectious, which pose a potential risk to human and/or environmental health. The hazardous materials will be generated mainly by the project's construction, mining, and processing operations and include hydrocarbons (i.e. fuels, waste oils, and lubricants) and oil containers, batteries, reagents, explosives, etc. The leaks, spills or other types of accidental releases of hazardous materials may have negative impact on soils, surface water, and groundwater resources.

The existing explosives magazine is located in the eastern part of the mining area. It consists of the explosives storage, detonator storage, a guard tower, a firewater pool, and surrounding walls, with a total capacity of 40 t. The explosives magazine lies approximately 2 km away from the mining headquarters, and the nearest residential area is approximately 6 km to the northwest, in Village 6, Rongdoi township. The external distance between the explosives magazine and external protection targets meets the requirements of the Blasting Safety Regulations.

The main hazardous wastes generated by the project include waste oil, and waste oil drums. These hazardous wastes should be temporarily stored in the hazardous waste storage facility and be regularly disposed of by qualified organisations. The anti-seepage measures for the hazardous waste temporary storage area should comply with the requirements of the Pollution Control Standards for the Storage of Hazardous Wastes (GB 18597-2001). SRK recommends the collected waste oil and diesel tank be stored with secondary containment, which is in line with recognised international industry management practices.

13.5.6 Occupational health and safety

SRK has reviewed the safety assessment reports, safety production procedures and environmental emergency response plan as provided by the Company, and considers the reports cover items that are generally in line with recognised Chinese industry practices and Chinese safety regulations. These documents cover the basic safety production managements for blasting, drilling, transportation, mineralised waste dumping, vehicle maintenance, crushing, screening, grinding, flotation, and TSF maintenance. SRK notes that these proposed safety management measures are the basis for the operational OHS (Occupational Health and Safety) management system/procedures.

The Company reported to SRK that no safety accidents had occurred on site in the last 3 years. SRK recommends safety records be compiled and incident analysis reports developed for possible injuries in future. The proposed reports analyse the cause of injuries and identified measures to prevent a recurrence, which are in line with international recognised occupational health and safety (OHS) accident monitoring practice.

13.5.7 Mine closure and rehabilitation

The Chinese national requirements for mine closure are covered under Article 21 of the M*ineral Resources Law of People's Republic of China (2023)*, the Rules for Implementation of the Mineral Resources Law of the People's Republic of China, the *Mine Site Geological Environment Protection Regulations (2019)*, and the *Land Rehabilitation Regulation (2011)* issued by the State Council. In summary, these legislative requirements cover the need to conduct land rehabilitation, to prepare a geological environmental protection and reclamation plan, and to submit it for assessment and approval. In addition, the Company should establish a mine geological environment treatment and restoration fund account.

SRK has sighted the geological environmental protection and land reclamation plan for the Mengya'a mine produced in August 2023. The plan includes the management of mine geological disasters, land reclamation in the mining area, aquifer damage restoration, soil and water environmental pollution remediation, mining geological environment monitoring, and land reclamation monitoring in the mining area. The construction cost for the mine geological environment protection proposed in the plan is 904,700 RMB, while the construction cost for the land reclamation project is 1,631,300 RMB. In addition, the Company prepared land reclamation reports for both the existing TSF and the new TSF, both of which were completed in December 2020.

According to the land reclamation reports, the estimated static total investment for land reclamation of the existing TSF is RMB 6.3788 million, while that for the new TSF RMB 25.1313 million.

13.5.8 Social considerations

The Mengya'a mine is located approximately 120 km southwest of Jiali County in Nagqu Prefecture, Xizang, and is within the jurisdiction of Rongdoi township. The mine is approximately 25 km from Rongdoi township and is easily accessible by road. Rongdoi township is connected to Lhasa by a road that extends approximately 220 km southwest through Linzhou County.

The mining area is situated in a pastoral region with several scattered pastures. The population is sparse, with most residents being Xizang. The local economy is underdeveloped, and the main sources of income for residents include animal husbandry and the collection of wild medicinal plants such as cordyceps and fritillaria.

The mining area does not intersect with any registered nature reserves, historical and cultural sites, or major scenic areas. The nearest residents to the project live in scattered households in Village 6 of Rongdoi township, with a total population of approximately 100 people. These households are located 3.75 km north of the mining headquarters and 3.95 km northeast of the Pb12 orebody industrial site. Domestic water is supplied through a centralised water supply system by the government, with the water drawn from a spring on the right bank downstream of the Mengya'a mining area.

Public consultation activities were included during the EIA process for the project. No objections were received during the announcement period. Stakeholder engagement plans should be ongoing throughout the mining development process. Effective and continuous engagement with stakeholders ensures transparency, builds trust, and helps address concerns that may arise during the various phases of the project.

14 Capital and operating costs

14.1 Capital cost

The project's historical and forecast capital cost is summarised in Table 14.1. Between 2022 and July 2025, a total capital cost of RMB 446.4 million was incurred.

In 2022, capital cost amounted to RMB 82.1 million, primarily allocated to the construction of the new TSF (RMB 51.3 million) and the Pb12 underground development (RMB 14.9 million). In 2023, capital costs were increased to RMB 219.5 million, driven by continued investment in the new TSF (RMB 140.2 million) and land use rights (RMB 53.3 million). In 2024, capital cost decreased to RMB 89.1 million, with spending focused on the continued construction of the new TSF (RMB 67.5 million).

The forecast capital cost for the period August 2025–2035 shows phased investments in underground development, infrastructure and sustaining capital.

Peak investment is expected to occur in the period of August-December 2025, with total capital cost expected to reach RMB 68.6 million, driven by the Pb12 underground development (RMB 13.4 million) and continued expenditure on the new TSF (RMB 52.6 million). Capital cost are projected to decline after 2025. Between 2026 and 2030, annual costs are expected to range between RMB 22.2 million and RMB 56.2 million, with this expenditure focused on processing equipment upgrades (RMB 10.5-25.2 million in 2027), underground development (RMB 9.8-31.5 million). Between 2031 and 2035, capital cost is forecast to stabilise at RMB 18.9-38.0 million per year, primarily allocated to sustainable capital (RMB 2.7-3.6 million annually) and the Pb12 underground development. The forecast capital cost includes a 15% contingency.

This historical and forecast capital cost profile reflects the project's upfront infrastructure costs, followed by a gradual reduction in capital spend as operations shift to sustaining investments. SRK considers the forecast capital cost to be reasonable and achievable.

Table 14.1: Historical (2022-July 2025) and forecast (August 2025-35) capital cost (RMB million)

Cost Centre	2022	2023	2024	Jan-Jul 2025	Aug-Dec 2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Pb12 UG development	14.9	14.7	21.1	49.1	13.4	9.8	26.6	31.0	31.5	26.1	16.2	25.0	33.4	33.8	34.4
Power infrastructure and access roads	0.9	-	-	-	0.3	-	-	-	-	-	-	-	-	-	-
Auxiliary facilities	2.6	3.5	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing equipment upgrade	-	-	-	-	-	10.5	14.7	25.2	15.1	14.5	-	-	-	-	-
Workforce facilities upgrade	2.2	5.1	0.0	-	-	-	-	-	-	-	-	-	-	-	-
Retaining wall construction	-	-	-	-	1.0	0.4	-	-	-	-	-	-	-	-	-
Existing TSF	0.8	0.0	0.0	-	1.2	-	-	-	-	-	-	-	-	-	-
New TSF	51.3	140.2	67.5	5.8	52.6	-	-	-	-	-	-	-	-	-	-
Land use rights and other rights of use	9.2	53.3	0.2	0.5	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital	-	-	-	-	-	-	-	-	-	-	2.6	3.0	3.3	3.5	3.6
Other	0.3	2.7	0.3	0.3	-	-	-	-	-	-	-	-	-	-	-
Total	82.1	219.5	89.1	55.7	68.6	22.2	41.2	56.2	46.7	40.6	18.9	28.1	36.7	37.3	38.0

Source: Zhihui, 2025

Note: The capital cost is directly associated with the Project and includes expenditures related to the development, construction and operation of the open-pit mine and underground project. Capital costs related to headquarters and other non-project-specific expenditures are excluded.

14.2 Operating cost

The project's historical operating cash costs (2022–July 2025) are summarised in Table 14.2.

Total operating expenditure increased from RMB 224.5 million in 2022 to RMB 249.9 million in 2023, primarily driven by higher mining costs during that year. In 2024, expenditures declined significantly to RMB 130.9 million, reflecting a reduced volume of ore mined from the open-pit operations.

Processing operating cash costs remained stable at around RMB 78.5–78.9 million between 2022 and 2023 but decreased in 2024, when the processing of stockpiled oxide ores contributed to lower processing costs of RMB 53.5 million. During the period from January to July 2025, the total operating cash cost amounted to RMB 98.9 million, which excluded the cost of externally purchased ore.

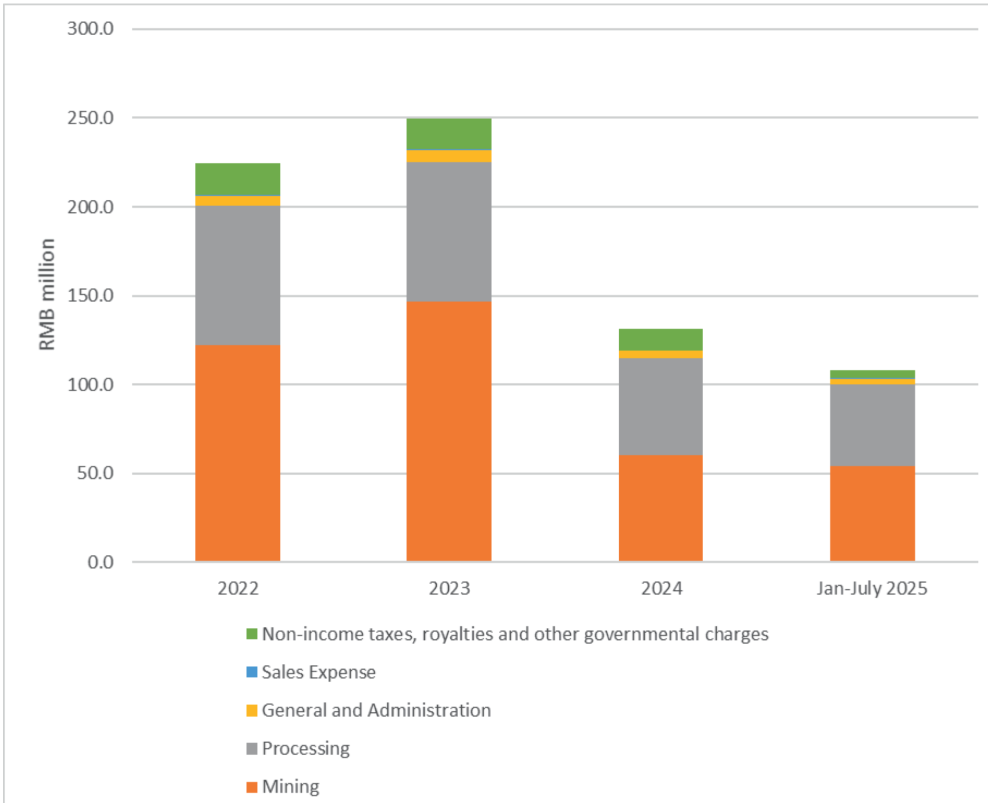
Table 14.2: Historical operating cash cost (2022–July 2025) (RMB million)

Operating cash cost by types	**2022**	**2023**	**2024**	**Jan-Jul 2025**
Mining	121.9	146.3	60.5	54.0
Processing	78.5	78.9	53.5	37.2
General and Administration	5.7	6.8	4.6	3.1
Sales expense	1.0	0.4	0.2	0.3
Non-income taxes, royalties and other governmental charges	17.3	17.4	12.1	4.3
Total	224.5	249.9	130.9	98.9
Operating cash cost by activity				
Contract mining	96.3	116.0	41.6	40.3
Workforce employment	23.7	27.0	24.0	17.4
Consumables	28.1	29.0	21.0	17.1
Fuel, electricity, water and other services	38.4	36.7	18.6	6.3
On-site and off-site administration	5.7	6.8	4.6	3.1
Environmental protection and monitoring	0.0	0.9	0.6	0.6
Transport of workforce	-	-	-	-
Product marketing and transport	15.0	16.0	8.3	9.7
Non-income taxes, royalties and other governmental charges	17.3	17.4	12.1	4.3
Total	**224.5**	**249.9**	**130.9**	**98.9**

Source: Zhihui, 2025

Note: A negligible amount of workforce transportation cost has been incurred and has been accounted for within the on-site and off-site administration costs. Cost of externally purchased ore is excluded. Figures may not add up to the total due to rounding.

Figure 14.1: Historical cash operating cost (2022–July 2025)



Source: Zhihui, 2025

Note: Cost of externally purchased ore is not included

The forecast operating costs for the project (August 2025–2035) are summarised in Table 14.3. The forecast, as prepared by Zhihui, is based on historical operating costs, contractual agreements with mining contractors, and government-prescribed resource tax rates and other applicable tax rates. Inflation has also been factored. Operating costs exclude expenses related to the purchase and processing of third-party ores, as well as the processing of ores from the ore sorter, which SRK considers to be at the trial stage.

Mining activities dominate the operating cost profile, influenced by the stripping ratio (open pit) and development costs (underground). Processing operating costs are driven primarily by energy consumption and consumables, while resource taxes and other applicable taxes consistently account for 10–16% of total operating costs.

Between August 2025 and 2035, total operating cash costs are forecast to range from RMB 140.3 million to RMB 285.4 million. Operating cash unit costs are expected to range from RMB/t 473 ore to RMB/t 798 ore. The operating cash unit cost for copper concentrates is forecast to range from RMB/t 6,865 concentrate to RMB/t 11,483 concentrate, while lead concentrate unit costs range from RMB/t 4,669 concentrate to RMB/t 7,761 concentrate. Zinc concentrate unit costs are planned to vary between RMB/t 3,311 concentrate and RMB/t 5,324 concentrate, with fluctuations largely driven by variability of grades over time.

Table 14.3: Physicals and forecast operating cost (August 2025–2035)

Production profile	Unit	Total LOM	Aug-Dec 2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Open Pit													
Ore	kt	1438	113	199	206	200	214	197	208	101	-	-	-
Waste	kt	8534	1640	3744	2029	384	329	248	90	70	-	-	-
Underground													
Ore	t	10623	111	199	197	199	198	200	193	298	398	403	409
Processing	t												
Feed ore	t	12061	224	397	403	399	412	397	402	399	398	403	409
Pb	%	2.71	2.17	2.25	3.18	2.04	2.37	3.01	2.06	3.29	2.97	3.83	4.49
Zn	%	4.23	4.13	4.68	4.74	3.94	4.53	5.09	3.77	4.28	3.32	4.27	4.69
Cu	%	0.20	0.17	0.19	0.18	0.17	0.16	0.15	0.11	0.18	0.20	0.22	0.25
Ag	g/t	32.03	18.27	22.52	23.06	18.35	17.79	19.76	14.06	25.11	32.32	33.40	38.70
Products													
Copper concentrate	WMT	67,852	962	2,069	1,986	1,843	1,739	1,500	1,172	2,076	2,129	2,526	2,752
Lead concentrate	WMT	510,026	6,203	13,968	18,771	11,691	13,165	16,307	11,807	20,859	17,966	24,645	28,161
Zinc concentrate	WMT	1,539,805	23,721	51,352	56,243	42,751	49,404	55,248	41,323	54,742	44,171	58,982	65,592
Total	WMT	2,117,684	30,886	67,389	77,001	56,285	64,308	73,054	54,301	77,677	64,266	86,154	96,505
Operating cash cost by types													
Mining	RMB million	4747.1	75.1	141.4	113.6	86.2	89.0	88.7	87.8	106.5	124.4	129.7	135.5
Processing	RMB million	3343.0	46.4	72.4	75.1	72.9	78.2	80.1	79.6	85.1	84.6	92.0	97.5

Production profile	Unit	Total LOM	Aug-Dec 2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
General and Administration	RMB million	254.5	5.1	5.4	5.5	5.6	5.7	5.9	6.1	6.3	6.4	6.6	6.8
Sales Expense	RMB million	10.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.3	0.3	0.3
Non-income taxes, royalties and other governmental charges	RMB million	1268.7	13.4	29.0	31.7	24.0	28.8	33.3	25.6	34.6	28.7	39.4	45.2
Total	RMB million	9623.3	140.3	248.4	226.2	189.0	201.9	208.2	199.3	232.7	244.4	268.0	285.4
Operating cash cost by activity													
Contract mining	RMB million	3392.9	55.0	112.2	83.8	56.1	57.3	56.9	54.8	72.6	89.6	93.5	97.9
Workforce employment	RMB million	1246.1	25.1	26.4	26.7	27.2	28.0	28.9	29.7	30.6	31.5	32.5	33.5
Consumables	RMB million	1096.8	13.4	23.8	24.5	24.7	26.2	26.0	27.1	27.7	28.5	29.7	31.1
Fuel, electricity, water and other services	RMB million	1797.3	21.4	38.8	40.1	40.5	43.1	42.7	44.5	45.5	46.6	48.7	50.9
On and off-site administration	RMB million	254.5	5.1	5.4	5.5	5.6	5.7	5.9	6.1	6.3	6.4	6.6	6.8
Environmental protection and monitoring	RMB million	44.7	0.9	1.2	1.0	0.9	0.9	1.0	1.0	1.1	1.2	1.2	1.3
Transport of workforce	RMB million	-	-	-	-	-	-	-	-	-	-	-	-
Product marketing and transport	RMB million	522.4	5.9	11.6	12.8	10.0	11.8	13.6	10.5	14.3	12.0	16.4	18.7
Non-income taxes, royalties and other governmental charges	RMB million	1268.7	13.4	29.0	31.7	24.0	28.8	33.3	25.6	34.6	28.7	39.4	45.2
Contingency allowances	RMB million	-	-	-	-	-	-	-	-	-	-	-	-
Total	RMB million	9623.3	140.3	248.4	226.2	189.0	201.9	208.2	199.3	232.7	244.4	268.0	285.4
Operating cash unit cost													
Mining	RMB/t ore	394	336	356	282	216	216	223	219	267	313	322	331
Processing	RMB/t ore	277	207	182	186	182	190	202	198	213	213	228	238

Production profile	Unit	Total LOM	Aug-Dec 2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
General and Administration	RMB/t ore	21	23	14	14	14	14	15	15	16	16	16	17
Sales expense	RMB/t ore	1	1	1	1	1	1	1	1	1	1	1	1
Non-income taxes, royalties and other governmental charges	RMB/t ore	105	60	73	79	60	70	84	64	87	72	98	110
Total	RMB/t ore	798	627	625	561	473	490	524	496	583	615	665	697
Co-product operating cash cost													
Copper concentrate	RMB million	717.6	9.1	15.9	12.9	12.8	11.5	9.5	9.4	14.0	18.6	18.1	19.2
Lead concentrate	RMB million	3533.1	44.2	80.8	85.3	61.5	62.9	69.9	64.8	97.2	111.2	121.6	132.3
Zinc concentrate	RMB million	5372.7	86.9	151.7	128.0	114.7	127.5	128.8	125.2	121.5	114.6	128.3	133.9
Co-product operating cash unit cost													
Copper concentrate	RMB/t concentrate	11,483	10,314	8,348	7,035	7,548	7,171	6,865	8,691	7,339	9,469	7,801	7,559
Lead concentrate	RMB/t concentrate	7,546	7,761	6,305	4,952	5,728	5,208	4,669	5,978	5,076	6,744	5,373	5,119
Zinc concentrate	RMB/t concentrate	5,324	5,073	4,142	3,442	3,782	3,569	3,311	4,257	3,606	4,457	3,770	3,598

Source: Zhihui and SRK analysis, 2025

Note: operating cost in nominal terms, WMT: wet metric tonne. Operating costs exclude expenses related to the purchase and processing of third-party ores, as well as the processing of ores from the ore sorter.

14.3 Economic viability analysis

The project's economic viability was assessed, incorporating capital and operating costs, the production schedule, and the processing of fresh and oxide ores from both open pit and underground operations and the processing recoveries based on the historical performance (Table 11.4). This assessment considered the production of copper, lead and zinc concentrates from 1 August 2025 to the end of LOM (2055 for a total of 31 years).

A 10% nominal discount rate is derived from a 1.65% risk-free rate (10-year PRC Government Bond) , 2–4% mining project risk and 2–4% country risk, which is considered by SRK as appropriate.

Using a base case scenario with forecast metal prices from SMM[1] and the after-tax net present value (NPV) at a 25% corporate tax rate and a 10% nominal discount rate yielded a positive result as at 1 August 2025.

It is important to note that this economic viability analysis is limited to evaluating the technical and operational feasibility of the Project. It does not consider financial metrics such as debt, interest or IPO expenses. Additionally, it does not evaluate the project's profitability, market value, investor returns, or overall financial viability.

Additionally, a post-tax sensitivity analysis was conducted to evaluate the impact of changes to the sale prices (copper, lead and zinc), operating and capital costs.

The analysis shows:

- 1% increase in copper sale price results in a 0.09% increase in NPV.
- 1% increase in lead sale price results in a 0.47% increase in NPV.
- 1% increase in zinc sale price results in a 0.84% increase in NPV.
- 1% increase in operating cost leads to a 0.46% decrease in NPV.
- 1% increase in capital cost causes a 0.09% decrease in NPV.

Table 14.4: Post-tax sensitivity analysis

Variance	Cu Sale Price	Pb Sale Price	Zn Sale Price	Opex	Capex
25%	4,147	4,530	4,906	3,594	3,966
20%	4,129	4,435	4,736	3,686	3,984
15%	4,111	4,340	4,566	3,779	4,002
10%	4,092	4,245	4,396	3,871	4,020
5%	4,074	4,151	4,226	3,963	4,038
0%	4,056	4,056	4,056	4,056	4,056
-5%	4,037	3,961	3,886	4,148	4,074
-10%	4,019	3,866	3,716	4,241	4,092
-15%	4,001	3,771	3,546	4,333	4,110
-20%	3,982	3,677	3,376	4,425	4,128
-25%	3,964	3,582	3,206	4,518	4,146

Source: SRK

Note: The commodity price forecast is based on data from SMM presented in the industry view of this prospectus.

Figure 14.2: Post-tax sensitivity analysis

Source: SRK

Table 14.5 presents the NPVs of the project's post-tax cash flows at different discount rates, expressed in RMB. The derived NPVs evident across all discount rates confirm that the project is economically viable and support the declaration of the Ore Reserve as outlined in Section 9 of this Report.

Table 14.5: Post-tax NPVs at different discount nominal rates (RMB million)

6%	8%	10%	12%	14%
3,975	5,264	4,056	3,204	2,590

Source: SRK

15 Conclusion

Zhihui's Mengya'a Lead and Zinc Project is located approximately 120 km northeast of Lhasa, the capital of the Xizang Autonomous Region, China. The project has been operating successfully via open-pit mining means since 2007. The project is covered by a granted mining licence, spanning an area of 4.4544 km^2 with a permitted annual production capacity of 400 kt. The mining licence is surrounded by a granted exploration licence covering 58.45 km^2.

The project lies within the Northern Gangdese Metallogenic Belt (NGMB) which is characterised by Late Carboniferous to Jurassic carbonate and clastic sedimentary rocks intruded by granite porphyry. Base metal mineralisation primarily consists of layered or lenticular skarn deposits containing vein-hosted to massive sphalerite and galena.

The discovery of mineralisation during reconnaissance exploration in 2002 led to comprehensive exploration efforts from 2004 to 2021, which identified a number of base metal mineralised prospects and two key deposits (Pb14 and Pb12 deposits).

As at 31 July 2025, the project was estimated to host a Measured Mineral Resource of 6,726 kt at 2.70% Pb, 4.73% Zn, in addition to an Indicated Mineral Resource of 7,198 kt at 3.98% Pb, 5.00% Zn, and an Inferred Mineral Resource of 2,888 at 2.97% Pb, 3.87% Zn, which have collectively been estimated and reported in accordance with the JORC Code (2012).

Mining operations are currently focused on open-pit extraction at the Pb14 deposit, with future underground mining planned for the Pb12 deposit. Collectively, these two mining operations are planned to supply 400 kt of ore annually to the processing plant.

As at 31 July 2025, the Pb14 deposit was estimated to host Proved and Probable Ore Reserves of 1,438 kt at 0.69% Pb and 4.90% Zn, while the Pb12 deposit contains Proved and Probable Ore Reserves of 10.623 kt at 2.99% Pb and 4.14% Zn, which have been reported in accordance with the JORC Code (2012). At the current designed production capacity, the Pb14 OP is estimated to have an LOM of 8 years and the Pb12 UG is estimated to have an LOM of 31 years.

The processing plant, with a current processing nameplate capacity of 3,000 t/d, uses a Cu-Pb mixed-separation flotation and Zn flotation process to produce Cu, Pb and Zn concentrates with an average concentrate grade of 20%, 60% and 46%, respectively.

Key infrastructure includes access to National Highway 349 for concentrate transportation, water sourced from a nearby river and mountain springs, a base camp complex, and two TSFs. The existing TSF will be supplemented by a new TSF once the existing TSF reaches full capacity in 2 years' time. Together, these facilities ensure sufficient tailings storage for the entirety of the project's expected lifespan. No significant environmental or social risks have been identified that have the potential to disrupt ongoing mining and processing operations.

SRK conducted a detailed technical review of the project, evaluating historical process designs, technical studies, production performance and cost forecasts. SRK's assessment confirmed the processing plant designs and infrastructure plans as being technically sound, with open-pit operations well established and underground mining plans progressing. The exploration potential of various prospects within the exploration licence provides a solid foundation for the project's future growth. SRK concluded the project is economically viable, reflecting a balanced and well-executed approach to resource extraction, processing, and environmental management. The integration of phased infrastructure development, including the planned commissioning of the new TSF, underscores the project's strategic planning for long-term sustainability.

16 Risk assessment

This section presents risks that were identified and described in sections above. Risks have been classified from major to minor, defined as follows:

- **Major risk**: The factor poses an immediate danger of a failure which, if uncorrected, will have a material effect (>15% to 20%) on the project cash flow and performance and could potentially lead to project failure.
- **Moderate risk:** The factor, if uncorrected, could have a significant effect (10% to 15–20%) on the project cash flow and performance unless mitigated by some corrective action.
- **Minor risk:** The factor, if uncorrected, will have little or no effect (<10%) on project cash flow and performance.

In addition to the risk factor, the likelihood of risk must also be considered. Likelihood of occurrence within a 7-year timeframe can be considered as:

- likely: will probably occur.
- possible: may occur.
- unlikely: unlikely to occur.

Table 16.1: Risk assessment matrix

Likelihood	Consequence		
	Minor	**Moderate**	**Major**
Likely	Medium	High	High
Possible	Low	Medium	High
Unlikely	Low	Low	Medium

The results of the risk assessment rating are presented in Table 16.2. The rating of the risks is presented before implementation of control recommendations.

Table 16.2: Project risk assessment

Section	Risk	Description	Control Recommendations	Likelihood	Consequence	Rating
Mineral Resource and Ore Reserve	Relationship between density and combined Pb+Zn grade	Analysis of 316 samples from Pb12 and Pb14 deposits showed no significant correlation in Pb12 oxide ore, but a marginal correlation in fresh ore.	Conduct a detailed study to evaluate correlations, focusing on the Pb12 deposit.	Possible	Minor	Low
Mining	Poor production plan	Failure to meet targets due to mine access delays or equipment shutdown.	Implement short-term planning to identify/resolve delays and meet ore targets.	Unlikely	Moderate	Low
	High altitude challenges	Operational difficulties (e.g. health risks, equipment inefficiency) due to altitude.	Implement acclimatisation programs, emergency medical facilities, and equipment modifications.	Possible	Moderate	Medium
	Poor project implementation	Construction delays due to resource shortages or poor planning.	Monitor project plans, establish contingency budgets, and standby resources.	Unlikely	Moderate	Low
	Unfavourable mine design	Existing infrastructure (designed for exploration) may not support mining.	Conduct infrastructure assessments, upgrade ventilation and safety systems and ensure regulatory compliance is achieved.	Possible	Moderate	Medium
	Equipment shortage or shutdown	Insufficient equipment due to unstable ROM production.	Implement maintenance schedules and inventory management.	Unlikely	Moderate	Low
	Lack of skilled labour	Labour shortages leading to operational failures.	Train local workers, retain skilled labour, and hire domestic experts.	Possible	Moderate	Medium
	Adverse micro-geological conditions	Faults/disturbances causing disruption to mine plans.	Implement real-time geological mapping and update underground working plans.	Unlikely	Moderate	Low
	Bad weather interruptions	Operational shutdowns due to extreme weather.	Use weather-resistant facilities, maintain ROM stockpiles, and plan for delays.	Possible	Minor	Low
Processing	Low metal recoveries for oxidised ore	Concentrator optimised for sulfide ore struggles with oxidised ore.	Adjust reagent regimes to improve oxidised ore beneficiation.	Possible	Minor	Low
	Recycled water impacts	Recycled water reduces Cu-Pb-Zn recovery rates.	Optimise reagent regimes to minimise water quality impacts.	Unlikely	Moderate	Low
Tailings storage facilities	Earthquakes	Seismic events exceeding design resistance capacity.	Monitor warnings, establish emergency plans, and prevent secondary disasters.	Unlikely	Moderate	Low
	High altitude and cold weather	Winter conditions cause tailings discharge issues.	Halt production in extreme cold, control discharge speed, and follow protocols.	Unlikely	Minor	Low
	Flood or seepage accidents	Damage to facilities due to the use of substandard materials, poor structural integrity, inadequate construction quality, or insufficient chute sealing in flood discharge and seepage prevention systems.	Improve inspections, maintenance, and quality control.	Possible	Moderate	Medium

Section	Risk	Description	Control Recommendations	Likelihood	Consequence	Rating
	Damaged impermeable layer	Membrane damage during phased installation.	Improve supervision and maintenance during construction.	Possible	Minor	Low
	Changes in tailings discharge volume or tailings characteristics.	Changes in mining, mineral processing, or tailings treatment processes can result in an increased annual volume of tailings storage or a decline in tailings quality. Substantial variations may lead to a reduction in dam strength.	Improve supervision and implement countermeasures promptly.	Unlikely	Moderate	Low
Environment and social	Absence of key licences and permits	New permits required for refurbishment and expansion project.	Proactively identify and apply for all required permits/licences before project commencement and maintain compliance monitoring.	Unlikely	Moderate	Low
	Habitat damage (flora/fauna)	Land disturbance from infrastructure development/production, leading to biodiversity loss.	Minimise land footprint, implement phased land reclamation and restore habitats with native vegetation post-disturbance.	Possible	Moderate	Medium
	Water pollution (surface/ground)	Untreated mine dewatering, leachate from mineralised waste or stockpiles or beneficiation wastewater leaks.	Treat and re-use dewatering water, install leachate collection systems, prevent leaks via containment and monitor water quality quarterly.	Possible	Moderate	Medium
	Air pollution	Dust emissions from mining, transportation, and beneficiation activities.	Spray water on roads/sites, install dust suppression systems (e.g. mist cannons) and enclose processing facilities.	Possible	Minor	Low
	Hazardous substance pollution	Materials that are corrosive, reactive, explosive, toxic and flammable may pose a potential risk to environmental health.	Store hazardous materials in secure, dedicated facilities; register and track waste, and engage licensed contractors to carry out disposal.	Unlikely	Moderate	Low
	Community opposition	Complaints/resistance from local population due to project impacts (noise, dust, land use).	Establish a grievance redress mechanism, fund community infrastructure (e.g. water supply, schools), and engage in transparent stakeholder dialogue.	Unlikely	Moderate	Low
Capital and operating costs	High operating costs	Costs exceed forecasts, reducing profits.	Secure long-term contracts with suppliers/contractors.	Unlikely	Moderate	Low
	Product price volatility	Declining demand reduces prices.	Optimise processes, reduce costs, or scale down unprofitable operations.	Possible	Moderate	Medium

Source: SRK

Closure

This report, Independent Technical Report on the Mengya'a Lead and Zinc Project, was prepared by

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This signature has been scanned. The author has given permission to its use for this document. The original signature is held on file

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All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.

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Table 1 - JORC Code 2012

Table 1 – JORC Code 2012

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	■ Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as downhole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling. ■ Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. ■ Aspects of the determination of mineralisation that are Material to the Public Report. ■ In cases where 'industry standard' work has been done, this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases, more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.	The analytical results used to derive the Mineral Resource estimate at the Mengya'a Project were collected from two exploration periods: the historical 2004–2021 exploration (2004 and 2006–2021exploration), and the 2024 exploration. ■ In the 2004–2021 exploration: All drill holes were completed using XY-2 and XY-4 drill rigs. All drill holes collars begin with PQ size to penetrate the topsoil, followed by HQ size for deeper drilling. Sample intervals were generally 1.5 m. ■ In 2024 exploration: Twinned holes were drilled using XY-4 drill rigs on pit bench floor. Core samples were extracted at PQ and HQ size. Diamond core samples were collected by sawing in half, lengthwise, and were considered representative. Sample intervals were generally 1.5 m. Trench samples collected from the pit bench face and underground channel samples collected in drives were all continuous channel intervals of consistent width, depth and length of approximately 10 cm × 5 cm × 1.5 m, channelled either by chisels or saws.
Drilling techniques	■ Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).	■ In the 2004–2010 exploration period, drilling was conducted by PQ and HQ standard tube diamond core. Core was not oriented. The core samples collected from holes drilled prior to 2010 (70 holes) were not retained/preserved. ■ In the 2011–2021 exploration period, drilling was conducted by PQ and HQ standard tube diamond core. Core was not oriented. All core samples are meticulously boxed in accordance with established regulations. Each core box is clearly labelled in red paint with the drill hole number, box number, and the start and end depths. ■ In the 2024 exploration, drilling was conducted by PQ and HQ standard tube diamond core. Core was not oriented. All core samples are meticulously boxed in accordance with established regulations. Each core box is clearly labelled in red paint with the drill hole number, box number, and the start and end depths. Once boxed at the drill site, the core samples are transported to the core storage facility.
Drill sample recovery	■ Method of recording and assessing core and chip sample recoveries and results assessed. ■ Measures taken to maximise sample recovery and ensure representative nature of the samples. ■ Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain	■ In the 2004 period, core recovery ranged from 71% to 100%, with an average recovery of 78%. ■ In the 2006–2010 period, core recovery ranged from 69.1% to 98.4%, with an average recovery of 85%. ■ In the 2011–2018 period, core recovery ranged from 80.8% to 94.8%, with an average of

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
	of fine/coarse material.	recovery 90%. ▪ In the 2019–2021 period, core recovery ranged from 73.8% to 98.7%, with an average recovery of 91%. ▪ In the 2024 exploration, core recovery ranged from 95% to 100%, with an average recovery of 98%. ▪ No significant correlation between recovery and grade was evident.
Logging	▪ Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. ▪ Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography. ▪ The total length and percentage of the relevant intersections logged.	▪ For the 2004–2021 and 2024 exploration, the drill hole core was logged by onsite geologists, with detailed documentation covering lithology, mineral composition, structural features, texture, vein type, oxidation levels, alteration minerals, and rock quality designation (RQD) prepared. Intervals do not cross lithological or mineralisation boundaries. ▪ All hole cores were logged to an adequate level of detail to support Mineral Resource estimation.
Sub-sampling techniques and sample preparation	▪ If core, whether cut or sawn and whether quarter, half or all core taken. ▪ If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry. ▪ For all sample types, the nature, quality and appropriateness of the sample preparation technique. ▪ Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. ▪ Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. ▪ Whether sample sizes are appropriate to the grain size of the material being sampled.	In the 2004–2021 and 2024 exploration program: ▪ Diamond core samples were collected by sawing in half, lengthwise, and were considered representative. In the 2004–2021 exploration program: ▪ The 2004–2010 samples were analysed by the Xizang Autonomous Region Geological and Mineral Exploration and Development Bureau Central Laboratory (Xizang Centra Laboratory); the 2011–2012 and 2017–2021 samples were analysed by the Lhasa Jinzang Rock and Mineral Analysis and Testing Co., Ltd. (Jinzang Laboratory), also known as Lhasa Rock and Mineral Analysis and Testing Co., Ltd., and the 2014–2015 samples were analysed by the Yanjiao Central Laboratory of the North China Nonferrous Geological Exploration Bureau (Yanjiao Central Laboratory). ▪ Sample preparation at the various laboratories adhered to the prevailing Chinese standard protocols. Initially, the samples were crushed to a size of 2 mm using a jaw crusher. They were then pulverised to a fine particle size of 140 mesh (106 μm). Following this, the powdered samples were thoroughly blended and divided into two 500 g aliquots: a primary sample and a duplicate sample. The primary samples were further pulverised to a finer particle size of 200 mesh (74 μm). Finally, pulp samples weighing 20 g each were analysed using atomic absorption spectroscopy (AAS) to determine the concentrations of Pb, Zn, Cu and Ag. ▪ To ensure the accuracy for samples with a Pb+Zn grade greater than 20%, the titration method was also used. ▪ In 2014–2015 exploration period, approximately 10.54% of pulp duplicates were inserted as internal quality control measures and

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
		approximately 4.85% duplicates were sent for inter-laboratory checks. ▪ In the 2017–2018 exploration period, approximately 11.46% of pulp duplicates were inserted as internal quality control measures and approximately 8.09% duplicates were sent for inter-laboratory checks. In the 2024 exploration program: ▪ The primary drill samples were half-core cut by diamond saw, along the longitudinal axis. ▪ Samples were labelled, securely bagged and shipped to SGS Tianjin Laboratory for sample preparation and assaying. ▪ Pulp duplicates were inserted at a rate of 1 in 7 samples, blanks, and certified reference material (CRM) standard samples were inserted at a rate of 1 in 15 samples. ▪ Initially, samples were dried and crushed to a size of 3 mm, then homogenised. A 1.0 kg portion was extracted and further ground to a 200-mesh size (74 μm). ▪ For analysis, pulp samples weighing 0.20 g each were placed in a 50 mL PTFE (polytetrafluoroethylene) container. Then 2 mL of nitric acid (HNO_3), hydrochloric acid (HCl), hydrofluoric acid (HF), and perchloric acid ($HClO_4$) were added to each sample. The solution was evaporated to dryness, and the remaining residue was cooled. Subsequently, 2 mL of HCl, 1 mL of nitric acid (HNO_3), and 10 mL of water were added, and the mixture was heated to dissolve any precipitate. The resulting solution was then diluted with 12 mL of water, making it ready for ICP-OES testing. The SGS code for this testing method is ICP40Q. ▪ Sample sizes in the 2004–2024 exploration programs are appropriate for the grain size of the material being sampled.
Quality of assay data and laboratory tests	▪ The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. ▪ For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. ▪ Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.	In the 2004–2021 exploration programs: ▪ Samples were analysed for Pb, Zn, Cu and Ag contents in the laboratories of Xizang Centra Laboratory, Jinzang Laboratory and Yanjiao Central Laboratory. Inter-laboratory duplicates checks were sent to the Hebei Provincial Geological and Mineral Centre Laboratory (Hebei Laboratory). ▪ Results from internal duplicates and inter-laboratory checks showed good correlation. In the 2024 exploration: ▪ All samples were analysed for Pb, Zn, Cu and Ag at the SGS Tianjin Laboratory. ▪ Overall, quality control data collected do not indicate there are any significant problems with accuracy or precision. ▪ The assaying and laboratory procedures were considered appropriate.

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
Verification of sampling and assaying	■ The verification of significant intersections by either independent or alternative company personnel. ■ The use of twinned holes. ■ Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. ■ Discuss any adjustment to assay data.	■ For the 2004–2010 period, no core or pulp sample were retained. For verification of the results from these campaigns, SRK relied on information from twinned hole drilling and channel sampling. ■ For 2011–2021 period, 52 pulp samples and 107 core samples were collected and sent to SGS Tianjin Laboratory for validation. Results from these duplicates showed a good correlation. ■ In the 2024 exploration, samples were assayed under the QA/QC protocol in place at SGS Tianjin Laboratory. The quality control data collected from this verification campaign do not indicate there are any significant problems with accuracy or precision.
Location of data points	■ Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. ■ Specification of the grid system used. ■ Quality and adequacy of topographic control.	■ Unless otherwise specified, all coordinates in this report were in CGCS 2000/3-degree Gauss-Kruger zone 31 datum. Topographic control was considered adequate. In the 2004–2010 exploration period: ■ All drill holes and channel samples were surveyed using traditional optical surveying instruments or a total station. ■ Downhole surveys to measure dip and azimuth were conducted at 100 m intervals using KXP-2A type digital compass inclinometer. In the 2011–2021 exploration period: ■ All drill holes and channel samples were surveyed using OTS－632 type total stations. ■ Downhole surveys to measure dip and azimuth were conducted at 50 m intervals using KXP-2A type digital compass inclinometer. In 2024 exploration: ■ Collar positions were established using real-time kinematic (RTK) technology. ■ Downhole surveys to measure dip and azimuth were conducted at 50 m intervals.
Data spacing and distribution	■ Data spacing for reporting of Exploration Results. ■ Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. ■ Whether sample compositing has been applied.	■ In 2004–2010, exploration focused on the Pb14 deposit, with a general drill spacing of approximately 50–100 m × 50–100 m. ■ In 2011–2021, exploration focused on the Pb12 deposit, with a general drill spacing of approximately 50 m × 50–100 m. ■ The twinned drill holes and channel sampling completed in 2024 did not significantly affect the sampling spacing. ■ The combined spacing of the historical and 2024 programs is deemed adequate for estimation of the Mineral Resource.
Orientation of data in relation to geological structure	■ Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. ■ If the relationship between the drilling orientation	■ Drill core was not oriented. Orientation of core was deemed unnecessary on account of the low angle of orientation. ■ Drilling is at a sufficiently high angle to the mineralised structures to minimise the risk of

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
	and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	such bias. ■ Core structural measurements were not performed. ■ Folds and faults pattern have not been established from current observations.
Sample security	■ The measures taken to ensure sample security.	■ None of the 2004–2010 historical exploration samples were preserved. ■ The halved drill cores of 2011–2024 exploration period were stored in a well-maintained core shed at base camp.
Audits or reviews	■ The results of any audits or reviews of sampling techniques and data.	■ SRK undertook an audit of the assays, including standards, blanks and QA/QC of laboratory reporting. ■ No other audits of sampling techniques and data have been undertaken.

Section 2 Reporting of Exploration Results

(Criteria listed in section 1 also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	■ Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. ■ The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	■ The current mining licence (C5400002011043210111466) is valid until 25 November 2045. The mining licence covers an area of 4.4544 km^2 and allows exploitation between 4,964 m and 5,400 m asl. The permitted production capacity is 0.4 Mt/a. ■ The project is currently in operation. There are no known impediments to operation in the area at the time of reporting.
Exploration done by other parties	■ Acknowledgment and appraisal of exploration by other parties.	■ All resource exploration from 2002 to 2024 were carried out by the Second Brigade of the Xizang Autonomous Region Geological and Mineral Exploration and Development Bureau (Second Brigade).
Geology	■ Deposit type, geological setting and style of mineralisation.	■ A total of 37 prospects or deposits have been identified in the project area to date, most of which are layered or lenticular, occurring in skarns within the Laigu and Luobadui formations and along the contacts between limestones and siltstones. The ore textures are predominantly euhedral granular, with structures mainly of veinlet and massive types. The ore minerals include sphalerite, galena and minor amounts of pyrrhotite, pyrite and chalcopyrite, while the gangue minerals are primarily skarn minerals and calcite. ■ The Pb12 deposit is buried at depths ranging from -5 m to -350 m, with a thickness ranging from 0.6 m to 43.5 m. The entire deposit exhibits a steep dip angle, ranging from approximately 9° to 42°. ■ The Pb14 deposit is buried at depths ranging from 0 m to -130 m, with a thickness ranging from 7 m to 32 m. The entire layer exhibits a gentle dip angle, ranging from approximately 8° to 19°.
Drill hole Information	■ A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes: – easting and northing of the drill hole collar – elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar – dip and azimuth of the hole – downhole length and interception depth – hole length. ■ If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	■ All 255 drill holes were drilled from surface. Among them, 221 holes were drilled vertically, and the remaining 34 drill holes were drilled with a dipping angle ranging from 67.4° to 87.5°. Drill hole depths range from 43.58 m to 800.14 m. ■ Channel samples were collected horizontally on the drives wall with 1 m height from the floor. ■ All intervals were used in the interpretation of mineralised domains, variogram modelling and grade estimation.

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
Data aggregation methods	▪ In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated. ▪ Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. ▪ The assumptions used for any reporting of metal equivalent values should be clearly stated.	▪ To define the extension of mineralised domains in the Pb12 deposit, a nominal cut-off of either Pb ≥0.5% or Zn ≥1.0% was applied. A lower nominal cut-off of either Pb ≥0.3% or Zn ≥0.5% was used to delineate the mineralised intervals in Pb14 deposit. ▪ A grade capping level of 39% Pb was used, and no capping was applied to Zn, Cu and Ag.
Relationship between mineralisation widths and intercept lengths	▪ These relationships are particularly important in the reporting of Exploration Results. ▪ If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. ▪ If it is not known and only the downhole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').	▪ The mineralised domains exhibit a tabular or lenticular shape, with dips ranging from approximately 9° to 42° in the Pb12 deposit and 8° to 19° in the Pb14 deposit. As a result, the intersections from drill holes correspond to the true width of the mineralisation in the Pb14 deposit, while in the Pb12 deposit, the intersections have an apparent thickness greater than the true width of the mineralisation.
Diagrams	▪ Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	▪ Appropriate maps and sections were reported in this Report.
Balanced reporting	▪ Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	▪ Details of individual drill holes are not considered Material to the overall Mineral Resource estimate presented in this Report and are omitted.
Other substantive exploration data	▪ Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	▪ Metallurgical tests have been carried out for Pb14 and Pb12 deposits since 2007 in various institutions. Details are described in chapters 10 and 11.
Further work	▪ The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large-scale step-out drilling). ▪ Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	▪ No further exploration programs of infill or extension drilling are planned.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	▪ Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. ▪ Data validation procedures used.	▪ SRK conducted spot checks of the database against historical exploration program tables and maps, as well as the 2024 program assay certificates, and found no flaws in the data. ▪ During the process of uploading the database into Leapfrog software, various checks for internal inconsistencies (such as overlapping intervals and missing collars) are automatically performed. Visual checks of the different generations and types of sampling data against each other also ensure database integrity.
Site visits	▪ Comment on any site visits undertaken by the Competent Person and the outcome of those visits. ▪ If no site visits have been undertaken, indicate why this is the case.	▪ Site visits were undertaken on 2–5 September and 4–7 November 2024 (in two groups, respectively).
Geological interpretation	▪ Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. ▪ Nature of the data used and of any assumptions made. ▪ The effect, if any, of alternative interpretations on Mineral Resource estimation. ▪ The use of geology in guiding and controlling Mineral Resource estimation. ▪ The factors affecting continuity both of grade and geology.	▪ A total of 270 drill holes and underground channels were completed, resulting in the collection of 4,891 samples. This dataset provides a good definition of the dimensions for resource estimation. ▪ SRK has adequate confidence in the geological interpretation. The overall interpretation of the continuity, extent, and orientation of the mineralised domains, based on the various phases of exploration, has been confirmed by operation in Pb14 deposit since 2006, and underground drives in the Pb12 deposit. ▪ For the interpolation of attributes into the Mineral Resource block model, the anisotropies of the variogram models and search neighbourhoods were aligned to follow the approximately flat-lying geology.
Dimensions	▪ The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	▪ The dimensions of the mineralised domains are presented in Table 7.3.
Estimation and modelling techniques	▪ The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen, include a description of computer software and parameters used. ▪ The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. ▪ The assumptions made regarding recovery of by-	▪ Between 2004 and 2021, the Second Brigade in collaboration with Huaxia Mining, undertook several exploration activities and seven resource estimations. However, these estimates were not prepared under the reporting standards and definitions of the JORC Code. ▪ SRK's 3D block modelling and estimation was undertaken in Leapfrog Edge software (version 2024.1). The elements estimated were Pb, Zn, Cu and Ag. ▪ The estimation domains were built using Pb and Zn intervals in Leapfrog Edge software. A nominal cut-off of either Pb ≥0.5% or Zn ≥1.0% was applied to define the mineralised domains in

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
	products. ■ Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulphur for acid mine drainage characterisation). ■ In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. ■ Any assumptions behind modelling of selective mining units. ■ Any assumptions about correlation between variables. ■ Description of how the geological interpretation was used to control the resource estimates. ■ Discussion of basis for using or not using grade cutting or capping. ■ The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	the Pb12 deposit. A lower nominal cut-off of either Pb ≥0.3% or Zn ≥0.5% was used to delineate the mineralised domains in Pb14 deposit. ■ Three-dimensional (3D) block grade estimation approach was applied for the Mineral Resource estimation. ■ A grade capping level of 39% Pb was used, and no capping was applied to Zn, Cu and Ag. ■ No assumptions regarding correlation between variables were made. ■ The samples within domains were composited at intervals of 1.5 m. ■ The block models for all domains with dimensions of 20 m × 20 m × 4 m (East × North × Elevation) and sub-blocking with dimensions of 2 m × 2 m × 2 m (East × North × Elevation) were employed, and no rotation has been allowed. ■ Block grade values were interpolated using the Ordinary Kriging (OK) method or squared Inverse Distance Weighted (IDW) method. ■ SRK conducted visual validation of the longitudinal views and cross section view of the drill holes or channel grades and block model grades, which demonstrated good correlation between local block estimations and nearby samples, without excessive smoothing in the block model. ■ SRK compared recent (2024) production records and historical depletion against the model for reconciliation.
Moisture	■ Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	■ Tonnages are estimated on a dry basis.
Cut-off parameters	■ The basis of the adopted cut-off grade(s) or quality parameters applied.	■ A domain modelling threshold of either Pb ≥0.5% or Zn ≥1.0% was applied to define the mineralised domains in the Pb12 deposit. A lower nominal threshold of either Pb ≥0.3% or Zn ≥0.5% was used to delineate the mineralised domains in Pb14 deposit. These thresholds are suggested by the Chinese standard and have been applied on site for two decades.
Mining factors or assumptions	■ Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	■ Open-pit mining operations have been ongoing at the Pb14 deposit since 2007, using the conventional truck-and-shovel method. Material is blasted and then loaded into trucks. ■ The Pb12 deposit will be mined using underground methods, specifically room-and-pillar with delated backfill and sublevel open stoping.
Metallurgical factors	■ The basis for assumptions or predictions	■ The processing plant has been operating for

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
or assumptions	regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	more than 10 years. It uses a traditional three-stage, closed-circuit crushing process followed by a two-stage, closed grinding circuit. The mill feed ore is crushed and ground to the particle size of 70–75% passing approximately 74 µm. The Cu-Pb mixed flotation, separation flotation and Zn flotation process produces saleable Cu, Pb and Zn concentrates, with Ag being concentrated in the Cu and Pb concentrates. ▪ Based on historical performance, the forecast recoveries of Cu in copper concentrate is 51%, Pb in lead concentrate is 88% and Zn in zinc concentrate is 91%.
Environmental factors or assumptions	▪ Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	▪ Mineralised waste is disposed of in designated mineralised waste dumps within the mining area, while tailings is deposited in the tailings storage facility (TSF). The new TSF is lined with HDPE (high density polyethylene) geomembranes and equipped with various safety features. Based on the solid waste toxicity leaching test, the low-grade ore and tailings generated by this project are classified as Type I general industrial solid waste.
Bulk density	▪ Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. ▪ The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc.), moisture and differences between rock and alteration zones within the deposit. ▪ Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	▪ Bulk density and specific gravity measurements were conducted in historical programs and the density of ore rock was determined based on squared IDW method and supplemented by a fitted correlation between combined Pb+ Zn grade and density.
Classification	▪ The basis for the classification of the Mineral Resources into varying confidence categories. ▪ Whether appropriate account has been taken of all relevant factors (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). ▪ Whether the result appropriately reflects the Competent Person's view of the deposit.	▪ SRK considered the following factors in Mineral Resource classification: – Geological continuity and reliability of interpretation – Sample support and exploration workings density – OK attributes (kriging variance, slope of regression, kriging efficiency). ▪ Measured Mineral Resource is classified in the dimension area defined by sampling spacing within 50 m, or slope of regression greater than 0.85. ▪ Indicated Mineral Resource is classified in the dimension area defined by sampling spacing within 100 m, or slope of regression greater than

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
		0.6. ▪ Inferred Mineral Resource is classified in areas defined only by sampling spacing more than 100 m, or the domains estimated using IDW method.
Audits or reviews	▪ The results of any audits or reviews of Mineral Resource estimates.	▪ No external audits or reviews of the Mineral Resource estimate have been taken place. ▪ SRK carried out an internal peer review on the Mineral Resource estimate.
Discussion of relative accuracy/confidence	▪ Where appropriate, a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. ▪ The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. ▪ These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	▪ The relative accuracy of the Mineral Resource estimate is reflected in the Mineral Resource classification categories applied. ▪ The Mineral Resource Statement reflects the global estimates of in situ tonnes and grade.

Table 1 – JORC Code 2012

Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	▪ Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve. ▪ Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.	▪ A Mineral Resource estimate for the Mengya'a mine was completed by SRK Consulting (Hong Kong) Limited. The estimate is dated 31 December 2024. ▪ The reported Mineral Resource is inclusive of potential Or Reserve material.
Site visits	▪ Comment on any site visits undertaken by the Competent Person and the outcome of those visits. ▪ If no site visits have been undertaken, indicate why this is the case.	▪ Mr Falong Hu, the Competent Person for the Ore Reserve Statement, is a full-time employee of SRK Consulting (China) Ltd. ▪ Mr Falong Hu participated in the 4–7 November 2024 site visit.
Study status	▪ The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves. ▪ The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.	▪ Three technical studies are available to review: – Feasibility Study of the Mengya'a Lead-Zinc Mine in Jiali County, Nagqu, Xizang, May 2013, Guangdong Metallurgical & Architectural Design Institute – Preliminary Design for Underground Mining of the Pb12 Deposit at the Mengya'a Lead-Zinc Mine, Jiali County, Nagqu, Xizang., April 2022, CINF Engineering Corporation Limited – Preliminary Design for Underground Mining of the Pb12 Deposit at the Mengya'a Lead-Zinc Mine, Jiali County, Nagqu, Xizang, August 2024, Xinjiang Engineering & Research Institute of Nonferrous Metals Co., Ltd. ▪ For the Pb12 UG, the level of accuracy of the Modifying Factors described in the 22 CINF Mining Design, the 24 XERI Mining Design, and the existing underground infrastructure, and for the Pb14 OP, the level of accuracy of the Modifying Factors described in the 13 GDMADI FS and ongoing production, are considered by SRK to be akin to a pre-feasibility level study (PFS), prepared in accordance with the JORC Code (2012).
Cut-off parameters	▪ The basis of the cut-off grade(s) or quality parameters applied.	▪ The Mineral Resources of Mengya'a Mine contain lead, zinc, copper and silver elements. The lead equivalent (EqPb) grade was used for the cut-off grade to define the 'ore'.

Optimisation parameter	Unit	Inputs	Notes
Revenue and selling costs			
Lead price	RMB/t metal	14,300	50% Pb concentrate, SMM price excluding VAT
Treatment charge	RMB/t metal	-250	Grade premium applied due to higher Pb concentrate grade (≥ 60% Pb)
Royalties	%	5.9	22 CINF Mining Design
Mining parameters and costs			
Mining recovery	%	5	13 GDMADI FS, Huaxia Mining
Dilution	%	5	13 GDMADI FS, Huaxia Mining
Mining and stripping cost	RMB/t TMM	21	24 XERI Mining Design

Table 1 – JORC Code 2012

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
<table>
<tr><td>Overall slope angle</td><td>°</td><td>42</td><td>13 GDMADI FS, Huaxia Mining</td></tr>
<tr><td colspan="4">Processing parameters and costs</td></tr>
<tr><td>Design throughput capacity</td><td>kt/a</td><td>200</td><td>Huaxia Mining</td></tr>
<tr><td>Process plant recovery</td><td>%</td><td>88.0</td><td>Huaxia Mining</td></tr>
<tr><td>Processing cost</td><td>RMB/t ROM</td><td>154</td><td>Huaxia Mining</td></tr>
<tr><td>General and Administration cost</td><td>RMB/t ROM</td><td>66.8</td><td>24 XERI Mining Design</td></tr>
</table>
■ SRK is of the opinion that the cut-off grades used to distinguish ore from waste, set at EqPb 1.8% for material in the Pb14 OP, 6.0% for oxide and 3.7% for fresh material in the Pb12 UG, can be defined as economically extractable under these specific conditions.</td></tr>
<tr><td>Mining factors or assumptions</td><td>■ The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).
■ The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.
■ The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.
■ The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).
■ The mining dilution factors used.
■ The mining recovery factors used.
■ Any minimum mining widths used.
■ The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.
■ The infrastructure requirements of the selected mining methods.</td><td>Pb14 OP:
■ To develop an optimal open-pit design, an optimised open-pit shell was prepared using the Pseudoflow algorithm. The pit design was under the guide of optimisation and the design inputs, then manual modified by engineer. A EqPb cut-off grade and lead price of RMB14,300/t were used.
■ Dilution: Mining dilution is estimated at 5%.
■ Mining recovery: A 95% mining recovery rate was applied.
■ An OSA of 42% was applied.
■ The mine design treats Inferred Mineral Resources and sterilised material as waste materials.
Pb12 UG:
■ Based on the geological conditions, the following mining methods are proposed:
<table>
<tr><th>Thickness</th><th>Dip (°)</th><th>Mining Method</th></tr>
<tr><td><8 m</td><td><=30</td><td>Room and Pillar with Delayed Backfill</td></tr>
<tr><td>>=8 m</td><td><=30</td><td>Sublevel Open Stope with Delayed Backfill (Gently Dip Orebody& Longitudinal)</td></tr>
<tr><td><15 m</td><td>>30</td><td>Sublevel Open Stope with Delayed Backfill (Steep Dip Orebody& Longitudinal)</td></tr>
<tr><td>>=15 m</td><td>>30</td><td>Sublevel Open Stope with Delayed Backfill (Steep Dip Orebody& Transverse)</td></tr>
</table>
■ The stope is arranged along the strike, with a length of 40 m and a width of 15 m. The level spacing is 40 m/30 m.
■ For Pb12 UG, the mining factors applied to the Ore Reserve estimates are:
– Mining Design Scope:
– Only Measured and Indicated Mineral Resources are considered for the estimation.
– Deswik. was used to optimise the mineable shapes and refinement of these shapes was conducted as part of the stoping scope.
– Design loss:
– Stopes with grades below the cut-off grade will be filtered out as design loss.
– Resources that are isolated and located far from the main design are difficult to reach.
– Mining dilution:
– A dilution skin of 0.5 m on the hanging wall or back, and a dilution skin of 0.2 m on the footwall or floor, were considered to simulate</td></tr>
</table>

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
		the overbreak during mining. This contributes approximately 6%, which is less than the 22 CINF Mining Design at 12%. – Mining loss: – A loss rate of 8% is applied to account for the loss incurred during the transportation from the workface handover to the ROM bin, which is consistent with 22 CINF Mining Design. An additional 4% of the planned loss was considered for room and pillar mining, which includes the loss attributed by the pillars. ▪ The development system for the Pb12 UG will use existing exploration adits. A decline system will be constructed to connect levels where a level adit is not available and the level spacing is 40 m/30 m. ▪ The mine design treats the Inferred Mineral Resources as waste materials. ▪ The mine design takes mine service infrastructure, including backfill, ventilation, power supply, water supply, air compression and dewatering systems, into account.
Metallurgical factors or assumptions	▪ The metallurgical process proposed and the appropriateness of that process to the style of mineralisation. ▪ Whether the metallurgical process is well-tested technology or novel in nature. ▪ The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied. ▪ Any assumptions or allowances made for deleterious elements. ▪ The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole. ▪ For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?	▪ The ore is a Cu-Pb-Zn polymetallic sulfide ore, with the primary target minerals being chalcopyrite, galena and sphalerite. The process involves Cu-Pb bulk flotation followed by separation and Zn flotation, to produce Cu, Pb and Zn concentrates. The beneficiation process and operational parameters have been determined through multiple beneficiation tests. This process is mature and widely adopted for deposits with similar mineralisation. ▪ Arsenic is a harmful element in the ore, but its content is very low and does not affect the quality of the concentrates. Cu, Pb and Zn are separated from each other. This separation process enhances the quality of each concentrate and increases the recovery rate of each element. ▪ Gold and silver are beneficial elements. The silver content in the Cu and Pb concentrates is high enough to be considered payable, but silver does not reach a payable level in the Zn concentrate. The gold grade is low and does not reach a payable level in any of the concentrates. ▪ Although ore oxidation can affect the processing recovery of target elements, it is difficult to establish a clear relationship between the degree of oxidation and the recoveries. The proportion of oxidised ore is very small, and it has a minimal impact on the project's overall economics. Huaxia Mining is conducting beneficiation tests on oxidised ore and plans to process the mined oxidised ore in the future.
Environmen-tal	▪ The status of studies of potential environmental impacts of the mining and processing operation. Details of low-grade ore characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.	▪ The EIA reports comply with relevant Chinese legal requirements and the project has obtained the necessary government approvals. The environmental studies primarily cover the main production facilities (mine sites, processing plant and TSFs). ▪ Solid waste toxicity leaching tests were conducted on the mineralised waste from the Pb14 orebody and the tailings from the processing plant. The mineralised waste and tailings generated by this project are classified as Type I general industrial solid waste.

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
Infrastructure	■ The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.	■ Primary power supply: The Tangjia 110 kV substation, located 80 km from the site, provides primary power via two self-constructed 35 kV double-circuit transmission lines. ■ Domestic power supply: Power for the processing plant, living areas, and base camp is independently sourced from the Rongdoi township 10 kV rural distribution grid, ensuring separate residential and operational power networks. ■ Mining area water supply: Water is drawn from the nearby Qizu River using a riverside sump and pump system, which delivers water to an elevated tank for production and fire suppression needs. ■ Processing plant water supply: Water for the processing plant is sourced from the Langgezhanlong River. ■ Site access: The project site is approximately 165 km from Lhasa, accessible via the well-paved G4218 Ya'an–Yecheng Expressway to Mozhugongka County, followed by National Highway 349. ■ Land availability: No significant issues were identified regarding the availability of land for development. ■ Labour supply: Adequate labour is sourced locally and from other regions of China.
Costs	■ The derivation of, or assumptions made, regarding projected capital costs in the study. ■ The methodology used to estimate operating costs. ■ Allowances made for the content of deleterious elements. ■ The source of exchange rates used in the study. ■ Derivation of transportation charges. ■ The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc. ■ The allowances made for royalties payable, both Government and private.	■ The project has been underway for some time, and the construction of the underground infrastructure is largely complete. The projected capital costs primarily involve the refurbishment and replacement of equipment at the processing plant, as well as the construction of underground development infrastructure at the Pb12 deposit. The forecast was prepared by the Company's technical and financial team. ■ The operating costs are based on the Company's historical performance. ■ The treatment charges are based on forecasts provided by SMM, an independent market research company. ■ Resource tax is set at 5.90% of revenue.
Revenue factors	■ The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc. ■ The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.	■ The commodity price forecast is provided by SMM, an independent market research company. ■ The treatment charges are based on forecasts provided by SMM, an independent market research company. ■ The payable metals are based on the terms set out in offtake agreements with various clients.
Market assessment	■ The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.	■ A market study has been conducted by SMM, an independent market research company. ■ The forecast prices for copper, lead and zinc are based on SMM's market study, while the silver forecast price is derived from Consensus Economics Inc.

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
	▪ A customer and competitor analysis along with the identification of likely market windows for the product. ▪ Price and volume forecasts and the basis for these forecasts. ▪ For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	
Economic	▪ The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc. ▪ NPV ranges and sensitivity to variations in the significant assumptions and inputs.	▪ The inputs included the mining schedule, processing production schedule, operating and capital costs and the forecast prices by SMM. ▪ A base case economic analysis was undertaken at a 10% discount rate. ▪ A range of positive NPVs was derived at various discount rates, with key sensitive parameters being sale price and processing recovery.
Social	▪ The status of agreements with key stakeholders and matters leading to social licence to operate.	▪ Public participation was carried out during the EIA process for the project, and no objections were raised by the public during the announcement period.
Other	▪ To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves: ▪ Any identified material naturally occurring risks. ▪ The status of material legal agreements and marketing arrangements. ▪ The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	▪ The identified risks are tabulated in Chapter 16. ▪ No material issues in relation to the status of the mining and exploration licences are stated in the legal due diligence report.
Classification	▪ The basis for the classification of the Ore Reserves into varying confidence categories. ▪ Whether the result appropriately	▪ Measured Mineral Resource in the pit design or stopes are classified as Proved Ore Reserves. Indicated Mineral Resources in the pit design or stopes are classified as Probable Ore Reserves. ▪ The classification of Ore Reserves appropriately reflects the Competent

Table 1 – JORC Code 2012

Criteria	JORC Code explanation	Commentary
	reflects the Competent Person's view of the deposit. ▪ The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	Person's view of the deposits. ▪ There is no Measured Mineral Resource classified as Probable Ore Reserves.
Audits or reviews	▪ The results of any audits or reviews of Ore Reserve estimates.	▪ An internal review of the Ore Reserve has been conducted.
Discussion of relative accuracy/ confidence	▪ Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate. ▪ The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. ▪ Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage. ▪ It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	▪ The Ore Reserves estimates are based on the technical studies carried out in 2013, 2022 and 2024 as well as the ongoing open-pit operation. The Ore Reserves estimates are at a PFS level. ▪ All Modifying Factors have been applied for Ore Reserves estimates on a pit or stopes basis, on a global estimate. ▪ As is the case for most mining projects, the extent to which the estimate of Ore Reserves may be affected by mining, metallurgical, infrastructure, permitting, market and other factors could vary from major gains to total losses of Ore Reserves. ▪ There are no issues known to the Competent Person of this section that are expected to have a material impact on the Ore Reserve estimates.