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AZARGA METALS CORP

NI 43-101 TECHNICAL REPORT

PRELIMINARY ECONOMIC ASSESSMENT OF THE UNKUR COPPER DEPOSIT, ZABAIKALSKY KRAI, RUSSIAN FEDERATION

August 2021

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August 2021

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CONTENTS

TABLES

FIGURES

APPENDICES

1 EXECUTIVE SUMMARY

Azarga Metals Corp (AZR) has commissioned Wardell Armstrong International (WAI) to carry out a Preliminary Economic Assessment (PEA) of its mineral resource base and strategic assessment of the Unkur Copper Project. The study has aimed to assess and test the combined potential of open pit transitioning to underground mining methods considering four process options for treatment of oxide material.

The following scenarios have been considered as a part of this study:

- Fast-tracking oxide only operation, using cyanide heap leach-SART processing option. This option is associated with the relatively low capital costs, fast construction and quicker payback period;
- A combined oxide and sulphide open pit operation with the following oxide processing options:
  - Option 1: Agitated cyanide leach-SART
  - Option 2: Sequential acid and cyanide agitated leach;
  - Option 3: Cyanide heap leach-SART;
  - Option 4: Sequential acid and cyanide heap leach.
- A combined open pit and underground operation for the shortlisted oxide processing options.

Process parameters for all the above options are outlined in the relevant sections of this report. As a result of the performed trade-off analysis Options 1 and 2 have not proven to be economically viable. Therefore Options 3 and 4 were considered in detail for the combined oxide with sulphide operation using open pit and underground mining methods.

Therefore, this report presents the outcome of the preliminary economic assessment for the following Scenarios:

- Scenario 1: Open pit operation of oxide materials only, followed by the underground development.
- Scenario 2: A combined open pit operation of both oxide and sulphide materials, followed by underground development (Oxide Processing Options 3);
- Scenario 3: same as the Scenario 2 but with the Processing Option 4; and
- Scenario 4 (OP oxide only) with Processing Option 3.

The key elements included within the assessment are listed below:

- Mineral Resource Estimation;
- Hydrological and hydrogeological review;
- Mining geotechnical review;
- Open pit mining study;
- Transition to underground mining study;

- Mine production scheduling;
- Mining capital and operating cost estimation;
- Mineral processing review including disposal of tailings
- Environmental and social review; and,
- Financial analysis.

1.1 Mineral Resource Estimate

The estimate is based on 10,388m of diamond core drilling (from 33 drill holes) and 253m of channel sampling (from four trenches and two outcrops), completed during AZR campaigns of 2016-2017 and 2019-2020. Historical sampling does not directly inform the estimation, but the geological model and conceptual framework for preparing the estimation are influenced by consideration of historical data.

From logging and copper phase analysis (ratio of oxide Cu to total Cu), AZR prepared a wireframe surface of the contact between oxide and sulphide mineralisation. The depth of this boundary varies but is generally down to 200m below the topographic surface. A post-mineralisation fault interpretation was also incorporated which coincides with a discordance between the projected positions of the North and South mineralised zones.

The core of mineralisation was modelled in three zones (Figure 1.1):

- A North domain, based on 22 intersections;
- Parallel to the main part of the North domain, a less extensive parallel zone (possibly a splay, and separated by a few metres) referred to as the West domain, based on two (2) intersections; and
- A South domain, based on six (6) intersections.

A threshold of 0.2% Cu was used for defining the North and West domains, and a higher threshold (0.35% Cu) was used for defining the South domain.

Two halo zones, at 0.1% Cu threshold, were modelled around the combined North core domains and the West core domain. A third standalone 0.1% Cu threshold domain was modelled around a second and separate West domain (all assays for this domain were in the range 0.1 to 0.2% Cu).

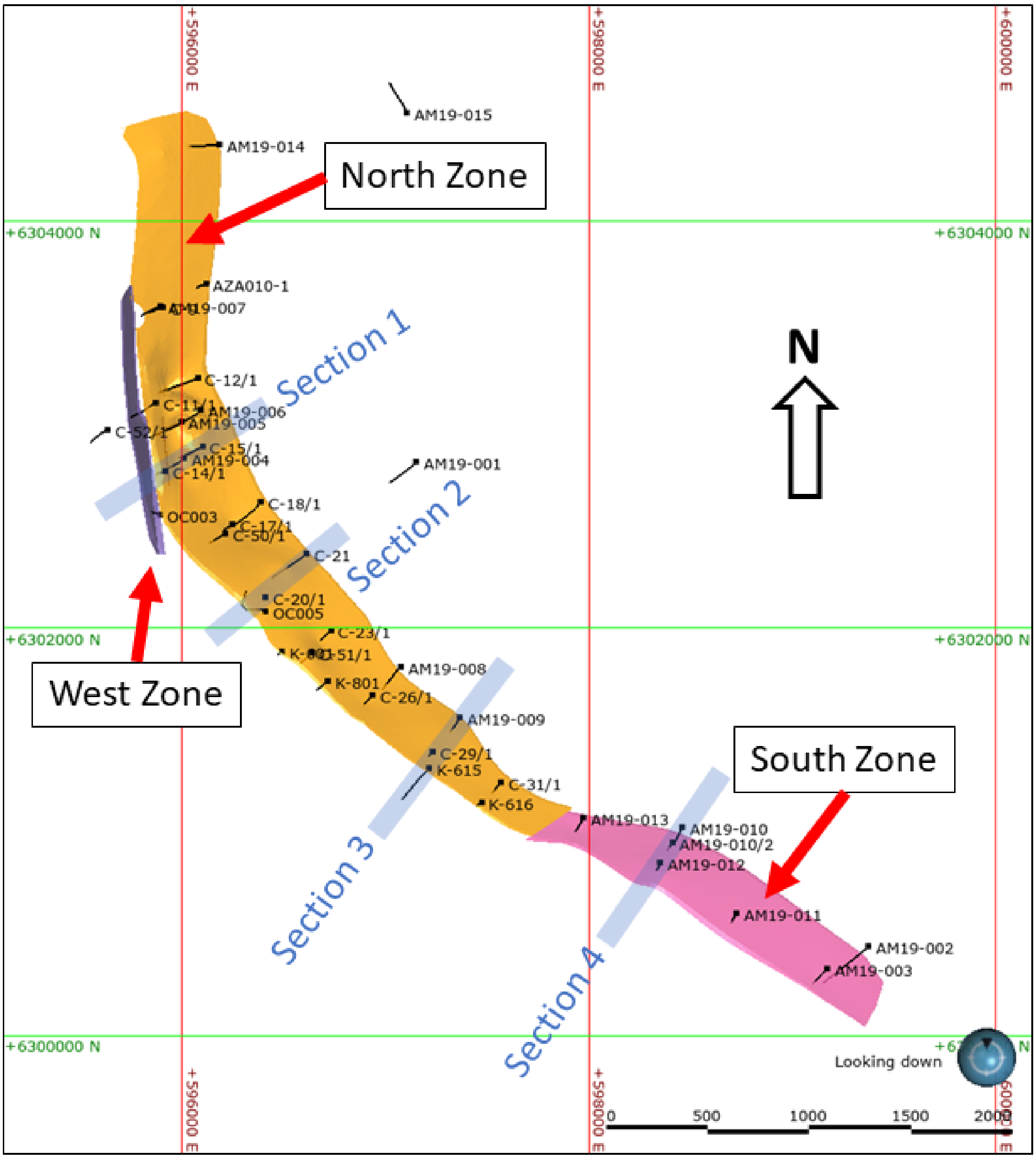


Figure 1.1: Plan Showing Collar Locations and Drill Hole Traces and Relation to Modelled Mineral Domains

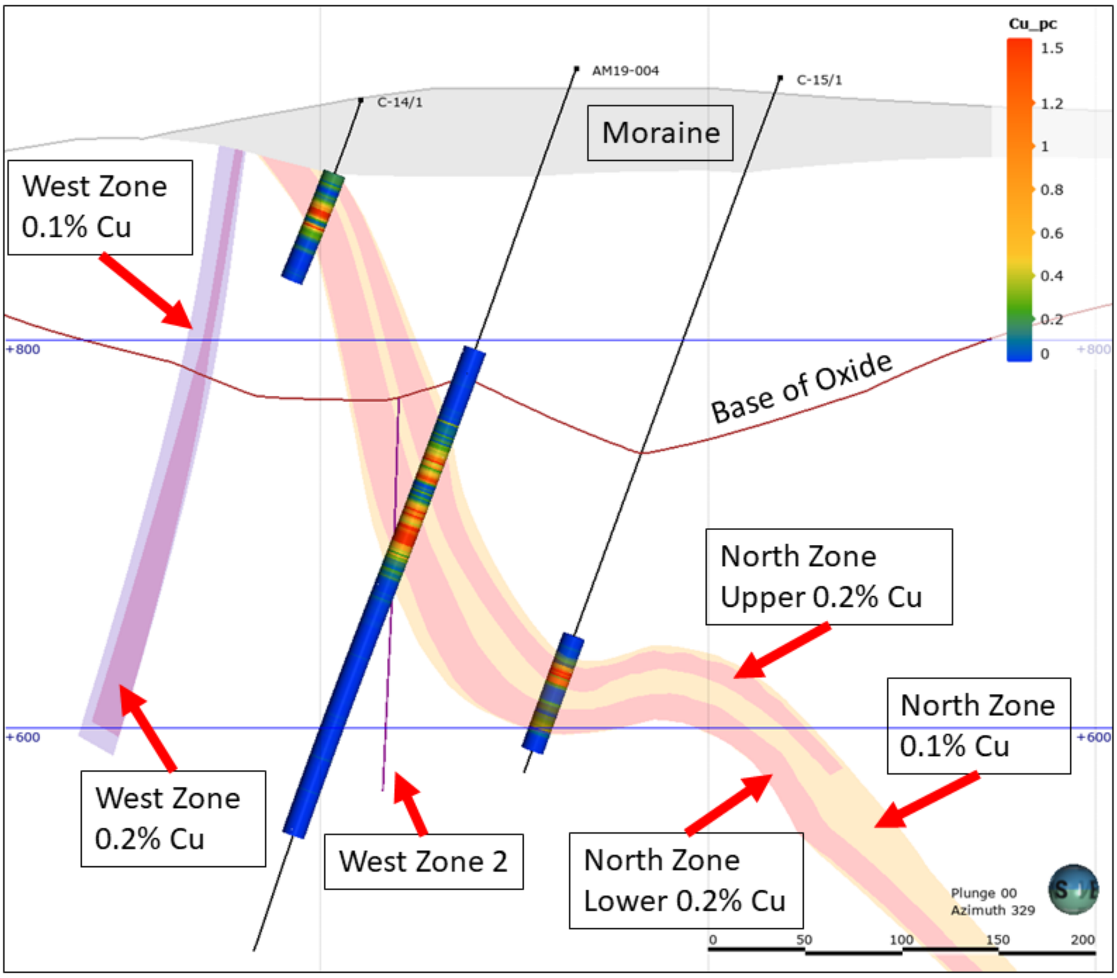


Figure 1.2: Example Section (corresponding to Section 1 on plan in Figure 1.1) Showing Northern Mineral Domains

The domains prepared by SRK Consulting Ltd. (SRK) were used as hard boundaries to constrain estimation (i.e., block grade estimates within a domain would only be influenced by composites from the same domain). Copper and silver grades within the four main mineralised domains were estimated by 2D Ordinary Kriging. Model cell dimensions selected are as follows (Table 1.1).

Table 1.1: Block Model Dimensions

	X	Y	Z
Minimum	600,200	6,305,300	265
Maximum	605,900	6,311,350	1,190
Estimation block size (m)	10	50	25
Sub-blocking (m)	Variable to fit wireframe	5	5
Discretisation	1	8	8

Density measurements were carried out by wet hydrostatic method on over 500 samples from core collected by AZR during the 2019-2020 drilling campaign.

From these sample averages, the following density factors were used to convert volumes in the block model to dry bulk tonnages:

- Oxide mineralised: 2.60 (59 samples)
- Oxide waste: 2.58 (76 samples)
- Sulphide mineralised: 2.67 (90 samples)
- Sulphide waste: 2.68 (270 samples)
- Moraine overburden: 2.00 (assumed)

The mineral resource estimate for the Unkur project is presented in Table 1.2. These mineral resources have been estimated in conformity with generally accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines and reported in accordance with the Canadian Securities Administrators' National Instrument 43-101.

Reasonable prospects for eventual economic extraction are supported through open pit optimisation using metal prices of US$8,500/t Cu and US$25/oz Ag, with underground resources based on cut-off grade of 0.54% CuEq.

Table 1.2: Mineral Resource Estimate for the Unkur Project

Classification	Method	Type	COG (CuEq %)	Tonnes (Mt)	Cu (%)	Ag (g/t)	CuEq (%)
Inferred	OP	Oxide	0.18	15.7	0.61	45	1.05
		Sulphide		17.1	0.59	49	1.03
	UG North	Oxide	0.54	0.4	0.51	23	0.73
		Sulphide		14.2	0.55	30	0.83
	UG South	Oxide		-	-	-	-
		Sulphide		3.7	0.64	20	0.82
	TOTAL	**OXIDE**		**16.1**	**0.61**	**44**	**1.04**
	TOTAL	**SULPHIDE**		**35.0**	**0.58**	**38**	**0.93**
	TOTAL	**ALL**		**51.1**	**0.59**	**40**	**0.96**

Notes:

1. Figures have been rounded to reflect this is an estimate.
2. Inferred Mineral Resources have been reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definition standards for Mineral Resources and Reserves and have been completed in accordance with the Standards of Disclosure for Mineral Projects as defined by National Instrument 43-101.
3. No Measured or Indicated Resources have been estimated.
4. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
5. Mineral Resources are based on a CuEq grade of 0.18% for the Open Pit resources and of 0.54% for the Underground resources using metal prices of US$3.86/lb Cu and US$25/oz Ag, the equivalence formula for Oxide is CuEq = Cu + (0.0097 x Ag) and for Sulphide is CuEq = Cu + (0.009 x Ag).
6. The Mineral Resource is effective as of 31st July 2021.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. Mineral Resources may be subject to legal, political, environmental and other risks and uncertainties.

1.2 Hydrological & Hydrogeological Review

The Site is located within the catchments of the River Kemen, which flows through the Site, and River Unkur which are sub-catchments of the Chara River catchment (674km^2). Rivers within the Chara catchment flow all year round including below ice cover during the winter months.

Surface water sampling was carried out in 2016 within the Site. The results exhibit typical water quality traits for mid-mountain, sparsely populated areas: low-mineralisation, soft water and low-radioactive.

Glacial sediments, in the form of moraines, cover most of the Site area. The average thickness of the moraine cover is 40 m; however, this cover increases to 180m to 200m thickness in the northwest and southeast of the Site.

Bedrock is reported to be confined with phreatic surfaces at 140m to 110mbgl. The hydraulic conductivity in the confined aquifer is controlled by fractures (such as the Kemensky fault) and is highly variable (0.01 to 22m/d). However, the presence of permafrost across the Site and surrounding area drastically reduces hydraulic conductivity and acts as an aquitard. The permafrost is widespread and continuous across the area with thickness typically ranging between 200m and 400m. The seasonal thaw of the upper permafrost layer is reported not to exceed 2-3m within the Site area. Previous hydrogeological studies have identified crossflows along 'talik' structures within the permafrost likely to be associated with thaw zones below large lakes and rivers. Measurements in the Soviet Era holes showed sub-zero temperatures of 0 - -1.5°C down to a depth of 200m and more.

Groundwater data available from the Udokan deposit exhibits hydro-carbonate calcium and calcium-sodium water, mineralization is 0.05-0.1 g/l and general hardness is 0.27-2.8 q/l. Confined groundwater conditions were encountered at depths of 250m and 284m in wells 122 and 123 respectively in the northwest area of the Site. Groundwater levels are reported to have risen over 100m.

Currently, there is limited site-specific hydrology and hydrogeology data however, it is anticipated that raw water will either be pumped from a point in the local river system where water is available all year round (in winter it flows below the ice cap); or it will be drawn from boreholes around the pit, depending upon the depth of any continuous permafrost. Studies at nearby Udokan deposit investigated groundwater availability from unconsolidated sediments within the thaw zones around river and streams in the valley of the Lower Ingamakit. Groundwater availability was calculated to be 207,399m^3/day. Further investigation will be required to obtain:

a. Surface water quality data;
b. Surface water flow data;
c. Groundwater quality data;
d. Aquifer properties, and
e. Permafrost/talik investigation.

A reliable water supply meeting the projects technical and potable demands need to be developed at PFS stage. A water balance and a water management plan will need to be established for the Project.

There is the potential for ARD issue due to the sulphidic ore. Therefore a suitable test programme will need to be established starting with static test work and testing sufficient representative samples to be commensurate with the inventory of potential acid forming materials (ores and waste). A groundwater model will be beneficial to establish impact predictions on the hydrological and hydrogeological receptors.

1.3 Geotechnical Review

WAI understands limited geotechnical investigation has been carried out at the site and further investigation is required to determine the geotechnical conditions for the design of the open pit, any underground studies, mine site and waste management facilities.

1.4 Mineral Processing

Testwork was conducted by VNIItsvetmet (Kazakhstan) on samples of oxide and sulphide ore in 2020. The total sample weight was approximately 730kg.

The oxide sample head grade was 0.56% Cu and 30.8g/t Ag. The sulphide sample assayed 0.56% Cu and 39.0g/t Ag. Diagnostic acid leaching indicated that 96.4% of the copper is acid soluble in the oxide sample.

Acid bottle roll testwork was conducted on three sizes of the oxide sample: -20mm, -10mm and -71 microns. Copper recovery was 73.6%, 80.6% and 96.4%, respectively, but at very high rates of acid consumption - 46.6, 54.2 and 81.9kg/t, respectively.

A whole ore cyanidation bottle roll test (BRT) was carried out at a grind size of 80% passing 71 microns using a 1% initial cyanide concentration. Copper and silver recovery was 55.6% and 96.7% respectively but at a high cyanide consumption of 40.4 kg/t.

Detailed flotation studies were carried out on the sulphide sample, including several Locked Cycle Tests (LCT). The results from LCT No 4 have been used for design purposes. The flowsheet tested in the LCT No 4 is shown in Figure 1.3.

The pH was in the range of 9-10 and Potassium Amyl Xanthate (PAX) was used as the collector with MIBC as the frother, both reagents added only to the rougher and scavenger stages. A total of 60g/t PAX was used for the roughing and scavenger stages with 30g/t MIBC frother.

The results indicated 89.1% copper recovery to a concentrate grading 25.8% Cu. Silver recovery to the copper concentrate was 82.7% grading 1,634g/t Ag. From this analysis, there does not appear to be any significant penalty elements although the Sb content was not analysed.

Due to the limited testwork available to demonstrate cyanide leaching of the oxide samples and particularly at coarser crush sizes for potential heap leaching, an additional testwork programme was conducted by VNIItsvetmet in 2021 for both fine and coarse ore BRTs using the same samples from the 2020 programme. The sample size for the fine ore tests was 80% passing 71 microns. Varying cyanide concentrations were tested. The fine ore fraction results are summarised in the following Table 1.3.

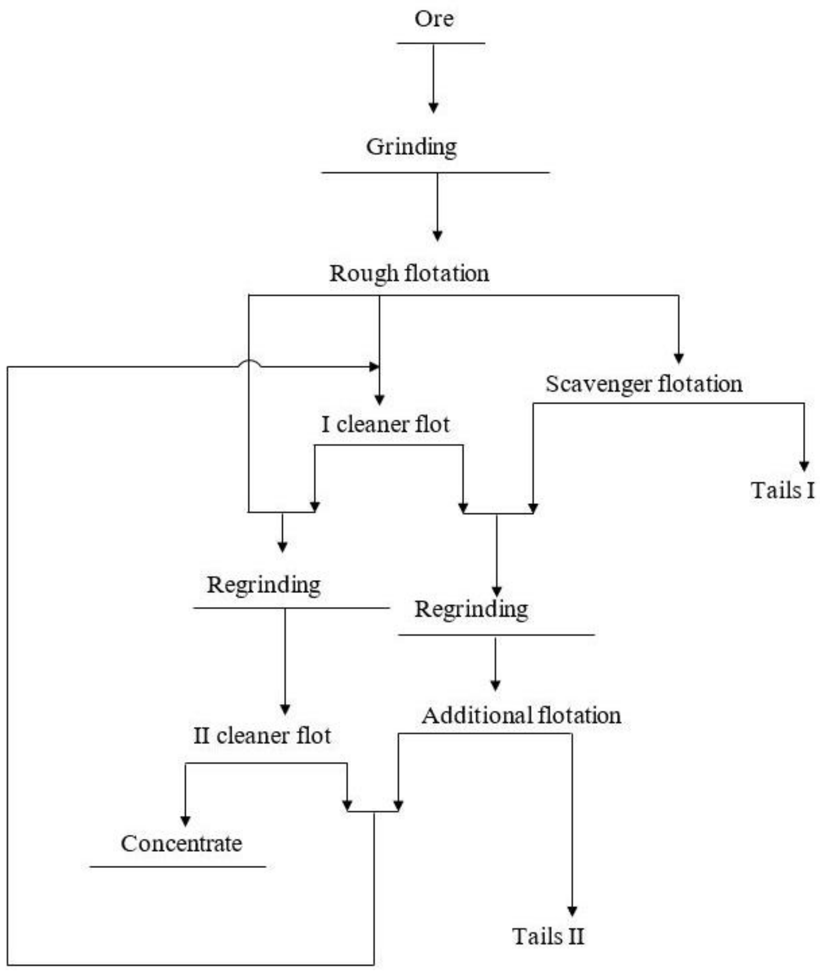


Figure 1.3: Flotation Testwork Flowsheet

Table 1.3: Cyanide Bottle-Roll Test Results for Fine Fractions

NaCN, %	Retention time, hr	Grade in cake		Recovery, %		NaCN consumption, kg/t ore
		Cu, %	Ag, g/t	Cu	Ag	
With carbon						
0.05	24	0.41	11.3	31.7	66.0	1.41
Without carbon						
0.05	24	0.41	13.6	31.7	56.4	1.32
0.1	48	0.29	8.6	49.5	72.7	4.60
0.5	48	0.26	1.5	54.5	95.2	6.97
1.0	48	0.24	1.3	58.0	95.9	9.41
1.0*	***48***	***0.25***	***1.0***	***55.6***	***96.7***	***40.40***

Oxide ore - 0.56% Cu, 30.9 g/t Ag
L:S=1.5:1
*As per 2020 test by VNIITSVETMET

In summary, the results show that recovery of both copper and silver increases to a concentration of 0.5% NaCN. At 1.0% NaCN, the recovery improvements are marginal, although personal communication reports from the laboratory suggest that there was human error in calculating NaCN consumption. **For scoping level purposes, copper and silver recoveries of 58.0% and 95.9%, respectively, are assumed for the fine grind size.**

For the coarse ore tests, three crush sizes were selected: -25mm, -12.5mm and -6.5mm. The tests were run for 21 days with two different cyanide concentrations and the results are summarised in the Table 1.4.

The best metal recoveries were obtained at 0.2% NaCN concentration and at the finest crush size of -6.5mm.

Table 1.4: Cyanide Leaching Results for Coarse Fractions

Fraction, mm	NaCN, %	Grade in cake		Recovery, %		NaCN consumption, kg/t ore
		Cu, %	Ag, g/t	Cu	Ag	
25.0	0.05	0.35	12.9	39.9	59.9	4.30
12.5		0.34	11.8	41.7	63.3	4.36
6.5		0.31	10.0	46.8	68.9	4.51
25.0	0.2	0.35	14.0	40.0	56.5	5.58
12.5		0.27	10.5	53.7	67.4	6.08
6.5		0.24	7.3	58.8	77.3	8.43

Oxide ore - 0.56% Cu, 30.9 g/t Ag
L:S=1.5:1
Retention time 21 days

For scoping level purposes, copper and silver recoveries of 58.8% and 77.3%, respectively, are assumed for the coarse crush size of -6.5mm.

WAI was requested to incorporate testwork results on oxide ore presented by VNIItsvetmet testwork carried out in Q2 2021 to consider the viability of four oxide process options proposed by Tetra Tech in 2018, input parameters for which are summarised in Table 1.5.

Table 1.5: Input Parameters of the Four Process Options (from Tetra Tech 2018)

Description	Unit	Option 1 Cyanide Tank Leach (Base Case)	Option 2 Tank Leach SX/EW	Option 3 Heap Leach SART	Option 4 Heap Leach SX/EW
Total Plant Capital Cost	US$ million	128.14	187.07	77.43	128.89
Total Plant Operating Cost	US$/t	19.18	28.64	13.86	22.45
Overall Metallurgical Recovery*	Ag %	95	95	65	65
Overall Metallurgical Recovery*	Cu %	95	95	65	65

The four options are summarized as follows:

1. Option 1 is conventional crushing, grinding and agitated cyanide leaching of the copper and silver, followed by CCD washing and processing of the solution using the SART process to recover a saleable synthetic copper/silver concentrate and recycling the cyanide solution back for heap leaching.
2. Option 2 is based on sequential acid and cyanide leaching of the finely ground ore. After crushing and grinding, agitated acid leaching is affected followed by CCD washing and SX/EW treatment of the solution for copper cathode production. The solids are then agitation leached with cyanide for silver recovery and, after CCD washing, the solution is processed via Merrill Crowe to produce silver bullion. The solids report as final tailings.
3. Option 3 is for a cyanide heap leach at a coarse crush size for silver recovery but with the solutions processed by SART (as for Option 1) to recover a combined Cu/Ag concentrate. After three stages of crushing, the ore is agglomerated if required and heap leached with sulphuric acid to recover the copper and silver into solution. This is then treated via the SART process as per Option 1 to produce a combined Cu/Ag concentrate and the cyanide recovered and recycled back to the heaps.
4. Option 4 is for sequential acid and cyanide heap leaching. After crushing and agglomeration as required, the ore is stacked, and heap leached with sulphuric acid for copper recovery and the pregnant solution treated by SX/EW to produce copper cathodes. The ore is then removed from the pads, neutralised, and re-stacked for leaching with cyanide solution. The pregnant solution is then treated by Merrill Crowe to produce silver bullion.

A processing rate of up to 3.5Mtpa was selected for open pit (OP) mining and 2.0Mtpa for underground (UG) mining. A SART recovery of 95% was applied for both copper and silver recoveries obtained from the testwork assuming a copper grade of at least 65% Cu will be achieved in the concentrate, based on benchmarking data from current operating SART plants.

A summary table of the updated key parameters for all four oxide ore processing options is shown in Table 1.6.

Table 1.6: Physicals for Oxide Processing Options

OXIDE ORE	Option 1 (Agitated Leach)	Option 2 (Sequential Agitated Leach)	Option 3 (Heap Leach)	Option 4 (Sequential Heap Leach)
Copper Recovery, %	55.1	95.0	55.9	80.6
Silver Recovery, %	91.1	95.0	73.4	73.4
Capital Cost, $M	164.3	229.8	52.3	97.3
Operating Cost, $/t	18.3	27.4	5.8	15.4

For sulphide ore, a conventional copper flotation processing plant is indicated, producing a single copper concentrate product for sale to market and containing significant silver credits. The copper concentrate grade is estimated as 25.8% Cu, containing 1,634g/t Ag (testwork results).

A summary of the key parameters for sulphide ore processing is shown below in Table 1.7.

Table 1.7: Summary of Key Parameters for Sulphide Ore Processing

SULPHIDE ORE	Copper Flotation Plant
Copper Recovery, %	89.1
Silver Recovery, %	82.7
Copper Grade, % Cu	25.8
Silver Grade, g/t Ag	1,634

The processing of oxide ore presents the biggest challenge due to the carbonate content, resulting in very high acid consumptions from conventional acid leaching, either by agitated or heap leaching. The focus has been on process routes employing cyanide leaching first for silver and copper dissolution, with two options studied for agitated and heap leaching, followed by processing of the leach solutions with SART technology.

A two-stage SART process has been assumed, as per the TT PEA report, whereby sequential copper and silver sulphide concentrates are produced, but combined for filtering into just one concentrate product for sale to market. No testwork has been conducted on SART processing at this stage of study.

Going forwards, a comprehensive metallurgical testwork programme is required on representative oxide ore samples to confirm expected copper and silver recoveries through heap leaching (to include optimisation of crush size, requirement for agglomeration, cyanide consumption and copper and silver recoveries prior to SART processing). This will include extensive bottle roll, agglomeration, and column testwork programmes.

The current project is based on a maximum process throughput of 3.5Mtpa with a resultant Life of Mine of a combined OP and UG operation at 12 to 14 years (depending on the selected Scenario Option), equating to a total production of around 22Mt of tailings. The facility will be operated in a "closed" system with no water discharge to the environment with recirculation of the supernatant water to the process plant and collection of seepage water. The natural topography will be utilised where possible to provide containment with additional storage provided by engineer embankments or containment structures. For the purpose of this study, it has been assumed that the tailings will be in the form of a thickened slurry with approximately 65% w/w solid content.

1.5 Open Pit Mining

A summary of the in-situ tonnages and grades contained within the five optimal pit shell runs is provided in Table 1.8. The methodology and assumptions used were as follows:

- Not limited by sinking rate – no constraints put on incremental costs as the pit deepens;
- One 15m3 bucket shovel production rate at 10Mtpa;
- Both lines for oxide and sulphide ore types can run both lines simultaneously after 4 years of operation (depending on the particular option) with 3.5Mtpa limit per line;
- No replacements assumed due to short LOM 6-7 years. Assuming up to 45-50k working hours for single unit, no sustaining CAPEX applied;
- Waste profile has been smoothed using NPV Milava algorithm;
- Discount rate at 8%;
- GA and infrastructure costs are included on annual basis (US$9Mpa); and
- Royalty has been calculated per metal.

Table 1.8: Summary of Optimization Open Pit Output

Item	Unit	Scenario 1, Option 3	Scenario 2, Option 1	Scenario 2, Option 2	Scenario 2, Option 3	Scenario 2, Option 4
Waste	t	97,420,044	243,664,112	252,363,018	228,138,962	254,164,978
Ore_oxide	t	10,980,525	9,219,308	9,230,153	11,949,639	10,232,932
Cu_grade	%	0.59	0.73	0.70	0.57	0.66
Ag_grade	g/t	48.24	61.24	58.69	46.72	54.10
Ore_sulphide	t	-	7,502,343	7,498,946	7,020,898	7,530,834
Cu_grade	%	-	0.80	0.73	0.74	0.74
Ag_grade	g/t	-	67.36	66.02	67.48	66.06
Stripping ratio	t/t	8.87	15.00	15.00	12.03	14.31

The higher average grade reported in Table 1.8, for Options 1 and 2, is a reflection of higher copper recoveries assumed for these options. These options are also associated with the highest CAPEX values required at 3.5Mtpa as a target level applied.

1.6 Underground Mining

WAI propose to consider mechanised sub-level open stoping (SLOS). The method offers favourable results in safety, cost, and dilution control. Stopes will be extracted in a retreat, top-down sequence, with adequate in-situ rock pillars left unmined for localised and regional stability. To reduce losses, on 5th -7th production year, top mineout levels have to be backfilled. This action reduces loses, giving the option to mine up to 60-70% pillars during the last 3-4 production years.

Due to the geological characteristics of the deposit (10 – 15m wide), it is challenging to achieve a steady state 2Mtpa production level, and much underground development is likely required.

1.7 Mine Production Schedule

The production profiles and ramp up for the combined open pit and underground operation scenarios are summarised in Figure 1.4 to Figure 1.6. Underground works are assumed to follow after the developed open pits identified by the shortlisted options.

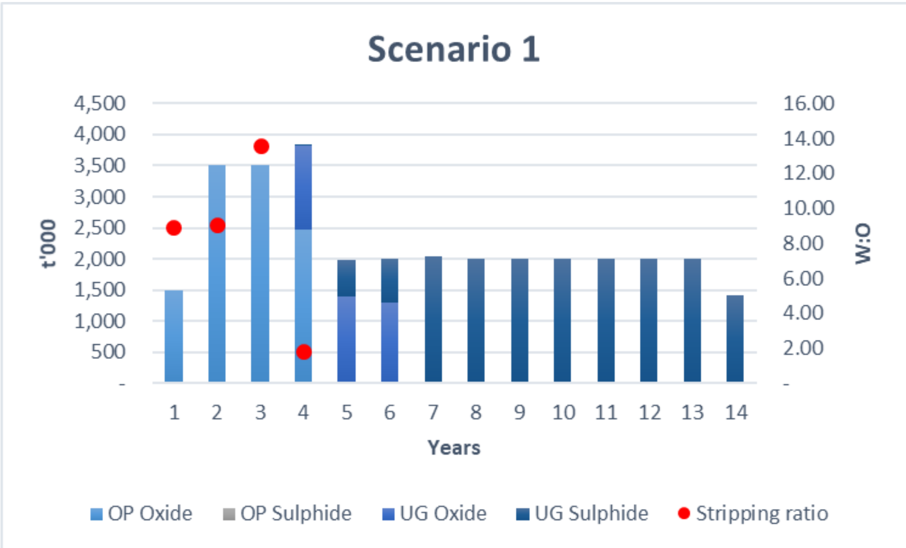


Figure 1.4: Production Profile Scenario 1 (Oxide Option 3)

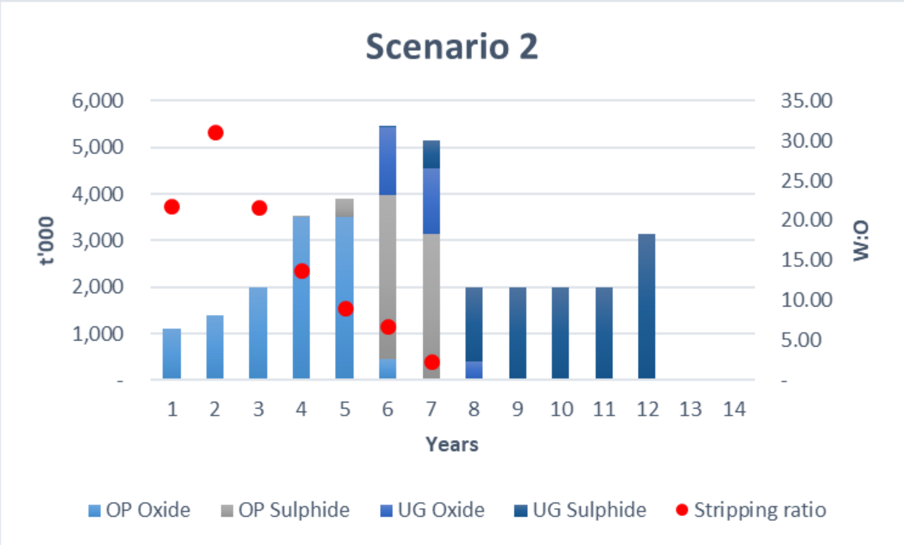


Figure 1.5: Production Profile Scenario 2, Oxide Processing Option 3

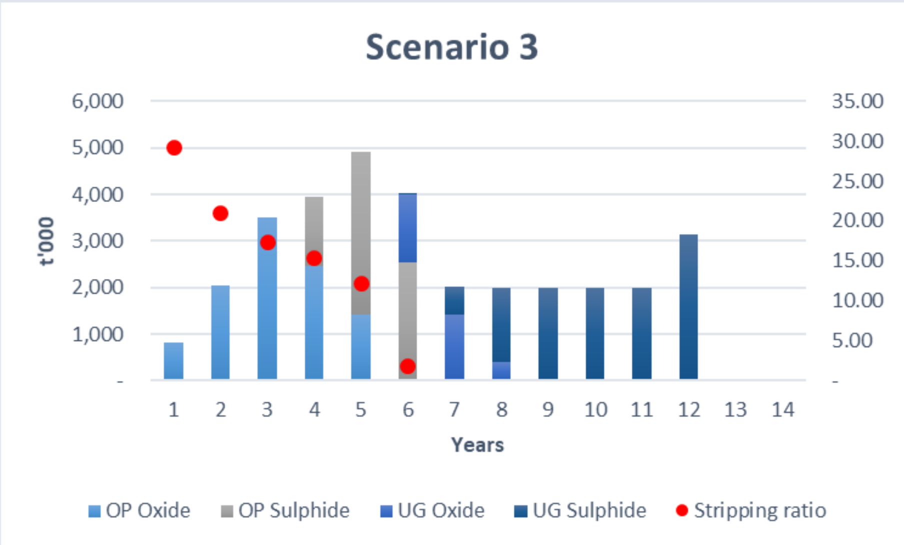


Figure 1.6: Production Profile Scenario 3, Oxide Processing Option 4

1.8 Infrastructure

The site is accessed from the main highway via an 8km unpaved road which continues through the licence area accessing the land to the South of the Project. In order to maintain this access there is the potential to provide ≈4.5km of track to the North of the open pit excavation as indicated in Figure 1.7.

Similarly, a new road will require construction providing approximately 3.5km of access to the indicated process plant and Administration areas. While the diverted track will only require reconstituting to the existing standards, the remainder as identified will require upgrading to a recognised standard supporting two-way traffic movements and significant logistical routing of up to 40 tonne vehicles.

There will be a requirement for approximately 11km of oversite roadways to meet the operational requirements, including connecting to and from the mineral process plant, warehouse(s), maintenance, and administration buildings. Despite the reasonably close proximity to the road and rail network and similarly the high voltage transmission line that traverses the north-western corner of the licence area there are no available utility services that may be directly utilised by the project.

The electrical power for the entire Project will be sourced from the transmission network via a new ≈4km, overhead supply line to a newly constructed transformer sub-station, located adjacent to the process building. The sub-station will have sufficient installed capacity to enable a duty /standby provision of 100% redundancy. An over-site 6kv reticulation is supplied from the substation and is transmitted via constructed overhead transmission lines to the various satellite locations for the site requirements. While the definitive equipment, plant and building requirements have yet to be developed it is considered by WAI that the operational demands and total production supply requirement is ≈25MW.

Currently there is no suitably identified source of raw water for the project location. The water requirements for the mine operations are substantial and surety of supply is of utmost importance for the continuity of operations. The following requirement is estimated for the maximum capacity achieved at sulphide Process plant: ≈2.5m^3/t at 3.5 Mtpa = 8.75million m^3/annum or ≈1,000m^3/hr. Assuming that 80% recirculation is achievable then the make-up requirement for raw water is 200m^3/hr and the quantity to pump from the TSF to the process plant is ≈800m^3/hr, assuming no losses. These values are estimated to be lower for the lower throughput maintained in the later years of the project development.

The permanent fuel storage facilities will be located in three areas:

- The Contractor mobile equipment area and compound;
- The primary crusher ROM pad; and
- A smaller storage facility at the process plant to primarily service the Emergency generator and refuel non-mining equipment.

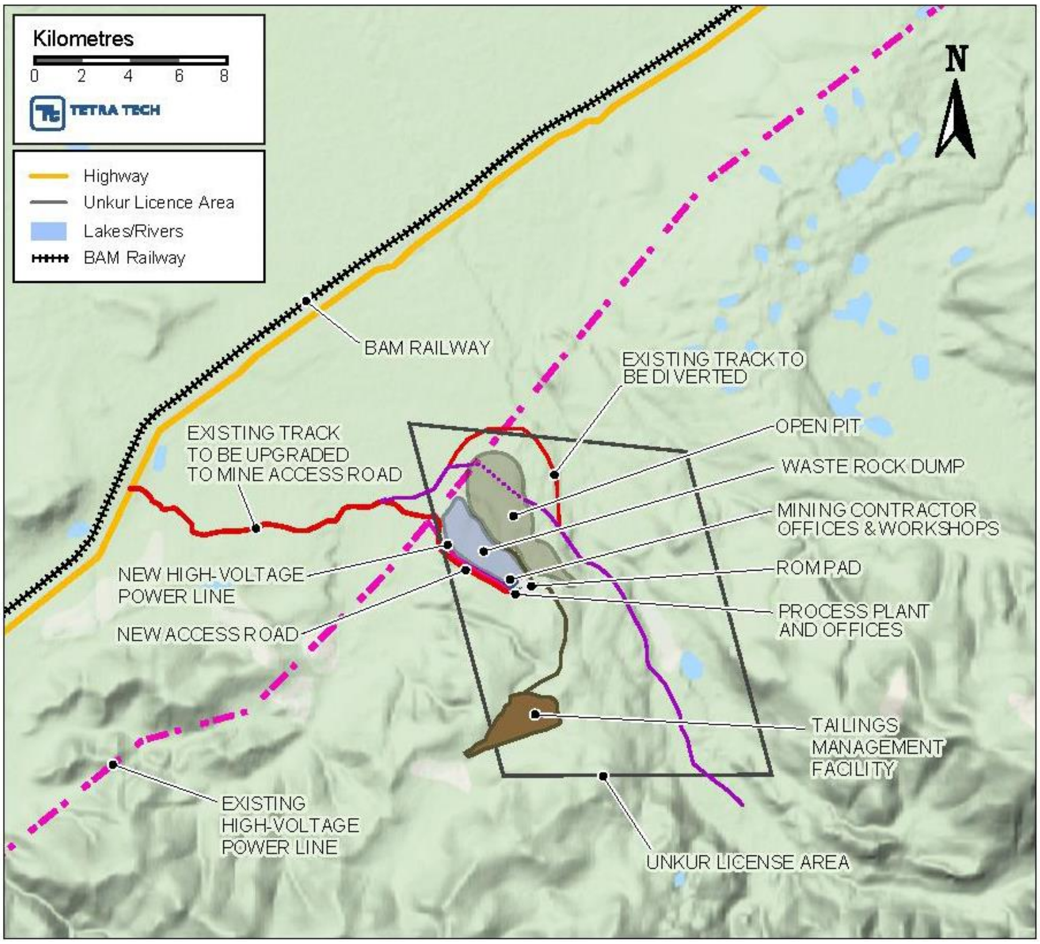


Figure 1.7: Proposed Indicative Site Layout

1.9 Environmental considerations

The Unkur Project operates in accordance with ЧИТ02522БР License for geological study, exploration and mining of copper, silver and associate components registered in 02.09.2014 and valid until 31.12.2039.

Currently the deposit is subject to extensive prospecting and evaluation activities that are planned to be completed in the first quarter of 2022 carried out in accordance with the Prospecting and Evaluation Programme approved by the government as evidenced by the expert conclusion report No. 025-02-11-2015 dated 17.08.2015.

Russian legislative requirements to the prospecting/exploration activities do not stipulate a process of environmental impact assessment ('OVOS'). However, according to the Rules of Mineral Deposit Exploration Programme Designing approved by the Order No.352 of the Russian Federation Ministry of Natural Resources and Ecology dated 14.06.2016 a prospecting/exploration programme should include an environment protection section describing the work area, nature and scale of potential environmental impact and mitigation measures suggested for the work period.

A prospecting and evaluation Programme for the Unkur deposit is developed in compliance with the mentioned requirement and describes environmental protection measures as evidenced by the provided document.

The deposit development will have a negative environmental impact. Therefore as the Project develops it will require an OVOS in accordance with the regulations, guidelines, and standards of the Russian Federation. OVOS consists of three main stages and includes public consultations at the stages of initial information submission, environmental impact assessment and preparation of justifications.

Also, current Russian environmental legislation and regulations require industrial environmental monitoring to be carried out within the zone of potential impact of industrial facilities at all stages of project implementation. Sanitary Rules 11-102-97 “Engineering and Environmental Survey for Construction” envisage the following stages of industrial environmental monitoring:

- Pre-construction (zero) monitoring;
- Monitoring at the construction stage (construction monitoring); and
- Monitoring at the operations stage.

Pre-construction (zero) monitoring was conducted in 2016 over the Project area and included preliminary baseline studies of ambient air, surface and groundwater, and soils conditions. The monitoring network included 9 points of snow cover sampling, 13 points of surface water sampling, 11 points of bottom sediments sampling. The results are presented in the provided report.

As for the required environmental permits, Russia's environmental permitting regime was revised on 1 January 2019 with an integrated permitting regime being introduced for the heaviest-polluting industrial facilities. This new regime is based on the existing classification of all emitting facilities into one of four categories, ranging from I to IV (that is, from highest to lowest environmental impact), with different levels of regulatory obligations for each category. Category I facilities are obliged to obtain a single integrated environmental permit instead of three separate permits for emissions to air, wastewater discharges and waste disposal. Applications can be made from 2019, and there is a transitional period until 1 January 2025.

At the operational stage, the Unkur Project will belong to the Category I facilities and therefore it will require an integrated environmental permit. In view of this WAI recommends considering the application of the best available techniques in the Project design at the early stage to avoid extra environmental charges in the future and ensure further compliance. To advance Project to prefeasibility level it will be necessary to include project permitting requirements and a timeline for environmental approvals including the required operating licenses and if these will cause the project to be deferred or cancelled. Developing an Environmental and Social Action Plan will assist in managing these aspects to ensure PFS requirements are satisfied.

Dependent on the future Project funding arrangements, and in accordance with the IFC[1] Performance standards (2012), the Project would be categorised as a Category 'A' industrial project, where it is *"likely to have significant adverse environmental impacts that are sensitive, diverse, or unprecedented"* and as such an ESIA would be required to be prepared based on comprehensive baselines studies examining the projects potential negative and positive environmental impacts, compared to feasible alternatives.

Data that are usually contained in an OVOS report is largely sufficient to provide a preliminary evaluation of the projects impact on the environment and evaluate the project setting for potentially significant environmental constraints. Some gaps to international standards exist, and some additional studies will be needed namely:

- Geochemistry;
- Soils;
- Hydrology and hydrogeology;
- Cultural heritage and archaeology;
- Socioeconomic baseline - stakeholder mapping;
- Biodiversity and ecosystem services;
- Climate and Energy Use;
- Air Quality; and
- Noise and vibrations.

1.10 Capital and Operating Costs – Mining

Mining Capital Costs were estimated based on WAI's cost database and project experience of similar operations. A summary of total CAPEX based on initial and sustaining capital discussed in Section 15.9.3 is presented in Table 1.9.

Table 1.9: Summary of Total Mining Capital Costs

	Open Pit (USD'000)	Underground (USD'000)		
		Equipment	Capital Development	Total Cost
CAPEX Scenario 1, Processing Option 3	**70,035**	105,000	85,000	**190,000**
CAPEX Scenario 2, Processing Option 3	**66,465**	95,000	85,000	**190,000**
CAPEX Scenario 3, Processing Option 4	**80,535**	95,000	80,000	**175,000**
CAPEX Scenario 4 (OP only), Processing Option 3	**70,035**	0		**0**

Open pit operating costs were estimated by WAI based on the generated production schedule, equipment operating cost estimates, consumable price estimates and labour estimates. Overall open pit costs are in the region of US$1.75/t rock mined, based on a separate earthmoving cost for ore ($/t)

[1] International Finance Corporation

and lower cost for hauling waste ($/m^3) from data recently derived by WAI from benchmarking real costs from similar-sized operations in the region.

Underground operating costs have been estimated at US$22/t with an additional backfilling of US$3/t.

1.11 Capital and Operating Costs – Processing

A summary table of the key parameters (revised by WAI) for all four oxide ore processing options is shown in Table 1.10.

Table 1.10: Physicals for Oxide Processing Options

OXIDE ORE	Option 1 (Agitated Leach)	Option 2 (Sequential Agitated Leach)	Option 3 (Heap Leach)	Option 4 (Sequential Heap Leach)
Copper Recovery, %	55.1	95.0	55.9	80.6
Silver Recovery, %	91.1	95.0	73.4*	73.4*
Capital Cost, $M	164.3	229.8	52.3	97.3
Operating Cost, $/t	18.3	27.4	5.8	15.4

* Represents 95% of 77.3% silver recovery from SART

For sulphide processing costs developed from Cost Mine data, the capital cost for a 10,000 tpd single-product flotation plant is **US$93.2** million and the operating cost is **US$8.98/t**. These costs are considered reasonable for scoping level accuracy.

Estimated Capital Costs for on-site and off-site infrastructure outside of the process plant battery are presented in Table 1.11. Power costs are based on an average of ₽2.0 per kW hour supplied from the Federal grid.

Costs are built on the basis of an oxide plus sulphide operation. The costs are reduced to a nominal 30% assuming a reduced oxide-only footprint for Scenario 1.

Table 1.11: Infrastructure Costs

	Scenario 1 oxide only	Scenario 2 oxide + sulphide	
	$	₽	$
CAPEX Estimate	10,000,000	2,365,891,370	30,171,505
OPEX Estimate/annum	1,500,000	297,150,322	4,015,545

1.12 Financial Analysis

WAI has undertaken a Preliminary Economic Assessment of the Unkur Project, using Discounted Cash Flow (DCF) analysis approach. As a part of this analysis, WAI performed the Project Trade-off study on processing options for oxide material treatment, and a solely open pit versus combination of the open pit and underground mining methods.

The Project Financial Model ("Model") has been developed using the production schedule developed by WAI, with all costs being estimated in 2021 US Dollars based on the available databases and previously completed studies.

All costs and cash flows reported in this section are shown in fixed US Dollars, with no inflation being incorporated.

Commodity Prices used in the valuation are as shown in Table 1.12.

Table 1.12: Commodity Price Assumptions Adopted in the Preliminary Economic Assessment		
Scenarios	**Consensus Price Assumption (as of May 2021)**	**Spot Price Assumption (avg. May 2021)**
Ag (US$ / oz)	25.00	28.00
Cu (US$ / t)	8,500	10,000
Cu (US$ / lb)	3.86	4.54

Oxide Processing Options considered are as following:

- Option 1: Agitated Leach
- Option 2: Tank Leach SX/EW
- Option 3: Heap Leach SART
- Option 4: Heap Leach SX/EW

A stand-alone Oxide Project with a targeted capacity of up to 3.5Mt of ore p.a., and processing Option 3 the financial results are as follows:

- At the Consensus Prices applied, the Project generates a *Free Cash Flow of US$149M* and after-tax **NPV of US$95.1M** at 8% discount rate (assuming the Company would be able to benefit from the tax relief in the initial years of project development).
- At the Spot Prices (as of May 2021) the Project generates US$237.9M Free Cash Flow and US$162.2M after-tax NPV at 8%.

This cash can be used to pay for the capital cost required for the sulphide flotation plant construction and underground development CAPEX.

A combined oxide and sulphide operation performed by open pit works has also demonstrated positive NPV results for oxide processing Options 3 and 4 and consequently have been included in the further analysis where underground mining is included following open pit depletion.

Overall, the Project can be strategically split into two main stages:

- Stage I: Oxide operation ("OP only", shown separately under Scenario 4 in Table 1.14); and

- Stage II: Sulphide operation undertaken by a combination of both OP and UG (Represented by Scenarios 1-3).

The payback period for the standalone Oxide Open Pit operation is expected to be achieved in the first year of ore production (at Spot Prices) or second year at the Consensus Prices. This coincides with the timing for capital investments required for the Second Stage of the project development, i.e., underground capital development, equipment procurement and sulphide plant construction.

The Stage II payback period is estimated between three to four years (depending on the Scenario Option), following second tranche of investments.

The capital costs, required to get Unkur project into the full production and ensure a self-sufficient operation are shown in Table 1.13 (summarised in Figure 1.8 through Figure 1.13). These costs cover all capital costs required for investments until the project generates enough cash flow to sustain production.

Table 1.13: Initial Capital Expenditures Summary (Excluding Sustaining)

Scenario	**Units**	**1**	**2**	**3**	**4**
Mining Options		OP (oxide Only) And UG	OP (Oxide and Sulphide) And UG	OP (Oxide and Sulphide) And UG	OP Oxide Only
Processing		*Heap Leach SART*	*Heap Leach SART*	*Heap Leach SX/EW*	*Heap Leach SART*
Stage I: Oxide only, OP	**US$'000**	**152,407**	**159,479**	**239,536**	**152,407**
OP Mining Equipment (Initial Payments for Leasing)	**US$'000**	70,035	55,755	80,535	70,035
Capitalised Interest Rate on Leasing (pre-production period)	**US$'000**	136	451	286	136
Oxide Plant	**US$'000**	52,300	52,300	97,300	52,300
General Infrastructure	**US$'000**	10,057	30,172	30,172	10,057
Contingency at 15%	**US$'000**	19,879	20,802	31,244	19,879
Stage II: inclusion of Sulphide Material, and UG	**US$'000**	**249,111**	**279,680**	**279,680**	**0**
Underground Capital Development	**US$'000**	75,000	75,000	75,000	0
Underground Equipment Cost	**US$'000**	75,000	75,000	75,000	0
Sulphide Plant	**US$'000**	66,618	93,200	93,200	0
Contingency at 15%	**US$'000**	32,493	36,480	36,480	0
Total Initial Capital Cost	**US$'000**	**401,518**	**439,159**	**519,216**	**152,407**
Including Pre-production Capital Cost, payable in Year 0	**US$'000**	77,615	103,355	154,915	77,615
Residual Value	**US$'000**	19,671	12,200	22,112	13,671

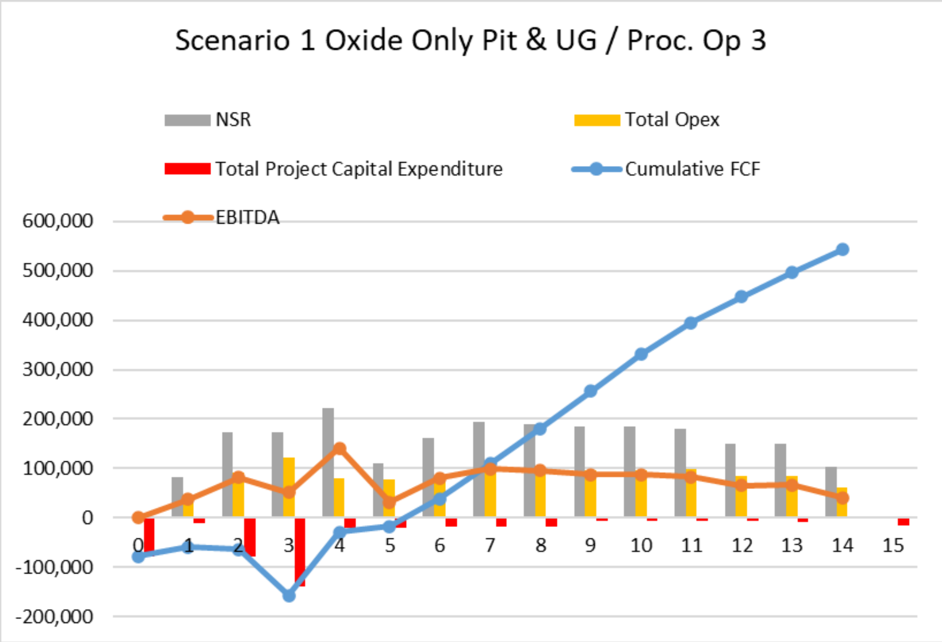


Figure 1.8: Financial Results for Scenario 1, Consensus Prices

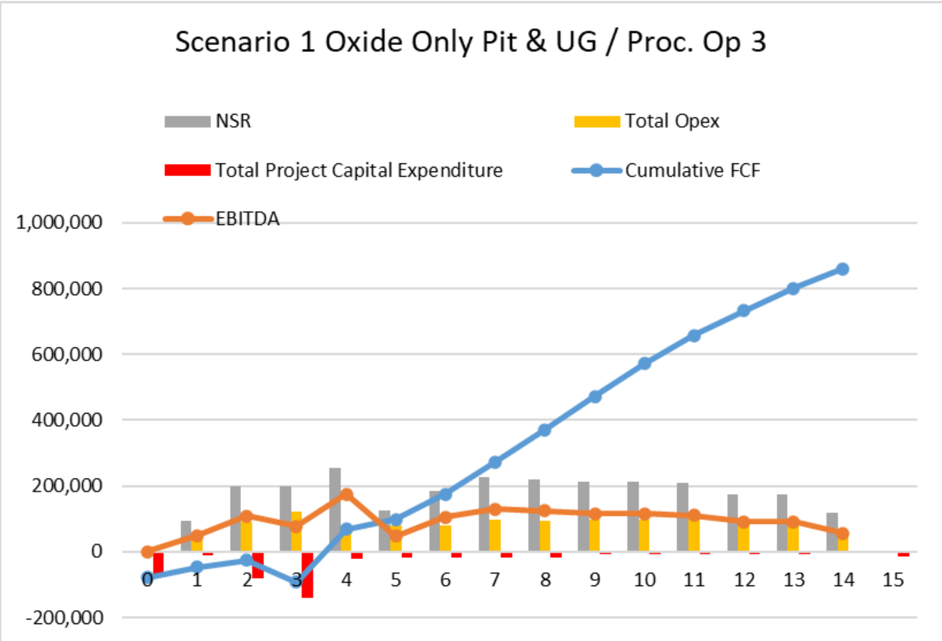


Figure 1.9: Financial Results for Scenario 1, Spot Prices

For the **Scenario 1**, the initial capital cost required to enable Stage I (OP operation) achieving full production capacity is estimated at **US$152.4M**, this includes US$70M for OP equipment, US$52.3M for Oxide Plant and US$10M for general infrastructure. CAPEX required to get **Stage II** in full production (scheduled for Year 3 onwards) has been estimated at **US$249M**, and includes 75M for UG capital development, US$75M for UG Equipment, and Sulphide Plant Cost of US$66.6M. These costs also include contingency at 15%, and capitalised interest rate, assumed for leasing programme.

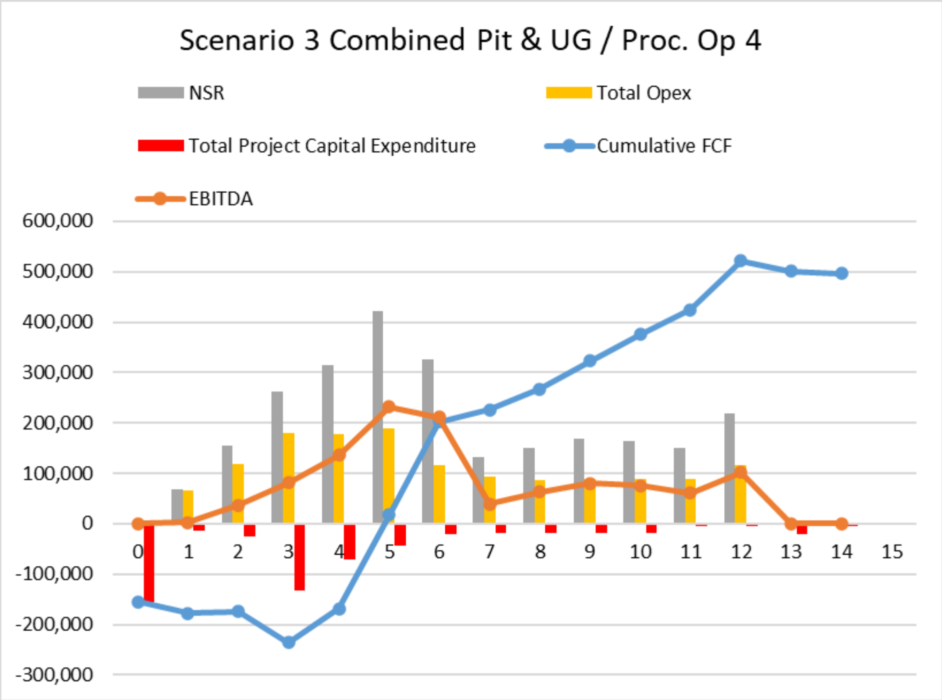


Figure 1.10: Financial Results for Scenario 3, Consensus Prices

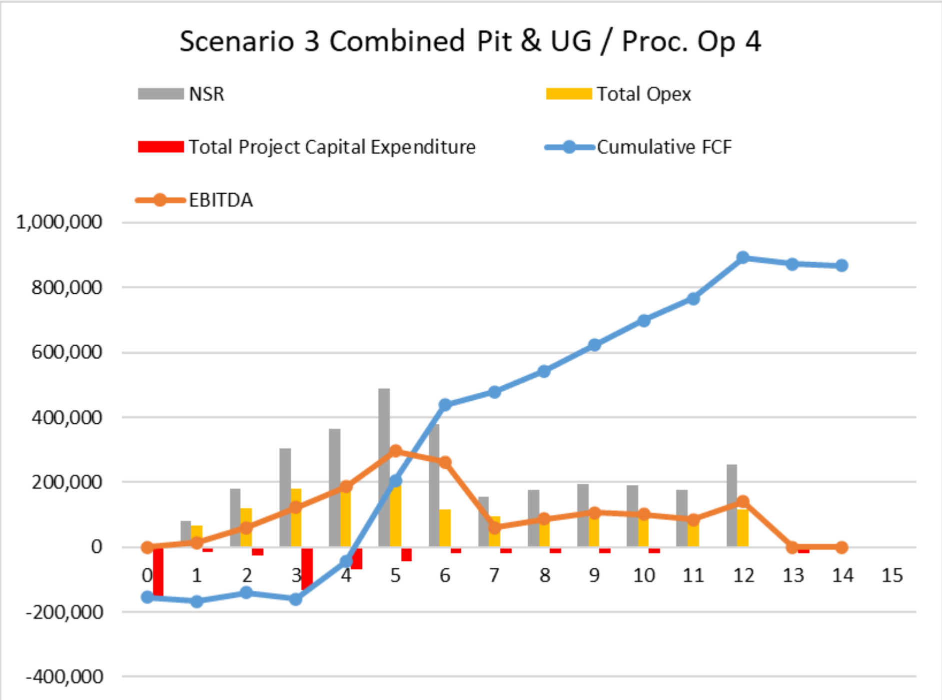


Figure 1.11: Financial Results for Scenario 3, Spot Prices

Hence the **Scenario 3** is associated with more expensive processing method (Heap Leach SX/EW) and bigger open pit (given that both oxide and sulphide materials are mined via OP prior switching to the UG), it attracts highest capital costs required for Oxide Processing Plant, compared to all other scenarios.

The initial capital cost required for enable Stage I (OP operation) and achieve production capacity target is estimated at **US$239.5M**, this includes US$80.5M for OP equipment (dictated by the bigger pit), US$97.3M for Oxide Plant and US$30M for general infrastructure.

CAPEX required to get Stage II in full production (allocated from Year 3 onwards) has been estimated at **US$249M**, and includes 75M for UG capital development, US$75M for UG Equipment, and Sulphide Plant Cost of US$66.6M (including contingency at 15%, and capitalised interest rate, assumed for leasing programme).

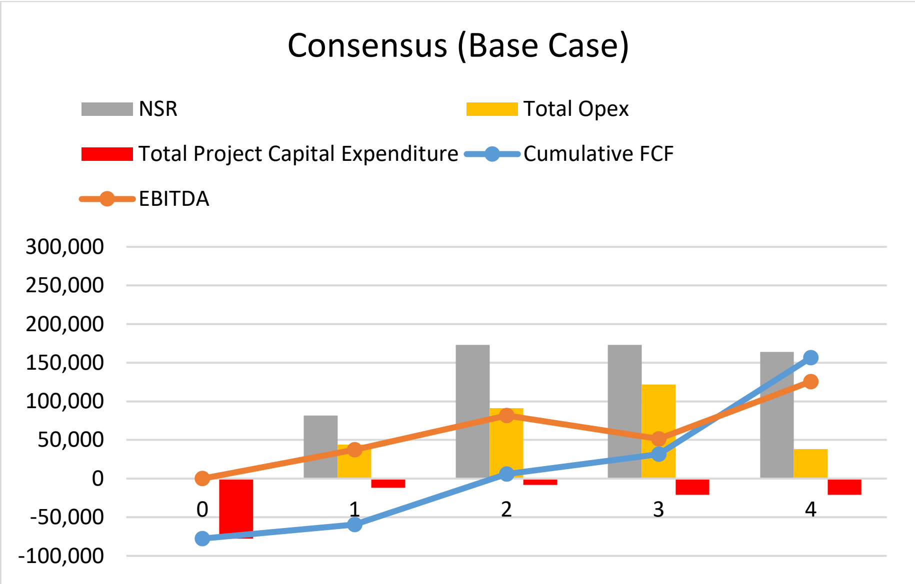


Figure 1.12: Financial Results for Scenario 4, Consensus Prices

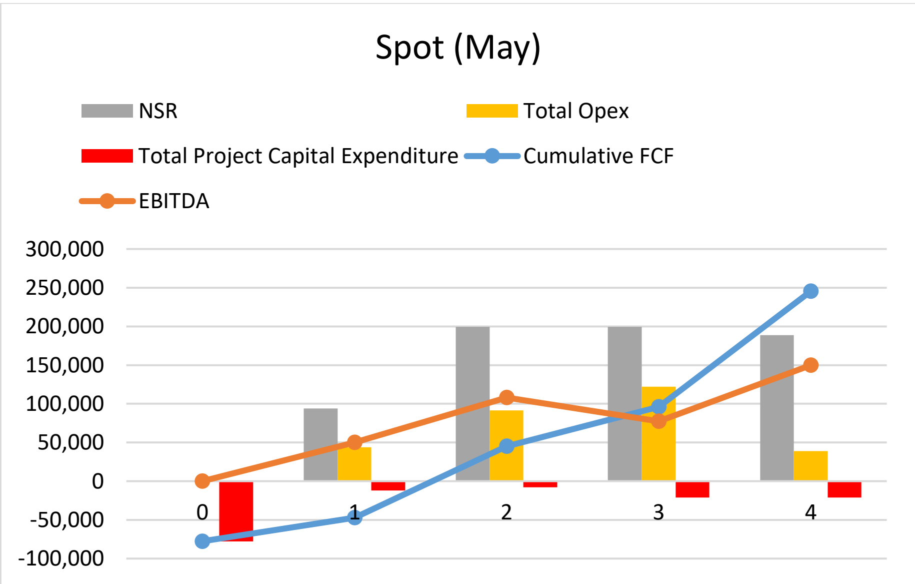


Figure 1.13: Financial Results for Scenario 4, Spot Prices

Following the above, for the Standalone operation of Oxide Materials (**Scenario 4**) the capital requirements have been estimated at **US$152.4M**, this includes US$70M for OP equipment, US$52.3M for Oxide Plant and US$10M for general infrastructure (including contingency at 15%, and capitalised interest rate, assumed for leasing programme).

WAI has run a trade-off assessment of the Unkur project both for solely open pit operation (where oxide and sulphide materials are mined) versus a combination of open pit and underground mining (where the deeper materials are mined via underground method). Given the depth of the ore body and associated higher stripping costs, the solely open pit option shows lower economic return compared to the options with an inclusion of the underground operation. Table 1.14 provides summary of the financial results compared between considered scenarios. These financial results are shown for the entire project as a whole, including both stages. However, the payback period has been considered for each stage individually, given the requirement for two tranches of major capital investments.

Table 1.14: Comparative analysis of the Project Financial Results for Open Pit and Underground Scenarios (Post-Tax, including Tax Relief Benefits)

Scenarios		**1**	**2**	**3**	**4**
Mining Options		OP (oxide Only) And UG	OP (Oxide and Sulphide) And UG	OP (Oxide and Sulphide) And UG	OP Oxide Only
Processing		*Heap Leach SART*	*Heap Leach SART*	*Heap Leach SX/EW*	*Heap Leach SART*
Consensus Prices					
NPV at Discount Rate of 8.00%	**US$ M**	205,562	143,400	193,442	95,122
IRR	%	26.7%	16.4%	20.9%	46.3%
Stage I Payback period (FCF)	Year	3	3	3	3
Stage II Payback Period (FCF)	Year	3	4	2	n/a
Spot Prices (as of May 2021)					
NPV at Discount Rate of 8.00%	**US$ M**	380,410	322,637	412,729	162,231
IRR	%	44.4%	25.6%	34.1%	70.1%
Stage I Payback period (FCF)	Year	2	2	2	2
Stage II Payback period (FCF)	Year	2	4	3	n/a

This report provides a summary of the shortlisted Scenarios and combined OP and UG operations. Figure 1.14 and Figure 1.15 provide a summary of the financial results achieved for both price decks.

The preliminary economic analysis shows that the Consensus Prices **Scenario 1** results in the highest NPV and IRR numbers, benefiting from the lower capital and operating costs even at the lower processing recovery rates achievable (compared to the Processing Option 4: Heap Leach SX/EW).

However, **Scenario 3** (Processing Option 4) takes an advantage at the higher price deck. As the trade-off between higher processing capital, operating costs and increased recovery rate works towards project economics improvement at the higher commodity prices applied.

WAI recommends that the options with the inclusion of underground operations are selected for further analysis at subsequent stages of the project development.

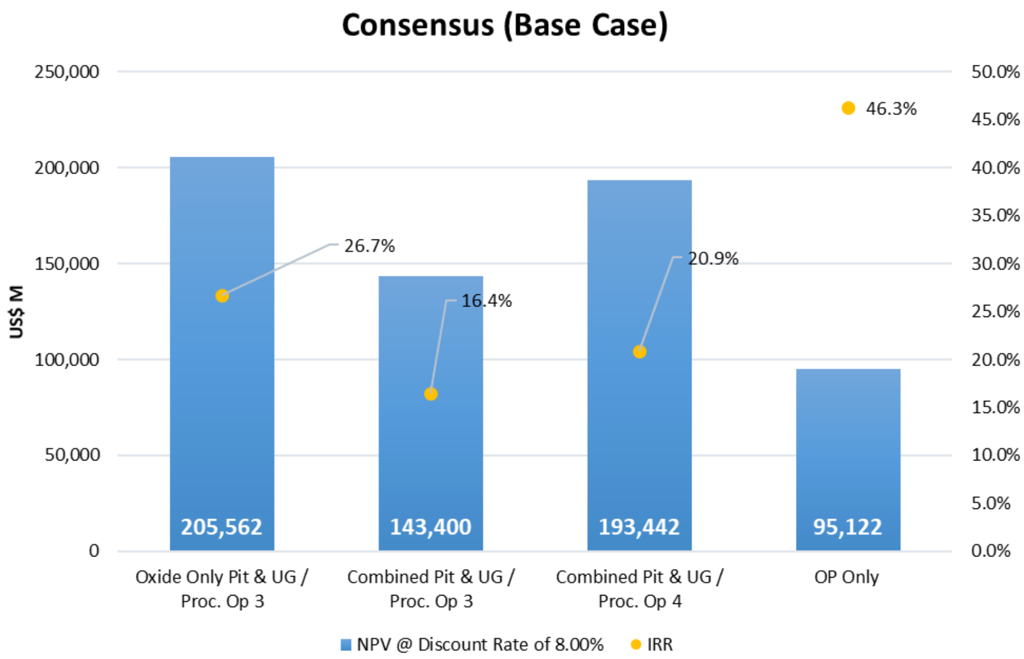


Figure 1.14: Financial Results shown for the Consensus Prices Deck

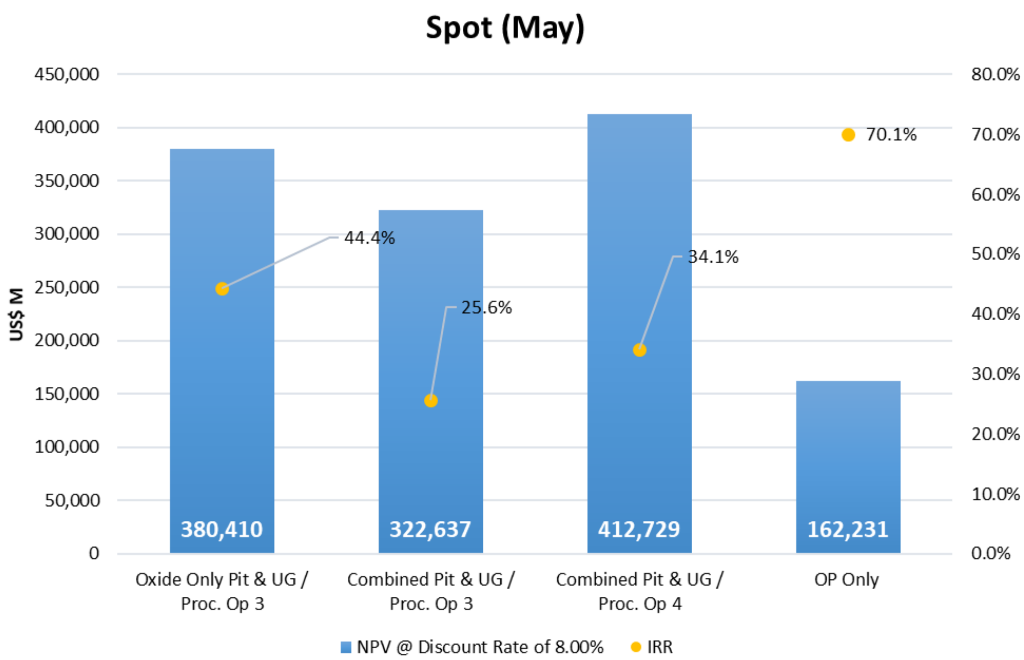


Figure 1.15: Financial Results shown for the Spot Prices Deck

2 INTRODUCTION

2.1 Terms of Reference and Reporting Aims

Azarga Metals Corp. (AZR) is listed on the Toronto Stock Exchange TSX Venture Exchange (TSX-V: AZR) and has the right to undertake exploration and development of the Unkur copper-silver deposit under the terms of an evaluation and mining licence held by its local subsidiary LLC Tuva Cobalt valid until December 2039. Unkur represents one of 7 prospective tracts identified by the USGS for undeveloped copper in the Kodar-Udokan basin, as summarised in Table 2.1.

Table 2.1: USGS Prospective Tracts for Undeveloped Copper in the Kodar-Udokan Basin

Name	Status	Resource category[2]	Tonnes, Mt	Cu grade %	Ag grade g/t
Krasnoe	Prospect	P1	33	1.50	NA
Ingamakit	Partial delineation of resources	C2+P1	123	0.88	24
Sakin	Prospect	P1+P2	122	0.90	7
Sulban	Prospect	P1	95	1.07	NA
Udokan	Development	B+C1+C2	1,310	1.44	13
Unkur	Deposit with upside	C2+P1	143	0.75	64
Burpala	Partial delineation of resources	C2+P1	43	1.19	56

This report was prepared jointly as a National Instrument 43-101 (NI 43-101) Technical Report for AZR by Wardell Armstrong International (WAI), Items 1 – 8 and 13 – 26, and SRK Consulting Ltd., Items 9 – 12 (see Section 2.2).

The aim of this study is to assess and test the combined potential of open pit transitioning to underground mining methods considering four process options and two economic scenarios:

- Scenario 1 is fast-tracking oxide only using Option 3 (cyanide heap leach-SART) with minimum CAPEX/fast construction and lower site infrastructure CAPEX; and
- Scenario 2 is best case, max NPV (minimise CAPEX if possible) using the best-case option for oxide out of four oxide options, namely:
    - Option 1: Agitated cyanide leach-SART
    - Option 2: Sequential acid and cyanide agitated leach;
    - Option 3: Cyanide heap leach-SART;
    - Option 4: Sequential acid and cyanide heap leach.

Which is determined through financial modelling/trade-off study and also to include conventional sulphide processing. A low-tax regime will also be considered as part of the financial trade-off.

[2] These estimates should not be relied upon and has been superseded by the mineral resource discussed in Item 14 of this report. See section 6.6.1 for definition of GKZ resource/reserve definition and classification.

Process parameters for all the above options will be presented but it became clear in early trade offs and results of previous studies that Options 1 and 2 were not economically viable. So Options 3 and 4 were considered in detail for the combined oxide with sulphide operation.

The key elements included within the assessment are listed below:

- Mineral Resource Estimation;
- Hydrological and hydrogeological review;
- Mining geotechnical review;
- Open pit mining study;
- Transition to underground mining study;
- Mine production scheduling;
- Mining capital and operating cost estimation;
- Mineral processing review including disposal of tailings
- Environmental and social review; and,
- Financial analysis.

This report presents the two scenarios and a clear distinction between Scenario 1, Option 3 and Scenario 2 combining the oxide and sulphide production considering the trade-off between oxide Options 3 and 4.

WAI has adopted the definition of Mineral Resources as outlined within the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves (CIM 2014) in order to classify the Mineral Resources.

2.2 Qualifications of Consultants

The Consultants preparing this technical report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, underground mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in AZR. The Consultants are not insiders, associates, or affiliates of AZR. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between AZR and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience, and professional association, is considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and is a member in good standing of appropriate professional institutions.

The QP responsible for specific items are as follows:

- Ché Osmond, Technical Director, WAI is the QP responsible for Items 1 through 8, and Items 15 through 26 (except 21.3);
- Robin Simpson, Principal Resource Geologist, SRK Consulting is the QP responsible for Items 9 through 12;
- James Turner, Technical Director, WAI is the QP responsible for Items 13, 17, and 21.3; and
- Alan Clarke, Associate Director, WAI is the QP responsible for Item 14.

2.2.1 Details of Inspection

Robin Simpson visited the site during December 10, 2014 and October 13, 2016, accompanied by representatives of AZR. During this visit AZR drilling and trenching teams were active, and the qualified person was able to observe the protocols in action for collecting, handling, analysing, and storing samples.

Mr Simpson previously visited site during December 10, 2014. This earlier site visit included an inspection of outcropping copper-bearing horizons, and examination of historical drill core from the 1969-1971 and 1975-1978 exploration campaigns.

WAI consultants have not conducted a site visit to the exploration area at the time of writing this report. It has not been possible for WAI to access the site due to international and regional travel restrictions in light of the Covid-19 pandemic, namely:

- A ban on foreign citizens entering the Russian Federation for ordinary travel purposes since March 2020;
- A suspension of direct flights between UK and Russia for specific travel purposes since December 22, 2020, making travel for UK Citizens not resident in Russia for repatriation or emergency purposes only; and
- Regional restrictions and a mandatory quarantine enforced by regional authorities in Zabaikal imposed on any citizens arriving from outside of the region for much of 2020.

This report is therefore prepared in lieu of a recent site inspection by WAI and relies on the recent witness of Mr Simpson.

2.2.2 Sources of Information and Extent of Reliance

Supporting information has been sourced from:

- Geological wireframes and resource block model effective 2020 provided by SRK;
- Discussions with personnel from AZR;
- Observations made by SRK during two visits to site (December 10, 2014, and October 13, 2016);

- Observations made by SRK during a visit to SGS Laboratories in Chita, the primary laboratory for 2016-2017 Azarga Metals' samples (October 14, 2016);
- A 2020 metallurgy tests report by VNIItsvetmet;
- A 2021 metallurgy tests report by VNIItsvetmet;
- An August 2018 Technical Report and Preliminary Economic Assessment by Tetra Tech;
- A March 2018 Technical Report and Mineral Resource Estimate by Tetra Tech;
- A March 2016 Technical Report for the Unkur Copper-Silver Deposit by SRK;
- A March 2017 Technical Report for Unkur Copper-Silver Deposit by SRK;
- A February 2015 report by ZAO SGS Vostok Ltd., describing the results from metallurgical testwork done on a 350kg sample collected from an outcrop of oxidised mineralised material on the Unkur property;
- A two-volume report from the results of exploration undertaken by the Naminginskaya expedition team at the Unkur copper project in 1969-1971;
- A three-volume report from the results of exploration undertaken by the Lukturskaya expedition team at the Unkur copper project and Klyukvennoye deposit in 1975-1978;
- Information from WAI's own sources for benchmarking based on similar operations in the region including the Kodar District, and
- Information obtained from the public domain.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in WAI's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by subject to the terms and conditions of its contract with WAI and relevant securities legislation. The contract permits AZR to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with SBR. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

2.3 Effective Date

The effective date for issue of this report is 16 August 2021. The effective date for reliance on information contained in this report is 31 July 2021 as no data or material information used in its compilation was considered after this date.

2.4 Terms and Units of Measurement

All currency amounts are stated in US dollars ($ or USD or US$) or Russian Rubles (RUB₽ or ₽) unless otherwise specified. The units of measure presented in this report are metric units except for bullion prices which are quoted in troy ounces (toz). Silver values are reported in grams per tonne (g/t) or parts per million (ppm), respectively. Gold is also reported in grams per tonne (g/t). Tonnage is reported as metric tonnes (t), unless otherwise specified.

List of Abbreviations and Acronyms:

A.P. Karpinsky Russian Geological Research Institute	VSEGEI
Ammonium Nitrate-Fuel Oil	ANFO
Acid Rock Drainage	ARD
ALS Global	ALS
Atomic Absorption Spectroscopy	AAS
Azarga Metals Corp	AZR
Baikal-Amur Mainline	BAM
Bottle roll test	BRT
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Capital Cost	CAPEX
Central Geological Research Institute	TsNIGRI
Certified Reference Material	CRM
Commonwealth of Independent States	CIS
Copper Equivalent	CuEq
Copper Sulphide	Cu2S
Copper	Cu
Counter Current Decantation	CCD
Cyanide	CN
Discounted Cash Flow	DCF
environmental and social impact assessment (also known as a Russian OVOS)	ESIA
General and Administrative	G&A
Global Positioning System	GPS
Gold	Au
Heating Ventilation and Cooling	HVAC
Hydrogen Cyanide	HCN
Inductively Coupled Plasma	ICP
Internal Rate of Return	IRR
International Finance Corporation	IFC
Institute of Geotechnologies	IGT
International Organization for Standardization	ISO
International Union for Conservation of Nature	IUCN
Lerchs-Grossmann	LG
Life-of-Mine	LOM
Locked Cycle Tests	LCT

Methyl isobutyl carbinol	MIBC
Mineral Extraction Tax	MET
National Instrument 43-101	NI 43-101
Net Present Value	NPV
Net Smelter Royalty/Return	NSR
NPV Scheduler	NPVS
Open Pit	OP
Operating Cost	OPEX
Preliminary Economic Assessment	PEA
Preliminary Feasibility Study	PFS
Pole-Dipole	PDIP
Potassium Amyl Xanthate	PAX
Qualified Person	QP
Quality Assurance/Quality Control	QA/QC
Run-Of-Mine	ROM
Russian State Commission on Mineral Resources	GKZ
SGS Vostok Limited	SGS
Silver	Ag
Silver sulphide	Ag2S
Sodium cyanide	NaCN
Sodium hydrogen	NaOH
Sodium hydrogen sulphide	NaSH
Solvent extraction and electrowinning	SX/EW
SRK Consulting (Russian) Ltd.	SRK
Strengths, Weaknesses, Opportunities and Threats	SWOT
Sub-Level Open Stoping	SLOS
Sulpidisation, Acidification, Recycling and Thickening	SART
Tailings Management Facility	TMF
Tailings Storage Facility	TSF
Three Dimension/al	3D
Treatment Charge	TC
Two Dimension/al	2D
Udokan Copper Operating Company	UCOC
United Kingdom	UK
Underground	UG
World Health Organisation	WHO
Waste Rock Dump	WRD
Whole Ore Leach	WOL
World Geodetic System	WGS
X-Ray Fluorescence	XRF

3 RELIANCE ON OTHER EXPERTS

The Author's opinion contained herein is based on information provided to the Author by AZR Corporate (Dr Alexander Yakubshuk), and SRK Consultants Ltd., throughout the course of the investigations. WAI has relied upon the work of other consultants in the project areas in support of this Technical Report. The sources of information include data and reports supplied by AZR and SRK personnel as well as documents referenced in Section 2.3.1 of this report.

Historic information provided to WAI and used to prepare this report was acquired through SRK and AZR from a variety of sources that have had access to geologic, metallurgical, environmental and engineering studies and electronic copies of historical reports prepared in the 1970's from predecessor companies.

For the purpose of this Technical Report, WAI has relied on ownership information provided by AZR. The ownership information is relied upon in Section 4 and the relevant sections of the Summary. WAI has not researched property title or mineral rights for the Project and expresses no opinion as to the ownership status of the property.

Except for the purposes legislated under provincial securities laws, any use of this Technical Report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

Information from this section is drawn from Tetra Tech (2018) and reliant thereupon the accuracy of this information.

4.1 Property Description and Location

The deposit is located in the north of Zabaikalsky Krai some 600km north of the regional centre at Chita. The licence area comprises 53.9km2 in relatively flat, seasonally swampy taiga terrain in the north from 400-500mRL rising to medium mountain relief up to 1200m in the south approaching the rolling foothills of the Udokan Range. The Kemen River cuts the licence north-south which is flanked by larch forest. The Baikal-Amur Railway runs within 7km of the licence area with the nearest station at the local centre at Nova Chara, 12km away (Figure 4.1).

Figure 4.1: Property Location Map (after Tetra Tech, 2018)

4.2 Licence Tenure

The licence is an evaluation and mining licence held by LLC Tuva Cobalt, the local subsidiary of AZR, valid until December 2039 with the evaluation phase completed by September 2020. Azarga Metals currently owns 100% of the project through 100% ownership of Azarga Metals Corp (AZR). AZR owns 100% of Shilka Metals (Cyprus) LLC which in turn owns 100% of LLC Tuva Cobalt. The coordinates of the exploration license are shown in Table 4.1.

Table 4.1: Exploration License Coordinates

Mining Licence ЧИТ02522БР		
Corner no	**Northing Coordinate**	**Easting Coordinate**
1	56˚48’01”	118˚34’20”
2	56˚52’36”	118˚32’03”
3	56˚52’14”	118˚38’45”
4	56˚47’59”	118˚40’45”

The license (No. ЧИТ02522БР) covers an area of 53.9 km^2 and allows the owner to perform geological study, exploration, and production of copper, silver, and associated components.

The licence details and conditions are shown in Table 4.2 . The license area is shown Figure 3.2.

The exploration licences give the recipient the authority to use the subsoil for the purposes of geological investigation within the licence area, for exploration, and appraisal of the gold and silver deposits. The licence area has the status of a “geological allotment” with the preliminary borders outlined and an unlimited licenced depth for investigation. There are no specially protected natural territories within the limits of the licence.

Table 4.2: Licence Details

Item	Description	
License No.	ЧИТ02522БР	
License Name	Licence Agreement on conditions of subsoil use for mining of copper, silver, and associated minerals in the Unkur Project	
Valid From	02/09/2014	
Expiry	31/12/2039	
Area	53.9 km^2	
GKZ Resource Approval	Not included in the State Balance Sheet	
The GKZ Prognostic Resources[3], 2014	Prognostic Resources:	
	Block No 1	P1 – tonnage is 16,516.5 kt, Cu grade - 0.90%, Cu content – 148.6 kt
	Block No 2	P1 - tonnage is 3,964, Cu grade - 0.65%, Cu content - 25.8 Kt
	Total P1	P1 - tonnage is 20.480.5, Cu grade - 0.85%, Ag grade - 77.96g/t, Cu content - 174.4 Kt, Ag content - 1,600 t
Conditions	Compliance with the Russian legislation, advanced geological survey, full-extraction of on-balance Mineral Reserves/Resources	
	Industrial and occupational safety	
	Environmental protection	
	Social and economic development of region	

4.3 Royalties, Agreements and Encumbrances

The licence states the charges and taxes relating to subsoil use, which include the following:

- Mineral extraction tax as per Russian Federation laws;
- Water tax as per Russian Federation laws;
- A single payment of RUB₽20.856 million for the right to use subsoil for mining copper and associated minerals; and
- Other charges and taxes prescribed by the tax laws of the Russian Federation.

According to the license conditions, the holder of the license (LLC Tuva Cobalt) shall pay the following rates:

- Early-stage Exploration: For the entire subsoil area, except for the deposit areas at the exploration stage, the rate for the first year is RUB₽50/km2; for years 2 to 5 the rate will be RUB₽162/year/km2; and from the fifth year RUB₽225/year/km2;
- Exploration Stage: RUB₽1,900/km2 for the first year; then
- RUB₽8,707/year/km2 for the second and third years of work.

[3] These estimates should not be relied upon and has been superseded by the mineral resource discussed in Item 14 of this report. See section '5.1.9 Resource/reserve classification system of the Soviet Union' for definition of GKZ resource/reserve definition and classification.

The royalties to be paid to the Russian Federation for extracting copper and silver are 8% and 6.5%, respectively. In addition, as described in more detail in Section 4.6, the vendors who sold part of their shareholding to European Uranium Resources Ltd. will retain a 5% net smelter return (NSR) royalty. WAI understands that as of October 2020 the RF Tax authority has introduced a 'Krent' factor to the royalty and increased royalties by a factor of 3.5 which if applied may mean an effective tax rate of 23-28% on NSR for producers. The implications for this is discussed further in the financial analysis 'Section 22'.

4.4 Environmental Liabilities and Permitting

WAI is not aware of any existing liabilities arising from previous industrial activity and land use and it is not part of the scope of this study to investigate historical impacts caused by project activities to date.

Currently the deposit is subject to extensive prospecting and evaluation activities that are planned to be completed in the first quarter of 2022. Prospecting and evaluation works include walkovers, trenching, core drilling, sampling, assaying, etc. and are carried out in accordance with the Prospecting and Evaluation Programme approved by the government as evidenced by the expert conclusion report No. 025-02-11-2015 dated 17.08.2015.

Russian legislative requirements to the prospecting/exploration activities do not stipulate a process of environmental impact assessment ('OVOS'). However, according to the Rules of Mineral Deposit Exploration Programme Designing approved by the Order No.352 of the Russian Federation Ministry of Natural Resources and Ecology dated 14.06.2016 a prospecting/exploration programme should include an environment protection section describing the work area, nature and scale of potential environmental impact and mitigation measures suggested for the work period.

Prospecting and evaluation Programme for the Unkur deposit is developed in compliance with the mentioned requirement and describes environmental protection measures as evidenced by the provided document.

The deposit development will have a negative environmental impact. Therefore, as the Project develops it will require an OVOS in accordance with the regulations, guidelines, and standards of the Russian Federation.

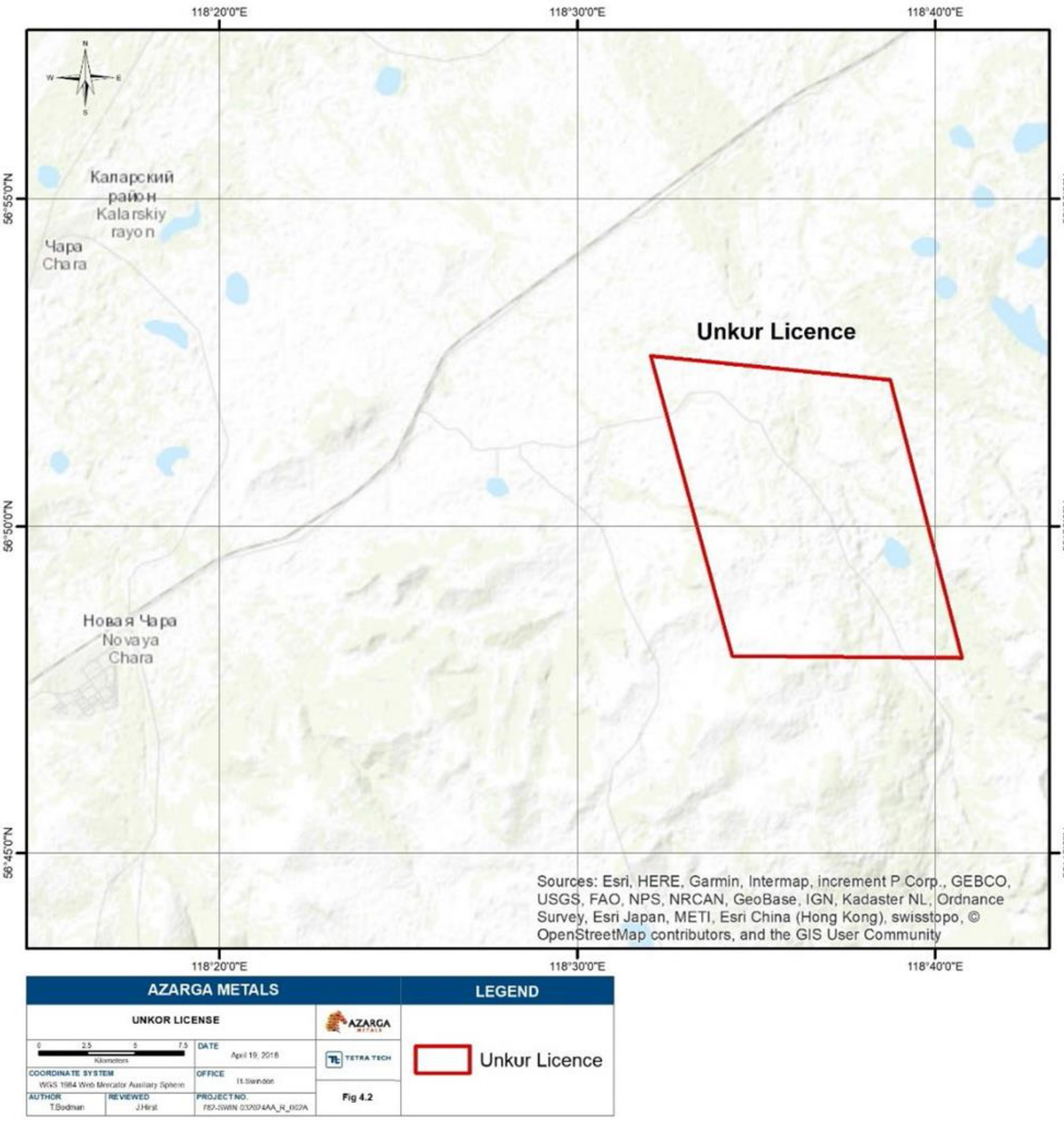


Figure 4.2: Unkur Licence Boundary (after Tetra Tech, 2018)

No permitting is required until the Project reaches the Feasibility Study stage. The exploration stage only requires observation of existing environmental laws and regulations. Details of the approvals process for project development are outlined as follows (after Tetra Tech report, 2018):

- Approval of a project design for geological investigation of subsurface mineral resources (early-stage exploration) which has previously received a positive conclusion in accordance with Article 36.1 of the Russian Federation Subsoil Law;
- Submission of prepared documents, no later than 02/09/2020, based on geological study of the subsurface mineral resources to the State Appraisal of Reserves of Commercial Minerals (GKZ) in accordance with Article 29 of the Russian Federation Subsoil Law. AZR was granted a new deadline of March 2021 and it can be further

extended due to COVID-19 situation. Approval of a project design for detailed exploration, no later than 02/09/2020, which has previously received a positive government conclusion in accordance with Article 36.1 of the Russian Federation Subsoil Law of the Russian Federation. This will be delayed in accordance with delayed GKZ approval;

- Submission of prepared documents, no later than 02/09/2024, based on detailed exploration results to the State Appraisal of Reserves of Commercial Minerals in accordance with Article 29 of the Russian Federation Subsoil Law;
- Preparation and approval, no later than 02/09/2026, of the technical project of deposit exploration arranged in accordance with Article 23.2 of the Russian Federation Subsoil Law;
- Preparation and approval of the technical project of abandonment and suspension of workings, drillholes, and other underground workings arranged in accordance with Article 23.2 of the Russian Federation Subsoil Law a year ahead of the planned completion of the deposit development;
- Submission of the annual information report on the works carried out on site, no later than January 15 of the year following the reporting period; the order of presentation of these materials is determined by the Federal Agency on Subsoil Use and its territorial bodies; and
- Submission of annual statistical reporting (5-GR, 70-TP, 71-TP, 2-LS, 2-GR, 7-GR forms, etc.) within the prescribed time limits.

The Project is not in a protected woods territory and AZR expects that no tree cutting will be required for the purposes of exploration. Therefore, it should be possible for exploration to proceed without a forestry permit at this stage.

5 ACCESSIBILITY, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Information from this section is drawn from Tetra Tech (2018) and reliant thereupon the accuracy of this information.

5.1 Physiography

The Project area is located in the northern slopes of the Udokan Range in the catchment of the Kemen and Unkur Rivers, which are right-bank tributaries of the Chara River. The area of the deposit is characterized by low and medium mountain relief with absolute elevations of 1,100 to 1,200 m, with local differences in elevation of 100 to 200 m. Flat watersheds and smooth hillsides are found in the northern portion of the area with an elevation of 400 m (Figure 5.1). The deposit and surrounding area is covered by taiga vegetation (swampy coniferous forest), as is typical between the tundra and steppes of Siberia. The main forest-forming species is Dahurian larch.

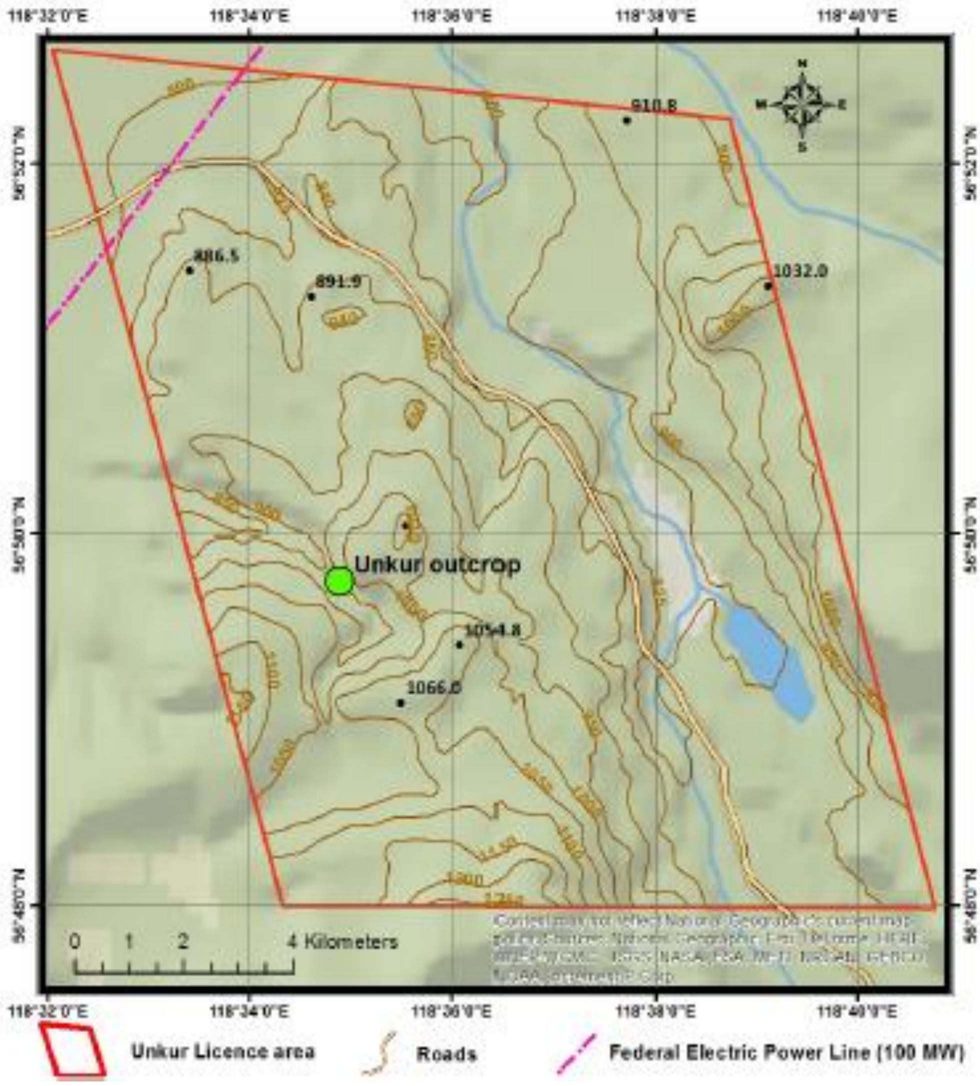


Figure 5.1: Unkur Topography and Drainage (after SRK, 2015)

5.2 Operating Season

The Project area has a harsh continental climate with very cold and long winters and short hot summers. During the cold period, the terrain is dominated by a stable Siberian anticyclone with significant temperature inversions. The air temperature varies depending on the relief. Average air temperature in January ranges from –27.8°C at altitude in the Project area and –33.2°C in the Chara valley. The winter air temperature minimum is –57°C at lower levels and –47°C at altitude. The July air temperature maximum is +32°C and at the foothills it is +27°C. The cold and long winters (October to April) are characterised by high air pressure.

Yearly precipitation distribution is very uneven with first snow usually falling in mid-September. By mid-October a stable snow cover typically forms. The snow cover typically melts in mid-April at lower elevations and in May at higher elevations. The project is in a zone of permafrost which needs to be factored into the costs of design and construction of infrastructure.

5.3 Sufficiency of Surface Rights

Exploration and development of mineral deposits is generally not possible without the use of the ground surface. Under Russian law, relevant subsoil use licences do not automatically entitle a company to occupy the land necessary for mining and associated industrial activities. The issue of obtaining the necessary land rights is addressed by a company separately from, but in parallel with, obtaining the subsoil licence. Land use rights are obtained for the parts of the licence area being used, including the plot to be mined, access areas, and areas where other mining-related activities will occur.

Russian legislation on land does not definitively state at what stage the subsoil user should initiate the procedure for obtaining land rights. Under existing subsoil legislation, the formalisation of a subsoil user's land rights for the purposes of geological exploration and subsoil use are carried out under the procedure stipulated by the Land Code.

5.4 Accessibility

The Property is accessible all-year round from Chara village and the town of Novaya Chara by the unpaved highway that runs close to the Baikal-Amur Railway. A regional airport at Chara provides links to Chita 800km southwest, and Irkutsk 1,030 km south-southwest.

There is also a winter road for transport of supplies used to access the city of Chita and the town of Taksimo in Buryatia.

5.5 Infrastructure

5.5.1 Transport

The road distance from the site to Novaya Chara is approximately 22km, and to Chara is approximately 33km. The airport in Chara has a paved airstrip that accommodates regular flights from Chita and then on to Moscow and other municipalities within Russia.

The Baikal-Amur Mainline railway runs parallel to the road with the nearest station at Novaya Chara linking Bratsk (1,356km), Krasnoyarsk and Moscow to the west and Khabarovsk and Vladivostok to the east. The nearest ice-free port is at Vladivostok.

The Project area can be accessed by a winter road that is usable from mid-January until mid-April.

5.5.2 Power

There is access to the main power grid on the Property whereby a 100 MW federal electric power line passes through the north-western corner of the licence area (Figure 4.1, dashed line). There is not a significant industrial demand and drawdown on power on this line although the Udokan Mine site currently being developed by the Udokan Copper Operating Company (UCOC) will have a significant bearing on future usage. UCOC has recently upgraded this grid to develop the Udokan Copper Deposit.

5.5.3 Water

There is potential from the size of the borefield catchment surrounding the deposit to supply enough water for construction and process within the catchment area of the Kemen-Unkur-Chara Rivers to attain 90,000m^3/day although it is not known if there are existing or past resources on the Federal State Balance.

5.5.4 Labour

Given the relatively isolated location of the Property use of local resources is limited. The Kalarsky district is sparsely populated with an estimated population of 8,253 as of January 2016, spread across an area of 56,000km^2. However, the main towns of Novay Chara and Chara have approximately 4,300 and 2,200 inhabitants, respectively.

5.5.5 Seismicity

The area of the deposit and adjacent areas is quoted as being 9 points on the 12-point Russian MSK-64 scale of seismicity used throughout the Commonwealth of Independent States (CIS). This constitutes a severe earthquake potential zone, with at least one catastrophic earthquake likely to occur over a 25-year period.

6 HISTORY

6.1 Discovery and Initial Work

Unkur copper mineralization was discovered by geologists of the All-Union Aerogeological Trust in 1962 during the course of 1:200,000 geological mapping (Shulgina et al., 1962). The mineralized layer was observed in two outcrops of copper-bearing sandstone within a canyon of the Unkur River, occurring 1km apart. In these exposures, the thickness of the layer varied from about 5 to 8m. Based on the chemical assays of channel and chip samples an average copper grade of 1% was determined. It was established that the mineralization is stratabound within the Lower Sakukan sub-formation.

In 1963, the Udokan Expedition (a stated-owned company that included Lukturskaya, Naminginskaya, and other exploration teams) carried out trenching every 200-300m to further define the copper mineralization zone for 1.2km. Sampling from the trenches showed mineralized intervals of 10-12m thick with an average copper grade of 1.02%. Also in 1963, the Udokan Expedition carried out magnetic and electric geophysical surveys over limited areas of the south-eastern syncline at 100m spacing between profiles and 20m spacing between measurement points. The magnetic survey identified distinct magnetic suites but did not directly reveal the zone of copper mineralization.

In 1966, a group of geologists from A.P. Karpinsky Russian Geological Research Institute (VSEGEI) visited the Unkur site. Based on a number of lithological characteristics, the sediments hosting the mineralized layer were classified as shallow-marine and deltaic strata.

6.2 The 1969-1971 Campaign

The studies mentioned above formed the basis for carrying out substantial prospecting works at the Unkur Project, at 250-800m profile spacing, from 1969 to 1971 by the Naminginskaya Exploration Team. These studies (Table 6.1) included drilling of vertical holes only, mapping and geophysics.

Table 6.1: Exploration Works on the Unkur Project, 1969-1978

Period	Unit	1969-1971	1975-1978
Mapping drilling (56 holes)	m	1,200	
Core drilling	m	5,549.1	1,154
Trench volume	m^3	20,524.3	19,144
Mapping traverses	km	50	
Core sampling	samples	194	36
Trench sample length	m	62.7	192
Geochemical sampling	samples	370	580
Chemical analysis	samples	2,486	100
Combined sampling for silver grade	samples	8	11
Composite sampling	samples	51	

From the 1969-1971 works, the geological setting of the mineralized area, internal structure and geochemical characteristics of mineralization became better understood. Based on the new drilling and trenching data, the copper-bearing horizon of 20-50 m thick was traced along the southwestern limb of the Unkur Syncline (Figure 6.1) from southeast to northwest for 4-6 km to a depth of 350m. The average copper grade for the mineralized zone was determined as 0.75%. Silver grade was identified only in composite samples and no routine assaying was performed for silver. Geophysical methods identified a potential extension of the copper-bearing horizon for a further 4km northwest under the moraine sediments 150-180m thick. Based on the results from the 1969-1971 works an estimate of copper and silver resources was prepared by geologists of the Naminginskaya Exploration Team.

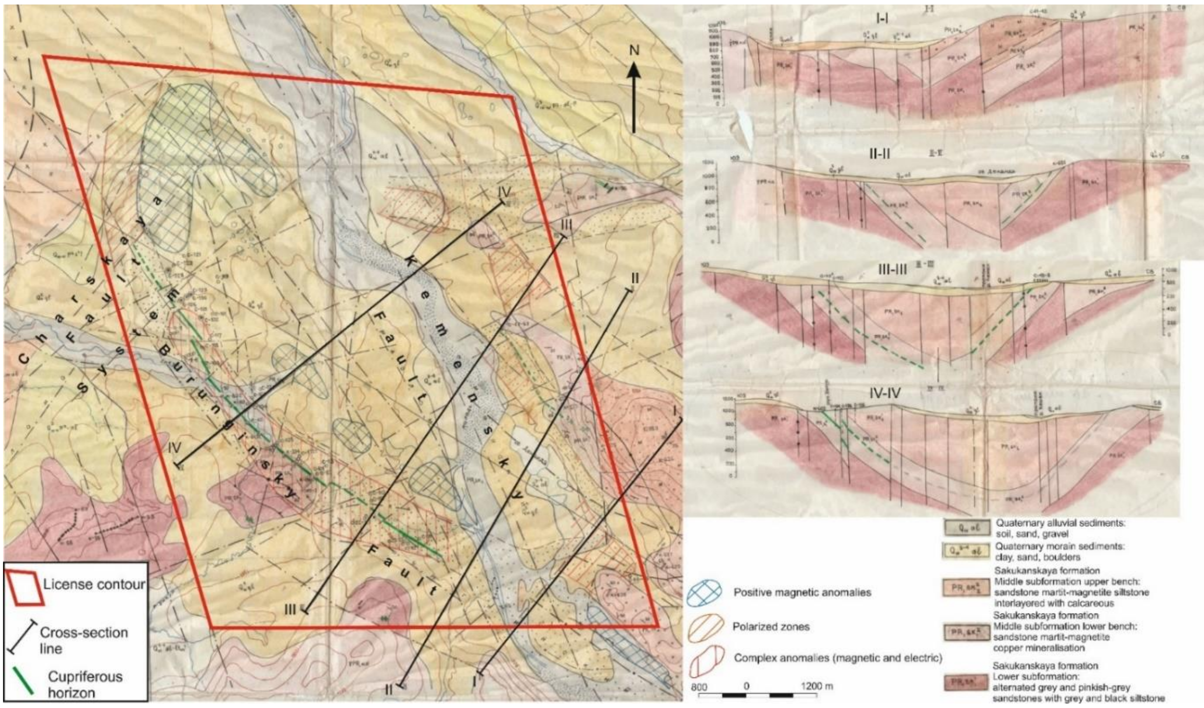


Figure 6.1: Geology of the Unkur Syncline (modified by SRK from Berezin, 1979)

6.3 The 1975-1978 Campaign

During the 1975-1978 campaign, detailed exploration works at a 25m profile spacing were carried out by geologists of the Lukturskaya Exploration Team (Berezin G., 1978) in order to assess the potential of the Klyukvenny copper-bearing deposit, 6km northwest of the Udokan deposit, and the potential of the Luktursky gabbroid massif, which borders the northwest flank of the Unkur deposit. The Klyukvenny and Luktursky prospects fall outside the licensed area owned by AZR, but secondary to the focus on Klyukvenny and Luktursky, further sampling and geophysical assessments took place on the Unkur deposit. The Unkur works included drilling of 4 additional vertical core holes. The aim of this drilling was to test the lateral extents of the deposit. Only one of these holes (C-102) intersected the copper-bearing horizon, at a depth of 250m.

The summary of the exploration works from the 1969-1971 and 1975-1978 programmes is given in Table 6.2. The surface position of the copper-bearing horizon, derived from mapping, drilling and trenching, is shown in this figure as a green line.

6.4 Drilling

Historical drilling at the Unkur Project was mostly carried out during the 1969-1971 campaign (Table 6.2).

SRK notes that the reports from the 1969-1971 and 1975-1978 campaigns list no coordinates for drillhole collars. Previously, the drill holes were depicted on maps and sections. SRK has derived the location data by scanning and georeferencing the historical hard copy maps. SRK estimates that the x and y collar coordinates derived in this manner could have an uncertainty of up to 100m. In 2019, AZR geologists located some historical collars (Figure 6.2), and their coordinates were taken using hand-held GPS devices. The accepted accuracy of these readings is 8m.

Location of 56 mapping holes drilled in 1969-1971 could be depicted only from the historical maps. They are not included into the database.

Table 6.2: Unkur Project Diamond Drilling

Type	1969-1971	1975-1978
Mapping holes (m)	1,200	
Core drilling (m)	5,549	1,154

Figure 6.2: Collars of historical Holes Drilled in the 1970s, Identified and Surveyed in Autumn 2019, Some 45-50 Years After They Were Drilled

Table 6.3: Summary of Unkur Drill Holes, 1969-1978

Hole ID	Azimuth	Dip	Depth (m)	Easting* (m)	Northing* (m)	Elevation* (m)	Line	Date	Core Recovery %
C-103	-	-90	202.5	595911	6302747	924	1	1971	72
C-104	-	-90	296.9	596542	6302212	960	2	1971	88
C-105	-	-90	341.9	596625	6302299	1000	2	1971	
C-107	-	-90	148.7	597511	6301180	1055	4	1971	76
C-108	-	-90	329.6	597646	6301331	1040	4	1971	
C-22	-	-90	12.5	598199.1	6300466	1034	5	1971	
C-110	-	-90	192	598252	6300790	1009	5	1971	
C-110a	-	-90	265	598269	6300694	1014	5	1971	
C-111	-	-90	284.4	599159	6300317	963	6	1971	31
C-111split	-	-90	21.8	599159	6300317	708	6	1971	31
C-112	-	-90	250	599052	6300245	965	6	1971	
C-102	-	-90	272	595760.5	6303455	913	7	1971	50
C-117	-	-90	231	595975.8	6303153	925	8	1971	
C-118	-	-90	274	595821.8	6303014	933	8	1971	58
C-119	-	-90	219.7	596026	6303975	888	9	1971	
C-123	-	-90	284	595616.3	6303795	908	9	1971	
C-121	-	-90	21	595577.2	6304311	901	10	1971	
C-122	-	-90	254.7	595413.9	6304192	903	10	1971	
C-126	-	-90	275	595511.1	6303686	910		1978	
C-128	-	-90	345	595371.4	6304106	906		1978	
C-130	-	-90	275	595532	6303784	910		1978	
C-1	-	-90	35.7	595912	6302664	907			
C-2	-	-90	30	595896	6302664	905			
C-12	-	-90	30.4	596890	6301522	955			
C-13	-	-90	28	596909	6301548	953			
C-14	-	-90	34.5	596927	6301572	950			
C-15	-	-90	38.5	596938	6301586	948			
C-16	-	-90	21.9	596874	6301500	957			
C-101	-	-90	192.7	595981.8	6303495	905			
C-106	-	-90	188	596957	6301606	950			
C-125	-	-90	200	596812.9	6302165	1001			
Total			**5,596**						

Note: * Coordinates in Pulkovo 1942 datum, Zone 20.

A total of 8 drill holes intersected significant copper mineralization in the bedrock. The deepest mineralized intersection is from hole C-104, from a down hole depth of 242.4m (Figure 6.3).

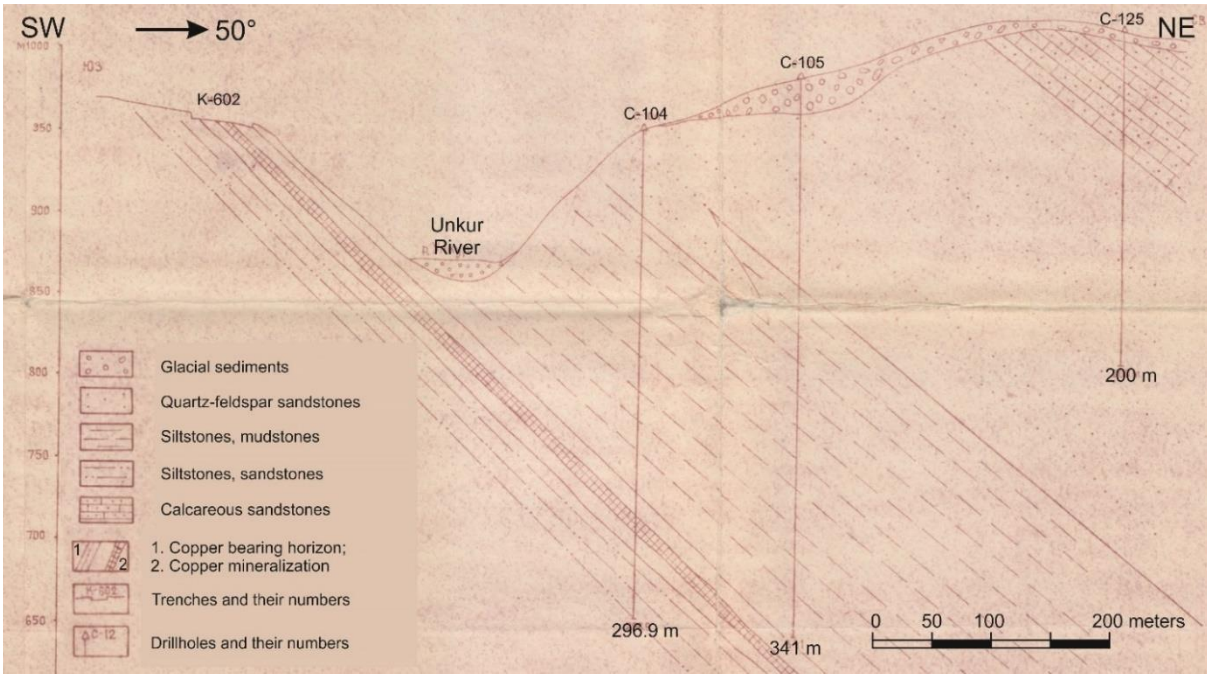


Figure 6.3: Typical Historical Geological Cross-Section, Central Part of the Unkur Project
(modified by SRK from Melnichenko, 1972)

Core drilling during 1969-1971 campaign aimed to assess the copper-bearing horizon under the moraine sediments. All these drill holes are vertical.

As part of the 1969-1971 campaign, a set of “mapping” holes were drilled to 30-40m depth. The profile spacing for this group of holes was 400 m, with a distance between holes of 15–20m. This drilling was carried out by UPB-25 rigs using a single-tube core barrel. A hard metal bit (76 mm diameter) was used for drilling through the sedimentary cover, and then a diamond bit (59 mm diameter) for the bedrock. The total length of the mapping hole drilling was 1,200 m.

A deeper set of drill holes was drilled in 1969-1971 to define copper mineralization to 200-350m depth. This single-tube drilling was carried out by ZIF-300, ZIF-650 and SBA-500 rigs. The distance between the profiles of these drillholes was 400-800m, and the distance between holes was 80-200m. A 146 mm diameter bit was used for the sedimentary cover, a 90 mm bit was used for bedrock, and a 76 mm bit was used for the mineralized zone. The core recoveries for the drillholes which intersected mineralization are shown in Table 6.3. Figure 6.4 shows a summary of the historical holes, section lines and geophysics lines.

A deviation survey was carried out for all drillholes. The dip deviations from vertical did not exceed 1-2°.

From 1969-1978, 31 drill holes were drilled in the Project area (excluding the 56 mapping holes that are omitted from the database). The drilling method was single-tube core barrel. The average length-weighted core recovery from the mineralized intersections was 65.2%.

The mineralized zone in the area covered by the historical drilling generally dips to the northeast at 40-60°, therefore the vertical drill holes were not at the optimum orientation for testing this zone.

A total of 11 composite samples were made from the core sample duplicates in order to determine the grades of associated elements (primarily silver). Results are presented in Table 6.5.

Table 6.4: Assay Results for Core Sampling of Mineralized Intervals

Hole-ID	From	To	Sample-ID	Sample length, m	Cu grade, %
C-1	18.0	24.4	2	6.4	0.5
C-1	24.4	30.0	3	5.6	0.3
C-1	30.0	35.7	4	5.7	0.9
C-12	8.0	10.1	9	2.1	1.7
C-12	10.1	12.0	10	1.9	1.6
C-12	12.0	13.4	11	1.4	1.6
C-12	13.4	16.7	12	3.3	
C-12	16.7	19.0	13	2.3	
C-12	19.0	21.0	14	2.0	0.7
C-13	19.0	23.2	19	4.2	0.9
C-13	23.2	27.4	20	4.2	
C-13	27.4	33.3	21	5.9	
C-13	33.3	38.0	22	4.7	1.8
C-103	88.0	90.0	131	2.0	0.7
C-103	90.0	92.5	132	2.5	0.3
C-103	92.5	93.6	133	1.1	0.4
C-103	93.6	97.0	134	3.4	1.3
C-103	97.0	98.5	135	1.5	0.7
C-104	242.4	245.4	182	3.0	0.6
C-104	245.4	248.6	183	3.2	0.9
C-106	152.0	154.7	156	2.7	0.9
C-107	85.6	87.5	165	1.9	0.3
C-107	87.5	89.5	166	2.0	0.2
C-107	89.5	91.5	167	2.0	0.7
C-118	136.4	138.9	241	2.5	1.4
C-118	138.9	140.4	242	1.5	2.4
C-118	140.4	141.7	243	1.3	1.3
C-118	141.7	143.1	244	1.4	0.8
C-118	143.1	145.1	245	2.0	0.7
C-118	145.1	146.3	246	1.2	0.4
C-118	146.3	148.3	247	2.0	2.3
C-118	148.3	151.1	248	2.8	1.3
C-118	151.1	153.4	249	2.3	1.1
C-118	153.4	155.2	250	1.8	0.3
C-118	155.2	160.0	251	4.8	1.1
C-118	160.0	163.5	252	3.5	2.5
C-118	163.5	164.2	253	0.7	0.5
C-118	164.2	167.5	254	3.3	3.3
C-118	167.5	169.6	255	2.1	2.4
C-118	169.6	171.6	256	2.0	3.1
C-118	171.6	173.8	257	2.2	1.8
C-118	173.8	175.7	258	1.9	2.1

Table 6.4: Assay Results for Core Sampling of Mineralized Intervals

Hole-ID	From	To	Sample-ID	Sample length, m	Cu grade, %
C-118	175.7	178.0	260	2.3	3.5
C-118	196.0	200.5	266	4.5	1.7
C-118	200.5	203.6	267	3.2	3.3
C-118	203.6	205.6	268	2.0	1.5
C-118	205.6	207.6	269	2.0	2.6
C-111	240.0	242.9	202	2.9	0.7
C-111	242.9	243.9	203	1.0	0.3
C-111	243.9	245.2	204	1.3	0.6

Table 6.5: Results from 1975-1978 Composite Sampling for Silver

Hole-ID	Sample	Silver grade, g/t
C-103	1	135.0
C-107	2	11.2
C-106	3	164.6
C-104	4	20.0
C-111	5	21.4
C-118	6	41.6
C-118	7	95.0
C-118	8	87.0
C-102	9	76.8
C-102	10	32.8
C-102	11	56.0
Average		67.4

6.4.1 Sample Preparation and Analyses

Sampling of historical drill holes and trenches was performed by geologists of the Naminginskaya and Lukturskaya Parties of the Udokanskaya Expedition. The intervals selected for sampling included the mineralized zone, as identified by the geologists, and the host rock for 2-4m either side.

The average sample length for the exploration drillholes (200 - 350m deep) was 2m but varied to fit lithology and mineralization intensity boundaries. Intersections of reasonably intact core were manually halved: one half was used as a sample, and the other half was stored as a duplicate. Frequently though, the core returned from drilling was very broken, with poor recovery from 30%-70%, and for these intersections all the available chips were included in the sample.

Sample lengths for the mapping drillholes (hole depths of up to 30m) were typically close to 6 m, but the exact sampling boundaries were chosen with regard to mineralization intensity zones, as identified by the geologists. The longer length of the samples from mapping drill holes was adopted to compensate for the smaller core diameter (26 - 28mm), compared to the exploration drill hole diameter (59mm), in order to obtain comparable sample weights.

Samples were prepared by the Central Chemical Laboratory, Chita. The historical information available for the Project does not include a description of sample preparation procedures and equipment. Trench, core, and composite samples (composed of several core samples) were analysed for copper; geochemical samples were submitted for a semiquantitative spectral analysis for 10 elements. Composite samples were fire assayed for gold and silver and analysed by spectral analysis for 36 elements.

No information on the historical certification of the Central Chemical Laboratory is available.

6.4.2 Quality Control Programmes

Quality control on the historical sample preparation and analytical testwork of the Unkur samples was not done to presently accepted international best practices.

During the 1969-1971 campaign, the Central Chemical Laboratory inserted its own duplicate samples, at a rate of 17% of the total primary sampling. This limited set of results does not show a significant problem with precision.

No quality control samples were analysed for the Unkur Project from the 1975-1978 campaign.

6.5 Geophysical Surveys

Ground geophysical surveys at the Unkur Project were carried out in 1963 and during the 1969-1972 and 1975-1978 exploration campaigns. Geophysical methods included electric logging (induced polarization, dipole electric profiling), time-variable natural magnetic field, magnetic and gravity survey (Figure 6.4).

In order to study physical properties of the copper-bearing horizon, samples were taken from outcrops and drillhole core. These samples were used to determine magnetization, chargeability, resistivity, and specific gravity.

Based on geological description of outcrops, trenches and drillhole core, the geological unit underlying the copper-bearing horizon was identified as highly pyritized over a greater width than copper-bearing horizon. Disseminated pyrite will potentially act as a geophysical marker for induced polarization, in particular, that may broadly identify the position of the copper-bearing horizon.

The results from magnetic and historical IP surveys are shown in Figure 6.5 and Figure 6.6.

Cumulative data on gravity, magnetic, and electric surveys helped to determine trends for fold hinges at the north-western and south-eastern margins of the deposit and defined a series of northeast- and northwest-striking faults which break the Unkur Syncline into several blocks.

Radioactivity of the Udokan Series in the Unkur area is low.

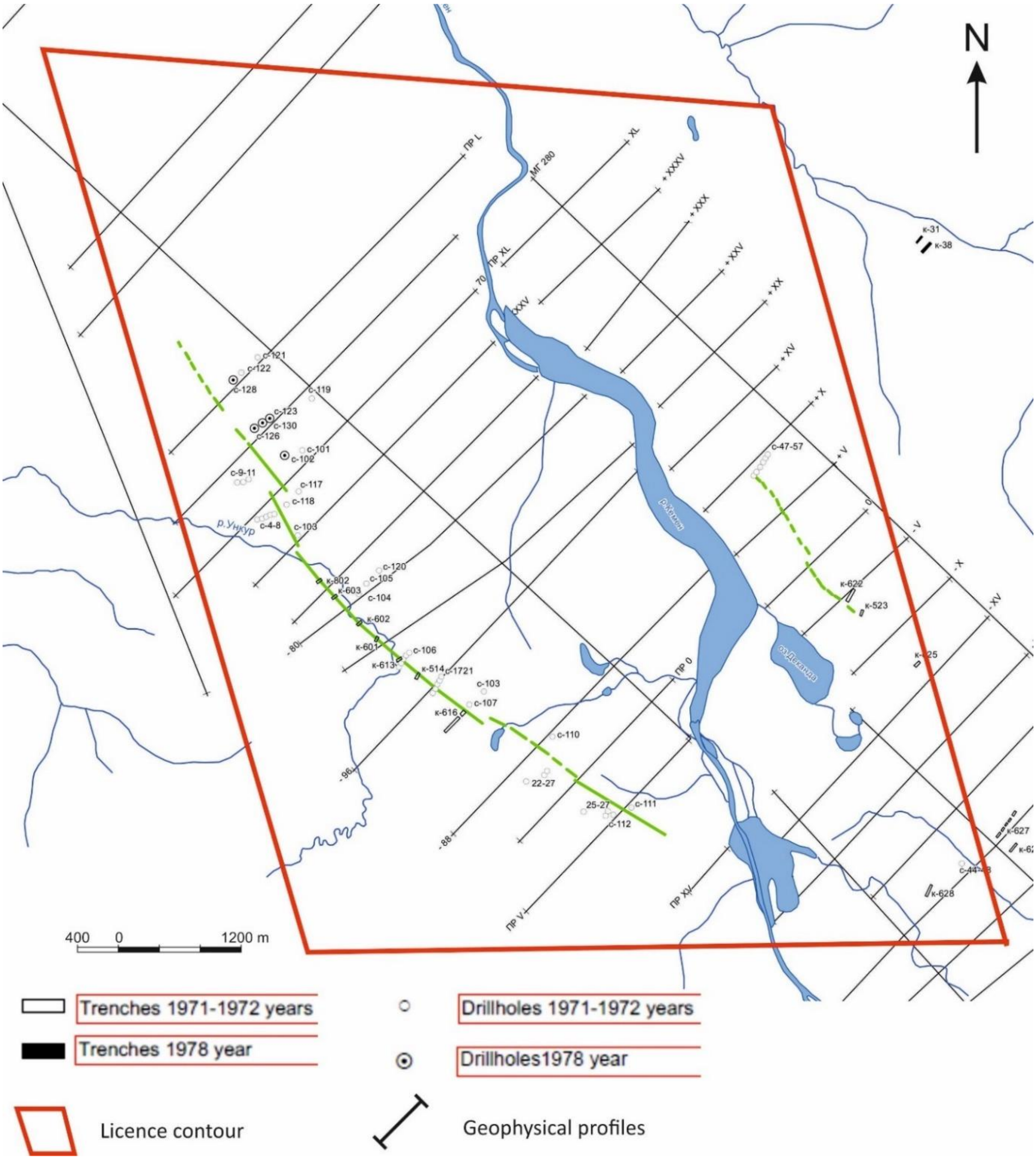


Figure 6.4: Unkur Project Drillholes, Trenches and Geophysical Survey Profiles (LLC GeoExpert Ltd., 2014)

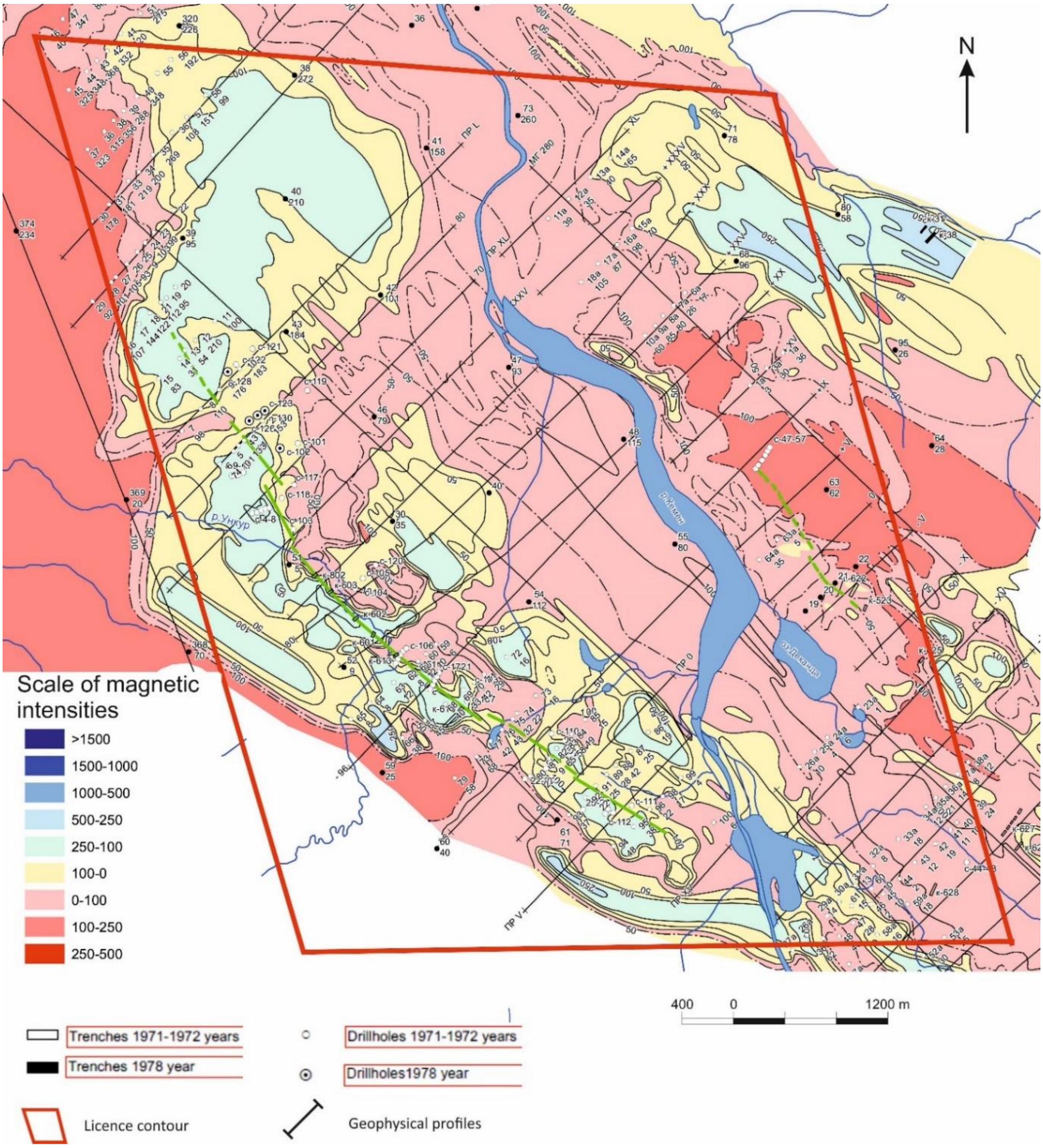


Figure 6.5: Unkur Project Area Magnetic Survey (LLC GeoExpert Ltd., 2014)

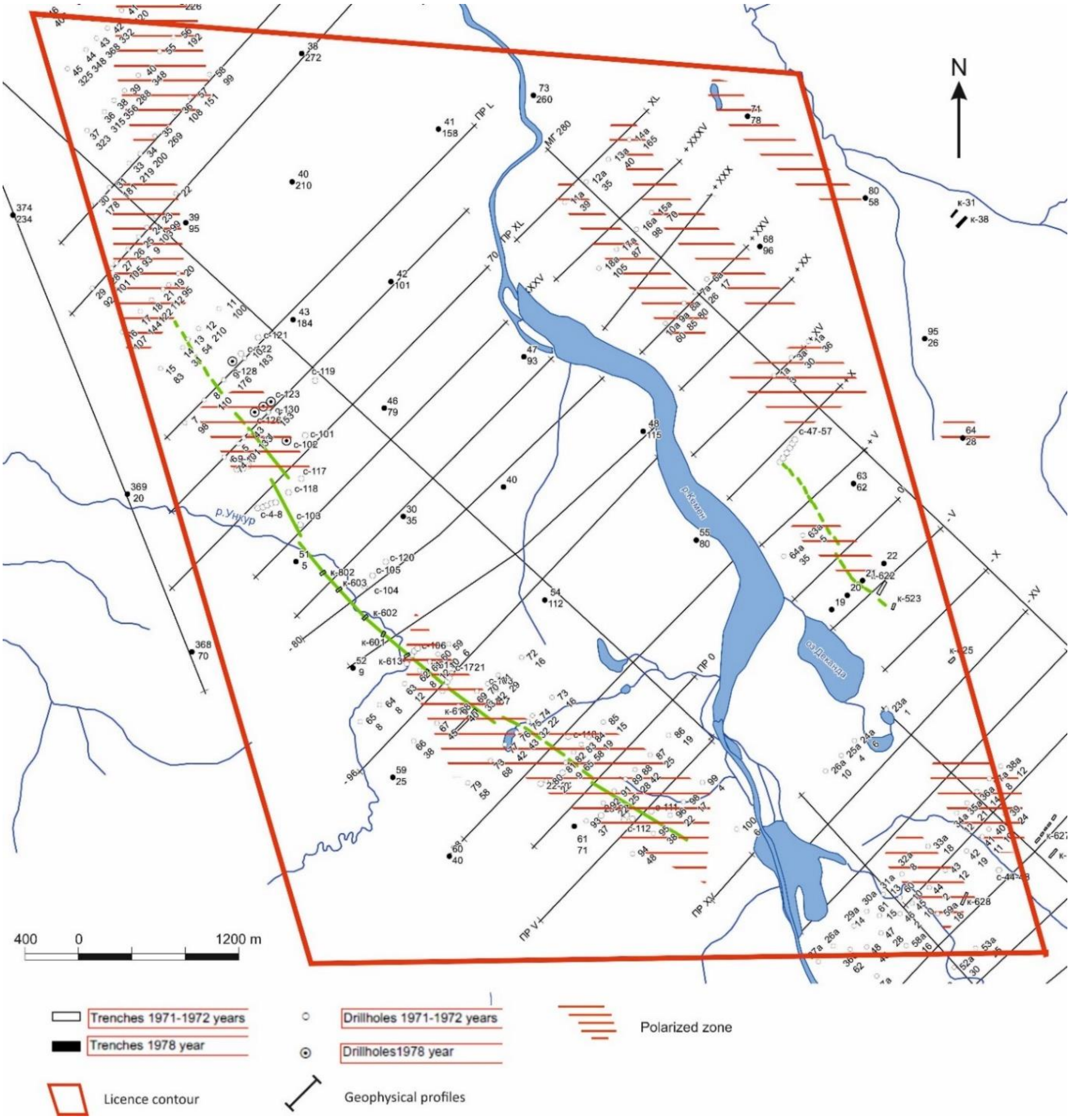


Figure 6.6: Zones of High Chargeability (LLC GeoExpert Ltd., 2014)

6.6 Historical Estimates

Four historical (Non-NI 43-101 Compliant) estimations of copper and silver mineralization for the Unkur Project have been prepared: Melnichenko (1972), Berezin (1979), a 1988 estimate, and a 2014 estimate by the Central Geological Research Institute (TsNIGRI) for the Unkur licence agreement (Table 4.2). These estimates are all based on polygonal methodology and were prepared in accordance with the procedures and definitions of the Soviet Union resource/reserve estimation and reporting system. The qualified person has not done sufficient work to classify the historical estimates as current mineral resources or mineral reserves, and the issuer is not treating the historical estimates as current mineral resources or mineral reserves.

Historical mineral resource estimates presented in this section have been superseded by the mineral resource estimate discussed in Item 14. The historical estimates presented in this section are relevant to provide context but should not be relied upon.

National Instrument 43-101 requires mineral resource reporting to adhere to the resource category definitions of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) in the *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines*. The categories in the Soviet resource/reserve system are incompatible with these definitions, and the estimation methods mandated by the Soviet system are different to the geological modelling and geostatistical estimation methods the qualified person would recommend as optimal for the Unkur deposit. Furthermore, the poor quality of the core remaining from the previous exploration programmes, and the difficulty of doing detailed verification of historical results, means that any future programme of resource definition drilling is likely to replace rather than build on the historical drilling data. In addition, silver was not assayed routinely along with copper. Therefore, the historical estimates reported here should be regarded as an indication of deposit geometry and exploration potential, instead of an inventory that will necessarily be converted into mineral resources.

6.6.1 Resource/reserve classification system of the Soviet Union

The summary of the Soviet resource/reserve categories below is quoted from Henley (2004). Note that Prognostic Resources in the 1960 version of the classification system were a single category; this category was split into three after the 1980 revision to the classification system.

Category A: The reserves in place are known in detail. The boundaries of the deposit have been outlined by trenching, drilling, or underground workings. The quality and properties of the ore are known in sufficient detail to ensure the reliability of the projected exploitation.

Category B: The reserves in place have been explored but are only known in fair detail. The boundaries of the deposit have been outlined by trenching, drilling, or underground workings. The quality and properties of the ore are known in sufficient detail to ensure the basic reliability of the projected exploitation.

Category C1: The reserves in place have been estimated by a sparse grid of trenches, drill holes or underground workings. This category also includes reserves adjoining the boundaries of A and B reserves as well as reserves of very complex deposits in which the distribution cannot be determined even by a very dense grid. The quality and properties of the deposit are known tentatively by analyses and by analogy with known deposits of the same type. The general conditions for exploitation are known. The ore tonnage is derived from estimates of strike length, dip length and average thickness of the ore body. Allowance for barren blocks may be made statistically.

Category C2: These reserves are based on an extremely loose exploration grid, with little data. The limits of the orebody are defined mainly by extrapolation within known geological structures, and from comparison with other similar deposits in the vicinity. The grade and mineral properties of the orebody are determined from core samples and comparison with similar mineral deposits in the area.

The reserves have been extrapolated from limited data, sometimes only a single hole. This category includes reserves that are adjoining A, B, and C1 reserves in the same deposit.

Prognostic Resources are estimated for mineralization outside the limits of areas that have been explored in detail and are often based on data from trenches and from geochemical and geophysical surveys.

Category P1: Resources in the P1 category may extend outside the actual limits of the ore reserves defined in the C2 category. The outer limits of P1-type resources are determined indirectly by extrapolating from similar known mineral deposits in the area. P1 is the main source from which C2 reserves can be increased.

Category P2: These resources represent possible mineral structures in known mineral deposits or ore-bearing regions. They are estimated based on geophysical and geochemical data. Morphology, mineral composition, and size of the orebody are estimated by analogy with similar mineralized geologic structures in the area.

Category P3: Any potential ore-bearing deposits are classified as resources in the P3 category. The presence of these resources relies on the theoretical definition of a "favourable geological environment". Resource figures are derived from figures of similar deposits in the region.

6.6.1.1 *The 1972 Estimate*

The results of the estimation based on the 1972 data are presented in Table 6.6. Prognostic silver resources were estimated within the copper mineralization domain. Average silver grades were determined based on the chemical assays of eight composite samples. The arithmetic mean of these samples is 73.3 g/t, and this grade was equally applied to all the blocks. Therefore, the prognostic resources of silver amount to 10.1 Kt Ag. The mineralized envelope was extrapolated 1300 m downdip, e.g., much deeper than the depth of drilling.

Table 6.6: Results From the 1972 Estimate for the Unkur Project (Melnichenko V., 1972)

Category*	Block No.	Zone Thickness, m	Tonnes, Kt	Average Cu Grade, %	Contained Metal, Kt
C2	Block 1	12.4	77,760	0.80	622
	Block 2	4.3	9,978	0.60	60
Total, C2 Category		**9.8**	**87,738**	**0.78**	**682**
Prognostic resources	Block 3	12.4	33,849	0.80	271
	Block 4	8.3	16,409	0.75	123
Total, prognostic resources		**10.7**	**50,258**	**0.78**	**394**
Total		**20.5**	**137,996**	**0.78**	**1,076**

* classified according to the Soviet Union resource/reserve classification system of 1960

This estimate should not be relied upon as it has been superseded by the mineral resource discussed in Item 14 of this report.

6.6.1.2 *The 1979 Estimate*

Upon completion of the second phase of exploration works for the Unkur Project in 1979, the second resource/reserve estimate for the Unkur deposit was performed with regard to the new (albeit limited) drilling data (Table 6.7). Prognostic silver resources were estimated within the copper mineralization domain. Average silver grades were determined based on the chemical assays of eleven composite samples. The arithmetic mean of these samples is 68.3 g/t, and this grade was applied to all the blocks. Therefore, the prognostic resources of silver amount to 9.7Kt Ag. The mineralized envelope was extrapolated 1,300m downdip, e.g., much deeper than the depth of drilling.

Table 6.7: Results From the 1979 Estimate for the Unkur Project (Berezin G., 1979)

Category	Block No.	Zone Thickness, m	Tonnes, Kt	Average Cu Grade, %	Contained Metal, Kt
C2*	Block 1	12.9	91,820	0.80	725
	Block 2	4.3	9,978	0.60	60
Total, C2 Category		**8.6**	**101,798**	**0.77**	**785**
Prognostic resources	Block 3	12.9	24,685	0.80	195
	Block 4	8.3	16,409	0.75	123
Total, prognostic resources		**10.6**	**41,095**	**0.77**	**318**
Total		**10.1**	**142,893**	**0.77**	**1,103**

* classified according to the Soviet Union resource/reserve classification system of 1960

This estimate should not be relied upon as it has been superseded by the mineral resource discussed in Item 14 of this report.

6.6.1.3 *The 1988 Estimate*

In 1980, the Soviet resource/reserve classification system was updated. The changes primarily affected the definitions of the C2 resource category and prognostic resources: under the new system, the C2 category was grouped with estimated reserves, and the prognostic resources were divided into three categories: P1, P2, and P3. In 1988, the Unkur deposit was re-estimated and re-classified in accordance with the new classification system. A consequence of this revision was the entire inventory was classified as prognostic resources (Table 6.8).

For the 1988 estimate, a 0.4% Cu grade threshold was used for defining the resource domain, compared to the 0.6% Cu threshold used for the 1972 and 1979 estimates.

Soviet resources were calculated on the southwestern limb only. Exploration was not followed up on the northeastern limb due to poor results from shallow drilling.

Table 6.8: Results From the 1988 Estimate for the Unkur Project (Unkur Licence Agreement)				
Category*	**Component**	**Tonnes, Kt**	**Average Grade**	**Metal Contained**
P1	Copper	83,500.9	0.79%	660 Kt
	Silver		68.3 g/t	5,703 t
P2	Copper	58,107.7	0.75%	436 Kt
	Silver		68.3 g/t	3,969 t
P3	Copper	87,532.5	0.77%	674 Kt
	Silver		68.3 g/t	5,979 t

* classified according to the Soviet Union resource/reserve classification system of 1980

This estimate should not be relied upon as it has been superseded by the mineral resource discussed in Item 14 of this report.

6.6.1.4 *2014 Estimate*

The most recent assessment of the prognostic copper and silver resources for the Unkur Project before the issue of the current Unkur licence was completed by the geologists of TsNIGRI. The results of this estimate are presented in Table 6.9. The data supporting the 2014 estimate are the same as for the 1979 and 1988 estimates (there have been no material additions to the supporting data since 1978); the resource/reserve reporting system is the same as was in place for the 1988 estimate; the threshold for defining the resource domain (0.4% Cu) is also the same as used for the 1988 estimate, but the estimated tonnes and metal in 2014 were an order of magnitude lower than in the 1988 estimate.

The differences between the prognostic resource statements of 1988 and 2014 are due to different interpretations of how the Russian resource/reserve reporting system should be applied to the Unkur deposit. The main reasons for the substantially lower tonnage of the 2014 estimate are:

1) The 1988 estimate included a substantial portion of P3 material, representing mineralization on the northeast limb of the Unkur Syncline. All of this northeast limb material was omitted from the 2014 estimate as it was not supported by assays. All copper mineralization was recorded only in slope scree material.
2) From the southwest limb of the Unkur Syncline, the P2 category of the 1988 estimate included about 1,000m of interpolation along strike, between areas covered by drilling and trenching, and about 1,000m extrapolation along strike to the northwest. The down dip extrapolation was to 1,000m vertical depth (1,300 m down dip). This along strike/downdip interpolation and extrapolation was not included in the 2014 estimate.
3) For the 2014 estimate, extrapolation down dip was limited to 300 m below surface on the assumption that this would be the maximum depth of open pit mining. A greater depth limit of 1,000m vertically below surface was used to constrain the 1988 and earlier estimates, on the basis that the deposit could potentially be mined by underground methods.

Table 6.9: Results From the 2014 Estimate for the Unkur Project (Volchkov and Nikeshin, 2014)

Category*	Block No.	Component	Tonnes, Kt	Average Grade	Metal Contained
P1	1	Copper	16,516.5	0.90%	148.6 Kt
	2		3,964	0.65%	25.8 Kt
Total P1		Copper	20,480.5	0.85%	174.4 Kt
		Silver		77.96 g/t	1,600 t

* classified according to the Russian resource/reserve classification system of 1980

This estimate should not be relied upon as it has been superseded by the mineral resource discussed in Item 14 of this report.

6.6.2 SRK 2017 Estimate

In March 2017, the following MRE was prepared by SRK classified within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) and were NI 43-101 compliant, as summarised in Table 6.10.

Mr Robin Simpson prepared this MRE on behalf of AZR and as presented in "Technical Report for the Unkur Copper-Silver Deposit, Kodar-Udokan Area, Russian Federation" with an effective date of March 31, 2017.

Table 6.10: Unkur Cu-Ag Project Mineral Resource Statement as of March 2017

Domain	Classification	Tonnes (Mt)	Cu (%)	Ag (ppm)	CuEq (%)	Cu Metal (mlb)	Ag Metal (Million tr oz)
Zone 1, Near Surface	Inferred	23	0.54	40	0.93	270	29
Zone 2 North, Near Surface	Inferred	9	0.47	43	0.89	90	12
Zone 2 South, Near Surface	Inferred	1	0.42	4	0.46	10	0
Total Near Surface	Inferred	33	0.52	39	0.9	380	41
Zone 1, Underground	Inferred	8	0.53	34	0.86	100	9
Zone 2 North, Underground	Inferred	1	0.47	43	0.89	10	2
Total Underground	Inferred	10	0.52	35	0.87	110	11
Zone 1	Inferred	31	0.54	38	0.91	37	28
Zone 2	Inferred	11	0.46	38	0.84	120	14
Total	**Inferred**	**42**	**0.52**	**38**	**0.9**	**480**	**52**

Notes:

1. CIM Definition Standards were followed for Mineral Resources.
2. Reporting of near surface Mineral Resources is constrained by a conceptual pit shell.
3. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
4. Mineral Resources are reported at a cut-off grade of 0.3% CuEq for near surface and .7% CuEq for underground.
5. Copper and Silver equivalent grades were estimated using US$3.00/lb Copper price, US$20.00/oz silver prices, and assuming 100% recovery for both; the equivalence formula is CuEq = Cu + (0.009722 x Ag)
6. Number may not add up due to rounding

The following density factors were used to convert volumes in the block model to dry bulk tonnages:

- Oxide mineralised: 2.60 (59 samples)
- Oxide waste: 2.58 (76 samples)
- Sulphide mineralised: 2.67 (90 samples)
- Sulphide waste: 2.68 (270 samples)
- Moraine overburden: 2.00 (assumed)

6.6.3 Tetra Tech 2018 Estimate

The 2018 MRE was prepared by Tetra Tech and classified within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM 2014) and were NI 43-101 compliant, as summarised in Table 6.11. This MRE incorporates all the data from the 2016-17 drilling and represents an increase by 50% on the 2017 resource but without an upgrade in confidence. However, it does not incorporate any of the pre-2014 Soviet-era data and unlike the 2014 estimate does not differentiate inventories for near-surface and underground potential.

Mr Joseph Hirst prepared this MRE on behalf of AZR and as presented in "Technical Report and Mineral Resource Estimate for the Unkur Copper-Silver Project, Kodar-Udokan, Russian Federation", with an effective date of 27th March 2018.

Table 6.11: Unkur Cu-Ag Project Mineral Resource Statement as of March 2018

Class	Tonnes (t)	Density	Cu Grade (%)	Ag Grade (g/t)	CuEq (%)	Cu Metal (t)*	Ag Metal (troy oz)
Inferred	62,000,000	2.67	0.53	38.6	0.9	328,600	76,881,000

Notes:

- The effective date of the Minerals Resources is 7th March 2018.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
- *1,328,600 t Cu = 724,234,400 lb

A mean density value of 2.57 g/cm^3 was used for all blocks, except for glacial moraine which was given a value of 1.8 g/cm^3.

In order to demonstrate that the deposit has reasonable prospect for economic extraction, a cut-off grade of 0.3% CuEq was applied for Mineral Resources constrained by the second search pass. The cut-off grade is based on the following assumptions:

- silver price of US$20/tr oz;
- copper price of US$3.00/lb; and
- silver and copper recovery of 100%.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The Unkur Project is situated in the centre of the 200 x 60 km Kodar-Udokan sedimentary basin (Figure 7.1) in the southeast of the Siberian platform. Within this zone, Archean, Paleoproterozoic, Neoproterozoic (Vendian), Lower Cambrian, Mesozoic and Cenozoic formations are present.

The bedrock in the vicinity of the Project is dominated by Paleoproterozoic weakly metamorphosed clastic rocks. This sedimentary succession is intruded by Paleoproterozoic, Neoproterozoic, late Palaeozoic and Mesozoic igneous complexes.

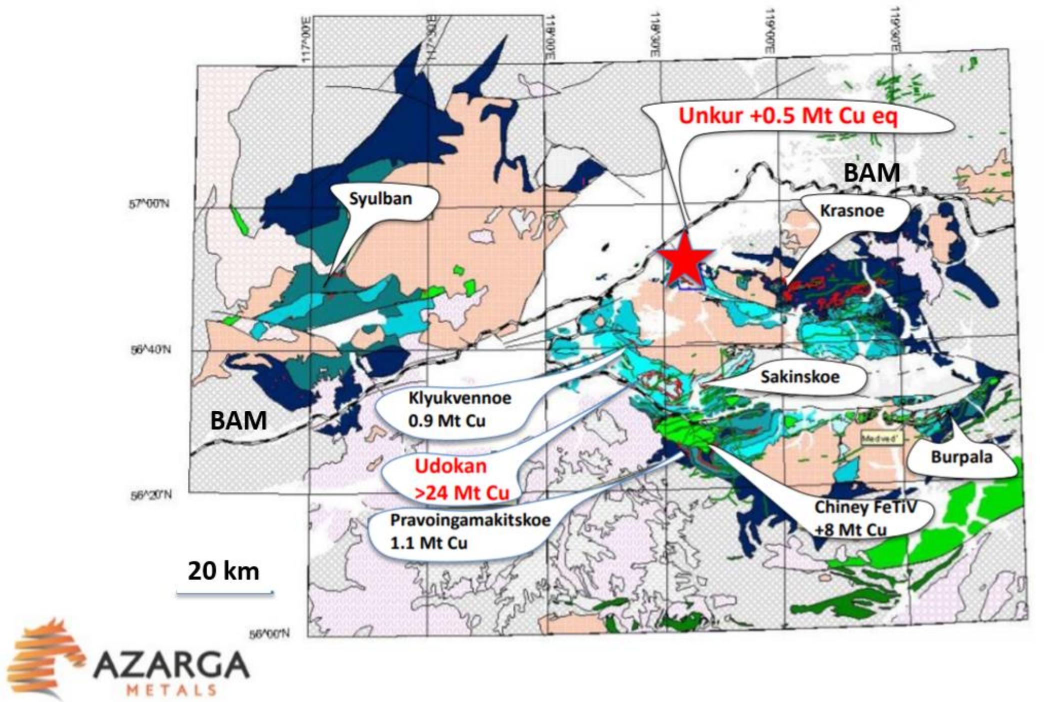


Figure 7.1: Regional Geology of the Kodar-Udokan Sedimentary Basin and its Copper Deposits and Prospects

7.2 Local Geology

Locally, the geology is composed of Paleoproterozoic metamorphosed sedimentary rocks of the Udokan Group, Paleoproterozoic granitoids of the Chuisko-Kodarsky complex, gabbroid massifs of the Paleoproterozoic Chiney complex and Neoproterozoic mafic dykes. There is extensive Quaternary alluvial and glacial cover developed in relation to the continental rifting in the adjacent Chara graben depression (Figure 7.2).

The sediments of the Udokan Group were deposited in a shallow marine environment. In ascending stratigraphic order, the formations of the group are named as the Ikabyinskaya, Inyrskaya, Chitkandinskaya, Alexandrovskaya, Butunskaya, and Sakukanskaya. Their overall thickness is 5,350m.

The copper-bearing horizon is confined to sedimentary rocks of the Lower sub-formation of the Sakukanskaya formation. This sub-formation is a 500m thick package of alternating pinkish-grey medium-grained sandstones and grey to black siltstones.

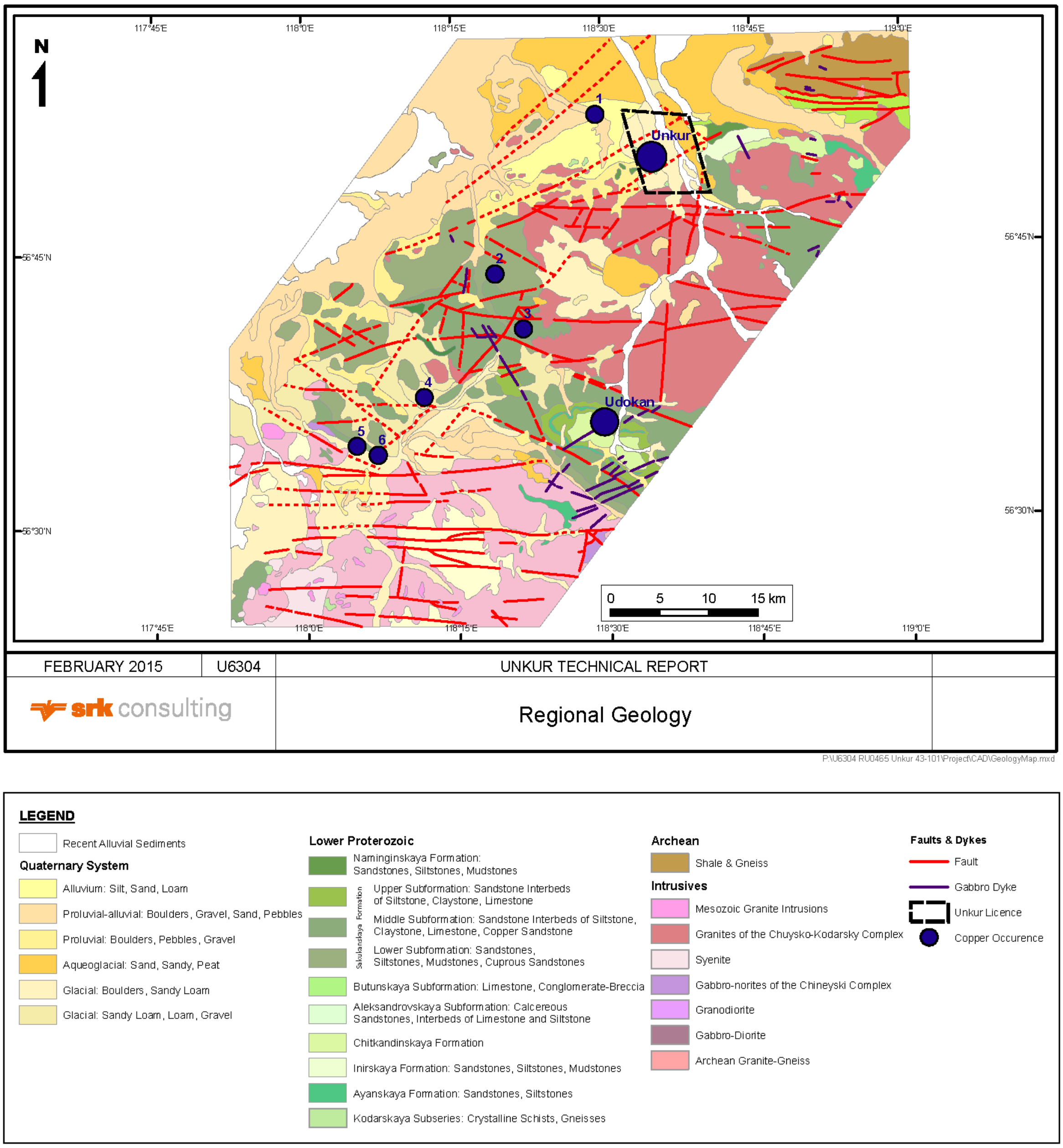


Figure 7.2: Regional Geology Setting (modified by SRK from Melnichenko, 1972)

In addition to the Unkur and Udokan deposits, the other copper occurrences shown on the map are: Luktursky (1); Nirungnakanskaya group (2 and 3); Ingamakitskaya group (4, 5 and 6)

7.3 Property Geology

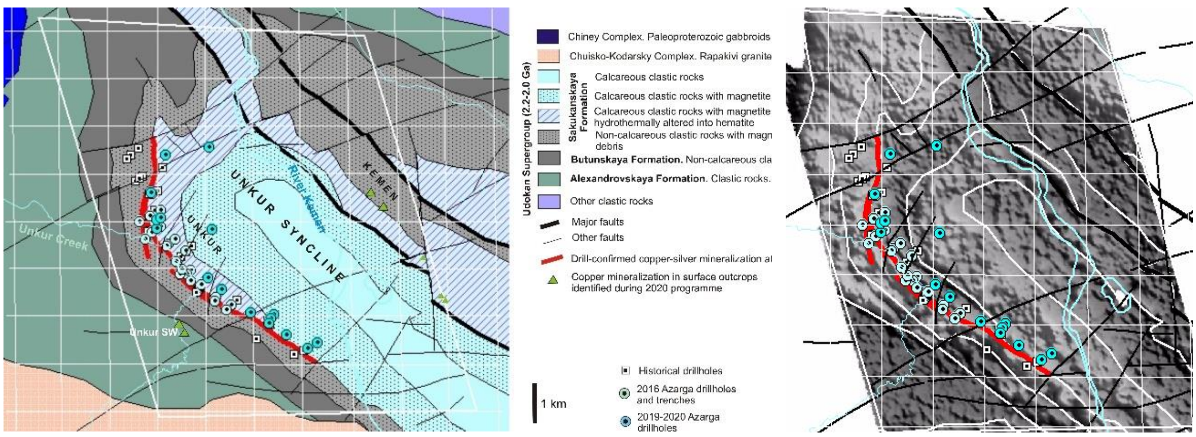
7.3.1 Udokan Group

In the vicinity of the Unkur deposit, the sediments of the Udokan Group are folded into a broad, doubly plunging syncline, with a steeply inclined axial plane striking northwest Figure 7.3. The northwest-southeast extent of this synclinal structure within the licence is about 12km.

Three of the Udokan Group formations have been identified within the Unkur Project area: Alexandrovskaya, Butunskaya and Sakukanskaya.

The rocks of the Alexandrovskaya formation are exposed in the south-western limb of the syncline and comprise a package of interstratified siltstone and sandstone, with locally developed quartzites (about 1m thick), occurring every 25-30m. The formation is characterized by a magnetic low (Figure 7.3A). Based on geophysical data, the thickness of the formation in the project area is about 450-600m.

Most of the Butunskaya formation is covered by Quaternary moraine. The upper part of the Butunskaya formation is exposed in the canyon of the Unkur Creek and occurs as a package of alternating siltstone and fine-grained sandstone. The formation is characterized by a magnetic high (Figure 7.3B), reflecting widespread presence of clastic magnetite. Based on the geophysical data, the thickness of the formation in the project area is 500-600m.



A B

Figure 7.3: Interpreted Geology of the Unkur Syncline (A) and Reduced-to-Pole Grey Scale Magnetic Map (B) (AZR)

[Non-magnetic units correspond to darker colours]

The Sakukanskaya formation hosts copper mineralization and occupies most of the Unkur Project area. The Sakukanskaya formation is mainly medium-grained grey sandstone.

Of the Sakukanskaya sub-formations, the Middle and Lower have been identified in the project area. The Lower sub-formation is characterized by grey and pinkish-grey sandstones alternating with grey and black siltstone and is 1,000 to 1,200m thick. The Middle sub-formation mainly consists of grey and pinkish-grey sandstones interlayered with calcareous sediments, whereas they are not calcareous in the Lower sub-formation. Rough crossbedding is characteristic of the sandstone. The overall thickness of the Middle sub-formation is about 1,000m. The pinkish rock colour is due to hydrothermal oxidation of magnetite into hematite.

7.3.2 Intrusive Rocks

In the south, the rocks of the Udokan Group are intruded by the Paleoproterozoic Chuya-Kodar granitoids.

Paleoproterozoic gabbroids of the Luktur massif (part of Chiney Complex) occur under Quaternary moraine cover, some 250m west from the north-western corner of the Unkur licence.

The Udokan Group rocks are also intruded by Neoproterozoic gabbro-diorite dykes of the Doross Complex. Dyke thicknesses range from metres to tens of meters, with observed strike lengths of 200-1,000m. The dykes strike northeast and northwest, corresponding to the strikes of the two main fault systems. The dykes were intercepted by AZR drill holes on two occasions.

7.3.3 Quaternary Cover

Glacial sediments cover >95% of the project area and form numerous moraines. The average thickness of the moraine cover is 40m. However, this cover is almost absent or very thin (3-5m) over a strike length of 2.5km near the central part of the Unkur prospect. The thickness sharply increases to 180-200m thickness in both the northwest and southeast of the project area after passing the northeast-trending faults, subparallel to the continental rift of the Chara Graben.

Recent alluvial sediments have been deposited by the Unkur and Kemen Rivers cutting into glacial sediments. The alluvial sediments are composed of gravel and sandy soil and form 5-20m high terraces above floodplains.

7.3.4 Structure

As noted above, the major structure of the deposit is an Unkur Syncline with a northwest-striking axial plane (Figure 7.3A). The southwest limb of the fold dips to the northeast at 40-60° and is complicated by higher order folding.

The Butunskaya and Sakukanskaya formations outcrop in the northeast limb of the fold, and dip 15-30° southwest, increasing to 35-60° closer to the axial plane. As shown on historical maps, and confirmed by AZR fieldwork in 2020, the north-eastern limb dips at 60-70^o to the southwest.

To the southeast the syncline is open and extends for at least another 10km. In the northwest, geophysical evidence implies the syncline is closed and cut by a branch of the Kemen Fault. The Kemen

Fault is one of three large northwest-striking faults. The other in this group is the Burunginsky Fault. The displacement in vertical direction on these major faults does not exceed 300m.

The Unkur Syncline is also cut by the Chara northeast-striking fault system. Displacements on these faults do not exceed 150-200m. They offset all structural elements. Satellite and aerial images show distinct open linear features. There are several hot springs along these faults near the Luktur gabbroid massif to the north. These faults are interpreted as developed in relation to the Quaternary movements in the Chara depression, part of the Baikal rift system.

All the faults have undergone tectonic-magmatic re-activation at various stages. There is no reliable information on the cross-cutting relationships between faults.

7.4 Mineralization

The main copper-bearing horizon (Zone 1) was initially identified and traced along the south-western limb of the Unkur Syncline. It is confined to weakly metamorphosed clastic rocks of the Lower Sakukanskaya.

sub-formation. Stratigraphically, the position of the copper-bearing horizon is 80-100m above the base of the Sakukanskaya formation. Copper oxide minerals were identified in 2020 in five outcrops over 1 to 3km length on the opposite (northeast) limb of the Unkur syncline (Figure 7.4). They were previously recorded only among Pleistocene sediments. It is clear that they belong to more than one stratigraphic level.

Figure 7.4: AZR geologist at one of the Cleared Outcrops and Rocks with Visible Cu oxide at Kemen, October 2020

The Zone 1 (Main) mineralized horizon dips northeast at 45-60° (Figure 7.5), with higher order flexures identified during detailed logging of the drill core. As a result of 2020 drilling programme, Zone 1 has been traced along the strike for 6.5km by drill hole and trench intersections. The maximum drillhole intersection depth is 550m. The true thickness of the horizon ranges from 7 to 50m.

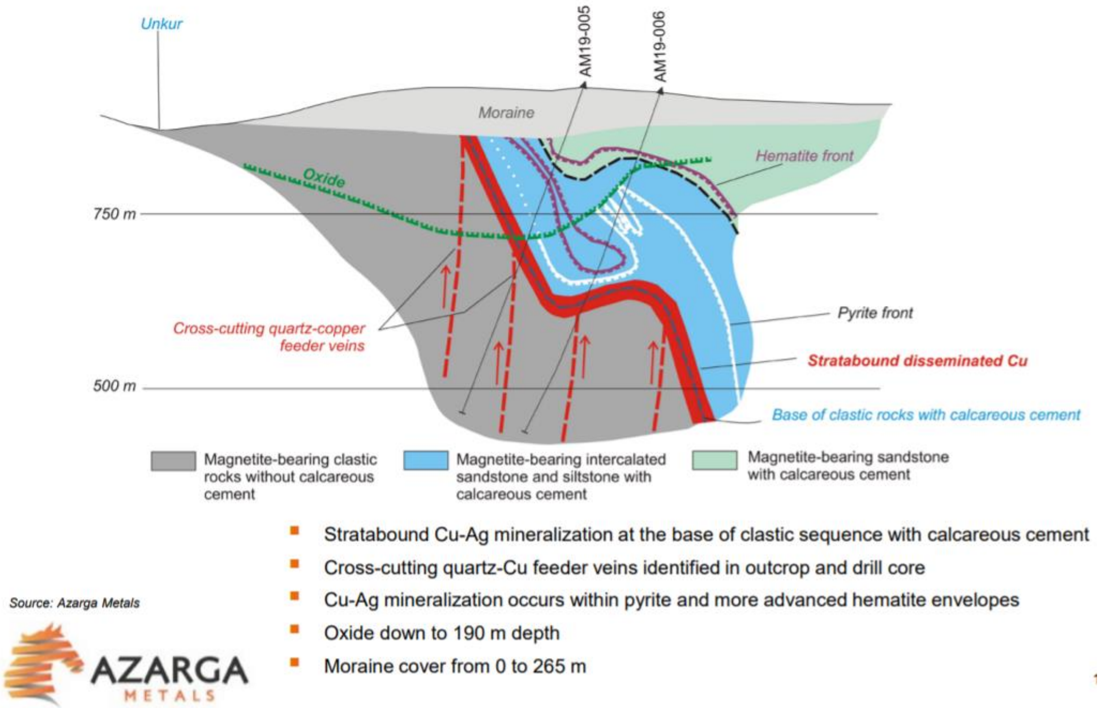


Figure 7.5: Lithological-Structural Setting, Alteration, Depth of Oxide and Position of Two Types of Unkur Cu-Ag Mineralization

The main stratabound copper-bearing horizon occurs at the transition from non-calcareous to calcareous rhythmically-layered sandstone and siltstone (Figure 6.5). The true thickness of the layers varies from 1 to 40m.

In the Outcrop 3 (Figure 7.6) on the left bank of the Unkur Creek (where mineralization was discovered back in 1962), the copper mineralization is present in quartz veins, cross-cutting the lithological bedding at 45 to 90°. These veins are individually 0.5 to 2cm thick, but they form up to 20-30m wide clusters. The veins are interpreted as feeders to the stratabound copper-silver lode. The cross-cutting veins were also recorded in the drill core. This area was previously interpreted as Zone 2, subparallel to and stratigraphically below the stratabound Zone 1

Figure 7.6: A South-Looking View of Outcrop 3 (OC003) and its Field Drawing Showing Position of Cross-Cutting Quartz-Copper Veins Relative to Lithological Boundaries

Detailed logging revealed a wide hematite envelope, starting some 50m above the stratabound mineralization. It can be much wider (≈150m) if there are higher order flexures. Hematite was microscopically recorded to replace magnetite. When drilling, first pyrite occurs some 30 to 50m above the copper sulphides. First appearances of hematite and pyrite were successfully used by AZR to control the depth of drilling during the 2019-2020 programme.

The recorded relationships suggest that emplacement of mineralization was probably syn-deformational, not syn-depositional with the clastic rocks. This was confirmed by Perello et al. (2015) at Udokan, where U-Pb dating of titanite collected from disseminated stratabound and cross-cutting mineralization showed their synchronism at 1.89 Ga, some 300 m.y. after deposition of the hosting 2.2 Ga clastic sequence.

At Unkur, the mineralized fluids are interpreted to have passed through the clastic package along the feeder veins and then were deposited within the stratified package at the non-calcareous/calcareous transition. Clastic magnetite was hydrothermally altered into hematite within a large volume of rocks, whereas pyrite formed a much narrower envelope.

Sulphides are usually oxidized down to 180m depth below surface irrespective of presence or absence of moraine cover. This indicates that oxidation event is pre-Quaternary, most likely Jurassic to Cretaceous.

From geophysical methods, the copper-bearing horizon has been traced under moraine sediments for more than 6km. It clearly corresponds to magnetic low, internal to a wider magnetic high of the Sakukan formation. This magnetic high corresponds to recorded presence of clastic magnetite. The internal magnetic low within this package (Figure 6.7) coincides with the stratabound mineralization and recorded dominance of hematite instead of magnetite. This feature was used to trace Cu-Ag mineralization south and north of the historically drilled area. This factor was underestimated in Soviet times and historical exploration works missed the target horizon while trying to intercept it in the north.

The recent sampling by AZR has not defined a consistent, continuous high-grade zone within the overall mineralised zone, but there are three shoots with the highest grades (>0.5% Cu). At a larger scale, the northern part of the deposit (north of 6302300N) tends to be higher grade than the southern part, and the relatively high grade and thick intersection in drill hole AM-001 coincides with a change in strike, from approximately northwest-southeast, to approximately north-south.

Sulphide copper minerals comprise chalcopyrite, bornite, chalcocite and covellite. Oxide copper minerals include malachite and brochantite. Accessory minerals include magnetite, hematite, and ilmenite.

A hypogene zonation is noted in the distribution of the copper minerals: a chalcopyrite-pyrite-bornite-chalcocite association is found in the centre; either side of this there is a monomineral chalcopyrite association (usually lower grade). Higher grade Cu-Ag areas seem to correspond to intersection with the cross-cutting (feeder) veins. Pyrite forms a wider (thicker) envelope than copper mineralization. The mineralization remains open along strike and downdip.

Copper oxide minerals are consistently observed to a downhole depth of 180m.

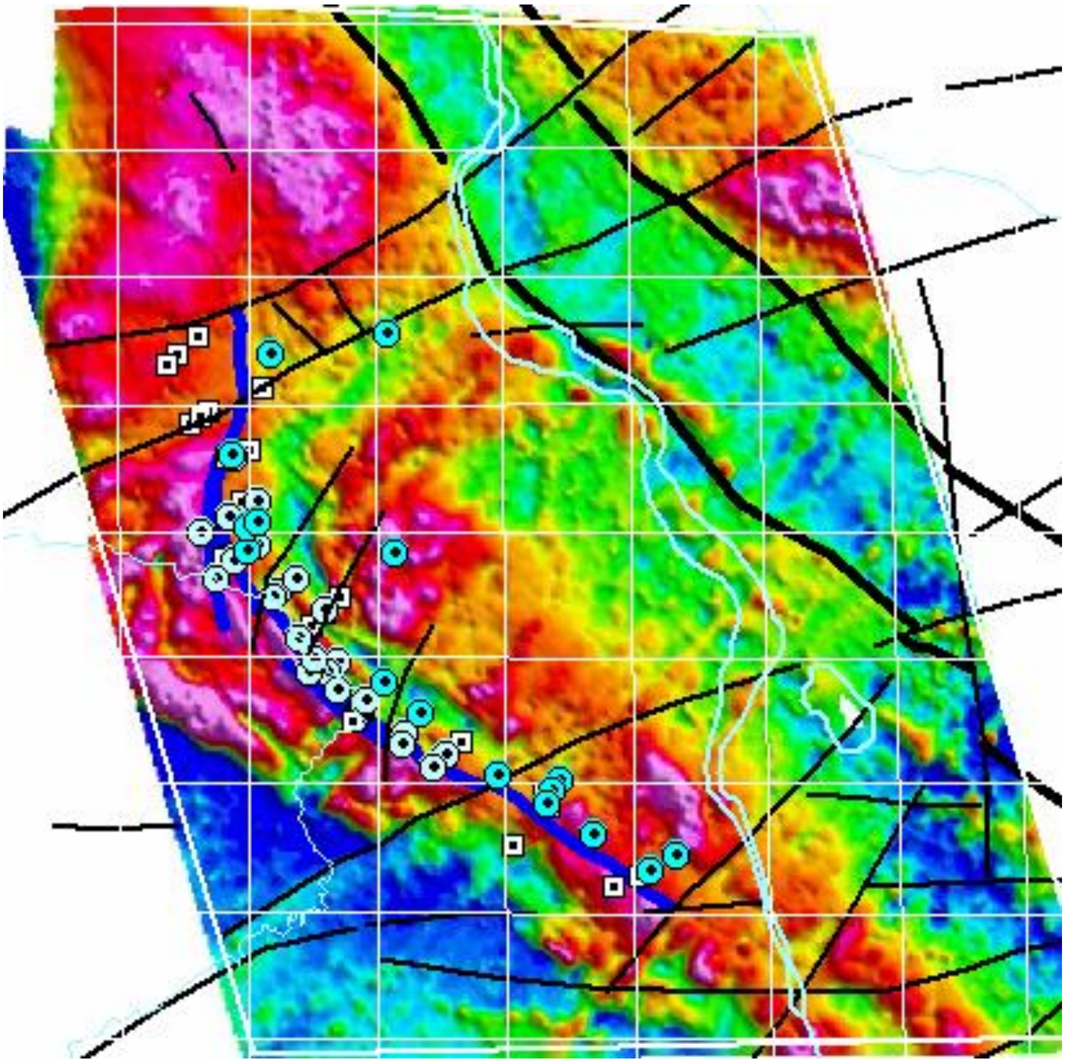


Figure 7.7: Position of Drill-Tested Cu-Ag Zones at Unkur (dark blue) on Reduced-to-Pole Magnetic Image

[The internal magnetic low corresponds to hematite area, interpreted as product of altered magnetite. Best Cu and Ag grades correspond to most intense internal magnetic low]

The mineralized zone is displaced by northeast-striking fault and breccia zones. The displacements are typically 20-70 m, but for some faults displacements are as much as 150m. The thickness of overlying moraine sharply increases across such faults.

Based on samples collected by AZR from drill holes, trenches, and outcrops, a second mineralised horizon (Zone 2) has been identified to the west, stratigraphically 100 to 150m below Zone 1 in its southern central part. The sparse information available so far for Zone 2 suggest that this zone has a similar orientation, thickness, intensity, and mineralogy to Zone 1.

8 DEPOSIT TYPE

The Unkur deposit is interpreted as a stratabound cupriferous sandstone type with copper and silver mineralization likely to have been deposited during diagenesis and early deformation at a moving redox boundary resulting in cross-cutting feeder veins. This geological model is considered appropriate for the deposit because of the following observations:

1. There is a clear stratigraphic control on Main copper mineralization, which is confined to the upper part of the Lower Sakukanskaya sub-formation.
2. Several sedimentary features (such as crossbedding),
3. Wave rippling and desiccation cracks) imply a shallow and relatively low-energy depositional environment. This facies type is a key requirement for many models of other stratiform copper deposits.
4. The cross-cutting feeder veins were identified, but their extent requires further exploration work.
5. Absence of obvious igneous or structural first order controls on mineralization. The faulting in the Unkur Project area generally appears to be post-mineralization.
6. A simple copper mineral composition, which is characteristic of sandstone-hosted copper deposits.

The nearby Udokan copper deposit is also an example of a sediment-hosted stratiform copper deposit with cross-cutting cupriferous quartz veins. Globally, other prominent examples of this deposit type are the Dzhezkazgan copper deposits in Kazakhstan, the Zambian copper belts, and the Kupferschiefer in Central Europe.

9 EXPLORATION

9.1 Channel sampling of trenches and outcrops

In 2016-2017, AZR collected channel samples from two exposures of the mineralised zone in the bank of the Unkur Creek (Outcrop 3 (see Figure 7.6) and 5 (Figure 9.1); and from four sites of historical trenching that were cleared to re-expose the bedrock. In total, 67m of samples were collected from the outcrops, and 186m from the trenches. Sampling was done on one-meter lengths, with a nominal width of 5cm and depth of 3cm. Sample locations were derived based on several hand-held GPS measurements along each sampling profile.

Figure 9.1: Outcrop 5 (OC005) with Clear Northeast-Dipping Bedding of Cupriferous Sandstone in the Left Bank of Unkur Creek

This outcrop was used to collect an oxide sample for metallurgical tests.

The outcrop channel samples were approximately orientated along the strike of the mineralisation, and the irregular outcrop surface meant that it was difficult to obtain a consistent sample width and depth. Nevertheless, the outcrop samples were retained in the database to inform the geological modelling and geostatistical estimation. The overall Inferred classification applied to the Mineral Resource estimate is considered to adequately cover such risks to sample quality.

The trenches are oriented on azimuths approximately perpendicular to the mineralisation. The trench sampling information was merged into the drill hole database, effectively as a set of drill holes parallel to the topographic surface. Three of the trenches intersected copper-silver mineralisation (Table 9.1). None of the samples from trench K801 returned results indicating significant copper-silver mineralisation. The channel samples from the trenches, which the qualified person considers to be similarly reliable and representative as samples obtained from drill core, were used for both modelling the contacts of the mineralisation domains and for the geostatistical grade estimation within these domains.

Table 9.1: Trench and Outcrop Intersections Used for Mineral Resource Estimation (0.1% Cu Threshold)

Trench ID	Zone intersected	From m	To m	Length m	Cu %	Ag ppm	True Thickness m
K-601	North	0	10	10	0.73	2	7
K-615	North	8	17	9	0.30	14	7
K-616	North	18	29	11	0.41	6	7
OC005	North	172	198	26	0.69	35	17
OC003	West	0	32	32	0.15	14	28

9.2 Ground Magnetic Survey

A total of 53 square kilometres of detailed ground magnetics data were collected during AZR's first phase exploration programme in 2016-2017 and during the second phase in 2019. The traverse lines were oriented northeast across the strike of the Unkur Syncline. The distance between the lines was 100m, with sampling sites every 25m along the lines. The data were collected by AZR contractors using precision complex MagniProX4 and proton magnetometers ММП-203 (Figure 9.2). The data were processed by Condor Consulting Inc, a specialized geophysical firm based in Colorado, USA. The resulting maps show that copper-silver mineralisation is associated with a strong magnetic signature (Figure 6.3) and that ground magnetics is a useful targeting tool on the project that was successfully tested in 2019-2020.

Figure 9.2: Precision Complex MagniPro X4 and Modernized Proton Magnetometer ММП-203

9.3 Induced Polarization Survey

In September-October 2019, e.g., before the 2019-2020 drilling campaign, Tien Shan Ltd, a Bishkek-based contractor of AZR, conducted an induced polarization survey at Unkur using pole-dipole (PDIP) technique. This technique allows to scan the project area to a depth of 500m. Tien Shan Ltd completed 9 profiles (2 of which were continuations of other profiles on different banks of the Kemen River) 1km apart with 50m spacing between the reading stations along the profile. The profiles were oriented across the strike of lithologies interpreted from magnetic survey completed in August 2019 by the same contractor. The total length of PDIP profiles is 28km, and they covered the larger part of the Unkur licence (Figure 9.3).

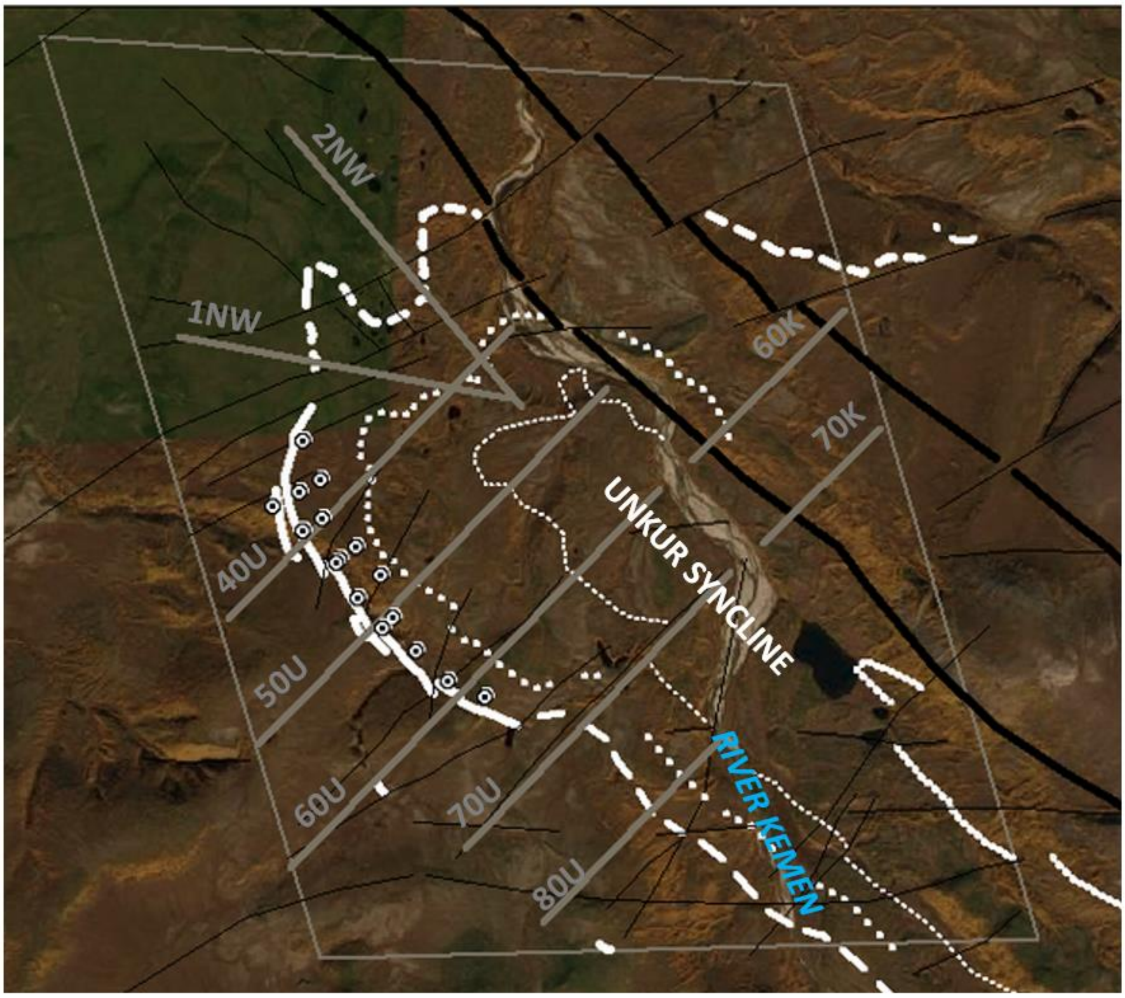


Figure 9.3: Position of PDIP Profiles on the Satellite Image of the Unkur Project Area
[Dashed white lines correspond to stratigraphy interpreted from magnetic images. Solid white lines correspond to the extent of the 2018 Inferred resource based on 2016-2017 drill holes by AZR. Interpreted faults are shown in black.]

Tien Shan Ltd used an 8.8 kW petrol-electric Aksa Generator ABE 110 M with 1000V and 6A. The measurements were taken with ЭИН-209М device, produced by Institute for Geophysical Exploration, National Nuclear Center, Republic of Kazakhstan (Figure 9.4). The data were collected directly into computer. The time length of current impulse was 1.6 sec, with measurements of transmission voltage collected during 1.25 sec, and measurements IP voltage at 0.04375, 0.0875, 0.175, and 0.350 sec.

Figure 9.4: Portable Electric Survey Devices ЭИН-209М

The collected data were processed using Tien Shan Ltd.'s in-house software and MapInfo package. The inverted geoelectric models were further processed in RES2DINV (M. Loke) by GEOTOMO SOFTWARE (www.geotomosoft.com).

Profiles 40U and 50U are type sections across the mineralized zone at Unkur. They were used to calibrate the chargeability and resistivity values against the known mineralization.

Profile 40U (Figure 8.5) crosses the Unkur mineralization at stations 95-110 where chargeability is 3%. It appears that the anomaly is much wider than copper mineralization. It is interpreted to map a pyrite envelope which is also wider than copper mineralization. There is no chargeability anomalism within ≈100m from the topography surface, corresponding to most intense oxidation. Profile 40U also has another anomaly in the eastern part. This was drill tested by hole AM19-001, which showed presence of pyrite without copper sulphides.

The interpreted magnetic susceptibility section (Figure 9.5, top profile) shows position of mineralization at the magnetic gradient. This correlates well with the plan view interpretation discussed above. The resistivity section (Figure 9.5, bottom profile) maps a 100-150m thick sub-horizontal anomaly (>5000 Ohm*m) extending for 3-4 km. Underneath this anomaly, the resistivity shows 500 Ohm*m values. The near-surface anomaly is interpreted to map the permafrost. Resistivity data can be therefore used to map the extent of permafrost.

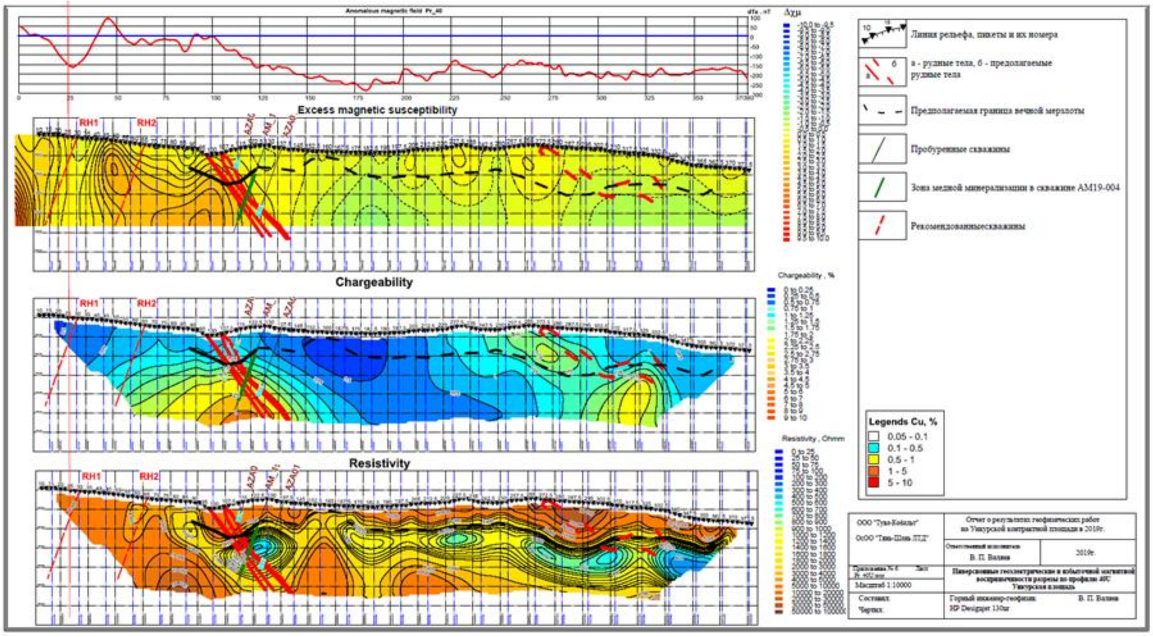


Figure 9.5: Profile 40U, Showing Excess Magnetic Susceptibility, Chargeability and Resistivity Sections and Position of the Unkur Mineralization

This approach was applied to all other profiles. The chargeability and resistivity isolines were then imported into Datamine software along with magnetic gradient lines and 2018 mineralized envelope (Figure 9.6). This model was used during 2019-2020 drilling when AZR targeted the north-western and south-eastern extents of Unkur mineralization.

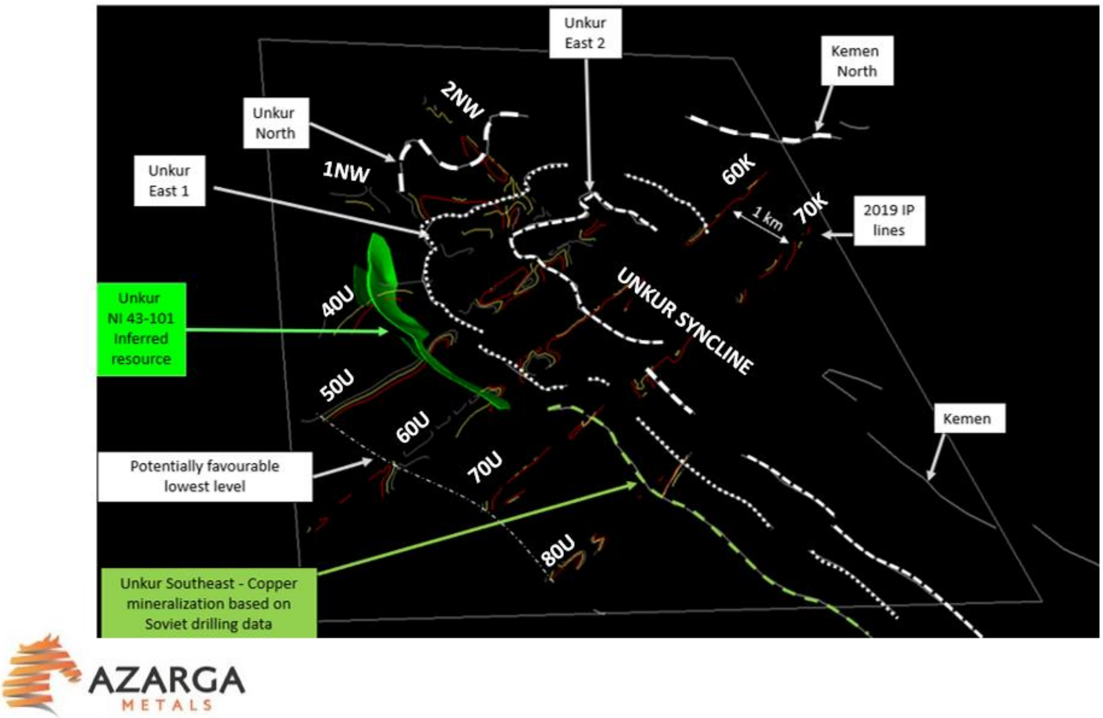


Figure 9.6: 3D Diagram of the Unkur Licence Area Showing Chargeability Data and Magnetic Gradient Lines against the Mineralized Envelope of the 2018 Inferred Resource (AZR, 2019)

9.4 Geochemical Sampling

In October to early November 2020, AZR conducted a reconnaissance geochemical work. The objective of the programme was to identify new areas of copper and silver mineralization outside of the known area of recent drilling by taking lithochemical samples just below the surface over widespread areas of the property and analysing the -1mm fraction. Due to COVID-19 restrictions AZR could not start the work in summer 2020. The start of the programme was delayed until autumn when there was already frost and snow on site. The original design of the programme was reduced but despite this, 180 rock-chip samples and 28 channel samples were taken from outcrops at 6 locations.

The results of the channel sampling showed copper mineralization at surface on the east side of the Kemen River (Kemen area) as well as areas on the southwestern side (Unkur SW) of the property (Table 9.2; Figure 9.7).

In the Kemen area, several lithochemical samples with more than 0.05% copper highlighted three anomalies extending over strike lengths of 2km, 1.1km, and 0.9 km. This mineralization possibly occurs at different stratigraphic levels.

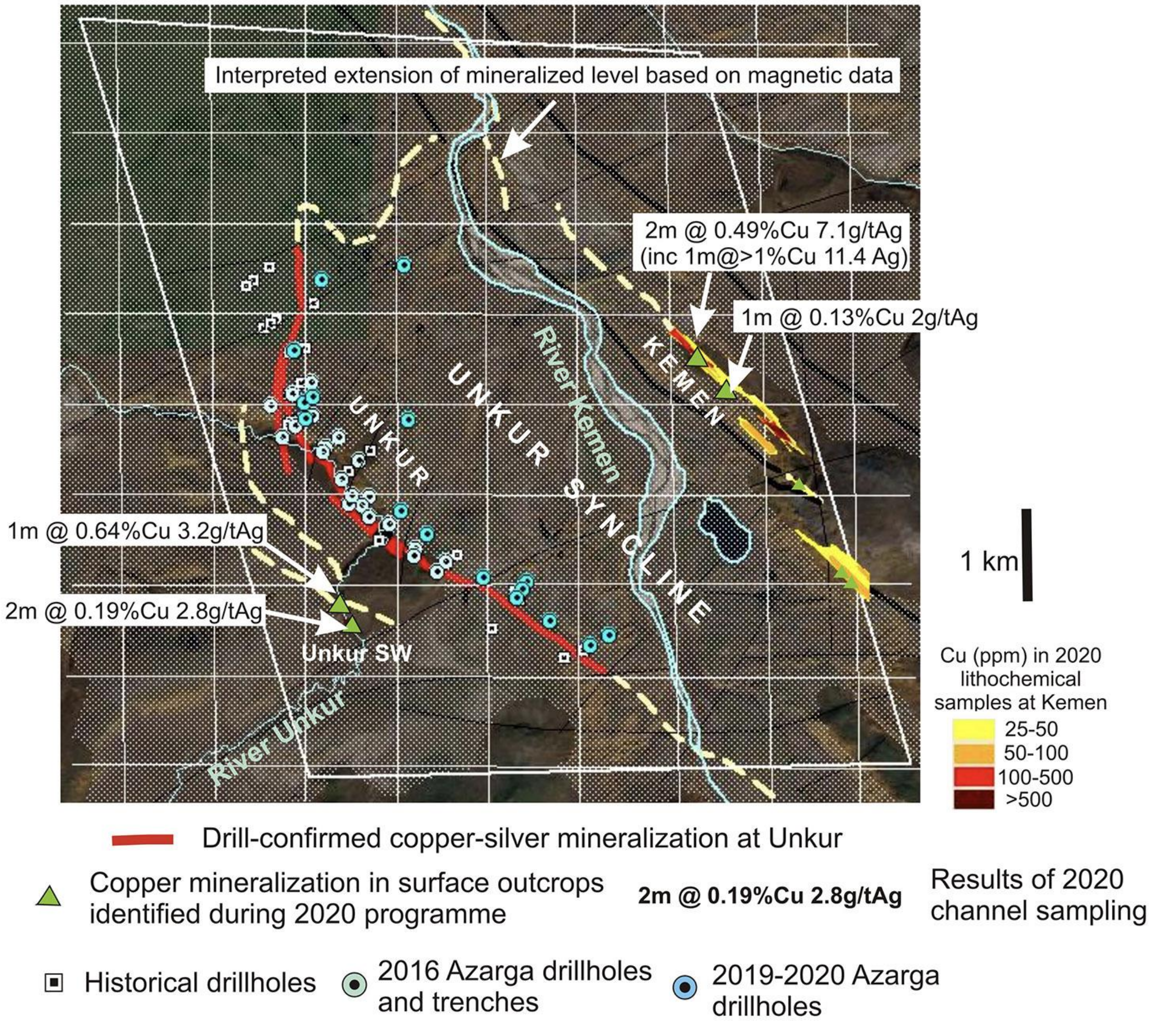


Figure 9.7: Satellite Image of the Unkur Licence Area Showing Extent of Identified Lithochemical Anomalies vs Extent of Moraine

Cu mineralization, previously drilled in the southwestern limb of the Unkur Syncline, is now discovered on its north-eastern limb at Kemen.

Copper mineralization in outcrops was found at several locations on which channel samples were taken (Table 9.2). This finding confirms, for the first time, that mineralization is present in bedrock on the eastern limb of the Unkur Syncline.

Table 9.2: Results of 2020 Channel Sampling at Kemen and Unkur SW
All Samples Represent 1 m Intervals

Sampling site	WGS 84 coordinates	Pulkovo 42 coordinates	Sample ID	Cu ppm	Ag ppm	Intercept
AM200168 Kemen	56.841405	20600384.87	201681	461	0.2	
	118.645723	6303492.73	201682	8045	11.0	2m at 0.49% Cu 7.1 g/t Ag
			201683	1760	3.1	
AM200064 Kemen	56.837931	20600649.32	200641	247	<0.2	
	118.649904	6303112.086	200642	1280	2.0	1m at 0.13% Cu 2.0 g/t Ag
			200643	236	0.7	
AM200108 Kemen	56.828305	20601513.98	201081	541	2.5	
	118.663646	6302060.653	201082	69	<0.2	
AM200135 Kemen	56.819174	20602041.98	201351	14	<0.2	
	118.671889	6301056.294	201352	699	2.0	
			201353	104	<0.2	
			201354	39	<0.2	
			201355	87	<0.2	
AM200140 Kemen	56.819092	20602215.6	201401	18	<0.2	
	118.674729	6301051.403	201402	61	<0.2	
			201403	72	<0.2	
AM200150 Yuktokan	56.859652	20601916.63	201501	2430	2.6	
	118.671639	6305562.638	201502	796	1.0	
AM200153 Unkur SW	56.815975	20596490.4	201531	2230	3.3	2m at 0.19% Cu 2.8 g/t Ag
	118.580829	6300568.023	201532	1480	2.2	
AM200160 Unkur SW	56.81771	20596409.67	201601	526	<0.2	
	118.57958	6300759.436	201602	6420	3.2	1m at 0.64% Cu 3.2 g/t Ag
			201603	769	<0.2	
AM200161 Unkur SW	56.81843	20596402.94	201611	39	<0.2	
	118.57950	6300839.488	201612	67	<0.2	
			201613	38	<0.2	

At Unkur SW, new bedrock mineralization was discovered over an area of 250 x 160m in four closely spaced outcrops on which channel samples were taken (Table 9.2). This mineralization appears to be stratigraphically below the main Unkur mineralization and implies a second perhaps parallel zone of mineralization to the known body hosting the current resources at Unkur. Comparison with the 2016-2017 trench samples suggests that the copper values in the channel samples are more indicative of underlying grade due to the greater mobility of silver over copper at surface.

At Kemen, Cu oxide rocks (including residual chalcopyrite, bornite and chalcocite) were discovered in 4 separate outcrops over a total distance of 3km. In all cases the stratigraphic layering was measured to dip southwest at 40 to 60°.

These discoveries generated new exploration targets at the Unkur Project.

10 DRILLING

The main source of information for the mineral resource estimate presented in this report is 10,388 meters of diamond core drilling (from 33 drill-holes) completed during AZR exploration campaigns: from August 2016 until February 2017, and from October 2019 to April 2020. Section lines for drilling are mostly spaced 300m to 500m apart. In the north, there are three fences drilled 150 m apart.

10.1 Type and extent

Summary information for individual holes and intersections is listed in Table 10.1 and Table 10.2. Figure 10.1 shows a plan of the collar locations, and representative sections are presented in Figure 10.2 to Figure 10.5.

In 2016-2017, the holes were drilled by two Christensen CS14 rigs. Core was collected on 3.0m drilling lengths, using a double tube core barrel. Drilling through the loose sediments of the moraine was done at PQ diameter. The hole diameter was reduced to NQ, or (less frequently) HQ, for drilling the bedrock. Hole collars were surveyed using a hand-held GPS device. The down hole orientation was surveyed using an IMMN-42 magnetometric inclinometer.

In 2019-2020, the holes were drilled by one Boart Longyear LF90 rig. Core was collected on 3.0m drilling lengths, using a double tube core barrel. Drilling through the loose sediments of the moraine was done at PQ diameter. The hole diameter was reduced to NQ, or (less frequently) HQ, for drilling the bedrock. Hole collars were surveyed using a hand-held GPS device. The down hole orientation was surveyed using a Roschen EZ-SHOT №1552 inclinometer. The deviation of azimuth was less than 10°. The dip of holes was flattening at depth in comparison with the start of the hole.

10.2 Location, Spacing, Distribution and Orientation of Data

All data was supplied in the Gauss Kruger, Pulkovo 42, Zone 20 Projection and it is understood that collar positions for all holes were laid out by the on-site surveyor using a differential GPS and then checked again once drilling was completed. Downhole surveys were carried out for all of the diamond drillholes using Reflex Ez-Shot equipment over a nominal interval of 20m in general.

Data spacing is nominally on fence lines spaced 200-500m over a strike length of 4.5km with spacing down to 100-200m along fence lines parallel to dip direction in the central and western parts of the deposit with some area of infill drilling to 50m. In the southern zone the data spacing is wider between 500m x 200m and prospective in nature with 1-2 holes along fence lines. Given the style of mineralisation this spacing is sufficient to establish geological and mineralisation continuity appropriate for the reporting of Mineral Resources.

In general, drilling is carried out so that the intersections of holes with mineralised zones occurs at a high angle which results in limited sample bias. The general strike of mineralisation is to northwest at 310° with sub-vertical steeply dipping mineralisation zone hence drilling is generally inclined at

-50-60° towards the strike of the zones. Intercepts are reported as apparent thicknesses except where otherwise stated.

10.3 Factors that could materially affect the accuracy and reliability of results

SRK has considered drilling, sampling and recovery factors that could materially affect the results from AZR sampling. In the opinion of SRK, the sampling procedures used by AZR are consistent with generally accepted industry best practice. All drilling sampling was conducted under the direct supervision of appropriately qualified geologists.

The core from the mineralised zones is often very broken, so it is often not practical to estimate recovery by piecing together the fragments and measuring the length. Instead, recovery is estimated based on comparisons of actual to theoretical sample weights. The average recovery from the mineralised zone is estimated to be approximately 90%. Given the style and grade of mineralisation at Unkur, SRK considers this recovery to be sufficient for the samples to support mineral resource estimation.

Table 10.1: Drill Hole Location, Maximum Depth, and Orientation							
Hole ID	Campaign	Collar coordinates (Pulkovo 42 datum, Zone 20)			Maximum depth (m)	Starting dip	Starting azimuth
		x	y	z			
AM19-001	2019-2020	597153	6302819	1018	301	-69	241
AM19-002	2019-2020	599377	6300437	935	502	-70	248
AM19-003	2019-2020	599171	6300329	957.7	213	-72	242
AM19-004	2019-2020	596012	6302834	940	485.3	-70	242
AM19-005	2019-2020	595996	6303013	932	500	-71	241
AM19-006	2019-2020	596093	6303071	927	532	-69	221
AM19-007	2019-2020	595893	6303584	905	237	-70	222
AM19-008	2019-2020	597074	6301813	979	450	-72	228
AM19-009	2019-2020	597366	6301559	996	304	-69	224
AM19-010	2019-2020	598462	6301026	979	318	-68	223
AM19-010/2	2019-2020	598410	6300948	703	162	-68	220
AM19-011	2019-2020	598725	6300600	986	200	-68	217
AM19-012	2019-2020	598351	6300847	1002	220	-71	230
AM19-013	2019-2020	597975	6301070	1014	270	-73	241
AM19-014	2019-2020	596186	6304375	886	486.9	-69	224
AM19-015	2019-2020	597106	6304535	886	473.9	-70	249
C-9	2016-2017	595906.1	6303578	902.7	284.9	-70	249
C-11/1	2016-2017	595871.3	6303108	929.7	400.5	-69	241
C-12/1	2016-2017	596076.7	6303227	919.4	520.5	-70	248
C-14/1	2016-2017	595918	6302775	924	100	-72	241
C-15/1	2016-2017	596100	6302892	935.5	382.9	-69	242
C-17/1	2016-2017	596246.8	6302510	913.7	160	-71	238
C-18/1	2016-2017	596388.3	6302620	955.1	572	-69	221
C-20/1	2016-2017	596410.6	6302155	927.7	80	-70	222
C-21	2016-2017	596610.8	6302365	1000	601.3	-72	228
C-23/1	2016-2017	596731	6301991	960	238	-69	224
C-26/1	2016-2017	596936.3	6301672	952.3	178.5	-68	223
C-29/1	2016-2017	597233.3	6301394	995.8	100	-68	220
C-31/1	2016-2017	597566.7	6301246	1042	201	-68	217
C-50/1	2016-2017	596210.9	6302467	916.1	277.5	-71	230
C-51/1	2016-2017	596639.3	6301879	938.9	226.7	-69	224
C-52/1	2016-2017	595635.3	6302977	937.8	256.6	-73	241
AZA010-1	2016-2017	596118	6303693	902.7	152.7	-70	248

Table 10.2: Drill Hole Intersections (0.1% Cu threshold) used for Mineral Resource Estimation							
Zone	Hole ID	From (m)	To (m)	Length (m)	Composite Grades		True Thickness (m)
					Cu (%)	Ag (ppm)	
North	AM19-004	194	277	83	0.62	60	68.2
North	AM19-005	247	269	22	0.74	55	18.0
North	AM19-006	305	336	31	0.40	26	25.9
North	AM19-008	328.8	340.8	12	0.17	3	9.6
North	AM19-009	231	241	10	0.19	4	7.9
North	AM19-014	445	447	2	0.14	150	1.5
North	C-9	231	241	10	0.68	38	7.5
North	C-11/1	82.5	125.5	43	0.65	62	35.8
North	C-12/1	432.5	472.5	40	0.32	13	34.1
North	C-14/1*	40.501	76.5	36	0.44	41	29.9
North	C-15/1	319.5	358.5	39	0.35	22	31.7
North	C-18/1	468.5	484.5	16	0.34	11	13.3
North	C-20/1	47	60	13	0.25	17	10.5
North	C-21	352.3	364.3	12	0.24	6	9.8
North	C-26/1	145.5	153.9	8.4	0.92	62	7.0
North	C-29/1	61	78	17	0.31	12	14.2
North	C-31/1	133	145	12	0.26	4	10.0
North	C-50/1	189.5	202.5	13	1.28	104	10.5
North	C-51/1	32	49	17	0.33	8	14.0
West	C-11/1*	311.5	346.5	35	0.48	44	2.6
West 2	AM19-005	293	295	2	0.15	7	0.8
South	AM19-002	324	330	6	0.94	23	5.5
South	AM19-003	172	180	8	0.33	22	7.1
South	AM19-010/2	65	83	18	0.27	8	14.5
South	AM19-011	159	192	33	0.18	6	25.5
South	AM19-012	162	177	15	0.31	13	11.1
South	AM19-013	232	250	18	0.28	6	14.4

* Denotes intersections that begin immediately below the base of moraine, and therefore are possibly truncated by erosion.

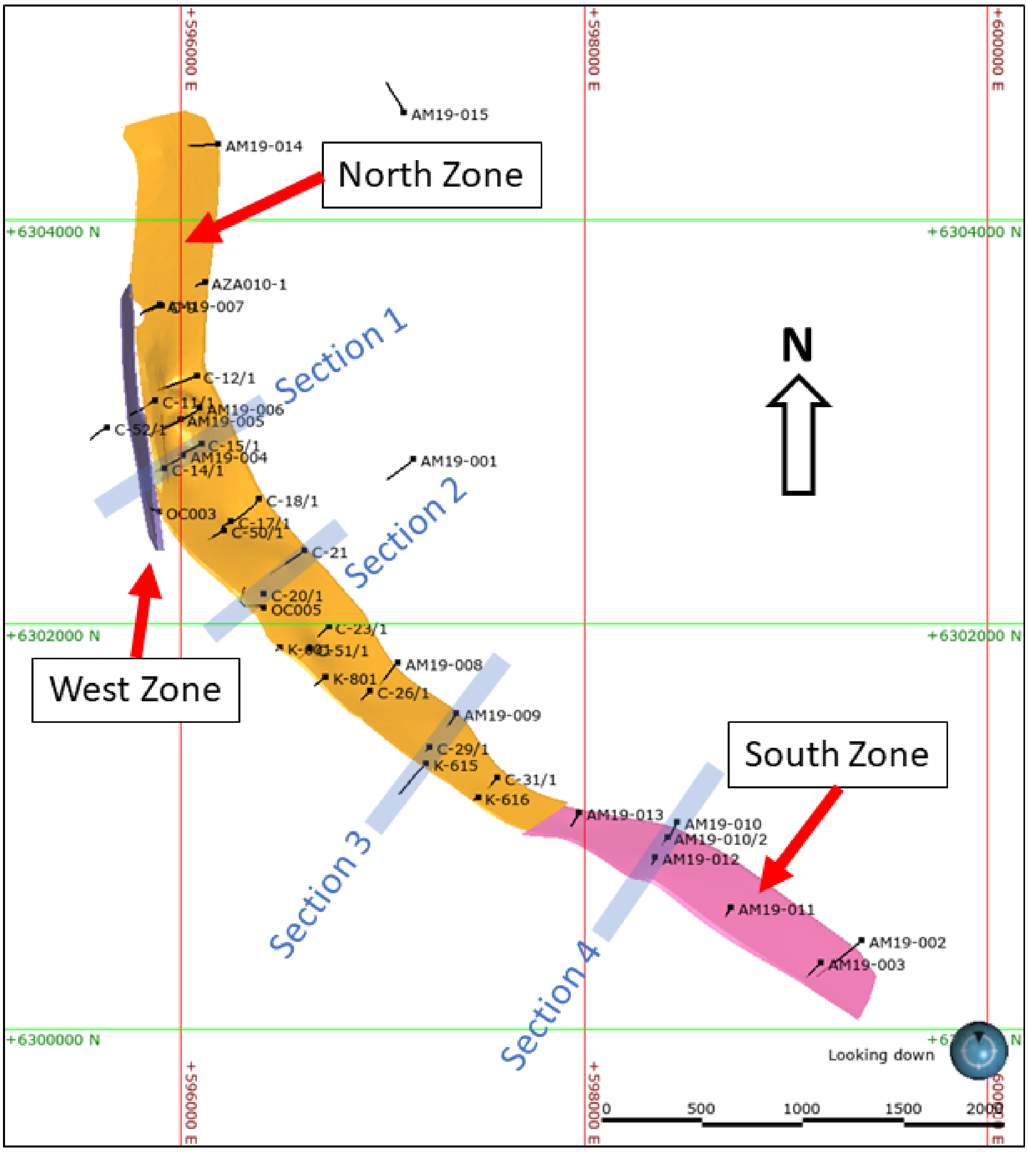


Figure 10.1: Plan Showing Collar Locations and Drill Hole Traces in Relation to Modelled Mineralisation Domains

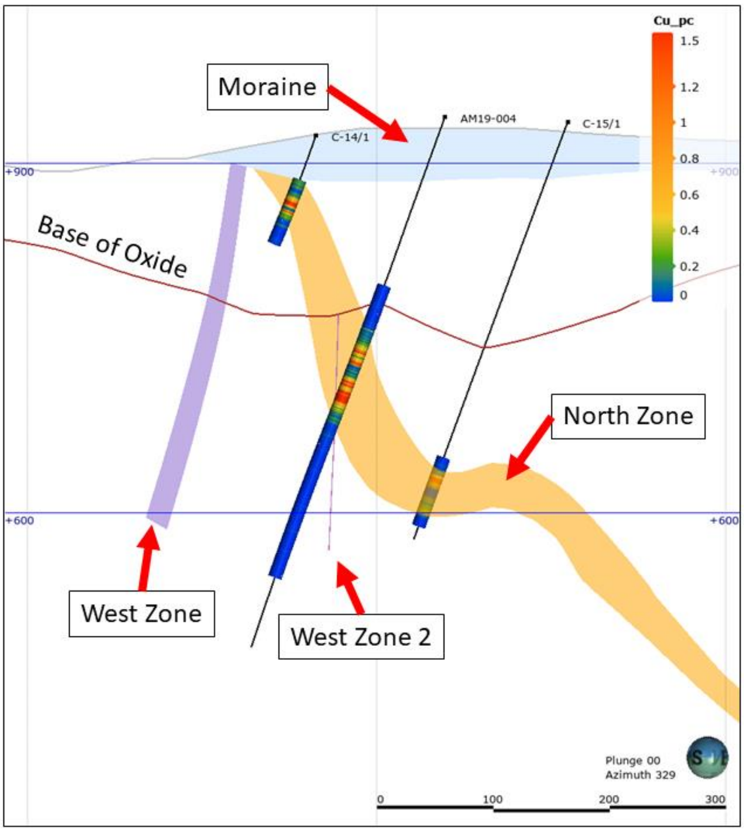


Figure 10.2: Vertical Cross Section 1. View Looking NNW

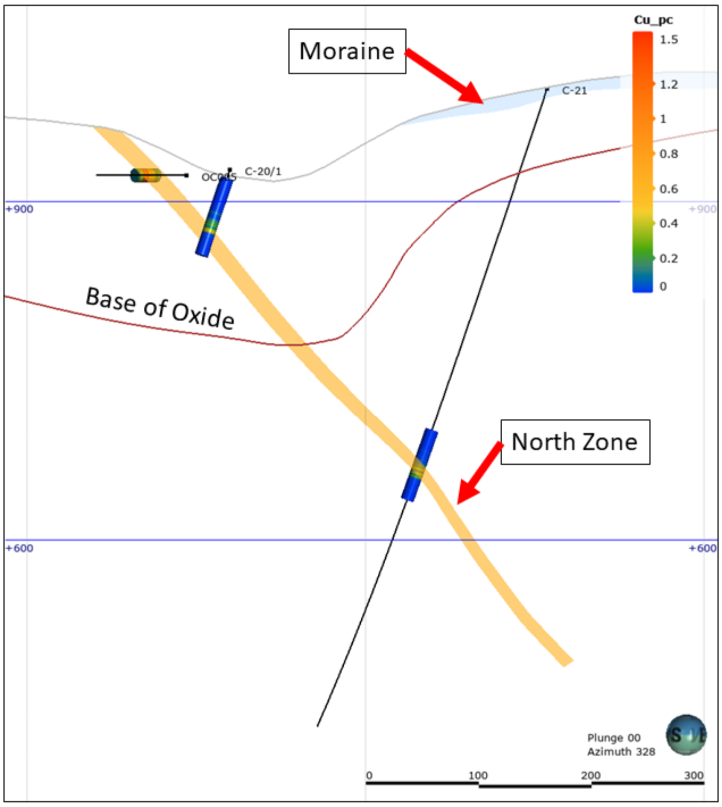


Figure 10.3: Vertical Cross Section 2. View Looking NNW

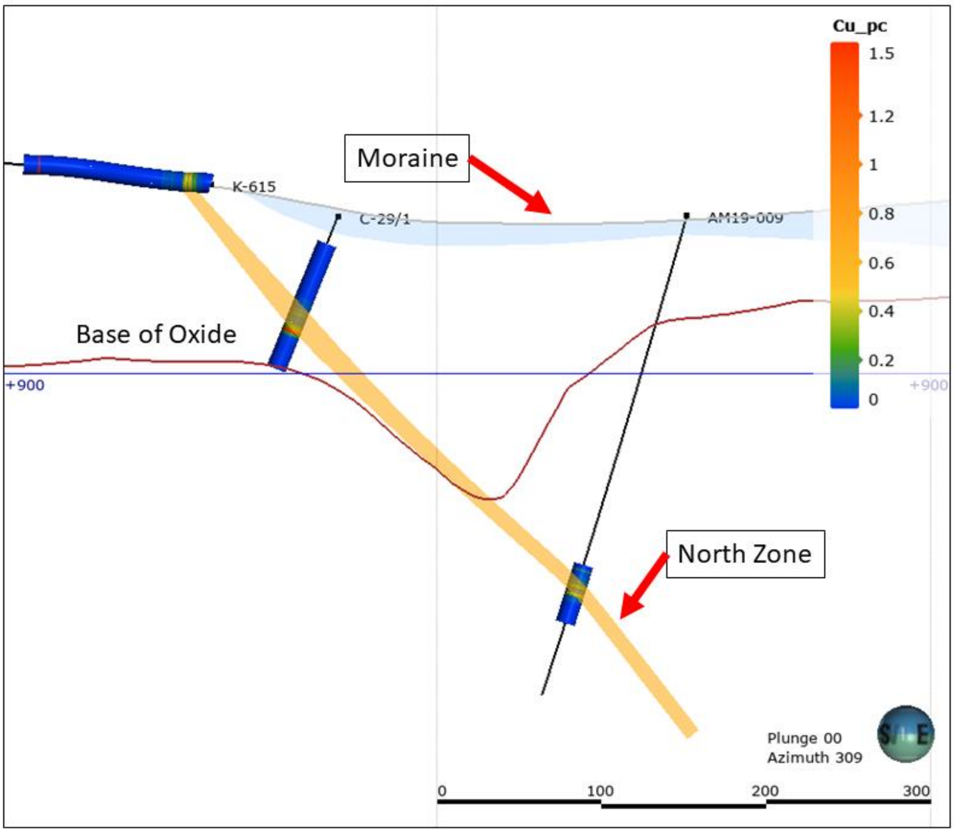


Figure 10.4: Vertical Cross Section 3. View Looking NW

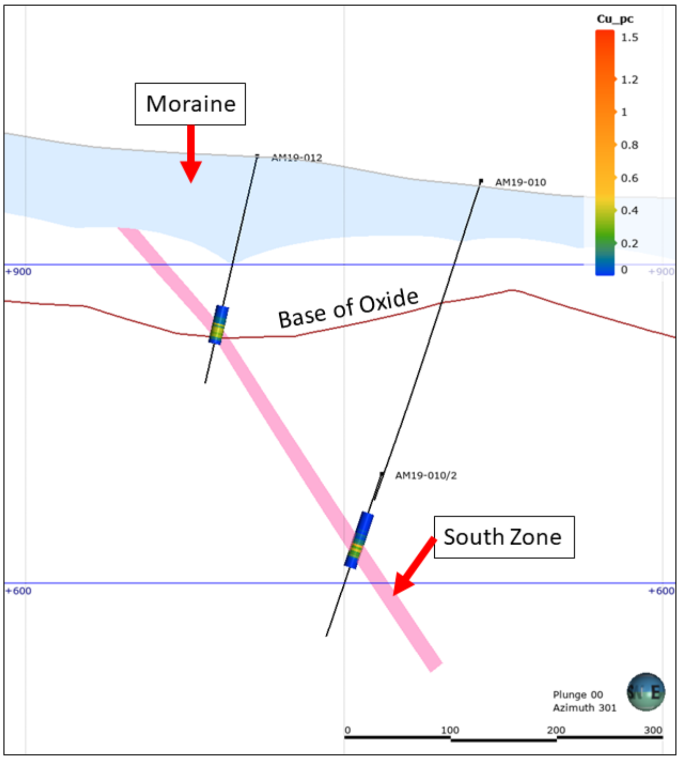


Figure 10.5: Vertical Cross Section 4. View Looking WNW

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Sample preparation on site

In 2016-2017, core trays were transported from the rigs to the AZR exploration camp. This transportation distance was up to 3km. All core was digitally photographed. Intervals identified by the geologists as likely to be mineralised were selected for sampling, and the sampling interval was extended for at least 10m beyond the limits of the identified mineralisation. Hand-held XRF measurements were used as a further check, to ensure that all mineralised zones were identified for sampling. Several hand-held XRF readings of copper content were taken within each meter of core. XRF copper readings were used as a logging tool, not in the resource estimate calculations.

In 2019-2020, core trays were transported from the rigs to the AZR logging facility in Chara. This transportation distance was up to 40-45km. Institute of Geotechnologies (IGT, Moscow) was employed to log the core under the guidance of AZR geologists. All core was digitally photographed. After some training, mineralized intervals could be easily identified by the geologists due to obvious presence of green copper oxides and well identifiable copper sulphides. The mineralised intervals were selected for sampling, and the sampling interval was extended for at least 10m beyond the limits of the identified mineralisation. No hand-held XRF measurements were used in 2019-2020.

The 2019-2020 core is kept in the rented storage facility in Chara. All 2016-2017 core was also transported to Chara. For consistency, all 2016-2017 core was completely relogged by IGT geologists. Both in 2016-2017 and 2019-2020, core selected for sampling was cut with a core saw. Sample lengths were nominally 1.0m, but adjustments to the lengths were made in order to honour geological boundaries. The minimum sample length was 0.4m and the maximum length was 1.3m. 970 of the 971 samples from the 2019-2020 drilling programme were 1.0m long. Half-core from the intervals selected for sampling was dispatched by railroad to SGS and ALS Laboratories in Chita. Trays of the retained half core were closed with covers marked.

11.2 Sample preparation and analysis at laboratory

The primary laboratory used for analysing AZR 2016-2017 samples is SGS Vostok Limited (SGS) in Chita. The laboratory is independent from AZR and has ISO/IEC 17025 certification for the specific procedures used.

Samples received by SGS were dried at 105 ± 5˚C. Samples up to 4.0kg were then crushed to 85% passing 2.0mm, and ground to 90% passing 0.7mm. Sieving checks were done on 3 – 5% of the samples. Samples of more than 4.0kg went through the same crushing stage but were split to 4.0kg before proceeding to the grinding stage.

A subsample of 0.5 to 1.0kg was collected using a rotary splitter. This subsample went through a further stage of fine grinding, to 95% passing 75µm. A 50% split of this subsample (250 to 500g) was used for analysis.

SGS analysed the samples for copper and silver. The copper content was determined by SGS method ICP90A (sodium peroxide fusion, then inductively coupled plasma - atomic emission spectroscopy). The silver content was determined by SGS method AAS12E (two acid digest, then atomic absorption spectroscopy).

The primary laboratory used for analysing AZR samples in 2019-2020 was ALS-Chita (ALS). ALS is independent from AZR and has International Organisation for Standardisation/International Electrotechnical Commission (ISO/IEC) 17025 certification for the specific procedures used.

ALS analysed the samples for copper and a set of 16 other elements using ME-ICP81 method. The silver content was determined by Ag-AA45 method (two acid digest, then atomic absorption spectroscopy).

All 2016-2017 and 2019-2020 samples with >0.1% Cu values were analysed for oxide/sulphide copper.

External quality control samples used by AZR included certified reference material (CRM) and blanks, submitted to SGS with the primary samples, and check assays by an umpire laboratory (ALS in Chita and Alex Stewart Assayers in Bishkek, Kyrgyzstan).

11.3 Quality control / Quality assurance

11.3.1 2016-2017 Campaign

The control samples AZR submitted during the 2016-2017 campaign included 73 samples from four different certified reference materials (prepared by laboratory Udokanskaya Med), and 90 pulp duplicates submitted to an umpire laboratory (ALS in Chita). Compared to the 1,799 primary samples analysed during this campaign, these two types of control samples were submitted at rates of approximately 4% and 5% respectively. The specifications and results of the 2016-2017 control samples were presented and discussed in the 2017 (SRK) and 2018 (Tetra Tech) technical reports for the project.

In the opinion of the QP for this section of the current technical report (and who is also the QP for the equivalent sections in those previous technical reports), the quality control sampling results from the 2016-2017 campaign do not imply any bias or precision problems significant enough to materially affect confidence in the Mineral Resource estimate.

11.3.2 2019-2020 Campaign

The control samples used during the 2019-2020 campaign included certified reference materials (CRMs), blanks, and quarter-core duplicates re-submitted to the primary laboratory. Compared to the 970 primary samples submitted to SGS during 2019-2020, the CRM, blanks, and core duplicates were added at a rate of approximately 4%.

Certified reference materials

Forty-one samples from five different CRMs (sourced from Geostats Pty Ltd, Australia) were analysed. The results are presented in Table 11.1 and Figure 11.1.

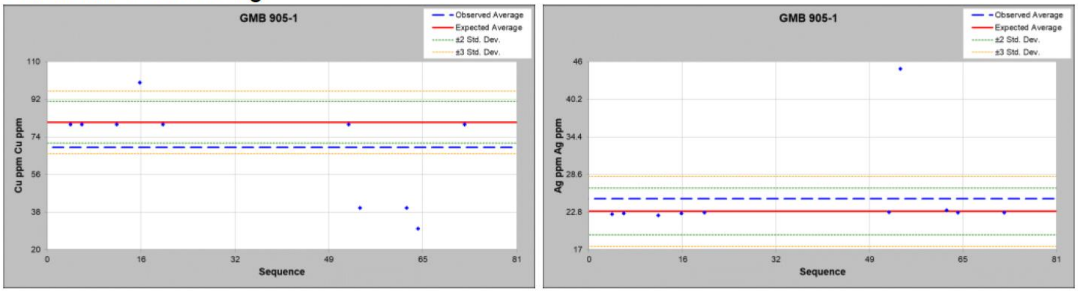
Table 11.1: Summary of Results from Analyses of Certified Reference Materials

Quality Control Sample ID	Number of Analyses by SGS	Element	Certified Value	Mean SGS Analysis	Minimum SGS Analysis	Maximum SGS Analysis
GM 905-1	10	Cu %	0.0081	0.0069	0.0030	0.0100
		Ag g/t	22.9	24.9	22.3	23.1
GMB 399-6	8	Cu %	2.14	2.09	2.03	2.14
		Ag g/t	15.5	15.7	15.0	16.2
GMB 311-3	3	Cu %	1.009	1.027	1.025	1.030
		Ag g/t	20.4	19.3	19.2	19.5
GMB 311-2	9	Cu %	0.227	0.224	0.216	0.234
		Ag g/t	10.4	9.9	9.4	10.8
GMB 310-2	11	Cu %	0.694	0.697	0.651	0.889
		Ag g/t	45.5	44.2	40.8	48.0

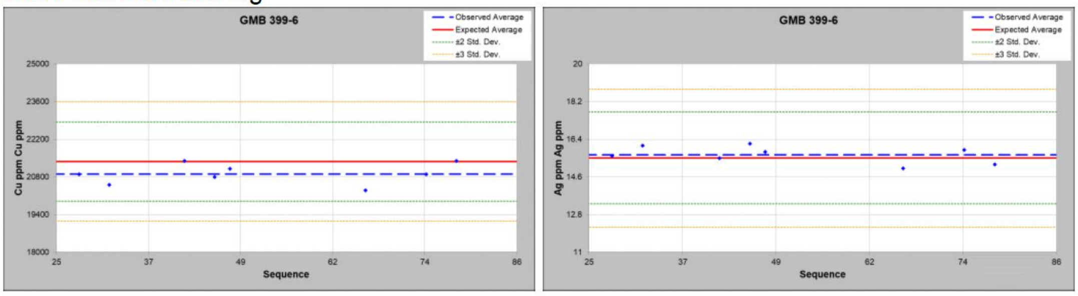
Blanks

Forty-one blanks were inserted. Two blanks returned results >0.010% Cu (maximum 0.226%), and one blank returned a result >10 g/t Ag (35 g/t). The anomalous Ag result came from a sample that also returned one of the two anomalous Cu results.

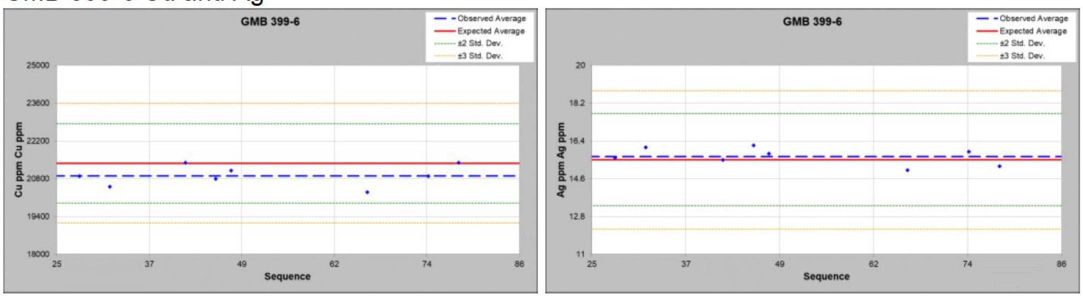
GMB 905-1 Cu and Ag



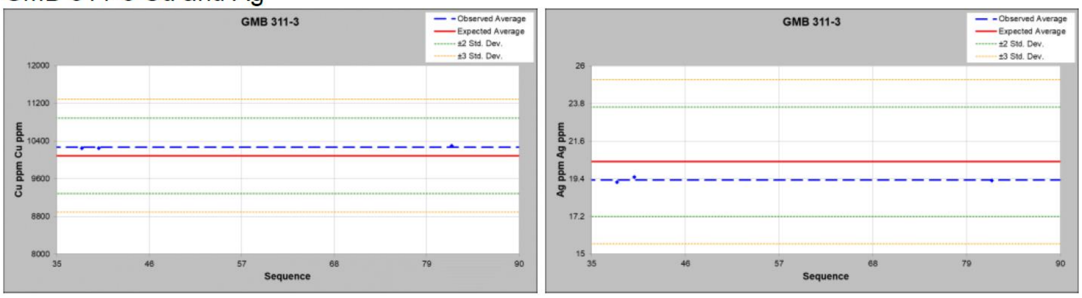
GMB 399-6 Cu and Ag



GMB 399-6 Cu and Ag



GMB 311-3 Cu and Ag



GMB 310-2 Cu and Ag

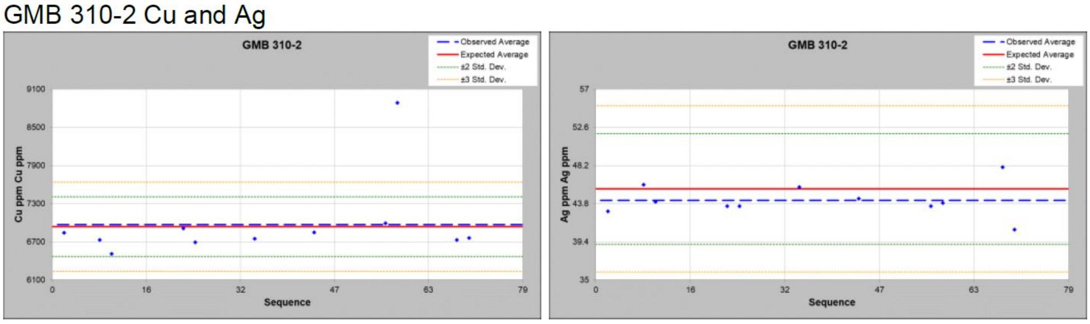


Figure 11.1: Charts of CRM Results from 2019-2020

Core duplicates re-submitted to SGS

The Cu and Ag results from thirty-seven quarter-core duplicates are shown in Figure 11.2, with logarithmic scale axes. Summary statistics from these duplicates are listed in Table 11.2.

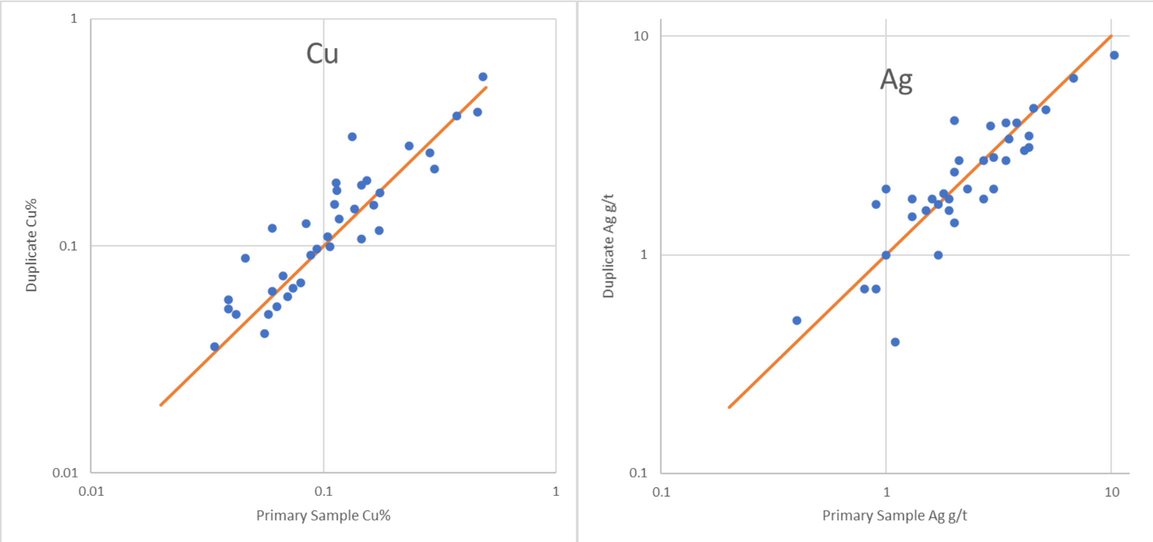


Figure 11.2: Results from Quarter-Core Duplicates

Table 11.2: Summary Statistics for Pulp Duplicates				
	Cu% Primary	**Cu% Duplicate**	**Ag g/t Primary**	**Ag g/t Duplicate**
Count	37	37	37	37
Min	0.03	0.04	0.40	0.40
Max	0.49	0.55	10.30	8.20
Mean	0.14	0.15	2.68	2.57
Std Dev	0.11	0.11	1.90	1.63

Conclusion from review of 2019-2020 quality control results

In the opinion of the QP, the quality control sampling results from the 2019-2020 campaign do not imply any bias or precision problems significant enough to materially affect confidence in the Mineral Resource estimate.

11.4 QP Comments

The sample preparation, security and analytical procedures used by AZR are consistent with generally accepted industry best practices and are, therefore, adequate for the purpose of mineral resource estimation.

12 DATA VERIFICATION

12.1 Data verification by the qualified person

The qualified person visited site on December 10, 2014, and October 13, 2016. The 2016 visit included a visit to the primary assay laboratory (SGS in Chita) the following day.

The qualified person has verified the database the mineral resource estimate is based on. This verification was done by personal inspection of drill core, drill sites and trenches during the 2016 site visit, by analysing the results from quality control samples, and by checking database content against primary data sources and historical information.

A follow-up site visit during 2020 was not possible, because of COVID-19 travel restrictions. For data verification, the qualified person has relied on observations from the previous site visits, and validation checks on the digital data. During the process of importing into the database used for modelling and estimation, the information from the 2019-2020 campaign was checked for internal consistency (such as having complete collar and survey information, and absence of overlapping intervals) and for consistency with the data from the previous campaign (for example, reasonably compatible locations of mineralised intersections).

12.2 Limitations on data verification

During the 2014 site visit, SRK visited an old core storage facility (Figure 12.1) and inspected the state of the historical core (Figure 12.2). The historical sampling could not be verified because of the poor condition of the core, due to poor recovery during drilling, deterioration of the core and core trays over the subsequent four decades, and collapse of the core storage shed. Also, it appears that the intervals of most interest (the mineralised intersections) were generally entirely consumed by sampling during the historical exploration programmes.

Because of the limitations on the confidence in the quality of the historical data, this information was not used by SRK to prepare the mineral resource estimation.

Figure 12.1: Old Core Storage, the Unkur Project (source: SRK, December 2014)

Figure 12.2: Core Recovered from hole C-118 (source: SRK, December 2014)

12.3 Opinion on Data Adequacy

The QP is satisfied that data collection, security, spacing and orientation of sample collection is sufficient to support the Mineral Resource classification presented herein.

13 MINERAL PROCESSING AND METALLURGICAL TESTWORK

13.1 Procedures

The most recent metallurgical testwork was conducted at "VNIItsvetmet" laboratory in Kazakhstan in March/April 2021 as part of this scoping level study. This included fine and coarse ore bottle roll tests using cyanide leaching on oxide ore samples only. The other recent testwork, also conducted at "VNIItsvetmet" in 2020, included additional oxide ore leaching tests and flotation testwork on sulphide ore samples. The most recent study report is the Tetra Tech PEA issued in August 2018 and reference should be made to this report.

13.2 Historical Testwork – SGS Vostok

Most of the historical testwork is focussed on oxide ores and was conducted by SGS Vostok in February 2015. The head grade of the sample tested, a single 350kg oxide sample, was 1.31% Cu and 28.2g/t Ag.

The main copper minerals are malachite and azurite, both oxide minerals, and diagnostic acid leaching indicated that 95.4% of the copper is present as these oxide minerals. Significant amounts of carbonate minerals in the ore results in very high acid consumptions. Chrysocolla and secondary sulphides were identified (bornite + chalcocite + covellite) with traces of sulphides (chalcopyrite + pyrite + acanthite) and native silver. Both copper and silver grades were distributed evenly through the size fractions at -1.7mm with most of the mass yield being in the size fraction -1.2+0.6mm.

Both gravity and flotation were shown to be non-viable processes for oxide ore treatment.

The only technically viable flowsheet tested for oxide ores was sequential sulphuric acid leaching followed by cyanide leaching of the acidic residue (with neutralisation and lime addition) at a grind size of 80% passing 74 microns. At a pH of 2.0 and a subsequent cyanide concentration of 0.2%, copper and silver recovery were 95.9% and 96.0% respectively, at acid and cyanide consumptions of 90.5 kg/t and 0.84 kg/t respectively. The high acid consumption is due to the carbonate minerals, accounting for 78% of the reported acid consumption. This high acid consumption would likely render this flowsheet option uneconomic. Table 13.1 summarizes the testwork carried out by SGS. These represent the best values and none of the testwork was optimized.

Table 13.1: Summary of Testwork Carried out by SGS and Reported in 2015

Test	Conditions	Recovery Cu	Recovery Ag	Comment
DIAGNOSTIC ACID LEACH	Solution in sulphuric acid	95.4	-	Separation of copper phases
FLOTATION	Open 2 stage, NaSH	40-43	66-70	Unsaleable concentrate <7% Cu
GRAVITY	10 kg charges in Knelson centrifuge	<2%	<3.5%	
WHOLE ORE LEACH	9 tests considering sulphuric acid consumption, grind size, pH, pulp density	95.8	-	High acid consumption at 90.5 kg/t due to carbonate content
WOL-CYANIDE	Cyanide bottle roll on WOL residue	-	96	CN consumption 0.84 kg/t
FLOTATION-HYDROMET	Bulk float followed by acid leaching and cyanidation	39	71	Manageable acid consumption but at expense of Cu and Ag recovery

13.3 Historical Testwork – VNIItsvetmet

Testwork was conducted on samples of oxide and sulphide ore in 2020. The total sample weight was approximately 730kg.

The oxide sample head grade was 0.56% Cu and 30.8 g/t Ag. The sulphide sample assayed 0.56% Cu and 39.0 g/t Ag. Diagnostic acid leaching indicated that 96.4% of the copper is acid soluble in the oxide sample; for the sulphide sample, 42.8% is in the form of secondary copper sulphides and 53.6% as primary sulphides.

Acid bottle roll testwork was conducted on three sizes of the oxide sample: -20mm, -10mm and -71 microns. Copper recovery was 73.6%, 80.6% and 96.4% respectively. Acid consumption was 46.6, 54.2 and 81.9 kg/t respectively.

A whole ore cyanidation bottle roll test was carried out at a grind size of 80% passing 71 microns using a 1% initial cyanide concentration. Copper and silver recovery was 55.6% and 96.7% respectively at a cyanide consumption of 40.4 kg/t.

Detailed flotation studies were carried out on the sulphide sample, including several Locked Cycle Tests (LCT). The results from LCT No 4 have been used for design purposes. The flowsheet tested in the LCT No 4 is shown in Figure 13.1.

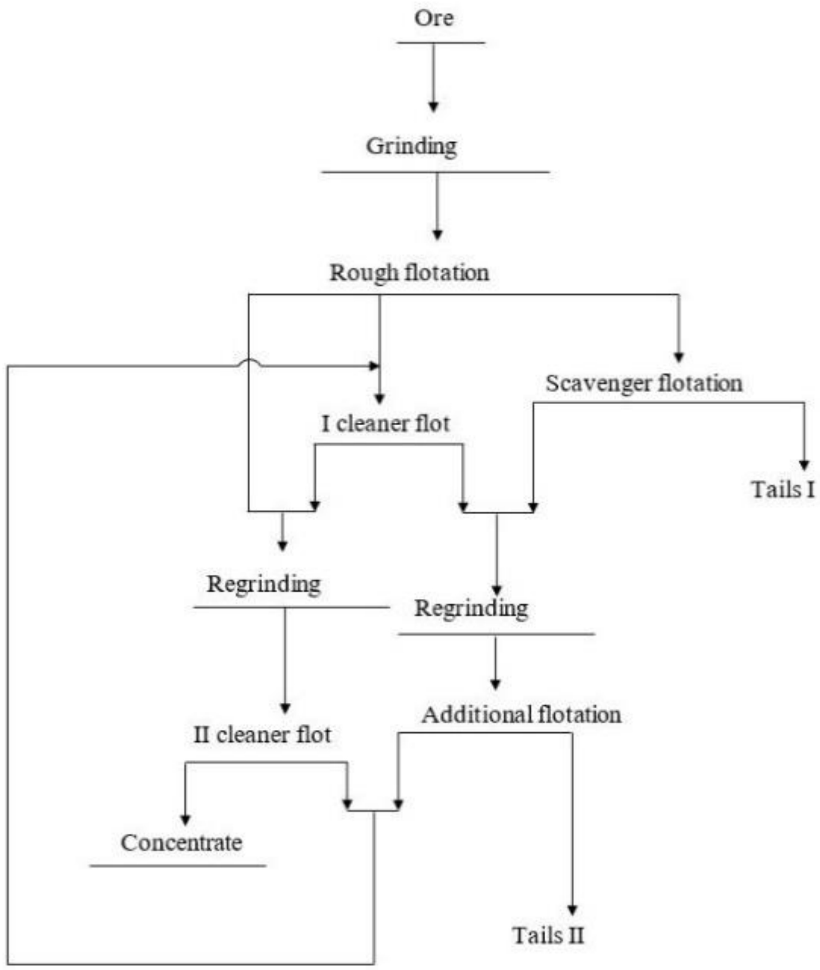


Figure 13.1: Flotation Testwork Flowsheet

The pH was in the range of 9-10 and Potassium Amyl Xanthate (PAX) was used as the collector with MIBC as the frother, both reagents added only to the rougher and scavenger stages. A total of 60 g/t PAX was used for the roughing and scavenger stages with 30g/t MIBC frother.

The results indicated 89.1% copper recovery to a concentrate grading 25.8% Cu. Silver recovery to the copper concentrate was 82.7% grading 1,634 g/t Ag.

A detailed analysis of the copper concentrate from this test is summarised below in Table 13.2.

From this analysis, there does not appear to be any significant penalty elements. However, the Sb content was not analysed, and this can be a potential significant penalty element.

Table 13.2: Summary of Properties of Copper Flotation Concentrate								
Component	Concentrate		Final tailings I		Tailings II		Head tailings (estimate)	
	Grade, %	Recovery, %	Grade, %	Recovery, %	Grade, %	Recovery, %	Grade, %	Recovery, %
Cu	25,8	89,1	0,053	8,4	0,14	2,5	0,56	100,0
Ag, g/t	1633,6	82,7	5,7	13,2	15,6	4,1	38,1	100,0
Au, g/t	0,87	26,0	0,044	60,1	0,09	13,9	0,06	100,0
Zn	0,035	1,5	0,045	89,9	0,038	8,6	0,04	100,0
Fe	17,7	13,9	2,00	71,8	3,5	14,2	2,45	100,0
Ni	0,0073	2,5	0,0053	84,0	0,0075	13,5	0,01	100,0
Co	0,0054	-	<0,005	-	<0,005	-	<0,005	100,0
Mo	0,004	7,2	0,0010	82,5	0,0011	10,3	0,00	100,0
Cd	<0,002	-	<0,002	-	<0,002	-	<0,002	100,0
Ca	1,22	0,8	2,87	89,6	2,71	9,6	2,82	100,0
Al	2,33	0,8	5,39	88,8	5,55	10,3	5,35	100,0
Mg	0,43	0,9	0,96	87,8	1,09	11,3	0,96	100,0
SiO_2	17,46	0,5	64,94	90,4	57,3	9,0	63,26	100,0
S	20,53	57,6	0,24	30,8	0,8	11,6	0,69	100,0
As	<0,03	-	<0,03	-	<0,03	-	<0,03	100,0
Re	<0,00045	-	<0,00045	-	<0,00045	-	<0,00045	100.0

This testwork programme demonstrated the technical viability of conventional flotation for the sulphide sample to produce a saleable copper concentrate with significant payable silver values at good recoveries.

For the oxide sample, acid leaching produces high copper recovery but at uneconomically high acid consumptions (due to the significant carbonate content in the ore). Cyanide leaching offers the best potential route for high silver recovery but lower copper recovery without additional processing. Unfortunately, the one cyanide test conducted indicated a very high cyanide consumption.

13.4 Current Testwork - VNIItsvetmet

Due to the limited testwork available to demonstrate cyanide leaching of the oxide samples and particularly at coarser crush sizes for potential heap leaching, an additional testwork programme was conducted for both fine and coarse ore BRTs using the same samples from the 2020 programme. The sample size for the fine ore tests was 80% passing 71 microns. Varying cyanide concentrations were tested. The fine ore results are summarised in the following Table 13.3.

Table 13.3: Cyanide Bottle-Roll Test Results for Fine Fractions

NaCN, %	Retention time, hr	Grade in cake		Recovery, %		NaCN consumption, kg/t ore
		Cu, %	Ag, g/t	Cu	Ag	
With carbon						
0.05	24	0.41	11.3	31.7	66.0	1.41
Without carbon						
0.05	24	0.41	13.6	31.7	56.4	1.32
0.1	48	0.29	8.6	49.5	72.7	4.60
0.5	48	0.26	1.5	54.5	95.2	6.97
1.0	48	0.24	1.3	58.0	95.9	9.41
1.0*	***48***	***0.25***	***1.0***	***55.6***	***96.7***	***40.40***

Oxide ore - 0.56% Cu, 30.9 g/t Ag
L:S=1.5:1
*As per 2020 test by VNIITSVETMET

In summary, the results show that recovery of both copper and silver increases to a concentration of 0.5% NaCN. At 1.0% NaCN, the recovery improvements are marginal. Cyanide consumption increases with the increased cyanide concentration. Of particular note is that the cyanide consumption is significantly less than the 2020 result, although the reason for this is unexplained. However, the cyanide consumptions reported above are consistent.

For scoping level purposes, copper and silver recoveries of 58.0% and 95.9% respectively are assumed for the fine grind size.

The lower copper recovery compared to that for acid leaching is indicative of the copper mineralogy, i.e., less cyanide-soluble copper than acid-soluble copper. Silver recoveries remain high but with high cyanide consumptions, i.e., 9.41 kg/t for the result at 1% NaCN concentration and which gave the highest metals recoveries.

For the coarse ore tests, three crush sizes were selected: -25mm, -12.5mm and -6.5mm. The tests were run for 21 days with two different cyanide concentrations and the results are summarised in Table 13.4.

The best metals recoveries were obtained at 0.2% NaCN concentration and at the finest crush size of -6.5mm. Recoveries and cyanide consumptions generally increase with the finer crush size as expected.

Table 13.4: Cyanide Leaching Results for Coarse Fractions

Fraction, mm	NaCN, %	Grade in cake		Recovery, %		NaCN consumption, kg/t ore
		Cu, %	Ag, g/t	Cu	Ag	
25.0	0.05	0.35	12.9	39.9	59.9	4.30
12.5		0.34	11.8	41.7	63.3	4.36
6.5		0.31	10.0	46.8	68.9	4.51
25.0	0.2	0.35	14.0	40.0	56.5	5.58
12.5		0.27	10.5	53.7	67.4	6.08
6.5		0.24	7.3	58.8	77.3	8.43
Oxide ore - 0.56% Cu, 30.9 g/t Ag L:S=1.5:1 Retention time 21 days						

For scoping level purposes, copper and silver recoveries of 58.8% and 77.3% respectively are assumed for the coarse crush size of -6.5mm.

13.5 Limitations

Reliance and emphasis on the 2015 testwork results which were not optimized and not necessarily representative of the major ore zones/types is acceptable for the current level of study but will need to be reassessed as part of a programme at the level of a Definition Phase Study.

13.6 Opinion on Data Adequacy

It is WAI's opinion that the metallurgical testwork provides a scoping level of accuracy for the basis of developing the process flowsheet and 'reglament'.

14 MINERAL RESOURCE ESTIMATES

The mineral resource estimate is reported from a block model of Cu and Ag grades, prepared by SRK, using Leapfrog software (Geo and EDGE). The input data, and methods, assumptions and parameters used to prepare the block model are summarised below.

14.1 Exploration database

The estimate is based on 10,388m of diamond core drilling (from 33 drill holes) and 253m of channel sampling (from four trenches and two outcrops), completed during AZR campaigns of 2016-2017 and 2019-2020. Historical sampling does not directly inform the estimation, although the conceptual framework for preparing the estimation, including assumptions about the orientation and continuity of mineralisation, is influenced by consideration of historical data.

Location and intersection information for channel samples and drill holes is tabulated in Sections 9 and 10 of this report.

14.2 Geological Model

The components of the wireframed geology model are discussed below.

14.2.1 Topography

The topographic surface was created from publicly available radar data downloaded from NASA website.

14.2.2 Base of moraine

From the logging, AZR prepared a wireframe surface of the base of the moraine overburden. SRK reviewed and accepted this surface and used this contact to code the block model and as a limit on the extent of mineralisation. Above the mineralisation, the thickness of the moraine overburden is generally in the range 0 to 100m.

14.2.3 Base of oxidation

From logging and assays (ratio of oxide Cu to total Cu), AZR prepared a wireframe surface of the contact between oxide and sulphide mineralisation. SRK reviewed and accepted this surface and used this contact to code the block model. The depth of oxidation varies but is generally down to 200m below the topographic surface.

14.2.4 Post-mineralisation fault

A sub-vertical, northeast-southwest striking fault is interpreted from the magnetic survey information, and this structure coincides with a discordance between the projected positions of the North and South mineralised zones. In plan view, the apparent offset of mineralisation is dextral and approximately 30m.

14.2.5 Mineralisation domains

North of the fault, the mineralisation domains were defined using a 0.2% Cu threshold to represent the core of the mineralisation, and a 0.1% Cu threshold to define the halo of lower grade mineralisation around this core (Figure 14.1). These thresholds were set after testing various alternatives, and chosen (with consideration of the likely range of open pit and underground cut-offs that would be applied to the estimate), as the best compromise between:

1. Maintaining continuity – at higher thresholds, fewer drill holes have an above-threshold intersection, and the interpretation of the mineralised structure breaks up.
2. Avoiding capturing too much weakly mineralised material in the core mineralisation domain.

The core of mineralisation was modelled in three zones (see Figure 1.1 and Figure 10.1):

- A North domain, based on 22 intersections;
- Parallel to the main part of the North domain, a less extensive parallel zone (possibly a splay, and separated by a few metres) referred to as the West domain, based on two (2) intersections; and
- A South domain, based on six (6) intersections.

A threshold of 0.2% Cu was used for defining the North and West domains, and a higher threshold (0.35% Cu) was used for defining the South domain.

Two halo zones, at 0.1% Cu threshold, were modelled around the combined North core domains and the West core domain.

A third standalone 0.1% Cu threshold domain was modelled around a second and separate West domain (all assays for this domain were in the range 0.1 to 0.2% Cu).

South of the fault (Figure 14.2), the higher threshold (0.35% Cu) for the core mineralisation was chosen as a better compromise between maintaining continuity and excluding weaking mineralised material, than the 0.2% Cu threshold used North of the fault. A 0.1% Cu halo domain was also modelled around the South core mineralisation.

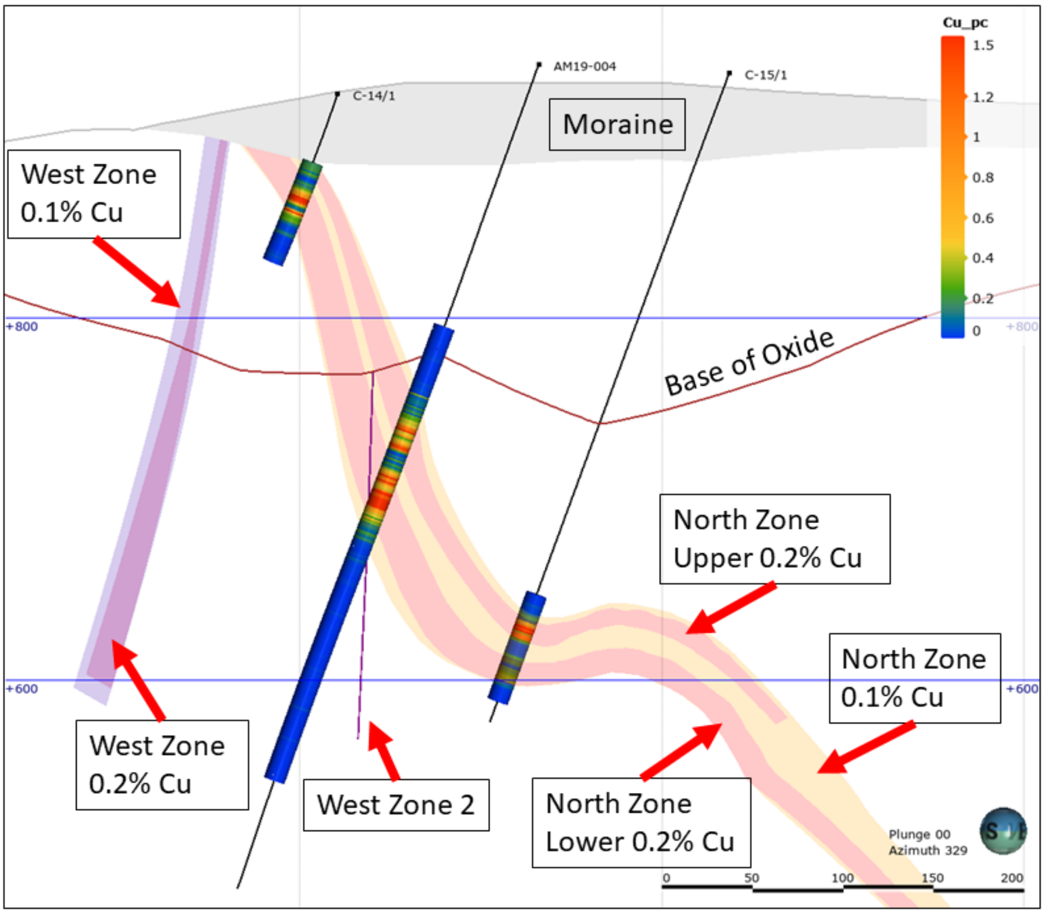


Figure 14.1: Example Section (Corresponding to Section 1 on Plan in Figure 10.1) Showing Northern Mineralisation Domains

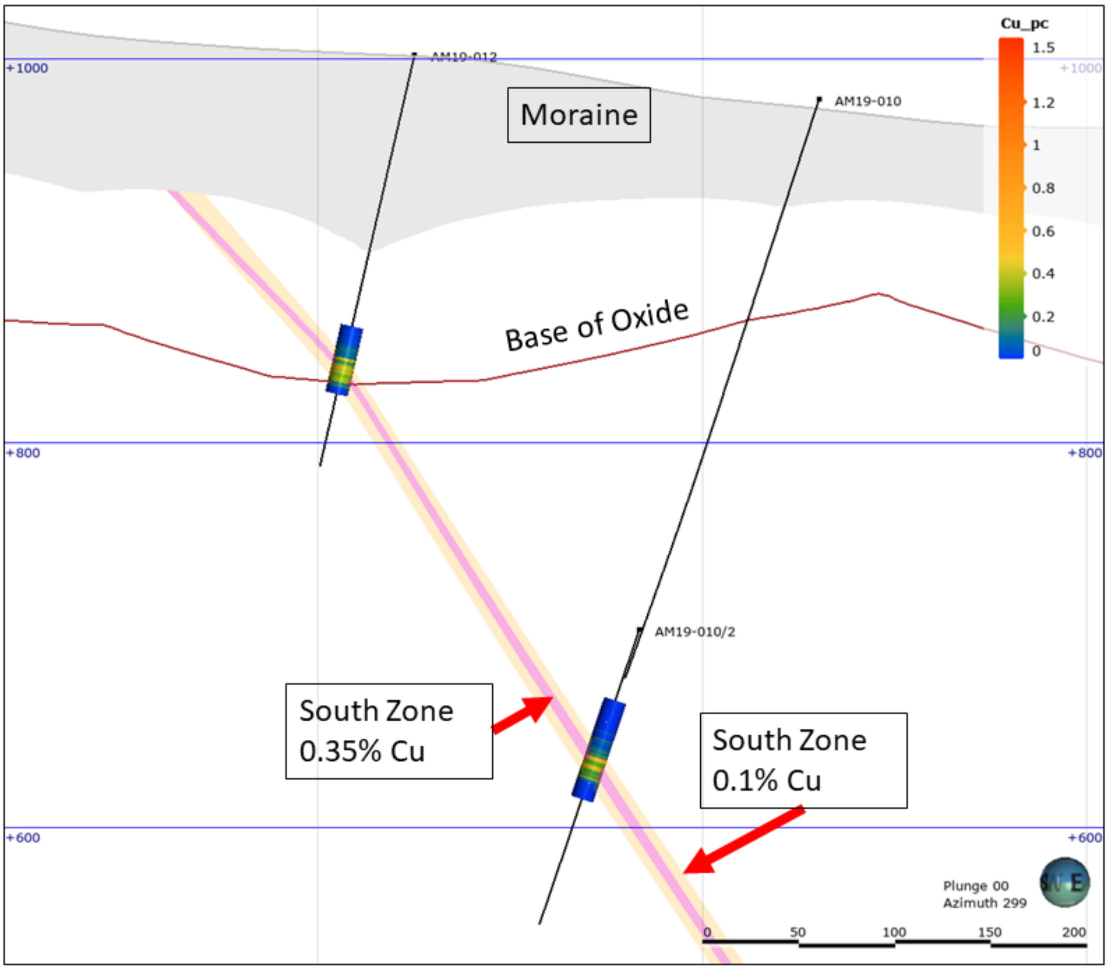


Figure 14.2: Example Section (Corresponding to Section 4 on Plan in Figure 10.1) Showing Southern Mineralisation Domains

14.3 Geostatistical Grade Estimation

Statistical analysis, variogram modelling and grade estimation were done using Leapfrog Geo and EDGE software.

14.3.1 Estimation Method

The domains listed in the previous section were used as hard boundaries to constrain estimation (i.e., block grade estimates within a domain would only be influenced by composites from the same domain). Copper and silver grades within the four main mineralised domains were estimated by 2D Ordinary Kriging. The 2D approach was chosen because of the relatively low thickness of the mineralised zones (average true thicknesses for North Lower, North Upper, West, and South domains are, respectively: 8.9m, 12.6m, 3.4m and 3.6m) compared to the spacing between intersections (300m or more between drilling sections, and 100m or more down dip), and compared to the nominal 1m sampling length.

For 2D estimation, one composite is generated per intersection. Grade is not estimated directly from these composites (because the varying lengths of the composites mean that direct grade estimates would not comply well with geostatistical assumptions the kriging estimation method is built on), but the full intersection composites are suitable inputs for estimating metal accumulation (product of grade and true thickness) and true thickness. Block model grade estimates can then be obtained by dividing each block estimate for metal accumulation by the block estimate for true thickness. The advantages the 2D approach has (compared to 3D estimation using fixed-length composites) is that decisions regarding composite length, block size, search neighbourhood, and estimation anisotropy are simplified.

For the domains of halo mineralisation (>0.1% Cu threshold), surrounding the four main mineralised domains, the grades of Cu and Ag were estimated by Inverse Distance Squared estimation, from 1m composites. Although the Cu and Ag grade estimates are constrained by the same Cu grade shells, within these shells the two elements are estimated independently of each other.

14.3.2 Capping

Summary statistics of the raw sample Cu and Ag grades for all mineralised domains are presented in Table 14.1 and Table 14.2. For the four main mineralised domains (>0.2% Cu shell in the North and West, and >0.35% Cu shell in the south) histograms are shown in Figure 14.3. Based on a review of the high-grade tails of the copper and silver grade distributions, and assessment of how the highest grades were distributed spatially, generally no grade capping was applied to either the samples or the composites. The highest grades of Cu and Ag tend to be clustered together, and the highest Cu and Ag grades also tend to correlate reasonably well with each other (Figure 14.4). The only exception to the absence of capping was the low-grade halo domain for the North, where one 1m composite Ag grade (289g/t) was capped at 50g/t.

14.3.3 Estimation parameters

For the 2D kriging, the same variogram model and search neighbourhood was applied to the estimates of Cu metal accumulation, Ag metal accumulation, and true thickness, for all four domains. The search ellipse radii were 1,500m x 600m. The maximum number of full intersection composites per estimate was four, and all blocks were estimates with at least two composites. For all domains, the ellipse was oriented to have a moderate (30°) plunge to the north. The variogram model had a nugget proportion of 20%, a single structure with the remaining 80% of variance, and anisotropy ellipse ranges of 1,000m and 400m, also oriented to plunge 30° to north.

For the 3D estimations of Cu and Ag halo mineralisation, the ranges of the search ellipsoid were set at 1,500m x 600m x 300m (oriented to follow the average plane of mineralisation for each domain, and plunge 30° to north), and the maximum number of 1m composites per estimate was set at 12.

Table 14.3 lists the dimensions of the block model used to store the Cu and Ag estimates.

Table 14.1: Summary Statistics for Sample Cu Grades

Mineralised Domain	Total Length (m)	Mean (% Cu)	Standard deviation (% Cu)	Coefficient of variation	Variance (% Cu)^2	Minimum (% Cu)	Median (% Cu)	Maximum (% Cu)
South 0.1 to 0.35% Cu	72.0	0.19	0.10	0.55	0.01	0.04	0.17	0.45
South >0.35% Cu	26.0	0.59	0.31	0.52	0.09	0.24	0.52	1.68
North 0.1 to 0.2% Cu	153.5	0.11	0.07	0.64	0.00	0.00	0.11	0.49
North >0.2% Cu lower	247.9	0.67	0.65	0.97	0.43	0.01	0.38	3.71
North >0.2% Cu upper	91.0	0.65	0.60	0.92	0.36	0.06	0.43	3.24
West 0.1 to 0.2% Cu	30.0	0.11	0.05	0.41	0.00	0.03	0.11	0.20
West >0.2% Cu	37.0	0.50	0.60	1.19	0.36	0.00	0.34	2.46
West >0.1% Cu (Zone 2)	5.0	0.14	0.03	0.23	0.00	0.10	0.14	0.18

Table 14.2: Summary Statistics for Sample Ag Grades

Mineralised Domain	Total Length (m)	Mean (g/t Ag)	Standard deviation (g/t Ag)	Coefficient of variation	Variance (g/t Ag)^2	Minimum (g/t Ag)	Median (g/t Ag)	Maximum (g/t Ag)
South 0.1 to 0.35% Cu	72.0	7.0	7.0	1.00	49.1	0.4	3.8	32.3
South >0.35% Cu	26.0	17.5	11.6	0.67	135.2	2.2	15.6	47.1
North 0.1 to 0.2% Cu	153.5	6.6	23.6	3.57	554.9	0.4	2.5	289.0
North >0.2% Cu lower	247.9	47.2	71.3	1.51	5087.4	0.4	15.0	366.0
North >0.2% Cu upper	91.0	47.5	70.7	1.49	4998.4	0.8	20.6	356.0
West 0.1 to 0.2% Cu	30.0	14.6	9.7	0.66	94.1	3.1	11.2	45.5
West >0.2% Cu	37.0	42.3	55.0	1.30	3023.5	0.6	17.8	202.0
West >0.1% Cu (Zone 2)	5.0	5.6	1.7	0.30	2.9	3.3	5.3	7.8

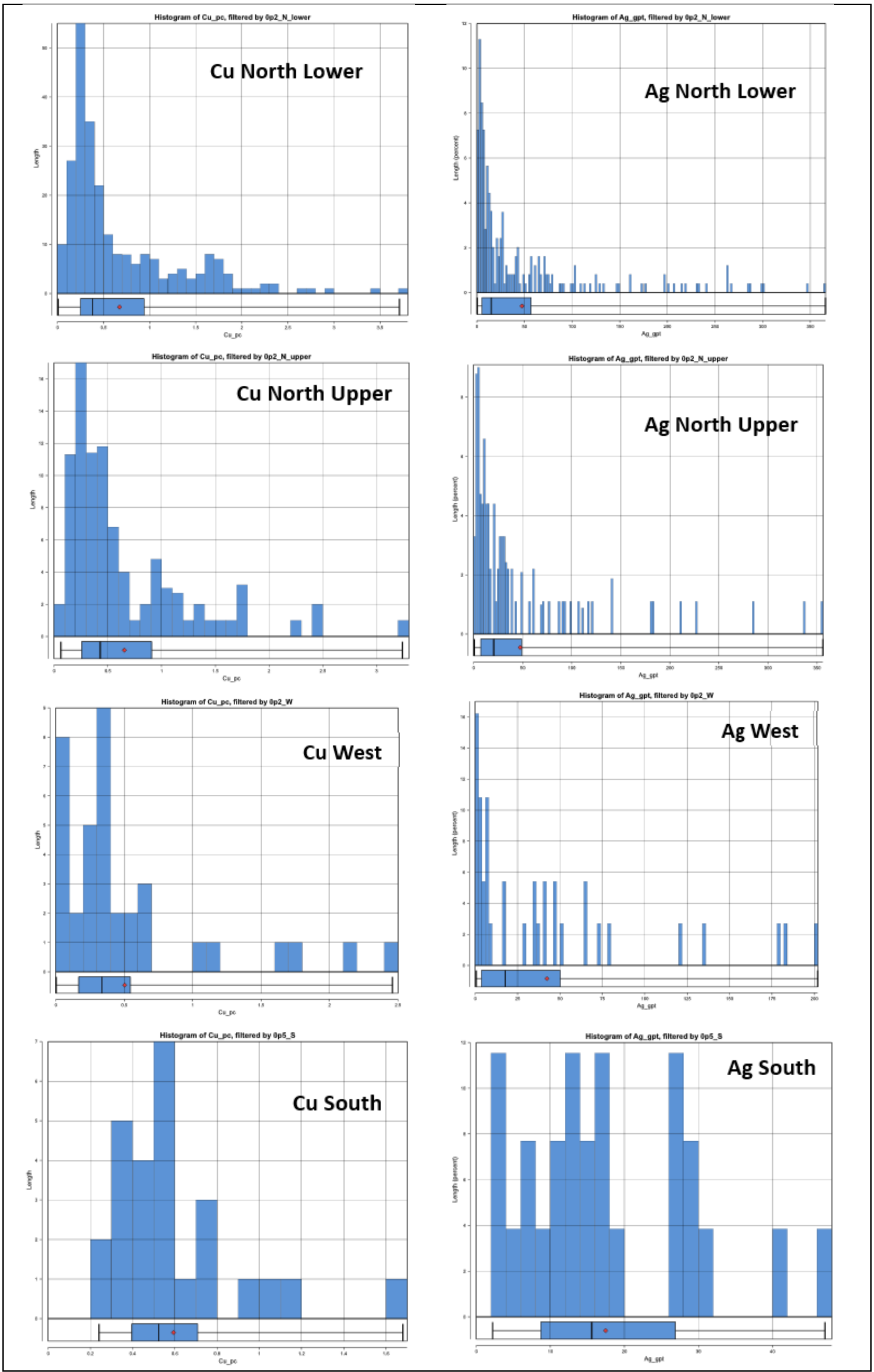
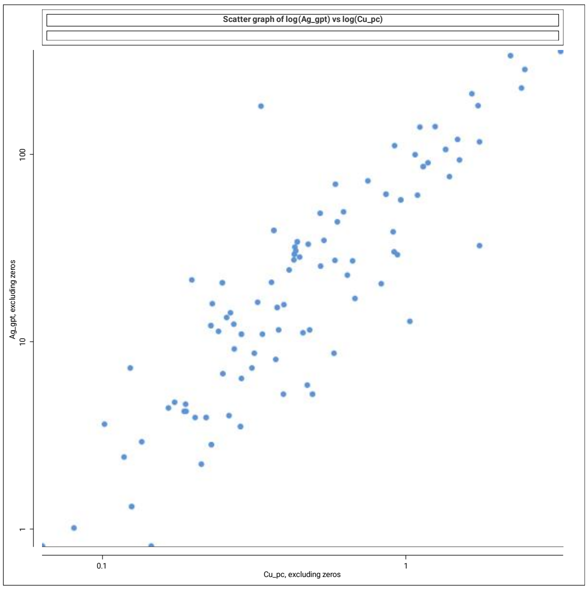
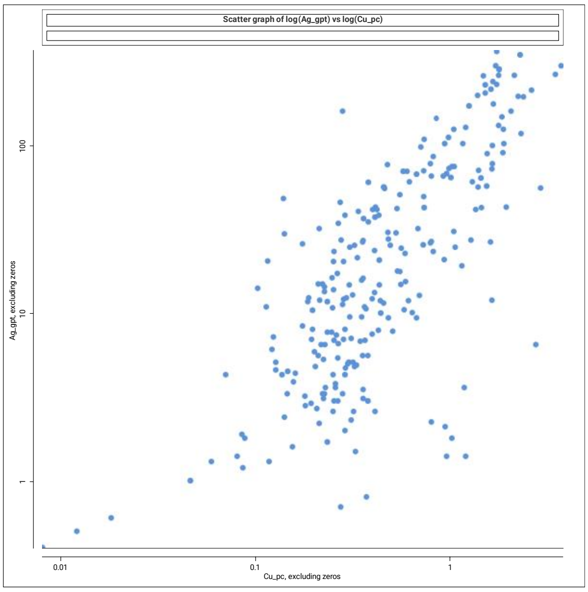


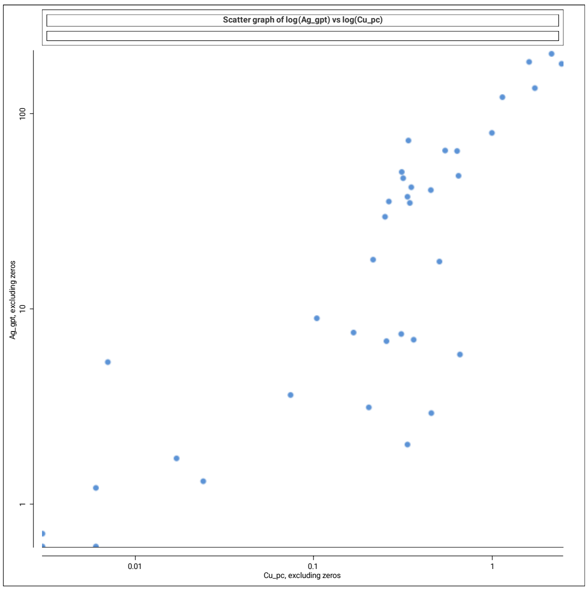
Figure 14.3: Histograms of Raw Sample Grades in Main Mineralised Domains



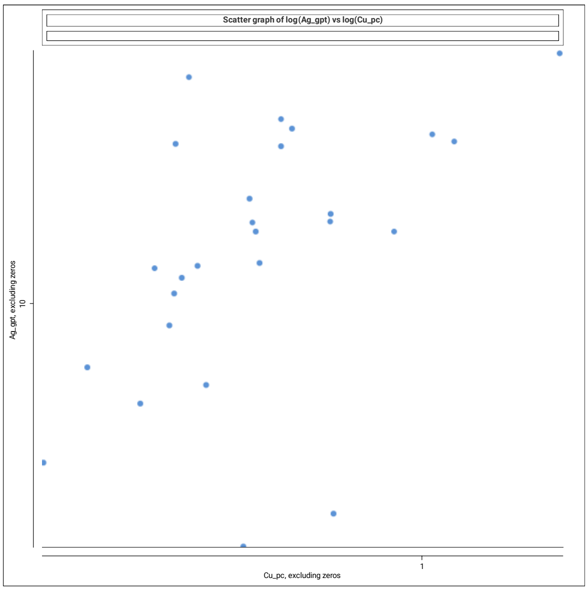
North Lower



Noth Upper



West



South

Figure 14.4: Scatterplots (log scale) of Ag Versus Cu Grade

Table 14.3: Block Model Dimensions			
	X	Y	Z
Minimum	600200	6305300	265
Maximum	605900	6311350	1190
Estimation block size (m)	10	50	25
Sub-blocking (m)	Variable to fit wireframe	5	5
Discretisation	1	8	8

14.4 Density

During the 2019-2020 campaign, AZR carried out over 500 hydrostatic density measurements on core. This information supersedes density measurements from earlier campaigns. After excluding some outlier values (<2 and >3.4), average values were calculated, for the oxide and sulphide domains, and inside and outside the >0.1% Cu envelope. From these sample averages, the following density factors were used to convert volumes in the block model to dry bulk tonnages:

- Oxide mineralised: 2.60 (59 samples)
- Oxide waste: 2.58 (76 samples)
- Sulphide mineralised: 2.67 (90 samples)
- Sulphide waste: 2.68 (270 samples)
- Moraine overburden: 2.00 (assumed)

14.5 Validation

The estimate was validated by visual and statistical checks of the block model against the sampling information, and against the composite and wireframe files created during the modelling process. Long sections of composite and block grades from of the two largest mineralised domains are shown in Figure 14.5.

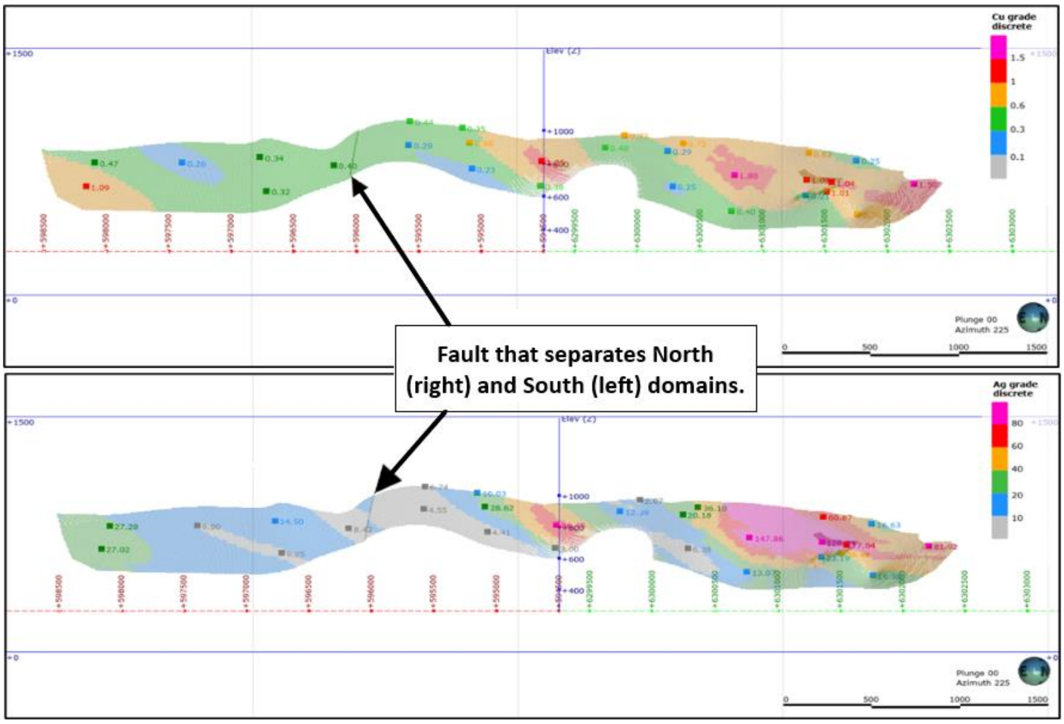


Figure 14.5: Long Section (view towards SW), Showing Full Intersection Composite Grades and Block Model Grade Estimates for Cu (top) and Ag (bottom)

[For clarity, only the main mineralised domains North Lower (>0.2% Cu shell) and South (>0.35% Cu shell) are displayed]

14.6 Classification

Block model quantities and grade estimates for Unkur were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). The portion of the mineralisation model that met the CIM definition of a mineral resource ("…reasonable prospects for eventual economic extraction") was established by SRK and reviewed by WAI experts, Mr Alan Clarke.

All Unkur mineral resources were classified as Inferred, based on the intersection spacing relative to the interpreted continuity, and potential complexity, of mineralisation and geology.

14.7 WAI Mineral Resource Review

14.7.1 Scope of Work

AZR commissioned WAI to review data pertaining to the Unkur project, in particular block models generated by Tetra Tech in 2018 and SRK in 2020.

The 2018 Tetra Tech model formed the basis of a Mineral Resource estimate reported as 62Mt at 0.53% Cu and 38.6g/t Ag (0.9% Cu equivalent). Tetra Tech classified this Mineral Resource as inferred as defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves (CIM 2014). The Tetra Tech Mineral Resource estimate was reported to a cut-off grade of 0.3% Cu equivalent, but no other economic or technical parameters were used to limit Mineral Resources.

The 2020 SRK model provided the basis of an Inferred Mineral Resource estimate totalling approximately 44Mt at 0.65% Cu and 44g/t Ag (1.04% Cu equivalent). Reporting from the SRK model was limited by a cut-off grade of 0.18% Cu equivalent inside an optimised open pit shell generated by suitable economic and technical parameters and by a cut-off grade of 0.57% Cu equivalent outside and below this pit shell to account for the possibility of underground extraction.

Following the data review, WAI was required to give an opinion on the models provided and the potential scope for the Inferred Mineral Resource to be increased as a basis for a Preliminary Economic Assessment ("PEA") of the Unkur project. To complete this work, WAI:

- Reviewed the background to the Resource Estimates as a high-level gap analysis to identify any potential issue that would preclude the reporting of Mineral Resources at an Inferred level;
- Reviewed the Mineral Resource models against the geological model and exploration data used to generate those models; and
- Carried out additional open pit optimisations and reporting of potential Mineral Resources at varying metal prices to determine if reported Mineral Resources are sensitive to these changes and to what degree.

14.7.2 High Level Gap Analysis

WAI reviewed the 2018 Tetra Tech Mineral Resource estimate report in a high-level gap analysis. The purpose of this review was to provide comfort that the available background and exploration data was to a standard to allow reporting of Mineral Resources.

As a result of this review, WAI concludes that:

- An appropriate exploration license covers the area of the Mineral Resource estimate;
- The geological setting that acts as the basis of the Mineral Resource models is well understood and the exploration programmes used as the basis for the Mineral Resource estimate have been completed in a manner so as to explore along the projected strike of the mineralised zones, intersecting these zones roughly perpendicular to their dip;
- Sample data acting as the basis of the Mineral Resource estimate was collected from diamond drilling completed by AZR;
- Sample selection, preparation and analysis was carried out following international best practice. Analysis of primary samples was carried out by SGS in Chita; and
- QA/QC procedures included the insertion of certified reference materials and the analysis of check assays at an umpire laboratory (ALS Chita). Results of the QA/QC sample analysis were generally good.

In summary, based upon the available data, WAI is of the opinion that the data used for the generation of the Mineral Resource has been collected in a suitable and robust manner with appropriate quality control measures in place. Analysis for copper and silver has been carried out at an internationally accredited laboratory. WAI agree that sufficient confidence can be placed in the exploration database for the reporting of Mineral Resources.

14.7.3 Model Review

WAI reviewed the Tetra Tech and SRK models alongside the associated mineralisation and geological wireframes and input exploration data used as the basis for the Mineral Resource estimates with the following conclusions:

- Given the number of drillholes available and the data spacing at Unkur, WAI believes that the interpretation of the mineralised domains by both SRK and Tetra Tech is generally reasonable, but with two points to note:
    - Tetra Tech have wireframed a small domain in the northern part of the project area that is known to be modelled dipping in the wrong orientation (towards the east as opposed to the west); the SRK model has this in the accepted orientation.
    - SRK have omitted a small domain located in the footwall of the main mineralised zone in the centre of the project area that Tetra Tech included. This additional zone is only intersected by one drillhole and a surface trench, but may add a small amount of tonnes to the Mineral Resource if included. However, WAI believes that even in the

Tetra Tech model, this zone is modelled quite optimistically in terms of strike length and believes that the strike direction is not correct as it does not follow the trend indicated by the intersection identified in the surface trench.

- Extrapolation of mineralised zones down dip and along strike is generally reasonable in the northern part of the project area. The interpretation of the most northerly part of the main zone of mineralisation is based upon a single drillhole some distance from the last drilling profile, but little mineralisation of interest is estimated in this area.
- Interpretation of the mineralised domain in the southern part of the project area is based upon wide spaced profiles. However, drillholes in this area seem to show a reasonably consistent dip and strike of this domain and WAI believes these would support an Inferred Mineral Resource classification.
- The oxide and sulphide domain wireframes appear reasonable interpretations, but the surface representing the base of overburden has a tendency to “balloon” in places between drillholes. This is a common problem in automated wireframing with sparse data where wireframes may be accurate at drillhole locations but bulge out between these known data points. Whilst this surface may benefit from reinterpretation at a later date, it is unlikely this issue has a major effect on the Mineral Resource estimate.
- The primary estimation methodology of Inverse Distance Weighting is appropriate given the limitation in data. The models are seen to validate reasonably well against the input exploration data.
- Density values for tonnage estimation are based upon a small data set. In WAI’s opinion there is no conclusive evidence in the results to act as a basis for developing a regression equation for density based upon copper content. Density values used in the block models appear reasonable.
- Classification limited to Inferred as appropriate (for those areas deemed to have reasonable expectations of eventual economic extraction) given the drillhole spacing at Unkur.

Further work by WAI concentrated on the SRK model as this was known to better match the dip and strike of mineralised domains in the northern part of the project area. In WAI’s opinion, the 2020 SRK model is generally a good representation of the current interpretation of the Unkur project and suitable as a basis for reporting an inferred Mineral Resource.

14.8 Mineral Resource Estimate

14.8.1 Introduction

Having determined that the exploration data and SRK model based on that data was suitable for the reporting of an Inferred Mineral Resource, WAI carried out a review of the optimisation parameters used for limiting the reported Mineral Resource based on the idea that a Mineral Resource should have reasonable expectations of eventual economic extraction.

14.8.2 Optimisation Parameters

The (open pit) optimisation parameters used by WAI to constrain the resource model for are presented in Table 14.4.

Table 14.4: Optimisation Parameters Used by WAI

	Unit	Parameters
Slope (Overburden)	Degrees	30
Slope (Rock)	Degrees	45
Mining Dilution	%	5
Mining Recovery	%	95
Recovery Cu (oxide)	%	96.4
Recovery Cu (sulphide)	%	89.1
Recovery Ag (oxide)	%	96.7
Recovery Ag (sulphide)	%	82.7
Mining cost (waste)	$/t	1.2
Mining cost (mineralised)	$/t	1.5
Inc mining cost (per m)	$/m	0.005
Processing	$/t	10
G&A	$/t	2
Royalty (Cu)	%	8
Royalty (Ag)	%	6.5
Copper Price	$/lb	3.86
	$/tonne	8,500
Silver Price	$/oz	25
	$/g	0.80

14.8.3 Copper Equivalent Equation

The optimisation parameter necessitated a copper equivalent equation and marginal open pit and underground cut-offs to be used for reporting of results. Differing metal recoveries for oxide and sulphide material required separate equivalent calculations and cut-off grades for each material type. The parameters used in these equations are shown in Table 14.5.

Table 14.5: Basis of Cut Off and Cu Equivalent

Underground			**Open Pit**		
Cu price	USD/lb	3.86	Cu price	USD/lb	3.86
lbs per kilo	lbs	2.2	lbs per kilo	lbs	2.2
recovery	fraction	0.97	recovery	fraction	0.89
dilution (zero grade)	fraction	0.1	dilution (zero grade)	fraction	0.05
royalty	fraction	0.08	royalty	fraction	0.08
processing	USD/t	10	processing	USD/t	10
UG mining	USD/t	25	mining	USD/t	0
GA	USD/t	2	GA	USD/t	2
Cost per tonne	USD/t	37	Cost per tonne	USD/t	12
Revenue per tonne at1% Cu eq	USD/t	69.04	Revenue per tonne at1% Cu eq	USD/t	66.36
Marginal cut-off	**Cu_eq %**	**0.54**	**Marginal cut-off**	**Cu_eq %**	**0.18**
Cu Equivalent - Oxide			**Cu Equivalent – Sulphide**		
	Cu	Ag		Cu	Ag
Price	3.86	25	Price	3.86	25
Price Unit	lb	oz	Price Unit	lb	oz
gram per price unit	453.59	31.1	gram per price unit	453.59	31.1
price/g	0.01	0.8	price/g	0.01	0.8
Recovery	0.96	0.97	Recovery	0.89	0.83
Royalty	0.08	0.07	Royalty	0.08	0.07
Grade Unit	%	ppm	Grade Unit	%	ppm
Revenue at 1 grade unit per tonne	75.16	0.73	Revenue at 1 grade unit per tonne	69.68	0.62
Multiplying Factor for Ag		**0.0097**	**Multiplying Factor for Ag**		**0.009**

14.8.4 Mineral Resource Reporting

The mineral resource estimate for the Unkur project is presented in Table 14.6. These mineral resources have been estimated in conformity with generally accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines and reported in accordance with the Canadian Securities Administrators' National Instrument 43-101.

Reasonable prospects for eventual economic extraction are supported through open pit optimisation using metal prices of US$8,500/t Cu and US$25/oz Ag, with underground resources based on cut-off grade of 0.54% CuEq.

Table 14.6: Mineral Resource Estimate for the Unkur Project

Classification	Method	Type	COG (CuEq %)	Tonnes (Mt)	Cu (%)	Ag (g/t)	CuEq (%)
Inferred	OP	Oxide	0.18	15.7	0.61	45	1.05
		Sulphide		17.1	0.59	49	1.03
	UG North	Oxide	0.54	0.4	0.51	23	0.73
		Sulphide		14.2	0.55	30	0.83
	UG South	Oxide		-	-	-	-
		Sulphide		3.7	0.64	20	0.82
	TOTAL	**OXIDE**		**16.1**	**0.61**	**44**	**1.04**
	TOTAL	**SULPHIDE**		**35.0**	**0.58**	**38**	**0.93**
	TOTAL	**ALL**		**51.1**	**0.59**	**40**	**0.96**

Notes:

1. Figures have been rounded to reflect this is an estimate.
2. Inferred Mineral Resources have been reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definition standards for Mineral Resources and Reserves and have been completed in accordance with the Standards of Disclosure for Mineral Projects as defined by National Instrument 43-101.
3. No Measured or Indicated Resources have been estimated.
4. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
5. Mineral Resources are based on a CuEq grade of 0.18% for the Open Pit resources and of 0.54% for the Underground resources using metal prices of US$3.86/lb Cu and US$25/oz Ag, the equivalence formula for Oxide is CuEq = Cu + (0.0097 x Ag) and for Sulphide is CuEq = Cu + (0.009 x Ag).
6. The Mineral Resource is effective as of 31st July 2021.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. Mineral Resources may be subject to legal, political, environmental and other risks and uncertainties.

14.9 Summary and Conclusions

The potential tonnages as stated above are lower than those declared in the 2018 Tetra Tech Mineral Resource estimate (62 Mt at 0.53% Cu and 39g/t Ag for 0.9% Cu equivalent). However, in WAI's opinion, the 2018 Tetra Tech Mineral Resource estimate overstates Mineral Resource tonnage as no limitation to report only areas with reasonable prospects for eventual economic extraction has been carried out.

If the SRK model (all domains) were to be reported to a similar single cut-off grade to the Tetra Tech Mineral Resource estimate of 0.3% Cu equivalent (i.e., without using open pit optimisation and a suitable underground cut-off grade for reporting) the SRK model would report similar to the Tetra Tech model at approximately 67 Mt at 0.54% Cu and 33g/t Ag (0.85% Cu Equivalent).

WAI believes the SRK model is generally a good representation of the mineralisation at Unkur given the limited amount of data available and with appropriate economic and technical parameters applied, an inferred Mineral Resource estimate could be declared based upon this model.

WAI concludes that any significant increase in reported Mineral Resources without additional exploration would be limited to refining optimisation parameters rather than by carrying out any reinterpretation or re-estimation. Increasing metal prices and refining metal recovery factors during optimisation of the block model, alongside associated changes in copper equivalent calculation and marginal cut-off grade for reporting, indicates that a slight increase in reported Mineral Resources would be possible.

15 MINERAL RESERVE ESTIMATES

Estimation of mineral reserves has not formed part of this study and are not reported here.

It should be noted that 'minable tonnage estimates' are not Ore Reserves and are not demonstrative of technical and economic viability.

The use of 'minable tonnage estimate' or minable inventory and its relationship to Mineral Resource Estimates is discussed further in Section 16.1 and 16.5.

16 MINING METHODS

16.1 Introduction

WAI has carried out a scoping level mining study to define a mineable tonnage estimate for the Unkur deposit. Two scenarios will be considered for optimisation and economic analysis:

- Underground transitions from the oxide only open pit; and
- Underground transitions from the ultimate sulphide + oxide open pits (different processing scenarios applied).

WAI completed a mining study to define an underground mineable tonnage estimate for Unkur below the open pit battery. This study considered the volume of mineralised material below the best-case open pit shell (after 40m crown pillar applied) based on mineable tonnage estimate (different cut-off grade numbers were investigated). The final result reflects the assessment of supporting development/infrastructure and constitutes only a high-level conceptual design given that 'minable tonnage estimates' are not Ore Reserves and are not demonstrative of technical and economic viability.

16.2 Hydrology and Hydrogeology

16.2.1 Introduction

The hydrological and hydrogeological review of the Unkur deposit is primarily based on Tetra Tech Preliminary Economic Assessment[4] (2018) and Tetra Tech Mineral Resource Estimate[5] (2018). In addition, the Unkur 2016 baseline report has also been reviewed[6]. The hydrological and hydrogeological setting is supplemented by SRK Consulting Report[7] on the Udokan deposit located 25km south-west of Unkur.

The Project licence area of the Unkur deposit (hereafter referred to as the Site) is located in the Kalarsky district of the Zabaikalsky administrative region of Russia, 15km east of the Novaya Chara town.

The Unkur deposit is a sediment-hosted stratiform copper deposit situated on the northern slopes of the Udokan Range. Mineralisation is reported to be confined to the south-western limb of the Unkur syncline.

[4] Tetra Tech. 2018. Technical Report and preliminary economic assessment for the Unkur Copper-Silver Project, Kodar-Udokan, Russian Federation
[5] Tetra Tech. 2018. Technical Report and Mineral Resource Estimate for the Unkur Copper-Silver Project, Kodar-Udokan, Russian Federation
[6] Geoexpert LLC. 2016. Information Report 2016 on the Program of Creation and Management of MMTPI and Production economic control when carrying out search and evaluation works for copper, silver and associated components at Unkursk ore mining
[7] SRK Consulting. 2010. Preliminary Environmental and social assessment of Udokan Deposit Project (2010)

The Site is located in a mountainous area within the Chara River catchment characterised by relatively gentle sloping terrain with rocky outcrops and taiga vegetation (swampy coniferous forest) comprising of heathland and peat bogs.

The climate at Site is described as subarctic, with long, cold winters and warm, mild summers with average daily temperatures ranging from -32°C in the winter to 16.5°C in the summer. Typically snow falls in mid-September and begins melting in mid-April. Snow thaw from mid-April onwards could lead to significant river flows within the catchment with the highest flow rates anticipated in June.

16.2.2 Hydrogeology

16.2.2.1 *General*

The Site is located within the catchments of the River Kemen, which flows through the Site, and River Unkur which are sub catchments of the Chara River catchment (674km^2). Rivers within the Chara catchment flow all year round including below ice cover during the winter months.

Surface water sampling was carried out in 2016 within the Site. The results exhibit typical water quality traits for mid-mountain, sparsely populated areas: low-mineralisation, soft water and low-radioactive.

Glacial sediments, in the form of moraines, cover most of the Site area. The average thickness of the moraine cover is 40m; however, this cover increases to 180m to 200m thickness in the northwest and southeast of the Site. Alluvial sediments within the vicinity of the Unkur and Kemen rivers comprise of gravel and sandy soil which form 5m to 20m high terraces above floodplains. No groundwater levels or quality is available for the superficial glacial deposits.

Bedrock is reported to be confined with phreatic surfaces at 140m to 110mbgl. The hydraulic conductivity in the confined aquifer is controlled by fractures (such as the Kemensky fault) and is highly variable (0.01 to 22 m/d). However, the presence of permafrost across the Site and surrounding area drastically reduces hydraulic conductivity and acts as an aquitard. Due to the presence of permafrost, groundwater quality samples could not be obtained within the Site however, a groundwater sample was obtained from a 1,000m deep well at Luktur thermal spring within the Chara rift depression 7 km west of the Site. The water quality of the thermal spring exhibited alkaline condition (pH = 8.7), mineralized (0.63 g/l), hard (9.10) with a geochemically increased content of dissolved uranium (90 ng/l). Other trace elements were below detection limits except sulphur.

Groundwater data available from the Udokan deposit exhibits hydro-carbonate calcium and calcium-sodium water, mineralization is 0.05-0.1 g/l and general hardness is 0.27-2.8 eq/l.

16.2.2.2 *Permafrost Conditions*

The Permafrost is widespread and continuous across the area with thickness typically ranging between 200m and 400m. WAI has reviewed the basal surface generated by electrical resistivity data which provides a good tool to map the extent of the base of the permafrost across the deposit. The thickness of the permafrost depends on location (i.e. valleys or mountain ridges), ground type, gradient of terrain and slope direction. The seasonal thaw of the upper permafrost layer is reported not to exceed 2-3m within the Site area. The presence of permafrost will affect groundwater movement within the bedrock aquifer and create a boundary to flow. However, previous hydrogeological studies have identified crossflows along talik structures within the permafrost likely to be associated with thaw zones below large lakes and rivers. These taliks could carry significant quantities of groundwater however this has not been confirmed within the Site.

Confined groundwater conditions were encountered at depths of 250m and 284m in wells 122 and 123 respectively in the northwest area of the Site. Groundwater levels are reported to have risen over 100m. However, many Soviet Era holes had thermometric measurements downhole, with most temperatures ranging from 0°C to -1.5°C.

In the region of the Kemensky fault, water has been mapped issuing from a fault which, during the winter period, can form extensive ice accumulations in the Kemen River.

Permafrost and talik studies within the region show the chemical composition of groundwater in the under-channel talik to be similar to the composition of river waters.

16.2.2.3 *Water Supply and Management*

Currently, there is limited site-specific hydrology and hydrogeology data however, it is anticipated that raw water will either be pumped from a point in the local river system where water is available all year round (in winter it flows below the ice cap); or it will be drawn from boreholes around the pit, depending upon the depth of any continuous permafrost. Studies at nearby Udokan deposit investigated groundwater availability from unconsolidated sediments within the thaw zones around river and streams in the valley of the Lower Ingamakit. Groundwater availability was calculated to be 207,399 m^3/day.

Within limited water quality for the Site, Tetra Tech proposed using a 20 m^3/h reverse osmosis water treatment system with 100% standby capacity to provide all site potable water requirements reliably.

Process water on Site is proposed to largely be recycled from the TMF and site runoff supplemented with 'top-up' water from the raw water source. Appropriate water management on Site will prevent adverse impacts to the surrounding water. Water management will include: separation of clean water flows from potentially dirty areas; any water arising in dusty areas around the mine, process plant, or WRD will be collected in settling ponds to reduce the particulate levels; and treatment of wastewater to meet water quality standards before it is discharged.

16.2.3 Potential Risks with Pit Geometries and Interaction with Groundwater

The setting of the proposed mine is in close proximity to watercourses, within a mountainous area with harsh climate leading to potential significant flows during snow thaw and is within a region susceptible to earthquakes. These conditions could lead to potential water environmental impacts affecting water quality and quantity. Exploration results and geological studies reveal sulphide mineralisation and, therefore, the potential for Acid Rock Drainage (ARD) leaching metals into surface and groundwater. In addition to the quality of the surface water, aquatic ecology studies of the Chara River catchment and sub-catchments have recorded 20 fish species including two protected species which could also be affected by the Project.

16.2.4 Conclusions and Recommendations

1. Site specific hydrological and hydrogeological information is limited, therefore further investigation will be required to obtain:
    a. Surface water quality data;
    b. Surface water flow data;
    c. Groundwater quality data;
    d. Aquifer properties, and;
    e. Permafrost/talik investigation.
2. A reliable water supply meeting the projects technical and potable demands need to be developed at PFS stage.
3. A water balance and a water management plan will need to be established for the Project.
4. There is the potential for ARD issue due to the sulphidic ore therefore a suitable test programme will need to be established starting with static test work and testing sufficient representative samples to be commensurate with the inventory of potential acid forming materials (ores and waste);
5. A groundwater model will be beneficial to establish impact predictions on the hydrological and hydrogeological receptors.

16.3 Geotechnical

WAI understands limited geotechnical investigation has been carried out at the Project site and further investigation is required to determine the geotechnical conditions for the design of the open pit, any underground studies, mine site and waste management facilities.

16.4 Net Smelter Return Model and Factors

For the current ore processing options being considered for oxide mineralisation Options 1 and 3 (the cyanide leach – SART options), there is only one concentrate produced without silver bullion, namely a high-grade copper concentrate with silver credits.

For the other two oxide Options 2 & 4 with sequential leaching, the products are silver bullion and copper cathodes.

For the flotation plant processing the sulphide mineralisation two products would be produced:

- Copper concentrate;
- Silver (from Copper/Silver middlings).

A basic net smelter return (NSR) calculation was performed which considered grade, metal price, metallurgical recovery, and metal payability. The payable metal includes the applicable concentrate and refining charges but does not include price participation or penalty element payments. The metal price assumptions were derived by WAI and approved by AZR. All metallurgical recoveries/costs used in the NSR calculation are based on data agreed with AZR.

The NSR factors are shown in the breakdown of optimisation parameters discussed in Section 16.6 and Table 16.1.

16.5 Mineable Inventories

The mineable inventories represent all resources that have the potential to be economic in the future as upside in the scoping study for long term financial forecasting. WAI has based conceptual open pit designs and a combined conceptual design on the in-pit and underground inventories.

The mineable inventory (tonnage) estimate is based on a more optimistic set of cost parameters developed downstream to the MRE and considered for the future conceptual design.

Considering the approach to economically constraining underground resources, the approach will be to apply a set of underground operating parameters applied to block grades below the open pit shell and classified accordingly as potentially economic. It will not include a design or development, but the mineable inventory will incorporate a stope optimiser to simulate stoping and considers development.

Inventories for the PEA nearly all the oxide ore is from OP (15.7Mt) only 0.4Mt is sulphide ore from OP and for oxide only Scenario 1 is stockpiled for future sulphide processing. Sulphide ore is basically split 50:50 between OP and UG.

16.6 Open Pit Optimisation

16.6.1 Overview

WAI has carried out open pit optimisation for the Unkur Deposit using the Datamine NPV Scheduler v4 (NPVS) software package.

The pit optimisations were carried out on the resource block model generated by SRK and driven on the calculated block NSR values. Optimisations were driven on all Inferred resources.

NPVS utilises the Lerchs-Grossmann (LG) algorithm to produce a pit shell yielding the highest undiscounted profit; subject to a fixed set of selling prices (NSR values), mining costs, processing costs and slope angle constraints. NPVS provides the ability to parametrise the commodity selling price (NSR values) and run successive applications of the LG algorithm to generate a sequence of nested pit shells; commonly known as LG phases.

16.6.2 Optimisation Parameters

A breakdown of the costs and parameters used in the Unkur Deposit pit optimisation are presented in Table 16.1, and Processing Factors in Table 16.2.

Table 16.1: Open Pit Optimisation Parameters

Parameter	Unit	Cost
Copper price - Base case	USD/lb	3.86
Silver price - Base case	USD/oz	25.0
Recovery mining	%	95
Dilution (zero grade)	%	5.0
Royalty - Cu	%	8.0*
Royalty - Ag	%	6.5
L&H, D&B - rock	USD/t mined	1.5
L&H, rip - soft	USD/t mined	0.5
Tech services	USD/t mined	0.25
Total Mining - hard rock	**USD/t mined**	**1.75**
Total Mining - soft rock	**USD/t mined**	**0.75**
Bench slope angle - waste	°	75
Bench slope angle - ore	°	75
Bench slope angle - soft	°	45
G&A and Infrastructure Costs	US$ p.a.	9 M

* Note: Does not include x3.5 factor that may be added under RF legislation in Q3 2020. More details on taxes are given in the Financial Analysis Section

Table 16.2: Processing Factors Applied

Oxide Ore Treatment	**Cu Recovery**	**Ag Recovery**	**Opex, US$/t**
Option 1	55.1%	91.1%	18.30
Option 2	95.0%	95.0%	27.40
Option 3	55.9%	73.4%	5.80
Option 4	80.6%	73.4%	15.40
Sulphide Ore Treatment	**Cu Recovery**	**Ag Recovery**	**Opex, US$/t**
Sulphide Ore Treatment	89.1%	82.7%	8.98

All optimisation cost parameters were provided by WAI in conjunction with AZR. A processing rate of up to 3.5Mtpa was assumed for all scenarios.

Smelter and realisation terms have not been considered in the optimisation parameters (other than mining royalty cost and smelter recovery) and are applied at the financial modelling stage.

16.6.3 Optimization Runs

A summary of the in-situ tonnages and grades contained within the five optimal pit shell runs is provided in Table 16.3. The methodology and assumptions used were as follows:

- Not limited by sinking rate – no constraints put on incremental costs as the pit deepens;
- One 15m3 bucket shovel production rate at 10Mtpa;
- Both lines for oxide and sulphide ore types can run simultaneously with 3.5Mtpa limit per line;
- No replacements assumed due to short LOM 6-7 years. Assuming up to 45-50k working hours for single unit, no sustaining CAPEX applied;
- Waste profile has been smoothed using NPV Milava algorithm;
- Comments 16.07.2021Discount rate at 8%;
- GA and infrastructure costs are included on annual basis (US$9Mpa); and
- Royalty has been calculated per metal.

Table 16.3: Summary of Optimization Open Pit Output

Item	Unit	Scenario 1, Option 3	Scenario 2, Option 1	Scenario 2, Option 2	Scenario 2, Option 3	Scenario 2, Option 4
Waste	t	97,420,044	243,664,112	252,363,018	228,138,962	254,164,978
Ore_oxide	t	10,980,525	9,219,308	9,230,153	11,949,639	10,232,932
Cu_grade	%	0.59	0.73	0.70	0.57	0.66
Ag_grade	g/t	48.24	61.24	58.69	46.72	54.10
Ore_sulphide	t	-	7,502,343	7,498,946	7,020,898	7,530,834
Cu_grade	%	-	0.80	0.73	0.74	0.74
Ag_grade	g/t	-	67.36	66.02	67.48	66.06
Stripping ratio	t/t	8.87	15.00	15.00	12.03	14.31

Note that the relative higher metal grades obtained for Options 1 and 2 are a function of higher metal recoveries derived from these options (see Section 16.2).

16.7 Open Pit Design

WAI has not prepared a tactical conceptual pit or waste dump design as part of this study. Waste disposal strategies should be examined in greater detail in further engineering studies.

16.7.1 Satisficer Oxide Only Pit Shell

A plan view of the ultimate pit shells for Scenario 1 (Oxide only, Pit Shell 8) is presented in Figure 16.1.

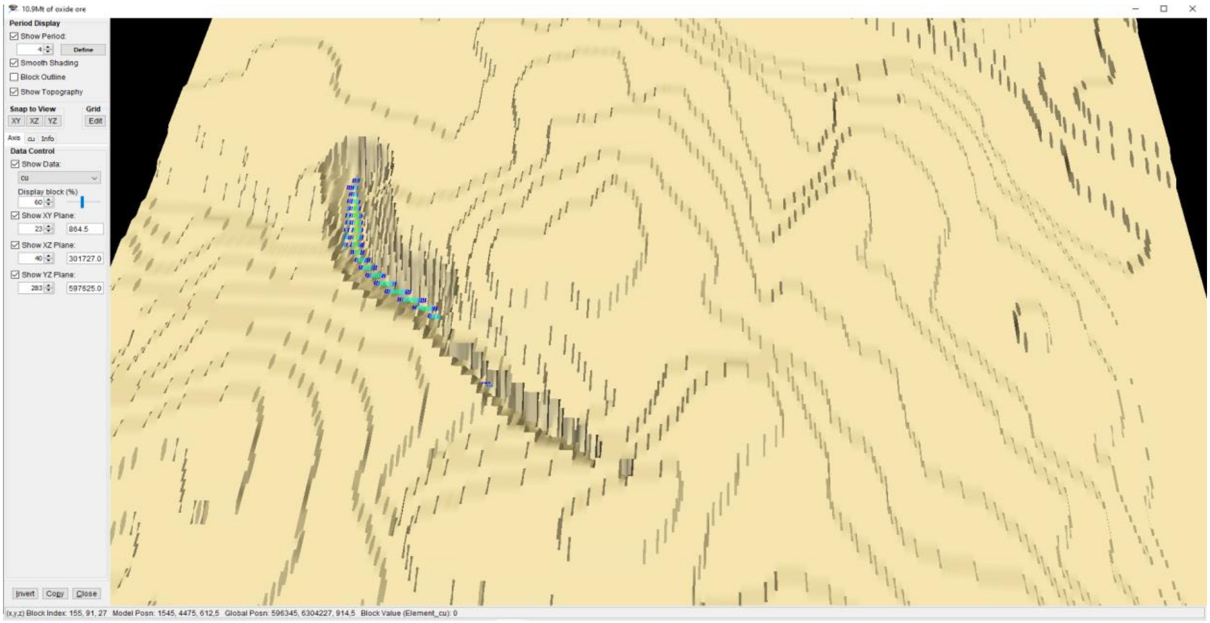


Figure 16.1: Screenshot of Optimal Blocks (Blue) in Pit Shell 8 for Scenario 1

16.7.2 Satisficer Combined Oxide-Sulphide Only Pit Shell

A plan view of Scenario 2 Option 4 (Pit Shell 13) in Figure 16.2. Pit Shell 13 was used as the open pit battery and footprint for use in the following sections to develop infrastructure and TSF locations.

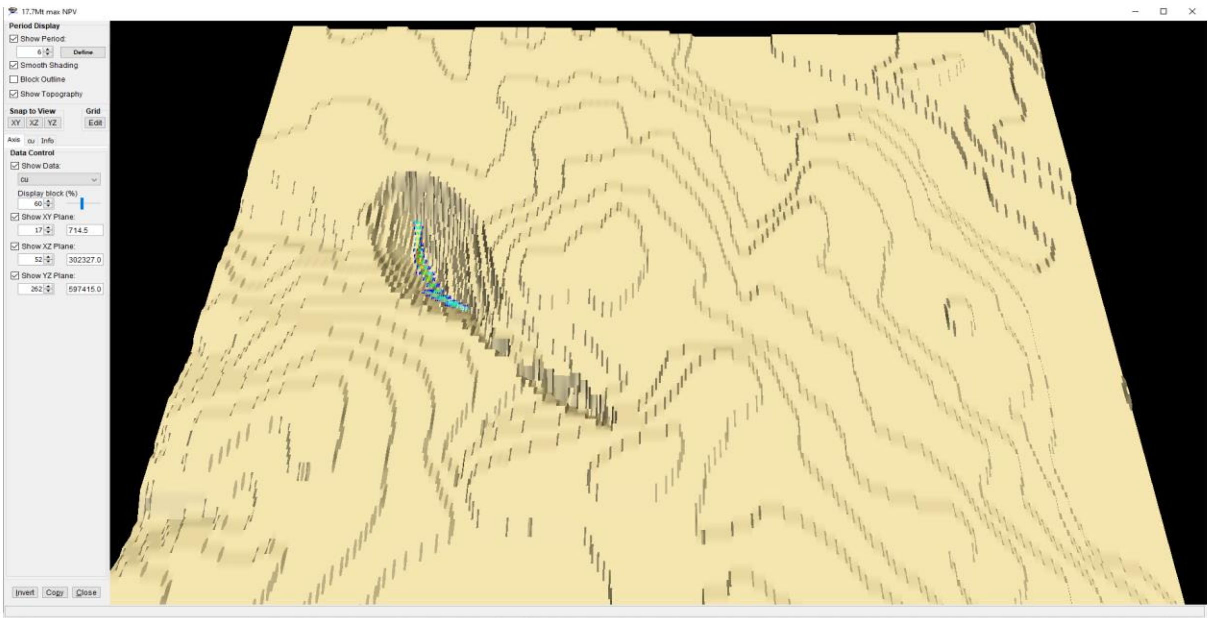


Figure 16.2: Screenshot of Optimal Blocks (Blue) in Pit Shell 13 for Scenario 2, Process Option 4

16.8 Underground Mining

16.8.1 Underground Mining Method

WAI propose to consider mechanised sub-level open stoping (SLOS). The method offers favourable results in safety, cost, and dilution control. Stopes will be extracted in a retreat, top-down sequence, with adequate in-situ rock pillars left unmined for localised and regional stability. WAI has considered 2 major options: to run an oxide-only open pit and apply underground afterwards, and to run different options of sulphide and oxide ore mined and processed from open pit to kick-of the underground. Thus, three scenarios have been considered in the economic analysis:

- Scenario 1: Underground transitions from the oxide only open pit; and
- Scenario 2 and 3: Underground transitions from the ultimate sulphide open pit for oxide processing options 3 and 4.

16.9 Mine Production Scheduling and Equipment Requirements

Mine production scheduling was carried out using the Geovia MineSched mine scheduling software package. A combined open pit and underground production schedule was generated utilising the optimal pit shells for the 5 scenarios. A scheduling block model was prepared in which the mineralised material was split by cut-off grade (i.e., above/below) and rock type (i.e., oxide/sulphide). A nominal cut-off grade of 0.1% copper was selected

16.9.1 Production Schedules

Schedules were prepared for oxide only Scenario 1 taking the lowest CAPEX processing Option 3. Two variants of Scenario 2 where prepared on the premise that a flotation circuit will be implemented to process the sulphide feed following depletion of the oxides to be available as of mid-Year 4.

Results of the combined open pit and sulphide production schedule are summarised in Table 16.4 to Table 16.6. WAI notes that scheduling has been reported annually and sufficient for the level of study.

WAI has also assessed options with a lower throughput and production rate of 2Mtpa and 2.5Mtpa, however, these rates have not proven to be viable and therefore were excluded from further analysis. This was caused by the effect from lower throughput scenarios resulting in the extended the life of mine and consequently lower overall NPVs, impacted by positive cashflows being pushed further towards the later years, and additional sustaining CAPEX requirements.

Table 16.4: Combined Production Schedule Scenario 1 (Option 3)

		Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Open Pit																
Waste	kt	**97,420**	13,431	31,789	47,677	4,522	-	-	-	-	-	-	-	-	-	-
Oxide Ore	**kt**	**10,981**	1,500	3,500	3,500	2,481	-	-	-	-	-	-	-	-	-	-
Cu grade	%	**0.59**	0.63	0.56	0.54	0.68	-	-	-	-	-	-	-	-	-	-
Ag grade	g/t	**48.24**	46.21	42.90	44.20	62.72	-	-	-	-	-	-	-	-	-	-
Sulphide Ore	kt	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Cu grade	%	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag grade	g/t	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Stripping ratio	t/t	**8.87**	8.95	9.08	13.62	1.82	-	-	-	-	-	-	-	-	-	-
Total Ore OP	kt	**10,981**	**1,500**	**3,500**	**3,500**	**2,481**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Underground																
Oxide Ore	**kt**	**4,067**	**-**	**-**	**-**	**1,342**	**1,398**	**1,291**	**36**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Cu grade	%	**0.71**	-	-	-	0.63	0.71	0.80	0.76	-	-	-	-	-	-	-
Ag grade	g/t	**41.34**	-	-	-	26.11	34.77	63.59	66.10	-	-	-	-	-	-	-
Sulphide Ore	kt	**16,712**	**-**	**-**	**-**	**0**	**590**	**710**	**2,000**	**2,000**	**2,000**	**2,000**	**2,002**	**2,000**	**2,000**	**1,410**
Cu grade	%	**0.77**	-	-	-	0.54	0.65	0.81	0.81	0.81	0.80	0.80	0.80	0.73	0.73	0.64
Ag grade	g/t	**53.11**	-	-	-	12.67	29.26	63.60	63.60	61.95	58.80	58.80	57.10	39.87	39.87	46.14
CAPEX	US$ M	**150**	-	50	75	25	-	-	-	-	-	-	-	-	-	-
Total Ore UG	kt	**20,779**	**-**	**-**	**-**	**1,342**	**1,987**	**2,001**	**2,036**	**2,000**	**2,000**	**2,000**	**2,002**	**2,000**	**2,000**	**1,410**
Overall Ore OP & UG	kt	**31,759**	**1,500**	**3,500**	**3,500**	**3,823**	**1,987**	**2,001**	**2,036**	**2,000**	**2,000**	**2,000**	**2,002**	**2,000**	**2,000**	**1,410**

Table 16.5: Combined Production Schedule Scenario 2 (Option 3)														
		Total	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**
Open Pit														
Waste	kt	**228,139**	23,906	43,599	43,010	48,142	35,389	26,707	7,386	-	-	-	-	-
Oxide Ore	**kt**	**11,950**	1,094	1,401	1,990	3,500	3,500	465	-	-	-	-	-	-
Cu grade	%	**0.57**	0.56	0.59	0.53	0.55	0.59	0.61	-	-	-	-	-	-
Ag grade	g/t	**46.72**	34.75	41.57	43.86	47.83	52.18	53.20	-	-	-	-	-	-
Sulphide Ore	**t**	**7,021**	**-**	**-**	**-**	**0**	**386**	**3,500**	**3,134**	**-**	**-**	**-**	**-**	**-**
Cu grade	%	**0.74**	-	-	-	0.75	0.75	0.77	0.71	-	-	-	-	-
Ag grade	g/t	**67.48**	-	-	-	67.00	67.00	67.52	67.50	-	-	-	-	-
Stripping ratio	t/t	**12.03**	21.85	31.13	21.61	13.75	9.11	6.74	2.36	-	-	-	-	-
Total Ore OP	kt	**18,971**	**1,094**	**1,401**	**1,990**	**3,500**	**3,886**	**3,965**	**3,134**	**-**	**-**	**-**	**-**	**-**
Underground														
Oxide Ore	kt	**3,283**	**-**	**-**	**-**	**-**	**-**	**1,472**	**1,420**	**391**	**-**	**-**	**-**	**-**
Cu grade	%	**0.68**	-	-	-	-	-	0.64	0.70	0.75	-	-	-	-
Ag grade	g/t	**34.12**	-	-	-	-	-	27.07	35.91	54.25	-	-	-	-
Sulphide Ore	kt	**11,332**	**-**	**-**	**-**	**-**	**-**	**0.04**	**590**	**1,611**	**2,000**	**2,000**	**2,000**	**3,130**
Cu grade	%	**0.74**	-	-	-	-	-	0.54	0.65	0.75	0.80	0.78	0.73	0.69
Ag grade	g/t	**39.69**	-	-	-	-	-	12.67	29.26	35.77	46.48	45.00	39.82	35.87
CAPEX	US$ M	**175**	-	-	-	50	75	25	-	-	-	-	25	-
Total Ore UG	kt	**14,615**	**-**	**-**	**-**	**-**	**-**	**1,472**	**2,010**	**2,002**	**2,000**	**2,000**	**2,000**	**3,130**
Overall Ore OP & UG	kt	**33,585**	**1,094**	**1,401**	**1,990**	**3,500**	**3,886**	**5,437**	**5,144**	**2,002**	**2,000**	**2,000**	**2,000**	**3,130**

Table 16.6: Combined Production Schedule Scenario 3 (Option 4)

		Total	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**
Open Pit														
Waste	kt	**254,165**	24,175	42,967	61,114	61,046	59,985	4,878	-	-	-	-	-	-
Oxide Ore	kt	**10,233**	825	2,033	3,500	2,455	1,420	-	-	-	-	-	-	-
Cu grade	%	**0.66**	0.77	0.66	0.66	0.68	0.56	-	-	-	-	-	-	-
Ag grade	g/t	**54.10**	55.39	56.57	54.20	55.00	48.00	-	-	-	-	-	-	-
Sulphide Ore	kt	**7,531**	**-**	**-**	**-**	**1,499**	**3,500**	**2,532**	**-**	**-**	**-**	**-**	**-**	**-**
Cu grade	%	**0.74**	-	-	-	0.68	0.75	0.77	-	-	-	-	-	-
Ag grade	g/t	**66.06**	-	-	-	61.40	67.00	67.52	-	-	-	-	-	-
Stripping ratio	t/t	**14.31**	29.31	21.13	17.46	15.44	12.19	1.93	-	-	-	-	-	-
Total Ore OP	kt	**17,764**	**825**	**2,033**	**3,500**	**3,954**	**4,920**	**2,532**	**-**	**-**	**-**	**-**	**-**	**-**
Underground														
Oxide Ore	kt	**3,283**	**-**	**-**	**-**	**-**	**-**	**1,472**	**1,420**	**391**	**-**	**-**	**-**	**-**
Cu grade	%	**0.68**	-	-	-	-	-	0.64	0.70	0.75	-	-	-	-
Ag grade	g/t	**34.12**	-	-	-	-	-	27.07	35.91	54.25	-	-	-	-
Sulphide Ore	kt	**11,332**	**-**	**-**	**-**	**-**	**-**	**0**	**590**	**1,611**	**2,000**	**2,000**	**2,000**	**3,130**
Cu grade	%	**0.74**	-	-	-	-	-	0.54	0.65	0.75	0.80	0.78	0.73	0.69
Ag grade	g/t	**39.69**	-	-	-	-	-	12.67	29.26	35.77	46.48	45.00	39.82	35.87
CAPEX	US$ M	**175**	-	-	-	50	75	25	-	-	-	-	25	-
Total Ore UG	kt	**14,615**	**-**	**-**	**-**	**-**	**-**	**1,472**	**2,010**	**2,002**	**2,000**	**2,000**	**2,000**	**3,130**
Overall Ore OP & UG	kt	**32,378**	**825**	**2,033**	**3,500**	**3,954**	**4,920**	**4,004**	**2,010**	**2,002**	**2,000**	**2,000**	**2,000**	**3,130**

16.9.2 Development Profile

Figure 16.3 through to Figure 16.5 show the production profiles and ramp up for the combined open pit and underground operation scenarios. Underground works are assumed to follow after the developed open pits.

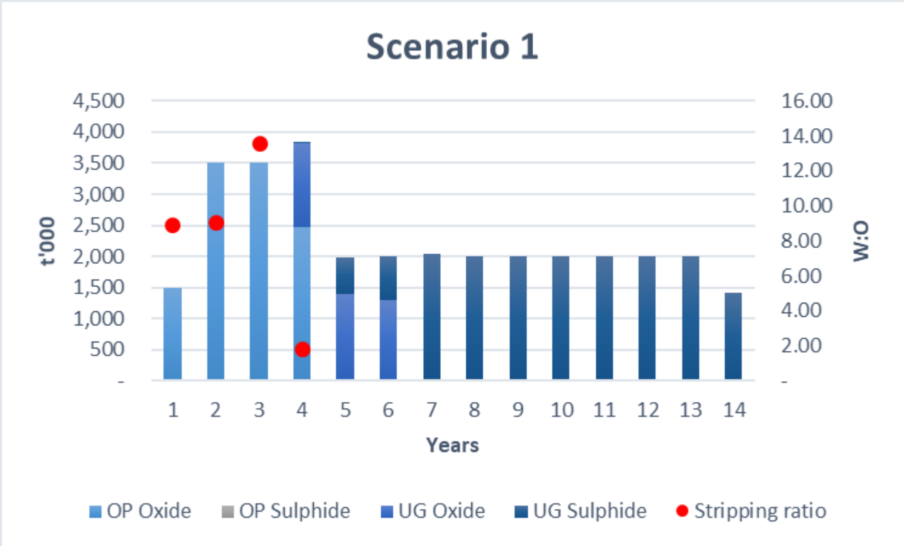


Figure 16.3: Production Profile Scenario 1 – Oxide Option 3

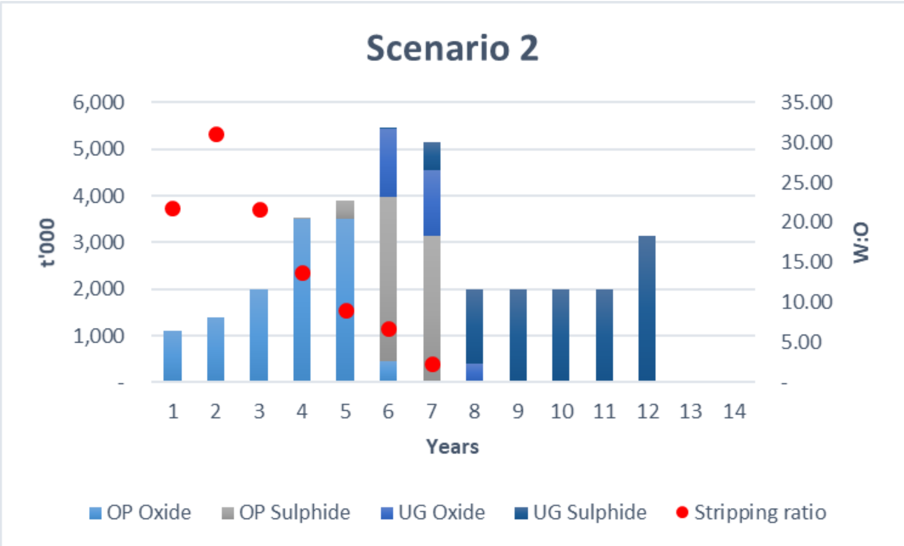


Figure 16.4: Production Profile Scenario 2 -Oxide Processing Option 3

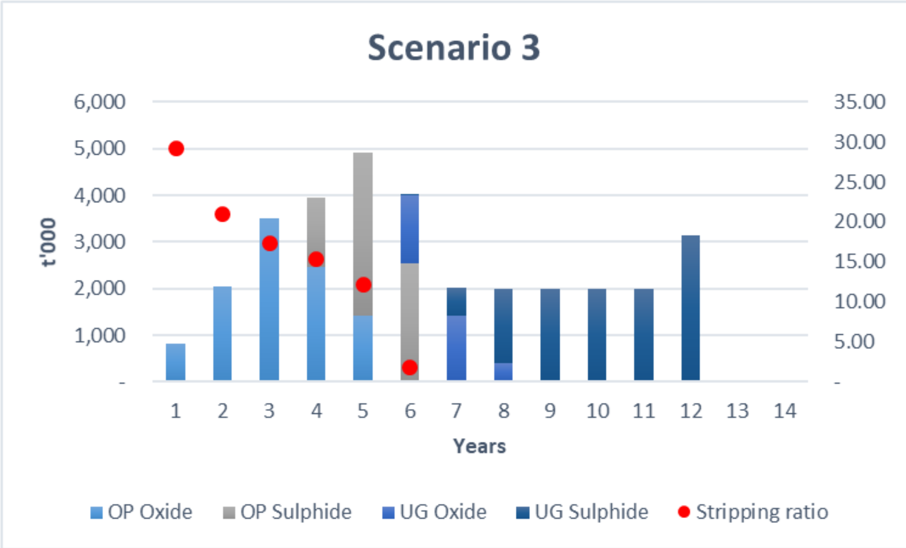


Figure 16.5: Production Profile Scenario 3 – Oxide Processing Option 4

From early runs of estimating the schedules and production profiles steep ramp ups in moving waste at the front end of the operation were observed leading to very high stripping ratios at the front end of the LOM. WAI therefore decided to lower the nominal cut-off grade (0.1% Cu), push out the waste movement more evenly to the back end of the schedule and drive the optimiser to high grade areas early in the schedule. This has captured previous marginal grade material as ore and produced a manageable strip ratio profile (particularly for oxide) but at the expense of lower grades. Less material has been captured in the RF1 pit shells compared to the Tetra Tech estimate largely as a result of lower but more accurate and reflective recoveries for oxide ore and the split in the resource between oxide and sulphide approximately 1:3. Recovery is a key driver in the optimization. Scenario 2, Option 3 presents the best case in terms of smoothest distribution of waste movement profile and lowest average stripping ratio for the combined oxide + sulphide routes – albeit still requiring movement of a high proportion of waste to expose ore.

16.9.3 Open Pit Equipment Requirements

Mine equipment requirements were estimated to achieve the open pit production schedules for each scenario. Equipment requirement estimates for drilling, loading, and hauling were calculated from first principles analysis. Key considerations made in estimation include:

- Utilisation of similar specification equipment to that currently available on site;
- Application of the current blast design parameters;
- Estimates of the annual haulage distances to the waste rock dump (WRD) and run-of-mine (ROM) pad; and
- Application of suitable productivity/utilisation factors and working hours.

Makes and models of equipment have not been specified here but sizing and pricing is typical within scoping level of accuracy for Russian, Chinese or Japanese suppliers assuming 7 years (up to 48K of working hours) operating life based on 85% availability and utilization. The ancillary equipment requirements were estimated based on previous experience of similar projects and approximate working hours required and set at 5% of the major fleet requirements. A summary of the estimated major fleet requirements for the shortlisted scenario options is provided in Table 16.7 through Table 16.9. The units per year represent the number of units required to be working on site for that year and not replacement units.

Table 16.7: Equipment Schedule Scenario 1, Option 3 (US$)

Equipment Type / Year	Unit Cost	1 year	2 year	3 year	4 year
Excavator, 15m3	2,500,000	2	3	5	1
Truck, 135t	1,600,000	5	12	16	3
Dozer, 100t	2,000,000	2	3	5	1
Loader, 10m3	1,800,000	1	1	2	1
Drill rig, 250mm	2,500,000	2	4	6	1
CAPEX Schedule	Total	Yr 1	Yr 2	Yr 3	Yr 4
Equipment Capital Cost	**70,035,000**	**24,990,000**	**21,735,000**	**23,310,000**	

Table 16.8: Equipment Schedule Scenario 2, Option 3 (US$)

Equipment Type / Year	CAPEX per unit	1	2	3	4	5	6
Excavator, 15m3	2,500,000	3	4	4	5	4	3
Truck, 135t	1,600,000	8	15	15	17	13	9
Dozer, 100t	2,000,000	3	4	4	5	4	3
Loader, 10m3	1,800,000	1	2	2	2	2	1
Drill rig, 250mm	2,500,000	2	3	3	4	3	2
CAPEX Schedule	Total	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6
Equipment Capital Cost	**66,465,000**	**34,755,000**	**21,000,000**		**10,710,000**		

Table 16.9: Equipment Schedule Scenario 2, Option 4 (USD)

Equipment Type / Year	CAPEX per unit	1	2	3	4	5	6
Excavator, 15m3	2,500,000	3	4	6	6	6	1
Truck, 135t	1,600,000	8	15	21	20	20	2
Dozer, 100t	2,000,000	3	4	6	6	6	1
Loader, 10m3	1,800,000	1	2	2	2	2	1
Drill rig, 250mm	2,500,000	2	3	5	5	5	1
CAPEX Schedule	Total	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6
Equipment Capital Cost	**80,535,000**	**34,755,000**	**21,000,000**	**24,780,000**			

WAI has assumed all mining equipment to be deployed on site is assumed to be leased. Although there are sometimes benefits from use of contract mining in saving on initial mining CAPEX (WAI assumed zero mining CAPEX in its initial assessments) escalating operating costs, contractor monopolies in the region and the degree of waste movement required initially would be unattractive to the local contractors – hence an owner-operated with supplier lease approach.

16.9.4 Underground Equipment Requirements

Mine equipment requirements have been benchmarked against the existing analogous operations to achieve the underground production schedules prepared for the three underground scenarios. In addition to the mobile equipment, fixed infrastructure crucial to the operation of the underground workings have been considered along with the underground equipment requirements.

Initial capital costs required for underground capital development works have been estimated at the total of $85M, with additional US$85M required for underground mining fleet (see Table 21.1).

Sustaining capital cost of US$25M have been allocated to replace equipment in every 5th year following equipment purchase.

Thus, total underground capital requirements for the considered scenarios have been estimated at US$190M for the Scenario 1 and US$175 for the Scenario 2 and 3. The latter are associated with the shorter LOM, and therefore no sustaining capital costs have been included.

16.10 Risks

The key mining risks associated with the Unkur project are summarised in the points below:

- Relatively high stripping ratios especially during key periods of push-back which cannot afford to experience delays. The schedule demonstrates getting into sulphide material as early and quickly as possible is desirable to improve NPV and the project economics.
- Mining capital and operating cost estimates are based on a Preliminary Economic Assessment (PEA) level of confidence (±45%). The study offers a valuable view in determining the merits of pursuing further engineering studies but should not be the sole reference for the purposes of economic decision making.

17 RECOVERY METHODS

17.1 Introduction

WAI was requested to incorporate testwork results on oxide ore presented by VNIItsvetmet testwork carried out in Q2 2021 to consider the viability of four oxide process options.

17.2 Tetra Tech PEA Process Flowsheet Oxide Ore Options

The Tetra Tech PEA summarises the oxide sample testwork conducted prior to its publication in August 2018, so did not include any of the later VNIItsvetmet testwork, in particular copper recovery from cyanide leaching.

Four process options were studied by Tetra Tech, and Table 17.1 summarises the main input parameters.

Table 17.1: Input Parameters of the Four Process Options					
	Unit	**Option 1 Cyanide Tank Leach (Base case)**	**Option 2 Tank Leach SX/EW**	**Option 3 Heap Leach SART**	**Option 4 Heap Leach SX/EW**
Total Plant Capital Cost	US$ M	128.14	187.07	77.43	128.89
Total Plant Operating Cost	US$ / t	19.18	28.64	13.86	22.45
Overall Metallurgical Recovery	Ag %	95	95	65	65
	Cu %	95	95	65	65

17.2.1 Tetra Tech Option 1

Option 1, selected as the base case, is conventional crushing, grinding and agitated cyanide leaching of the copper and silver, followed by CCD washing and processing of the solution using the SART process to recover a saleable synthetic copper/silver concentrate and recycling the cyanide solution back for heap leaching. This compensates for the high cyanide consumption in the heap leaching of the cyanide-soluble copper with the silver and allows the overall cyanide consumption to decrease to acceptable levels.

It should be noted that, while the Tetra Tech estimate for silver recovery is reasonable, the recovery for copper was significantly over-estimated, as cyanide leaching results for copper were not available at that time.

The schematic flowsheet for Option 1 is shown in Figure 17.1.

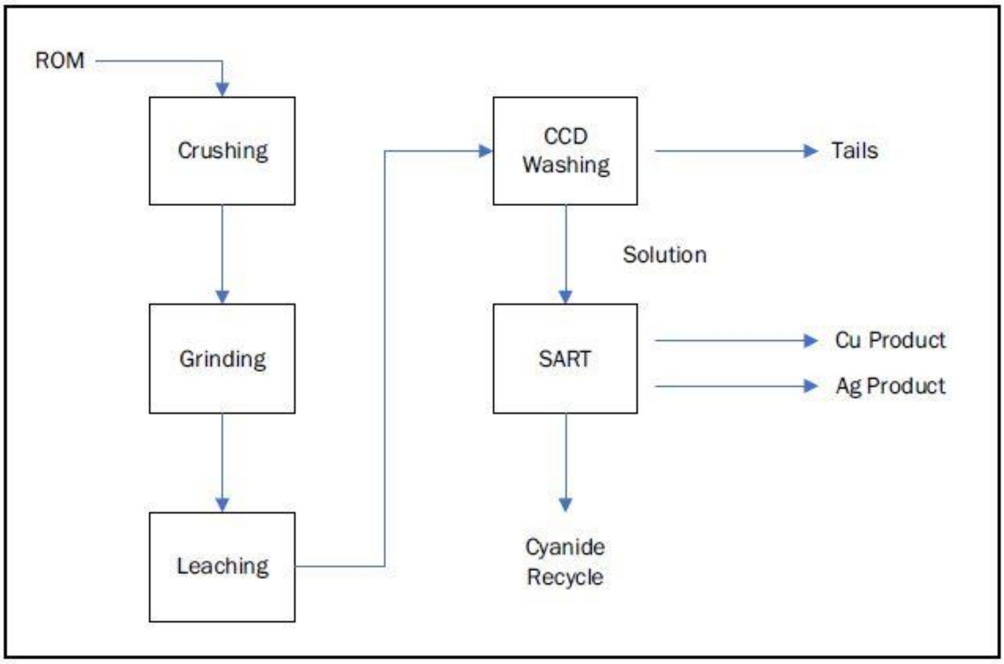


Figure 17.1: Schematic Flowsheet for Option 1

17.2.2 SART

The SART process includes **S**ulphidisation-**A**cidification-**R**ecycling-**T**hickening.

A schematic diagram (Figure 17.2) showing the principles of the SART process is shown below.

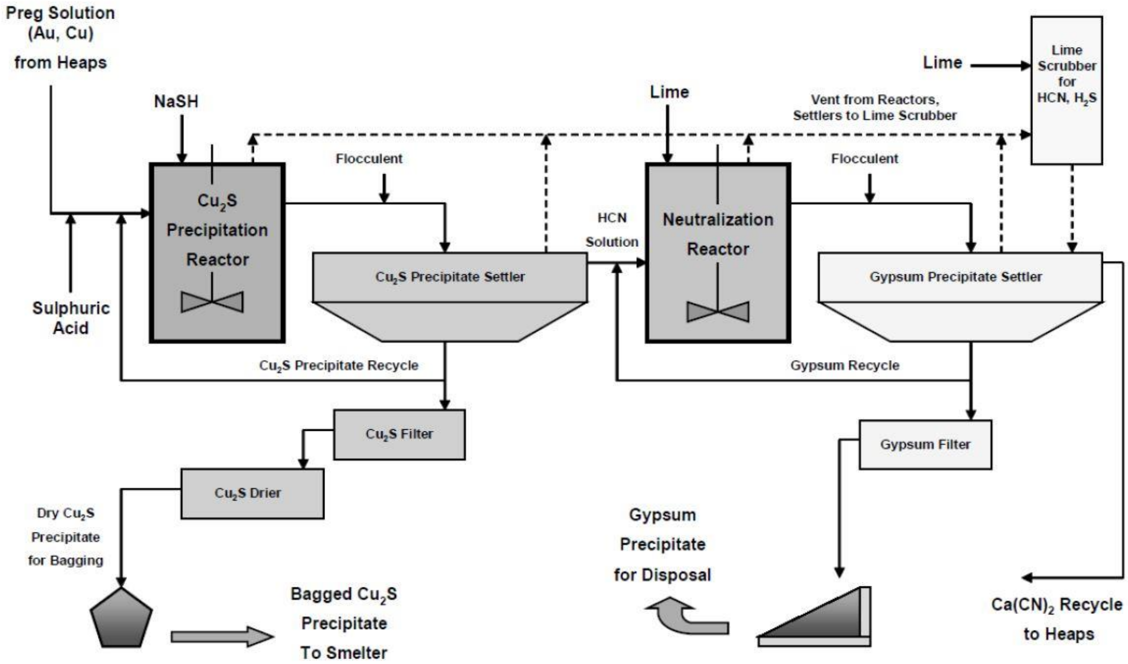


Figure 17.2: Principles of SART

Copper sulphide is precipitated and thickened to produce a filtered Cu2S concentrate by treating the leach solution with sodium hydrogen sulphide (NaSH) at low pH (4-5) using sulphuric acid. A portion of the thickened concentrate is recycled back to the reactor tank to act as seeding for precipitation. Sufficient NaSH must be added to prevent any oxidation of the copper sulphides. Flocculant is added for the thickening process.

Although not shown in the diagram above, an identical arrangement can be included for silver sulphide (Ag2S) precipitation by treating the Cu2S thickener overflow solution by adding further NaSH and sodium hydroxide (NaOH) to increase the pH in the range of 5-10. In other words, a two-stage SART process is suggested.

Although the Cu2S and Ag2S are produced separately, the thickener underflows can be combined for filtration and therefore only one concentrate product for sale to market, effectively a high-grade copper concentrate with high-grade silver credits.

Finally, the Ag2S thickener overflow solution is neutralised with lime to produce gypsum for disposal as tailings and with the cyanide solution recycled back to the heaps.

Vented air from the reactors and thickeners containing Hydrogen cyanide and Hydrogen sulphide gas is scrubbed with lime and the resulting solids report to the gypsum thickener/settler, while the scrubbed air can be exhausted to atmosphere.

Future SART testwork will determine whether a single or two-stage SART process is required.

17.2.3 Tetra Tech Option 2

The block flowsheet for this option is shown in Figure 17.3.

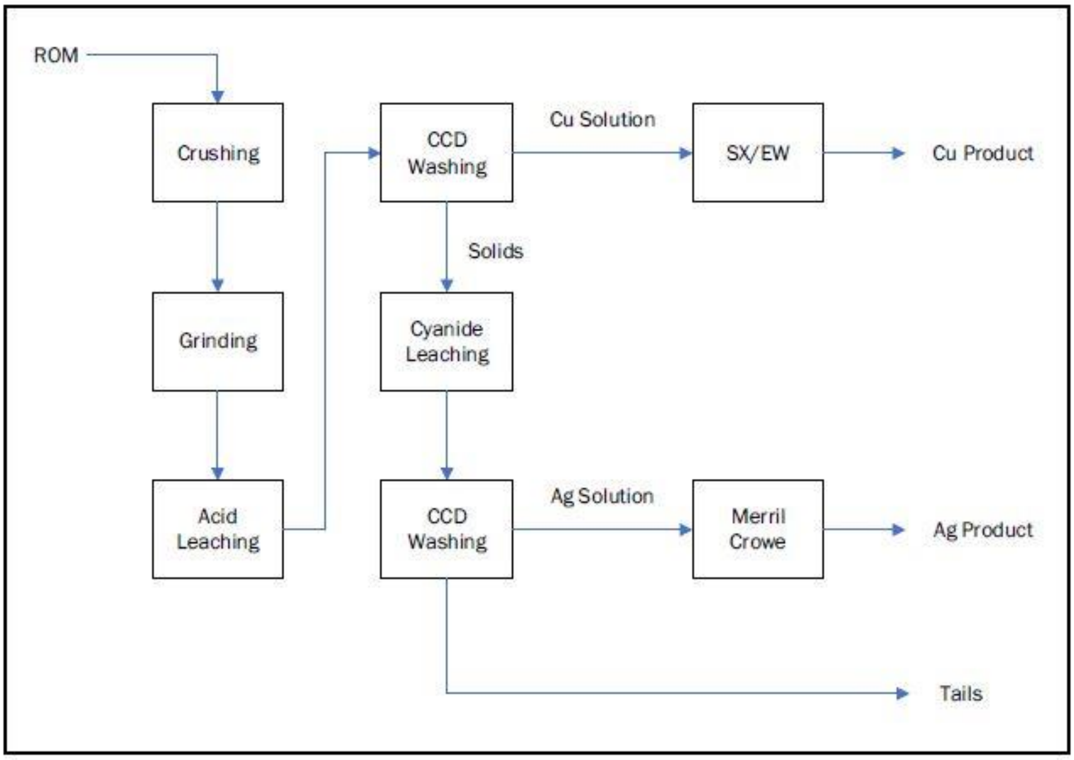


Figure 17.3: Process Flowsheet for Option 2

This option is the most capital-intensive and with the highest operating cost (in particular due to the high acid consumption in the acid leach circuit) and is based on sequential acid and cyanide leaching of the finely ground ore. After crushing and grinding, agitated acid leaching is affected followed by CCD washing and SX/EW treatment of the solution for copper cathode production. The solids are then agitation leached with cyanide for silver recovery and, after CCD washing, the solution is processed via Merrill Crowe to produce silver bullion. The solids report as final tailings.

The Tetra Tech estimated copper and silver recoveries of **95%** each are reasonable based on the historical testwork results. Although the highest silver and copper recoveries are obtained, the capital and operating costs are significantly higher.

17.2.4 Tetra Tech Option 3

This option is for a cyanide heap leach at a coarse crush size for silver recovery but with the solutions processed by SART (as for Option 1) to recover a combined Cu/Ag concentrate.

The flowsheet in presented in Figure 17.4.

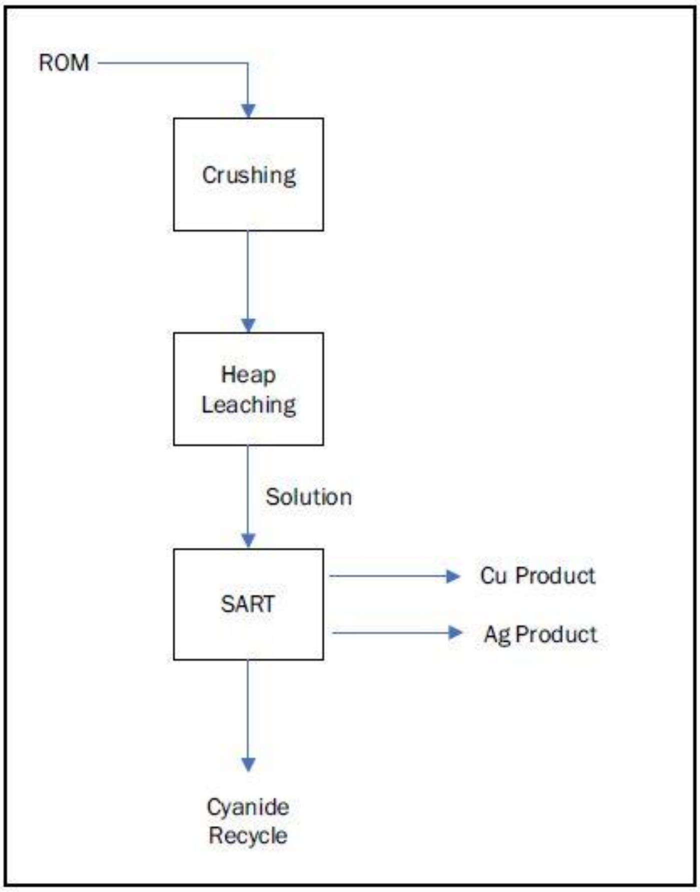


Figure 17.4: Process Flowsheet for Option 3

After three stages of crushing, the ore is agglomerated if required (depending on the level of fines) and heap leached with sulphuric acid to recover the copper and silver into solution. This is then treated via the SART process as per Option 1 to produce a combined Cu/Ag concentrate and the cyanide recovered and recycled back to the heaps.

This option has the lowest capital and operating costs.

The copper and silver recoveries were previously over-estimated and under-estimated respectively as no coarse ore bottle roll test results for this option were available at the time of the Tetra Tech report.

An alternative flowsheet (as per Kinross's Maricunga operation) is to treat only a bleed stream from the heap leach pregnant solution via SART for producing the combined Cu/Ag concentrate, with the majority of solution conventionally processed through a carbon-in-column (CIC) and carbon elution/electrowinning circuit for silver bullion production.

17.2.5 Tetra Tech Option 4

This option is for sequential acid and cyanide heap leaching). The block flowsheet for this option is shown in Figure 17.5.

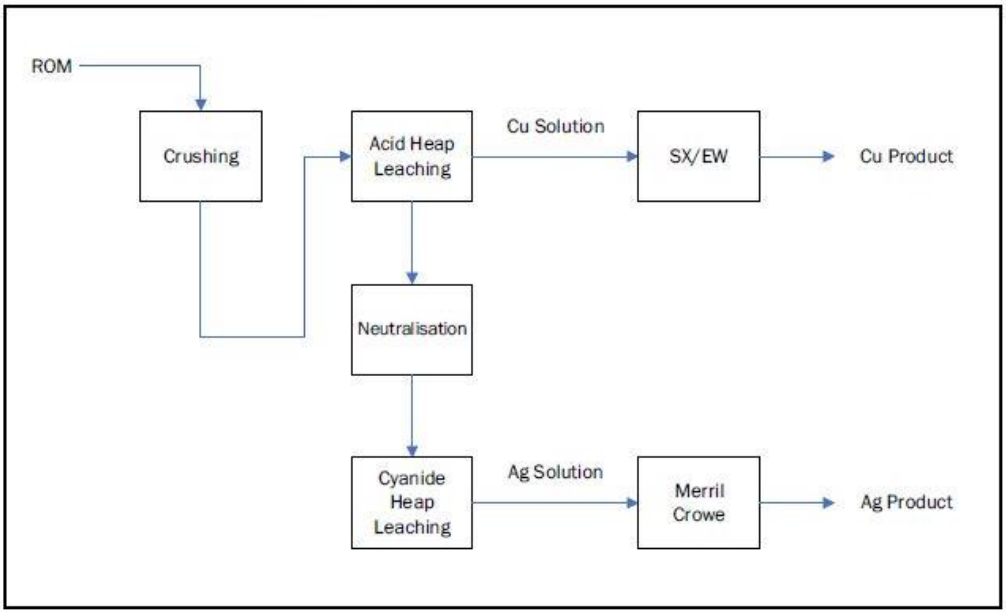


Figure 17.5: Process Flowsheet for Option 2

After crushing and agglomeration as required, the ore is stacked, and heap leached with sulphuric acid for copper recovery and the pregnant solution treated by SX/EW to produce copper cathodes. The ore is then removed from the pads, neutralised, and re-stacked for leaching with cyanide solution. The pregnant solution is then treated by Merrill Crowe to produce silver bullion. A neutralisation step is required after the acid heap leaching is complete and necessitates the washing, neutralisation, and removal/restacking of the ore to another pad for cyanide heap leaching, which is additional rehandling cost.

The estimated cyanide-leached silver recovery remains at **73.4%** (same as for Option 3) but the copper recovery, based on acid leaching rather than cyanide leaching, is estimated as **80.6%** at 54.2 kg/t acid consumption (this is taken from the historical testwork results at a crush size of -10mm).

Operating costs are high due to the high acid consumption.

The recoveries for copper and silver were under-estimated by Tetra Tech, as testwork data for this option was not available at the time. This includes later testwork on coarse ore acid leach bottle roll tests, as well as coarse ore cyanide bottle roll tests.

As a result of the Tetra Tech options analysis, using the best data available at that time, **Option 1** was selected as the base case and **Option 3** was also considered for further economic analysis.

17.3 Process Options for Oxide Ore Selected for Updated Evaluation

AZR has indicated that, for the oxide ore, capital costs should ideally not exceed approximately US$50 million while still trying to maximise NPV. The only option with a capital cost near that objective is cyanide heap leaching followed by SART (**Tetra Tech Option 3**).

However, agitated cyanide leaching followed by SART (**Tetra Tech Option 1**) is still to be considered due to the better overall recoveries.

Importantly, for both these options, the latest testwork results can be used to update the recovery estimates used by Tetra Tech to study these two options more accurately, with updated capital and operating costs where appropriate.

Sequential acid leaching followed by cyanide leaching, whether by agitated leaching or heap leaching, is not considered viable due to the higher capital costs, excessive acid consumption with consequent high operating costs and greater complexity. The SX/EW options are also generally more expensive than the equivalent SART options.

The two options selected are discussed in more detail below.

A processing rate has been selected of up to 3.5Mtpa for open pit (OP) mining and 2.0Mtpa for underground (UG) mining.

A summary of the current oxide resources is shown in Table 17.2, together with the estimated recovery parameters obtained from the latest testwork programme. It should be noted that **a SART recovery of 95%** has been applied to both options for both the copper and silver recoveries obtained from the testwork (as summarised in the section on metallurgical testwork).

This is based on a reported copper recovery from SART of 80-95%, used as the basis for design at Kinross's Maricunga operation, and assumed to apply for silver recovery also. The maximum recovery of 95% was selected for scoping level purposes. This SART recovery is also in reasonable agreement with other published data from operating SART plants.

It is also assumed that a copper grade of at least 65% Cu will be achieved in the concentrate, based on benchmarking data from current operating SART plants.

Table 17.2: Summary of oxide mineral resource inventory

Ore	Mt	% Cu	g/t Ag	Option 1 Cu Rec, %	Option 1 Ag Rec, %	Option 3 Cu Rec, %	Option 3 Ag Rec, %
Oxide OP	15.7	0.61	44	55.1	91.1	55.9	73.4
Oxide UG	0.4	0.51	23	55.1	91.1	55.9	73.4
Total	**16.1**	**0.61**	**43**	**55.1**	**91.1**	**55.9**	**73.4**

17.4 Process Options for Sulphide Ore

For sulphide ore, a conventional copper flotation processing plant is indicated, producing a single copper concentrate product for sale to market and containing significant silver credits.

A summary of the current sulphide resources is shown in the Table 17.3, together with the estimated recovery parameters obtained from the latest testwork programme.

Table 17.3: Summary of Sulphide Mineral Resource Inventory

Ore	Mt	% Cu	g/t Ag	Cu Rec, %	Ag Rec, %
Sulphide OP	17.1	0.59	49	89.1	82.7
Sulphide UG	17.9	0.57	28	89.1	82.7
Total	**35.0**	**0.58**	**38**	**89.1**	**82.7**

A processing rate has been selected of 3.5Mtpa for open pit (OP) mining and 3.0Mtpa for underground (UG) mining.

The copper concentrate grade is estimated as 25.8% Cu, containing 1,634g/t Ag (testwork results).

A summary of the key parameters for sulphide ore processing is shown below in Table 17.4.

Table 17.4: Summary of Key Parameters for Sulphide Ore Processing

SULPHIDE ORE	Copper Flotation Plant
Copper Recovery, %	89.1
Silver Recovery, %	82.7
Copper Grade, % Cu	25.8
Silver Grade, g/t Ag	1,634
Capital Cost, $M	93.2
Operating Cost, $/t	8.98

A generic block flowsheet for a typical copper flotation plant is illustrated in Figure 17.6.

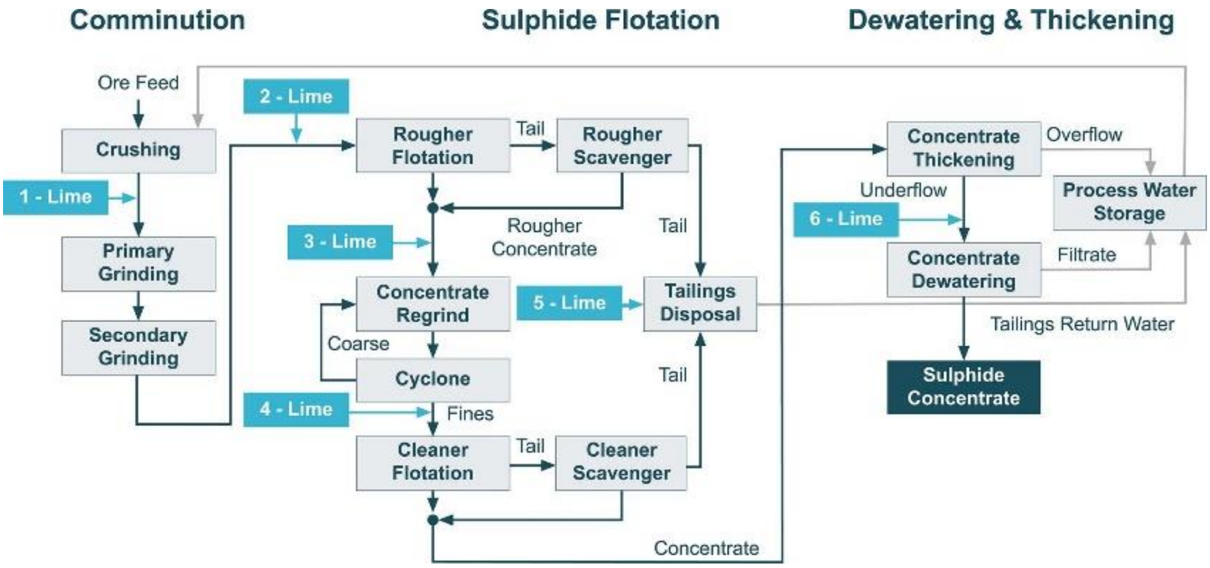


Figure 17.6: Schematic Copper Flotation Circuit

17.5 Conclusions

Current resources indicate 35Mt of sulphide ore at 0.58% Cu and 38g/t Ag and 16.1Mt of oxide ore at 0.61% Cu and 43g/t Ag. Both open pit and underground operations are envisaged. Oxide ore is predominantly from open pit with sulphide ore equally split between underground and open pit production.

The processing of oxide ore presents the biggest challenge due to the carbonate content, resulting in very high acid consumptions from conventional acid leaching, either by agitated or heap leaching. Therefore, as per the Tetra Tech PEA report, the focus has been on process routes employing cyanide leaching first for silver and copper dissolution, with two options studied for agitated and heap leaching, followed by processing of the leach solutions with SART technology.

During the course of this current study, a bottle roll testwork programme was conducted to estimate copper and silver recoveries from cyanide leaching at fine and coarse sizes for the agitated and heap leaching options. This data was not available at the time of the Tetra Tech PEA report, so these recovery estimates have been revised from the initial PEA assumptions, together with updated capital and operating costs.

It is clear from simple relative NPV calculations for the two oxide process options, that heap leaching is significantly better than agitated leaching, in terms of higher NPV and significantly lower capital costs (in line with AZR's target of approximately US$50M capital cost for an oxide processing plant).

For SART performance, benchmarking data has been used for the capital and operating cost estimates and for the estimate of 95% copper and silver recovery, to achieve a single concentrate grading 65% Cu. A two-stage SART process has been assumed, as per the Tetra Tech PEA report, whereby sequential copper and silver sulphide concentrates are produced, but combined for filtering into just one concentrate product for sale to market. No testwork has been conducted on SART processing at this stage of study.

Going forwards, a comprehensive metallurgical testwork programme is required on representative oxide ore samples to confirm expected copper and silver recoveries through heap leaching (to include optimisation of crush size, requirement for agglomeration, cyanide consumption and copper and silver recoveries prior to SART processing). This will include extensive bottle roll, agglomeration, and column testwork programmes.

A comprehensive testwork programme is also required in particular for the SART process to define the expected copper and silver recoveries, grade of concentrate produced and recycling of recovered cyanide back for heap leaching. Details to be investigated include whether only a bleed stream of the pregnant solution for SART processing should be considered, with the inclusion of conventional carbon adsorption and elution/electrowinning/refining for the bulk of the leach solution for conventional silver bullion production (as per the Maricunga flowsheet).

In addition, the capital and operating costs for SART are influenced strongly by the metals and free cyanide content in the feed solutions. Indications from the limited testwork conducted to-date are that both these factors will be at high levels. With the silver levels in the feed solution significantly lower than the copper, it may be worth considering just a single-stage SART process to decrease capital and operating costs.

All these issues should be considered in the comprehensive SART testwork programme based on the leach solutions obtained from the preceding heap leach testwork programme.

SART is a commercialised process and there are several examples worldwide. However, it is noted that performance has often been less than expected, particularly in terms of higher NaSH consumption. Part of this problem has been attributed to oxidation of the copper sulphides through longer residence times within the reactors and thickeners than in the laboratory/pilot scale set-ups.

The sulphide ore processing is straightforward, and a conventional copper flotation plant is indicated with estimated capital and operating costs developed from Cost Mine data. The copper recovery and concentrate grade (with associated silver credits) has been defined by testwork conducted to-date. However, a comprehensive testwork programme on representative samples of sulphide ore is required to confirm metallurgical performance, develop grindability data and conduct variability sampling.

17.5.1 Risks

Some of the risks to be considered are the following:

- Tonnes of ore captured in the pit are sensitive to recoveries. Rigorous assessment of SART needs to be carried out at a higher level of study to ensure commercial viability on representative samples on the scale of several tonnes;
- Whilst the oxide and sulphide composites selected for the most recent 2020 where geographically representative, attention needs to be given to defining ore type boundaries based on recoveries given the comment above – namely oxide-transition-sulphide boundaries. Also, composites representing other major 'ore' types, such as high secondary sulphides, high penalty element concentrations or high concentrations of harder material, which may influence selection of the process design. In short, more targeted testwork on more representative major 'ore' types.

17.6 Tailings Management Facilities

17.6.1 Introduction

The sulphide process plant has a maximum throughput of up to 3.5Mtpa, the majority of which once processing is complete will report to a Tailings Management Facility (TMF) as tailings. In order to comply with international best practice and guidance the TMF design concept will have to be inclusive of environmental, social, geotechnical, and economic considerations and be design for closure so that the tailings are stored in a safe and stable facility for perpetuity with minimal impact on the environment.

This assessment is a desk-top study based on the information provided by the Client and did not include a site visit to the Project Area. The data sources have included:

- Technical Report and Preliminary Economic Assessment for the Unkur Copper-Silver Project, Kodar-Udokan, Russian Federation, Tetra Tech 2018;
- Technical Report and Mineral Resource Estimate for the Unkur Copper-Silver Project, Kodar-Udokan, Russian Federation, Tetra Tech 2018;
- Supplement to Prospecting and Evaluation Programme for Unkur Ore Occurrence, 2020;
- Udokan Project Environmental and Social Scoping Study, SRK Consulting, 2010; and
- VNIItsvetmet, Ore Processing Technology Development for Unkur Deposit, Ministry of Industry and Infrastructural Development of the Republic of Kazakhstan, 2020.

17.6.2 Basis of Design

The current project is based on a maximum process throughput of up to 3.5Mtpa with a Life of Mine of 12-14 years (sulphide processing 6-9 years), equating to a total production of around 22Mt of tailings. The facility will be operated in a "closed" system with no water discharge to the environment with recirculation of the supernatant water to the process plant and collection of seepage water. The natural topography will be utilised where possible to provide containment with additional storage provided by engineer embankments or containment structures. Tailings will be deposited continually throughout the year. The facility will be designed for closure and will comply with international standards.

The facility will be located in an area that will have the least potential impact on the local water bodies, will not be located immediately upstream of the mine infrastructure or other sensitive receptors but will minimise pumping distances and be located entirely within the mine licence area.

The main methods of tailings deposition are slurry, thickened tailings, paste, dry stack and co-disposal, which in effect represent a reduction in water content in the tailings for each subsequent deposition method. Co-disposal can occur at varying water content and requires the blending of waste rock and tailings waste streams to create amalgamated waste. The preferred and optimum waste storage method is dependent on several factors including production rates, tailings rheology and geochemistry, climatic, environmental and site conditions, water availability and the site water balance, energy cost, geotechnical and stability issues and finally the topography and availability of land.

For the purpose of this study, it has been assumed that the tailings will be in the form of a thickened slurry with approximately 65% w/w solid content, as this is a widely established deposition method, optimises the water removal from the tailings prior to deposition but has a lower power demand than paste or dry stack. A trade-off study including disposal method and site selection should be undertaken at the next stage which considers paste, dry stack and co-disposal options.

17.6.3 Site Selection

The Project location and topography has been detailed in previous sections, but in summary it is characterised by mountainous and hilly terrain with relatively gentle sloping terrain with rocky outcrops and taiga vegetation (swampy coniferous forest) comprising of heathland and peat bogs. The topography means that a cross valley TMF is the preferred storage facility configuration.

The eastern area of the Project site is dominated by the south-north running Kemen River. The open pit is located on the western side of the river with the waste dumps beyond located away from the river but as close as possible to the open pit to minimise haul routes. The process plant and mine infrastructure are located on the southwestern side of the open pit and waste dumps. The site constraints mean that there are limited options for the TMF location, and there for the valley identified in the Tera Teck 2018 report is still considered the preferred location 2.4km southwest of the process site as it is relatively close to the process plant, is away from and does not cross the Kemen River, is not immediately upstream of the mine infrastructure and is located entirely within the mining licence area. The approximate location is indicated in Figure 18.1. The volumetric capacity of this valley will have to be confirmed at the next project stage when topographical data is available.

17.6.4 Climate

The subarctic conditions are characterised by long, cold winters and warm, mild summers with average daily temperatures ranging from -32°C in the winter to 16.5°C in the summer. Typically snow falls in mid-September and begins melting in mid-April. Snow thaw from mid-April onwards could lead to spring melt flows and large seasonal variations in the site water balance. The freezing winter temperatures will require the tailing pipeline to be lagged and heated to allow year-round tailings deposition.

The Site is identified as being within an area of continuous permafrost with thickness typically ranging between 200m and 400m. The thickness of the permafrost depends on location (i.e. valleys or mountain ridges), ground type, gradient of terrain and slope direction. The seasonal thaw of the upper permafrost layer is reported not to exceed 2-3m within the Site area. The presence of permafrost will impact on the stability of the TMF, and the design will have to mitigate for thawing of the permafrost and subsequent settlement of the natural strata. This typically includes the removal of the permafrost from within the upper few metres beneath the embankment footprint by over dill and recompaction of the superficial material.

17.6.5 Geology

The superficial strata across the Site are typically Glacial sediments, in the form of moraines, with an average thickness of 40 m; however, this cover increases to 180m to 200m thickness in the northwest and southeast of the Site. Alluvial sediments within the vicinity of the Unkur and Kemen rivers comprise gravel and sandy soil which form 5m to 20m high terraces above floodplains. The bedrock geology is Lower Proterozoic weakly metamorphosed clastic sedimentary strata intruded by Paleoproterozoic, Neoproterozoic and Mesozoic igneous intrusions.

17.6.6 Seismicity

The Project site is identified as being located within a severe earthquake potential zone with 1 catastrophic earthquake predicted every 25 years and a seismic rating of 9 points on the 12 point MSK -64 Scale. The Tetra Tech report indicates a Peak Ground Acceleration of between 1.6 - 4.0m/s2 for an earthquake of 10% probability of exceedance in 50-year return period.

In order to comply with international standards the TMF will have to be designed to withstand an Operating Basis Earthquake (OBE) which for a facility of this size is typically a 1 in 500 year earthquake (based on Canadian Dam Association CDA) or 1 in 10000 (Global Industry Standard on Tailings Management) and a Maximum Design Earthquake (MDE) of 1 in 2500 years or 1 in 10,000 years respectively at closure.

17.6.7 TMF Design

In order to reduce CAPEX a phased containment structure will be developed with a starter embankment which will be raised with subsequent downstream lifts. A typical downstream raise strategy is presented in Figure 17.7.

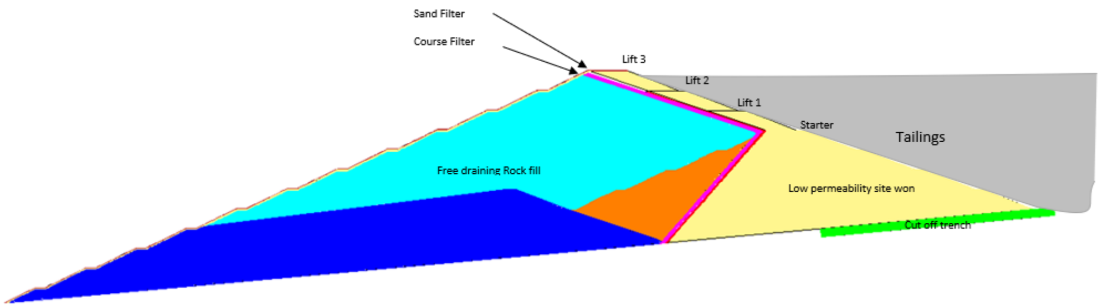


Figure 17.7: Downstream Embankment Raise

The embankment will be zone with a low permeability upstream zone formed from site won silts and clays located in borrow pits within the impoundment area. A filter zone of graded sand and coarse material will prevent fines migration from the from the low permeability zone with the downstream toe of the embankment formed from mine waste rock forming free draining toe drain zone. A HDPE liner will be placed on the upstream slope to prevent a build-up of a phreatic surface through the embankment and improve stability.

A cut off trench will be keyed into the upstream toe excavated down to the permanent permafrost zone, backfilled with low permeability material to prevent seepage beneath the embankment.

No geochemical tests of the tailings are available, but it is considered that the SART process should remove heavy metals and cyanide from the tailings and as such leaching of heavy metals should not occur and the tailings can be considered non-hazardous and non-acid generating. On this assumption lining of the impoundment floor would not be required. This assumption will have to be confirmed at the next study stage.

The storage capacity of the facility will have to be confirmed at the next design stage, but it is considered that the valley will have sufficient storage capacity for the proposed total tailings production.

An upstream dam and diversion ditches will be required to manage the surface water run off through the valley. A spillway will be constructed in the TMF with sufficient capacity within the TMF for a 1 in 10,000-year storm event.

The supernatant water will be ponded away from the embankment and decanted via a decant tower and pumping system piped to the process plant for recirculation in the process.

17.6.8 Closure

The facility will be closed by lowering the phreatic surface within the tailings as low as possible. All pipework and infrastructure will be decommissioned before the facility is capped. A typical capping system of a 3-layer system will be utilise with an initial 3.0m thick reprofiling layer formed from non-acid generating waste rock. Over 1.0m thick low permeability layer of site won material will be placed to prevent ingress of surface water into the facility. Above this low permeability layer, a 0.5m thick topsoil layer will be placed from the stripped and stockpiled topsoil from within the impoundment area.

The re-freezing of the permafrost and eventually the tailings material will prevent seepage into the groundwater system and assist with long term stability of the facility post closure.

17.6.9 Conclusions and Recommendations

A downstream raised cross valley TMF is proposed for the 22Mt tailings assuming a maximum production rate of 3.5Mtpa and a total Life of Mine of 12-14 years. The tailings will be a thickened slurry deposited subaerially within the facility and all supernatant water returned to the process plant in a closed system. The embankments will be formed from site won material and mine waste rock in a zoned configuration. The concept design has been developed on assumptions of the site conditions, tailings properties and international guidance and best practice.

The site is characterised by valleys with glacial moraines in the valley bottom and rock hill sides with coniferous trees. The superficial deposits range from approximately 40m up to approximately 200m in some area of the Project Site.

The Project is located within a seismically active area and continuous permafrost. The TMF design must consider the appropriate seismic and permafrost conditions to ensure stability both during operations and after closure.

The TMF design should be subject to a trade-off study at the next stage to confirm the optimum deposition method and site selection. A detailed ground investigation should then be undertaken of the preferred site to establish both geotechnical, hydrogeological and permafrost characteristics of the site and all geo hazards, including earthquakes, rock fall and surface water management.

18 PROJECT INFRASTRUCTURE

18.1 Access to the Project Licence Area

The site is accessed from the main unpaved highway via an 8km unpaved road which continues through the licence area accessing the land to the South of the Project. In order to maintain this access there is the potential to provide ≈4.5km of track to the North of the open pit excavation as indicated in Figure 18.1.

Similarly, a new road will require construction providing approximately 3.5km of access to the indicated process plant and Administration areas.

While the diverted track will only require reconstituting to the existing standards, the remainder as identified will require upgrading to a recognised standard supporting two-way traffic movements and significant logistical routing of up to 40 tonne vehicles.

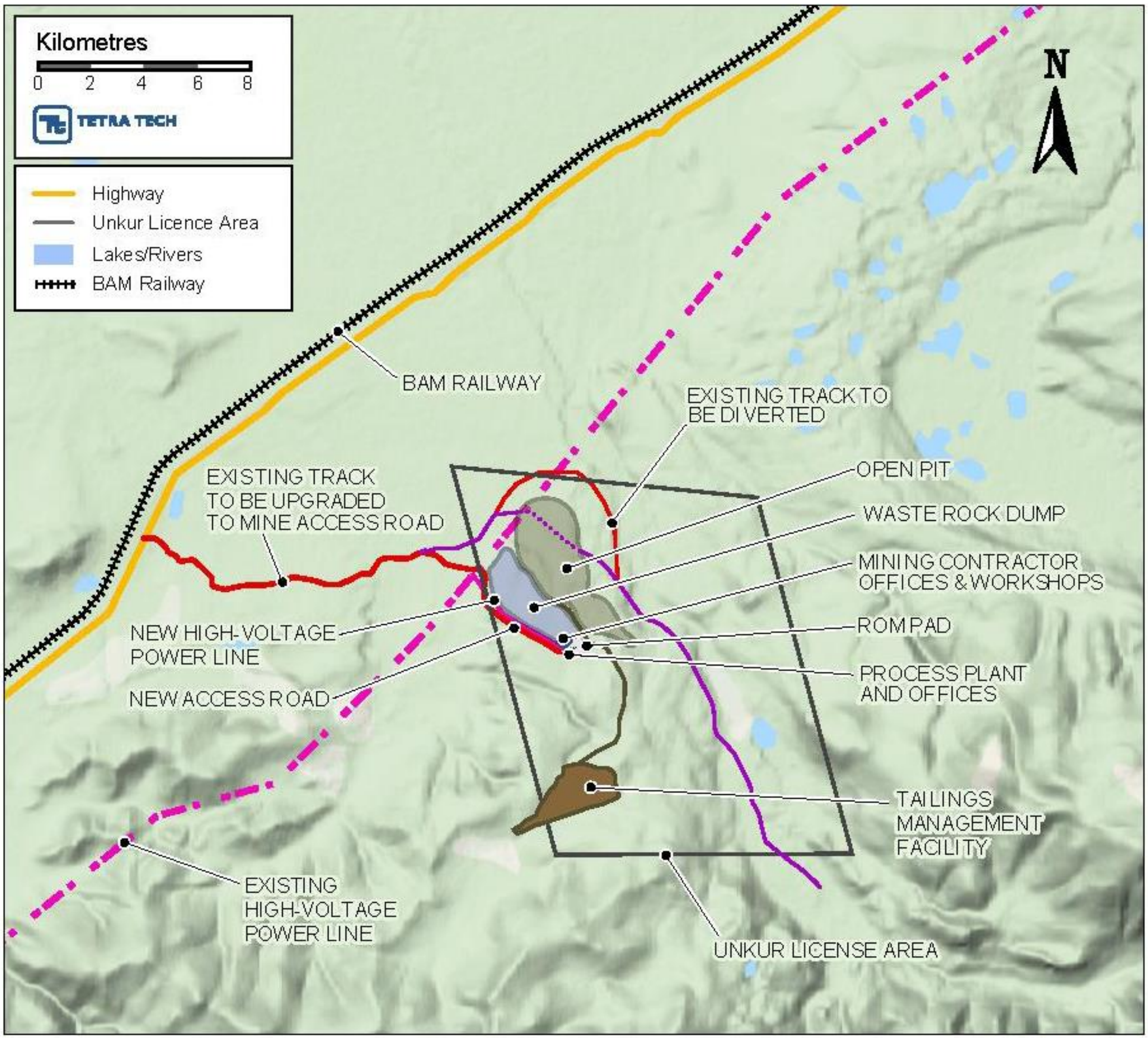


Figure 18.1: Proposed Indicative Site Layout

18.2 Site roads

There will be a requirement for approximately 11km of oversite roadways to meet the operational requirements, including connecting to and from the mineral process plant, warehouse(s), maintenance and administration buildings, Contractors area and similarly, from the excavation sites to the primary crusher locations, TSF, pump houses etc.

The site roads are specified as either Permanent or Temporary roadways.

18.2.1 Permanent roads

Roads that connect the mine with landfills, mineral preparation facilities, and loading stations, must be constructed to suit the maximum load of the transport machinery. These roadways will be generally constructed utilising a compacted gravel dressing over a similarly compacted substrate.

18.2.2 Temporary roads

Roads and ramp access routes from mine excavation areas, landfills and connections with permanent roads must not be loaded more than the bearing capacity and construction specification. All of the site roads are unpaved.

18.2.3 Dimensions

In general, the width of the internal and external roadways for site traffic shall be 7.5m for both one-way and two-way traffic.

18.2.4 Drainage

All roads will have lateral drains suitable for rainwater run-off and discharge management.

18.3 Site Security and Control

A metal fence boundary fence supported by metal posts of approximately 2m in height, will be installed around the perimeter of the operations. A single checkpoint will be installed to monitor access to and from the site.

The checkpoint will consist of a single 34m^2 building, with accompanying access barriers, and shall be located on the main access road at the licence boundary.

A weighbridge and control centre is proposed to facilitate the recording of materials transported but there are no design details available at this juncture. It is suggested that the location site may be adjacent to the concentrate storage area, but this will be confirmed during future study development.

18.4 Mining operations

In general terms the proposal is to mine the deposit by conventional open pit methodologies and later to diversify to include production from underground stoping methods to fully exploit the resource. The maximum ROM production capacity to be considered is 3.5Mtpa.

The waste material is transported to the WRD while the extracted ore is transferred from either operation to a surface ROM pad for primary crushing prior to stockpiling and subsequent feed to the Process Plant.

The ore concentrates shall be packaged for onward transport whilst the tailings produced shall be transported via pipelines (gravity/pumped) to the TSF facility.

It is proposed to minimise the water requirements of the Process plant by recycling from the TSF and it is assumed that an 80% recovery rate can be achieved.

18.5 Available Infrastructure

Despite the reasonably close proximity to the road and rail network and similarly the high voltage transmission line that traverses the north-western corner of the licence area there are no available utility services that may be directly utilised by the project.

18.6 Power

The main electrical load for the mining and processing system is from the primary crushers operating at the open pit site(s), crushers at secondary ore crushing, mills, and flotation machines within the process facility. Additional loads will be imposed later by the instigation of underground mining and the associated equipment requirements, particularly ventilation, crushing and pumping systems.

Seasonal loads imposed by reticulation and building heating requirements place additional loads to the system it is anticipated that an estimated requirement of ≈4 - 6MW is sufficient to address the situation.

While the definitive equipment, plant and building requirements have yet to be developed it is considered by WAI that the operational demands and total production supply requirement is ≈25MW.

Although there is no identified electrical power source available to the Mine site, there is proximity to a high voltage power line which traverses the licence area in the northwest corner of the site. It is assumed for the purpose of this evaluation that sufficient capacity and utilisation of this facility is achievable.

Other options would be the potential for:

- Supply from the electrical substation at Chara; and
- On site electricity generation fuelled by Arctic diesel.

However, both of the above are discounted due to significant negative CAPEX indications.

The electrical power for the entire Project will be sourced from the transmission network via a new ≈4km, overhead supply line to a newly constructed transformer sub-station, located adjacent to the process building. The sub-station will have sufficient installed capacity to enable a duty /standby provision of 100% redundancy.

From the substation and associated motor control centre (MCC), the electrical reticulation supplies the process plant equipment at medium voltage levels (6kv) and via step down transformers low and control voltages as required within the facility.

An over-site 6kv reticulation is similarly supplied from the substation and is transmitted via constructed overhead transmission lines to the various satellite locations for the site requirements. The individual supplies terminating at localised substations containing stepdown transformers and MCC equipment as dictated by the end usage.

These will predominantly be located at the following areas:

- TSF Pumping stations;
- TSF Water treatment facility
- Contractors’ area;
- Open pit pump stations;
- Primary crushers

Smaller ancillary supplies will be taken from the localised sub-stations and reticulated as required, either as a buried service or an overhead installation. These will be utilised to provide power and lighting to outlying or remotely located facilities such as:

- Biological treatment installations;
- Water reticulation pumps; and
- Explosive storage facilities.

18.6.1 Emergency power supply

The mineral processing plant will also be equipped with an emergency generator to provide electrical power in case of a main system power failure. The backup generator will have 3 -5Mva installed power which will be sufficient to maintain emergency lighting, critical pumps, and infrastructure, i.e., thickeners and selected auxiliary equipment, air compressors and cranes, etc.

The emergency generator will have specific diesel storage tank with minimum capacity for 24 hours of continual operation at nominal power.

The provision for and specification of the emergency generating facilities are to be developed and finalised within the scope of further studies.

18.7 Water

Currently there is no suitably identified source of raw water for the project location. While it is intimated that supply may be available from either local water courses and/or cased borehole supplies there is no available data that would substantiate such an option.

The water requirements for the mine operations are substantial and surety of supply is of utmost importance for the continuity of operations.

The supply from either source will undoubtedly require some form of filtration and minimal treatment to satisfy the process water requirements but further specific treatment shall be necessary for the potable supplies.

This may be satisfied by a further simple filtration and chlorination system and or ultraviolet whichever provides the optimum cost-effective solution, as the treatment by chlorination is expensive for reduced quantities.

The deciding factors will be contained within the resultant analysis of the groundwater compositions.

The following quantities are estimated for supply and reticulation purposes:

Estimated process plant requirement:

> ≈2.5m^3/t at 3.5Mtpa = 8.75 million m^3/annum or ≈1,000m^3/hr. Assuming that 80% recirculation is achievable then the make-up requirement for raw water is 200m^3/hr and the quantity to pump from the TSF to the process plant is ≈800m^3/hr, assuming no losses.

Additional supply quantities shall be required for the administration and service buildings as well as the requirements for both the surface mining operations and later the underground facilities.

The definitive requirement shall be identified within further detailed studies.

18.7.1 Hot Water

The provision of hot water to the necessary areas will be provided by in-building reticulation systems from to be specified localised heating systems that will be electrically powered.

18.8 Over site reticulation

The oversite reticulation networks for the distribution of both industrial and potable water have yet to be developed and require further design detailing and rationalisation to finalise the specifications.

At this stage, the industrial water and the potable water are assumed to be contained within the storage facilities at the Process plant location and shall be distributed to the final network via localised pumps and tank/reservoir facilities.

The industrial water supply serves to satisfy the residual demands of the process facility in the event of a shortfall in recycled quantities and may also be pumped to other ancillary storage facilities to supply local requirements associated with the mining and crushing operations. A further storage facility is anticipated to be provided at the Contractor area for similar purposes.

The provision of water storage at these areas has the potential to provide fire-fighting resources within the general location of the re-fuelling stations and this should be further evaluated within the context of further study development.

18.9 Waste, effluent, and sewage

Wastewater from any showers and wash hand basins is piped from the buildings and combines with sewage from the toilets into a single effluent stream that subsequently discharges into the biological treatment facilities which shall be located in two separate areas:

- In proximity to the administration building; and
- In proximity to the Contractors area.

The biological treatment units will be either containerised or will be housed in a suitable building comprising a light structural steel framework with colour coated, corrugated galvanised mild steel roof and sidewall sheeting.

The treatment facilities process the sewage to a waste product that conforms to WHO environmental standards and the treated liquid effluent will discharge and drain via the drainage system transferring to the wastewater treatment area. The residual solid component being collected and disposed of by the contractor.

The total quantity of wastewater within the context of the overall water supply quantities will be insignificant and the benefit to the project of further treating this water will provide no significant benefit.

18.9.1 Pipe Routing

In general, pipes will be run above ground on pipe racks. Where pipe contents are susceptible to freezing, the pipes will be insulated, and heat traced.

Pipes in buildings will be supported on elevated pipe supports. To prevent excessive numbers of pipe supports, or sagging of pipes between supports, the smallest pipe size used for services distribution will be 50 mm NB.

Hot water pipelines will be insulated for protection of personnel and to reduce heat loss. Expansion loops will be provided.

Where possible, flanges and couplings will be located at easily accessible positions.

Valves will be positioned to provide good access for operating personnel, platforms will be provided where necessary.

18.10 Buildings

The definitive overall site plan requires developing and will do so throughout successive studies and detailed engineering phases of the Project. For the purposes of defining a quantum for this document the following are indicated for inclusion any amendments and additions remain subject to clarification and location requirement.

The buildings and infrastructure facilities identified in Table 18.1 are included for the Project to date.

Table 18.1: Main Building Facilities

Building	Quantity	Length (m)	Width (m)
Re-agent storage warehouses etc.	2	35	20
Welfare facilities.	1	35	20
Administration building.	1	40	30
Laboratory.	1	30	15
General and technical services building.	1	40	20
Electrical sub-stations/transformer buildings	3	7	5
MCC Buildings	3	7	5
Vehicle repair workshops	1	50	40
Vehicle wash-down facility	2	18	10
Mechanical/electrical workshops	2	20	15
Security building, lighting, and fences.	1	8	5
Weighbridge and attendant facility.	1	8	5
Spare parts warehouse	2	30	20
Tyre bay	1	20	15
Water supply facilities, reservoirs, tanks, pump houses etc.	2	8	3
Sewage treatment.	2	8	3
Fuel storage and distribution facilities. (addressed elsewhere)			
Fire station and Ambulance station.	2	20	5
Explosives magazine.	1	20	10
Medical centre	1	10	7

Although the basic design is not yet developed it is generally considered that some or all of the following will be applicable (subject to specification) to the majority of the building requirements.

18.11 Site Layout

The following general requirements are made for the development of the design of the surface facilities.

- Good access will be provided to all areas, but excessive numbers of roads will be avoided.
- The mine will be laid out in accordance with operational and other requirements. Good engineering practice will be followed to minimise pipe and cable runs and maximise gravity flow. Facilities will be located close to the areas they serve.
- The designated mine operational areas will be safety fenced to prevent casual access and other specific facilities will be security fenced.
- Mine workshops, offices and stores will be located close to the operational area.
- Emergency and rescue facilities will be located close to the operational areas.
- Movement and transport of persons shall wherever practicable be separated from the flow of minerals and material.

- Where possible, the arrangements of individual facility areas will make allowance for potential future expansion.

18.11.1 Civil and Structural

Wherever appropriate and cost effective, the Designer will seek to maximise the use of locally available materials.

The Designer will incorporate appropriate features within the design for protection against inclement weather.

Modes of construction will be compatible with available skills.

18.11.2 Foundations

Foundations will be determined from data and recommendations of the soil's investigation report.

18.11.3 Concrete

All structural concrete will be designed and constructed in accordance with the appropriate regulations.

18.11.4 Reinforcement

All carbon steel and mesh fabric reinforcement will comply with the appropriate regulations.

18.11.5 Steelwork

All structural steelwork will be designed and constructed in accordance with the appropriate regulations.

The Designer will maximise the use of steel sections produced by in country mills for all buildings and equipment structures. Where this is not possible, all structural steel will be specified in accordance with appropriate international standards.

18.11.6 Cladding

Wherever dictated by the design, proprietary profiled mild steel sheeting will be specified for cladding. Surface coating/protection will be incorporated appropriate to the environmental conditions prevailing at the mine site.

18.11.7 Access Flooring

Where duty requirements necessitate, standard mild steel open grid or raised pattern plate floor panels will be specified. Surface coating/protection will be incorporated as required and appropriate to the duty conditions.

The minimum structural thickness of steel section in open grid flooring shall be 6mm.

The minimum structural thickness of raised pattern floor panels shall be 6mm and appropriately stiffened to prevent deflection/springing when loaded.

18.11.8 Floor Finishes

Floor finishes will be specified appropriate to the duty conditions and wherever possible will maximise the use of local materials. These finishes will include, but are not limited to, the following:

- Concrete – wood float finish.
- Concrete – steel float finish.
- Concrete – steel float finish to receive specified finish:
    - Non-slip
    - Acid resistant epoxy-based paint
    - Epoxy based paint (non-acidic areas)
    - Ceramic tiles

In order to maintain hygiene standards in toilets, showers and locker areas, galvanised mild steel open mesh flooring panels will be specified to cover open drainage channels facilitating ease of removal and cleaning. All flooring panels in wet areas will be painted to prevent corrosion.

18.11.9 Drainage

A positive sewage and storm water drainage system will be installed across the mine site.

Drainage channels and pipes within the building perimeter will be designed to accommodate peak flows without surcharging.

It is unlikely that all showers and other sanitary facilities will operate simultaneously and therefore, a diversity factor will be applied for design purposes derived from industry norms.

Manholes are susceptible to blockage in this environment. In view of the heavy industrial application, manholes of the catch pit type will be constructed to facilitate trapping suspended solids and to facilitate pipe cleansing.

18.11.10 HVAC

Heating should be provided for winter conditions. Heating will be by hot water radiators, or electrically powered space heaters where appropriate.

Ventilation should be provided where necessary, to ensure a clean and healthy environment.

For areas subject to significant quantities of dust, a positive extraction system will be provided.

In general, air conditioning should be provided by bulk evaporative cooling systems.

In exceptional circumstances (for example the Control Room and/or the administration Building) individual refrigerated type air conditioning systems should be provided.

18.11.11 Mechanical and Electrical Workshops

Provision shall be provided for mechanical and electrical repair/overhaul facilities, including the normal maintenance requirements, switchgear overhaul, cable manufacture, welding, and small-scale fabrications.

Each building shall be equipped with suitably sized lifting beams, located by design for the utilisation of manual or small-scale electronic winches according to the designated duty requirements.

It is considered that major overhaul and substantial fabrications and/or repair will be outsourced to the local facilities.

18.11.12 Tyre Bay

Tyre storage and changing facility is considered for inclusion within the surface building infrastructure.

The building shall be equipped with installed lifting beams located by design and suitable for the manual operations and or small scale electrically operated winch systems.

18.12 Compressed Air

18.12.1 Mine Area

There is no requirement for an oversite compressed air reticulation network from centrally located compressors. Compressed air will be provided as required by mobile diesel-powered units to specific locations. If necessary small reticulation networks with suitably sized pipe and fittings may be utilised.

18.12.2 Process Plant Area, Workshops and Tyre Bay

The compressed air requirement for the process plant is achieved via dedicated compressors and reticulation systems. A compressed air ring main system will also supply the Process Plant Workshop.

Ancillary workshops and tyre bay facilities, which may be under the control of a Contractor, will also have dedicated compressed air systems and reticulation installations to suit the requirements.

18.13 Explosives Storage

All of the requisite materials and products will be transported as non-explosive components from a designated supply and manufacturing plant.

The surface magazine complex shall be located a suitable distance from other amenities (500m) and will be utilised to store Ammonium Nitrate prill and emulsion agents and similarly the initiating systems i.e. electric and non-electric detonators and explosive boosters. The storage facilities shall be compliant with the regulatory requirements regarding separation distances and security controls.

An ANFO and Emulsion mixing, and “in-the-hole” delivery vehicle will be located on site and will be utilised to mix and deliver the explosives to the blast holes.

While the potential location of the facility is to be identified it is anticipated that further detail with regard to the layout and access routes will be developed during the course of future studies.

18.14 Diesel Fuel Supply and Distribution

18.14.1 Requirement

The basic requirement is to maintain sufficient refuelling facilities to ensure that the vehicle fleet is serviced as and when necessary. While initially, refuelling may be suitable via mobile containment and transfer facilities, as the fleet and operations expand, designated fuel storage and fuelling stations will be required.

18.14.2 Permanent Storage

The permanent fuel storage facilities will be located in three areas:

- The Contractor mobile equipment area and compound;
- The primary crusher ROM pad; and
- A smaller storage facility at the process plant to primarily service the Emergency generator and refuel non-mining equipment.

18.14.3 Tank Design

The tank design shall be that of a fully bunded shell structure such that all leakage of fuel can be contained within the tank shell. A monitoring system shall be installed to the bund that alarms subject to a pre-determined level being achieved within the shell void. Tanks shall be fitted with suitable vent facilities, inspection manholes and access ladder-ways, gauges and indicators for temperature and level as a minimum requirement.

18.14.4 Refuelling Station

Fuel levels within the tanks, amounts dispensed, to which vehicle, at specific times, to specific operators, shall be recorded by a computerised electronic key systems. All dispensing equipment shall be of leak-proof fail-safe design. Where necessary filtration systems and electronic pumps shall be installed both for dispensing, replenishing and if required rapid transfer of containment.

The refuelling station shall be so constructed as to provide a flat level surface of hard standing. If this cannot be achieved, then consideration shall be taken with regard to forming a concrete floor sufficiently reinforced to withstand repeated vehicular access.

The entire area around the refuelling station shall be protected from accidental fuel spills by a channelled or raised sill at least 150mm deep/high and 150mm wide. This channel shall be inclined to a catchment interceptor or alternatively an area containing removable steel bins for the collection and removal of such a spillage. The channel shall be maintained clear of debris at all times.

Suitable and adequate lighting shall be provided for the entire refuelling area to maintain a high standard of safety and for inspection purposes of both vehicles and refuelling equipment.

18.15 Fire Detection and Alarm

Automatic fire detection and alarm shall be installed in all buildings, switch rooms and re-fuelling areas, to provide warning in the event of an abnormal occurrence of smoke or heat. If appropriate, water sprinkler systems, or automatic dry power extinguishers, will be specified as part of the engineering development process. Any circuits and equipment required for such systems shall be specified as part of any future design development.

In the absence of dedicated fire-fighting reticulation networks, the requirement can be satisfied by either a pumped water supply and portable foam generators or the provision of suitable extinguishers in sufficient quantities.

18.16 Transport of concentrates

An automatic bagging plant shall be located adjacent to the Process plant building for the packaging of concentrate products that will be stored in the concentrate warehouse facility.

The packaged products will be loaded into standard shipping containers, which will be subsequently loaded to conventional road haulage transport for delivery to the Novaya Chara rail despatch facility.

It is anticipated that suitable off-loading and storage facilities can be made available pending onward transport to a final destination.

18.17 Conclusions and Recommendations

- Greater accuracy of infrastructure requirements and costs will be defined at a higher level of study. Some preparation groundwork should be made to design preliminary ground investigation programmes for heavy load bearing items (leach pads, waste dumps, centre line for tailings dam, sulphide mill etc.) once strategic pit designs and optimal process designs have been established.
- Assess the Udokan effect. There can be both opportunities with sharing off-site infrastructure and supply routes and also risks in the pressure that the Udokan power requirements may put on the power line. There may be restrictions on Megawatts available.

19 MARKET STUDIES AND CONTRACTS

19.1 Product Realisation

The main products from the Unkur Project are proposed to be silver bullion and copper cathode or high-grade silver bearing copper concentrate depending on the adopted scenario.

Silver bullion as precious metals is always in demand among the Russian banks.

Copper sulphide concentrate is expected to be produced at 26% copper and average 1,633g/t silver and is considered to be saleable based on typical western smelter contracts, although there has not been detailed characterization of penalty elements.

19.2 Commodity Market Outlook

All costs assumptions and commodity prices used in this study have been estimated as of the end of May 2021.

Table 19.1 provides a summary of commodity prices used in the preliminary economic assessment (PEA) with consideration of the World Bank Commodity Market Outlook.

Latest World Bank's Commodity Market Outlook for metals ('pink sheet' published in April 2021) suggests that early gains in Q1 could flatten as faster-than expected withdrawal of stimulus by some major emerging market economies could pose a downside risk to prices; however, a major infrastructure programme in the United States could support prices for metals, including aluminium, copper, and iron ore. An intensification of the global energy transition to decarbonization could further strengthen demand for metals in the medium term.

According to the Commodity Market Outlook published by the World Bank (as of April 2021), Copper is expected to reach US$8,500 per t in 2021, with silver reaching US$25/oz. Therefore, for the purpose of this study, WAI has adopted the following price decks.

Table 19.1: Commodity Price Assumptions Adopted in the Preliminary Economic Assessment

Scenarios	Consensus Price Assumption (as of May 2021)	Spot Price Assumption (avg. May 2021)
Ag (US$ / oz)	25.00	28.00
Cu (US$ / t)	8,500	10,000
Cu (US$ / lb)	3.86	4.54

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The following section of the study summarises the review and assessment of the environmental and social aspects of the Unkur Project carried out from an international best practice perspective and in consideration of any future development options. The assessment was a desk-top study based on the information provided by the Client and did not include a site visit to the Project area.

20.1.1 Scope of Assessment

The sources of information used to complete the assessment included:

- Baseline Studies of ambient air, surface water, and soil conditions, 2016;
- Technical Report and Preliminary Economic Assessment for the Unkur Copper-Silver Project, Kodar-Udokan, Russian Federation, Tetra Tech 2018;
- Supplement to Prospecting and Evaluation Programme for Unkur Ore Occurrence, 2020;
- Interactive Subsoil Use Map of Russian Federation; and
- Publicly available information.

Environmental and social impacts were assessed against the level of detail available for the initial design phase of works. Based on the limited data available at this stage in the project development, the focus of the assessment was to identify any potential significant environmental or social risks that may impact the future development design, or implementation schedule of the Project. Given the limited data available at this stage it was only possible to have an overview of the Project and focus of the assessment was to identify any potential significant environmental or social risks that may impact the future development design, or implementation schedule of the project. Only when the technological and financial aspects of the Project are better defined can the full extent of environmental and social aspects and impacts be investigated in detail.

20.2 Location Setting

20.2.1 Environmental Setting

The Project is located in the Kalarskiy district in the north of Zabaikalsky Krai some 600km north of the regional centre at Chita. The licence area comprises 53.9km^2 in relatively flat, seasonally swampy taiga terrain in the north from 400-500mRL rising to medium mountain relief up to 1200m in the south approaching the rolling foothills of the Udokan Range. The Kemen River, the right tributary of the Chara River, cuts the licence north-south which is flanked by birch forest. The license area has a plenty of glacial lakes formed in frontal moraines.

The study region has an extremely continental climate with rapid increase of average annual temperatures in spring and rapid reduction in autumn. Permafrost has a significant influence on the thermal mode of the area. The coldest month is January (absolute minimum is down to -50 to -57°C), and the hottest month is July (maximum temperature in Chara valley is 32-35°C). The amount of precipitation fluctuates annually and throughout the year, most precipitation falls during the warm period. Transition from winter to spring is characterized by increase of wind velocity and a reduction of time with no wind conditions. Rivers and brooks become clear of ice in the beginning-middle of May.

The Unkur concession is in the mountainous taiga or forest vegetation that is typical of the zone between 1,100 and 1,400m in the Kalarsky District. Flora and fauna at the deposit and in adjacent areas is typical for Siberian taiga. The Kalarsky district is located in the sparsely populated part of Eastern Siberia where pristine natural communities prevail over vast areas. Absence of large settlements or operations, and extremely low population density in the deposit area has facilitated the preservation of natural terrains and habitats. Rocky outcrops may be present and host to communities such as lichens and moss. Boggy ground is common due to permafrost and shallow rock preventing infiltration. Forest trees include Dahurian Larch, fir trees, birches and Chosenia arbutifolia. Shrubs growing in the forests include red bilberry, blueberry, Arctic raspberry, willow, alder, dwarf arctic birch and dwarf Siberian pine. Faunal species that frequent the taiga habitat include Siberian stag, musk deer, lynx, Gulo gulo, grouse, black-capped marmot with water birds in the valleys. Some species such as bear, ermine, weasel, and small mammals may be found.

In the Kalarsky district, there are 14 regional protected territories that have the status of “natural heritage” according to Russian classification, however none of them is in the proximity to the Project area. Some IUCN Red List species are known to occur in Kalarsky District but no information about whether protected species are found on the concession is available.

The deposit location area has a risk of significant seismic activity with seismicity assessed at 9-12 magnitude by MSK-94 scale (equals to magnitude 7.5 on the Richter scale). Severe earthquake potential in case of an earthquake could trigger geohazards such as rock falls, mudslides, avalanches, and slope failures, all of which will need to be considered with reference to safety and in layout of surface facilities and infrastructure.

20.2.2 Social Setting

The Kalarsky district is the largest and most northern district of the Transbaikal region. The district centre is the Chara settlement, located 650 km to the north of Chita and 20 km from the railway station and settlement of Novaya Chara. Industry and transport are poorly developed in the district, in spite of large potential and various explored mineral deposits. The main transport line of the district is the BAM which passes from the south-west (town of Severobaikalsk) to the north-east (town of Tynda).

The 2020 census of Kalarskiy district gives a population of 7,666, which demonstrates the population is sparse. The population is concentrated mainly along the railroad, in the settlements of Novaya Chara, Syulban, Kuanda, and Ikabia. Other settlements are located at a distance from the railroad: Chara (district centre), Kyust-Kemda, Udokan, Chapo-Ologo, Sredny Kalar and Nelyady. The nearest settlement to the Unkur deposit is the Udokan settlement with population of about 200. Novaya Chara, where more than half of the district population lives, is located 15 km to the north-west from the deposit. Chara settlement, with population of about 2,500, is located 25 km to the north-west from the deposit, Ikabia settlement is at about 20 km to the north-east. Currently, the main sources of employment include jobs associated with the maintenance of the East Siberian Railway, State authorities, and in the service sector.

There are 466 indigenous Evenk people living in the Kalarsky district, mainly in the settlements of Kyust-Kemda, Chapo-Ologo, Nelyaty and Sredny Kalar. These settlements have legislative status of residential and economic areas of indigenous minorities of Russia. The main occupations of the Evenk people are reindeer breeding and hunting.

20.2.3 Archaeology and Modern Social Heritage

Twenty or so archaeological objects (tangible assets) were discovered in the Chara River during the 1970s and 1980s and included settlements, camps, and rock art. In the 1990s, modern religious objects were found by State researchers near Udokan. There is no available archaeological nor cultural heritage information on the Unkur concession.

20.2.4 Site Infrastructure

20.2.4.1 *Location and Access*

The Property is accessed from Chara village and the town of Novaya Chara by the year-round natural road passing along the BAM at 5km from the deposit. The road distance from the site to Novaya Chara is approximately 22km, and to Chara is approximately 33km. In Chara there is an airport with a paved airstrip that accommodates regular flights from Chita, approximately 800km to the southwest. Novaya Chara railway station is accessed by the BAM from Bratsk (1,356km) through the town of Severobaikalsk (637km). In winter snow roads are used to access the city of Chita and the town of Taksimo.

20.2.4.2 *Site Infrastructure*

The Project has only been subject to exploration activity and onsite infrastructure (within the license area) only includes access tracks. However, the Project will benefit from substantial investments that have already been made in infrastructure, particularly by the Udokan mining project which is approximately 30km from Unkur (e.g., the regional airport at Chara, roads, and expansion of facilities in Novaya Chara). A national grid powerline crosses the concession to the northeast and there is a high-voltage substation at Novaya Chara (approximately 10km from the concession) with a 200MW capacity. Established villages and towns will have some capacity to provide goods and services albeit that most materials and equipment for construction will probably be imported from elsewhere in Russia or other countries.

20.3 Environmental Impact Assessment and Permitting

The Unkur Project operates in accordance with ЧИТ02522БР License for geological study, exploration and mining of copper, silver and associate components registered in 02.09.2014 and valid until 31.12.2039.

Currently the deposit is subject to extensive prospecting and evaluation activities that are planned to be completed in the first quarter of 2022. Prospecting and evaluation works include walkovers, trenching, core drilling, sampling, assaying, etc. and are carried out in accordance with the Prospecting and Evaluation Programme approved by the government as evidenced by the expert conclusion report No. 025-02-11-2015 dated 17.08.2015.

Russian legislative requirements to the prospecting/exploration activities do not stipulate a process of environmental impact assessment ('OVOS'), however according to the Rules of Mineral Deposit Exploration Programme Designing approved by the Order No.352 of the Russian Federation Ministry of Natural Resources and Ecology dated 14.06.2016 a prospecting/exploration programme should include an environment protection section describing the work area, nature and scale of potential environmental impact and mitigation measures suggested for the work period.

Prospecting and evaluation Programme for the Unkur deposit is developed in compliance with the mentioned requirement and describes environmental protection measures as evidenced by the provided document.

The deposit development will have a negative environmental impact therefore as the Project develops it will require an OVOS in accordance with the regulations, guidelines, and standards of the Russian Federation. OVOS consists of three main stages and includes public consultations at the stages of initial information submission, environmental impact assessment and preparation of justifications.

Also, current Russian environmental legislation and regulations require industrial environmental monitoring to be carried out within the zone of potential impact of industrial facilities at all stages of project implementation. Sanitary Rules 11-102-97 "Engineering and Environmental Survey for Construction" envisage the following stages of industrial environmental monitoring:

- Pre-construction (zero) monitoring;
- Monitoring at the construction stage (construction monitoring); and
- Monitoring at the operations stage.

Pre-construction (zero) monitoring was conducted in 2016 over the Project area and included preliminary baseline studies of ambient air, surface and groundwater, and soils conditions. The monitoring network included 9 points of snow cover sampling, 13 points of surface water sampling, 11 points of bottom sediments sampling. The results are presented in the provided report.

As for the required environmental permits, on 1 January 2019, in the context of broader reforms of government oversight of various sectors, Russia's environmental permitting regime was revised, with an integrated permitting regime being introduced for the heaviest-polluting industrial facilities. This new regime is based on the existing classification of all emitting facilities into one of four categories, ranging from I to IV (that is, from highest to lowest environmental impact), with different levels of regulatory obligations for each category.

From 1 January 2019, Category I facilities are obliged to obtain a single integrated environmental permit instead of three separate permits for emissions to air, wastewater discharges and waste disposal. Applications can be made from 2019, and there is a transitional period until 1 January 2025.

At the operational stage, the Unkur Project will belong to the Category I facilities and therefore it will require an integrated environmental permit. In view of this WAI recommends considering the application of the best available techniques in the Project design at the early stage to avoid extra environmental charges in the future and ensure further compliance. To advance Project to prefeasibility level it will be necessary to include project permitting requirements and a timeline for environmental approvals including the required operating licenses and if these will cause the project to be deferred or cancelled. Developing an Environmental and Social Action Plan will assist in managing these aspects to ensure PFS requirements are satisfied.

20.4 Environmental and Social Risks

20.4.1 Impact Level Categorisation

Dependent on the future Project funding arrangements, and in accordance with the IFC[8] Performance standards (2012), the Project would be categorised as a Category 'A' project, where it is "likely to have significant adverse environmental impacts that are sensitive, diverse, or unprecedented" and as such an ESIA would be required to be prepared based on comprehensive baselines studies examining the projects potential negative and positive environmental impacts, compared to feasible alternatives.

Data that are usually contained in an OVOS report is largely sufficient to provide a preliminary evaluation of the projects impact on the environment and evaluate the project setting for potentially significant environmental constraints.

Some gaps to international standards exist, and some additional studies will be needed namely:

- Geochemistry;
- Soils;
- Hydrology and hydrogeology;
- Cultural heritage and archaeology;
- Socioeconomic baseline - stakeholder mapping;

[8] International Finance Corporation

- Biodiversity and ecosystem services;
- Climate and Energy Use;
- Air Quality; and
- Noise and vibrations.

20.4.2 Priority Baseline Surveys and Management Plans

20.4.2.1 *Summary*

It is understood that further definition of the deposit is required prior to preparing a Prefeasibility Study. To ensure an effective use of resources the following environmental and social aspects are recommended to be advanced in parallel with the next Project development phase.

20.4.2.2 Hydrology and Hydrogeology

The Project is located within the Kemen and Unkur River catchments in the Udokan Range. The Kemen and Unkur rivers are tributaries of the Chara River and sub-catchments of its large catchment. The water courses hydroregime is unstable. In the periods of spring and autumn floods rivers and brooks are almost unfordable.

As the area is mountainous there is potential for significant flows particularly during the spring thaw of snow, which reaches the highest rate in June. The hydrogeological environment comprises aquifers in superficial deposits such as fluvio-glacial sediments, and moraines; and confined aquifer based on borehole evidence (phreatic surfaces at 140 and 110mbgl). There is a perennial permafrost layer varying in thickness from 200 to 400m; the upper layer varies seasonally but this has not been measured to date.

Based on available information, water resources are potentially plentiful and indeed, managing water will be one of the key environmental issues (quantity and quality) of the Project. Comprehensive groundwater and surface water baseline studies with modelling will be essential so that reliable impact predictions and management methods can be developed. Baseline studies about water quality should consider the local and regional uses of water (domestic, industrial, urban, agricultural, recreational, others) and assess water quality as part of the ecosystem (in relation to the life of plant and animal communities). Quantity must reflect several aspects such as watershed distribution, hydrological processes, and the availability for different water uses at local and regional levels. A detailed understanding of hydrogeology with groundwater modelling is also required to assess dewatering requirements.

The Kemen River is included in the list of the water bodies subject to the regional state supervision in water bodies use and protection. The information about aquatic biodiversity of the river is not available however according to the criteria for being qualified as a water body subject to the regional state supervision it can be assumed that Kemen has a commercial fishing importance or is a habitat of anadromous and catadromous fish species. Therefor the Project might be subject to the requirements of IFC Performance Standard 6 "Biodiversity Conservation and Sustainable Management of Living Natural Resources" and have to compile a Biodiversity Action Plan with the project's mitigation strategy especially if the river diversion is required as part of the mine design.

20.4.2.3 Geochemical Categorisation – Mine Waste

A brief geological description of the Unkur deposit is presented in the Addendum to Prospecting and Evaluation Programme. Primary copper mineralisation is comprised by chalcopyrite, pyrite, bornite, and chalcocite and covellite. Oxide minerals include malachite and brochantite. Accessory minerals include magnetite, magnetite, hematite, and ilmenite. The presence of pyrite, and other associated minerals, suggests that Acid Rock Drainage (ARD) could occur at the Project site.

WAI is not aware that baseline geochemical studies for the deposit have been conducted to date and that the ARD potential has not yet been qualified for either waste or ore. Therefore, a geochemical screening assessment will be required to determine the potential for ARD. Significant design measures may be necessary to mitigate the potential risks from ARD, and these should be integrated into the overall design of the Project. Features of the Project that may require additional design to mitigate the risks from ARD may include the waste rock dump, ROM/ore stockpile, dewatering programme, and conceptual closure designs, all of which may have an impact on the Project implementation.

During the next stage of exploration and Project development, the opportunity to collect representative waste samples for ARD testing will arise. It is recommended that a limited number of waste samples be tested for screening the ARD potential.

20.4.2.4 Socioeconomics Surveys

Information about any form of interaction between the Project owners/operators, the existing other land users, and the nearest communities including the community of Evenks is unavailable. As part of the future development planning for the Project, and in line with international best standards, WAI would recommend stakeholder mapping is completed to determine the number of stakeholders directly or indirectly affected by the development and confirm their legal and cultural status.

Stakeholder mapping and engagement is considered best international practice, and dependent on the future funding mechanism for the Project, will be required to satisfy International Finance Institution requirements. As such, a formalised Stakeholder Engagement Plan (SEP) should be developed as part of the next stage of the Project development.

Considering the distance from the settlements to the license area it is unlikely that direct resettlement will be required as part of the Project development, though this will need to be confirmed. There will however be land use change and local community members that rely on the Project area for resources (e.g. grazing) will need to be assessed, as they will be indirectly impacted by the Project.

About 5% of the study area population are comprised by Indigenous Peoples, Evenks. One of their main livelihoods is reindeer herding that involves moving animals across extensive tracts of land. Affected Communities of Indigenous Peoples may be particularly vulnerable to the loss of, alienation from or exploitation of their land and access to natural and cultural resources, so it is essential to establish if the Evenk herders use or cross land in the concession (for hunting or herding). If so under IFC PS7 the Client will have to will obtain the formal, prior, and informed consent of the Affected Communities of Evenks, and a Livelihood Restoration Plan is likely to be required for the Project. The objectives of PS7 include predicting and avoiding adverse impacts on indigenous communities and respecting and preserving the culture, knowledge, and practices of the communities. Its guidelines would be a key reference for the social studies.

20.4.2.5 Conceptual Mine Closure Plan

International best practice stipulates that prior to the commencement of a project a conceptual mine closure plan, including a satisfactory financing mechanism, should be established to ensure sufficient provision of funds. It is recommended that mine closure and rehabilitation is considered following the next phase of the Project to ensure that any measures regarding closure are integrated into the design of the mine and mine facilities, thus reducing liability costs and potential impacts. In order to comply with International best practice, the conceptual closure plan should be developed as early as possible.

In developing a mine closure plan, and cost estimate, a significant component is associated with earthworks and land reclamation. Planning for closure from the Project outset can lead to cost-effective plans being developed (e.g., costs for rehabilitation of worked out areas of the mine during the operational phase are covered by operating costs rather than a dedicated closure budget). Closure and rehabilitation plans should address planned and unplanned for closure. Post-closure land-use should be included as a topic for discussion during public consultation.

21 CAPITAL AND OPERATING COST ESTIMATE

Capital and operating costs reported this section in US Dollars (US$) are shown in fixed 2021 US Dollars. These costs assumptions have been used in the preliminary economic assessment with no inflation being incorporated.

21.1 Mining - Introduction

The open pit production schedules were used as the basis for cost estimation for the Unkur Deposit. The cost estimates were developed by WAI from its internal cost database.

The calculated costs are estimated to have an accuracy equivalent to a Preliminary Economic Assessment (PEA) level of detail. The study offers a valuable view in determining the merits of pursuing further engineering studies but should not be the sole reference for the purposes of economic decision making.

21.1.1 Open Pit and Underground Mining Capital Cost Estimates

Mining Capital Costs were estimated based on WAI's cost database and project experience of similar operations. A summary of total CAPEX based on initial and sustaining capital, discussed in Section 16.9.3 and 16.9.4, is presented in Table 21.1.

Table 21.1: Summary of Total Mining Capital Costs

	Open Pit (USD'000)	Underground (USD'000)		
		Equipment	Capital Development	Total Cost
CAPEX Scenario 1, Processing Option 3	70,035	105,000	85,000	**190,000**
CAPEX Scenario 2, Processing Option 3	66,465	95,000	85,000	**190,000**
CAPEX Scenario 3, Processing Option 4	80,535	95,000	80,000	**175,000**
CAPEX Scenario 4 (OP only), Processing Options 3	70,035	0		

Underground Capital costs requirements have been outlined in the tables below along with the production schedule.

WAI notes that the OP and UG equipment is proposed to be leased. Leasing payments schedule have been included in the Financial Model accordingly.

A production and costs summary of the different scenarios are presented in Table 21.2 to Table 21.4.

Table 21.2: Scenario 1: Open Pit Oxide Only, followed by Underground Mining. Processing Op 3

		Total	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
OP																
Waste	t	97,420,044	13,431,340	31,789,436	47,676,998	4,522,270										
Ore oxide	**t**	**10,980,525**	**1,500,000**	**3,500,000**	**3,500,000**	**2,480,525**										
Cu grade	%	0.59	0.63	0.56	0.54	0.68										
Ag grade	g/t	48.24	46.21	42.90	44.20	62.72										
Ore sulphide	**t**	**0**														
Cu grade	%	0.00														
Ag grade	g/t	0.00														
Stripping ratio	t/t	**8.9**	8.95	9.08	13.62	1.82										
Total Ore OP		**10,980,525**	**1,500,000**	**3,500,000**	**3,500,000**	**2,480,525**	**0**	**0**	**0**							
UG																
Oxide Ore	**t**	**4,066,589**				**1,342,071**	**1,397,518**	**1,290,764**	**36,235**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Cu	%	0.71				0.63	0.71	0.80	0.76							
Ag	g/t	41.34				26	35	64	66							
Sulphide Ore	**t**	**16,712,017**				**43**	**589,666**	**710,000**	**2,000,000**	**2,000,366**	**2,000,000**	**2,000,000**	**2,002,047**	**2,000,000**	**2,000,000**	**1,409,895**
Cu	%	0.77				1	1	1	1	1	1	1	1	1	1	1
Ag	g/t	53.11				13	29	64	64	62	59	59	57	40	40	46
UG CAPEX	US$ M	**190**		50	75	25	0	0	0	25	0	0	0	0	15	0
Equipment (UG)	US$ M	**105**			50	25				20	0				10	
UG Capital Development	US$ M	**85**		50	25					5					5	
Total Ore UG	**t**	**20,778,606**	**0**	**0**	**0**	**1,342,114**	**1,987,184**	**2,000,764**	**2,036,235**	**2,000,366**	**2,000,000**	**2,000,000**	**2,002,047**	**2,000,000**	**2,000,000**	**1,409,895**
Overall Ore OP & UG	**t**	**31,759,131**	**1,500,000**	**3,500,000**	**3,500,000**	**3,822,639**	**1,987,184**	**2,000,764**	**2,036,235**	**2,000,366**	**2,000,000**	**2,000,000**	**2,002,047**	**2,000,000**	**2,000,000**	**1,409,895**

Table 21.3: Scenario 2: Oxide and Sulphide Combined Open Pit, followed by Underground. Processing Option 3

		Total	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**
OP														
Waste	t	228,138,962	23,905,732	43,599,317	43,009,900	48,142,187	35,389,217	26,707,062	7,385,547					
Ore oxide	**t**	**11,949,639**	**1,094,268**	**1,400,683**	**1,990,100**	**3,500,000**	**3,500,000**	**464,588**						
Cu grade	%	0.57	0.56	0.59	0.53	0.55	1	1						
Ag grade	g/t	46.72	34.75	41.57	43.86	47.83	52	53						
Ore sulphide	**t**	**7,020,898**				**48**	**386,358**	**3,500,000**	**3,134,492**					
Cu grade	%	0.74				0.75	0.75	0.77	0.71					
Ag grade	g/t	67.48				67	67	68	68					
Stripping ratio	t/t	**12.0**	21.85	31.13	21.61	13.75	9.11	6.74	2.36					
Total Ore OP		**18,970,537**	1,094,268	1,400,683	1,990,100	3,500,048	3,886,358	3,964,588	3,134,492					
UG														
Oxide Ore	**t**	**3,282,956**						1,472,222	1,420,073	390,661	0	0	0	0
Cu	%	0.68						0.64	0.70	0.75				
Ag	g/t	34.12						27	36	54				
Sulphide Ore	**t**	**11,331,557**						43	589,666	1,611,169	2,000,000	2,000,284	2,000,000	3,130,395
Cu	%	0.74						1	1	1	1	1	1	1
Ag	g/t	39.69						13	29	36	46	45	40	36
UG CAPEX	US$ M	**175**				50	75	25	0	0	0	25	0	0
Equipment (UG)	US$ M	**95**				0	50	25				20		
UG Capital Development	US$ M	**80**				50	25					5		
Total Ore UG	**t**	**14,614,514**	0	0	0	0	0	1,472,264	2,009,740	2,001,830	2,000,000	2,000,284	2,000,000	3,130,395
Overall Ore OP & UG	**t**	**33,585,051**	**1,094,268**	**1,400,683**	**1,990,100**	**3,500,048**	**3,886,358**	**5,436,852**	**5,144,232**	**2,001,830**	**2,000,000**	**2,000,284**	**2,000,000**	**3,130,395**

Table 21.4: Scenario 3: Oxide and Sulphide Combined Open Pit, followed by Underground. Processing Option 4

		Total	1	2	3	4	5	6	7	8	9	10	11	12
OP														
Waste	t	254,164,978	24,175,226	42,966,534	61,113,508	61,046,107	59,985,229	4,878,374						
Ore oxide	**t**	**10,232,932**	824,774	2,033,466	3,500,000	2,454,950	1,419,742	0						
Cu grade	%	0.66	0.77	0.66	0.66	0.68	1							
Ag grade	g/t	54.10	55.39	56.57	54.20	55.00	48							
Ore sulphide	**t**	**7,530,834**				1,498,943	3,500,000	2,531,891						
Cu grade	%	0.74				0.68	0.75	0.77						
Ag grade	g/t	66.06				61	67	68						
Stripping ratio	t/t	**14.3**	29.31	21.13	17.46	15.44	12.19	1.93						
Total Ore OP		**17,763,766**	**824,774**	**2,033,466**	**3,500,000**	**3,953,893**	**4,919,742**	**2,531,891**						
UG														
Oxide Ore	**t**	**3,282,956**						**1,472,222**	**1,420,073**	**390,661**	**0**	**0**	**0**	**0**
Cu	%	0.68						0.64	0.70	0.75				
Ag	g/t	34.12						27	36	54				
Sulphide Ore	**t**	**11,331,557**						**43**	**589,666**	**1,611,169**	**2,000,000**	**2,000,284**	**2,000,000**	**3,130,395**
Cu	%	0.74						1	1	1	1	1	1	1
Ag	g/t	39.69						13	29	36	46	45	40	36
UG CAPEX	US$ M	**175**				50	75	25	0	0	0	25	0	0
Equipment (UG)	US$ M	**95**				0	50	25				20		
UG Capital Development	US$ M	**80**				50	25					5		
Total Ore UG	**t**	**14,614,514**	**0**	**0**	**0**	**0**	**0**	**1,472,264**	**2,009,740**	**2,001,830**	**2,000,000**	**2,000,284**	**2,000,000**	**3,130,395**
Overall Ore OP & UG	**t**	**32,378,280**	**824,774**	**2,033,466**	**3,500,000**	**3,953,893**	**4,919,742**	**4,004,155**	**2,009,740**	**2,001,830**	**2,000,000**	**2,000,284**	**2,000,000**	**3,130,395**

21.1.2 Operating Cost Estimates

Open pit operating costs were estimated by WAI based on the generated production schedule, equipment operating cost estimates, consumable price estimates and labour estimates.

WAI notes that in order to reduce CAPEX, but maintain control of the mining fleet in the financial analysis, the financial model considered leasing of the OP and UG equipment required to carry out the production schedule from suppliers. Operating costs for these additional items of equipment have been increased to allow for the interest payment on leasing. Where possible a residual value has been applied as available for sales.

Estimated overall open pit costs are in the region of US$1.75/t rock mined, based on a separate earthmoving cost for ore (US$/t) and lower cost for hauling waste (US$/m^3) from data recently derived from WAI from benchmarking real costs from similar-sized operations in the region.

21.2 Underground Operating Costs

A fixed underground cost of US$25.0/t ore mined has been accepted in the financial analysis, including US$22/t of the operation cost and US$3/t for backfilling. These costs have been derived by benchmarking against similar scale operations in the Zabaikal Region. The backfill has been included to reduce losses. Costs are considered to be adequate for the level of scale of this study.

21.3 Processing Costs

21.3.1 Option 1: Agitated Cyanide Leach - SART

Tetra Tech estimated the capital costs for Option 1 as us$128.14 million based on a throughput of 3.27Mtpa.

For 3.5Mtpa and using recent Cost Mine data, a 5,000tpd agitated leaching plant with Merrill Crowe and including a starter TSF facility has an estimated capital cost of US$108.5 million. Therefore, applying the 6/10ths rule, for 10,000tpd (3.5Mtpa) the capital cost is estimated as US$164.5 million. However, the cost of the Merrill Crowe circuit must be replaced by that for a two-stage SART circuit and the TSF costs removed.

From published data, the capital costs of SART plants based on US$/m^3/h capacity vary widely, but are typically in the range of approximately 30 - 90 k$/m^3/h. With the requirement for a potential two-stage SART process, a figure of 60 k$/m^3/h is selected.

For a throughput of 3.5Mtpa, 91% plant availability (439tph), a head grade of 0.61% Cu and 43g/t Ag, recoveries of 55.1% Cu and 91.1% Ag, and assuming a solids content in leaching of 45%, this equates to a SART feed volume rate of approximately 537m^3/h. The feed metals grades then calculate as approximately 2,748ppm Cu and 32ppm Ag.

Therefore, the capital cost calculates as 60,000 * 537 = **US$32.2 million**.

Tetra Tech estimated a turnkey cost for the Merrill Crowe plant at US$25.85 million. Furthermore, the starter TSF capital cost was estimated as US$6.6 million.

Therefore, the revised scoping level Option 1 capital cost for the process plant (ex TSF) is 164.5 - 25.85 - 6.6 + 32.2 = <u>US$164.3 million.</u>

The main difference between this and the Tetra Tech estimate (US$128.1 million) is the higher estimated cost for the SART plant (Tetra Tech estimate US$3.9 million).

With the relatively low level of silver in solution, it may be more economical to consider a one-stage SART process for production of a copper concentrate with silver credits. This can be determined through future testwork.

For 3.5Mtpa and using recent Cost Mine data, a 5,000tpd agitated leaching plant with Merrill Crowe and including a starter TSF facility has an estimated operating cost of US$17.99/t. Therefore, applying the 6/10ths rule, for 10,000tpd (3.5Mtpa) the operating cost is estimated as US$11.9/t. However, the operating cost of the Merrill Crowe circuit must be replaced by that for a two-stage SART circuit and the TSF costs also removed. The operating cost for the TSF is estimated as US$0.6/t and the Merrill Crowe plant at US$0.31/t.

Published data indicates a wide range of operating costs based on m^3 of solution treated by SART, varying from US$0.4/$m^3$ to US$6/$m^3$ for high metals levels. Operating costs are dictated by the levels of metal and free cyanide in solution. In this case, the copper level is estimated at approximately 2,748ppm Cu which is very high (silver levels are relatively low) and free cyanide levels are also very high, as the testwork indicated a minimum cyanide concentration of 0.5% required for acceptable metals recoveries. Therefore, a cost of US$6/$m^3$ is selected. For 537m^3/h of solution and 439tph solids throughput, this calculates as US$7.3/t.

The main operating cost in the SART process is the reagent consumptions, with NaSH consumption being the main contributor.

Therefore, the revised scoping level Option 1 operating cost for the process plant (ex TSF) is 11.9 - 0.6 – 0.31 + 7.3 = <u>US$18.3/t.</u>

This is in reasonable agreement with the Tetra Tech estimate of US$19.2/t.

21.3.2 Option 2: Sequential Acid and Cyanide Agitated Leaching

The estimated capital cost for this hybrid circuit using Cost Mine data and Tetra Tech data from the PEA where appropriate, and for 3.5Mtpa, is US$229.8 million. This is a slight increase compared to the Tetra Tech estimate of US$187 million. The operating cost estimate is US**$27.4/t**, very similar and slightly lower than the Tetra Tech estimate of US$28.6/t. The acid consumption represents the largest component of this cost – assuming an acid consumption of 90kg/t and an acid price of US$90/t, this equates to approximately US$8.1/t ROM ore for the acid consumption alone.

21.3.3 Option 3: Cyanide Heap Leach - SART

Tetra Tech estimated the capital costs for Option 3 as US$77.4 million based on a throughput of 3.27Mtpa.

Using WAI's database of heap leach projects, the capital cost for a 3.5Mtpa conventional heap leach operation is estimated as US$44.5 million. However, the conventional carbon-in-solution columns, carbon elution/regeneration and electrowinning/refining circuits for silver bullion production will be replaced by the two-stage SART circuit. The capital cost component for the conventional carbon treatment circuit is estimated at US$7.2 million.

For the SART plant, the estimated flowrate to be treated is 250 m^3/h (based on an irrigation rate of 10 l/h/m^2 and a pad size of typically 250m x 50m with two pads concurrently in use (irrespective of the maximum pad stack height)). Therefore, for the same capital cost as for Option 1, i.e., 60 k$/$m^3$/h, the capital cost for the SART plant is estimated as 60,000 * 250 = US**$15 million.**

Therefore, the revised scoping level Option 3 capital cost for the process plant is 44.5 – 7.2 + 15 = <u>US$52.3 million.</u>

This is slightly lower than the Tetra Tech estimate of $77.4 million.

For a throughput of 3.5Mtpa, 91% plant availability (439 tph), a head grade of 0.61% Cu and 43 g/t Ag, recoveries of 55.9% Cu and 73.4% Ag, and a SART feed volume rate of 250m^3/h, the average feed metals grades calculate as approximately 5,988ppm Cu and 55ppm Ag.

Applying the same operating costs estimate as for Option 1, i.e., US$6/$m^3$, then for 250$m^3$/h of solution and 439 tph solids throughput, this calculates as US**$3.4/t.**

From WAI's database of projects, the operating cost for a conventional 3.5Mtpa heap leach operation is estimated as US$5.9/t. The operating cost component for the conventional carbon treatment circuit is estimated as US$3.5/t

Therefore, the revised scoping level Option 3 operating cost for the process plant is 5.9 – 3.5 + 3.4 = US$5.8/t.

This is lower than the Tetra Tech estimate of US$13.9/t.

21.3.4 Option 4: Sequential Acid and Cyanide Heap leaching

The estimated capital cost for this hybrid circuit using WAI data and Tetra Tech data from the PEA where appropriate, and for 3.5Mtpa, is **US$97.3 million**. This is a decrease compared to the Tetra

Tech estimate of US$128.9 million. The operating cost estimate is estimated at US**$15.4/t**, similar but slightly lower than the Tetra Tech estimate of US$22.5/t. The acid consumption represents a significant component of this cost – assuming an acid consumption of 54 kg/t and an acid price of US$90/t, this equates to approximately US$4.7/t ROM ore for the acid consumption alone.

A summary table of the key parameters for all four oxide ore processing options is shown in Table 21.5.

Table 21.5: Physicals for Oxide Processing Options

OXIDE ORE	Option 1 (Agitated Leach)	Option 2 (Sequential Agitated Leach)	Option 3 (Heap Leach)	Option 4 (Sequential Heap Leach)
Copper Recovery, %	55.1	95.0	55.9	80.6
Silver Recovery, %	91.1	95.0	73.4	73.4
Capital Cost, $M	164.3	229.8	52.3	97.3
Operating Cost, $/t	18.3	27.4	5.8	15.4

A simple excel NPV calculation shows that Option 3 delivers a significantly higher NPV compared to Option 1 (approximately 40% higher on a relative difference basis).

21.3.5 Sulphide processing Costs

A processing rate has been selected of 3.5Mtpa for open pit (OP) mining and 3.0Mtpa for underground (UG) mining.

The copper concentrate grade is estimated as 25.8% Cu, containing 1,634g/t Ag (testwork results).

From Cost Mine data, the capital cost for a 10,000 tpd single-product flotation plant is **US$93.2 million** and the operating cost is **US$8.98/t**. These costs are considered reasonable for scoping level accuracy.

21.4 Infrastructure Costs

21.4.1 Capital Costs

Estimated Capital Costs for on-site and off-site infrastructure outside of the process plant battery are presented in Table 21.7. Power costs are based on an average of ₽2.0 per kW hour supplied from the Federal grid.

Costs are presented in Table 21.6 and built on the basis of an oxide plus sulphide operation. The costs are reduced to a nominal 30% assuming a reduced oxide-only footprint for Scenario 1.

21.4.2 Operating Costs

Estimated Operating Costs are presented in Table 21.8 and estimated on an annual basis also for a maximum oxide plus sulphide operation. The costs are also reduced for a nominal reduced oxide-only footprint for Scenario 1 summarized in Section 20.3.5.

21.4.3 Cost Summary

Table 21.6 shows the summary of infrastructure costs for Scenarios 1 and 2.

Table 21.6: Infrastructure Costs

	Scenario 1 oxide only	Scenario 2 oxide + sulphide	
	US$	₽	US$
CAPEX Estimate	**10,000,000**	**2,365,891,370**	**30,171,505**
OPEX Estimate/annum	**1,500,000**	**297,150,322**	**4,015,545**

- The reticulation network is not fully detailed but nominal values are included in order to register the requirement.
- The water requirement sheet is not populated as it is only a function of demand to establish the contingent power requirement i.e. to recycle tailings water at 800m^3/hr
- Heat values have been utilised previously for Russian projects but the definitive requirement will undoubtedly alter.
- All building sizes and requirements are estimated but are deemed necessary in some form or other to satisfy the project needs.

Table 21.7: Infrastructure Capital Costs

Item	Manufacturer/Supplier	Source	Quantity	Length	Width	Wall height	Ridge Height	Total Quantity	Unit	Unit price (US$)	Total price (US$)	Total price ₽
Mine Site Infrastructure												
Planning												
Engineering & Design		Estimate	1					1	LS	2,459,109	2,459,109	181,974,066
Procurement of equipment		Estimate	1					1	LS	1,000,000	1,000,000	74,000,000
											3,459,109	**255,974,066**
Construction Phase												
Site preparation for construction	Contractor	Estimate	1					1	LS	1,500,000	1,500,000	111,000,000
Site Offices & Welfare Temporary facility (included above)	Contractor		1					1	LS			
Power 2 x 2Mw generators	Contractor	Database	2					2	LS	410,000	820,000	60,680,000
Water & Sewerage Containerised facilities? Small scale?	Contractor	Estimate	2					250	m^3	170	42,500	3,145,000
Concrete batch plant 45m^3/hr	Contractor	Database	1					1	LS	90,896	90,896	6,726,304
											2,453,396	**70,551,304**
Permanent facilities Roads etc												
Main site access roadway	Contractor	Estimate	1					3500	m	400	1,400,000	103,600,000
Over site roads	Contractor	Estimate	1					11000	m	400	4,400,000	325,600,000
Light vehicle parking	Contractor	Estimate	1					3000	m^2	250	750,000	55,500,000
											6,550,000	**484,700,000**
Electrical Power												
4km HV transmission line	by design requirement	Estimate	1					4	km	150,000	600,000	44,400,000
Electrical Substation	by design requirement	Estimate	1	40	20	5	7	1	LS	1,500,000	1,500,000	111,000,000
Electrical MCC	by design requirement	Estimate	1	20	10	4	6	1	LS	1,000,000	1,000,000	74,000,000
Site reticulation	by design requirement	Estimate	1					1	LS	1,500,000	1,500,000	111,000,000
Oversite lighting	by design requirement	Estimate	1					2	LS	125,000	250,000	18,500,000

Table 21.7: Infrastructure Capital Costs

Item	Manufacturer/Supplier	Source	Quantity	Length	Width	Wall height	Ridge Height	Total Quantity	Unit	Unit price (US$)	Total price (US$)	Total price ₽
Mine Site Infrastructure												
											4,850,000	**358,900,000**
Fuel Farm												
Fuel storage and transfer equipment	by design requirement	Estimate	4					4	LS	187,500	750,000	55,500,000
Fuel distribution network	by design requirement	Estimate	2					2	LS	50,000	100,000	7,400,000
Fuel dispensation and control systems	by design requirement	Estimate	3					3	LS	30,000	90,000	6,660,000
											940,000	**69,560,000**
Buildings etc												
Water reservoirs, treatment facilities.	by design requirement	Estimate	2					6,000	m³	300	1,800,000	133,200,000
Re-agent storage warehouses etc.	by design requirement	Estimate	2	35	20	5	7	1400	m²	300	420,000	31,080,000
Welfare facilities.	by design requirement	Estimate	1	35	20	4	6	700	m²	1,800	1,260,000	93,240,000
Administration building.	by design requirement	Estimate	1	40	30	7	9	1200	m²	1,000	1,200,000	88,800,000
Laboratory.	by design requirement	Estimate	1	30	15	4	6	450	m²	1,000	450,000	33,300,000
General and technical services building.	by design requirement	estimate	1	40	20	4	6	800	m²	500	400,000	29,600,000
Electrical sub-stations/transformer buildings	by design requirement	Estimate	3	7	5	5	7	105	m²	500	52,500	3,885,000
MCC Buildings	by design requirement	Estimate	3	7	5	4	6	105	m²	500	52,500	3,885,000
Vehicle repair workshops	by design requirement	Estimate	1	50	40	12	15	2000	m²	1,000	2,000,000	148,000,000
Vehicle wash-down facility	by design requirement	Estimate	2	18	10	5	7	360	m²	100	36,000	2,664,000
Mechanical/electrical workshops	by design requirement	Estimate	2	20	15	6	8	600	m²	1,000	600,000	44,400,000
Security building, lighting, and fences.	by design requirement	Estimate	1	8	5	5	7	40	m²	500	20,000	1,480,000
Weighbridge and attendant facility.	by design requirement	Estimate	1	8	5	5	7	40	m²	500	20,000	1,480,000
Spare parts warehouse	by design requirement	Estimate	2	30	20	5	7	1200	m²	400	480,000	35,520,000
Tyre bay	by design requirement	Estimate	1	20	15	5	7	300	m²	400	120,000	8,880,000

Table 21.7: Infrastructure Capital Costs

Item	Manufacturer/Supplier	Source	Quantity	Length	Width	Wall height	Ridge Height	Total Quantity	Unit	Unit price (US$)	Total price (US$)	Total price ₽
Mine Site Infrastructure												
Water supply facilities, reservoirs, tanks, pump houses etc.	by design requirement	Estimate	2	8	3			48	m²	1,000	48,000	3,552,000
Sewage treatment.	by design requirement	Estimate	2	8	3	4		48	m²	1,000	48,000	3,552,000
Fuel storage and distribution facilities.	In fuel farm											0
Fire station and Ambulance station.	by design requirement	Estimate	2	20	5	4	6	200	m²	400	80,000	5,920,000
Explosives magazine.	by design requirement	Estimate	1	20	10	5	7	200	m²	600	120,000	8,880,000
Medical centre	by design requirement	Estimate	1	10	7	4	6	70	m²	400	28,000	2,072,000
											9,235,000	**683,390,000**
Water supply and oversite reticulation												
Water supply	by design requirement	Estimate	1					3000	m	300	900000	66600000
Pump requirement ≈800m³/hr	by design requirement	Estimate	2					4		600,000	2400000	177600000
Raw water storage facilities Constructed Concrete	by design requirement	Estimate	2					1000	m³	300	300,000	22,200,000
Raw water treatment plant (infeed) ≈200m³/hr	by design requirement	Estimate	1					200	m³	170	34,000	2,516,000
Process water recirculation facility		Estimate	1					2,400	m³	170	408,000	30,192,000
Reticulation pumping scheme ≈800m³/hr	by design requirement	Estimate	1					3000	m	35	105,000	7,770,000
Potable water storage & distribution 100m³ Tank	by design requirement	Estimate	1					2	LS	38,000	76,000	5,624,000
Industrial water storage & distribution 1500m³ Tank	by design requirement	Estimate	2					2	LS	120,000	240,000	17,760,000
Over site reticulation	by design requirement	Estimate	1					3000	m	35	105,000	7,770,000
Domestic Waste-water recovery (reticulation)	by design requirement	Estimate	1					3000	m	35	105,000	7,770,000
Domestic Waste-water treatment	by design requirement	Estimate	1					300	m³	170	51,000	3,774,000
Industrial/Tailings and open pit water recovery	by design requirement	Estimate	1					3000	m	35	105,000	7,770,000

Table 21.7: Infrastructure Capital Costs												
Mine Site Infrastructure												
Item	**Manufacturer/Supplier**	**Source**	**Quantity**	**Length**	**Width**	**Wall height**	**Ridge Height**	**Total Quantity**	**Unit**	**Unit price (US$)**	**Total price (US$)**	**Total price ₽**
Industrial/Tailings Waste-water treatment	by design requirement	Estimate	1					2400	m^3	375	900,000	66,600,000
Heating Reticulation 2 pipe system Ring Main	by design requirement	Estimate	2					6000	m	30	180,000	13,320,000
Hot water Reticulation 2 pipe system Ring main	by design requirement	Estimate	2					3000	m	25	75,000	5,550,000
											2,684,000	**442,816,000**
											30,171,505	**2,365,891,370**

Table 21.8: Infrastructure Operating Costs										
Item	**Requirement**	**Quantity**	**Installed Power (kW)**	**Total Installed power (kW)**	**System Efficiency (%)**	**Absorbed Power (kW)**	**Utilisation (%)**	**Demand (hrs/yr)**	**kWh/y**	**Cost at ₽2/kWh**
Mine Infrastructure										
Electrical Power										
4km HV transmission line	Maintenance only									
Electrical Substation	lighting, heating maintenance	1	6	6	90	5	15	1,314	7,096	14,191
Electrical MCC	lighting, heating maintenance	1	6	6	90	5	15	1,314	7,096	14,191
Site reticulation	Maintenance only			0						
Oversite lighting	lighting, maint'	1	50	50	85	43	35	3,066	130,305	260,610
				62		**53**			**144,496**	**288,992**
Fuel Farm										
Fuel storage and transfer equipment	Included in Buildings section									
Fuel distribution network	Included in Buildings section									
Fuel dispensation and control systems	Included in Buildings section									
Process Plant										
Process Building (not equipment)by design	All equipment	1	20000	20000	85	**17,000**	85	7,446	**126,582,000**	**253,164,000**
Process water storage and distribution	Included in water and buildings									
ReAgent storage building	Included in buildings									
Raw ore stockpile and delivery equipment	Included in Mineral circuit									
				20000		**17,000**			**126,582,000**	**253,164,000**
Comminution and Mineral circuit prior to Plant										
Primary crushers	lighting, heating maintenance	1	200	200	85	170	90	7,884	1,340,280	2,680,560
Vibro Feeders	lighting, heating maintenance	1	20	20	85	17	90	7,884	134,028	268,056
overband magnet	lighting, heating maintenance	1	8	8	85	7	90	7,884	53,611	107,222
Conveyors to Dome	lighting, heating maintenance	1	90	90	85	77	90	7,884	603,126	1,206,252

Table 21.8: Infrastructure Operating Costs

Item	Requirement	Quantity	Installed Power (kW)	Total Installed power (kW)	System Efficiency (%)	Absorbed Power (kW)	Utilisation (%)	Demand (hrs/yr)	kWh/y	Cost at ₽2/kWh
Geo dome storage facility	lighting, heating maintenance	1	6	6	90	5	90	7,884	42,574	85,147
Vibro Feeders dome discharge	lighting, heating maintenance	3	20	60	56	34	90	7,884	264,902	529,805
Conveyor to Process plant stockpile area or to secondary crushing	lighting, heating maintenance	1	25		85	0	90	7,884	0	0
Ore and Concentrate stockpiles/bins etc.	lighting, heating maintenance									
				384		**309**			**2,438,521**	**4,877,042**
Buildings etc										
Water reservoirs, treatment facilities.	lighting + small pumps	**2**	6	12	95	11	40	3,504	39,946	79,891
Re-agent storage warehouses etc.	lighting	**2**	5	10	80	8	30	2,628	21,024	42,048
Welfare facilities.	lighting + small pumps	**1**	50	50	80	40	35	3,066	122,640	245,280
Administration building.	lighting + small pumps	**1**	30	30	80	24	35	3,066	73,584	147,168
Laboratory.	Specific + lighting + HVAC	**1**	10	10	90	9	50	4,380	39,420	78,840
General and technical services building.	lighting HVAC	**1**	20	20	95	19	55	4,818	91,542	183,084
Electrical sub-stations/transformer buildings (satellite)	specific	**3**	0.5	1.5	90	1.35	25	2,190	2,957	5,913
MCC Buildings (satellite)	specific	**3**	3	9	85	7.65	30	2,628	20,104	40,208
Vehicle repair workshops	specific	**1**	30	30	85	25.5	25	2,190	55,845	111,690
Vehicle wash-down facility	lighting	**2**	5	10	95	9.5	45	3,942	37,449	74,898
Mechanical/electrical workshops	specific	**2**	20	40	85	34	50	4,380	148,920	297,840
Security building, lighting, and fences.	lighting HVAC	**1**	6	6	85	5.1	20	1,752	8,935	17,870
Weighbridge and attendant facility.	specific	**1**	6	6	85	5.1	20	1,752	8,935	17,870
Spare parts warehouse	lighting	**2**	2	4	85	3.4	15	1314	4467.6	8,935
Tyre bay	lighting, compressor, hand tools	**1**	3	3	85	2.55	20	1752	4467.6	8,935

Table 21.8: Infrastructure Operating Costs

Item	Requirement	Quantity	Installed Power (kW)	Total Installed power (kW)	System Efficiency (%)	Absorbed Power (kW)	Utilisation (%)	Demand (hrs/yr)	kWh/y	Cost at ₽2/kWh
Water supply facilities, reservoirs, tanks, pump houses etc.	General requirement	**2**	35	70	80	56	75	6,570	367,920	735,840
Sewage treatment.	Biological treatment	**2**	15	30	80	24	25	2190	52560	105,120
Fuel storage and distribution facilities.	Pumps and control circuitry		20	0	85	0	95	8,322	0	0
Fire station and Ambulance station.	lighting	**2**	3	6	85	5.1	95	8,322	42,442	84,884
Explosives magazine.	lighting	**1**	3	3	85	2.55	95	8,322	21,221	42,442
Medical centre	lighting, general requirement	**1**	7	7	85	5.95	95	8,322	49,516	99,032
				357.5		**299.15**			**1,213,895**	**2,427,790**
				20,804		**17,662**			**130,378,913**	**260,757,825**
Estimated Heat requirement for buildings				1,382		1,313			14193802	28,387,603
Estimated Heat requirement for reticulation				112		106			734,237	1,468,474
Estimated fuel cost for concentrate transport										6,536,420
				22,298		**19,081**			**145,306,951**	**297,150,322**
									US$	**4,015,545**

22 FINANCIAL ANALYSIS

22.1 Overview

WAI has undertaken a Preliminary Economic Assessment of the Unkur Project, using Discounted Cash Flow (DCF) analysis, from which the Net Present Value (NPV), payback period and other measures of project viability have been determined.

The financial analysis has been performed to reflect valuation as of April 2021.

The Project Financial Model ("Model") has been developed using the production schedule developed by WAI, with all costs being estimated in 2021 US Dollars based on the available databases.

All costs and cash flows reported in this section are shown in fixed US Dollars, with no inflation being incorporated.

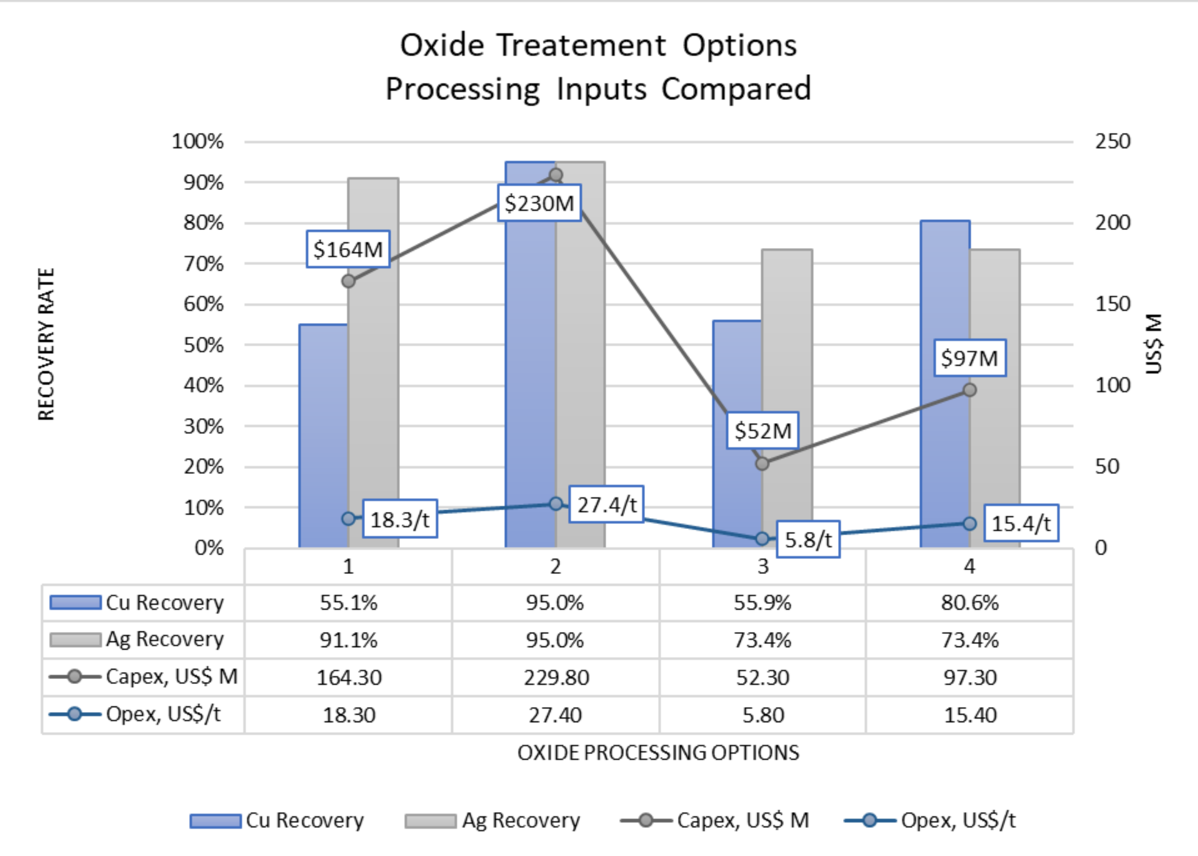
The Preliminary Economic Assessment is preliminary in nature, it is based on Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

22.2 Methodology of the Project Analysis

WAI has undertaken a Trade-off study in order to identify the most viable option for solely Open Pit operation at Unkur Deposit at various oxide processing options. A summary of the considered scenarios for the open pit operation of the oxide and sulphide ores.

- Scenario 1 is an oxide open pit operation with Cyanide Heap Leach-SART treatment (Option 3). This option is associated with the lowest capital costs requirements for oxide processing plant, infrastructure, and mining fleet;
- Scenario 2 is a combined oxide and sulphide open pit operation with the following oxide processing options (at up to 3.5Mtpa capacity):
    - Option 1: Agitated Leach
    - Option 2: Tank Leach SX/EW
    - Option 3: Heap Leach SART
    - Option 4: Heap Leach SX/EW

Summary of key processing parameters for the Oxide Operation is shown in Figure 22.1.



	1	2	3	4
Cu Recovery	55.1%	95.0%	55.9%	80.6%
Ag Recovery	91.1%	95.0%	73.4%	73.4%
Capex, US$ M	164.30	229.80	52.30	97.30
Opex, US$/t	18.30	27.40	5.80	15.40

Figure 22.1: Processing Inputs and Assumptions Compared for the Selected Oxide Options

Option 2 allows for the highest recovery for Copper and Silver at 95%, at the price of the highest capital and operating costs of US$230M and US$27.4/t respectively.

Option 1 provides 95% recovery for Silver, but only 55% recovery for Copper. This options is also associated with high capital cost requirements of US$164M and relatively high operating costs (US$18.3/t), however, the Cu recovery is significantly less than at the Option 2.

Based on the performed economic analysis Options 1 and 2 did not prove to be viable for the Unkur Project. Therefore, WAI selected processing Options 3 and 4 for the further analysis, with open pit works being followed by the underground development. Thus, the financial analysis results, reported in this section are provided for the following Options:

- **Scenario 1 (OP & UG)**: Open Pit (Oxide Only) followed by Underground with Processing Option 3;
- **Scenario 2 (OP & UG)**: Open Pit (Oxide and Sulphide) followed by Underground with Processing Option 3; and
- **Scenario 3 (OP & UG)**: Open Pit (Oxide and Sulphide) followed by Underground with Processing Option 4.

Key Inputs and Assumptions

The Sales Revenue has been estimated on the production of the following commodities:

Oxide:

- Processing Option 1 and 3 consider production of a combined concentrate with 68% Copper grade and rich silver content (from oxide operation).
- Options 2 and 4 consider production of silver bullions and cathode copper (from oxide operation)

Sulphide:

- Sulphide ores are considered to be treated at the separate Plant with 3.5Mtpa designed capacity and a production of a standard coper concentrate containing 26% Cu.

WAI notes that the lower mining rate maintained at UG operation (at 2Mtpa) has been reflected in the analysis as associated with a smaller plant capital cost and adjusted operating costs.

Commodity Price Assumptions have been considered as following:

- Consensus Prices: US$8,500/t (US$3.86/lb) for Copper and US$25/oz for silver; and
- Spot Prices (May 2021): Copper at US$10,000/t and Silver at 28/oz.

Consensus Price Deck has been selected by the Client as the Base Case of this analysis and fits well with the expected long term price forecasts.

Realisation Terms for copper and silver have been based on benchmarked data and products assumed to be sold to a smelter located in China or Kazakhstan.

Due to the limited data on impurities contained in concentrates, no penalties have been included in this valuation.

- 98.0% - Payable for Cu value in Concentrate;
- 97.0% - Payable for Ag value in Concentrate;
- 99% Payable for Cu Cathode and Ah Bullions (Processing Option 2 and 4);
- Treatment Charge (TC) at US$ 21/t of 68% Cu Concentrate;
- Treatment Charge (TC) at US$ 115/t of 26% Cu Concentrate;
- 0.07/lb Cu Refining Charge;
- 0.5/oz Ag Refining Charge;
- Concentrate Moisture content at 8%; and
- Concentrate Transportation Cost at US$ 15/wmt.

Fiscal Regime and Taxes

WAI has estimated both Pre-Tax and Post Tax NPV and IRR. Summary of the taxes applied in the financial analysis is outlined below.

The Unkur Project is expected to be benefiting from the tax relief available at the current government programme. Thus, Corporate Income Tax for the first five years would be estimated at 0%, with the following five years at 10% (reduced rate) and 20% (basic rate) thereafter. This is subject to the Company being able to sign up for TOR status.

Mineral Extraction Tax is also subject to the tax relief programme. The base case tax rates are 8% and 6.5% for payable Cu and Ag values, respectively. The tax relief has been accounted for in the current financial model as following:

- 0% of the base rate has been applied for the first two years of operation,
- followed by 80% discount to the base rates for the following two years,
- 60% for the following two years,
- And so on, gradually increasing to the full base tax rates by year 11.

WAI notes that the Mineral Extraction Tax (MET) can potentially be higher than considered in the current financial analysis due to ongoing changes in the Russian Fiscal Legislation. Thus, there is a possibility that a factor of 3.5x will be introduced in additional to the existing royalty charges.

The decision on tax benefits is made on the individual project basis following the application submitted to the special Committee. WAI considers that given the Unkur project location and utilisation of the innovation processing technology (SART Leaching), there is a good chance that the Unkur project will benefit from the tax relief.

WAI notes that fiscal regime for the Unkur Project should be monitored closely during subsequent studies of the project development, and in the event of failing to gain approval for the tax holidays, the Unkur Project financial results reported within this report would be decreased.

No other taxes have explicitly been included in the Unkur Project economic assessment, given the high level of the study. Other taxes and fees are assumed to be covered under the operating cost assumptions, including social tax, land and property tax, environmental fees etc.

All capital costs are estimated exclusive of VAT, and no VAT rebate is considered in the financial model.

Depreciation has been assumed at flat 9.5% rate based on the capital expenses. No sunk costs or depreciation balance has been included in valuation.

Summary of the Project Operating Costs:

Mining operating costs are seen to be comprising the largest part of the overall Operating Costs structure (Figure 22.2). Operating costs summary is outlined below:

- Mining **OP** operating cost at US$ 1.75/t rock;
- Mining **UG** operating cost at US$ 22/t ore;
- Backfilling **(UG)** cost at US$ 3/t ore;
    - Processing operating costs for **oxide** as per outlined options:
    - Option 3: US$ 5.80/t (production target 3.5Mtpa); and
- Option 4: US$15.40/t (estimated for the production target of 3.5Mtpa);
- Processing operating costs for **sulphide** material at US$ 8.98/t (for 3.5Mtpa production or
- US$ 12.6/t for smaller sulphide plant with 2Mtpa capacity);
- G&A and General Infrastructure cost at US$ 9Mpa (including 4Mtpa for Infrastructure);
- Mineral Extraction Tax as outlined above.

Working Capital was estimated on the basis of two months value of the annual production costs.

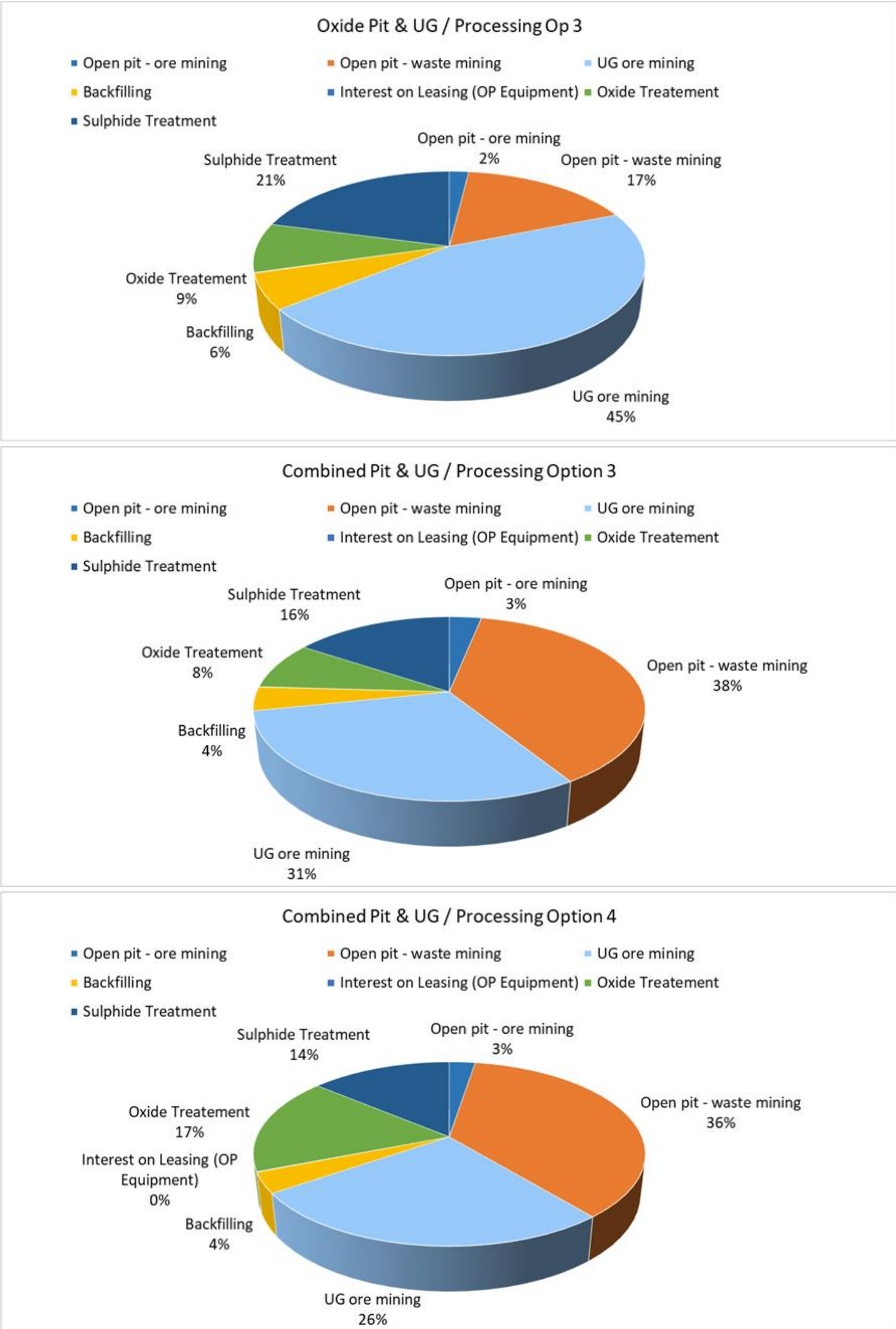


Figure 22.2: Project Operating Costs Pie Chart for the Respective "OP & UG Scenarios"

A summary of the Project Capital Costs is shown in Table 22.1:

Table 22.1: Total Project Capital Costs Summary (Including Sustaining)

Scenario (OP & UG)	1 (OP oxide only, UG)	2 (Ox & Sulph OP, and UG)	3 (Ox & Sulph OP, and UG)	4 (Oxide, OP Only)
Processing Option	**Option 3**	**Option 3**	**Option 4**	**Option 3**
Mining Capital Costs OP (Leased)	70,035	66,465	80,535	70,035
Capitalised Interest on OP Leased Equipment	136	451	286	136
Mining UG Capital Costs	190,000	175,000	175,000	0
Processing (Oxide)	52,300	52,300	97,300	52,300
Processing Sustaining (Oxide)	3,138	3,923	7,298	785
Processing (Sulphide)	66,618*	93,200	93,200	0
Processing Sustaining CAPEX (Sulphide)	8,993	9,786	9,786	0
TSF Wall Rise	2,831 *	3,240	3,240	0
General Infrastructure Initial CAPEX	10,057	30,172	30,172	10,057
Closure and Reclamation Cost (US$0.5/t ore mined)	16,000	17,000	16,000	5,000
Contingency (at 15% of Initial Costs)	57,472	62,638	71,474	19,879
Assets Residual Value**	-19,671	-12,200	-22,112	-13,671
Total Capital Expenditures	451,909	501,974	562,178	144,521

* Note: WAI notes that for the Oxide Only OP followed by UG and Processing Option 3 the capacity of the sulphide Plant is expected to be 2Mtpa versus the designed 3.5Mtpa, assumed for the open pit operations. To reflect this, the capital and operating costs for the sulphide processing and TSF have been adjusted for a lower throughput requirement.
** Residual Value represents a value of the equipment leased at the end of the mine life, where the period of use has been less than five years (maximum assumed useful life). This is a preliminary and indicative assumption. More detailed leasing programme will be included in the analysis at the following study stages.

22.3 Financial Results Summary

The full version of the Financial Model is presented as Appendix 1 of this report. The Unkur DCF Model has been developed with the number of drivers and switches that could be flexed to provide a summary of the results for all considered scenarios and options, particularly with the most sensitive indices. Thus, the model will enable to switch between selected scenarios, metal price forecasts and tax benefits options.

WAI notes that the Unkur Project financially benefits from the involvement of the underground works for sulphide ore mining. A summary of the key production and cost inputs along with the comparative results is provided in Table 22.2 through Table 22.4.

Table 22.2: Financial Model Inputs Summary

Scenario		1	2 & 3		4
Mining Option		**Oxide Only OP, Followed by UG**	**Oxide & Sulphide OP, Followed by UG**		**OP Only**
Processing Option		**Option 3**	**Option 3**	**Option 4**	**Option 3**
LOM	years	14	12	12	4
Open Pit Operation					
Oxide Ore	t'000	10,981	11,950	10,233	10,981
Cu Grade	%	0.59%	0.57%	0.66%	0.59%
Ag Grade	g/t	48.24	46.72	54.10	48.24
Cu In-situ	t	64,846	67,562	67,425	64,846
Ag In-situ	kg	529,734	558,261	553,581	529,734
Sulphide	t'000		7,021	7,531	
Cu Grade	%		0.74%	0.74%	
Ag Grade	g/t		67.48	66.06	
Cu In-situ	t		52,183	56,017	
Ag In-situ	kg		473,787	497,488	
Total Ore Mined (OP)	t'000	**10,981**	**18,971**	**17,764**	**10,981**
Waste	t'000	97,420	228,139	254,165	97,420
Rock	t'000	108,401	247,109	271,929	108,401
Strip Ratio	W:O	8.87	12.03	14.31	8.87
Underground Operation					
Oxide Ore	t'000	4,067	3,283	3,283	
Cu Grade	%	0.71%	0.68%	0.68%	
Ag Grade	g/t	41.34	34.12	34.12	
Cu In-situ	t	28,963	22,234	22,234	
Ag In-situ	kg	168,105	112,030	112,030	
Sulphide	t'000	16,712	11,332	11,332	
Cu Grade	%	0.77%	0.74%	0.74%	
Ag Grade	g/t	53.11	39.69	39.69	
Cu In-situ	t	128,129	83,522	83,522	
Ag In-situ	kg	887,600	449,772	449,772	
Total Ore Mined (UG)	t'000	**20,779**	**14,615**	**14,615**	
Combined Operation					
TOTAL ORE MINED OP & UG	t'000	**31,759**	**33,585**	**32,378**	
Total Mined Cu	t	221,939	225,502	229,199	
Total Mined Ag	oz	50,973,028	51,243,475	51,855,025	

Table 22.3: Cash Flow Summary (Consensus Prices)					
Mining Option		**Oxide Only OP, Followed by UG**	**Oxide & Sulphide OP, Followed by UG**		**OP Only, No UG**
Processing Option		**Option 3**	**Option 3**	**Option 4**	**Option 3**
LOM	years	14	12	12	4
Cu Price	US$/t	8,500	8,500	8,500	8,500
Ag Price	US$/t	25	25	25	25
Net Revenue (after sales and realisation)	**US$'000**	**2,254,346**	**2,295,195**	**2,530,740**	**605,104**
Less Operating Costs:					
Mining	US$'000	-712,647	-802,645	-845,748	-193,302
Processing	US$'000	-297,229	-253,154	-377,529	-63,687
G&A	US$'000	-70,000	-60,000	-60,000	-20,000
Closure and Reclamation Fees*	US$'000	0	0	0	0
Infrastructure	US$'000	-56,218	-48,187	-48,187	-16,062
Royalty (Mineral Extraction Tax)	US$'000	-69,495	-82,588	-78,321	-4,984
Total Operating Costs	US$'000	**-1,205,589**	**-1,246,574**	**-1,409,785**	**-298,036**
Project EBITDA	US$'000	1,048,757	1,048,621	1,120,955	293,540
Taxable Income	US$'000	774,636	860,049	829,243	261,435
Less Corporate Income Tax (w/Tax Holidays)	US$'000	-69,499	-84,070	-54,283	
Net Operating Profit	US$'000	979,258	964,551	1,066,672	293,540
Total Project Capital Expenditures	US$'000	-451,909	-501,974	-562,178	-144,521
Free Cash Flow to Firm	US$'000	528,048	462,889	504,997	149,020
NPV at Discount Rate of 8.00%	US$'000	205,562	143,400	193,442	95,122
IRR	%	26.7%	16.4%	20.9%	46.3%
Payback period of capital (FCF)	Years	6	7	5	3

* Note: Closure and reclamation annual fees are subject to local regulations, established following a submission of the budgeted Closure Plan by the Company. Current version of evaluation considers a one off payment payable at the end of the mine life for the purpose of PEA study.

Table 22.4: Cash Flow Summary (Spot Prices)					
Mining Option		**Oxide Only OP, Followed by UG**	**Oxide & Sulphide OP, Followed by UG**		**OP Only**
Processing Option		**Option 3**	**Option 3**	**Option 4**	**Option 3**
LOM	years	14	12	12	4
Cu Price	US$/t	10,000	10,000	10,000	10,000
Ag Price	US$/t	28	28	28	28
Net Revenue (after sales and realisation)	US$'000	**2,615,851**	**2,664,215**	**2,940,763**	**694,768**
Less Operating Costs:					
Mining	US$'000	-712,647	-802,645	-845,748	-193,302
Processing	US$'000	-297,229	-253,154	-377,529	-63,687
G&A	US$'000	-70,000	-60,000	-60,000	-20,000
Closure and Reclamation Fees*	US$'000				
Infrastructure	US$'000	-56,218	-48,187	-48,187	-16,062
Royalty (Mineral Extraction Tax)	US$'000	-80,370	-95,567	-90,749	-5,734
Total Operating Costs	US$'000	-1,216,464	-1,259,553	-1,422,213	-298,785
Project EBITDA	US$'000	1,399,387	1,404,662	1,518,550	382,455
Taxable Income	US$'000	1,125,266	1,183,309	1,216,014	350,350
Less Corporate Income Tax (w/Tax Holidays)	US$'000	-102,507	-116,300	-81,144	0
Net Operating Profit	US$'000	1,296,880	1,288,362	1,437,406	382,455
Total Project Capital Expenditures	US$'000	-451,909	-501,974	-562,178	-144,521
Free Cash Flow to Firm	US$'000	845,670	786,701	875,732	237,935
NPV at Discount Rate of 8.00%	US$'000	380,410	322,637	412,729	162,231
IRR	%	44.4%	25.6%	34.1%	70.1%
Payback period of capital (FCF)	Years	4	6	5	2

* Note: Closure and reclamation annual fees are subject to local regulations, established following a submission of the budgeted Closure Plan by the Company. Current version of evaluation considers a one off payment payable at the end of the mine life for the purpose of PEA study.

A performed preliminary economic analysis shown that at the Consensus Prices Scenario 1 results in the highest NPV and IRR numbers, benefiting from the lower capital and operating costs that compensate lower processing recovery rates (compared to the Processing Option 4: Heap Leach SX/EW).

However, Scenario 3 (Processing Option 4) takes an advantage at the higher price deck. As the trade-off between higher processing capital, operating costs and increased recovery rate works towards project economics improvement at the higher commodity prices applied.

WAI recommends that the options with an involvement of the underground works are selected for further analysis at the later stages of the project development.

A summary of the Financial Results shown for the Consensus and Spot Price Decks are presented in 3 Figure 22.3 and Figure 22.4 respectively.

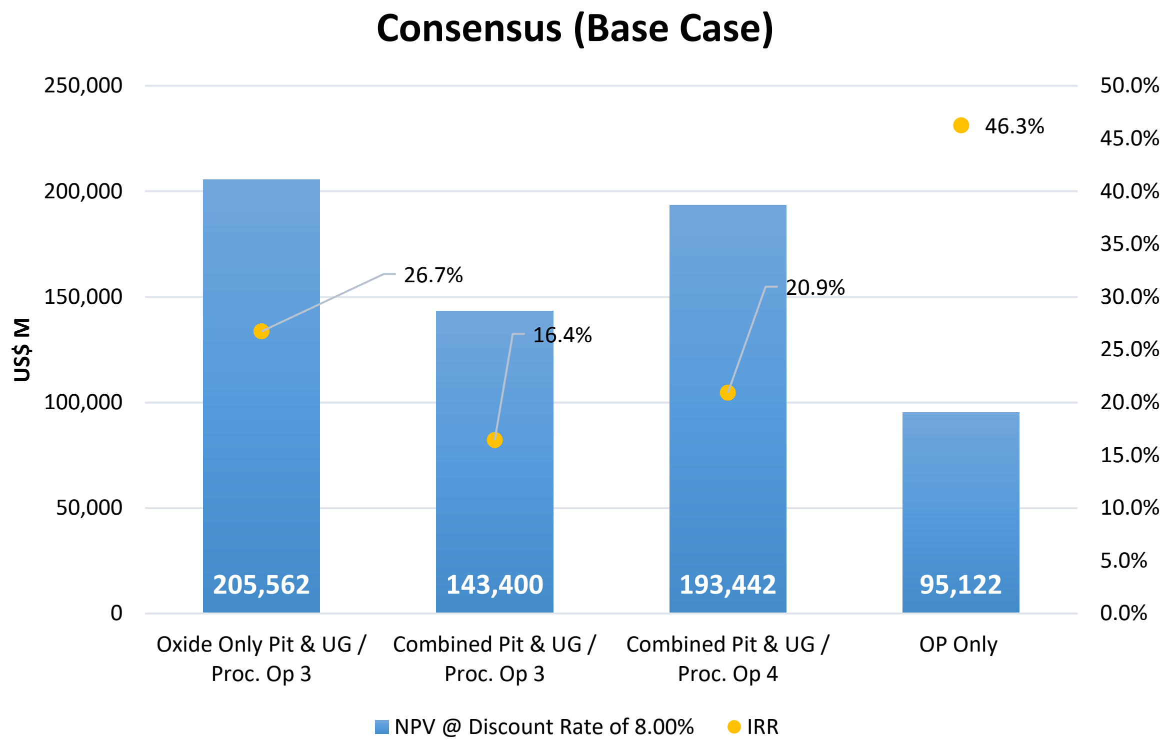


Figure 22.3: Financial Results shown for the Consensus Prices Deck

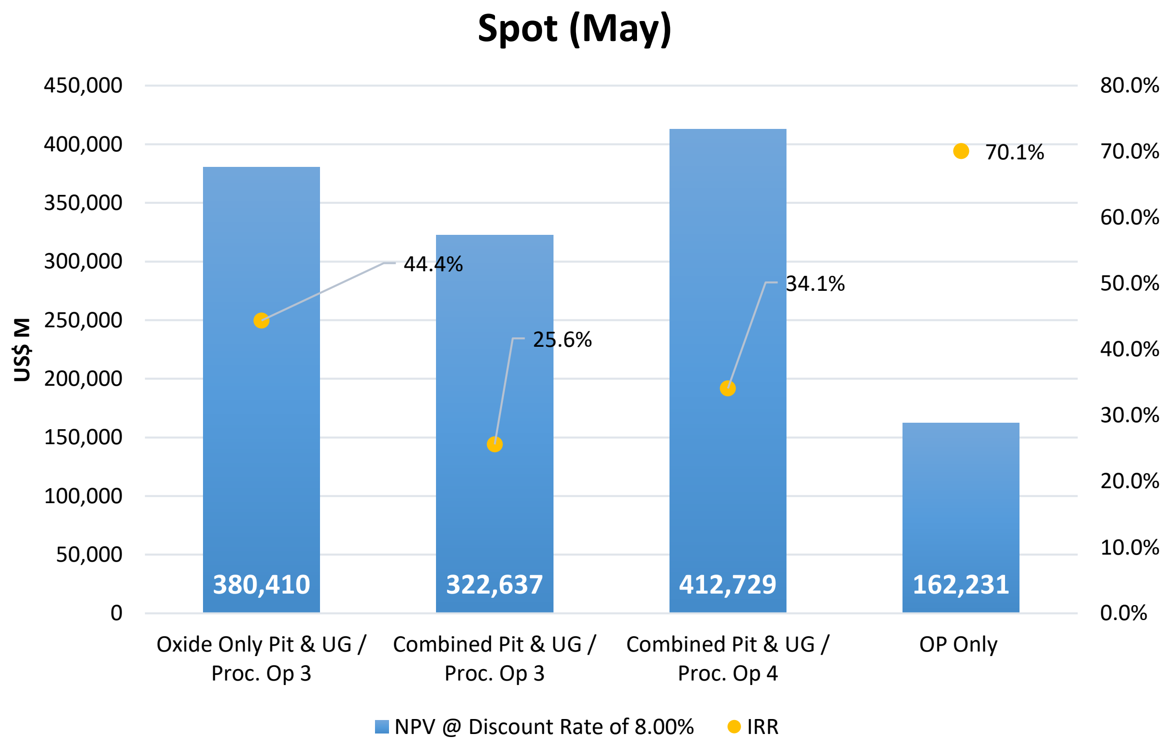


Figure 22.4: Financial Results shown for the Spot Prices Deck

The current project timeframe considers that the oxide ore processing capacity is reached fully in year two, with underground feed being added from year four / six (subject to the selected Scenario); and sulphide plant target feed being achieved in year seven / nine.

Capital costs for sulphide plant are assumed to be funded by cash generated from the open pit operation. Capital required for underground development and equipment, as well as sulphide plant construction is assumed to be paid back in three/ four years following investment (project year five / seven) once the production target is fully achieved.

Summary of the Initial Capital Costs, required to get the Unkur project into the full production are shown in Table 22.5.

Table 22.5: Initial Capital Expenditures Summary

Capital Costs		1	2	3	4
Stage I: Oxide only, OP	**US$'000**	**152,407**	**159,479**	**239,536**	**152,407**
OP Mining Equipment (Initial Payments for Leasing)	**US$'000**	70,035	55,755	80,535	70,035
Capitalised Interest Rate on Leasing (pre-production period)	**US$'000**	136	451	286	136
Oxide Plant	**US$'000**	52,300	52,300	97,300	52,300
General Infrastructure	**US$'000**	10,057	30,172	30,172	10,057
Contingency at 15%	**US$'000**	19,879	20,802	31,244	19,879
Stage II: inclusion of Sulphide Material, and UG	**US$'000**	**249,111**	**279,680**	**279,680**	**0**
Underground Capital Development	**US$'000**	75,000	75,000	75,000	0
Underground Equipment Cost	**US$'000**	75,000	75,000	75,000	0
Sulphide Plant	**US$'000**	66,618	93,200	93,200	0
Contingency at 15%	**US$'000**	32,493	36,480	36,480	0
Residual Value Sales	**US$'000**				
Total Initial Capital Cost	**US$'000**	**401,518**	**439,159**	**519,216**	**152,407**
Including Pre-production Capital Cost; payable in Year 0	**US$'000**	77,615	103,355	154,915	77,615
Residual Value	**US$'000**	-19,671	-12,200	-22,112	-13,671

The Financial Results are presented in Figure 22.5 to Figure 22.10.

For the Scenario 1, the initial capital cost required to enable **Stage I** (OP operation) achieving full production capacity is estimated at **US$152.4M**, this includes US$70M for OP equipment, US$52.3M for Oxide Plant and US$10M for general infrastructure. CAPEX required to get **Stage II** in full production (scheduled for Year 3 onwards) has been estimated at **US$249M,** and includes 75M for UG capital development, US$75M for UG Equipment, and Sulphide Plant Cost of US$66.6M. These costs also include contingency at 15%, and capitalised interest rate, assumed for leasing programme.

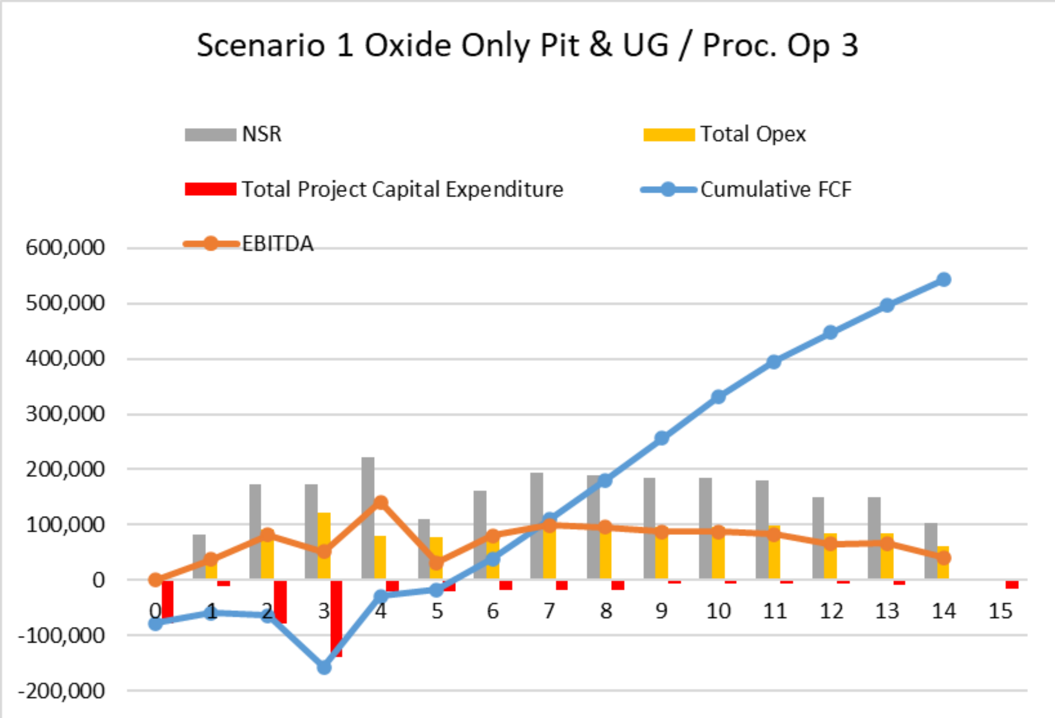


Figure 22.5: Financial Results for Scenario 1, Consensus Prices

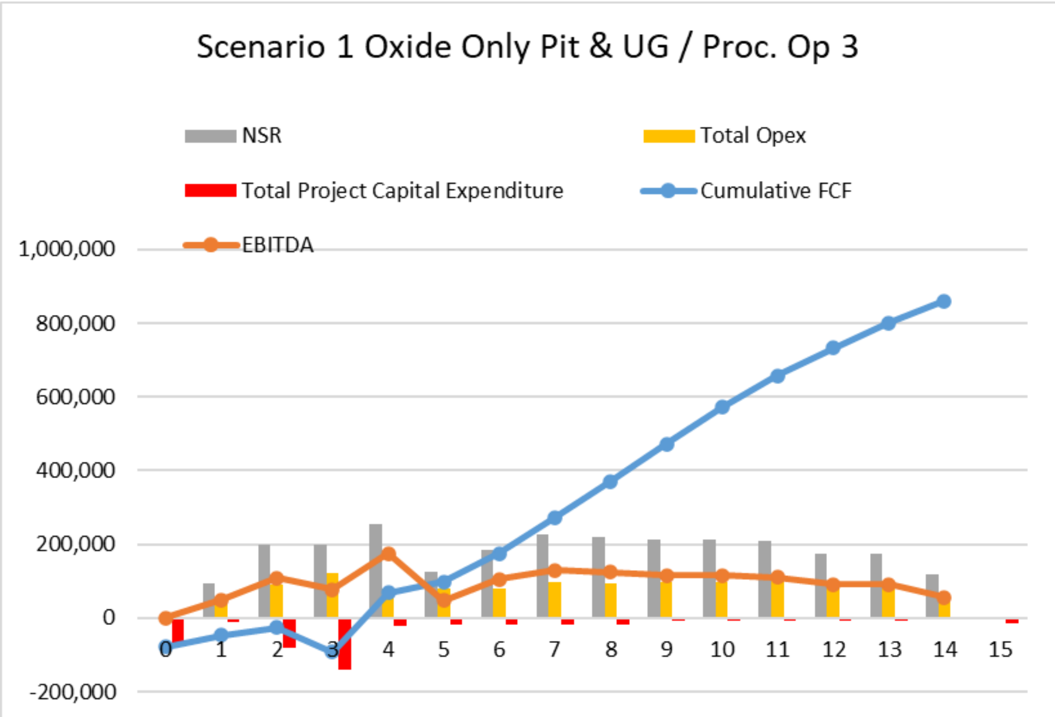


Figure 22.6: Financial Results for Scenario 1, Spot Prices

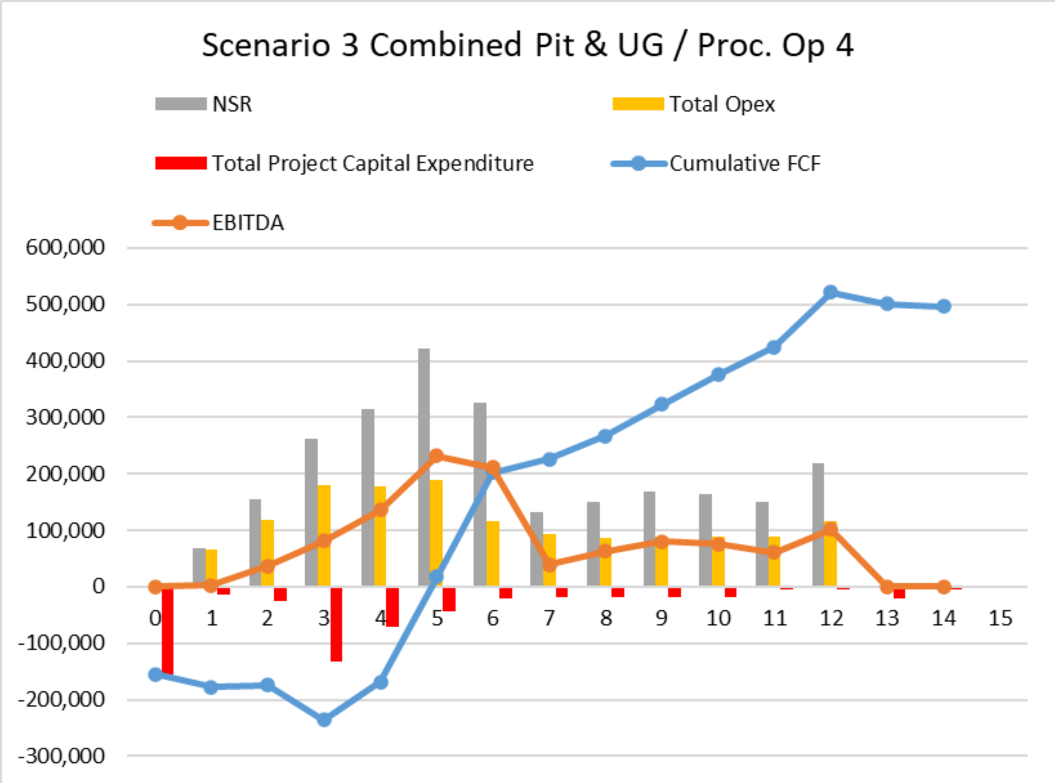


Figure 22.7: Financial Results for Scenario 3, Consensus Prices

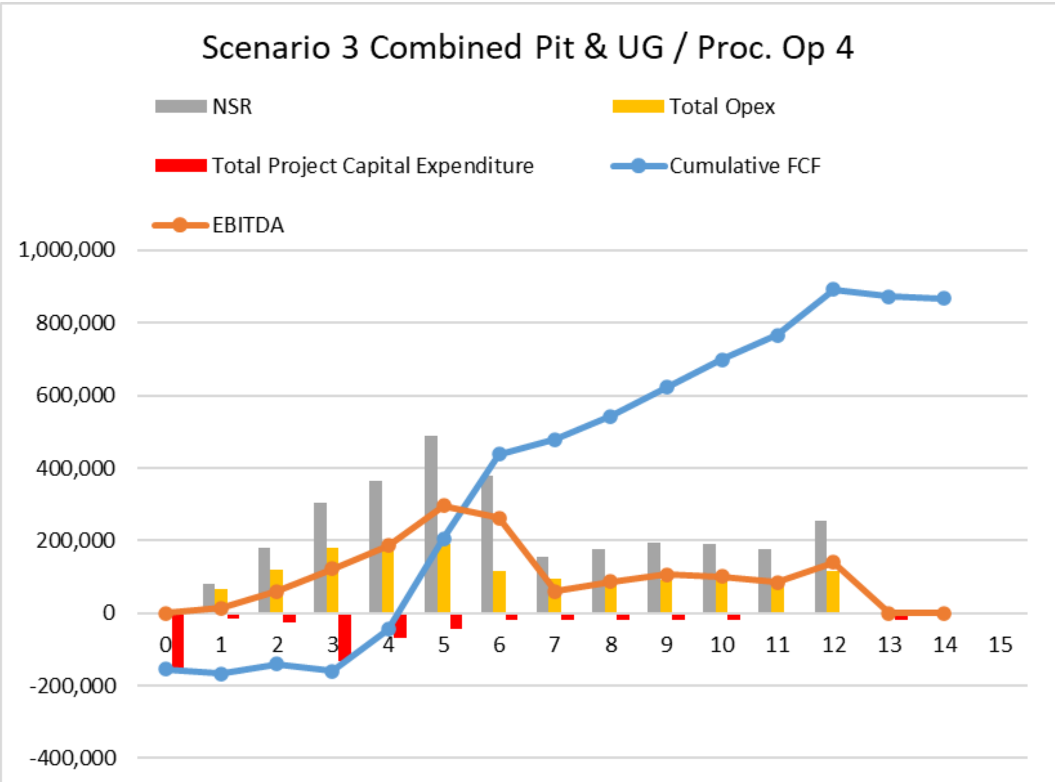


Figure 22.8: Financial Results for Scenario 3, Spot Prices

Hence the Scenario 3 is associated with more expensive processing method (Heap Leach SX/EW) and bigger open pit (given that both oxide and sulphide materials are mined via OP prior switching to the UG), it attracts highest capital costs required for Oxide Processing Plant, compared to all other scenarios.

The initial capital cost required for enable Stage I (OP operation) and achieve production capacity target is estimated at **US$239.5M**, this includes US$80.5M for OP equipment (dictated by the bigger pit), US$97.3M for Oxide Plant and US$30M for general infrastructure.

CAPEX required to get Stage II in full production (allocated from Year 3 onwards) has been estimated at **US$249M,** and includes 75M for UG capital development, US$75M for UG Equipment, and Sulphide Plant Cost of US$66.6M (including contingency at 15%, and capitalised interest rate, assumed for leasing programme).

Following the above, for the Standalone operation of Oxide Materials (Scenario 4) the capital requirements have been estimated at **US$152.4M**, this includes **US$70M** for OP equipment, **US$52.3M** for Oxide Plant and **US$10M** for general infrastructure (including contingency at 15%, and capitalised interest rate, assumed for leasing programme).

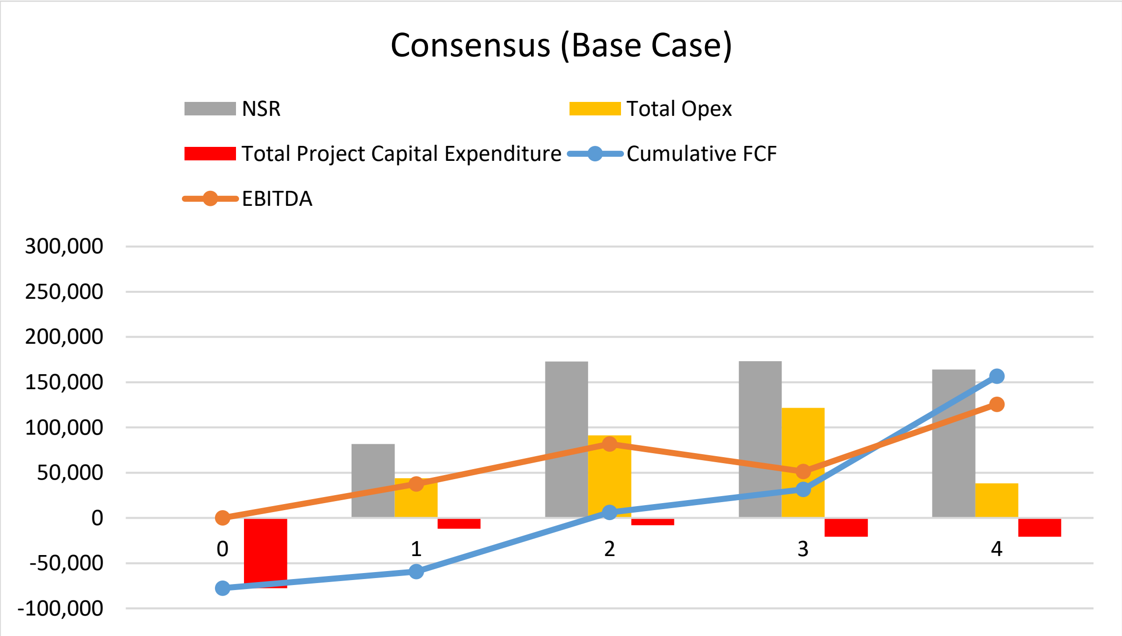


Figure 22.9: Financial Results for Scenario 4, Consensus Prices

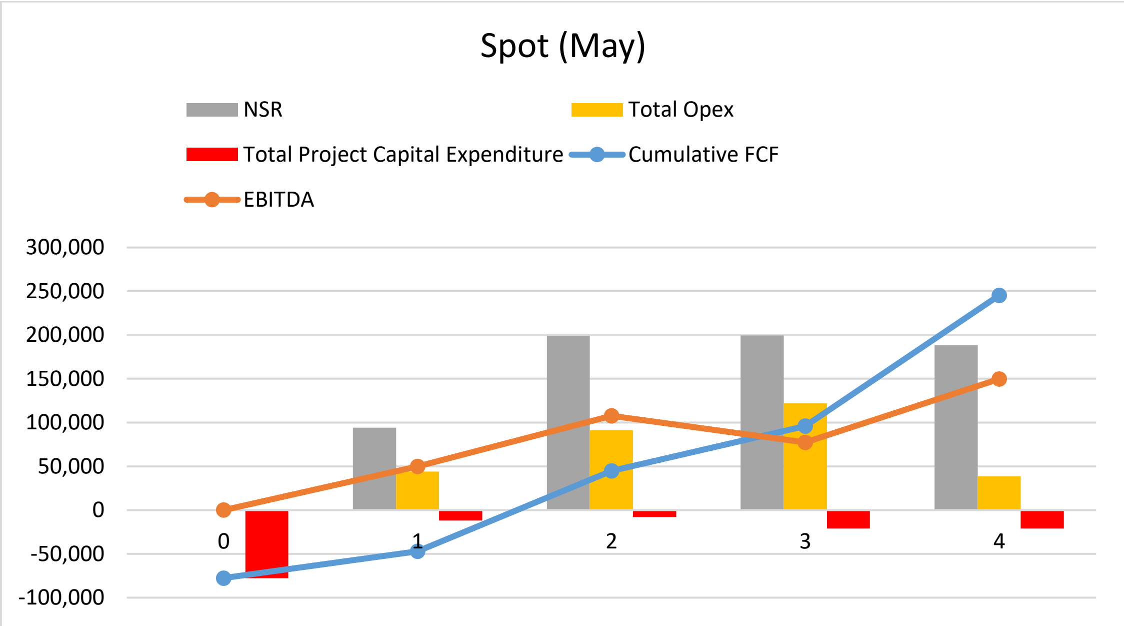


Figure 22.10: Financial Results for Scenario 4, Spot Prices

22.4 Sensitivity Analysis

A sensitivity analysis was performed on the key parameters within the financial model to assess the impact of changes upon the Net Present Value of the project (at a base case 8% discount rate). These parameters are as follows:

- Cu Price;
- Ag Price;
- Mining Opex (OP);
- Mining Opex (UG);
- Processing Opex (Oxide);
- Processing Opex (Sulphide);
- G&A;
- Infrastructure; and
- Project CAPEX.

Each factor was variated within a range of +/25% (while other parameters remained unchanged) to examine the sensitivity of the model to changing economic and operational conditions.

Sensitivity Analysis Results are shown in Figure 22.11 and Figure 22.12.

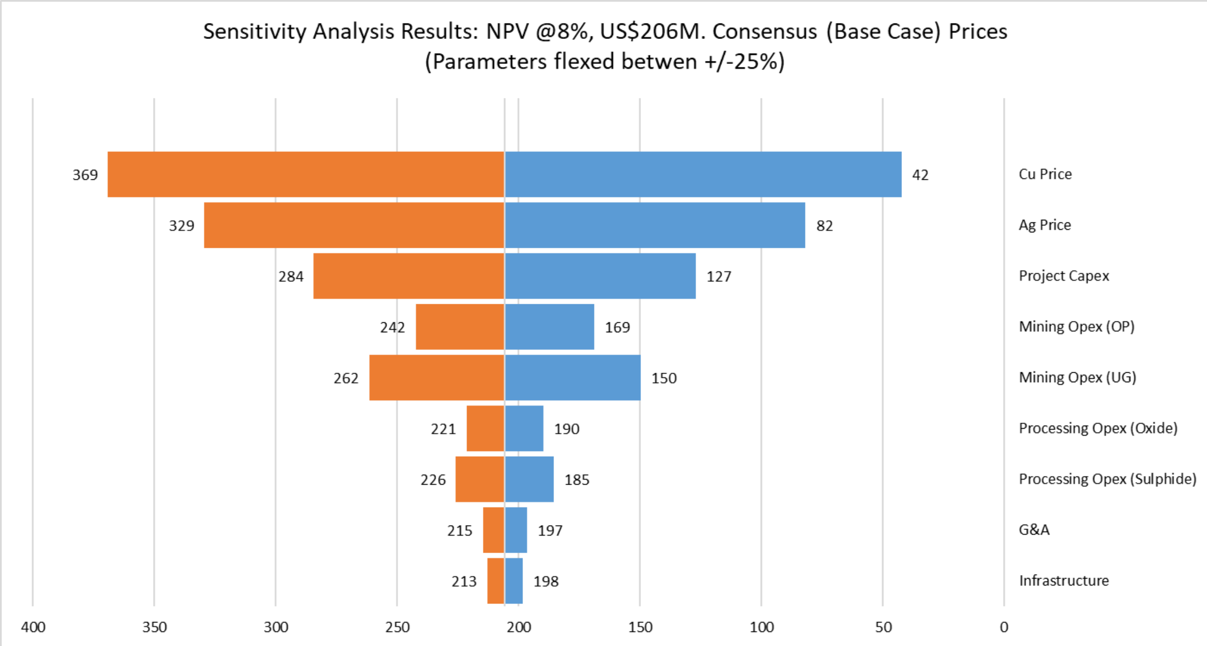


Figure 22.11: NPV Sensitivity Results for Scenario:
Oxide OP followed by UG, Processing Option 3, Consensus Prices

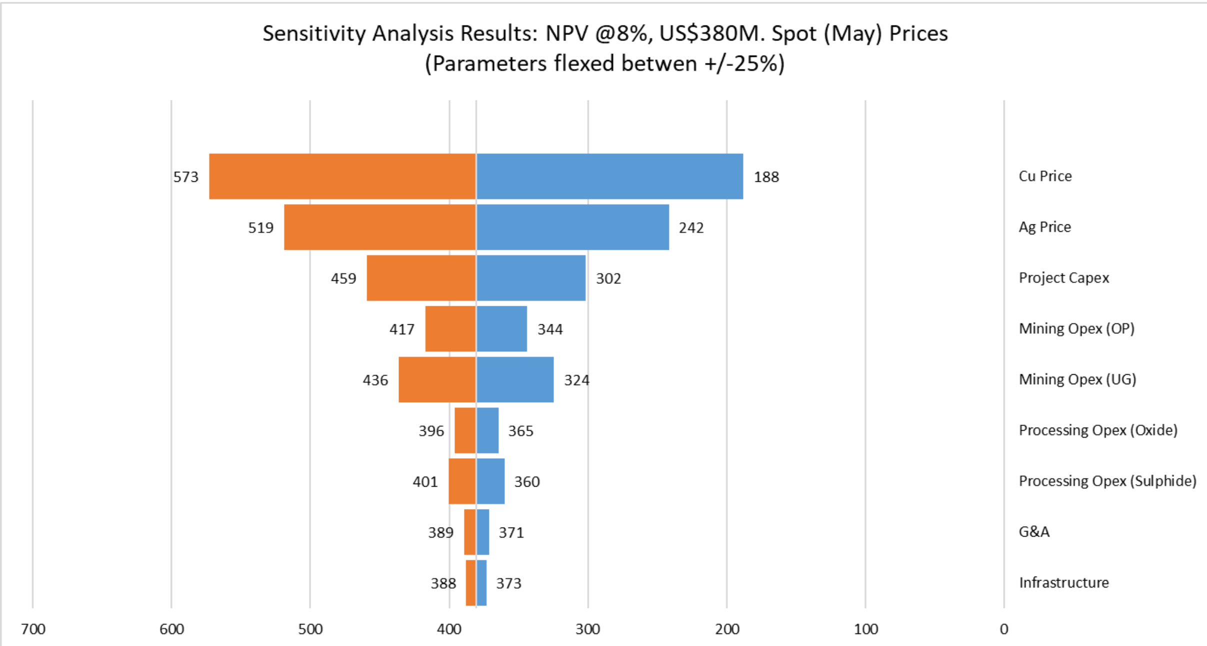


Figure 22.12: Sensitivity Analysis Results for Scenario:
Oxide OP followed by UG, Processing Option 3, Spot Prices

Based on the sensitivity analysis performed, the Project is mostly sensitive to change in Copper Price and Silver Price, as well as overall project capital costs and underground mining operating cost.

23 ADJACENT PROPERTIES

WAI is not aware of any properties immediately adjacent to the Unkur EL.

24 OTHER RELEVANT DATA AND INFORMATION

All relevant data and information regarding the Unkur Project are included in other sections of this Technical Report.

The QP is not aware of any other data that would make a material difference to the quality of this Technical Report or make it more understandable, or without which the report would be incomplete or misleading.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource Estimate

In WAI's opinion, the established understanding of the geological and grade continuity is sufficient to support the classification of the Mineral Resources as Inferred.

Current resources indicate 35Mt of sulphide ore at 0.58% Cu and 38g/t Ag and 16.1Mt of oxide ore at 0.61% Cu and 43g/t Ag, all classified as Inferred. Both open pit and underground operations are envisaged. Oxide ore is predominantly from open pit with sulphide ore equally split between underground and open pit production.

Much of upside exploration potential lies in developing the northeastern limb of the system. This has been underexplored in the past due to poor results from shallow prospecting, lack of outcrop and more difficult access to drill on the right bank of the Kemen River. With known variability in oxide at Udokan, poor prospecting results does not preclude improvement in mineralization at depth. Moreover, it is not certain whether historic work properly targeted the correct ore-bearing horizon.

Target generation would also benefit from a study of sequence stratigraphy to augment the structural study to determine the basin shoreline and thus where the mineralized horizon laterally pinches out and swells.

25.2 Hydrological & Hydrogeological Review

Site specific hydrological and hydrogeological information is limited therefore further investigation will be required to obtain:

- Surface water quality data;
- Surface water flow data;
- Groundwater quality data;
- Aquifer properties, and;
- Permafrost/talik investigation.

A reliable water supply meeting the projects technical and potable demands need to be developed at PFS stage.

A water balance and a water management plan will need to be established for the Project.

There is the potential for ARD issue due to the sulphidic ore therefore a suitable test programme will need to be established starting with static test work and testing sufficient representative samples to be commensurate with the inventory of potential acid forming materials (ores and waste).

A groundwater model will be beneficial to establish impact predictions on the hydrological and hydrogeological receptors.

25.3 Geotechnical Review

A review of geotechnical requirements has not formed part of this study.

25.4 Open Pit Optimization

A breakdown of the costs and parameters used in the Unkur Deposit pit optimisation are presented in Table 16.1 in Section 16.6.2

Recommendations to improve the confidence of the NSR model are listed below:

- Marketability of concentrate products (especially lead concentrate due to low lead assay);
- Identify concentrate off-takers and generation of agreements in principle; and
- NSR input parameters (i.e., concentrate moisture content, metal payability, metal deductions and penalties, transport costs, treatment, and refining charges, etc.).

25.5 Open Pit Mining

WAI has carried out an open pit mining study to define a mineable tonnage estimate for the Unkur deposit. As per approved initial data and scenarios 3 pits hells were evaluated from a 10Mt oxide only scenario, 17.6Mt for various processing options, and the largest scenario of 19Mt.

The different scenarios and schedules started from 4 years for oxide pit and +7 years for the largest scenario open pit. Different duration and volumes dictates different CAPEX for every option (sustaining and initial) – all values were based on the most inexpensive and affordable equipment.

A long ramp up to 3.5Mtpa is essential due to the deposit geology and ore bodies – pit development is not limited by sinking rate (considered 60Mtpa as a technical limit). The stripping ratio varies from 9 - 14 t/t, considered typical for this type of scenario/operation, and the annual rock (ROM) for 3.5Mtpa are thus between 45 and 60Mtpa (15-25M BCM[9] per annum).

Open pit optimisation was carried out using the Datamine NPV Scheduler v4 (NPVS) software package. Pit optimisations were carried out on the Resource block models generated for the two deposits and driven on the calculated block NSR values. The optimisations included Inferred resources.

The key recommendations to improve the confidence of the open pit mining study are listed below:

- Conduct dilution and loss study specific to the Unkur pit;

[9] Bulk Cubic Metres

- Generate and implement a tactical pit design criteria for the Unkur pit following geotechnical data collection, investigation, and analysis;
- Carry out waste dump design and positioning exercise to improve confidence in the waste disposal strategy; and,
- Carry out optimisation on Measured and Indicated Resources to determine influence of Inferred Resources and identify measures to improve geological confidence.

25.6 Underground Mining

WAI has not carried out a mining study to define an underground mineable tonnage estimate for the Unkur deposit at time of writing.

The key recommendations for the underground mining study are listed below, should trade off studies prove that underground mining methods are warranted economically:

- Geotechnical studies are required to optimise the stope dimensions, identify the in-situ pillar requirements to ensure regional underground stability, identify stand-off distance of access declines from mineralised zones, etc.;
- Ventilation studies are required to understand airflow requirements, identify suitable primary/secondary fan sizes, generate more detailed ventilation costs, etc.;
- It will be necessary to carry out further stope optimisation on Measured and Indicated Resources to determine influence of Inferred Resources and identify measures to improve geological confidence; and
- Tailor underground development dimensions and physicals to the likely Scenario + Process Option selection to define a workable production rate target for underground scheduling.

25.7 Mine Production Scheduling & Equipment Requirements

The generated mine designs were used as the basis for developing a combined open pit and underground production schedule. Effort was made to sequence the operations such that a steady flow of plant feed is maintained over the life-of-mine. Key points noted from the generated production schedule include:

- Overall oxide only mine life anticipated and combined oxide + sulphide at 5-6 years with Scenarios assuming depletion of oxide before sulphide.
- Depletion of oxide feed from Unkur pit anticipated at the mid Y4; indicating the point at which floatation plant would likely need to be established.
- Equipment choice based on strong local suppliers (Belaz, Kamaz, Komatsu, Doogan) represented in the region at expense of suppliers currently subject to sanctions.
- High grading initially with movement of waste push back after year 2.

No trade off was performed transitioning from open pit to underground.

25.8 Capital and Operating Costs – Mining

A mining cost model was developed to assess the open pit mining capital and operating expenditures for the Unkur Project.

The calculated costs are estimated to have an accuracy equivalent to a PEA level of detail. The study offers a valuable view in determining the merits of pursuing further engineering studies but should not be the sole reference for the purposes of economic decision making.

A summary of total CAPEX based on initial and sustaining capital discussed in Section 15.9.3 is presented in Table 25.1.

Table 25.1: Summary of Mining CAPEX

Mining Equipment	Initial Yr1 USD	Sustaining USD	Total USD
CAPEX Scenario 1	24,990,000	45,045,000	70,035,000
CAPEX Scenario 2 opt 3	34,755,000	31,710,000	66,465,000
CAPEX Scenario 2 opt 4	34,755,000	45,780,000	80,535,000

Estimated overall open pit costs are in the region of US$1.75/t rock mined. This is based on an earthmoving cost of US$1.5/t rock mined for hard rock plus an additional US$0.25 for technical services.

Total Underground Capital costs have been estimated at US$150-175M, depending on the selected option life of mine.

Operating costs for underground mining has been estimated at US$25/t, including US$3/t for backfill.

25.9 Processing

The processing of oxide ore presents the biggest challenge due to the carbonate content, resulting in very high acid consumptions from conventional acid leaching, either by agitated or heap leaching.

Therefore, the focus has been on process routes employing cyanide leaching first for silver and copper dissolution, with two options studied for agitated and heap leaching, followed by processing of the leach solutions with SART technology.

For SART performance, benchmarking data has been used for the capital and operating cost estimates and for the estimate of 95% copper and silver recovery, to achieve a single concentrate grading 65% Cu. A two-stage SART process has been assumed, as per the Tetra Tech PEA report, whereby sequential copper and silver sulphide concentrates are produced, but combined for filtering into just one concentrate product for sale to market. No testwork has been conducted on SART processing at this stage of study.

Going forwards, a comprehensive metallurgical testwork programme is required on representative oxide ore samples to confirm expected copper and silver recoveries through heap leaching and in particular for the SART process to define the expected copper and silver recoveries, grade of concentrate produced and recycling of recovered cyanide back for heap leaching. Details to be investigated include whether only a bleed stream of the pregnant solution for SART processing should be considered, with the inclusion of conventional carbon adsorption and elution/electrowinning/refining for the bulk of the leach solution for conventional silver bullion production (as per the Maricunga flowsheet).

A summary table of the key parameters for all four oxide ore processing options is shown in Table 25.2.

Table 25.2: Summary of Key Processing Parameters

OXIDE ORE	Option 1 (Agitated Leach)	Option 2 (Sequential Agitated Leach)	Option 3 (Heap Leach)	Option 4 (Sequential Heap Leach)
Copper Recovery, %	55.1	95.0	55.9	80.6
Silver Recovery, %	91.1	95.0	73.4	73.4
Capital Cost, $M	164.3	229.8	52.3	97.3
Operating Cost, $/t	18.3	27.4	5.8	15.4

From Cost Mine data, the capital cost for a 10,000 tpd single-product flotation plant is US$93.2 million and the operating cost is **US$8.98/t**. These costs are considered reasonable for scoping level accuracy.

25.10 Financial Analysis

The two mining scenarios are moderately positive at a base case price deck and do not preclude a 'showstop' at this stage but afford some constraints as to the direction to take for the project at a higher level of study.

Thus, at the Consensus prices, Scenario 1 (namely SART plant for treatment of oxide material extracted from the open pit, and underground mining of the sulphide materials thereafter) has demonstrated best NPV and IRR Results. At the base case (consensus prices) the project value has been estimated at US$ 205.5M (at 8% DR) and IRR of 26.7%. Or US$ 380M NPV8% and IRR of 44.4% at spot price deck.

However, **Scenario 3** (Processing Option 4) takes an advantage at the higher price deck. As the trade-off between higher processing capital, operating costs and increased recovery rate works towards project economics improvement at the higher commodity prices applied.

Decision making between Scenario 1 and Scenario 3 must be balanced against the initial capital cost requirements, i.e. at US$401.5M vs US$519.2M respectively.

As a standalone oxide project, **Scenario 4** is associated with the lowest capital, lowest operating cost scenario and generates a post-tax free cash flow of US$ 238M (Consensus Prices). Which is sufficient to pay for the sulphide plant capital and commencing underground operation, involvement of which has shown a positive impact on the overall project economics.

The current project timeframe considers that the oxide ore processing capacity is reached fully in year two, with underground feed being added from year four/ six (subject to the selected Scenario); and sulphide plant target feed being achieved in year seven / nine (Figure 25.1 and Figure 25.2).

Capital costs for sulphide plant are assumed to be funded by cash generated from the open pit operation. Capital required for underground development and equipment, as well as sulphide plant construction is assumed to be paid back in three / four years following investment (project year five / seven) once the production target is fully achieved.

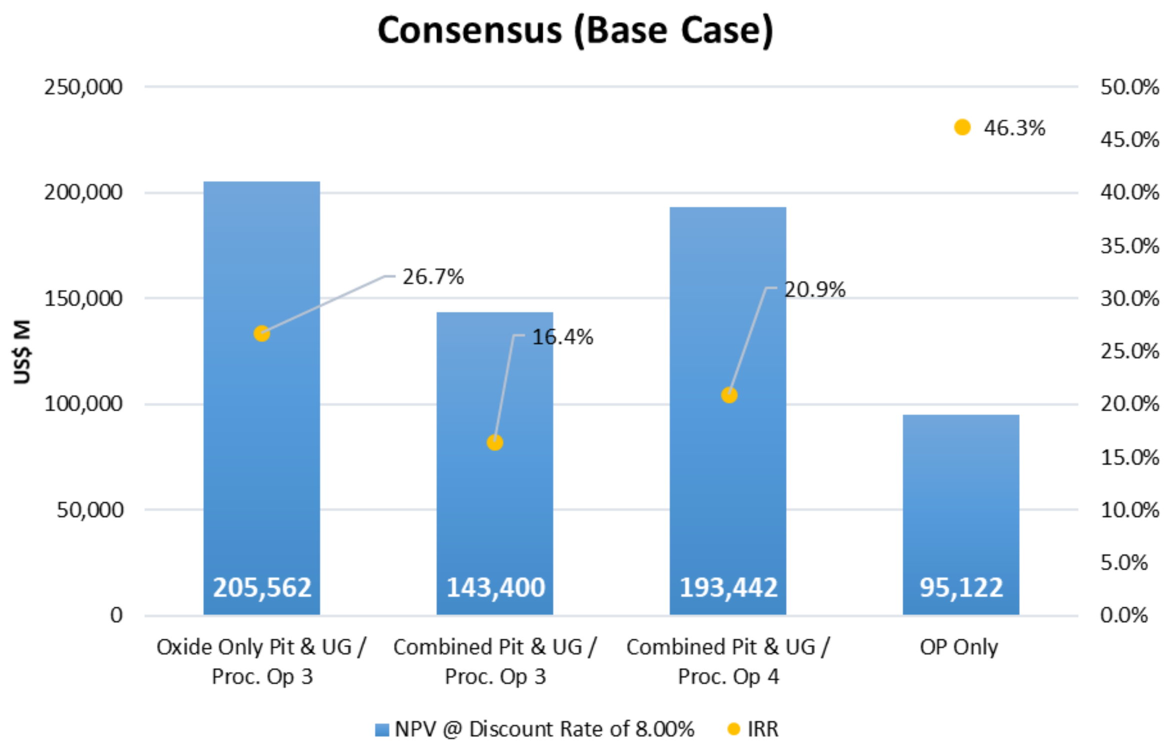


Figure 25.1: Financial Results Shown for the Consensus Prices Deck

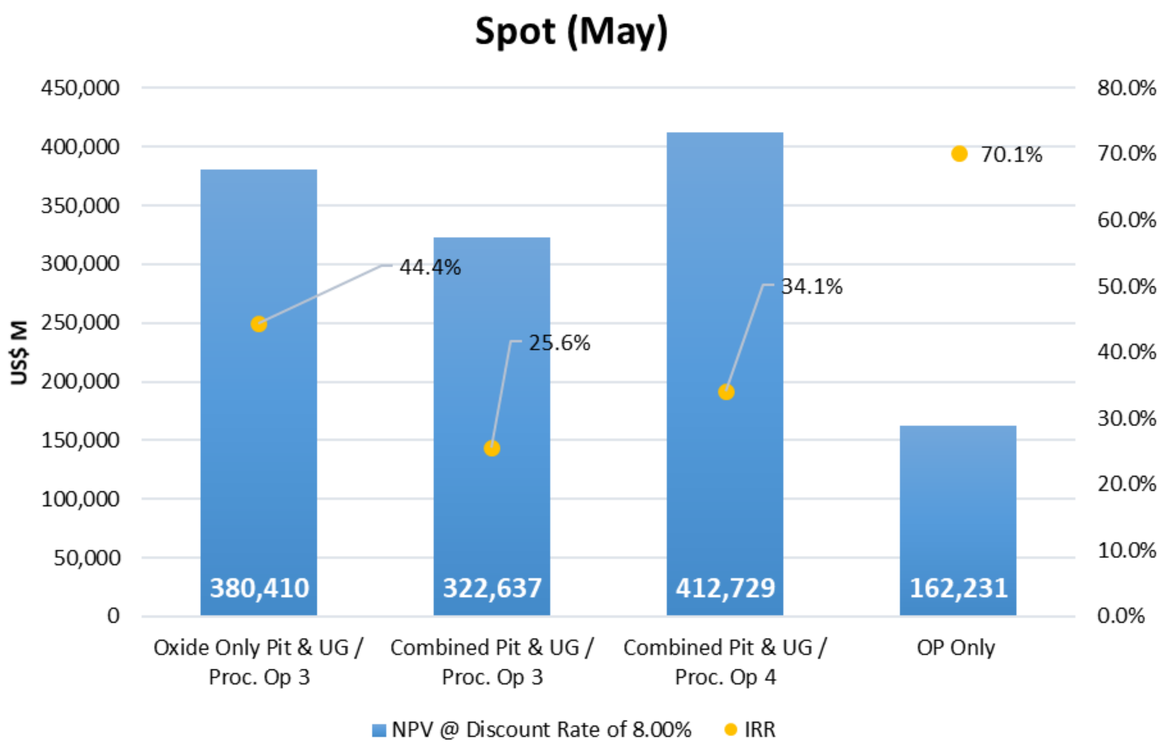


Figure 25.2: Financial Results Shown for the Spot Prices Deck

Based on the sensitivity analysis performed, the Project is mostly sensitive to change in Copper Price and Silver Price, as well as overall project capital costs and underground mining operating cost.

25.11 Risks and Opportunities

Areas of risk and opportunity material to the project are set out in Table 25.4 within the framework of the Strengths, Weaknesses, Opportunities and Threats (SWOT) analysis. The legend for the SWOT analysis is set out in Table 25.3.

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Table 25.3: Legend for SWOT Analysis

	Element related to Data
	Element related to Geology and Mineral Resources
	Element related to Mining
	Element related to Processing and Infrastructure
	Element related to Financial
	Element related to Other Modifying Factors

Table 25.4: SWOT Analysis for the Unkur Project	
Strengths	Adequate exploration SOPs and QA/QC procedures since 2014 with good recovery of drill core. Low risk to provenance of data for PEA level of study.
	Stratabound model is open and by its nature should demonstrate some degree of uniformity along strike, down-dip and on the opposite northern flank of the Unkur syncline.
	Robust contained copper grades and higher silver credits compared to Udokan.
	Good supply chain for equipment and access to suppliers in the region
	Process Option 3 is cash positive for oxide only option.
	Well-developed infrastructure and proximity to a developing operation.
Weaknesses	A lack of Measured and Indicated Resources defined for Unkur.
	Increasing depth of moraine a barrier to exploration on the southern flank of the deposit.
	The mining schedules all indicate a significant increase in ramp up of waste material to be moved in years 1-3 in order to expose enough ore will put pressure on the haulage fleet and availability of equipment in order to strip the required volumes of material.
	Lack of detailed geotechnical data and analysis for the Unkur Pit.
	The mineable tonnage does not represent Ore Reserves.
	Insufficient geotechnical data and analysis to consider for an underground geotechnical study which will need to be addressed at a higher level of study.
	Lack of variability testwork conducted on sulphide ore for hardness/grindability.
	Geometallurgical uncertainties and lack of phase analysis or representative testwork yet conducted to define ore sub types and oxide/sulphide boundary.
	No penalties have been considered in the PEA valuation due to limited geological data and undefined payment terms.
Opportunities	Sequence stratigraphy and basin analysis to correlate northern Unkur limb to south and open the northern area up for exploration.
	Robust and increasing copper price forecast over the next 5 years
	SART technology to increase copper recovery from oxide and recycle cyanide, lowering cyanide consumption and process operating costs.
Threats	Capability to increase the resource base with moraine cover, lack of all-year access to north of the deposit and increasing costs and risks in drilling downdip, although the deposit remains open in all directions.
	High waste movement during concentrated periods – little room for slack in the schedule.
	Relative lack of commercial benchmarking for SART process. Testwork needs to be done on representative bulk samples and synthetic mixes of the expected blends of oxide:sulphide to mitigate technological risk if this route is chosen.
	Effect of penalty elements in the final concentrates and constraints on smelter contracts.
	Sensitivity to commodity prices and mining costs
	Optimal stand-alone oxide process route does not transpose as the best option for a combined oxide-sulphide process route but may present a satisficer option.

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12. License ЧИТ02522БР (geological study, exploration and production of copper, silver, and associated components for the Unkur Project).

27 DATE AND SIGNATURE PAGE

This report titled "Technical Report Preliminary Economic Assessment of the Unkur Copper Deposit, Zabaikalsky Krai, Russian Federation" with an effective date of July 31, 2021 was prepared and signed by the following authors:

(Signed & Sealed) *Ché Osmond*

Dated at
August 16,2021

Ché Osmond
Technical Director
Wardell Armstrong International

(Signed & Sealed) *Robin Simpson*

Dated at
August 16,2021

Robin Simpson
Principal Resource Geologist
SRK Consulting (Russia) Ltd.

(Signed & Sealed) *James Turner*

Dated at
August 16,2021

James Turner
Technical Director
Wardell Armstrong International

(Signed & Sealed) *Alan Clarke*

Dated at
August 16,2021

Alan Clarke
Associate Director
Wardell Armstrong International

28 CERTIFICATE OF QUALIFIED PERSON

28.1 Ché Osmond, CGeol

As a Qualified Person of the Technical Report on the Unkur Copper Deposit, Zabaikalsky Krai, Russian Federation, I, Ché Osmond, do hereby certify:

- I am a Technical Director with Wardell Armstrong International, with a business address at Baldhu House, Wheal Jane Earth Science Park, Baldhu, Truro, Cornwall, United Kingdom TR3 6EH.
- This certificate applies to the technical report entitled "Technical Report Preliminary Economic Assessment of the Unkur Copper Deposit, Zabaikalsky Krai, Russian Federation" dated 16th August 2021 (the "Technical Report").
- I am a graduate of Oxford Brookes University, (BSc (Hons) Geology & Cartography, 1995) and Camborne School of Mines (Exeter University) (MSc Mining Geology, 1997). I have practiced my profession continuously since 1997. During my employment with Wardell Armstrong International I have frequently authored or reviewed for a variety of commodities, including copper.
- I have read the definition of "qualified person" set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "Qualified Person" for the purposes of the Instrument.
- I am a registered member in good standing of the Geological Society of London as a Fellow and Chartered Geologist and a registered European Geologist as elected by the European Federation of Geologists (Fellowship number 1016839).
- I have not personally inspected the Property that is the subject of this Technical Report.
- I am responsible for Items 1 through 8, and 15 through 26 (except 21.3) of the Technical Report.
- I am independent of Azarga Metals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this certificate, to the best of my knowledge, information, and belief, the sections of Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th day of August 2021

(Signed & Sealed) *Ché Osmond*

Ché Osmond, CGeol
Technical Director
Wardell Armstrong International

28.2 Robin Simpson, MAIG

As a Qualified Person of the Technical Report on the Unkur Copper Deposit, Zabaikalsky Krai, Russian Federation, I, Robin Simpson, MAIG, of Moscow, Russia, do hereby certify:

- I am a Principal Resource Geologist with SRK Consulting (Russia) Ltd. with a business address at 4/3 Kuznetsky Most, Building 1, 125009, Moscow, Russia.
- This certificate applies to the technical report entitled "Technical Report Preliminary Economic Assessment of the Unkur Copper Deposit, Zabaikalsky Krai, Russian Federation" dated 16th August 2021 (the "Technical Report").
- I am a graduate of the University of Canterbury, New Zealand (BSc (Hons) Geology, 1996) and Leeds University (MSc Geostatistics, 2004). I am a member in good standing of the Australian Institute of Geoscientists (No. 3156). I have practiced my profession continuously since 1996. I worked as a mine and exploration geologist at gold and copper mines in Australia for seven years, and then joined SRK Consulting Ltd. In 2005 as a resource geologist. During my employment in SRK's Perth, Cardiff, and Moscow offices I have frequently authored or reviewed mineral resource estimates for a variety of commodities, including copper and silver.
- I have read the definition of "qualified person" set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "Qualified Person" for the purposes of the Instrument.
- I am a Member of the Australian Institute of Geoscientists, membership number 3156.
- I personally inspected the Property that is the subject of this Technical Report on December 10, 2014, and October 13, 2016.
- I am responsible for Items 9 through 12 of the Technical Report.
- I am independent of Azarga Metals Corp. as defined by Section 1.5 of the Instrument.
- I have had prior involvement with the Property that is the subject of this Technical Report having authored previous Technical Reports dated March 1, 2016; March 31, 2017; March 27, 2018; and August 30, 2018.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this certificate, to the best of my knowledge, information, and belief, the sections of Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th day of August 2021

(Signed & Sealed) *Robin Simpson*

Robin Simpson, MAIG
Principal Resource Geologist
SRK Consulting (Russia) Ltd.

28.3 James Turner, CEng

As a Qualified Person of the Technical Report on the Unkur Copper Deposit, Zabaikalsky Krai, Russian Federation, I, James Turner, do hereby certify:

- I am a Technical Director with Wardell Armstrong International, with a business address at Baldhu House, Wheal Jane Earth Science Park, Baldhu, Truro, Cornwall, United Kingdom TR3 6EH.
- This certificate applies to the technical report entitled "Technical Report Preliminary Economic Assessment of the Unkur Copper Deposit, Zabaikalsky Krai, Russian Federation" dated 16th August 2021 (the "Technical Report").
- I am a graduate of the Camborne School of Mines (BSc (Hons) Mineral Processing Technology, 1984, MSc Minerals Engineering, 1993). During my employment with Wardell Armstrong International I have frequently authored or reviewed for a variety of commodities, including copper.
- I have read the definition of "qualified person" set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "Qualified Person" for the purposes of the Instrument.
- I am a member in good standing of the Institute of Materials, Minerals and Mining and a Chartered Mineral Processing Engineer registered with the Engineering Council of the U.K. (registration number 413242). I have practiced my profession continuously since 1984.
- I have not personally inspected the Property that is the subject of this Technical Report.
- I am responsible for Items 13, 17 and 21.3 of the Technical Report.
- I am independent of Azarga Metals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this certificate, to the best of my knowledge, information, and belief, the sections of Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th day of August 2021

(Signed & Sealed) ***James Turner***

James W.G. Turner, B.Sc. (Hons), ACSM, M.Sc., MCSM, MIMMM, CEng
Technical Director
Wardell Armstrong International

28.4 Alan Clarke, CGeol

As a Qualified Person of the Technical Report on the Unkur Copper Deposit, Zabaikalsky Krai, Russian Federation, I, Alan Clarke, do hereby certify:

- I am an Associate Director with Wardell Armstrong International, with a business address at Baldhu House, Wheal Jane Earth Science Park, Baldhu, Truro, Cornwall, United Kingdom TR3 6EH.
- This certificate applies to the technical report entitled "Technical Report Preliminary Economic Assessment of the Unkur Copper Deposit, Zabaikalsky Krai, Russian Federation" dated 16th August 2021 (the "Technical Report").
- I am a graduate of Edinburgh University, (BSc (Hons) Geology, 2002) and Camborne School of Mines (Exeter University) (MSc Mining Geology, 2003). I have practiced my profession continuously since 2003. During my employment with Wardell Armstrong International I have frequently authored or reviewed for a variety of commodities, including copper.
- I have read the definition of "qualified person" set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "Qualified Person" for the purposes of the Instrument.
- I am a registered member in good standing of the Geological Society of London as a Fellow and Chartered Geologist and a registered European Geologist as elected by the European Federation of Geologists (Fellowship number 1014124).
- I have not personally inspected the Property that is the subject of this Technical Report.
- I am responsible for Item 14 of the Technical Report.
- I am independent of Azarga Metals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this certificate, to the best of my knowledge, information, and belief, the sections of Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th day of August 2021

(Signed & Sealed) *Alan Clarke*

Alan Clarke, CGeol
Associate Director
Wardell Armstrong International

APPENDIX 1: FINANCIAL MODEL

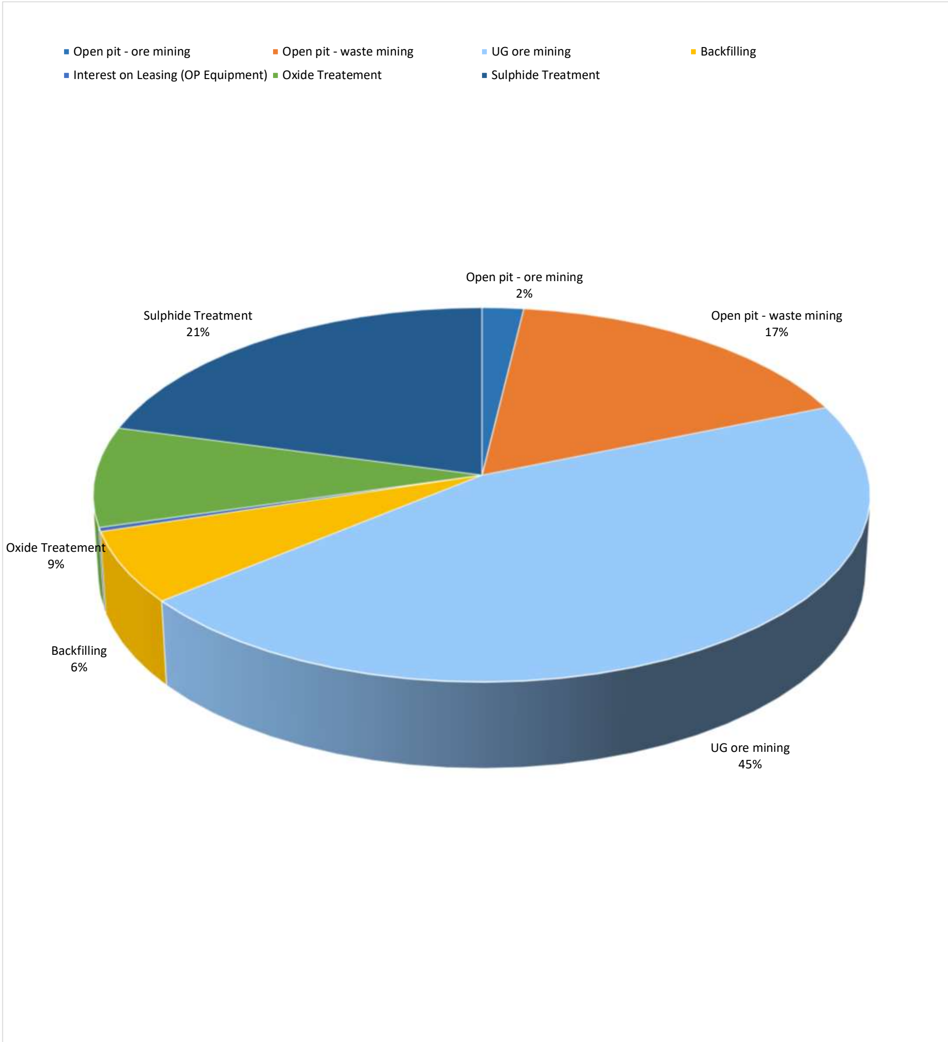
Selected Price Deck:		Scenario Oxide Only Pit & UG / Proc. Op 3	Consensus (Base Case)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Metal Prices	**Consensus (Base Case)**		**Units**	**TOTAL LOM**																	
	0%	Cu Price Selected Scenario per t	3.86 /lb	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500	8,500
	0%	Ag Price per oz		25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25
Production Schedule																					
		LOM	years	14																	
Open Pit		**Oxide Ore**	t'000	**10,981**		1,500	3,500	3,500	2,481	0	0	0	0	0	0	0	0	0	0	0	0
		Cu Grade	%	0.59%		0.63%	0.56%	0.54%	0.68%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
		Ag Grade	g/t	48.24		46.21	42.90	44.20	62.72	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
		Cu In-situ	t	**64,846**		9,393	19,534	19,023	16,897	0	0	0	0	0	0	0	0	0	0	0	0
		Ag In-situ	kg	**529,734**		69,315	150,147	154,689	155,584	0	0	0	0	0	0	0	0	0	0	0	0
		Sulphide	t'000	**0**		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Cu Grade	%	0.00%		0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
		Ag Grade	g/t	0.00		0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
		Cu In-situ	t	**0**		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Ag In-situ	kg	**0**		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Total Ore Mined (OP)	t'000	**10,981**		1,500	3,500	3,500	2,481	0	0	0	0	0	0	0	0	0	0	0	0
		Waste	t'000	**97,420**		13,431	31,789	47,677	4,522	0	0	0	0	0	0	0	0	0	0	0	0
		Rock	t'000	**108,401**		14,931	35,289	51,177	7,003	0	0	0	0	0	0	0	0	0	0	0	0
		Strip Ratio	W:O	**8.87**		8.95	9.08	13.62	1.82	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Underground added on following Oxide OP																					
		Oxide Ore	t'000	4,067		0	0	0	1,342	1,398	1,291	36	0	0	0	0	0	0	0	0	0
		Cu Grade	%	0.71%		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Ag Grade	g/t	41.34		0	0	0	26	35	64	66	0	0	0	0	0	0	0	0	0
		Cu In-situ	t	**28,963**		0	0	0	8,473	9,853	10,362	275	0	0	0	0	0	0	0	0	0
		Ag In-situ	kg	**168,105**		0	0	0	35,038	48,588	82,084	2,395	0	0	0	0	0	0	0	0	0
				8.36																	
		Sulphide	t'000	16,712		0	0	0	0	590	710	2,000	2,000	2,000	2,000	2,002	2,000	2,000	1,410	0	0
		Cu Grade	%	0.77%		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Ag Grade	g/t	53.11		0	0	0	13	29	64	64	62	59	59	57	40	40	46	0	0
		Cu In-situ	t	**128,129**		0	0	0	0	3,828	5,752	16,203	16,153	16,048	16,048	15,934	14,600	14,600	8,963	0	0
		Ag In-situ	kg	**887,600**		0	0	0	1	17,256	45,153	127,192	123,930	117,599	117,599	114,313	79,749	79,749	65,059	0	0
		Total Ore Mined (UG)	t'000	**20,779**		0	0	0	1,342	1,987	2,001	2,036	2,000	2,000	2,000	2,002	2,000	2,000	1,410	0	0
	Royalty Rate	**TOTAL ORE MINED OP & UG**	t'000	**31,759**		**1,500**	**3,500**	**3,500**	**3,823**	**1,987**	**2,001**	**2,036**	**2,000**	**2,000**	**2,000**	**2,002**	**2,000**	**2,000**	**1,410**	**0**	**0**
Mined Metals	8.00%	Total Mined Cu	t	**221,939**		9,393	19,534	19,023	25,370	13,681	16,115	16,479	16,153	16,048	16,048	15,934	14,600	14,600	8,963	0	0
	6.50%	Total Mined Ag	oz	**50,973,028**		2,228,514	4,827,322	4,973,360	6,128,657	2,116,929	4,090,755	4,166,325	3,984,432	3,780,894	3,780,894	3,675,244	2,564,003	2,564,003	2,091,696	0	0
		Total Sulphide		**16,712**																	

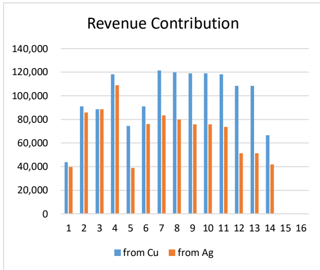
Selected Price Deck:		Scenario Oxide Only Pit & UG / Proc. Op 3	Consensus (Base Case)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Concentrate Production and Realisation																					
		Total Oxide	t'000	15,047		1,500	3,500	3,500	3,823	1,398	1,291	36	0	0	0	0	0	0	0	0	0
		Total Sulphide	t'000	16,712		0	0	0	0	590	710	2,000	2,000	2,000	2,000	2,002	2,000	2,000	1,410	0	0
Oxide Ore Treatment		Cu Recovery Rate	%	55.90%																	
		Ag Recovery Rate	%	73.40%																	
		Cu Recovered from Oxide Ore	t	52,439		5,251	10,919	10,634	14,182	5,508	5,793	154	0	0	0	0	0	0	0	0	0
		Ag Recovered from Oxide Ore	kg	512,213		50,877	110,208	113,542	139,917	35,664	60,249	1,758	0	0	0	0	0	0	0	0	0
		Ag grade in Concentrate (for Processing Optic	g/t	6,642		6,589	6,863	7,261	6,709	4,403	7,073	7,767	0	0	0	0	0	0	0	0	0
	68%	**Cu Concentrate Produced (for Processing Op**	dmt	77,117		7,722	16,058	15,638	20,856	8,099	8,518	226	0	0	0	0	0	0	0	0	0
Sulphide Ore Treatement		Cu Recovery to 26% Cu Concentrate	%	89%																	
		Ag Recovery to 26% Cu Concentrate	%	83%																	
		Cu Recovered to Cu Concentrate	t	114,163		0	0	0	0	3,411	5,125	14,437	14,392	14,299	14,299	14,197	13,008	13,008	7,986	0	0
		Ag Recovered to Cu Concentrate	kg	734,045		0	0	0	0	14,271	37,342	105,188	102,490	97,254	97,254	94,537	65,953	65,953	53,804	0	0
		Ag grade in Concentrate	g/t	1,672		0	0	0	566	1,088	1,894	1,894	1,851	1,768	1,768	1,731	1,318	1,318	1,752	0	0
	26%	**Cu Concentrate Produced**	dmt	439,090		0	0	0	1	13,119	19,712	55,528	55,356	54,996	54,996	54,605	50,031	50,031	30,715	0	0
Products		**Total Cu Concentrate Produced**	dmt	516,207		7,722	16,058	15,638	20,856	21,219	28,231	55,754	55,356	54,996	54,996	54,605	50,031	50,031	30,715	0	0
	99.0%	Total Payable: Cu in Cathode	t	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	99.0%	Total Payable: Ag in Bullions	kg	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	98%	Total Payable Cu in Cu Concentrate	t	163,271		5,146	10,701	10,421	13,898	8,740	10,699	14,299	14,105	14,013	14,013	13,913	12,748	12,748	7,826	0	0
	97%	Total Payable in Cu Concentrate	kg	1,208,871		49,351	106,901	110,135	135,719	48,436	94,663	103,737	99,415	94,337	94,337	91,701	63,974	63,974	52,190	0	0
Payable Assay Volumes		Payable for Cu recovered	t	163,271		5,146	10,701	10,421	13,898	8,740	10,699	14,299	14,105	14,013	14,013	13,913	12,748	12,748	7,826	0	0
		Payable for Ag Recovered	oz	38,866,100		1,586,658	3,436,956	3,540,933	4,363,483	1,557,259	3,043,494	3,335,238	3,196,272	3,032,995	3,032,995	2,948,244	2,056,818	2,056,818	1,677,937	0	0
Realisation Costs - Oxide Con	21	Treatment Charge for Oxide Concentrate	US$'000	1,619		162	337	328	438	170	179	5	0	0	0	0	0	0	0	0	0
Realisation Costs - Sulphide C	115	Treatment Charge for Sulphide Concentrate	US$'000	50,495		0	0	0	0	1,509	2,267	6,386	6,366	6,324	6,324	6,280	5,754	5,754	3,532	0	0
$/lb	0.07	Cu Refining Charge	US$'000	25,197		794	1,651	1,608	2,145	1,349	1,651	2,207	2,177	2,163	2,163	2,147	1,967	1,967	1,208	0	0
	0.5	Ag Refining Charge	US$'000	19,433		793	1,718	1,770	2,182	779	1,522	1,668	1,598	1,516	1,516	1,474	1,028	1,028	839	0	0
	8%	Concentrate available for Transportation	wmt	557,503		8,339	17,342	16,889	22,525	22,916	30,489	60,214	59,784	59,395	59,395	58,974	54,034	54,034	33,172	0	0
	15	Concentrate Transportation Cost	US$'000	8,363		125	260	253	338	344	457	903	897	891	891	885	811	811	498	0	0
Gross Revenue Payable		**Gross Revenue**	US$'000	2,359,453		83,405	176,881	177,101	227,223	113,223	167,032	204,925	199,796	194,934	194,934	191,970	159,778	159,778	108,471	0	0
		from Cu	US$'000	1,387,801		43,738	90,957	88,578	118,136	74,292	90,945	121,544	119,889	119,110	119,110	118,264	108,358	108,358	66,523	0	0
		from Ag	US$'000	971,652		39,666	85,924	88,523	109,087	38,931	76,087	83,381	79,907	75,825	75,825	73,706	51,420	51,420	41,948	0	0
		Realisation Costs	US$'000	-105,107		-1,875	-3,967	-3,960	-5,103	-4,150	-6,076	-11,168	-11,037	-10,894	-10,894	-10,785	-9,560	-9,560	-6,077	0	0
Net Smelter Return		**NSR**	US$'000	**2,254,346**		**81,530**	**172,914**	**173,141**	**222,120**	**109,073**	**160,956**	**193,757**	**188,758**	**184,040**	**184,040**	**181,184**	**150,219**	**150,219**	**102,395**	**0**	**0**

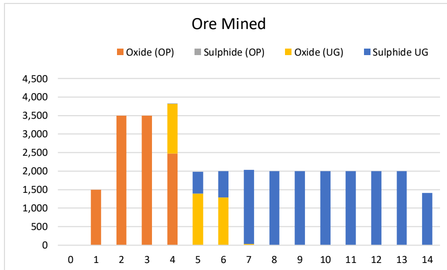
Selected Price Deck:		Scenario Oxide Only Pit & UG / Proc. Op 3	Consensus (Base Case)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Production Costs																					
Mining																					
0%	1.75	Open pit - ore mining	US$'000	**19,216**		2,625	6,125	6,125	4,341	0	0	0	0	0	0	0	0	0	0	0	0
	1.75	Open pit - waste mining	US$'000	**170,485**		23,505	55,632	83,435	7,914	0	0	0	0	0	0	0	0	0	0	0	0
0%	22.00	UG ore mining	US$'000	**457,129**		0	0	0	29,527	43,718	44,017	44,797	44,008	44,000	44,000	44,045	44,000	44,000	31,018	0	0
	3.00	Backfilling	US$'000	**62,336**		0	0	0	4,026	5,962	6,002	6,109	6,001	6,000	6,000	6,006	6,000	6,000	4,230	0	0
		Interest on Leasing (OP Equipment)	US$'000	**3,481**		6	-191	509	709	409	229	149	469	389	309	269	229	0	0	0	0
Processing		Total Mining Opex	US$'000	**712,647**		**26,136**	**61,565**	**90,068**	**46,516**	**50,088**	**50,248**	**51,054**	**50,478**	**50,389**	**50,309**	**50,320**	**50,229**	**50,000**	**35,247**	**0**	**0**
	5.8	Oxide Treatement	$/t	**5.80**																	
	12.6	Sulphide Treatment	$/t	**12.56**																	
0%		Oxide Treatement	US$'000	**87,273**		8,700	20,300	20,300	22,171	8,106	7,486	210	0	0	0	0	0	0	0	0	0
0%		Sulphide Treatment	US$'000	**209,956**		0	0	0	1	7,408	8,920	25,126	25,131	25,126	25,126	25,152	25,126	25,126	17,713	0	0
		Total Processing Opex	US$'000	**297,229**		8,700	20,300	20,300	22,172	15,514	16,406	25,337	25,131	25,126	25,126	25,152	25,126	25,126	17,713	0	0
0%	5,000	G&A	US$'000	**70,000**		5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	0	0
	0.00	Fees to Closure and Reclamation	US$'000	**0**		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0%	4,016	Infrastructure	US$'000	**56,218**		4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	0	0
Tax Relief >>	yes	Royalty (Mineral Extraction Tax)	US$'000	**69,495**		0	0	2,568	3,308	3,390	4,889	9,086	8,871	11,566	11,566	14,252	0	0	0	0	0
Total Operating Costs		**Total Opex**	US$'000	**1,205,589**		43,852	90,881	121,952	81,012	78,007	80,558	94,492	93,495	96,096	96,016	98,739	84,370	84,142	61,976	0	0
				-872,278																	
Capital Expenditure																					
		Total Mining Capex		**260,171**																	
		Mining Capital Costs OP & UG (Leased Equ	US$'000	**169,035**	4,998	10,432	18,202	28,202	33,202	15,000	15,000	15,000	9,000	4,000	4,000	4,000	4,000	2,000	2,000	0	0
		Including Estimated OP Equipment Cost	*US$'000*	*70,035*	*24,990*	*21,735*	*23,310*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*
		Including Estimated UG Equipment Cost	*US$'000*	*105,000*	*0*	*0*	*0*	*50,000*	*25,000*	*0*	*0*	*0*	*20,000*	*0*	*0*	*0*	*0*	*10,000*	*0*	*0*	*0*
		Capitalised Interest	US$'000	**136**	136	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		UG Mining Capital Development Cost	US$'000	**85,000**		0	50,000	25,000	0	0	0	0	5,000	0	0	0	0	5,000	0	0	0
	52.3	**Processing Capex - Oxide**	US$'000	**52,300**	52,300																
	1.50%	Processing Sustaining Capex Oxide	US$'000	**3,138**			785	785	785	785	0	0	0	0	0	0	0	0	0	0	0
	66.62	**Processing Capex - Sulphide**	US$'000	**66,618**	0	0	0	66,618	0	0											
	1.50%	Processing Sustaining Capex Sulphide	US$'000	**8,993**					999	999	999	999	999	999	999	999	999	0	0	0	0
	0.26 Mpa	TSF Wall Raise (Initial is included in Cost abov	US$'000	**2,831**	0	0	0	0	257	257	257	257	257	257	257	257	257	257	257	0	0
		Total UG		190,000																	
	10,057	**General Infrastructure Initial Capex**	US$'000	**10,057**	10,057																
		Closure and Reclamation Cost	US$'000	**16,000**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	16,000	0
	0%	EPCM	US$'000	**0**	0																
	15%	Contingency	US$'000	**57,472**	10,124	1,565	10,230	17,973	4,980	2,250	2,250	2,250	2,100	600	600	600	600	1,050	300	0	0
		Assets Residual Value (Sold at the end of min	US$'000	**-19,671**	0	0	0	0	0	0	0	0	0	0	0	0	-19,671	0	0	0	0
0%		**Total Project Capital Expenditure**	US$'000	**451,909**	**77,615**	**11,997**	**79,217**	**138,577**	**40,223**	**19,291**	**18,507**	**18,507**	**17,357**	**5,857**	**5,857**	**5,857**	**-13,814**	**8,307**	**2,557**	**16,000**	**0**
Taxes & Depreciation																					
		Working Capital																			
		2 Months of Production				1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0
		Working Capital	US$'000	**190,602**		7,309	15,147	20,325	13,502	13,001	13,426	15,749	15,583	16,016	16,003	16,457	14,062	14,024	0	0	0
						7,309	7,838	5,179	-6,823	-501	425	2,322	-166	434	-13	454	-2,395	-38	-14,024	0	0
		Corporate Income Tax																			
Select Tax Option >>		Tax Rate Applied in the Valuation		8.98%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	10.0%	10.0%	10.0%	10.0%	10.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%
	1	Tax Relief Applied			0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	10.0%	10.0%	10.0%	10.0%	10.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%
	2	Tax Rate - Basic			20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%
	2	Post-Tax			1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	1	Pre-Tax			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	2	Post-Tax			1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

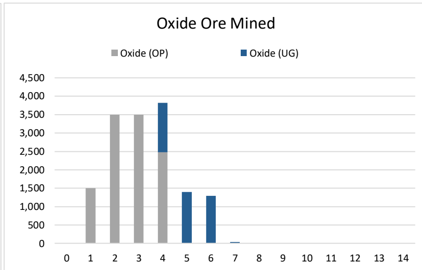
Selected Price Deck:	Scenario Oxide Only Pit & UG / Proc. Op 3	Consensus (Base Case)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
CASH FLOW MODEL			US$'000																	
	Summary of Cash Flows	US$'M	TOTAL LOM																	
	Gross Revenue Payable		**2,359,453**	0	83,405	176,881	177,101	227,223	113,223	167,032	204,925	199,796	194,934	194,934	191,970	159,778	159,778	108,471	0	0
10	Less Smelter and Realisation Costs		**-105,107**	0	-1,875	-3,967	-3,960	-5,103	-4,150	-6,076	-11,168	-11,037	-10,894	-10,894	-10,785	-9,560	-9,560	-6,077	0	0
	NSR		**2,254,346**	0	81,530	172,914	173,141	222,120	109,073	160,956	193,757	188,758	184,040	184,040	181,184	150,219	150,219	102,395	0	0
53%	**Less Operating Costs, including:**		**-1,205,589**	0	-43,852	-90,881	-121,952	-81,012	-78,007	-80,558	-94,492	-93,495	-96,096	-96,016	-98,739	-84,370	-84,142	-61,976	0	0
59%	Mining		**-712,647**	0	-26,136	-61,565	-90,068	-46,516	-50,088	-50,248	-51,054	-50,478	-50,389	-50,309	-50,320	-50,229	-50,000	-35,247	0	0
25%	Processing		**-297,229**	0	-8,700	-20,300	-20,300	-22,172	-15,514	-16,406	-25,337	-25,131	-25,126	-25,126	-25,152	-25,126	-25,126	-17,713	0	0
6%	G&A		**-70,000**	0	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	0	0
0%	Closure and Reclamation Fees		**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
5%	Infrastructure		**-56,218**	0	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	0	0
6%	Royalty (Mineral Extraction Tax)		**-69,495**	0	0	0	-2,568	-3,308	-3,390	-4,889	-9,086	-8,871	-11,566	-11,566	-14,252	0	0	0	0	0
	Cash Costs (C1, including Silver Credits)		2,077		206	367	3,108	-2,013	4,374	518	1,208	1,391	1,867	1,862	2,215	2,942	2,924	2,697	0	0
	Cash Costs (C1, excluding Silver Credits)		6,962		7,389	7,864	11,061	5,308	8,046	6,722	5,720	5,732	5,959	5,953	6,195	5,684	5,666	6,738	0	0
	EBITDA		**1,048,757**	**0**	**37,678**	**82,033**	**51,189**	**141,109**	**31,066**	**80,398**	**99,264**	**95,263**	**87,944**	**88,024**	**82,445**	**65,848**	**66,077**	**40,419**	**0**	**0**
	Less Depreciation		**-282,983**	0	-7,373	-7,813	-14,596	-26,374	-27,690	-26,892	-26,095	-25,374	-24,613	-22,831	-21,218	-19,759	-16,570	-15,785	0	0
Select >> Post-Tax	**Taxable Income**		**765,774**	0	30,305	74,221	36,593	114,734	3,376	53,506	73,169	69,889	63,331	65,193	61,227	46,089	49,507	24,634	0	0
Tax Rate with Holidays	*Less Corporate Income Tax*		**-68,800**	0	0	0	0	0	0	-5,351	-7,317	-6,989	-6,333	-6,519	-12,245	-9,218	-9,901	-4,927	0	0
	Less Change in NWC (Net Working Capital)		**0**	0	-7,309	-7,838	-5,179	6,823	501	-425	-2,322	166	-434	13	-454	2,395	38	14,024	0	0
Post-Tax	**Net Operating Profit**		**979,957**	0	30,370	74,195	46,010	147,932	31,567	74,623	89,625	88,441	81,177	81,518	69,746	59,025	56,213	49,516	0	0
	Less Project Capital Costs		**-451,909**	-77,615	-11,997	-79,217	-138,577	-40,223	-19,291	-18,507	-18,507	-17,357	-5,857	-5,857	-5,857	13,814	-8,307	-2,557	-16,000	0
Post-Tax	**Free Cash Flow to Firm**		**528,048**	-77,615	18,373	-5,021	-92,567	107,709	12,276	56,116	71,118	71,084	75,320	75,661	63,889	72,840	47,906	46,958	-16,000	0
Post-Tax	**Cumulative FCF**			-77,615	-59,242	-64,263	-156,830	-49,121	-36,845	19,271	90,389	161,473	236,793	312,454	376,344	449,183	497,089	544,048	528,048	528,048
			1																	
Post-Tax	**NPV @ Discount Rate of 8.00%**	US$ M	205,562																	
Post-Tax	IRR	%	26.73%																	
	Payback period of capital (FCF)	Years	6																	
	By-Product Basis																			
	Cash Costs per t of Copper																			
	Mining Costs	US$/t Cu payable	4,365		5,079	5,753	8,643	3,347	5,731	4,696	3,570	3,579	3,596	3,590	3,617	3,940	3,922	4,504	0	0
	Processing Costs	US$/t Cu payable	1,820		1,691	1,897	1,948	1,595	1,775	1,533	1,772	1,782	1,793	1,793	1,808	1,971	1,971	2,263	0	0
	Smelting, refining, transportation costs	US$/t Cu payable	644		364	371	380	367	475	568	781	783	777	777	775	750	750	776	0	0
yes	**By Product Credit (Deduct)**	US$/t Cu payable	**-5,951**		**-7,709**	**-8,030**	**-8,495**	**-7,849**	**-4,454**	**-7,111**	**-5,831**	**-5,665**	**-5,411**	**-5,411**	**-5,297**	**-4,034**	**-4,034**	**-5,360**	**0**	**0**
yes	Mining Royalty	US$/t Cu payable	426		0	0	246	238	388	457	635	629	825	825	1,024	0	0	0	0	0
yes	On-site G&A and Infrastructure	US$/t Cu payable	773		1,752	843	865	649	1,032	843	630	639	643	643	648	707	707	1,152	0	0
	Cash Operating Costs (C1)	US$/t Cu payable	2,077		1,178	834	3,588	-1,653	4,946	986	1,558	1,746	2,224	2,218	2,574	3,335	3,317	3,335	0	0
	Cash Operating Costs (C1)	US$/lb Cu payable	0.94																	
	Sustaining Capital Costs (LOM)	US$M	**50.39**																	
59%	Co-Product Basis																			
	Copper																			
	Cash Costs																			
	Mining Costs	US$/t Cu payable	2,567		2,988	3,384	5,084	1,969	3,371	2,762	2,100	2,105	2,115	2,112	2,127	2,318	2,307	2,649	0	0
	Processing Costs	US$/t Cu payable	1,071		994	1,116	1,146	938	1,044	902	1,042	1,048	1,055	1,055	1,063	1,159	1,159	1,331	0	0
	Smelting, refining, transportation costs	US$/t Cu payable	379		214	218	224	216	279	334	459	460	457	457	456	441	441	457	0	0
yes	Mining Royalty	US$/t Cu payable	250		0	0	145	140	228	269	374	370	485	485	603	0	0	0	0	0
yes	On-site G&A and Infrastructure	US$/t Cu payable	455		1,031	496	509	382	607	496	371	376	378	378	381	416	416	678	0	0
	Cash Operating Costs (C1)	US$/t Cu payable	4,722		5,227	5,213	7,107	3,644	5,529	4,763	4,346	4,359	4,491	4,488	4,630	4,334	4,323	5,115	0	0
	Cash Operating Costs (C1)	US$/lb Cu payable	2.14		2.37	2.36	3.22	1.65	2.51	2.16	1.97	1.98	2.04	2.04	2.10	1.97	1.96	2.32	0.00	0.00
41%	Co-Product Basis																			
	Copper																			
	Cash Costs (Silver)																			
	Mining Costs	US$/t oz payable	7.55		6.78	7.38	10.48	4.39	13.25	6.80	6.30	6.50	6.84	6.83	7.03	10.06	10.01	8.65	0.00	0.00
	Processing Costs	US$/t oz payable	3.15		2.26	2.43	2.36	2.09	4.10	2.22	3.13	3.24	3.41	3.41	3.51	5.03	5.03	4.35	0.00	0.00
	Smelting, refining, transportation costs	US$/t oz payable	1.11		0.49	0.48	0.46	0.48	1.10	0.82	1.38	1.42	1.48	1.48	1.51	1.91	1.91	1.49	0.00	0.00
					0.00	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
yes	Mining Royalty	US$/t oz payable	0.74		0.00	0.00	0.30	0.31	0.90	0.66	1.12	1.14	1.57	1.57	1.99	0.00	0.00	0.00	0.00	0.00
yes	On-site G&A and Infrastructure	US$/t oz payable	1.34		2.34	1.08	1.05	0.85	2.38	1.22	1.11	1.16	1.22	1.22	1.26	1.81	1.81	2.21	0.00	0.00
	Cash Operating Costs (C1)	US$/t oz payable	13.89		11.87	11.36	14.64	8.13	21.73	11.72	13.05	13.47	14.53	14.52	15.30	18.81	18.76	16.70	0.00	0.00

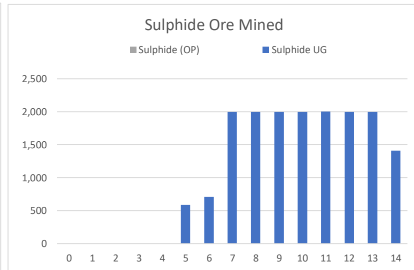
Selected Price Deck:	Scenario Oxide Only Pit & UG / Proc. Op 3	Consensus (Base Case)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16

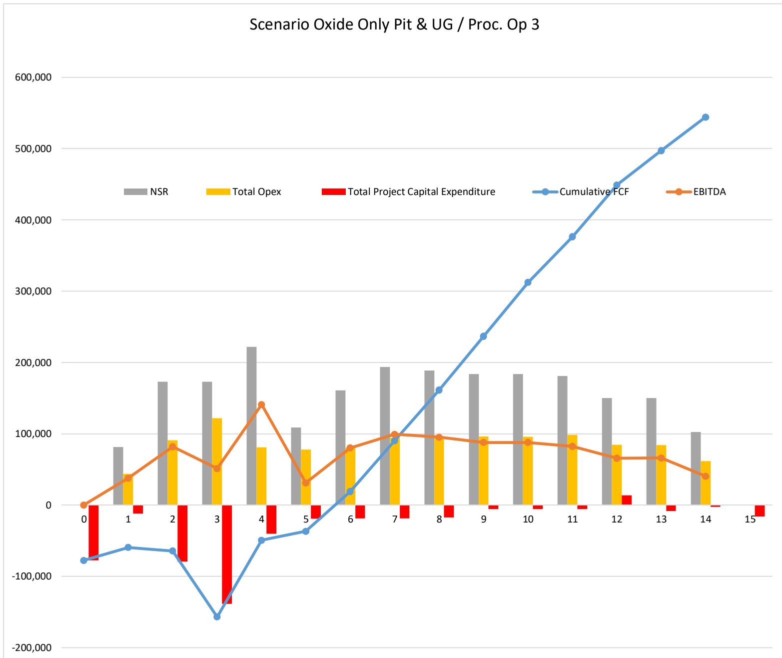


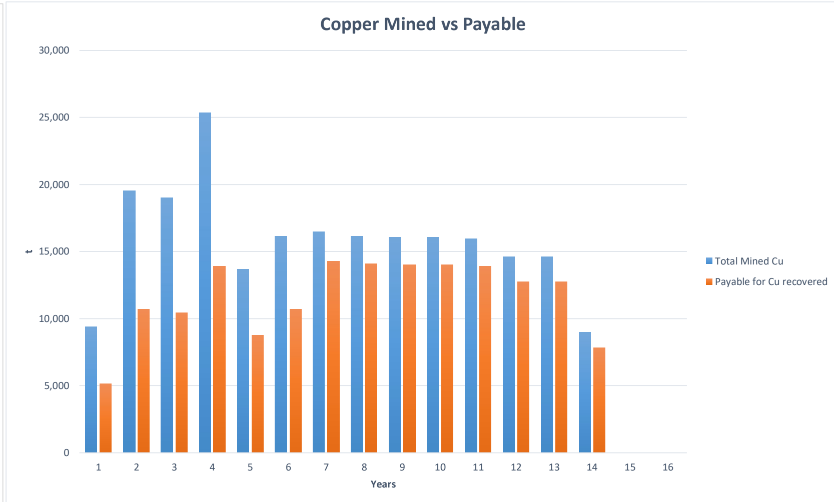


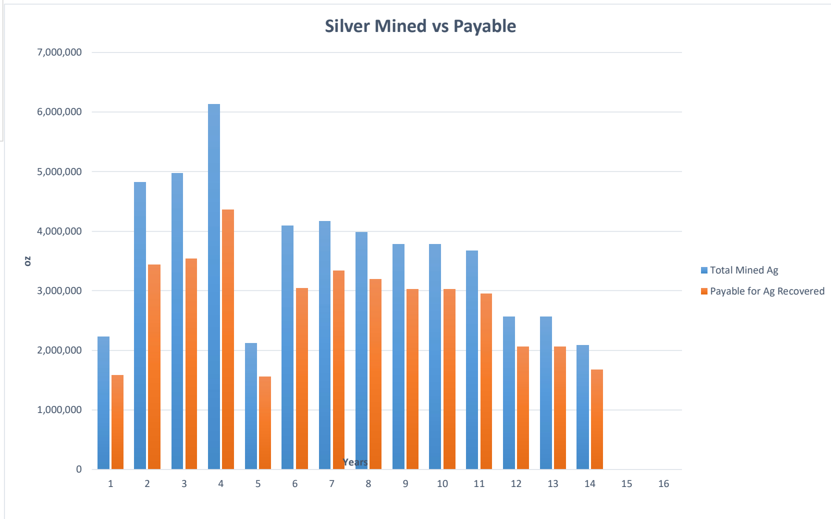












Selected Price Deck:		Scenario Oxide Only Pit & UG / Proc. Op 3	Spot (May 2021)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Metal Prices	**Spot (May 2021)**		**Units**	**TOTAL LOM**																	
	0%	Cu Price Selected Scenario per t	4.54 /lb	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000
	0%	Ag Price per oz		28	28	28	28	28	28	28	28	28	28	28	28	28	28	28	28	28	28
Production Schedule																					
		LOM	years	14																	
Open Pit		**Oxide Ore**	t'000	**10,981**		1,500	3,500	3,500	2,481	0	0	0	0	0	0	0	0	0	0	0	0
		Cu Grade	%	0.59%		0.63%	0.56%	0.54%	0.68%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
		Ag Grade	g/t	48.24		46.21	42.90	44.20	62.72	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
		Cu In-situ	t	**64,846**		9,393	19,534	19,023	16,897	0	0	0	0	0	0	0	0	0	0	0	0
		Ag In-situ	kg	**529,734**		69,315	150,147	154,689	155,584	0	0	0	0	0	0	0	0	0	0	0	0
		Sulphide	t'000	**0**		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Cu Grade	%	0.00%		0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
		Ag Grade	g/t	0.00		0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
		Cu In-situ	t	**0**		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Ag In-situ	kg	**0**		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Total Ore Mined (OP)	t'000	**10,981**		1,500	3,500	3,500	2,481	0	0	0	0	0	0	0	0	0	0	0	0
		Waste	t'000	**97,420**		13,431	31,789	47,677	4,522	0	0	0	0	0	0	0	0	0	0	0	0
		Rock	t'000	**108,401**		14,931	35,289	51,177	7,003	0	0	0	0	0	0	0	0	0	0	0	0
		Strip Ratio	W:O	**8.87**		8.95	9.08	13.62	1.82	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Underground added on following Oxide OP																					
		Oxide Ore	t'000	4,067		0	0	0	1,342	1,398	1,291	36	0	0	0	0	0	0	0	0	0
		Cu Grade	%	0.71%		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Ag Grade	g/t	41.34		0	0	0	26	35	64	66	0	0	0	0	0	0	0	0	0
		Cu In-situ	t	**28,963**		0	0	0	8,473	9,853	10,362	275	0	0	0	0	0	0	0	0	0
		Ag In-situ	kg	**168,105**		0	0	0	35,038	48,588	82,084	2,395	0	0	0	0	0	0	0	0	0
				8.36																	
		Sulphide	t'000	16,712		0	0	0	0	590	710	2,000	2,000	2,000	2,000	2,002	2,000	2,000	1,410	0	0
		Cu Grade	%	0.77%		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		Ag Grade	g/t	53.11		0	0	0	13	29	64	64	62	59	59	57	40	40	46	0	0
		Cu In-situ	t	**128,129**		0	0	0	0	3,828	5,752	16,203	16,153	16,048	16,048	15,934	14,600	14,600	8,963	0	0
		Ag In-situ	kg	**887,600**		0	0	0	1	17,256	45,153	127,192	123,930	117,599	117,599	114,313	79,749	79,749	65,059	0	0
		Total Ore Mined (UG)	t'000	**20,779**		0	0	0	1,342	1,987	2,001	2,036	2,000	2,000	2,000	2,002	2,000	2,000	1,410	0	0
	Royalty Rate	**TOTAL ORE MINED OP & UG**	t'000	31,759		1,500	3,500	3,500	3,823	1,987	2,001	2,036	2,000	2,000	2,000	2,002	2,000	2,000	1,410	0	0
Mined Metals	8.00%	Total Mined Cu	t	**221,939**		9,393	19,534	19,023	25,370	13,681	16,115	16,479	16,153	16,048	16,048	15,934	14,600	14,600	8,963	0	0
	6.50%	Total Mined Ag	oz	**50,973,028**		2,228,514	4,827,322	4,973,360	6,128,657	2,116,929	4,090,755	4,166,325	3,984,432	3,780,894	3,780,894	3,675,244	2,564,003	2,564,003	2,091,696	0	0
		Total Sulphide		**16,712**																	

Selected Price Deck:		Scenario Oxide Only Pit & UG / Proc. Op 3	Spot (May 2021)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Concentrate Production and Realisation																					
		Total Oxide	t'000	15,047		1,500	3,500	3,500	3,823	1,398	1,291	36	0	0	0	0	0	0	0	0	0
		Total Sulphide	t'000	16,712		0	0	0	0	590	710	2,000	2,000	2,000	2,000	2,002	2,000	2,000	1,410	0	0
Oxide Ore Treatment		Cu Recovery Rate	%	55.90%																	
		Ag Recovery Rate	%	73.40%																	
		Cu Recovered from Oxide Ore	t	52,439		5,251	10,919	10,634	14,182	5,508	5,793	154	0	0	0	0	0	0	0	0	0
		Ag Recovered from Oxide Ore	kg	512,213		50,877	110,208	113,542	139,917	35,664	60,249	1,758	0	0	0	0	0	0	0	0	0
		Ag grade in Concentrate (for Processing Optic	g/t	6,642		6,589	6,863	7,261	6,709	4,403	7,073	7,767	0	0	0	0	0	0	0	0	0
	68%	**Cu Concentrate Produced (for Processing Op**	dmt	77,117		7,722	16,058	15,638	20,856	8,099	8,518	226	0	0	0	0	0	0	0	0	0
Sulphide Ore Treatement		Cu Recovery to 26% Cu Concentrate	%	89%																	
		Ag Recovery to 26% Cu Concentrate	%	83%																	
		Cu Recovered to Cu Concentrate	t	114,163		0	0	0	0	3,411	5,125	14,437	14,392	14,299	14,299	14,197	13,008	13,008	7,986	0	0
		Ag Recovered to Cu Concentrate	kg	734,045		0	0	0	0	14,271	37,342	105,188	102,490	97,254	97,254	94,537	65,953	65,953	53,804	0	0
		Ag grade in Concentrate	g/t	1,672		0	0	0	566	1,088	1,894	1,894	1,851	1,768	1,768	1,731	1,318	1,318	1,752	0	0
	26%	**Cu Concentrate Produced**	dmt	439,090		0	0	0	1	13,119	19,712	55,528	55,356	54,996	54,996	54,605	50,031	50,031	30,715	0	0
Products		**Total Cu Concentrate Produced**	dmt	516,207		7,722	16,058	15,638	20,856	21,219	28,231	55,754	55,356	54,996	54,996	54,605	50,031	50,031	30,715	0	0
	99.0%	Total Payable: Cu in Cathode	t	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	99.0%	Total Payable: Ag in Bullions	kg	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	98%	Total Payable Cu in Cu Concentrate	t	163,271		5,146	10,701	10,421	13,898	8,740	10,699	14,299	14,105	14,013	14,013	13,913	12,748	12,748	7,826	0	0
	97%	Total Payable in Cu Concentrate	kg	1,208,871		49,351	106,901	110,135	135,719	48,436	94,663	103,737	99,415	94,337	94,337	91,701	63,974	63,974	52,190	0	0
Payable Assay Volumes		Payable for Cu recovered	t	163,271		5,146	10,701	10,421	13,898	8,740	10,699	14,299	14,105	14,013	14,013	13,913	12,748	12,748	7,826	0	0
		Payable for Ag Recovered	oz	38,866,100		1,586,658	3,436,956	3,540,933	4,363,483	1,557,259	3,043,494	3,335,238	3,196,272	3,032,995	3,032,995	2,948,244	2,056,818	2,056,818	1,677,937	0	0
Realisation Costs - Oxide Con	21	Treatment Charge for Oxide Concentrate	US$'000	1,619		162	337	328	438	170	179	5	0	0	0	0	0	0	0	0	0
Realisation Costs - Sulphide C	115	Treatment Charge for Sulphide Concentrate	US$'000	50,495		0	0	0	0	1,509	2,267	6,386	6,366	6,324	6,324	6,280	5,754	5,754	3,532	0	0
$/lb	0.07	Cu Refining Charge	US$'000	25,197		794	1,651	1,608	2,145	1,349	1,651	2,207	2,177	2,163	2,163	2,147	1,967	1,967	1,208	0	0
	0.5	Ag Refining Charge	US$'000	19,433		793	1,718	1,770	2,182	779	1,522	1,668	1,598	1,516	1,516	1,474	1,028	1,028	839	0	0
	8%	Concentrate available for Transportation	wmt	557,503		8,339	17,342	16,889	22,525	22,916	30,489	60,214	59,784	59,395	59,395	58,974	54,034	54,034	33,172	0	0
	15	Concentrate Transportation Cost	US$'000	8,363		125	260	253	338	344	457	903	897	891	891	885	811	811	498	0	0
Gross Revenue Payable		**Gross Revenue**	US$'000	2,720,957		95,883	203,243	203,355	261,161	131,005	192,212	236,379	230,542	225,053	225,053	221,685	185,071	185,071	125,244	0	0
		from Cu	US$'000	1,632,707		51,457	107,008	104,209	138,983	87,402	106,994	142,993	141,046	140,129	140,129	139,134	127,480	127,480	78,262	0	0
		from Ag	US$'000	1,088,251		44,426	96,235	99,146	122,178	43,603	85,218	93,387	89,496	84,924	84,924	82,551	57,591	57,591	46,982	0	0
		Realisation Costs	US$'000	-105,107		-1,875	-3,967	-3,960	-5,103	-4,150	-6,076	-11,168	-11,037	-10,894	-10,894	-10,785	-9,560	-9,560	-6,077	0	0
Net Smelter Return		**NSR**	US$'000	**2,615,851**		**94,008**	**199,276**	**199,395**	**256,058**	**126,855**	**186,136**	**225,211**	**219,504**	**214,158**	**214,158**	**210,899**	**175,511**	**175,511**	**119,168**	**0**	**0**

Selected Price Deck:		Scenario Oxide Only Pit & UG / Proc. Op 3	Spot (May 2021)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Production Costs																					
Mining																					
0%	1.75	Open pit - ore mining	US$'000	19,216		2,625	6,125	6,125	4,341	0	0	0	0	0	0	0	0	0	0	0	0
	1.75	Open pit - waste mining	US$'000	170,485		23,505	55,632	83,435	7,914	0	0	0	0	0	0	0	0	0	0	0	0
0%	22.00	UG ore mining	US$'000	457,129		0	0	0	29,527	43,718	44,017	44,797	44,008	44,000	44,000	44,045	44,000	44,000	31,018	0	0
	3.00	Backfilling	US$'000	62,336		0	0	0	4,026	5,962	6,002	6,109	6,001	6,000	6,000	6,006	6,000	6,000	4,230	0	0
		Interest on Leasing (OP Equipment)	US$'000	3,481		6	-191	509	709	409	229	149	469	389	309	269	229	0	0	0	0
Processing		Total Mining Opex	US$'000	**712,647**		**26,136**	**61,565**	**90,068**	**46,516**	**50,088**	**50,248**	**51,054**	**50,478**	**50,389**	**50,309**	**50,320**	**50,229**	**50,000**	**35,247**	**0**	**0**
	5.8	Oxide Treatement	$/t	5.80																	
	12.6	Sulphide Treatment	$/t	12.56																	
0%		Oxide Treatement	US$'000	87,273		8,700	20,300	20,300	22,171	8,106	7,486	210	0	0	0	0	0	0	0	0	0
0%		Sulphide Treatment	US$'000	209,956		0	0	0	1	7,408	8,920	25,126	25,131	25,126	25,126	25,152	25,126	25,126	17,713	0	0
		Total Processing Opex	US$'000	297,229		8,700	20,300	20,300	22,172	15,514	16,406	25,337	25,131	25,126	25,126	25,152	25,126	25,126	17,713	0	0
0%	5,000	G&A	US$'000	70,000		5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	0	0
	0.00	Fees to Closure and Reclamation	US$'000	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0%	4,016	Infrastructure	US$'000	56,218		4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	4,016	0	0
Tax Relief >>	yes	Royalty (Mineral Extraction Tax)	US$'000	80,370		0	0	2,956	3,812	3,931	5,639	10,506	10,261	13,384	13,384	16,497	0	0	0	0	0
Total Operating Costs		**Total Opex**	US$'000	1,216,464		43,852	90,881	122,340	81,516	78,548	81,309	95,912	94,885	97,915	97,835	100,984	84,370	84,142	61,976	0	0
				-883,153																	
Capital Expenditure																					
		Total Mining Capex		260,171																	
		Mining Capital Costs OP & UG (Leased Equ	US$'000	169,035	4,998	10,432	18,202	28,202	33,202	15,000	15,000	15,000	9,000	4,000	4,000	4,000	4,000	2,000	2,000	0	0
		Including Estimated OP Equipment Cost	*US$'000*	*70,035*	*24,990*	*21,735*	*23,310*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*
		Including Estimated UG Equipment Cost	*US$'000*	*105,000*	*0*	*0*	*0*	*50,000*	*25,000*	*0*	*0*	*0*	*20,000*	*0*	*0*	*0*	*0*	*10,000*	*0*	*0*	*0*
		Capitalised Interest	US$'000	136	136	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
		UG Mining Capital Development Cost	US$'000	85,000		0	50,000	25,000	0	0	0	0	5,000	0	0	0	0	5,000	0	0	0
	52.3	**Processing Capex - Oxide**	US$'000	52,300	52,300																
	1.50%	Processing Sustaining Capex Oxide	US$'000	3,138			785	785	785	785	0	0	0	0	0	0	0	0	0	0	0
	66.62	**Processing Capex - Sulphide**	US$'000	66,618	0	0	0	66,618	0	0											
	1.50%	Processing Sustaining Capex Sulphide	US$'000	8,993					999	999	999	999	999	999	999	999	999	0	0	0	0
	0.26 Mpa	TSF Wall Raise (Initial is included in Cost abov	US$'000	2,831	0	0	0	0	257	257	257	257	257	257	257	257	257	257	257	0	0
		Total UG		190,000																	
	10,057	**General Infrastructure Initial Capex**	US$'000	10,057	10,057																
		Closure and Reclamation Cost	US$'000	16,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	16,000	0
	0%	EPCM	US$'000	0	0																
	15%	Contingency	US$'000	57,472	10,124	1,565	10,230	17,973	4,980	2,250	2,250	2,250	2,100	600	600	600	600	1,050	300	0	0
		Assets Residual Value (Sold at the end of min	US$'000	-19,671	0	0	0	0	0	0	0	0	0	0	0	0	-19,671	0	0	0	0
0%		**Total Project Capital Expenditure**	US$'000	451,909	**77,615**	**11,997**	**79,217**	**138,577**	**40,223**	**19,291**	**18,507**	**18,507**	**17,357**	**5,857**	**5,857**	**5,857**	**-13,814**	**8,307**	**2,557**	**16,000**	**0**
Taxes & Depreciation																					
		Working Capital																			
		2 Months of Production				1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0
		Working Capital	US$'000	192,415		7,309	15,147	20,390	13,586	13,091	13,551	15,985	15,814	16,319	16,306	16,831	14,062	14,024	0	0	0
						7,309	7,838	5,243	-6,804	-495	460	2,434	-171	505	-13	525	-2,769	-38	-14,024	0	0
		Corporate Income Tax																			
Select Tax Option >>		Tax Rate Applied in the Valuation		9.12%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	10.0%	10.0%	10.0%	10.0%	10.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%
	1	Tax Relief Applied			0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	10.0%	10.0%	10.0%	10.0%	10.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%
	2	Tax Rate - Basic			20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%
	2	Post-Tax			1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	1	Pre-Tax			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
	2	Post-Tax			1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

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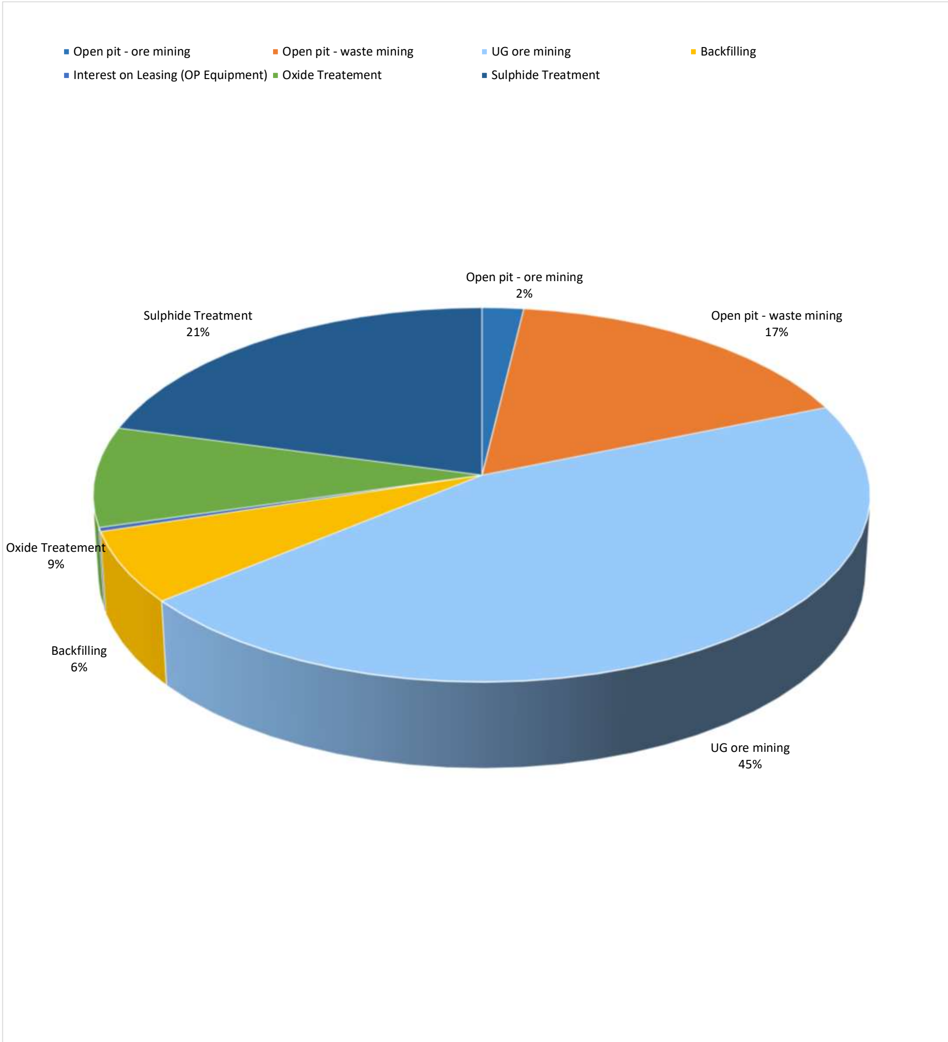
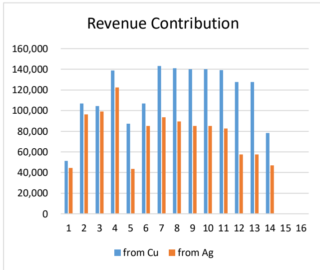
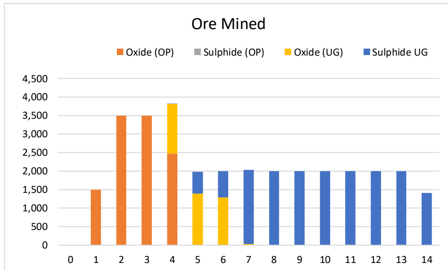
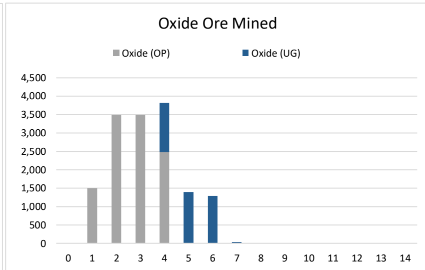
	Selected Price Deck:	Scenario Oxide Only Pit & UG / Proc. Op 3	Spot (May 2021)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
	CASH FLOW MODEL			US$'000																	
	Summary of Cash Flows	US$'M		TOTAL LOM																	
	Gross Revenue Payable			**2,720,957**	0	95,883	203,243	203,355	261,161	131,005	192,212	236,379	230,542	225,053	225,053	221,685	185,071	185,071	125,244	0	0
10	Less Smelter and Realisation Costs			**-105,107**	0	-1,875	-3,967	-3,960	-5,103	-4,150	-6,076	-11,168	-11,037	-10,894	-10,894	-10,785	-9,560	-9,560	-6,077	0	0
	NSR			**2,615,851**	0	94,008	199,276	199,395	256,058	126,855	186,136	225,211	219,504	214,158	214,158	210,899	175,511	175,511	119,168	0	0
47%	**Less Operating Costs, including:**			**-1,216,464**	0	-43,852	-90,881	-122,340	-81,516	-78,548	-81,309	-95,912	-94,885	-97,915	-97,835	-100,984	-84,370	-84,142	-61,976	0	0
59%	Mining			**-712,647**	0	-26,136	-61,565	-90,068	-46,516	-50,088	-50,248	-51,054	-50,478	-50,389	-50,309	-50,320	-50,229	-50,000	-35,247	0	0
24%	Processing			**-297,229**	0	-8,700	-20,300	-20,300	-22,172	-15,514	-16,406	-25,337	-25,131	-25,126	-25,126	-25,152	-25,126	-25,126	-17,713	0	0
6%	G&A			**-70,000**	0	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	-5,000	0	0
0%	Closure and Reclamation Fees			**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
5%	Infrastructure			**-56,218**	0	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	-4,016	0	0
7%	Royalty (Mineral Extraction Tax)			**-80,370**	0	0	0	-2,956	-3,812	-3,931	-5,639	-10,506	-10,261	-13,384	-13,384	-16,497	0	0	0	0	0
	Cash Costs (C1, including Silver Credits)			1,429		-719	-597	2,126	-2,918	3,901	-265	608	810	1,348	1,342	1,741	2,458	2,440	2,053	0	0
	Cash Costs (C1, excluding Silver Credits)			7,028		7,389	7,864	11,099	5,344	8,108	6,792	5,820	5,831	6,089	6,083	6,357	5,684	5,666	6,738	0	0
	EBITDA			**1,399,387**	**0**	**50,157**	**108,395**	**77,055**	**174,543**	**48,307**	**104,827**	**129,299**	**124,619**	**116,244**	**116,324**	**109,915**	**91,141**	**91,369**	**57,192**	**0**	**0**
	Less Depreciation			**-282,983**	0	-7,373	-7,813	-14,596	-26,374	-27,690	-26,892	-26,095	-25,374	-24,613	-22,831	-21,218	-19,759	-16,570	-15,785	0	0
Select >> Post-Tax	**Taxable Income**			**1,116,404**	0	42,783	100,583	62,459	148,169	20,617	77,935	103,204	99,245	91,631	93,493	88,697	71,382	74,800	41,408	0	0
Tax Rate with Holidays	*Less Corporate Income Tax*			**-101,808**	0	0	0	0	0	0	-7,794	-10,320	-9,925	-9,163	-9,349	-17,739	-14,276	-14,960	-8,282	0	0
	Less Change in NWC (Net Working Capital)			**0**	0	-7,309	-7,838	-5,243	6,804	495	-460	-2,434	171	-505	13	-525	2,769	38	14,024	0	0
Post-Tax	**Net Operating Profit**			**1,297,579**	0	42,848	100,557	71,811	181,347	48,802	96,573	116,545	114,866	106,575	106,988	91,651	79,633	76,447	62,934	0	0
	Less Project Capital Costs			**-451,909**	-77,615	-11,997	-79,217	-138,577	-40,223	-19,291	-18,507	-18,507	-17,357	-5,857	-5,857	-5,857	13,814	-8,307	-2,557	-16,000	0
Post-Tax	**Free Cash Flow to Firm**			**845,670**	-77,615	30,852	21,341	-66,766	141,124	29,511	78,067	98,038	97,510	100,719	101,131	85,795	93,448	68,140	60,377	-16,000	0
Post-Tax	**Cumulative FCF**				-77,615	-46,763	-25,422	-92,188	48,935	78,446	156,513	254,551	352,061	452,780	553,911	639,705	733,153	801,293	861,670	845,670	845,670
				1																	
Post-Tax	**NPV @ Discount Rate of 8.00%**	US$ M		380,410																	
Post-Tax	IRR	%		44.40%																	
	Payback period of capital (FCF)	Years		4																	
	By-Product Basis																				
	Cash Costs per t of Copper																				
	Mining Costs	US$/t Cu payable		4,365		5,079	5,753	8,643	3,347	5,731	4,696	3,570	3,579	3,596	3,590	3,617	3,940	3,922	4,504	0	0
	Processing Costs	US$/t Cu payable		1,820		1,691	1,897	1,948	1,595	1,775	1,533	1,772	1,782	1,793	1,793	1,808	1,971	1,971	2,263	0	0
	Smelting, refining, transportation costs	US$/t Cu payable		644		364	371	380	367	475	568	781	783	777	777	775	750	750	776	0	0
yes	**By Product Credit (Deduct)**	US$/t Cu payable		**-6,665**		**-8,634**	**-8,993**	**-9,514**	**-8,791**	**-4,989**	**-7,965**	**-6,531**	**-6,345**	**-6,060**	**-6,060**	**-5,933**	**-4,518**	**-4,518**	**-6,003**	**0**	**0**
yes	Mining Royalty	US$/t Cu payable		492		0	0	284	274	450	527	735	727	955	955	1,186	0	0	0	0	0
yes	On-site G&A and Infrastructure	US$/t Cu payable		773		1,752	843	865	649	1,032	843	630	639	643	643	648	707	707	1,152	0	0
	Cash Operating Costs (C1)	US$/t Cu payable		1,429		253	-130	2,606	-2,559	4,473	203	958	1,165	1,705	1,699	2,100	2,851	2,833	2,692	0	0
	Cash Operating Costs (C1)	US$/lb Cu payable		0.65																	
	Sustaining Capital Costs (LOM)	US$M		**50.39**																	
60%	Co-Product Basis																				
	Copper																				
	Cash Costs																				
	Mining Costs	US$/t Cu payable		2,619		3,048	3,452	5,186	2,008	3,439	2,818	2,142	2,147	2,158	2,154	2,170	2,364	2,354	2,702	0	0
	Processing Costs	US$/t Cu payable		1,092		1,015	1,138	1,169	957	1,065	920	1,063	1,069	1,076	1,076	1,085	1,183	1,183	1,358	0	0
	Smelting, refining, transportation costs	US$/t Cu payable		386		219	222	228	220	285	341	469	470	467	467	465	450	450	466	0	0
yes	Mining Royalty	US$/t Cu payable		295		0	0	170	165	270	316	441	437	573	573	711	0	0	0	0	0
yes	On-site G&A and Infrastructure	US$/t Cu payable		464		1,051	506	519	389	619	506	378	384	386	386	389	424	424	691	0	0
	Cash Operating Costs (C1)	US$/t Cu payable		4,857		5,332	5,319	7,273	3,740	5,678	4,901	4,493	4,506	4,659	4,656	4,820	4,421	4,411	5,218	0	0
	Cash Operating Costs (C1)	US$/lb Cu payable		2.20		2.42	2.41	3.30	1.70	2.58	2.22	2.04	2.04	2.11	2.11	2.19	2.01	2.00	2.37	0.00	0.00
40%	Co-Product Basis																				
	Copper																				
	Cash Costs (Silver)																				
	Mining Costs	US$/t oz payable		7.33		6.59	7.16	10.17	4.26	12.86	6.60	6.12	6.32	6.64	6.63	6.83	9.77	9.72	8.40	0.00	0.00
	Processing Costs	US$/t oz payable		3.06		2.19	2.36	2.29	2.03	3.98	2.16	3.04	3.14	3.31	3.31	3.41	4.89	4.89	4.22	0.00	0.00
	Smelting, refining, transportation costs	US$/t oz payable		1.08		0.47	0.46	0.45	0.47	1.07	0.80	1.34	1.38	1.44	1.44	1.46	1.86	1.86	1.45	0.00	0.00
						0.00	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
yes	Mining Royalty	US$/t oz payable		0.83		0.00	0.00	0.33	0.35	1.01	0.74	1.26	1.28	1.76	1.76	2.24	0.00	0.00	0.00	0.00	0.00
yes	On-site G&A and Infrastructure	US$/t oz payable		1.30		2.27	1.05	1.02	0.83	2.32	1.18	1.08	1.13	1.19	1.19	1.22	1.75	1.75	2.15	0.00	0.00
	Cash Operating Costs (C1)	US$/t oz payable		13.60		11.53	11.04	14.27	7.94	21.24	11.48	12.84	13.25	14.35	14.34	15.16	18.26	18.22	16.22	0.00	0.00

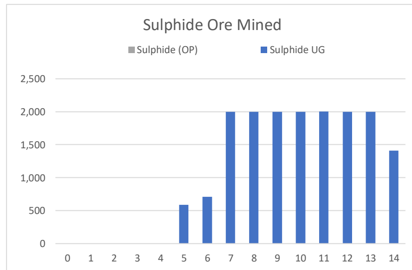
Selected Price Deck:	Scenario Oxide Only Pit & UG / Proc. Op 3	Spot (May 2021)	LOM	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16

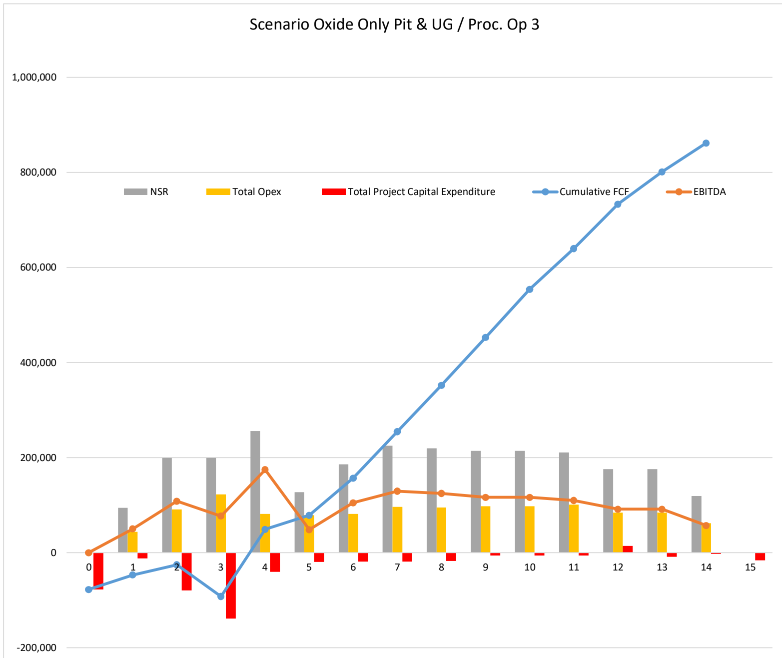


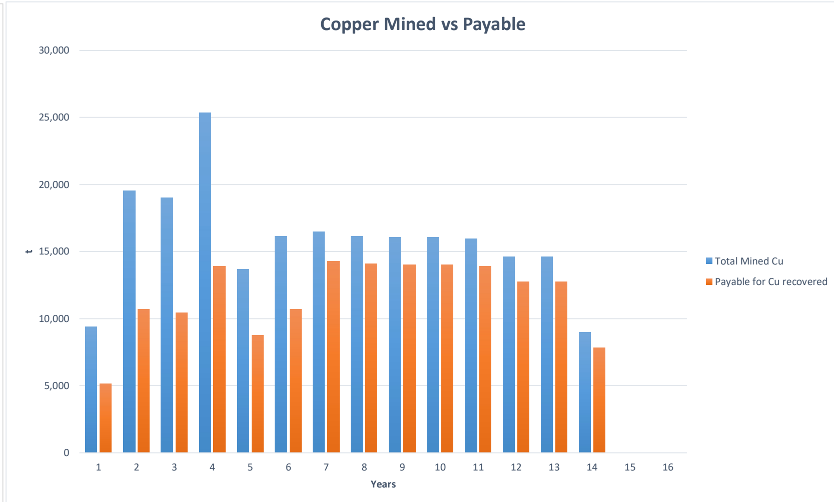


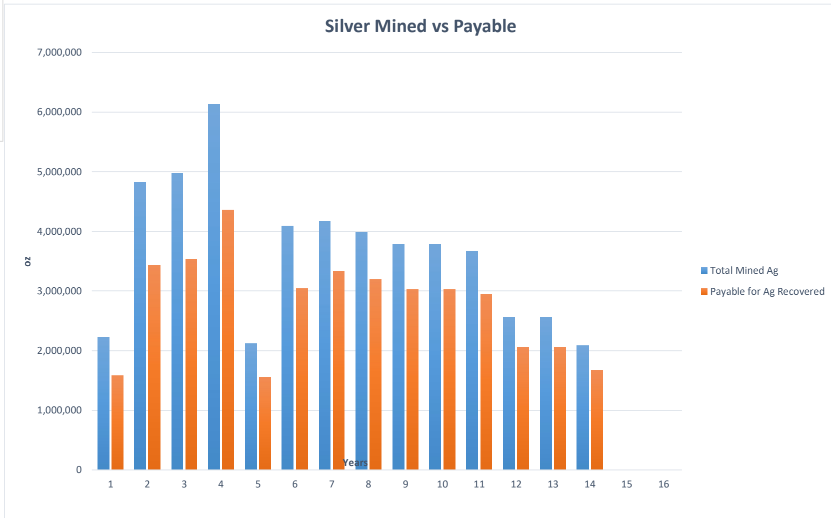












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