February 22, 2016

TECHNICAL REPORT

TECHNICAL REPORT AND OCTOBER 2015 MINERAL RESOURCE ESTIMATE

THE KAJARAN COPPER-MOLYBDENUM MINE, KAJARAN ARMENIA

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Distribution:

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1.0 SUMMARY

1.1 Executive Summary

Golder Associates Ltd. (Golder) was contracted by Cronimet Mining AG (Cronimet) to provide an independent Mineral Resource estimate and Technical Report on the Zangezur Copper-Molybdenum Mine deposit, located near the town of Kajaran, in the southern region of the Republic of Armenia. The Kajaran Mine is currently operated by Zangezur Copper-Molybdenum Combine (ZCMC).

The last previous resource estimate (non-NI 43-101) was completed in 2005. The resource estimate supported by this Technical Report uses diamond drill data from 776 exploration holes, approximately 120 of which were completed between 2006 and 2012, as well as updated metal pricing, process recoveries, and production costs.

The qualified persons (QP) responsible for the preparation of this Technical Report (dated February 22, 2016) is Greg Greenough, P.Geo., of Golder.

1.2 Property Description and Location

The Kajaran project is situated in the South-Eastern part of Armenia, with the closest borders being Azerbaijan and Turkey. The ZCMC Kajaran Project is located in the Voghji valley which is in the Syunik province, the second largest in Armenia. The Syunik province occupies 4,506 square km. (15.15% of the territory of Armenia) and has a population of about 153 thousand.

Access is by paved highway to the town of Kajaran where the processing plant is located, and the open pit is 2km from the town. Total travel time to the Kajaran Project from Yerevan is approximately five and half hours. The only other access is a small airport operating at Kapan, 30km from the mine. Non-regular flights between Yerevan and Kapan take about an hour.

The Kajaran climate is relatively continental, with dry, hot summers and cold and short winters.

The topography of the Kajaran area is one of high relief with mountains up to nearly 4,000m elevation in close proximity to the mine. The actual open pit operates at an elevation of between 1,920 to 2,360 m, while the processing plant is located between the old Kajaran site and the new town, at approximately 1,750 m above sea level.

Mineral tenure is wholly owned by ZCMC under all required governmental permits and contracts, as detailed in Section 4.2.

Royalties are paid to the Republic of Armenia under the Nature Protection and Nature Usage Payments law, as defined by Article 13.3., Calculation Basis of Royalty, and Article 13.4., Rate of Royalty, the details of which can be found in Section 4.3.

With regard to the environmental policy and spending, the ZCMC Kajaran operations are fully compliant with national legislation, and the Company does not have any outstanding litigation cases, legacy issues or historical claims.

The mitigation of environmental liabilities is an important part of the Kajaran project, and is carried out through structured processes such as community consultation, grievance mechanisms, and ongoing improvements.

Details of some of the recent steps taken by ZCMC to mitigate environmental impacts are included in Section 4.4.

1.3 Infrastructure

The Kajaran project is located in a well-developed region of Armenia, where the power supply is secured through the national electricity network.

Other than during severe winter storms, paved road access from the plant to Yerevan provides reliable transportation for people, supplies and concentrate.

Both process and drinking water is obtained reliably year round from the Voghji and Geghi Rivers.

There are three waste rock dumps at the Kajaran Project. The “North-West” dump was the first one established, and low grade oxide ore and barren rock were disposed here between 1951 and 1970. The Spitak Djur (previously called Sakhkar Su) dump is located on the South-Eastern edge of the mine in the Sakhkar River valley, and was started in 1971. The Dzorategh (previously called Darazami) dump is located 1.5 km to the East from the Spitak Djur dump and has been in operation from 1979 to today.

The silty sand tailings slurry from the process plant is transported by pipeline about 30km E-NE from the mine to the Artsvanik tailings dam, and has been since 1977. There is still about 120 Mm^3 of space available to tailings to reach the design elevation of 964 m.

1.4 Geology and Mineralization

The Kajaran Cu-Mo porphyry deposit is located in Southern Armenia, part of the Lesser Caucasus, and is hosted by the Tertiary Meghri (Meghri-Ordubad) pluton, lying within the Tethyan metallogenic belt.

The Zangezur-Ordubad region, sitting astride the territories of southern Armenia and Nakhichevan, is bounded to the east by the northwest-oriented Khustup-Giratakh fault. Basement rocks in the region include Devonian, Permian, Jurassic and Lower Cretaceous sedimentary rocks, followed by Upper Cretaceous and Paleocene sedimentary rocks. Into these sedimentary basement rock successive Eocene to Miocene magmatic pulses of olivine gabbro, gabbro-monzonite, syenite, syenogranite, monzonite, quartz-diorite, monzodiorite, granodiorite, and porphyritic granite were introduced.

The Kajaran deposit is hosted by one of these monzonite intrusions (31.83 ±0.02Ma; Moritz et al. 2013) belonging to the Meghri composite pluton. After emplacement of the monzonite intrusion, the tectonic evolution of the Kajaran area was characterized by a succession of faults and fractures, which controlled further emplacement of magmatic rocks and mineralization. These structures were the controlling factor for the successive introduction of intrusive dykes. The magmatic evolution at the Kajaran property and deposit ends with the formation of thick, coarse-grained granodiorite-porphyry dykes, cross-cutting the deposit’s mineralization.

The economic material of the deposit is located on the eastern side of the north-west oriented (320-340^{O}) Tashtun (previously called Debaklin) fault, which dips at approximately 45 degrees to the east-north-east. Copper-molybdenum stock-works are recognized along a more than 3.5 km long and 2 km-wide corridor. West of the fault, the porphyry granite is devoid of any mineralization and pronounced alteration.

Mineralization at the Kajaran deposit consists of a number of quartz-sulphide assemblages, with the economic stages being quartz-molybdenite, quartz-molybdenite-chalcopyrite, and quartz-chalcopyrite. Some oxidation of the sulphides occurs, most notably closer to surface, but also coincidental with more major vertical structures.

1.5 Exploration

Exploration at Kajaran started in the 1930's and continued in phases, starting with underground drifting, trenches pits and short adits (1931-1951) following quartz-molybdenite veins and zones of intensive mineralization. Underground drifting continued from 1952, with the addition of some underground drilling.

Phase by phase, underground exploration was replaced by drilling of vertical holes from surface, and from 1967 to 1984 the deposit was explored only by drill holes. Holes were drilled along the South-North exploration lines, ranging from 50, to 100, to 200 m spacing.

The current owner initiated diamond drilling in 2005 and continued till 2012. The main purpose of the drilling was to infill existing drilling in the Shlorkut area, along the edges of the Central area, and in the Eastern part of Left Bank area. A total of 131 holes, which included a few twinned holes, were drilled within this period.

In the autumn of 2015, the company drilled 9 holes, 3 for hydrological purposes and other 6 were drilled at the Northern edge of the Central Area of the deposit to fill gaps and verify the historical data. This 2015 drill hole data is not included in the current resource estimation as assay results were not yet completed.

Due to the shortage of data verification information on much of the historic drilling, an important part of this Mineral Resource estimate was to confirm assay results through a program of half-core and pulp duplicate assays at an accredited lab, the details of which can be found in Section 12.0

1.6 Mineral Resource and Mineral Reserve Estimates

The October 1, 2015 Mineral Resource estimate for the Kajaran project is the first NI 43-101 resource estimate on the property. The estimate was completed by G. Greenough, P.Geo. of Golder using traditional geostatistical block modelling techniques using Datamine Studio v3 (Datamine), and reviewed by Golder senior geological staff.

A site visit was conducted in July 2015 by G. Greenough (Section 12.2).

Geological modelling consisted of a mineralization envelope, with its boundary defined by the natural break in Cu and Mo mineralization. This was determined in large part by the location of the Tashtun on the west, and the general depth of diamond drill holes on the bottom. Due to the complexities in shape of the cross-cutting dykes, these were modelled using a statistical approach that generated small blocks (5m x 5m x 5m) following the general trend of the dykes (Section 14.3.1), and honouring the expression:

total dyke block volumes / *total block model volume* = *total drill hole dyke length* / *total drill hole length*

Block estimates included Total Copper (Cu), Copper Oxide (Cu Oxide), Total Molybdenum (Mo), Molybdenum Oxide (Mo Oxide), Gold (Au) and Silver (Ag). Although there is no differentiation between sulphide and oxide processing at Kajaran, there is a threshold above which the high oxide content material must be stockpiled and blended. The oxide grades are included in the model therefore to provide assistance in the mine planning process.

Cu, Mo, Cu Oxide and Mo Oxide modelling used a total of 198,590 m of historic and recent drill data, from 121,830 intervals in 750 holes (includes dyke intervals), satisfactorily confirmed by the Cu and Mo check assay program described in Section 12.0. Au and Ag estimates used a total of 43,264 m of 925 *composite* assayed intervals from 221 holes, also used in the previous Mineral Resource estimate (Kaputin, 2005). Results from the pulp check assay analysis (Section 12.0) shows a significant increase in the Au global average from pulp assays when compared to the Au global average of the composites used in the estimate and a moderate decrease in pulp Ag assay results. A direct hole-to-hole comparison was not possible since the pulps are from holes drilled after the Au composites were analyzed. Considering the other factors involved, such as process recoveries and contribution to value, Golder is satisfied that although the Au and Ag composite grades used may produce conservative Au and slightly optimistic Ag interpolations, they will not have a material effect on this resource estimate. This does however highlight the need for further work on Au and Ag assay data (see Conclusions and Recommendations).

Due to the polymetallic nature of the Kajaran deposit, and since Cu is the greatest overall economic contributor to value, Copper Equivalent (CuEq%) grades were calculated in the estimated block model, based on projected selling prices, agreed upon by ZCMC and Golder and actual metal recoveries and selling costs (historical data) provided by ZCMC) (Table 1-1). The use of the CuEq allows for a more robust method of determining the mineral resource for Cu, Mo, Au and Ag and long-term planning.

Table 1-1: Kajaran CuEq Economic Parameters

Element	Recovery	Selling Price	Selling Cost
Cu	78.7%	$2.68/lb.	$0.98/lb.
Mo	83.6%	$7.67/lb.	$0.12/lb.
Au	70%	$1250/oz.	$570.75/oz.
Ag	75%	$18/oz.	$10.43/oz.

Specific Gravity (SG) data from 76 samples, where the variation in SG was very small) was used to provide a default value of 2.55 to all solid material in the block model.

Variogram analysis was done for all interpolated elements, and the results used to determine search parameters and orientations used in the Ordinary Kriged estimates. Cu and Mo continuity (variography) suggested a minor preferred orientation direction, generally following the orientation of the Tashtun fault. Cu and Mo Oxides used a dynamic search approach, where the search parameters follow the original topographic surface. Nearest Neighbour estimates were also done for each element for block model validation purposes.

Validation of the block model was done through visual inspection, swath plots, and global statistical comparisons of Kriged and Nearest Neighbour estimates. These steps all showed that the block model estimates performed as expected.

Since mill production records are available for the Kajaran project, reconciliation to the mill was possible on quarterly and annual bases from 2012 to 2015 (Section 14.9.1). Results of this exercise were positive, with quarterly differences between the block model and mined chosen volumes within the industry accepted guideline of 15%. This supported the classification of Measured Resources in certain areas of the block model.

Classification was generally based on those blocks estimated for Cu and Mo in the first search (or, within the sill range of the variogram) being assigned a classification of Measured Resource. Visual inspection of the Measured areas shows support for this classification, with much higher drill densities, as well as proximity to those areas used in the mill reconciliation.

In order to support the definition of resource as having the potential for eventual economic extraction, a preliminary open pit assessment of the Kajaran deposit was performed on the resource block model using Whittle v4.5.2 (Whittle), with the open pit costs, selling prices and costs, and metallurgical recovery factors (Section 14.12.1). An overall average slope angle of 45 degrees was assumed, and a base case (break-even) pit shell generated using all blocks in the resource model lying below the October 1, 2015 surveyed mining surface.

As mentioned above, a CuEq value was applied to the block model and a CuEq cut-off determined for the reporting of resources within the Whittle pit shell, which are presented in Table 1-2.

Table 1-2: Kajaran Mineral Resources October 1, 2015

Cut-off CuEq%	Classification	Tonnes (000)	Cu%	Mo%	Au g/t	Ag g/t	Cu Oxide%	Mo Oxide %
0.21	Measured	605,420	0.278	0.036	0.014	1.413	0.022	0.003
	Indicated	1,328,401	0.246	0.031	0.018	1.660	0.020	0.002
	Total Resources	1,933,821	0.256	0.032	0.017	1.582	0.021	0.002

Note: Cu% and Mo% are Total Cu and Total Mo

Of the 1,328 billion tonnes of Indicated Resource, 1,017 billion tons @ 0.252% Cu, 0.028% Mo, 0.020g/t Au, and 1.797g/t Ag lie north of the Voghji River and will require diversion of the river and relocation of the village. ZCMC is confident that the permitting and logistics required is feasible, therefore it is reasonable to assume that this resource has the potential for eventual economic extraction.

1.7 Conclusions

An NI 43-101 Mineral Resource estimate has been successfully carried out on ZCMC's Copper-Molybdenum deposit at Kajaran Armenia. Golder is satisfied that the data has been confirmed to a level that is sufficient for a Mineral Resource estimate, through a comprehensive program of half-core and pulp reject assays, as well as reconciliation to mill production for the last 5 years.

The Mineral Resource is based on reasonable economic parameters, including reliable production costs from historical data, and the application of Whittle open pit optimization.

The resources stated are based on a Whittle analysis covering the entire deposit. Current mine plans are constrained by current permitting, and recovery of the 1,017 billion tonnes of Indicated Resource north of the

Voghji River relies on required permitting. Geological information north of the river is also not as detailed, and will need additional drilling to increase confidence in all economic minerals in the deposit, as well as density.

SG determinations are only available for 76 samples, and although widely scattered, with a low variation in values, almost all are in areas south of the Voghji River. Also, none of the 76 SG samples represent cross-cutting dyke material. It has been assumed that the dyke SG will not be dissimilar to the mineralized and un-mineralized porphyries, but confirmation of this is required.

Check assays of pulp rejects show the possibility of a minor negative bias in the historic Mo grades.

Check assays from 2,165 pulp rejects showed through global average comparison to the composite Au assays used in the resource estimate that there is the probability that Au grades in the resource are being under-reported. This is supported by the fact that monthly composite Au assay (12 month period) average at the mill agrees quite favourably to that of the pulp assays. The average of the pulps Ag grades would indicate a possible slight over-reporting in the Ag resource grades, but this is not supported as well by mill grades since none were available.

The pulp assays also suggest a potential correlation between Cu and Au (30%), as well as Cu and Ag (54%) These correlations increase substantially with the removal of a few outliers in the data (62% for Au, 60% for Ag). This raises the potential that with enough data and a stronger correlation, Au content could be applied to the resource model (and planning) through its association with Cu.

The pulp analysis also shows a high correlation (89%) between Mo and Rhenium (Re), which is recovered from the Mo concentrate. This correlation is enough to include Re in the resource block model using a regression formula, but additional assay data to confirm this correlation is suggested.

A CuEq attribute has been implemented to include Cu, Mo, Au, and Ag in the cut-off determination for more robust economic evaluations and mine planning. With Re contributing value to the ore, adding it to the CuEq calculation would make economic evaluation of the deposit even more robust.

1.8 Recommendations

Based on findings during the course of this resource estimate and conclusions listed above, Golder recommends the following actions to improve the Kajaran project:

- Diamond drilling north of the Voghji River to increase confidence, possibly increasing some resources to Measured. A suggested initial phase would concentrate in areas of higher grade indicated in the block model:
  - 40 holes @ depth of 500 m, at a cost of US$ 200/m (drilling, handling, assaying), for a total of 20,000 m at a cost of US$4.0 million.
  - If there is potential to deepen the pit even further than current plans, then drilling to extend below the limits of existing drilling is suggested. Amount and costs would be determined at that time.
- Additional SG sampling, particularly north of the river and in dyke material, through representative grab sampling and drilling;

- Additional Au and Ag assay data from remaining pulp rejects, production holes, new diamond drill holes; also daily mill feed and tailings Au and Ag assays to more accurately calculate and predict recoveries;
- Microscopic and high resolution material analysis, such as QEMSCAN (Quantitative Evaluation of Minerals by SCANning electron microscopy) to further understand relationships between Au and Ag to Cu, as well as other relationships that may arise;
- Obtain additional pulp reject assays to confirm the Re – Mo correlation, and include Re in the CuEq calculation and future resource estimates.
- Assay as many remaining pulp rejects as possible to further investigate the possibility of a negative bias on historic Mo grades, and provide support for adjustments to the historic grades if required.

2.0 INTRODUCTION

Golder Associates Ltd. (Golder) has completed an independent mineral resource estimate in conformance with the CIM Mineral Resource and Mineral Reserve definitions referred to in National Instrument NI 43-101 (Standards of Disclosure for Mineral Projects) for the Zangezur Copper-Molybdenum Mine, located close to the town of Kajaran, in the republic of Armenia, along with this supporting technical report as defined by NI 43-101 and the Form 43-101F1 (the “technical report”).

The qualified person (QP) responsible for the preparation of the October 1, 2015 Kajaran Mineral Resource Estimate and Technical Report is Greg Greenough, P.Geo.

Mr. Greenough completed a site visit of the Zangezur property on July 6, 2015. Although no diamond drilling was being performed at the time of the site visit, half-core stored from post-2005 exploration drill programs was available and inspected. Production RC drilling was also inspected during the site visit, along with exposed faces and blasted material within the existing open pit. Sulphide mineralization containing Cu and Mo was observed in quantities consistent with the reporting of this deposit.

The effective date of the Zangezur Mine Technical Report is February 22, 2016.

2.1 Sources of Information

The original sources of information that were used in the preparation of the independent Mineral Resource estimate and Technical Report were provided by ZCMC under the direction of Oscar Alvarado. Some historic information was also referenced from sections of the December 2005 report by IFB Mining Consultants titled “Resource Modeling and Estimation, Calculation of Recoverable Reserves”.

The mineral resource estimate is based primarily on the historical diamond drill hole data provided by ZCMC. Golder has reviewed the sources of the data provided by ZCMC through validation checks of the digital data provided (comma-delimited files from ZCMC’s Access database) and against a selection of the original source data provided by ZCMC. Observations made during the site visit and data validation procedures indicate that ZCMC’s Access database is sufficiently free of errors to be used in the 2015 mineral resource estimate.

2.2 Terms of Reference (Abbreviations)

All units of measure (Figure 2-1) used in this report are in the metric system, unless stated otherwise. The contained metal quantities shown in the mineral resource estimate for Copper (Cu) and Molybdenum (Mo) are in percentage (%) and pounds (lb), while Gold (Au) and Silver (Ag) are expressed in grams/tonne (g/t) and troy ounces (oz). Currencies outlined in the report are in US dollars unless otherwise stated.

Capital expenditure	CAPEX
Centimetre	cm
Cubic centimetre	cm^3
Cubic metre	m^3
Degree	°
Degrees Celsius	°C
Gold	Au
Gram	g
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Internal rate of return	IRR

Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per square metre	kg/m^2
Kilometre	km
Less than	<
Metre	m
Metres above sea level	masl
Millimetre	mm
Million	M
Million tonnes	Mt
Million tonnes per annum	Mtpa
Operating expense	OPEX
Ounce	Oz
Ounce per tonne	opt
Per cent	%
Parts per million	ppm
Parts per billion	ppb
Relative Percentage Difference	RPD
Square km	km^2
Square metre	m^2
Short Tons (907 kgs)	tons
Tonnes (1000 kgs)	t
Tonnes per day	t/d

Figure 2-1: Units of Measure and Abbreviations

2.3 Conversions

The following conversions were used in the preparation of this report.

- 1 troy ounce = 31.1035 g
- 1 pound = 14.5833 troy ounce
- 1 tonne = 2,204.627 lb

Specific gravity (SG) has been used throughout this report and refers to the bulk density as g/cm^3.

3.0 RELIANCE ON OTHER EXPERTS

This report has been prepared for ZCMC by Golder, and the information, conclusions, opinions and estimates contained within are based on:

- Information available to Golder at the time of this report's preparation;
- Assumptions, conditions and qualifications as set forth in this report; and,
- Data, reports and other information provided by ZCMC and other third party sources.

ZCMC were relied upon with respect to information on mineral tenure, environment, accessibility and permitting.

Other than for purposes legislated by securities laws, the use of this report by any third party is solely at that party's risk.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Area and Location

The Kajaran project is situated in the South-Eastern part of Armenia (see Figure 4-1). The country is landlocked in the Lesser Caucasus Mountains (part of Armenian Highland), bordering Georgia, Azerbaijan, Iran and Turkey, with only the latter two open for trade.

The country's borders with Azerbaijan and Turkey have been closed since 1991 and 1993 respectively, as a result of Armenia's conflict with Azerbaijan. Armenia's economy declined in the 2009 global downturn, but growth has been positive since then.

Armenia is a presidential representative republic, independent since 21 September 1991. It has 11 provinces, a population of 3,060,631 (July 2014 est.), with Yerevan as the capital city.

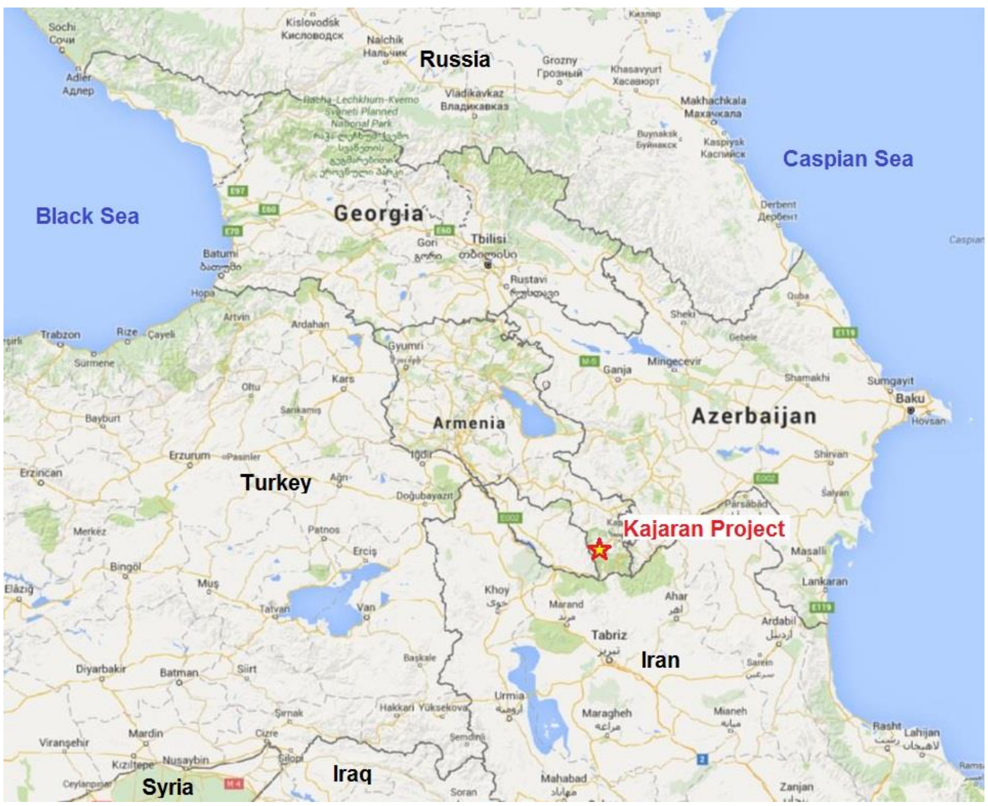

Figure 4-1: Kajaran Property Location (large scale)

The country's modern mining industry was established during the period when the country was part of the Soviet Union, with major mining and refining complexes being established at Agarak, Kajaran, Kapan, Zod and Alaverdi.

The ZCMC Kajaran Project is located in the Voghji valley in the southeast corner of Armenia, close to the borders with Iran and the Azerbaijan-Nakhichevan exclave. Kajaran is situated 330 km from Yerevan and approximately 25 km from Kapan, the administrative center of Syunik province. Syunik is the second largest province in Armenia and occupies 4,506 square km. (15.15% of the territory of Armenia) and has a population of about 153 thousand (2012 Armenian Statistics Service data).

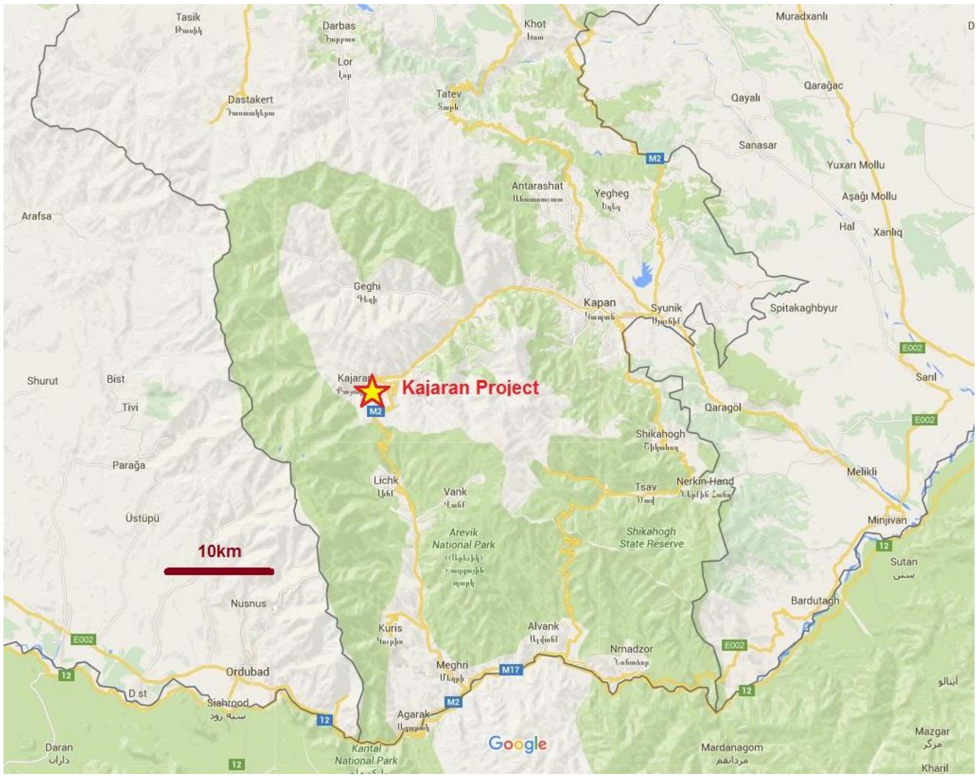

Figure 4-2: Kajaran Property Location (small scale)

4.2 Mineral Tenure

The Mining rights of the Company are approved by the Mining Permit, “Mining Contract and Mining Land Allotment Act”. The company is carrying out its mining activities based on the terms of the three approved core documents:

- The Mining Permit for Minerals Mining No ShATV-29/232, issued on November 27, 2012 for the period till May 30, 2030;
- The Mining Land Allotment Act No LV-232, issued on November 27, 2012 for the period till May 30, 2030; and,
- The Mining Contract No PV-232, signed on November 27, 2012 between the Company and RA Ministry of Energy and Natural Resources, with validity term till expiration date of the above Mining Permit.

The above mentioned documents were dated November 27, 2012, as they were issued (re-issued) based on new Law requiring that all the existing licenses in the mining sphere issued prior to 2012 shall be re-issued.

4.3 Royalties and Other Agreements

4.3.1 Nature Protection and Nature Usage Payments

According to the Republic of Armenia Law on Nature Protection and Nature Usage Payments the royalty and its rate are defined by the following articles:

4.3.1.1 Article 13.3., Calculation Basis of Royalty

1) The basis for the calculation of royalty is considered to be revenue received from the mining product delivered within the reporting period without VAT and profit before tax.

2) The mining product is considered to be metal concentrates (hereinafter concentrates).

3) If it is delivered not the concentrate but the alloying or any other final product received in the result of processing the concentrate or alloy, then the calculation size of the used concentrate is the basis for the calculation.

4) In accordance with this article the calculation order of revenue from the realization is defined by the Government of the Republic of Armenia.

4.3.1.2 Article 13.4., Rate of Royalty

Percentage rate is defined towards the basis of royalty calculation, the size of which is decided by the following formula:

Royalty = 4 + [P/(Rx8)] x100; where,

Royalty = rate of royalty in percentage;

P = profit before tax in Armenian drams. It is decided as the positive difference of royalty calculation basis and applicable deductions defined by RA law on Profit Tax (except for the costs on financial activity and tax damages of previous years); and,

R = Revenue from the delivering of the goods without VAT in Armenian drams.

4.4 Environmental

The environmental responsibilities, along with other obligations of the company are defined by the Legislation of the Republic of Armenia and the Mining Contract signed between ZCMC and RA Ministry of Energy And Natural Resources. Pursuant to the mentioned acts the main environmental responsibilities of the company during mining processes are the following:

1) to carry out the mining activities in accordance with the provisions of Mining Contract and Mine Design;
2) to perform the instructions of the relevant Authority in accordance with the requirements of RA Legislation;
3) to comply with the requirements of mining, ore transportation and enrichment standards approved in the Republic of Armenia;

4) to keep geological, surveying and other records during carrying out of all kinds of subsoil usage activities;
5) to keep daily records for extracted resources;
6) to present to the relevant Authority quarterly and annual reports on extracted resources;
7) to hand to the relevant Authority the needed geological information;
8) to gather, keep and provide to the relevant Authority information on compound, quality and quantity of explored and extracted mineral resources, as well as about those lost in the bowels;
9) to ensure the safety of the subsoil usage activities;
10) to ensure the protection of the bowels, air, lands, forests, water and other objects of the environment from the negative impact of mining activities;
11) to ensure the protection of the natural, historical and cultural monuments from the negative impact of mining activities;
12) to re-cultivate and develop the lands damaged during the mining activates, in accordance with the requirements of Mining Contract and Mine Design;
13) to ensure the implementation of Mine Closure project;
14) to ensure the payments for monitoring the areas of extracted resources, placement of tailings in result of mining activities, as well as the surrounding, with the aim to ensure the safety of adjacent communities.

At the same time, it is important to highlight, that the mining activities are subject to licensing in the Republic of Armenia. The process of obtaining Mining license (Mining Permit) requires the company to pass several procedures, including Environmental Impact Assessment (EIA) and Expertise, and Technical Safety Expertise. EIA and Expertise are mandatory activities conducted by the state; its main goal is to predict, prevent or reduce to the minimum the hazardous impact of an intended activity or procedure on human health, the environment, regular economic and social development. The environmental impact assessment and expertise are based upon the right of human beings to have favourable environment for health, life and creative activity, the requirement of efficient, complex and reasonable use of natural resources, the necessity of maintaining the equilibrium of ecological systems, preserving all species of flora and fauna, taking into account the interests of the current and future generations. The EIA and Expertise are based upon the principles of scientific justification, legitimacy, and transparency of decision making.

As a result of the processes of EIA and Expertise the relevant Authorities (RA Ministry of Energy and Natural Resources) provide a positive or negative decision. Based on a positive EIA and Expertise decision, the Mining Permit is provided. A positive Environmental Impact Assessment Expertise decision can also contain several environmental requirements, which are considered conditions of the positive EIA and Expertise decision, and are binding for the Company.

Based on requirements of the above mentioned acts and documents, ZCMC is carrying out its activities and performing the obligations appearing from them. Below are presented some examples of those activities.

The ZCMC Department of Environment is involved in stakeholder engagement with regard to the land acquisition process for the tailings pond expansion project.

Community consultation is accomplished by Martun Harutyunyan, ZCMC Head of Environmental Department, who consults with the mayors of Kajaran and Kapan, members of the local community, the Kajaran Municipal Environment Department, the Syunik region Environmental Department in Kapan, and local ENGOs. According to Mr.Harutyunyan the primary concerns of stakeholders are the following:

- Leak or break in the slurry pipeline and subsequent soil or water contamination;
- Spill from the tailings ponds;
- Dust from mining operations and trucks travelling along the highway; and,
- Water pollution.

ZCMC has undertaken measures over the past several years to mitigate any impacts. For example, a back-up slurry pipeline was built (if the slurry pipeline leaks or breaks the slurry will transfer automatically into the backup pipeline). Roads are frequently watered during the dry season to reduce dust and particulate matter in the air.

There is also a grievance mechanism in place for stakeholders to voice their issues and concerns. As previously stated, Armenia's culture involves verbal discussion. Similar to the mechanism used for both the community investments and the land acquisition process, stakeholders can discuss their issues with ZCMC personnel or the General Director. If the issue is considered serious, the stakeholder will file a written statement or complaint. In both cases, ZCMC tries to address their issues and solve the problem.

With regard to the environmental policy and spending, the ZCMC Kajaran operations are fully compliant with national legislation, and the Company does not have any outstanding litigation cases, legacy issues or historical claims.

Additionally, the following actions have been recently taken to minimize or eliminate the potential for environmental liability:

- Tailings and Waste Dam Site Assessment completed. A review of the stability of the Artsvanik facility has been completed. This document has been submitted to the Armenian Government and has been reviewed by Worley Parsons in their September International Finance Corporation (IFC) compliance review;
- Environment Policy has been completed and reviewed by WorleyParsons;
- The Environmental Management Plan (EMP) is to be updated to bring it into compliance with IFC standards;
- The Environmental Risk and Impact Register as well as the Waste Management Plan are under development;
- The EMP will include ground water use and quality, biodiversity, air and vibration. A new water sampling regime has been put in place; this includes bacteriological analysis of drinking water supplies for the entire Kajaran town;
- A new nursery facility has been established. A section of this will be dedicated to the growth of Red Book (Endangered) species native to the Syunik region; and,
- The Yerevan State University have been approached to assist in undertaking studies into the endangered Molluscs of the Syunik region.

4.5 Permits and Future Work

Current permits limit the mine's production to 12.5 million tonnes per year. To address the company's planned production increase, updated permitting limiting production to 22 million tonnes per year for 25 years is currently being processed.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Kajaran mine is located some 330 km southeast of the capital Yerevan. Access is by paved highway to the town of Kajaran where the processing plant is located. The open pit is 2 km from the town. Total travel time to the Kajaran Project from Yerevan is approximately five and half hours. No operating railway existed in the region. There is a small airport operating at Kapan, 30 km from the mine. Non-regular flights take place by helicopter. A flight from the capital to Kapan airport takes about an hour. Small jet flights are also possible from this airport.

5.2 Climate

The Armenian climate is relatively continental, with dry, hot summers and cold and short winters.

Three climatic zones are distinguished in the region of the Kajaran deposit. The deposit is located in a Middle Steppe-Meadow zone at elevations up to heights of 2,300 m above sea level. Annual average precipitation is 550-750 mm and the average annual humidity is 60-70%.

Above the Kajaran deposit, a high mountainous Alpestrine zone occurs at elevations between 2300 and 2500 m above sea level. This zone can be described with moderate cold climate. From 2,500 to 3,000 m above sea level a high Mountainous Alpine zone can be described as cold mountainous and humid. Precipitation (700-800 mm) exceeds evaporation (250-300 mm).

Within the period from 1961 to 2011 the average annual temperature in the Kajaran deposit area (in the valley of the Voghji River) was $+6.8^0$C. Average annual precipitation for the same given period was 605 mm, and the annual average relative humidity was 50-60%.

Wind regime including its direction and speed mainly depends on the relief. Mountain-valley winds are observed on clear days. Wind average speed makes 1.3-4.2 m/sec, while the maximum is 23 m/sec.

5.3 Physiography

The Kajaran Copper-Molybdenum deposit is situated in the South-Eastern part of Armenia, Syunik Marz, in the eastern offset of Zangezur range within the Voghji River water intake.

The highest point of the region is Kaputjugh peak (3,921 m) located in the headwaters of Voghji River, west of the deposit.

The topography of the Kajaran area is one of high relief with mountains up to nearly 4,000 m elevation in close proximity to the mine. The actual open pit operates at an elevation of between 1,920 to 2,360 m, while the processing plant in Kajaran is located between the old Kajaran site and the new town, at approximately 1,750 m above sea level.

The Voghji River flows from the Ranges and along the periphery of the active mine site through the center of the town. The minor Sakhkar-Su and Miakan-Su Rivers have been diverted beneath the open pit operation. Voghji River basin embraces Voghji and Tsav Rivers with its channel inflows. The Voghji River starts from sources and small lakes of the Kaputjugh mountain slopes in the Zangezur mountain range. The entire river length is 82 km (on the territory of RA: 52 km), and the surface area of the catchment basin is 2,337 km^2 (on the territory of RA: 1,240.5 km^2). The largest inflow to the Voghji River in Armenia is the Geghi River. The average flow rate of the Voghji river near Kajaran is 3.14 m^3/sec, but maximum and minimum flow rates of 11.2 and 0.36 m^3/sec have been measured.

The major portion of the territory (~ 80%) is covered by layers of soil and diluvium (1-5*m*). Up to elevations of 1,800 m above sea level soil cover is mainly represented by brown soils, and at elevations of 1,800-2,400 m above sea level by chestnut soils

This high relief environment is typified by the vegetation which up to an elevation of 2,000 m consists predominantly of forest in non-cultivated areas, and bushes and grasses in the upper regions above the tree line. The lower parts in the vicinity of populated areas are mostly cleared of trees.

5.4 Infrastructure

The Kajaran project is located in a well-developed region of Armenia, where the power supply is secured by the national electricity network. The Armenia-Iran high-voltage power line passes nearby, and a second line is planned for constructed. A concrete high quality “North-South” highway connecting Iranian border in the South and Georgian Border in the North of Armenia is currently under construction, starting from Yerevan towards both Southern and Northern borders. The highway will reach Kajaran in a few years and will have an 8 km long tunnel close to the mine.

5.4.1 Transportation/Roads

There is no rail connection to the project area and all goods entering the mine and concentrates leaving the mine are transported by road. The M2 highway passes close to the mine.

5.4.2 Power supply

Electrical power to the Kajaran area is supplied by two 110 kV high voltage lines (“Zangezur” and “David-Bek”). These two lines feed ZCMC's two substations (N1 110/35/6 kV and N2 110/10 kV). Both ZCMC substations were built in 2010 by AREVA and are fully equipped with modern equipment.

5.4.3 Water supply

The process water supply is currently obtained from two rivers, the Voghji River and the Geghi River.

A weir with an abstraction capacity of approximately 360 m^3/h is installed in the Voghji River, upstream of the processing plant. Water from the Voghji River flows without pumping (drift) and is stored in a reservoir located higher up in the valley above the mill.

Abstraction from the Voghji River is year round with additional supply coming from the Geghi River during the dry season (September to March).

Water from the Geghi River is pumped in two stages by electric pumps:

- From N1 Pump station to N2 Pump station and N3 Pump station; and,
- From N2 and N3 pump stations to the processing plant.

Pump stations details are:

- The N1 pump station has water reservoirs, 7 pumps 14M8x4-1 type (Q=600 m^3/hour. H=380, P= 1000 kW), out of which 4 pumps are from N3 Pump Station, and 3 from the N2 Pump station. The water from N1 Pump station is being pumped to N3 Pump Station and N2 Pump station by two water pipelines – 7 km in length and 720 mm and 612 mm in diameter;
- The N2 pump station is equipped with water reservoir and three 14M8x4-1 type water pumps. The water pumped from Station N1 is them pushed to the water reservoir of processing plant by a pipeline – 8 km on length 612 mm in diameter;
- N3 pump station is equipped with the water reservoir and four 14M8x4-1 type pumps. The water is being pumped to the processing plant by a pipeline – 8 km on length 720 mm in diameter; and,
- The water is provided to the processing plant by three drift lines.

5.4.4 Waste Dumps

There are three waste rock dumps at Kajaran Project. The “North-West” dump was the first established, and low grade oxide ore and barren rock (monzonites and granodiorite porphyry dykes) were disposed there between 1951 and 1970.

The Spitak Djur dump is located on the South-Eastern edge of the mine in the valley of Sakhkar River. Its operation started in 1971.

The Dzorateghn dump is located 1.5 km to the East from Spitak Djur dump. It has been in operation from 1979 to today.

A Drilling campaign was organized in 1980s to explore all the three waste rock dumps as potential resources for heap leaching. a report was prepared in 1991, with estimation results as followings:

- North-West Dump, 68.4 Mt @ 0.247% Cu;
- Spitak Djur Dump, 72.3 Mt @ 0.219% Cu;
- Dzorategh Dump, 61.0 Mt @ 0.193% Cu.

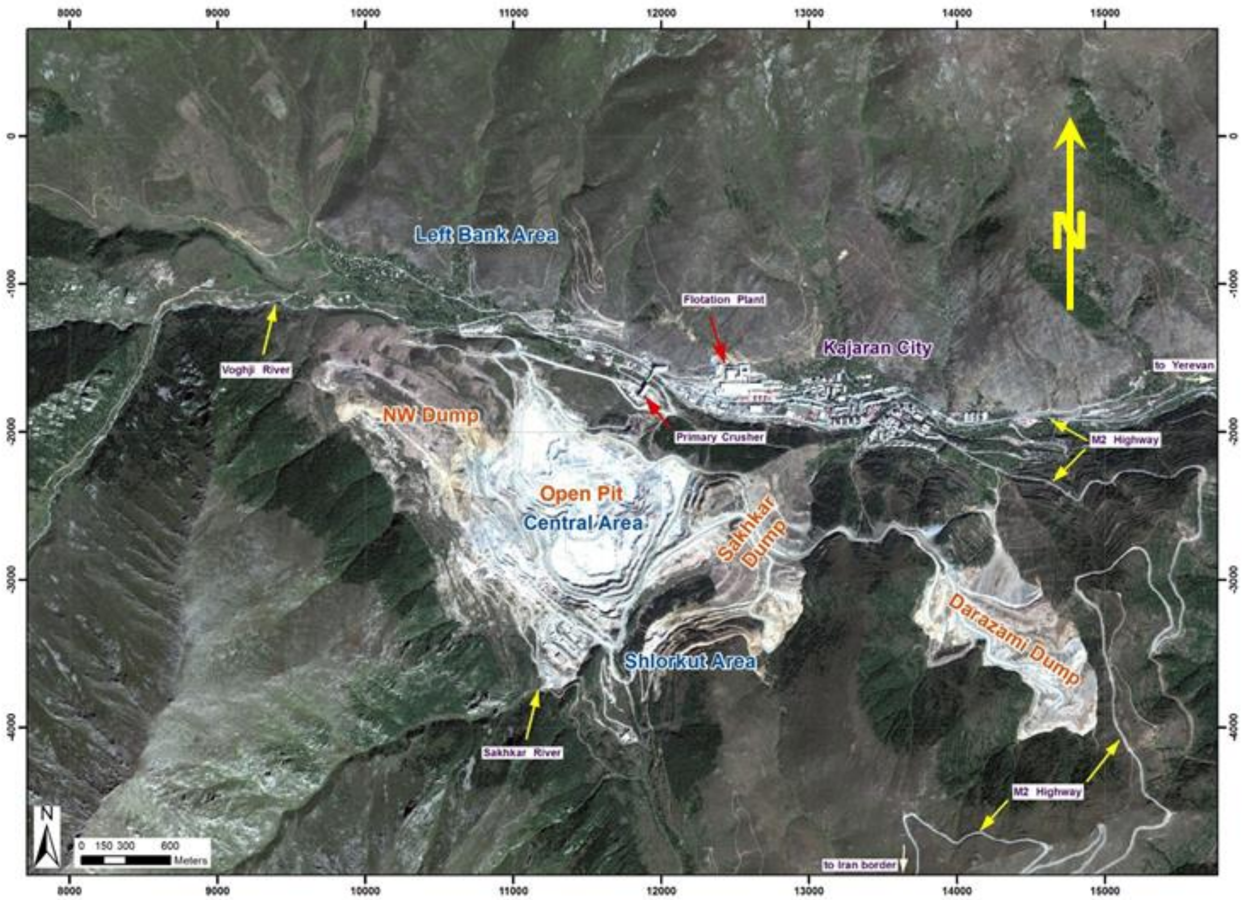

Figure 5-1: Kajaran Property Site Plan

5.4.5 Tailings Management

Kajaran tailings are classified as silty sand with about 60% of grain sizes between 2 mm to 0.05 mm. Since 1977 ZCMC has operated the Artsvanik tailing dam located about 30km E-NE from the mine. Tailing slurry is being transported by pipeline having 34.5km total length17.6km of which are tunnels and 5.2km are siphons.

At the present pond elevation of around 898 m there is still about 120 Mm^3 of space available to tailings to reach the design elevation of 964 m.

5.5 Local Resources

The existence of the Voghji River with its streams at the mine, and the Geghi River with its reservoir, ensures both industrial and drinking water for the project. Areas for the waste rock disposal and tailing are deemed to be large enough for the project. Kajaran is located in close proximity (30 km) to Kapan, the administrative center of Syunik region. Historically Kapan has a high level of education, with regional branches of a number universities and professional educational institutions. In addition, two other large mining companies are located in this region and traditionally people are comfortable with work in the mining industry.

6.0 HISTORY

6.1 Ownership and Exploration History

Operation of the Kajaran deposit started on 1951 and the first concentrate was produced in 1952. Until 2004 the Kajaran project was operated by state companies. By the Decree No 1677-Ն of RA Government, dated December 9, 2004, the shares of "Zangezur Copper Molybdenum Combine" were privatized. Pursuant to that Decree the Government decided to privatize 100 (one hundred) percent shares of the company to Cronimet Mining LLC (Germany), Pure Iron Plant OJSC, Armenian Molybdenum Production LLC, and Zangezur Mining LLC.

The first geological study in Kajaran region was carried out in the second half of the 19th century following the discovery in 1925. Preliminary exploration on the Kajaran deposit started on 1931 and was preliminarily completed after five major exploration periods. Each of the exploration phases was completed with a corresponding calculation and statement of reserves.

The first stage of preliminary exploration was completed between 1931 and 1937.

The second stage, from 1938 to 1944, consisted of detailed exploration in the central area of the deposit up to 100m depth, covering an area of 0.5 km^2. The surface of the deposit was investigated by trenches and pits while other exploration work was mainly in underground workings. As a result of this exploration work reserves were calculated and confirmed by State Committee of Reserves of USSR (Statement N3354 dated 11.05.46).

The third stage of exploration was carried out within the period from 1945 to 1951. In this stage the exploration was expanded to 1.0 km^2 on surface and to a depth of 200 m (up to the horizon of 2,075 m). Exploration on the 2,165 m horizon (adit #7 and #8) was completed, and exploration of the 2,075 m and 2,065 m horizons were continued by horizontal underground workings, whereas exploration of lower horizons of Kajaran Central Area was covered by drill holes. Upon completion of the third stage reserves were calculated and approved by the State Committee of Reserves of USSR (Statement N7267 dated 15.03.52).

The fourth stage of detailed exploration (1953-1959) was directed towards exploration of deeper horizons and flanks of the deposit, and the study of prospective areas on the left banks of the Voghji and Spitak Djur Rivers. This work was intended to support disclosure and outline of potential economic areas, and expanding the reserves of the deposit for increased capacity of the plant. During this period exploration on the 2,025 m horizon was completed (adits #36 and #38) and the exploration horizon of 1,875 m started (adit Kapitalnaya). Deep horizons were explored by core drilling. Completion of this stage extended the coverage of exploration to 3.5 km^2 area and depths of 300-400 m. Reserves were estimated as of July 1, 1961 and approved by the State Committee of Reserves of USSR (Statement N3747 dated 30.08.62).

Exploration on the deposit was suspended in 1959, but started again in 1967.

The fifth stage (1967-1985, advanced geological exploration) consisted solely of drill holes located on grids of 200x400m, 100x200m, 100x100m and in some places a grid of 50x100m. As a result, the area of exploration covered approximately 6.0 km^2 to depths of 350-750 m. Reserves were estimated as of April 1, 1985 and approved by the State Committee of Reserves of USSR (Statement N9975 dated 25.05.1986).

A few drill holes were completed on Kajaran deposit (within the Central and Shlorkut areas) between 2003 and 2004, but produced no material changes to the Kajaran resources.

The latest exploration phase was initiated after privatization of ZCMC. Drilling, which was mainly an infill drilling program in the Shlorkut area, started in 2005 and continued until 2012 along the edges of the Central area and in the Eastern part of Left Bank zone (Northern to Voghji River). In this phase a few twinning holes also were drilled for data verification. A total of 131 holes were drilled within this period.

In the autumn of 2015 the company drilled another 9 holes, 3 of which were for mainly hydrological purpose, and other 6 were drilled at the Northern edge of the Central Area of the deposit to infill gaps and check the historical data. This 2015 drill data is not included in the current resource estimation as assay results were not yet completed.

Table 6-1 summarizes known exploration programs on the Kajaran deposit.

Table 6-1: Kajaran Exploration Summary

Stages	1931 To 1937	1938 To 1944	1945 To 1951	1952 To 1961	1967 To 1984	2003 To 2004	2005 To 2012	2015	Total
Drifts (m)	253	8,053	3,027	5,913	256				17,502
Core drilling (m)		3,600	4,088	40,688	168,356	1,723	21,387	1,220	241,064
Trenches (m³)	3,439	7,408		3,952	8,121	19			22,920

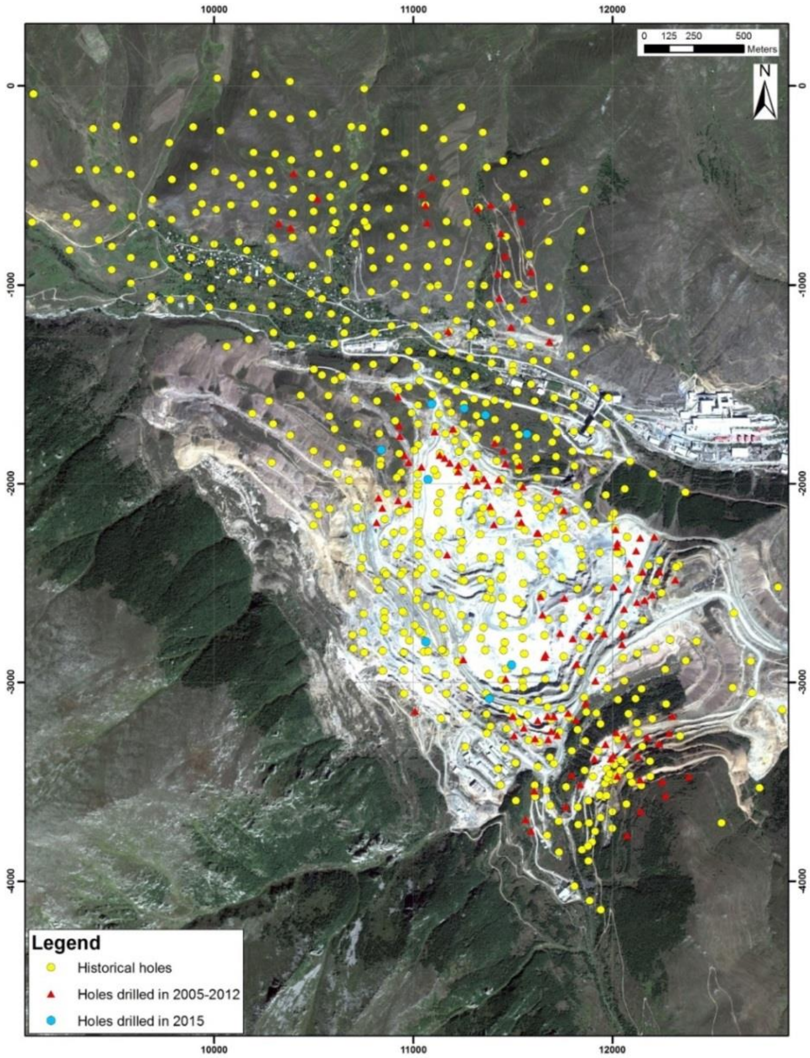

Figure 6-1: Kajaran Exploration Drill Collar Locations

6.2 Historical Resource Estimates

Resource estimations for the Kajaran deposit were done after each phase of exploration. Up to 2005 all official Resource Estimations were done by a polygonal method of horizontal or vertical sections block by block (100 x 100 x 50 m or 100 x 200 x 50 m), using weighted average grades. Each block was outlined by limiting underground working or drill holes and by horizons (usually each 50 m). Estimations and all calculations were done according the guidelines of State Committee of Reserves of USSR and State Agency of Reserves of Armenia.

The results of the last three historic resource estimates are presented below in Table 6-2 to Table 6-4.

Table 6-2: **Kajaran Mineral Resources as of July 1, 1961 (USSR State Committee of Reserves, Statement N3747 Aug 30, 1962)**

Sulphide

Balance	Category	Ore, Mt	Mo, %	Cu, %
Economic	B	83.5	0.09	0.47
	C1	624.0	0.06	0.33
	C2	317.3	0.06	0.28
	B+C1+C2	1,024.8	-	-
Non-economic	C1	216.0	0.02	0.16
	C2	216.0	-	-
Oxide				
Economic	C1	12.8	0.066	0.58
	C2	12.8	-	-
Non-economic	C1	132.2	0.037	0.44
	C2	132.2	-	-
Mixed				
Non-economic	C1	0.7	0.011	0.37
	C2	0.7	-	-

Table 6-3: Kajaran Mineral Resources Apr 1, 1985 (USSR State Committee of Reserves, May 25, 1986)

Sulphide

	Balance	Category	Ore, Mt	Mo, %	Cu, %	Mo eq., %
Within the designed pit	Economic	B	101,695	0.055	0.300	0.071
		C1	1,652,606	0.038	0.240	0.050
		C2	21,360	0.030	0.240	0.042
		B+C1+C2	1,775,661	0.039	0.250	0.051
	Non-economic	C1	159,731	0.017	0.200	0.027
Out of pit	Non-economic	C1	693,504	0.031	0.195	0.041
		C2	554,477	0.029	0.180	0.038

Table 6-4: Kajaran Mineral Resources July 1, 2005 (GeoEconomica)

Sulphide

Global Resources

	Category	Ore, Mt	Mo %	Cu %	MoEq %
Economic	B	259,416	0.037	0.28	0.065
	C1	3,628,154	0.029	0.22	0.051
	C2	665,264	0.025	0.20	0.045
	B+C1+C2	4,552,833	0.029	0.22	0.051

Sulphide					
Non-economic	C1	609,887	0.009	0.11	0.011
	C2	152,364	0.013	0.06	0.013
Within the Economic Pit					
Economic	B	259,416	0.037	0.28	0.065
	C1	1,984,618	0.032	0.23	0.055
	C2	-	-	-	-
	B+C1+C2	2,244,034	0.033	0.23	0.056
Non-economic	C1	190,827	0.009	0.11	0.011
	C2	-	-		

6.3 Previous Production

The Kajaran deposit operations were started in 1951 by Kajaran Copper-Molybdenum Combine (State Company) of the Ministry of Non-Ferrous Metallurgy (USSR). From 1951 to 1954 ore was produced only by underground, and from 1955 to 1958 production was a combination of both underground and open pit. Starting in 1959 production was from only open pit mining.

Kajaran mine production capacity of ore mining and processing increased continuously from the beginning. In 1966 the first planned production capacity (5.1 Mt per year) was reached. Other than a cutback in production during the collapse of USSR (1992-1993), production increased gradually to 9.1 Mt per year in 2005. From 2005 onward production increased further to a total of 18 Mt in 2015.

The mining and processing summary for the 64 years of production (1951-2014) are presented in Table 6-5 and Table 6-6 respectively.

Table 6-5: Kajaran Mining Summary, 1951-2014

Year	Mined Reserves			Tonnes Losses (000)	Tonnes Dilution (000)	Extracted ore			Waste, m^3	Stripping Ratio, m^3/m^3
	Ore, Kt	Mo, %	Cu, %			Tonnes (000)	Mo %	Cu %		
1951	20	0.306	1.00	6	0	14	0.302	1.01	0	0.0
1952	291	0.081	0.86	123	38	206	0.067	0.89	5	0.1
1953	373	0.075	0.72	46	28	355	0.069	0.74	262	1.9
1954	432	0.088	0.91	1	115	547	0.070	0.58	881	4.1
1955	753	0.085	0.72	176	138	715	0.069	0.67	911	3.2
1956	721	0.084	0.65	69	133	786	0.070	0.56	514	1.7
1957	985	0.083	0.66	174	135	946	0.072	0.58	1569	4.2
1958	1264	0.078	0.59	45	99	1318	0.072	0.56	1816	3.5
1959	1518	0.086	0.54	49	214	1683	0.075	0.46	1209	1.8
1960	1880	0.084	0.50	94	164	1950	0.077	0.46	2345	3.1
1961	2444	0.080	0.49	65	124	2503	0.076	0.42	2687	2.7

Year	Mined Reserves			Tonnes Losses (000)	Tonnes Dilution (000)	Extracted ore			Waste, m^3	Stripping Ratio, m^3/m^3
	Ore, Kt	Mo, %	Cu, %			Tonnes (000)	Mo %	Cu %		
1962	2694	0.077	0.58	171	169	2693	0.072	0.50	3314	3.1
1963	3022	0.069	0.59	336	295	2982	0.062	0.51	3565	3.0
1964	3935	0.070	0.63	334	370	3971	0.064	0.49	4540	2.9
1965	4703	0.069	0.54	278	399	4823	0.063	0.45	4001	2.1
1966	5129	0.068	0.49	239	417	5307	0.062	0.45	4073	2.0
1967	5213	0.069	0.46	160	453	5507	0.063	0.41	3914	1.8
1968	5430	0.061	0.43	240	366	5557	0.057	0.34	3368	1.5
1969	5651	0.060	0.48	229	290	5712	0.057	0.33	2540	1.1
1970	5738	0.060	0.47	224	285	5798	0.057	0.34	4032	1.8
1971	5832	0.060	0.40	219	278	5891	0.057	0.32	4037	1.7
1972	5764	0.062	0.39	251	315	5829	0.059	0.32	3632	1.6
1973	5935	0.060	0.37	211	277	6001	0.057	0.32	3564	1.5
1974	6073	0.060	0.37	193	249	6129	0.057	0.32	3711	1.5
1975	6395	0.059	0.35	223	244	6416	0.057	0.31	3604	1.4
1976	6512	0.059	0.36	214	266	6563	0.057	0.31	3563	1.4
1977	7122	0.057	0.33	255	302	7169	0.055	0.32	3571	1.3
1978	7481	0.056	0.33	325	364	7520	0.053	0.32	3488	1.2
1979	7833	0.055	0.34	310	385	7908	0.052	0.32	3101	1.0
1980	7968	0.054	0.35	289	373	8052	0.052	0.31	3078	1.0
1981	8025	0.055	0.31	298	378	8105	0.052	0.29	3171	1.0
1982	7876	0.056	0.30	245	592	8224	0.052	0.27	3142	1.0
1983	7984	0.057	0.30	255	587	8316	0.053	0.27	3166	1.0
1984	8082	0.057	0.28	254	602	8430	0.053	0.26	3087	0.9
1985	8188	0.056	0.29	247	584	8525	0.053	0.26	3136	0.9
1986	8175	0.057	0.27	246	615	8544	0.053	0.25	3162	0.9
1987	8147	0.058	0.26	247	653	8553	0.053	0.24	3068	0.9
1988	8357	0.057	0.25	255	629	8731	0.053	0.23	3009	0.9
1989	8701	0.056	0.23	291	672	9083	0.052	0.21	2835	0.8
1990	7525	0.059	0.23	248	633	7910	0.054	0.21	2009	0.6
1991	5219	0.059	0.22	178	437	5478	0.054	0.20	1484	0.7
1992	1082	0.068	0.21	28	77	1132	0.064	0.20	0	0.0
1993	508	0.067	0.20	6	13	514	0.066	0.18	0	0.0
1994	1451	0.067	0.23	46	104	1508	0.062	0.21	28	0.0
1995	2538	0.062	0.22	81	160	2617	0.058	0.20	989	1.0
1996	3199	0.064	0.23	84	212	3327	0.060	0.21	403	0.3
1997	3570	0.062	0.27	102	202	3670	0.059	0.22	209	0.1
1998	4965	0.060	0.23	137	383	5211	0.056	0.21	698	0.3

Year	Mined Reserves			Tonnes Losses (000)	Tonnes Dilution (000)	Extracted ore			Waste, m^3	Stripping Ratio, m^3/m^3
	Ore, Kt	Mo, %	Cu, %			Tonnes (000)	Mo %	Cu %		
1999	5934	0.061	0.23	283	426	6077	0.057	0.20	781	0.3
2000	6833	0.062	0.20	256	484	7062	0.058	0.19	1104	0.4
2001	7645	0.056	0.23	268	375	7753	0.054	0.21	1050	0.3
2002	7816	0.058	0.22	300	456	7971	0.055	0.21	954	0.3
2003	7883	0.056	0.22	345	547	8086	0.052	0.20	1401	0.4
2004	7912	0.056	0.20	297	601	8216	0.052	0.19	2309	0.7
2005	8070	0.053	0.19	180	618	8509	0.049	0.17	2780	0.8
2006	9088	0.051	0.17	209	761	9640	0.047	0.16	2724	0.7
2007	9631	0.050	0.16	196	703	10138	0.046	0.15	3460	0.9
2008	10829	0.048	0.15	241	858	11446	0.044	0.14	3500	0.8
2009	12462	0.042	0.19	224	907	13145	0.039	0.18	3403	0.7
2010	13292	0.038	0.21	261	969	14000	0.036	0.20	4346	0.8
2011	14000	0.038	0.21	194	831	14638	0.036	0.20	4820	0.8
2012	16672	0.036	0.23	189	690	17173	0.034	0.22	3926	0.6
2013	17944	0.036	0.24	234	719	18430	0.034	0.23	4506	0.6
2014	18531	0.035	0.24	238	674	18966	0.034	0.23	4551	0.6
TOTAL	**397244**	**0.054**	**0.29**	**11128**	**19790**	**409976**	**0.050**	**0.26**	**160102**	**1.4**

Table 6-6: Kajaran Processing Summary, 1951-2014

Year	ORE PROCESSED					CONCENTRATE PRODUCED						TAILINGS		
	Tonnes (000)	Grade %		Metal (tonnes)		Tonnes (000)		Grade %		Metal (tonnes)		Quantity	Contents, %	
		Mo	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Kt	Mo	Cu
1951														
1952	215	0.067	0.89	143	1904	198	4693	48.6	19.0	96	891	209	0.017	0.28
1953	352	0.069	0.74	243	2593	351	8325	48.1	17.0	169	1411	343	0.017	0.27
1954	549	0.070	0.72	382	3943	471	12000	47.9	16.7	225	2001	536	0.018	0.32
1955	714	0.069	0.67	493	4782	758	18967	49.0	14.7	371	2797	693	0.015	0.24
1956	785	0.070	0.60	551	4684	841	17160	49.4	14.3	415	2450	766	0.014	0.25
1957	947	0.072	0.60	678	5709	1072	23021	48.8	15.6	523	3590	922	0.014	0.20
1958	1313	0.075	0.54	979	7109	1558	29428	48.8	15.7	760	4619	1282	0.014	0.18
1959	1678	0.075	0.46	1255	7761	2000	33577	48.8	15.2	975	5118	1643	0.015	0.15
1960	1943	0.076	0.46	1484	8933	2391	41983	48.8	14.1	1167	5920	1899	0.014	0.14
1961	2507	0.076	0.42	1911	10591	3074	52102	48.9	13.8	1503	7198	2452	0.014	0.13
1962	2684	0.072	0.50	1942	13359	3099	63158	49.0	15.0	1517	9479	2619	0.013	0.15

Year	ORE PROCESSED					CONCENTRATE PRODUCED						TAILINGS		
	Tonnes (000)	Grade %		Metal (tonnes)		Tonnes (000)		Grade %		Metal (tonnes)		Quantity	Contents, %	
		Mo	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Kt	Mo	Cu
1963	2984	0.062	0.51	1851	15130	2996	71841	49.2	15.4	1473	11041	2911	0.010	0.14
1964	3960	0.063	0.49	2512	19361	4062	86629	48.9	16.0	1984	13843	3870	0.011	0.14
1965	4829	0.063	0.45	3038	21906	4852	114770	48.3	13.1	2341	15032	4711	0.012	0.14
1966	5302	0.062	0.45	3309	23636	5389	113388	49.0	14.9	2639	16870	5183	0.011	0.13
1967	5509	0.063	0.41	3458	22719	5577	113683	49.1	14.2	2738	16140	5390	0.011	0.12
1968	5557	0.056	0.34	3138	19005	5183	110280	48.6	12.2	2518	13468	5441	0.010	0.10
1969	5712	0.057	0.33	3246	19134	5351	92717	48.7	14.9	2606	13783	5614	0.010	0.09
1970	5801	0.057	0.34	3324	19558	5442	94414	48.9	15.2	2661	14326	5700	0.011	0.09
1971	5889	0.057	0.32	3353	18862	5599	89203	48.5	15.8	2717	14059	5793	0.010	0.08
1972	5835	0.059	0.32	3422	18685	5671	89965	49.0	15.3	2776	13792	5738	0.010	0.08
1973	5992	0.057	0.32	3442	19034	5678	87245	49.4	16.4	2804	14329	5899	0.010	0.08
1974	6125	0.058	0.32	3521	19407	5733	87539	50.2	16.9	2879	14778	6030	0.010	0.07
1975	6411	0.057	0.31	3623	19896	5879	88828	50.5	17.1	2970	15176	6315	0.010	0.07
1976	6561	0.057	0.31	3716	20149	6068	92347	50.2	16.7	3049	15436	6461	0.010	0.07
1977	7166	0.055	0.32	3922	22850	6399	98224	50.4	18.0	3225	17664	7061	0.009	0.07
1978	7517	0.053	0.32	4007	23739	6524	99194	50.6	18.6	3301	18451	7410	0.009	0.07
1979	7907	0.052	0.32	4128	24994	6684	101682	50.8	19.2	3398	19476	7797	0.009	0.07
1980	8046	0.052	0.31	4177	25268	6790	102857	51.1	19.1	3466	19684	7936	0.008	0.07
1981	8109	0.052	0.29	4244	23461	6944	97756	50.5	18.6	3507	18159	8004	0.009	0.07
1982	8219	0.052	0.27	4282	22488	6973	95253	50.7	18.3	3537	17408	8116	0.009	0.06
1983	8317	0.053	0.27	4368	22518	7181	93603	50.3	18.6	3613	17430	8216	0.009	0.06
1984	8434	0.053	0.26	4474	21901	7344	93786	50.4	18.1	3700	16951	8332	0.009	0.06
1985	8528	0.053	0.26	4481	22258	7346	94976	50.5	18.1	3710	17229	8425	0.009	0.06
1986	8552	0.053	0.25	4504	21342	7351	91313	50.7	18.0	3730	16451	8453	0.009	0.06
1987	8554	0.053	0.24	4552	20811	7405	91956	51.0	17.4	3774	15964	8455	0.009	0.06
1988	8741	0.053	0.23	4663	20525	7567	87602	51.2	17.8	3871	15552	8646	0.009	0.06
1989	9076	0.052	0.21	4726	19170	7581	80731	51.8	17.7	3923	14282	8988	0.008	0.05
1990	7882	0.054	0.21	4277	16554	6922	69057	51.3	17.6	3552	12165	7806	0.009	0.06
1991	5482	0.054	0.20	2969	10943	4853	44992	50.8	16.7	2464	7518	5432	0.009	0.06
1992	1139	0.063	0.20	723	2242	1173	9380	49.7	15.2	583	1421	1129	0.012	0.07
1993	505	0.066	0.18	331	911	563	3144	48.1	16.8	271	528	501	0.016	0.07
1994	1504	0.062	0.21	935	3164	1445	11399	49.0	17.3	707	1970	1491	0.014	0.08
1995	2623	0.058	0.20	1533	5323	2376	18628	50.7	18.7	1204	3483	2601	0.012	0.07
1996	3366	0.060	0.21	2032	6922	3171	20860	50.1	20.9	1589	4350	3342	0.013	0.08

Year	ORE PROCESSED					CONCENTRATE PRODUCED						TAILINGS		
	Tonnes (000)	Grade %		Metal (tonnes)		Tonnes (000)		Grade %		Metal (tonnes)		Quantity	Contents, %	
		Mo	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Mo	Cu	Kt	Mo	Cu
1997	3638	0.059	0.22	2130	7939	3332	24090	49.9	21.1	1663	5087	3610	0.012	0.08
1998	5215	0.056	0.21	2902	10777	4587	31908	50.1	21.7	2299	6938	5179	0.011	0.07
1999	6073	0.057	0.20	3433	12185	5508	34065	50.4	23.4	2774	7981	6033	0.010	0.07
2000	7062	0.058	0.19	4105	13339	6665	32240	50.4	28.0	3359	9032	7023	0.010	0.06
2001	7773	0.054	0.21	4163	16579	6757	40556	50.3	28.2	3397	11430	7725	0.009	0.07
2002	8004	0.055	0.21	4394	16864	7187	41913	50.1	27.7	3598	11616	7954	0.010	0.07
2003	8060	0.053	0.20	4231	16316	6789	41769	51.0	27.4	3462	11450	8011	0.009	0.06
2004	8203	0.052	0.19	4254	15232	6986	38976	50.0	27.6	3492	10773	8157	0.009	0.05
2005	8501	0.049	0.17	4203	14828	6890	38126	50.0	27.6	3443	10517	8456	0.009	0.05
2006	9642	0.047	0.16	4539	15433	7512	40324	49.6	27.7	3726	11157	9594	0.008	0.04
2007	10139	0.046	0.15	4710	15077	7839	39719	49.5	27.5	3879	10934	10091	0.008	0.04
2008	11443	0.044	0.14	5034	16072	8039	40735	50.9	27.8	4096	11335	11394	0.008	0.04
2009	13157	0.039	0.18	5099	23313	8587	62422	49.2	27.3	4226	17064	13086	0.007	0.05
2010	14000	0.036	0.20	4980	27588	8440	79069	49.0	26.6	4134	21045	13913	0.006	0.05
2011	14637	0.036	0.20	5228	29411	8809	83950	49.1	26.4	4322	22187	14544	0.006	0.05
2012	17172	0.034	0.22	5891	38110	9932	110219	49.3	26.4	4893	29069	17051	0.006	0.05
2013	18379	0.034	0.23	6328	41897	10699	131731	49.5	25.1	5299	33096	18144	0.005	0.05
2014	18712	0.034	0.23	6310	43121	10768	137180	49.0	24.7	5276	33944	18565	0.005	0.05
TOTAL	**409624**	**0.050**	**0.26**	**206303**	**1059342**	**337242**	**4092676**	**49.9**	**19.0**	**168369**	**778368**	**405090**	**0.009**	**0.07**

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Kajaran Cu-Mo porphyry deposit is located in Southern Armenia, part of the Lesser Caucasus, and is hosted by the Tertiary Meghri (Meghri-Ordubad) pluton, lying within the Tethyan metallogenic belt (Figure 7-1).

The Tethys orogenic belt was formed during convergence of the African-Arabian and Eurasian plates. The Lesser Caucasus is a key area to understand the lateral connection of the Iranian belts with the Western and Central Tethys orogenic and metallogenic belt, including the Balkans, Rhodopes and Taurides-Anatolides.

The Zangezur-Ordubad region of the southernmost Lesser Caucasus, along the Armenian and Nakhichevan borders with Iran, is a unique location within the Tethys orogenic belt, where magmatism remained stationary from an Early Tertiary subduction setting to a Late Tertiary post-collision environment, in a place where the Gondwana-derived South Armenian block collided with the Eurasian margin.

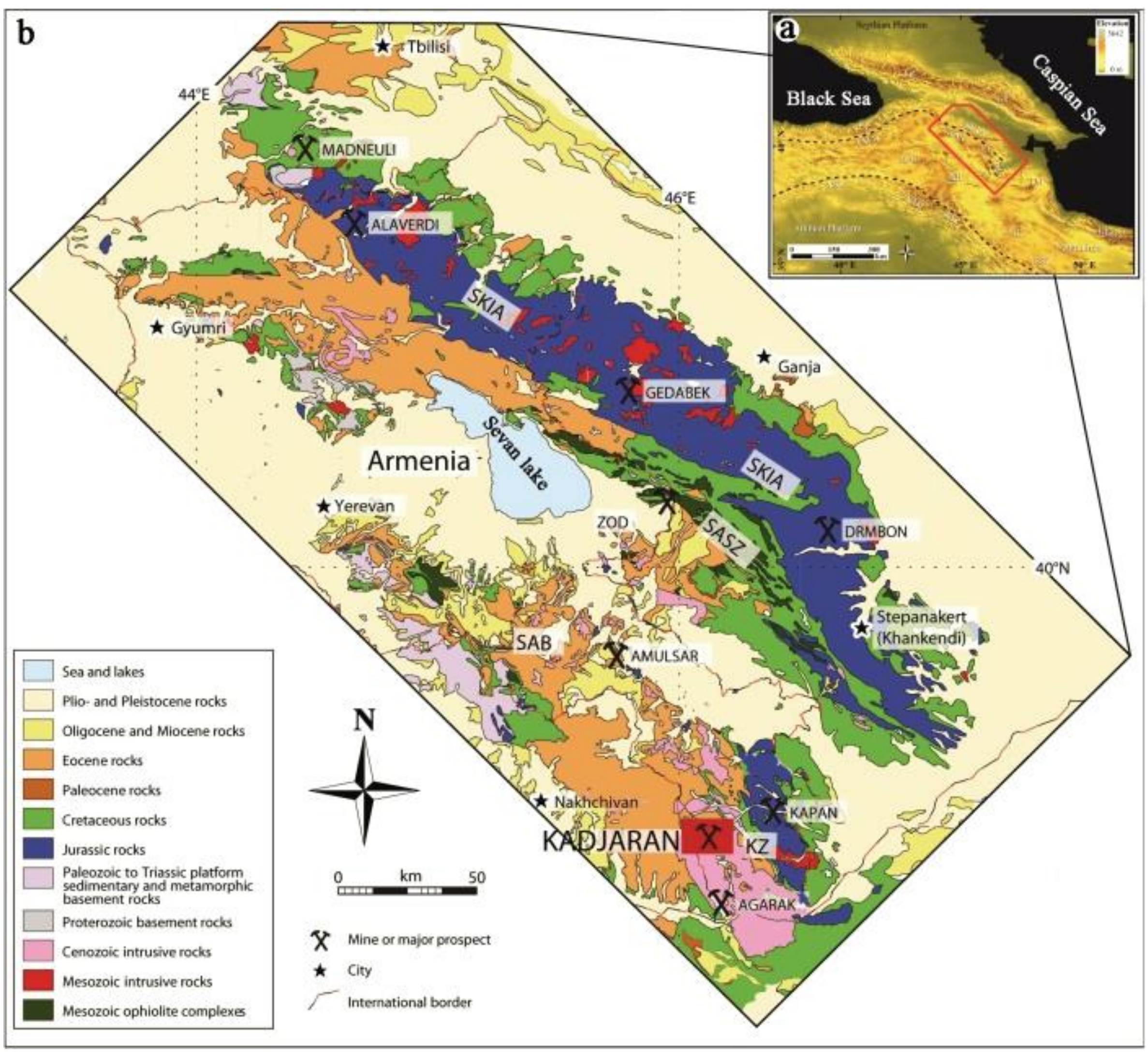

Figure 7-1: Regional Geology of the Kajaran Porphyry Copper Deposit

The long-lived, Eocene to Pliocene pulsed magmatic system generated the composite Meghri-Ordubad and Bargushat plutons at the contact of both tectonic zones (Figure 7-2).

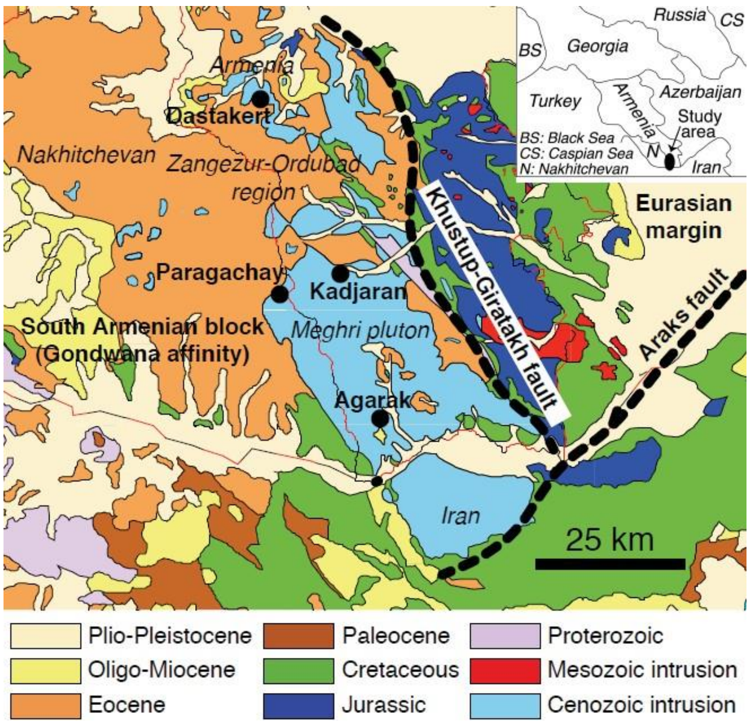

Figure 7-2: Geology of Southern Lesser Caucasus and Ore Deposits

The Zangezur-Ordubad region, sitting astride the territories of southern Armenia and Nakhichevan, is bounded to the east by the northwest-oriented Khustup-Giratakh fault along the Kapan block belonging to the Eurasian margin (Figure 7-2). Basement rocks include Devonian, Permian, Jurassic and Lower Cretaceous sedimentary rocks, followed by Upper Cretaceous and Paleocene terrigenous sedimentary rocks. A sequence of terrigenous flysch of up to 2.5 km thick was deposited during the Lower Eocene, and is overlain by Middle Eocene terrigenous sedimentary and pyroclastic rocks. The end of the Middle Eocene is characterized by voluminous calc-alkaline to subalkaline volcanic activity, including basalt, andesite, and trachyandesite, accompanied by sub-volcanic bodies. Upper Eocene to Oligocene olivine basalt and andesite are overlain by Oligo-Miocene molasse-type rocks, and Mio-Pliocene sandy carbonate, volcanic and terrigenous rocks (Djrbashyan et al. 1976; Tayan et al. 1976).

The composite Meghri-Ordubad and Bargushat plutons intrude pre-Middle Eocene rocks, and outcrop predominantly in the uplifted Zangezur block. Karamyan et al. (1974), Tayan et al. (1976) and Babazadeh et al. (1990) recognized successive Eocene to Miocene magmatic pulses of olivine gabbro, gabbro-monzonite, syenite, syenogranite, monzonite, quartz-diorite, monzodiorite, granodiorite, and porphyritic granite.

New high-precision TIMS U-Pb zircon ages confirm the magmatic sequence recognized by previous Rb-Sr isochron and whole-rock K-Ar dating (Melkonyan et al. 2008, 2010). A 44.03 ± 0.02 Ma-old granite and 48.99 ± 0.07 Ma-old granodiorite from the southern part of the composite plutons belong to the initial Eocene magmatic pulse. The Oligocene magmatic pulse was dated by U-Pb zircon at 31.82 ± 0.02 Ma and 33.49

± 0.02 Ma for monzonite and gabbro, and the Miocene by porphyritic granodioritic dyke and granite yielding ages of 22.46 ± 0.02 Ma and 22.22 ± 0.01 Ma, respectively.

The Zangezur-Ordubad region hosts major porphyry Cu-Mo deposits and prospects (Karamyan 1978; Babazadeh et al. 1990), including: the producing Kajaran and Agarak deposits; the past-producing Paragachay deposit; a range of smaller Cu+/-Mo porphyry deposits and prospects (Aygedzor, Lichk, Dastakert); and, Epithermal gold-polymetallic vein style deposits and prospects (Lichkvaz-Tey, Terterasar, Marjan). Re-Os dating of molybdenite from porphyry Cu-Mo deposits and occurrences, reported in Moritz et al. (2013), yielded Eocene ages for Agarak (44.2 ± 0.2 Ma), Hankasar (43.14 ± 0.17 Ma), Aygedzor (42.62 ± 0.17 Ma) and Dastakert (40.22 ± 0.16 to 39.97 ± 0.16 Ma), therefore they are coeval with the initial Eocene magmatic pulse in the Zangezur-Ordubad region. Late Oligocene ages were obtained for the Kajaran (27.2 ±0.1to 26.43 ± 0.11 Ma) and Paragachay deposits (26.78± 0.11 Ma), thus broadly coinciding with the Oligo-Miocene magmatic pulse. Precious and base metal epithermal occurrences hosted by volcanic and plutonic rocks are spatially associated with some of the porphyry deposits, but are of lesser economic interest (e.g. Babazadeh et al. 1990).

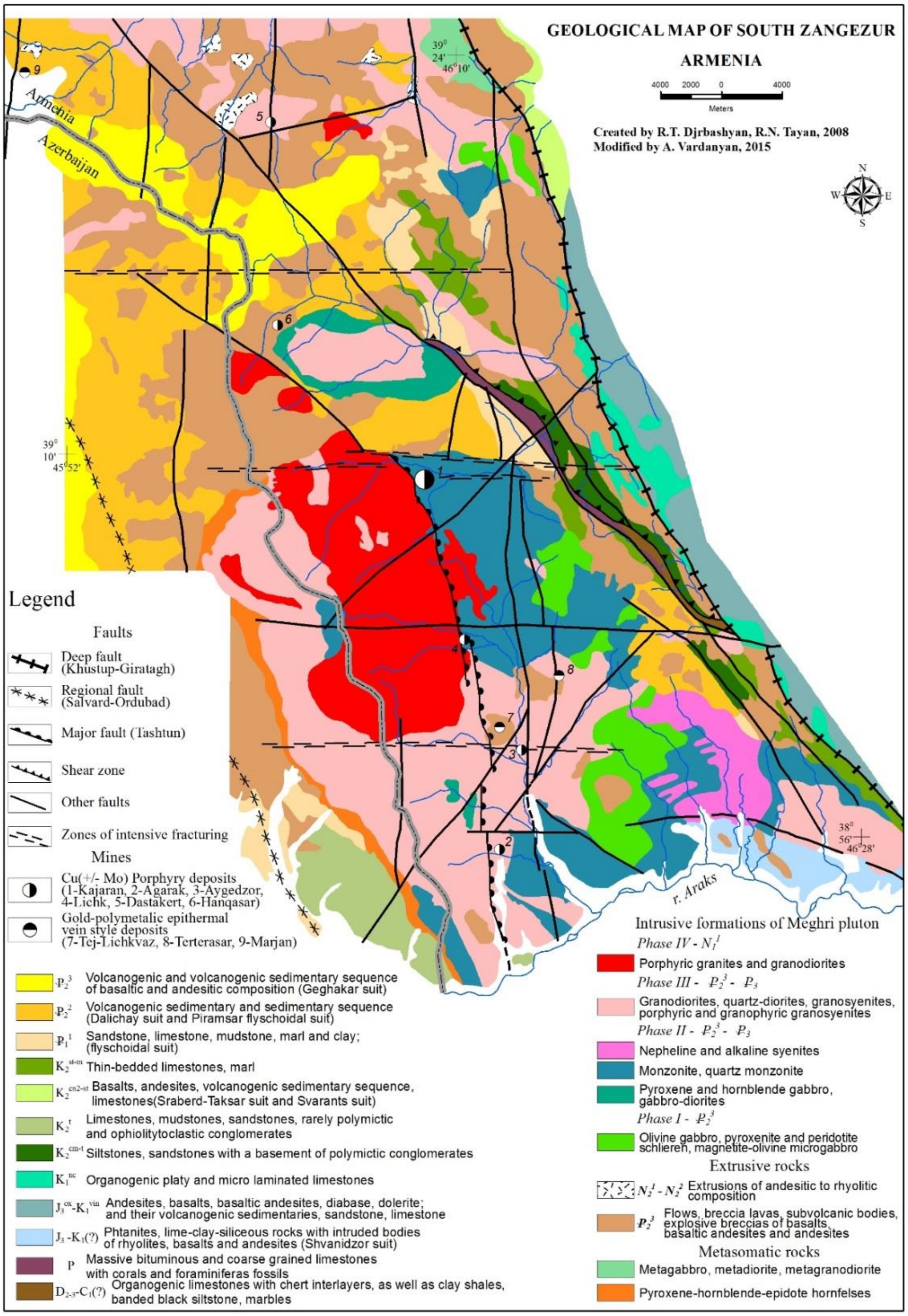

Figure 7-3: Geological Map of South Zangezur

7.2 Property Geology

The Kajaran deposit is hosted by a monzonite intrusion (31.83 ±0.02Ma; Moritz et al. 2013) belonging to the Meghri composite pluton. After emplacement of the monzonite intrusion, the tectonic evolution of the Kajaran area was characterized by a specific succession of fault and fracture development, which controlled further emplacement of magmatic rocks and mineralization.

Early activation of east-west oriented fractures along the northern exocontact of the monzonite intrusion, and roughly north-south oriented structures in the eastern part, resulted in the emplacement of elongated zones of migmatites, which represent large apophyses of the monzonite. Further magmatic activity along the endocontact of the monzonite intrusion, resulted in the emplacement of isolated dykes of microsyenite and aplite (Tayan 1984).

The earliest dyke stage consists of porphyritic diorite dykes, located only in the eastern part of the Kajaran property, within roughly north-south and north-east oriented structures. The same structures also controlled the emplacement of early spessartite dykes, and later abundant medium-grained granodioritic porphyry dykes.

The regional northwest-oriented Tashtun fault zone was active during the Lower Miocene, and was accompanied by the formation of a complex of 22 Ma-old porphyry granites. The Tashtun fault traces the contact between the 31.83 Ma-old host monzonite and 22 Ma-old porphyritic granites, and constitutes the western limit of the Kajaran deposit.

The magmatic evolution at the Kajaran property and deposit ends with the formation of thick, coarse-grained granodiorite-porphyry dykes (Figure 7-4). These dykes are interpreted to be genetically related to the 22 Ma-old porphyry granitoids (Harutyunyan et al. 2002; Moritz et al 2013).

The above mentioned different structural controls of the different magmatic stages provide a basis to consider the Kajaran ore field as an area of increased long-lived magmatic and tectonic activity which experienced a complex and multi-stage structural-geological evolution.

The Kajaran deposit is structurally controlled by a steeply dipping (65-85^{o}) orthogonal system of east-west, north-south and northeast-oriented (45-65^{o}) fractures.

The major ore-bearing faults of the deposit host a stock-work mineralization, with veins ranging in thickness from a few mm up to 5 cm, attaining a length up to a few tens of meters. They are predominantly shallow dipping (25-40°, rarely up to 60°), forming parallel to sub-parallel oriented systems of veins reaching thicknesses of a few tens of meters.

The dykes have played an important structural control on the location of mineralization, which is located within the hanging wall, on the eastern side of the north-west- oriented (320-340°) Tashtun fault. Copper-molybdenum stock-works are recognized along a more than 3.5 km long and 2 km-wide corridor. Along the footwall of the fault, the porphyry granite is devoid of mineralization and pronounced alteration.

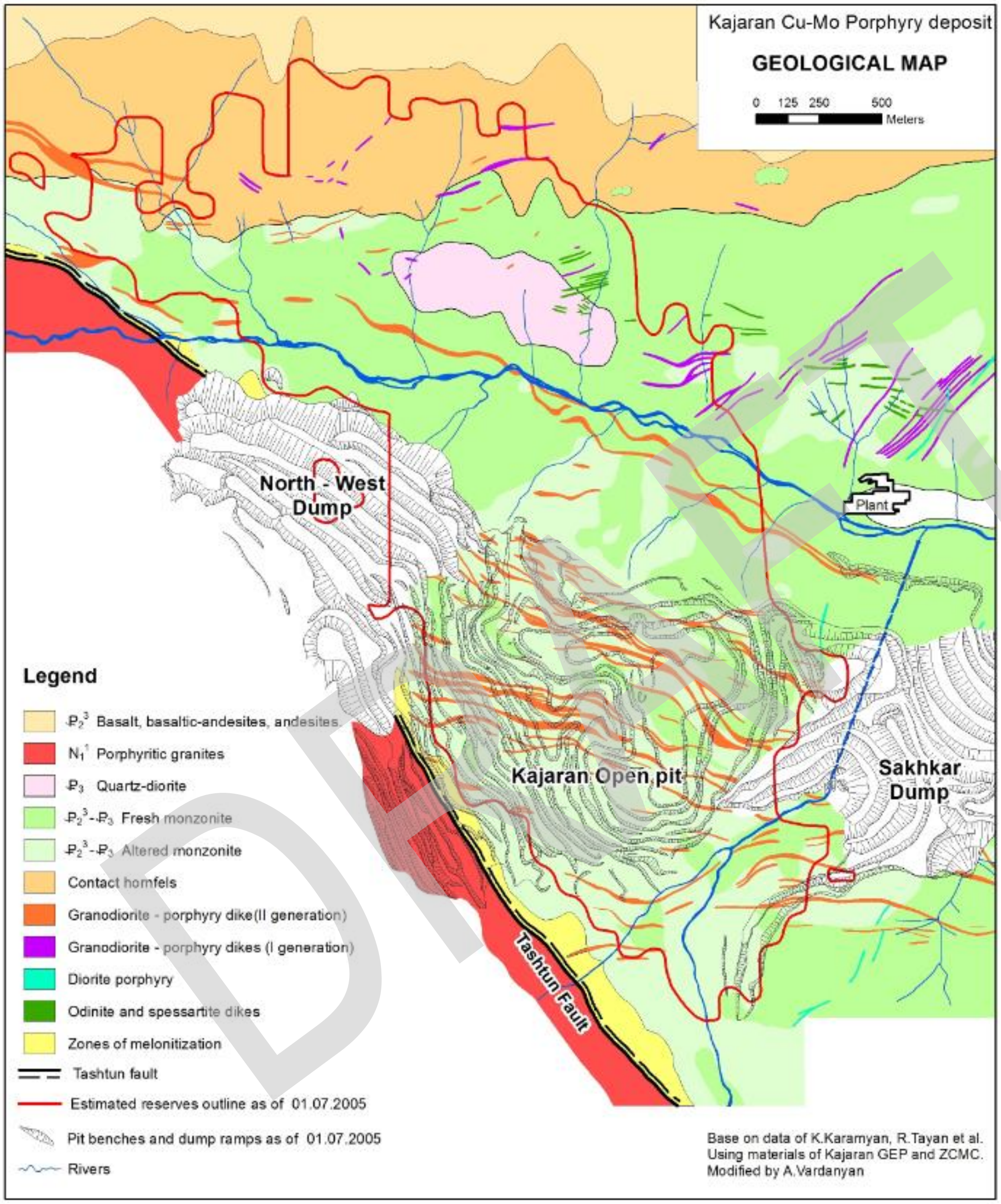

Figure 7-4: Property Geology of the Kajaran Porphyry Copper Deposit

7.3 Mineralization

Crosscutting and displacement relationships of the mineralized fractures and detailed studies of the age relationship between different paragenetic mineral associations were the criteria for distinction of ten stages of mineralization at the Kajaran deposit (Karamyan & Faramazyan 1960).

The main paragenetic association of the Kajaran deposit includes pyrite, molybdenite, chalcopyrite, bornite, chalcocite, covellite, digenite, tennantite, tetrahedrite, luzonite, sphalerite, and galena, generally in a gangue of quartz, followed by a late carbonate and gypsum stage (Figure 7-5). Forms of mineralization are subdivided in different stages, the economic stages being: quartz-molybdenite, quartz-molybdenite-chalcopyrite, and quartz-chalcopyrite.

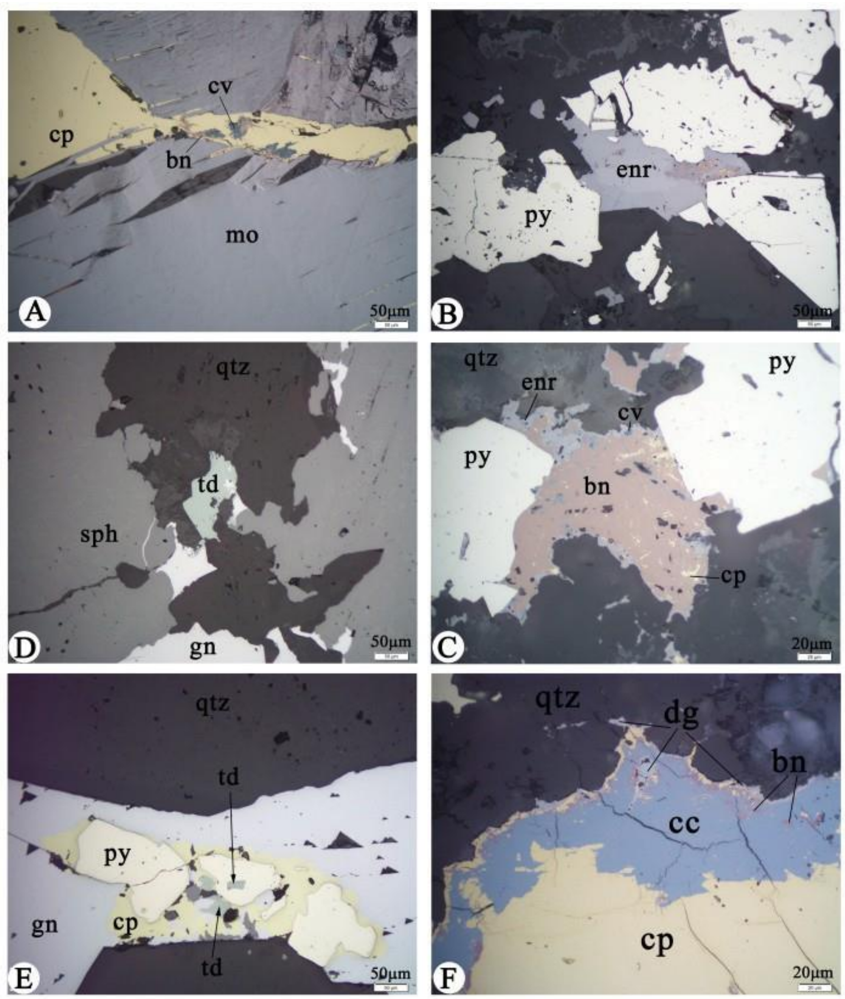

Figure 7-5: Reflected Light Plane Polarized Micrographs

Abbreviations: mo – molybdenite ; cp – chalcopyrite ; py – pyrite ; enr – enargite ; bn – bornite ; sph – sphalerite ; gn – galena; td – tetrahedrite ; dg – digenite ; cc - chalcocite ; qtz - quartz .

Fluid inclusion petrography and microthermometry were carried out at the University of Geneva, Switzerland. Measurements were made using a Linkam THM-600 combined heating and freezing stage with a temperature range of −196°C to 600°C, mounted on a Leica DM LB petrographic microscope. Cathodoluminescence (SEM- CL) was carried out at the University of Geneva and University of Lausanne.

Quartz from the different mineralization stages contains a large variety of fluid inclusions, which can be classified into five families according to their nature, bubble size, and daughter mineral content (Hovakimyan et al. 2013): vapour-rich (VR), aqueous-carbonic (Ac), brine (B1), polyphase brine (B2) and liquid-rich (LR) inclusions (Figure 7-5).

The size of fluid inclusions ranges from 2 µm up to 100 µm. In some cases they can be larger. Generally, they have a negative crystal or irregular shape, tubular and spherical shapes are also present.

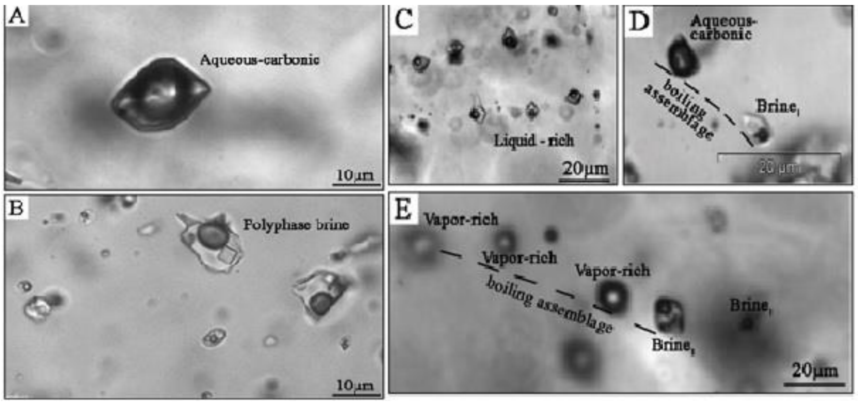

Figure 7-6: Kajaran Fluid Inclusion Types

A- Aqueous-carbonic fluid inclusions; B- Polyphase brine inclusions; C- Secondary liquid-rich inclusions; D- Brine inclusion coexisting with Ac in boiling assemblages; E- Brine inclusions coexisting with VR in boiling assemblages

Detailed fluid inclusion petrography and microthermometry combined with SEM-CL images (Scanning electron microscope - cathodoluminescence) of quartz, revealed several characteristics of the ore-forming fluids at Kajaran.

Cathodoluminescence (SEM-CL) images reveal four generations of quartz. Molybdenite is associated with a dark luminescent quartz generation (Q2), which contains typical brine and aqueous-carbonic fluid inclusions, with some of them coexisting locally as boiling assemblages. Final homogenization of all brine inclusions occurs by halite dissolution. Dissolution of halite between 356±8 and 422±10°C in brine (B1) inclusions of quartz - molybdenite veins indicates salinities between about 42.6 to 50.7 wt % NaCl equiv. The vapour bubbles homogenized between 278±4°C and 327±5°C.

In quartz of the quartz-molybdenite-chalcopyrite veins, typical primary inclusions include vapour-rich brine and aqueous-carbonic fluid inclusions. Secondary inclusions are of the liquid-rich type, and of the polyphase brine type. The temperature of CO_2 melting of aqueous-carbonic fluid inclusions ranges from -56.9 to -56.6°C. Final clathrate melting occurs from +7.3 to +8.3°C. Homogenization temperature of CO_2 could not be measured reliably, because homogenization was to the vapour, therefore phase boundaries were difficult to observe. The bulk homogenization was to the vapour phase between 398±10 and 427±8°C. The clathrate melting temperatures yield salinities from 3.3 to 5.2 wt % NaCl equiv.

The main copper deposition at Kajaran is associated to quartz-chalcopyrite veins. Total homogenization of brine inclusions occurs by halite dissolution between 361±4°C and 385±3°C. Salinities lie between 43.4 and 45.9 wt % NaCl equivalent.

8.0 DEPOSIT TYPES

The Kajaran Cu-Mo mineralization belongs to the type of disseminated and stock work deposits associated with plutonic porphyritic intrusives which is well known under the name "Porphyry Copper". Molybdenum may or may not be present in porphyry coppers. If it reaches economic levels Mo-rich porphyry may be called "Porphyry Molybdenum". Together with the other porphyry Cu-Mo deposits of the Lesser Caucasus like Agarak (in production), Lichk (starting operation on 2016) and Aygedzor (not in operation), Kajaran is part of a belt of porphyry copper deposits, that stretches roughly from Bor in Serbia, over Medet in Bulgaria, to Sar Cheshmeh in Iran. Other deposits of Armenia within this metallogenetically fertile belt are polymetallic gold bearing like Kapan (in operation) and Terterasar, Lichkvaz-Tey, Marjan (not in operation).

9.0 EXPLORATION

9.1 Historical Exploration

In early stages of exploration of Kajaran deposit the main method was underground drifting. In the first stage (1931-1937) trenches, pits and few short adits were excavated to chase quartz-molybdenite veins and zones of intensive mineralization along the granodiorite-porphyric dykes. But exploration didn't go beyond the oxide zone.

Within next two phases of exploration (1938-1944 and 1945-1951) a network of underground exploration drifts were formed in horizons 2,175 m, 2,165 m, 2,125 m and 2,075 m by driving adits in opposite directions from two sides of Gandzasar mountain (in place of recent open pit). 100 x 100 m blocks were formed by crosscuts in each horizon. At the 2,075 m horizon the drift network was not regular as workings mainly chased thick quartz-molybdenum veins discovered at this depth of the deposit. From the west of the deposit adit #30 was driven to study the zone of the Tashtun fault. It crosscut the fault zone from the porphyric granites and entered into mineralized monzonites.

In the next phase (1952-1961) of exploration 2,025 m and 1,875 m horizons of the central area was explored by underground workings, and underground holes were drilled from adits of the 2,075 m and 2,025 m horizons. Within this period the Left Bank Area (NW from Voghji River) was explored by drill holes and adits.

Before 1938 underground workings were driven by hand at 2.7 m^2 cross sections. Later hand work was replaced by machines and the cross section of adits became 5.1 m^2, and after 1981 adits were driven at 6.4 m^2 cross sections.

Phase by phase, underground exploration was replaced by drilling of vertical holes. And from 1967 to 1984 the deposit was explored only by drill holes. Holes were drilled along the South-North exploration lines at a spacing of 100 m. The distances between drill holes along lines were 200 m, 100 m and 50 m.

9.2 Exploration by Current Owner

The current owner initiated a drilling program which was started in 2005 and continued till 2012. The aim of the drilling was to infill existing drilling in the Shlorkut area, along the edges of the Central area, and in the Eastern part of Left Bank area. In this phase few twinning holes also were drilled for data verification. A total of 131 holes were drilled within this period.

In the autumn of 2015 the company drilled 9 holes, 3 of which were for mainly hydrological purposes, and the other 6 were drilled at the Northern edge of the Central Area of the deposit to infill gaps here and to check the historical data. The 2015 drill hole data is not included in the current resource estimation, as assay results were not completed.

10.0 DRILLING

10.1 Historical Drilling

Exploration core drilling in the central part of the license was conducted from about 1940 on until 1984 with conventional Russian ZIF 300M, ZIF 650A, ZIF 1200A and URB-ZAM rigs. With a few exceptions, all holes were drilled vertically. Drill grids were 50 x 100, 100 x 100 and 100 x 200 m. The deepest hole reached 850 m. Core drilling on the flanks of the deposit started around 1967. A few holes were drilled from underground openings.

Drilling diameter was 150 mm for up to 40 m depth. After 40 m the diameter was reduced to 110 mm. For weak or barren zones 91 mm drilling was used. The average core recovery for historic drilling is about 70%. At the intensively broken or fractured intervals core recovery decreased to 55-60%. In the case of poor recovery drilling mud also was sampled, but it always had higher grade than the core, therefore never used for resource estimation.

From the start of mining until 2003 percussion drilling using simple cable tool drill devices was carried out, filling in a grid of 50 x 50 m with Russian BU-20M rigs to 60 m depths.

In 2003-2004 a few vertical holes were drilled in the Shlorkut area and on the Northern and Eastern rims of the open pit by Kolchedan CJSC based in Kapan city. This was the first time double tube drilling was applied at Kajaran project. Drilling diameter started from HQ and it was changed to NQ after passing overburden intervals. Russian SKB-4 drill rigs were used

10.2 Current Owner Drilling

Both 2005-2012 and 2015 diamond drilling programs were implemented by the Kolchedan Company. All holes are vertical at HQ starting diameter, with below overburden drilling diameter reduced to NQ. Russian SKB-4 and ZIF rigs were used with the Atlas Copco rods and bits.

Blast hole drilling for grade control has been conducted since 1958 until now, with Russian SBSH-250MN rotary drill rigs within 12 x 12, 10 x 10 or 6 x 6 m grids. In the last few years the company has also used an Atlas Copco ROC F9 rotary rig. The holes were 17 or 22 m deep depending on the height of the benches (bench height plus 2 m over-drill).

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Method and Approach

Channel Samples

In the early stage of exploration all underground workings were sampled on each three meters of advance. Three channel samples were taken from each round of advance: two vertical channels from each wall, and one horizontal channel from the roof connecting two verticals. The cross section of channels was 5x10cm.

After 1941 horizontal continuous channel samples (5 x 10 cm across) were taken from both walls of the workings, each sample one meter long. Where the workings were parallel to veinlets, vertical or inclined channels were taken from both of walls. In both cases, the two channels from the opposite walls were composited as one sample for assay.

Starting in 1951 it was decided to reduce the channel cross section to 4 x 6 cm, and to take 2 m channels from only one of the two walls.

Drill Core Samples

Core samples were taken along the whole drill hole except soil, diluvium and dyke intervals. Before 1961 sampling intervals were 1-2m. After it was decided to sample each drilling run, the average run length was 1.5m. Half cores were taken as a sample by cutting cores along the long axis using chisels. If core recovery was less than 60% drilling mud also was sampled but never used in resource calculations. From the old drilling activities no cores were kept.

For the post 2003 core drilling 1m intervals were sampled. Cores (mostly NQ size) were cut by core saw and half cores were taken as a sample. Post 2003 drill half cores are partially preserved at the ZCMC core shed. Most of pulp duplicates for this period are kept at Kolchedan Company facilities (Kapan), which drilled all the exploration diamond holes after 2003. The same company did geological logging and sample preparation for the holes drilled, with pulps sent to ZCMC lab for analysis.

2015 diamond holes also were drilled by Kolchedan but were logged and sampled by ZCMC geological staff. 2 m intervals were sampled along the whole drill holes including dykes. Only soils and overburden were not sampled. A core saw was used to cut HQ and NQ (mostly) size cores. Samples were packed in calico bags, coarse blank, pulp blank and standard in each 20 samples were inserted before they have been sent to SGS lab for preparation and assay. Coarse duplicates were taken to the SGS lab after crushing the core sample.

Composite samples were formed from the rejects of original samples. As a result, Au and Ag grades were assayed only in composite form for use in the resource estimate (same as the 2005 estimate).

Percussion drill samples

Wet percussion drill samples were collected during infill percussion drilling. Samples were taken from each 0.7-0.8 m intervals and then combined into 3 m samples. The sample was caught in a collection tray and 5 kg of chips were collected randomly by a digger and put into a bucket. These samples were not used in the resource estimate.

Blasthole Samples

One blasthole sample is taken from every second production hole (17 or 22 m deep) of the 6 x 6 m production drill grid. The wet sample is grabbed by shovel along a grid from the drill chip pile around the collar and 5 kg are collected in a bucket. These samples were also not used in the resource estimate.

SG Samples

To define specific gravity or bulk density values 10 - 20 cm pieces of cores were selected from the holes drilled in the central area and were sent to labs. From the Left Bank area bulk samples were taken from underground adits and bulk density measurements done in the field.

11.2 Sample Preparation

Historically, all diamond drill sample preparation was done at the Kajaran Expedition prep lab, equipped with jaw and cylinder crushers, ball mills and pulverizers. Splitting was done manually by “cone and ring” method, or rarely by riffle splitter. Samples were crushed and reduced according the Richard-Chechot formula: $Q=kd^2$, where “Q” is a weight of sample after splitting in kg, “d” is the max particle size in the sample, and “k” is a coefficient which is equal to 0.2 for Kajaran.

After 2003 all drill core sample preparation was done at the Kolchedan Company lab in Kajaran. 2015 core samples were sent to the SGS Kapan lab for both preparation and analysis.

All blasthole sample preparation is done at the ZCMC prep lab.

11.3 Analysis

All exploration samples from the Kajaran deposit were analyzed for Molybdenum and Copper, both total and oxide using the following analytical laboratories:

- Chemical labs of Armenian Geological Survey, up to 1947;
- “SevKavCvetMetRazvedka” (Ordjonikidze), 1948-1949;
- “KavCvetMetRazvedka” (Tbilisi), 1949-1958;
- “Kajaran Copper-Molybdenium Combine” (Now ZCMC), 1958-1977 and 1985-2015; and,
- “Kapan Geological Exploration”, 1978-1984.

The hydrogen sulfide method of molybdenum assay was applied up to 1938. After 1938 thiosulfate and iodine methods were used for molybdenum and copper respectively. In 1940 colorimetric methods started to be applied for both copper and molybdenum.

The analytical lab of ZCMC used two types of analytical methods:

- Express analysis on Mo and Cu for samples of the processing plant to control the current process of producing concentrates and tailings; and,
- Routine analysis on Mo and Cu for core and blast hole samples.

Express analysis methods: Cu is assayed by titration of iodine using sodium-thiosulfate-solution ($Na_2S_2O_3$) after separation of iron with $NH_4(OH)$ and precipitation of Cu as Cu(I)-jodide (CuJ). The complex procedure lasts approximately 2 hours for each sample. Mo is titrated as $PbMoO_4$-precipitate by using $Pb(NO_3)_2$-solution in the presence of tannine.

Routine analysis methods: Low grade molybdenum (range of detection: 0,005 – 5 % Mo) of the geological samples is assayed by colorimetric titration using ammonium thiocyanate (NH_4SCN). Copper of the geological samples (0,005 – 5 % Cu) is analyzed by Atomic Absorption Spectrometry (AAS) using an old Perkin Elmer Instruments AAnalyst 100.

Core samples from 2015 drilling program were prepared (PRP86 SGS code) and assayed at SGS Kapan lab (30km form the mine) operated by SGS Bulgaria and owned by Dundee Precious Metals Armenia. ICP12B analytical method (2 acid digestion and ICP-OES finish) is used for Cu, Mo, Ag and another 31 elements. AAS72B method is used for Oxide Copper and FAI505 is used for gold (fire assay+ICP-OES finish). As was mentioned previously, 2015 data is not including in this Mineral Resource estimation.

11.4 Quality Assurance and Quality Control

Prior to 2015, only duplicates were systematically used for QA/QC analysis. Two types of duplicates were selected during the exploration phases for assay at the main (internal), and at the external labs.

11.4.1 Certified Reference Material

Historically, no Certified Reference Material (CRM) were used for QA/QC. For the 2015 drill program and Golder's check sample program (Section 12.0) three types of CRMs (501b, 503b and 52c) were purchased from Ore Research & Exploration Pty Ltd (OREAS) Australia.

11.4.2 Blanks

Historically, no blank samples were systematically used for QA/QC. Coarse blank material (perlite gravel) and pulp blank samples (23a by OREAS) were used for 2015 drilling program and Golder's check sample program (Section 12.0).

11.4.3 Duplicates

Systematically selected duplicates of samples were coded (logged by new sample ID) and assayed at the same lab by the same method as the original samples. Selected duplicates for each year were grouped by ranges of original sample contents. After the control assay relative standard deviation was calculated for each group per period. Results were compared with the upper limits provided by State Agency of Reserves of USSR.

Table 11-1: Groups by ranges of Cu and Mo contents for QA/QC analysis

Element	Range 1	Range 2	Range 3	Range 4	Range 5	Range 6	Range 7
Mo, %	<0.01	0.01-0.02	0.02-0.026	0.026-0.10	0.10-0.25	0.25-1.00	>1.00
Cu, %	<0.10	0.10-0.50	0.50-0.65	0.65-3.00	>3.00		

Between 1967 and 1985 about 6,500 duplicates were assayed for Mo (6.46% of all samples for the period) and about 6,700 duplicates were assayed for Cu (6.63% of all samples for the period). Analysis showed that most standard deviation values don't exceed the provided upper limits.

11.4.4 External Lab Check

External lab checks were performed to control a Systematic Error for the particular group of control samples for each year (period). For the external lab check a number of duplicates (about 5% from all original samples) were selected and grouped by same ranges of grades as for internal lab checks. Analysing original and control lab results a Student Criteria was calculated for the each group by the following formula:

$$t_{calc} = \frac{[d]\sqrt{m}}{sd}$$

Where: t_{calc} is the calculated Student Criteria;

d is the absolute systematic error for the group;

m is the number of control samples in the group; and,

sd is the standard deviation.

If "t_{calc}" is greater than the theoretical value of "t" provided by the State Agency of Reserves the Systematic Error is significant for the group.

Different labs for external lab control were used over the period of exploration of Kajaran deposit. Lab names by the periods are:

- 1951: Central Chem. Lab of "VostSibCvetMetRavedka"
- 1952: Chem. Lab of "GinCvetMet", Moscow
- 1970-Feb, 1974: Central Chem. Lab of North Osetian Geological Expedition, Ordjonikidze
- March 1974-Aug1974: Central Chem. Lab of Geological Survey, Yesentuki
- Sept 1974-1977: Central Chem. Lab of Bronnic Expedition, Bronnici
- 1978- 1984: Central Chem. Lab of "SevKavCvetMetRazvedka", Ordjonikidze
- 2003-2012: Central Lab of Analytic CJSC, Yerevan

For the period from 1967 to 1984 about 5,150 samples (5.1% of all samples for that period) were sent to external labs for control assay on Cu and Mo. In some groups systematic errors were defined but no correcting coefficients were applied on the particular ranges of samples for the particular period of time when resources were calculated, because; a) the values of systematic errors were lower than the values of random errors caused by an element variability and by the lab work, and b) the grades with the systematic errors belonged to ranges "<0.01%Mo" and "0.01-0.02%Mo", which were lower than the cut-off grades for that period, and these samples did not participate in resource calculation.

Since 2003 duplicates for external lab check are being sent to the Central Lab of Analytic CJSC, which is a state company and it is regulated by the Armenian Government, ruling that it is mandatory for all exploration and mining companies to send 5% of their samples to this lab for external checks. Since 2003 no significant systematic error have been found within groups of samples sent to Analytic Central Lab from ZCMC.

11.5 Sample Security

For the duplicates selected for both internal and external lab checks new sample IDs were applied by the geologists and lab staff didn't know the original sample ID. No other special security action is performed at the project.

12.0 DATA VERIFICATION

Due to the fact that the bulk of historic drill data was analyzed at the uncertified lab in Kajaran and lack of accompanying laboratory certificates, it was important to verify the grade information through a program of check assays from half-core and pulp rejects remaining from post 2005 diamond drill programs, and stored at the Kajaran mine site.

A total of 81 holes remaining from post 2005 drilling are stored at the site. Some have some boxes missing, but most are complete.

A selection of 204 half-core samples collected from 13 holes was chosen by Golder to represent reasonable coverage of the deposit, and submitted to the accredited SGS laboratory in Kapan, Armenia. Included in this process was the insertion of 12 standards, 26 blanks, and 12 duplicates.

As shown in Figure 12-1, correlation between the check samples and original assays for Cu is more scattered than anticipated with a correlation of only 64%. Golder is satisfied with the opinion however that the major factor for this is that collecting exact duplicates from the half-core boxes was made difficult due to the lack of depth markings. This conclusion is supported by the fact that when composited into 50m lengths, the correlation rises significantly to 80%.

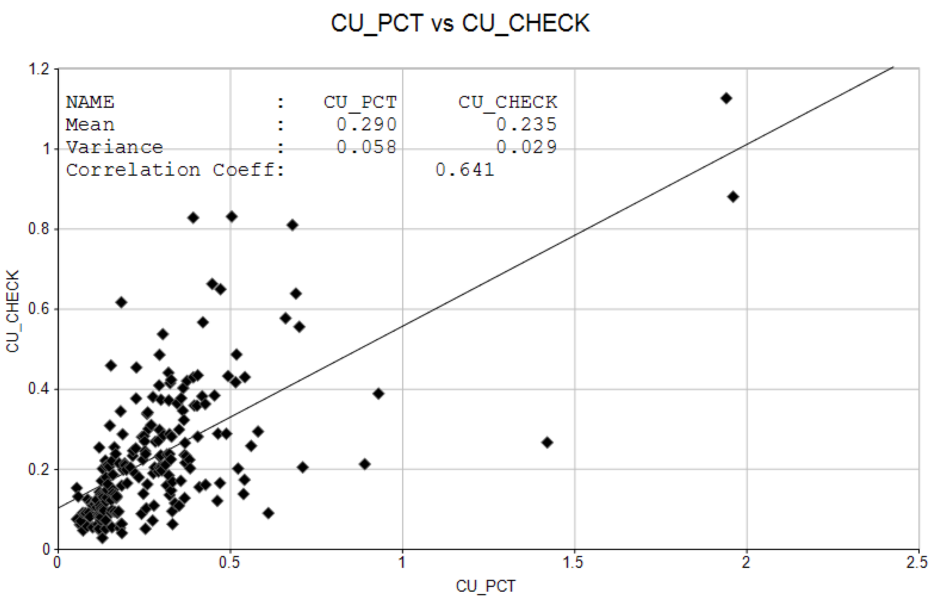

Figure 12-1: Original Cu (CU_PCT) vs Half-Core Check Cu (CU_CHECK)

Post 2005 drilling pulp rejects were also available, and a representative selection of 2,165 from 14 holes (Figure 12-2) was also submitted to the SGS lab in Kapan. It should be noted that of the 2,165 pulps chosen, only 1,811 of the historic assays had values for Mo due to detection limits at the Kajaran lab. A total of 113 standards were also inserted into the batches of submitted pulps.

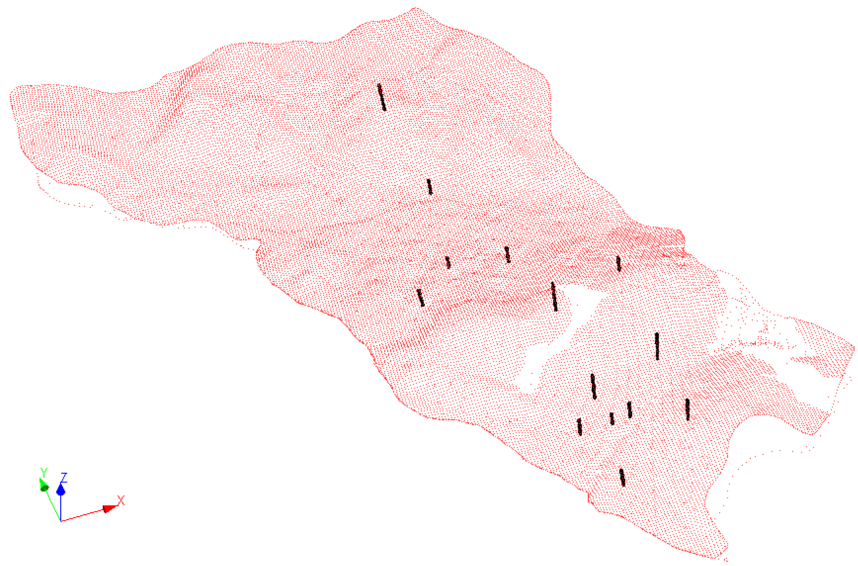

Figure 12-2: Locations of Pulp Check Assay Drill Holes

Results of the pulp check assays are much better (Figure 12-3), with a correlation of 91% for Cu and similar grade variances. When sorted by increasing values of the check samples however, the trend line of historic Cu values (left side of Figure 12-4) would seem to indicate a negative bias above 0.4% when compared to the pulp check assays. Good reconciliation to the mill would suggest that any bias in the historical data is minimal, with no material effect on the resource, but additional check assaying of pulp rejects is recommended to confirm and quantify any potential Cu grade bias for minor corrective action in future resource estimates.

The similar trend lines of historic assays used in the resource estimate and pulp checks (right side of Figure 12-4) and matching mean grades and variances would seem to indicate a favourable verification of the historic resource Mo grades. However, a correlation of 71%, a scatter plot favouring a small bias in the historic data (right side of Figure 12-3), and historic higher than anticipated Mo grades at the mill would suggest a slightly negative bias in the historic assays.

Golder is confident that any bias in Cu or Mo will be minimal and not have a material effect on the resource estimate. Before considering any bias correction to the historic data however, it is strongly recommended that all available pulps be assayed by an accredited laboratory, to provide additional support.

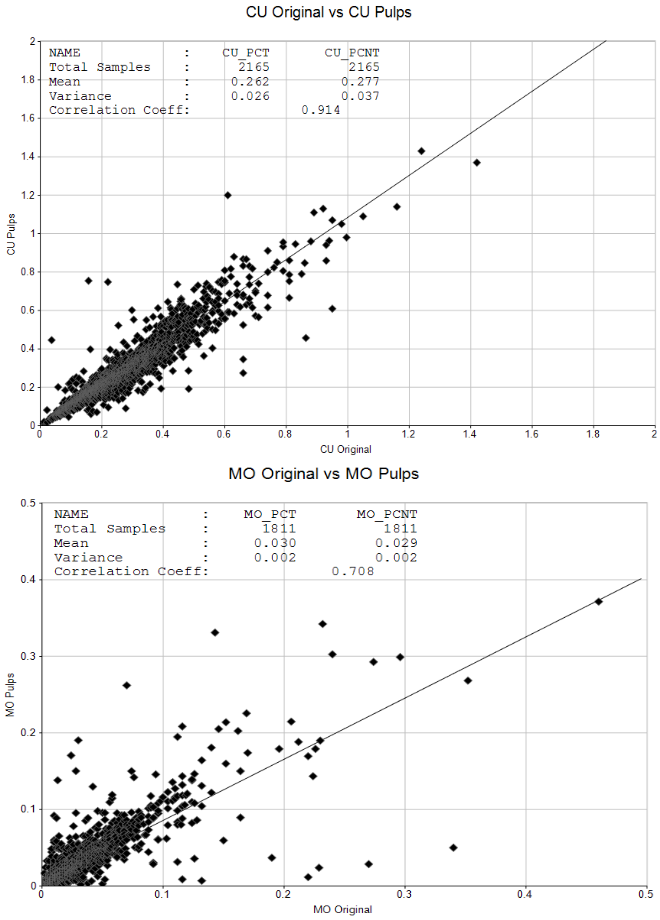

Figure 12-3: Comparison of Cu and Mo Pulp Check Assays to Original Assays

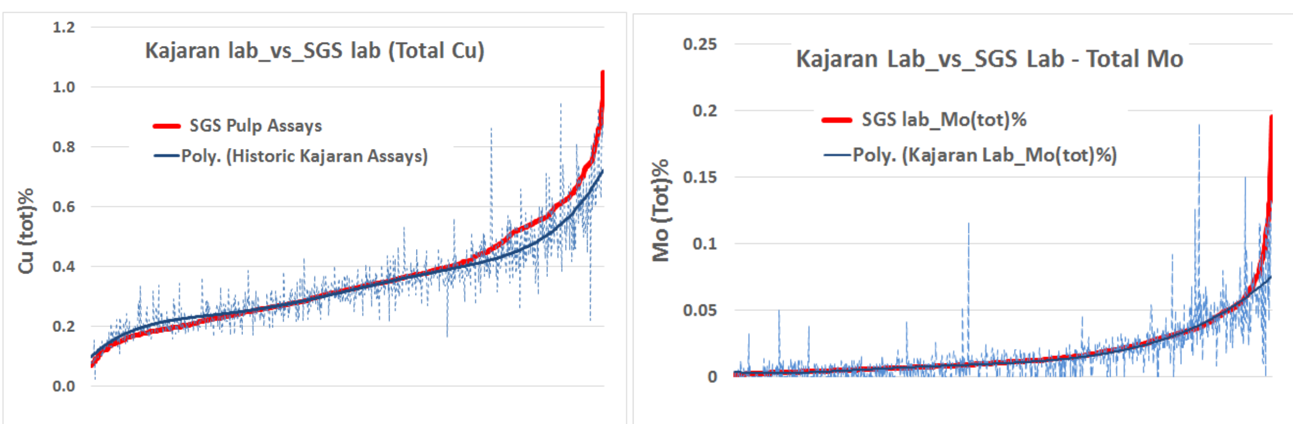

Figure 12-4: Comparison of Cu and Mo Pulp Check Assays to Original Assays By Grade

Pulp check assays (2,165) were also submitted for Au and Ag. Although the pulps were obtained from different (newer) drill holes than the historic data composites used in the resource estimate, Au pulp assays show an increase in the global mean value of more than 200% (Table 12-1). Ag pulp check assays however have a global mean that is approximately 80% less than the historic composites.

Table 12-1: Pulp Check Au and Ag Assays vs Historic Composites

	Records	Length (mean)	Au	Ag
Historic Composites	5,078	8.80	0.017	1.663
Pulp Checks	2,165	1.02	0.044	0.909

Given that there are a greater number of historic composites covering a more representative volume of the deposit, it is considered premature to make adjustments to the Au and Ag grades used in this resource estimate. It should be noted that due to their economic contributions to value at current economic parameters the suggested increase in Au, when combined with the suggested decrease in Ag, would result in an overall increase in equivalent value. The pulp check results suggest then that use of the historic Au and Ag grades is providing understated Au grades, overstated Ag grades, and a slightly conservative estimate overall.

This underscores the requirement for as much additional Au and Ag data as possible, through re-assay of all available pulps and additional drilling.

A correlation matrix of the pulp assay data was generated which showed an approximate correlation of 30% between Au and Cu, and 54% between Ag and Cu. These correlations increase substantially with the removal of a few outliers in the data (62% for Au, 60% for Ag). Sufficient increase in this correlation with additional data could provide the potential for reasonably calculating resource Au and/or Ag values from Cu grades in the future.

12.1 Database Verification

The drill data extracted from ZCMC's Access database was provided to Golder in csv format for input to Datamine. Transfer to Datamine included numerous checks for issues such as duplicate data, interval overlaps, and erratic downhole survey readings.

A few minor issues were found initially and corrective action to the Access database taken, so that the final transfer of drill data to Datamine was clear of errors and suitable for Mineral Resource estimation.

12.2 Site Visit

Greg Greenough visited the ZCMC mine site from June 29 to July 3, 2015. A thorough tour of the open pit mining operations was carried out, and observations included:

- Overall extents of the open pit and waste stockpiles
- Exposures of mineralization and cross-cutting barren dykes

- Production drilling and sampling procedures
- Post 2005 diamond drill half core (Figure 12-6)
- Freshly blasted ore

The visit also included visits with ZCMC geological staff, where characteristics of the deposit were discussed and data was collected and reviewed.

Inspection of the post 2005 half-core remaining at site exhibited abundant areas of finely disseminated chalcopyrite, although quite a few zones were broken, whether from original structural weakness or from the core-halving procedure is uncertain.

Bench faces and freshly blasted ore in the pit also contained disseminated chalcopyrite.

Figure 12-5: ZCMC Open Pit South Wall at Time of Site Visit

Figure 12-6: Post 2005 Drilling Half-core Stored at Kajaran Core shed

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Historical Mineral Processing

The Kajaran mine operation was started in 1951. In 1952 the company produced its first 4,700 t copper concentrate and 200 t molybdenum concentrate. To date the company uses only flotation to produce copper and molybdenum concentrates from the Kajaran ores.

See Section 17 for details on current mineral processing and recovery for the Kajaran project.

14.0 MINERAL RESOURCE ESTIMATE

14.1 Introduction

This resource estimate was completed by Greg Greenough, P.Geo. of Golder and is the first independent NI 43-101 mineral resource estimate and Technical Report for the Kajaran Cu-Mo deposit. The estimate incorporated data analysis, 3-dimensional solids modelling, and geostatistical block model techniques utilizing Datamine Studio v3 (Datamine) in extended (double) precision.

The modelling consisted of one 3D mineralized solid (wireframe) constructed by Golder, based on drill hole geology and assay data, and extending from approximately 89,000E/ -4,200N to 13,000E/ 100N (in the mine's local coordinate system). The dimensions of the modelled deposit are approximately 4,500 m in length and 1,500 m in width, striking in a northwest-southeast direction. Average depth of the deposit is approximately 450 m, limited only by the depth of drilling information.

Diamond drill spacing ranges from approximately 50 m in the central area of the deposit to 100 m in the lesser drilled areas. This resource estimate includes 121 diamond drill holes completed between 2006 and 2012 (inclusive), that had not been included in the previous resource modelling exercise (Kaputin, 2005). A representative selection of remaining half-core samples from these holes was also assayed by certified laboratories to assist in the validation (Section 12.0) of the historic data used in this resource estimate.

14.2 Drill Hole Data

The Kajaran property final drill hole database was supplied to Golder by ZCMC in Excel workbook format on September 2, 2015. These files were extracted from ZCMC's Microsoft Access database, and are summarized in Table 14-1.

Table 14-1: Kajaran Drill Hole Database

File name	Last Modified Date	Comments
Collar.xlsx	Sept 2/2015	775 hole collars; includes 123 post-2005 holes
Survey.xlsx	Sept 2/2015	7,420 down-hole surveys;
Assay.xlsx	Sept 2/2015	129,729 assay intervals;
Geology. xlsx	Aug 24/2015	2,737 logged geological intervals (rock type, alteration type, etc.)

The above files were converted to comma-delimited (csv) format by Golder and imported to Datamine, where they were validated and used to produce 3-D hole data by a combination of custom scripts and internal Datamine merging and de-surveying processes.

Validation of the input data prior to de-surveying consisted of checks including:

- duplicate entries in all data files;
- overlaps in the assay and geology files;
- missing collar coordinate data;

- missing FROM and/or TO data in the assay and geology files; and,
- the unreasonable deviation of adjacent downhole survey intervals.

There were a number of holes with collar information but no corresponding assay or geology data. The final de-surveyed drillhole file consisted of 763 drill holes.

Of the total drillhole database supplied, 124 were drilled after the previous resource estimate in 2005. Of these, 115 were used in the resource estimate which, based on total sample length, equates to a 10% increase in assay data over the previous estimate.

The assay data file contained the column ‘Note’, which identified intervals logged as barren Dyke material. Part of the data validation procedure was to make this attribute value consistent (DYKE) for appropriate modelling of dyke material within the mineral envelope.

Missing data for Cu, Mo, Cu Oxide and Mo Oxide in the ZCMC database was represented by -1 for more than one reason. Grade assignments applied to these instances are summarized in Table 14-2.

Table 14-2: Missing Data Assignments

Reason	Assignment
Not sampled and assumed barren	Assigned 0.0 (includes dyke material)
Mined prior to 2005 and removed from database	Assigned absent data (-)

These intervals accounted for only 5% of the data (length) used in the resource estimate.

Gold (Au) and Silver (Ag) values were not included in the September 2, 2015 ZCMC database, but were available from data files used in the previous resource estimate (Kaputin 2005). These were batch assay values with an average length of approximately 53 m. This data was available only in Datamine format from the Kaputin 2005 resource estimate and contained assay values for other elements, including Rhenium (Re), Selenium (Se), Tellurium (Te), Bismuth (Bi), and Sulphur (S).

It should be noted that Cu and Mo grades in the database are Total Cu and Mo.

Production percussion drill hole data was provided by ZCMC late in the exercise, but was not used in the resource estimation process due to uncertainty in the sampling and assaying procedures. Also, most of this data represented areas that were already mined, therefore would have minimal influence on the Mineral Resource.

In the future however, further analysis of this production drill data may aid in the characterization of the deposit.

14.3 Geological Interpretation

The original topographical surface (prior to mining) with a nominal accuracy of 10 m was provided by ZCMC, and used to confirm collar elevations for those holes drilled from original topography. Surveyed contours of open pit mining and waste dump areas at various stages of the project were also provided by ZCMC and used to construct wire frame surfaces.

A wire frame surface of the Tashtun Fault was also provided by ZCMC, and served as the western limit of the mineralization envelope modelling. Visual inspection of the surface showed correct adherence to the de-surveyed drill contacts.

A bedrock surface was constructed by Golder from the drill data logged overburden intervals, and used as the upper boundary of the deposit (where mineralization exists), including those areas already removed in order to facilitate block model reconciliation to mining.

The mineralization envelope was generated from a combination of points and outer edge control strings and, since the outer limits of the mineralization were 'squared off' rather than tapered, a distance of 1/3 the nominal drill spacing to the closest un-mineralized hole, or from the last mineralized hole, was used for the outer edge control strings. A Cu cut-off grade of 0.1% was used as a guide, however sub-cut-off grades were used where necessary to remove irregular shapes in the mineralized wire frame (Figure 14-1).

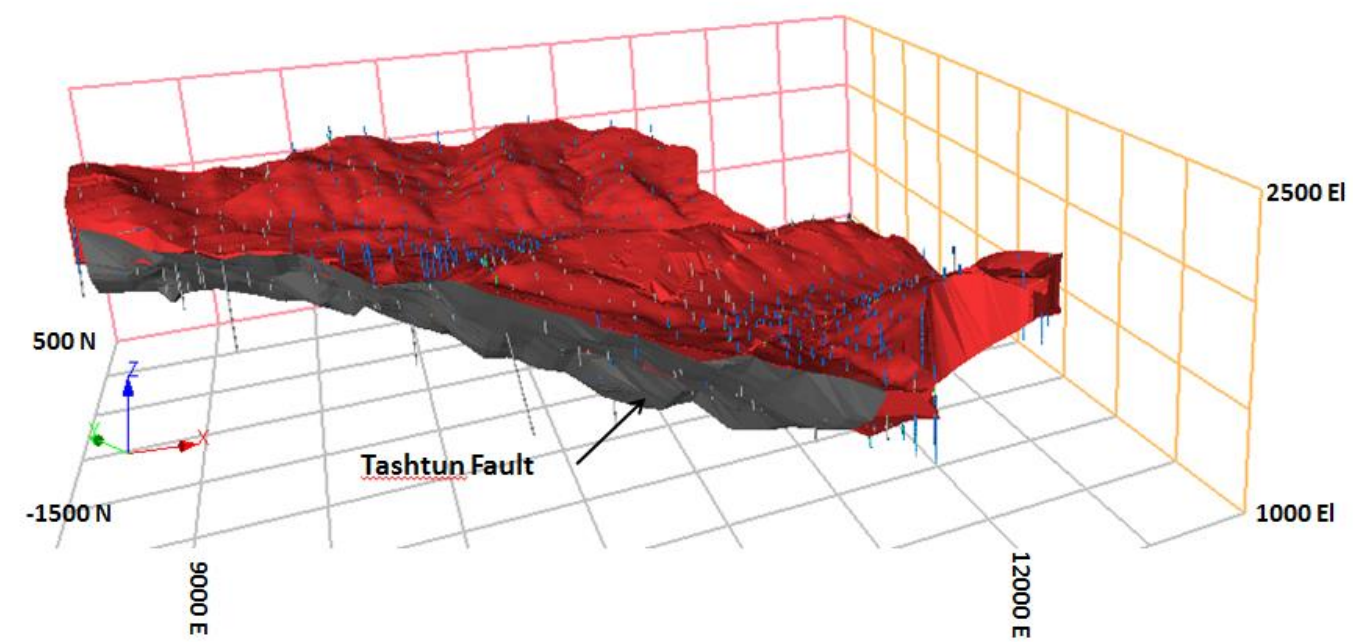

Figure 14-1: Isometric view of Kajaran Mineralization Envelope with Drill Coverage

Generally, the extents of the mineral envelope are controlled by:

- first significantly mineralized interval below bedrock at the top;
- last mineralized drill hole to the north, east and south;
- the Tashtun Fault to the west; and,
- last significantly mineralized interval at the drill hole bottoms.

Note that three or four holes with some mineralization extended well below the bottom of the model, but these were too widely spaced to support including them in definition of the resource model.

It is important to note here that the final model used for Whittle analysis and the Mineral Resource statement is limited on the top to the October 1, 2015 surveyed mined surface, as supplied by ZCMC.

14.3.1 Dyke Modelling

Due to the cross-cutting nature of the barren dykes in the Kajaran deposit, it was important to define them separately from the mineralization, while maintaining spatial accuracy as much as possible. Golder agrees with comments in the previous resource estimate report (Kaputin 2005) that suggest, due to the erratic nature of the dyke material it is extremely difficult to spatially represent the dyke domains and maintain accurate volumetric representation using traditional solids modelling techniques. Regardless of the method used, the ratio of dyke volume to modelled mineralization volume should be as close as possible to the length ratio of dyke to mineralization in the drill data.

To achieve the desired dyke representation in the resource model Golder employed Datamine's dynamic anisotropy search interpolation to generate dyke model blocks to be superimposed into the resource block model:

- Bench mapping dyke strings provided by ZCMC were used to generate trend lines;
- A model with block sizes of 5 x 5 x 5 m was defined using the same model limits as the resource model, and trends in strike and dip assigned to each block based on the trend lines above;
- Intervals logged as dyke material (NOTE = DYKE) were extracted from the drill data and composited to 2.5 m;
- A narrow rectangular search volume (140m strike x 70m dip x 3m thickness) was defined (note that various search sizes were tested, with the one described here achieving the desired volume of modelled dyke material); and,
- Blocks were captured by Nearest Neighbour interpolation, using the dyke composites and dynamic searching were flagged as dyke material and extracted.

Figure 14-2 illustrates a plan projection of the dyke pit mapping (blue), the trend lines used in the interpolation (green), and the interpolated dyke blocks (brown).

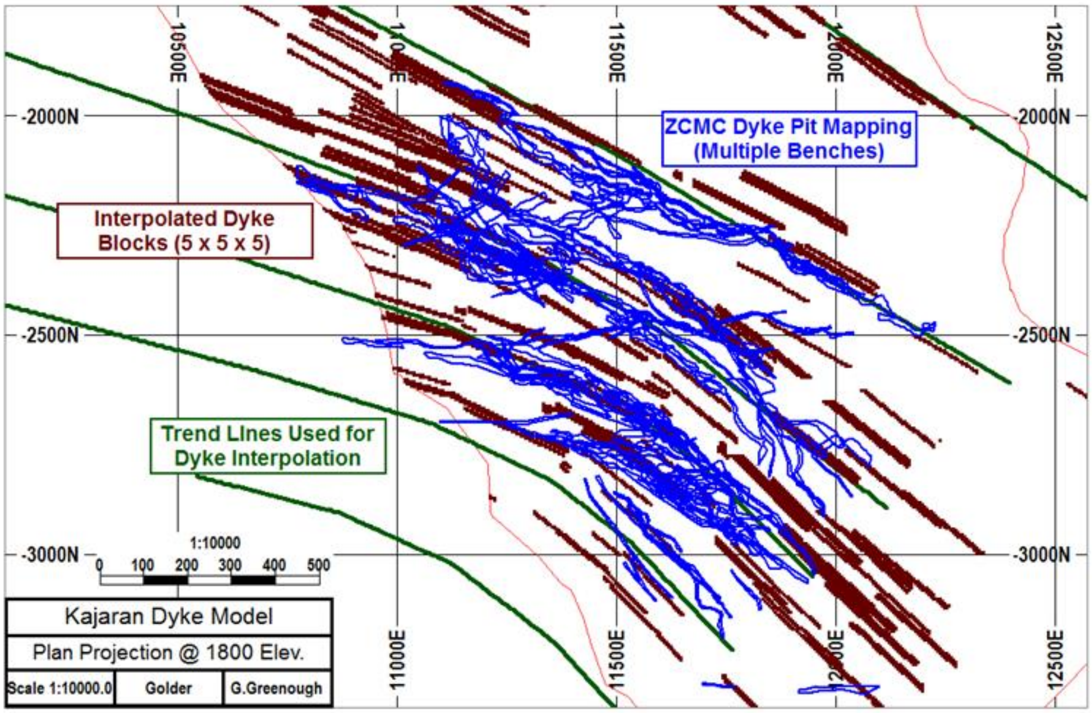

Figure 14-2: Barren Dyke Modelling Plan View

Analysis shows that the ratio of 5.8% for block model dyke vs. non-dyke volumes closely matches the ratio of 5.7% for drill data dyke vs. non-dyke lengths. Golder is satisfied that, given the erratic cross-cutting nature of the dykes, this method provides a reasonable spatial and volumetric representation of dyke material used in the Mineral Resource estimate.

14.4 Data Capture and Exploratory Data Analysis (EDA)

Table 14-3 summarizes the primary statistics of the drill hole data intervals lying within the mineralization envelope (Figure 14-1), and used in the resource estimate.

The Cu and Mo data (sulphide and oxide) in the de-surveyed drill holes generated from the ZCMC database (top 5 rows of Table 14-3) was captured separately from the Au, Ag and Sulphur (S) data (bottom 4 rows of Table 14-3) obtained from the previous resource estimate data (Kaputin 2005). There are samples from 750 holes contained in the captured Cu-Mo data and samples from 221 holes in the Au-Ag data.

Note that before capture of the Cu and Mo data with the mineral solid, all intervals noted as DYKE were removed.

Visual checks of drill hole grade values during the modelling showed higher than reasonable grade values, particularly for Cu and Mo oxides. This hole was therefore eliminated from the resource estimate by removal from the database prior to capture.

Table 14-3: Kajaran Primary Statistics (Captured Data)

FIELD	Number of Records	Null Values	Samples > 0	Minimum Value	Maximum Value	Total	Mean	Variance	Standard Deviation	Weighting Field
LENGTH	121,830	0	121,830	0.10	260.0	198,590	1.63			None
Cu %	121,830	871	120,845	0.00	5.680		0.203	0.032	0.180	LENGTH
Mo %	121,830	6,739	114,991	0.00	3.010		0.027	0.003	0.052	LENGTH
Cu Oxide %	121,830	103,754	18,033	0.00	0.860		0.026	0.001	0.036	LENGTH
Mo Oxide %	121,830	117,962	3,824	0.00	0.165		0.003	0.000	0.007	LENGTH
LENGTH	4,906	0	4,906	0.10	10.0	43,264	8.82			None
AU g/t	4,906	4	4,902	0.00	0.40		0.017	0.001	0.023	LENGTH
AG g/t	4,906	4	4,902	0.01	10.00		1.663	0.927	0.963	LENGTH
S %	4,906	4	4,902	0.02	3.38		0.660	0.165	0.406	LENGTH

A correlation matrix analysis was conducted on both the Cu-Mo and Au-Ag captured data. There were no correlations greater than 30% between any of the elements tested.

14.5 Composites

The mean length of the Cu-Mo samples (includes oxides) captured with the mineralization solid was 1.63 m (Table 14-3). A composite length of 2 m was chosen, resulting in only 5% of the captured data longer than 2m and composited into smaller lengths. A composite method was used that eliminates any samples less than 1m at the end of the composited hole from the process. The amount of this eliminated material was tabulated and deemed to be too minimal to have any material effect on the estimates (<0.08% of the total length, with the average grades eliminated equal to the average grades captured).

Note that since dyke intervals were excluded from data capture with the mineralization envelope (Section 14.4) composites did not include any barren dyke material.

As previously noted, Au and Ag values were taken from ‘batch’ (composite) samples, resulting in an mean length of 46.8 m. For consistency in estimation methods a composite length of 2 m was chosen also for the captured Au and Ag samples. The same method of compositing was used as with Cu-Mo, eliminating 0.13% of total length and at the ends of holes, with average grades eliminated the same as the average grades captured.

Prior to compositing, Au was capped at 0.1g/t and Ag was capped at 4 g/t.

Primary statistics of the composite sample files are summarized in Table 14-4.

Table 14-4: Kajaran Primary Composite Statistics

FIELD	Number of Records	Null Values	Samples > 0	Minimum Value	Maximum Value	Total	Mean	Variance	Standard Deviation	Weighting Field
LENGTH	99,453	0	99,453	1.0	2.000	198,433	1.995	0.004	0.063	None
Cu %	99,453	5,935	93,302	0.0	5.660		0.203	0.027	0.165	LENGTH
Mo %	99,453	8,443	90,807	0.0	2.191		0.027	0.002	0.044	LENGTH
Cu Oxide %	99,453	89,349	9,964	0.0	0.508		0.026	0.001	0.034	LENGTH
Mo Oxide %	99,453	96,405	2,902	0.0	0.165		0.003	0.000	0.006	LENGTH
LENGTH	21,647	0	21,647	1.000	2.000	43,206	1.996	0.003	0.056	None
AU g/t	21,647	1	21,646	0.001	0.100		0.016	0.000	0.016	LENGTH
AG g/t	21,647	1	21,646	0.010	4.000		1.647	0.778	0.882	LENGTH
S %	21,647	1	21,646	0.020	3.380		0.660	0.165	0.406	LENGTH

14.6 Variography

Variogram contours in the horizontal plane were calculated for the Cu and Mo composites to analyze for preferred continuity. Since there is a relative abundance of very low grade material defined by the mineralization only Cu grades greater than 0.5% were used in the analysis. Results are illustrated in Figure 14-3.

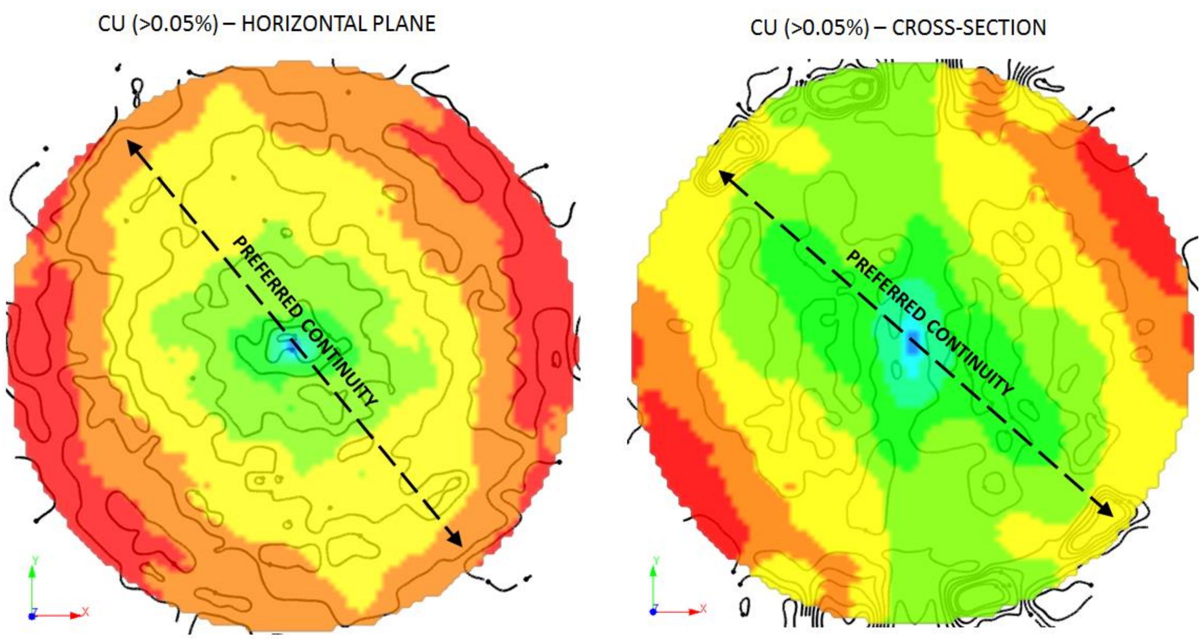

Figure 14-3: Kajaran Variogram Contours

Although weak, a slight elongation in the green and yellow coloration (left half of Figure 14-3) suggests a preferred orientation of continuity in the NW-SE direction, which coincides with the general strike direction of the deposit. Variogram contour generated in the vertical plane, normal to this strike direction, also suggests preferred continuity at a dip of 40-45^{o}, similar in orientation to the Tashtun fault.

Based on these observations experimental grade variograms for Cu and Mo were calculated using the parameters listed in Table 14-5, with rotations applied such that the strike direction was 140^{o}-330^{o}, with a dip of 45^{o}. It should be noted that in the local coordinate system with these rotations applied, in all aspects of the modelling exercise (variography, search volumes, etc.), the X-axis is in the strike direction, the Y-axis is the down-dip direction, and the Z-axis is across the thickness of the preferred orientation.

Table 14-5: Grade Variogram Parameters

Distance	50
Number of Lags	10
Sub-lag Distance	5
Number Lags to be Sub-lagged	5
Azimuth regularization angle	30
Number of Azimuths	2
Cylindrical search radius	25

Interactive fitting of models to the experimental variograms was carried out using the enhanced geostatistical tools in Datamine. In addition to the variograms, the process calculates pair-wise relative variograms (PWRVGRAM), which are the same except that every term in the calculation is divided by the average value of the two samples contributing to that term.

Variogram modelling assumed the best fit using a two structure anisotropic model. As can be seen by the variogram modelling results for Total Cu in Figure 14-4 ranges in the dip and thickness directions are not as clearly defined as the strike, but Golder considers them sufficient to define anisotropic characteristics to the resource estimation process. The second structure ranges were used to define the search distances in the three directions for both the Nearest Neighbour (NN) and Ordinary Kriged (OK) grade interpolations.

Modelling of Mo variograms showed similar shapes to that of Cu, but with slightly shorter ranges in all directions.

Although experimental variograms for Au and Ag were calculated, the relatively limited amount of sample data in ‘batch’ lengths prevented the reliable fitting of variogram models. Although indicated ranges of approximately 150m in the strike and dip directions seem greater than expected for Au and Ag, Golder is satisfied that the low variance of grade, as well as higher than anticipated pulp check assay results (Section 12.0), will allow reasonable estimation results using isotropic search ranges indicated, and similar to those used in the previous resource estimate (Kaputin 2005).

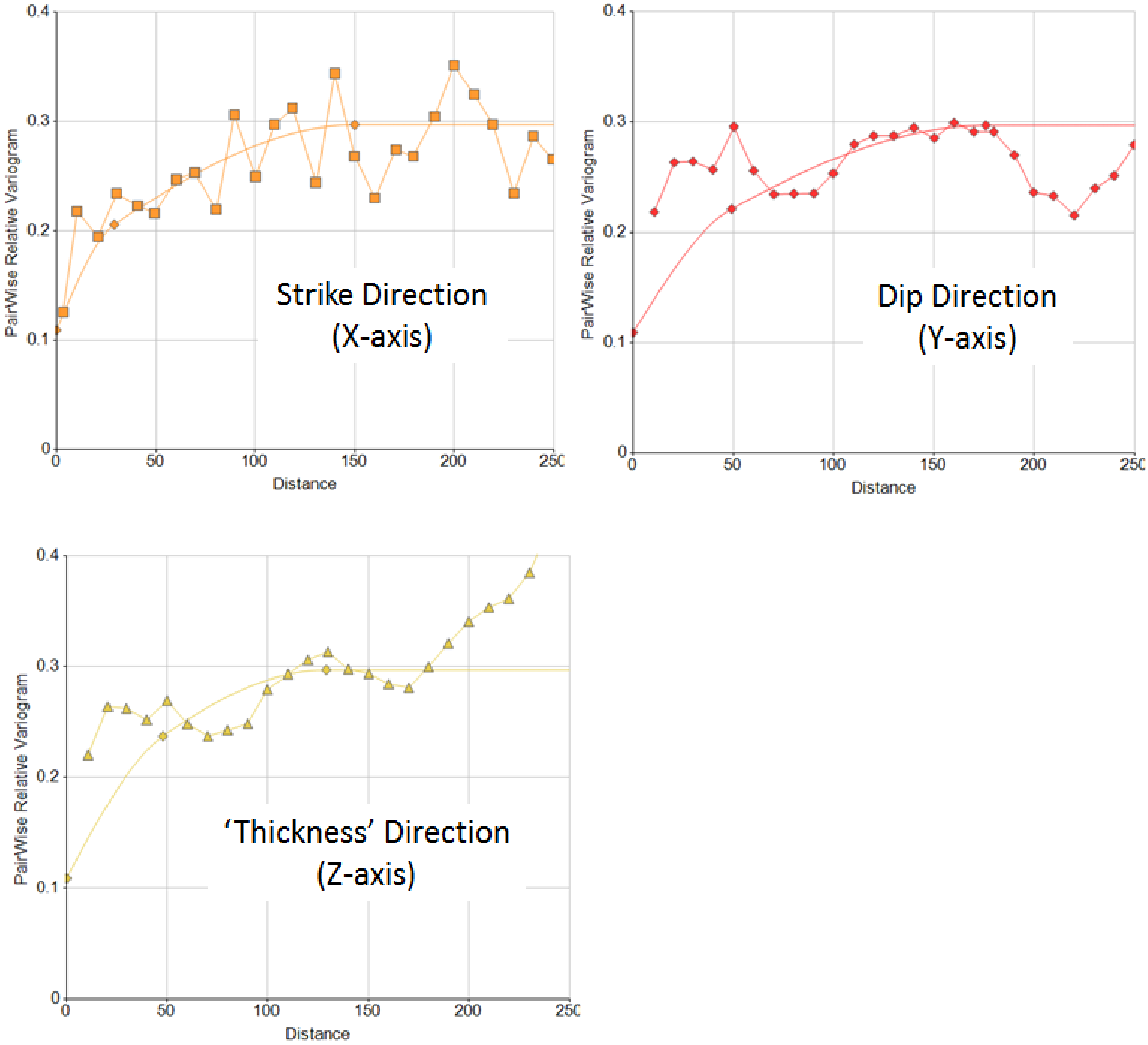

Figure 14-4: Kajaran Cu Grade Variogram

14.7 Bulk Density

ZCMC provided Golder with the same Specific Gravity (SG) data, 120 measurements from 47 holes, used in the previous resource estimate (Kaputin 2005). Most of the data is central to the deposit, in the areas of current and previous mining (Figure 14-5). The histogram illustrated in Figure 14-5 shows a narrow range of distribution with a mean in-situ density of 2.55 g/cm^3. Although historic documentation provided by ZCMC indicates a bulk density in the northern parts of the deposit of 2.63, it was decided to apply the average SG of 2.55 to the entire deposit, as was done in the previous resource estimate.

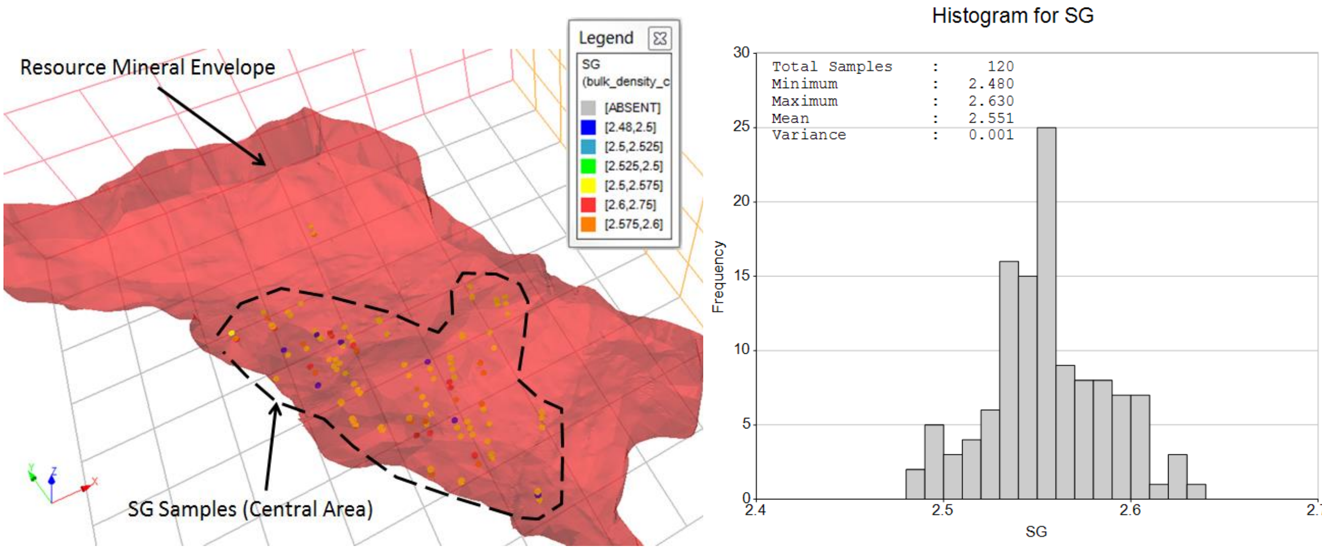

Figure 14-5: Kajaran SG Sample Locations and Distribution

14.8 Resource Estimate

14.8.1 Resource Block Model Definition

A block model was created to cover a 3-D block of the Kajaran mineralized geometry in the local ZCMC grid system. The mineralization extends from, approximately 9,000 to 13,000 m east, -4,000 to 0 m north, and 1,300 to 2,300 m elevation. Table 14-6 shows the mineral resource block model definition.

Drill hole spacing in the more densely drilled areas ranges from 50 m to 60 m in the plane of the deposit, so a block size of 25 m horizontally and 15 m vertically was used to provide an appropriate level of smoothing during distance estimates, with a block height equal to the general bench height of current open pit mining.

Table 14-6: Kajaran Block Model Definition

Model Origin			Block Size			Number Of Cells		
X (East)	Y (North)	Z (Elev.)	X	Y	Z	X	Y	Z
8800	-4,300	1,040	25	25	15	169	193	85

Blank cells were generated by filling the wireframe models, allowing sub-cells along the boundaries to get an accurate volume representation. To allow more efficient grade estimates and more thorough validation techniques, the blocks were then regularized (all blocks put back to the parent cell size), with the Datamine field FILLVOL representing the volume of the block lying within the wire frame, and the Datamine field VOIDVOL representing the volume outside the wire frame.

A volume check of the regularized block model vs. 3D Kajaran mineralized geometry used to generate it revealed a very good representation of the volume (Table 14-7).

Table 14-7: Kajaran Block Model vs Wire Frame Volume Check

No. Blocks	Fill Volume (m^3)	Wireframe Volume (m^3)	Difference
323,090	2,845,559,354	2,845,361,232	0.01%

14.8.2 Interpolation Plan

Block model grades for Cu, Mo, Au and Ag were interpolated using Ordinary Kriging using the variogram models described in Section 14.6, with anisotropic search ranges and orientations based on those variogram models.

Since data for Cu and Mo Oxide grades was insufficient for reliable variogram analysis Inverse Power of Distance (IPD) squared was used for interpolation. Visual inspection of the Cu and Mo oxide grades however suggests that oxide content generally decreases with depth from surface. The assumption was made therefore that the Cu and Mo oxide estimates should use a search volume flattened in the vertical direction (see Table 14-8). To maintain search volume orientations that follow the Kajaran high topographic relief, Datamine's 'dynamic anisotropy' search option was used, whereby a series of control surfaces parallel to the topography were used to estimate strike and dip directions into each block in the model, and those directions used to orient the search volume for each block's estimate of Cu and Mo oxides. To limit the influence vertically in the Cu and Mo oxide estimates a maximum restriction of 5 composites per drill hole was applied.

Three searches were utilized to grade as many blocks in the model as possible. The first search reflected the ranges determined in the variogram modelling, the second and third searches were factored as shown in Table 14-8.

Nearest Neighbour (NN) estimates for each estimated element was also carried out, providing declustered sample grades for global block model validation.

In addition, a '1-sample Nearest Neighbour' estimate was carried out to provide an approximation of a polygonal estimate.

Table 14-8: Kajaran Block Model Interpolation Search Parameters

ELEMENTS	ANGLE 1 [1]	ANGLE 2	AXIS 1	AXIS 2	OCTANTS [2]	1st SEARCH					2nd SEARCH			3rd SEARCH		
						X RANGE	Y RANGE	Z RANGE	MINIMUM [4]	MAXIMUM [4]	FACTOR [3]	MINIMUM [4]	MAXIMUM [4]	FACTOR [3]	MINIMUM [4]	MAXIMUM [4]
Cu%	60	45	Z	X	Yes	75	125	100	12	24	2	12	24	4	24	10
Mo%	60	45	Z	X	Yes	70	100	50	12	24	2	12	24	4	24	10
Cu + Mo Oxides%	-30	-	Z		No	100	100	50	8	20	2	8	20	4	8	20
Au g/t	-	-			Yes	150	150	150	12	24	2	12	24	4	24	10
Ag g/t	-30	-	Z		Yes	150	200	200	12	24	2	12	24	4	24	10

Notes:

1. Axis rotations use 'left-hand rule'; first rotation uses the vertical (Z) axis; second rotation uses the strike (X) axis;
2. Those estimates using octant restriction required a minimum of 5 octants, with a minimum of 1 and maximum of 4 composites per octant;
3. 1st search ranges multiplied by 2nd and 3rd search factors for 2nd and 3rd search ranges;
4. Minimum and maximum composites required for an estimate;

14.9 Block Model Validation

Statistical comparisons between NN and OK estimates (IPD^2 for Cu and Mo oxides), as well as the composites, are presented in Table 14-9. Since the barren dyke material was added to the mineralized blocks after estimation, block model validation (not reconciliation) used only the estimated mineralized blocks.

Table 14-9: Kajaran Resource Block Model Primary Statistics

		RECORDS	MIN.	MAX.	MEAN	VARIANCE
Cu %	Composites	93,518	0.00	5.66	0.203	0.027
	1 Sample NN	292,232	0.00	1.26	0.184	0.011
	NN	317,817	0.00	5.66	0.183	0.029
	OK	317,817	0.008	1.60	0.183	0.009
Mo %	Composites	91,010	0.00	2.19	0.027	0.002
	1 Sample NN	310,498	0.00	0.27	0.025	<0.001
	NN	310,228	0.00	2.19	0.025	0.002
	OK	310,228	<0.001	0.530	0.025	<0.001
Cu Oxide	Composites	10,050	0.00	0.508	0.025	0.001
	1 Sample NN	220,554	0.00	0.508	0.019	0.001
	NN	201,182	0.00	0.508	0.019	0.001
	IPD^2	201,182	0.00	0.359	0.020	<0.001
Mo Oxide	Composites	3,048	0.00	0.165	0.003	<0.001
	1 Sample NN	179,517	0	0.165	0.002	<0.001
	NN	161,632	0	0.165	0.002	<0.001
	IPD^2	161,632	0	0.048	0.002	<0.001
Au g/t	Composites	21,646	0.001	0.100	0.016	<0.001
	1 Sample NN	230,673	0.001	0.100	0.015	<0.001
	NN	230,673	0.001	0.100	0.015	<0.001
	OK	230,673	0.000	0.100	0.015	<0.001
Ag g/t	Composites	21,646	0.010	4.000	1.647	0.778
	1 Sample NN	241,688	0.010	4.000	1.540	0.999
	NN	241,688	0.010	4.000	1.531	1.023
	OK	241,688	0.019	3.854	1.542	0.472

Note: Block model grades are weighted by tonnes.

Differences between the interpolated (OK, IPD^2) and NN global mean grades are negligible and well within best practice acceptable limits of +-5%. Slightly higher mean grades for the Cu and Mo composites indicate minor increased drill hole clustering of higher grade, most likely in areas around the open pit where drill data density is increased.

It should be reiterated here that for the purposes of reconciliation to production the block model includes all areas of mineralization below the bedrock surface, including those areas already mined.

Swath plots comparing average composite grades, NN and OK estimates were also generated and reviewed, and showed reasonably good representation of expected trends and smoothing. As examples, Figure 14-6 includes plots of Cu in the West-East direction and Mo in the South-North direction.

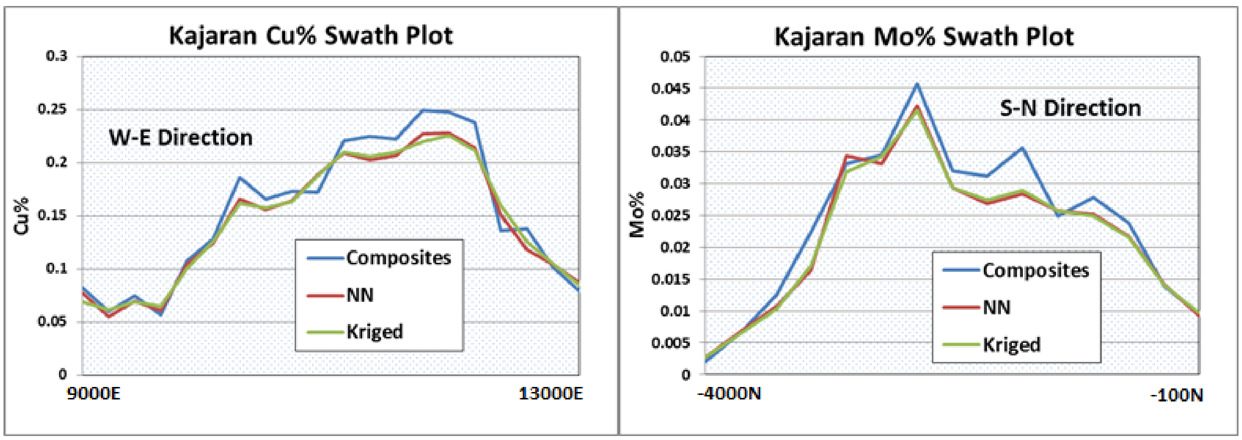

Figure 14-6: Kajaran Block Model Swath Plots (Examples)

Visual inspection comparing block model estimates to the drill data revealed no issues.

Representative plan and section plots showing block model Cu and Mo estimates can be found in Appendix A.

14.9.1 Reconciliation to Production

Mill feed production data (Tonnes, Cu% and Mo% only) was provided by ZCMC, monthly for 2012, 2014, 2015, and annually for 2014. Mining surveys corresponding to these time periods were also provided by ZCMC in the form of polygons, and used to capture resource model blocks for comparison to corresponding mill feed production data. Figure 14-7 illustrates the location of resource model blocks, relative to the entire mineral envelope, selected by mining volumes for the reconciliation analysis.

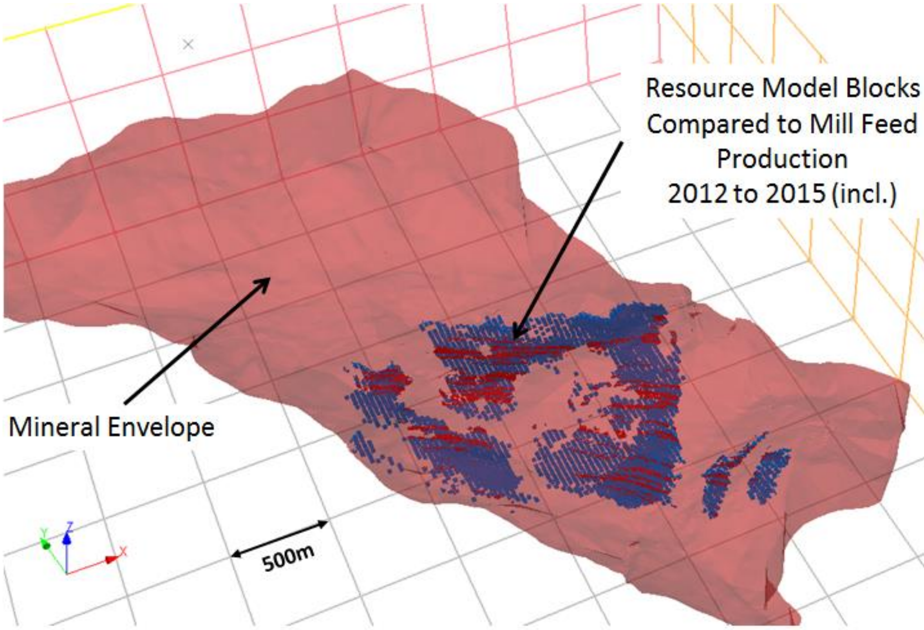

Figure 14-7: Kajaran Model Blocks Used in Mill Reconciliation Study

Table 14-10 tabulates resource block model reconciliation for the various production periods and shows that, other than for a couple of occurrences, block model tonnages and grades comparisons to quarterly mill feed production are within the 15% considered indicative of a confidence in the estimate to support Measured Resource classification. Comparison of the block model to annual production volumes for 2012, 2013 and 2014 are also within the 15% suggested by best practice to be supportive of Indicated Resources.

Table 14-10: Kajaran Resource Block Model Reconciliation to Production

PERIOD	ACTUAL			BLOCK MODEL			DIFFERENCE		
	TONNES	CU%	MO%	TONNES	CU%	MO%	TONNES	CU%	MO%
Q1 2012	3,761,260	0.209	0.035	4,495,740	0.238	0.035	20%	14%	1%
Q2 2012	4,423,200	0.225	0.034	4,083,802	0.216	0.031	-8%	-4%	-7%
Q3 2012	4,403,500	0.236	0.034	4,438,972	0.222	0.031	1%	-6%	-9%
Q4 2012	4,583,500	0.217	0.035	4,040,672	0.225	0.032	-12%	4%	-9%
Q1 2014	4,352,700	0.236	0.034	4,467,346	0.225	0.031	3%	-4%	-8%
Q2 2014	4,794,000	0.227	0.033	4,488,499	0.219	0.030	-6%	-3%	-10%
Q3 2014	4,722,500	0.232	0.034	4,735,800	0.214	0.033	0%	-8%	-3%
Q4 2014	4,842,720	0.227	0.034	3,935,718	0.239	0.033	-19%	5%	-2%
2012	17,171,460	0.222	0.034	17,371,008	0.222	0.032	1%	0%	-7%
2013	18,378,500	0.228	0.034	15,772,176	0.234	0.032	-14%	3%	-7%
2014	18,711,920	0.230	0.034	18,148,557	0.221	0.031	-3%	-4%	-8%

PERIOD	ACTUAL			BLOCK MODEL			DIFFERENCE		
	TONNES	CU%	MO%	TONNES	CU%	MO%	TONNES	CU%	MO%
2015*	13,126,320	0.254	0.031	12,596,942	0.220	0.025	-4%	-13%	-19%

* Includes production for first nine months only

14.10 Mineral Resource Classification

The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guideline for resource classification includes the following definitions (CIM, May 10, 2014), which are pertinent to the classification for the Kajaran resources:

A **Mineral Resource** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of modifying factors (mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors, etc.) in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of modifying factors (mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors, etc.) to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

As one measure of confidence, Golder recorded the search ellipses used during the estimate (Table 14-8), with the first search representing distances supported by the variogram analysis. Approximately 35% of the model blocks were estimated for Cu in the first search, represented by red (Measured) in Figure 14-8, with the bulk of the remainder estimated using the second search, represented by green (Indicated).

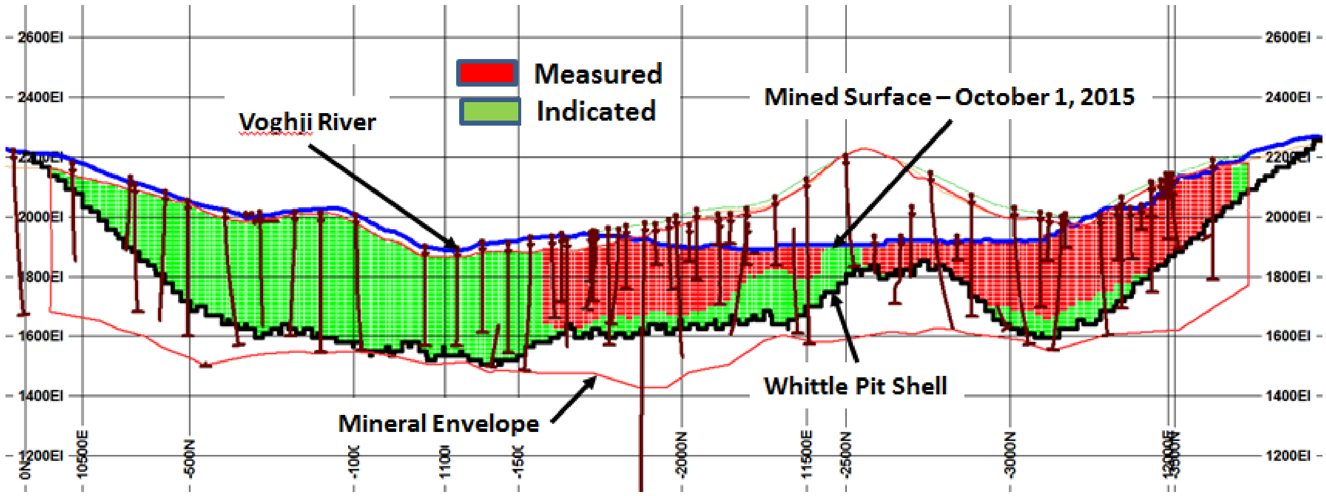

Figure 14-8: Kajaran Longitudinal Section Showing Resource Classification

The longitudinal section (100m thickness) in Figure 14-8 shows that the Measured Resource areas are coincidental with significantly higher drill densities. These areas are also near existing mining, where the reconciliation study (Section 14.9.1) shows positive results on both quarterly and annual bases.

14.11 Cut-Off

14.11.1 Copper Equivalent Calculations

In order to account for all recoverable metals in the reporting of tonnage and grade above selected cut-offs, a Copper Equivalent (CuEq%) value was calculated. Values for Mo, Au and Ag were calculated to the equivalent of 1% Cu using the following long-term metal pricing, selling costs (refining, etc.) and process recoveries:

Element	Recovery	Selling Price	Selling Cost
Cu	78.7%	$2.68/lb.	$0.98/lb.
Mo	83.6%	$7.67/lb.	$0.12/lb.
Au	70%	$1,250/oz.	$570.75/oz.
Ag	75%	$18/oz.	$10.43/oz.

The following conventions were also used in the CuEq% calculations:

- Grams per troy ounce – 31.1035
- Troy ounces per lb – 14.5833
- Lbs per tonne – 2,204.627

The in situ revenue per tonne for Cu and Mo at 1% and Au and Ag at 1 g/t is as follows:

Cu	1% * LbsPerTonne * (Metal Price - Sell Cost) = 1/100 * 2,204.627 * (2.68-0.98) =	**$37.48 /tonne at 1% Cu**
Mo	1% * LbsPerTonne * (Metal Price - Sell Cost) = 1/100 * 2,204.627 * (7.67-0.12) =	**$166.45 per tonne at 1% Mo**
Au	1g / GramsPerOz * Sell Price(1,250 - 570.75) =	**$21.84 per tonne at 1 g/t Au**
Ag	1g / GramsPerOz * Sell Price(18 - 10.43) =	**$0.24 per tonne at 1 g/t Ag**

With consideration to metal recoveries, the CuEq% calculation was defined as follows and applied to the block model for use as the cut-off:

CuEq% = CU * Recovery(78.7)/100
+ (MO * Recovery(83.6)/100 * MO Revenue Factor(166.45) / 37.48
+ (AU * Recovery(70)/100 * AU Revenue Factor(21.84) / 37.48)
+ (AG * Recovery(75)/100 * AG Revenue Factor(0.24) / 37.48)

14.11.2 Cut-off Calculation

CuEq% cut-offs for the Kajaran resource tabulation are based on actual operating costs supplied by ZCMC, and summarized in Table 14-11. All pricing and costs are in $US, so no exchange rate factor is required.

Table 14-11: Kajaran Cut-off Cost Assumptions

	Open Pit
Mining	$1.99
Transportation/Process	$5.12
General & Administration	$0.76
Total Cost ($US)*	**$7.87**

Based on the metal recoveries, pricing and costs, the CuEq% cut-off then becomes:

Cost per Tonne = (Mining Cost(1.99) + Process Cost(5.12) + G & A(0.76)) * Exchange Rate(1) = $7.87
Units / Tonne = (Total Cost per Tonne(7.87) / CU Metal Price(1.70) = 4.63

CUEQ% Cut-off = Units / Tonne(4.63) / (LbsPerTonne / 100) = **0.21**

14.12 Resource Statement

14.12.1 Whittle Pit Optimization

A preliminary open pit assessment of the Kajaran deposit was performed on the resource block model using Whittle v4.5.2, with the open pit costs, selling prices and costs, and metallurgical recovery factors indicated above. An overall average slope angle of 45 degrees was assumed, and a base case (break-even) pit shell generated using all blocks in the resource model lying below the October 1, 2015 surveyed mining surface.

In order to provide Whittle with a model sufficient for analysis, some modifications to the resource block model were required:

- Regularization of the resource model – the combined mineralized and barren dyke blocks were regularized (all blocks the same 'parent' block size – 25 x 25 x 15 m). Where mineral and dyke subcells are combined to produce the parent block size grades are volumetrically averaged;
- Waste blocks (all grades = 0.0) below the bedrock surface added to the regularized resource model;
- Overburden blocks between bedrock and original topographic surfaces added;
- Waste dump blocks between original topographic and surveyed waste dump surfaces added; and,
- Removal of all blocks above the October 1, 2015 surveyed mining surface.

Although some constraints such as permitting and water diversion may be considered in the short and mid-term mine planning process, it is reasonably assumed that these can be addressed over the life of the project and no such constraints were used in the Whittle analysis for Mineral Resource reporting.

A cut-off grade of 0.21% CuEq (Section 14.11.2) was used to report the Measured and Indicated Resources lying within the Whittle pit shell, and is reported in Table 14-12.

Table 14-12: Kajaran Mineral Resources October 1, 2015

Cut-off CuEq%	Classification	Tonnes (000)	Cu%	Mo%	Au g/t	Ag g/t	Cu Oxide%	Mo Oxide %
0.21	Measured	605,420	0.278	0.036	0.014	1.413	0.022	0.003
	Indicated	1,328,401	0.246	0.031	0.018	1.660	0.020	0.002
	Total Resources	1,933,821	0.256	0.032	0.017	1.582	0.021	0.002

Note: Cu% and Mo% are Total Cu and Total Mo

Of the 1.328 billion tons of Indicated Resource, 1.017 billion tons @0.252% Cu, 0.028% Mo, 0.020g/t Au, and 1.797g/t Ag lie north of the river and will require diversion of the river and relocation of the village. ZCMC is confident that the permitting and logistics required is feasible, therefore it is reasonable to assume that this resource has the potential for eventual economic extraction.

In addition to the Mineral Resources within the Whittle pit shell there are 702 million tonnes of waste material, including external waste, internal waste (mineral below cut-off and dyke), and overburden.

For comparative purposes, Table 14-13 shows changes in resources resulting from approximately 10% changes in CuEq cut-offs.

Table 14-13: Kajaran Mineral Resource Sensitivities to Cut-off

Cut-off CuEq%	Classification/ Location*	Tonnes (000)	Cu%	Mo%	Au g/t	Ag g/t	Cu Oxide%	Mo Oxide%
0.19	Measured/South	622,600	0.275	0.035	0.018	1.646	0.022	0.003
	Indicated/South	326,728	0.223	0.038	0.012	1.210	0.019	0.002
	Total Res./South	949,329	0.257	0.036	0.014	1.343	0.021	0.003
	Indicated/North	1,097,084	0.246	0.027	0.019	1.776	0.021	0.002
	Total Resources	2,046,412	0.251	0.031	0.017	1.575	0.021	0.002
0.21	Measured/South	605,420	0.278	0.036	0.014	1.413	0.022	0.003
	Indicated/South	311,010	0.226	0.039	0.012	1.219	0.019	0.002
	Total Res./South	916,430	0.260	0.037	0.014	1.347	0.021	0.003
	Indicated/North	1,017,391	0.252	0.028	0.020	1.795	0.021	0.002
	Total Resources	1,933,821	0.256	0.032	0.017	1.582	0.021	0.002
0.23	Measured/South	579,404	0.282	0.037	0.015	1.415	0.022	0.003
	Indicated/South	289,841	0.230	0.041	0.013	1.236	0.019	0.002
	Total Res./South	869,245	0.265	0.038	0.014	1.355	0.021	0.003
	Indicated/North	917,944	0.260	0.030	0.020	1.810	0.020	0.002
	Total Resources	1,787,189	0.262	0.034	0.017	1.589	0.021	0.002
0.25	Measured/South	545,077	0.288	0.038	0.015	1.422	0.022	0.003
	Indicated/South	262,497	0.236	0.042	0.013	1.263	0.018	0.002
	Total Res./South	807,574	0.271	0.039	0.014	1.371	0.021	0.003
	Indicated/North	809,392	0.268	0.031	0.020	1.820	0.020	0.002
	Total Resources	1,616,965	0.269	0.035	0.017	1.595	0.020	0.002
0.27	Measured/South	503,571	0.294	0.039	0.015	1.434	0.022	0.003
	Indicated/South	230,715	0.243	0.044	0.013	1.292	0.018	0.002
	Total Res./South	734,286	0.278	0.041	0.015	1.389	0.021	0.003
	Indicated/North	700,005	0.277	0.033	0.021	1.826	0.019	0.002
	Total Resources	1,434,291	0.277	0.037	0.017	1.602	0.020	0.002
0.29	Measured/South	457,619	0.302	0.040	0.015	1.450	0.023	0.003
	Indicated/South	197,968	0.251	0.046	0.014	1.324	0.018	0.002
	Total Res./South	655,587	0.287	0.042	0.015	1.412	0.021	0.003
	Indicated/North	593,949	0.285	0.034	0.021	1.828	0.019	0.002

Cut-off CuEq%	Classification/ Location*	Tonnes (000)	Cu%	Mo%	Au g/t	Ag g/t	Cu Oxide%	Mo Oxide%
	Total Resources	1,249,536	0.286	0.038	0.018	1.609	0.020	0.002
0.31	Measured/South	410,208	0.311	0.042	0.016	1.463	0.023	0.003
	Indicated/South	164,996	0.259	0.049	0.014	1.344	0.018	0.002
	Total Res./South	575,204	0.296	0.044	0.015	1.429	0.021	0.003
	Indicated/North	489,518	0.294	0.036	0.021	1.828	0.018	0.002
	Total Resources	1,064,722	0.295	0.040	0.018	1.612	0.020	0.002

* Resources differentiated between North and South of the Voghji River

15.0 MINERAL RESERVE ESTIMATE

There are no Mineral Reserves stated for the Kajaran property in this report.

16.0 MINING METHODS

Current and projected mining methods for the Kajaran project consist of blasthole open pit mining.

17.0 RECOVERY METHODS

17.1 Current Mineral Processing

Copper and Molybdenum are the main economic elements of the Kajaran mineralization. Secondary components are rhenium, gold, silver, selenium, tellurium, bismuth and sulfur. The rhenium is mainly presented in the molybdenite in the form of isomorph mixture and causes the most concentration in its coarse-crystalline and course-scaled intergrowths. The gold occurs in the sulfides of ores, mostly in the form of fine mechanical mixtures, while the silver occurs in the form of isomorph mixtures. The selenium and tellurium are associated with peacock ore and iron sulphide and less associated with molybdenite, sphalerite and galenite. The both elements are available in the form of isomorph mixtures.

According to the mineral composition and technological properties, there are two types of ores in the Kajaran deposit, sulphide and oxide. Where the degree of oxidation exceeds 20% the ore is considered ‘oxidized’ for the purposes of mine planning. Sulphide and Oxide mineralization types within the shipped ore are not extracted separately and are processed as one material according to the same flow-chart.

Oxidized ore in the central section of the deposit was abundant near surface and covered the sulphide ores in the form of shell with the thickness of 40-60 m. Mining has removed the majority of oxidized ore, which remains only in the pleural region of open pit, as well as in the contacts of granodiorite-porphyritic dykes and large fissures up to depths of 200-250 m.

A bulk-differential floatation method is used for the beneficiation of the Kajaran ore, producing Copper concentrate with 26-28% Cu grade and Molybdenum concentrate with 50-51% Mo grade.

Rhenium concentration in the molybdenum concentrate reaches 200 g/t.

Secondary minerals in the copper concentrate reach values of:

- Gold, 4.0 g/t;
- Silver, up to 75-80 g/t;
- Selenium, 100-140 g/t; and,
- Tellurium, 35-50 g/t.

17.1.1 Processing Plant and Facilities

The processing plant is located down-gradient from the open pit in the centre of all infrastructure facilities and in the direct vicinity to administration buildings, mills and transportation facilities.

After comminution and classification, the copper-molybdenum concentrate is processed by using froth flotation with sodium sulphide as main reagent in different flotation cells. Quality control of the processing results is conducted by record keeping of the amounts of reagents used during the processing and evaluating the copper-molybdenum outcome.

Concentrate Dewatering Facility

The facility for concentrate dewatering is located in a separate building. Tailings are thickened using three large cylindrical thickeners. The concentrate is then pressed using an older vacuum filter press and newer chamber filter presses with modern technical control system. After dewatering the concentrate dry cake is collected and loaded on trucks for transport.

Boiler plant

A modern boiler plant is located in the vicinity of the processing plant.

Laboratory

The laboratory is located near the main administration building. Separate areas are available to conduct specific analysis on tailings, copper and molybdenum. The laboratory also has a special area for weighing and for assaying with furnaces.

Crushing Facility

The main crushing facility is located downhill near the Voghji River. The ore is transported either by man-operated trains using the underground rail haulage system from the open pit to the crushing plant or by trucks. Four underground tunnels connect the open pit with the crushing facility. The ore is then transported using conveyor belts from the crushing facility to the grinding facility.

Grinding

Grinding facilities (old and new) are currently in operation. Feeders control the rate of material entering the process. Comminution is accomplished using steel ball mills. After comminution the crushed ore is classified by spiral classifiers and cyclones.

Background

ZCMC is planning to increase plant throughput from 18Mtpa to 22Mtpa.

Conveyor CV42 is reported to be at full capacity and a project is planned to increase the belt width from 1.6m to 2.0m.

Primary Crusher

The primary crusher model is a KKD 1500/180 supplied by Uralmash with the technical parameters noted in Table 17-1.

Table 17-1: Primary Crusher Parameters

Model	**KKD 1500/180**
Feed Opening	1,500 mm
Discharge size from the open side	180 mm
Feed lump size(by 5% rest on square hole)	
Feed	1,200 mm
Product	310 mm
Capacity if material compression strength of 100-150 Mtpa and	2,240 m^3/hr or 3,580 t/hr

Model	**KKD 1500/180**
Moisture content not more than 4%,in open cycle	
Main motor power	400 kw

With 2 primary crushers, each capable of crushing 3,500 tonnes per hour (tph), there is sufficient capacity within the actual crushers for up to 40 Mtpa. Each of the primary crusher has two feeders underneath it to withdraw ore.

Conveyors CV40, 41 & 42

Conveyor CV42 conveyor receives all primary crushed material prior to discharging into the crushed ore storage bin. The full load capacity of this conveyor is determined to be 3,700 tph.

Checks of the edge distance (the distance between the material and the edge of the belt) during operation have determined that it is 120 mm. This distance confirms the current operating throughput of CV42 as 3,580 tph, or 97% of full load capacity.

It is planned to increase the width of the conveyor to 2.0m so as to increase the capacity of the conveyor. The capacity would be expected to be 5,870 tph with a 2.0m belt, a 30 degree idler angle and a belt speed of 2.53 m/s.

The full load capacity of conveyors CV40 and 41 is proven to be 3,480 tph each.

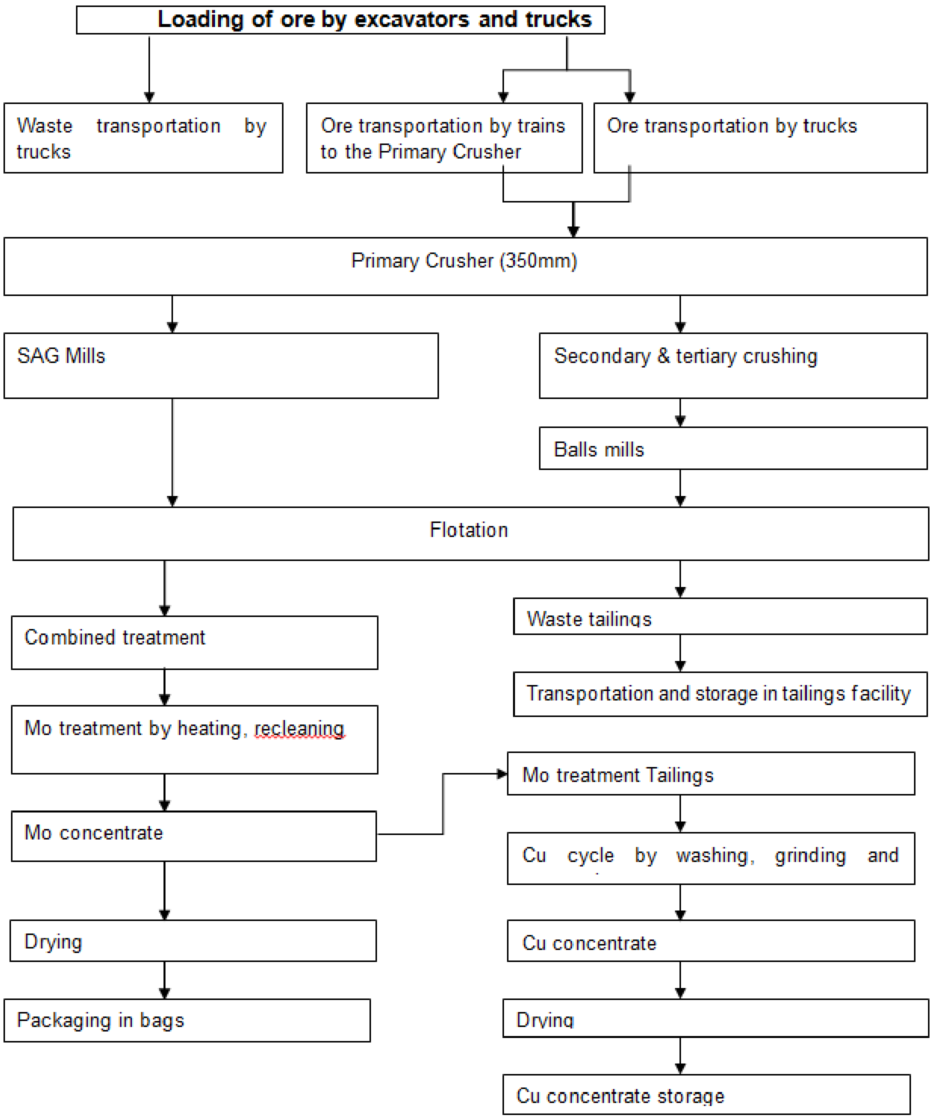

Figure 17-1: Kajaran Production Flow Chart

17.1.2 Copper Molybdenum Ore Enrichment

Collective flotation is done by xanthogenate in a weakly alkaline media created with a small amount of lime, as well as depression of pyrite and activation of oxidized forms of copper with sodium sulfide. Desorption is carried out by means of steaming of the concentrate with sodium sulphide. In the cycle of selection kerosene and sodium sulfide is used as depressor of pyrite and chalcopyrite. Polypropylene glycol monobutyl ether frother (PGMEF) is used as a frother.

Regrinding of the concentrate to 80% size grade (0.08 mm) is implemented to increase recovery. The concentrate with content of 2.54% molybdenum and 12.7% copper is sent to the copper-molybdenum separation section. Floatation tails of the bulk scavenging flotation and 2nd and 3rd cleaner scavengers, with

0.006% molybdenum and 0.041% copper content are final (discardable) tailings. The flotation froth goes back to the beginning of bulk floatation.

Separation of the collective concentrate includes preliminary steaming at 40°C, basic and three control molybdenum flotation, sevenfold cleaning cycle, refining of first Mo cleaning up to 80% size grade, and re-flotation of the first molybdenum cleaning.

Molybdenum concentrate is a foamy product of the seventh cleaning with a content of 50% Molybdenum and 0.4% Copper, and is sent to filtration and drying. Water processed in the molybdenum concentrate filtration and drying is reused in the flotation cycle.

The copper separation is simplified and optimized with the aim of stabilization and facilitation of controlling the flotation process. Industrial products are redirected into the operations where feeding is closer to content of copper. In the control slimes cycle cleaning operations are excluded, control concentrates are wrapped onto the main slimes flotation.

Tailings from the third molybdenum flotation feeds the copper flotation cycle that is being performed by separate processing of sand and slime fractions, including the following main technological operations:

- division of feeding to sand and slime fraction in the hydro-cyclones;
- de-mudding of sand fraction in the classifier and verification de-mudding - in the hydro-cyclones;
- regrinding of sand fraction;
- thickening of overflow of hydro-cyclones of the first and second classification;
- aeration and agitation warmed up to 40–45°C of re-grinded sand fraction;
- Main and control sand copper flotation;
- direction of industrial product of sand flotation composed of cleaning tailings and control concentrate towards the feed of the process, i.e. to aeration and agitation before main sand flotation;
- cleaning of concentrate of main sand Cu flotation with the aim of getting a Cu concentrate with composition of 27.5% copper;
- direction of the tailings of sand flotation to the control slime one;
- aeration and agitation with lime, main and control slurry copper flotation after thickening;
- two cleanings of concentrate of first slurry flotation;
- direction of tailings of first cleaning and concentrate of control flotation to aeration and agitation before main slime flotation;
- direction of tailing of second cleaning to the first cleaning; and,
- depending on composition of copper in it direction of tailings of second copper slime, either to the copper concentrate or copper sand cycle.

Tailings of the control slime flotation are waste, and together with tailings from collective flotation are sent to the plant tailings dump.

Copper concentrate is sent to thickening and filtration, after which filtered copper concentrate with a composition of 26.5% Copper and moisture content of 9-10% goes to consumer.

17.1.3 Increased Capacity Projections

Projected increased throughput and metal balance (Table 17-3) compared to current actual throughput and metal balance (Table 17-2) is planned to result in the following:

- Quality of Cu concentrate – 26.5% at recovery 78.2% (recovery increases by 2.23%).
- Composition of molybdenum in molybdenum concentrate made 49% at 83.06% recovery
- Growth of annual acquisition of Cu and Mo metals in to the metals of the same name made correspondingly 7.72 Kt/year and 1,1Kt/year
- Quality of copper and molybdenum concentrate acquired complies with the main requirements re composition of admixtures including arsenic
- Copper concentrate corresponds to KM5-KM6 mark.
- Molybdenum concentrate corresponds to KTS3 mark.

Table 17-2: ***Kajaran Actual Metal Balance***

Product name	Yield		Grade %		Recovery %		Metal Kt	
	Kt	%	Cu	Mo	Cu	Mo	Cu	Mo
Cu Concentrate	110.4	0.65	26.500	0.035	76.00	0.69	28.42	0.04
Mo Concentrate	9.5	0.06	0.400	49.000	0.10	83.06	0.04	4.66
Sum of the concentrates	**119.9**	**0.71**	**24.430**	**3.918**	**76.10**	**83.75**	**28.46**	**4.70**
Tailings	16,880	99.29	0.053	0.005	23.90	16.25	8.94	0.91
Feed ore	**17,000**	**100**	**0.220**	**0.033**	**100.00**	**100.00**	**37.40**	**5.61**

Table 17-3: ***Kajaran Design Metal Balance***

Product name	Yield	Grade %	Recovery, %	Metal, Kt

Product name	Yield		Grade %		Recovery, %		Metal, Kt	
	Kt	%	**Cu**	**Mo**	**Cu**	**Mo**	**Cu**	**Mo**
Cu Concentrate	129.90	0.65	26.500	0.035	78.23	0.69	34.42	0.05
Mo Concentrate	11.19	0.06	0.400	49.000	0.10	83.06	0.04	5.48
Sum of the concentrates	**141.09**	**0.71**	**24.430**	**3.918**	**78.34**	**83.75**	**34.47**	**5.53**
Tailings	19,858	99.29	0.048	0.005	21.66	16.25	9.53	1.07
Feed ore	**20,000**	**100**	**0.220**	**0.033**	**100.00**	**100.00**	**44.00**	**6.60**

Table 17-4: ***Kajaran Floatation Process (20M Tonnes Annual)***

#	Operation	Pulp volume, m^3/hour	float per min, t	Equipment	Qty.
Collective Cycle					
Sections I-V of plant for 12.5Mtpa capacity					
1	Contacting	4575.9	***3***	**KCH 40**	**5**
2	Main collective floatation	4575.9	***15***	**RIF 25**	**55**
3	Control collective floatation	4201.7	***19***	**RIF 25**	**60**
Section for 4Mtpa instead of cleaning, 16+4=20					
4	Contacting	1465.0	***4***	**KCH 40**	**2**
5	Main collective floatation	1465.0	***15***	**RIF 25**	**16**
6	Control collective floatation	1346.0	***19***	**RIF 25**	**18**
7					
Collective cleaning for 21Mt					
8	1 cleaning of collective concentrate	1080.6	***9***	**RIF 25**	**8**
9	2 cleaning of collective concentrate	327.5	***17***	**RIF 25**	***4***
10	3 cleaning of collective concentrate	144.7	***22***	**RIF 25**	**2**
11	1 re-floating of tailings	808.4	***8***	**RIF 25**	**6**
12	2-3 re-floating of tailings	536.1	***21***	**RIF 25**	**8**
Moly cycle					
Section I					
13	Steaming	144.8	***5***	**KCH 25**	**1**
14	Main Mo floatation	144.8	***15***	**RIF 8.5**	**4**
15	Control Mo floatation	117.0	***20***	**RIF 8.5**	**5**
16					
Section II (reserve)					
17	Main Mo floatation	144.8	***15***	**RIF 8.5**	**4**
18	Control Mo floatation	117.0	***20***	**RIF 8.5**	**5**
Section I for Mo cleaning					

#	Operation	Pulp volume, m^3/hour	float per min, t	Equipment	Qty.
19	1 cleaning of Mo concentrate	48.8	***13***	**RIF 3.5**	**3**
20	Re-floatation of tailings	36.9	***18***	**RIF 3.5**	**3**
21	2 cleaning of Mo concentrate	19.9	***12***	**RIF 1.5**	**3**
22	3 cleaning of Mo concentrate	12.8	***14***	**RIF 1.5**	**2**
23	4 cleaning of Mo concentrate	8.2	***17***	**RIF 1.5**	**2**
24	5 cleaning of Mo concentrate	5.9	***23***	**RIF 1.5**	**2**
25	6 cleaning of Mo concentrate	4.4	***21***	**RIF 1.5**	**2**
26	7 cleaning of Mo concentrate	2.8	***33***	**RIF 1.5**	**2**
Section II for Mo cleaning (reserve)					
27	1 cleaning of Mo concentrate	48.8	***13***	**RIF 3.5**	**3**
28	Re-floatation of tailings	36.9	***18***	**RIF 3.5**	**3**
29	2 cleaning of Mo concentrate	19.9	***12***	**RIF 1.5**	**3**
30	3 cleaning of Mo concentrate	12.8	***14***	**RIF 1.5**	**2**
31	4 cleaning of Mo concentrate	8.2	***17***	**RIF 1.5**	**2**
32	5 cleaning of Mo concentrate	5.9	***23***	**RIF 1.5**	**2**
33	6 cleaning of Mo concentrate	4.4	***21***	**RIF 1.5**	**2**
34	7 cleaning of Mo concentrate	2.8	***33***	**RIF 1.5**	**2**
Copper cycle					
Sand floatation					
Section I					
35	Aeration	210.1	***4***	**KCH 25**	**0**
36	Main sand floatation	210.1	***15***	**RIF 25**	**2**
37	Cleaning	103.8	***15***	**RIF 25**	**1**
38	Control sand floatation	109.4	***20***	**RIF 25**	**2**
Section II					
39	Aeration	210.1	***5***	**KCH 25**	**2**
40	Main sand floatation	210.1	***15***	**RIF 25**	**2**
41	Cleaning of concentrate of main sand floatation	103.8	***15***	**RIF 25**	**1**
42	Control sand floatation	109.4	***20***	**RIF 25**	**1**
Slurry floatation					
43	Aeration	204.4	***5***	**KCH 25**	**1**
44	Aeration	204.4	***5***	**KCH 25**	**1**
45	Main slurry floatation	204.4	***16***	**RIF 25**	**3**
46	1 cleaning of concentrate of main slurry floatation	157.4	***12***	**RIF 25**	**1**
47	2 cleaning of concentrate of main slurry floatation	85.5	***15***	**RIF 25**	**1**

#	Operation	Pulp volume, m^3/hour	float per min, t	Equipment	Qty.
48	Control slurry floatation	138.5	***25***	**RIF 25**	**4**

18.0 PROJECT INFRASTRUCTURE

18.1 Transportation/Roads

There is no rail connection to the project area and all goods entering the mine and concentrates leaving the mine are transported by the M2 highway, which passes close to the mine.

18.2 Power Supply

The Kajaran project is located in a well-developed region of Armenia, where the power supply is secured through the national electricity network.

Electrical power to the Kajaran area is supplied by two 110kV high voltage lines (“Zangezur” and “David-Bek”). These two lines feed ZCMC's two substations (N1 110/35/6 kV and N2 110/10 kV). Both ZCMC substations were built in 2010 by AREVA and are fully equipped with modern equipment.

The Armenia-Iran high-voltage power line passes nearby, and a second line is planned for constructed.

18.3 Water Supply

The process water supply is currently obtained from the Voghji and the Geghi Rivers.

A weir with an abstraction capacity of approximately 360m^3/h is installed in the Voghji River, upstream of the processing plant. Water from the Voghji River flows without pumping, through a tunnel, and is stored in a reservoir located higher up in the valley above the mill.

Abstraction from the Voghji River is year round, with additional supply coming from the Geghi River during the dry season (September to March).

18.4 Waste Dumps

There are three waste rock dumps at Kajaran Project. The “North-West” dump was the first established, and low grade oxide ore and barren rock (monzonites and granodiorite porphyry dykes) were disposed here between 1951 and 1970.

The Spitak Djur dump is located on the South-Eastern edge of the mine in the Spitak Djur River valley. Its operation started in 1971.

The Dzorategh dump is located 1.5km to the East from Spitak Djur dump, and has been in operation from 1979 to today.

18.5 Tailings Management

Kajaran tailings are classified as silty sand with about 60% of the grain sizes between 0.05 to 2 mm. Since 1977 ZCMC has operated the Artsvanik tailing dam located about 30km E-NE of the mine. Tailings slurry is being transported 34.5 km by pipeline, 17.6 km of which are tunnels and 5.2 km are siphons.

At the present pond elevation of 898 m there is still 120 Mm^3 of space available for tailings to reach the design elevation of 964 m.

18.6 Local Resources

The existence of the Voghji River at the mine, and the Geghi River with its reservoir, ensures both industrial and drinking water for the project. Areas for the waste rock disposal and tailing are deemed to be large enough for the project. Kajaran is located in close proximity (30 km) to Kapan, the administrative center of the Syunik region. Historically, Kapan has a high level of education with regional branches of a number universities and professional educational institutions. In addition, two other large mining companies are located in this region, and people are traditionally comfortable with work in the mining industry.

18.7 Buildings/Offices

The Kajaran mining and processing complex owned by ZCMC was built based on historic reserves filed in 1951 occupying a 50 km^2 area. This complex includes mining, crushing and transportation, and processing facilities. Each of these facilities has his own Administration building, which includes offices for administrative staff, dining, change rooms and bathrooms for employees. There are mechanical repair shops for machinery and equipment such as drilling rigs, loaders and excavators, dump trucks and bulldozers. There are also heavy machine shops where larger mining, crushing and processing equipment is maintained.

19.0 MARKET STUDIES AND CONTRACTS

This section is not applicable for this report.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This section is not applicable for this report.

21.0 CAPITAL AND OPERATING COSTS

This section is not applicable for this report.

22.0 ECONOMIC ANALYSIS

This section is not applicable for this report.

23.0 ADJACENT PROPERTIES

There are no properties adjacent to the Kajaran project having a material effect on this resource estimate.

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no known other relevant data and information on the Kajaran project having a material effect on this resource estimate.

25.0 INTERPRETATION AND CONCLUSIONS

An NI 43-101 Mineral Resource estimate has been successfully carried out on ZCMC's Copper-Molybdenum deposit at Kajaran Armenia. Golder is satisfied that the data has been confirmed to a level that is sufficient for a Mineral Resource estimate, through a comprehensive program of half-core and pulp reject assays, as well as reconciliation to mill production for the last 5 years.

The Mineral Resource is based on reasonable economic parameters, including reliable production costs from historical data, and the application of Whittle open pit optimization.

The resources stated are based on a Whittle analysis covering the entire deposit. Current mine plans for the entire Zangazur deposit are constrained by current permitting, and recovery of the 1.017 billion tonnes of Indicated Resources north of the Voghji River relies on required permitting. Geological information north of the river is also not as detailed, and will need additional drilling to increase confidence in all economic minerals in the deposit, as well as density.

SG determinations are only available for 76 samples, and although widely scattered, with a low variation in values, almost all in areas south of the Voghji River. Also, none of the 76 SG samples represent cross-cutting dyke material. It has been assumed that the dyke SG will not be dissimilar to the mineralized and un-mineralized porphyries, but confirmation of this is required.

Check assays of pulp rejects show the possibility of a minor negative bias in the historic Mo grades.

Check assays from 2,165 pulp rejects showed, through global average comparison to the composite Au assays used in the resource estimate, that there is the probability that Au grades in the resource are being under-reported. This is supported by the fact that 2014 monthly composite Au assay (12 month period) average at the mill agrees quite favourably to that of the pulp assays. The average of the pulps Ag grades would indicate a possible slight over-reporting in the Ag resource grades, but this is not supported as well by mill grades since none were available.

The pulp assays also suggest a potential correlation between Cu and Au (30%), as well as Cu and Ag (54%) These correlations increase substantially with the removal of a few outliers in the data (62% for Au, 60% for Ag). This raises the potential that with enough data and a stronger correlation, Au content could be applied to the resource model (and planning) through its association with Cu.

The pulp analysis also shows a high correlation (89%) between Mo and Rhenium (Re), which is recovered from the Mo concentrate. This correlation is enough to include Re in the resource block model using a regression formula, but additional assay data to confirm this correlation is suggested.

A CuEq attribute has been implemented to include Cu, Mo, Au, and Ag in the cut-off determination, for more robust economic evaluations and mine planning. With Re contributing value to the ore, adding it to the CuEq calculation would make economic evaluation of the deposit even more robust.

26.0 RECOMMENDATIONS

Based on findings during the course of this resource estimate and conclusions listed above, Golder recommends the following actions to improve the Kajaran project:

- Diamond drilling north of the Voghji River to increase confidence, possibly increasing some resources to Measured. A suggested initial phase would concentrate in areas of higher grade indicated in the block model:
  - 40 holes to a depth of 500 m, at a cost of US$ 200/m (drilling, handling, assaying), for a total of 20,000 m and a cost of US$4.0 million.
  - Drilling south of the Voghji River is also recommended potentially convert Indicated Resources to Measured Resources, and explore below the limits of existing drilling. Amount and costs would be determined at that time.
- Additional SG sampling, particularly north of the river and in dyke material, through representative grab sampling and drilling;
- Additional Au and Ag assay data from same remaining pulp rejects, production holes, new diamond drill holes; also daily mill feed and tailings Au and Ag assays to more accurately calculate and predict recoveries; a higher correlation (+85%) will result in more accurate Au and Ag resource estimates without the need for extensive drilling and assaying.
- Microscopic and high resolution material analysis, such as QEMSCAN (Quantitative Evaluation of Minerals by SCANning electron microscopy) to further understand relationships between Au and Ag to Cu, as well as other relationships that may arise; should be done on concentrate, tailings, drill core and/or grab samples.
- Obtain additional pulp reject assays to confirm the Re – Mo correlation, and include Re in the CuEq calculation and future resource estimates.
- Assay as many remaining pulp rejects as possible to further investigate the possibility of a negative bias on historic Mo grades, and provide support for adjustments to the historic grades if required.

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28.0 DATE AND SIGNATURE PAGE

This report titled “Technical Report and October 2015 Mineral Resource Estimate, The Kajaran Copper-Molybdenum Mine, Kajaran, Armenia” was prepared and signed by the following authors. The signatures and the effective date of the Technical Report is February 22, 2016. The effective date of the Mineral Resource estimate in the Technical Report is October 1, 2015.

Signed at Mississauga, Ontario, Canada, February 22, 2016

Greg Greenough, BSc. Hons., P.Geo.

Signed at Kajaran, Armenia, February 22, 2016

Ashot Margaryan, Ph.D.
Technical Director

29.0 CERTIFICATES OF QUALIFIED PERSONS

GREG GREENOUGH

I, Greg Greenough, P.Geo., as an author of this report titled Technical Report and October 2015 Mineral Resource Estimate, The Kajaran Copper-Molybdenum Mine, Kajaran, Armenia, prepared for Zangezur Copper-Molybdenum Combine (ZCMC), and dated February 22, 2016, do certify that:

1. I am a Senior Resource Geologist with Golder Associates Ltd., 6925 Century Avenue, Mississauga, Ontario, Canada L5N 7K2.
2. I graduated in 1976 with a Bachelor's degree (Hons) in Geology from Laurentian University in Sudbury, Ontario.
3. I am a practicing member of the Association of Professional Geoscientists of Ontario (#0825).
4. I have worked as a geologist in the mineral resource industry for a total of forty years since my graduation from university. My relevant experience for the purpose of the Technical Report includes Mineral Resource estimation and Technical Report authoring of numerous mining and exploration projects, including massive sulphides, hydrothermal gold, VMS, graphite, and iron
5. I am responsible for the Mineral Resource estimate, all sections of the report except Section 13, and the overall preparation of the report Technical Report and October 2015 Mineral Resource Estimate, The Kajaran Copper-Molybdenum Mine, Kajaran, Armenia, dated February 22, 2016 (the "Technical Report").
6. I visited the property June 29 to July 3, 2015.
7. The effective date of the Mineral Resource estimate is October 1, 2015, and to that date I am not aware of any material fact or change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
8. I am independent of the issuer applying all the tests in section 1.4 of the National Instrument (NI) 43-101, and have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements of a Qualified Person for the purposes of NI 43-101.

Signed this 22nd day of February, 2016 at Mississauga, Ontario, Canada

Greg Greenough, P.Geo.

Senior Resource Geologist

ASHOT MARGARYAN

I, Ashot Margaryan, Ph.D., as a person who is responsible for reliability of provided information for report titled Technical Report and October 2015 Mineral Resource Estimate, The Kajaran Copper-Molybdenum Mine, Kajaran, Armenia, prepared for Zangezur Copper-Molybdenum Combine (ZCMC), and dated February 22, 2016, do certify that:

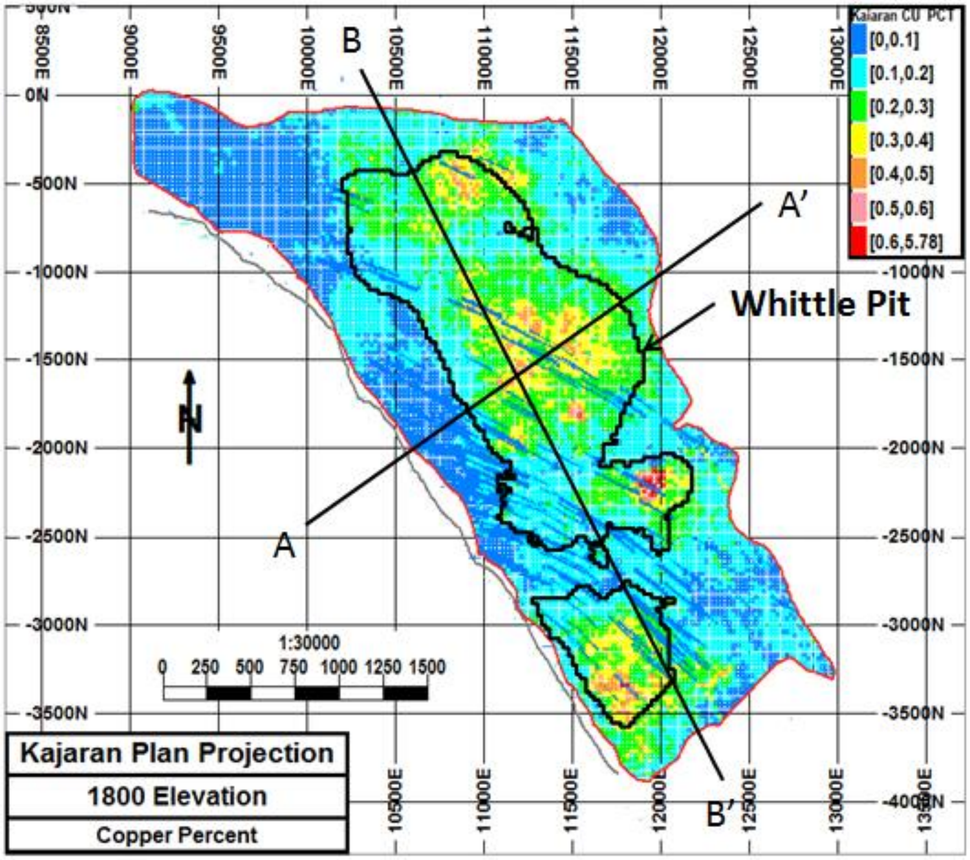
1. I am a Technical Director of ZCMC CJSC, Lernagorcneri str. 18, Syunik Marz, 3309 Kajaran Armenia.
2. I graduated from Donetsk National Technical University (formerly Donetsk Polytechnic Institute), Ukraine in 1980 as Mining Engineer-Surveyor.
3. I defended a Ph.D. thesis in Moscow Mining University in 1992.
4. I work as Technical Director of ZCMC since 2001. My relevant experience for the purpose of the Technical Report includes Geology, Open Pit and Underground Mining, Power and Mechanical Engineering, Haulage, Mineral Processing and Operation of Hydro-technical Facilities.
5. I am responsible for Sections 13 and 17 of the report Technical Report and October 2015 Mineral Resource Estimate, The Kajaran Copper-Molybdenum Mine, Kajaran, Armenia, dated February 22, 2016 (the "Technical Report").
6. The effective date of the Mineral Resource estimate is October 1, 2015, and to that date I am not aware of any material fact or change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

Signed this 22nd day of February, 2016, at Kajaran, Armenia

Ashot Margaryan, Ph.D.
Technical Director

APPENDIX A

Model Plots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Whittle Pit
A
A'
B
B'
N
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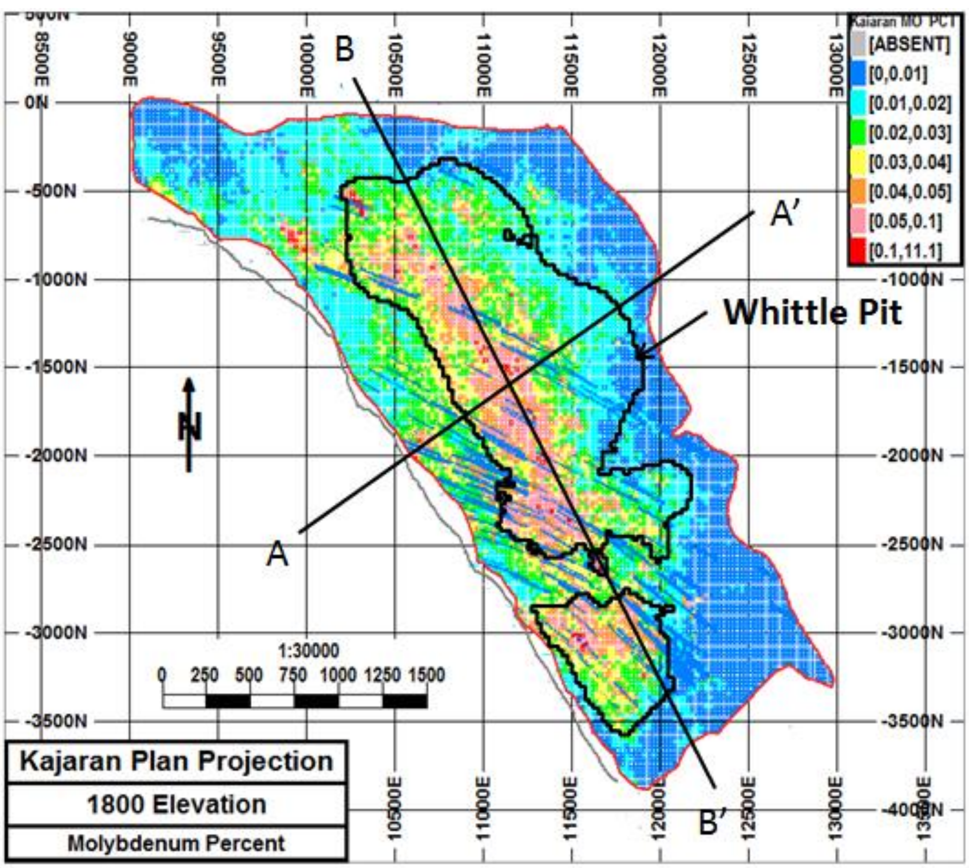
Kajaran Plan Projection
1800 Elevation
Copper Percent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Whittle Pit
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A'
B
B'
N
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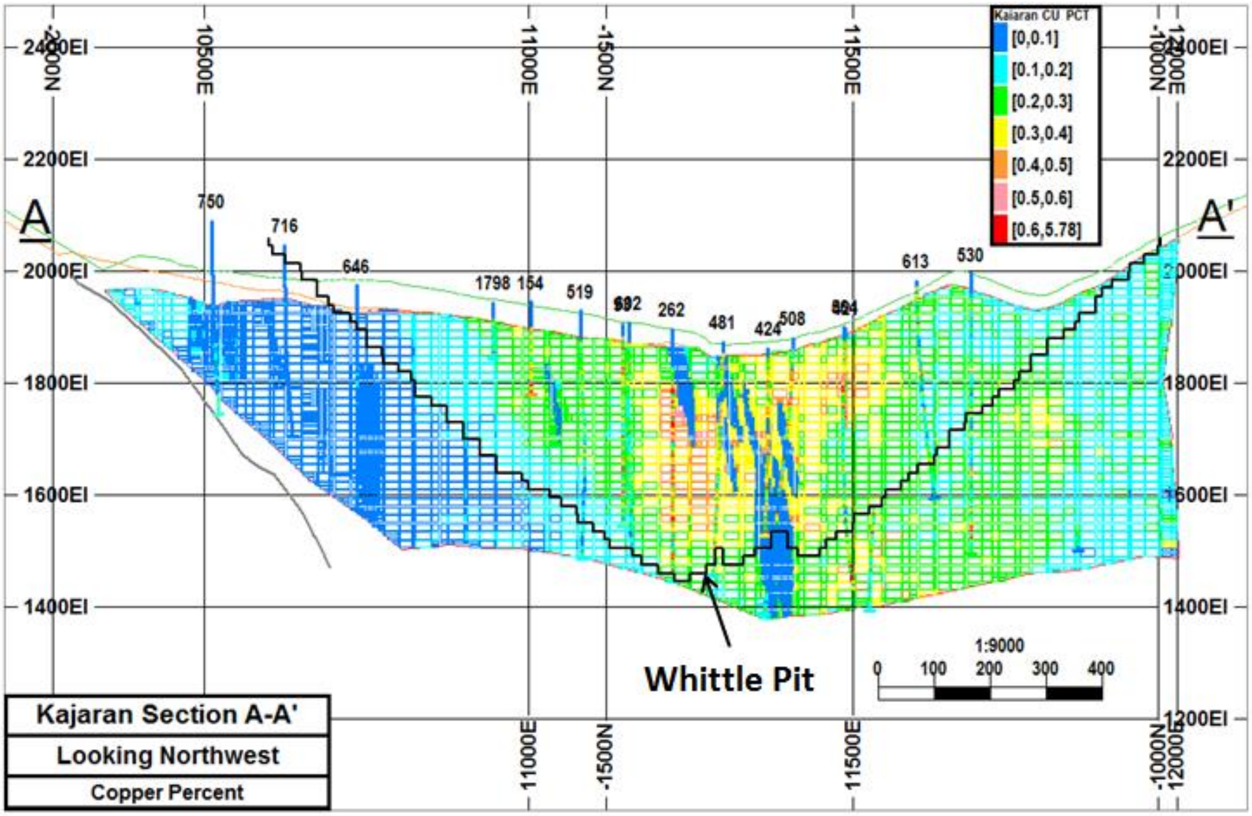
Kajaran Plan Projection
1800 Elevation
Molybdenum Percent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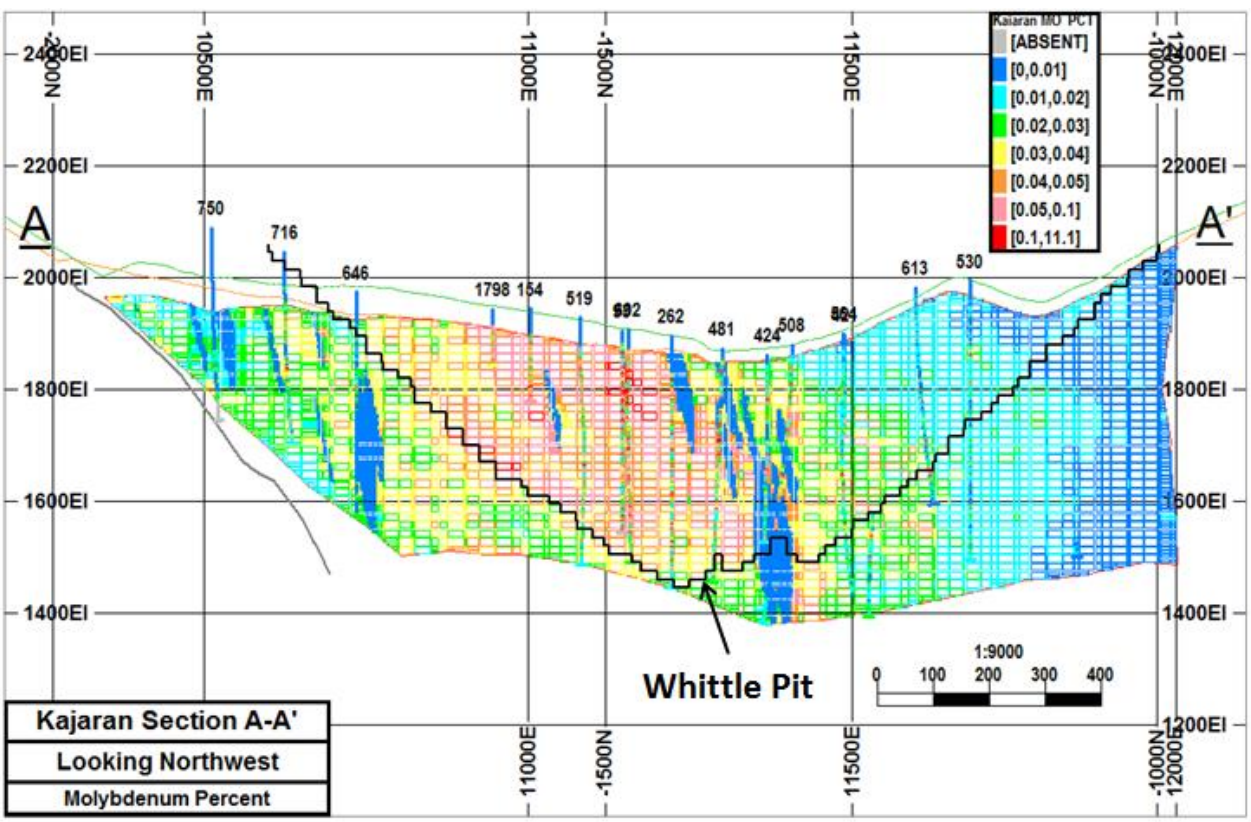
A
A'
Whittle Pit
1:9000
Kajaran Section A-A'
Looking Northwest
Copper Percent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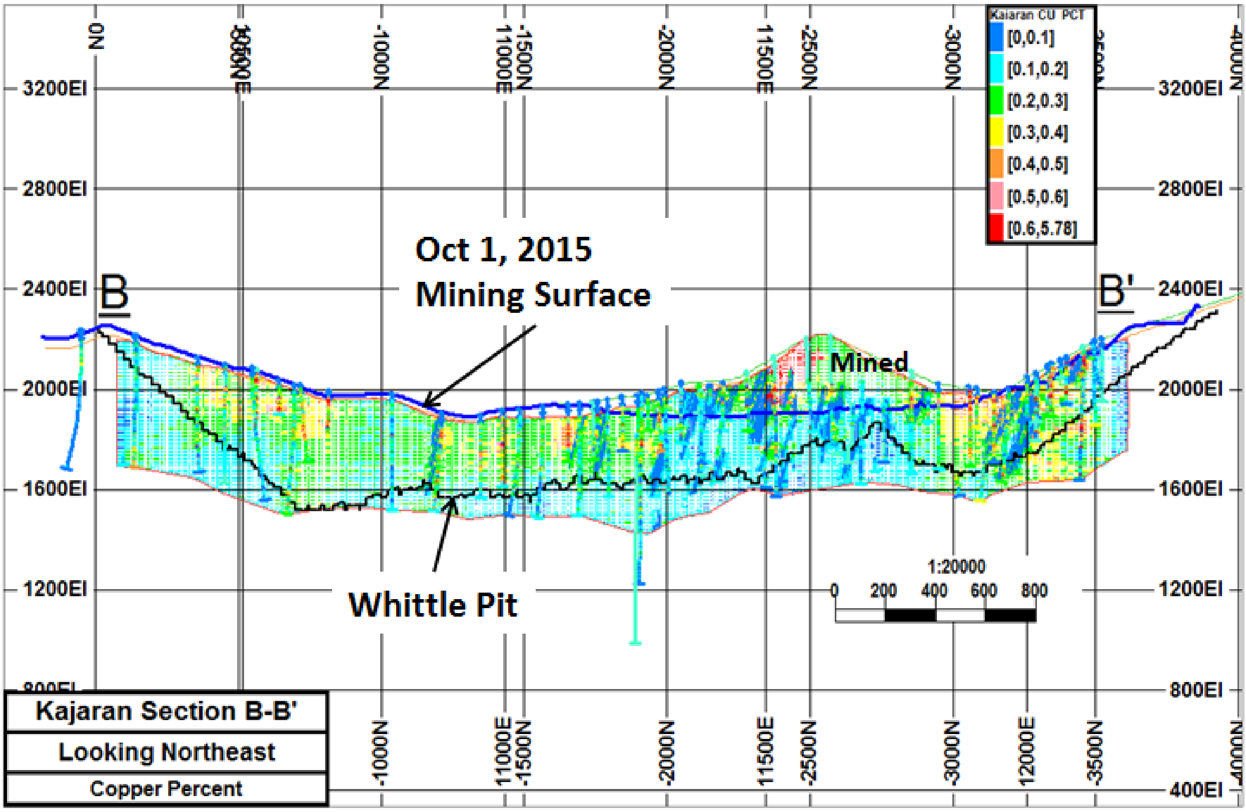
A
A'
Whittle Pit
1:9000
Kajaran Section A-A'
Looking Northwest
Molybdenum Percent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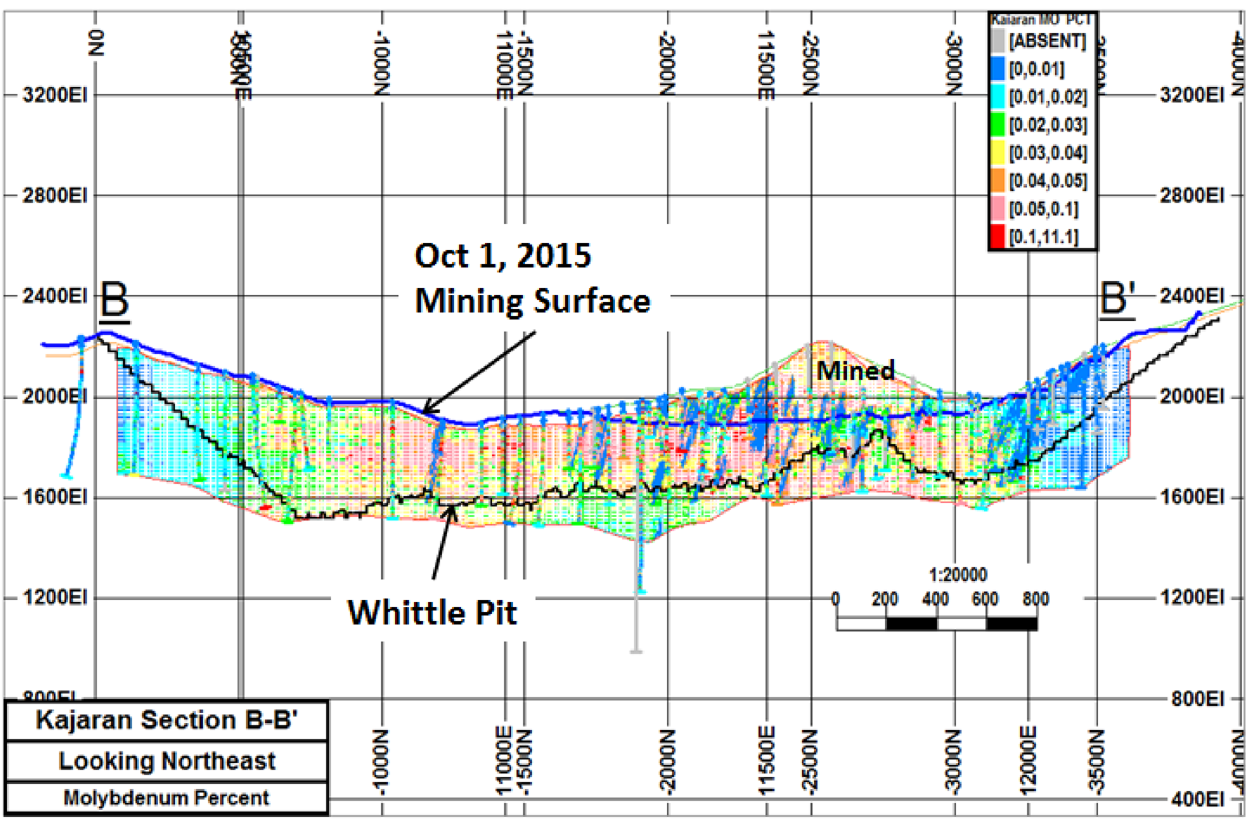
Oct 1, 2015
Mining Surface
Mined
Whittle Pit
B
B'
1:20000
0 200 400 600 800
Kajaran Section B-B'
Looking Northeast
Copper Percent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[0.1,11.1]
Oct 1, 2015
Mining Surface
Mined
Whittle Pit
B
B'
1:20000
0 200 400 600 800
Kajaran Section B-B'
Looking Northeast
Molybdenum Percent

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