

**AMC Mining Consultants (Canada) Ltd**

BC0767129

Suite 1330, 200 Granville Street  
Vancouver BC V6C 1S4  
CANADA

**T** +1 604 669 0044  
**F** +1 604 669 1120  
**E** amccanada@amcconsultants.ca



**NI 43-101 TECHNICAL REPORT ON THE  
GC AG-ZN-PB PROJECT IN  
GUANGDONG PROVINCE  
PEOPLE'S REPUBLIC OF CHINA**

**SILVERCORP METALS INC**

Report Prepared by

**Brian O'Connor P.Geo**

**Peter Mokos MAusIMM (CP)**

**Alan Riles MAIG**

**Owen Watson MAusIMM (CP)**

**Mo Molavi P.Eng**

**Patrick Stephenson P.Geo**

Report with Appendices for Silvercorp Internal Purposes

**AMC File 711009**

**Effective Date: 23 January 2012**

ADELAIDE  
+61 8 8201 1800

BRISBANE  
+61 7 3839 0099

MELBOURNE  
+61 3 8601 3300

PERTH  
+61 8 6330 1100

TORONTO  
+1 416 640 1212

VANCOUVER  
+1 604 669 0044

MAIDENHEAD  
+44 1628 778 256

## 1 SUMMARY

### Introduction

AMC Mining Consultants (Canada) Ltd (AMC) was commissioned by Silvercorp Metals Inc. (Silvercorp) to review a report titled "Mining and Dressing Project of Gaocheng Lead-Zinc Ore in Yun'an County, Guangdong Province" prepared by the Guangdong Metallurgical & Architectural Design Institute (GMADI) in January 2011, and to prepare an independent Technical Report on the property incorporating its findings on the GMADI report. AMC prepared a previous Technical Report on the Gaocheng (GC) property in June 2009 titled "NI 43-101 Technical Report Update on the GC Ag-Zn-Pb Project in Guangdong Province, People's Republic of China" (AMC report number 709003).

The six authors of this report all qualify as independent Qualified Persons. Four visited the GC property in May 2011 (P Mokos revisited in July-August 2011) and undertook various validation tasks. For its 2009 report, AMC also undertook detailed validation work, including taking quarter split core samples from 34 diamond drillholes and having them check assayed at the Eco Tech Laboratory in Kamloops, British Columbia.

Yangtze Mining Ltd (Yangtze Mining), which is wholly owned by Yangtze Gold Ltd (Yangtze Gold), acquired the GC property in 2005 through a 95% interest in a Sino-Foreign joint venture company, Anhui Yangtze Mining Co. Ltd (Anhui Yangtze). Silvercorp acquired 100% of the shares in Yangtze Gold in 2008. An Environmental Permit and 30-year Mining License were received in 2010 and Silvercorp is currently in the process of constructing the mine. Initial production of 700 tonnes per day mining capacity is expected to be achieved in 12 months with full capacity of 1,600 tonnes per day to be achieved in 18 months.

The GC project is located around Gaocheng Village of Gaochun Township, Yun'an County, Guangdong Province, China. It is west of the city of Guangzhou, the capital of Guangdong Province. Access to the project from Guangzhou is via 178 km of four lane express highway to Yunfu, then 48 km of paved road to the project site. A railway connection from Guangzhou to Yunfu is also available. The region has a sub-tropical, monsoon climate and the project would be able to operate year round.

### Geology, Exploration and Mineral Resources

Geologically, the deposit occurs at the intersection of two major fault zones within sedimentary clastic and carbonate rocks. There are three predominant sets of structures in the region, all of which host some mineralization, but the lead-zinc-silver-(tin) deposits are contained mainly within the third structure set, which comprise arc or ring structures surrounding a granite zone.

GC mineralization belongs to the mesothermal, vein-infill style of deposit and occurs as veins which are structurally controlled within broader alteration zones. The veins have a sharp contact with host rocks and dip at angles of between 60° and 85°. They have true widths varying from 0.1 m to over 10 m and have been traced for over 1,200 m along strike and over 500 m down dip. Mineralization is 95% primary and is mainly composed of galena-sphalerite-silver minerals which occur sparsely, disseminated, and as veinlets and lumps. Pyrite typically comprises a few percent to 13% of the vein.

There are 25 individual veins included in the current mineral resource estimate.

Various state-sponsored Chinese Geological Brigades and companies conducted geological and exploration work in the project area between 1959 and the 2000s. Illegal mining activity during this period resulted in the excavation of several tunnels and small scale extraction of ore. In 2002, the Guangdong Provincial Institute of Geological Survey (GIGS) developed 66 m of tunnels to crosscut certain veins, and sampled and mapped a number of adits. During 2006 and 2007, GIGS, under contract to Yangtze Mining, completed a 36-hole, 11,470m surface diamond drilling program. During 2008, Silvercorp undertook a major soil geochemical program and completed a 22 hole, 10,083 m drilling program, which resulted in the discovery of an additional 15 mineralized veins.

A total of 65 diamond drill holes have now been completed on the GC property since 2001, totalling 23,546 m. Core is stored in a dedicated core shack in the town of Gaocun, which is locked when not in use with two guards on patrol. Core recoveries range from acceptable to excellent, averaging greater than 95%.

Drill core is logged initially at the drill site and the intervals of economic interest are moved to the core shack where they are logged, photographed and split by diamond saw. The split samples are individually secured in sample bags and then collectively secured in rice bags for shipment to ALS Chemex in Guangzhou (an ISO 9001:2000 accredited laboratory), located approximately 180 km southeast of the GC property site. Sample preparation consists of drying, crushing and splitting with a riffle splitter to 150 g, then pulverizing to 200 mesh. Silver, lead and zinc are analyzed by aqua regia digestion and AAS. Tin is analysed by peroxide fusion with an AAS finish.

Check samples including field duplicates and pulps are routinely sent to the Laboratory of the Henan Institute of Geological Survey (the "Henan Laboratory"), located in Zhengzhou, Henan Province, Central China.

AMC is satisfied that drilling, sampling, sample security, sub-sampling, analysing and QA / QC procedures meet accepted industry standards, that the QA / QC results show no material bias or imprecision and that the assay results may be relied upon for mineral resource estimation.

Current mineral resource estimates have been prepared by W Qiang, Chief Geologist of Yangtze Mining, and M Gao, P.Geo, President & Chief Operating Officer of Silvercorp, who is a Qualified Person, as defined by NI 43-101. B O'Connor of AMC, P.Geo, who qualifies as an independent Qualified Person, has reviewed the estimation procedures and results and takes responsibility for the estimates.

A polygonal method has been used for resource estimation (a common approach in China), based on detailed long-sections constructed for each of the veins. The minimum cut-off thickness used is 0.20 m, although only around 5% of resource blocks have a thickness between 0.2m and 0.3m, the mineral reserve minimum mining width and Silvercorp has experience at mining to such widths. To achieve the minimum mining width, the rescue mining method, a highly selective extraction method, is planned to be employed to mine veins with thicknesses less than 0.8 m. Silvercorp uses a "recovered equivalent-silver" value (AgEq Recovered), which takes into account metallurgical recoveries, to assess and compare the vein resources. The formula and parameters are shown in the text of this report and are the same for mineral resources and mineral reserves.

The lower cut-off grade is 100 g/t AgEq Recovered. A top-cut has been applied to silver, zinc, and lead assays. Dilution at zero grade has been applied to 11 individual resource blocks less than 0.20 m in horizontal width. Resource classification is based primarily on the type of samples and distance beyond underground vein exposure or drillhole samples.

The independent QP is satisfied that the resource estimates comply with reasonable industry practice, subject to a qualification with respect to use of the polygonal method. Although this is a common estimation method in China, the technique tends to produce estimates that are higher in grade and lower in tonnage than methods in common use in Canada, such as kriging. However, the independent QP does not consider that any differences from a likely block model estimate would be material to the project.

**Table 1 Summary of Mineral Resources**

Resource Classification	Tonnes	Grade			Contained Metal		
		Ag (g/t)	Pb %	Zn %	Ag (kg)	Pb (t)	Zn (t)
Measured	592,800	230	1.41	3.33	136,600	8,400	19,800
Indicated	7,038,700	113	1.31	3.06	797,700	92,500	215,500
<b>Total</b>	<b>7,631,500</b>	<b>122</b>	<b>1.32</b>	<b>3.08</b>	<b>934,300</b>	<b>100,900</b>	<b>235,300</b>
Inferred	7,959,800	123	1.41	2.66	976,800	112,500	211,900

Metal prices used: silver US\$18.00/troy oz, lead US\$1.00/lb, zinc US\$1.00/lb

Inclusive of resources converted to mineral reserves

Lower cut-off grade, 100 g/t AgEq Recovered

Rounding of some figures may lead to minor discrepancies in some totals

## Mining and Mineral Reserves

Mineral reserve estimates have been prepared by P Mokos, MAusIMM (CP) of AMC, who qualifies as an Independent QP, and are based on a mine design prepared by Guangdong Metallurgical & Architectural Design Institute (GMADI) as part of its January 2011 study (modified by AMC where required). Overall, the study work undertaken to date meets preliminary feasibility study levels.

The reserves encompass 21 veins that are categorized as Measured and Indicated Resources and are based on the highly selective extraction methods of shrinkage stoping and rescue stoping. The shrinkage method uses the blasted ore as the working platform for each stope lift. The ore is removed on completion of stope mining leaving an empty void. The rescue method uses blasted waste from the footwall (to achieve the minimum mining width) as the working platform for each stope lift. The waste remains in the stope at completion of stope mining. Mine access will be by decline (ramp) and shaft.

The mineral reserve has been estimated using a 135 g/t AgEq Recovered cut-off grade, which is approximately equivalent to the estimated operating breakeven. Those mineral resources that have been excluded from conversion to reserves are above +100 mRL (to maintain a surface crown pillar), below -300 mRL (being the extent of the Silvercorp mine design to date), and within certain planned pillars.

Average mining dilution and recovery for shrinkage stoping are 10.3% and 90% respectively, while average mining dilution and recovery for resue stoping are 25% and 88.1% respectively AMC considers the dilution estimates to be reasonable and commensurate with the stoping methods. Dilution control will be assisted by selective hand sorting of waste from ore, a practice also conducted at Silvercorp's Ying Mine. AMC considers that mining recoveries may be less than predicted in certain situations, but that there is likely to be less than a 3% impact on the stoping tonnage.

**Table 2 Summary of Mineral Reserves**

Reserve Classification	Tonnes	Grade			Contained Metal		
		Ag (g/t)	Pb %	Zn %	Ag (kg)	Pb (t)	Zn (t)
Proven	464,000	199	1.12	3.18	92,300	5,200	14,800
Probable	4,285,700	113	1.33	2.93	482,400	56,900	125,500
<b>Total</b>	<b>4,749,700</b>	<b>121</b>	<b>1.31</b>	<b>2.95</b>	<b>574,700</b>	<b>62,100</b>	<b>140,200</b>

Metal prices used: silver US\$18.00/troy oz, lead US\$1.00/lb, zinc US\$1.00/lb  
 Lower cut-off grade, 135 g/t AgEq Recovered  
 Based on studies that, overall, are to a standard of a preliminary feasibility study  
 Rounding of some figures may lead to minor discrepancies in some totals

The mining component of the project will be developed in two stages. Stage 1 targets fast-tracking the project into production and is developed by mobile rubber-tired, diesel-powered equipment (development jumbo, loader, truck) with surface decline access down to -50 mRL. Stage 2 is developed using conventional tracked equipment (electric locomotive, rail cars, electric rocker shovels, pneumatic hand held drills) with shaft access from -50mRL to -300 mRL.

The life-of-mine (LOM) production duration will be 12 years. The average production rate will be 496,000 tonnes per annum (tpa) of ore from 2013 to 2021 inclusive. The steady state mine production rate will be 518,000 tpa of ore from 2016 to 2021 inclusive.

The total LOM waste produced will be 2.3 Mt with 0.9 Mt produced from Stage 1. The +215 mRL waste dump tip head capacity accommodates the initial two years of waste produced. AMC was not provided with details for the final waste dump capacity, but its site visit and review of the surface plans indicate there appears to be room to accommodate all waste produced. Waste could also be disposed of into shrinkage stope voids although this is not in the current mine plan.

Primary mine ventilation will generally flow from west to east using the main levels interconnected by dedicated level vent raises (plus active stope accesses). The upper level(s) where stoping has been completed will be used as return airways to separate the fresh and exhaust air. A series of air doors and sealed walls will be utilized in the ventilation system. Secondary ventilation will consist of auxiliary fans for ventilating development faces, infrastructure chambers, loading and tipping areas and stope faces.

AMC considers that the geotechnical aspects of the GMADI mine design are generally reasonable for mining study purposes. However, given the limited nature of the data, the geotechnical knowledge at the project is not considered to be at the level of detail normally associated with a mining feasibility study in Canada, and is more in line with a scoping level

study. Recommendations for further work include: collection of additional detailed geotechnical logging data from drill core and from mapping of underground workings, development of a three dimensional geological model with interpretations of primary lithologies and structures (such as faults and shear zones), geotechnical investigations of proposed shaft locations to determine site suitability and ground support requirements, geotechnical investigations of the surface crown pillar, particularly in the vicinity of the Hashui Creek (also referred to as Hashui River in some documents) valley, further hydrogeological assessments, and further investigation of in situ stresses to confirm assumptions made in the mine design and stability assessments.

### **Metallurgical Testwork and Processing**

Metallurgical testing for the GC project was carried out in 2007 and in 2009. Results are summarized in the GMADI January 2011 report. AMC is satisfied that the testwork samples are adequately representative of the main part of the deposit and of the mineral reserves. Although no grinding testwork was carried out as part of these studies, AMC has made adequate allowance in the process design section.

The overall process consists of crushing, grinding, flotation of lead, zinc and pyrite concentrates, and concentrate dewatering, with the option of a tin recovery gravity separation circuit on pyrite flotation tails. Two-stage crushing, with the second stage in closed circuit, from run of mine ore at 350 mm produces a -10 mm crushed product. This is followed by two-stage grinding in ball mills to a product size of 80% passing 75 µm.

The flotation process consists of a standard sequential flotation of lead, zinc and pyrite with three-stage cleaning of the lead and zinc concentrates and single stage cleaning for pyrite. Concentrates are dewatered by conventional thickening and filtration. The optional tin recovery circuit comprises spiral concentration followed by coarse and fine shaking tables with a final stage of flotation to remove residual sulphides.

The process design assumes 1,000 tpd feed base case (likely expansion to 1,600 tpd), 330 days per year operation, crushing 18 hrs/day and grinding-flotation 24 hrs/day. No availabilities as such are cited; however the above parameters translate to 68% for crushing and 90% for grinding which AMC considers to be reasonable and in line with normal mining industry practice. In all sections of the plant design, space / capacity has been allocated for an expansion to 1,600 tpd.

Total installed power amounts to 5,043 kW and the estimate for actual power drawn is 3,657 kW. From AMC's analysis of the equipment sizing, the comminution circuit, especially grinding, is undersized for the 1,600 tpd throughput level. An additional 600 kW capacity in the grinding circuit ball mills is recommended to handle 1,600 tpd.

With the use of dry stacking of tailings there is minimal lock-up of water in tailings and a close to 90% recycle of water; however there is a requirement for fresh water for e.g. pump seals, cooling and reagent mixing. Detailed circuit water balances have been derived for the 1,000 tpd case and from this a net fresh water demand of 1,200 m<sup>3</sup>/d has been estimated. With the conservative assumption that the fresh water demand is proportional to the throughput then the demand at 1,600 tpd would be approximately 1,900 m<sup>3</sup>/d.

## **Infrastructure, including Tailings Dam**

The tailings management facility (TMF) proposed is for dry stacking and filling with concurrent rolling and compaction, in a location immediately to the south of the mine and concentrator in the Daken valley. Although AMC believes that the basic concept is reasonable, the work carried out to date towards the TMF design does not meet feasibility study standards. The main areas that require addressing are: tailings properties and their suitability for dry stacking, site geotechnical assessments, the Gaocheng River Class II water resource classification and implications for the TMF location and design, a more detailed site-specific TMF design, safety calculations, and further consideration of the impact of monsoonal rains. In addition, the two XA90/920 filters selected have been sized for 1,000 tpd ore feed and AMC considers that these two units are of inadequate size for the tonnage to be filtered. With the production schedule based on 1,600 tpd then, as a first approximation, AMC recommends that the filtration capacity be doubled.

Power will be provided from a 110 kv substation from the Guangdong Province electrical grid system near Gaocun, about 6 km from the mining area. This will be supplemented by a new 10kv substation to be built in the mining area. Two 1,500kw diesel generators are designated for emergency backup to the man-hoist, underground ventilation system and essential services in the plant. As mentioned above, AMC believes that an additional 600 kW capacity in the grinding circuit ball mills is required to handle a throughput of 1,600 tpd.

The final products from the process plant are metallic ore concentrates. The concentrate will be either packaged or shipped as bulk to the Yunfu Railway Station under escort.

Water for the mine will be sourced from local creeks and gullies that flow into the Hashui Creek. The water quality and quantity from the local creeks will be sufficient to meet the project requirements, which are estimated be 2,093 m<sup>3</sup>/day. Silvercorp has proposed a 1 km long diversion tunnel with two dams on the Hashui Creek to relocate the course of this creek beyond the projected subsidence zone of influence of the mine.

Silvercorp plans to operate the mine using contractors for development, production and the operation and maintenance of Silvercorp's fixed plant, with Silvercorp providing its own management, technical services and supervision staff to manage the mine operation. Estimates for the overall workforce indicate peak requirements of 624 people for Phase 1 and 679 people for Phase 2.

## **Market Studies and Contracts**

Initial concentrate sales contracts were put in place for the lead and pyrite concentrates with Jinan Wanyang Smelting (Group) Co., Ltd and for the zinc concentrate with Henan Yuguang Zinc Industry Co., Ltd. Silvercorp has advised AMC that all three contracts have been renewed with a three year term to 31 December 2014. Although AMC would have preferred to have seen the contracts as part of a life-of-mine frame agreement, it also understands that they should be viewed in the context of the existing operations and concentrate sales to these smelters and therefore does not view the three-year term of the contracts as a material issue.

There are some residual concerns / improvement opportunities regarding concentrate quality, with both copper and zinc levels being higher than ideal in the lead concentrates but these are commercial issues rather than material issues of concentrate marketability. With current silver

prices, there is also an opportunity to improve financial returns by lowering the concentrate lead grade to maximize payable silver recovery to the lead concentrate. Acceptable arsenic levels in Chinese base metal concentrates are generally of the order of 0.5% As and the GC lead and zinc concentrates are right at the limit of general acceptability. However AMC has been advised by Silvercorp that the renewed contracts referred to above specify arsenic penalty levels in the lead and zinc concentrates as 1.0% As which alleviates concerns about arsenic levels in these concentrates..

### **Environmental, Permitting, Social / Community Impact**

An Environmental Impact Assessment (EIA) report on the GC project was prepared by the Guangdong Environmental Technology Centre (GETC) and received by the Yunfu Environment Protection Bureau (Yunfu EPB) for comment. The Yunfu EPB states the mining area does not cover any natural conservation zones, ecological forests, and strict land control zones and gave consent to operate the GC project with the stipulation that the scope, site, processing technique, and environmental protection measures are followed as written in the report. An Environmental Permit was subsequently issued by the Department of Environmental Protection of Guangdong Province in June 2010.

The GC Mine will operate under, *inter alia*, 15 National and Provincial environmental laws and regulations, including the Law of Environmental Protection of the People's Republic of China (1989.12) and the Law of the People's Republic of China on Environmental Impact Assessment (2003.9). Both the mine waste rock dump and the TMF will be covered by soil and vegetated after completion of the operations.

A mine environmental protection department will be set up, with both full time and part time personnel. As part of mine site preparations, the Hashui Creek will be closed and diverted through a new 510 m water tunnel.

Overflow and excess water from the processing and mining operations will be reused as much as possible, with remaining water treated according to the Surface Water Quality Standards. The treated water is then stored in nearby reservoirs to be used as irrigation water for nearby woodland and farmland. That water needing to be discharged will be directed to the Hashui Creek and will be treated to remove heavy metals such as mercury, cadmium, chromium etc.

Silvercorp has received all the permits required to allow it to commence construction of the mine, including an Environmental Permit and a Mining Permit. The remaining permitting activities required to enable production to commence include completion of a review of the health and safety production measures by the Guangdong Provincial Safety Production Bureau (GPSPB), a safety measure inspection by the GPSPB to ensure that the construction of the mine, mill and tailing facility has followed the "Mine Design" in terms of safety measures, and an inspection of the TMF, mill, and other engineering works by the Guangdong Environmental Bureau.

Silvercorp has posted a \$200,000 bond to cover certain reclamation costs. Soil conservation and land reclamation costs are estimated as \$1.672M, while geological environment protection and rehabilitation costs are estimated as \$0.917M.

Based on the Chinese National requirements, a site decommissioning plan will be produced at least one year before mine closure. Site rehabilitation and closure cost estimates will be made in the site closure plan.

### Capital and Operating Costs

Total initial capital expenditure is estimated to be \$67.4M including mining, mill, infrastructure, owner's costs and contingency, as summarized in Table 3. The currency exchange rate used for the estimate is 1.00:6.35 (US\$: RMB).

**Table 3 Summary of Capital Costs**

	US\$000	Notes
Mining	18,490	
Mill and Infrastructure	29,619	
Owners Costs	11,655	
Contingency	7,589	
Total Initial Capital Expenditure	67,352	
Working Capital	5,714	based on 3 months yr 1 operating costs
Sustaining	25,321	
Total LoM Capital Cost	92,673	Working Capital netted to zero

Total operating cost for the project is estimated at \$40.6/t milled. The estimate includes mining, process, general and administration (G&A) and surface service costs, as summarized in Table 4.

**Table 4 Summary of Operating Costs**

	US\$/t milled
Mine	15.2
Mill	16.8
G&A	8.6
Environment (incl tailings)	
Total	40.6

### Economic Analysis

Metal prices assumed for AMC's economic analysis are summarized in Table 5.

**Table 5 Metal Prices used for Economic Analysis**

	2012	2013	2014	2015	After 2015
Silver (US\$/oz)	40.00	30.00	25.00	18.00	18.00
Lead (US\$/lb)	1.11	1.16	1.14	1.15	1.00
Zinc (US\$/lb)	1.05	1.12	1.11	1.15	1.00

The exchange rate has been set at the November 2011 value of USD:RMB of 6.35.

Based on these parameters and operating the mine as described in previous sections, the pre-tax Base Case economic model shows an NPV of \$73.7M using a discount factor of 8%. The IRR is 33% and the payback period is 2.4 years.

Key input variances have been analyzed for project sensitivity and the results synthesized into credible scenarios. Scenario probabilities based on industry experience have been assigned and a weighted average of key project financial return parameters calculated. This results in an NPV at 8% discount rate of \$52.95M, an IRR of 28% and a payback period of 3.3 years.

These probability-weighted average metrics are positive and demonstrate that the project is robust in the face of the possible scenarios that typically impact on a mining operation.

### **Main Recommendations**

(Stated costs are estimated for those recommendations not covered by operational activities).

- Undertake variography studies to refine the understanding of the grade distribution and utilize a kriging or inverse distance weighting approach to grade interpolation prior to future resource and reserve estimations
- Collect additional detailed geotechnical for mining purposes
- Undertake geotechnical investigations of proposed shaft locations
- Undertake further hydrogeological assessments. Estimated cost \$75,000.
- Undertake further investigation of in situ stresses to confirm assumptions made in the mine design and stability assessments.
- Install an additional 600 kW of grinding power to address the under-sizing of the comminution circuit. Estimated cost \$500,000 installed and this has already been included in the capital cost estimate as it is deemed essential.
- Give consideration to a small increase in lead cleaner and filtration capacity to allow for optimization of silver recovery to payable lead concentrates. Estimated cost \$100,000 and this has not been included in the capital cost estimate as further validation is required.
- Double the tailings filtration capacity. Estimated cost \$580,000 and this has been included in the capital cost estimate as it is also deemed essential.
- Undertake additional testwork of tailings properties and suitability for dry stacking. Estimated cost \$34,000.
- Undertake further TMF site investigations. Estimated cost \$50,000.
- Reassess the factor of TMF safety calculations using standard industry practice finite element numerical modeling
- Prepare a more detailed water balance for the TMF on a month-by-month basis.
- Provide emergency backup power for essential critical services such as man cage, mine ventilation, mine dewatering pumps and thickeners. Estimated cost \$800,000.

## CONTENT

1	SUMMARY .....	1
2	INTRODUCTION .....	17
3	RELIANCE ON OTHER EXPERTS .....	19
4	PROPERTY DESCRIPTION AND LOCATION .....	20
4.1	Exploration and Mining Permits.....	20
5	ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY .....	23
6	HISTORY.....	24
6.1	History of Mining.....	25
6.2	History of Mineral Resources .....	26
7	GEOLOGICAL SETTING AND MINERALIZATION.....	28
7.1	Regional Geology .....	28
7.2	Property Geology .....	29
7.2.1	Structures .....	30
7.3	Mineralization .....	31
7.4	Characteristics of the Mineralized Veins .....	32
7.5	Other Mineralization Features .....	32
8	DEPOSIT TYPES .....	33
9	EXPLORATION.....	34
9.1	Soil Geochemical Program.....	34
9.2	Topographic and Geological Mapping.....	35
9.3	Trenching and Pitting .....	35
9.4	Drilling .....	37
10	DRILLING .....	38
11	SAMPLING, SAMPLE PREPARATION, ANALYSES AND SECURITY .....	44
11.1	Sampling .....	44
11.2	Sample Preparation, Analysis and Security .....	45
12	DATA VERIFICATION.....	46
13	MINERAL PROCESSING AND METALLURGICAL TESTING.....	51
13.1	Introduction.....	51
13.2	Metallurgical Samples .....	51
13.3	Mineralogy.....	52
13.4	Metallurgical Testwork.....	53
13.4.1	Lead Flotation Conditions.....	53
13.4.2	Zinc and Pyrite Flotation Conditions.....	54
13.4.3	Sulphide Circuit Flotation Tests.....	54
13.4.4	Copper-Lead Separation Tests.....	56
13.4.5	Tin Recovery Tests.....	56
13.4.6	Optimization Opportunities .....	56
13.5	Concentrate Quality Considerations.....	58

13.6	Summary of Testwork Outcomes .....	59
	13.6.1 Metallurgical Samples.....	59
	13.6.2 Mineralogy .....	59
	13.6.3 Laboratory Testwork.....	59
	13.6.4 Optimization Opportunities .....	60
	13.6.5 Concentrate Quality .....	60
14	MINERAL RESOURCE ESTIMATES .....	61
	14.1 Resource Data and Statistics .....	63
	14.2 Resource Estimates .....	64
15	MINERAL RESERVE ESTIMATES .....	66
	15.1 Introduction.....	66
	15.2 Resource Extraction Limits.....	66
	15.3 Cut-off.....	67
	15.4 Bulk Density .....	68
	15.5 Mine Dilution and Losses .....	68
	15.6 Mineral Reserve Estimate .....	69
16	MINING METHODS.....	72
	16.1 Conventions .....	72
	16.2 Introduction.....	72
	16.3 Geotechnical Conditions .....	72
	16.3.1 Introduction .....	72
	16.3.2 Available Data.....	73
	16.3.3 Data Analysis.....	73
	16.3.4 Characterization of Geotechnical Conditions.....	76
	16.3.5 In Situ Stress .....	77
	16.3.6 Hydrogeology .....	77
	16.3.7 Mine Design Considerations.....	77
	16.4 Extraction Sequence .....	81
	16.5 Production Rate.....	81
	16.6 Mining Methods .....	81
	16.6.1 Shrinkage Stoping .....	81
	16.6.2 Resue Stoping .....	82
	16.7 Mine Design .....	84
	16.7.1 Pre Existing Development .....	86
	16.7.2 Mine Access .....	87
	16.8 Mine Development .....	87
	16.8.1 Development.....	88
	16.8.2 Shafts .....	90
	16.8.3 Capital and Operating Development.....	91
	16.8.4 Development Stages .....	91
	16.9 Mine Production .....	92
	16.10 Rock Handling .....	94
	16.10.1 Shaft Hoisting .....	94
	16.10.2 Waste Dump.....	94
	16.10.3 Stage 1 Rock Transportation.....	95
	16.10.4 Stage 2 Rock Transportation.....	95
	16.10.5 Stope Backfill.....	96
	16.11 Mine Services.....	96

16.11.1	Ventilation.....	96
16.11.2	Water Supply.....	99
16.11.3	Dewatering.....	99
16.11.4	Power Supply.....	101
16.11.5	Fuel Storage and Dispensing.....	101
16.11.6	Compressed Air.....	101
16.11.7	Communications.....	101
16.11.8	Explosive Storage.....	102
16.11.9	Mine Equipment Maintenance.....	102
16.11.10	Roadway Maintenance.....	102
16.12	Mine Equipment.....	102
16.12.1	Equipment Productivities.....	104
16.13	Mine Personnel.....	104
16.14	Mine Safety.....	105
16.14.1	Fire Prevention.....	105
16.14.2	Mine Rescue.....	106
16.14.3	Dust.....	106
16.14.4	Emergency Egress.....	106
16.14.5	Mine Refuge Station.....	107
16.14.6	Ablutions.....	107
17	RECOVERY METHODS.....	108
17.1	Introduction.....	108
17.2	Process Flowsheet.....	108
17.3	Process Description.....	110
17.3.1	Summary.....	110
17.3.2	Crushing.....	112
17.3.3	Grinding.....	112
17.3.4	Flotation.....	112
17.3.5	Concentrate handling.....	115
17.3.6	Tin Recovery Circuit.....	115
17.4	Process Control and Automation.....	115
17.5	Ancillary Facilities.....	116
17.5.1	Laboratory.....	116
17.5.2	Maintenance Workshop(s).....	116
17.6	Key Inputs.....	116
17.6.1	Power.....	116
17.6.2	Water.....	116
17.6.3	Reagents.....	117
17.7	Summary.....	117
18	PROJECT INFRASTRUCTURE.....	118
18.1	Tailings Management Facility (TMF).....	118
18.1.1	Overview.....	118
18.1.2	Tailings Properties.....	118
18.1.3	Site Selection.....	119
18.1.4	TMF Design.....	120
18.1.5	Tailings Delivery.....	121
18.1.6	Water Balance Considerations.....	121
18.2	Waste Rock Dump.....	122
18.3	Power supply.....	122

18.4	Roads .....	123
18.5	Transportation .....	123
18.6	Water supply .....	124
18.7	Sewage treatment .....	124
18.8	Mine dewatering .....	125
18.9	Site communications .....	125
18.10	Camp	125
18.11	Dams and Tunnels .....	125
	18.11.1 Surface Maintenance Workshop .....	125
	18.11.2 Underground workshop .....	126
18.12	Explosives magazines .....	126
18.13	Fuel farm .....	127
18.14	Mine dry .....	127
18.15	Administration Building .....	127
18.16	Open Area Storage .....	127
18.17	Assay Laboratory .....	127
18.18	Security/Gatehouse .....	127
18.19	Communication System .....	127
18.20	Compressed air .....	128
19	MARKET STUDIES AND CONTRACTS .....	129
19.1	Concentrate Marketing .....	129
19.2	Smelter Contracts .....	129
19.3	Commodity Prices .....	130
20	ENVIRONMENTAL, PERMITTING AND SOCIAL / COMMUNITY IMPACT .....	131
20.1	Introduction .....	131
20.2	Laws and Regulations .....	131
20.3	Waste and Tailings Disposal .....	132
20.4	Site Monitoring .....	133
	20.4.1 Monitoring Plan .....	133
	20.4.2 Water Management .....	133
	20.4.3 Groundwater .....	133
	20.4.4 Waste Water .....	133
20.5	Permitting Requirements .....	134
20.6	Social .....	135
20.7	Remediation and Reclamation .....	135
20.8	Site Closure Plan .....	136
21	CAPITAL AND OPERATING COSTS .....	137
21.1	Summary .....	137
21.2	Mining Capital Cost .....	137
21.3	Mining Operating Cost .....	138
21.4	Mill Capital Costs .....	138
21.5	Mill Operating Costs .....	139
22	ECONOMIC ANALYSIS .....	141
22.1	Key Assumptions .....	141
22.2	Key Metrics .....	141
22.3	Taxes and Royalties .....	144
22.4	Sensitivity Analysis .....	144

23	ADJACENT PROPERTIES.....	146
24	OTHER RELEVANT DATA AND INFORMATION.....	147
25	INTERPRETATION AND CONCLUSIONS .....	148
26	RECOMMENDATIONS .....	151
27	REFERENCES .....	153
28	CERTIFICATES OF QUALIFIED PERSONS .....	154

## TABLES

Table 2.1	Persons who Prepared or Contributed to this Technical Report.....	17
Table 4.1	Mining Permit Boundary of the GC Property .....	21
Table 4.2	GC Mining Permit owned by Anhui Yangtze.....	21
Table 6.1	Historical Exploration Work 2001-2007 .....	25
Table 6.2	Record of Drilling 2001-2007 .....	25
Table 6.3	Details of Underground Workings.....	26
Table 6.4	SRK April 2008 Resource Estimates for the GC Project .....	27
Table 6.5	AMC June 2009 Resource Estimates for the GC Project.....	27
Table 7.1	Dimensions and Occurrences of Mineralized Veins .....	32
Table 9.1	Main Programs Conducted on the GC Property by Silvercorp .....	34
Table 9.2	Trenches and Pits Completed by Silvercorp in 2008.....	36
Table 9.3	2008 Drilling in Metres by Core Diameter.....	37
Table 10.1	Meters of various drill core sizes drilled in different years .....	39
Table 11.1	Individual Vein Characteristics.....	45
Table 11.2	Detection Limits, Aqua Regia / AAS .....	46
Table 11.3	Variation in Bulk Density Measurements .....	47
Table 12.1	Independent Core Sample Verification .....	48
Table 13.1	Metallurgical Samples Relative to GC Reserve.....	52
Table 13.2	Silver Associations.....	53
Table 13.3	Closed Circuit Flotation Test Results.....	56
Table 13.4	Closed Circuit Flotation Test Results.....	59
Table 14.1	Top-cuts of different veins .....	62
Table 14.2	Number of Samples and Assays in Resource Estimates for Different Veins.....	63
Table 14.3	Mineral Resources at 100g/t and 150 g/t Recovered Silver Equivalent Cut-off Grades .....	65
Table 14.4	Mineral Resources 100g/t Recovered Silver Equivalent Cut-off Grade.....	65
Table 15.1	Resource Extraction Boundary Limits.....	67
Table 15.2	Mineral Reserve Cut-off Estimate.....	67
Table 15.3	Mine Width, Dilution and Recovery by Stope Method .....	69
Table 15.4	Mineral Reserve Summary .....	70
Table 16.1	Rock Mass Classification Parameters by Geotechnical Domain .....	77
Table 16.2	Co-ordinates of Pre Existing Development.....	87
Table 16.3	Lateral Development Profiles and Categories .....	88
Table 16.4	Vertical Development Profiles and Categories .....	89
Table 16.5	Development Waste .....	89
Table 16.6	Mine Shafts.....	90

Table 16.7	LOM Capital and Operating Development Summary .....	91
Table 16.8	LOM Production Summary .....	92
Table 16.9	Mining Contractor Typical Key Equipment Summary .....	103
Table 16.10	Silvercorp Typical Fixed Plant Summary .....	103
Table 16.11	Activity Productivities .....	104
Table 19.1	Key Elements of Smelter Contracts .....	130
Table 21.1	Summary of Capital Costs .....	137
Table 21.2	Summary of Operating Costs .....	137
Table 21.3	Mining Capital Costs .....	138
Table 21.4	Mill & Infrastructure Capital Costs .....	139
Table 21.5	Mill Operating Costs .....	140
Table 22.1	Metal Prices .....	141
Table 22.2	Project Production Schedule and Cash Flow (Pre-Tax) .....	142
Table 22.3	Scenario Sensitivity Analysis .....	145

## FIGURES

Figure 4.1	GC Property Location Map .....	20
Figure 7.1	Tectonic Geology Map of Southern China .....	28
Figure 7.2	Regional Geological Map .....	29
Figure 7.3	Property Geology Map .....	30
Figure 7.4	Fault Planes .....	31
Figure 9.1	Soil Geochemical Anomalies of Ag, Pb, And Zn on the GC Property .....	35
Figure 9.2	Locations of Trenches and Pits on the GC property .....	36
Figure 10.1	Drill-hole locations in the GC property .....	38
Figure 10.2	Drill Core Storage Facility .....	39
Figure 10.3	Cross Section on Exploration Line 24 .....	40
Figure 10.4	Cross Section on Exploration Line 28 .....	41
Figure 10.5	Cross Section on Exploration Line 32 .....	42
Figure 10.6	Cross Section on Exploration Line 406 .....	43
Figure 13.1	Closed Circuit Flotation Test Flowsheet .....	55
Figure 13.2	Concentrate Value vs %Pb (Ag \$18/oz) .....	57
Figure 13.3	Concentrate Value vs %Pb (Ag \$30/oz) .....	58
Figure 16.1	Section looking west at 93500 mE, showing '100-RQD' histograms plotted on drillhole traces .....	74
Figure 16.2	Distribution Analysis for All Geotechnical Logging Data .....	75
Figure 16.3	RQD Data Distribution .....	76
Figure 16.4	Typical Shrinkage Stope Layout .....	82
Figure 16.5	Typical Resue Stope Layout .....	83
Figure 16.6	GC Mine Design in 3D Perspective .....	84
Figure 16.7	GC Mine Design Stages .....	85
Figure 16.8	GC Veins and Development Plan View +50mRL .....	86
Figure 16.9	Development Profile by Type .....	90
Figure 16.10	Development Stages .....	91
Figure 16.11	LOM Production Profile by Type .....	93
Figure 16.12	LOM Production Profile by Stage .....	93
Figure 16.13	GC Mine Operation Labour Estimates by Stage .....	105
Figure 17.1	Process Flowsheet .....	109
Figure 17.2	Processing Plant Site Plan .....	111

Figure 17.3	Grinding/Flotation Plant General Arrangement.....	114
Figure 18.1	Plan of Surface Facilities, GC Project.....	120
Figure 23.1	Zonation of Mineralization in the Daganshan Mineralization Field .....	146

## 2 INTRODUCTION

AMC Mining Consultants (Canada) Ltd (AMC) was commissioned by Silvercorp Metals Inc. (Silvercorp) to review a report titled "Mining and Dressing Project of Gaocheng Lead-Zinc Ore in Yun'an County, Guangdong Province" prepared by the Guangdong Metallurgical & Architectural Design Institute (GMADI) in January 2011, and to prepare an independent Technical Report on the property incorporating its findings on the GMADI report. AMC prepared a previous Technical Report on the GC property in June 2009 titled "NI 43-101 Technical Report Update on the GC Ag-Zn-Pb Project in Guangdong Province, People's Republic of China" (AMC report number 709003).

**Table 2.1 Persons who Prepared or Contributed to this Technical Report**

Qualified Persons responsible for the preparation of this Technical Report						
Qualified Person	Position	Employer	Independent of Silvercorp?	Date of Last Site Visit	Professional Designation	Sections of Report
Mr B O'Connor	Principal Geologist	AMC Mining Consultants (Canada) Ltd	Yes	23 – 31 May 2011	PGeo, BSc MCIM	Sections 2 to 12, 14, 20, 23, 24
Mr P Mokos	Principal Mining Geologist	AMC Mining Consultants (Canada) Ltd	Yes	23 – 31 May 2011	BSc (Eng), DipEng (Mining), MAusIMM (CP)	15, 16
Mr A Riles	Principal Metallurgical Consultant	Riles Integrated Resource Management Ltd	Yes	23 – 31 May 2011	BSc (Hons) Grad Dipl Business Management, MAIG	13, 17, 19, 21, 22, part of 18
Mr O Watson	Senior Geotechnical Engineer	AMC Mining Consultants (Canada) Ltd	Yes	23 – 31 May 2011	BEng (Geological) (Hons) MAusIMM (CP), MCIM	Parts of 15, 16
Mr M Molavi	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd	Yes	No visit	PEng, M Eng, B Eng	Parts of 18
Mr P Stephenson	General Manager	AMC Mining Consultants (Canada) Ltd	Yes	No visit	PGeo, BSc (Hons), FAusIMM (CP), MAIG, MCIM	1, 25, 26
Other Experts upon whose contributions the Qualified Persons have relied						
Expert	Position	Employer	Independent of Silvercorp	Visited Site	Sections of Report	
Mr B Fallaw	Senior Tailings and Backfill Consultant	AMC Consultants Pty Ltd	Yes	No visit	Part of 18	
Mr S Wyllie	Senior Mining Consultant	AMC Mining Consultants (Canada) Ltd	Yes	No visit	Parts of 15, 16	

Brian O'Connor, Owen Watson, Peter Mokos and Alan Riles of AMC visited the GC property in May 2011 (P Mokos revisited in July-August 2011). All aspects of the project were examined by

the Qualified Persons, including drill core, exploration sites, underground workings, processing plant and surface infrastructure.

In preparing this report, AMC has relied on various geological maps, reports and other technical information provided by Silvercorp. AMC has reviewed and analysed the data provided and drawn its own conclusions augmented by its direct field observations. The key information used in this report is listed in the References chapter at the end of this report.

Much of the geological information in this report was in Chinese, prepared by Mr. Qiang Wang, Chief Geologist of Anhui Yangtze, and his staff. A translation of key technical documents and data into English was completed by Mr. Hongen Ma, M Sc. (Geology), and Mr. Myles Gao, P Geo. Both are fluent in Chinese and competent in English. The technical data was derived from a Chinese report by the GMADI Report: 'Mining and Dressing Project of Gaocheng Lead-Zinc Ore in Yun'an County, Guangdong Province - Preliminary Design (GD1371CS) Volume I', January 2011. An English translated report was provided to AMC by Silvercorp. Legends and other text on many of the maps were translated by Silvercorp. The Qualified Persons have no reason to believe that the translations are not credible and believe they are generally reliable but cannot attest to their absolute accuracy.

Silvercorp's internal technical information reviewed by AMC was adequately documented, comprehensive and of good technical quality. It was gathered, prepared and compiled by competent technical persons. Silvercorp's external technical information was prepared by reputable companies and AMC has no reason to doubt its validity. AMC used its professional judgement and made recommendations in this report where further work is warranted.

All currency amounts and commodity prices are stated in US dollars. Quantities are stated in metric (SI) units. Commodity weights of measure are in grams (g) or percent (%) unless otherwise stated.

This report includes the tabulation of numerical data which involves a degree of rounding for the purpose of resource estimation. AMC does not consider any rounding of the numerical data to be material to the project.

This report has been produced in accordance with the Standards of Disclosure for Mineral Projects as contained in NI 43-101 and accompanying policies and documents. NI 43-101 utilises the definitions and categories of mineral resources and mineral reserves as set out in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves Definitions and Guidelines (CIM Standards).

A draft of the report was provided to Silvercorp for checking for factual accuracy. The report is effective at 23 January 2012.

### 3 RELIANCE ON OTHER EXPERTS

The Qualified Persons have relied, in respect of legal aspects, upon the work of an Expert listed below. To the extent permitted under NI 43-101, the Qualified Persons disclaim responsibility for the relevant section of the Report.

The following disclosure is made in respect of this Expert:

Audrey Chen, Partner, Jun He Law Offices, Beijing.

**Report, opinion or statement relied upon:** information on mineral tenure and status, title issues, royalty obligations, etc.

**Extent of reliance:** full reliance following a review by the Qualified Person(s).

**Portion of Technical Report to which disclaimer applies:** Section 4.

#### 4 PROPERTY DESCRIPTION AND LOCATION

The GC property is located in Yunfu City, Yun'an County, Guangdong Province, People's Republic of China.

Figure 4.1 GC Property Location Map



#### 4.1 Exploration and Mining Permits

In 2008, Silvercorp acquired 100% of the shares of Yangtze Gold Ltd. (Yangtze Gold), a private British Virgin Island (BVI) company, which in turn wholly owns Yangtze Mining Ltd. (Yangtze Mining). Yangtze Mining owns a 95% interest in a Sino-Foreign joint venture company, Anhui

Yangtze Mining Co. Ltd. (Anhui Yangtze). Anhui Yangtze's main asset is the GC exploration permit for the GC Project, which was subsequently converted mining permit in November 2010.

The boundaries of the exploration permit have not been surveyed and no boundary markers have been staked in the ground.

On June 14, 2010 Silvercorp announced that it has been issued an Environmental Permit for the project from the Department of Environmental Protection of Guangdong Province, an essential document required for a mining permit application.

A Mining License was issued by the Ministry of Land and Resources of China on November 24, 2010. The license is valid for 30 years to November 24, 2040, covers the entire 5.5238 km<sup>2</sup> area of the GC project and permits mining from 315m to minus 530 m elevations. The permit was issued on the terms applied for, and allows for the operation of an underground mine to produce silver, lead and zinc.

Approximate boundaries of the GC project area are as follows.

**Table 4.1 Mining Permit Boundary of the GC Property**

Point	Geographic Coordinates		Gauss Coordinates	
	Longitude E	Latitude N	X	Y
1	111°53'45"	22°55'45"	2537018.36	37591895.13
2	111°55'30"	22°55'45"	2537036.88	37594887.25
3	111°55'30"	22°54'45"	2535190.97	37594898.85
4	111°53'45"	22°54'45"	2535172.45	37591906.37

The grid system used for the GC project is the Beijing Geodetic Coordinate System 1954. Altitude is referred to the Yellow Sea 1956 Elevation System. The project survey control points were generated from three nearby national survey control points.

Key Information contained in the Mining License is as follows:

**Table 4.2 GC Mining Permit owned by Anhui Yangtze**

Permit No.	No. C1000002010113210083333
Owner	Anhui Yangtze Mining Co. Ltd.
Owner's address	#102-21 Building He Tai Xing town, Chizhou city, Anhui Province.
Name of the Mine	Anhui Yangtze Mining Co. Ltd GC Lead and Zinc Mine
Business Category	Sino-Foreign cooperative enterprises
Types of ore mined	Zinc, Lead and Silver ore
Mining Method:	Underground mining
Production Capacity:	330,000 tonne/year
Mine Area	5.5238 Km <sup>2</sup>
Valid Period	2010-11-24 to 2040-11-24
Issued Date	Nov. 24 2010

The Licensee is subject to the charge of a Mining-right using fee (\$158 /km<sup>2</sup>), a Mineral-resource compensation fee (2% of sales) and applicable mineral resource taxes (\$2/t milled).

The Guangdong Metallurgical & Architectural Design Institute, a qualified Chinese engineering firm finalized the design of a 1,600 tonne per day mechanized underground mine, a flotation mill, and a dry stack tailing facility. The estimated capital cost was about \$30 million (however, see Section 21 for AMC's updated estimate of capital costs). With the support of the local County government, Silvercorp has completed the acquisition of surface rights required for the construction of the mine and mill and is preparing the site and hiring contractors for the construction. Initial production of 700 tonnes per day mining capacity is expected to be achieved in 12 months with full capacity of 1,600 tonnes per day to be achieved in 18 months.

AMC is not aware of any additional royalties, back-in rights, payments, agreements, environmental liabilities or encumbrances particular to the property other than those stated above.

## **5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY**

The GC project is located around Gaocheng Village of Gaochun Township, Yun'an County, Guangdong Province, China. Altitudes in the region range from 78.0m to 378.0m above sea level (ASL), usually 150-250m ASL, with relative differences of 50-150m. Vegetation is in the form of secondary forests of pine and hardwoods, bushes and grasses. Top soil covers most of the ground. Outcrops of bedrocks can only be observed in valleys.

The GC project is located west of the metropolitan city of Guangzhou, the capital of Guangdong Province. Guangzhou is located about 120 km northwest of Hong Kong and has a population of almost 12 million registered residents and temporary migrant inhabitants in December of 2007, according to an economic report released by the Guangzhou Academy of Social Science. It is serviced by rail and daily flights from many of China's larger population centres. Access to the GC project from Guangzhou is via 178 km of four lane express highway to Yunfu, then 48 km of paved road to the project site. A railway connection from Guangzhou to Yunfu is also available.

The region belongs to sub-tropical monsoon climate with average annual temperature of 20-22°C. Rainfall is mainly concentrated in spring and summer from March to August. Winters feature short periods of frosting. The GC project would be able to operate year round

Streams are well developed in the area, the Hashui Creek flows in the GC project area. There is a reservoir upstream of the GC project area. Small hydro power stations are developed in the region that are connected to the provincial electrical grid. There is a 10KV power line across through the project area. Silvercorp is also having discussions with a utility company regarding power supply and related facilities.

The economy of Yun'an County mainly relies upon agriculture and some small township industrial enterprises. Labour is locally available, and technical personnel are available in Yunfu and nearby cities. The Gaocheng village is located within the GC project area.

## 6 HISTORY

Various state-sponsored Chinese Geological Brigades and companies have conducted geological and exploration work in the project area. Systematic regional geological surveys covering the area started in 1959. The following is a brief history of the exploration work in the area:

During 1959 to 1960, No. 763 Geological Brigade of Guangdong Bureau of Geology conducted a 1:200,000 regional geological survey and mapping, and regional prospecting of mineral resources in the area. A geological map and geological reports were published.

In 1964-1967, Comprehensive Study Brigade of Guangdong Bureau of Geology conducted general prospecting and 1:50,000 geological mapping in the area including the project area, and submitted a geological report.

In 1983, Geophysical Survey Brigade of Guangdong Bureau of Geology and Mineral Resources conducted a 1:200,000 airborne magnetic survey covering the project area.

In 1988, the Regional Geological Survey Brigade of Guangdong Bureau of Geology and Mineral Resources conducted a 1:200,000 stream sediment survey, which covers the project area.

In 1991, Geophysical Survey Brigade of Guangdong Bureau of Geology and Mineral Resources conducted a 1:200,000 gravity survey covering the project area.

In 1995, Ministry of Geology and Mineral Resources completed the compilation and interpretation of 1:1,000,000 geochemical, geophysical and remote sensing surveys covering the area.

During 1995 and 1996, Geophysical Survey Brigade of Guangdong Bureau of Geology and Mineral Resources conducted a 1:50,000 soil survey, and defined some large and intensive Pb, Zn, Ag, Sn, W and Bi geochemical anomalies, which covers the project area.

During 1990 and 2000, Guangdong Provincial Institute of Geological Survey (GIGS) conducted a 1:50,000 stream sediment survey which covers the project area, and defined several intensive anomalies of Pb-Zn-Ag-Sn-Mn, leading to the discovery of GC deposit.

During 2001 and 2002, and again in 2004 and 2005, GIGS conducted general prospecting at the GC project area, and defined some mineralized bodies and estimated mineral resources for the GC deposit.

During 2006 and 2007, contracted by Yangtze Mining, GIGS conducted a detailed prospecting at the GC project area, completed a 36-hole, 11,470m surface diamond drilling program and 1,964m<sup>3</sup> of trenching and surface stripping, to update and upgrade the mineral resources of the GC deposit.

A summary of the historical work between 2001 and 2007 is shown in Table 6.1. Table 6.2 contains a drill record for the same period.

**Table 6.1 Historical Exploration Work 2001-2007**

Program	Unit	Work completed			
		2001-2002	2004-2005	2006-2007	Total
1:10,000 Soil survey	km <sup>2</sup>	4.0			4.0
1:10,000 soil profiling	km	12.4			12.4
1: 5,000 Hydrogeological survey	km <sup>2</sup>			12.5	12.5
1:10,000 Geological survey	km <sup>2</sup>	15.0			15.0
1:5000 Geological survey	km <sup>2</sup>		5.3		5.3
1:2000 Geological survey	km <sup>2</sup>			4.5	4.5
1:2000 topographic survey	km <sup>2</sup>			4.5	4.5
Exploration line survey	km	3.8	2.0	6.7	12.5
Diamond drilling	m	1,010.3	983.6	11,469.8	13,463.7
Trenching (pitting)	m <sup>3</sup>	3,888.0	3,582.5	1,964.1	9,434.6
Tunnelling	m	66.0			66.0
Tunnel clean-up	m	197.2		302.6	499.8
Tunnel geological mapping and sampling	m	637.2		632.6	1,269.8
Control point survey	point			42.0	42.0
Survey of engineering points	point		23.0	222.0	245.0
Hydrogeological observation hole	point			2.0	2.0
Soil samples	sample	1,116.0			1,116.0
Chemical analysis samples	sample	454.0	182.0	2,849.0	3,485.0
Internal checking samples	sample	17.0		275.0	292.0
External checking samples	sample			150.0	150.0
Rock survey samples	sample	41.0			41.0
Thin and polishing sections	piece	34.0	8.0	27.0	69.0
Composite samples	sample	2.0		26.0	28.0
Spectrum analysis sample	sample			1.0	1.0
Small specific gravity samples	sample			62.0	62.0
Artificial heavy mineral sample	sample			1.0	1.0
Multiple element samples	sample			3.0	3.0
Water quality samples	sample			11.0	11.0
Rock and ore samples	sample			38.0	38.0
Sample for metallurgical test	sample			1.0	1.0
Metallurgical testing	test			1.0	1.0

**Table 6.2 Record of Drilling 2001-2007**

Year Drilled	PQ (m) 85 mm	HQ (m) 63.5 mm	NQ (m) 47.6 mm	Total (m)
2001-2005		1,993.9		1,993.9
2006-2007	420.3	5,179.7	5,869.9	11,469.8
Total (m)	420.3	7,173.6	5,869.9	13,463.7

## 6.1 History of Mining

Prior to Yangtze Mining acquiring the GC Property, illegal mining activity resulted in the excavation of several tunnels and small scale mining of veins V2, V2-2, V3, V4, V5, V6 and

V10. GIGS reported that a total of 1,398 m of excavation comprised of 10 adits and tunnels had been completed on the property through the illegal activity.

In 2002, GIGS developed 66m of tunnel to crosscut veins V5 and V5-1. GIGS sampled and mapped adits ML1 to ML5, ML6, ML7, ML9, and PD12.

Yangtze Mining, after its purchase of the property in 2005, mapped and sampled the accessible tunnels ML5 and ML8. Tunnel ML5 has exposure to vein V10 and tunnel ML8 has exposure to vein V2-2. Assay results of tunnel samples were used in resource estimation. Table 6.3 details the underground workings and work completed.

**Table 6.3 Details of Underground Workings**

Tunnel/Adit	Length of Tunnel/Adit (m)	Vein intersected	Samples No. Collected	Mapped and Sampled By
ML1	156	V4	12	GIGS
ML2	70	V3	1	GIGS
ML3	2	V4	6	GIGS
ML4	41	V4	3	GIGS
ML5	324	V10	13	Yangtze
ML6	438	V2	25	GIGS
ML7	45	not named, parallel to V4		GIGS
ML8	246	V2-2	19	Yangtze
ML9	46	V4		GIGS
PD12	28	V6	3	GIGS
PD4401	66	V5	5	GIGS

AMC visited the tunnel sites but was not able to access them as most entrances were blocked for safety reasons. There is no detailed reconciliation data available for any of the mineralization extracted.

## 6.2 History of Mineral Resources

GIGS prepared a resource estimate for nine mineralized veins for the GC project after the 2004-2005 exploration season. The GIGS has its own classification system of mineral resources / reserves which is different from the CIM Standards. AMC does not believe those resources are material to this report.

Prior to the current report, resource estimates for the GC project were reported in a Technical Report by SRK Consulting (SRK) dated April, 2008 (entitled "Technical Report on Gaocheng Ag-Zn-Pb Project and Shimentou Au-Ag-Zn-Pb Project, Guangdong Province, People's Republic of China") and in AMC's June 2009 Technical Report. Table 6.4 summarizes the SRK resource estimates for thirteen better-explored veins, while Table 6.5 summarizes AMC's 2009 resource estimates.

**Table 6.4 SRK April 2008 Resource Estimates for the GC Project**

Category	Tonnage	Ag (g/t)	Pb (%)	Zn (%)	Sn (%)	Ag (kg)	Pb (t)	Zn (t)	Sn (t)
Measured	499,793	206.63	1.31	3.94	0.14	103,270	6,546	19,712	709
Indicated	1,329,903	100.45	1.51	2.36	0.13	133,590	20,040	31,362	1,772
Inferred (o)	707,237	238.50	1.01	0.10	0.08	168,682	7,121	673	561
Inferred (s)	6,574,146	<b>112.92</b>	1.42	3.32	0.13	742,361	93,669	218,002	8,619
Measured + Indicated	1,829,695	129.45	1.45	2.79	0.14	236,860	26,587	51,075	2,481
Inferred	7,281,383	125.12	1.38	3.00	0.13	911,043	100,790	218,675	9,179

(o) – oxidized (s) – sulphide

**Table 6.5 AMC June 2009 Resource Estimates for the GC Project**

Resource Classification	Tonnes	GRADE		
		Ag (g/t)	Pb %	Zn %
Measured	374,000	233	1.83	3.19
Indicated	6,034,000	132	1.47	3.35
<b>Total</b>	<b>6,408,000</b>	<b>138</b>	<b>1.49</b>	<b>3.34</b>
Inferred	7,892,000	121	1.45	2.70

Current mineral resources and mineral reserves are discussed in the respective sections of this report.

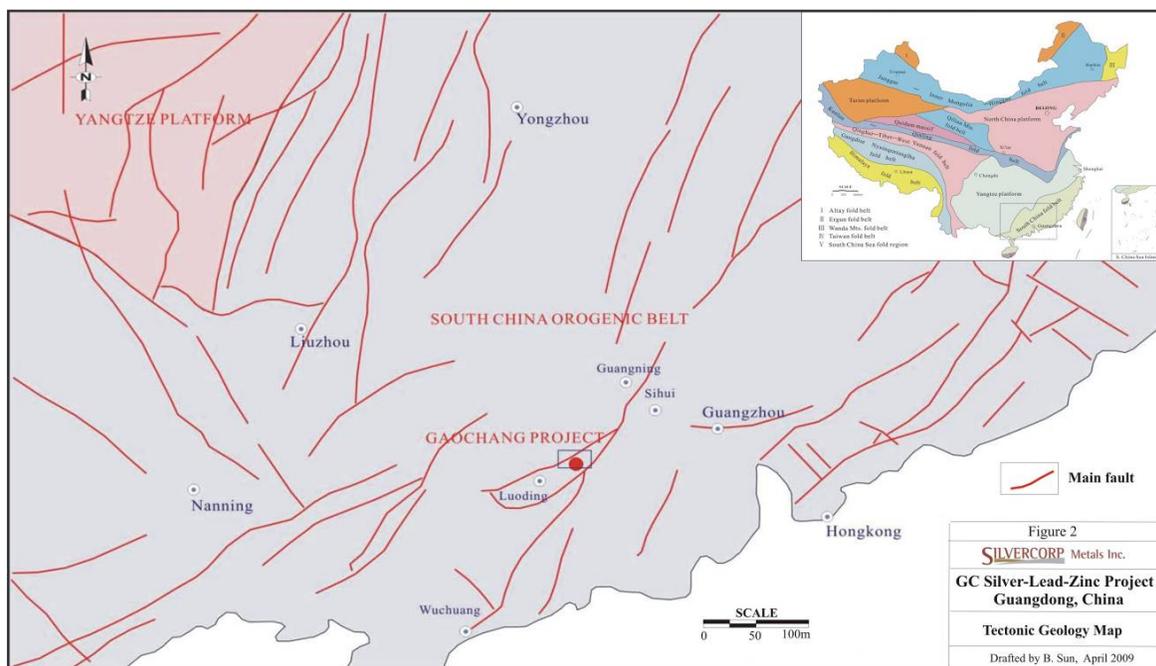
## 7 GEOLOGICAL SETTING AND MINERALIZATION

This section is a summary of the geological setting and mineralization descriptions presented in the AMC's June 2009 Technical Report titled "NI 43-101 Technical Report on the GC Ag-Zn-Pb Project in Guangdong Province, People's Republic of China". The original source of data was derived from the previous SRK Technical Report on the property titled "Technical Report on Gaocheng Ag-Zn-Pb Project, and Shimentou Au-Ag-Zn-Pb Project, Guangdong Province, People's Republic of China", dated April, 2008.

### 7.1 Regional Geology

The property is located at the north portion of the Yunkai uplift of the Southern China folding system, the east margin of the Luoding basin and east of the Wuchuan – Sihui major fault. Structurally, the deposit occurs at the intersection of a northeasterly striking fault zone and a near east-westerly striking fault zone. Northeast striking structures and arc structures form the basic geological framework of the region.

**Figure 7.1 Tectonic Geology Map of Southern China**



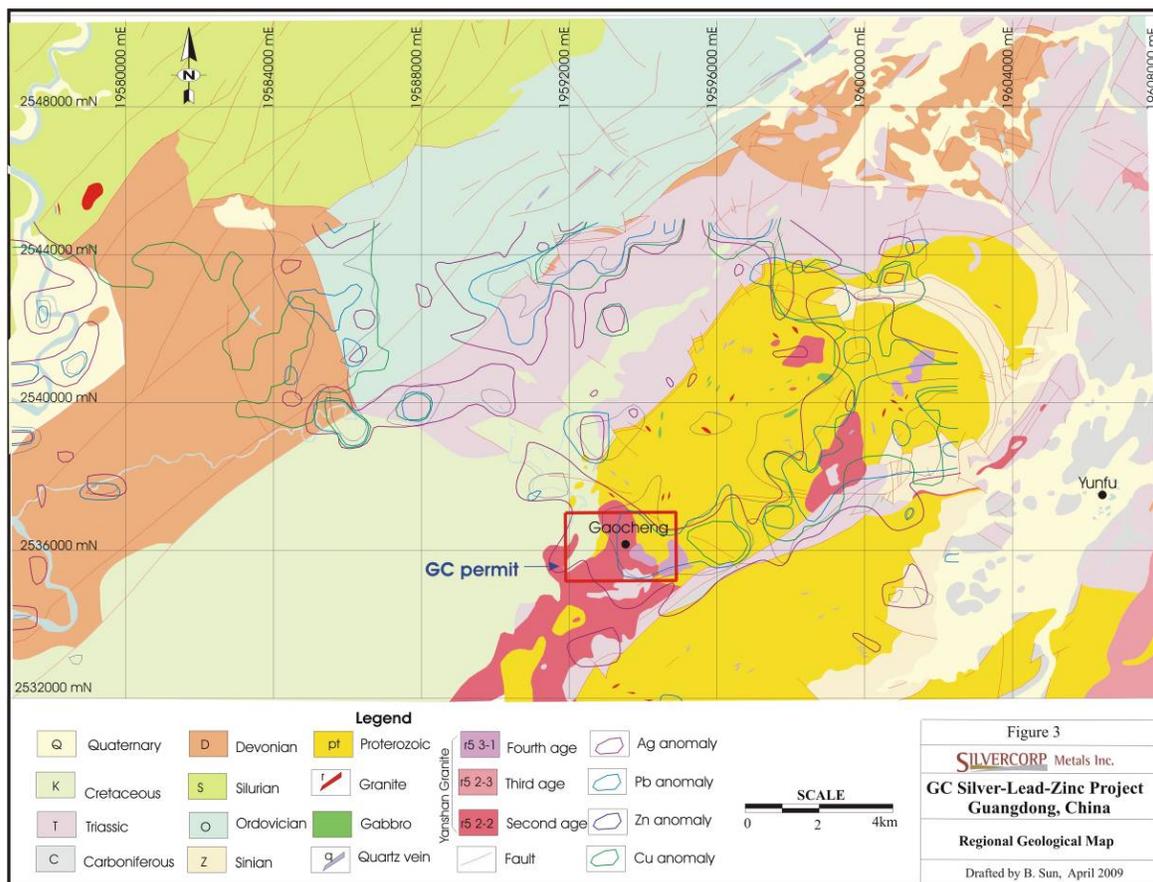
Regional outcrop includes Later Proterozoic Sinian sedimentary clastics and carbonate rocks, which host Ag-Pb-Zn multi-metallic deposits in the region, and Paleozoic Ordovician, Silurian, Devonian, Carboniferous sedimentary clastics and carbonate rocks which host some Cu-Pb-Zn, Mn and Au-Ag deposits, and Mesozoic Triassic coal-bearing clastic rocks, and Cretaceous red clastic rocks.

There are three sets of structures in the region. The north-easterly striking structure is comprised of a series of folds and faults that host some mineralized bodies. The near east-westerly striking structure dips steeply, and contains structural breccias within the faulting zones. Alteration zones are formed along both sides of the regional structures. Typically quartz is the filling in the faults. The third structure set is the arc or ring structures including folds and

faults surrounding the Daganshan granite zone. The Pb-Zn-Ag-Sn deposits and mineralization showings, as well as geochemical anomalies of Au-Ag-Pb-Zn etc. occur in the arc or ring structural zone.

Magma intruded in the centre of the arc/ring structure. The intrusives include Palaeozoic granite batholiths, and Mesozoic granite stocks and dykes. The Mesozoic stocks and dykes intruded in the inner zone of the arc/ring structure, and are associated with the Pb-Zn-Ag multi-metallic mineralization in the region.

**Figure 7.2 Regional Geological Map**



## 7.2 Property Geology

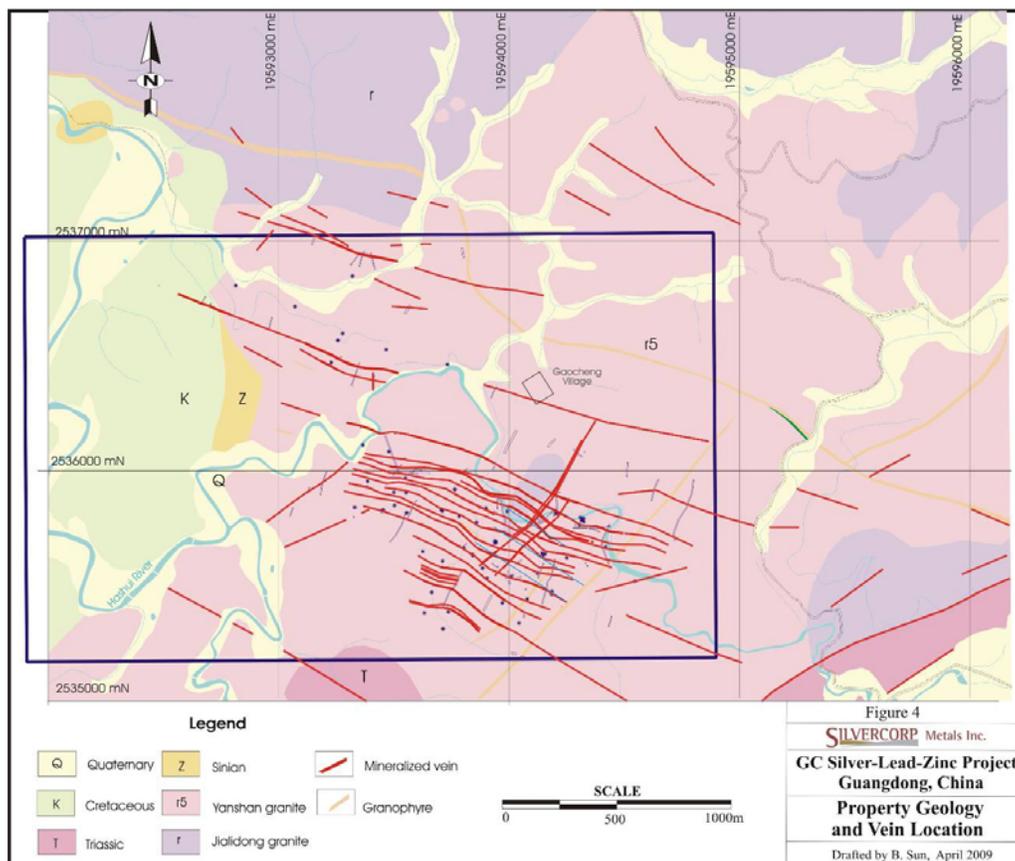
The GC Project is located in the northeastern margin of the Luoding basin, which is at the middle portion of the Yunkai uplift in the Hua'nán (South China) Fold System. The deposit is located at the intersection between Wuchuan-Sihui Deep Fault zone and Daganshan Arc-ring structural zone.

Outcrop in the project area includes the Sinian Daganshan Formation which is composed of quartz sandstone, meta-carbonaceous siltstone, carbonaceous phyllite, calcareous quartzite, argillaceous limestone; the Triassic Xiaoyunwushan Formation which is made up of quartz sandstone and shale; and the Cretaceous Luoding Formation of sandy conglomerate and conglomerate.

A series of magmatic events occurred on the GC property. Intrusives include Palaeozoic gneissic, medium-grained biotite granite, and Mesozoic fine- to medium-grained adamellite, brownish, fine-grained, biotite mylonite, granite porphyry, quartz porphyry, diabase, and aplite. The Mesozoic intrusives intruded along the south and southwest contacts of the Palaeozoic granites. The majority of Ag-Zn-Pb mineralization is hosted by the Mesozoic granite.

The granite dips to south and strikes to west northwest, parallel to the majority of mineralized veins on the GC property.

**Figure 7.3 Property Geology Map**



### 7.2.1 Structures

The project area is situated in the southwest part of the Daganshan uplift. Structures developed in the area are mainly the NWW striking Gaocheng Fault zone, the NE striking Baimei Fault zone, and the Songgui Fault zone.

The NWW-striking fault zone is the main mineralization bearing structure in the deposit. The fault zone is approximately 4.8 km long and 2000m wide. It is the main mineralization hosting structure in the area.

The NE-striking faults cut through the NWW-striking structures with no or minor displacement. Mineralization veins such as V10, V10-1, V11 and V12 are part of this trend.

The following two photos are of the NWW and NE striking faults. The faults demonstrate a sharp contact between the veins and the host rock.

**Figure 7.4 Fault Planes**



a) Vertical and smooth fault plane of V10 Vein in an open stope of Tunnel ML5.



b) Vertical and smooth fault plane of V2-2 Vein in an open stope of Tunnel ML5

### 7.3 Mineralization

The mineralized veins in the GC area occur in relatively permeable fault-breccia zones and are extensively oxidized from the surface to depths of about 40m. Within this zone, the veins show many open spaces with boxwork lattice textures resulting from the leaching and oxidation of sulphide minerals. Secondary minerals present in varying amounts in this zone include kaolinite, hematite, and limonite.

The dominant sulphide is pyrite, typically comprising a few percent to 13% of the vein. Other constituents are a few percent of sphalerite, galena, pyrrhotite, arsenopyrite, magnetite and less than a percentage of chalcopyrite and cassiterite. Metallic minerals in much smaller amounts include argentite, native silver, bornite, wolframite, scheelite, and antimonite. The minerals occur in narrow massive bands, veinlets or as disseminations in the gangue. Gangue minerals include chlorite, quartz, fluorite, feldspar, mica, hornblende, etc., with a small amount or trace amount of kaolinite, tremolite, actinolite, chalcedony, garnet, zoisite, apatite and tourmaline, etc.

The Ag-Zn-Pb mineralization in the deposit can be divided into two types: primary and oxidized. The primary mineralization is mainly composed of galena-sphalerite-silver ore minerals which occur sparsely, disseminate, and as veinlets and lumps. The type accounts for 95% of the entire mineral resource. The oxidized mineralization occurs on and near the surface topography as a result of oxidization of the primary mineralization.

The alteration minerals associated the GC vein systems include silica, sericite, pyrite and chlorite, together with clay minerals and limonite. Silicification is common near the center of the veins, chlorite and sericite occur near and slightly beyond the vein margins.

Silica, pyrite, fluorite, and chlorite are closely related to the mineralization.

## 7.4 Characteristics of the Mineralized Veins

Table 7.1 presents a summary of the characteristics of the veins on the GC property.

**Table 7.1 Dimensions and Occurrences of Mineralized Veins**

Vein #	Length (m)	Defined Inclined Depth (m)	Elevation of defined Depth (m)	Strike/Dip	Average Dip Angle	Average True Thickness (m)
V2	948	516	-364	175-215° ∠60-84°	72	3.35
V2-0	596	458	-326	200° ∠60-84°	69	0.55
V2-1	890	446	-312	130-200° ∠41-84°	70	1.28
V2-2	932	569	-420	180-200° ∠42-84°	70	1.14
V3	158	238	-96	200° ∠80-85°	81	0.53
V4	291	318	-142	200° ∠70-88°	80	0.93
V5	706	430	-172	180-200° ∠70-88°	74	0.46
V5-1	464	380	-96	200° ∠60-88°	73	1.18
V6	1106	659	-485	165-265° ∠60-84°	70	2.24
V6-0	800	445	-329	200° ∠65-70°	67	1.45
V7	800	383	-180	128-200° ∠39-81°	70	0.76
V7-0	720	350	-165	200° ∠63-81°	72	1.31
V7-1	796	342	-172	200° ∠61-83°	73	0.77
V8	208	272	-28	200° ∠56-88°	75	1.07
V8-0	100	211	19	200° ∠78-88°	84	0.29
V8-1	203	87	135	200° ∠78-88°	85	0.56
V9	300	442	-245	200° ∠70-85°	76	1.36
V9-0	198	441	-239	200° ∠83°	83	1.99
V9-1	300	402	-232	200° ∠66-82°	76	0.58
V10	298	281	-94	55-200° ∠65-75°	70	2.58
V10-1	493	230	-56	123° ∠62-70°	68	0.77
V11	185	201	-229	123° ∠66-70°	69	1.08
V13	200	391	-179	200° ∠77-81°	79	1.15
V14	200	255	-80	200° ∠77-78°	78	0.83
V15	100	253	-68	200° ∠77-78°	77	3.9
V15-1	100	135	84	200° ∠78°	78	0.57

## 7.5 Other Mineralization Features

The following are additional mineralization features of the GC deposit.

1. The high grade Ag-Zn-Pb mineralized shoots usually occur at the intersections of the NWW and east-west striking faults, in which the shoots plunge to east.
2. Within breccia zones of a fault, Ag-Zn-Pb mineralization is intensive, continuous, and wide.
3. Individual metal grade contours of the mineralized veins indicate that the Zn mineralization is more common than Ag and Pb. Usually Ag and Pb are locally concentrated.

## 8 DEPOSIT TYPES

This section is a summary of the geological setting and mineralization descriptions presented in the Technical Report titled "NI 43-101 Technical Report on the GC Ag-Zn-Pb Project in Guangdong Province, People's Republic of China" dated June 2009. The original source of data was derived from the previous Technical Report on the property titled "Technical Report on Gaocheng Ag-Zn-Pb Project, and Shimentou Au-Ag-Zn-Pb Project, Guangdong Province, People's Republic of China", dated April, 2008.

The poly-metallic mineralization of the GC deposit belongs to the mesothermal vein infill style of deposit and exhibits the following characteristics:

- The mineralization occurs as veins which are structurally controlled within broader alteration zones. The alteration can reach more than a few meters along the faults distributing in both hanging wall and footwall.
- The veins have a sharp contact with the host rocks and steeply dip at angles between 60-85°.
- In general, the Ag-Zn-Pb mineralization occurs along the strike of the faults. The veins have true widths varying from just over 0.1m to over 10 m. They have been traced for over about 1,250 m along the strike, and approximately 550 m down dip.

## 9 EXPLORATION

Exploration work by Silvercorp on the GC property was carried out in 2008. The program is summarized in Table 9.1. No material exploration has been carried out on the property since that time.

**Table 9.1 Main Programs Conducted on the GC Property by Silvercorp**

Program	Unit	Work Completed
		2008
1:10,000 soil profiling	km	10
Diamond drilling	m	10,083
Trenching (pitting)	m <sup>3</sup>	740
Soil samples	sample	535
Chemical analysis samples	sample	2,139
Metallurgical testing	test	1

The diamond drilling undertaken in 2008 represented 43% of all the diamond drilling on the property.

### 9.1 Soil Geochemical Program

A 1:10,000 scale geochemical survey was done by taking samples from C-layer of top soils and the samples were assayed for Au, Ag, Cu, Pb, Zn, Mo, Sb, and As, etc. The program resulted in outlining significant Ag, Pb, and Zn anomalies, providing targets for surface trenching and pitting. Most of the veins on the property were discovered using the soil geochemical results.

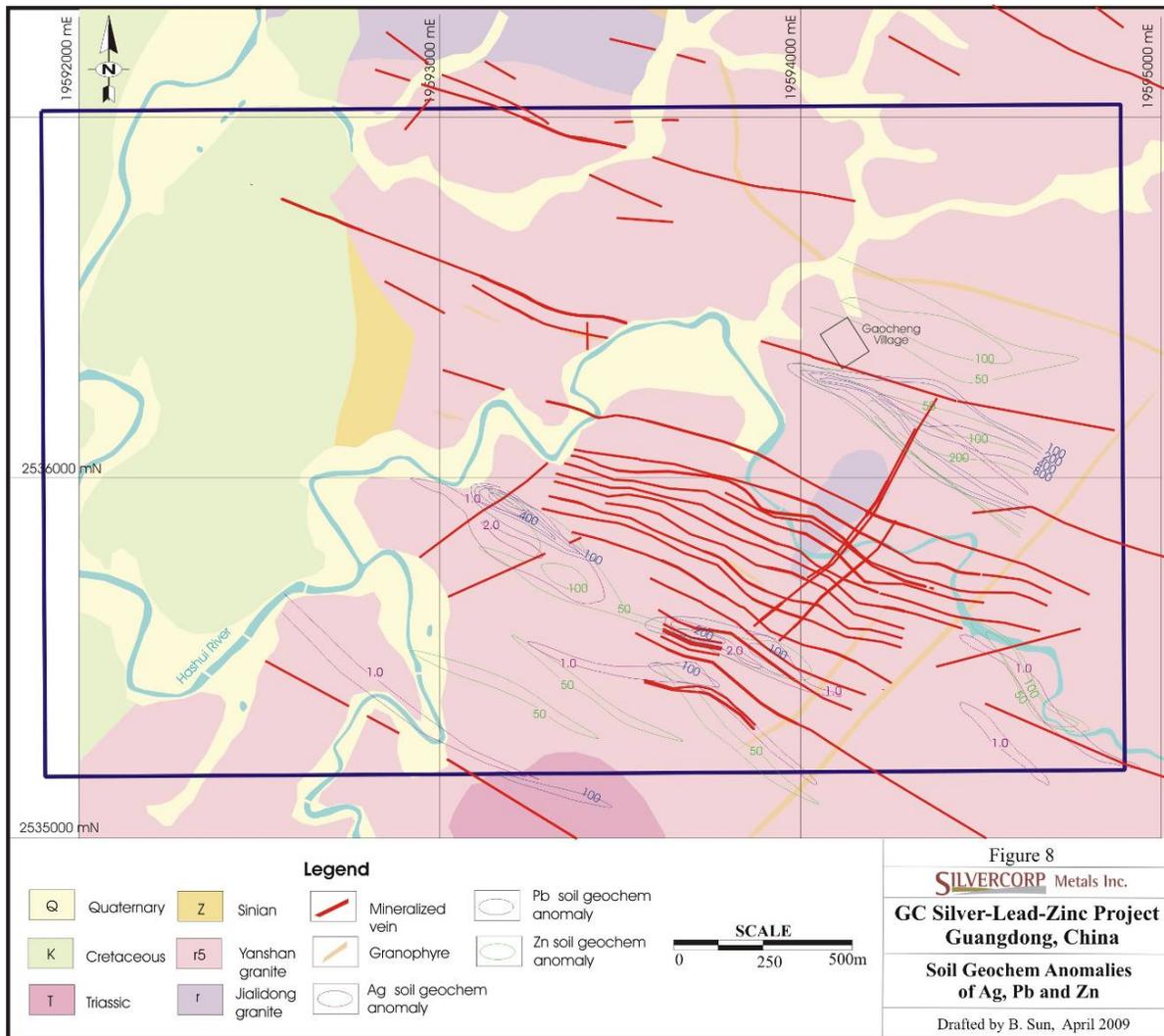
In 2008, soil geochemical survey (1:10,000 scale) was carried out by Silvercorp through the collection of 535 samples within a 2.22 km<sup>2</sup> area in the southern part of the property where no drilling had been previously performed. Three new Ag-Zn-Pb geochemical anomalies observed to be over 500m long and up to 250m wide were identified, providing priority drill targets with the potential to host additional veins.

Anomaly AS1 is located at the east of V4 vein along F4 fault. The anomaly is about 500m in length and 50 to 100m in width. The peak values of Ag, Pb and Zn are 2.1 ppm, 0.19% and 0.03% respectively. Trenching was carried out over the anomaly and mineralization was confirmed by the sample assay result.

AS2 anomaly is located between exploration line 1 and 12. It measures about 500m in length and 20 to 200m in width. The maximum values of Ag, Pb and Zn are 14.5 ppm, 0.11% and 0.02%, respectively.

AS3 anomaly is between exploration lines 28 to 44. Its length is about 500m. The anomaly ranges 20 to 50m in width from exploration lines 36 to 44 and expands to 250 wide at exploration 44.

**Figure 9.1 Soil Geochemical Anomalies of Ag, Pb, And Zn on the GC Property**



## 9.2 Topographic and Geological Mapping

GIGS conducted a 1:10,000, 1:5,000 and 1:2,000 geological mapping programs, and a 1: 2,000 topographic survey covering the GC project area. The geological mapping programs established stratigraphic sequences, size, and distributions of intrusive rocks and faults.

The grid system used for the GC project is BeiYing Geodetic Coordinate System 1954. Altitude is referred to Yellow Sea 1956 Elevation System. The project survey control points were generated from three nearby national survey control points. The control points were surveyed using four NGS-9600 GPS receivers. Survey machines used for topographical survey and geological points, trenches, adits, and drillhole collars were Topcon GTS-Serial Total Station Instrument – XJ0747 and one NX2350 and Sokkia SET-230PK Total Station Instrument.

## 9.3 Trenching and Pitting

Based on the soil geochemical and surface mapping, Silvercorp conducted trenching and pitting programs on the GC property. The program exposed the mineralized veins on the surface and



## 9.4 Drilling

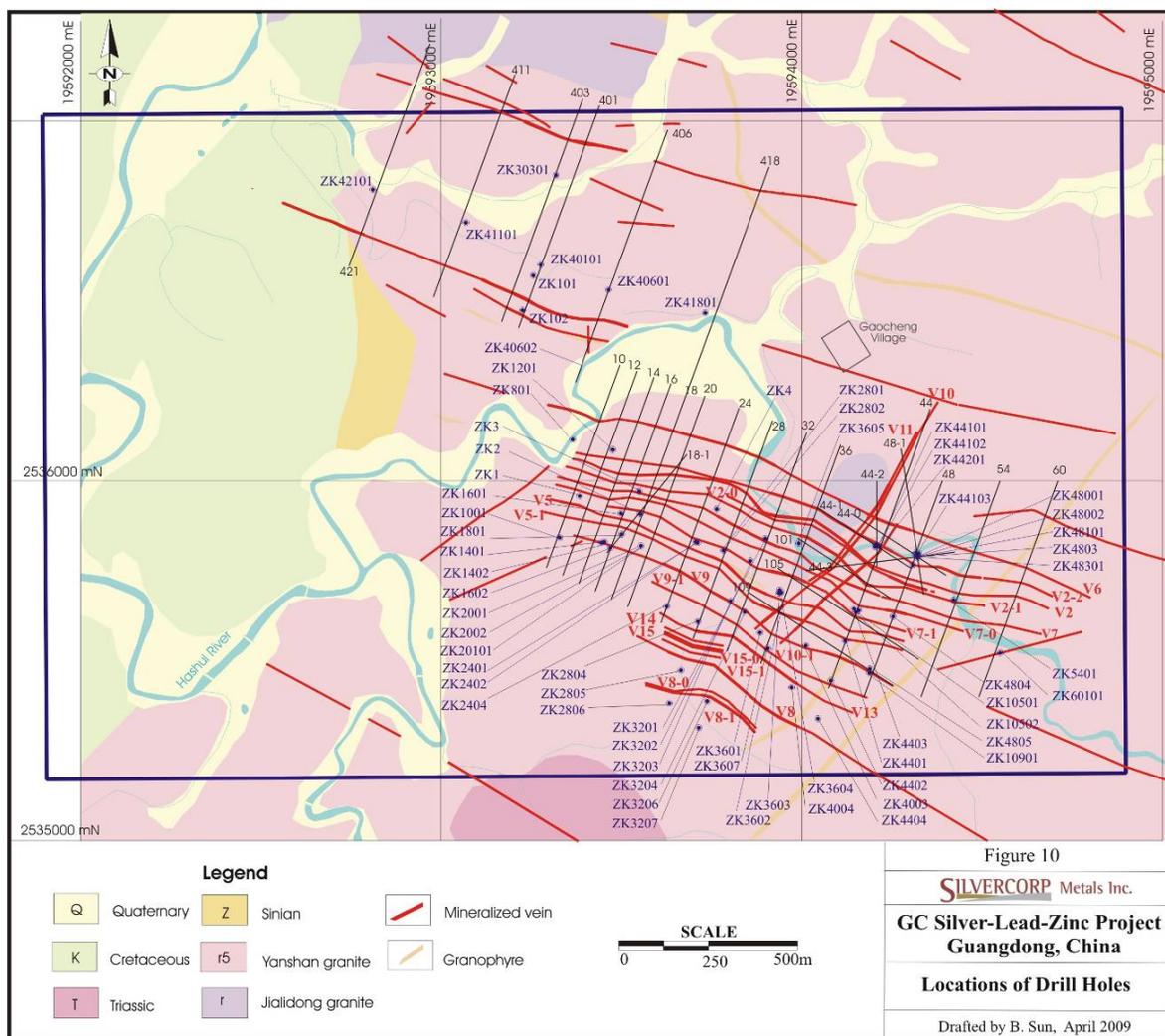
**Table 9.3 2008 Drilling in Metres by Core Diameter**

Year Drilled	PQ (m) - 85mm	HQ(m) - 63.5mm	NQ(m) - 47.6mm	Total(m)
2008	286.1	1,841.8	7,955.2	10,082

## 10 DRILLING

A total of 65 diamond drill holes have been completed on the GC property since 2001 totalling 23,546.34m. During 2008, Silvercorp completed 22 hole, 10,082.6m drilling program, which resulted in the discovery of an additional 15 mineralized veins. Drillhole locations are shown in Figure 11.1. The drill hole collar, downhole survey and core recoveries are listed in Appendix VI and VII in AMC's 2009 Technical Report. A program of surface drilling commenced in the last quarter of 2011 at a budget of \$2.5 million. No results of the program were available at the time of this report.

**Figure 10.1 Drill-hole locations in the GC property**

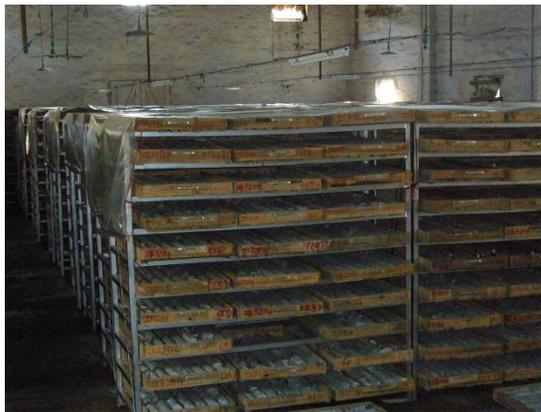


Diamond drill holes were drilled using PQ size in overburden, then reduced to HQ size for up to 100m depth. The remainder of a hole was drilled using NQ size unless the hole was required to drill over 600m in length. Drill core recoveries vary from 85 to 100% and average 99%.

Down hole surveys for a drill hole were done at every 50m using a Chinese made equivalent of a Sperry-Sun downhole survey tool. Drillhole collars were cemented after completion and locations of drill holes were marked using 50x30x20cm concrete blocks.

The drill cores were stored in a clean and well-maintained core shack in the town of Gaochun (Figure 10.2). Core shack are locked when nobody works inside and two attendants are on duty around the clock to maintain good security.

**Figure 10.2 Drill Core Storage Facility**



a) Drill core racks in core shack



b) Drill core racks in core shack

Meters of various drill core sizes are shown in Table 10.1.

**Table 10.1 Meters of various drill core sizes drilled in different years**

Year Drilled	PQ -85 mm (m)	HQ-63.5 mm (m)	NQ-47.6 mm (m)	Total (m)
2001-2005		1993.91		1993.91
2006-2007	420.27	5179.68	5869.88	11469.83
2008	286.1	1841.76	7955.2	10082.6
<b>Total (m)</b>	<b>706.37</b>	<b>7021.44</b>	<b>13825.08</b>	<b>23546.34</b>

Figures 10.3 to 10.6 show drill-holes on various cross sections at the GC property. The widths in the figures are horizontal widths.

Figure 10.3 Cross Section on Exploration Line 24

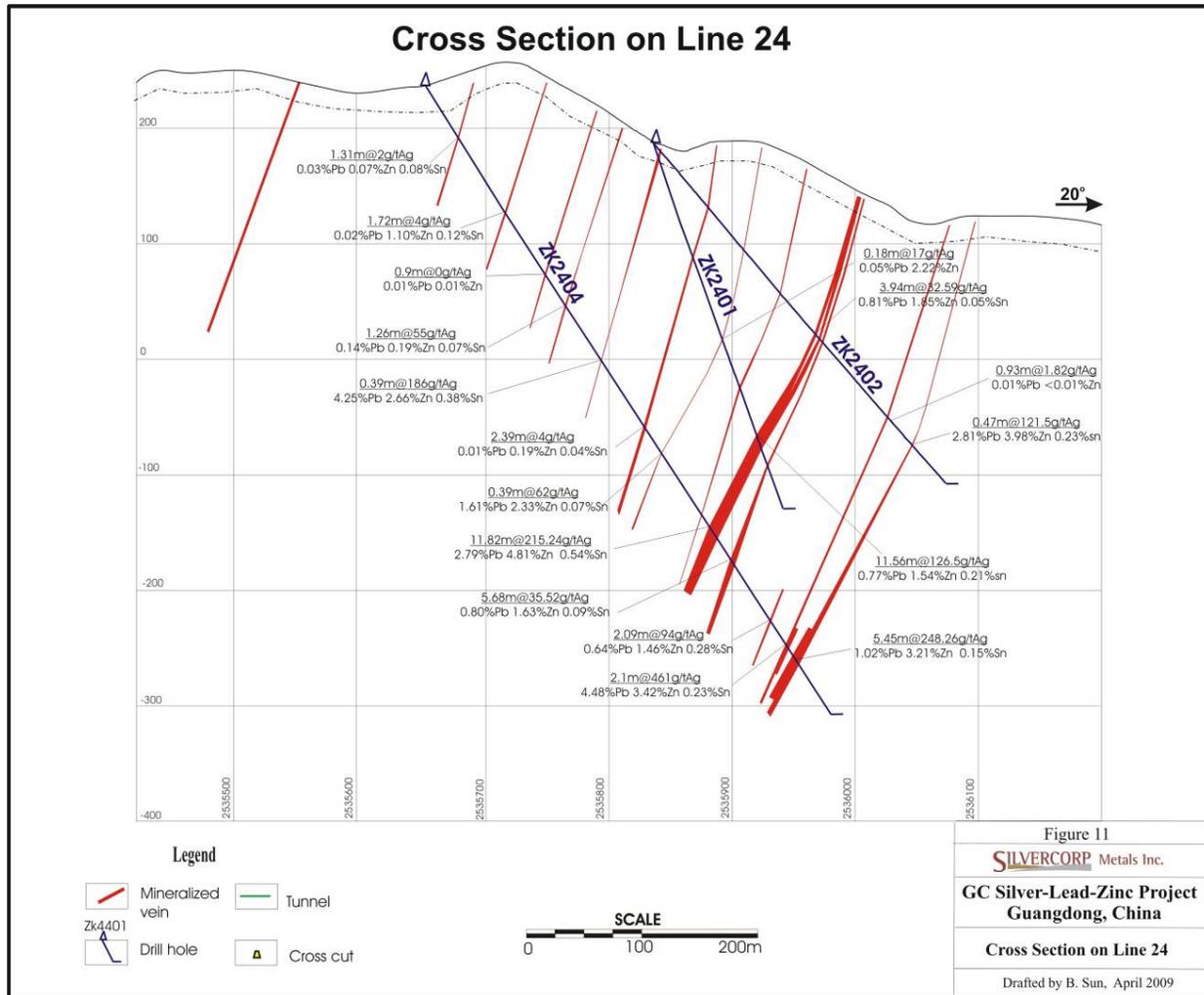


Figure 10.4 Cross Section on Exploration Line 28

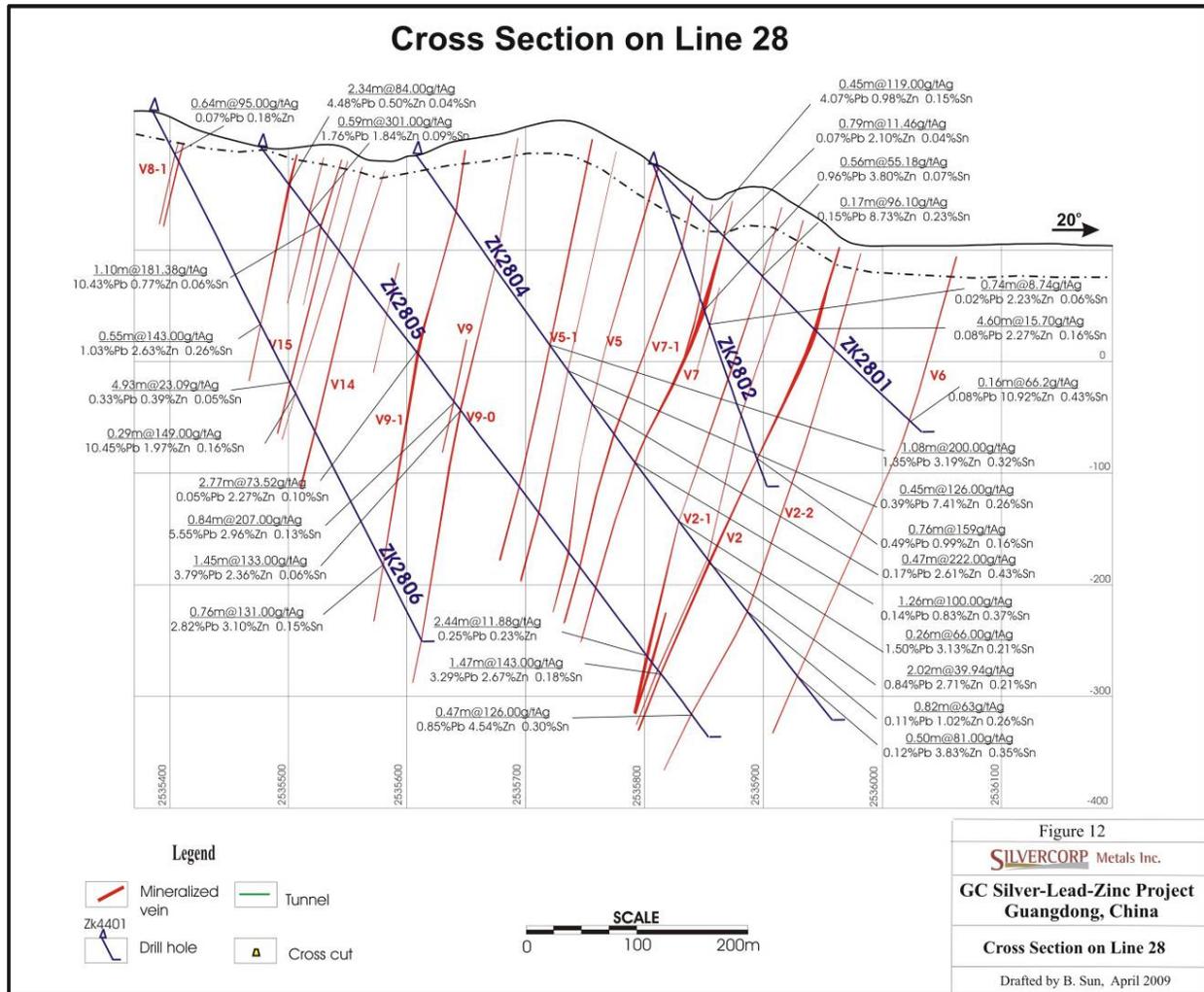


Figure 10.5 Cross Section on Exploration Line 32

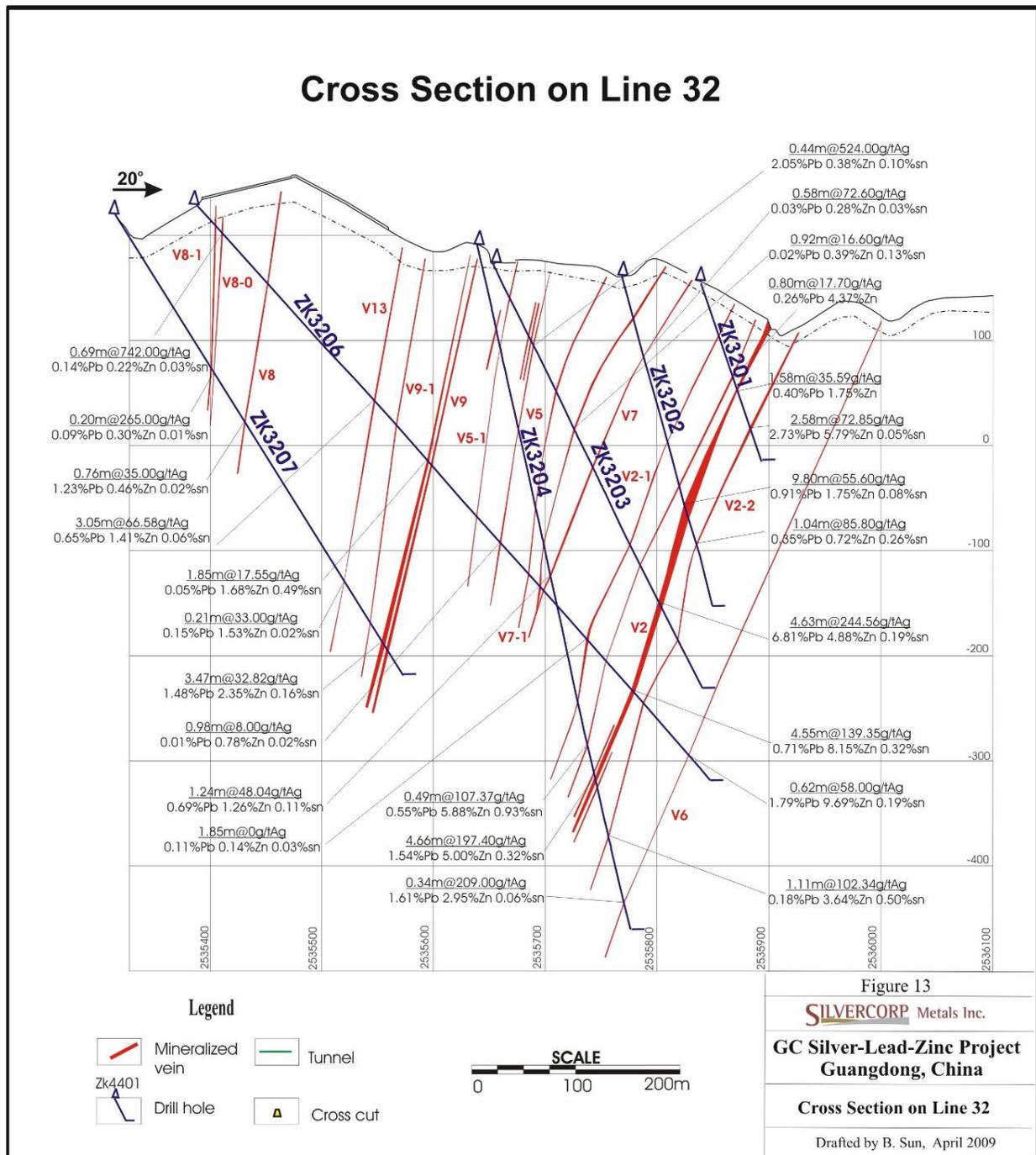
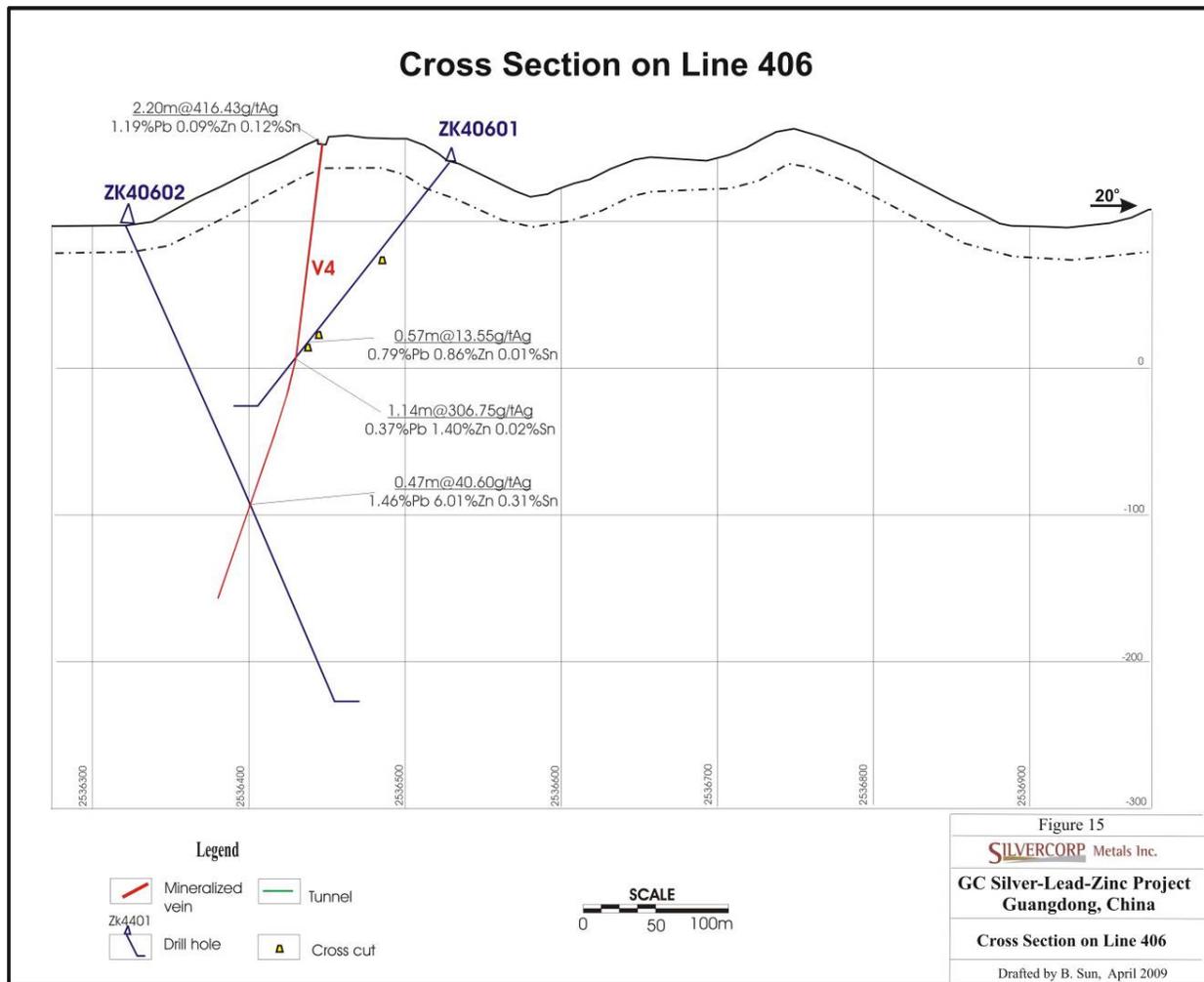


Figure 10.6 Cross Section on Exploration Line 406



## 11 SAMPLING, SAMPLE PREPARATION, ANALYSES AND SECURITY

### 11.1 Sampling

The drill core is logged initially at the drill site and the mineralized or favourably altered intervals are moved to the surface core shack where they are logged, photographed and sampled in detail. Samples are taken prepared by cutting the core in half with a diamond saw. One half of the core is returned to the core box for archival storage, the other half is placed in a labelled cotton bag with the sample number written on the bag. The bagged core sample is then shipped to the laboratory for assaying.

Individual samples from the drill core are from veins that range in width from 0.05m to 12.03m. The veins consist of either massive sulphides or sulphide-bearing materials and can be easily identified and separately sampled from non-mineralized wall rock. Mineralized veins intercepted by drill cores were sampled in 1.5m maximum intervals and the distances cut where warranted by apparent wallrock.

Core recoveries are determined by measuring the actual amount of core recovered versus the length of the drilled interval from which the core was obtained. Core recoveries (calculated as percentage) are documented in the log. In general, the recoveries range from acceptable to excellent; although the recoveries vary somewhat from vein to vein.

Samples appear to have no apparent sampling or recovery difficulties that would affect the reliability of results. The samples appear to be representative and results of check samples show no apparent evidence of sample bias. Rocks sampled trenches, tunnels or in drill core are sulphide-rich veins that follow structures (faults). These veins are easily identified because of their bright metallic sulphides and they can be sampled with little difficulty.

The angle of the vein to core is determined by using the vein to core angles and cross-sectional correlations to determine the dip of the veins. The apparent thickness is then corrected to true thickness using simple trigonometry.

In AMC's opinion, the sampling procedures and controls meet accepted industry standards. In general, the trench, chip and core samples appear to be representative of the areas examined and suitable for use in resource estimation.

**Table 11.1 Individual Vein Characteristics**

Vein #	Thickness		Ag		Pb		Zn	
	Min (m)	Max (m)	Min (g/t)	Max (g/t)	Min (%)	Max (%)	Min (%)	Max (%)
V2	0.24	12.03	3	1110	0.01	13.2	0.01	17.4
V2-0	0.20	2.18	5	550	0.01	11.4	0.05	9.85
V2-1	0.17	4.28	6	780	0.04	2.24	0.2	8.73
V2-2	0.13	6.74	4	968	0.03	3.83	0.01	10.75
V3	0.29	2.55	34	1626	0.04	13.74	0.01	10.74
V4	0.47	3.46	41	584	0.06	1.62	0.04	6.01
V5	0.16	3.38	17	836	0.03	7.15	0.11	12.05
V5-1	0.12	2.58	5	409	0.01	2.62	0.03	21.96
V6	0.16	11.22	11	863	0.03	3.73	0.05	24.31
V6-0	0.16	4.84	6	461	0.02	8.25	0.58	9.14
V7	0.18	5.6	5	261	0.02	6.94	0.18	15.3
V7-0	0.21	2.12	13	670	0.02	8.37	0.02	13.84
V7-1	0.07	5.84	17	1200	0.02	4.62	0.41	10.65
V8	0.55	2.34	35	293	0.02	4.48	0.02	2.63
V8-0	0.20	0.69	265	742	0.09	0.14	0.22	0.3
V8-1	0.46	0.64	95	396	0.07	2.22	0.18	1.14
V9	0.29	3.66	17	285	0.03	4.88	0.23	15.35
V9-0	0.25	3.47	24	207	0.05	5.55	0.88	6.58
V9-1	0.28	0.75	71	273	0.07	3.24	0.84	7.92
V10	1.12	9.86	4	690	0.03	9.23	0.01	11.47
V10-1	0.16	1.53	11	505	0.01	17.51	0.4	23.3
V11	0.80	3.06	21	300	0.48	1.4	0.05	4.96
V13	0.05	1.02	3	489	0.04	1.6	0.05	5.2
V14	0.42	1.08	31	149	0.37	10.45	0.7	2.79
V15	3.81	4.34	29	194	0.16	1.64	0.18	4.42
V15-1	0.59	0.59	301	301	1.76	1.76	1.84	1.84

## 11.2 Sample Preparation, Analysis and Security

Drill core samples were taken from sawn half core for every 1.5m or limited by apparent wall rock and mineralization contact. Half of the core was sent to the laboratory for analysis and the other half retained for archive. The samples are individually secured in sample bags and then collectively secured in rice bags for shipment to the laboratory. Employees of Yangtze Mining collect and split the core for sampling. No officer or director of either Silvercorp or Yangtze Mining has contact with any of these samples prior to shipment to the laboratory.

The samples are shipped directly in security sealed bags to ALS Chemex in Guangzhou, China (Certification ISO 9001:2000), located approximately 180 km southeast of the GC property site.

The sample preparation consists of drying, crushing and splitting of the sample with a riffle splitter to 150g then pulverizing the sample to 200 mesh. Ag, Pb and Zn in drill core samples were analyzed by aqua regia digestion and AAS. The prepared sample is digested in aqua regia (HNO<sub>3</sub>-HCl). After cooling, the resulting solution is diluted with de-ionized water, mixed and then analysed by inductively coupled plasma-atomic emission spectrometry (ICP-AES).

Detection ranges for this method are:

**Table 11.2 Detection Limits, Aqua Regia / AAS**

Element	Symbol	Units	Lower Limit	Upper Limit
Silver	Ag	g/t	1	1500
Lead	Pb	%	0.01	20
Zinc	Zn	%	0.01	60

Soil samples were analysed by aqua regia digestion and ME-ICP.

Tin was analysed by fusing with peroxide, then leaching the melt and acidifying to precipitate out the tin for AAS finish.

Check samples including field duplicates and sample rejects are routinely sent to Laboratory of the Henan Institute of Geological Survey (the "Henan Laboratory"), located in Zhengzhou, Henan Province, Central China. In the Henan Laboratory, lead, zinc, tin, and silver are all analysed with using AAS after a three-hour hot aqua regia digestion on a 30g split of the pulverized portion. A gravimetric finish is done on samples with silver values in excess of 1,500g/t. On samples containing more than 30% lead, an acid dissolution and titration is used to complete the analysis. Henan Laboratory's lower detection limits are 3 g/t for silver, 0.03% for lead and zinc.

Silvercorp's check procedures include (a) inserting purchased standards and blanks that were prepared by Yangtze Mining in the every 40-sample batches submitted to the ALS Chemex Laboratory on a regular basis, (b) submitting duplicate pulps to the ALS Chemex Laboratory on a regular basis, (c) submitting 1/4 core samples as sample duplicates to the ALS Chemex Laboratory for every 40-sample batches, and (d) submitting duplicate pulps to an independent external lab on an intermittent basis.

AMC has reviewed the procedures used by Silvercorp for the preparation, security and assaying of samples and believes that they are adequate and conform to standard industry practices. AMC has also reviewed the QA / QC results and is satisfied that they confirm the acceptability of the assay database. AMC would recommend that standards and blanks be purchased from an independent source than the assay laboratory in future drill programs.

A total of 62 samples were taken for bulk density measurement. The tests were done using the wax-immersion method by Guangdong Material Test Centre, a Chinese government certified lab located in Guangzhou, Guangdong, China. Samples ranged in size from 470 to 2,690g. Based on a cutoff grade of 100g/t AgEq (no recoveries included), the results of 56 samples were used to calculate the average bulk density for each vein on the GC property. The average bulk density is determined to be 3.57t/m<sup>3</sup>. Note that one extreme high grade sample, returning a value of 5.51 t/m<sup>3</sup> and containing 2,793 g/t Ag, 53.04% Pb, 6.44% Zn was excluded from the overall bulk density calculation..

The average grades for these 56 samples are 176 g/t Ag, 1.99% Pb, and 4.47% Zn. In theory, bulk density is related positively to metal contents, especially lead and zinc. However, bulk density is sometimes high in low grade material if the pyrite content is high and it is noted that the GC deposit is rich in pyrite.

Bulk density values were assigned to three different grade groups as shown in Table 11.3.

AMC is satisfied with the procedures and approach taken by Silvercorp to determining bulk density values for resource estimation purposes. Nevertheless, AMC recommends that additional bulk density determinations be done on a regular basis with checks from different independent laboratories.

**Table 11.3 Variation in Bulk Density Measurements**

Grade (Pb+Zn)	≥10%	4-10%	≤4%
No. of Samples	12	17	27
% of samples	21.4%	30.4%	48.2%
SG	3.79	3.73	3.37
Average based on % of samples in different grade group	3.57		

## 12 DATA VERIFICATION

AMC's site visit was conducted by AMC consultants and independent Qualified Persons B O'Connor, O Watson, P Mokos and A Riles in May 2011 (P Mokos re-visited in July-August 2011). All aspects of the project were examined during the visit, including drill core, exploration sites, underground workings, processing plant and surface infrastructure.

For the 2009 Technical Report, B O'Connor conducted a site visit and undertook the following validation tasks. There have been no material changes to the original data since that time.

- Review of the state of geological and mineralization knowledge.
- Tour ALS Chemex facility in Guangzhou. Review of methodology and process flow.
- Visit surface expressions of mineralized veins V4 (ML3), V2 (BT-3), adit to V10 and site ML8 (V2-2).
- Review core logging and processing procedures, chain of custody for samples.
- Data set and compilation review.
- Review of the resource model block estimations.

AMC could not verify the underground workings and sample data at the time of the site visit as the access to the underground workings was barricaded for safety reasons.

AMC requested 34 diamond drill samples from the 2008 drill campaign be quarter split. The quarter split samples were delivered to Eco Tech Laboratory in Kamloops, British Columbia, Canada for analysis. Eco Tech is a fully accredited to ISO 9001:2000 and is currently working toward 17025 accreditation. The results are listed in Table 12.1.

**Table 12.1 Independent Core Sample Verification**

Vein	Hole ID	Sample #	Eco Tech (1/4 Core)			Original (1/2 Core)		
			Ag (g/t)	Pb %	Zn %	Ag (g/t)	Pb %	Zn %
V7-1	ZK2804	71730	160	0.13	2.03	222	0.17	2.61
V2	ZK2804	71772	92.3	1.08	4.60	76.0	1.07	4.10
V15-1	ZK2805	71231	196	1.00	1.31	301	1.76	1.84
V15	ZK2805	71235	204	0.48	0.63	178	0.55	0.64
V14	ZK2805	71246	72.0	1.15	1.95	91.0	1.39	2.79
V13	ZK2805	71305	7.1	0.05	0.06	3.0	0.04	0.05
V9	ZK2805	71389	39.3	0.35	0.46	39.0	0.38	0.65
V8	ZK2806	71864	150	0.68	2.48	143	1.03	2.63
V8-1	ZK3206	71208	346	1.78	1.16	396	2.22	1.14
V8-0	ZK3206	71213	723	0.15	0.32	742	0.14	0.22
V2-2	ZK3206	71636	66.0	0.05	0.14	82.0	0.04	0.14
V6	ZK3206	71673	40.0	0.83	6.22	58.0	1.79	9.69
V8	ZK3207	71945	14.1	0.95	0.20	35.0	1.23	0.46
V9-1	ZK3207	72055	68.7	1.65	2.38	71.0	1.92	2.16

			Eco Tech (1/4 Core)			Original (1/2 Core)		
Vein	Hole ID	Sample #	Ag (g/t)	Pb %	Zn %	Ag (g/t)	Pb %	Zn %
V9-0	ZK3207	72072	44.0	1.10	4.85	59.0	1.79	6.58
V9	ZK3207	72075	144	0.83	10.9	205	0.72	11.8
V2-1	ZK3606	70649	126	0.10	1.15	140	0.18	2.15
V2	ZK3606	70681	24.2	0.53	3.83	26.0	0.55	3.98
V2-2	ZK3606	70710	63.8	0.88	2.10	55.0	0.35	1.88
V5-1	ZK3607	70520	56.6	1.28	2.98	50.0	1.11	3.82
V7-1	ZK3607	70550	25.7	0.59	0.87	57.0	0.83	1.10
V2-1	ZK3607	70588	144	1.60	2.48	113	1.57	2.80
V2	ZK4001	70034	21.3	0.98	2.15	42.0	2.02	4.43
V5-1	ZK4003	70404	62.0	0.60	0.49	47.0	0.85	0.33
V5	ZK4003	70420	183	0.91	2.38	193	1.27	3.51
V10-1	ZK4004	71171	58.3	0.01	4.03	52.0	0.02	6.85
V10	ZK4004	71202	16.4	0.80	0.95	17.0	0.89	0.91
V7-0	ZK4004	71352	14.9	0.23	3.40	18.0	0.26	4.32
V2-0	ZK4004	71476	37.4	1.60	4.25	32.0	1.78	2.17
V5	ZK4402	70789	965	7.54	9.28	836	7.15	12.05
V7-0	ZK4402	71059	52.0	0.70	4.98	54.0	0.67	5.16
V6	ZK4403	70947	116	2.23	1.05	144	3.73	1.04
V7	ZK4805	71654	64.7	1.03	1.73	47.0	0.70	1.41
V2-0	ZK4805	71660	59.2	0.03	5.03	46.0	0.04	4.18
	Minimum Value		7	0.01	0.06	3	0.02	0.05
	Average Value		131	1.00	2.73	137	1.18	3.22
	1st Quartile		39	0.38	0.98	46	0.36	1.06
	Median Value		64	0.83	2.13	59	0.87	2.39
	3rd Quartile		144	1.10	3.98	144	1.71	4.16
	Maximum Value		965	7.54	10.90	836	7.15	12.05
	Standard Deviation		197	1.28	2.50	187	1.33	3.09
	Correlation Coefficient		0.98	0.95	0.95			

In general, the silver values returned from the quartered core analysis were in agreement with the original half core assay results. The lead and zinc values returned from the quarter core analysis were on average elevated with respect to the original half core assays. It is AMC's opinion that there is not a systemic bias in the data for lead and zinc assays, but that the variations seen are a reflection of the distribution of the lead and zinc mineralization within the vein. AMC notes that the quartile analysis for lead was anomalously elevated only in the third quartile and zinc was elevated anomalously in the median (2<sup>nd</sup>) quartile and the maximum (4<sup>th</sup>) quartile. The other quartile results were within a +/- 5 to 10% error range.

AMC's review of the QA / QC data did not reveal any major deficiencies that are likely to have a material impact on the assay results used in the resource database.

Silvercorp's data is stored in digital format, for both internal and external audit purposes, hardcopy output of the raw and interpreted data in the form of tables, plans and sections was readily available.

AMC compared 40% of the original assay certificates with the assays used in the block grade estimates and noted only minor discrepancies. AMC's check of 15% of the grade, area, volume and tonnage calculations did not reveal any major deficiencies that are likely to have a material impact on the resource estimates. AMC recommends that Silvercorp undertake variography studies to refine the understanding of the grade distribution and utilize a kriging or inverse distance weighting approach to grade interpolation prior to future resource and reserve estimations.

AMC did note some discrepancies in the tabled assays on some sections where the lead and zinc assays were inverted. AMC investigated this finding and found that these typographic errors were not incorporated into the resource estimation.

AMC did not take samples from the trenches during the site visit. In AMC's opinion the trench data influence is less than 2% of the Measured and Indicated Resource, thus not material to the viability of the project.

## 13 MINERAL PROCESSING AND METALLURGICAL TESTING

### 13.1 Introduction

The metallurgical testing for the GC project was carried out by the Hunan Research Institute of Non-Ferrous Metals and reported in May 2009 in the report “Development and Research of the Comprehensive Recovery Test of Lead Zinc Silver Tin Sulphur for the Lead Zinc Ore Dressing in GC Mine Area”. This was made available in English translation by Silvercorp. The testwork was summarized in the January 2011 GMADI report as part of the “Design Instructions” for the plant design; however AMC drew on the original Hunan Institute report in preparing this section of the report.

The objectives of the testwork were, following on the previous testwork of 2007 on samples from artisanal mining dumps, to i) maximize silver recovery to the lead concentrate, ii) investigate the potential for tin recovery, iii) develop a process flow sheet with appropriate operating parameters as a basis for the industrial scale implementation of lead, zinc, sulphur (and possibly tin) recovery, iv) determine the product quality characteristics relative to the relevant national standards.

### 13.2 Metallurgical Samples

The mineralization and vein structure has been well-summarized in this report and in AMC's June 2009 Technical Report. The majority of the veins strike west-north-west dipping steeply to the south-west with exception of veins V10 and V10-1 which cross-cut the main vein with a north-easterly strike. Figure 10.1 of this report shows the veins and drillhole locations.

For the purposes of assessing the representativeness of the metallurgical samples it is worth noting the following:

- The samples derive from 152 drillhole intersections drilled along lines 24-48, representing the central main cluster of veins
- The main metal tonnage in the resource is contained in veins V2, V6, V7 and V10, although V6 is not as well represented in the reserve, presumably because of its depth.
- The high grade Ag-Pb-Zn shoots occur at the intersections of the WNW and E-W striking faults.
- The Zn mineralization is more pervasive; Ag and Pb are more locally concentrated but intensive, continuous and wide within the breccias zones of a fault.
- The distribution of the metallurgical samples relative to the reserve tonnages and grades for the major veins are shown in Table 13.1.

**Table 13.1 Metallurgical Samples Relative to GC Reserve**

Vein Complex	Reserve				Metallurgical Sample			
	Wt%	Ag g/t	%Pb	%Zn	Wt%	Ag g/t	%Pb	%Zn
V2	58.1				32.5	145	1.74	3.82
V6	5.3				7.0	487	3.42	6.69
V7	14.5				14.4	81	0.75	2.08
V9	10.8				12.8	114	1.68	4.00
V10	4.9				5.0	103	1.99	7.62
Total	93.6	<b>114</b>	<b>1.28</b>	<b>2.85</b>	71.7	<b>147</b>	<b>1.42</b>	<b>3.26</b>

As can be seen from Table 13.1 the metallurgical sample grades approximate closely to the average reserve grades and although V2 is somewhat under-represented, the weight distribution of the samples follows fairly closely the distribution by weight of the main veins that make up the mining reserve. And as V2 grade is close to the reserve average in any case, AMC does not consider the under-representation to be a critical issue.

### 13.3 Mineralogy

The sulphide mineralization is typical of mesothermal silver-lead-zinc-quartz-pyrite veins and has been described in general terms by O'Connor (2009) and in this report. However AMC notes that the sphalerite is described as having very fine inclusions of chalcopyrite; this "diseased" sphalerite will promote general sphalerite flotation and inhibit selectivity against it in the lead (and copper) flotation.

The main focus of the Hunan Research Institute mineralogical work was on the silver deportment, rightly so given the importance of silver revenue, especially in the first 2-3 years where a combination of high grades and high prices means that silver constitutes almost 60% of the total revenue.

The occurrence of silver is in three main forms, as summarized below together with the approximate weight distribution:

- Elemental silver, 23%
- Silver sulphides e.g. acanthite and argentite (both Ag<sub>2</sub>S), 41%
- Silver in sulphides i.e. as solid solution or inclusions, 33%

In liberation terms, the principal elemental silver and silver sulphide associations are 13% free, 40% with galena, 14% with sphalerite and 30% with other sulphides, mainly pyrite, also pyrrhotite and arsenopyrite.

Of the silver in sulphides the occurrence is summarized in Table 13.2:

**Table 13.2 Silver Associations**

Mineral	Wt%	Ag g/t	Ag % distribution
Galena	1.6	2897	46.2
Sphalerite	4.8	352	16.8
Other sulphides (pyrite, pyrrhotite, arsenopyrite)	16.7	207	34.5

These mineralogical results with the silver spread across the sphalerite and pyrite as well as the galena have implications for the expected metallurgical performance. As silver is only paid in the lead concentrates, it will probably be necessary to compromise lead concentrate grades (and zinc recovery) in order to maximize silver recovery to a payable (i.e. lead) concentrate. This is dealt with further in Section 13.4 in consideration of optimization of the process flowsheet. Also AMC considers that the presence of elemental silver and silver sulphides would benefit from the use of a precious metal specific collector like a dithiophosphinate (e.g. Cytec 3418A) in addition to the standard dithiophosphate collectors used in this current round of testwork.

The tin mineralogy is dominated by cassiterite (75%) with minor amounts of stannite (14%), tin in silicates (6%) and colloidal tin (5%). However the granulometric distribution of the tin is very fine (<75 $\mu$ ) which does not augur well for effective gravity concentration.

### 13.4 Metallurgical Testwork

AMC notes that no comminution testwork has been carried out so there is no work index data or similar available for grinding circuit design. This is further discussed in Section 17.

The prime focus of the flotation testwork was on lead (and therefore silver) recovery and both open circuit and closed circuit flotation tests were conducted to derive the final metallurgical performance predictions in line with normal practice. Some investigations into copper-lead separation and tin recovery were also carried out.

#### 13.4.1 Lead Flotation Conditions

A series of rougher-scavenger flotation tests were performed to determine the optimum grind size, collector selection and dosage, and modifier regime. These were followed by kinetic rougher tests to determine the flotation residence time required.

Initial tests on various grind sizes ranging from 65% passing 75 $\mu$  to 90% passing 75 $\mu$  showed that based on lead recovery and the silver grade in the lead concentrate, the optimum grind size was 80% passing 75 $\mu$ . AMC notes however that silver recovery was still increasing at finer sizes and investigations into regrinding the rougher concentrate may be warranted.

The basic chemical regime selected was based on lime for pH adjustment and pyrite depression with a combination of zinc sulphate and sodium sulphite for depressing sphalerite and a modest dosage of sodium sulphide to enhance flotation of any oxidized ores (note that an excess of sodium sulphide depressed lead). The use of cyanide in combination with zinc sulphate, the preferred combination in western complex sulphide flotation plants for sphalerite and pyrite depression, was not considered for environmental reasons.

Optimum rougher dosages were found to be:

- Lime 2000 g/t
- Sodium sulphide 500 g/t
- Zinc sulphate 1000 g/t
- Sodium sulphite 500 g/t

Based on this regime, investigations were carried out into the collector type and dosage from which it was concluded that the best results in terms of lead and silver recovery was a combination of a dithiocarbamate (AMC notes that this is more usually used for selective copper flotation) and a dithiophosphate at 25 g/t and 10 g/t respectively. AMC also notes that no tests were carried out with a precious metals specific collector of the type previously mentioned and considers this to be an improvement opportunity.

Conditions for cleaner flotation were determined to be 700 g/t lime and 400 g/t zinc sulphate with no further additions of sodium sulphide or sodium sulphite.

The kinetic rougher tests showed that a laboratory flotation time of 5 minutes was required (subject to the usual scale-up factors for industrial design).

#### **13.4.2 Zinc and Pyrite Flotation Conditions**

Only limited testwork on zinc and pyrite flotation was carried out, based on the 2007 testwork and industry practice of copper sulphate as an activator and sodium iso-butyl xanthate (SIBX) under alkaline conditions as the collector for zinc flotation. This was followed by lowering the pH to 8 with sulphuric acid and flotation of the pyrite with more SIBX.

Acceptable zinc concentrate grades (52% Zn) at reasonable open-circuit recoveries and high pyrite recoveries were achieved.

#### **13.4.3 Sulphide Circuit Flotation Tests**

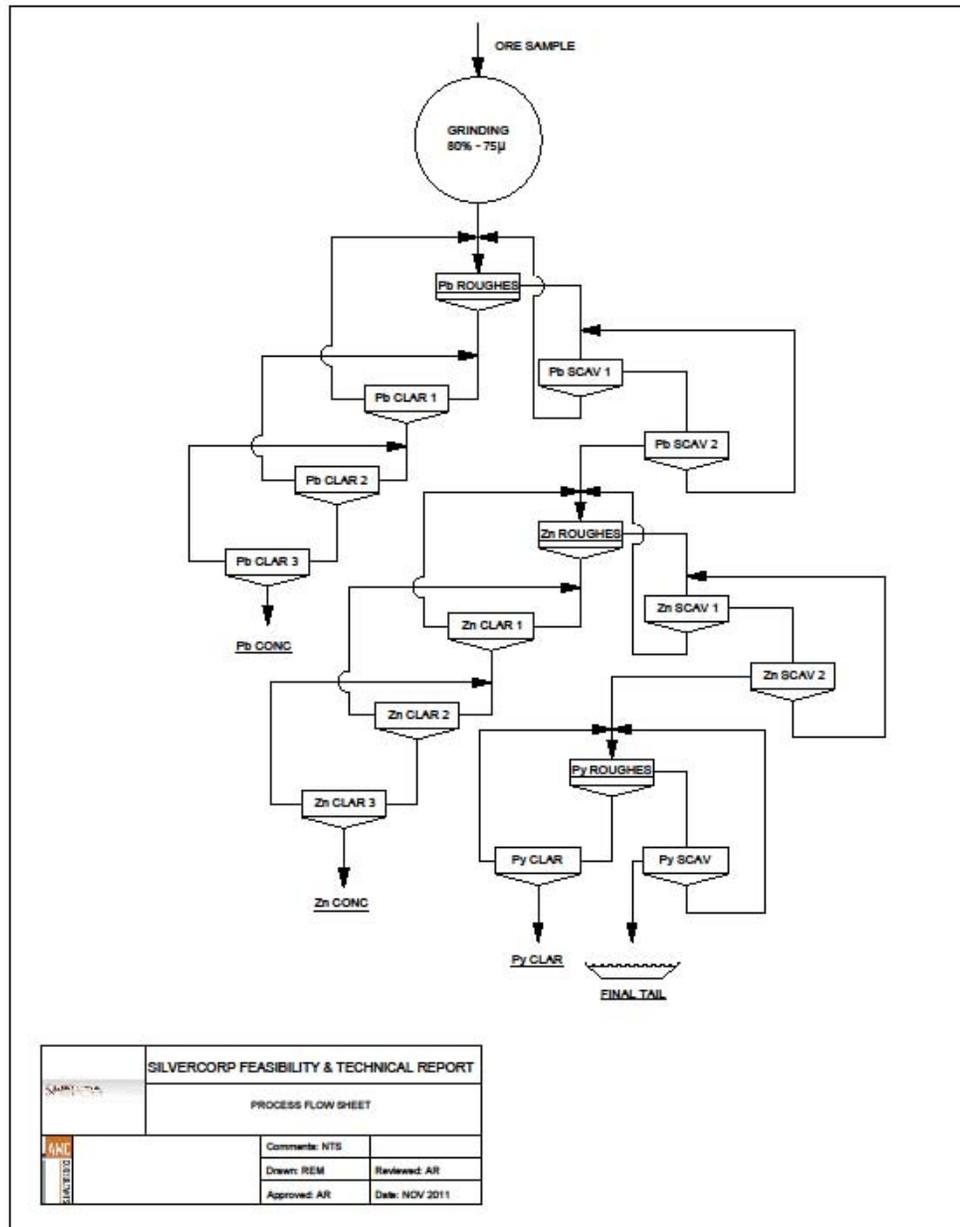
Based on these conditions established for lead, zinc and pyrite flotation and with three stages of cleaning for lead and zinc flotation and one cleaner for pyrite flotation, a full open-circuit test of sulphide minerals flotation was conducted, as a proof of concept of the overall circuit.

A 48% Pb concentrate at 72% recovery was achieved; zinc recovery was lower (49%) to a 52% Zn concentrate with a substantial amount of the zinc tied up in lead circuit cleaner tails and scavenger concentrates that in practice would be recycled. The remainder of the sulphur was largely recovered to a 48% S pyrite concentrate.

This test demonstrated that sulphide flotation to saleable lead and zinc concentrates at acceptable (for batch tests) recoveries and that a high recovery of the balance of the sulphur to a pyrite concentrate was also possible.

Determination of likely recoveries in an actual industrial scale flotation plant with recycling of intermediate "middlings" streams such as cleaner tails and scavenger concentrates requires closed circuit flotation testing. This was carried out according to the flowsheet shown in Figure 13.1.

Figure 13.1 Closed Circuit Flotation Test Flowsheet



It is not clear to AMC from the testwork data and report the extent to which this closed circuit test approached the locked cycle test standards commonly used in western laboratories. There is no information on how many cycles were performed (the usual minimum is six) and whether circuit stability was in fact achieved as the circulating loads of middlings approached a sort of equilibrium. The most likely consequence of not attaining equilibrium is that concentrate grades may be over-estimated and recoveries under-estimated.

Notwithstanding this, the results as presented in Table 13.3 appear reasonable and in accord with expectations from the mineralogy and experience of similar ores. These results constitute the design basis for the flowsheet and the financial model.

**Table 13.3 Closed Circuit Flotation Test Results**

Product	Wt %	Grades				Recoveries			
		% Pb	% Zn	% S	Ag g/t	Pb	Zn	S	Ag
Pb Conc	2.63	46.35	9.53		3009	84.7	7.74		62.8
Zn Conc	5.84	0.92	48.95		268	3.73	88.2		12.4
Pyrite Conc	14.65	0.81	0.41	42.52	190	8.24	1.85	61.3	22.1
Tailings	76.88	0.06	0.09	0.53	4.5	3.38	2.18	4.01	2.74
Feed	100.0	1.44	3.24	10.16	126	100.0	100.0	100.0	100.0

#### 13.4.4 Copper-Lead Separation Tests

The closed circuit test produced a lead concentrate with 3% Cu; accordingly some preliminary investigations were carried out into producing separate copper and lead concentrates. No details of the experimental conditions are available but the results showed that an 18.5% copper concentrate was produced at 67.6% recovery, but with 7.2% Pb and 16.6% Zn, not attractive to a smelter. The lead concentrate assayed 57.2% Pb at 89% recovery, with negligible copper levels. There is no information on silver deportment.

It appears that no further work has been done and no consideration given to incorporating a copper recovery circuit; AMC would consider that reasonable, given the results.

#### 13.4.5 Tin Recovery Tests

Despite the low tin head grade and fine size distribution of the tin previously referred to, an extensive series of tests to recover tin were performed.

Attempts to produce a saleable grade tin concentrate through either froth flotation or centrifugal "high-g" gravity devices were unsuccessful.

Finally a concentrate of >50% Sn was obtained by spiral concentration followed by tabling of sized streams of the spiral concentrate then froth flotation on the table concentrate to remove the sulphides also concentrated by the gravity processes. Overall tin recovery from these batch tests was of the order of 30%. It was estimated that in a closed circuit tin recovery to a saleable concentrate would be 37%.

AMC considers that the concentrate grade is still relatively low but that if an appropriate smelter customer can be found a tin recovery circuit could in fact be potentially economically viable.

#### 13.4.6 Optimization Opportunities

It is AMC's experience with silver-lead-zinc concentrates that the optimum grade-recovery point for the lead concentrate is driven by maximizing silver recovery and is often at a surprisingly low lead grade. This is particularly so in a fully integrated mine-concentrator-smelter operation and at currently high silver prices.

In the case of GC selling concentrates under commercial terms then there are constraints on the minimum concentrate grade acceptable to the customer. Nevertheless AMC has carried out a preliminary assessment of the optimization opportunity for moving to a lower grade point on the grade-recovery curve based on the following:

- Analysis of the open-circuit flotation test results to derive grade-recovery data for the lead concentrate and lead-zinc selectivity
- Polynomial curve-fitting of this data to derive predictive formulae (relying on interpolation only, not extrapolation)
- Estimation from the flotation test data covering a range of lead and silver grades of the silver content of the lead concentrate at various lead grades (approx. 65g/t Ag per 1% Pb)
- Calculation of the concentrate value at a range of concentrate grades from 30% to 60% Pb allowing for:
  - Incremental increases in silver recovery with increasing lead recovery (and lower lead grade)
  - Declining lead and silver payables as grade falls
  - Zinc losses from the zinc concentrate to the lead concentrate (Zn non-payable).
- No allowance has been made for incremental transport costs (no data available).

The results are summarized in Figures 13.2 and 13.3 below where two silver prices are considered, the long-term of US\$18/oz and the current high, conservatively set at US\$30/oz. Note that in both cases the long-term lead and zinc prices of US\$1.00/lb are used.

**Figure 13.2 Concentrate Value vs %Pb (Ag \$18/oz)**

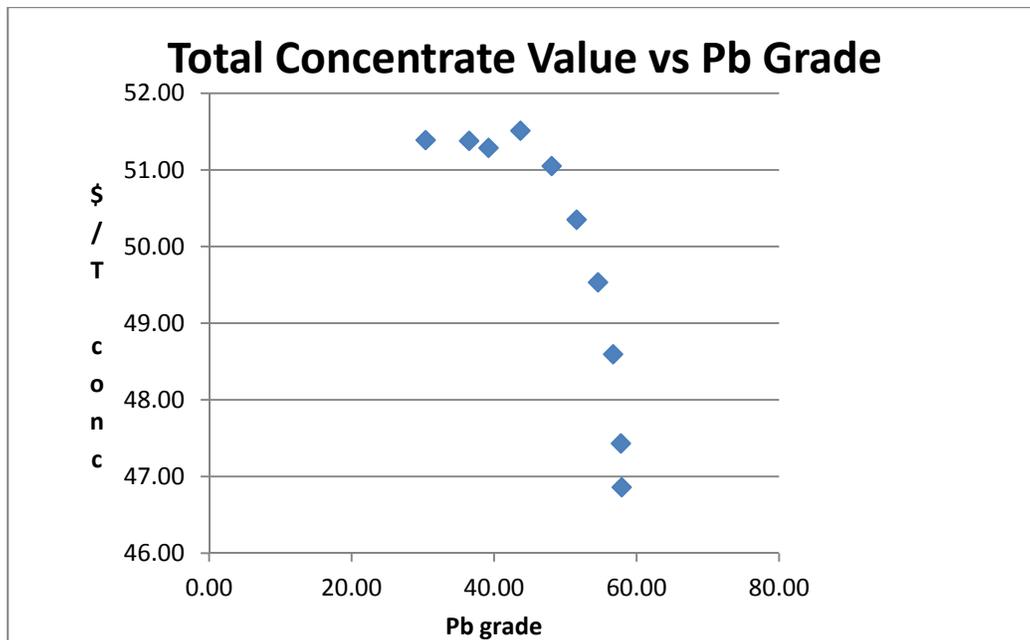
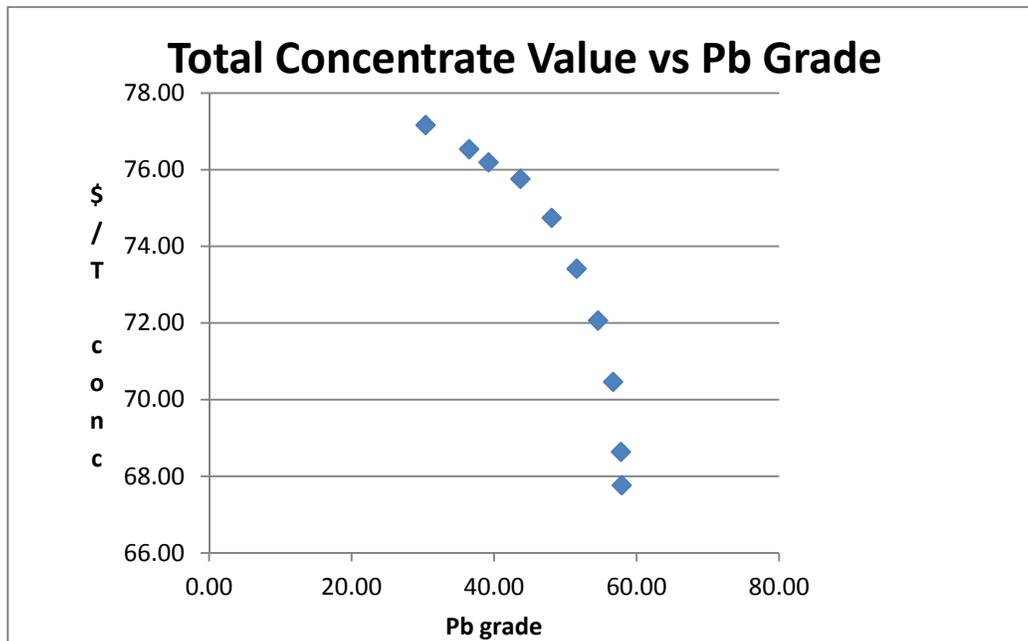


Figure 13.3 Concentrate Value vs %Pb (Ag \$30/oz)



It is clear from the above graphs, and accepting the preliminary nature of this evaluation, that the optimum grade-recovery point is sensitive to silver prices. At the long-term price, the current strategy targeting a lead concentrate grade of 46% Pb is close to the optimum of around 43%; however at the current high silver prices of around US\$30/oz, it makes more sense to pursue a strategy of maximizing lead and silver recovery, at the expense of zinc recovery and, within smelter contract constraints (min 35% Pb grade), lead concentrate grade.

Clearly this analysis requires validation with transport cost inputs and with operating data once the plant is running but AMC believes that this optimization opportunity should be pursued and that appropriate smelter contracts should be negotiated, if possible, to enable this.

### 13.5 Concentrate Quality Considerations

The main issues highlighted by GMADI with respect to concentrate quality relative to the national standards are:

- Copper and zinc levels in the lead concentrate (3% and 9% respectively) which AMC considers to be more of a commercial issue rather than a material quality issue, as discussed in Section 19.
- Arsenic levels in the zinc concentrate (0.57% As) which exceed the 0.4% As level for an otherwise clean grade 3 concentrate
- Arsenic levels in the pyrite concentrate (1.15% As) which exceed the 0.07% As level in the top category (grade 1) of the standard.

AMC considers the only potential quality issue is arsenic and this is discussed further in Section 19.

## 13.6 Summary of Testwork Outcomes

The key outcomes of the metallurgical testwork are as follows:

### 13.6.1 Metallurgical Samples

The metallurgical samples are adequately representative of the main part of the ore body and of the reserves.

### 13.6.2 Mineralogy

The mineralogy is more challenging than Silvercorp's Ying mine in Henan province, mainly because the silver content is more widely spread across the mineral suite i.e. in the sphalerite and pyrite as well as being of payable content in association with the galena

### 13.6.3 Laboratory Testwork

#### 13.6.3.1 Grinding

No grinding testwork has been carried out. Although this would normally be a standard inclusion in any feasibility study, AMC has made adequate allowance to compensate for this deficiency in the process design section.

#### 13.6.3.2 Flotation

- Batch flotation tests have established a workable set of flotation conditions and reagents although an opportunity exists to pursue the use of a precious metal specific collector like Cytex A3418A
- Closed circuit flotation tests have derived reasonable predictions of concentrate grades and recoveries, as summarized in Table 13.4

**Table 13.4 Closed Circuit Flotation Test Results**

Product	Wt %	Grades				Recoveries			
		% Pb	% Zn	% S	Ag g/t	Pb	Zn	S	Ag
Pb Conc	2.63	46.35	9.53		3009	84.7	7.74		62.8
Zn Conc	5.84	0.92	48.95		268	3.73	88.2		12.4
Pyrite Conc	14.65	0.81	0.41	42.52	190	8.24	1.85	61.3	22.1
Tailings	76.88	0.06	0.09	0.53	4.5	3.38	2.18	4.01	2.74
Feed	100.0	1.44	3.24	10.16	126	100.0	100.0	100.0	100.0

Copper-lead separation is not commercially viable.

#### 13.6.3.3 Tin Recovery

Despite the fine grain size and resulting low gravity recoveries, a tin recovery circuit appended to the end of the main circuit would be low cost and potentially viable.

#### **13.6.4 Optimization Opportunities**

Particularly in the current high silver price environment, attention should be paid to increasing further the silver recovery even at the expense of a lower %Pb concentrate grade and the smelter contracts negotiated accordingly

#### **13.6.5 Concentrate Quality**

Copper and zinc levels in the lead concentrate are a commercial rather than a material quality issue; however arsenic levels in the lead and zinc concentrates are potentially material and, as discussed in Section 19, merit further investigation.

## 14 MINERAL RESOURCE ESTIMATES

The mineral resource categories used in this report are those established by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) in the CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines (CIM Standards) as adopted by the CIM Council dated December 2005.

Mineralization in the GC project consists of narrow vein type deposits which occur as discrete planes of variable grade and variable thickness. The resources were outlined using polygonal methods on longitudinal sections constructed for each vein. The resource estimates reported herein were prepared using such methods by Mr. Wang Qiang, Chief Geologist of Yangtze, and Mr. Myles J. Gao, P. Geo, President & Chief Operating Officer of Silvercorp, who is a Qualified Person, as defined by NI 43-101. B O'Connor of AMC has reviewed Silvercorp's methodologies and data used to prepare the resource estimates and is satisfied that they comply with reasonable industry practice, subject to a qualification with respect to use of the polygonal method. Although this is a common estimation method in China and its use by Silvercorp therefore accords with common industry practice in that country, the technique tends to produce estimates that are higher in grade and lower in tonnage than methods in common use in Canada, such as kriging or inverse distance weighting.

Following is an explanation with comments regarding the parameters and assumptions used to prepare the resource estimations reported in this Technical Report:

- A polygonal block model was used in this resource estimation.
- The polygonal block model utilizes detailed long-sections constructed for each of the veins. The topographic control for these sections is taken from 1:2,000 topographic map.
- Polygonal resource blocks drawn on long-sections of the vein were constructed, and their areas measured, using MapGIS, a MapInfo-like GIS software application widely used in China.
- Sulphide resources are estimated using only the assays obtained from drilling and historical tunnelling. A small portion of samples (41 assays) from 17 surface trenches were used for the oxide block resource estimates. Channel samples from tunnels were taken by GIGS from 2003 to 2005. Yangtze Mining performed a check by re-sampling the channels and found the GIGS results were reliable.
- The minimum cut-off thickness used for mineralization is 0.20m. Although this is relatively narrow, the rescue mining method employed by Silvercorp at the mine makes it feasible to extract veins of this thickness. Also, only around 5% of resource blocks have a thickness between 0.2m and 0.3m (the mineral reserve minimum mining width and Silvercorp has experience at mining to such widths).
- The veins are polymetallic containing several payable metals. Although contents of each of the payable metals are separately reported in the resource estimations, Silvercorp uses a "recovered equivalent-silver"<sup>1</sup> (AgEq Recovered) value to assess and compare the vein resources. The formula and metal prices / metallurgical recoveries used are the same as those used for the mineral reserves and are shown in Section 15.3.

---

<sup>1</sup> The term "recovered" is used because the formula takes into account metallurgical recoveries

- Potentially payable tin and sulphur concentrates have not been included in the silver equivalent calculation.
- Refinery costs have not been included in the silver equivalent calculation.
- Metal prices used in this report are the median prices from selected technical reports on similar deposit types filed on SEDAR between November 2008 and April 2009 (see footnote).

A top-cut has been applied to silver, zinc, and lead assays. Values of the top-cuts for each of the veins and commodities are listed in Table 14.1.

**Table 14.1 Top-cuts of different veins**

Vein #	Top cuts			No. of Assays Exceeding Top-cuts		
	Ag (g/t)	Pb (%)	Zn (%)	Ag	Pb	Zn
V2	968	10.94	22.30	1	2	0
V2-0	660	9.96	18.89	0	1	0
V2-1	697	3.59	9.93	1	0	0
V2-2	421	5.92	18.03	1	0	0
V3	1318	40.30	36.56	1	0	0
V4	1444	6.49	14.43	0	0	0
V5	1453	8.28	21.53	0	0	0
V5-1	472	3.49	14.03	0	0	1
V6	840	5.32	19.17	1	0	1
V6-0	1123	11.80	15.33	0	0	0
V7	419	6.71	20.88	0	1	0
V7-0	678	5.56	21.93	0	1	0
V7-1	483	5.42	16.75	1	0	0
V8	669	14.88	4.25	0	0	0
V8-0	3329	0.78	1.85	0	0	0
V8-1	1314	5.13	3.33	0	0	0
V9	675	7.54	23.91	0	0	0
V9-0	353	12.89	17.50	0	0	0
V9-1	927	14.30	25.74	0	0	0
V10	1116	10.77	8.34	0	0	1
V10-1	475	7.29	44.44	1	2	0
V11	790	5.66	20.17	0	0	0
V13	781	5.84	13.83	0	0	0
V14	472	19.51	9.37	0	0	0
V15	396	3.53	4.69	0	0	0
V15-1	2107	12.32	12.88	0	0	0
<b>Total</b>				<b>7</b>	<b>7</b>	<b>3</b>

- No dilution has been applied with the exception of the 11 individual resource block occurrences below 0.20m in horizontal width in the dataset used for the resource estimate. Those 11 occurrences were diluted at zero grade to 0.20m in horizontal width.
- Any interpolations are based upon vein thickness and grade.
- The wall rock surrounding the veins is in sharp contact with the veins and commonly silicified.
- The data and methods employed are adequate to allow resources to be categorized as Measured, Indicated and Inferred.
- Resource blocks categorized as “Measured” are defined by assays from tunnel samples on vein and drill holes samples. These blocks are projected up to 25m above and below a given tunnel where warranted, and along strike from a given tunnel intersection or projected from a drill hole intercept within 50m of a tunnel sample.
- Resource blocks categorized as “Indicated” begin either above or below a Measured Resource block or are projected from a drill intercept. For blocks projected from the Measured Resource blocks, the distances are not greater than 50m. For blocks projected from drill holes, the distances are not greater than 50 to 60m. Block boundaries are defined as the midpoint between drill holes.
- Resource blocks categorized as “Inferred” use grades and thicknesses derived from the average of all the Measured and Indicated blocks along the vein. For veins intersected by deep holes, the Inferred Resource blocks are projected 100m down-dip from the Indicated blocks.

#### 14.1 Resource Data and Statistics

The information used to estimate project resources is maintained in a series of linked Excel worksheets maintained for all exploration areas. The worksheets contain individual sample information such as sampling dates, locations, sample number, elevation, width, and assay results, and additionally, for drill holes, collar information, downhole survey data, sample intervals, and assay results.

There are 5,280 samples in the database. 740 of these samples are used to define 26 veins in which 492 samples are used for resource estimates (Table 14.2).

**Table 14.2 Number of Samples and Assays in Resource Estimates for Different Veins**

Vein #	No. of channel samples	No. of drill hole Samples	No. of channel and drill holes samples	No. of assays for each metal			
				Ag	Pb	Zn	Sn
V2	25	120	145	145	145	145	132
V2-0	0	17	17	17	17	17	15
V2-1	1	33	34	34	34	34	34
V2-2	5	33	38	38	38	38	36
V3	4	1	5	5	5	5	4
V4	9	3	12	12	12	12	12
V5	2	12	14	14	14	14	13
V5-1	3	11	14	14	14	14	12

Vein #	No. of channel samples	No. of drill hole Samples	No. of channel and drill holes samples	No. of assays for each metal			
V6	4	25	29	29	29	29	27
V6-0	0	12	12	12	12	12	12
V7	1	39	40	40	40	40	40
V7-0	0	20	20	20	20	20	20
V7-1	0	21	21	21	21	21	21
V8	2	3	5	5	5	5	5
V8-0	0	2	2	2	2	2	2
V8-1	0	2	2	2	2	2	2
V9	0	11	11	11	11	11	11
V9-0	0	5	5	5	5	5	5
V9-1	0	4	4	4	4	4	4
V10	11	13	24	24	24	24	14
V10-1	0	14	14	14	14	14	14
V11	2	1	3	3	3	3	2
V13	0	7	7	7	7	7	7
V14	0	3	3	3	3	3	3
V15	0	10	10	10	10	10	10
<b>Total</b>	<b>69</b>	<b>422</b>	<b>491</b>	<b>491</b>	<b>491</b>	<b>491</b>	<b>457</b>

## 14.2 Resource Estimates

The Ag-Zn-Pb metals are reported separately in the resource estimates (Table 14.3). The resources at a cut-off grade of 150 AgEq Recovered are also shown to enable a comparison with the June 2009 resources (see Table 6.5 in Section 6). The reduction in cut-off grade from 150 g/t to 100 g/t AgEq Recovered arises from the application of updated metal prices and metallurgical recoveries.

AMC is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues that would materially affect the mineral resource estimate in Table 14.4. The GC project contains two additional commodities that have not been incorporated into the silver equivalent calculation. The first is sulphur which would take the form, after processing, of a sulphur concentrate containing silver. The silver in the sulphur concentrate has been accounted for in the silver equivalent equation. The second commodity that has not been accounted for in the silver equivalent equation is tin. Silvercorp is researching the potential to produce tin as a saleable product but at the time of the production of the resource estimate the issue remained undetermined.

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

**Table 14.3 Mineral Resources at 100g/t and 150 g/t Recovered Silver Equivalent Cut-off Grades**

Measured and Indicated Resources	Tonnes	Ag (g)	Pb %	Zn %
100g/t Recovered AgEq Cut-off	<b>7,632,000</b>	<b>122</b>	<b>1.32</b>	<b>3.08</b>
150g/t Recovered AgEq Cut-off	5,812,400	144	1.50	3.50

Rounding of some figures may lead to minor discrepancies in some totals

The estimated mineral resources for the 26 veins of the GC project are summarized in the following table using the 100 g/t AgEq Recovered cut-off. Note the subtotals have been rounded and may not sum to the totals due to the rounding.

**Table 14.4 Mineral Resources 100g/t Recovered Silver Equivalent Cut-off Grade**

Resource Classification	Tonnes	Grade			Contained Metal		
		Ag (g/t)	Pb %	Zn %	Ag (kg)	Pb (t)	Zn (t)
Measured	592,800	230	1.41	3.33	136,600	8,400	19,800
Indicated	7,038,700	113	1.31	3.06	797,700	92,500	215,500
<b>Total</b>	<b>7,631,500</b>	<b>122</b>	<b>1.32</b>	<b>3.08</b>	<b>934,300</b>	<b>100,900</b>	<b>235,300</b>
Inferred	7,959,800	123	1.41	2.66	976,800	112,500	211,900

Metal prices used: silver US\$18.00/troy oz, lead US\$1.00/lb, zinc US\$1.00/lb

Inclusive of resources converted to mineral reserves

Lower cut-off grade, 100 g/t AgEq Recovered

Rounding of some figures may lead to minor discrepancies in some totals

The differences between the 2011 mineral resources and the 2009 mineral resources is due to updated metal prices and metallurgical recoveries and to a lower cut-off grade in 2011.

## 15 MINERAL RESERVE ESTIMATES

### 15.1 Introduction

The mineral reserve estimates are the conversion of the mineral resource estimates above a nominated cut-off after applying mining modifying factors such as dilution and losses.

Mineral reserve estimates are based on employing highly-selective stoping methods.

The resource footprint area is approximately 1.2 km west-east and 0.6 km south-north.

The mineral reserve includes 21 veins that are categorized as Measured and Indicated Resources and are named as; V2, V2-0, V2-1, V2-2, V5, V5-1, V6, V6-0, V7, V7-0, V7-1, V8, V8-0, V9, V9-0, V9-1, V10, V10-1, V11, V13 and V14. The veins have a general east-west orientation (dipping generally south) with the exceptions of V10, V10-1 and V11 which have a general north-south orientation (generally dipping east). Veins generally dip at 60° – 80°. The main vein is V2.

The following Measured and Indicated Resources are excluded from the mine plan:

- Resource above +100 mRL (to maintain a surface crown pillar).
- Resource below -300 mRL (being the extent of the Silvercorp mine design to date).
- Resource with averaged grade below 150 g/t recovered AgEq.
- Resource without continuity between two mine levels and potential stoping tonnage less than 1,431t for Stage 1 and 1,646t for Stage 2 (refer to Section 16.2 for a detailed description of the various mine stages). The basis is breakeven tonnage to pay for stope access.
- Resource with minor strike extensions between levels and/or immediately below a level that is not practical to access.
- Resource within pillars for the Main Shaft, Ramp, Hashui Creek (east end of V2, west end of V6) and GC village.

Overall, the study work undertaken to date meets PFS study levels, justifying the conversion of mineral resources to mineral reserves.

### 15.2 Resource Extraction Limits

The underground lease boundary limit for regulatory resource extraction is summarized in Table 15.1 and is valid for a 30 year term. AMC's review confirms the mine design is well within the underground extraction limits. The surface mining-lease rights boundaries are negotiated with the various land owners and Silvercorp surface plans show these to cover the appropriate mining areas.

**Table 15.1 Resource Extraction Boundary Limits**

Boundary Point	Easting	Northing
1	2,536,958.82	37,591,830.45
2	2,536,977.34	37,594,822.59
3	2,535,131.42	37,594,834.19
4	2,535,112.90	37,591,841.69
Depth	-540 mRL	

### 15.3 Cut-off

The mineral reserve has been estimated using a 135 g/t AgEq Recovered cut-off grade, which is equivalent to the operating breakeven. The basis for this is summarized in Table 15.2. AMC believes this cut-off is not likely to be optimal to maximize project value.

The cut-off used is specified by the following.

$$\begin{aligned} \text{Cut-off grade AgEq (g/t)} = & (\text{mining cost} \\ & + \text{milling cost} \\ & + \text{sustaining capital} \\ & + \text{environmental cost} \\ & + \text{G\&A cost} + \text{selling cost}) / (\text{Ag recovery} * \text{Ag price}) \end{aligned}$$

**Table 15.2 Mineral Reserve Cut-off Estimate**

Cut-off Estimate	Unit	Unit Value
Foreign Exchange Rate	RMB:US\$	6.35
Contract Development Cost	US\$/t-ore	12.13
Contract Stopping Cost	US\$/t-ore	7.04
Silvercorp Mine Labour Cost	US\$/t-ore	2.04
Mine Sustaining Capital Cost	US\$/t-ore	0.52
Process Cost	US\$/t-ore	17.83
Process Sustaining Capital Cost	US\$/t-ore	0.72
Tailings & Environmental Cost	US\$/t-ore	0.50
G&A and Product Selling Cost	US\$/t-ore	8.32
<b>Total Costs</b>	<b>US\$/t-ore</b>	<b>49.10</b>
Ag Metal Price	US\$/oz	18.00
Ag Metal Price	US\$/gm	0.58
Ag Recovery (payable)	%	62.78
Revenue (after recovery)	US\$/gm	0.36
<b>AgEq Cut-off (recovered)</b>	<b>AgEq</b>	<b>135.1</b>

Note: ^ For equipment replacement and/or refurbishment cost only.

Vein development that occurs within the mineral reserve is subjected to 45 g/t AgEq Recovered cut-off to categorize the vein development as either vein ore or vein waste. Vein waste is

assigned zero grade. The development cut-off is based on RMB113.56/t-ore process cost, US\$12.00/oz silver metal and RMB to US\$ exchange rate of 6.45 to 1.00 (February 2011 assumptions).

The derivation for the AgEq Recovered is described in the following.

$$\text{AgEq Recovered} = \{ \text{Ag (g/t)} * \text{Ag (\$/gm)} * \text{Ag (Rec\%)} \} / \text{Ag (\$/gm)} + \{ \text{Pb (\%)} * \text{Pb (\$/lb)} * \text{Pb (Rec\%)} * 22.0462 \} / \text{Ag (\$/gm)} + \{ \text{Zn (\%)} * \text{Zn (\$/lb)} * \text{Zn (Rec\%)} * 22.0462 \} / \text{Ag (\$/gm)}$$

The metal prices used for the AgEq Recovered grade estimate are:

- Ag – US\$18.00/troy oz
- Pb – US\$1.00/lb
- Zn – US\$1.00/lb

The recoveries used for the AgEq cut-off are:

- Ag – 62.78%
- Pb – 84.57%
- Zn – 88.42%

The unit conversions used are:

- 1 troy oz = 31.1035 gm
- 1 tonne = 2204.62 lb
- 1 US\$ = 6.45 RMB

The cut-off and AgEq Recovered estimates reflect updates to the metal prices, exchange and recovery as specified in Table 22.1 (November 2011).

#### 15.4 Bulk Density

Resource estimates use a bulk density of 3.57t/m<sup>3</sup> which is assumed constant for all veins and areas and is also assumed to not be oxidized. AMC notes that the grade and relative distribution of the three key payable elements; Ag, Pb and Zn, can vary significantly (>10%) from vein to vein, but does not consider the potential impact of varying grade on density to be material (<5%) on the resource tonnage estimates.

Waste density is 2.64t/m<sup>3</sup>.

#### 15.5 Mine Dilution and Losses

The minimum stoping extraction and mining widths are:

- Shrinkage stoping is 0.8m vein extraction and 0.8m mining width
- Resue stoping is 0.3m vein extraction and 0.8m mining width

AMC's review of dilution, operational losses and mining width is summarized by mine method in Table 15.3. Dilution and recovery are stated as the percentage of the mineable in situ ore tonnes. AMC considers the dilution estimates to be reasonable and commensurate with the

stopping methods, which is also assisted by selective hand sorting of waste from ore, a practice conducted at Silvercorp's Ying Mine, and also proposed at the GC mine.

**Table 15.3 Mine Width, Dilution and Recovery by Stope Method**

Mine Method	Shrinkage	Resue
Mine Width (m) ^	4.0	0.8
Mine Dilution (%)	10.3	25.0
Mine Recovery (%)	90.0	88.1

Note: ^ Average for life-of-mine.

AMC makes the following comments regards GC mine recovery estimates:

- Parts of the stope access pillars (Shrinkage and Resue stopes) are considered by GC to be recoverable but AMC believes this may not be practical to achieve due to adjacent void or adjacent rock fill (Shrinkage and Resue respectively).
- AMC estimates the semi-mechanized shrinkage stopes in Stage 1 (using LHD mucking units) will have marginally higher mine losses relative to the conventional shrinkage stopes in Stage 2 (using rail mounted over-throw units). This is due to larger draw point rib pillars to accommodate the larger development profile for LHD mucking (with larger operational clearance requirements) and hence larger ore cones between the draw points along the sill alignment which are assumed not recovered (without remote loader capability and/or due to stope wall fall-off dilution).
- The stope designs assume a 3-5m height (apparent vertical) crown pillar is left in situ for regional stability purposes and for down-dip dilution control. AMC considers this to be a fair allowance based on the average mine widths for each stopping method.

AMC however does not consider the above mine recovery issues to be material regards the stopping inventory estimates, representing less than approximately 3% impact on the stopping tonnage.

### 15.6 Mineral Reserve Estimate

Table 15.4 summarizes the mineral reserve estimate from the scheduled mine plan. Approximately 10% of the mineral reserve estimate is categorized as Proven and approximately 88% is categorized as stope extraction with the remainder being development extraction.

The conversion of mineral resource to mineral reserve is described in the following.

$$\text{Mineral Reserve} = \{\text{Mineral Resource} \times \text{Recovery \%}\} / \{1 + \text{Dilution \%}\}$$

**Table 15.4 Mineral Reserve Summary**

Category	Mineral Reserve (t)	Ag (g/t)	Pb (%)	Zn (%)	Ag (kg)	Pb (t)	Zn (t)	AgEq (g/t)*	Percent by Tonnes
<b>By Reserve Category</b>									
Proven	463,976	199	1.12	3.18	92,315	5,183	14,753	268	10%
Probable	4,285,689	113	1.33	2.93	482,432	56,906	125,481	212	90%
<b>Proven + Probable #</b>	<b>4,749,665</b>	<b>121</b>	<b>1.31</b>	<b>2.95</b>	<b>574,747</b>	<b>62,089</b>	<b>140,234</b>	<b>218</b>	<b>100%</b>
<b>By Level</b>									
+50mRL	716,614	136	1.28	2.74	97,261	9,165	19,638	219	15%
0mRL	889,408	126	1.22	3.04	111,706	10,824	27,039	220	19%
-50mRL	877,557	128	1.11	2.68	112,259	9,786	23,486	206	18%
-100mRL	588,095	97	1.19	2.45	57,046	7,018	14,405	182	12%
-150mRL	452,724	119	1.58	2.85	54,009	7,171	12,902	222	10%
-200mRL	491,922	122	1.66	3.65	60,112	8,148	17,970	253	10%
-250mRL	359,349	119	1.51	3.40	42,767	5,407	12,217	238	8%
-300mRL	373,996	106	1.22	3.36	39,587	4,570	12,577	219	8%
<b>All Levels</b>	<b>4,749,665</b>	<b>121</b>	<b>1.31</b>	<b>2.95</b>	<b>574,747</b>	<b>62,089</b>	<b>140,234</b>	<b>218</b>	<b>100%</b>
<b>By Vein</b>									
V2	2,389,960	144	1.32	3.20	343,420	31,477	76,506	240	50.3%
V2-0	104,348	104	2.05	2.40	10,809	2,143	2,501	212	2.2%
V2-1	255,518	98	0.74	2.23	25,028	1,889	5,710	161	5.4%
V2-2	104,776	59	1.49	4.39	6,209	1,558	4,600	233	2.2%
V5	71,210	99	0.76	2.79	7,077	543	1,984	181	1.5%
V5-1	40,876	126	0.99	2.98	5,133	403	1,220	211	0.9%
V6	88,584	72	1.58	2.39	6,409	1,404	2,114	177	1.9%
V6-0	182,969	183	2.09	2.04	33,399	3,827	3,728	251	3.9%
V7	285,166	52	0.88	3.44	14,723	2,505	9,816	177	6.0%
V7-0	172,733	63	0.60	2.91	10,892	1,034	5,026	157	3.6%
V7-1	131,039	103	0.83	2.62	13,499	1,087	3,438	180	2.8%

**SILVERCORP METALS INC**  
**NI 43-101 Technical Report Update on the GC Ag-Zn-Pb Project in**  
**Guangdong Province, People's Republic of China**

Category	Mineral Reserve (t)	Ag (g/t)	Pb (%)	Zn (%)	Ag (kg)	Pb (t)	Zn (t)	AgEq (g/t)*	Percent by Tonnes
V8	44,932	88	2.90	1.00	3,934	1,304	449	182	0.9%
V8-0	12,288	293	0.07	0.16	3,596	8	20	191	0.3%
V9	313,307	77	1.15	2.62	23,984	3,611	8,212	173	6.6%
V9-0	46,439	127	3.32	2.75	5,875	1,542	1,278	279	1.0%
V9-1	90,093	84	1.27	3.36	7,549	1,141	3,028	207	1.9%
V10	177,804	223	2.26	2.22	39,691	4,022	3,953	288	3.7%
V10-1	76,387	45	0.87	4.43	3,436	664	3,383	205	1.6%
V11	30,338	17	0.39	4.00	506	117	1,213	158	0.6%
V13	80,846	84	0.79	1.86	6,779	639	1,507	141	1.7%
V14	50,052	56	2.34	1.09	2,799	1,171	548	147	1.1%
<b>All Veins</b>	<b>4,749,665</b>	<b>121</b>	<b>1.31</b>	<b>2.95</b>	<b>574,747</b>	<b>62,089</b>	<b>140,234</b>	<b>218</b>	<b>100%</b>
<b>By Stope Method</b>									
Shrinkage	4,295,122	125	1.33	2.97	537,829	56,917	127,455	221	90%
Resuing	454,543	81	1.14	2.81	36,918	5,172	12,779	182	10%
<b>All Methods</b>	<b>4,749,665</b>	<b>121</b>	<b>1.31</b>	<b>2.95</b>	<b>574,747</b>	<b>62,089</b>	<b>140,234</b>	<b>218</b>	<b>100%</b>

Notes: \* AgEq (g/t) is the recovered silver equivalent. # 135g/t AgEq cut-off for Stope Reserves & 45g/t AgEq for Development Reserves.  
Rounding of some figures may lead to minor discrepancies in some totals

## 16 MINING METHODS

### 16.1 Conventions

All measurement units in this report are metric SI units.

The GC project has a local mine section grid (Mine Section) orientated at 200° – 20° bearing whereby the section numbers increase with easting and the section spacing is at 50m intervals between even numbered sections (e.g. Section 10 to Section 12 is a 50m interval).

### 16.2 Introduction

Mining will be conducted in two stages. The stages are generally subdivided as follows:

1. Stage 1 - +100 mRL to -50 mRL between local Mine Sections 10 to 36 for development and 12 to 32 for production. West side of project.
2. Stage 2 - +100 mRL to -50 mRL between Mine Sections 36 to 54 for development and 32 to 54 for production. For -50m RL to -300 mRL between Mine Sections 12 to 50 for both development and production.

Stage 1 targets fast-tracking the project into production and is developed by mobile rubber-tired diesel-powered equipment (development jumbo, loader and truck) with surface decline access down to -50 mRL.

Stage 2 is developed using conventional tracked equipment (electric locomotive, rail cars, electric rocker shovels and pneumatic hand held drills) with shaft access from -50mRL down to -300 mRL.

Selective stoping methods such as Shrinkage and Resue are employed with stope production drilling conducted with pneumatic jackleg drilling. In-stope rock movement will be by gravity to draw points or hand-carting to steel lined passes.

Stage 1 production mucking uses load-haul-dump loaders (LHD) with trucks hauling ore to the surface ROM stockpile. Ore is re-handled from the ROM stockpile to the primary crusher feed bin using a ROM front-end-loader (FEL).

Stage 2 production mucking uses electric-powered over-throw rail loaders with rail cars and electric locomotives transporting ore to the Main Shaft ore pass. Ore is skip hoisted to surface and conveyed to the surface crusher feed bin.

### 16.3 Geotechnical Conditions

#### 16.3.1 Introduction

AMC's geotechnical review included the following:

- Review of existing available data and reports provided by Silvercorp.
- Preliminary characterization of rock mass conditions based on available geotechnical data.
- Preliminary assessment of ground support requirements for lateral mine development.
- Review of mine design parameters including stope spans and pillar dimensions.

Presented within this section is a summary of the methodology and results of the various geotechnical assessments undertaken, and recommendations for further work.

It should be noted, that due to the limited available geotechnical data, AMC's review is considered high-level and not to the level of detail normally associated with a mining feasibility study in Canada. As such, AMC's geotechnical review incorporated preliminary assessments aimed at assessing the "reasonableness" of the geotechnical aspects of the GMADI design for the mining study.

### 16.3.2 Available Data

AMC's geotechnical assessments were based on observations made during a site visit to the GC Project in May 2011 by Owen Watson (AMC Senior Geotechnical Engineer), together with reports and data provided by Silvercorp.

As part of the site visit, geotechnical observations were made of the following:

- Rock mass exposures in portal areas of previously mined adits ML-5 and ML-8.
- Selections of core from drillholes ZK1401, ZK1001, ZK40204, and ZK101.

The following data was provided by Silvercorp:

- GMADI Report: 'Mining and Dressing Project of Gaocheng Lead-Zinc Ore in Yun'an County, Guangdong Province - Preliminary Design (GD1371CS) Volume I', January 2011.
- Detailed geotechnical interval data of Q-System rock mass classification parameters (after Barton et al, 1974) from two drillholes (ZK2002 and ZK3604) collected by Silvercorp.
- RQD logging data from 35 drillholes collected by Silvercorp. RQD data had been recorded in long intervals based on lithological units, rather than shorter intervals based on drilling runs. As a result, the data provides an indication of the overall rock quality for the entire lithology unit, but detail on the variation of RQD within the logged lithology unit can not be determined.
- Additional geotechnical data including degree of weathering, compressive strength and rock quality index data. The compressive strength data and rock quality index data was not directly used for the analysis as AMC was unable to establish the specific procedures used to obtain the data, and therefore could not determine the reliability of the data.
- Wireframe interpretations of the mineralized veins.
- Wireframe of surface topography.

### 16.3.3 Data Analysis

AMC's analysis of the geotechnical logging data involved the following:

- Generation of sections (on 250m spacing, looking west) showing RQD histograms plotted along drillhole traces, to investigate the spatial variation in RQD values relative to the mine design and interpreted veins. RQD was plotted as '100-RQD' in order that zones of lowest RQD values are displayed as the tallest histograms. Presented in Figure 16.1 is a north-south section located towards the western limit of the mine design showing RQD histograms. Low RQD values appear to be generally related to weathered

material near surface and locally throughout the rock mass, possibly related to veins and/or vein contacts.

- Generation of distribution plots of the logged rock mass classification parameters to investigate the statistical distribution of the logged parameters for each of the main geotechnical domains. The distribution plots for 'all data' are shown in Figure 16.2.
- Generation of distribution plots of RQD logging data for each of the main geotechnical domains. These plots are presented in Figure 16.3.

**Figure 16.1 Section looking west at 93500 mE, showing '100-RQD' histograms plotted on drillhole traces**

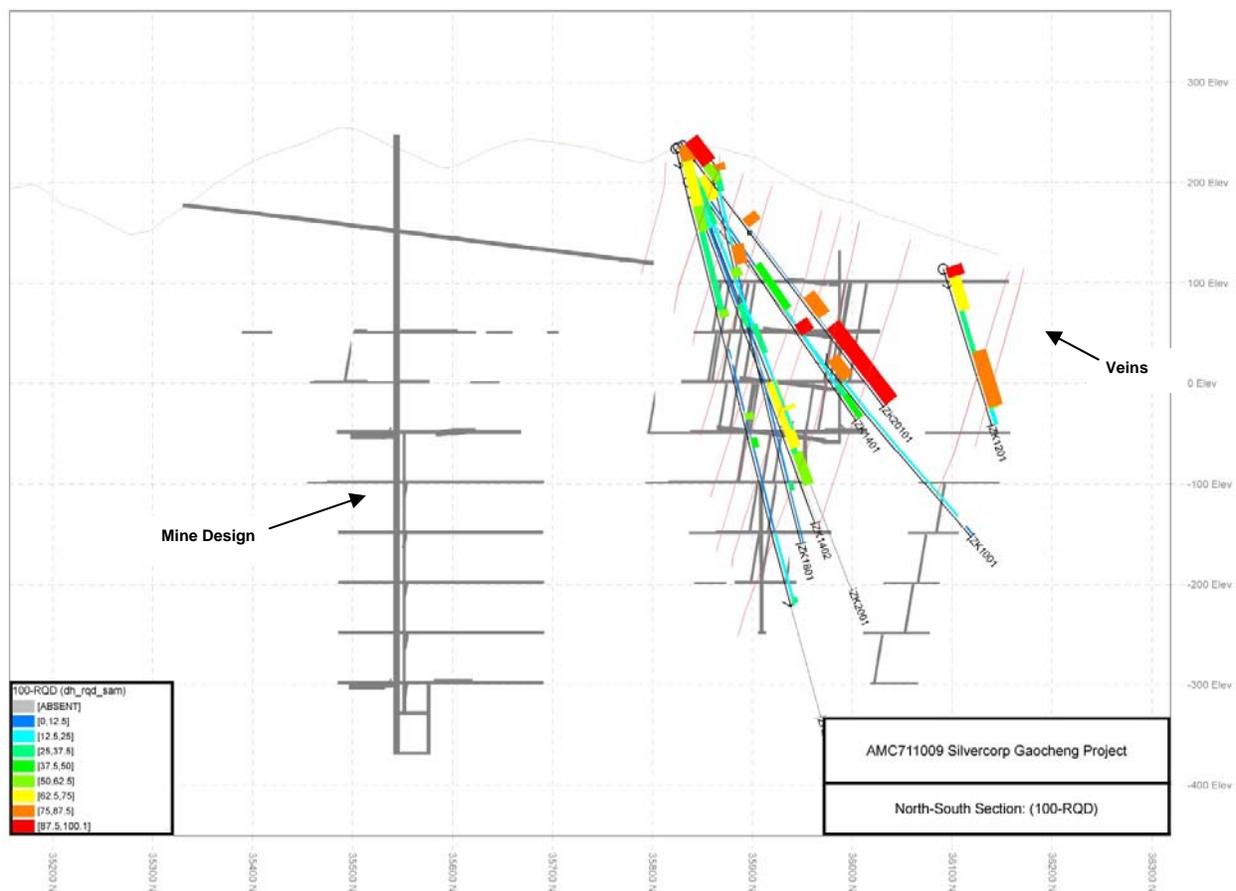
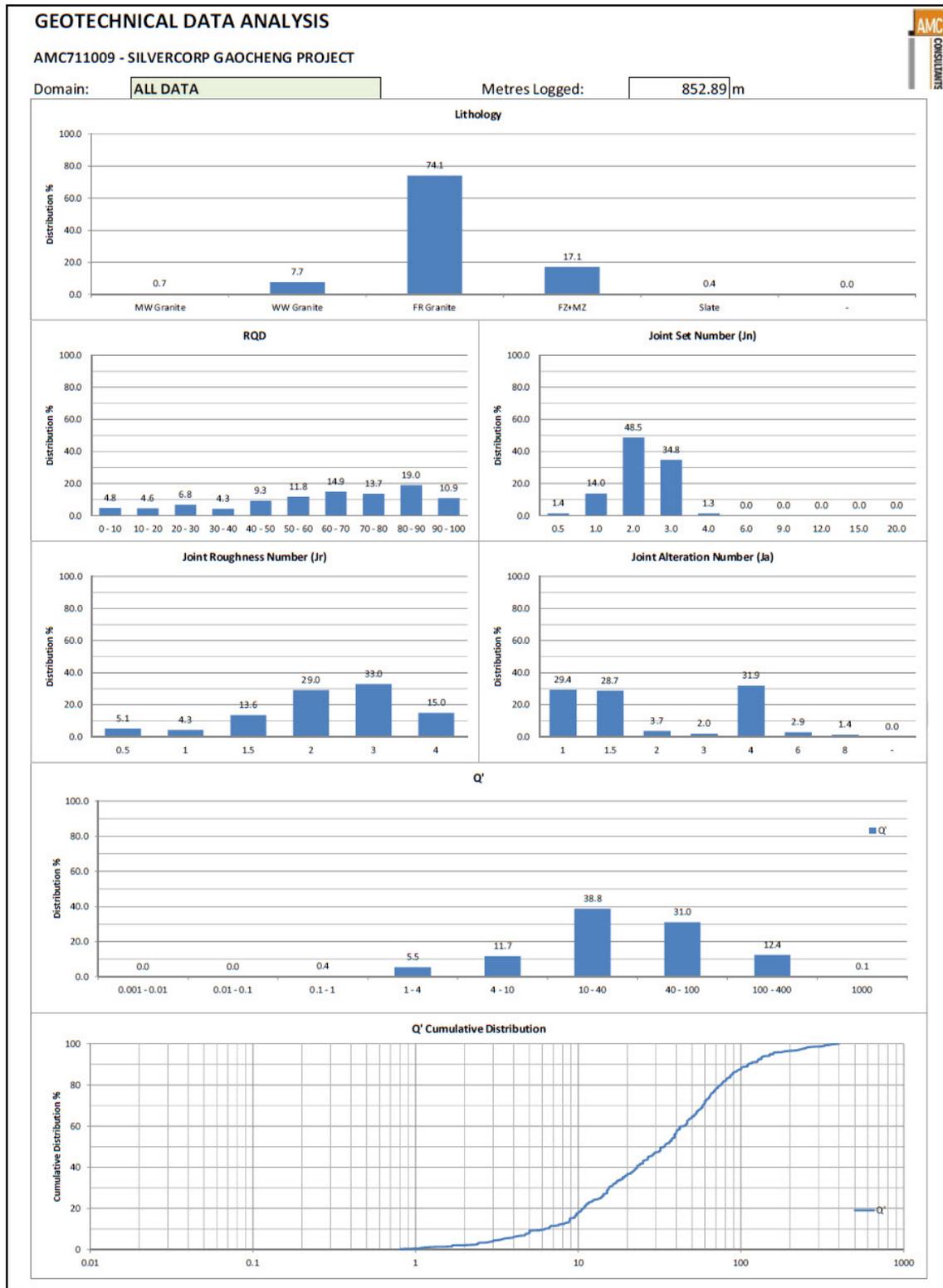
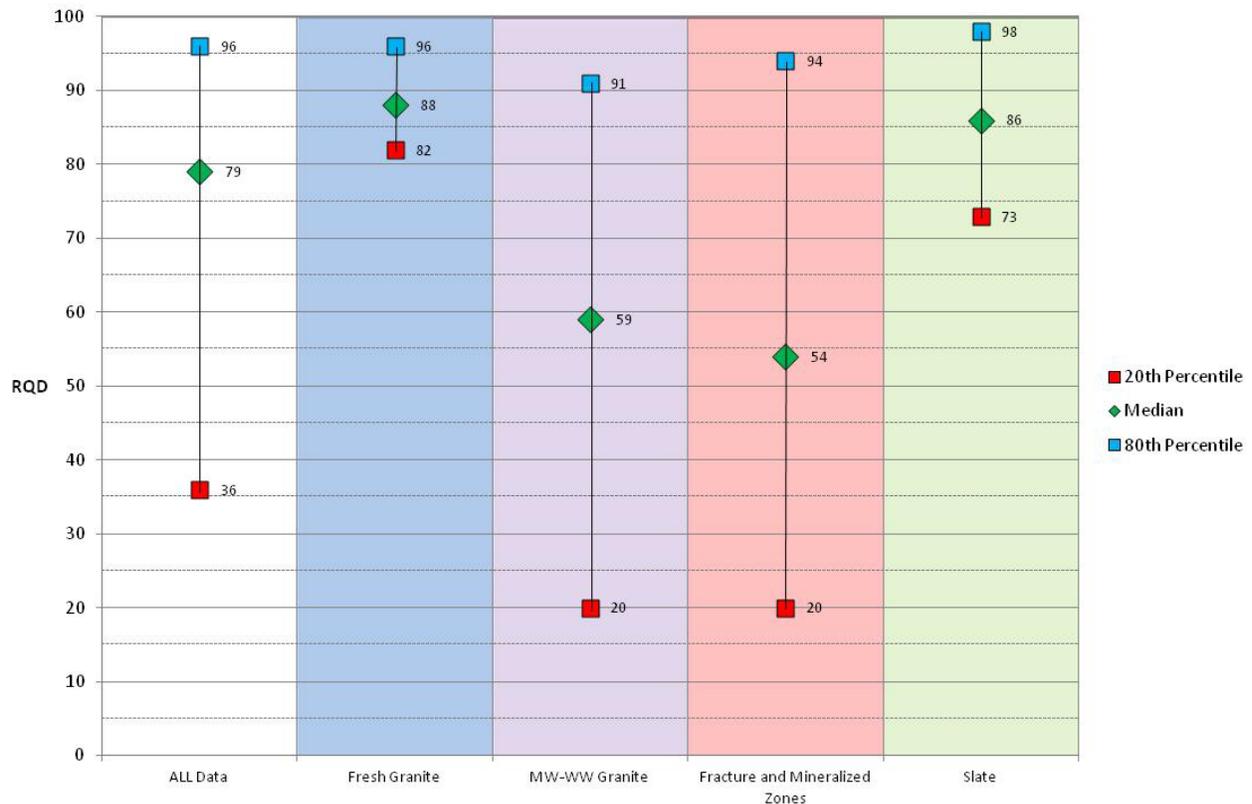


Figure 16.2 Distribution Analysis for All Geotechnical Logging Data



**Figure 16.3 RQD Data Distribution**



### 16.3.4 Characterization of Geotechnical Conditions

Geotechnical domains were assigned according to lithology and degree of weathering as follows:

- Moderately and weakly weathered granite
- Fresh granite
- Fracture and mineralized zones

These domains are considered 'preliminary' based on the limited available geotechnical data. Collection of additional data may result in definition of additional geotechnical domains, particularly in the immediate hangingwall of the veins, where there is currently insufficient detailed geotechnical information, and also beyond the footwall contact of the vein system in which there is limited drilling coverage. Geological logging of existing drill core indicates the presence of an argillaceous slate unit beyond the footwall of the veins, however there is insufficient geotechnical data to characterize rock mass conditions in the slate.

Depth of weathering is variable across the project. Completely to Highly Weathered material generally extends up to approximately 20m below surface, with Moderate to Weakly Weathered Granite extending to depths of approximately 100m.

AMC understands that Fresh Granite forms the primary host rock of mineralization and is the domain in which the majority of waste development will take place.

Mineralized veins which comprise the orebody have been included in the domain 'Fracture and Mineralized Zones'.

The 'Q' rock mass classification parameters (after Barton et al, 1974) which characterize expected rock mass conditions within each domain are summarized in Table 16.1. These values are based on the geotechnical logging data provided by Silvercorp, and observations of drill core made by AMC during the site visit. It should be noted that AMC increased the logged values of Joint Set Number for all rock types by one joint set based on observations made during the site visit.

**Table 16.1 Rock Mass Classification Parameters by Geotechnical Domain**

Domain	Moderate to Weakly Weathered Granite	Fresh Granite	Fracture and Mineralized Zones
RQD	20% to 60%	80% to 90%	20% to 50%
Joint Set Number (Jn)	2 to 3 Joint Sets ( $J_n = 4$ to $J_n = 9$ )	2 to 3 Joint Sets ( $J_n = 4$ to $J_n = 9$ )	2 to 3 Joint Sets ( $J_n = 4$ to $J_n = 9$ )
Joint Roughness (Jr)	Smooth to Rough, Undulating ( $J_r = 2$ to $J_r = 3$ )	Rough and Planar to Smooth and Undulating ( $J_r = 1.5$ to $J_r = 2$ )	Rough and Planar to Smooth and Undulating ( $J_r = 1.5$ to $J_r = 2$ )
Joint Alteration (Ja)	Hard, non-softening coating to soft, sheared coating ( $J_a = 1.5$ to $J_a = 4$ )	Hard, non-softening coating to soft, sheared coating ( $J_a = 1.5$ to $J_a = 4$ )	Non-softening coating to soft, sheared coating ( $J_a = 2$ to $J_a = 4$ )
Rock Mass Classification	<b>Poor to Good</b> rock mass quality	<b>Poor to Good</b> rock mass quality	<b>Poor to Fair</b> rock mass quality

### 16.3.5 In Situ Stress

No specific data was available for the study regarding *in situ* stresses at the project. However, the GMADI report states that direction and magnitude of major principal stress is expected to be consistent with dead-weight loading of overburden. This assumption formed the basis for estimating mining induced stresses as part of stope stability and ground support assessments presented below.

### 16.3.6 Hydrogeology

AMC has not conducted any specific hydrogeological investigations for this study. The GMADI report presents discussion of hydrogeological conditions at the project and states that hydrogeological exploration in the district is relatively inadequate. For AMC's preliminary geotechnical assessments, minor water inflows (less than 5 litres per minute locally) were assumed.

### 16.3.7 Mine Design Considerations

#### 16.3.7.1 Rock Mass Conditions

The rock mass condition is categorized as fair to good and it is anticipated that the vein and host rocks in the mine area will be competent and require minimal ground support. In general, the mine development and stopes will be left unsupported. However, where poor ground conditions exist in the shafts, long-term lateral development or infrastructure chambers, ground support will be provided. This may be either shotcrete only, shotcrete with rock bolts or shotcrete with rock bolts and mesh.

Surface shafts collars will traverse approximately 10m of soil coverage and 20m of oxidised ground.

#### **16.3.7.2 Surface Requirements**

AMC understands that surface subsidence is not permitted at the GC Project. The GMADI design incorporates a surface crown pillar with the upper stoping limit set at +100 mRL. The local topography above the mine plan area varies between approximately 105 mRL (at river level) to 340 mRL (hill top).

The Hashui Creek traverses the north-eastern portion of the mine area inside the Stage 1 and Stage 2 potential subsidence zones between mine sections 26 to 56. The creek will be diverted via a tunnel (579m) to the north-east to fall outside of the Stage 2 potential subsidence zone. The creek tunnel diversion will be implemented during Stage 1 before production commences.

AMC considers the allowance for the surface crown pillar made in the design is generally appropriate for the feasibility study. However, AMC recommends that a detailed investigation and assessment of crown pillar requirements be undertaken for input into the detailed mine design (an 'executable design'), with particular focus on surface pillar requirements in the vicinity of Hashui Creek valley, and any other streams (or drainage paths) that traverse the mine area.

#### **16.3.7.3 Stability Assessment for Stoping**

A preliminary stability assessment of the proposed shrink stoping configuration was undertaken using the Modified Stability Graph method as described by Hutchinson and Diederichs (1996). The input parameters used for the assessment were based on median rock mass conditions estimated from distribution plots of geotechnical logging data .

The proposed shrink stoping layout consists of mining panels 50m in length on strike, and 50m in height, resulting in a hangingwall with hydraulic radius (HR) of 12.5. Each shrink stope remains filled with broken ore until excavation is completed to full height, at which time the broken ore is removed from the stope via cross-cuts established on the mucking horizon. On completion of production, the stope remains open and unfilled.

AMC's preliminary assessment indicates that an open stope hangingwall of HR=12.5 is at the upper limit to achieve stability without the requirement for cable bolt support. Because ground conditions will be variable (locally better or worse than the median values used for the assessment), instances of local hangingwall instability are expected. Hangingwall instability may result in unacceptable levels of dilution of the broken ore stocks, or loss of ore within the stope. Based on AMC's limited understanding of the variability of the rock mass conditions, the average allowance for dilution presented in Table 15.3 is considered reasonable.

Shrink stope end walls and back are forecast to be stable without requirement for cablebolt support for the majority of expected rock mass conditions.

It should be noted that this assessment is concerned with the 'rock mass' and does not consider possible destabilizing effects associated with major structures such as faults or shear zones. These should be considered on a case by case basis.

#### 16.3.7.4 Stope Pillars

Stope crown pillars for both Shrinkage and Resue stopes will be approximately 3-5m in height on-dip at the prevailing mining width and vein dip.

For the Shrinkage stoping method the travel way access pillars will be approximately 3m height on-dip by 2m width on-strike by the prevailing mining width and vein dip.

For the Resue stoping method a secondary sill pillar will be employed (located above the vein drive which is at the access level elevation) and will be approximately 3m in height on-dip at the prevailing mining width and vein dip.

Based on AMC's understanding of the rock mass conditions and the generally narrow mining widths envisaged, the pillar allowances are considered reasonable. It should be noted however, that variability of rock mass conditions will likely dictate that locally larger pillars are necessary where poor rock mass conditions are encountered. In addition, as mining progresses to greater depths, increases in *in situ* and mining induced stresses may also result in the requirement for larger pillars.

#### 16.3.7.5 Main Shaft Pillar

A pillar will be maintained around the Main Shaft. Development may occur within the pillar zone however stope production will not be allowed. The shaft pillar is an expanding conical with depth from the collar elevation at an 80° dip. The radius of the conical at surface (248mRL) is 13m and the Main Shaft radius is 3m.

#### 16.3.7.6 Ground Support Requirements

Indicative ground support requirements were estimated for the lateral development using the Q-system (after Barton, Lien and Lunde, 1974) and the Tunnelling Support Guidelines developed by Grimstad and Barton (1993).

Assessments were conducted for each geotechnical domain for median and lower 20<sup>th</sup> percentile rock mass conditions estimated from distribution plots of geotechnical logging data.

Based on AMC's experience, where drift development is by conventional drill and blast methods, installing a minimum standard of ground support on a round by round basis in all mine development is the most effective and reliable method of reducing the exposure of mine personnel to rock fall hazard, particularly at the working heading. This is the approach AMC recommends for any new mine, regardless of the mine's location and local mining practices.

However, AMC understands that in general, the mine development at the GC project will be left unsupported unless ground conditions warrant otherwise - as is common mining industry practice in China. AMC's ground support assessment indicates that for the relatively small dimensioned drift development proposed, excavations are forecast to be stable without installation of support for the majority of expected rock mass conditions. Where 'poor' ground conditions are encountered, the assessment indicates that pattern bolting on a spacing of 1.5m and shotcrete support (50-70 mm thickness) will be necessary.

In lieu of installing ground support in all underground development on a round by round basis, AMC makes the following recommendations:

- Assess ground conditions on a round by round basis in all development headings (ore and waste) to determine the requirement for ground support. Doing so will help prevent the occurrence of significant failures from backs and walls, which require timely rehabilitation and expose the work force to rock fall hazard.
- Ensure scaling of the development heading on a round by round basis.
- Conduct routine check scaling of all unsupported development at the mine. This process can help identify areas of the mine in which rock mass deterioration is occurring and allow rehabilitation works to be planned.
- Where possible, avoid mining development intersections in fault zones, and design drifts to cross fault zones at right angles (to minimize the exposure length within the drift).

In addition to the above, AMC recommends that specific rock mass conditions be assessed for critical underground infrastructure, including shafts and chambers, to determine ground support requirements to ensure serviceability of the excavation for the life of mine.

#### **16.3.7.7 Conclusions**

Based on the review of the available geotechnical data and high-level assessments undertaken, AMC considers that the geotechnical aspects of the GMADI mine design are generally reasonable for mining study purposes. However, given the limited nature of the data, the geotechnical knowledge at the project is not considered to be at the level of detail normally associated with a mining feasibility study in Canada, and is more in line with a scoping level study.

Further geotechnical investigations are recommended to advance the current mine design to an 'executable design'. In particular, AMC recommends that the following work is undertaken:

- Collection of additional detailed geotechnical logging data from drill core and mapping of underground workings, to allow improved characterization of rock mass conditions within the immediate stope hangingwall zone, and the mineralized veins. This should incorporate collection of structural orientation data. Data collection should allow rock mass classification using an internationally recognized system, such as the Q-System (after Barton et al, 1974) or RMR (after Bieniawski, 1989).
- Development of a three-dimensional geological model with interpretations of primary lithologies and structures (such as faults and shear zones).
- Geotechnical investigations of proposed shaft locations to determine site suitability and ground support requirements. This should incorporate more detailed assessment of shaft pillar requirements.
- Geotechnical investigations of the surface crown pillar, particularly in the vicinity of the Hashui Creek valley, and any other streams or drainage paths that traverse the mine area.
- Further hydrogeological assessments, particularly to assess hydraulic connectivity between the Hashui Creek valley (and any other streams or drainage paths that traverse the mine area) and the underground mine workings.

Further investigation of *in situ* stresses to confirm assumptions made in the mine design and stability assessments.

## 16.4 Extraction Sequence

The global extraction sequence is top-down from +100 mRL extending down to -300 mRL and generally west to east for Stage 1 and Stage 2 above -50mRL. It is centrally outwards from the Main Shaft location for Stage 2 below -100 mRL.

The macro stope extraction sequence is bottom-up for both stoping methods.

## 16.5 Production Rate

Mine operations will be conducted 365 days of the year but mine production is scheduled on the basis of 330 days per year at approximately 1,500 tpd for approximately 496 ktpa for the first eight years, rising to approximately 1,570 tpd for approximately 518 ktpa for the last four years. The production life is estimated to be 12 years.

Production is expected to be approximately 80 tonnes per day per stope for Shrinkage stopes and 75 tonnes per day per stope for Resue stopes with production per level capped at approximately 25% of the available stopes and up to 20 stopes concurrently over all active levels.

The production rate from each stope is dependent on the vein width, and as such, the production rate and schedule assumes a balance of wide and narrow vein stopes (generally Shrinkage and Resue respectively).

AMC's high-level review indicates a mineral resource endowment of 12,829 tonnes per vertical metre equating to a vertical advance rate of approximately 42m per year which is within industry performance for like operations.

## 16.6 Mining Methods

Shrinkage stoping and Resue stoping will be the methods employed.

To confirm AMC's understanding of Silvercorp's application of the stoping methods and also their suitability for the GC mine environment, AMC observed the application of these stoping methods at Silvercorp's Ying mine operation during May 2011. The Ying mine is located in Luoning County, in the Henan Province, about 10 km South-East of Xiayu and about 60 km South-East of Luoning. AMC believe the methods and their proposed application to be appropriate for the GC mine environment.

### 16.6.1 Shrinkage Stoping

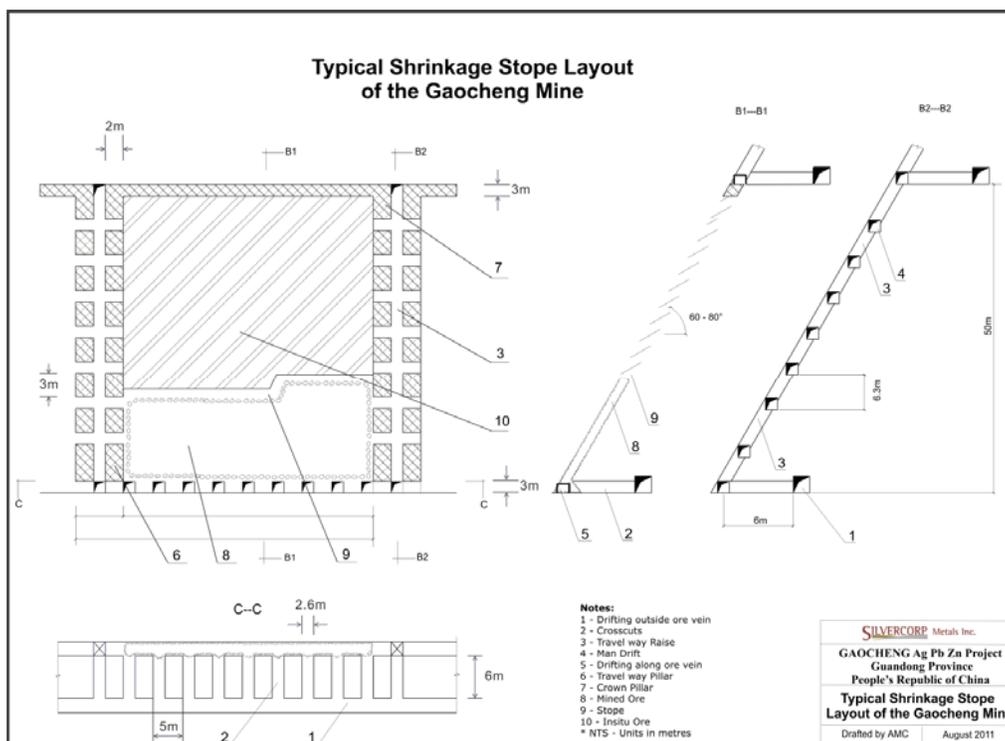
The method begins with establishing a sill drive along the vein to expose the vein. An access drive (conventionally a footwall drive) is also developed parallel to the vein. It is located away from the vein so as not to be affected by the stoping void and also to enable loader access clear of the access drive (footwall drive). Crosscuts between the access and vein drives are expected to be developed at approximately 7m strike spacing (but this is dependent on the loader used, loader dimensions and the rib pillar thickness required for rib stability). The crosscuts act as draw points for the mucking of the stope ore. Travel way raises that are also used for services are established between the levels at each end of the stope block. Each stoping block is 50m strike length by 50m height.

Jackleg miners using pneumatic drills to drill a 1.8 – 2.2m stope lift that is drilled and blasted as inclined up-holes with a forward inclination of 75-85° ("half-uppers"). The typical drill pattern

uses a drill burden 0.6 – 0.8m and spacing 0.8 – 1.2, pending the vein thickness. Holes are charged with cartridge explosives and ignited with tape fuse. The powder factor is expected to be 0.4 – 0.5 kg/t. Stope blasting fills the void below with ore as the mining proceeds upwards. While mining upwards, only 30-40% of the stope ore may be mucked until the entire stope is mined-out. At this point, all ore is mucked from the stope, pending unplanned dilution does not occur, leaving the stope void empty. A crown pillar is maintained for the stope to provide regional stability and to minimise dilution from up-dip stopes. Ventilation, compressed air and water are carried up the travel way raises to the stoping level. Loading of the ore from the draw points is by rubber tired LHD into trucks (Stage 1) or electric rail over-throw loaders into rail cars (Stage 2).

Figure 16.4 depicts the Shrinkage stoping method as proposed at GC Mine.

**Figure 16.4 Typical Shrinkage Stope Layout**



### 16.6.2 Resue Stoping

Vein and access development preparation is essentially the same as for Shrinkage stoping except an additional elevated sill drive (3m on-dip height) is established and draw points are limited to approximately two to enable access to the raise positions (used as steel lined mill holes) in order to minimize waste development.

Resue stoping method veins are typically high-grade and typically between 0.20 and 0.80m width. Resue stoping involves separately blasting and mucking the high grade narrow vein and waste required to achieve a minimum stoping work width.

The mining crew consists of Jackleg miners using pneumatic drills. Half-upper lifts are drilled and blasted in essentially the same manner as for Shrinkage stoping. After an ore lift is blasted and mucked, the footwall is blasted and used to fill the space mined out. This process is

repeated until the crown pillar is reached. The entire stope is left filled with waste from the slashing of the footwall.

The blasted ore is transported by wheel barrow and/or hand shovelling to the steel lined pass. The steel pass is constructed in lift segments as the stope is mined upwards. The base of the steel pass is held in place with a timber set. The footwall waste is then slashed (blasted) to maintain a minimum mining width (typically 0.8m for GC) and to provide the working platform for the next stope lift. In contrast to Shrinkage stoping, the mined out stope is left filled with waste from the slashing of the footwall necessary to maintain a minimum mining thickness.

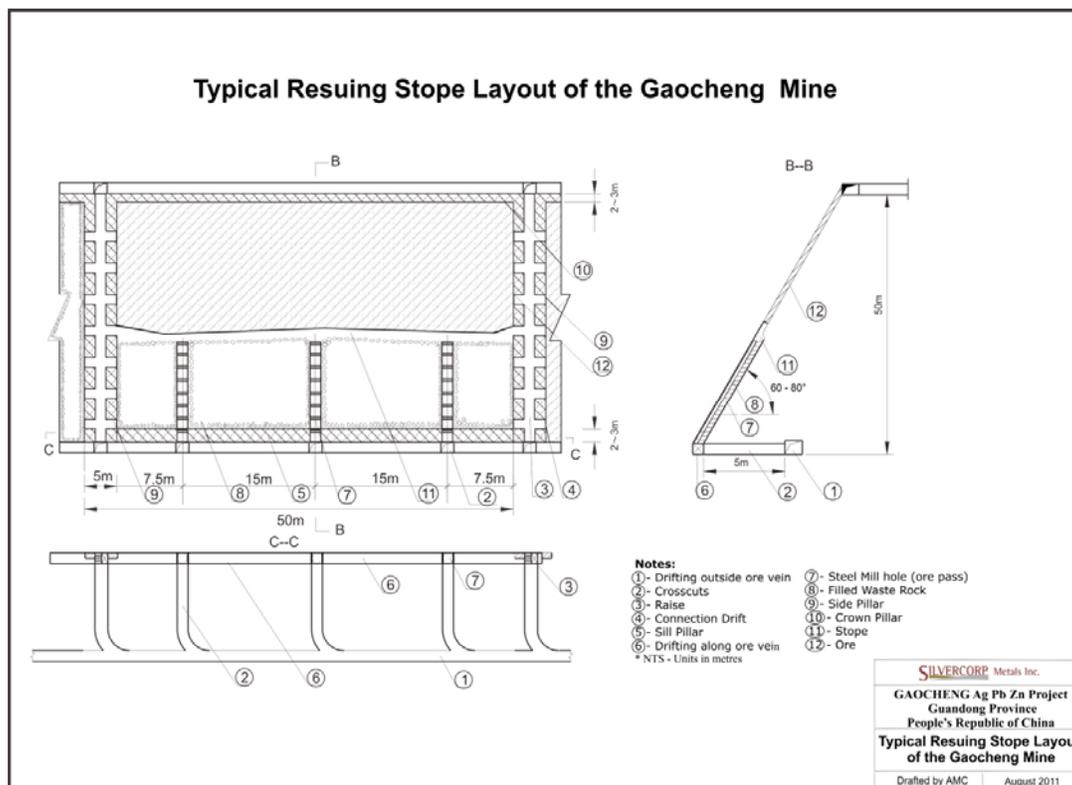
The order of vein extraction and footwall slashing is generally dependent on the condition of the vein hangingwall contact. Where the vein hangingwall contact is distinct and stable, the vein is extracted first; otherwise the footwall waste is extracted first followed by vein slashing.

Rubber mats and/or belting are placed on top of the waste after each waste lift to minimise ore intermingling with the waste (ore losses) and also to minimise over-mucking of the waste (dilution). Mucking of the ore consists of hand lashing (shovelling) and hand carting to a steel pass which gravitate to the mill hole crosscut. The rubber mats and/or belt are rolled up and removed prior to slashing the footwall forming the next platform lift.

In-stope ore transporting may potentially be improved by using scraper winches with small hoes.

Figure 16.5 depicts the Resue stoping method as proposed at GC Mine.

**Figure 16.5 Typical Resue Stope Layout**



## 16.7 Mine Design

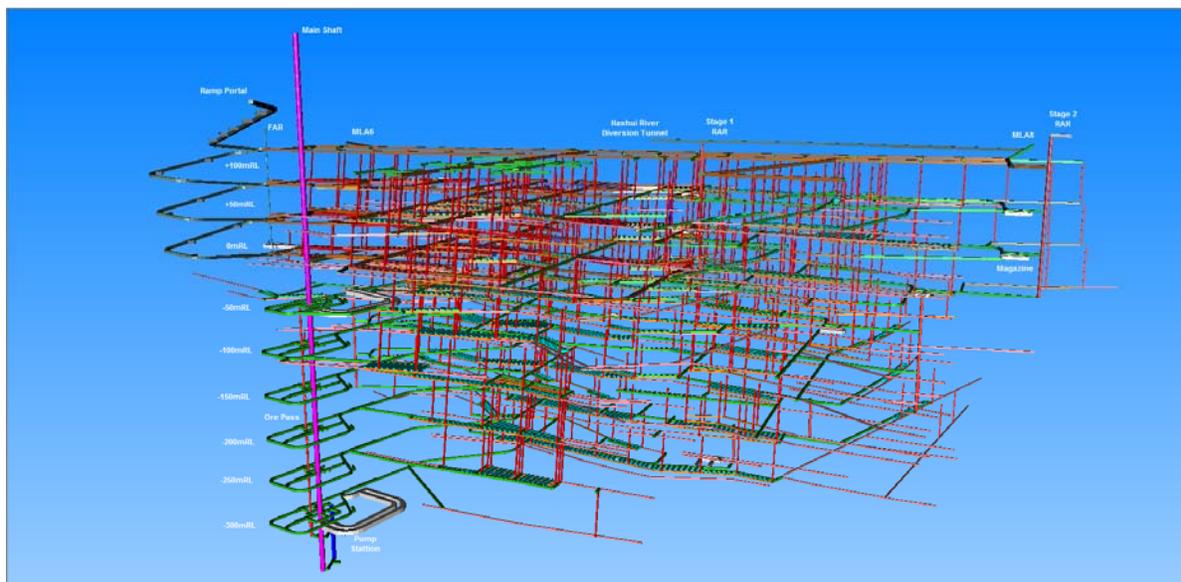
The mine design is based on the engineering work completed by the local official provincial design institute GMADI (April 2011). Refinements such as profile dimensions, alignments, fleet sizing, etc have been made by Silvercorp technical personnel on an as-needed basis during the project preliminary construction phase.

The mine design provided is considered by AMC to be below feasibility study standard (within +/-10-15% on the inputs) with respect to knowledge of the vein location and vein peripheral extents and missing minor miscellaneous development items such as travel way refuges, stripping, service holes and the like. AMC expects that the design will be progressively refined as the mine is developed, but does not anticipate a material change in the development requirements (e.g. <5%). The exception to this is the placement of the development relative to the veins. The issues related to placement (potential re-development, potentially longer than planned draw points, etc) are anticipated to be managed by the common practice of development on the vein prior to the development of the waste accesses.

In plan view, the mine development covers an area approximately 600m by 1,200m between Mine Sections 8 and 56. The mine design total lateral and vertical requirement is 115.6 km and 28.9 km respectively. A surface plan showing key mine infrastructure locations is provided in Figure 18.1.

Figure 16.6 illustrates the mine design in 3D perspective looking generally north-west.

**Figure 16.6 GC Mine Design in 3D Perspective**



The design strategy is two staged with Stage 1 being predominantly mechanized development to fast track production while the longer term Stage 2 at depth reverts to Chinese conventional tracked development methods.

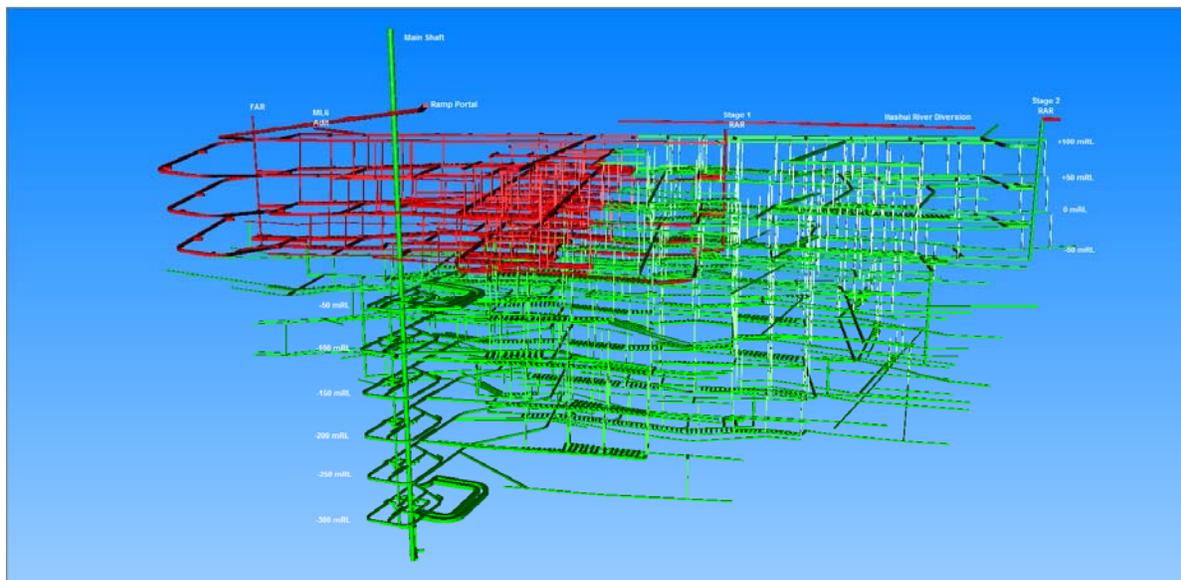
The Stage 1 Ramp will be used for hauling ore, waste rock, materials, equipment, personnel and provide access for key services like dewatering lines, feed water, power, communications

and ventilation. Ramp access is from +176 mRL down to -64mRL. There is no plan to extend the ramp for Stage 2, but the opportunity exists if required.

The Stage 2 Main Shaft will be used for skip hoisting ore, cage hoisting waste, hoisting materials, equipment, personnel and provide access for key services like dewatering lines, feed water, power, compressed air, communications, egress ladder ways and ventilation. The Main Shaft collar is at +248 mRL and sump bottom is at -370 mRL.

Figure 16.7 illustrates the mine design stages in 3D perspective looking generally north-north-west. Red colour indicates Stage 1 development and bright green colour indicates Stage 2 development.

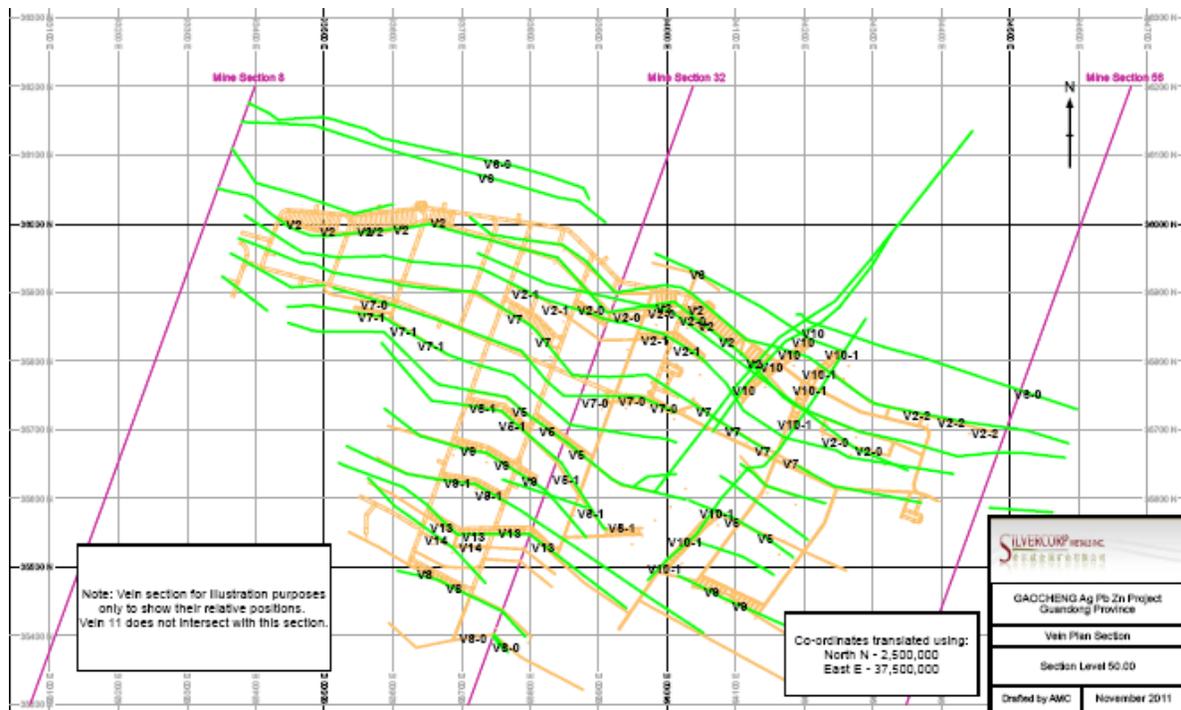
**Figure 16.7 GC Mine Design Stages**



The veins included in the mine design are V2, V2-0, V2-1, V2-2, V5, V5-1, V6, V6-0, V7, V7-0, V7-1, V8, V8-0, V9, V9-0, V9-1, V10, V10-1, V11, V13 and V14.

Figure 16.8 illustrates the relative vein positions on the +50 mRL. Note that not all veins are present on this level (e.g. V11) and the extent of veins illustrated are not all mined within the current study design.

Figure 16.8 GC Veins and Development Plan View +50mRL



### 16.7.1 Pre Existing Development

In the immediate area of the current mine design there is some minor pre existing development off three adits called ML5, ML6, ML8. The ML6 adit is located at the immediate west side of the Gaocheng Village. The development from the three adits has a combined void volume of approximately 13,000m<sup>3</sup> (as advised by the GMADI study). Table 16.2 summarizes the complete list of pre existing development in the GC project resource area with a surface location plan.

The pre-existing ML6 and ML8 development will be used in the mine design (e.g. ventilation and egress purpose). The ML6 adit will be used as a ventilation airway (intake initially reverting to exhaust) and environmental pre cautions will be required to protect the amenities of the Gaocheng Village. ML6 and ML5 have been used in the past for sampling to validate some of the vein resource estimates (V2 and V10 respectively).

Precautions will be undertaken to prevent the pre-existing development becoming water logged or exposed to surface run-off inundation to eliminate the possibility of mud or water in-rush in the un-likely event of subsidence from the stoping areas immediately below. This is partially ameliorated by the Hashui Creek diversion tunnel.

**Table 16.2 Co-ordinates of Pre Existing Development**

Adit	Northing	Easting	Elevation
ML1	2,536,522	37,593,270	139
ML2	2,536,898	35,793,310	97
ML3	2,536,444	37,593,459	129
ML4	2,536,759	37,592,679	120
ML5	2,535,758	37,594,035	116
ML6	2,536,156	37,593,460	101
ML7	2,536,450	37,593,241	148
ML8	2,535,761	37,594,440	117
ML9	2,536,375	37,593,315	110

### 16.7.2 Mine Access

Mine accesses for rock transport, materials supply and labour access is provided by a decline (Ramp) and a shaft (Main Shaft). Secondary mine access for labour emergency egress is provided by the Ramp Shaft, Stage 1 and Stage 2 return airway shafts, and the +100 mRL level connections from the ML6 and ML8 adits.

The initial ramp alignment is approximately parallel with the local grid Line 10 at the western end of the project area. The ramp portal co-ordinate is at approximately 37,593,581m east, 2,535,330m north, +176 mRL elevation. The ramp provides access to the +100 mRL, +50 mRL, 0 mRL, -50 mRL and -100 mRL levels. The ramp spirals to the bottom located at -64 mRL. The ramp profile is 4.2m width by 3.6m height (approximately 13.9m<sup>2</sup> profile area). The average gradient is 12% (1 in 8.3) with minimum radius of 20m. The total ramp access length is 2,224m (excluding stockpiles).

The ramp includes 10m length remuck stockpiles at approximately 100m intervals with travel way refuges excavated between the remuck stockpiles. The ramp spirals at the northern end make connections to a blind sunk shaft (Ramp Shaft) at approximately +100 mRL, 0 mRL, -50 mRL and -100 mRL. The Ramp Shaft with 3.5m diameter (9.6m<sup>2</sup>) acts as a return ventilation airway during ramp development and reverts to an intake ventilation airway prior to Stage 1 production. The Ramp Shaft also provides secondary egress and will be used for mine services (piping for air and water, electrical cables and ladders).

The Main Shaft collar is located at +248 mRL elevation at approximately 37,593,562m east, 2,535,544m north with the shaft sump at -370 mRL for a 618m sink depth. The shaft diameter is 6.0m diameter.

### 16.8 Mine Development

The mine design is based on the mineral resources above a 150 g/t AgEq Recovered with the addition of vein exploration development (which in some part, is also used for stope access). Vein exploration development is categorized as development that occurs outside of the mineral resource categorization. Vein exploration development is reported as development waste and assigned zero grade irrespective of its resource grade. Vein exploration development represents approximately 51% of the total 44.7 km of vein development in the mine plan.

The mine levels will be placed at 50m vertical intervals. Levels will be graded at 0.3% from either the Ramp or Main Shaft access however the mine design provided does not incorporate this feature. AMC does not consider this to be material with respect to estimates for development quantities.

The Hashui Creek diversion tunnel will be developed concurrently from both ends and will be completed early in Stage 1 prior to production commencing.

### 16.8.1 Development

Table 16.3 summarizes the lateral development requirement, profiles and their various categories.

**Table 16.3 Lateral Development Profiles and Categories**

Lateral Development Type	Stage	Width (m)	Height (m)	Length (m)	Material Type	Cost Category
<b>Jumbo Development</b>						
Ramp	1	4.2	3.6	2,224	Waste	Capex
Ramp Stockpile	1	4.2	3.6	269	Waste	Capex
Level (Jumbo)	1	4.2	3.6	13,963	Waste	Capex
Vein Drive - Ore (Jumbo)	1	4.2	3.6	1,937	Ore	Opex
Vein Drive - Waste (Jumbo)	1	3.9	3.7	298	Waste	Opex
Drawpoint (Jumbo)	1	3.9	3.7	652	Waste	Opex
<b>Subtotal</b>				<b>19,344</b>		
<b>Jackleg Development</b>						
Level (Jackleg) - Large Profile	1	2.6	2.8	29,517	Waste	Capex
Lateral Development Type	Stage	Width (m)	Height (m)	Length (m)	Material Type	Cost Category
Vein Drive - Ore (Jackleg) - Small Profile	1&2	1.6	1.8	20,088	Ore	Opex
Lateral Development Type	Stage	Width (m)	Height (m)	Length (m)	Material Type	Cost Category
Vein Drive - Waste (Jackleg) - Small Profile	1&2	1.6	1.8	22,254	Waste	Opex
Drawpoint (Jackleg) - Small Profile	1&2	2.6	3.1	18,164	Waste	Opex
Drawpoint (Jackleg) - Large Profile	1&2	3.9	3.1	3,586	Waste	Opex
Ventilation Adit	2	3.9	3.7	22	Waste	Capex
Explosive Magazine	1&2	3.9	3.7	800	Waste	Capex
Pump Station and Sumps	1&2	6.1	5.0	1,015	Waste	Capex
Electrical Substation	1&2	4.6	4.0	233	Waste	Capex
Hashui Creek Diversion Tunnel	1	3.0	3.7	579	Waste	Capex
<b>Subtotal</b>				<b>96,258</b>		
<b>Total Lateral Development</b>				<b>115,601</b>		

Table 16.4 summarizes the vertical development requirement, profiles and their various categories.

**Table 16.4 Vertical Development Profiles and Categories**

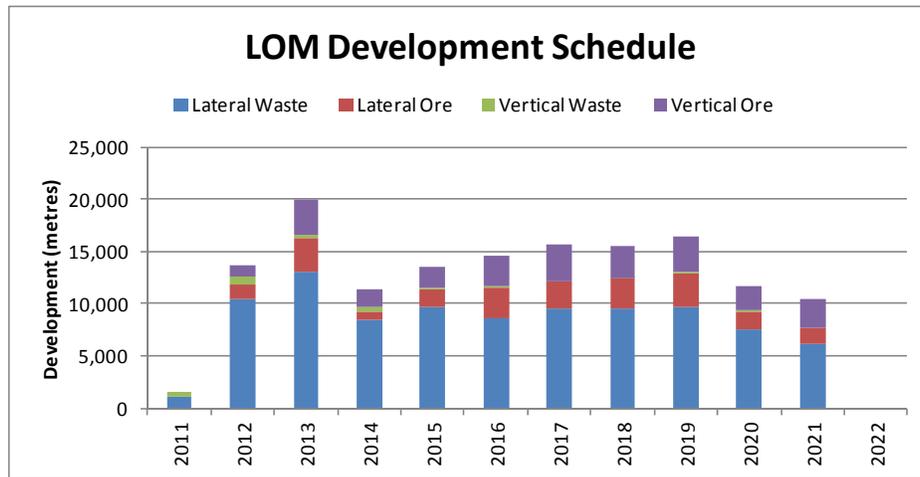
Vertical Development Type	Stage	Width (m)	Height (m)	Length (m)	Material Type	Cost Category
<b>Conventional Sink</b>						
Ramp Shaft (Intake)	1	3.5	na	192	Waste	Capex
Ventilation Shaft (Exhaust)	1	3.5	na	170	Waste	Capex
Ventilation Shaft (Exhaust)	2	3.5	na	936	Waste	Capex
Ore Pass	2	2.5	na	100	Waste	Opex
Waste Pass	2	2.5	na	100	Waste	Opex
Main Shaft	2	6.0	na	618	Waste	Capex
Main Shaft Ore Pass	2	2.5	na	325	Waste	Capex
Main Shaft Ore Bin	2	3.5	na	30	Waste	Capex
Main Shaft Sump Access	2	4.17	3.4	70	Waste	Capex
<b>Subtotal</b>				<b>2,541</b>		
<b>Jackleg Development</b>						
Stope Raise	1&2	1.6	1.8	26,350	Ore	Opex
Water Drainage Chute	1&2	1.0	na	45	Waste	Capex
<b>Subtotal</b>				<b>26,395</b>		
<b>Total Vertical Development</b>				<b>28,936</b>		

Table 16.5 summarizes the development waste produced by stage.

**Table 16.5 Development Waste**

Development Waste	Tonnes	LOM %
Stage 1 Development	821,553	40
Stage 2 Development	1,255,503	60
<b>Total Waste</b>	<b>2,077,056</b>	<b>100</b>

Figure 16.9 Development Profile by Type



### 16.8.2 Shafts

Several shafts are planned for the LOM design with four of the shafts connected to surface. All shafts will be by conventional underhand sink method.

Table 16.6 summarizes the general details for each shaft.

Table 16.6 Mine Shafts

Shaft Name	Mine Stage	Diameter (m)	Collar Elevation (mRL)	Bottom Elevation (mRL)	Depth (m)	Profile	East Collar Co-ordinate	North Collar Co-ordinate	Collar Access
Ramp Shaft	1	3.5	132	-50	182	Circular	37,593,379	2,535,987	Surface
Stage 1 Ventilation Raise	1	3.5	120	-50	170	Circular	37,594,018	2,535,748	Surface
Stage 2 Ventilation Raise	2	3.5	136	-50	186	Circular	37,594,396	2,535,595	Adit
Stage 2 Ventilation Raise	2	3.5	-50	-300	250	Circular	37,593,803	2,535,916	Internal
Stage 2 Ore Pass	2	2.5	50	-50	100	Circular	37,593,972	2,535,755	Internal
Stage 2 Waste Pass	2	2.5	50	-50	100	Circular	37,593,977	2,535,769	Internal
Shaft Name	Mine Stage	Diameter (m)	Collar Elevation (mRL)	Bottom Elevation (mRL)	Depth (m)	Profile	East Collar Co-ordinate	North Collar Co-ordinate	Collar Access
Main Shaft	2	6.0	248	-370	618	Circular	37,593,562	2,535,544	Surface
Main Shaft Ore Pass	2	2.5	-100	-300	200	Circular	37,593,548	2,535,552	Internal
Main Shaft Ore Bin	2	3.5	-300	-330	30	Circular	37,593,548	2,535,552	Internal
Main Shaft Sump Access	2	4.17 x 3.40	-300	-370	70	Rectangular	37,593,567	2,535,575	Internal

### 16.8.3 Capital and Operating Development

The capital development is notionally the Ramp, level access and level rock transportation routes.

The operating development is notionally the portions of the level access that provide immediate access to a stope (i.e. the strike extent of the stope), the draw point accesses and the vein development including exploration vein development.

Table 16.7 summarizes the capital and operating development requirements for the LOM design.

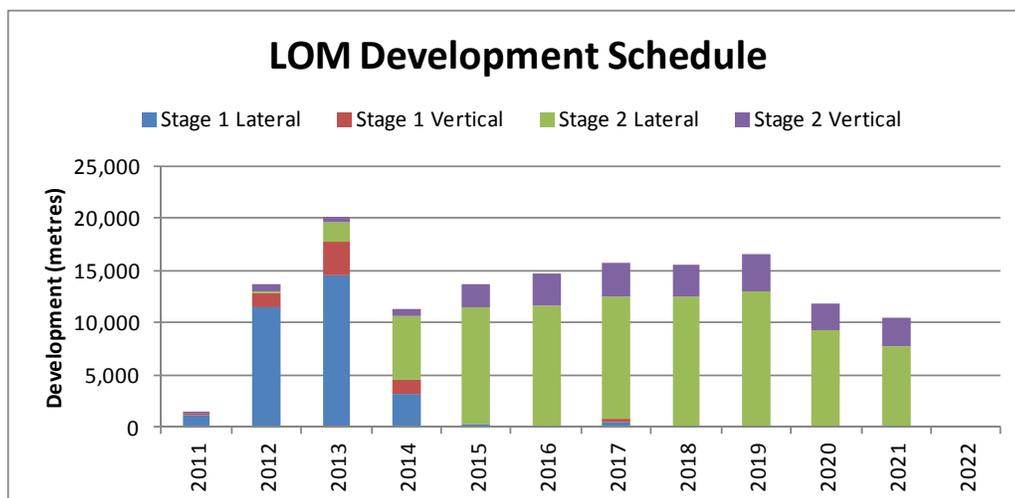
**Table 16.7 LOM Capital and Operating Development Summary**

Year	Capital Lateral	Capital Vertical	Operating Lateral	Operating Vertical	Total
2011	1,063	463	0	0	1,527
2012	8,169	768	3,592	1,154	13,682
2013	8,242	169	8,014	3,586	20,011
2014	6,656	383	2,604	1,658	11,300
2015	5,902	153	5,477	2,058	13,590
2016	4,023	150	7,430	3,017	14,620
2017	3,733	0	8,390	3,517	15,640
2018	3,545	50	8,881	3,030	15,506
2019	3,275	100	9,703	3,405	16,484
2020	2,525	150	6,670	2,382	11,726
2021	1,490	0	6,218	2,743	10,452
2022	0	0	0	0	0
Total	48,622	2,386	66,979	26,550	144,537

### 16.8.4 Development Stages

Figure 16.10 summarizes the development stages for the LOM design.

**Figure 16.10 Development Stages**



## 16.9 Mine Production

Mine production is the combination of development ore and stope ore.

AMC conducted a high-level review of the mine schedule using mine specific scheduling software (Mine2-4D and Earthworks Production Scheduler). The mine production schedule is considered by AMC to be below feasibility study standard (notionally within +/-10-15% on the inputs) with respect to the production volatility indicated and protracted overlap of production for the mine stages. AMC expects that the production schedule will be progressively refined as the mine is developed, but does not anticipate any material change in the production levels achieved (<10%).

Table 16.8 LOM summarizes the LOM production.

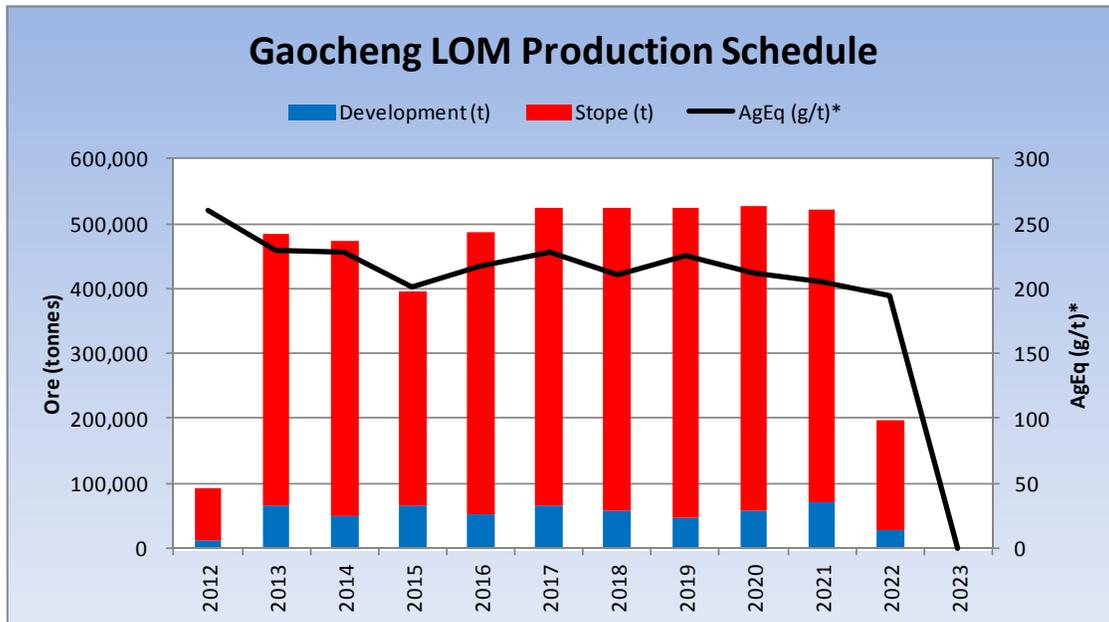
**Table 16.8 LOM Production Summary**

Year	Production (t)	Ag (g/t)	Pb (%)	Zn (%)	Ag(kg)	Pb (t)	Zn (t)	AgEq (g/t)*
2012	91,799	184	0.91	3.43	16,902	839	3,147	260
2013	484,547	159	1.02	2.87	77,065	4,942	13,929	230
2014	472,387	150	1.19	2.84	70,686	5,623	13,399	228
2015	395,153	115	1.36	2.55	45,439	5,362	10,080	202
2016	487,570	110	1.43	3.03	53,602	6,945	14,787	217
2017	524,317	121	1.50	3.08	63,683	7,865	16,148	228
2018	525,385	120	1.34	2.74	62,906	7,039	14,379	211
2019	525,005	112	1.38	3.30	58,558	7,250	17,340	226
2020	526,510	110	1.30	2.99	57,709	6,841	15,768	212
2021	520,672	97	1.38	2.96	50,320	7,163	15,391	205
2022	196,319	91	1.13	2.99	17,876	2,220	5,867	194
2023	0	0	0.00	0.00	0	0	0	0
<b>Total</b>	<b>4,749,665</b>	<b>121</b>	<b>1.31</b>	<b>2.95</b>	<b>574,747</b>	<b>62,089</b>	<b>140,234</b>	<b>218</b>

The key issue for the production schedule will be the production dip in 2014. The dip will be due to the Main Shaft development (shaft, ore pass, ore bin, loading pockets, preliminary level plats, pump station, etc) being scheduled for completion in 2014. There will be six months of shaft furnishing with no rock hoisting assumed during this phase, which is estimated to finish towards the end of 2014. There will then be significant development required from the Main Shaft initial plat development to access the stoping areas plus establishment of a primary vent circuit prior to production commencing from the -100 mRL and lower levels.

The life-of-mine (LOM) production duration will be 12 years. The average production rate will be 496 ktpa of ore from 2013 to 2021 inclusive. The steady state mine production rate will be 518 ktpa of ore from 2016 to 2021 inclusive. Figure 16.11 summarizes the LOM production profile for the development and stope ore starting from 2012, the first year of significant production. Note (\*) that the stated AgEq grade is the recovered metal equivalent.

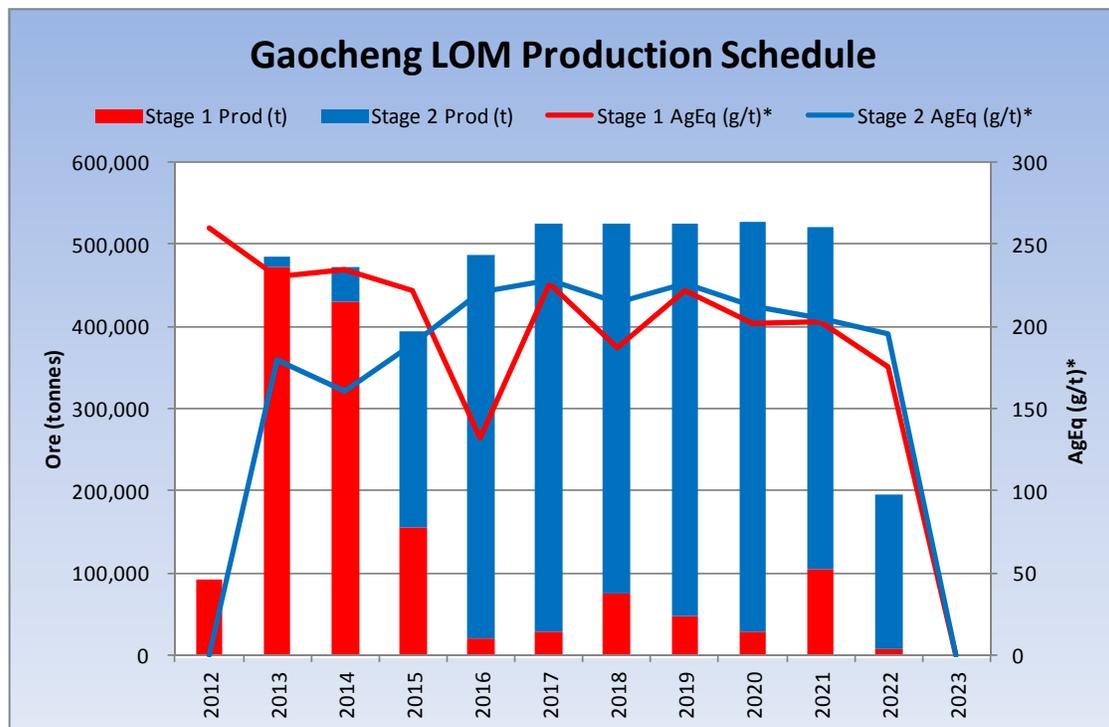
Figure 16.11 LOM Production Profile by Type



The Stage 1 production ramp up period to the LOM average production rate will be approximately two years from the project start in August 2011.

Figure 16.12 summarizes the LOM production profile by stage. Note (\*) that the stated AgEq grade is the recovered metal equivalent.

Figure 16.12 LOM Production Profile by Stage



## 16.10 Rock Handling

Total ore and waste produced from the mine design is approximately 4.7 Mt and 2.3 Mt respectively.

Stage 1 ore will be trucked to surface, representing approximately 31% of the LOM production, with the remainder shaft hoisted to surface.

All waste for the mine plan is assumed to be disposed of at surface. Stage 1 waste will be trucked to surface, representing approximately 37% of the LOM waste, with the remainder transported by rail car cage hoisting.

Hand sorting of waste from ore at surface will be conducted opportunistically.

### 16.10.1 Shaft Hoisting

The shaft will have two tower mounted multi-rope friction winders. One will be used for skip hoisting of ore (400 kW), and the other for cage hoisting (252 kW) of waste, labor, materials and mine equipment access for below the -50 mRL. The shaft will also be used for intake air, services access (ladder, cables and pipes) and labour emergency egress.

The shaft diameter is 6.0m diameter with the collar at +248 mRL and sump at -370 mRL. Skip loading facilities are located at -330 mRL. A spillage shaft from -300 mRL to the sump will have a hoist and ladderway for sump cleaning activities. Shaft spillage will be tipped into the ore bin.

The Main Shaft hoisting capacity is estimated to be approximately 629 ktpa skip ore (106 t/hr) and 106 ktpa cage waste (320 t/d). The capacities are estimated based on 330 days per year, 3 shifts per day and 8hr shifts.

The Main Shaft ore pass (2.5m diameter) is located 15m north-west of the shaft. The ore pass will be vertical (200m) with inclined tipping passes from each shaft level. Ore will gravitate to the ore bin below -300 mRL (240m<sup>3</sup>) which will feed via a vibratory feeder (460-519 t/hr) to a transfer conveyor (719 t/hr) to the skip loading weighing flask with a maximum 350 mm lump size. Skips (3.2m<sup>3</sup>) will discharge into a surface ore bin which will feed onto a conveyor (719 t/hr) to the crusher feed bin.

Waste that is cage hoisted in rail cars to surface will be transferred to the rail waste dump tip head by locomotive that is within 200m of the Main Shaft.

### 16.10.2 Waste Dump

The total LOM waste produced will be 2.3 Mt with 0.9 Mt produced from Stage 1.

The +215 mRL waste dump tip head capacity is 275,000m<sup>3</sup> (~558 kt). This accommodates the initial two years of waste produced. Silvercorp has not provided AMC details for the final waste dump capacity, but AMC's site visit and review of the surface plans indicate there appears to be room for a downstream extension of the waste dump location and/or ability to increase the waste dump height to approximately +300 mRL to accommodate all waste produced.

Waste may opportunistically be disposed of into the Shrinkage stope voids (with approximately 1.2 Mm<sup>3</sup> or 2.3 Mt void capacity) but this is not in the current mine plan, and the method for

potential tipping of waste into the stopes has not been clarified by Silvercorp. The potential for waste disposal into Shrinkage stope voids represents 100% of the LOM waste produced.

Waste may also be consumed for local construction works such as roadways, hardstand areas, tailings dam retainer walls and other miscellaneous infrastructure foundations.

### 16.10.3 Stage 1 Rock Transportation

Stage 1 development will be predominantly jumbo development with jackleg development required for the smaller profile development such as the stope draw points and vein exploration development.

Stage 1 will use LHD for mucking and trucks for hauling rock either to the surface run-of-mine (ROM) ore stockpile located approximately 100m north of the Main Shaft at +248 mRL or to the waste dump tip head located approximately 200m east-south-east of the Main Shaft at +215 mRL (for Stage 1) and +300 mRL (for Stage 2).

Process plant personnel will re-handle the ROM ore by FEL to the primary crusher feed bin. The ROM pad (approximately 1600m<sup>2</sup>) will have a notional 5 day production capacity.

Ore truck haulage from the +100 mRL, +50 mRL, 0 mRL and -50 mRL levels to the Ramp portal entrance at +176 mRL will be at a 12% maximum gradient. Surface haulage from the Ramp portal entrance to the ROM stockpile (approximately 700m) will be via a concrete paved roadway (approximately 400m concrete pavement at 200 mm thickness at maximum 12% gradient and 300m of compacted gravel pavement at maximum 10% gradient). Ore loaded trucks will pass via a weighbridge on their way to the ROM stockpile.

Waste truck haulage to the waste dump uses the same Ramp portal access roadway until the Process Thickener Plant access junction (approximately 550m) at which point the trucks divert to the waste dump tip head (approximately 150m of compacted gravel pavement at flat gradient).

Trucks for ore or waste will either be directly loaded adjacent to the development and production faces or loader re-handled from stockpiles within 200m of the development or production face.

Stage 1 jackleg development will be mucked using a small LHD (2m<sup>3</sup>) to a stockpile with 100% re-handle by a larger LHD (3m<sup>3</sup>) into the trucks (20t) for haulage to the surface ROM stockpile or waste dump tip head.

Stockpiles for LHD operation will be generally within 200m of the mucking face; however, if mucking performance is inadequate, rock transportation to the LHD stockpile may be assisted by a small truck (5t).

### 16.10.4 Stage 2 Rock Transportation

Stage 2 development is entirely with jackleg.

Stage 2 development and production will use rail mounted electric over-throw loaders (0.3m<sup>3</sup>) to load side-tipping rail cars (10 x 0.7m<sup>3</sup>) that will be transported to the Main Shaft ore pass by electric locomotive (3t) using light rail (0.6m gauge and 15kg/m). Ore gravitates to the Main Shaft ore bin between -300 mRL to -330 mRL (288 m<sup>3</sup> capacity) and is skip hoisted to surface discharging onto a conveyor that feeds directly to the Process Plant primary crusher.

For ore production on and below -50 mRL, the ore is directly transported to the Main Shaft ore pass tip (located at approximately Mine Section 22 between -100 mRL and -300 mRL).

For ore production above -50 mRL, the ore is transported by rail cars and electric locomotive to the Stage 2 ore pass (located at approximately Mine Section 36 between +50 mRL and -50 mRL) and then re-handled on the -50 mRL to the Main Shaft ore pass, again by locomotive and rail car.

For development waste on and below -50 mRL, the rail cars will be taken to the Main Shaft for cage hoisting. Waste rail cars will be loaded two at a time into the shaft cage. At surface the rail cars are transported by locomotive to a rail tip head for side tipping via a 400m rail loop that runs adjacent to the waste dump.

For development waste above -50 mRL, the waste is transported by rail cars and electric locomotive to the Stage 2 waste pass (located at approximately Mine Section 36 between +50 mRL and -50 mRL) and then re-handled on the -50 mRL to the Main Shaft for surface cage hoisting. There is potential for Stage 2 waste above -50 mRL to be handled by truck to surface, but is not part of the current design.

There is rail-siding development adjacent to the Main Shaft for approximately 12 rail cars either side of the shaft for loaded and empty return rail cars.

AMC considers the proposed waste management design to be inefficient and may ultimately require an independent waste pass system, in parallel with the Main Shaft ore pass system, with campaigned waste skip hoisting to be established. AMC has not determined the impact of this, but does not believe it will materially affect the project schedule or economics.

#### **16.10.5 Stope Backfill**

Backfill such as tailings or development waste will not be required for the Shrinkage and Resue stoping methods.

The Shrinkage stope method uses the blasted ore as the working platform for each stope lift. The ore is removed on completion of stope mining leaving an empty void. There is potential to opportunistically dispose of development waste into these voids, but current mine plans do not make allowance for this.

The Resue stoping method uses blasted waste from the footwall (to achieve the minimum mining width) as the working platform for each stope lift. The waste remains in the stope at completion of stope mining.

#### **16.11 Mine Services**

The mine services are described in the following sub sections.

##### **16.11.1 Ventilation**

Mine ventilation will be practiced as set out by Chinese laws and regulations. The key ventilation regulations (but not limited to) are; minimum ventilation volume per person (4 m<sup>3</sup>/min/person), minimum ventilation velocity (typically 0.25-0.50 m/sec dependent on location or activity) and minimum diluting volume for diesel emissions (4 m<sup>3</sup>/min/kW).

The primary ventilation will generally flow from west to east using the main levels interconnected by dedicated level vent raises (plus active stope accesses). The upper level(s) where stoping has been completed will be used as return airways to separate the fresh and exhaust air. A series of air doors and sealed walls will be utilised in the ventilation system. Inactive development headings and draw points will be sealed to enhance the ventilation circuit by minimising leakage.

AMC notes there is a protracted overlap in the mine production stages. AMC reviewed the ventilation criteria for each stage's peak requirements by simulating the development and operation air requirements in those years (2015 for Stage 1 and 2018 for Stage 2 using Ventsim software) and made adjustments to the Silvercorp design and costs estimates to reflect the following findings.

#### 16.11.1.1 Stage 1 Primary Ventilation

The ventilation volume for Stage 1 is predominantly influenced by the diluting volume for the diesel equipment fleet. The peak ventilation volume is estimated to be 205 m<sup>3</sup>/sec, inclusive of 30% air leakage.

The key issue for the Stage 1 ventilation is the Ramp intake airway velocity being maintained below a regulatory 8 m/sec. This will be managed by appropriately sizing the Ramp Shaft diameter and level accesses to the Ramp Shaft.

The fresh air intake airways are:

- Ramp (4.2m x 3.6m located approximately at Mine Section 26) of 73 m<sup>3</sup>/sec at the portal.
- Ramp Shaft (3.5m diameter located at Mine Section 10) of 132 m<sup>3</sup>/sec at the collar. Friction factor assumes the shaft is furnished with a ladder way

The return air exhaust airways are:

- Stage 1 Ventilation Shaft (3.5m diameter located approximately at Mine Section 36). The fan duty point is 155 m<sup>3</sup>/sec at 1,960 Pa. Friction factor assumes the shaft is furnished with a ladder way. The exhaust fan configuration will be axial (200 kW – 380V) mounted vertically with a fan diffuser for silencing.
- ML6 Adit drift (4.2m x 3.6m located approximately at Mine Section 10). The fan duty point is 50 m<sup>3</sup>/sec at 960 Pa. The exhaust fan configuration will be axial (75 kW – 380V) mounted horizontally with a fan diffuser for silencing.

The key airway regulation requirements are:

- Man access door within the +100 mRL level access development.
- Vehicle doors (airlock system) within the +50 mRL level access development.
- Exhaust regulators in the access development to the Stage 1 Ventilation Shaft on the +100 mRL and +50 mRL.
- All rock passes are assumed to be filled with rock for leakage purposes.
- All stope and inter-level ventilation raises include ladderway resistances.

### 16.11.1.2 Stage 2 Primary Ventilation

The ventilation volume for Stage 2 is predominantly influenced by the minimum air velocity for the various development and production activities. No diesel equipment is required for Stage 2 mining. The peak ventilation volume is estimated to be 140 m<sup>3</sup>/sec inclusive of 30% air leakage.

The fresh air intake airways are:

- Main Shaft (6.0m diameter located approximately at Mine Section 22) of 88 m<sup>3</sup>/sec at the collar. Friction factor assumes hoisting furniture in the shaft. For hoisting intake airways, there is a regulatory requirement for air purification prior to a level receiving fresh air from the Main Shaft.
- Ramp (4.2m x 3.6m located approximately at Mine Section 26) of 18 m<sup>3</sup>/sec at the portal.
- Ramp Shaft (3.5m diameter located at Mine Section 10) of 33 m<sup>3</sup>/sec at the collar.

The return air exhaust airways are:

- Stage 2 Ventilation Shaft (3.5m diameter located approximately at Mine Section 52). The fan duty point is 140 m<sup>3</sup>/sec at 2070 Pa (total pressure). The friction factor assumes the shaft is furnished with a ladder way. The exhaust fan configuration will be axial (200 kW – 380V) mounted horizontally with a fan diffuser for silencing. The development on the inlet side is configured to enable emergency egress. The Stage 2 Ventilation Shaft will be developed internally within the adit with the fan installation also established within the adit drift development.

The key airway regulation requirements are:

- Vehicle access doors (airlock system) placed in the Ramp level accesses for the +100 mRL, +50 mRL and 0 mRL levels.
- Two regulators on the -100 mRL level and one on the -50 mRL level to force air to the lower level working areas.
- The Stage 1 Vent Shaft is sealed at the collar and is used as an internal exhaust in Stage 2.
- All rock passes are assumed to be filled with rock for leakage purposes.
- All stope and inter-level ventilation raises include ladderway resistances.

### 16.11.1.3 Secondary Ventilation

The secondary ventilation will consist of auxiliary fans for ventilating development faces, infrastructure chambers, loading and tipping areas and stope faces.

Mechanized development faces (Stage 1) will be ventilated using domestically manufactured fans (twin 30 kW - 380V, with 9 m<sup>3</sup>/sec at 950 Pa capacity). The maximum secondary ventilation run will be approximately 740m in the Ramp (within the fan stated duty). Ventilation ducting (600 mm diameter) will consist of rigid sections around curves and long flexible duct sections for the straights to minimize resistance and to also enable high development advance rates via ease of installation and repair.

Conventional development faces (Stage 1 and 2) will be ventilated using domestically manufactured fans (5.5 kW – 380V). A combination of forced and exhaust ventilation will be applied for long blind-heading distances as required.

Stopes will be force ventilated using domestically manufactured fans (4 kW – 380V) via the access timber cribbed travel way. The stope air returns to the upper level via a second access travel way at 50m strike spacing.

### 16.11.2 Water Supply

The source of water for the mine will be from local creeks and gullies that flow into the Hashui Creek. The flows will typically vary from 11,000 m<sup>3</sup>/day (dry season) to 69,000m<sup>3</sup>/day (wet season) with the wet season being from April to September inclusive. The annual average rainfall varies 1400-1734 mm. The water quality and quantity from the local creeks will be sufficient to meet the project requirements which is estimated be 2,093m<sup>3</sup>/day.

Water will be drawn from the Bai Mai reservoir (at approximately Mine Section 56 and elevation 105 mRL) and pumped to an elevated hill top (at approximately 343 mRL) for water treatment – filtration and surge capacity storage. The treated water is then gravity fed to the mine site and treatment plant (at approximately 248 mRL).

The key specifications of the water supply system are:

- Bai Mai reservoir water tank with 100m<sup>3</sup> of settle capacity and 200m<sup>3</sup> clean water capacity.
- Hill top water tank with 300m<sup>3</sup> storage and water filtration capacity of 85 t/hr (via two filtration units).

Water consumption at the mine will primarily be for drilling and suppressing dust.

Potable water will be provided underground adjacent to the Main Shaft with water quality conforming to sanitary standards. Labour will carry drinking water as required to remote work place in water containers.

### 16.11.3 Dewatering

Underground water will be discharged to surface using conventional centrifugal pumps via pipe lines installed in the Ramp, Ramp Shaft and Main Shaft. Underground water pumped to surface will be collected in ponds at the Ramp portal or Main Shaft for sediment settling prior to being pumped to the process plant water treatment station.

Three pumps will be installed in each of pump chamber. Under normal water inflow conditions one unit will be running, one unit will be under maintenance, and the other will be on standby. Under maximum water inflow conditions, two pumps will be running. Underground pumps will be specified for clean water discharge, so each pump station will have its own twin compartment sediment settling arrangement. The capacity of these will be equivalent to 6 to 8 hours of normal water inflow condition (Safety Regulations on Metal and Nonmetal Mining Operation – National standard GB16423-2006).

Quality monitoring of the mine waters and the surrounding receiving surface waters will be conducted on a regular basis.

The normal mine ground water inflow conditions for Stage 1 and 2 are estimated to be 2830 and 3081 m<sup>3</sup>/d respectively (e.g. -50 mRL and -300 mRL respectively), and the maximum mine water inflow conditions for Stage 1 and 2 are estimated to be 5661 and 6161 m<sup>3</sup>/d respectively. Estimates are based on three hydro-geology holes which AMC does not consider to be feasibility study accuracy. AMC however does not believe there will be a material change to the estimates based on the topography relief. Mine make water is estimated to be 600 m<sup>3</sup>/d for both stages.

Pump demand under normal conditions will be approximately 13 to 14 hours per day for Stage 1 and 2 respectively and under maximum conditions will be approximately 12 hours per day for both stages. Pump station sumps will provide 6 hours of water inflow capacity.

#### **16.11.3.1 Stage 1 Primary Dewatering**

For Stage 1 the rising main will trace the decline to +50 mRL and then to surface at +113 mRL via the Ramp Shaft. A single lift discharge has been specified from the -50 mRL pump station. The -50 mRL pump discharge capacity will be 280 m<sup>3</sup>/hr for a 344m head (400 kW).

There will be a drainage hole adjacent to the Ramp accesses connecting the levels to gravitate the mine water to the -50 mRL pump station, and to prevent water entering the Ramp.

#### **16.11.3.2 Stage 2 Primary Dewatering**

Stage 2 will have two pump stations; one located on the -100 mRL level and the other on the -300 mRL level. There will also be a minor shaft sump station at -370 mRL. The rising main will be via the Main Shaft.

A single lift discharge has been specified from the -100 mRL pump station to surface (+248 mRL). The -100 mRL pump discharge capacity will be 280 m<sup>3</sup>/hr for a 396m head (500 kW), the same as for the -50 mRL station.

The -300 mRL pump discharge capacity is notionally 155 m<sup>3</sup>/hr for a 335m head (220 kW) and will be fully specified during the operation phase.

There will be a drainage hole adjacent to the Main Shaft accesses connecting the levels to gravitate the mine water to the -300 mRL pump station, and to prevent water entering the Main Shaft.

There will be a Main Shaft sump pump arrangement that will discharge into the -300 mRL pump station. The pump discharge capacity will be 30 m<sup>3</sup>/hr for a 110m head (18.5 kW). Sump silt removal will be by rail car (0.5m<sup>3</sup>) hoisted to surface.

#### **16.11.3.3 Secondary Dewatering**

Conventional compressed air diaphragm and/or electric submersible pumps will be used for Ramp face dewatering on an as-needed basis. Water will be stage discharged via a pump line to the surface settling pond or the -50 mRL pump station.

Levels will be self draining (0.3 % gradient) to either the Ramp access or Main Shaft access drainage holes. Drains will be constructed from 245 mm diameter half pipes.

#### **16.11.4 Power Supply**

Power will be provided from a 110 kV substation near Gaochun town, about 6 km from the mine site which is fed from the Guangdong Province electrical grid system.

High voltage supply will be 10 kV to the surface sub-stations. The mine will have standby diesel generator power for essential mine facilities (pump stations, shaft operations, primary ventilation fans).

Underground sub-stations will be located on each level adjacent to the V2 vein. Level development utilizing jumbo development will incorporate additional sub-stations along the level to manage any voltage drop from the sub-station.

Low voltage supply from the underground sub-stations will be 415V (jumbo), 380V (pumps and fans) and 220-250V (lighting and rail operation).

#### **16.11.5 Fuel Storage and Dispensing**

No fuel will be stored underground. Stage 1 jumbo drills will be re-fuelled using a dedicated purpose vehicle with appropriate safety equipment.

Stage 1 trucks and loaders will be re-fuelled at the surface fuel farm and dispensing facility.

#### **16.11.6 Compressed Air**

Compressed air will be primarily used for drilling blast holes. Jackleg drilling will be used in the stopes and conventional development faces. There will be some minor use for shotcreting, blast hole cleaning and ANFO charging of blast holes.

Compressed air will be reticulated to all levels and to the emergency refuge stations.

Compressors (electrically powered two-stage piston compressors) will be located adjacent to the Ramp portal (2 x 20 m<sup>3</sup>/min, 0.8 Mpa, 110 kW) and Main Shaft brace area (2 x 40 m<sup>3</sup>/min, 0.8 Mpa, 250 kW). Compressed air will be reticulated using steel and plastic piping for air distribution via the Ramp and Ramp Shaft for Stage 1 and the Main Shaft for Stage 2.

AMC's review indicates additional compressor capacity may be required to cater for an estimated demand of approximately 140m<sup>3</sup>/min. AMC does not consider the impact to be material.

#### **16.11.7 Communications**

Mine surface communications will be available by landline and mobile phone services.

Mobile service coverage will be available for the GC mine site and Silvercorp's GC project development offices in the Gaochuntownship.

Telephones will be the base means of communicating with the underground. Phones will be located adjacent to the Ramp level accesses for Stage 1 and adjacent to the Main Shaft level accesses (plat area) for Stage 2.

Silvercorp intends to install a leaky feeder communication system for safety, operations supervision and maintenance activities.

### **16.11.8 Explosive Storage**

Silvercorp estimates approximately 182t of bulk explosive and 182,000 detonators will be consumed annually. The surface explosives magazine will be permitted to hold 10t of bulk explosive and 15,000 detonators representing approximately 15 days and 30 days of supply respectively. Security services will be used and detonators will be scanned on release from the magazine for security audit purposes.

Underground working party magazines will be located adjacent to each level's return air shaft (approximately Mine Section 36 for Stage 1 and Mine Section 54 for the upper levels of Stage 2) and will be limited to one day of requirement for bulk explosives and three days of requirements for blasting ancillaries.

### **16.11.9 Mine Equipment Maintenance**

The mining contractor will have its own mobile equipment workshop for repairs and servicing located adjacent to the Ramp portal. This predominantly caters for the Stage 1 requirements. There will also be underground drill service bays established in redundant stockpiles to minimize tramming delays for the slower moving drills.

Mobile equipment (trucks, loaders, etc), other equipment breakdowns and equipment major services will be conducted in the mining contractor's surface workshop with minor services conducted in redundant stockpiles for Stage 1.

Minor equipment (such as jacklegs, secondary fans, development pumps, etc) will be serviced in the mining contractor's surface workshop.

The electric locomotive and rail cars for Stage 2 will be serviced and repaired in a service rail sidings located adjacent to the Main Shaft.

Other fixed and mobile plant (primary pumps, surface electric locomotive, rail cars, vehicles, etc) will be serviced in Silvercorp's surface workshop located adjacent to the Main Shaft. This will be fully equipped with overhead crane, welding, electrical, hydraulic, lathe services, etc.

### **16.11.10 Roadway Maintenance**

Underground roadway maintenance for trackless equipment in Stage 1 (Ramp and levels down to -50 mRL) will be undertaken by the mining contractor using manual hand cleaning methods and LHD as required. No grader is allowed for.

The roadway base will be approximately 200m depth of size graded material predominantly sourced from development waste.

## **16.12 Mine Equipment**

All mobile equipment and some minor fixed plant will be provided by the mining contractor.

The Silvercorp fixed plant will be predominantly domestically manufactured and locally sourced (Guangdong Province). The equipment manufacturers are well known and commonly used for China mine operations.

Table 16.9 summarizes the typical type and capacity of the contractor equipment and

**Table 16.9 Mining Contractor Typical Key Equipment Summary**

Contractor Equipment	Stage	Units	Manufacturer	Model	Capacity
Single Boom Jumbo	1	2	Atlas Copco	Boomer 281	3660 mm rod
LHD small	1	2	Anhui Tongguan Machinery Co. Ltd.	TCY-2A	2m <sup>3</sup>
LHD large	1	3	Anhui Tongguan Machinery Co. Ltd.	TCY-3	3m <sup>3</sup>
Haul Truck	1	8	Anhui Tongguan Machinery Co. Ltd.	JZC-20	20t
Services Platform	1	1	Anhui Tongguan Machinery Co. Ltd.	JY-5	5t
Personnel Carrier	1	1	Anhui Tongguan Machinery Co. Ltd.	JY-5YR-16	16 persons
Shotcreter	1 & 2	2	Hunan Changde Shotcrete Machinery Factory	HPZ-6	6 m <sup>3</sup> /hr
Electric Locomotive	2	6	Jilin Longed Iron Alloy Co. Ltd.	ZK3-6/250	20.9t
Electric Loader	2	30	Nanchang Hengye Mining and Metallurgical Machinery Factory	Z-30	0.3m <sup>3</sup>
Rail Cars	2	100	Henan Hebi Mishi Machinery Co. Ltd.	YCC0.7-6	0.7m <sup>3</sup>
Auxillary Stopping Fan	1 & 2	40	Zib Ventilation Machine Plant Ltd.	JK56-N <sub>0</sub> 4	0.1~3.4 m <sup>3</sup> /hr
Auxillary Development Fan	1 & 2	20	Zib Ventilation Machine Plant Ltd.	JK-58N <sub>0</sub> 4	2.2~3.5 m <sup>3</sup> /hr
Mobile Refuge Chamber	1 & 2	3	Chongqing Research Institute of China Kegong Group Corp.	JYZY-96/12A	12 person, 96 hrs
Static Refuge Chamber	1 & 2	5	Xuzhou Yongxing Mechanical Manufacture Ltd.	ACYX-24	24 person, 120 hrs

Table 16.10 summarizes the typical type and capacity of the fixed plant as proposed by Silvercorp to develop and produce at 1,600 t/d.

**Table 16.10 Silvercorp Typical Fixed Plant Summary**

Silvercorp Equipment	Stage	Units	Manufacturer	Model	Capacity
JKMD2.6x4 Multirope Friction Hoist - Skip	2	1	Citic Heavy Machinery Manufacturing Company Ltd	Z4-315-22	106t/hr - ore
JKMD2.25x4 Multirope Friction Hoist - Cage	2	1	Citic Heavy Machinery Manufacturing Company Ltd	Z4-450-32	320t/day - waste
Primary Fan	1 & 2	2	Shandong Befeng Ventilation Machine Ltd	DK-40-8-N <sub>0</sub> 25/2x200	62.9-150.4 m <sup>3</sup> /sec
Ore Skip	2	1	Xuzou Coal mine safety equipment Limited	DjD1/2-3.2	3.2m <sup>3</sup>
Waste and Service Cage	2	1	Xuzou Coal mine safety equipment Limited	4# lengthen cage	
Multiple Stage Centrifugal Pump -50 mRL	2	3	Changsha Canon General Pumps Company Ltd	D280-43x8	Q=280m <sup>3</sup> /hr, 344m head
Multiple Stage Centrifugal Pump -100 mRL	1	3	Changsha Canon General Pumps Company Ltd	200D65Bx6	Q=280m <sup>3</sup> /hr, 396m head
Multiple Stage Centrifugal Pump -300 mRL	2	3	Changsha Canon General Pumps Company Ltd	D155-67x5	Q=155m <sup>3</sup> /hr, 335m head

Silvercorp Equipment	Stage	Units	Manufacturer	Model	Capacity
Multiple Stage Centrifugal Pump -370 mRL	2	2	Changsha Canon General Pumps Company Ltd	D25-30x4	Q=30m <sup>3</sup> /hr, 110m head
Air Compressor - Ramp	1 & 2	2	Nanking Compressor Company Ltd	SDA110	20m <sup>3</sup> /min at 0.8Mpa
Air Compressor - Main Shaft	1 & 2	2	Nanking Compressor Company Ltd	SDA250	40m <sup>3</sup> /min at 0.8Mpa

### 16.12.1 Equipment Productivities

Table 16.11 summarizes the productivities assumed for the development and production activities.

**Table 16.11 Activity Productivities**

Development or Production Activity	Unit	Schedule Rate	Machine Type
Jumbo - Single Heading (Ramp)	m/mth	180	Single Boom Electric-Hydraulic
Jumbo - Single Heading (Levels)	m/mth	80	Single Boom Electric-Hydraulic
Jumbo - Multi Heading (Levels)	m/mth	240	Single Boom Electric-Hydraulic
Jackleg - Single Heading (Levels)	m/mth	70	Jackleg (YT-24 )
Jackleg - Multi Heading (Levels)	m/mth	200	Jackleg (YT-24 )
Jackleg - Slope Raises	m/mth	60	Jackleg (YT-24 )
LHD - Slope Mucking	t/mth	30,000	TCY-3 (3m <sup>3</sup> )
LHD - Development Mucking	t/mth	16,000	TCY-2A (2m <sup>3</sup> )
Truck - Production (to ROM Stockpile)	t/mth	14,000	JZC-20 (20t)
Truck - Development (to Waste Dump)	t/mth	13,000	JZC-20 (20t)
Rail Loader - Development & Slope Mucking	t/mth/level	13,000	Z-30 (0.3m <sup>3</sup> )
Rail Trucks - Production & Waste (to Main Shaft)	t/mth/level	24,000	ZK3-6/250 Loco & 10xYCC0.7m <sup>3</sup> cars

### 16.13 Mine Personnel

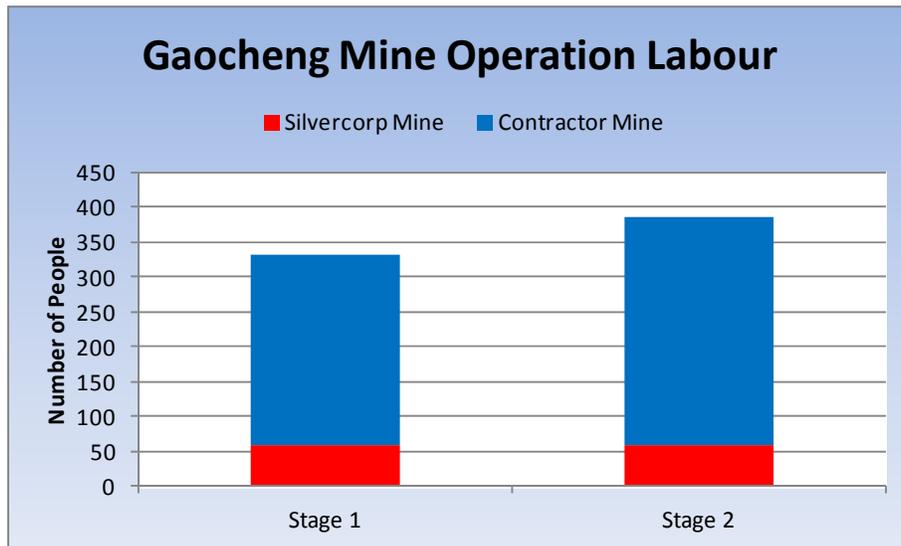
Silvercorp plans to operate the mine using contractors for development, production and the operation and maintenance of Silvercorp's fixed plant with Silvercorp providing its own management, technical services and supervision staff to manage the GC mine operation.

The mine will be operated on a continuous roster for 365 days per year working three 8 hour shifts per day.

Figure 16.13 summarizes the direct mine operation labour estimates by stage. The estimates have a peak requirement of 322 people during Stage 1 and a peak requirement of 377 people during Stage 2. The estimates exclude General and Administration (G&A) personnel, geological drilling, external consultants and process plant operation. The estimate depicts people on-site at

any point in time and does not account for the off-site labour panels, sick leave, absenteeism, annual leave, turn-over, etc.

**Figure 16.13 GC Mine Operation Labour Estimates by Stage**



#### 16.14 Mine Safety

Mine safety will be practiced as set out by Chinese health and safety laws and regulations.

There will be an OHS department for the GC mine, staffed with three mine safety trainer officers.

The mine and mining contractors will provide appropriate Personal Protective Equipment (PPE) to their own staff or miners. The PPE includes hard hats, safety boots, work gloves, face masks, and ear plugs.

The OHS departments will provide safety training, enforce the OHS policies and procedures, makes recommendations on mine safety issues and carry out daily inspections of the underground workings and explosive usages.

Safety committees will be headed by the General Manager and made up of the Deputy General Manager, Mine Superintendent, Safety Department Supervisor, and representatives of the mining contractor. The committees will be co-ordinated by the GC Safety Department. The mine management and the safety officers will be required to have valid mine safety training certificates issued by the Provincial Bureau of Safe Production and Inspection.

##### 16.14.1 Fire Prevention

Water for fire protection will be provided via the Ramp Shaft and Main Shaft with 216t surge capacity. Primary reticulation and secondary reticulation will be by 108 mm and 89 mm nominal bore pipes respectively and will be installed and maintained in accordance to national safety standards (Safety Regulations on Metal and Nonmetal Mining Operation – National standard GB16423-2006).

Fire extinguishers will be provided and maintained in accordance with regulations and good practice at the electrical installations, pump stations, service workshops, and locomotive garage and wherever a fire hazard is identified to exist.

Visible fire signs and fire safety notices will be posted in appropriate areas.

A suitable number of fire extinguishers will be provided and maintained at each stationary diesel motor and transformer substation.

Every light duty vehicle will carry at least one fire extinguisher of adequate size and proper type.

All heavy duty mobile mine equipment; loaders, trucks, drills, charge-up machines, etc., will be equipped with on-board fire suppression systems.

A mine-wide warning system will be installed at the main mine intake airway entries to alert underground workers to the event of an emergency. This will consist of audible alarms, ventilation status lights and stench gas.

#### **16.14.2 Mine Rescue**

Fully trained and equipped mine rescue teams will be site based with team members provided by the mining contractor and maintained on-site at all times. The mine rescue teams will be trained for surface and underground emergencies.

A mine rescue Emergency Response Plan will be developed, kept up to date, and followed in an emergency.

A mine rescue room will be provided in the surface mine offices adjacent to the Main Shaft.

An emergency clinic will also be maintained on-site and manned by a physician 24 hrs per day. Silvercorp will also have a contract established with the Yunfu General Hospital to provide emergency services and ambulance extraction to the hospital.

#### **16.14.3 Dust**

All broken rock will be wet down using hoses and sprays after blasting, prior to mucking and during mucking.

Decline roadway dust suppression will use a water cart with sprays on an as-needed basis.

Regular dust monitoring will be conducted as per regulatory requirements.

Personnel working in dust generating work areas will be provided with personal dust respirators.

#### **16.14.4 Emergency Egress**

Egress to surface will be available via all ventilation shafts, the Ramp, the Main Shaft and the ML6 adit connected to the +100 mRL level.

The Main Shaft and ventilation shafts will be equipped with staged ladder ways incorporating general mine services and will be partitioned from other shaft activities and provided with appropriate ventilation profile clearance and established in accordance with good practices.

Lateral egresses will be appropriately sign posted and maintained for walking access.

For Stage 1, personal self rescuer device will be provided as each person's personal protective equipment to gain access to a mine refuge station.

#### **16.14.5 Mine Refuge Station**

Static and/or mobile refuge stations will be established on each mine level with the exception of the +100mRL, which is not a production level.

The static refuge stations or mobile refuge chambers will be established in accordance with good practices with independent air supply (compressed or oxygen), communications, first aid, etc and of appropriate capacity to cater for the labour numbers in the active mine areas.

For the +50 mRL, 0 mRL and -50 mRL levels, mobile mine refuge chambers will be located in close proximity to the active development and production stopes in redundant stockpiles.

For the remaining levels from -100 mRL to -300 mRL static mine refuge stations will be located adjacent to the Main Shaft.

#### **16.14.6 Ablutions**

Facilities will be provided on each working level in the middle section adjacent (approximately Mine Section 32) to a return airway and will be cleaned and disinfected on a regular basis.

## 17 RECOVERY METHODS

### 17.1 Introduction

The key outcomes from the metallurgical testwork were summarized in Section 13.6 and of direct pertinence to this section on recovery methods are the following:

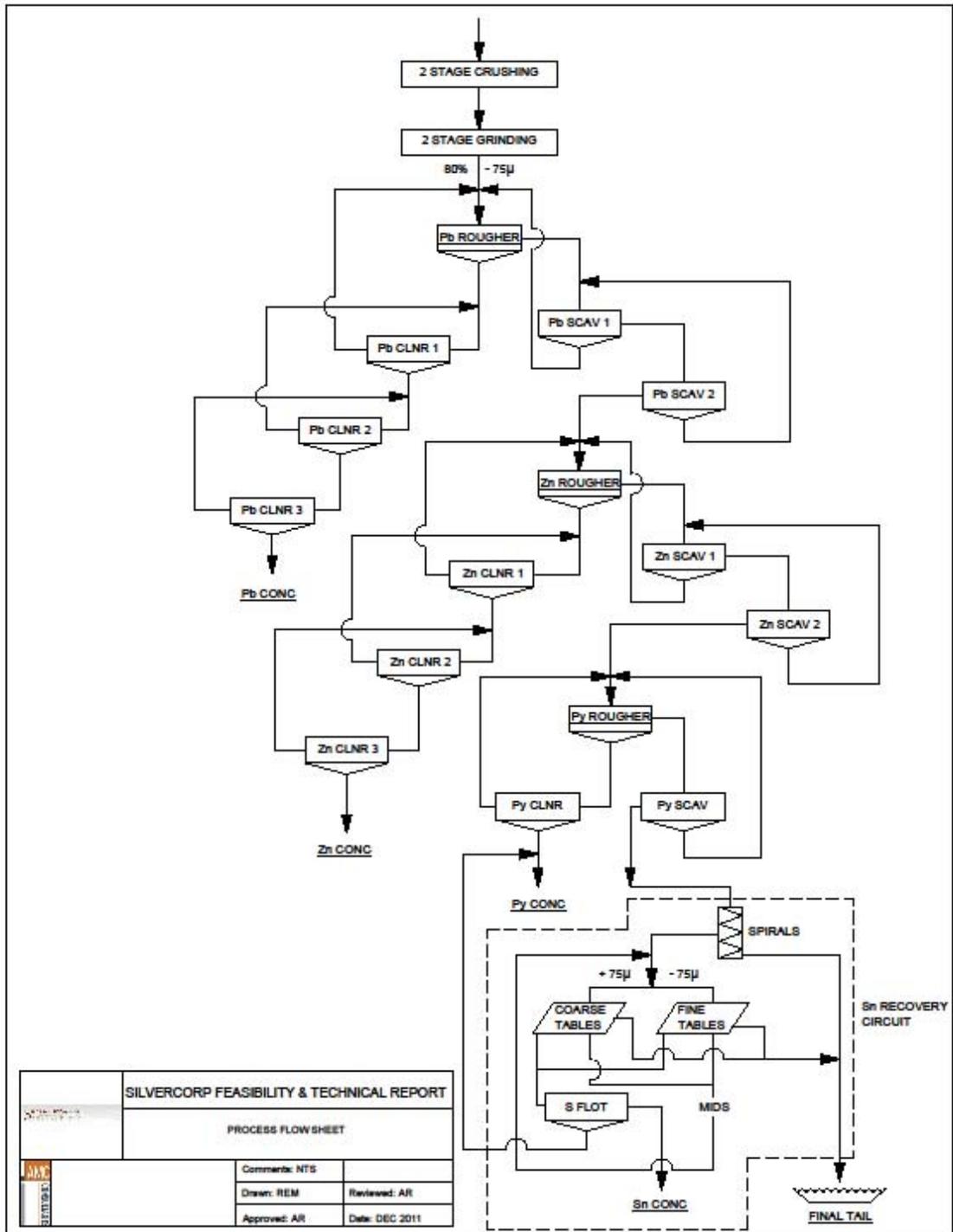
- The silver mineralogy indicates an optimization opportunity in increasing silver recovery from all species, including sphalerite and pyrite, to the lead concentrate, within the constraints of the minimum %Pb specifications. This has implications for lead cleaner circuit and filtration capacity.
- There is no comminution testwork as a basis for the crushing and grinding circuit design
- The flotation testwork culminating in the closed circuit test does provide an adequate basis for the flotation process design.
- Some circuit options were investigated, specifically copper-lead separation and tin recovery, and although these have been included in the GMADI Design Instructions, neither has been included in the financial model supplied by Silvercorp.
- AMC considers the copper-lead separation not to be viable, but in any case, is of such a small scale and therefore of such limited materiality that it is of little consequence to this report. Moreover there is only limited Cu resource data to support any copper recovery process.
- On the other hand, AMC believes a tin recovery circuit does have potential merit and although the base case of the report, consistent with the financial model, does not include it, it is considered as an opportunity and a material circuit option.

Note that in preparing this section the QP has also drawn on the visit to Silvercorp's Ying mine as evidence of a workable process design and appropriate equipment for a similar one.

### 17.2 Process Flowsheet

The process flowsheet is shown schematically in Figure 17.1, being essentially the same process as was adopted in the closed circuit flotation tests described in Section 13.4.3, with however the optional tin recovery circuit also shown.

Figure 17.1 Process Flowsheet



## 17.3 Process Description

### 17.3.1 Summary

The overall process consists of crushing, grinding, flotation of lead, zinc and pyrite concentrates, and concentrate dewatering, with the option of the tin recovery gravity separation circuit on pyrite flotation tails.

Two-stage crushing, with the second stage in closed circuit, from run of mine ore at 350 mm produces a -10 mm crushed product. This is followed by two-stage grinding in ball mills to a product size of 80% passing 75 $\mu$ m.

The flotation process consists of a standard sequential flotation of lead, zinc and pyrite with three-stage cleaning of the lead and zinc concentrates and single stage cleaning for pyrite. Concentrates are dewatered by conventional thickening and filtration.

The optional tin recovery circuit comprises spiral concentration followed by coarse and fine shaking tables with a final stage of flotation to remove residual sulphides.

The process design is based on the following overall throughput assumptions:

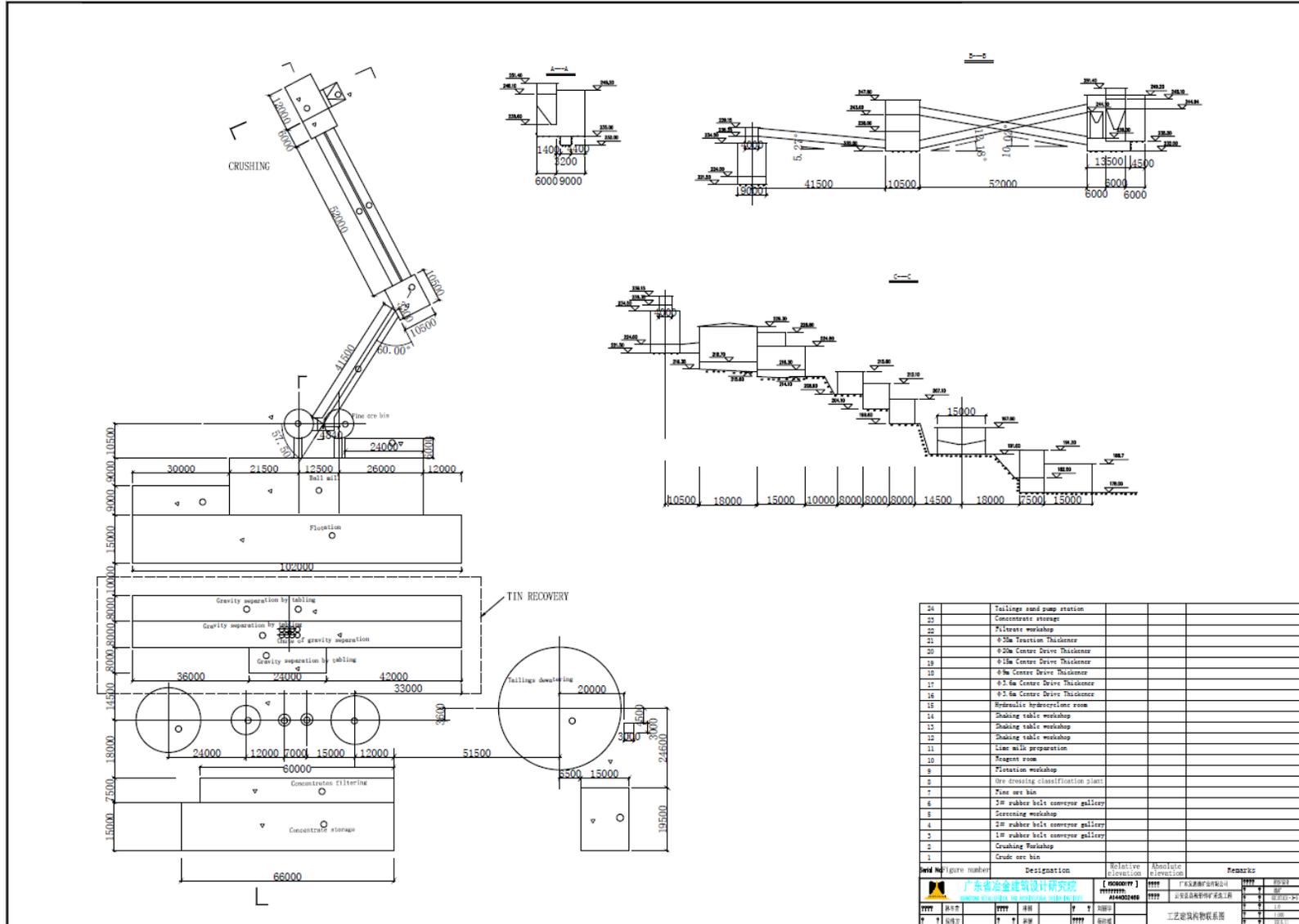
- 1,000 tpd feed base case (likely expansion to 1,600 tpd)
- 330 days per year
- Crushing 18 hrs/ day
- Grinding-flotation etc 24 hrs /day

No availabilities as such are cited; however the above parameters translate to 68% for crushing and 90% for grinding which AMC considers to be reasonable and in line with normal mining industry practice.

In all sections, space/capacity has been allocated for a expansion to 1,600 tpd. However the mine schedule very quickly ramps up to 500,000 tpa and the mill feed schedule naturally must closely match this. The implications of this are discussed in each section description.

A general site plan is shown in Figure 17.2.

Figure 17.2 Processing Plant Site Plan



### 17.3.2 Crushing

The crushing circuit design is based on Swedish Sandvik equipment in preference to Chinese crushers for reasons of more efficient crusher design and consequent savings in foundations and installation costs as well as better performance.

In the absence of comminution data, the primary jaw crusher and secondary cone crusher have been selected from the manufacturer's brochure and capacity ranges at the appropriate closed side setting. Although the GMADI document states that both the CJ408 jaw and the CH430 cone crusher been sized for 1,600 tpd, AMC notes that this appears to have been based on the upper end of the manufacturer's range and therefore should be considered optimistic.

The crushing circuit consists of a run-of-mine ore bin from which the ore is drawn by a vibratory feeder into the primary jaw crusher. The jaw crusher product is screened on a vibrating screen with the -10 mm fines being conveyed forwards to the fine ore bin while the +10 mm material feeds the secondary cone crusher via a buffer storage bin to maintain choke feeding of the crusher. The fine ore bin has a capacity of 1,600t i.e. 24 hrs at the higher throughput level

### 17.3.3 Grinding

In a similar fashion to the crushing section, the two-stage grinding circuit is claimed to be sized for 1,600 tpd. With no ball mill work index data available, AMC has attempted to back calculate operating work indices to test the reasonableness of the mill sizing, which is understood to be based on a scale-up from the similar operating Ying mine. At the 1,000 tpd base case, even allowing for 25% losses/design margin the back-calculated work index is 20.8 which gives considerable margin on the work index expected for this type of mid-sulphidation epithermal quartz vein deposit of around 16. However at 1,600 tpd and with no allowance for losses etc the calculated work index is 16.1 so once again the sizing would appear to be optimistic at the higher throughput.

Given that 1,600 tpd is the likely throughput and that an option was in fact considered of 2 x 2 x 7'x3.6' mills with 400 kW motors instead of the 2 x 2 x 7'x2.7' mills with 310 kW motors selected, AMC recommends that the 400 kW option be re-evaluated. This represents the minimum requirement to provide sufficient design margin; to cover the uncertainty of there being no testwork, the total additional power requirement should be 600 kW.

Typical of Chinese practice and conforming to the design successfully used at the Ying mine, the grinding circuit consists of a grate-discharge ball mill in closed circuit with screw classifier followed by an overflow ball mill in closed circuit with cyclones to achieve the desired flotation feed size of 80% passing 75µm in the cyclone overflow. The primary mills have weightometers fitted to the feed conveyors linked to a variable speed belt motor for mill feed control.

The circuit is configured in two parallel trains of 800 tpd each, for reasons of flexibility and ease of maintenance.

### 17.3.4 Flotation

Following on from the grinding circuit the flotation circuit is similarly configured in two parallel trains.

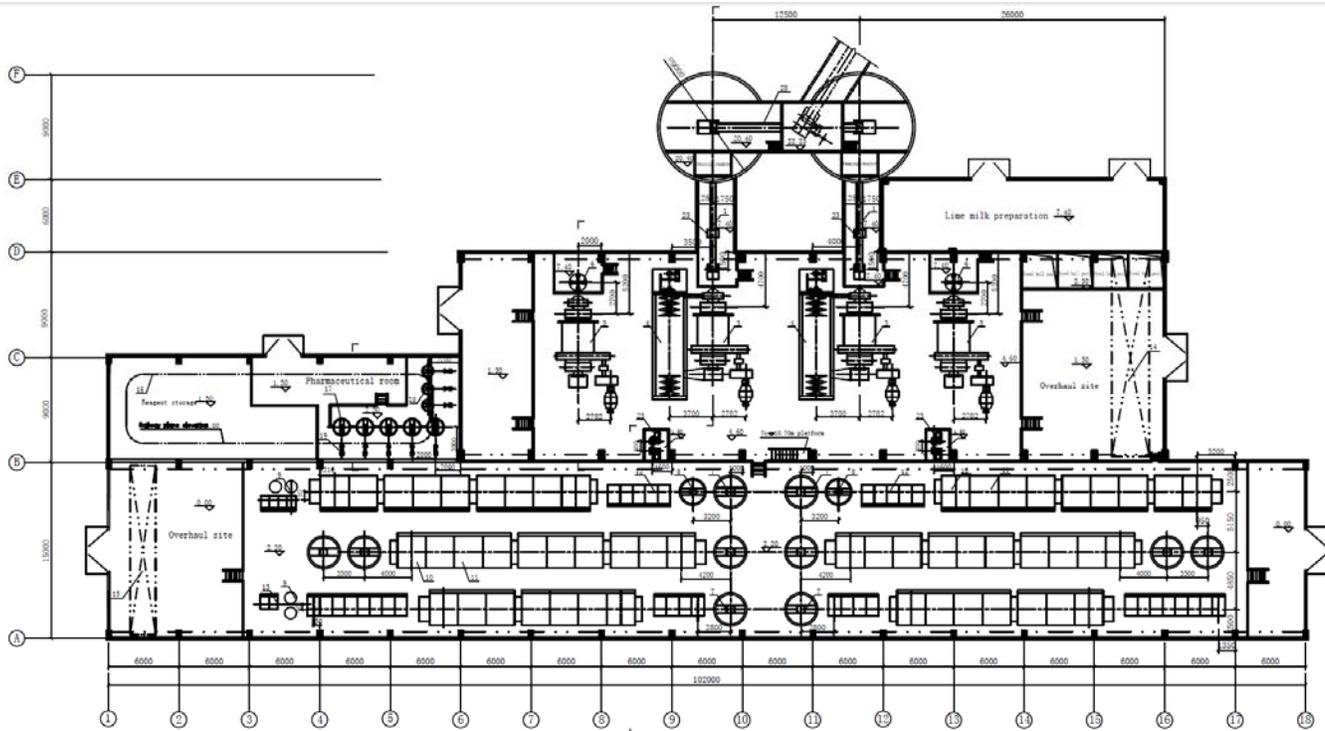
In this case the flotation cell sizing appears adequate for 1,600 tpd, with rougher residence time designed to be a minimum of 15 mins plus scavenger time also of 15mins, representing a 3x scale-up factor from the laboratory results. In practice the available cell sizes means these residence times are generally exceeded by about 20%. Conditioning times of the order of five minutes apply.

The general layout is compact and efficient making use of gravity and the sloping terrain as the banks follow successive parallel contour lines.

However in light of previous comments about recovering more silver to the lead concentrate which will result in increased concentrate mass flows, it would be desirable to have the flexibility to increase lead cleaner circuit capacity if required. This is a very small increase relative to total flotation cell volumes and would need only a few meters allowance of additional space.

Figure 17.3 shows a general arrangement plan of the grinding and flotation section.

Figure 17.3 Grinding/Flotation Plant General Arrangement



Relative elevation of  $\pm 0.00m$  is equal to absolute elevation 214.1m in this figure.  
 All elevations are process internal elevations in this figure.  
 Elevation is measured in meters in this figure.  
 Distance is measured in millimeters in this figure.  
 Lime milk preparation is showed in the figure of GD1371CS-X-15.

29	MD-0.5A handle pulley Q10.5t	1				0.007	0.007	
28	4J Roundtable belt conveyor with workman 4000*1150	1		7.5	1	3.50	3.50	
27	CD11-4D electric hoist Q1t	1		1.7	1	0.137	0.137	
26	Storage medicine barrel $\phi 1500 \times 1500$	5				0.36	4.45	
25	Nuclear Belt Balance (BMS20)	2		3.0	2	0.90	1.80	
24	Reagent Feeder JY-32	1		2.0	1	7.20	7.20	With automatic control device
23	Slurry Pump 4/5C-4R(R)	4		30.0	4	0.72	2.88	2.5m of lift, maximum and 1.5m of flow, maximum 2.5m of length
22	SPCL Vertical Slurry Pump	2		6.9	2	0.25	0.50	Used for the flotation working
21	Slurry Pump 20LS-A	2		4.2	2	0.36	0.72	Used for the concentrate, 1.5m of lift, 1.5m of maximum flow
20	Pipeline Sampler	7						
19	Chemical Pump DHS0-32-10EA	5		1.91				
18	Reagent Agitating Vat $\phi 1000 \times 1000$	3	Y90L-6	1.1	3	0.532	1.60	
17	Reagent Agitating Vat $\phi 1500 \times 1500$	5	Y132D-6	3.0	5	1.36	6.8	
16	CD12-4D Electric Hoist Q2t	1		5.4	1	0.232	0.232	
15	1 Type Electric Single-beam Crane Q=10-15.5t	1	20021-4	1.6	1	0.50	2.50	
14	8 Type Electric Sling Brakes Q=20/30/14-24.5t	1		27.4			18.0	
13	Flotation Machine BF-0.37	6	Y90L-4	1.5	6	0.47	2.82	
12	Flotation Machine BF-1.2	24	Y132M-6	5.5	24	1.37	46.58	
11	Flotation Machine XFP-10	36		22.0	36	4.334	156.02	
10	Flotation Machine XFP-10	16		30.0	16	4.894	78.31	
9	One Slurry Mixing Bucket $\phi 1000 \times 1000$	4	Y90L-6	1.1	4	0.532	2.12	
8	One Slurry Mixing Bucket $\phi 2000 \times 2000$	2	Y132M-6	5.5	2	1.47	3.34	
7	One Slurry Mixing Bucket $\phi 2500 \times 2500$	10	Y190L-6	13	10	3.44	34.4	
6	Hydraulic Cylinder $\phi 100mm$	5				0.17	1.36	1 series, 3 for utilization
5	Wet Overflow Ball Mills M02700 $\times$ 2700	2	JR440-0	310	2	46.0	132.0	1.5m of lift, maximum and 1.5m of flow, maximum 2.5m of length
4	Radwair Type Single-spiral Classifier FC-28	2	T100L-4	13.0	4	25.65	51.2	1.5m of lift, maximum and 1.5m of flow, maximum 2.5m of length
3	Wet Grate Ball Mills M02700 $\times$ 2700	2	JR440-0	310	2	46.0	132.0	1.5m of lift, maximum and 1.5m of flow, maximum 2.5m of length
2	Disk Ore Feeder DSE10	4	Y190L-6	1.5	4	0.79	3.16	
1	S/R (6#) Belt Conveyor TD75-6550 L=115	2		3	2	1.9	3.8	
Serial No	Designation and specification	Quantity	Model	Power (KW)	Quantity	Power weight	Crane weight	Remarks
Equipment list						Total: 706.1 t		
广东省冶金建筑设计研究院 GUANGDONG METALLURGICAL AND ARCHITECTURAL DESIGN INSTITUTE		[ EIC000117 ] A144022469		广东银禧有色金属有限公司 云南昆明白银矿采选工程		磨矿浮选车间平面图 1:100 2011.11		

### 17.3.5 Concentrate handling

The respective concentrates are thickened and then filtered on ceramic disk filters sized at 9m<sup>2</sup>, 15m<sup>2</sup>, and 30m<sup>2</sup> for the lead, zinc and pyrite concentrates respectively.

Once again with respect to lead concentrates AMC recommends increasing filter capacity (to 15m<sup>2</sup>) so there are no capacity constraints in maximizing silver recovery.

The filters are positioned above the concentrate storage shed for direct discharge and from which the concentrates are loaded by front-end loader into trucks for transport to the smelter customers.

### 17.3.6 Tin Recovery Circuit

The tin recovery circuit, if implemented, would treat the pyrite flotation tailings.

After an initial pre-concentration stage on 8 sets of 4-start spirals, the material is cyclone to split at 75µm and then the +75µm size fraction fed over 25 sand shaking tables and the -75µm material fed over 51 fine shaking tables.

Given the fine particle size AMC considers this shaking table circuit, although labour intensive, to be appropriate especially in the context of low Chinese labour costs and a long tradition in China of this sort of circuit. The number of tables also appears to be adequate for 1,600 tpd. AMC also notes that the low volume of tin concentrate coupled with the low labour costs indicates a possible application of the batch process of kieving (high frequency vibration in tubs as practiced many years ago at the Renison Tin concentrator in Australia), for upgrading of the concentrates.

The final step is a batch flotation stage to remove the residual sulphides also concentrated by the gravity separation processes. This would take place in a small unit in the main flotation building

## 17.4 Process Control and Automation

The planned level of process control and automation is basic but adequate, recognizing that the process separation is complex and that operating labour to monitor process variables is low cost and plentiful. It will consist of the following key components:

- A central control room in the grinding-flotation building from which TV imaging of key points in the production flow can be monitored
- Centralized monitoring of equipment run status
- On/off interlocking of the main crushing and grinding system flows
- Measurement and control of key parameters, including:
  - Ball mill feed tonnage
  - Critical bin and tank levels
  - Critical densities e.g. screw classifier, thickener underflow densities
  - Flotation cell pulp levels and reagent dosage

- Automatic sampling of key metallurgical accounting streams e.g. flotation feed, concentrates and tailings

## **17.5 Ancillary Facilities**

### **17.5.1 Laboratory**

The laboratory will be equipped with the usual sample preparation and wet chemistry and basic photometric analytical equipment as well as crushing, grinding, flotation and gravity separation metallurgical testing equipment.

It will conduct routine analysis of ores and concentrates as well as water quality and other environmental testing. It will also provide a technical service to the processing plant in monitoring plant conditions, solving production problems and investigating new technology and new processes to assist with the improvement efforts.

### **17.5.2 Maintenance Workshop(s)**

Daily maintenance requirements will be serviced through section specific workshops, each equipped with craneage, welding capability and basic machine-shop facilities.

More extensive maintenance and major overhaul needs will be met through use of appropriate contractors

## **17.6 Key Inputs**

### **17.6.1 Power**

Total installed power amounts to 5,043 kW (includes standby equipment) and the estimate for actual power drawn is 3,657 kW which corresponds to 28,963,000 kWh per annum.

Note that this includes tailings return water pumping.

From AMC's analysis of the equipment sizing, an additional 600 kW capacity in the grinding circuit ball mills is recommended to handle 1,600 tpd, which would increase installed power to 5,643 MW and actual power drawn (assuming the same ratio of drawn to installed) to 4,092 kW. Annual consumption would be 32,409,000 kWh.

### **17.6.2 Water**

With the use of dry stacking of tailings there is minimal lock-up of water in tailings and a close to 90% recycle of water; however there is a requirement for fresh water for e.g. pump seals, cooling and reagent mixing and it is this requirement that sets the overall fresh water demand.

Detailed circuit water balances have been derived for the 1,000 tpd case and from this a net fresh water demand of 1,200 m<sup>3</sup>/d has been estimated. With the conservative assumption that the fresh water demand is proportional to the throughput then the demand at 1,600 tpd would be approximately 1,900 m<sup>3</sup>/d.

### 17.6.3 Reagents

Reagent storage and mixing is located adjacent to the grinding/flotation plant and comprises a storage area with hoisting equipment to lift bags and drums through into the mixing area.

The reagents in this area are:

- Depressant/modifiers:
  - Sodium sulphide,
  - Zinc sulphate,
  - Sodium sulphite
  - Copper sulphate
- Collectors:
  - Di-ethyl dithiocarbamate
  - Ammonium dibutyl dithiophosphate
  - Butyl xanthate
- Frother – no. 2 oil (added directly)

From the mixing area the reagents are pumped up to the dosing station located above the flotation section for dosing and gravity feeding to the various addition points.

Since the usage of lime is large (8 kg/t) the lime storage and milk of lime mixing area is separate, but also adjacent to the grinding/flotation plant. Milk of lime storage tanks have grit removal submersible pumps.

The sulphuric acid tank and dosing pumps are also located separately, for reasons of safety.

### 17.7 Summary

The recovery methods proposed for the GC deposit are generally appropriate for the ore characteristics as tested. The following specific comments apply:

- The proposed flowsheet is fit for purpose and should achieve the targeted recoveries and concentrate grades
- The comminution circuit, especially grinding, is undersized for the 1,600 tpd throughput level, especially as there is no testwork data and therefore a degree of conservatism is warranted. AMC recommends an additional 600 kW grinding power.
- Drawing on the design data provided and on the site visit to the operating Ying mine, AMC concludes that, apart from the comments above pertaining to comminution, appropriate equipment has been selected for an operation in China and that the plant layout is practical and functional.
- AMC recommends consideration be given to a small increase in lead cleaner and filtration capacity to allow for optimization of silver recovery to payable lead concentrates.
- Ancillary facilities are adequate.
- The option of the tin recovery circuit merits inclusion.

## 18 PROJECT INFRASTRUCTURE

### 18.1 Tailings Management Facility (TMF)

#### 18.1.1 Overview

The tailings deposition method proposed is dry stacking and filling (from bottom to top and stacking by bench to form the embankment) with concurrent rolling and compaction.

Although AMC believes that the basic concept proposed is reasonable and in any case dry stacking usually has less onerous requirements than slurry tailings storage, nevertheless the work carried out to date towards the TMF design does not meet feasibility study standards. AMC considers that the following areas of deficiency need addressing:

- Tailings properties determination is critical for dry stacking as the tailings are effectively their own containment and so requires additional testwork including:
  - Proctor compaction tests to derive target moisture levels for the required compacted density
  - Shear tests to assess the internal strength of the tailings as an input to stability analysis
  - More comprehensive size analysis, to include potential clay component size range
  - Geochemical characterization e.g. metal leaching tests
  - Filtration tests to assist in the pressure filter sizing to meet target moisture levels
- Site investigations are required including:
  - Geotechnical evaluation of underlying bedrock etc
  - An assessment of the implications of the Gaocheng River class II water resource classification for the TMF location and design
- Although the TMF design meets storage capacity requirements, the following work is still required:
  - A site-specific risk assessment as opposed to the generic grade III design criteria within the Chinese volume-height categories.
  - A re-assessment of factor of safety calculations using standard industry practice finite element numerical modeling
- A more detailed water balance on a month-by-month basis is required since the project is situated in the monsoon belt with 70% of annual precipitation falling in the summer months.

#### 18.1.2 Tailings Properties

Physical tailings properties have been summarized by the GMADI but are limited to the following:

- Dry solids: S.G. 2.7 t/m<sup>3</sup>
- Dry density (estimated): 1.5 t/m<sup>3</sup>

- Tailings sizing: 80% - 75µm, average diameter 50 µm.

Although the dry density estimate of 1.5 t/m<sup>3</sup> is considered reasonable by AMC, there is no testwork basis for this e.g. Proctor compaction tests to derive moisture-density relationships. The compaction and ultimate density is normally quite sensitive to the moisture content and the optimum moisture can be fairly tightly constrained in the +/- 1-2% range. Nor are there any shear tests to investigate the internal strength of the tailings, important for the stability analysis. The tailings size analysis should also be extended to finer sizes to better measure clay content.

Geochemical properties of the tailings consist of a multi-element analysis, the only element of concern apart from Pb and Zn of course, being As. No leaching tests have been carried out to determine the potential for metal leaching, and leachate As limits in particular are difficult to meet.

### 18.1.3 Site Selection

Two possible sites for the TMF were considered, one immediately to the south of the mine and concentrator in the Daken valley and the second 5 km to the south-east in the Heliken valley.

These sites were evaluated with respect to the following criteria:

- Not being upstream of any significant residential, industrial, water storage areas or areas of historic or natural beauty conservation area
- Avoiding areas of complex geological structures and geological hazards
- Avoiding sterilization of mineral resources
- Having adequate storage capacity and minimum catchment area.

The Daken valley site was selected on the basis of:

- Proximity to the concentrator
- No residential or industrial developments although there is some small scale farming within the proposed site
- Small catchment area and adequate storage capacity.

The main disadvantage of this site is being upstream of the Gaocheng River with its Class II environmental classification. This Class II water resource category is highly restrictive of industrial development.

The Heliken valley site has superior storage capacity but a much larger catchment area. Its main disadvantage was the distance from the concentrator.

Data on site conditions consists of:

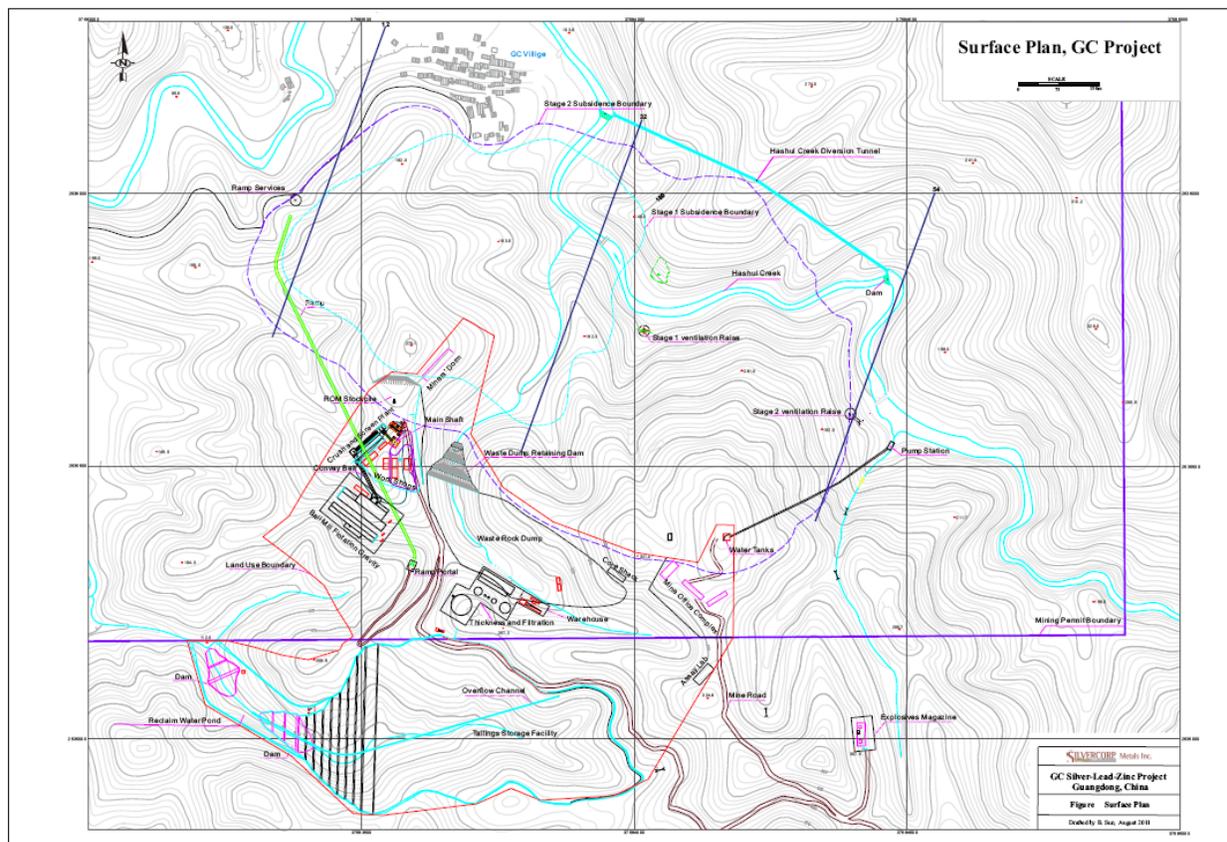
- Rainfall data (annual average 1493 mm with 70% of that occurring in the April to August period)
- Surficial geology: quaternary residual overlying shales and schists, no known structures

- Seismic intensity rating according to the Earthquake Intensity Zoning Map of China (2002) of VI (the intensity scale is similar to the Modified Mercalli, ie. in this case “slightly damaging”)

No site specific geotechnical field investigations have been carried out, e.g. geotechnical drilling to bedrock beneath the main containment structures.

The TMF location and other surface infrastructure is shown in Figure 18.1.

**Figure 18.1 Plan of Surface Facilities, GC Project**



### 18.1.4 TMF Design

Storage capacity calculations for the Daken valley site under the bottom to top dry stacking by bench method proposed provide a total storage volume of 3.4 Mm<sup>3</sup>, which at the assumed dry density of 1.5 t/m<sup>3</sup> is equivalent to 5.1 Mt of tailings. This is more than adequate for the tonnage of tailings in the production schedule proposed in this report.

The design criteria under the Chinese system are based solely on the height and volume, which places it within a Grade III facility (i.e. mid-range in the I-V system). However no site-specific risk assessment has been carried out which would be the normal process under western standards.

The TMF will consist of an initial earth retaining dam, behind which the tailings will be delivered by a system of conveyors and then spread by bulldozer on a bench by bench basis with

concurrent rolling and compaction to the desired dry density standards, to construct the tiered tailings embankment gradually rising up the valley.

Seepage control is effected by geomembrane and geotextile impervious layers together with an intercepting drain and collector system discharging into a downstream water storage dam for pumping to the concentrator.

A safety and reliability analysis for the TMF has been carried out in accord with the Safety Technical Regulations for Tailings Ponds (AQ2006-2005) and under the Grade III requirements. These stipulate minimum Factors of Safety, as determined by the Swedish Circular Arc Method for assessing the potential for slip rotation failure, in the 1.05-1.20 range. Although the calculated factors of safety are generally around 1.3 (which AMC considers should be the minimum) the method used is now considered outdated and industry practice would be to conduct finite element numerical modeling, even if just in two-dimensions.

AMC recommends that the TMF safety analysis be re-assessed in accord with industry practice.

Flood calculations have been performed appropriate to the Grade III classification of the TMF, under a dry stacking scenario, which requires the flood control measures to meet a 1 in 100 year recurrence interval for design purposes with a 1 in 500 year probable maximum flood criterion too.

#### **18.1.5 Tailings Delivery**

The concentrator tailings are thickened in a conventional rake thickener and then filtered in plate and frame pressure filters of Chinese manufacture at a filtration plant to be situated immediately adjacent to the TMF.

The two XA90/920 filters selected have been sized for 1,000 tpd ore feed and AMC considers that, based on the manufacture data and subject to further work on tailings properties and moisture level requirements, these two units are of inadequate size for the tonnage to be filtered. With the production schedule based on 1,600 tpd then, as a first approximation, AMC recommends that the filtration capacity be doubled.

The filtered tailings will be conveyed to the TMF with three conveyor belts for spreading and stacking as previously described.

#### **18.1.6 Water Balance Considerations**

Ultimately two main factors determine the overall water balance: i) the lock-up of water in the tailings and ii) the requirement for fresh water quality for e.g. pump gland seals, cooling, reagent mixing.

In the case of GC, a dry stacking tailings strategy minimizes the volume of water locked up in tailings and from the tailings properties previously mentioned. Making some reasonable assumptions about void ratios etc, AMC estimates that around 15-20% moisture will pertain to the filtered tailings and that at 1,600 tpd this will amount to 250-300 m<sup>3</sup>/d of water lock-up.

Fresh water requirements have been estimated at around 1,100 m<sup>3</sup>/d so unless there is some compromise in water quality requirement that allows the use of recycled water there will be a small positive water balance. The flood calculations have assumed that flood water recovered

from the water storage dam will displace fresh water until the floods have subsided. In any case AMC recommends that a more detailed evaluation of the water balance be carried out and on a monthly basis as rain is seasonal, to ensure that water needs and discharge requirements if necessary, can be properly managed.

## 18.2 Waste Rock Dump

The waste rock dump is located a short distance to the east of the mine portal. Silvercorp estimates that the total volume of phase-1 and phase-2 waste rock will be up to 450,000m<sup>3</sup>. The period of capital construction is two years. A total of 245,100m<sup>3</sup> waste is produced during this period (including waste produced by site leveling). Silvercorp intends to utilize 196,800m<sup>3</sup> for engineering backfilling and 50,000m<sup>3</sup> for railway, highway, sites, retaining wall, and raceway and buildings construction. Basically, there will be no waste surplus during the period of capital construction.

The total LOM waste produced will be 2.3 Mt with 0.9 Mt produced from Phase 1.

The +215 mRL waste dump tip head capacity is 275,000m<sup>3</sup> (~558 kt). This accommodates the initial two years of waste produced. Silvercorp has not provided AMC details for the final waste dump capacity, but AMC's site visit and review of the surface plans indicate there appears to be room for a downstream extension of the waste dump location and/or ability to increase the waste dump height to approximately +300 mRL to accommodate all waste produced.

Waste may opportunistically be disposed of into the shrinkage stope voids (with approximately 1.2 Mm<sup>3</sup> or 2.3 Mt void capacity) but this is not in the current mine plan, and the method for potential tipping of waste into the stopes has not been clarified by GC mine. The potential for waste disposal into shrinkage stope voids represents 100% of the LOM waste produced. Sources of the solid wastes in this project are mainly the waste rocks produced during the mining and the mine tailing produced during the ore processing.

Based on the GC environmental assessment report, waste rock at the GC mine has no significant acid-generating potential.

Silvercorp intends to sell the waste rock as commercial stone in regions where such stone is in short supply. Waste rock derived from mine development can also be used to assist in the construction of the tailings dam.

## 18.3 Power supply

There is a 110 kv substation near Gaochun, about 6 km from the mining area. This is fed from the Guangdong Province electrical grid system. Silvercorp has advised that this substation can be used as the main source of power for the mine. Current plans are for two overhead power lines for the 6 km route. Two 1,500kv diesel generators are designated for emergency backup to the man-hoist, underground ventilation system and essential services in the plant.

A new 10kv substation will be built in the mining area to provide power service for the entire mining area. The power supply and distribution in the process plant, mining area, administrative and living areas will be configured based on needs.

For the first phase electrical equipment, the total installed capacity is approximately 11 MW, working capacity is 8.9 MW, calculated load is 5.8 MW, apparent power is 6,260 kva, and annual electricity consumption is 35,040,000 kWh.

According to Chinese standards the electrical loads are sub-divided into three classes. Underground dewatering and the man-cage belong to first class. For the second phase, at peak dewatering, the working capacity of first class load is estimated at 1.7 MW. The total installed capacity above the second class load is 12.3 MW, working capacity is 10.1 MW, calculated load is 6.7 MW, apparent power is 7,612kVA, and annual electricity consumption is 39,258,000 kWh.

Underground sub-stations will be located on each level adjacent to the V2 vein. Level development utilizing jumbo development will incorporate additional sub-stations along the level to manage any voltage drop from the sub-station. Low voltage supply from the underground sub-stations will be 415V (jumbo), 380V (pumps and fans) and 220-250V (lighting and rail operation).

Total installed power amounts to 5,043 kW (includes standby equipment) and the estimate for actual power drawn is 3,657 kW which corresponds to 28,963,000 kWh per annum. This includes tailings return water pumping.

From AMC's analysis of the equipment sizing, an additional 600 kW capacity in the grinding circuit ball mills is recommended to handle 1,600 tpd, which would increase installed power to 5,643 MW and actual power drawn (assuming the same ratio of drawn to installed) to 4,092 kW. Annual consumption would be 32,409,000 kWh.

The size of the emergency backup power appears to be small compared with the overall power demand of 12 MW. It is recommended that the emergency backup power be provided for essential critical services, such as man cage, mine ventilation, mine dewatering pumps and thickeners.

#### **18.4 Roads**

Access to the GC project from Guangzhou is via 178 km of four lane express highway to Yunfu, then 48 km of paved road to the project site. A railway connection from Guangzhou to Yunfu is also available.

There are 15 roads assigned to this project, some are site and others general access roads.

#### **18.5 Transportation**

Special security personnel will be dispatched to the mine for transportation of ore concentrates to Yunfu Railway Station under escort. Silvercorp states that trucks will be used to deliver concentrate from the mine to the railway station.

The final products from the process plant are metallic ore concentrates. Daily output of copper, lead, zinc, sulfur and tin ore concentrates is estimated as 2.7 t, 16.8 t, 47.0 t, 148.6 t and 1.3 t respectively. Annual output of copper, lead, zinc, sulfur and tin ore concentrate is estimated as 891 t, 5,544 t, 15,510 t, 49,038 t and 429 t respectively. The concentrate will be either packaged or shipped as bulk. Special container vehicles and drivers will be provided.

A loader will be used to load the concentrate from concentrate sheds near filters at the mill site to the concentrate shipping trucks.

## 18.6 Water supply

Streams are well developed in the area, the Hashui Creek flows in the GC project area. There is a reservoir upstream of the GC project area.

The source of water for the mine will be local creeks and gullies that flow into the Hashui Creek. The flows will typically vary from 11,000 m<sup>3</sup>/day (dry season) to 69,000m<sup>3</sup>/day (wet season) with the wet season being from April to September inclusive. The annual average rainfall varies from 1,400 to 1,734 mm. The water quality and quantity from the local creeks will be sufficient to meet the project requirements, which are estimated be 2,093 m<sup>3</sup>/day.

As water volume of the Hashui Creek is abundant, maximum daily water intake volume of this project shall be up to 2,092.5256 m<sup>3</sup>/d, accounting for 19.5% of the low water volume approximately. Potable water has to be filtered and sterilized with sodium hypochlorite before use.

Water will be drawn from the Bai Mai reservoir and pumped to an elevated hilltop for water treatment. The treated water is then gravity fed to the mine site and treatment plant

The key specifications of the water supply system are:

- Bai Mai reservoir water tank with 100 m<sup>3</sup> of settling capacity and 200 m<sup>3</sup> clean water capacity.
- Hill top water tank with 300 m<sup>3</sup> storage and water filtration capacity of 85 t/hr (via two filtration units).

Water consumption at the underground mine will primarily be for drilling and suppressing dust.

Potable water will be provided underground adjacent to the Main Shaft with water quality conforming to sanitary standards.

With the use of dry stacking of tailings there is minimal lock-up of water in tailings and a close to 90% recycle of water. However there is a requirement for fresh water for e.g. pump seals, cooling and reagent mixing and it is this requirement that sets the overall fresh water demand.

Detailed circuit water balances have been derived for the 1,000 tpd case and from this a net fresh water demand of 1,200 m<sup>3</sup>/d has been estimated. With the conservative assumption that the fresh water demand is proportional to the throughput then the demand at 1,600 tpd would be approximately 1,900 m<sup>3</sup>/d.

## 18.7 Sewage treatment

Sewage treatment station is not included in the design scope. Storm water from process plant and mining plant will be discharged with reinforced concrete pipe to the river at the periphery of the mining area.

## 18.8 Mine dewatering

Maximum water yield from the mine is estimated at 6,200 m<sup>3</sup>/day. Average yield is estimated at 3,100 m<sup>3</sup>/day. Underground sump capacity based on Chinese safety regulation is 6-8h at the average water yield.

## 18.9 Site communications

A level-1 dispatching system will be used to ensure production dispatching of the mine. A 200-gate digital dispatching exchange of programmed control will be deployed at the dispatching room of the office building under management production dispatching personnel. To facilitate external communication, 10 pairs of trunk lines will be provided.

Underground communication line will be in the form of communication cable laid out along the sidewall of the drift. Two communication cables will be fed to the underground distributing equipment from different shafts. If any communication cable fails, the other will have adequate capacity to assume communication of all underground communication terminal.

## 18.10 Camp

Silvercorp plans to operate the mine using contractors for development, production and the operation and maintenance of Silvercorp's fixed plant with Silvercorp providing its own management, technical services and supervision staff to manage the GC mine operation.

The estimates for the underground workforce indicate a peak requirement of 322 people during phase 1 and a peak requirement of 377 people during phase 2. The estimates exclude General and Administration (G&A) personnel, geological drilling, external consultants and process plant operation. The estimate depicts people on-site at any point in time and does not account for the off-site labour panels, sick leave, absenteeism, annual leave, turn-over, etc. Silvercorp indicates overall workforce of 624 for phase 1 and 679 for phase 2.

Administrative, Living and Welfare Facilities are composed of administrative office building, hostel, canteen, bathroom, residential building and proposed buffet as well as dining and entertainment services.

## 18.11 Dams and Tunnels

Silvercorp has proposed a 1 km long diversion tunnel with two dams on the Hashui Creek to relocate the course of this river beyond the projected subsidence zone of influence (Figure 18.1).

### 18.11.1 Surface Maintenance Workshop

Silvercorp proposes a workshop building area of 756.5 m<sup>2</sup>, in which the following auxiliary services are provided:

- Tyre processing, maintenance and servicing
- Welding
- Electrical
- Hydraulic
- Tools, parts and material warehouse

The comprehensive repair workshop is mainly responsible for maintenance of large-scale production equipment, vehicle repair, processing and repair of partial components, and the processing of emergency parts. One LD 10t electric single-beam crane, one BC6063B shaping machine, one CD6240A saddle bed lathe, one Z3040×16/l radial drilling machine, and one bench drilling machine are equipped in the workshop, as well as alternating current arc welding, rectification arc welding, snag grinding machine, cut-off machine, electric drying oven, mobile air compressor, etc, and maintenance equipment such as tool rack, working platform, gas cutting device, etc. Also, a dynamic balancing machine, tyre picking machine, tyre mending machine, battery charger, vehicle repair pit.

### **18.11.2 Underground workshop**

Mechanical Maintenance Facilities are composed of mining equipment maintenance workshop, equipment and spare parts store, dump oil depot, reserved electric locomotives and tramcars maintenance workshop and stockpile yard.

The mining contractor will have its own mobile equipment workshop for repairs and servicing located adjacent to the Ramp portal. This predominantly caters for the Phase 1 requirements. There will also be underground drill service bays established in redundant stockpiles to minimize tramming delays for the slower moving drills.

Mobile equipment (trucks, loaders, etc), other equipment breakdowns and equipment major services will be conducted in the mining contractor's surface workshop with minor services conducted in redundant stockpiles for Phase 1.

Minor equipment (such as jacklegs, secondary fans, development pumps, etc) will be serviced in the mining contractor's surface workshop.

The electric locomotive and rail cars for Phase 2 will be serviced and repaired in a service rail sidings located adjacent to the Main Shaft.

Other fixed and mobile plant (primary pumps, surface electric locomotive, rail cars, vehicles, etc) will be serviced in Silvercorp's surface workshop located adjacent to the Main Shaft. This will be fully equipped with overhead crane, welding, electrical, hydraulic, lathe services, etc.

### **18.12 Explosives magazines**

It is proposed that the explosive warehouse will be constructed in the valley to the southeast of the GC Mining Area.

Silvercorp estimates that approximately 182 t of bulk explosive and 182,000 detonators will be consumed annually. The surface explosives magazine will be permitted to hold 10 t of bulk explosive and 15,000 detonators representing approximately 15 days and 30 days of supply respectively. Security services will be used and detonators will be scanned on release from the magazine for security audit purposes.

Underground working party magazines will be located adjacent to each level's return air shaft and will be limited to one day of requirement for bulk explosives and three days of requirements for blasting ancillaries.

### **18.13 Fuel farm**

Diesel fuel will be required for the mobile mine equipment, some small trucks, and surface vehicles. The pumping station allows for refueling of both light vehicles and heavy-duty mining equipment.

Ordinarily a properly constructed containment for storage of fuel is designed in the vicinity of the diesel generators and fuel dispensing facilities. The storage facility must be located down-wind from the mine air intake fans and a reasonable distance from buildings, camp and mine portal (dependent upon local occupational health and safety regulations and fire fighting requirement). the lined containment area is constructed such that spills are confined and can readily be cleaned, and so that the need for extensive and costly remediation work can be avoided during site closure.

The UTM coordinates of the fuel farm are 2,535,168.1 m (easting) and 37,593,487.9 m (northing). No fuel will be stored underground. Phase 1 jumbo drills will be re-fuelled using a dedicated purpose vehicle with appropriate safety equipment. Phase 1 trucks and loaders will be re-fuelled at the surface fuel farm and dispensing facility.

### **18.14 Mine dry**

Facilities accommodating, lockers, change room, showers and washrooms for the miners are ordinarily placed near the portal. Provisions for personal protective equipment such as gloves, safety glasses, hard hats and cap lamps and batteries must be made by Silvercorp or its contractor.

### **18.15 Administration Building**

The mine office complex to the east of warehouse will be a modularized structure that will provide working space for management, supervision, geology, engineering, and other operations support staff.

### **18.16 Open Area Storage**

No open area storage area has been specifically allocated. However, there is area within the plant site that could be fenced off to provide extra storage for equipment and materials, if required.

### **18.17 Assay Laboratory**

An assay laboratory will be located in a separate modular building at the southeast side of the mill building. The laboratory will be a single-storey structure equipped to perform daily analyses of mine and process samples.

### **18.18 Security/Gatehouse**

A security/gatehouse will be located on the site access road at the plant site. The access road off a local village road will have a manual gate with signage indicating that they are now entering private property.

### **18.19 Communication System**

The site communications systems will be supplied as a design build package and the scope defined in further project phases.

Mine surface communications will be available by landline and mobile phone services.

Mobile service coverage will be available for the GC mine site and Silvercorp's GC project development offices in the Gaochun township.

Telephones will be the base means of communicating with the underground workings. Phones will be located adjacent to the Ramp level accesses for Phase 1 and adjacent to the Main Shaft level accesses (plant area) for Phase 2.

Silvercorp intends to install a leaky feeder communication system for safety, operations supervision and maintenance activities.

### **18.20 Compressed air**

Compressed air will be primarily used for drilling blast holes. Jackleg drilling will be used in the stopes and conventional development faces. There will be some minor use for shotcreting, blast hole cleaning and ANFO charging of blast holes.

Compressed air will be reticulated to all levels and to the emergency refuge stations.

Compressors (electrically powered two-Phase piston compressors) will be located adjacent to the Ramp portal (2 x 20 m<sup>3</sup>/min, 0.8 Mpa, 110 kW) and Main Shaft brace area (2 x 40 m<sup>3</sup>/min, 0.8 Mpa, 250 kW). Compressed air will be reticulated using steel and plastic piping for air distribution via the Ramp and Ramp Shaft for Phase 1 and the Main Shaft for Phase 2.

AMC's review indicates additional compressor capacity may be required to cater for an estimated demand of approximately 140m<sup>3</sup>/min. AMC does not consider the impact to be material.

## 19 MARKET STUDIES AND CONTRACTS

### 19.1 Concentrate Marketing

AMC understands that the concentrates will be marketed to existing smelter customers in Henan province and appropriate terms have been negotiated as detailed in Section 19.3 below.

As has been mentioned in Section 13, there are some residual concerns / improvement opportunities regarding concentrate quality. Both copper and zinc levels are higher than ideal in the lead concentrates.

With respect to copper, testwork has so far been unsuccessful in producing a saleable copper concentrate, but copper levels in the ore are low and this is not a material issue for concentrate quality.

In the case of zinc, this is an issue of economic optimization with current silver prices indicating that a lower lead grade (35-40% Pb) concentrate (with higher zinc and silver levels) should be targeted to maximize payable silver recovery to the lead concentrate. This should not pose a problem with concentrate marketing.

AMC had expressed some concerns about arsenic levels in all concentrates (0.5%As in the lead and zinc concentrates and >1%As in the pyrite concentrate), which would potentially pose concentrate marketing problems to western smelters. Although there was no mention of arsenic levels in the initial contracts with the Chinese smelters AMC has been able to verify from direct experience of recent Chinese smelter contracts that, notwithstanding the various grades within the national standards, arsenic levels up to 2-3% As are in fact acceptable in a precious metals bearing pyrite concentrate. In addition, AMC has been advised by Silvercorp that the renewed smelter contracts referred to in section 19.2 specify arsenic penalty levels as 3.0% As in the pyrite concentrate, so the GC concentrates are expected to be well within that.

With respect to the lead and zinc concentrates, AMC has also been advised that the renewed smelter contracts allow up to 1% As before penalties apply which allays AMC's previous concerns about their marketability.

### 19.2 Smelter Contracts

Initial sales contracts were in place for the lead and pyrite concentrates with Jinan Wanyang Smelting (Group) Co., Ltd and for the zinc concentrate with Henan Yuguang Zinc Industry Co., Ltd. AMC has been advised by Silvercorp that all three contracts have been renewed with a three year term to 31 December 2014, with identical terms to the initial ones apart from the arsenic penalties.

Although AMC would have preferred to have seen the contracts as part of a life-of-mine frame agreement, it also understands that they should be viewed in the context of the existing operations and concentrate sales to these smelters, and therefore does not view the three-year term of the contracts as a material issue.

All three contracts have freight and related expenses to the smelting company's account.

The key elements of the smelter terms (2011 contracts) are summarized in Table 19.1 below:

**Table 19.1 Key Elements of Smelter Contracts**

	Pb Concentrate				Zn Concentrate		Pyrite Concentrate	
	%Pb	Deduction RMB/T Pb	Ag(g/t)	% payable	%Zn	Deduction RMB/T Zn	%S	Payable RMB/T S
Minimum quality	35		500		45		35	
Payment Scales	>60	1800	>5000	92.0		Price<RMB 15000/T: 5200	>35	350 + 15 RMB per 1%
	55-60	1900	4500-5000	91.5		Price>RMB 15000/T: 5200 + 20% PP		
	50-55	2000	4000-4500	91.0	40-45	45 RMB per 1% <45		
	45-50	2100	3500-4000	90.5				
	40-45	2200	3000-3500	90.0				
	35-40	2700	2500-3000	89.5				
			2000-2500	89.0				
			1500-2000	88.0				
			1000-1500	86.0				
			500-1000	84.0				

With respect to lead and zinc terms, the above deductibles calculate out to 85-90% payable for the lead concentrate and approx 65% for zinc, at long-term prices. AMC considers these to be favourable terms relative to global smelter industry norms. Silver payables of approx 90% are similarly in accord with industry norms.

### 19.3 Commodity Prices

AMC normally relies on the client for metal price projections but would typically offer comment on the reasonableness of these projections. However, the lead and zinc prices proposed by Silvercorp seemed to AMC to be too optimistic in relation to its understanding of general market trends. AMC has therefore referenced an internationally respected monthly publication of consensus metal price forecasts and has used the prices shown in Table 22.1 for its economic analysis.

For the purposes of cut-off grade and NSR calculations AMC has used the following long-term prices: Ag \$18/oz, Pb \$1.00/lb, Zn \$1.00/lb.

## **20 ENVIRONMENTAL, PERMITTING AND SOCIAL / COMMUNITY IMPACT**

### **20.1 Introduction**

An Environmental Impact Assessment (EIA) report on the GC project was prepared by the Guangdong Environmental Technology Center (GETC) and received by the Yunfu Environment Protection Bureau (Yunfu EPB) for comment. The Yunfu EPB states the mining area does not cover any natural conservation zones, ecological forests, and strict land control zones. Based on the assessment of the EIA report and the recommendations of the provincial environmental technology center, that the remediation of the site is completed, no overflow of waste water occurs, and environmental protection is maintained, the Yunfu EPB gave consent to operate the GC project with the stipulation that the scope, site, processing technique, and environmental protection measures are followed as written in the report. An Environmental Permit was subsequently issued by the Department of Environmental Protection of Guangdong Province in June 2010.

The mining area does not cover any natural conservation zones, ecological forests, or strict land control zones. The current vegetation within the area is mainly secondary vegetation including farming vegetation. Larger wild mammals were not found in the region during the assessment. Small birds nesting and moving in the woodland were observed occasionally in the evaluation region. The surrounding villagers raise domestic animals, such as chickens, ducks, dogs, etc.

### **20.2 Laws and Regulations**

The GC Mine will operate under the following laws, regulations and guidelines:

1. Law of Environmental Protection of the People's Republic of China (1989.12);
2. Law of the People's Republic of China on Environmental Impact Assessment (2003.9);
3. Law of the People's Republic of China on Prevention and Control of Water Pollution (Amended in 2008.2);
4. Rules for Implementation of the Law of the People's Republic of China on the Prevention and Control of Water Pollution (2000.3);
5. Air Pollution Prevention and Control Law of the People's Republic of China (Amended in 2000.4);
6. Law of the People's Republic of China on Environmental Noise Pollution Prevention (1997.3);
7. Law of the People's Republic of China on the Prevention and Control of Environmental Pollution by Solid Wastes (Amended in 2004.12);
8. Cleaner Production Law of the People's Republic of China (2003.1);
9. Decision of the State Council on Several Issues Concerning Environmental Protection (1996.8);
10. Solid Waste Pollution Prevention and Control Act of Guangdong Province (2004);
11. Regulations on the Administration of Construction Project Environmental Protection of Guangdong Province (Tenth Standing Committee of the National People's Congress of Guangdong Province in 2004);

12. Notice to Strengthen Water Pollution Control of Guangdong Province (People's Government of Guangdong Province, office of Guangdong Government [1999.11.26]);
13. Environmental Protection Regulations of Guangdong Province (2005.1);
14. The Mineral Resources Law (1996),
15. The Rules of Implementation Procedures of the Mineral Resources Law Of The People's Republic Of China (2006).

#### Regulation Guidelines

1. Environmental Quality Standard for Surface Water (GB3838-2002);
2. Groundwater Environmental Quality Standards (GB/T14848-93);
3. Ambient Air Quality Standard (GB3095-1996, Amendment Sheet in 2000);
4. Environmental Quality Standard for Noise (GB3096-2008);
5. Emission Standard for Industrial Enterprises Noise at Boundary (GB12348-2008);
6. Noise limits for Construction Site (GB12523-90);
7. Standard for Pollution Control on the Storage and Disposal Site for General Industrial Solid Wastes (GB18599-2001);
8. Air Pollutant Emission Limit (DB44/27-2001);
9. Hygienic Standards for the Design of Industrial Enterprises (GBZ2-2002);
10. Prevention and Control on Tailings Environmental Pollution Prevention and Control (State Environmental Protection Administration in Oct., 1992);
11. Evaluating Indicator System for Lead and Zinc Industry Cleaner Production (Trial) (2007);
12. Air Pollutant Emission Limit of Guangdong Province (DB44/27-2001).

### **20.3 Waste and Tailings Disposal**

Sources of the waste for the project are mainly the waste rocks produced during the mining and the mine tailings produced during the processing.

Waste rock produced during mining will be mainly composed of silicon dioxide and calcium oxide. Some of the waste rock will be used as fill during the construction of the infrastructure and possibly as fill in the mined areas; the majority will be piled on the waste rock dump. The waste rock dump will be required to be covered by soil and vegetated after the dump is full. Retaining wall spats will be built downstream of the waste rock site for stabilization. An interception ditch will be constructed upstream to prevent the slope surface from washing out as well as to avoid water and soil loss. On closure, a soil cover will be placed and vegetation planted.

Mine tailings are discussed in section 18.1. After the completion of TMF, the facility will be earth covering and a vegetation program will be conducted. This is to ensure that all water flowing into the TMF does not directly contact the tailings and can be discharged to the downstream water system through the drainage ditch at the dam abutment.

## **20.4 Site Monitoring**

### **20.4.1 Monitoring Plan**

A monitoring plan is negotiated between the company and the local environmental protection department to meet the environmental management requirements of the project. A key component of the monitoring plan is water pollution monitoring; secondary is environmental air and noise monitoring. The monitoring work will be carried out by qualified persons and undertaken on a regular basis.

An environmental protection department will be set up for this project. The full time environment management personnel will be mainly responsible for the environment management and rehabilitation management work in the mining area, and part-time environmental protection personnel will be allocated in shifts for various workshops to coordinate the environmental protection work.

### **20.4.2 Water Management**

The Hashui Creek is shallow, and will be affected by the mining process which will have an impact on local villagers. A water retaining dam is built on the river and irrigation wastewater from the farmland is discharged into the river. During site investigation by the GETC, large size fish were not observed in the Hashui Creek; fish fry were found moving among the submerged plants. As part of mine site preparations, the Hashui Creek will be closed and diverted through a new water tunnel of approximately 510m in length.

Prior to the completion of the construction phase, drainage construction in the project water catchment area will be completed. Overflow water from the mill process waste water which is segregated by the thickener, and water generated from the tailings by the pressure filter, will be returned to the milling process to ensure that waste water (include tailings water) is not discharged. Water from mining operations will be reused for mining operations and the remaining water will be treated according to the Surface Water Quality Standards (GB3838-2002) to meet the requirement of Class III water quality. The treated water is then stored in nearby reservoirs to be used as irrigation water for nearby woodland and farmland. That water needing to be discharged will be directed to the Hashui Creek and will be treated to remove heavy metals such as mercury, cadmium, chromium etc. Sewage treated by the GC sewage treatment facility will be reused in mine forestation and irrigation prior to excess being discharged into the environment. Construction should be conducted during the dry season to reduce soil erosion.

### **20.4.3 Groundwater**

Groundwater guidelines are contained in the Groundwater Environmental Quality Standards (GB/T14848-93). The groundwater quality meets the Class III standard with the exceptions of zinc and fecal coliform. The zinc is related to the high background level at the site and the fecal coliform is related to the local villagers.

### **20.4.4 Waste Water**

There are three sources of waste water identified at the GC project; mining activities, mineral processing and domestic sewage. About 70% of mine water will be pumped from the underground sumps to the waste water treatment station, and remaining mine water will be pumped into surface water tanks for mill and mine production use. Treatment will primarily be

de-sedimentation and lime addition. Once the water reaches the required standard it will be used for forestry and agriculture irrigation or discharged into Hashui Creek. Process water is maintained in a closed circuit and is not discharged into the environment. After the treatment of the sewage water at the sewage treatment station, and testing indicates it has reached the required standard, it will be released into the environment to be used for watering lawns and gardens at the mine site.

## 20.5 Permitting Requirements

Silvercorp has completed the following permitting and contracting requirements to receive approval to extract ore from the GC Mine:

- Silvercorp obtained a Notice of Approval to start the process of the Application for Mining Permit from the Ministry of Land and Resources (MOLAR) in BeiYing on a designated mining area. Silvercorp received the Notice of Approval from MOLAR in 2008.
- The Resource Utilization Plan (RUP) Report on the GC project prepared by the Guangdong Institute of Metallurgical Industry was reviewed by a MOLAR design review organization, the China Non-Ferrous Metal Association, in 2008.
- The Health and Safety section of the RUP Report was reviewed by the Guangdong Provincial Safety Production Bureau in 2008. Both reviews indicated that the report satisfied the requirement for the mining permit application.
- An Environmental Assessment Report was completed in March 2009 and passed a review by an expert panel appointed by the Environmental Protection Bureau of Guangdong Province and by the local community.
- A Geological Hazards Assessment Report and Soil Conservation Plan Report prepared by a qualified geo-engineering firms, was reviewed and filed with Ministry of Land and Resources.
- A Geological Environment Protection and Rehabilitation and Reclamation Measure Report, prepared by a qualified geo-engineering firm, was reviewed and filed with Ministry of Land and Resources.
- A Land Reclamation Measure Report, prepared by a qualified engineering firm, was reviewed and filed with Ministry of Land and Resources.
- An Environmental Permit for the GC Silver-Lead-Zinc Project was issued by the Department of Environmental Protection of Guangdong Province in June 2010.
- A mining permit application for the GC Silver-Lead-Zinc Project was submitted to MOLAR in August, 2010.
- A mining permit for the GC Mine was issued by the Ministry of Land and Resources of China. The GC mining permit has a term of 30 years and covers the entire 5.5237 square kilometer area of the GC project. The permit was issued on the terms applied for, and allows for the operation of an underground mine to produce silver, lead and zinc ores.
- A qualified Chinese engineering firm has finalized the mine design of a 1,500 tonne per day mechanized underground mine, a flotation mill, and a dry stack tailing facility, which was reviewed and approved by the relevant government agents.
- Land usage and acquisition of land for the GC mine and milling sites has been completed.

- A qualified mining development contractor has been hired to build the mine.
- The same contractor, who built the Silvercorp's two mills (3,000 t/d) at the Ying Mining District, has hired to construct a 1,600 t/d capacity flotation mill that is capable of producing silver-lead, zinc, pyrite flotation concentrates and a tin gravity concentrate.
- An explosive permit was issued and an explosive magazine has been built followed the requirement of the Bureau of Public Security.

The following permits are required to be obtained by Silvercorp before production commences from the GC project:

- Completion of a review of the health and safety production measures in the mine design by the Guangdong Provincial Safety Production Bureau, after which it will be filed with the Guangdong Provincial Safety Production Bureau.
- Upon the completion of the mine construction, the work is subject to a safety measure inspection by the Guangdong Provincial Safety Production Bureau to satisfy that the construction of the mine, mill and tailing facility has followed the "Mine Design" in terms of safety measures. If satisfactory, the Guangdong Provincial Safety Production Bureau will issue a "Safety Production Permit".
- The Guangdong Environmental Bureau will also conduct an inspection of the tailing facility, flotation mill, and other environmental engineering works upon completion. If satisfactory, an environmental permit to operate will be issued.
- Land titles will be transferred to the subsidiary of Silvercorp

## 20.6 Social

Residents in the project area hold a positive attitude to the development of the project. Public participation methods of this project are the information disclosure, the inquiry form-sending, and the promotion and improvement of the reclamation consciousness.

Low-noise machinery and equipment, measures to minimize vibration, noise-proofing, noise reduction on the crusher, ball crusher, floater to ensure that the noise level of the mining area and the plant boundary meet the requirements of Class III function area limitation of emission standard for industrial enterprises noise at boundary (GB 12348-2008). The noise level inside the mine area and nearby inhabitant area should meet the requirements of Class II function area standard.

There are no residents in close proximity to the mining and processing areas. Noise is not expected to be of major concern to local residents.

## 20.7 Remediation and Reclamation

Remediation and reclamation plans have been discussed in the above text. Silvercorp has spent \$3.0M acquiring land for the project and has also posted \$200,000 to the Yunan County Government as bond for reclamation. Soil conservation and land reclamation costs are estimated as \$1.672M, while geological environment protection and rehabilitation costs are estimated as \$0.917M.

## 20.8 Site Closure Plan

Mine closure will comply with the Chinese National requirements. These comprise Article 21 (Closure Requirements) of the Mineral Resources Law (1996), and Articles 33 and 34 of the Rules of Implementation Procedures of the Mineral Resources Law of the People's Republic of China (2006).

The site closure planning process will include the following components:

- Identify all site closure stakeholders (e.g. government, employees, community etc.).
- Undertake stakeholder consultation to develop agreed site closure criteria and post operational land use.
- Maintain records of stakeholder consultation.
- Establish a site rehabilitation objective in line with the agreed post operational land use.
- Describe/define the site closure liabilities (i.e. determined against agreed closure criteria).
- Establish site closure management strategies and cost estimates (i.e. to address/reduce site closure liabilities).
- Establish a financial accrual process for site closure.
- Describe the post site closure monitoring activities/program (i.e. to demonstrate compliance with the rehabilitation objective/closure criteria).

Based on the Chinese National requirements, a site decommissioning plan will be produced at least one year before mine closure. Site rehabilitation and closure cost estimates will be made in the site closure plan.

## 21 CAPITAL AND OPERATING COSTS

### 21.1 Summary

Total initial capital expenditure is estimated to be \$67.4M including mining, mill, infrastructure, owner's costs and contingency, as summarized in Table 21.1

**Table 21.1 Summary of Capital Costs**

	US\$000	Notes
Mining	18,490	
Mill and Infrastructure	29,619	
Owners Costs	11,655	
Contingency	7,589	
Total Initial Capital Expenditure	67,352	
Working Capital	5,714	based on 3 months yr 1 operating costs
Sustaining	25,321	
Total LoM Capital Cost	92,673	Working Capital netted to zero

Total operating cost for the project is estimated at \$40.6/t milled. The estimate includes mining, process, G&A and surface service costs, as summarized in Table 21.2.

The unit costs are based on a nominal annual mill feed rate of 500,000 t/a or 1,515 t/d (equipment sized for 1,600 t/d instantaneous rate) and 330 d/a. The currency exchange rate used for the estimate is 1.00:6.35 (US\$: RMB).

**Table 21.2 Summary of Operating Costs**

	US\$/t milled
Mine	15.2
Mill	16.8
G&A	8.6
Environment (incl tailings)	
Total	40.6

### 21.2 Mining Capital Cost

The basis for the cost estimate (summarized in Table 21.3) is contractor mining with the contractor providing all trackless mobile fleet (Stage 1) and the owner providing all fixed plant and tracked fleet.

Costs are stated in Q1 2011 cost terms.

The capital cost estimate is prepared at a feasibility study level, which is considered to be at  $\pm 10\text{-}15\%$  level of accuracy on the inputs, based on the following:

- Equipment fleet and infrastructure requirement estimates
- Budget quotes obtained from equipment manufacturers
- Contractual quotes obtained from local contractors
- Project development and execution plans developed by the owner and refined by AMC

**Table 21.3 Mining Capital Costs**

Initial Capital Expenditure	US\$000	Notes
Capitalized operating costs	1,841	years -2 and -1
Project Establishment Capital	11,369	
Silvercorp Mine Capital	5,280	Ventilation, hoisting, underground infrastructure and miscellaneous equipment
Contingency	1,778	
Total	20,268	
<b>Sustaining Capital</b>		
Development	20,394	
Equipment Replacement	2,480	

### 21.3 Mining Operating Cost

The basis for the cost estimate is contractor mining and owner estimates for its management of the operation.

Costs are stated in Q2 2011 cost terms.

The operating cost estimate is prepared at a feasibility study level, which is considered to be at  $\pm 10\text{-}15\%$  level of accuracy on the inputs, based on the following:

- Silvercorp has a fixed price mine contract in place that is valid until the end of 2013 (Mining Engineering Construction Contract of Gaocheng Lead-Zinc Mine Project GF-2011-0225 dated 19 March 2011),
- Contractor operation has commenced in Q2 2011 and appears to be on-time and budget,
- Owner's in-house database of current operating costs for similar style of operation in China (Ying mine).

The underground mine operating costs exclude processing and G&A. The operating development is notionally the footwall drive portion that traces the stopes, the crosscuts to the stopes (draw points) and the stope portion of the vein sill drives (i.e. excluding the exploration vein development). The underground production includes direct stoping, rock handling, mine services, technical engineering and mine supervision.

### 21.4 Mill Capital Costs

The mill and infrastructure capital costs are summarized in Table 21.4.

The estimate is based on Q2 2011 costs and is considered to be of  $-5\% + 20\%$  accuracy and this is reflected in the sensitivity/scenario analysis in Section 22 in order to derive a robust overall estimate.

Major equipment quotations were supplied by Silvercorp and installed costs were based on a factoring approach developed by GMADI.

As discussed in Section 17, AMC has amended the estimate to allow for equipment undersizing of grinding mills and tailings filters.

In addition AMC has included an allowance for engineering, procurement construction management (EPCM) costs equal to 15% of direct installed costs

**Table 21.4 Mill & Infrastructure Capital Costs**

Initial Capital Expenditure	US\$000	Notes
Capitalized Operating Costs	2536	years -2 and -1
Mill	11,074	Installed costs based on Chinese equipment quotes supplied by Silvercorp for crushing, grinding, flotation, tails filtering major equipment items, adjusted by AMC to address undersizing
Tailings Dam	2,860	
Infrastructure	9,616	Site development, power distribution and ancillary buildings
EPCM	3,533	15% of Mill, tailings dam and infrastructure costs
Owners Costs	11,655	Land acquisition, permitting, administration and start-up/commissioning
Contingency (15%)	5,811	
Total	47,084	
Sustaining Capital	2,447	

## 21.5 Mill Operating Costs

The mill operating cost estimate is summarized in Table 21.5.

The estimate is based on Q2 2011 costs and is considered to be of +/-15% accuracy.

Labour costs were based on detailed manning schedules and wages/salary estimates supplied by Silvercorp.

Consumables costs were based on the metallurgical testwork results and unit costs for steel and reagents etc which, although lower than western costs, are in accord with AMC's experience of Chinese supply costs.

**Table 21.5 Mill Operating Costs**

	US\$/T milled	US\$000 (LoM)	Notes
Labour	2.45	11,628	
Variable, comprising:	14.35	68,138	
Wear steel	1.56		
Reagents	4.72		
Power / water	4.35		Based on power cost of US\$0.08/kWh
Resource Tax	1.54		
Other	2.18		
Total	16.79		

## 22 ECONOMIC ANALYSIS

### 22.1 Key Assumptions

The economic analysis in this section is based on the following key assumptions:

- The mine production schedule developed in Section 16 by AMC which quickly ramps up to around 500,000 tpa.
- The subsequent mill production schedule to match mine output at a nominal 500,000tpa throughput rate with generous ROM stockpile allowances in the early years to handle any mine development delays and still protect the metal outputs
- Mine development and mill construction take place during 2011 - 2012 with commercial production in 2013. The absolute dates will require adjustment but the relative timing remains valid.
- Recoveries as detailed in Section 13, and set out below:
  - Ag (to lead concentrate only, not payable in zinc concentrate): 62.8%
  - Pb: 84.7%
  - Zn: 88.2%
  - S: 61.3%
- Metal prices as set out in Table 22.1.

**Table 22.1 Metal Prices**

	2012	2013	2014	2015	After 2015
<b>Silver (US\$/oz)</b>	40.00	30.00	25.00	18.00	18.00
<b>Lead (US\$/lb)</b>	1.11	1.16	1.14	1.15	1.00
<b>Zinc (US\$/lb)</b>	1.05	1.12	1.11	1.15	1.00

- Foreign exchange rate was set at the November 2011 value of USD:RMB of 6.35 (Silvercorp had previously advised 6.50 but AMC has updated this with the most recent value in line with the gradually appreciating trend)
- Capital and operating costs as estimated in Section 21.

### 22.2 Key Metrics

Table 22.2 shows the critical input and output metrics for the GC project including the annual production schedule and the estimated cash flow and key pre-tax economic parameters.

**Table 22.2 Project Production Schedule and Cash Flow (Pre-Tax)**

Base Case		Total	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
<b>Mine Output</b>														
T		4,749,665	0	91,799	484,547	472,387	395,153	487,570	524,317	525,385	525,005	526,510	520,672	196,319
<b>Mill Feed</b>														
T		4,749,665			450,000	475,000	500,000	500,000	500,000	500,000	500,000	500,000	500,000	324,665
	Ag/g/t	121			163	152	124	110	121	120	112	110	99	94
	%Pb	1.31			1.00	1.15	1.31	1.42	1.50	1.35	1.38	1.31	1.36	1.22
	%Zn	2.95			2.96	2.86	2.62	3.02	3.08	2.76	3.25	3.03	2.97	2.98
(notional)	%S	9.13			9.13	9.13	9.13	9.13	9.13	9.13	9.13	9.13	9.13	9.13
<b>Concentrate Production</b>														
Pb Conc	T	113,404			8245	9979	11942	12973	13692	12315	12581	11962	12460	7255
	Ag g/t				5586	4555	3257	2673	2782	3054	2802	2886	2492	2644
	Ag % payable				92.0	91.5	90.0	89.5	89.5	90.0	89.5	89.5	89.0	89.5
	% Pb				46.35	46.35	46.35	46.35	46.35	46.35	46.35	46.35	46.35	46.35
	% Zn				12.5	10.5	8.5	9.0	8.7	8.7	10.0	9.8	9.2	10.3
Zn Conc	T	252,760			24027	24503	23657	27206	27756	24834	29285	27292	26754	17446
	% Pb				0.70	0.83	1.03	0.97	1.01	1.01	0.88	0.90	0.95	0.85
	% Zn				48.95	48.95	48.95	48.95	48.95	48.95	48.95	48.95	48.95	48.95
Py Conc	T	625,276			59241	62532	65823	65823	65823	65823	65823	65823	65823	42741
	%S				42.52	42.52	42.52	42.52	42.52	42.52	42.52	42.52	42.52	42.52
<b>Recovered Metals</b>														
	koz Ag	11,600			1481	1462	1251	1115	1225	1209	1133	1110	998	617
	T Pb	52,563			3822	4625	5535	6013	6346	5708	5831	5544	5775	3363
	T Zn	123,726			11761	11994	11580	13317	13587	12156	14335	13360	13096	8540
	T S	265,867			25189	26589	27988	27988	27988	27988	27988	27988	27988	18173

Continued on next page

**SILVERCORP METALS INC**  
**NI 43-101 Technical Report Update on the GC Ag-Zn-Pb Project in**  
**Guangdong Province, People's Republic of China**

Base Case		Total	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
<b>Revenue (US\$000)</b>	Ag	186,010			35540	29072	17618	15616	17157	17035	15879	15550	13904	8641
	Pb	89,655			7399	8778	10611	9798	10341	9301	9502	9034	9411	5479
	Zn	157,650			16878	16982	17283	16047	16371	14648	17273	16098	15780	0290
	S	12,449			1179	1245	1310	1310	1310	1310	1310	1310	1310	851
	Total	445,764			60996	56077	46822	42772	45179	42294	43965	41993	40405	25261
<b>Operating Costs US\$000</b>														
Mine	72,150	All Operating Costs Capitalized		9022	5969	6138	7881	8586	8658	9148	7622	7519	1608	
Mill	79,530			6745	7245	7771	7981	8198	8421	8650	8887	9133	6499	
Other	40,682			4833	4321	4152	3811	3964	4073	4368	4018	3883	3260	
Taxes	16,493			2257	2075	1732	1583	1672	1565	1627	1554	1495	935	
Total Costs (US\$M000)	208,855			22857	19609	19793	21255	22419	22717	23792	22080	22030	12302	
<b>Operating Cash Flow (US\$000)</b>	236,909	0	0	38139	36468	27029	21516	22760	19577	20172	19913	18376	12959	
<b>Capital Costs US\$000</b>														
<b>Total Capital</b>	92,673	5,639	61,713	11,839	5,071	2,986	1,991	2,622	1,688	1,657	2,230	847	-5,610	
<b>Project Cash Flow (US\$000)</b>	144,236	-5639	-61713	26300	31397	24043	19525	20138	17889	18516	17682	17529	18569	
<b>Cumulative Cash Flow</b>		-5639	-67352	-41052	-9655	14387	33913	54051	71940	90455	108138	125667	144236	
<b>NPV (8%)</b>	<b>\$73,712.36</b>													
<b>IRR</b>	<b>32.74%</b>													
<b>Project Payback yrs</b>	<b>2.40</b>													

Note that the Base Case pre-tax economic model shows an NPV of \$73.7M using a discount factor of 8%, considered by AMC to be a typical discount factor for a base-metals project. The IRR is 33% and the payback period is 2.4 years. The sensitivity of these returns and their robustness to credible scenarios are further discussed in 22.4

### 22.3 Taxes and Royalties

Metal prices originally supplied by Silvercorp were net of VAT (17% gross partially offset with a nominal 2% credit for goods purchased) and this has been retained by AMC in the economic analysis.

A City Tax equivalent to 10% of the VAT has been applied. A Resource Compensation tax of 2% of revenue (after VAT) has been applied; equivalent to a royalty.

No additional income or corporate taxes have been applied and therefore no depreciation schedule has been developed.

### 22.4 Sensitivity Analysis

The key input variances analyzed for the project sensitivity analysis were:

- Throughput (minus 20% relative)
- Mining dilution (plus 50% relative)
- Mill recovery (minus 10% and plus 5% relative)
- Metal prices (minus 20% and plus 10% relative to the prices shown in Table 22.1)
- Capital cost (plus 20%)
- Foreign exchange rate, USD:RMB (RMB appreciating to 6.00)

Note that AMC believes these are feasible sensitivities (some being only one-sided, others, such as mill recovery being +5% / -10% in line with plant realities) rather than the somewhat arbitrary +/- x % often applied.

Moreover the sensitivities were synthesized into credible scenarios using various combinations of the input variances rather than the one-at-a-time "spider-charts" that typically appear in sensitivity analyses. Scenario probabilities based on industry experience were assigned and a weighted average of key project financial return parameters was calculated. These are summarized in Table 22.3.

Note that shown separately from this scenario analysis in Table 22.3 (case 7) is the sensitivity to the application of the Silvercorp metal prices. As mentioned in Section 19, AMC would normally defer to the client for metal prices hence it is showing the sensitivity to applying the original Silvercorp prices.

**Table 22.3 Scenario Sensitivity Analysis**

Case	Scenario	IRR	NPV (8%)	Payback yrs	Scenario Details						Assigned Probability
					Throughput	Dilution	Mill Recovery	Metal Prices	Capex	FX	
	<b>Base Case</b>	<b>33%</b>	<b>\$73,712.360</b>	<b>2.4</b>						6.35	30%
1	Can't achieve mine output	26%	\$56,194.787	2.9	minus 10%					6.35	15%
2	Capex +20% to achieve throughput	24%	\$57,553.877	3.1					plus 20%	6.35	20%
3	Pessimistic case	11%	\$11,371.953	5.4		plus 50%	minus 10%		plus 20%	6.00	10%
4	Worst case	-1%	-\$28,567.647	11.0	minus 10%	plus 50%	minus 10%	minus 10%	plus 20%	6.00	5%
5	Optimistic case	46%	\$119,600.329	1.8			plus 5%	plus 10%		6.35	10%
6	Metal prices collapse	22%	\$40,595.590	3.3				minus 20%		6.35	10%
	<b>Weighted Average</b>	<b>27%</b>	<b>\$57,782.106</b>	<b>3.4</b>							
7	Silvercorp Metal Prices	43%	\$116,676.753	2.0							

AMC considers in general that the probability-weighted average of plausible scenarios is preferable to a single-point base case estimate which is often optimistic. In the case of GC AMC considers that because of some of the study deficiencies highlighted in this report it is necessary to adopt a conservative bias towards a slightly pessimistic view. The weighted average economic returns for the GC project are estimated to be:

- NPV (at 8% discount)                      \$52.950M
- IRR(%)    28%
- Payback (years)                                      3.3

These probability-weighted average metrics are positive and demonstrate that the project is robust in the face of the possible scenarios that typically impact on a mining operation.

## 23 ADJACENT PROPERTIES

The GC project is located within the Daganshan mineralization field featuring tungsten (W), tin (Sn), gold (Au), silver (Ag), lead (Pb), zinc (Zn) mineralization, Figure 23.1. The field is characterized by five “nested” zonation. From the centre outward, the mineralization zones are W, Sn, Sn-Pb-Zn, Ag-Pb-Zn, and Au. The following are a list of deposits that have been discovered and mined within the field.

*Dajinshan Tungsten Deposit.* The deposit is located in the centre of the Daganshan field.

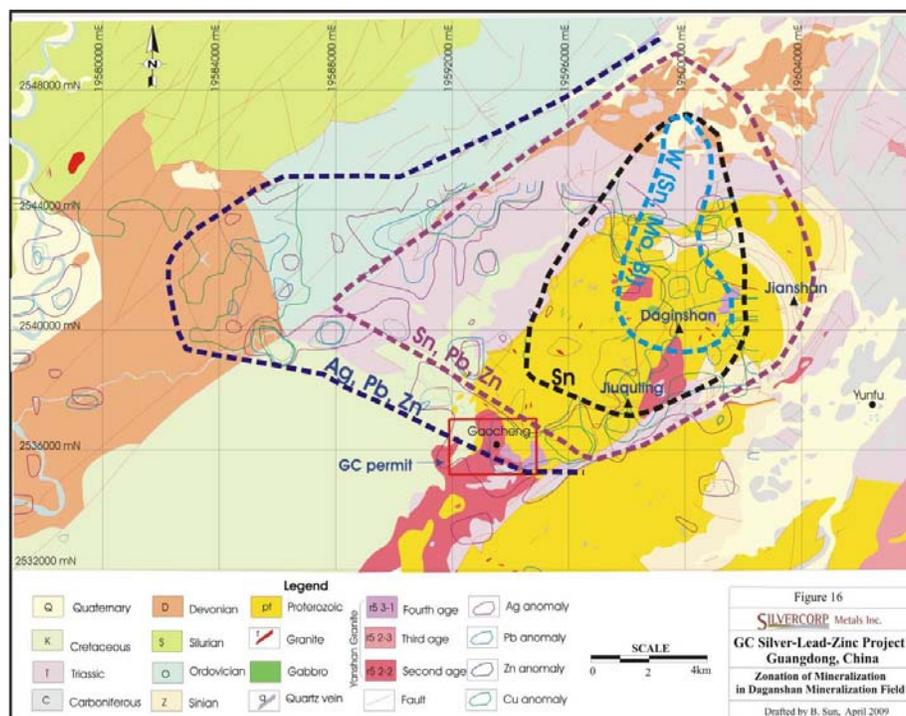
*Jiuquling Tin Deposit.* The deposit is a quartz vein type and surrounds the tungsten mineralization zone. It is reported that the Jiuquling deposit has been developed and is production, however detailed information such as grade, deposit size, tonnage; metal recovery, etc. are not available at this time.

*Jianshan Tin-Lead-Zinc-Silver Deposit.* The deposit is located in the tin-lead-zinc mineralization zone. It is sedimentary type of deposit.

*Yunfu Pyrite Deposit.* The Yunfu pyrite mine is an open pit mine located 4.5 km northwest of the city of Yunfu. Mine production began in 1988.

Figure 23.1 illustrates the general geological understanding of adjacent properties to the GC project. AMC is not aware of any immediate adjacent properties which would directly affect the interpretation or evaluation of the mineralization and anomalies found on the GC project property.

**Figure 23.1 Zonation of Mineralization in the Daganshan Mineralization Field**



## **24 OTHER RELEVANT DATA AND INFORMATION**

There is no additional information or explanation required to make the Technical Report understandable and not misleading.

## 25 INTERPRETATION AND CONCLUSIONS

Polymetallic mineralization at the GC project comprises over 20 distinct veins, ranging in thickness from a few centimetres to several metres, with a general east-west orientation and dipping generally south at 60° – 80°. The most recent mineral resource estimates were prepared using a polygonal method by Silvercorp and reviewed, classified and signed off by Mr B O'Connor, P.Geol of AMC, who qualifies as a Qualified Person for this purpose. Using a 100g/t recovered silver equivalent (AgEq Recovered) cut-off grade, Measured and Indicated Resources are 7.63 million tonnes grading 122g/t Ag, 1.32% Pb and 3.08% Zn, and Inferred Mineral Resources are 7.96 million tonnes grading 123g/t Ag, 1.41% Pb and 2.66% Zn.

In AMC's opinion, the quality and reliability of the data underpinning the resource estimates meets accepted industry standards and there are no issues that are likely to materially impact on mineral resource estimates.

Mineral reserves have been estimated at a 135 g/t AgEq Recovered cut-off grade, based on a mine design prepared by the Guangdong Metallurgical & Architectural Design Institute (GMADI) in January 2011 and modified by AMC where required, by Mr P Mokos, MAusIMM (CP) of AMC who qualifies as a Qualified Person for this purpose. Proven and Probable Reserves are 4.75 million tonnes grading 121g/t Ag, 1.31% Pb and 2.95% Zn.

Mining will be by highly selective approaches, mainly shrinkage and resue stoping methods. Some hand sorting of ore from waste will also be conducted. The shrinkage method uses the blasted ore as the working platform for each stope lift. The ore is removed on completion of stope mining leaving an empty void. There is potential to opportunistically dispose of development waste into these voids, but current mine plans do not make allowance for this. The resue method uses blasted waste from the footwall (to achieve the minimum mining width) as the working platform for each stope lift. The waste remains in the stope at completion of stope mining.

The life-of-mine (LOM) production duration will be 12 years. The average production rate will be 496,000 tonnes per annum (tpa) of ore from 2013 to 2021 inclusive. The steady state mine production rate will be 518,000 tpa of ore from 2016 to 2021 inclusive.

The project is currently advancing towards production, based largely on a study commissioned by Silvercorp and undertaken by the GMADI in January 2011 titled "Mining and Dressing Project of Gaocheng Lead-Zinc Ore in Yun'an County, Guangdong Province". AMC has reviewed this report and concluded that, overall, it meets the requirements of a pre-feasibility study (PFS) by CIM standards. Those components judged by AMC not to be at a PFS level are not considered to impact materially on the conversion of mineral resources to mineral reserves or on the economic viability of the project. Mill and mine construction is underway, including new underground accesses and a 1,600 tonnes per day flotation mill designed to produce silver-lead, zinc and pyrite flotation concentrates and a tin gravity concentrate. Silvercorp received a mining permit in December 2010.

The mining component of the project will be developed in two stages. Stage 1 targets fast-tracking the project into production and is developed by mobile rubber-tired, diesel-powered equipment (development jumbo, loader, truck) with surface decline access down to -50 mRL. Stage 2 is developed using conventional tracked equipment (electric locomotive, rail cars, electric rocker shovels, pneumatic hand held drills) with shaft access from -50mRL to -300 mRL.

The mine design provided is considered by AMC to be below feasibility study standard (within +/-10-15% on the inputs) with respect to knowledge of the vein location and vein peripheral extents and missing minor miscellaneous development items such as travel way refuges, stripping, service holes and the like. AMC expects that the design will be progressively refined as the mine is developed, but does not anticipate a material change in the development requirements (e.g. <5%). The exception to this is the placement of the development relative to the veins. The issues related to placement (potential re-development, potentially longer than planned draw points, etc) are anticipated to be managed by the common practice of development on the vein prior to the development of the waste accesses.

The key issue for the production schedule will be an anticipated production dip in 2015. The dip will be due to the Main Shaft development being scheduled for completion in 2014. There will be six months of shaft furnishing with no rock hoisting assumed during this phase, which is estimated to finish towards the end of 2014. There will then be significant development required from the Main Shaft initial plat development to access the stoping areas plus establishment of a primary vent circuit prior to production commencing from the -100 mRL and lower levels.

AMC considers that the geotechnical aspects of the GMADI mine design are generally reasonable for mining study purposes. However, given the limited nature of the data, the geotechnical knowledge at the project is not considered to be at the level of detail normally associated with a mining feasibility study in Canada, and is more in line with a scoping level study. Geotechnical recommendations are listed under Section 26 of this report.

Flotation tests on metallurgical samples have established a workable set of flotation conditions and reagents and enabled reasonable predictions of concentrate grades and recoveries. Despite the fine grain size and resulting low gravity recoveries, a tin recovery circuit appended to the end of the main circuit would be low cost and potentially viable. There may be an opportunity in the current high silver price environment to increasing further the silver recovery at the expense of a lower lead concentrate grade.

The recovery methods proposed are generally appropriate for the ore characteristics as tested, and the proposed flowsheet should achieve the targeted recoveries and concentrate grades. However, the comminution circuit, especially grinding, is undersized for the 1,600 tpd throughput level, especially as there is no testwork data and therefore a degree of conservatism is warranted.

The likely arsenic levels in all concentrates (0.5 %As in the lead and zinc concentrates and >1%As in the pyrite concentrate) would potentially pose concentrate marketing problems to western smelters. However there is no mention of arsenic levels in Silvercorp's marketing contracts with Chinese smelters and AMC has been able to verify from direct experience of recent Chinese smelter contracts that, notwithstanding the various grades within the national standards, arsenic levels up to 2-3 %As are in fact acceptable in a precious metals-bearing pyrite concentrate. AMC understands that acceptable arsenic levels in base metal concentrates are of the order of 0.5% As and therefore the GC lead and zinc concentrates are right at the limit of acceptability.

Although AMC believes that the concept of the proposed dry stacking tailings management facility (TMF) is reasonable, it considers that the work carried out to date towards its design does not meet feasibility study standards and that several areas need addressing as summarized under Section 26 of this report.

AMC has estimated initial capital expenditure of \$67.4M (including mining, mill, infrastructure, owner's costs and contingency) and total operating costs of \$40.6/t milled (including mining, processing, G&A and surface service costs).

Using long-term metal prices of silver \$18/oz, lead \$1.00/lb and zinc \$1.00/lb, and a USD:RMB exchange rate of 6.35, AMC has estimated an NPV at an 8% discount rate of \$73.7M, an IRR of 33% and payback in 2.4 years. If a probability-weighted average of plausible operational scenarios is taken instead of a single-point base case estimate, the results are an NPV at an 8% discount rate of \$53.0M, an IRR of 28% and payback in 3.3 years. These economic metrics are positive and demonstrate that the project is robust in the face of the possible scenarios that typically impact on a mining operation.

## 26 RECOMMENDATIONS

Stated costs are estimated for those recommendations not covered by operational activities.

### Geology / Mineral Resources

- Undertake variography studies to refine the understanding of the grade distribution and utilize a kriging or inverse distance weighting approach to grade interpolation prior to future resource and reserve estimations

### Mining / Geotechnical

- Assess ground conditions on a round by round basis in all development headings.
- Ensure scaling of the development heading on a round by round basis.
- Conduct routine check scaling of all unsupported development.
- Where possible, avoid mining development intersections in fault zones, and design drifts to cross fault zones at right angles.
- Assess specific rock mass conditions for critical underground infrastructure, including shafts and chambers, to determine ground support requirements to ensure serviceability of the excavation for the life of mine.
- Collect additional detailed geotechnical logging data from drill core and mapping of underground workings, to allow improved characterization of rock mass conditions within the immediate stope hangingwall zone, and the mineralized veins.
- Develop a three dimensional geological model with interpretations of primary lithologies and structures.
- Investigate proposed shaft locations to determine site suitability and ground support requirements.
- Further investigate the surface crown pillar, particularly in the vicinity of the Hashui Creek valley, and any other streams or drainage paths that traverse the mine area.
- Undertake further hydrogeological studies, particularly to assess hydraulic connectivity between the Hashui Creek valley (and any other streams or drainage paths that traverse the mine area) and the underground mine workings. Estimated cost \$75,000.

### Processing

- Install an additional 600 kW of grinding power to address the undersizing of the comminution circuit. Estimated cost \$500,000 installed and this has already been included in the capital cost estimate as it is deemed essential.
- Give consideration to a small increase in lead cleaner and filtration capacity to allow for optimization of silver recovery to payable lead concentrates. Estimated cost \$100,000 and this has not been included in the capital cost estimate as further validation is required.
- Double the tailings filtration capacity, as the two XA90/920 filters selected have been sized for 1,000 tpd ore feed. Estimated cost \$580,000 and this has been included in the capital cost estimate as it is also deemed essential.

### **Tailings Management Facility**

- Undertake additional testwork including:
  - Proctor compaction tests to derive target moisture levels for the required compacted density. Estimated cost \$2,000.
  - Shear tests to assess the internal strength of the tailings as an input to stability analysis. Estimated cost \$2,000.
  - More comprehensive size analysis, to include potential clay component size range. Estimated cost \$5,000.
  - Geochemical characterization e.g. metal leaching tests. Estimated cost \$10,000.
  - Filtration tests to assist in the pressure filter sizing to meet target moisture levels. Estimated cost \$15,000.
- Undertake further site investigations, including:
  - Geotechnical evaluation of underlying bedrock etc
  - An assessment of the implications of the Gaocheng River class II water resource classification for the TMF location and design. Estimated cost \$50,000 for both.
- Undertake a site-specific risk assessment as opposed to the generic grade III design criteria within the Chinese volume-height categories. Estimated cost \$30,000.
- Reassess the factor of safety calculations using standard industry practice finite element numerical modeling
- Prepare a more detailed water balance on a month-by-month basis.

### **Power**

- Provide emergency backup power for essential critical services such as man cage, mine ventilation, mine dewatering pumps and thickeners. Estimated cost \$800,000.

## 27 REFERENCES

- Anhui Yangzi Mining Co. Ltd., November 2007. The summary of the work at Shimentou project area.
- Guangdong Found Mining Co. Ltd., Mining Engineering Construction Contract of Gaocheng Lead-Zinc Mine Project, Mine Engineering Contract (GF-2011-0225), March 19 2011.
- Guangdong Provincial Institute of Geological Survey, April 2005. Geological report about general prospecting on Jianshan-Shimentou Pb-Zn multi-metallic deposit, Yunfu city, Guangdong province
- Guangdong Provincial Institute of Geological Survey, September 2007. Geological report about detailed prospecting on GC Pb-Zn-Ag deposit, Yunfu city, Guangdong province
- Jia, Shouyi and Songqing, Ye, 2003. Mineral Resource Exploration. Geological Publishing
- Liu, Jinhui, Niu, Lanliang, Xu, Anson, and Wang, Zhaojun (SRK Consulting China Ltd.), April 2008: Technical Report On Gaocheng Ag-Zn-Pb Project and Shimentou Au-Ag-Zn-Pb Project, Guangdong Province, People's Republic of China
- Mineral and Dressing Project of Gaocheng Lead-Zinc Ore in Yun'an County, Guangdong Province, Preliminary Design (GD1371CS) Volume 1 Instruction by GMADI, Guangdong Metallurgical & Architectural Design Institute, China, January 2011.
- Silvercorp Metals Inc. (April 28, 2008). Silvercorp Acquires Significant Silver-Lead-Zinc Resources in Guangdong Province, Southern China. Press Release. [http://www.silvercorpmetals.com/news/2008/index.php?&content\\_id=106](http://www.silvercorpmetals.com/news/2008/index.php?&content_id=106)
- Silvercorp Metals Inc. (June 6, 2008). Silvercorp Completes Acquisition of Significant Silver-Lead-Zinc Resources in Guangdong Province, Southern China. Press Release. [http://www.silvercorpmetals.com/news/2008/index.php?&content\\_id=110](http://www.silvercorpmetals.com/news/2008/index.php?&content_id=110)
- Silvercorp Metals Inc. (January 8, 2009). Drilling Intersects High-Grade Silver Mineralization in V2 and V6 Veins at the GC Silver-Lead-Zinc Project in Guangdong Province, Southern China. Press Release. [http://www.silvercorpmetals.com/news/index.php?&content\\_id=180](http://www.silvercorpmetals.com/news/index.php?&content_id=180)
- Silvercorp Metals Inc. (June 5, 2009). Annual Information Form. Annual Report. [http://www.silvercorpmetals.com/\\_resources/Silvercorp\\_AIF\\_rev\\_SEDAR\\_opt.pdf](http://www.silvercorpmetals.com/_resources/Silvercorp_AIF_rev_SEDAR_opt.pdf)
- Silvercorp Metals Inc. (June 3, 2009). Consolidated Financial Statements. Annual Report. [http://www.silvercorpmetals.com/\\_resources/fin/annual/2009\\_SVM\\_FS\\_final.pdf](http://www.silvercorpmetals.com/_resources/fin/annual/2009_SVM_FS_final.pdf)
- Silvercorp Metals Inc. (June 3, 2009). Management's Discussion and Analysis ("MD&A"). Annual Report. [http://www.silvercorpmetals.com/\\_resources/fin/annual/2009\\_SVM\\_MDA\\_final.pdf](http://www.silvercorpmetals.com/_resources/fin/annual/2009_SVM_MDA_final.pdf)
- Barton, N., Lien, and R., Lunde, J., (1974), 'Analysis of Rock Mass Quality and Support Practice in Tunneling and a Guide to Estimating Support Requirements' Geotech Inst. Report. No 54206.
- Hoek, E., Kaiser, P.K., and Bawden, W.F., (1995), 'Support of Underground Excavations in Hard Rock', A. A. Balkema, Rotterdam.
- Hutchinson, D.J., and Diederichs, M.S., (1996), 'Cablebolting in Underground Mines', BiTech Publishers Ltd., British Columbia, Canada.
- Mine Engineering Contract, (GF-2011-0225), Guangdong Found Mining Co., Ltd. Mining Engineering Construction Contract of Gaocheng Lead-zinc Mine Project March 19, 2011

## 28 CERTIFICATES OF QUALIFIED PERSONS

### A Riles

1. I, Alan Riles, MAIG, BMet (Hons), Grad Dipl Professional Management, do hereby certify that I am Associate Principal Consultant Metallurgist with AMC Mining Consultants (Canada) Ltd, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4, Canada.
2. The Technical Report to which this certificate applies is entitled "NI 43-101 Technical Report on the GC Ag-Zn-Pr Project in Guangdong Province, Peoples' Republic of China" and is effective 31 December 2011 (the "Technical Report").
3. I graduated with a Bachelor of Metallurgy (Hons Class 1) from Sheffield University, UK in 1974. I am a registered member of the Australian Institute of Geoscientists. I have practiced my profession continuously since 1974, with particular experience in study management and both operational and project experience in precious and base metal deposits. I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I visited the GC property in May 2011 for two days.
5. I am responsible for the preparation of Sections 13, 17, part of 18, 19, 21 and 22 of the Technical Report.
6. I am independent of the issuer as described in Section 1.5 of NI 43-101.
7. I have had no prior involvement, with the GC property.
8. I have read NI 43-101 and certify that the parts of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
9. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25<sup>th</sup> day of January 2011

*Original signed by*

---

Alan Riles, B.Met, MAIG

**M Molavi**

1. I, Mo Molavi P. Eng., M Eng, B Eng, of Vancouver, British Columbia do hereby certify that I am a Principal Mining Engineer with AMC Mining Consultants (Canada) Limited, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4.
2. The Technical Report to which this certificate applies is entitled "NI 43-101 Technical Report on the GC Ag-Zn-Pr Project in Guangdong Province, Peoples' Republic of China" and is effective 31 December 2011 (the "Technical Report").
3. I graduated with a B Eng in Mining Engineering from the Laurentian University in Sudbury Ontario in 1979 and an M Eng in Mining Engineering specializing in Rock Mechanics and mining methods from the McGill University of Montreal in 1987. I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of Saskatchewan and a Member of the Canadian Institute of Mining, Metallurgy and Petroleum. I have worked as a Mining Engineer for a total of 30 years since my graduation from university and have relevant experience in project management, feasibility studies and technical report preparations for mining projects in North America. I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I have not visited the GC property.
5. I am responsible for the preparation of parts of Section 18 of the Technical Report.
6. I am independent of the issuer as described in section 1.5 of NI 43-101.
7. I have had no prior involvement with the GC property.
8. I have read NI 43-101 and certify that the part of the Technical Report for which I am responsible has been prepared in compliance with the Instrument.
9. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the part of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25<sup>th</sup> day of January 2012

*Original signed and sealed by*

---

Mo Molavi P. Eng

**B O'Connor**

1. I, Brian F J O'Connor, P.Geo, BSc, of Vancouver, British Columbia do hereby certify that I am a Principal Geologist with AMC Mining Consultants (Canada) Limited, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4.
2. The Technical Report to which this certificate applies is entitled "NI 43-101 Technical Report on the GC Ag-Zn-Pb Project in Guangdong Province, Peoples' Republic of China" and is effective 31 December 2011 (the "Technical Report").
3. I graduated with a BSc in Geology from the University of New Brunswick in 1985. I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia. I have worked as a geologist for 26 years since my graduation from university and have relevant experience in geology, exploration and mineral resource estimation for base and precious metal deposits, with particular expertise in the exploration, evaluation and the mining of narrow vein deposits. I have 6 years experience in the collection and management of environmental data for a mining environment including the preparation of report filings for government departments. I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I visited the GC property in March 2009 for three days and in May 2011 for two days.
5. I am responsible for the preparation of Section 2 to 12, 14, 20, 23 and 24 of the Technical Report.
6. I am independent of the issuer as described in section 1.5 of NI 43-101.
7. I prepared a previous Technical Report on the GC property in June 2009.
8. I have read NI 43-101 and certify that the parts of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
9. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25<sup>th</sup> day of January 2012

*Original signed and sealed by*

---

Brian O'Connor P. Geo

**P Stephenson**

1. I, Patrick R Stephenson, P. Geo, BSc (Hons), FAusIMM (CP), MAIG, MCIM, of Vancouver, British Columbia, do hereby certify that I am General Manager and a Principal Geologist with AMC Mining Consultants (Canada) Limited, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4.
2. The Technical Report to which this certificate applies is entitled "NI 43-101 Technical Report on the GC Ag-Zn-Pr Project in Guangdong Province, Peoples' Republic of China" and is effective 31 December 2011 (the "Technical Report").
3. I graduated with a BSc (Hons) in Geology from Aberdeen University in Scotland in 1971. I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia. I have worked as a Geologist and Manager for a total of 40 years since my graduation from university and have relevant experience in geology, exploration and mineral resource estimation for base and precious metal deposits and in public reporting of mineral assets. I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I have not visited the GC property.
5. I am responsible for the preparation of Sections 1, 25 and 26 of the Technical Report.
6. I am independent of the issuer as described in section 1.5 of NI 43-101.
7. I peer reviewed the Technical Report prepared by AMC on the GC property in June 2009.
8. I have read NI 43-101 and certify that the parts of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
9. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25<sup>th</sup> day of January 2012

*Original signed and sealed by*

---

Patrick Stephenson, P.Geo

**P Mokos**

1. I, Peter Mokos, BSc (Eng), DipEng (Mining), MAusIMM (CP), of Vancouver, British Columbia do hereby certify that I am a Principal Mining Engineer with AMC Mining Consultants (Canada) Limited, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4.
2. The Technical Report to which this certificate applies is entitled "NI 43-101 Technical Report on the GC Ag-Zn-Pr Project in Guangdong Province, Peoples' Republic of China" and is effective 31 December 2011 (the "Technical Report").
3. I graduated with a Bachelor of Engineering (Mining) from the Ballarat College of Advanced Education in Australia in 1985. I am a Member of the Australasian Institute of Mining and Metallurgy with Chartered Professional status. I have worked as a Mining Engineer for a total of 27 years since my graduation and have relevant experience in underground mining, feasibility studies and technical report preparation for mining projects. I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I visited the GC property in May 2011 for two days and again in July-August 2011 for a further ten days.
5. I am responsible for the preparation of Sections 15 and 16 of the Technical Report.
6. I am independent of the issuer as described in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the GC property.
8. I have read NI 43-101 and certify that the parts of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
9. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25<sup>th</sup> day of January 2012

*Original signed by*

\_\_\_\_\_  
Peter Mokos, MAusIMM (CP)

**O Watson**

1. I, Owen Watson BEng (Geological) (Hons), MAusIMM (CP), MCIM of Vancouver, British Columbia do hereby certify that I am a Senior Geotechnical Engineer with AMC Mining Consultants (Canada) Limited, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4.
2. The Technical Report to which this certificate applies is entitled "NI 43-101 Technical Report on the GC Ag-Zn-Pr Project in Guangdong Province, Peoples' Republic of China" and is effective 31 December 2011 (the "Technical Report").
3. I graduated with a Bachelor of Engineering (Geological) from RMIT University, Melbourne, Australia, in 1998. I am a Member of the Australasian Institute of Mining and Metallurgy with Chartered Professional status. I have worked as a Geotechnical Engineer for a total of 14 years since my graduation from university and have relevant experience in geotechnical issues for underground base and precious metal mines. I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I visited the GC property in May 2011 for two days.
5. I am responsible for the preparation of parts of Sections 15 and 16 of the Technical Report.
6. I am independent of the issuer as described in section 1.5 of NI 43-101.
7. I have had no prior involvement with the property that is the subject of this Technical Report.
8. I have read NI 43-101 and certify that the parts of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
9. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25<sup>th</sup> day of January 2012

*Original signed by*

Owen Watson, MAusIMM (CP)