Report to:

MINCO SILVER CORPORATION

Fuwan Silver Project
Feasibility Study Technical Report

Document No. 0853900100-REP-R0007-05

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FUWAN SILVER PROJECT FEASIBILITY STUDY TECHNICAL REPORT

EFFECTIVE DATE: SEPTEMBER 1, 2009

ISSUE DATE: OCTOBER 23, 2009

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REVISION HISTORY

REV. NO	ISSUE DATE	PREPARED BY AND DATE	REVIEWED BY AND DATE	APPROVED BY AND DATE	DESCRIPTION OF REVISION
00	Sept. 18/09	R.K. Sept. 18/09	J.H. Sept. 18/09	P.W. Sept. 18/09	First draft issued to Client.
01	Sept. 28/09	R.K. Sept. 28/09	J.H. Sept. 28/09	P.W. Sept. 28/09	Second draft issued to Client.
02	Oct. 09/09	R.K. Oct. 09/09	J.H. Oct. 09/09	P.W. Oct. 09/09	Final draft issued to Client.
03	Oct. 16/09	R.K. Oct. 16/09	J.H. Oct. 16/09	P.W. Oct. 16/09	Final revised draft issued to Client.
04	Oct. 23/09	R.K. Oct. 23/09	J.H. Oct. 23/09	P.W. Oct. 23/09	Final issued to Client.
05	Oct. 23/09	R.K. Oct. 23/09	J.H. Oct. 23/09	P.W. Oct. 23/09	Final issued to Client.

TABLE OF CONTENTS

LIST OF APPENDICES

LIST OF TABLES

LIST OF FIGURES

GLOSSARY

UNITS OF MEASURE

Above mean sea level .... amsl
Acre .... ac
Ampere .... A
Annum (year) .... a
Billion .... B
Billion tonnes .... Bt
Billion years ago .... Ga
British thermal unit .... BTU
Centimetre .... cm
Cubic centimetre .... cm^3
Cubic feet per minute .... cfm
Cubic feet per second .... ft^3/s
Cubic foot .... ft^3
Cubic inch .... in^3
Cubic metre .... m^3

Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	Cdn$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh

Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million bank cubic metres per annum	Mbm^3/a
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D

Three Dimensional Model	3DM
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

Abbreviations and Acronyms

3-dimensional	3D
Abrasion resistant	AR
Acid base accounting	ABA
Acid base accounting	ABA
Acid Rock Drainage	ARD
Alphair Ventilating Systems Inc.	Alphair
Amino-dithiophosphate	ADTP
Ammonium dibutyl dithiophosphate	ADDP
Ammonium Nitrate and Fuel Oil	ANFO
Atomic absorption spectrophotometer	AAS
Barton's Tunnelling Quality Index	Q
Beijing General Research Institute for Mining and Metallurgy	BGRIMM
Bond Abrasion Index	AI
Bond Low-energy Impact	CWI
Bond Rod Mill Grindability	RWI
Canadian Institute of Mining, Minerals, and Petroleum	CIM
Changsha Engineering and Research Institute of Nonferrous Metallurgy	CINF
Closed circuit television	CCTV
Construction Management	CM
Copper sulphate	$CuSO_4$
Distributed control system	DCS
Effective grinding length	EGL
Energy and Metals Consensus Forecast	EMCF
Energy and Metals Consensus Forecast	EMCF
Engineering, Procurement, and Construction Management	EPCM
Environmental Management Plan	EMP
Environmental, Socio-Economic and (Community) Health Management Plan	ESHMP
EPCM Contractor	Contractor
Foot wall	FW
Free board marine	FOB
Free carrier	FCA
Front-end loader	FEL

General & Administrative	G&A
Guangdong Geological Exploration and Development Corporation	GGEDC
Guangdong Research Institute of Mineral Utilization	GRIMU
Hang wall	HW
Harris Exploration Services	Harris
Health and Safety Management Plan	HSMP
Health, Safety, and Environmental policy	HSE
Inductively Coupled Plasma	ICP
Internal Rate of Return	IRR
Institute of Geophysical and Geochemical Exploration	IGGE
Lehne and Associates Applied Mineralogy	Lehne
Lime slurry	$Ca(OH)_2$
Lime	CaO
Load-haul-dump	LHD
Locked cycle test	LCT
London Metal Exchange	LME
Low grade	LG
Material Takeoffs	MO
Minco Silver Corporation	Minco
Nanchang Engineering & Research Institute of Nonferrous Metals	NERIN
National Instrument 43-101	NI 43-101
Net neutralization potential	NNP
Net Present Value	NPV
Net Smelter Return	NSR
Non-electric	NONENL
Operator interface stations	OIS
Ore zone	Ore
P&E Mining Consultants Inc.	P&E
Peoples Republic of China	PRC
Potassium ethyl xanthate	PEX
Process and instrumentation drawings	P&IDs
Process Research Associates Ltd.	PRA
Project Management System	PMS
Project Management Team	PMT
Project Procedures Manual	PPM
Quality Assurance/Quality Control Plan	QA/QC
Quality Control	QC
Renminbi	¥
Request for Proposal	RFP
Rock Quality Designation	RQD
Run-of-mine	ROM
Semi-autogenous grinding	SAG
Semi-autogenous grinding-ball mill-pebble crushing circuit	SABC
Sewage Treatment Plant	STP
SGS Lakefield Minerals Services	SGS
Shanghai Gold Exchange	SGE

Sodium diethyl dithiocarbamate	SDTC
Sodium isopropyl xanthate	SIPX
SRK Consulting Canada	SRK
Tailings Management Facility	TMF
Traffic and Logistics	T&L
Ventsim Mine Ventilation Simulation Software	Ventsim
Wardrop, A Tetra Tech Company	Wardrop
Waste Rock Facility	WRF
Waste Rock Storage	WRS
Water treatment plant	WTP
Work Breakdown Structure	WBS
Zinc sulphate	$ZnSO_4$

1.0 SUMMARY

1.1 INTRODUCTION

The Fuwan silver-gold-lead-zinc deposit is owned by Minco Silver Corporation (Minco) and is located in Guangdong Province, southern China, about 45 km southwest of the provincial capital Guangzhou. The deposit was tested with 422 holes up to May 2008 with an aggregate length of approximately 96,000 m.

In November 2007, SRK Consulting Canada Inc. (SRK) completed a Preliminary Economic Assessment of the Fuwan deposit. In May 2008, Changsha Engineering and Research Institute of Nonferrous Metallurgy (CINF) completed a Pre-feasibility Study.

Minco has retained Wardrop, A Tetra Tech Company (Wardrop) to produce a Feasibility Study of the Fuwan property that is compliant with the reporting standards of National Instrument 43-101 (NI 43-101).

The principal consultants utilized by Minco in the preparation of this Fuwan Feasibility Study are as follows:

- Wardrop – mining, processing, capital cost (mining) and financial analysis
- P&E Mining Consultants Inc. – geology and resource estimation
- Environmental Resources Management (ERM) – environmental
- China Nerin Engineering Co. Ltd (NERIN)/Wardrop – infrastructure, overall site water management, hydrogeology, tailings and waste rock disposal, and capital cost (excluding mining).

1.2 GEOLOGY & RESOURCE ESTIMATION

The Fuwan silver deposit is located at the northwest margin of a triangular Upper Paleozoic fault basin at the juncture of the northeast-trending Shizhou fault to the northwest, the east-west trending Dashi fault to the south, and the northwest trending Xijiang fault to the northeast. There are known precious and base metal occurrences and deposits that occur predominantly along the margins of the basin.

The basin contains Lower Carboniferous limestone and unconformably overlying Triassic siliciclastic rocks. A low-angle fault zone (from several to tens of metres in thickness) is developed along the contact between the Lower Carboniferous unit and the Upper Triassic unit, and is occupied by lenticular zones of brecciated limestone and silicified sandy conglomerate. The fault zone may have acted as both a conduit for mineralizing fluids and as a host for the silver mineralization in the area. Second-order faults, parallel to the major fault and also containing silver mineralization, occur in the footwall limestone.

The Fuwan silver deposit falls into the broad category of sediment-hosted epithermal deposits and is characterized by vein and veinlet mineralization within zones of silicification. The predominant sulphide minerals are sphalerite and galena with lesser pyrite, as well as rare arsenopyrite, chalcopyrite, and bornite. The deposit is poor in gold (typically <0.2 ppm).

Diamond drill data from 231 out of a total of 422 holes were used for the resource calculation. Most holes were drilled at 80 m spaced sections and the central portion of the deposit was drilled at 40 m spaced sections that gave an effective 60 m x 60 m diagonal drill pattern.

Eight zones of silver mineralization have been identified:

- Zone 1, lying entirely within the fault plane, contains a relatively large volume of silver mineralization particularly in the west part.
- Zone 2, partially within the brecciated and silicified fault zone, contains the greatest volume of silver mineralization.
- Zone 3 occurs in the footwall of the main fault zone.
- Zones 4, 5, and 6 are situated entirely within the footwall along planar fractures in the limestone.
- Zone 7 is located in the Luzhou area, along strike to the southwest of the main Fuwan silver deposit.
- Zone 8 is located on the east side of the Xijiang River, along strike to the north east of the main Fuwan silver deposit.

Zones 7 and 8 are not included in the Fuwan resource estimate. The following is a summary of the May 2008 Fuwan resource estimate prepared by P&E Mining Consultants Inc. (P&E) at a cutoff of 40 g/t silver.

Table 1.1 Resource Estimate Summary at a 40 g/t Silver Cutoff – May 2008

Resource Area & Classification	Tonnes	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
Fuwan Permit Indicated	13,948,000	188	0.17	0.20	0.56
Fuwan Permit Inferred	10,241,000	171	0.26	0.26	0.72

Notes:

- Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
- The quantity and grade of reported inferred resources in this estimation are conceptual in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.
- The mineral resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council December 11, 2005.

1.3 MINING

1.3.1 RESERVE ESTIMATE

The resource estimate provided by P&E classified the resources for the Fuwan Zones 1 to 4 as indicated and inferred (Table 1.1). Only indicated mineral resources as defined in NI 43-101 were used to establish the probable mineral reserves. No reserves were categorized as proven.

Some of the wireframes for the resources provided geologically improbable shapes in the indicated resources in the May 2007 block model that would be difficult to mine. The mine design battery limit was to accept the resource estimate and interpretation at face value and prepare a mine design around it.

It will be essential for infill drilling to be undertaken during the basic engineering and detailed mine design phases for the production of detailed stope and development layouts for construction and mining. It is also Wardrop's opinion that there appeared to be no marker horizons to follow high grade zones within the limestone. It will be difficult if not impossible to follow economic mineralization visually during mining. Infill drilling will be essential to define the true orebody outlines ahead of development and stoping.

In order to obtain the mining permits in China, it is necessary to use an official Chinese resource estimate prepared according to Chinese codes. The Chinese resource may not be the same as the NI 43-101 resource used for this study.

Wardrop received the block model that was used for the P&E resource estimate then applied mining and economic parameters to the model in order to form the basis of the reserve estimate. Since the deposit is polymetallic, it was decided to estimate the net smelter return (NSR) for each block in the model in order to design the stope outlines and evaluate economic viability.

The NSR value was calculated assuming the three-year historical average metal prices from the London Metal Exchange (LME) as at May 31, 2009:

- US$13.00/oz for silver
- US$688.00/oz for gold
- US$0.88/lb for lead
- US$1.28/lb for zinc.

Factors for each contributing metal were calculated and input into the block model to calculate the NSR for each block within the model. The metallurgical and smelting metal recoveries, smelter and refining charges, and metal prices were incorporated into the following NSR formula:

NSR = (0.31 * in-situ g/t Ag grade) * (6.07 * in-situ g/t Ag grade) * (311.66 * in-situ % Pb grade) * (1,563.94 * in-situ % Zn grade)

NSR Cutoff Grade

A cutoff grade of US$37.13/t NSR was used for the reserve estimate and was selected based on estimated operating costs as shown in Table 1.2.

Table 1.2 Operating Costs for the Reserve Estimate

Area	Unit Cost (US$/t)
Mine	18.41
Process	10.77
Tailings Management	1.30
Surface Services	0.79
General & Administrative	5.86
Total	**37.13**

Wardrop used a stope recovery factor of 95%, an average mining extraction rate of 97%, and an average 7% internal dilution, 8% external dilution, and 3% fill dilution to estimate the total amount of diluted probable mineral reserves. Ore reserve calculations conservatively assumed dilution to contain no metal.

The probable mineral reserve estimate is 9,117,980 t at 189 g/t Ag, 0.146 g/t Au, 0.196% Pb, and 0.566% Zn. Table 1.3 lists the reserve estimate by zone.

Table 1.3 Probable Reserve Estimate Summary

Zone	Tonnes	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
1	1,327,580	186	0.180	0.064	0.324
2	4,806,443	192	0.167	0.177	0.568
3	2,451,699	192	0.105	0.257	0.636
4	532,259	150	0.068	0.421	0.822
Probable Mineral Reserve	**9,117,980**	**189**	**0.146**	**0.196**	**0.566**

1.3.2 GEOTECHNICAL

In general, the ground conditions within the orebody are predicted to be good with few localized stability problems. However, at the unconformity, particularly difficult ground conditions are expected with a fault zone that will probably be exposed in immediate stope backs.

The recommended support for waste development is as follows:

- backs – 2.4 m long bolts on 1.2 m by 1.2 m pattern
- walls – 2.4 m long bolts on 1.5 m by 1.5 m pattern
- allow 25% coverage with a welded wire mesh square measuring 100 mm by 100 mm with 4 mm diameter wire
- allow 25% coverage with shotcrete 50 mm nominal thickness.

Areas that intersect the unconformity will require full bolt, mesh, and shotcrete support.

Stopes have been sized to avoid the use of cable bolting. Drift-and-fill stopes will be 4 m wide with the unconformity in back, and 6 m wide with no unconformity. Any stope back with the unconformity exposed will require full bolt, mesh, and shotcrete support.

1.3.3 HYDROGEOLOGY

Wardrop performed a hydrogeological review of the available data. This review incorporated the results of field investigations undertaken by the local consultant 757 Exploration Team on both the Fuwan exploration area and the adjacent Changkeng exploration area in 2007 and 2008, as well as historic information from a variety of sources.

The scope of the recent hydrogeological investigations included the performance of 29 small scale pumping tests on a series of 13 geological exploration holes converted to groundwater monitoring and test wells. These tests were undertaken on multiple formations within each monitoring well, or at multiple pumping rates in order to allow for assessment of the hydrogeological characteristics of the various geological units. The results of these test indicated that the sandstone unit was in general a low conductivity unit, with limited potential for groundwater production. In some boreholes, high conductivities were noted, potentially due to interconnectivity with the underlying carbonate unit. The carbonate unit, which has been extensively affected by a shallow fault zone passing along the sandstone/ carbonate interface (referred to as the unconformity), and demonstrates karst conditions (i.e., solution enlarged fracturing and void spaces). Pumping tests performed on this unit suggested a moderate rate of groundwater production.

Two large scale pumping tests were subsequently undertaken, one in the Fuwan exploration area and one in the Chankeng exploration area. These tests involved the long term pumping (4 to 7 days) of a reamed out exploration borehole at rates of 15 to 24 L/s, and the regular monitoring of water levels in a series of surrounding monitoring wells completed in both the carbonate unit and the overlying sandstone. Analysis of the resulting pumping test data showed the carbonate unit to have a relatively high conductivity (1.1. x 10^{-5} m/s) and good hydraulic connectively over a large area (drawdown cone 9 m deep at the pumping well and extending at least 1.5 km in all directions. This pumping test data also suggests that the geological faults in the area do not have any significant influence on the drawdown cone so likely do not act as a source of groundwater recharge.

Although this testing did not identify any significant concern with respect to the faults, the scale of the pumping test response indicates that the karst formation is highly connected over a significant area with a transmissivity at the high end of the published range for carbonate systems.

Preliminary estimates for groundwater inflow into a simplified single stope running along the base of the mineral deposit (260 m below sea level) over its entire length (1100 m) were developed using a variety of standard formulas. These formulas applied to dewatering of a linear excavation, relative comparison to local recorded dewatering requirements, simplified water balance, and general inflow into a tunnel excavation. These preliminary estimates suggested that groundwater inflow could potentially be in the range of 4,550 to 27,011 m^3/day. Due to the natural heterogeneity of the subsurface conditions, inflow rates within certain excavation areas may be higher or lower than this average rate, with initial rates also being higher than later stage flow rates. There remains some unknown areas and further work is required to better understand the underground hydrogeology.

In order to refine the potential groundwater inflow rates, the existing geological and hydrogeological information, along with surface water and meteorological data, as collected by various parties should be compiled into a detailed hydrogeological model of the area, and calibrated against the existing large scale pumping test data set. Supplementary pumping test in the area of the F3 Fault should also be considered in order to complete the data set.

Due to the potential for large volume groundwater inflows into the proposed mine excavations, predictive and mitigations measures such as probe hole advancement in all proposed excavation areas, the installation of groundwater collection and drainage galleries, and installation of water tight doors or bulkheads at regular intervals will be required.

The potential for interconnection between the Xijiang River and proposed underground mine workings has been evaluated qualitatively from a geological point of view by 757 Team. Their interpretation was that the fine river bottom sediments (clay and silt) and low conductivity T3 unit underlying the river area would minimize direct hydraulic connection between the river and the Fx1 + C1 water bearing unit. The primary potential source of connection was therefore the apparent Changkeng, F2 and F3 fault traces which appear to extend out under the river. The SRK report indicated that the Xijiang River appears to be poorly connected hydraulically with the proposed underground mine envelope in the areas tested.

1.3.4 *Mining Methods*

Minco will develop a mechanized mine at the Fuwan deposit. A 2 m minimum mining height was adopted for mechanized mining. The selection of a mining method is dependent upon orebody geometry, ground conditions, and ore grade.

Drift-and-fill mining, and a small amount of room-and-pillar mining, will be used for flat lying zones. As the orebody has reasonably good grades, a trade-off study was undertaken to assess at what grade it would be worth backfilling with cemented fill and carrying out a primary/secondary drift-and-fill type mining method allowing 100% extraction without leaving any ore pillars.

Ore zones with lower grades will be mined by the room-and-pillar method. This method is selective and zones of low grade can be left as pillars. A variation of this method is post pillar cut-and-fill: where the ore height is greater than 6 to 7 m, the panel is taken in two cuts. The first cut is taken and backfilled, then a second cut is taken over the top of the first cut working off the backfill.

Stope and pillar dimensions, ground support in development headings, and stopes will depend on orebody geometry and ground condition.

The cut-and-fill method will be used for ore zones dipping between 15° and 50°.

In order to minimize waste development, Wardrop recommends using in-ore twin ramp development. Each panel will be about 100 m long and typically 60 m vertically. Twin ramps will be driven in ore from top and bottom access to meet in the middle of the stope. A minimum 3 m-wide pillar (or a 1:1 ore to pillar width) will be left between the ramps. The ramp below the pillar must always remain open for air passage and provide through-ventilation. After the ventilation airway is no longer needed, the pillar could be recovered; however, any estimate should only assume an effective 50% recovery of the pillar.

BACKFILL

All stopes will be backfilled after mining is completed. Free draining hydraulic backfill was selected as the most appropriate method due to the flat-lying and relatively large horizontal extent of the orebody, coupled with the distant location of the process plant and difficulties with accesss above the orebody. This backfilling method will allow up to 45 to 50% of the tailings to be disposed of as hydraulic backfill underground, reducing the required size of the surface tailings pond.

Backfill will be prepared from tailings produced in the plant and distributed to the underground stopes by a pipeline through the main access ramp. For primary stope filling in drift-and-fill, 5% cement will be added. Backfill for cut-and-fill, room-and-pillar, and secondary stopes of drift-and-fill mining will not be cemented.

1.3.5 MINE ACCESS

The mine will be accessed by a single decline developed at gradient of -15%. It will be used for access of personnel, equipment, materials, and services; it will also be utilized as an intake airway.

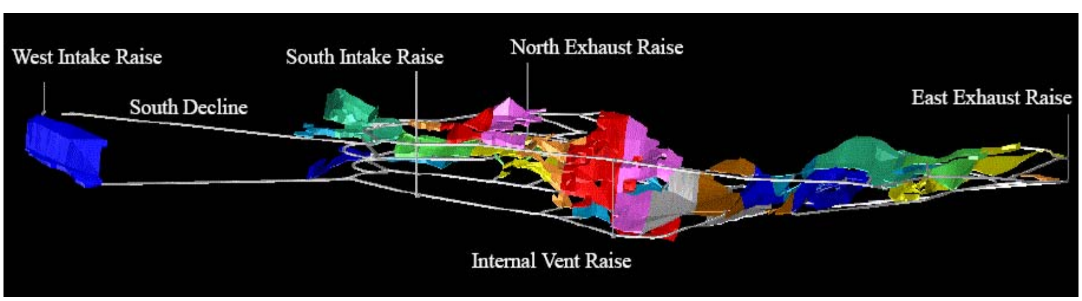
The location of the decline portal was selected on the south-west side of the deposit near the process plant. The size of the decline was selected at 4.5 m wide by 4.5 m high to accommodate the mining equipment and provide required clearances.

The four levels will be developed for haulage and for provision of fresh air supply to mining blocks. Ventilation access drifts will be excavated to connect the level development and ramps to the ventilation raises.

The 4.0 m diameter central south fresh air intake ventilation raise will have a man-way equipped with ladders and platforms to provide an auxiliary exit from the mine in case of emergency. Two 4.0 m diameter exhaust raises will be developed on the east and north side of the orebody and will be connected to the level development to provide flow-through ventilation. They will also be equipped with ladders.

Another 3.0 m diameter fresh air ventilation raise will be constructed in Year 6 of production on the west side of the deposit to provide intake air for mining block #201, and will be equipped with a man-way for emergency exit.

Figure 1.1 Access Development



Development headings will be driven with electro-hydraulic twin boom jumbos. Ventilation raise development will be done by raiseboring crews.

The broken rock will be mucked from the face by 7 t load-haul-dump (LHD) and hauled by 25 t trucks to the surface waste dump. The same equipment will be used for mucking broken ore from the production stopes and hauling to the mill for processing.

A 7 t capacity LHD with a 4.0 m^3 bucket and a 25 t underground mine truck with a 13.0 m^3 box were selected for ore and waste haulage.

A summary of ore and waste production is provided in Table 1.4.

Table 1.4 Production by Material Type

Year	Ore	Waste	Total
-2		83,515	83,515
-1		226,832	226,832
1	990,000	83,486	1,073,486
2	990,000	83,720	1,073,720
3	990,000	63,183	1,053,183
4	990,000	52,480	1,042,480
5	990,000	57,452	1,047,452
6	990,000	43,329	1,033,329
7	990,000	11,932	1,001,932
8	990,000	20,108	1,010,108
9	990,000	19,887	1,009,887
10	207,981		207,981
Total	**9,117,981**	**745,924**	**9,863,905**

Personnel requirement estimates are based on the mine production rate and mine schedule. A mining contractor will begin work in the pre-production development stage to allow time for the Owner to recruit staff for the project. The contractor will continue mine access development during production.

Underground staffing requirements peak at 54 personnel during full production, including 9 mine operating and 5 mine maintenance salaried dayshift personnel, 32 shift technical staff, and 8 shift supervisors. Underground hourly labour requirements peak at 312 in Year 5 during full production, including 248 mine operating and 64 mine maintenance hourly personnel. The personnel requirements do not include the labour required for access development performed by the contractor.

1.3.6 MINE SERVICES

A two-bay sump will be located at the bottom of the mine and will be constructed to allow suspended solids to settle out of the ground water before pumping. The sump will be equipped with four high-pressure pumps: two working and two on stand-by. A 300 mm (12″) diameter steel dewatering pipe will be installed in the main access decline to pump water from the sump to the final tailing pump box on surface.

Industrial-quality water will be distributed in 4″ and 2″ diameter pipelines throughout the underground workings for drilling equipment, dust suppression, and fire fighting.

The major electrical power consumption in the mine will be from the main and auxiliary ventilation fans, drilling equipment, and mine dewatering pumps

A high voltage cable will enter the mine via the main access decline and will be distributed from the main underground substation via boreholes to electrical sub-stations located on each sublevel. High voltage power will be reduced to 600 V at electrical sub-stations. All power will be three-phase; lighting and convenience receptacles will be single phase 127 kV power.

A leaky feeder communication system will be installed throughout the mine. The system will interface with the surface communication system. It will be also used for centralized blasting. Telephones will be located at key infrastructure locations such as the underground electrical sub-stations, refuge areas, lunchrooms, and pumping stations. Key personnel and mobile equipment operators will be supplied with an underground radio.

The mobile drilling equipment such as jumbos, rockbolters, and scissor lifts with ammonium nitrate and fuel oil (ANFO) loaders will be equipped with their own compressors. No reticulated compressed air system will be required. Six portable compressors will be used to satisfy compressed air consumption for miscellaneous underground operations.

Explosives will be stored on surface in permanent magazines. Detonation supplies (non-electric [NONEL] and electrical caps, detonating cords, etc.) will be stored in a separate magazine on surface.

The underground mobile equipment has an estimated average fuel consumption rate of approximately 8,556 L/d during the production period. Haulage trucks and all auxiliary vehicles will be fuelled at fuel stations on surface. The fuel/lube cassette will be used for the fuelling/lubing of LHDs and face equipment.

The personnel carriers will be used to shuttle employees from the surface to the underground workings and back during shift changes. Supervisors, engineers, geologists, and surveyors will use diesel-powered trucks as transportation underground. Mechanics and electricians will use the mechanics' truck and maintenance service vehicles.

A boom deck with a 10-t crane will be used to move supplies, drill parts, and other consumables from surface to active underground workings.

A mine service crew will perform mine maintenance and construction work, ground support control and scaling, mine dewatering, and safety work.

Mobile underground equipment will be maintained in a mechanical shop located on the surface outside of main ramp access portal. A small underground maintenance shop with an overhead crane will also be constructed underground to provide maintenance for drilling equipment. A mechanics truck will be used to perform emergency repairs underground. Major rebuild work will be conducted off site.

1.3.7 Development Schedule

The mine development is divided into two periods: pre-production development and ongoing development.

The pre-production development period runs from the start of the project to when the first ore is fed to the process plant. Pre-production development will be scheduled to:

- provide access for trackless equipment
- provide ventilation and emergency egress
- establish ore and waste handling systems
- install mining services (backfill distribution, power distribution, communications, explosives storage, fuel storage and distribution, water supply, mine dewatering)
- provide sufficient level development in advance of start-up to develop sufficient ore reserves to support the mine production rate.

All underground pre-production development will be done by contractor with use of a contractor’s equipment, personnel, and supervision. A 130 m per month advance rate was assumed for a jumbo crew developing a 4.5 m wide by 4.5 m high heading, and 90 m per month for a raiseboring crew to drill a pilot hole and ream it to the 4.0 m diameter.

Underground infrastructure development, such as dewatering sumps, maintenance shop, and explosives storage, will be completed prior to production.

It is estimated that pre-production development will be completed in two years. Ore development is not included in the development schedule as it will be part of ore production.

Ongoing sustaining development will continue to be performed by a contractor during the production stage. The contractor will demobilize from the site in Year 9 when all main access development is completed.

Table 1.5 Mine Development Schedule

Production Year	Unit	Pre-production	Year										Total
			1	2	3	4	5	6	7	8	9	10	
Annual Metres (Horizontal)	m	5,420	1,497	1,437	1,132	950	1,040	765	216	364	360	0	13,181
Annual Metres (Vertical)	m	462	45	214	37	0	0	61	0	0	0	0	819
Total Development	**m**	**5,882**	**1,542**	**1,651**	**1,169**	**950**	**1,040**	**826**	**216**	**364**	**360**	**0**	**14,000**

1.3.8 Production Schedule

The annual ore production rate of 990,000 t (including ore from development and stopes) was scheduled based on 330 mine operating days per year with three 8-h shifts.

Criteria for scheduling production included targeting the mining blocks with higher grade ore in the early stages of mine life in order to improve project economics. The production sequence of the mining blocks will be from the top down. The number of mining blocks in production will vary from 8 to 10 in most production years. On average, there will be five stopes in production for drift-and-fill mining and four in production for cut-and-fill. The only room-and-pillar block will be mined in Year 9.

Table 1.6 Production Schedule

	Unit	Year 1	2	3	4	5	6	7	8	9	10	Total
Operating Days per Year	d/a	330	330	330	330	330	330	330	330	330	70	
Mill Feed	t	990,000	990,000	990,000	990,000	990,000	990,000	990,000	990,000	990,000	207,981	**9,117,981**
Grade												
Ag	g/t	214	217	217	205	183	182	177	167	148	137	**189**
Au	g/t	0.171	0.169	0.158	0.157	0.150	0.157	0.151	0.138	0.079	0.076	**0.146**
Pb	%	0.194	0.194	0.146	0.148	0.120	0.189	0.235	0.242	0.263	0.372	**0.196**
Zn	%	0.584	0.614	0.506	0.541	0.483	0.487	0.615	0.595	0.637	0.709	**0.566**

1.4 MINERAL PROCESSING AND METALLURGICAL TESTING

Four main metallurgical testing programs were carried out on the multiple metal (silver/lead/zinc) mineralization samples from the Fuwan silver deposit in Guangdong province, China. Samples from different drill holes were composited for the metallurgical testing programs. The testwork includes ore hardness determination, mineralogical determination, flotation concentration, gravity separation, hydro-metallurgical process, and ancillary tests including settling tests and acid base accounting (ABA) tests.

The dominant sulphide minerals in the mineralization are: pyrite, sphalerite, galena, argentiferous tennantitete-trahedrite, miargyrite, proustite-pyrargyrite, marcasite, native gold, bournonite, stephanite, chalcopyrite, and covellite.

The flotation tests included open batch flotation condition optimization tests, locked cycle tests, and variability tests. The tests indicated that the mineralization responded well to conventional differential flotation: silver-lead flotation followed by zinc flotation. Although silver hydrometallurgical extraction was high when the head samples or the concentrate samples were pretreated by roasting and ultrafine regrinding, the hydrometallurgical processes had not been considered in the study due to high operating costs and potential environment issues.

A 3,000 t/d process plant has been designed for the Fuwan Project to process silver bearing lead and zinc sulphide mineralization. The deposit consists of eight major mineralization zones. The main value metals in the mineralization are silver, lead, zinc, and gold. The process plant will operate 330 d/a at an annual process rate of 990,000 t/d and three shifts per day. Overall process plant availability will be approximately 90%.

The run-of-mine (ROM) from the underground mine will be crushed by an 800 mm by 1,100 mm jaw crusher to 80% passing 150 mm, and then ground to 80% passing 100 µm in a semi-autogenous grinding (SAG, 5.5 m Dia x 3.0 m EGL, 1,250 kW)-ball mill (3.96 Dia x 6.56 L, 1,650 kW)-pebble crushing circuit (SABC). The silver, lead, and zinc minerals will be recovered by a conventional differential flotation process:

- silver-lead bulk rougher flotation followed by zinc rougher flotation
- the silver-lead rougher flotation concentrate will be reground and subject to three stages of cleaner flotation
- the zinc rougher flotation concentrate will be upgraded by three stages of cleaner flotation as well without regrinding.

The tailings produced from the zinc rougher scavenger flotation circuit will be sent to the tailings management facility (TMF) for the storage and to the underground mine for hydraulic backfilling. The produced silver-lead concentrate and zinc concentrate will be thickened and then pressure filtered separately prior to being transported to smelters. Depending on the lead head grade, the silver-lead concentrate may be further processed to produce a silver concentrate and a lead-silver concentrate.

The average dry concentrate production is forecast to be as follows:

- silver-lead concentrate – 15,900 t/a, including:
  - 154,700 kg/a (4,975,000 oz/a) silver
  - 1,600 t/a lead
- zinc concentrate – 9,300 t/a average, including:
  - 4,700 t/a zinc
  - 15,400 kg/a (495,400 oz/a) silver.

1.5 TAILINGS MANAGEMENT FACILITY

The Fuwan Silver project includes development of a new proposed land based TMF to store up to 2.6 M m^3 of the tailings. The tailings will be the fine fraction classified from the flotation tailings. The TMF will be developed in two stages:

- Stage 1 Facility - capable of storing initial 8.3 years of tailings deposition through three dam raises; and,
- Stage 2 – Final Facility capable of storing additional 0.9 years of tailings deposition by either raising the Stage 1 Facility or on-land storage in a separate facility.

Current cost estimate assumes that raising the Stage 1 TMF Dam (subject design) to accommodate additional 0.9 years of tailings deposition is feasible. However, this is to be confirmed by subsequent geotechnical and hydrogeological investigations.

Essentially the TMF Dam will be a 56 m high earth/rockfill structure with a 6 m wide crest and composite HFPE /clay core lining (Zone 1 / Zone 2) on the upstream slope. The HDPE membrane will be protected by woven bags filled with tailings (Zone 1).

The dam will be constructed in three stages:

- Stage 1 (3.1 years storage capacity) will be 38 m high with crest at El. 61 m.
- Stage 2 (2.7 years storage capacity) will add additional 10 m bringing the dam crest to El. 71 m.
- Stage 3 (2.5 years of storage capacity) will add another 8 m for the final crest at El. 79 m.

Storm water around the TMF will be managed using the following structures:

- Perimeter diversion ditch
- Decant tower and pipe.

The subject TMF designs have been developed in between the prefeasibility and feasibility levels. Detailed geotechnical engineering analyses have not been completed and this may have a potential impact on the current design and cost estimate accuracy because of potential design modifications to be developed when the results of geotechnical and hydrogeological investigations and laboratory testing have become available. It is recommended that the geotechnical engineering analyses be conducted to confirm the design before next phase engineering.

It is recommended identifying the location for storing the tailings produced during the rest of 0.9-year operation. The potential use of the tailings for making bricks for local infrastructure requirements should be further studied and confirmed.

1.6 INFRASTRUCTURE AND ANCILLARY FACILITIES

The project site is close to the Fuwan town, which has well developed paved village-level road network. The town is accessible by paved public highways to Guangzhou and other major cities. The haulage distance between the mine site and the Shanshui railway station, which connects the main stations, Guangzhou station and Zhanjiang station, is approximately 26 km. The deposit is adjacent to the Xijiang river which is accessible to international waterway in the South China Sea via the Zhujiang river.

Electrical power, water, telephone service, and supplies are available in Fuwan town.

The proposed minesite is large enough to accommodate proposed processing facilities, surface service facilities, waste rock storage areas, as well as approximately 8.3-year tailing surface storage pond. The surface service facilities will administration buildings, workshop, explosive magazine, power and water supply facilities, backfill station, waste water treatment facility and haulage road system.

All buildings of the project will be new ones and be built according to the Chinese construction codes.

Power to the project will be provided via an existing 110 kV utility substation located in Fuwan town, approximately 4 km from the mine. NERIN and Minco have contacted with the Fushan Power Supply Company of the South Grid and confirmed that the Fushan substation has a capacity to provide power to the Fuwan mining project.

This substation presently has a single incoming transmission line and will provide a single 35 kV power line to the mining project. The external 35 kV power line will be provided by the electrical utility to the mine site. At the mine, a step-down substation

(35 kV /10 kV) will be established consisting of equipment and facilities necessary to service the connected mine loads.

1.7 CAPITAL COST ESTIMATE

This estimate has been completed partially by NERIN and partially by Wardrop. The majority of the information used in the estimate is based on the quantities and pricing provided by NERIN to Wardrop on March 28, 2009 and additional information and clarifications via email between April 1, 2009 and April 8, 2009. .

NERIN indicated that its estimate has an accuracy range of ±25%. The estimate has sufficient detail to provide a suitable basis for controlling the Engineering, Procurement, and Construction Management (EPCM) phase of the project.

Table 1.7 provides a summary of capital costs for the Fuwan Project.

Table 1.7 Summary of Project Capital Costs

Area	Cost (US$)
Direct Works	
A – Mining (Wardrop)	21,636,951
B – Primary Crushing	659,816
C – Crushed Ore Stockpile and Reclaim	305,324
D – Secondary and Tertiary Crushing	51,736
E – Grinding, Flotation, Dewatering, Reagents & Service	9,139,827
F – Tailings Disposal Facilities	4,249,774
G – Plant Site, Infrastructure & Ancillary Facilities	8,626,643
H – Temporary Services	35,323
L – Site/Plant Mobile Equipment	1,190,204
N – Power Lines (Included in G1 – Power Supply)	0
Direct Works Subtotal	**45,895,598**
Indirects	
X – Project Indirects	13,330,282
Y1 – Land Acquisition	2,120,000
Y1 – Owner's Costs	5,663,442
Z – Contingency	6,050,500
Indirects Subtotal	**27,164,224**
TOTAL PROJECT	**US$73,059,822**

1.8 OPERATING COST ESTIMATE

The operating cost estimates are based on a process rate of 990,000 t of ore annually or 3,000 t/d of ore. All operating costs are shown in US$, unless otherwise specified.

Mining	$18.01/t
Processing	$9.90/t
Tailings	$1.13/t
G&A	$4.78/t
Surface Services	$0.60/t
Total	**$34.42/t**

The exchange rate for US and Canadian dollars to Chinese currency is US$1.00 = ¥6.82 = Cdn$0.82. Mine operating costs are shown in Table 1.8.

Table 1.8 Mine Operating Cost Summary – LOM

	Cost
Total Mine Operating Cost	**$164,234,279**
Average per Tonne	$18.01/t
Labour Cost	**$38,124,300**
Average per Tonne	$4.18/t
Mining Cost without Labour	**$126,109,979**
Average per Tonne	$13.83/t

On average, the annual process operating cost is estimated to be approximately $9.80 M or $9.90/t milled. The estimated process operating costs are in Table 1.9.

Table 1.9 Summary of Process Operating Costs

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Ore)
Labour			
Operating Staff	10	354,240	0.358
Operating Labour	46	427,680	0.432
Maintenance	46	469,440	0.474
Metallurgical Laboratory	3	38,160	0.039
Assay Laboratory	13	131,760	0.133
Sub-total Labour	**118**	**1,421,280**	**1.436**
Major Consumables			
Metal Consumables		2,347,140	2.371
Reagent Consumables		1,224,780	1.237

table continues…

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Ore)
Supplies			
Maintenance Supplies		597,000	0.603
Operating Supplies		125,000	0.126
Power Supply		4,085,866	4.127
Sub-total Supplies		**8,379,787**	**8.464**
Total Process	**118**	**9,801,067**	**9.900**

The average tailings operating cost is estimated to be $1.13/t milled

General and administrative (G&A) costs are the costs that do not relate directly to the mining or processing operating costs. The G&A costs are estimated at approximately $4.73 M/a or $4.78/t milled. The estimated personnel requirement is 61 persons, including supervision and services. The site service cost is estimated at $0.60/t milled or about $594,000/a.

1.9 FINANCIAL ANALYSIS

An economic evaluation of the Fuwan Project was prepared by Wardrop based on a pre-tax financial model. For the 9-year mine life and 9.1 Mt reserve, the following pre-tax financial parameters were calculated:

- 33.2% IRR
- 2.3 years payback on $73.1 M capital
- $US111.5 M net present value (NPV) at an 6% discount value.

The base case prices are the 3-year historical average price from the LME as at May 31, 2009:

- Silver – US$13.57/oz
- Gold – US$767.72/oz
- Zinc – US$1.18/lb
- Lead – US$0.91/oz.

Sensitivity analyses were carried out to evaluate the project economics for 2-year historical average metal prices (upside case) and the Energy and Metals Consensus Forecast (EMCF) published by Consensus Economics Inc. (downside case).

The analyses are presented graphically as financial outcomes in terms of NPV and IRR in Figure 1.2 and Figure 1.3. The project NPV (6% discount) is most sensitive to silver price and, in decreasing order: operating cost, capital cost, zinc price, gold price and lead price.

Figure 1.2 NPV Sensitivity Analysis

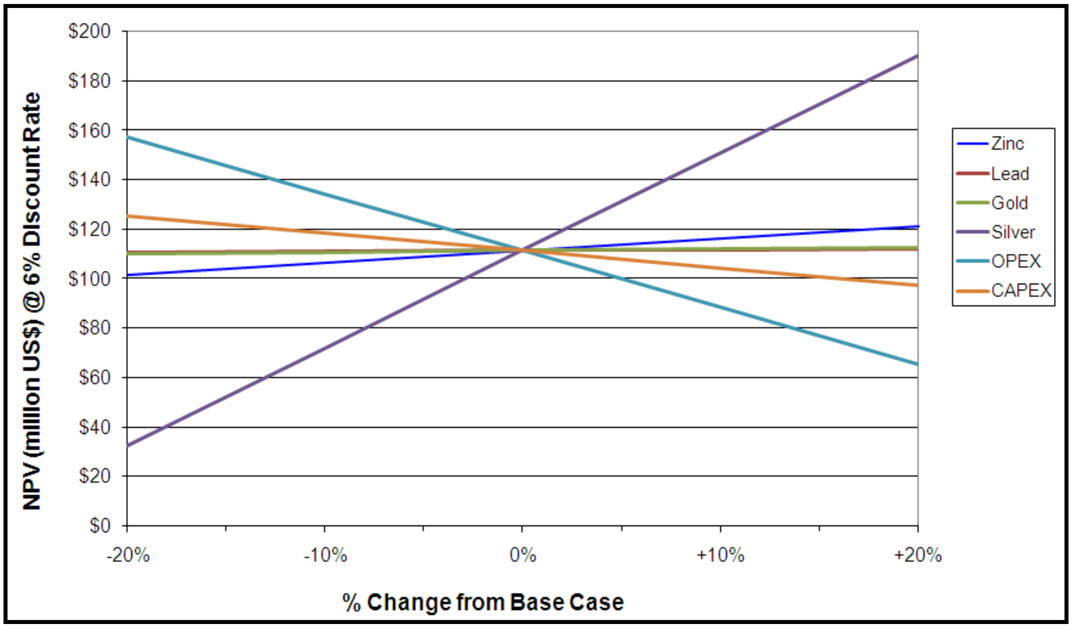
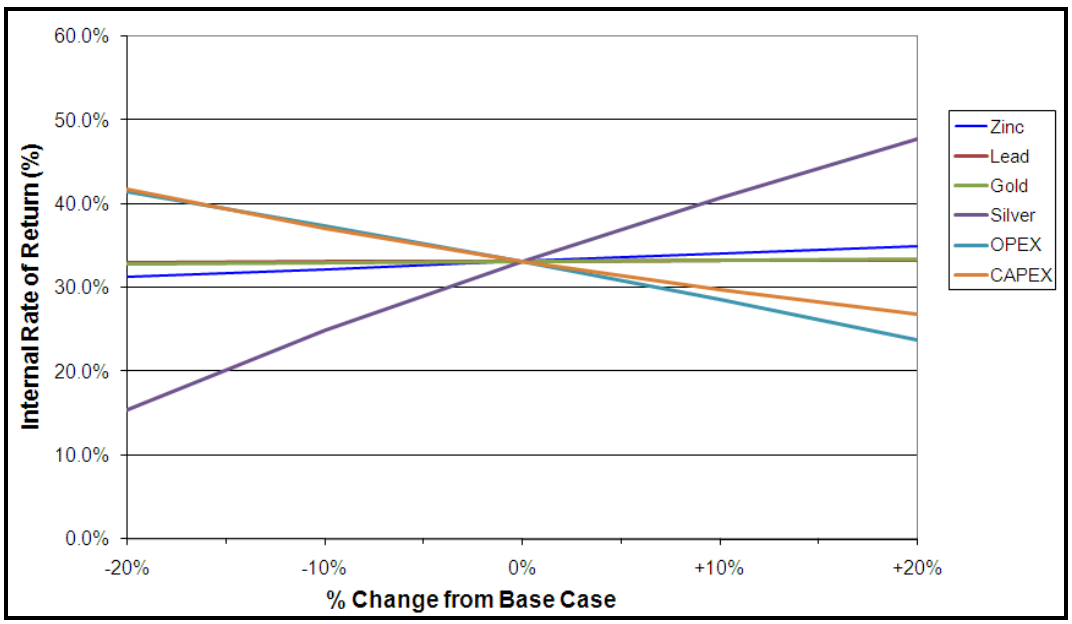


Figure 1.3 IRR Sensitivity Analysis



1.10 ENVIRONMENTAL

The detailed environmental assessment was completed by ERM and is presented in Appendix K

1.10.1 BACKGROUND

At the time of writing the Minco Silver Project ESHIA Report, project design was at the Feasibility Stage and hence some mine design details were not available to the ESHIA team and others were subject to change based on the evolving understanding of the geometry and grade distribution of the ore body (and hence mine plan) and technical issues relating to ore processing and site facilities' configuration. There is therefore, some uncertainty with respect some ESHIA findings and it is likely that further baseline investigations (as recommended in the ESHIA Report) and continuing work on the mine design will necessitate future revision of the ESHIA Report, likely in the form of an addendum, or of the Environmental, Socio-Economic and (Community) Health Management Plan (ESHMP).

1.10.2 PROJECT SETTING

The mine site area is typified by commercial plantation and secondary re-growth forest with some grassland areas. Numerous fish ponds are also located close to the mining and associated surface facility areas, the nearest of which is the Nankeng Reservoir, southeast of the TSF (Figure 1.4). Plantation forests and fish ponds represent primary and secondary income sources, respectively, for local communities. There are seven villages within one kilometer of the site as depicted in Figure 1.5.

Figure 1.4 Land Uses

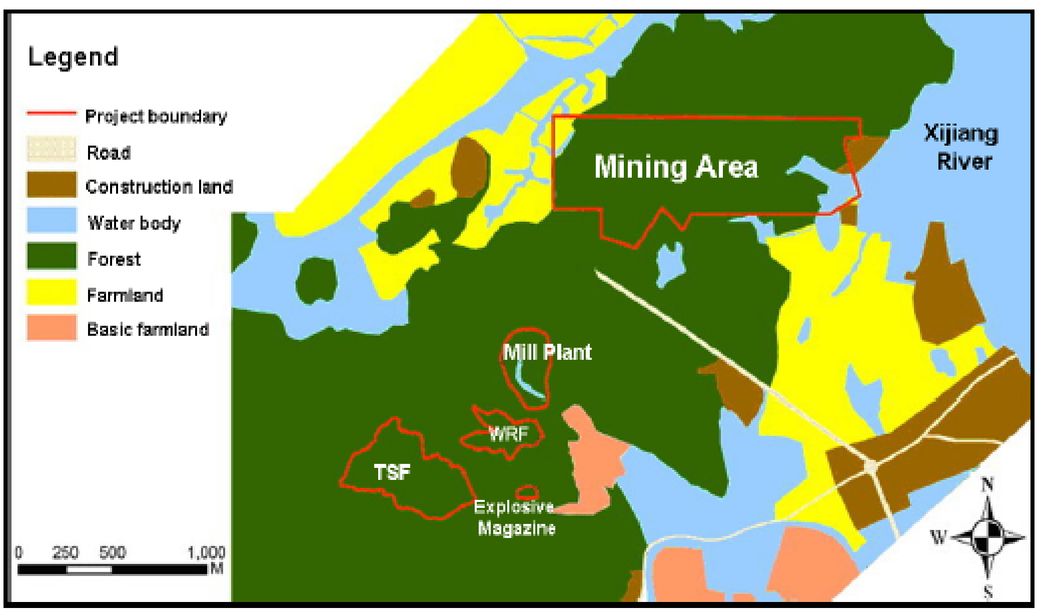
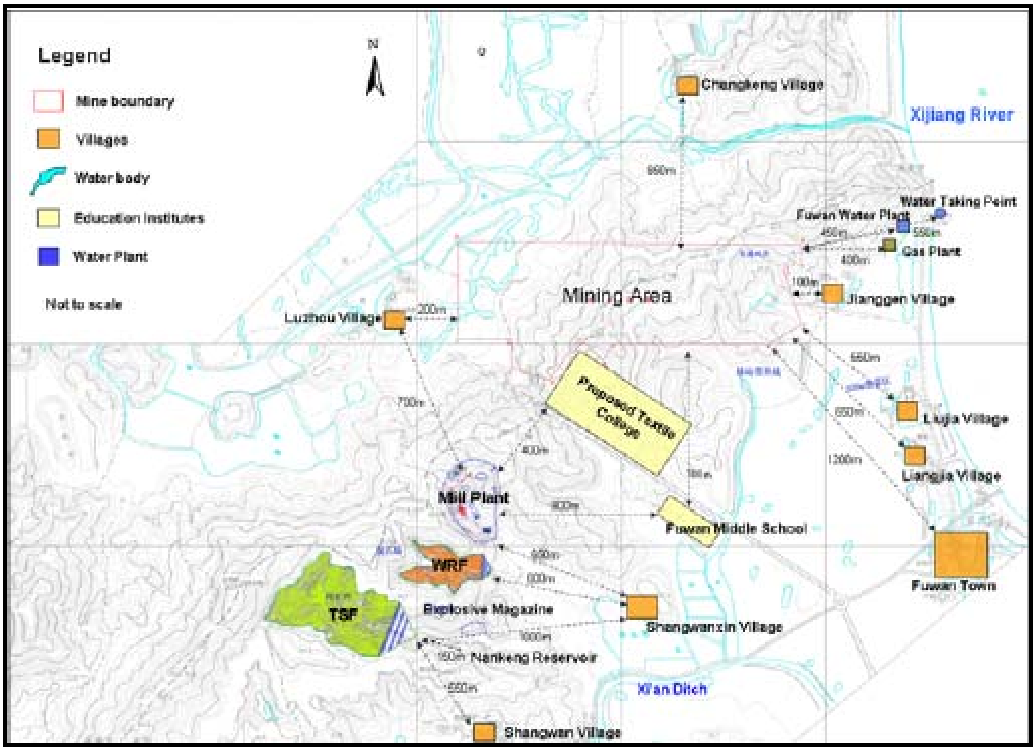


Figure 1.5 Nearby Villages



1.10.3 ESHIA FINDINGS

The ESHIA process assessed the project for all phases of its life cycle namely, development, operations and decommissioning. Broadly, the project has been assessed to not result in significant environmental, socio-economic or community health impacts assuming that industry best practice is implemented during execution and that additional control measures recommended within the ESHIA are satisfactorily implemented during all project phases.

The only issues where statutory limits have been predicted to be exceeded are in relation to dust and transport vehicle night time noise emissions at Shangwanxin Village. These impacts can however, be adequately mitigated by wetting down of the access road during dry and windy conditions and night time prohibition of transportation movement along the access road.

Notwithstanding the above, there are some aspects of the mine design and proposed development for which further investigation is considered warranted to be able to fully understand the environmental, socio-economic and (community) health issues and to confirm that there is no significant risk to receptors. These are summarized in the following sections.

Mine Blasting

The area to be mined is in close proximity to Luzhou and Jianggen Villages and the proposed Textile College site. Underground blasting in areas close to these receptors may result in plumb vibration levels that cause shaking of existing buildings or buildings that may be erected in the near future (i.e. within the college site). It is recommended that this risk be further evaluated and a blasting plan developed that prescribes and limits the weight of explosives, the number of holes to be blasted in a single shot and the time delay between blast shots to ensure that no adverse effects are caused.

Waste Rock and Tailings Storage Facilities

Laboratory tests have demonstrated that tailings and waste rock have traces of heavy metals and have a low potential to generate acid drainage.

A geotechnical survey of the TSF and WSF areas is yet to be conducted. Geotechnical survey data from the mining area suggests however, that the permeability of soil and rock in the general area is highly variable. There is therefore, some uncertainty regarding whether groundwater resources would be at risk from any leached metals or acid drainage from the TSF and WSF.

It is recommended that a geotechnical survey be undertaken to determine the permeability of TSF and WSF basement strata and, if found to be permeable, that natural (e.g. compacted clay) liners be introduced. It is also recommended that groundwater monitoring wells are installed down hydraulic gradient of the facilities and that these be sampled twice-yearly to confirm whether or not leaching of metals into groundwater or acid drainage is occurring. These monitoring wells can also be used during decommissioning.

Groundwater Drawdown

Groundwater that enters the mine void will be collected in a series of sumps and will be pumped to the surface for treatment and subsequent re-use in the process plant or disposal to the Xi’an Ditch.

The project geotechnical report states that maximum groundwater drawdown depth will be 283.83 m and that the permeability coefficient is 0.6815 m/day. The affected area will therefore, have a diameter of 2,343 m. Groundwater drawdown may result in surface subsidence, cave-ins or fracturing.

Existing groundwater wells within Shanwanxin and Jianggen Villages are within the predicted groundwater drawdown area and hence, groundwater availability may be affected by drawdown. As tap water has been provided to these villages, their reliance on the groundwater wells for potable water has reduced. Fish ponds in Shangwanxin Village are however, recharged using groundwater and hence may be affected if insufficient groundwater is available due to drawdown.

It is recommended that additional investigations into groundwater drawdown be conducted including a water balance study that assesses recharge rates against predicted draw down rates. The identified potential effects of drawdown should be further quantified where possible.

Geological Hazards –Surface Cave-In

Geological hazards in the mining area include landslides and surface cave-ins. A total of 19 sites where geological hazards have occurred have been identified including eight landslide sites and 11 cave-in sites. Among these, one landslide site and two collapse sites are defined as medium-severity and are in an unstable state. The three sites are respectively located near the Fuwan Water Plant, Gaoming-Gaoyao road and the mouth of the valley of the proposed Waste Rock Facility (WRF).

While the progressive backfilling of mine voids will assist in maintaining ground stability, it is recommended that additional work be undertaken to better understand the geotechnical state of ground above the proposed underground mine prior to the commencement of underground mining activities. The geo-technical survey should be aimed at identifying areas that may be prone to subsidence or cave-in and to determine what third party properties would be at risk in such a scenario.

1.11 Execution Plan

Minco continues to complete the remaining land acquisition and permitting requirements with the objective to commence construction activities by the first half of 2010. A 20 – 24 month construction schedule is envisaged with initial production anticipated for mid 2012. Refer to Section 18.6.

2.0 INTRODUCTION

The Fuwan silver-gold-lead-zinc deposit is owned by Minco and is located in Guangdong Province, southern China, about 45 km southwest of the provincial capital Guangzhou. Between 2005 and 2008, Minco explored the deposit with 422 drill holes with an aggregate length of about 96,000 m.

In November 2007, SRK Consulting Canada Inc. (SRK) completed a PEA and in May 2008, Changsha Engineering & Research Institute of Nonferrous Metallurgy (CNIF) completed a Chinese Pre-feasibility Study of the Fuwan deposit.

Minco has retained Wardrop, A Tetra Tech Company (Wardrop) to produce a Feasibility Study of the Fuwan property that is compliant with the reporting standards of National Instrument 43-101 (NI 43-101). The purpose of this report is to provide the technical basis of determining the feasibility of developing the Fuwan deposit.

Wardrop has followed standard professional procedures in preparing the contents of this Feasibility Study. Data used in this report has been verified where possible and Wardrop has no reason to believe that the data was not collected in a professional manner.

The principal consultants utilized by Minco in the preparation of this Fuwan Feasibility Study are as follows:

- Wardrop – mining, processing, capital cost (mining) and financial analysis
- P&E Mining Consultants Inc. – geology and resource estimation
- Environmental Resources Management (ERM) – environmental
- China Nerin Engineering Co. Ltd (NERIN)/Wardrop – infrastructure, overall site water management, hydrogeology, tailings and waste rock disposal, and capital cost (excluding mining).

3.0 RELIANCE UPON OTHER EXPERTS

Wardrop has relied upon others for information in this report. Information from third-party sources is referenced. Wardrop used information from these reports under the assumption that this information is accurate.

Data provided to Wardrop is listed in Section 21.0 (References) and the sections referenced are listed below:

- Section 18.4 by Environmental Resources Management (ERM)
- Sections 18.1.2, 18. 2, 18.3, 18.7 by China Nerin Engineering Co. Ltd (NERIN) and reviewed or reproduced by Wardrop.

4.0 PROPERTY DESCRIPTION AND LOCATION

This section has been excerpted, with modifications, from Puritch et al., 2007.

The Fuwan property is located in Gaoming City County, Guangdong Province, southern China, approximately 45 km southwest of Guangzhou, the capital city of Guangdong and 2 km northwest of the town of Fuwan (population 30,000) (Figure 4.1, Figure 4.2).

The Fuwan property is contained within the Luoke-Jilinggang Exploration Permit (T01120080420000336), which is 76.62 km^2 in area and is defined by the following geographic coordinates:

- northeast corner: 112°52'00" E, 23°03'00" N
- southeast corner: 112°52'00" E, 23°00'00" N
- southwest corner: 112°43'45" E, 23°00'00" N
- northwest corner: 112°43'45" E, 23°03'00" N.

This permit is in good standing until July 20, 2011.

China uses a map-based mineral title allocation system and therefore there are no physical survey markers on the boundaries of the permit.

Surface rights do not form part of the Exploration Permits and, for mining to occur, these rights will need to be secured.

On August 20, 2004, Minco, Minco Mining, Minco China, and Minco BVI entered into an assignment agreement (the “Assignment Agreement”) whereby Minco Mining, Minco BVI, and Minco China assigned to Minco their respective interests in each of the following:

- the preliminary Fuwan joint venture agreement dated April 16, 2004 and amended August 18, 2004 (the “Preliminary Fuwan JV Agreement”) between Minco BVI and the Guangdong Geological Exploration and Development Corporation (GGEDC)
- the right to earn the 51% interest in the silver mineralization to be acquired by Minco Mining pursuant to the Changkeng JV Agreement
- certain additional exploration permits identified by and to be acquired by Minco China, namely the Additional Permits (including a new permit in

respect of the Dadinggang property, for which an application has been made by Minco China to the Chinese governmental authorities).

In consideration for the assignment of the foregoing interests, Minco issued 14,000,000 common shares to Minco Mining.

Minco and GGEDC entered into a formal joint venture agreement dated September 28, 2004 (the "Fuwan JV Agreement"), which replaced and superseded the Preliminary Fuwan JV Agreement. The Fuwan JV Agreement provided for the establishment of a Sino-foreign joint venture with limited liability to be known as "Guangdong Minco-Nanling Mining Co., Ltd." (the "Fuwan JV"), which would serve as the vehicle through which the Fuwan JV would conduct further exploration and assess the economic viability of developing certain silver and polymetallic resources (other than gold).

In particular, the Fuwan JV Agreement contemplated the acquisition by the Fuwan JV of the Original Fuwan Silver Permit and the Additional Permits from the 757 Team. The Original Fuwan Silver Permit had previously been legally conferred to 757 Team on September 12, 2003, by the Guangdong Department of Lands and Resources.

The Fuwan JV Agreement provided for a total investment of 30 M renminbi (¥) (the "Fuwan Total Investment") and registered capital of ¥15 M. The Fuwan Total Investment was to be funded by the Company as to 70% and by GGEDC as to 30%.

The parties to the Fuwan JV Agreement also agreed that Minco would pay to 757 Team an acquisition payment in the amount of ¥1.5 M within 50 days of the Additional Permits being issued to Minco China.

On November 19, 2004, 757 Team and Minco China entered into an agreement (the "757 Transfer Agreement") pursuant to which 757 Team agreed to transfer and sell to Minco China the Original Fuwan Silver Permit for consideration of ¥10.33 M.

On November 19, 2004, 757 Team, GGEDC and Minco China entered into a confirmation agreement (the "Transfer Confirmation Agreement") which clarified that Minco China would transfer at cost the Original Fuwan Silver Permit to the Fuwan JV within one year after its receipt of the Original Fuwan Silver Permit pursuant to the 757 Transfer Agreement. The Transfer Confirmation Agreement also provided that any expenses incurred in connection with the transfer of the Original Fuwan Silver Permit would be borne by the Fuwan JV. On behalf of the Fuwan JV, Minco China also agreed to pre-pay ¥80,000 to the 757 Team as an appraisal fee in respect of the Original Fuwan Silver Permit.

On April 22, 2005, the application submitted by 757 Team and Minco China for the transfer of the Original Fuwan Silver Permit pursuant to the 757 Transfer Agreement was considered in accordance with all the state's requirements for a title transfer and approved by the Department of Land and Resources of Guangdong Province, thereby approving the transfer application.

On May 2, 2005, Minco Mining, Minco China, and Minco entered into a confirmation agreement (the "First Confirmation Agreement") pursuant to which, among other things, Minco China confirmed that it held the right to the Original Fuwan Silver Permit in trust for the Fuwan JV and that, upon the establishment of the Fuwan JV pursuant to the Fuwan JV Agreement and upon the written demand of the Fuwan JV, Minco China would transfer such permits to the Fuwan JV for no additional consideration. Minco Mining also agreed under the First Confirmation Agreement that it would ensure that Minco China remained a wholly-owned subsidiary of Minco Mining until such time as the permits were transferred to the Fuwan JV.

On January 10, 2006, Minco entered into a contract (the "Amending Contract") with GGEDC to amend the Fuwan JV Agreement. Pursuant to the Amending Contract, Minco and GGEDC agreed to not proceed with the establishment of the Fuwan JV. Rather, Minco agreed to be responsible for 100% of the exploration and development expenditures relating to the Fuwan Permits, including the entire ¥10.33 M purchase price for the Fuwan Silver Permit.

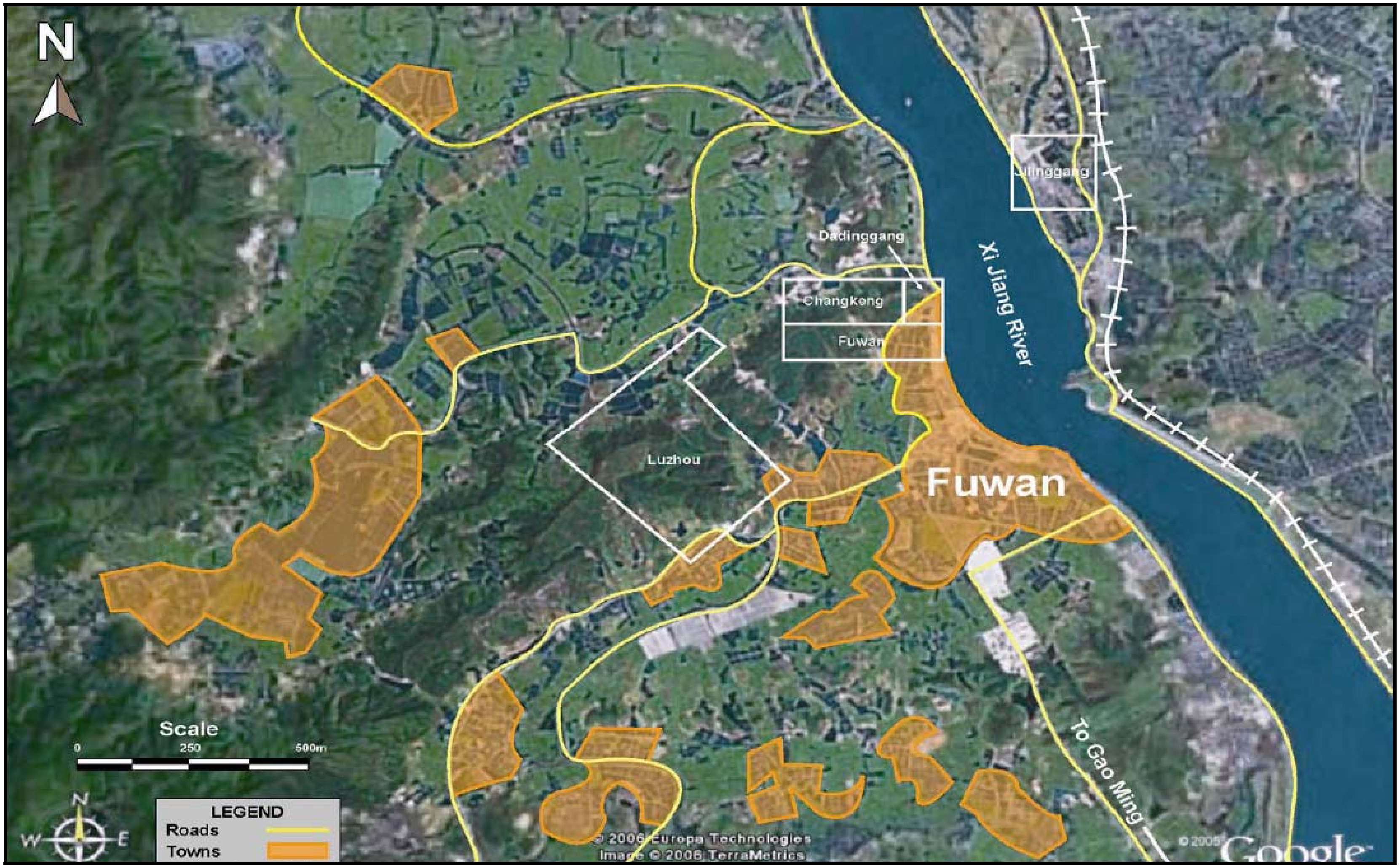
Pursuant to the Amending Contract, upon satisfaction of the Fuwan Purchase Price, Minco will hold, through Minco China, a 100% interest in the Fuwan Permits, subject to GGEDC retaining a 10% net profit interest in the properties subject to the Fuwan Permits.

On August 24, 2006, Minco, Minco China, and Minco Mining entered into a second confirmation agreement (the "Second Confirmation Agreement") pursuant to which the parties thereto confirmed, among other things, that Minco China holds the Fuwan Permits on behalf of and in trust for Minco and that Minco has the sole authority to direct Minco China in the future as to any transfer or other transaction relating to the Fuwan Permits. Minco Mining and Minco China agreed in the Second Confirmation Agreement not to transfer, sell, pledge, grant security interests in, or otherwise encumber, in any manner whatsoever, the Fuwan Permits. In addition, Minco Mining agreed, pursuant to the Second Confirmation Agreement, not to transfer or sell any of its ownership or equity interest in Minco China or encumber its interest in any way if any of the foregoing, individually or in combination, would have the effect of Minco Mining holding at any point in time less than, on an actual or a fully-diluted calculation basis, a 75% unencumbered ownership interest in Minco China. Likewise, Minco China agreed pursuant to the Second Confirmation Agreement not to enter into any agreement or grant any option or right for the purchase, sale, transfer or issuance of any ownership or equity interests in Minco China if any of the foregoing, individually or in combination, would have the effect of Minco Mining holding at any point in time less than, on an actual or a fully-diluted calculation basis, a 75% unencumbered ownership interest in Minco China.

There are no known environmental liabilities on the Fuwan property.

Figure 4.1 Fuwan Property Location Map

Figure 4.2 Fuwan/Changkeng Property Area

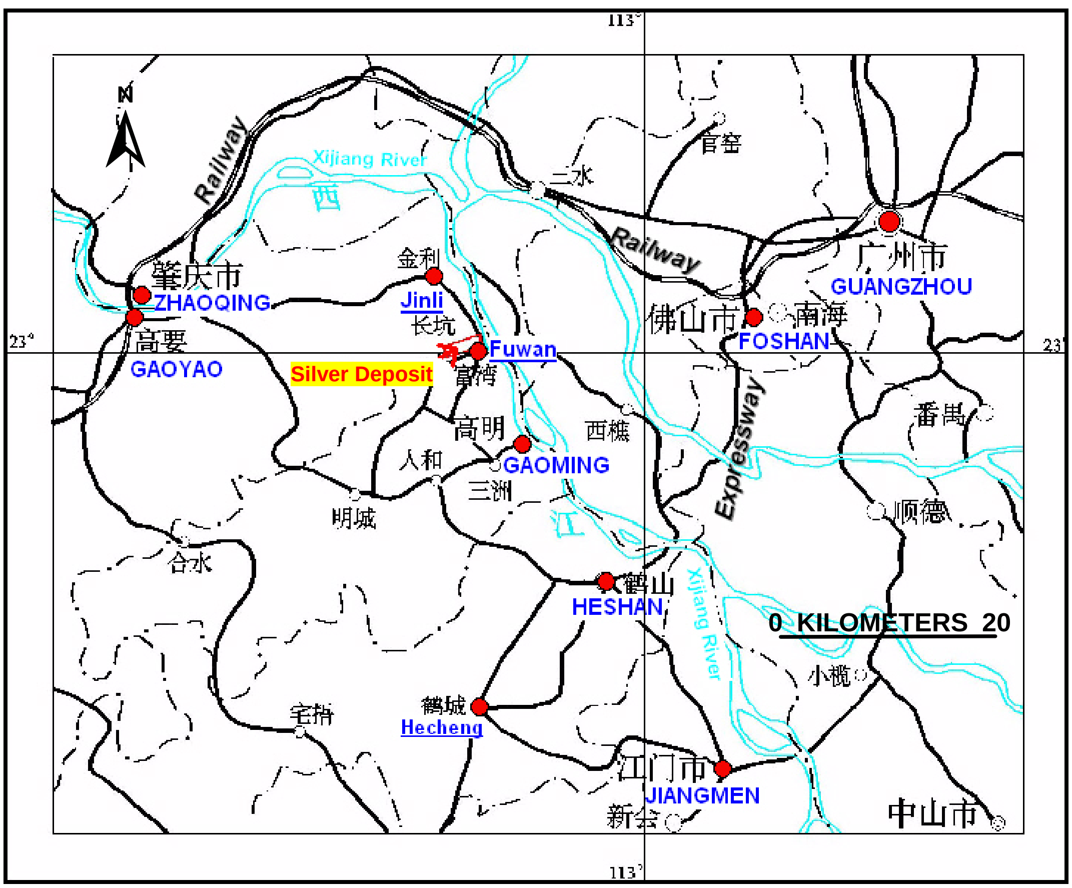


5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

This section has been excerpted, with modifications, from Puritch et al., 2007.

The Fuwan property is located approximately 45 km southwest of Guangzhou, the capital city of Guangdong province. Access to the property is via the Guangzhou-Zhuhai highway, which passes through Gaoming City. The property is located 2 km via gravel road northwest of the town of Fuwan (population 30,000). The town of Fuwan is well connected by paved highway and expressways to major cities, including Guangzhou (70 km highway distance), Gaoming (15 km), and Jiangmen (60 km), as shown in Figure 5.1. The Fuwan property is also accessible by water on the Xijiang River to major cities like Guangzhou, Zhaoqing and Jiangmen, as well as to international waterways in the South China Sea.

Figure 5.1 Local Area Map Showing Fuwan Property



The area is characterized by low hills, 60 to 90 metres above sea level (masl), with a maximum local elevation of 133.3 masl. Outcrop is scarce and most of the area is covered with 5 to 10 m of overburden.

The area is hot and humid with an annual average temperature of 21.5°C and an annual precipitation of about 1.7 m. Surface water is abundant in the valley areas due to irrigation and fish farming.

Like most of the coastal areas in southeast China, the property area is densely populated. Local residents are mainly engaged in farming and there are abundant rectangular aerated ponds for fish farming. General labour is readily available but labour more specialized in mining would need to be recruited and/or trained.

The town of Fuwan is located 2 km southeast of the Fuwan property along an unpaved road, which connects it to a major highway system. Electrical power, water, telephone service, and supplies are available in Fuwan. The property is large enough to accommodate potential tailings, waste disposal areas, and potential processing plant sites.

6.0 HISTORY

This section has been excerpted, with modifications, from Puritch et al., 2007.

6.1 PREVIOUS EXPLORATION

Since the establishment of the Peoples Republic of China (PRC) in 1949, mineral exploration has been undertaken at all scales by teams of government-employed geologists and engineers. Each team has been responsible for a certain region and within each team there were sub-teams with specific mandates such as geology, geochemistry, mineral deposit evaluation, diamond drilling, etc.

There is no historic record of mining in the property area before the discovery of gold within the Changkeng property, which Minco Gold has 51% interest, immediately to the north of the Fuwan property in early 1990. Illegal artisanal mining of oxidized gold mineralization began in 1991.

From 1994 until 2003, the Fuwan property was under the ownership of the Guangdong Department of Lands and Resources. In September 2003, land title was transferred to the 757 Team. In July 2005, the Fuwan Silver Permit was transferred to Minco China who holds it on behalf of Minco.

A brief history of recent exploration is as follows:

- **1959-1971:** Geological exploration for pyrite, coal, and uranium was carried out intermittently by different geological teams.
- **1986-1989:** The Regional Geological Survey Team of the Guangdong Bureau of Geological Exploration conducted a regional stream sediment sampling program at a 1:200,000 scale. Significant gold and silver geochemical anomalies were delineated in the Fuwan Silver Permit area. These anomalies were followed up with detailed soil sampling at a 1:50,000 scale, which demonstrated good potential for gold and silver mineralization in the area.
- **1990-1994:** The Fuwan silver deposit was discovered in 1990 during follow up of the 1:50,000 soil geochemical anomalies by the 757 Team. Detailed exploration was conducted at the Fuwan property and adjacent areas from 1990 to 1995.
- **1990:** The “Report on Reconnaissance Investigation of Gold Mineralization in Changkeng, Gaoyau County, Guangdong Province” was completed by the 757 Team.

- **1993-1995:** Prospecting of the Fuwan silver deposit was conducted by the 757 Team.

Sixteen holes were drilled on the Fuwan silver deposit, totalling 4,247 m.

There were many trenching programs undertaken on the property, as well as 2 holes drilled for the purposes of a metallurgical test on the Fuwan silver deposit. Geotechnical data were collected including core recovery, rock quality designation (RQD), and structural logging. Collar locations were surveyed using an EDM station with a survey accuracy of ±0.12 m.

6.2 HISTORICAL RESOURCE ESTIMATES

During the period 1993 to 1995, a total silver resource of about 5,100 t was estimated by the 757 Team for the Fuwan silver deposit (Category E, a regional estimation).

These resource calculations were done in 1995 before the application of NI 43-101. The Chinese classification system is not considered comparable to current Canadian Institute of Mining, Minerals, and Petroleum (CIM) definitions. As such, the resources are no longer considered relevant and have been replaced by the NI 43-101-compliant resource as reported in Section 17.0 of this report.

7.0 GEOLOGICAL SETTING

This section has been excerpted, with modifications, from Puritch et al., 2007.

7.1 REGIONAL GEOLOGY

The Fuwan silver deposit is located on the northwest margin of a triangular-shaped Upper Paleozoic sedimentary basin, near the junction of the northeast-trending Shizhou fault to the northwest, the east-trending Dashi fault to the south and the northwest-trending Xijiang fault to the northeast. There are known precious and base metal occurrences and deposits that occur predominantly along the margins of the basin.

The basin strata are comprised of two major sedimentary sequences: a lower, Upper Paleozoic-age siliceous and argillaceous carbonate sequence, and an overlying Mesozoic-age coal-bearing clastic sequence. On the northwest margin of the basin, these two units are separated by a low-angle fault. This fault zone hosts the known silver mineralization in the Fuwan area as well as to the southwest and northeast.

There are no outcrops of intrusive rocks in the immediate area of the Fuwan silver deposit. Late Mesozoic-age granites occur along the south margin of the Sanzhou basin.

7.2 PROPERTY GEOLOGY

Host rocks of the Fuwan silver deposit are comprised of Lower Carboniferous-age limestone and Upper Triassic-age terrestrial clastics.

The Lower Carboniferous Limestone sequence is comprised of three members:

- lower, neritic grey and dark-grey thickly-bedded bioclastic limestone
- middle terrestrial grey-whitish and reddish quartz sandstone intercalated with grey calcareous siltstone, mudstone, carbonaceous shale and coal
- an upper member comprised of grey and dark-grey medium to thickly bedded argillaceous limestone and mudstone, light-grey brecciated bioclastic limestone intercalated with yellowish silicified limestone and silty mudstone.

Some gold mineralization and most silver mineralization occur in brecciated bioclastic limestone.

The Upper Triassic clastic rocks are comprised of variegated sandstone, sandy conglomerate and conglomerate, dark-grey mudstone, carbonaceous mudstone, and siltstone.

A low-angle fault zone is developed along the contact between the Lower Carboniferous and overlying Triassic strata. The fault zone is from several metres to tens of metres thick and is occupied by lenticular zones of brecciated and silicified limestone and sandy conglomerate. The fault zone may have acted both as a conduit for mineralizing fluids and as the host structure for mineralization in the area. A set of subordinate faults parallel to the major fault occur in the limestone beneath the main fault. Silver mineralization also occurs in these footwall faults.

Within the main fault zone, the upper part of the Lower Carboniferous carbonate sequence and the lower part of the Upper Triassic clastic rocks are brecciated and mineralized with gold and silver. Most gold mineralization occurs in the Triassic clastic rocks while most of the silver mineralization occurs in the brecciated, siliceous fault zone that separates the two units and within parallel fractures in the underlying Lower Carboniferous carbonate sequence.

Typical alteration associated with the Fuwan silver deposits includes silicification, clay (mainly illite), barite, fluorite, carbonate, and pyrite. Alteration developed predominantly within the major fault zone between the Carboniferous limestone and Triassic clastic rocks and within second-order faults in the footwall carbonates. Silicification and sulphide mineralization are most closely associated with gold and silver mineralization.

8.0 DEPOSIT TYPES

This section has been excerpted, with modifications, from Puritch et al., 2007.

The Fuwan silver deposit is considered to belong to the class of sediment-hosted epithermal deposits and is characterized by vein and veinlet mineralization within zones of silicification. The predominant sulphide minerals are sphalerite and galena with lesser pyrite, and rare arsenopyrite, chalcopyrite, and bornite. Pyragyrite and freibergite are other important silver minerals in the deposit. The deposit is poor in gold (<0.2 ppm).

Eight zones of silver mineralization have been identified:

- Zone 1, lying entirely within the fault plane, contains a relatively large volume of silver mineralization, particularly in the west part.
- Zone 2, partially within the brecciated and silicified fault zone, contains the greatest volume of silver mineralization.
- Zone 3 occurs in the footwall of the main fault zone.
- Zones 4, 5, and 6 are situated entirely within the footwall along planar fractures in the limestone.
- Zone 7 is located in the Luzhou area, along strike to the southwest of the main Fuwan silver deposit.
- Zone 8 is located on the east side of the Xijiang River, along strike to the north east of the main Fuwan silver deposit.

Zones 7 and 8 are not included in the Fuwan resource estimate.

9.0 MINERALIZATION

This section has been excerpted, with modifications, from Puritch et al., 2007.

The mineralized zones at the Fuwan silver deposit have been divided into two types:

- Siliceous (silicified) material: this type of material is grey to dark grey in colour and mainly composed of secondary quartz, illite, argillaceous and carbonaceous material, and pyrite. Fractures and mariolitic cavities are highly developed.
- Calcareous-siliceous material (silicified limestone): this type of material is light grey to dark grey in colour and is composed of secondary quartz, residual limestone, calcite, and pyrite. Mineralization occurs in second-order faults in the footwall limestone of the contact zone.

10.0 EXPLORATION

Minco Mining began drill exploration of the Fuwan Deposit in 2005; however the most extensive phases of diamond drilling began in December 2005 and continued until 2008. No other exploration work apart from an IP geophysical program in 2008 was undertaken during the diamond drilling programs, details of which are covered in Section 11.0 of this report.

11.0 DRILLING

Up to May 2008, the Fuwan deposit area was tested by 422 drill holes with an aggregate length of 95,506 metres. Minco drilled 56,033 metres and the Chinese 757 Group drilled 39,473 metres. Drilling was completed on fifty three cross-sections (95W to 56E) on an 80 m x 80 m grid spacing with more than 60% of the deposit area in fill drilled at a spacing of 40 m by 40 m within the central portion of the deposit. The effective drill hole spacing on the 40 m sections was a diagonal 60 m x 60 m pattern.

Of the 422 drill holes, 231 totalling 59,341 metres were drilled within that portion of the deposit for which the Fuwan resource was estimated.

Details regarding the various drill programs at Fuwan can be found in the report titled "Technical Report and Updated Resource Estimate on the Fuwan Property, Guangdong Province, China" by P&E Mining Consultants Inc., dated December 2, 2007.

11.1 Final Results for Phase VI Drilling

The Phase VI drill program was finally completed in March 2008, with the last analyses received in May. Certain holes from Phase VI had been completed and were included in the last technical report titled, "Technical Report and Updated Resource Estimate on the Fuwan Property, Guangdong Province, China", dated January 25, 2008, and authored by P&E Mining Consultants Inc. The holes reported in the above cited report were FW0127 to FW0147. The goal of Phase VI was predominantly step-out drilling to expand the overall resource, with a minor component of in-fill drilling. The total number of metres drilled in Phase VI was 19,898 metres in 72 holes (Figure 11.1).

Results are presented in Table 11.1.

Table 11.1 Significant Mineralized Intersections for the Completion of Phase VI

Hole No.	From (m)	To (m)	Intercept (m)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
FW0100-1	176.22	177.22	1.00	0.63	55.00	0.02	0.01
	188.70	203.30	14.60	0.07	26.25	0.03	0.03
	270.40	271.40	1.00	0.03	63.00	0.04	0.07
	280.40	284.10	3.70	0.18	91.86	0.05	0.35
FW0135	85.18	111.87	26.69	0.09	87.50	0.13	0.47
FW0146	284.05	284.55	0.50	0.11	117.50	0.76	3.10
	292.85	293.85	1.00	0.05	227.50	0.40	0.54
	300.75	301.75	1.00	0.01	118.00	0.04	0.01
	173.05	174.05	1.00	0.15	232.50	0.77	2.05
	232.03	232.45	0.42	0.02	318.00	0.93	1.54
FW0151	223.30	224.10	0.80	0.08	101.50	0.24	0.24
FW0152	189.70	190.45	0.75	0.32	1264.00	2.63	5.64
	197.33	208.45	11.12	0.15	108.45	0.40	0.66
FW0153	186.80	187.80	1.00	0.11	150.00	0.05	0.33
	190.80	191.80	1.00	0.08	127.50	0.12	0.33
	195.20	196.20	1.00	0.06	105.00	0.06	0.09
	255.70	256.70	1.00	0.01	89.00	0.06	0.18
	263.81	264.96	1.15	0.02	101.00	0.15	0.32
FW0154	182.15	184.55	2.40	0.37	724.56	2.20	4.36
FW0155	296.40	297.43	1.03	3.29	275.50	3.00	4.72
FW0156	169.55	170.65	1.10	1.30	2142.00	0.58	1.33
	174.65	183.55	8.90	0.09	175.20	0.03	0.18
FW0159	183.95	185.62	1.67	0.03	251.13	0.09	0.12
FW0160	85.85	96.30	10.45	0.20	105.84	0.08	0.24
	98.30	99.15	0.85	0.21	94.50	0.16	0.62
	101.85	102.85	1.00	0.12	57.00	0.08	0.63
	127.02	127.52	0.50	0.21	379.50	1.49	0.15
	156.03	156.30	0.27	0.28	458.00	0.44	1.02
FW0161	160.70	161.60	0.90	0.08	133.50	0.11	0.49
FW0163	35.35	36.00	0.65	0.05	330.50	0.15	0.05
	47.60	48.40	0.80	0.16	366.50	0.09	5.18
	106.00	106.60	0.60	0.11	79.00	0.06	0.19
FW0164	92.35	93.20	0.85	0.06	168.00	0.14	0.40
	114.05	114.95	0.90	0.03	87.00	0.24	1.23
FW0165	195.03	196.03	1.00	0.15	427.00	0.37	0.97
FW0166	65.60	66.60	1.00	0.21	276.00	0.24	0.66
	69.60	70.60	1.00	0.40	308.00	0.28	0.85
FW0167	96.50	98.70	2.20	0.16	139.00	0.10	0.31
FW0168	28.57	46.00	17.43	0.29	185.35	0.07	0.20
FW0171	87.70	94.70	7.00	0.05	88.89	0.13	0.59
	107.10	108.80	1.70	0.22	76.41	0.09	0.42
FW0175	38.35	41.35	3.00	0.10	74.33	0.02	0.09
FW0176	73.87	76.87	3.00	0.06	98.17	0.02	0.10
	87.50	91.04	3.54	0.04	80.72	0.01	0.01
	129.30	140.74	11.44	0.04	89.50	0.07	0.13
	159.20	159.74	0.54	0.08	144.00	0.22	0.42
FW0177	139.30	141.30	2.00	0.37	1468.25	0.16	0.27
	144.80	145.90	1.10	0.29	429.50	0.22	1.32
	156.44	156.90	0.46	0.09	169.00	0.11	0.56
	158.80	159.25	0.45	0.18	328.50	0.18	0.47
	182.70	183.70	1.00	0.25	115.50	0.08	0.18
	185.31	187.00	1.69	0.25	485.79	0.41	1.31
	193.80	196.80	3.00	0.21	159.67	0.07	0.16

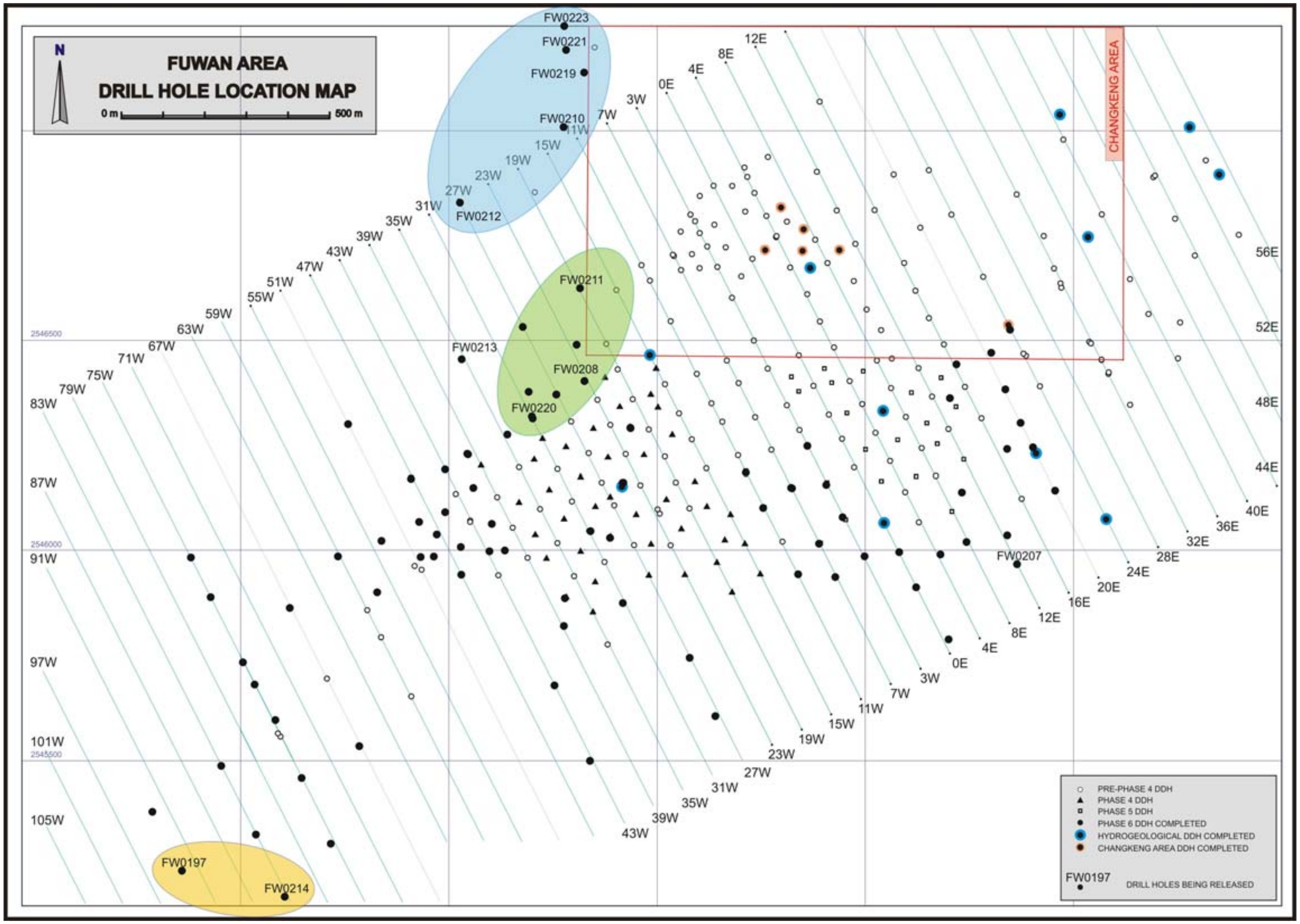
table continues...

Hole No.	From (m)	To (m)	Intercept (m)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
FW0178	188.15	189.95	1.80	0.11	187.65	0.04	0.28
	193.10	194.20	1.10	0.05	123.50	0.07	0.33
	280.80	281.20	0.40	0.02	361.50	0.25	0.11
	288.78	289.68	0.90	0.08	112.00	0.41	1.83
FW0179	210.10	225.30	15.20	0.29	474.12	0.10	0.19
FW0180	218.20	218.50	0.30	0.13	79.00	0.03	0.30
	222.82	226.36	3.54	0.09	218.54	0.13	0.69
	242.91	243.66	0.75	0.02	88.00	0.02	0.03
	251.64	253.80	2.16	0.04	185.42	0.04	0.05
	257.75	260.14	2.39	0.17	448.99	0.89	5.59
	261.99	262.79	0.80	0.12	57.00	0.13	0.20
FW0181	233.56	234.31	0.75	0.23	64.00	0.03	0.48
	235.80	236.70	0.90	0.05	66.00	0.04	0.05
	245.60	250.00	4.40	0.08	116.66	0.40	0.60
	267.75	267.97	0.22	0.10	76.00	0.21	0.35
	282.00	282.16	0.16	0.10	1999.50	1.46	0.04
	300.90	301.85	0.95	0.04	258.50	0.80	0.92
	304.10	305.15	1.05	0.19	135.52	0.47	0.94
	340.93	341.26	0.33	0.18	205.50	0.19	0.67
FW0183	189.50	195.50	6.00	0.64	304.50	0.17	0.32
	200.25	203.25	3.00	0.13	78.17	0.08	0.16
	198.80	199.30	0.50	0.03	343.00	0.58	0.86
	250.85	251.30	0.45	0.08	513.50	0.44	2.06
	256.60	257.25	0.65	0.16	628.00	1.01	3.02
	263.80	2664.20	2400.40	<0.01	142.00	0.01	0.30
	277.75	278.70	0.95	<0.01	54.00	0.06	0.12
	288.50	289.25	0.75	0.02	72.00	0.06	0.30
FW0186	25.35	27.35	2.00	0.39	117.25	0.02	0.02
	293.80	294.55	0.75	0.01	70.00	0.79	3.97
	85.60	99.90	14.30	0.02	39.65	0.01	0.03
	117.50	119.30	1.80	0.15	105.50	0.04	0.14
	128.95	131.25	2.30	0.21	103.50	0.04	2.62
	137.33	138.45	1.12	0.07	79.00	0.10	0.52
FW0158	130.58	132.90	2.32	0.03	211.46	0.46	0.78
	136.28	140.17	3.89	0.15	74.95	0.23	1.09
	161.90	165.21	3.31	0.03	202.72	0.21	0.41
	182.89	183.86	0.97	0.14	156.00	0.24	1.28
FW0172	111.40	143.95	32.55	0.59	183.67	0.11	0.79
FW0188	127.05	127.85	0.80	<0.01	58.00	0.00	0.01
	132.65	133.65	1.00	0.03	54.00	0.00	0.01
	143.90	144.16	0.26	0.08	494.00	0.00	0.03
	248.00	249.35	1.35	0.02	114.37	0.16	0.06
FW0189	98.05	99.05	1.00	<0.01	65.00	0.00	0.02
	103.05	107.05	4.00	0.00	50.25	0.00	0.02
FW0190	121.70	121.80	0.10	0.12	258.00	0.45	2.67
	130.20	130.35	0.15	0.06	315.00	0.41	2.02
FW0191	68.30	69.30	1.00	0.83	52.00	0.08	0.34
	152.10	152.80	0.70	0.09	125.00	0.36	1.43
FW0192	53.05	55.80	2.75	0.15	124.00	0.01	0.04
	66.30	72.10	5.80	0.15	89.56	0.02	0.04
	161.74	164.24	2.50	0.13	361.33	0.22	0.61
	169.24	171.20	1.96	0.26	388.78	1.29	3.26
	203.07	208.25	5.18	0.18	469.30	1.30	2.93
	215.40	215.72	0.32	0.14	934.00	0.45	1.49
FW0193	55.11	56.20	1.09	0.39	154.00	0.04	0.09
	72.83	73.17	0.34	<0.01	647.00	0.25	0.46

table continues...

Hole No.	From (m)	To (m)	Intercept (m)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
	78.13	79.38	1.25	0.13	89.60	0.06	0.15
FW0194	256.20	256.30	0.10	0.08	318.00	0.22	0.05
	295.50	295.90	0.40	0.05	105.00	0.60	0.58
FW0195	95.20	113.85	18.65	0.33	49.83	0.01	0.06
	161.35	165.35	4.00	0.10	229.50	0.83	0.60
	238.66	238.97	0.31	0.10	152.00	0.10	1.69
	252.55	253.55	1.00	0.35	168.00	0.12	6.06
FW0196	72.60	74.03	1.43	0.58	121.62	0.15	0.49
FW0198	101.10	110.45	9.35	0.18	79.03	0.02	0.04
	130.50	131.60	1.10	0.07	104.00	0.04	0.24
	205.85	206.85	1.00	0.18	385.00	0.35	2.45
	241.70	243.20	1.50	0.17	521.00	0.54	0.66
	262.80	263.50	0.70	0.03	198.00	0.36	0.52
FW0199	56.80	57.80	1.00	0.04	50.00	0.02	0.02
	68.70	69.70	1.00	0.02	50.00	0.02	0.12
	104.30	112.00	7.70	0.07	121.33	0.02	0.05
	120.60	120.90	0.30	0.06	56.10	0.01	0.01
	142.36	143.36	1.00	0.03	101.00	0.08	0.11
	154.40	156.40	2.00	0.38	306.58	0.25	5.08
	186.40	188.25	1.85	0.01	64.57	0.07	0.17
FW0200	97.75	102.95	5.20	0.17	316.44	0.30	0.57
FW0201	13.10	14.00	0.90	1.84	37.00	0.16	0.01
	272.60	273.35	0.75	0.06	71.00	0.08	0.86
FW0202	19.05	20.05	1.00	0.06	84.00	0.03	0.15
	142.80	143.50	0.70	0.13	124.00	0.02	0.03
FW0204	91.60	94.00	2.40	0.54	149.70	0.05	0.08
	101.45	107.40	5.95	1.21	70.64	0.06	0.07
	140.81	144.95	4.14	0.09	105.13	0.15	0.41
	165.80	166.80	1.00	0.23	173.00	0.03	3.50
	278.09	281.65	3.56	0.07	43.18	0.05	0.03
FW0205	18.14	26.34	8.20	0.10	87.48	0.03	0.08
	86.50	86.70	0.20	0.19	108.00	0.06	0.05
	249.65	250.70	1.05	0.03	243.00	0.22	0.27
FW0197	130.60	133.90	3.30	0.21	145.00	1.04	3.02
	163.80	167.30	3.50	0.85	216.77	0.25	0.50
FW0208	130.28	130.50	0.22	0.03	176.00	0.03	0.06
	173.90	175.10	1.20	0.10	356.00	0.13	0.84
FW0211	34.60	44.05	9.45	0.30	41.25	0.02	0.03
	185.10	185.95	0.85	0.05	51.20	0.11	0.08
FW0212	41.52	45.93	4.41	0.06	111.12	0.03	0.15
	58.70	62.10	3.40	0.03	48.53	0.02	0.10
	68.35	68.70	0.35	0.21	362.00	0.28	0.08
	246.68	247.20	0.52	0.04	61.00	0.04	0.15
FW0214	402.64	403.84	1.20	0.25	154.00	0.61	4.17
FW0220	75.90	77.00	1.10	0.03	62.00	0.01	0.03
	79.80	81.90	2.10	0.32	50.52	0.03	0.07
	86.10	94.05	7.95	0.12	203.86	0.13	0.11
	121.80	124.90	3.10	0.11	180.87	0.08	0.59
	135.80	136.10	0.30	0.03	97.00	0.04	0.23
FW0221	43.90	44.20	0.30	0.25	1079.00	0.18	0.44
	57.40	57.60	0.20	0.07	616.00	0.67	0.30
	119.87	120.05	0.18	0.86	3107.00	1.69	2.67
FW0223	83.10	84.00	0.90	0.02	155.00	0.14	0.84

Figure 11.1 Fuwan Drill Hole Location Map with all holes shown



12.0 SAMPLING METHOD AND APPROACH

12.1 HISTORICAL SAMPLING METHODS 757 TEAM

Drill core was logged by geologists, and sections with visible sulphides, alteration and structures were measured and marked for sampling. Generally, sampling of the drill core began just above the silicified breccia zone which marked the contact between the Lower Carboniferous limestone and the Upper Triassic clastic unit. Drill core above this contact was rarely sampled.

All core was sawn in half by a member of 757 Team at the core logging facility located at their previous base, situated approximately 20 kilometres northeast of Jiangmen City. The samples were sealed and transported by truck from the base to the central lab at 757 Team in Jiangmen City for analysis. Three to five percent of the samples were sent to the Guangdong Central Laboratory, Ministry of Geology and Mineral Resources in Guangzhou for external checks.

12.2 MINCO MINING SAMPLING METHODS PHASES I TO III

Drill core was shipped to the field camp from the drilling site at the end of each shift. At the field camp, routine logging was conducted by Minco Mining geologists. In most cases, the average sample length was one metre, although shorter samples were collected in some narrow fracture zones.

The core was cut in half with a diamond saw, which was located in the core logging building. The other half of the sawn core was kept in the original core box, which was numbered and kept in the same building. Samples were sealed in sample bags and shipped to the Central Laboratory of the Institute of Geophysical and Geochemical Exploration (IGGE) in Langfang, Hebei province for preparation and analysis.

Core recovery was deemed not to be an issue and there were no other issues impacting sample quality.

12.3 2007-2008 PHASE IV, V AND VI PROGRAMS

Due to the scale of the project, Minco moved all core logging, cutting and storage facilities to a former primary school compound in the town of Fuwan. The compound is comprised of several buildings. Core is logged outside in the compound, and once completed and sampled it is transferred to racks located inside a secure building.

Drill core was picked up from the site by pick up truck at the end of each shift and taken to the facilities.

Drill core was logged by geologists, and sections with visible sulphides, alteration and structures were measured and marked for sampling. The core was cut in half with a diamond saw, which was located in a separate building exclusively for that purpose. In most cases, the average sample length was one metre, although shorter samples were collected in some narrow fracture zones. The other half of the sawn core was kept in the original core box, which was numbered and transferred to the storage building.

Core recovery was examined at the drills during the 2008 site visit and found to be satisfactory, though sections of karstic collapse were common.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 HISTORICAL SAMPLE PREPARATION, ANALYSES AND SECURITY

For a full account of the historical sample preparation, analyses and security, the reader is referred to a previous technical report titled, “Technical Report and Updated Resource Estimate on the Fuwan Property, Guangdong Province, China”, and dated June 1, 2007 (the “2007 Technical Report”). This report has been filed on SEDAR.

13.2 PHASE IV, V AND VI MINCO SILVER

Samples for Phases IV through VI were sent to the PRA Kunming lab in Yunnan province, China, for analysis of silver and gold by fire assay with an atomic absorption or gravimetric finish. Approximately ten percent of the pulps were re-run at ALS Chemex in Vancouver, British Columbia, and at ALS Chemex in Guangzhou, PRC using an aqua regia digestion and AA finish. Lead and zinc were assayed using aqua regia at PRA Lab.

The assay protocol at PRA Kunming involved:

- Crushing the whole sample (not more than 2 kg) to 0.84 mm (-20 mesh) size with a routine jaw crusher;
- Taking a 250 gram split from the crushed material and pulverizing to minus 200 mesh;
- A 30 gram sub-sample is analyzed for gold and silver using fire assay with an atomic absorption spectrometry or gravimetric finish;
- For Pb and Zn, a composite sample was prepared by combining resource-grade samples (maximum 3 samples for a composite sample) in individual mineralized zones. A 0.1 to 0.5 gram sample from the composite sample is dissolved and analyzed for Pb and Zn with ICP mass spectrometry.

The authors consider that the sampling and assaying protocol was satisfactory.

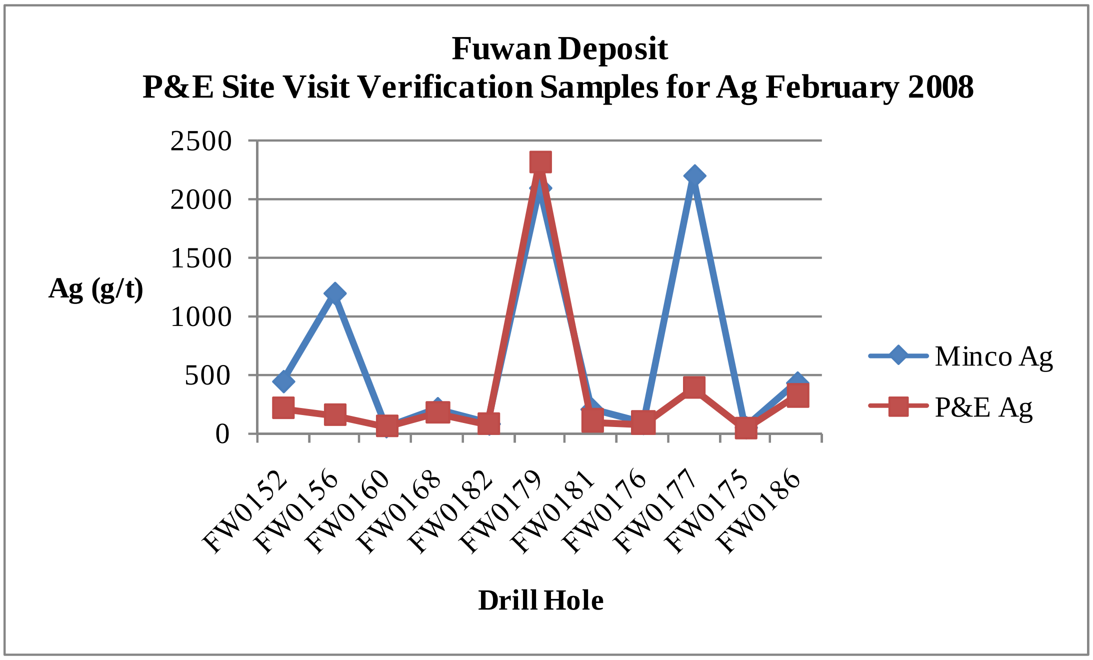
14.0 DATA VERIFICATION

14.1 SITE VISIT AND INDEPENDENT SAMPLING

The Fuwan Property was visited by Mr. Eugene Puritch, P. Eng., and Ms. Tracy Armstrong, P. Geo., on August 25th, 2005, by Ms. Armstrong on June 14 and 15, 2006, October 22 to 24, 2007 and February 4 and 5, 2008. Data verification sampling was done during all site visits.

Ten independent verification samples were taken during the February 2008 visit and results comparing the PRA Lab in Kunming, China to SGS Mineral Services Lab in Toronto, Canada are presented in Figure 14.1. All samples are from the latter half of the Phase VI program.

Figure 14.1 P&E Sample Verification Results – Phase VI Drill Program



A comparison of the P&E independent sample verification results versus the original assay results can be seen in Figure 14.1. The P&E results were satisfactory and demonstrate that the tenor of the silver is similar in most instances, to what was originally reported by Minco.

14.2 Minco QC Program – Phase VI

Minco instituted a quality assurance, quality control program which began for Phase I and has been maintained through all phases. To each batch of 17 samples Minco added one blank, one certified reference material sample and one field (core) duplicate sample. In addition, coarse and pulp duplicates were prepared and assayed regularly. A random ten percent of samples were sent to ALS Chemex Labs of Vancouver, British Columbia and ALS Chemex Labs in Guangzhou, PRC as an external monitor on the assaying.

All QC data from the Phase VI program were graphed and analyzed and results are presented in the following sections.

14.2.1 Blank Sample Results

The blank samples added to the batches sent to PRA lab were graphed. Results were reported for gold and silver. All values reported were less than 3 x detection limit for the element in question.

14.2.2 Certified Reference Material Results

Three certified reference materials were used to monitor lab accuracy in the latter half of Phase VI drilling. The values of the three reference materials were 446 g/t Ag, 220 g/t Ag and 54.9 g/t Ag. The two higher grade standards were obtained from IGGE, and the lower grade standard was supplied by CDN Resource Labs.

The reference material with a grade of 220 g/t Ag is close to the resource grade of 182 g/t Ag. The second IGGE standard used has a grade of 446 g/t Ag which is satisfactory for monitoring accuracy of the higher grades in the deposit, and the third reference material grade is close to the cut-off grade of 40 g/t Ag.

The author graphed and examined in detail all analytical results for the QC. There were 102 data points for the 446 g/t Ag reference material; 99 points remained within +/- two standard deviations from the mean with three failures greater than three standard deviations. As reported for previous drilling campaigns, the higher grade standard demonstrated a definite low bias.

The 220 g/t Ag standard had 106 data points and four failures. This standard demonstrated no bias.

The 54.9 g/t Ag standard had nine data points and all passed the QC.

The seven failures had no impact on the database as the samples surrounding them were either less than detection limit, or had the highest grades well below the cut-off grade of 40 g/t Ag. No action was taken.

14.2.3 DUPLICATE RESULTS

There were 225 field duplicate pairs. A simple scatter plot was done to examine precision. With the exception of two outliers, the data were clustered tightly around the 1:1 line.

It is the authors' opinion that the data were adequately verified for the purposes of the 2008 Technical Report.

15.0 ADJACENT PROPERTIES

The Changkeng Exploration Permit is surrounded completely by the Fuwan Property. The Changkeng Exploration Permit is owned by the Guangzhou Mingzhong Mining Co., (Mingzhong) a cooperative joint venture established between 5 partners, of which Minco Mining (China) Corporation, (Minco China) is the controlling shareholder with a 51% equity interest. Minco China is a wholly owned subsidiary of Minco Gold Corporation ("Minco Gold").

The following has been excerpted, with modifications, from Puritch et al, May, 2009, Technical Report and Updated Resource Estimate on the Changkeng Gold Property, Guangdong Province, China, NI 43-101 and NI 43-101F1 Technical Report prepared for Minco Gold Corporation, and Report No. 163.

Minco Gold has 51% ownership of the Changkeng Project which has 2 distinct and separate mineralized zones (a gold ("Au") dominant zone and a silver ("Ag") dominant zone). The gold portion of the resource estimate has been expanded and upgraded to contain indicated resources of 4.0 million tonnes @ 4.89 g/t Au for a total of 623,100 oz Au. The estimate also contains inferred resources of 4.0 million tonnes @ 3.01 g/t Au for a total of 386,800 oz Au (Table 15.1).

Table 15.1 March 2009 P&E Gold Dominant Portion of Resource Estimate @ 1.2 g/t AuEg Cut-off

Classification	Tonnes	Au (g/t)	Au (oz)	Ag (g/t)	Ag (oz)	AuEq** (g/t)	AuEq** (oz)
Indicated	3,961,000	4.89	623,100	11.2	1,423,000	5.08	646,800
Inferred	4,001,000	3.01	386,800	9.5	1,218,000	3.16	407,000

**The AuEq grade was calculated from Au US$800/oz and Ag US$14/oz with respective recoveries of 95% and 90%. The calculated Au:Ag ratio was 60:1 Pb and Zn values were too low to be of economic interest for resource reporting purposes.

1. Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. The quantity and grade of reported inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.

The Changkeng Project also contains a portion of a second distinct deposit which is silver dominant. Minco Gold assigned its 51% ownership in these resources to Minco Silver Corporation pursuant to the assignment agreement dated August 20, 2004. The deposit contains indicated resources of 5.6 million tonnes @ 170 g/t Ag for a total of 30,708,000 oz Ag and inferred resources of 1.1 million tonnes @ 220 g/t Ag for a total of 7,517,000 oz Ag (Table 15.2).

Table 15.2 March 2009 P&E Silver Dominant Portion of Resource Estimate @ 35 g/t Ag Cut-off

Classification	Tonnes	Ag (g/t)	Ag (oz)	Au (g/t)	Pb (%)	Zn (%)
Indicated	5,622,000	170	30,708,000	0.33	0.35	1.02
Inferred	1,063,000	220	7,517,000	0.24	0.61	1.36

1. Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. The quantity and grade of reported inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 METALLURGICAL REVIEW

Four main metallurgical testing programs were carried out on the multiple metal (silver/lead/zinc) mineralization samples from the Fuwan silver deposit in Guangdong province, China (Appendix N). The testwork includes ore hardness determination, mineralogical determination, flotation concentration, gravity separation, hydro-metallurgical process, and ancillary tests including settling tests and acid base accounting (ABA) tests. The testwork scopes are listed chronologically in Table 16.1.

Table 16.1 Main Testwork Programs for the Fuwan Silver Deposit

Laboratory	Testwork							
	Head Assay	Mineralogy	Hardness	Flotation	Gravity	Leach & Pretreatment	Settling	ABA
GRIMU, 1995	√	√		√		√		
Harris, 2007		√						
Lehne, 2007		√						
PRA, 2007	√	√	√	√	√	√		√
BGRIMM, 2008	√	√		√		√		
PRA, 2008/09	√			√			√	√
SGS, 2008			√					

Notes:
GRIMU = Guangdong Research Institute of Mineral Utilization
Harris = Harris Exploration Services
Lehne = Lehne and Associates Applied Mineralogy
PRA = Process Research Associates Ltd.
BGRIMM = Beijing General Research Institute for Mining and Metallurgy
SGS = SGS Lakefield Minerals Services

16.1.1 SAMPLE DESCRIPTION

Samples from different drill holes were composited for the metallurgical testing programs.

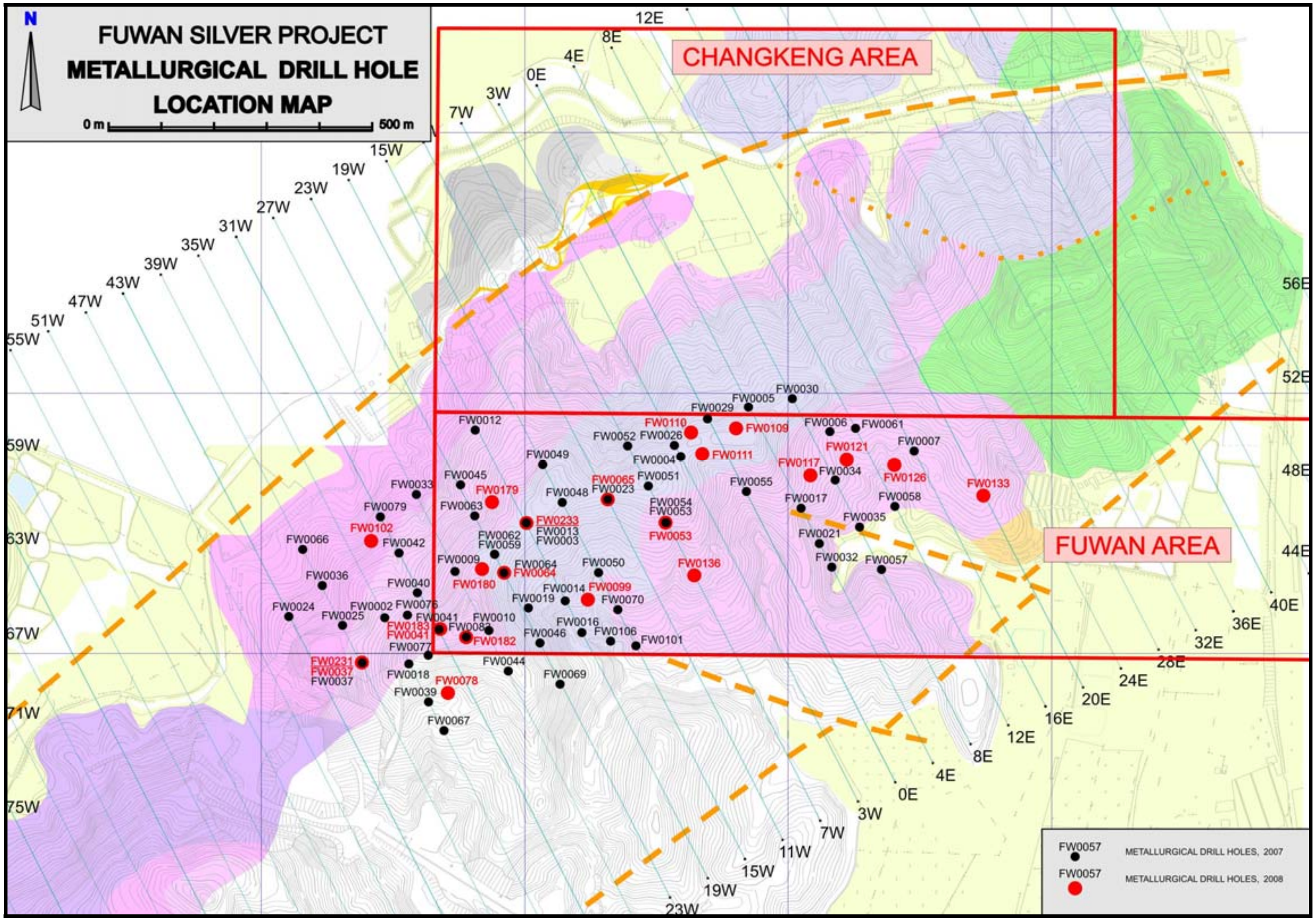
Table 16.2 summarizes sample resources, weights, and particle sizes in the reviewed testwork. The composite tested by BGRIMM was produced by blending 85 drilling hole interval samples. PRA used six composite samples containing high and average silver level for the 2007 testing program. For the 2008/2009 testing program, 16 samples were employed including master composite, zone composites, and variability test samples.

Table 16.2 Sample Description

Test Program	Sample Description	Weight (kg)	Top Particle Size (mm)
GRIMU 1995	1 composite from 2 drilling holes	200	2
PRA 2007	6 composite samples including 2 master composites from assay rejects	400	-
BGRIMM 2008	1 composite from 85 drilling hole intervals	71.5	2
PRA 2008/09	16 samples (including 1 master composite, 8 zone composites, 6 individual drill hole interval samples, and 1 hardness determination samples) from 253 drill core samples and assay rejects	298	19

The distribution of drill holes where metallurgical testing samples were collected for the 2007 and 2008/2009 testing programs is illustrated in Figure 16.1.

Figure 16.1 Distribution of Metallurgical Sample Drill Holes for the 2007 and 2008/2009 Testing Programs



16.1.2 Sample Characteristics

Chemical Element Analysis

The chemical element analyses of the composite samples were performed in each test program. Table 16.3 summarizes the main elements in the key composite samples. Silver grade varies between 200 g/t and 330 g/t, lead concentration ranges from 0.25% to 0.94%, and zinc content varies from 0.75% to 2.32%. Comparing with these significantly varied metal grades, gold content almost remains constant, ranging from 0.18 g/t to 0.21 g/t.

Table 16.3 Chemical Analysis of Composite Samples

Test Program	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)	TFe (%)	S (%)
Master Composite, GRIMU 1995	302	0.18	0.94	2.32	0.75	1.63
High Ag Composite 1, PRA 2007	330	0.21	0.39	1.15	-	-
Average Ag Composite 2, PRA 2007	245	0.20	0.28	1.16	-	-
Master Composite, BGRIMM 2008	200	0.18	0.25	0.75	1.16	1.62
Master Composite, PRA 2008/2009	255	0.15	0.44	1.38	-	-

Mineralogy

Composition

The dominant sulphide minerals in the mineralization are: pyrite, sphalerite, galena, argentiferous tennantitete-trahedrite, miargyrite, proustite-pyrargyrite, marcasite, native gold, bournonite, stephanite, chalcopyrite, and covellite. Argentiferous tennantitete-trahedrite, miargyrite, and proustite-pyrargyrite are the main silver carriers. Additional silver may be contained in stephanite, argentiferous galena, and black silver.

All the determinations indicate that galena and sphalerite are the major lead and zinc bearing minerals. GRIMU and BRIMM indicated that the gangue minerals include quartz, carbonate, and pyrite. The mineral species and distribution obtained in the testwork are summarized in Table 16.4.

Table 16.4 Key Minerals in Mineralization Samples

Laboratory	Silver Minerals (%)	Quartz (%)	Calcite (%)	Sphalerite (%)	Galena (%)	Pyrite (%)	Other (%)
GRIMU	trace	68	20	3.5	1.5	1	3
BGRIMM	0.06	61.81	26.06	1.07	0.32	2.21	8.47

PRA 2007	Ag (g/t)	Au g/t	Galena (%)	Sphalerite (%)	Pyrite (%)	Gangue
FW0018-256.4 m	193	0.08	2-2.5	5-7	~ 3	Vitric-tuff
FW0035-50.9 m	Colloidal Intergrowths			<0.01	> 15	Silty-tuff
FW0035-61.6 m	Sericitized Colloids			-	3-3.5	Sandstone
FW0035-75.7 m	Sedimentary Tuff			0.1	~ 3	Tuff-debris
FW0035-81.9 m	Rubble & Tuff Colloid			~0.05	2.5	Lava Breccia

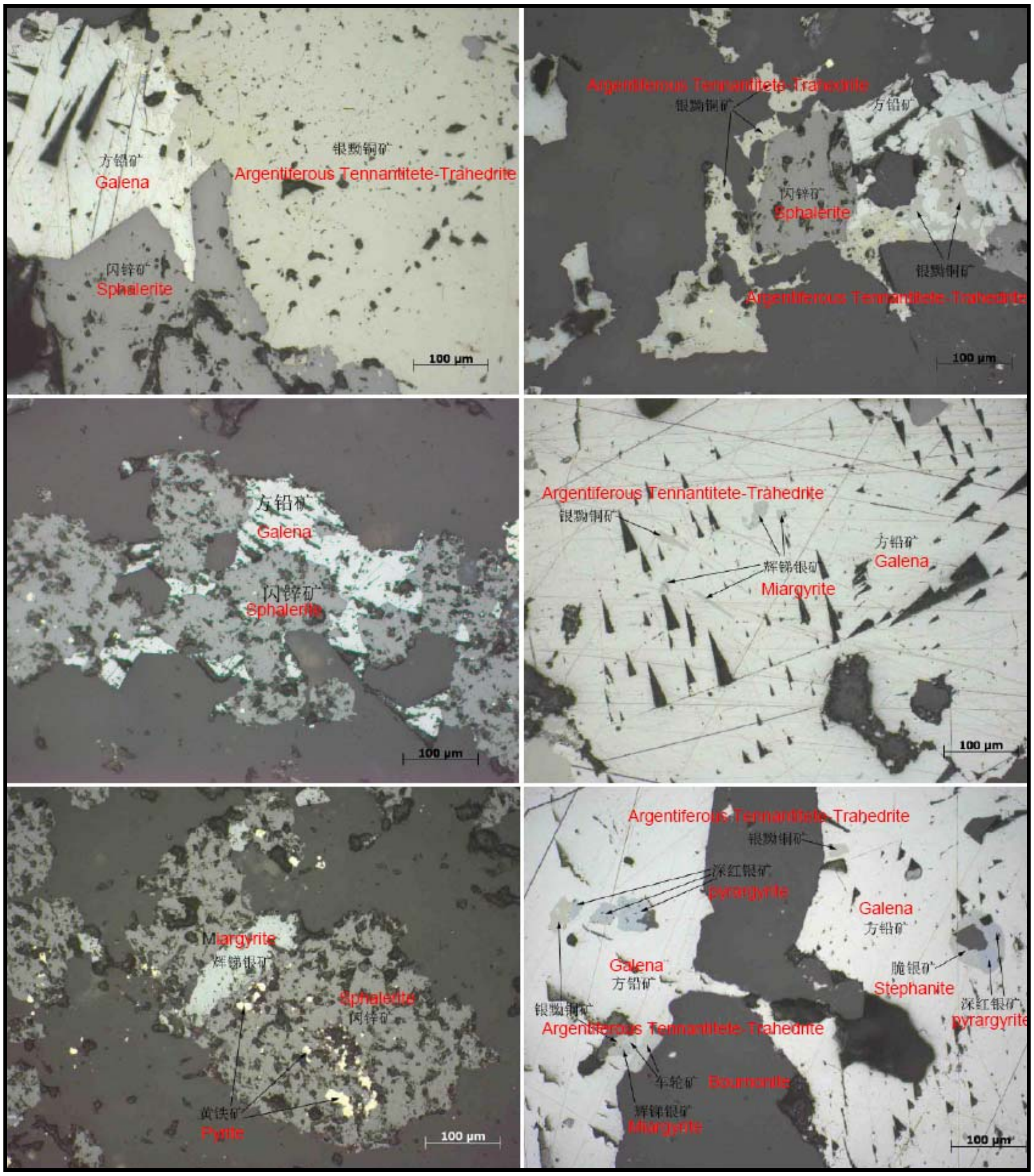
Furthermore, the determination results by PRA in 2007 indicate that the contents of sphalerite and pyrite vary significantly with the drill hole intervals. More than 15% pyrite was observed in sample FW0035-50.9 m, comparing with approximately 3% pyrite determined in the other samples. Sphalerite concentration varies from <0.01% to 5% to 7%.

BRIMM examined the liberation degree of silver, lead, and zinc minerals at various grinding particle sizes, varying from 60%, 70%, 80% to 90% passing 74 µm. At the grinding size of 80% passing 74 µm, approximately 54% of silver minerals are liberated from the other minerals. Higher liberation rates are observed for galena (81%) and sphalerite (88%). Mineral liberation is improved with further grinding to 90% passing 74 µm. At this particle size, the silver mineral liberation rate increases to 77%, galena to 90%, and sphalerite to 91%. The un-liberated silver minerals are found to be associated with galena (13%) and sphalerite (8%). Figure 16.2 shows some of typical relationships among the target minerals.

Specific Gravity

The ore specific gravity of 2.57 was reported by SGS as a part of the ore hardness tests.

Figure 16.2 Some Typical Relationships among Minerals



16.1.3 ORE HARDNESS

PRA, 2007

The ore hardness test was initially performed by PRA in 2007 on the high silver master composite 1. The obtained Bond Ball Mill Work Index is 17.8 kWh/t.

SGS, 2008

Detailed testwork was later carried out in 2008 by SGS. Four tests were conducted. The tests include Bond Low-energy Impact (CWI) Test, Bond Rod Mill Grindability (RWI) test, Bond Ball Mill Grindability (BWI) Test, and Bond Abrasion Index (AI) Test. Table 16.5 summarizes the test results on the five composite samples. SGS concluded that the composite samples are hard and moderately abrasive.

Table 16.5 Grindability Test Summary – SGS, 2008

Sample	Work Index (kWh/t)			AI (g)
	CWI	RWI	BWI	
Minco Comp.	14.6	15.9	15.4	0.774
FW-LOM	-	-	13.8	0.357
FW-Z1	-	-	16.7	-
FW-Z2	-	-	14.8	-
FW-5Y	-	-	14.3	-

16.1.4 *Flotation Tests*

Flotation Condition Tests

Bulk Flotation

Preliminary bulk flotation tests were conducted by GRIMU in 1995 and PRA in 2007 respectively. The flotation collectors used in the tests were conventional xanthates. The effects of feed particle size and collector consumption on the bulk flotation performance are discussed below.

GRIMU, 1995

Three different primary grinding particle sizes were tested by GRIMU: 49%, 61%, and 71% passing 74 µm. The test results are shown in Table 16.6. The metal recoveries were not significantly affected by the feed particle size.

Table 16.6 Metal Recoveries at Varied Feed Particle Sizes – GRIMU, 1995

Feed Particle Size (% passing 74 µm)	Recovery to Bulk Concentrate			Grade		
	Ag (%)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)
49	90.76	89.76	86.13	5,071	16.02	33.99
61	91.46	87.84	89.11	5,102	16.46	36.01
71	91.21	89.10	82.56	4,972	16.86	34.43

GRIMU also tested the effect of collector doses (butyl xanthate) on the silver metallurgical performance. The tested collector dosages were 150, 200 and 250 g/t. It was found that the highest silver recovery of 91% to the 4,969 g/t Ag bulk concentrates was obtained with adding 250 g/t butyl xanthate.

At a feed particle size of 61% passing 74 µm and 250 g/t collector consumption, GRIMU conducted a confirmation bulk flotation test. About 94% silver, 90% lead, and 90% zinc were recovered to the bulk concentrate, grading at 5,028 g/t Ag, 15.6% Pb, and 36.5% Zn.

PRA, 2007

The tests by PRA were carried out at particle sizes of 80% passing 74, 105 to 149 microns. As listed in Table 16.7, the results indicate that the metal recoveries to the bulk concentrates improved with reducing particle size.

Table 16.7 Recovery and Grade at Varied Feed Particle Sizes – PRA, 2007

Feed Particle Size (80% Passing, µm)	Recovery to Concentrate					Grade				
	Ag (%)	Au (%)	Pb (%)	Zn (%)	S (%)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)	S (%)
74	87.3	77.3	93.8	69.7	98.2	2,287	1.2	2.5	6.2	11.4
105	80.2	74.1	94.6	63.0	98.0	2,288	1.4	2.8	6.3	13.0
149	76.2	72.1	94.0	59.0	97.4	2,385	1.4	2.8	6.5	15.2

Figure 16.3 and Figure 16.4 plot the relationship between metal recoveries and flotation times at various grinding sizes. It is indicated that silver, gold, and lead recoveries increase rapidly in the first 10 minutes and then slow down. Zinc flotation recovery improved rapidly only after 10 minutes of flotation due to adding copper sulphate ($CuSO_4$) as zinc mineral activator. In addition, the finer feed had the better metal recoveries.

Figure 16.3 Silver and Gold Flotation Kinetics in Bulk Rougher Flotation – PRA

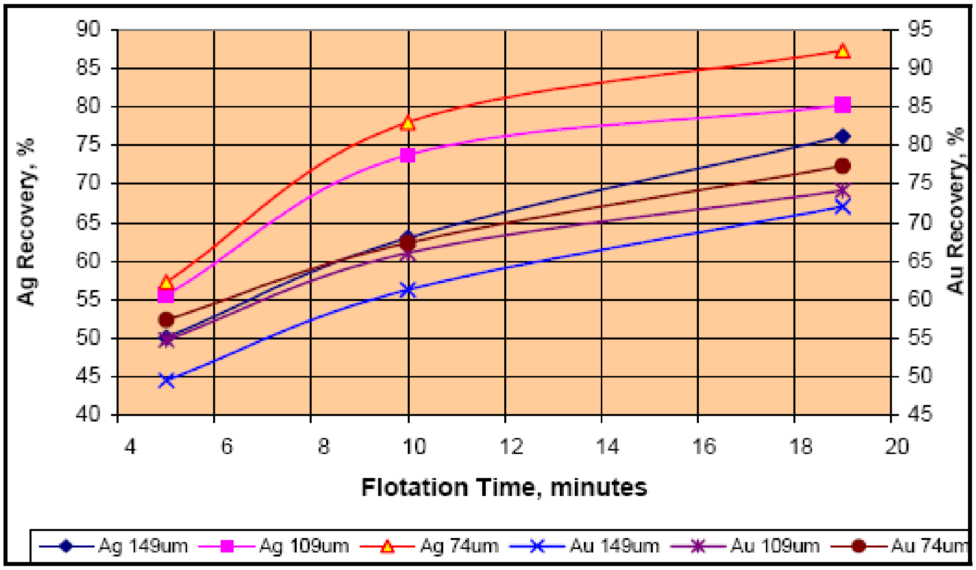
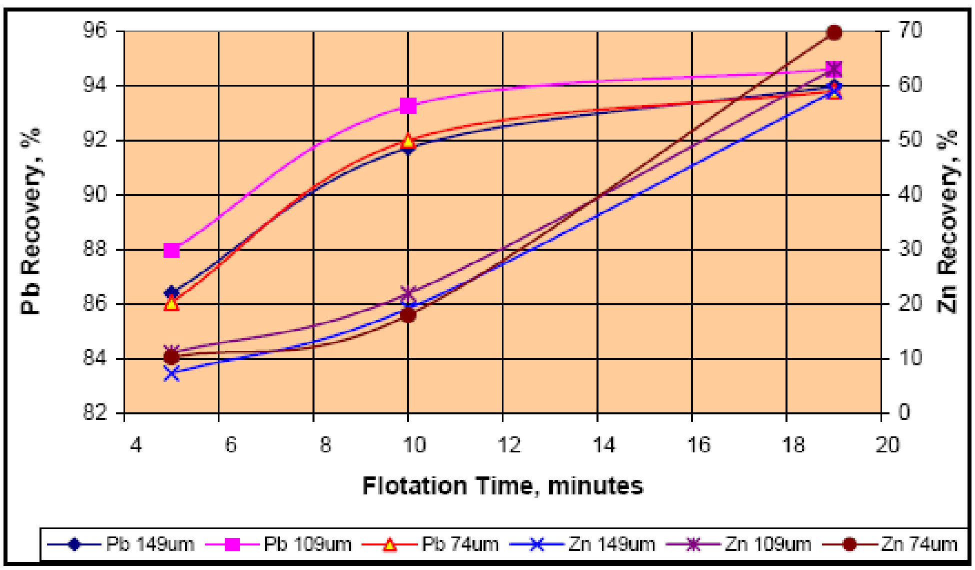


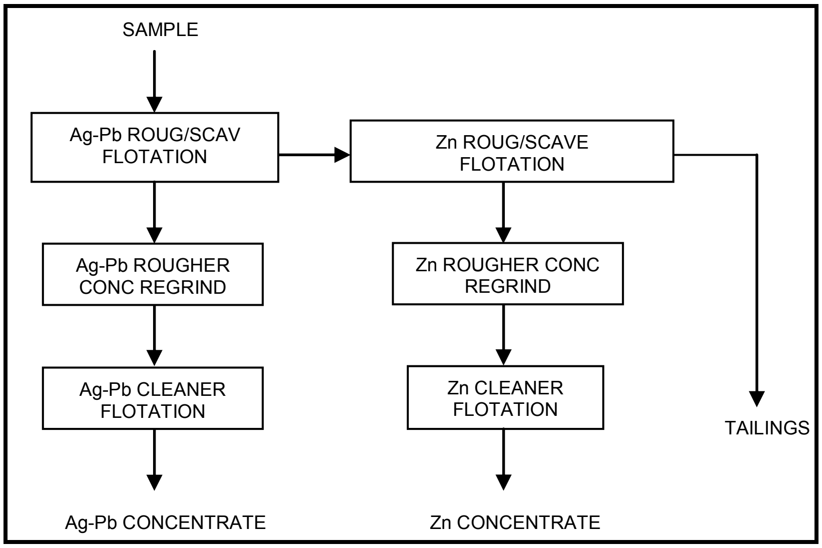
Figure 16.4 Lead and Zinc Flotation Kinetics in Bulk Rougher Flotation – PRA



Differential Flotation

All the testing programs tested the differential flotation in an effort to produce a silver-lead concentrate and a zinc concentrate. A typical differential flotation flowsheet tested is shown in Figure 16.5. In 1995, GRIMU explored the process and obtained promising results. The later testing programs by PRA and BGRIMM mainly focused on the process.

Figure 16.5 Typical Differential Flotation Flowsheet



Primary Grinding Particle Size – PRA, 2007

In the 2007 testwork, PRA tested the metallurgical responses of silver, lead, and zinc to differential flotation at three different primary grinding sizes of 80% passing 74 µm, 105 µm, to 149 µm. Sodium ethyl xanthate was used to as a collector while zinc sulphate ($ZnSO_4$) and lime (CaO) were employed as zinc minerals and pyrite suppressant in the silver-lead flotation circuit, and copper sulphate ($CuSO_4$) as an activator in the zinc flotation circuit. The slurry pH value was adjusted to 9.6 in the silver-lead circuit and to 11 in the zinc circuit by adding lime.

The test results are summarized in Table 16.8 and Table 16.9. The optimum grinding particle size selected was 80% passing 105 µm by PRA. At this particle size, silver and lead rougher recovery to the silver-lead rougher concentrate was 77%. Zinc recovery to the zinc rougher concentrate was 78%.

Table 16.8 Primary Grind Size – Silver- Lead Circuit, PRA 2007

Particle Size	Recovery to Ag-Pb Rougher Concentrate				Grade			
P_{80} (µm)	Ag (%)	Au (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
74	73.6	70.4	94.0	17.7	2894	1.61	3.96	2.45
105	77.0	64.9	93.8	20.8	2315	1.06	2.95	2.37
149	74.2	69.0	92.9	22.5	2755	1.78	3.31	2.99

Table 16.9 Primary Grind Size – Zinc Circuit, PRA 2007

Particle Size P_{80} (µm)	Recovery to Zn Rougher Concentrate				Grade			
	Ag (%)	Au (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
74	19.0	16.0	2.6	80.8	830	0.41	0.12	12.4
105	15.0	16.0	3.2	77.6	616	0.36	0.14	12.1
149	16.9	12.9	3.1	75.3	803	0.43	0.14	12.8

Primary Grinding Particle Size – BGRIMM, 2008

BGRIMM conducted the effect of primary grinding particle size on the silver, lead, and zinc metallurgical performance as well. The testwork was conducted at four particle sizes namely 60%, 70%, 80% to 90% passing 74 µm in the silver-lead rougher/scavenger circuit. The pH value was adjusted to 10.5 by adding lime solution. The other reagents used were $ZnSO_4$, amino-dithiophosphate (ADTP), sodium diethyl dithiocarbamate (SDTC), and BK204.

Table 16.10 summarizes the test results. The highest rougher silver and lead recoveries were obtained at the grind size of 80% passing 74 µm. About 60% of the silver and 88% of the lead were recovered to the Ag-Pb rougher flotation concentrate, grading at 3,312 g/t Ag and 5.8% Pb. Zinc recovery to the Ag-Pb scavenger flotation tailings was independent on the feed particle size.

Table 16.10 Effect of Primary Grind Size on Rougher/Scavenger Flotation – BGRIMM, 2008

Particle Size (% Passing 74 µm)	Concentrate	Recovery to Rougher Concentrate			Grade		
		Ag (%)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)
60	Rougher	59.7	82.4	8.0	4,558	8.0	2.3
	Scavenger	16.2	4.2	3.5	1,511	0.49	1.2
70	Rougher	55.7	85.4	6.6	3,841	7.4	1.8
	Scavenger	17.3	5.7	4.0	1,576	0.65	1.4
80	Rougher	59.8	88.3	6.7	3,312	5.8	1.5
	Scavenger	17.5	5.5	5.4	935	0.35	1.1
90	Rougher	28.8	77.4	4.5	1,850	6.2	1.1
	Scavenger	36.0	10.5	4.9	2,358	0.86	1.3

BGRIMM further conducted primary grind size confirmation tests using the optimized flotation conditions. The metal recoveries reporting to the rougher flotation concentrate and the rougher/scavenger flotation concentrate are compared in Table

16.11. It appears that in the tested primary grind size range, the effect of the grind size on the metal recoveries is insignificant.

Table 16.11 Effect of Primary Grind Size on Rougher/Scavenger Flotation (Confirmation Tests) – BGRIMM, 2008

Feed Particle Size (% Passing 74 µm)	Concentrate	Recovery to Rougher Concentrates		
		Ag (%)	Pb (%)	Zn (%)
60	Rougher	84.0	91.6	19.1
	Scavenger	2.4	1.7	3.8
70	Rougher	85.6	91.3	17.8
	Scavenger	2.3	1.9	3.9
80	Rougher	86.1	93.3	17.5
	Scavenger	2.3	1.3	3.9

BGRIMM also determined the liberation rates of the main metal bearing minerals at the grind sizes. The results are summarized in Table 16.12. The target mineral liberation rates improved substantially with increasing grinding fineness. At the grind size of 80% passing 74 µm, the silver liberation rate (including the silver associated with lead minerals) is over 80%. The liberation rates for the lead and zinc minerals reach approximately 90%.

Table 16.12 Mineral Liberation Percent at Varied Grind Sizes – BGRIMM, 2008

Particle Size (% Passing 74 µm)	Liberation Rate (%)					
	Silver Minerals		Lead Minerals		Zinc Minerals	
	Ag	Ag + Pb	Pb	Pb + Zn	Zn	Zn + Pb
60	21.7	51.6	75.2	86.5	72.0	81.6
70	26.8	54.0	78.3	86.8	75.5	82.4
80	53.8	80.2	80.8	89.1	88.2	90.7
90	77.4	90.2	90.4	95.6	90.8	93.2

Primary Grinding Particle Size – PRA, 2008/2009

In the 2008/2009 testwork, PRA conducted two primary grind size tests. The tested primary grinding particle sizes were 80% passing 70 and 148 µm. As shown in Table 16.13, the metal recoveries are not significantly affected by the particle size. Accordingly, PRA selected the particle size of 80% passing 100 µm as the primary grind size in the testing program.

Table 16.13 Effect of Primary Grind Size on Rougher/Scavenger Flotation – PRA, 2008/2009

Primary Grinding 80% passing (µm)	Roug + Scav Recovery (%)			Roug + Scav Conc Grades		
	Ag-Pb Conc		Zn Conc	Ag-Pb Conc		Zn Conc
	Ag	Pb	Zn	Ag (g/t)	Pb (%)	Zn (%)
71	83.8	95.2	68.9	1628	4.5	10.4
148	81.0	95.4	69.5	1923	4.9	19.0

Regrinding Particle Size

PRA examined the effects of the rougher concentrate regrinding on the silver-lead and zinc cleaner flotation. As shown in Table 16.14, the test results indicate that regrinding did not significantly affect the silver, lead, and zinc cleaner metallurgical performance. PRA recommended regrinding the silver-lead rougher concentrate only.

Table 16.14 Effect of Regrind Size on Metallurgical Performance – PRA, 2008/2009

Test ID	Regrind Time (min)		Recovery (%)			Grade		
	Ag-Pb Conc	Zn Conc	Ag	Pb	Zn	Ag (g/t)	Pb (%)	Zn (%)
F5	15	0	62.2	81.1	64.5	13122	36.8	61.2
F6	15	5	66.0	82.8	67.7	15538	45.9	60.2
F7	10	2.5	74.3	89.7	60.3	18396	46.0	60.3

Similar results were obtained in the 2007 testing program. However, the test results (Tests T26 and T28) appear to indicate that regrinding would help the rejection of zinc minerals from the silver-lead final concentrate.

BGRIMM investigated the liberation rate of the target minerals at various regrind sizes in an effort to project the optimum regrinding particle sizes for silver-lead rougher concentrate and zinc rougher concentrate. Table 16.15 shows the determination results. It appears that the target mineral liberation improves significantly with an increase in regrinding fineness. BGRIMM recommended that regrinding particle size for both the concentrates be approximately 87% passing 38 µm.

Table 16.15 Mineral Liberation Rate at Various Regrind Sizes – BGRIMM, 2008

% Passing 38 µm	Zn Mineral Liberation Rate in Reground Ag-Pb Rougher Concentrate (%)	% Passing 38 µm	Mineral Liberation Rate in Reground Zn Rougher Concentrate (%)	
			Zn Mineral	Ag/Pb Minerals
75	60.0	73	87.0	0
88	75.5	87	93.8	17.6
93	77.4	90	97.4	60.1
96	82.7	92	98.4	60.8

Flotation pH Value – PRA, 2007

In the 2007 testwork, the silver-lead rougher flotation was mostly carried out at a pH of 9.5. It appears that the zinc minerals were well suppressed at the pH. PRA investigated the effect of slurry pH on silver-lead concentrate and zinc concentrate upgrading. The tested pHs were 10.5 and 11.5 for the silver-lead cleaner circuit and 11.5 and 12.0 for the zinc cleaner circuit. The results, as shown in Table 16.16, indicate that the metal recoveries to the cleaner concentrates and the concentrate grades improve with increasing pH values.

Table 16.16 Effect of Slurry pH on Metal Recovery to Cleaner Concentrates – PRA, 2007

Flotation Stage	pH	Recovery to Concentrate (%)			Grade		
		Ag	Pb	Zn	Ag (g/t)	Pb (%)	Zn (%)
3rd Ag-Pb Cleaner	10.5	35.9	41.8	0.9	15,311	21.9	1.49
	11.5	54.3	55.0	1.1	19,137	22.9	1.52
3rd Zn Cleaner	11.5	5.3	1.0	53.2	1,538	0.34	53.2
	12.0	2.9	1.1	56.6	773	0.36	60.1

Flotation pH Value – BGRIMM, 2008

BGRIMM also studied the effects of slurry pH on the silver, lead, and zinc metallurgical performance. The tested pH values ranged from 7.7 to 10.5 in the silver-lead rougher flotation, 9 to 11.2 in the silver-lead cleaner flotation, and 11.4 to 11.9 in the zinc rougher flotation.

Table 16.17 shows the test results. A higher pH value would benefit for improving silver and lead recoveries at the silver-lead rougher flotation. Also increasing pH value from 9 to 11.2 improved silver and lead concentrate upgrading. The silver and lead grades in the silver-lead cleaner concentrate increased from 12,961 g/t Ag to 15,502 g/t Ag and from 13% Pb to16% Pb, respectively. In the tested pH range, it appears that pH had an insignificant impact on the zinc metallurgical response in the zinc rougher flotation.

Table 16.17 Effect of Slurry pH on Metal Recovery to Various Concentrates – BGRIMM, 2008

Flotation Stage	pH	Recovery to Concentrate			Grade		
		Ag (%)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)
Silver-Lead Rougher	7.7	59.5	85.8	6.7	4,022	6.9	1.8
	9.3	59.8	88.3	6.7	3,312	5.8	1.5
	10.5	62.8	88.3	7.3	3,607	6.2	1.7
Silver-Lead Cleaner	9.0	82.4	63.9	4.5	12,961	13.2	2.5
	11.2	79.9	69.4	3.6	15,502	16.0	2.4
Zinc Rougher	11.4	N/A	N/A	93.8	N/A	N/A	13.9
	11.9	N/A	N/A	93.9	N/A	N/A	12.6

Flotation pH Value – PRA, 2008/2009

In the 2008/2009 testing program, PRA conducted confirmation tests to further study the effect of slurry pH on the key metal metallurgical performance at the silver-lead rougher flotation stage. The test results are listed in Table 16.18. At pH 8.3, although both the silver and lead recoveries are high, most of the zinc reports to the silver-lead rougher concentrate. However, no zinc sulphate was added in the flotation. The optimum pH value was considered as 10.5.

Table 16.18 Effect of Slurry pH on Metal Recovery to Silver-Lead Rougher Concentrate – PRA, 2008/2009

Test ID	pH	Primary Grind 80% passing (µm)	$ZnSO_4$ (g/t)	Recovery (%)		
				Ag	Pb	Zn
F5	10.0	92	300	80.9	97.2	29.1
F6	10.0	92	300	83.8	97.3	29.0
F7	10.5	98	300	85.2	96.5	32.0
F2	10.8	86	350	82.1	92.5	28.3
F8	8.3	94	0	96.4	98.8	84.8

Reagent Schedule – Silver-Lead Flotation Circuit

ZINC DEPRESSANT – PRA, 2007

The 2007 testing program studied the impact of the depressant (zinc sulphate, $ZnSO_4$) dosage on the zinc mineral suppression in the silver-lead rougher/scavenger flotation. As shown in Table 16.19, the suppression of the zinc minerals in the mineralization is not very sensitive to the tested zinc sulphate dosage. However, the test results appear to indicate that the least zinc reporting the silver-lead concentrate was obtained when adding 200 g/t $ZnSO_4$.

Table 16.19 Effect of $ZnSO_4$ Dosage on Zinc Mineral Suppression – PRA, 2007

$ZnSO_4$ Addition (g/t)	Recovery to Concentrate (%)			Grade		
	Ag	Pb	Zn	Ag (g/t)	Pb (%)	Zn (%)
0	73.7	93.2	22.0	3,053	4.0	3.2
100	72.1	88.2	20.4	2,404	3.6	2.8
150	68.6	87.9	24.1	2,378	3.8	3.6
200	69.7	91.9	19.1	2,042	4.2	2.8

ZINC DEPRESSANT – BGRIMM, 2008

As shown in Table 16.20, when the $ZnSO_4$ dosage to the silver-lead rougher flotation was increased to 500 g/t, the silver minerals appeared to be suppressed although the zinc minerals were better suppressed. Accordingly, BRIMM suggested that the optimum $ZnSO_4$ dose should be 200 g/t.

Table 16.20 Effect of $ZnSO_4$ Dosage on Zinc Mineral Suppression – BGRIMM, 2008

ZnSO4 Dosage (g/t)	Recovery (%)		Grade	
	Ag	Zn	Ag (g/t)	Zn (%)
0	73.3	10.0	3,658	1.9
200	77.3	9.6	3,359	1.5
500	58.9	7.9	2,877	1.4

ZINC DEPRESSANT – PRA, 2008/2009

The 2008/2009 testing program had not conducted systematic testing to study the influence of the zinc mineral suppression reagents on the zinc mineral metallurgical performance. However, it was found that the zinc minerals in the FW-5Y and FW-Z1 samples were more difficult to suppress in the silver-lead flotation circuit compared to the other samples. When the total $ZnSO_4$ dosage was increased to 2,000 g/t in the lead rougher flotation, the zinc minerals reporting to the silver-lead concentrate reduced. Also it was found that when the zinc cleaner flotation pH was raised from 12.0 to 12.5, the zinc concentrate grade improved significantly (Tests F24 and F26).

COLLECTORS IN AG-PB ROUGHER FLOTATION – PRA, 2007

PRA tested the effects of sodium ethyl xanthate, ammonium dibutyl dithiophosphate (ADDP) and SDTC on silver and lead recoveries.

When only using sodium ethyl xanthate as a collector in the silver-lead flotation circuit, increasing the collector dosage was helpful to improve the metal recovery. The test results are summarized in Table 16.21. At a dosage of 100 g/t sodium ethyl xanthate, approximately 70% of the silver and 95% of the lead were recovered into the rougher flotation concentrate.

Table 16.21 Effect of Sodium Ethyl Xanthate Dosage on Metallurgical Performance – PRA, 2007

Sodium Ethyl Xanthate Dosage (g/t)	Recovery to Concentrate			Grade		
	Ag (%)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)
50	59.4	86.9	15.5	2,756	4.8	2.7
60	59.1	84.9	14.0	2,752	5.0	2.8
75	70.4	85.6	14.3	3,566	4.1	2.6
100	70.4	94.7	19.4	2,336	3.9	2.4

When ADDP was added with sodium ethyl xanthate (as shown in Table 16.22), both the silver and lead recoveries increased.

Table 16.22 Effect of ADDP Dosage on Metallurgical Performance – PRA, 2007

ADDP Dosage (g/t)	Recovery to Concentrate			Grade		
	Ag (%)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)
0	70.4	85.6	14.3	3,566	4.1	2.6
40	75.8	89.1	N/A	2,168	2.4	2.1

PRA also tested the effect of adding the SDTC collector, in conjunction with sodium ethyl xanthate and ADDP, on silver and lead rougher flotation. The test results shown in Table 16.23 indicate that SDTC played a key role in improving silver recovery. At an adding rate of 20 g/t SDTC, the silver recovery to rougher flotation concentrate was significantly improved to over 92%. The lead recovery appeared improvement as well.

Table 16.23 Effect of SDTC Dosage on Metallurgical Performance – PRA, 2007

SDTC Dosage (g/t)	Recovery to Concentrate			Grade		
	Ag (%)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)
20	92.0	89.6	23.3	3,305	3.3	3.1
30	90.7	92.9	23.6	3,117	4.1	3.4
40	90.3	88.7	20.8	3,055	2.6	2.9

COLLECTORS IN AG/PB ROUGHER FLOTATION – BGRIMM, 2008

BRIMM tested the effect of two flotation collector combinations on silver and zinc mineral metallurgical performance. The combinations were ADDP + SDTC, and ADTP + SDTC. The other reagents used were 200 g/t $ZnSO_4$, 70 g/t sodium hexametaphosphate and lime (maintaining pH at 9.8).

The test results are shown in Table 16.24. It seems that the silver recovery improved with increasing the reagent dosage. The mineralization responded well to both the collector combinations.

Table 16.24 Effect of Collector Regime on Silver and Zinc Metallurgical Performance – Rougher Flotation (BGRIMM, 2008)

Reagent Schedule	Dose (g/t)	Recoveries (%)		Grade	
		Ag	Zn	Ag (g/t)	Zn (%)
ADDP/SDTC/#2 Oil*	60/60/30	90.0	27.2	1168	1.3
ADDP/SDTC/#2 Oil	30/60/20	88.2	18.1	2488	1.8
ADTP/SDTC/#2 Oil	30/60/30	87.4	13.4	2880	1.7
ADDP/SDTC/KB204*	30/60/30	87.0	24.1	1245	1.3
ADTP/SDTC/KB204	20/60/50	86.1	17.6	2026	1.6
ADDP/SDTC/KB204	10/30/30	83.8	17.5	2410	1.9
ADTP/SDTC/KB204	10/60/50	86.3	20.6	1959	1.7
ADTP/SDTC/KB204	60/60/30	88.0	14.7	2727	1.7

* #2 oil and BK 204 are frothers.

BGRIMM also tested the effect of collector dosage (ADTP/SDTC/BK204) on silver-lead cleaner flotation. Table 16.25 summarizes the test results. The test that added more reagents produced better results for silver and lead but slightly more zinc reporting to the silver-lead concentrate.

Table 16.25 Effect of Collector Regime on Metallurgical Performance – Cleaner Flotation (BGRIMM, 2008)

Reagent Schedule	Reagent Addition (g/t)	Recoveries (%)			Grade		
		Ag	Pb	Zn	Ag (g/t)	Pb (%)	Zn (%)
ADTP/SDTC /BK204	20/20/0	84.0	84.5	24.2	10,885	13.0	3.0
ADTP/SDTC /BK204	80/20/10	90.9	90.4	26.6	9,906	11.2	2.6

COLLECTORS IN AG-PB ROUGHER FLOTATION – PRA, 2008/2009

PRA compared the effect of two different collector regimes (ADDP + SDTC and potassium ethyl xanthate [PEX] + 3418A) on silver and lead rougher flotation. The metal recoveries and grades of the silver-lead rougher flotation concentrates are shown in Table 16.26.

Both tests produced similar concentrate grades; however, when using the combination of PEX and 3418A collectors, silver recovery to the rougher flotation concentrate was reduced by 4.3%. Lead recovery and grade were not sensitive to the collector regimes.

Table 16.26 Effect of Collector Regime on Metallurgical Performance – Silver-Lead Rougher Flotation (PRA, 2009)

Test ID	Collectors	Recovery			Grade		
		Ag %	Pb %	Zn %	Ag g/t	Pb %	Zn %
F7	ADDP/SDTC	85.2	96.5	32.0	2,087	4.9	5.2
F9	PEX/3418 A	80.9	95.2	28.0	2,041	5.2	5.2

OTHER REAGENTS

BGRIMM also tested the effects of sodium hexametaphosphate on silver-lead rougher flotation. The tested sodium hexametaphosphate dosage ranged 0 g/t to 150 g/t. The test results indicate that the silver and lead metallurgical performances did not significantly responded to the reagent. The silver recovery and grade increased by 3.6% and 125 g/t by adding 70 g/t sodium hexametaphosphate, when compared to not adding the reagent. When the sodium hexametaphosphate dosage was increased to 150 g/t, no improvement was noticed in the metal recovery and grade. BGRIMM claimed that adding 70 g/t hexametaphosphate would benefit silver recovery.

Other collectors such as 9538 and Z200 were also tested by BGRIMM in an effort to improve silver and lead recovery. No significant improvement in silver and lead flotation was observed.

Reagent Schedule – Zinc Flotation Circuit

The testing programs used very common reagents for the zinc flotation circuit. All the tests used lime to adjust slurry pH, copper sulphate to activate zinc minerals, and xanthate to collect zinc minerals.

ZINC ACTIVATOR AND COLLECTOR – BGRIMM, 2008

BGRIMM conducted two tests to investigate the effect of zinc mineral activator and collector dosage on zinc cleaner flotation after regrinding the zinc rougher concentrate. Two stages of cleaner flotation were used in the tests at a slurry pH of approximately 12. The reagents tested were $CuSO_4$ and sodium butyl xanthate. As listed in Table 16.27, both the reagent schedules produced comparable zinc metallurgical performances.

Table 16.27 Effect of Reagent Schedules on Zinc Cleaner Flotation – BGRIMM, 2008

Reagent Schedule	Reagent Dosage (g/t)	Recovery* (Zn %)	Grade (Zn %)
$CuSO_4$/Butyl Xanthate	100/50	91.8	52.3
$CuSO_4$/Butyl Xanthate	200/75	95.0	50.3

* Zn Circuit Recovery.

ZN ACTIVATOR – PRA, 2007

In 2007, PRA tested the effects of a zinc activator ($CuSO_4$) on zinc rougher/ scavenger flotation. The zinc recovery and grade of the rougher/scavenger zinc concentrate are summarized in Table 16.28. PRA recommended adding 150 g/t $CuSO_4$ for zinc mineral activation.

Table 16.28 Effect of $CuSO_4$ Dosage on Zinc Rougher/Scavenger Flotation – PRA, 2008

Test ID	Flotation	$CuSO_4$ (g/t)	Recovery Zn (%)	Grade Zn (%)
T13	Zn Rougher	250	78.5	13.5
T14	Zn Rougher	200	74.6	14.5
T15	Zn Rougher	150	78.2	14.3
T16	Zn Rougher/Scavenger	150	77.4	10.3
T17	Zn Rougher/Scavenger	100	71.7	9.3
T18	Zn Rougher/Scavenger	50	44.1	6.5

Flotation Feed Pulp Solid Density

BGRIMM investigated the influence of rougher flotation pulp solids density on the silver metallurgical response at the silver-lead rougher flotation. The test results are tabulated in Table 16.29. It appears that the pulp solid density did not have a significant effect on silver flotation. BGRIMM recommended using 28% solids density in the silver-lead rougher flotation stage.

Table 16.29 Effect of Pulp Solid Density on Silver Rougher Flotation – BGRIMM, 2008

Solids Density (%)	Collector Schedule	Recovery (% Ag)	Grade (g/t Ag)
28	ADDP 30 g/t / SDTC 60 g/t	88.5	1,865
38	ADDP 38 g/t / SDTC 75 g/t	88.1	1,246
38	ADDP 45 g/t / SDTC 90 g/t	89.6	1,135

Variability Tests

PRA 2008/2009

The first batch variability flotation tests were performed on five zone composite samples, one low grade composite (LG) sample, and one composite which may be mined during early stages of mine life, as proposed by the 2008 Pre-feasibility Study by CNIF (the Pre-feasibility Study mining schedule has been changed in this Feasibility Study). The flowsheet used was similar to the one as illustrated in Figure 16.5 but without zinc rougher concentrate regrinding. Typical test conditions used are shown in Table 16.30. The test condition details can be found in the 2008/2009 PRA report.

Table 16.30 Typical Variability Flotation Test Conditions – PRA, 2008/2009

Reagents	Rougher/ Scavenger	1st Cleaner/ Scavenger	2nd Cleaner	3rd Cleaner
Primary Grind P_{80} (µm)			*100*	
Silver-Lead Flotation Circuit				
Rougher Ag/Pb Concentrate Regrind, 80% passing (µm)			*25*	
pH	10.5	11.5	11.5	11.5
$ZnSO_4$ (g/t)	300/-	200/-	200	100
ADDP (g/t)	40/10	30/10	-	-
SDTC (g/t)	20/10	40/10	-	-

table continues…

Reagents	Rougher/ Scavenger	1st Cleaner/ Scavenger	2nd Cleaner	3rd Cleaner
Zinc Flotation Circuit				
Rougher Zinc Concentrate Regrind P_{80} (µm)			*No Rougher Zinc Regrind*	
pH	11.5	12	12	12
$CuSO_4$ (g/t)	75/-	20/-	-	-
SIPX (g/t)	50/10	25/10	10	-

The variability test results are summarized in Table 16.31. The FW-LZ, FW-LG, FW-Z2, and FW-Z3 composite samples produced satisfactory metal recoveries and grades. For the FW-Z1, FW-5Y, and FW-UZ composite samples, it appears that the zinc minerals were more difficult to suppress, in particular for the upper level sample (FW-UZ), although the silver and lead responses are acceptable. At an enhanced zinc mineral depressant usage and a higher pH, the zinc minerals did not show reasonably well responses to the reagent dose changes.

Table 16.31 Composite Sample Variability Flotation Test Results – PRA 2008/2009

Test ID	Sample ID	Recovery (%)			Grade					
		Ag-Pb Conc		Zn Conc	3rd Ag-Pb Conc		3rd Zn Conc	Head		
		Ag	Pb	Zn	Ag (g/t)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)
F11	FW-Z1	85.6	87.2	35.7	21,003	13.4	23.5	262.5	0.16	0.56
F12	FW-5Y	90.0	82.5	26.4	10,321	3.5	16.8	239.2	0.09	0.37
F13	FW-5Y	79.5	79.2	36.7	28,461	15.9	13.1	246.0	0.14	0.36
F14	FW-LZ	83.4	94.9	69.5	24,122	37.6	64.6	175.9	0.23	1.00
F15	FW-UZ	91.9	88.5	29.2	22,298	16.1	13.1	284.0	0.20	0.46
F16	FW-LG	88.6	93.2	65.1*	16,819	22.9	58.5*	121.9	0.15	0.51
F17	FW-Z3	86.8*	97.1*	78.4*	15,946	43.6*	57.9*	172.4	0.41	1.26
F18	FW-Z2	75.0	88.7	78.6*	28,513	69.5	67.6*	329.5	0.65	2.89
F27	FW-Z2 B	87.5	97.5	78.6	8,446	33.8	39.6	289.5	0.95	1.76
F29	FW-Z2 B	90.5	97.2	75.7	7,920	27.1	51.2	288.6	0.86	1.70

* 2nd cleaner recoveries and grades.

The 2008/2009 PRA program also performed another batch variability tests on six samples randomly collected from various drill hole intervals. The test conditions were same as the previous tests. The test results are summarized in Table 16.32. These samples showed good metallurgical responses to the developed test conditions. The metal recoveries and grades from Tests F31 to F33 were higher than the first batch optimum results. The poor metal recoveries obtained from in Tests F28, F30 and F34 were dominantly caused by low head grades.

Table 16.32 Individual Sample Variability Flotation Test Results – PRA, 2008/2009

		Recovery (%)			Grade					
		Ag-Pb Conc		Zn Conc	3rd Ag-Pb Conc		3rd Zn Conc	Head		
Test ID	Sample ID	Ag	Pb	Zn	Ag (g/t)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)
F28*	FW-007 Z2	83.7	44.1	22.5	3,106	1.3*	4.7*	70.4	0.05	0.13
F30*	FW-0192 Z1	87.0	37.7	6.9	12,449	0.5	1.1*	121.9	0.02	0.07
F31	FW-0131 Z3	94.6	97.1	83.6	26,848	55.8	63.5	808.0	1.67	5.52
F32	FW-0013 Z2	95.2	94.0	77.7	25,789	12.4	51.6	793.8	0.38	1.62
F33*	FW-0192 Z2	93.2	91.3	61.2	47,293	32.5*	55.1*	376.4	0.25	0.78
F34*	FW-0131 Z2	29.2	28.2	15.7	148	0.1*	0.4*	43.2	0.03	0.08

* 2nd cleaner recoveries and grades.

Silver Recovery to Zinc Concentrate

The variability tests also indicated that silver grade in zinc concentrates increased with zinc grade of the concentrate. Some of the silver had a close relationship with the zinc minerals in the mineralization samples. Figure 16.6 shows this relationship.

In general, the higher than 50% zinc concentrates would contain approximately 500 g/t to 2,500 g/t Ag. Figure 16.7 shows the mineralogical relationship among sphalerite, galena, and silver minerals.

Figure 16.6 Silver Grade in Zinc Concentrate vs. Zinc Grade in Zinc Concentrate

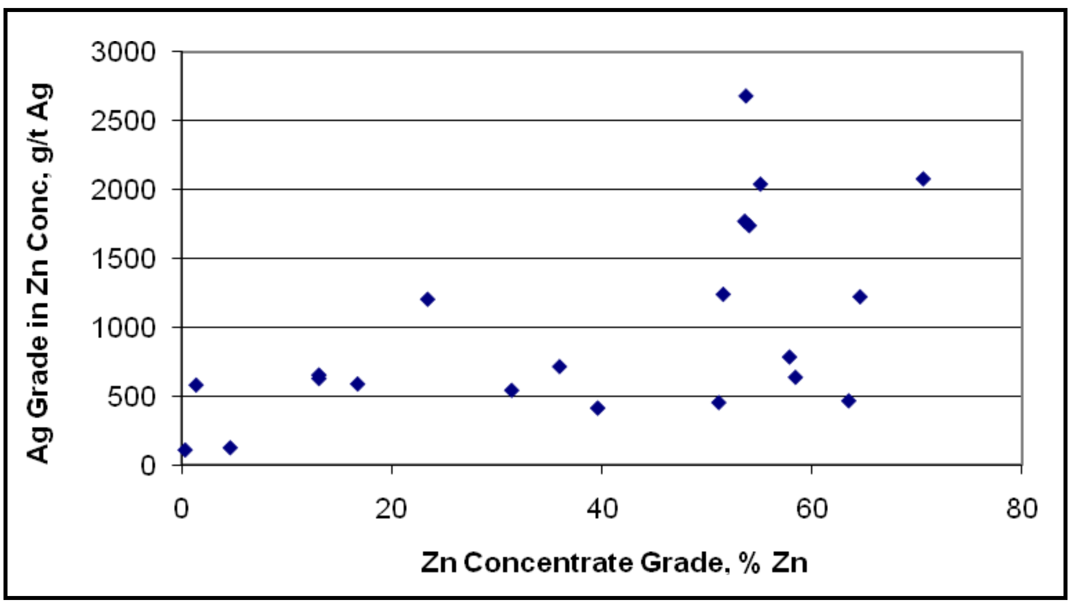
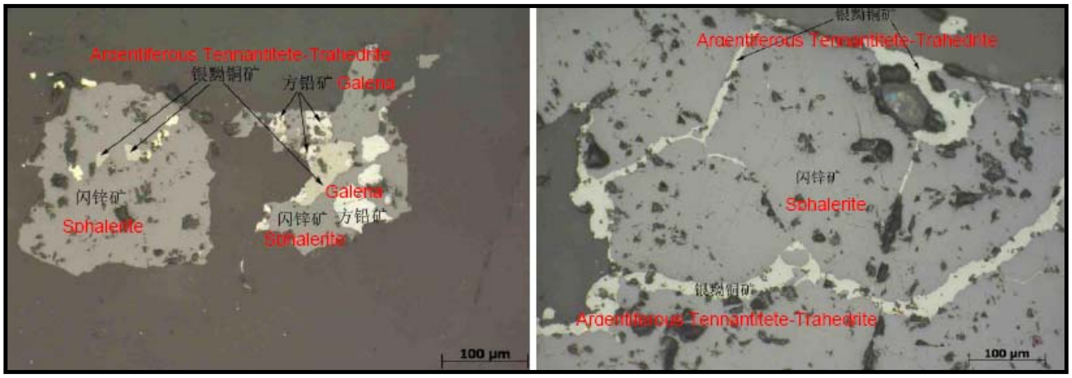


Figure 16.7 Mineralogical Relationship among Sphalerite, Galena, and Silver

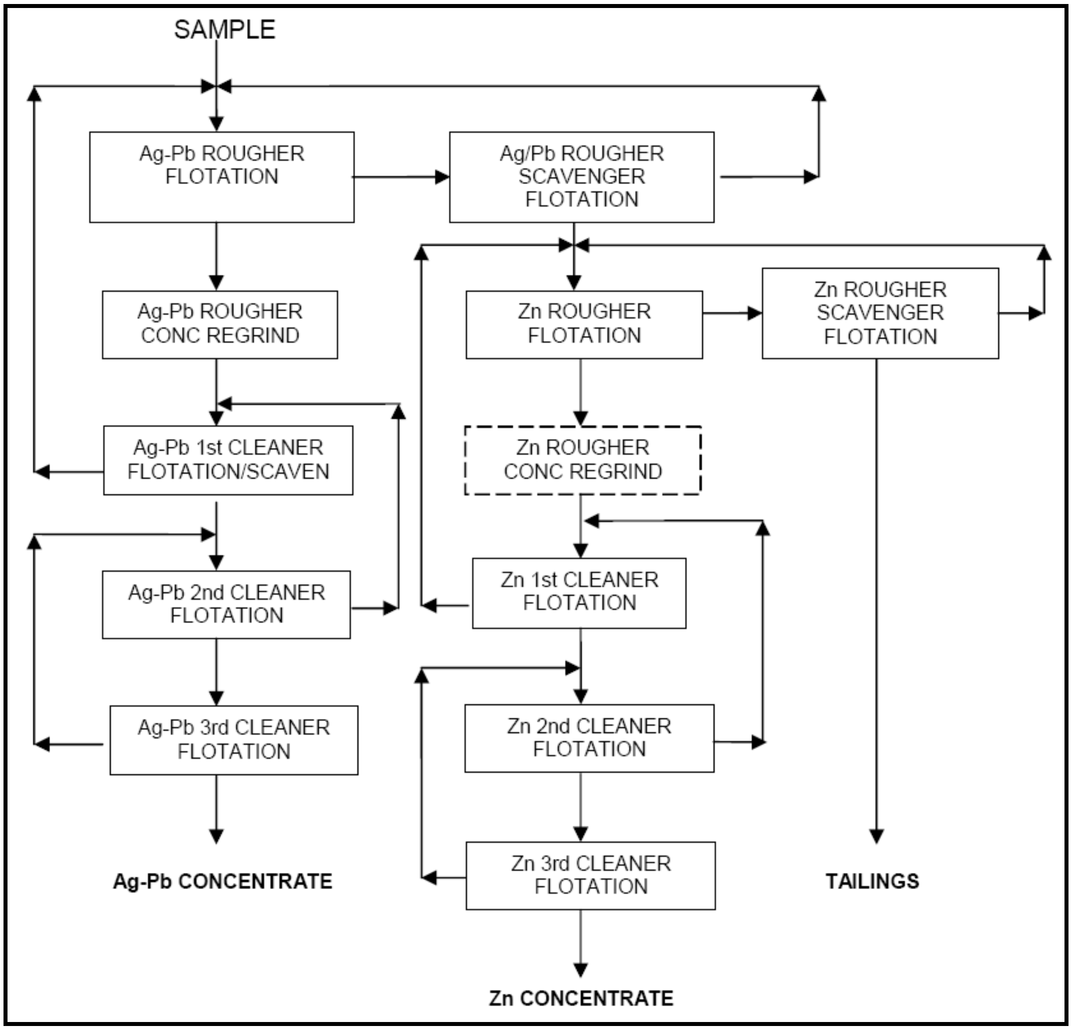


The mineralogical investigation by BGRIMM indicated that when the grind size is finer than 80% passing 74 μm, approximately 8% of the silver is still closely associated with the zinc minerals. BGRIMM concluded that a portion of the silver minerals would report to the zinc concentrate even at a fine grind size.

LOCKED CYCLE FLOTATION TESTS

Locked cycle flotation tests were carried out by BGRIMM and PRA respectively using various different composite samples. The typical locked cycle flowsheet is shown in Figure 16.8.

Figure 16.8 Typical Locked Cycle Differential Flotation Flowsheet



PRA, 2007

In 2007 PRA conducted locked cycle tests in two phases. The second phase used higher silver head grade samples compared to the average grade samples used in the first phase testing. A similar flowsheet as shown in Table 16.33 was employed. The zinc rougher concentrate was reground and also there were no silver-lead and zinc rougher scavenger flotation in these tests. The test conditions are summarized in Table 16.33.

Table 16.33 Locked Cycle Test Conditions – PRA, 2007

Reagents	Rougher/ Scavenger	1st Cleaner	2nd Cleaner	3rd Cleaner
Primary Grind P_{80} (µm)			*~105*	
Silver-Lead Flotation Circuit				
Rougher Ag/Pb Concentrate Regrind, 80% passing (µm)			*~45*	
pH	10.5	11.5	11.5	11.5
$ZnSO_4$ (g/t)	100/50	100	50	-
ADDP (g/t)	20/20	20	10	-
SDTC (g/t)	10/10	30	10	-
Zinc Flotation Circuit				
Rougher Zinc Concentrate Regrind P_{80} (µm)			*~45*	
pH	11.5	12.0	12.0	12.0
$CuSO_4$ (g/t)	100/50	200	-	-
Sodium Ethyl Xanthate (g/t)	50/25	75	25	-

The test results are listed in Table 16.34 and Table 16.35. Similar metal recoveries and grades were obtained from the two batches of tests. The silver recoveries were approximately 86%. The lead grades were low but seemed to be capable of further upgrading. In the subsequent concentrate mineralogy analysis, it was observed that the silver-lead concentrate was diluted with the liberated pyrite and gangue minerals.

Table 16.34 Phase One Locked Cycle Test Results – PRA, 2007

		Recovery (%)				Grade			
		Ag-Pb Conc			Zn Conc	Ag-Pb Conc			Zn Conc
Test No.	Products	Au	Ag	Pb	Zn	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
LCT1	Ag-Pb Cleaner Conc	46.1	83.1	87.4	4.3	4.0	10,961	12.3	2.8
	Zn Cleaner Conc	22.5	14.2	4.7	92.6	1.9	1,800	0.6	59.2
	Final Tailings	31.5	2.6	7.9	3.1	0.06	7.2	0.02	0.04
	Calculated Head	100	100	100	100	0.17	263.5	0.28	1.3
LCT2	Ag-Pb Cleaner Conc	49.1	85.4	75.5	7.1	3.9	8506	4.7	2.8
	Zn Cleaner Conc	17.8	10.8	4.4	89.4	2.1	1,562	0.4	51.4
	Final Tailings	33.2	3.7	20.1	3.5	0.06	7.9	0.03	0.03
	Calculated Head	100	100	100	100	0.16	205.9	0.13	0.82
LCT3	Ag-Pb Cleaner Conc	49.1	83.6	86.2	6.9	2.9	9,708	10.5	5.03
	Zn Cleaner Conc	24.6	14.4	5.4	90.9	1.3	1,513	0.6	59.7
	Final Tailings	26.2	2.0	8.4	2.1	0.04	6.1	0.03	0.04
	Calculated Head	100	100	100	100	0.14	284.8	0.30	1.77

Table 16.35 Phase Two Locked Cycle Test Results – PRA, 2007

Test No.	Products	Recovery (%)				Grade			
		Ag-Pb Conc			Zn Conc	Ag-Pb Conc			Zn Conc
		Au	Ag	Pb	Zn	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
LCT4	Ag-Pb Cleaner Conc	38.7	86.1	88.1	3.3	2.2	9,630	7.8	1.6
	Zn Cleaner Conc	41.0	12.1	4.7	94.6	2.4	1,402	0.43	47.0
	Final Tailings	20.3	1.8	7.1	2.2	0.04	6.4	0.02	0.03
	Calculated Head	100	100	100	100	0.17	336.3	0.26	1.45
LCT5	Ag-Pb Cleaner Conc	29.6	79.9	58.8	8.9	2.6	9,485	3.3	5.0
	Zn Cleaner Conc	36.5	17.6	22.9	87.8	3.2	2,037	1.27	48.2
	Final Tailings	33.9	2.6	18.4	3.3	0.08	7.8	0.03	0.05
	Calculated Head	100	100	100	100	0.22	290.1	0.14	1.37
LCT6	Ag-Pb Cleaner Conc	40.7	88.2	95.0	2.7	2.1	11,857	17.9	1.5
	Zn Cleaner Conc	26.3	9.8	1.7	94.0	1.3	1,249	0.30	49.2
	Final Tailings	32.9	2.0	3.3	3.3	0.04	5.8	0.01	0.04
	Calculated Head	100	100	100	100	0.11	277.7	0.39	1.14

BGRIMM, 2008

Five locked-cycle differential flotation tests were carried out by BGRIMM in 2008, each with a similar flowsheet as in Figure 16.5 but including some variations. Table 16.36 lists the typical test conditions.

Table 16.36 Typical Locked Cycle Test Conditions – BGRIMM, 2008

Reagents	Rougher/ Scavenger	1st Cleaner/ Scavenger	2nd Cleaner	3rd Cleaner
Primary Grind % Passing74 μm			*80/70*	
Silver-Lead Flotation Circuit				
Rougher Ag/Pb Concentrate Regrind, P_{90} (μm)			*38*	
pH	10.5	11.2	11.2	-
Hexametaphosphate (g/t)	100/50	-/50	-	-
$ZnSO_4$ (g/t)	200/100	200/-	200	100
ADDP (g/t)	30/10	5/5	-	-
ADTP * (g/t)	60/20	20/20		
SDTC (g/t)	60/20	20/10	-	-
Zinc Flotation Circuit				
Rougher Zn Concentrate Regrind P_{90} (μm)			*38*	
pH	11.5	12.0	12.0	12.0
$CuSO_4$ (g/t)	150/150	100	-	-
Butyl Xanthate (g/t)	60/50	50	-	-

* for 80%passing 74 μm feed only.

The test results are summarized in Table 16.37. Generally, the changes in the test conditions had a very insignificant influence on the silver, lead, and zinc metallurgical performance except for Test #1, which produced a lower zinc recovery. The results appear to indicate that regrinding on silver-lead rougher concentrate is necessary. The metallurgical benefits from the zinc rougher concentrate regrinding are not significant. Compared with Test #1, the zinc recovery from Test #2 improved by approximately 6.6%. It appears that using either ADDP or ADTP in conjunction with SDTC would produce similar metallurgical performance. No substantial effect of the primary grind sizes on the metallurgical responses was noticed.

Table 16.37 Locked Cycle Flotation Test Results – BGRIMM, 2008

Test #	Primary Grind % Passing 74 µm	Rougher Conc Regrinding		Recovery (%)			Grade					
				Ag-Pb Conc		Zn Conc	Ag-Pb Conc		Zn Conc	Tailings		
		Ag-Pb	Zn	Ag	Pb	Zn	Ag g/t	Pb %	Zn %	Ag g/t	Pb %	Zn %
1	80	No	No	87.5	94.5	84.6	9,200	12.0	54.3	10.1	0.01	0.07
2	80	Yes	No	87.4	93.4	91.2	11,145	13.5	59.0	9.3	0.01	0.03
3	70	Yes	Yes	86.2	93.5	90.0	11,447	15.7	52.0	11.0	0.01	0.03
4	70	Yes	Yes	86.8	94.1	92.1	11,971	15.3	52.6	9.06	0.01	0.03
5	70	Yes	Yes	87.1	92.6	92.7	9,330	11.6	54.1	9.4	0.02	0.03

PRA, 2008/2009

PRA conducted three locked-cycle tests in the 2008/2009 testing program on the FW-Master composite sample. Table 16.38 lists the locked-cycle test results. The metal recoveries ranged from 74% to 77% for silver, from 93% to 94% for lead, and from 86% to 91% for zinc respectively. The third locked-cycle flotation test was to produce tailings samples for environmental studies under the same condition as Test F20.

Table 16.38 Locked Cycle Flotation Test Results – PRA, 2008/2009

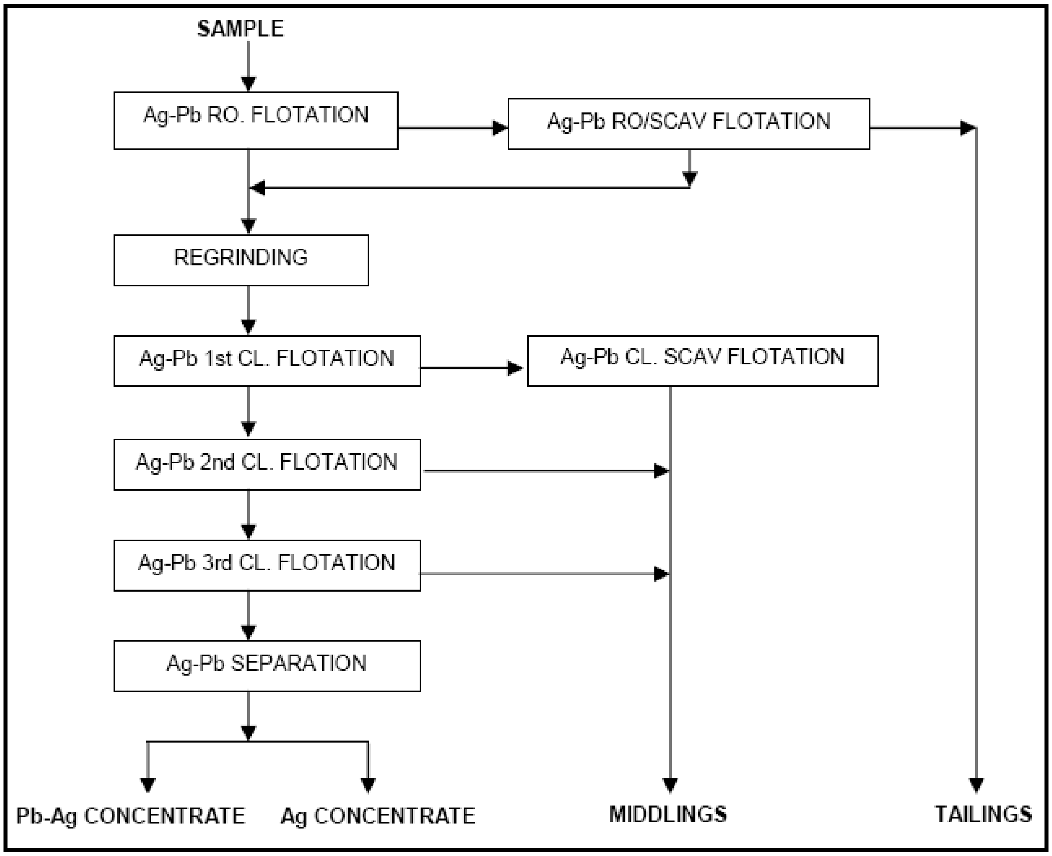
Test ID	Product	Mass Pull (%)	Assay			Recovery (%)		
			Ag (g/t)	Pb (%)	Zn (%)	Ag	Pb	Zn
F10 LCT1	Ag-Pb Cleaner Conc	1.7	9,823	28.7	4.3	74.0	92.6	3.8
	Zn Cleaner Conc	3.4	1,328	0.4	50.6	20.3	2.5	91.0
	Calculated Head	100	218.8	0.51	1.86	100	100	100
F20 LCT2	Ag-Pb Cleaner Conc	1.1	15,473	42.5	5.7	77.3	94.4	4.0
	Zn Cleaner Conc	2.9	1,446	0.4	48.1	19.5	2.2	90.0
	Calculated Head	100	213.4	0.48	1.54	100	100	100
F35 LCT3	Ag-Pb Cleaner Conc	1.2	13,155	36.8	8.0	76.3	93.9	6.6
	Zn Cleaner Conc	2.9	1,346	0.4	44.4	18.4	2.1	86.1
	Calculated Head	100	210.0	0.48	1.49	100	100	100

Compared with the data from the previous testing programs, less silver reported to the silver-lead concentrate although the total silver recovery to the silver-lead concentrate and the zinc concentrate was comparable.

THREE-PRODUCT FLOTATION TEST

BGRIMM tested a flotation flowsheet that produced three products including one silver/lead concentrate, one lead/silver concentrate, and one zinc rich tailings. The flowsheet is shown in Figure 16.9. The silver/lead rougher/scavenger concentrates were reground and then subjected to the four-stage cleaner flotation.

Figure 16.9 Simplified Open Cycle Flotation Test Flowsheet – BGRIMM, 2008



The produced lead-silver concentrate contains about 57% lead, compared to 34% lead in the silver-lead concentrate. The test results are summarized in Table 16.39.

Table 16.39 Open Cycle Differential Flotation Test Results – BGRIMM, 2008

	Recovery (%)			Grade		
Product	**Ag**	**Pb**	**Zn**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**
Pb-Ag Conc	9.2	21.4	0.28	17,770	57.3	2.7
Ag Conc	63.4	54.6	1.6	28,478	33.9	3.7

In the 2008/2009 testing program, PRA separated the third silver-lead concentrate produced from the locked cycle test (LCT 3) into a silver and a lead-silver concentrate. Table 16.40 lists the test results. It appears that a high lead grade concentrate can be produced from the silver-lead bulk concentrate. More lead minerals reported to the lead-silver concentrate when using either $ZnSO_4$+Na_2SiO_3 combination or $ZnSO_4$+NaCN combination as depressants. However, the silver grades were low in the silver concentrates produced from the two reagent regimes.

Table 16.40 Silver-Lead Concentrate Separation Test Results – PRA, 2008/2009

Test F35	Reagent	Pb-Ag Concentrate		Ag Concentrate	
		Pb (%)	Ag (kg/t)	Ag (kg/t)	Pb (%)
Cycle 1	$ZnSO_4$ – 200 g/t	72.9	23.9	12.8	34.8
Cycle 2	$ZnSO_4$ – 500 g/t	51.9	23.5	15.0	44.1
Cycle 3	$ZnSO_4$ – 200 g/t; Na_2SiO_3 - 200 g/t	54.9	21.6	2.0	3.0
Cycle 4	$ZnSO_4$ – 200 g/t; NaCN – 30 g/t	56.9	21.1	3.3	6.7
Cycle 5	MBS – 200 g/t	49.3	18.2	3.5	10.2

16.1.5 GRAVITY SEPARATION

PRA employed a three–pass procedure to evaluate silver mineral recovery by centrifugal gravity concentration using a batch Falcon SB40 machine. The centrifugal separation concentrate was panned for further upgrading. The tests were conducted at three different grind sizes of 80% passing 149, 100, and 74 µm to determine the optimum metal recovery conditions. Table 16.41 summarizes the test results. Although the target minerals were concentrated in some degree, the metal recoveries were lower. The primary grinding size had a slight impact on metal recoveries.

Table 16.41 Three-stage Falcon Concentration Test Results – PRA, 2007

Particle Size 80% Passing (µm)	Recovery (%)				Grade			
	Au	Ag	Pb	Zn	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
147	39.8	47.0	68.4	29.6	0.63	715	1.11	1.59
105	36.2	38.7	65.7	25.6	0.75	627	1.18	1.46
79	33.2	40.4	69.0	23.5	0.75	720	1.46	1.66

16.1.6 HYDROMETALLURGICAL EXTRACTION

WHOLE ORE DIRECT LEACHING TESTS

GRIMU, 1995

Direct cyanidation leaching on the composite samples was investigated by GRIMU in 1995. The tests were conducted at a feed particle size P_{90} of 75 µm, a pH value between 10 and 11, and a solids density of 33% with adding 10 kg/t NaCN. The feed samples containing 164 g/t to 1,004 g/t Ag were leached for 24 h. The silver extraction to the leaching solution was low, ranging from 16% to 27%.

PRA, 2007

In 2007, PRA also tested cyanidation leaching on the composite samples at varied feed particle sizes. The leaching test conditions were 5 g/L sodium cyanide, a pH higher than 11, and a 40% solid density. The test results are summarized in Table 16.42.

Table 16.42 Cyanide Leach Results on Master Composite 1 – PRA, 2007

Leach Time (h)	% Extraction					
	80% Passing 149 µm		80% Passing 109 µm		80% Passing 79 µm	
	Au	Ag	Au	Ag	Au	Ag
4	11.6	12.4	7.5	8.1	7.5	9.6
8	11.9	17.1	7.7	17.4	7.7	17.3
24	11.8	31.4	15.2	28.2	15.3	29.0
48	22.9	42.5	15.0	45.0	15.4	38.7
72	23.0	48.8	15.2	52.9	15.1	49.3

Approximately 50% of the silver was leached out after 72 h retention time. The extractions were not sensitive to grind size. It appears that leaching would continue if the leaching retention time was extended.

ROASTING PRE-TREATMENT AND LEACHING TEST ON HEAD SAMPLES

GRIMU investigated the effect of pre-treatment on silver and gold leaching extraction. The pre-treatment methods include direct oxidation roasting and chloride roasting.

The head sample was roasted at various temperatures for 2 h in an oxidation environment, and then the residue was cyanide leached in a 33% solids density for 24 hours. The slurry pH was between 10 and 11. The following sections will discuss the effect of roasting temperature, NaCN dosage, feed particle size, and roasting atmosphere on silver and gold extractions.

Roasting Temperature

The leaching test results at varied roasting temperatures are listed in Table 16.43. The silver extraction increased significantly after roasting compared to without pre-treatment. The extraction improved with increasing temperature. After being oxidized at 750°C, the silver extraction was improved to 71%.

Table 16.43 Effect of Oxidation Roasting Temperature on Silver Extraction – GRIMU, 1995

Roasting Temperature (°C)	Silver Grade in Residues (g/t)	Silver Extraction (%)
No Roasting	233	23.0
450	114	62.3
550	106	64.9
650	92	69.5
750	88	70.8

Cyanide Consumption

GRIMU also tested the effect of cyanide dosage on silver extraction after the head sample was roasted at 650°C. The cyanide dosage varied from 2 to 8 kg/t. The effect of the cyanide concentration on silver extraction was insignificant. Table 16.44 shows the silver leaching test results.

Table 16.44 Effect of Cyanide Dosage on Silver Extraction from Roasted Head – GRIMU, 1995

Cyanide Dosage (kg/t)	CN % in Leaching Solution	Grade of Residues (g/t Ag)	Extraction (% Ag)
2	0.08	108	64.3
4	0.27	95.8	68.3
6	0.28	96.7	67.9
8	0.38	84.4	71.9

Feed Particle Size

GRIMU test results showed that feed particle size did not significantly affect silver extraction after the head was pre-treated by roasting at 650°C. Table 16.45 lists the silver leaching test results.

Table 16.45 Effect of Feed Particle Size on Silver Extraction from Roasted Head – GRIMU, 1995

% Passing 74 µm	Grade in Residues (g/t Ag)	Extraction (% Ag)
59.0	130	56.9
80.0	114	62.3
91.4	102	66.3
96.5	110	63.6

Oxidation Roasting Atmosphere and Chloride Roasting

The effect of roasting atmosphere on silver extraction was investigated by GRIMU in 1995. The test results showed that the roasting atmosphere did not significantly affect silver extraction. It seems that silver extraction improved slightly when the head was chloride roasted.

In an effort to improve silver extraction, GRIMU also conducted various optimization tests including sodium chloride dosage, cyanide dosage, and leaching retention time. The test results indicated that the tested conditions did not substantially influence the silver extraction.

The test results obtained under the optimum test conditions are summarized in Table 16.46. Approximately 80% of the silver was extracted from the chloride roasted head.

Table 16.46 Chloride Roasting – Cyanidation Test Results, GRIMU 1995

Test No.	Head Grade (g/t Ag)	CN Concentration (%)	Grade in Residues (g/t Ag)	Extraction (% Ag)
1	302	0.16	62.7	80.3
2	302	0.16	63.3	80.1

Silver/Lead/Zinc Leaching on Flotation Concentrate Samples

GRIMU and BGRIMM investigated the silver and other metal recoveries from flotation concentrates. GRIMU focused on the silver-lead-zinc bulk flotation concentrate and BGRIMM on the silver-lead concentrate.

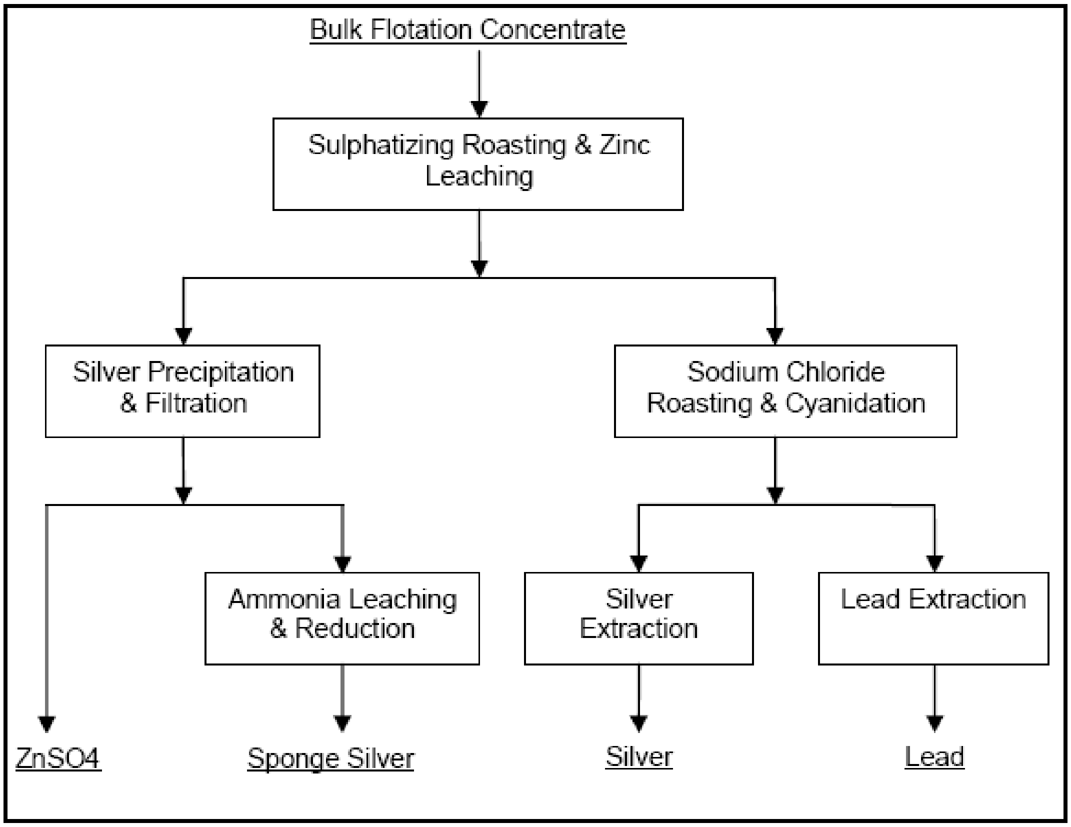
GRIMU

The flowsheet used by GRIMU consists of three stages:

1. bulk flotation concentrate sulphatizing roasting and sulphuric acid leaching
2. silver extraction from the sulphuric acid leached residues after chloride roasting
3. zinc and silver separation from sulphuric acid leaching solution.

Figure 16.10 shows the simplified flowsheet of the hydrometallurgical process.

Figure 16.10 Simplified Flowsheet for Silver/Lead/Zinc Recovery – GRIMU, 1995



Various roasting and leaching conditions were tested in the program. GRIMU recommended the following test conditions:

- sulphatizing roasting: 600°C to 650°C for 2 hours
- roasted residues leach: diluted sulphuric acid at 70°C to 80°C for 2 hours at a pH between 1.5 and 2
- chloride roasting on the sulphuric acid leaching residues: 630°C for 2 hours with adding 15 kg/t NaCl
- cyanidation on the chloride roasting residues: 25% solids, a pH of 10 to 11, a leach retention time of 24 hours, and a sodium cyanide dosage of 8 kg/t.

Two tests were conducted under the developed conditions. The test results showed that approximately 90% of the zinc and 97% of the silver were extracted under the test conditions.

BGRIMM

BGRIMM investigated silver extractions from the silver-lead concentrate by using various pre-treatments followed by cyanidation.

The tested technologies included:

- cyanidation with and without regrinding
- oxidation roasting followed by cyanidation
- oxidation roasting and residues regrinding followed by cyanidation
- cyanidation, oxidation roasting, and residues regrinding followed by cyanidation
- NaOH leach, cyanidation, oxidation roasting, and residue regrinding followed by cyanidation.

The test conditions and results are summarized in Table 16.47.

Table 16.47 Silver Extraction from Silver-Lead Concentrates – BGRIMM, 2008

Test Condition	Extraction (% Ag)
Cyanidation	13
Regrinding + Cyanidation	39 to 46
Cyanidation + Residue Regrinding + Cyanidation	45
Oxidation Roasting (500°C to 750°C) + Cyanidation	42 to 61
Oxidation Roasting (500°C to 750°C) + Regrinding + Cyanidation	73 to 87
Cyanidation + Oxidation Roasting (550°C) + Cyanidation	63 to 67
Regrinding + Cyanidation + Oxidation Roasting (550°C) + Cyanidation	73
Cyanidation + Oxidation Roasting (550°C) + Regrinding + Cyanidation	90 to 91
5% NaOH Leach + Cyanidation	18 to 26
Regrinding + 5% NaOH Leach + Cyanidation	46 to 51
5% NaOH Leach + Oxidation Roasting (550°C) + Cyanidation	62
5% NaOH Leach + Cyanidation + Oxidation Roasting (550°C) + Cyanidation	76
Regrinding + 5% NaOH Leach + Cyanidation + Oxidation Roasting (550°C) + Cyanidation	73
5% NaOH Leach + Cyanidation + Oxidation Roasting (550°C) + Regrinding + Cyanidation	92
Regrinding + 5% NaOH Leach + Cyanidation + Oxidation Roasting (550°C) + Cyanidation	78
Regrinding + 5% NaOH Leach + Cyanidation + Oxidation Roasting (550°C) + Regrinding + Cyanidation	92

These test results indicate that concentrate oxidation roasting and regrinding on the roasted residues are key factors for improving silver recovery. The direct cyanidation only recovered approximately 13% of the silver from the silver-lead concentrate. The extraction improved dramatically to over 90% after being treated by oxidation roasting and regrinding.

16.1.7 LOCKED CYCLE PRODUCTS CHARACTERISTICS

The concentrates obtained from the locked cycle tests were assayed for key chemical components. The test results from the 2008/2009 testing are presented in Table 16.48.

Table 16.48 Inductively Coupled Plasma (ICP) Assay Results on Locked Cycle Test Products – PRA, 2009

Elements	Units	Pb Conc (F10 Cyc 4-5)	Pb Conc (F20 Cyc 3-4)	Zn Conc (F10 Cyc 4-5)	Zn Conc (F20 Cyc 3-4)
Pb	%	35.9	43.2	0.43	0.48
Ag	ppm	9,823	8,450	2,053	1,621
Zn	%	4.43	6.37	48.6	46.1
Sb	ppm	17,737	18,006	1,861	13,32
As	ppm	1,374	2,999	<5	430
Bi	ppm	<2	<2	<2	<2
Cd	ppm	422	657	3,892	4,470
Ca	ppm	33,556	40,884	22,807	28,705
Cu	ppm	12,378	12,175	1,103	987
Hg	ppm	<3	<3	<3	4
Ni	ppm	110	153	35	182

The assay data indicates that antimony and arsenic in the silver-lead concentrate may receive penalties from some of smelters. Most of the impurities in the zinc concentrates would be lower than the penalty thresholds, except for caladium.

The tailings obtained from the PRA 2009 locked cycle tests were also assayed for chemical compositions. Table 16.49 shows assay results of the final tailings from Test F10 LCT1.

Table 16.49 ICP Assay on F10 LCT1 Final Tails – PRA 2008/2009

Elements	Units	F10 Final Tail
ST	**%**	0.18
S(-2)	%	0.17
Hg	ppb	<5
Se	%	<0.01
Al	ppm	9,642
Sb	ppm	30
As	ppm	21
Bi	ppm	<2
Cd	ppm	1.8
Ca	ppm	211,354

table continues…

Elements	Units	F10 Final Tail
Cr	ppm	281
Cu	ppm	48
Fe	ppm	4,486
Pb	ppm	191
Mg	ppm	1,638
Ag	ppm	20.1
Zn	ppm	1,188

16.1.8 *Settling Tests*

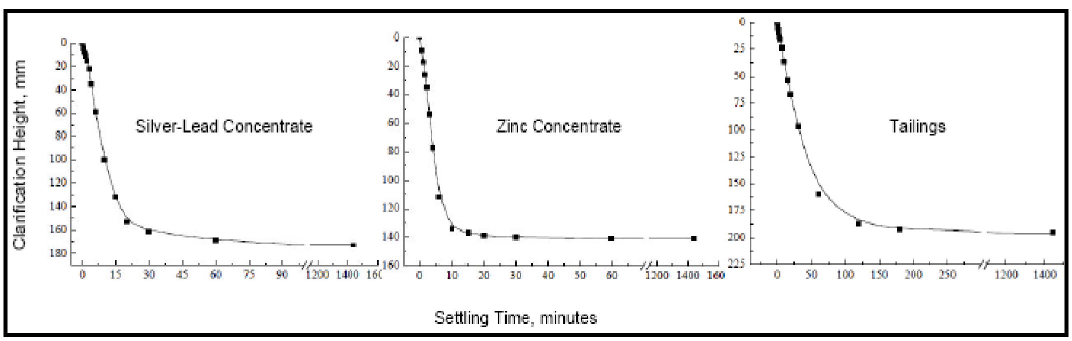
In the 2008/2009 testing program, PRA carried out settling tests on the locked cycle tailings (LCT 2) with or without flocculant addition. The test results are listed in Table 16.50. Tests ST1 and ST2 were on the finer than 45 µm fraction and Tests ST3 and ST4 were on the whole tailings.

Table 16.50 Settling Test Results – PRA, 2009

Test ID	Sample ID	Flocculant (g/t)	Unit Thickening Area (m^2/t/d)
STI	LCT2 Tailings - Fine Portion (-45 µm)	0	2.84
ST2	LCT2 Tailings - Fine Portion (-45 µm)	40	0.66
ST3	LCT2 Tailings - Whole	20	0.09
ST4	LCT2 Tailings - Whole	0	0.66

BGRIMM determined settling curves for the silver-lead concentrate, zinc concentrate, and tailings. The settling curves are presented in Figure 16.11.

Figure 16.11 Settling Curves – BGRIMM, 2008



16.1.9 Environmental Tests

In 2007 and 2009, PRA tested the acid generation potential on the mixed tailings from the locked cycle tests. In 2007, the ABA tests showed that the net neutralization potential (NNP) was about 239 kg/t. The tailings sample from the 2008/2009 testing program produced a slightly higher NNP value (467.1 kg/t). It is concluded by PRA that the samples would be non-acid generating in character.

16.1.10 Recommendations

According to the test results, the conventional differential flotation flowsheet is recommended for mineralization. Gravity separation and cyanidation, with or without pre-treatments, are not recommended for the project.

Further tests to determine the mineralization's resistance to SAG mill grinding are recommended.

It is also recommended to further investigate the smelting terms to optimize the flotation product schedule.

16.2 Process General Description

A 3,000 t/d process plant has been designed for the Fuwan Project to process silver bearing lead and zinc sulphide mineralization. The deposit consists of four major mineralization zones. The main value metals in the mineralization are silver, lead, zinc, and gold. The process plant will operate 330 d/a at an annual process rate of 990,000 t/d and three shifts per day. Overall process plant availability will be approximately 90%.

The ROM from the underground mine will be crushed by an 800 mm by 1,100 mm jaw crusher to 80% passing 150 mm, and then ground to 80% passing 100 µm in a semi-autogenous grinding (SAG)-ball mill-pebble crushing circuit (SABC). The silver, lead, and zinc minerals will be recovered by a conventional differential flotation process:

- silver-lead bulk rougher flotation followed by zinc rougher flotation
- the silver-lead rougher flotation concentrate will be reground and subject to three stages of cleaner flotation
- the zinc rougher flotation concentrate will be upgraded by three stages of cleaner flotation as well without regrinding.

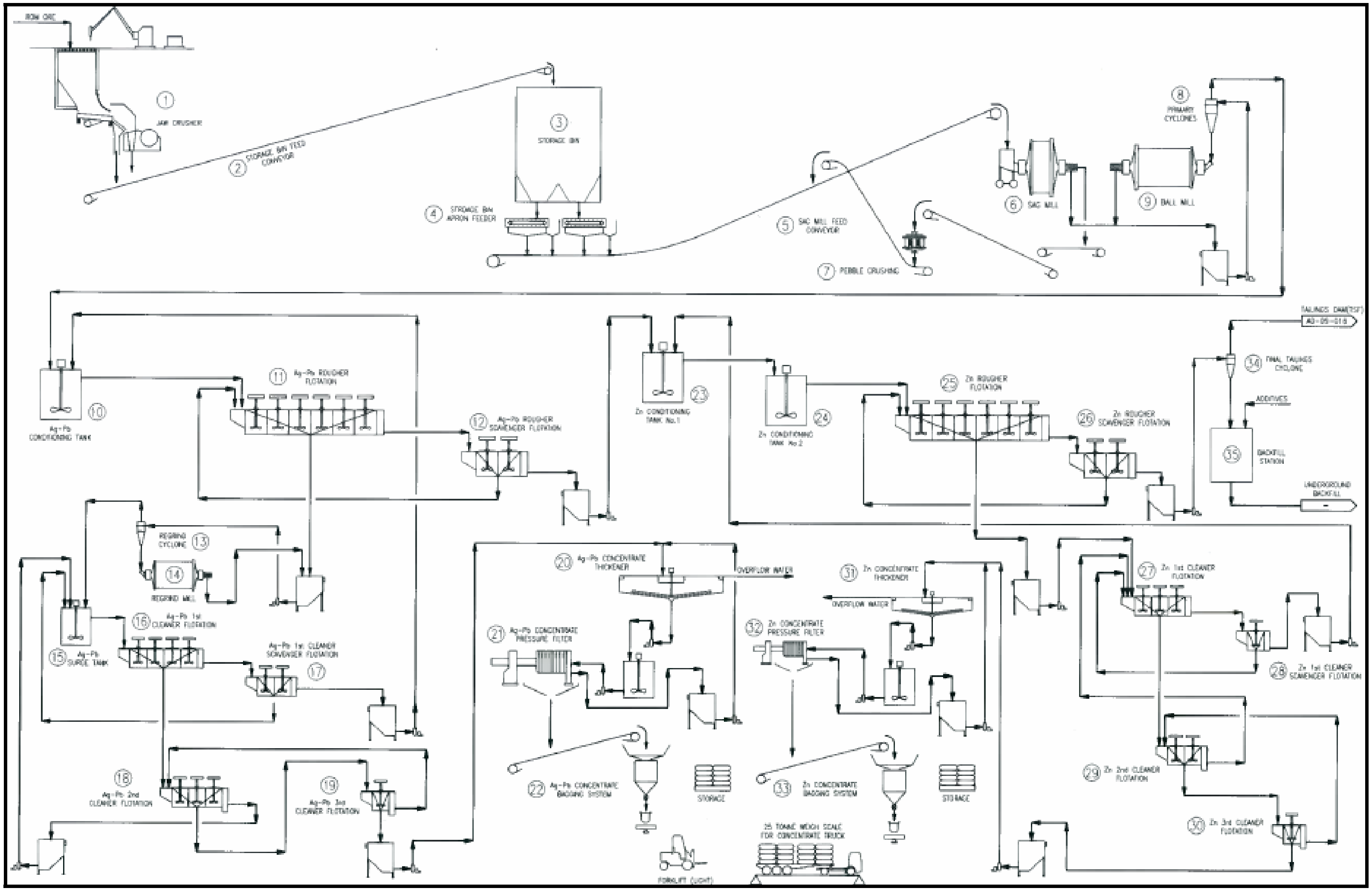
The tailings produced from the zinc rougher scavenger flotation circuit will be sent to the TSF for the storage and to the underground mine for hydraulic backfilling. The produced silver-lead concentrate and zinc concentrate will be thickened and then pressure filtered separately prior to being transported to smelters. Depending on the lead head grade, the silver-lead concentrate may be further processed to produce a silver concentrate and a lead-silver concentrate.

The average dry concentrate production is forecast to be as follows:

- silver-lead concentrate – 15,900 t/a, including:
  - 154,700 kg/a (4,975,000 oz/a) silver
  - 1,600 t/a lead
- zinc concentrate – 9,300 t/a average, including:
  - 4,700 t/a zinc
  - 15,400 kg/a (495,400 oz/a) silver.

A simplified process flowsheet is shown in Figure 16.12.

Figure 16.12 Simplified Process Flowsheet



16.3 PROCESS DESIGN CRITERIA

The process design criteria for this study has been developed based on a process rate of 3,000 t/d or 990,000 t/a. Table 16.51 presents the key design criteria. The detailed design criteria are presented in Appendix A.

Table 16.51 Key Design Criteria

Sources:

1 = Testwork
2 = Others
3 = Mining Plan
4 = Client
5 = Engineering Design
6 = Calculation

Description	Unit	Value	Sources
General			
Type Of Deposit	Silver Ore with Lead and Zinc		
Ore Characteristics			
Specific Gravity	g/cm^3	2.6	1
Bulk Density	t/m^3	1.65	2
Moisture Content	%	5.0	2, 3
Operating Schedule			
Shifts per Day		3	4
Hours per Shift	h	8	4
Hours per Day	h/d	24	4
Days per Year	d/a	365	4
Working Days per Year	d/a	330	4, 5
Plant Availability/Utilization			
Overall Plant Feed (330 Working Days)	t/a	990,000	4
Overall Plant Feed	t/d	3,000	4
Head Grades (life-of-mine)	Ag g/t	188.8	3
	Pb %	0.20	3
	Zn %	0.57	3
	Au g/t	0.15	3
Recovery (life-of-mine)	Ag % (in Ag-Pb Conc)	82.8	1
	Ag % (in Zn Conc)	8.2	1
	Pb %	80.6	1
	Zn %	83.4	1
	Au % (in Ag-Pb Conc)	38.6	1
	Au % (in Zn Conc)	23.0	1
Silver-Lead Concentrate Grade	Ag g/t	9,750	1
	Pb %	9.9	1
	Zn %	2.9	1
	Au g/t	3.5	1

table continues…

Description	Unit	Value	Sources
Zinc Concentrate Grade	Ag g/t	1,650	1
	Pb %	1.3	1
	Zn %	50.0	1
	Au g/t	3.6	1
Silver-Lead Concentrate Mass Recovery	%	1.60	6
Silver-Lead Concentrate Production	t/a	15,900	6
Zinc Concentrate Mass Recovery	%	0.94	6
Zinc Concentrate Production	t/a	9,300	6
Primary Crushing/Primary Grinding			
Crushing Circuit Availability (330 days)	%	68.8	5
Crushing Rate	t/h	181.8	6
Grinding Circuit Availability (365 days)	%	90.4	5
Grinding Rate	t/h	125.0	6
Crusher Type		Jaw Crusher	5
Grinding Type		SABC	5
Crushing Work Index	kWh/t	14.6	1
Bond Rod Mill Work Index (Design)	kWh/t	15.9	1
Bond Ball Mill Work Index (Design)	kWh/t	16.7	1
Abrasion Index		0.357	1
Product Size (P80)	µm	100	5
Primary Grinding Classification		Cyclones	5
Metal Recovery/Concentrate Dewatering			
Flotation Circuit Availability (365 days)	%	90.4	5
Flotation Circuit Process Rate	t/h	125	6
Metal Recovery		Differential Flotation	5
Concentrate Dewatering		High Rate Thickening + Pressure Filtration	5

16.4 PROCESS DESCRIPTION

16.4.1 CRUSHING

The ROM material will be hauled to the primary crushing area by 25-t trucks from the underground mine. The crushing area is closely located at the portal of the decline. The ROM material will be discharged onto a stationary grizzly feeder with 650 mm by 650 mm openings at an average rate of 182 t/h.

The grizzly oversize will be broken by a rock breaker with 37 kW of installed power. The grizzly undersize will report to a 45 m^3 dump hopper and feed to a vibrating grizzly feeder with 150 mm openings. The grizzly screen oversize will be gravity discharged to a jaw crusher where the oversize will be crushed to approximately 80% passing 150 mm. The grizzly screen undersize will join with the jaw crusher discharge and be conveyed to a coarse ore surge bin. The key equipment is as follows:

- one 800 mm x 1,100 mm jaw crusher with an installed power of 132 kW
- one 1,100 mm x 4,900 mm grizzly feeder with an installed power of 15 kW
- one 37 kW rock breaker.

At the decline porter platform, a ROM surge stockpile has been designed with a live capacity of 500 t for temporary storage.

The crusher and grizzly feeder will be housed in an enclosed building. Spray water or fogging water will be required to suppress futile dust generated at the dumping area and various transfer points.

The crushing operation is detailed in Drawing A0-09-002, available in Appendix B.

16.4.2 COARSE ORE HANDLING

The crushed ore will be transported by an 800 mm wide by 117 m long conveyor to an 1,800 m^3 live capacity surge bin. At a rate of 125 t/h, the ore will then be reclaimed by two 1,000 mm wide x 4,000 mm long apron feeders. The reclaimed ore will be conveyed to the main process facility by an 800 mm wide x 56.5 m long conveyor.

A dust collector, located at the top of the surge bin, and a water spray system will minimize dust emissions during coarse ore handling.

The handling system is shown in Drawing A0-09-002, available in Appendix B.

16.4.3 PRIMARY GRINDING

The SABC is designed to grind the crushed ore to a particle size of 80% passing 100 µm, which is required for effective liberation of the target minerals from gangue minerals.

The crushed ore that is reclaimed from the surge bin will feed to a 5.5 m diameter by 3.0 m effective grinding length (EGL) SAG mill. The pulp solid density in the SAG mill will be approximately 72% solids. The SAG mill discharge will feed onto a trommel screen. The designed transfer particle size is 80% passing 2,000 µm. The screen oversize (pebbles) will be transported to a 55 kW pebble crusher via the SAG mill trommel oversize conveyor and the pebble crusher feed conveyor. The SAG mill trommel oversize conveyor will be equipped with two belt magnets to remove any steel material which may damage the crusher. One metal detector will be installed on the crusher feeder conveyor to further protect the crusher as well. If there is any metal detected or the crusher needs maintenance, the pebbles can be bypassed to the SAG mill via a three-way automatic divert chute.

The SAG mill trommel screen undersize will gravity flow into the hydrocyclone feed pumpbox together with the ball mill discharge. The combined slurry will be pumped to two 500-mm hydrocyclones. The hydrocyclone underflow will gravity flow to a 3.96 m diameter by 6.56 m long ball mill. The hydrocyclone overflows will be directed to the silver-lead rougher flotation circuit at approximately 35% solids with a proposed particle size of 80% passing 100 µm. A particle size analyzer, connected with the grinding circuit control system, will monitor the particle size of the hydrocyclone overflow.

The ball mill grinding will be operated at a pulp density of 75% solids with a ball charge of approximately 40% by volume. The ball mill circulating load will be 200% of the feed.

The key equipment will include:

- one 5.5 m diameter by 3.0 m EGL SAG mill powered by a 1,250-kW variable speed motor
- one 3.96 m diameter by 6.56 m long ball mill driven by a 1,650-kW motor
- one 900 mm cone crusher with an installed power of 55 kW
- three 500 mm diameter hydrocyclones (2 operational, 1 standby)
- one particle size analyzer.

The steel grinding media (steel balls) will be added as necessary to maintain sufficient steel load for optimum grinding efficiency.

Lime slurry ($Ca(OH)_2$) will be added to the SAG mill to maintain a pulp pH of approximately 10.5. Zinc sulphate ($ZnSO_4$) will be added into the hydrocyclone feed pumpbox to suppress the zinc minerals.

The process is shown in Drawing A0-09-003, available in Appendix B.

16.4.4 *Silver-Lead Rougher/Scavenger Flotation Circuit*

The silver-lead rougher scavenger flotation circuit consists of one 3.5 m diameter by 3.5 m high conditioning tank, a bank of six 40 m^3 tank cells for rougher flotation, and two 40 m^3 tank cells for scavenger flotation. The hydrocyclone overflow from the grinding circuit (containing 35% solids), together with the 1st cleaner/scavenger flotation tailings, will be conditioned with the silver-lead mineral collectors prior to gravity flowing to the rougher flotation cells. The rougher flotation will produce a silver-lead rougher concentrate, which will be advanced to the silver-lead concentrate regrind circuit. The rougher flotation tailings will be further floated for the silver and lead minerals by the scavenger flotation. The scavenger flotation concentrate will be pumped to the silver/lead rougher flotation feed box while the tailings will leave the silver/lead flotation circuit and report to the zinc circuit.

The reagents used in the circuit will include lime as a pH regulator, ammonium dibutyl dithiophosphate (ADDP) and sodium diethyl dithiocarbamate (SDTC) as silver-lead mineral collectors, and #2 oil (pine oil) as a frother.

The flowsheet is shown in Drawing A0-09-004, available in Appendix B.

16.4.5 *Silver-Lead Rougher Flotation Concentrate Regrind Circuit*

The silver-lead rougher concentrate will be reground prior to being further upgraded. The regrind circuit will consist of:

- one 2.7 m diameter by 3.6 m long ball mill driven by a 400-kW motor
- 1 hydrocyclone cluster of seven 150 mm hydrocyclones (6 operational, 1 standby).

The rougher concentrate will gravity-flow into the hydrocyclone feed pumpbox where the regrind mill discharge will report. The combined slurry will be pumped to the hydrocyclones for classification. The overflow of the hydrocyclones with a particle size of 80% passing 35 µm will report to the silver-lead cleaner flotation circuit. The underflow will gravity-flow to the mill for further grinding. The regrind mill will be operated at a solids density of approximately 65%.

Zinc sulphate and lime will be added to the regrinding circuit to suppress the zinc minerals and pyrite. The particle size of the hydrocyclone overflow will be monitored by the same on-line particle analyzer that is used in the primary grind circuit.

The circuit is shown in Drawing A0-09-004, available in Appendix B.

16.4.6 *Silver-Lead Cleaner Flotation Circuit*

The silver-lead regrind hydrocyclone overflow, together with the 1st silver-lead cleaner scavenger flotation concentrate and the 2nd silver-lead cleaner tailings, will be conditioned with the silver-lead mineral collectors in a 2.5 m diameter by 2.5 m high tank. The pulp will then be fed to a bank of four 16 m^3 flotation tank cells. The resulting tailings will be scavenged in two 16 m^3 flotation tank cells. The scavenger flotation concentrate will be returned to the 1st cleaner flotation conditioning tank while the tailings will leave the cleaner circuit and be sent to the silver-lead rougher flotation conditioning tank.

The 1st silver-lead cleaner flotation concentrate will be further cleaned in the 2nd cleaner flotation stage, which consists of a bank of three 8 m^3 flotation cells. The 2nd cleaner tailings will return to the 1st silver-lead cleaner flotation conditioning tank.

The 2nd cleaner flotation concentrate will be further upgraded in an 8 m^3 flotation cell. The 3rd cleaner flotation tailings will return to the preceding cleaner cell feed box. According to the lead grade in the mill feed, the 3rd cleaner flotation concentrate will be either pumped to the silver-lead concentrate thickener as the final product, or subjected to an additional separation process to produce a silver concentrate and a lead-silver concentrate.

In the latter case, the 3rd cleaner flotation concentrate will be conditioned and then upgraded to produce a lead-silver concentrate (in which the lead grade is acceptable to most of the local smelters) and a silver concentrate.

The slurry pH of the cleaner flotation will be maintained at approximately 10.5.

The same reagents used in the silver-lead rougher flotation circuit will be used in the upgrading processes.

The key equipment in the silver-lead cleaner flotation circuits is summarized as follows:

- one 2.5 m diameter by 2.5 m high conditioning tank
- one bank of four 16 m^3 flotation tank cells for the 1st silver-lead cleaner flotation
- two 16 m^3 flotation tank cells for the 1st silver-lead cleaner scavenger flotation
- one bank of three 8 m^3 flotation cells for the 2nd silver-lead cleaner flotation
- one 8 m^3 flotation cell for the 3rd silver-lead cleaner flotation.

The flowsheet is shown in Drawing A0-09-004, available in Appendix B.

16.4.7 *Zinc Rougher/Scavenger Flotation Circuit*

The silver-lead rougher scavenger flotation tailings will feed the zinc flotation circuit. The tailings will be conditioned with lime to depress pyrite and copper sulphate ($CuSO_4$) to activate zinc minerals at a pH of approximately 11.5. Then, the slurry will be conditioned with sodium isopropyl xanthate (SIPX). The conditioned pulp will be advanced to a bank of six 40 m^3 cells for zinc rougher flotation. The produced zinc rougher flotation concentrate will be further upgraded in the zinc cleaner circuits. The zinc rougher flotation tailings will be further floated in two same size cells to produce the zinc rougher scavenger concentrate and the final flotation tailings. The scavenger concentrate will return to the zinc rougher flotation cell feed box. The tailings will be stored in the TSF or the mined underground by hydraulic backfilling. The main process equipment used in the zinc rougher/scavenger flotation circuit is:

- two 4 m diameter x 4.5 high conditioning tanks
- one bank of six 40 m^3 flotation tank cells
- two 40 m^3 flotation tank cells.

Reagents used in the zinc circuit include $CuSO_4$, lime, SIPX, and #2 oil (as a frother).

The flowsheet is shown in Drawing A0-09-004, available in Appendix B.

16.4.8 ZINC CLEANER FLOTATION CIRCUIT

The zinc rougher concentrate will be further upgraded in three stages of cleaner flotation. The 1st zinc cleaner flotation will be carried out in three 16 m^3 flotation tank cells. The concentrate from the 1st cleaner flotation will gravity flow to the 2nd cleaner flotation cells while the tailings will be further processed to produce a cleaner scavenger concentrate that will be recycled to the 1st zinc cleaner flotation head box. The cleaner scavenger tailings will be pumped to the zinc rougher flotation conditioning tank.

The 1st cleaner flotation concentrate will be further upgraded in the 2nd and 3rd cleaner flotation. The tailings from the cleaner flotation will return to the preceding flotation. The 3rd cleaner concentrate will be advanced to the zinc concentrate thickener as the final zinc concentrate.

The main process equipment used in the upgrading processes is:

- one bank of three 16 m^3 flotation tank cells for the 1st cleaner flotation
- two 16 m^3 flotation tank cells for the 1st cleaner scavenger flotation
- two 8 m^3 flotation tank cells for the 2nd cleaner flotation
- one 8 m^3 flotation tank cell for the 3rd cleaner flotation.

The same reagents used in the zinc rougher flotation will be used in the zinc cleaner flotation circuits. The processes are illustrated in Drawing A0-09-004, available in Appendix B.

16.4.9 CONCENTRATE DEWATERING

Both the silver-lead and zinc concentrates will be thickened in their respective thickeners and further dewatered to a moisture content of 8% by separate pressure filters. The filtered cakes will then be bagged in tote bags and stored in the separate concentrate storage areas prior to being shipped to smelters by trucks. A front-end loader (FEL) will service at the concentrate dewatering and concentrate storage area.

SILVER-LEAD CONCENTRATE DEWATERING

The final silver-lead concentrate will be pumped at a solid flow rate of 2 t/h from the 3rd silver-lead cleaner standpipe to the thickener feed tank, where the concentrate will be mixed with flocculant prior to entering a 8,000 mm diameter high-rate thickener. The thickener underflow, containing approximately 60% solids, will be pumped to a 4,000 mm diameter by 4,000 mm high concentrate stock tank. The tank is capable of holding the concentrate for up to 10 h. The silver-lead thickener overflow will be sent to the SAG mill discharge pumpbox as process make-up water.

The thickened concentrate slurry from the silver-lead concentrate tank will be further dewatered in a 30 m^2 pressure filter. The filtrate will return to the silver-lead concentrate thickener. The filtered cake will be gravity discharged onto a 600 mm-wide conveyer that transfers the concentrate to the silver-lead concentrate automatic bagging system, which will automatically record the concentrate weight.

If the silver-lead concentrate is further processed to produce a silver concentrate and a lead-silver concentrate, the resulting concentrates will be thickened in two separate thickeners. However, the thickened concentrates will be filtrated in the same filter.

The silver-lead concentrate storage area is designed to have a 5-day storage capacity to allow for storage build-up during potential transportation delays.

The key dewatering equipment is as follows:

- one 8 m diameter high-rate thickener
- one 4 m diameter by 4 m high stock tank
- one 30 m^2 pressure filter
- one bagging machine.

The dewatering process is shown in Drawing A0-09-006 available in Appendix B.

ZINC CONCENTRATE DEWATERING

Similarly, the final zinc concentrate will be mixed with flocculant solution and dilution water in the zinc thickener feed tank. The blended slurry will flow into the settling area in the 6.5 m diameter high-rate thickener at a 1.2 t/h solid flow rate. The zinc thickener overflow will be pumped to the zinc circuit and used as launder spray water. The thickener underflow with a density of 60% solids will be stored in a 2,400 mm diameter by 2,400 mm high concentrate slurry stock tank with a 10-h storage capacity. Subsequently, the zinc concentrate will pressure filtrated in a 17 m^2 pressure filter.

The filtered zinc concentrate will be bagged in a similar manner as the silver-lead concentrate previously described. The bagged zinc concentrate will be stored in a dedicated area.

The equipment used in the zinc dewatering will include:

- one 6.5 m diameter high-rate thickener
- one 2.4 m diameter by 2.4 m long stock tank

- one 17 m^2 pressure filter
- one bagging machine.

The dewatering process is illustrated in Drawing A0-09-007, available in Appendix B.

16.4.10 TAILINGS DISPOSAL

The final flotation tailings generated from the zinc rougher scavenger flotation will be directed to the backfill preparation plant at a solid flow rate of 122 t/h. The tailings will be pumped by a variable speed drive to six 250 mm hydrocyclones that classify the tailings into 2 approximately equal weight fractions. The fine fraction material will be stored in the TSF on the surface and the coarse fraction will be used as backfill material for the underground mining process.

The fine fraction or the hydrocyclone overflow will be pumped to the TSF for storage, which is located at approximately 800 m southwest of the process plant. The decant water from the TSF will be reclaimed to the process water tank by reclaim water pumps installed on a barge at the north end of the pond. All the water from the tailings (excluding the void water) will be reused. The estimated reclaimed water will be approximately 305 m^3/h.

The coarse material or the hydrocyclone underflow, containing approximately 50% of the tailings, will gravity-flow to the backfill surge tank. Depending on the backfill requirements, the coarse material will be hydraulically backfilled to the mined stopes with or without adding cement as binding material.

The backfill preparation plant will consist of the following major equipment:

- six 250 mm hydrocyclones (5 operational, 1 standby)
- one 50 t cement storage silo
- one 4 m diameter by 4.5 m high backfill surge tank with mixing blenders.

The tailings disposal flowsheets are shown in Drawings A0-09-008 and A0-09-016, available in Appendix B.

16.4.11 WATER

There will be two separate water supply systems – a fresh water supply system and a process water supply system. Both water supplies are designed to distribute the water by gravity. Water distribution flowsheets and water balance schematics are presented in Drawings A0-09-002 to A0-09-016, available in Appendix B.

Fresh Water

There are two fresh water systems – the process fresh water system and the potable water system. The process fresh water will be from the water treatment plant (WTP), which treats water from the underground mine as well as the mine site runoff water. One 1,000 m^3 fresh water and fire water storage tank has been designed to store the operating fresh water. Fresh water will be mainly utilized for the following:

- fire water for emergency purposes
- make-up water for the mill lubrication cooling water loop (water loss due to evaporation in the grinding mill lubrication cooling water tower)
- reagent preparation
- underground water requirement.

By design, the fresh water tank will be full at all times and will provide at least 2 h of fire water in an emergency.

The potable water will be supplied from the wells drilled within the mine site. The water will be chlorinated and stored in a dedicated holding tank.

Process Water

The process plant will require approximately 330 m^3/h of make-up water. On average, approximately 305 m^3/h of process water will be reclaimed from the tailings pond. The balance water will be from the treated water from the WTP. The silver-lead concentrate thickener overflow will be recycled directly to the primary grind circuit and the water from the zinc thickener will be recycled in the zinc circuit.

The reclaimed water from the TSF and the treated water from the WTP will be directed to a 1,000 m^3 process water tank, which is located at approximately 200 m northeast of the process plant. The process water will be distributed via gravity to the process plant and other service locations. The process water used for pump gland sealing will be filtrated prior to being pumped to various slurry pumps.

16.4.12 Air Service

Two separate air supply systems will service the flotation process and instrumentation:

- low pressure air:
  - provided by air blowers and used for the froth flotation process

- high pressure air:
  - crushing/coarse ore storage bin: the air for the dust suppression (fogging) system and other services will be provided by a separate air compressor
  - plant services: the air will be provided for main plant services (including instrument air) by two separate air compressors; the instrumentation air will be filtered, dried, and stored in a dedicated air receiver
  - concentrate filtering: the air for concentrate filter pressing and drying will be provided by dedicated air compressors
  - mill clutch air: from a dedicated air compressor.

16.4.13 PROCESS EQUIPMENT LIST

A list of major equipment, including equipment descriptions, technical specifications, and power requirements, is included in Appendix C.

16.4.14 QUALITY CONTROL

The intermediate and the final products from the process circuit will be routinely sampled and analyzed in the assay laboratory where the standard assays will be performed. The data obtained will be used for product quality control (QC) and routine process optimization. The mill feed and the tailings will also be collected and subjected to routine assay.

The assay laboratory will be equipped with a full set of assay instruments for base metal analysis and gold and silver assays, including atomic absorption spectrophotometer (AAS), analytical and precision balances, and other determination instruments such as pH and redox potential meters. The assay laboratory will service the mining and exploration departments for routine QC and conduct chemical analyses for overall site environmental monitoring.

The metallurgical laboratory will perform tests for QC (such as grind size measurements) and process optimization. The laboratory will be equipped with crushers, mills, particle size analysis devices, flotation cells, balances, and pH meters.

16.5 PROCESS CONTROL PHILOSOPHY

16.5.1 OVERVIEW

The plant control system will consist of a distributed control system (DCS) with PC-based operator interface stations (OIS) located in the control room within the main process plant. The DCS, in conjunction with the OIS, will perform all equipment and process interlocking, controlling, alarming, trending, event logging, and report generation.

16.5.2 PRIMARY CRUSHER

A local control panel in the primary crushing building will be provided with a single OIS. Control and monitoring of all primary and conveying operations (including discharging onto the crushed ore stockpile) will be conducted from this location.

Control and monitoring functions will include (but are not limited to) the following:

- plugged chute detection at all transfer points
- zero speed switches, side travel switches, and emergency pull cords
- weightometers on selected conveyors to monitor feed rates and quantities
- equipment bearing temperatures and lubrication system status
- vendors' instrumentation packages.

The crushing and coarse ore reclaim area will be equipped with closed circuit television (CCTV) system, which can be monitored from the central control room within the main process plant.

16.5.3 MILL

A central control room in the mill building will be provided with an OIS. Control and monitoring of all processes in the mill building and water treatment plant will be conducted from this location.

Control and monitoring functions will include (but are not limited to) the following:

- grinding mills (bearing temperatures, lubrication systems, clutches, motors, and feed rates)
- particle size monitoring (particle size analyzer for grinding optimization)
- pumpbox, tank, and bin levels
- variable speed pumps
- cyclone feed density controls
- thickeners (drives, slurry interface levels, underflow density, and flocculant addition)
- flotation cells (level controls, reagent addition, and air flow rates)
- pressure filters
- reagent handling and distribution systems
- tailings disposal

- water storage, reclamation, and distribution
- air compressors.

The control philosophy is detailed in the process and instrumentation drawings (P&IDs) A0-13-001 to A0-13-020, available in Appendix D.

16.6 Processing Facilities, Comminution Circuit Design, Online Control System

16.6.1 *Processing Facilities*

The processing plant is designed to process 990,000 t/a of ore and will consist of a primary crushing facility, a coarse core surge bin, a pebble crushing facility, and a process complex.

The primary crushing facility will be housed in a separate building on the surface. The crushed product will be transported to a surge bin via conveyors.

The main process complex houses grinding, flotation, dewatering, concentrate load-out, air supply, reagent make-up, central control, and the assay and metallurgical laboratories. The building will also include management offices and meeting rooms.

The pebble crushing circuit will be housed in a separate building, which is located in the northwest of the main process complex.

The process facility general arrangement drawings are included in Appendix E.

16.6.2 *Comminution Circuit Design*

A conventional comminution circuit, consisting of primary crushing and the SABC circuit, is designed for ore comminution in this study. This design was considered due to the space limitations of the plant site.

16.6.3 *Online Analysis System*

The online x-ray analysis system is recommended and spaces for the system installation have been allowed for in the design. However, based on operator experience, the installation decision should be made after plant commissioning. The capital cost of the system has not been included in the estimate.

16.7 Projected Metal Production

Table 16.52 summarizes the projected annual metal production. The projected metal recoveries and concentrate grades are based on the locked cycle test results and variability test results from various metallurgical testing programs.

On average, the LOM annual production is estimated to be approximately 15,900 t of Ag-Pb concentrate containing 4,975,000 oz Ag and 1,600 t Pb, and 9,300 t of Zn concentrate containing 495,000 oz Ag and 4,700 t Zn.

Table 16.52 Projected Metal Production

Year	Process Rate (000 t)	Head Grade				Recovery								Grade			
						Silver/Lead Concentrate				Zinc Concentrate				Silver/Lead Conc.		Zinc Concentrate	
		Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)	Mass (%)	Ag (%)	Au (%)	Pb (%)	Mass (%)	Ag (%)	Au (%)	Zn (%)	Ag (g/t)	Pb, (%)	Ag (g/t)	Zn (%)
1	990	214	0.17	0.19	0.58	1.55	83.8	40.0	79.9	0.98	8.4	23.0	84.2	11,587	10.0	1,839	50.0
2	990	217	0.17	0.19	0.61	1.55	83.6	40.0	79.9	1.05	8.7	23.0	85.5	11,712	10.0	1,803	50.0
3	990	217	0.16	0.15	0.51	1.04	84.6	40.0	71.1	0.81	7.7	23.0	80.2	17,655	10.0	2,058	50.0
4	990	205	0.16	0.15	0.54	1.05	83.8	40.0	71.4	0.89	8.0	23.0	82.1	16,327	10.0	1,856	50.0
5	990	183	0.15	0.12	0.48	0.98	83.3	40.0	65.0	0.76	7.5	23.0	78.7	15,545	8.0	1,798	50.0
6	990	182	0.16	0.19	0.49	1.49	83.3	40.0	79.1	0.77	7.5	23.0	79.0	10,144	10.0	1,782	50.0
7	990	177	0.15	0.23	0.62	2.00	81.7	40.0	85.3	1.05	8.7	23.0	85.6	7,231	10.0	1,468	50.0
8	990	167	0.14	0.24	0.60	2.08	81.2	40.0	86.0	1.01	8.6	23.0	84.7	6,522	10.0	1,416	50.0
9	990	148	0.08	0.26	0.64	2.31	79.2	20.0	88.0	1.10	8.9	23.0	86.4	5,077	10.0	1,205	50.0
10	208	137	0.08	0.37	0.71	3.38	77.1	20.0	91.0	1.26	9.6	23.0	88.6	3,113	10.0	1,042	50.0
Average	**9,118**	**189**	**0.15**	**0.20**	**0.57**	**1.60**	**82.8**	**38.6**	**80.6**	**0.94**	**8.2**	**23.0**	**83.4**	**9,749**	**9.9**	**1,650**	**50.0**

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 INTRODUCTION

P&E completed five resource estimates for the Fuwan silver deposit in November 2005, November 2006, June 2007, December 2007 and May 2008. Details of the 2005, 2006 and two 2007 estimates are included in reports that are referenced in Section 21.0 (References) of this report. The May 2008 estimate, which is an update of the December 2007 estimate on the basis of infill drilling, forms the basis of the feasibility study that is the subject of this Technical Report. The balance of this section has been prepared by P&E and is reproduced here with minor modifications for clarification, and to omit reference to mineralization other than that contained within the Fuwan deposit.

17.2 DATABASE

Drill data were provided by Minco Silver in the form of Microsoft Access files, Microsoft Excel files (including the 2005-2008 assay laboratory results), drill logs, digital photos of Chinese laboratory assay certificates. For the deposit area in which drilling has been carried out, there were 53 northeast-facing cross sections developed, named 95W to 56E, and with an azimuth of 63°. Of these, 24 sections (from 95W to 43W and from 20E to 56E) were created on a nominal 80 m spacing, while the remaining 29 sections (from 39W to 16E) were created on a nominal 40 m spacing. The effective drillhole spacing on the 40 m sections was a diagonal 60 m x 60 m pattern. A Gemcom database was constructed containing 422 diamond drill holes, of which 231 were utilized in the resource calculation. The remaining drill data fell outside the area that was modelled for this resource estimate and were drilled for delineation of the neighbouring Changkeng Deposit and for other regional exploration targets. Surface drill hole plans are shown in Appendix F.

The database was verified in Gemcom with minor corrections made to bring it to an error-free status. The assay table of the database contained 14,109 Ag, 13,994 Au, 2,814 Pb, and 2,770 Zn assays. Data are expressed in metric units and grid coordinates are in a Chinese UTM system.

17.3 Data Verification

Verification of assay data entry was performed on 9,313 assay intervals for Ag, Au, Pb, and Zn. Very few minor data errors were observed and corrected, with the overall impact to the database being negligible. The 9,313 verified intervals were verified with original assay laboratory certificates from the 757 Team assay certificates, Process Research Associates Ltd. (PRA) laboratory certificates from Kunming, China, and ALS Chemex laboratory certificates from Vancouver, Canada. The checked assays represented 85% of the data to be used for the resource estimate and approximately 66.0% of the entire database.

17.4 Domain Interpretation

Domain boundaries were determined from lithology, structure, and grade boundary interpretation from visual inspection of the drill hole sections. Six domains were developed and referred to as Zone 1 to Zone 6. The updated domains were created with computer screen digitizing on drill hole sections in Gemcom by P&E. The outlines were influenced by the selection of mineralized material above 40 g/t Ag that demonstrated a lithological and structural zonal continuity along strike and down dip. In some cases, mineralization below 40 g/t Ag was included for the purpose of maintaining zonal continuity. Smoothing was utilized to remove obvious jogs and dips in the domains and incorporated a minor addition of inferred mineralization. This exercise allowed for easier domain creation without triangulation errors from solids validation.

On each section, polyline interpretations were digitized from drill hole to drill hole but were not extended more than 80 m into untested territory. Minimum constrained true width for interpretation was 1.5 m. The interpreted polylines from each section were “wireframed” in Gemcom into 3-dimensional (3D) domains. The resulting solids (domains) were used for statistical analysis, grade interpolation, rock coding, and resource reporting purposes (Appendix F).

17.5 Rock Code Determination

The rock codes used for the resource model were derived from the mineralized domain solids. A list of the rock codes used is outlined in Table 17.1.

Table 17.1 Rock Codes

Rock Code	Description
0	Air
10	Fuwan Zone 1
20	Fuwan Zone 2
30	Fuwan Zone 3
40	Fuwan Zone 4
50	Fuwan Zone 5
60	Fuwan Zone 6

17.6 COMPOSITES

Length weighted composites were generated for the drill hole data that fell within the constraints of the above-mentioned domains. These composites were calculated for Ag, Au, and (wherever present) Pb and Zn over 1.0 m lengths starting at the first point of intersection between the assay data hole and hanging wall of the 3D zonal constraint. The compositing process was halted upon exit from the footwall of the aforementioned constraint. Un-assayed intervals were treated as null data. Any composites calculated that were less than 0.35 m in length were discarded so as to not introduce any short sample bias in the interpolation process. The composite data were transferred to Gemcom extraction files for the grade interpolation as an X, Y, Z, Ag, Au, Pb, Zn file for each domain.

17.7 GRADE CAPPING

Grade capping was investigated on the raw assay values in the database within each domain to ensure that the possible influence of erratic high values did not bias the database. Extraction files were created for constrained Ag data within each mineralized domain. The Au, Pb, and Zn data were sparse in some domains resulting in their being treated as one group. From these extraction files, lognormal histograms were generated (Appendix F).

Table 17.2 Ag Grade Capping

Domain	Capping Value Ag (g/t)	No. of Assays Capped	Raw COV*	Capped COV	Cumulative Percent for Capping
Zone 1	1,500	2	1.85	1.41	99.4%
Zone 2	3,000	3	2.45	1.86	99.7%
Zone 3	3,000	3	1.96	1.79	99.4%
Zone 4	2,500	1	2.00	1.96	99.5%
Zone 5	500	2	1.14	0.82	90.5%
Zone 6	1,000	1	2.31	1.57	96.9%

* COV = Coefficient of Variation.

Table 17.3 Au Grade Capping

Domain	Capping Value Au (g/t)	No. of Assays Capped	Raw COV	Capped COV	Cumulative Percent for Capping
All Zones	3.5	10	2.19	1.80	99.5%

Table 17.4 Pb Grade Capping

Domain	Capping Value Pb (%)	No. of Assays Capped	Raw COV	Capped COV	Cumulative Percent for Capping
All Zones	4.0	6	2.23	2.02	99.4%

Table 17.5 Zn Grade Capping

Domain	Capping Value Zn (%)	No. of Assays Capped	Raw COV	Capped COV	Cumulative Percent for Capping
All Zones	8.0	9	2.06	1.77	99.1%

17.8 VARIOGRAPHY

Variography was attempted on the constrained domain composites with reasonable success. Zones 1 to 4 allowed the creation of discernable variograms while the high variability and relatively low data population density of the remaining zones did not yield discernable variograms, resulting in their classifications defaulting to the inferred category (Appendix F).

17.9 BULK DENSITY

The bulk density used for the creation of a density block model was derived from 696 field measurements taken by Minco. In addition, representative samples obtained by P&E of the mineralized zones of the deposit were utilized. The average bulk density was calculated to be 2.62 t/m^3 from client samples and 2.64 t/m^3 from P&E samples.

17.10 BLOCK MODELLING

The Fuwan silver deposit block model has 38,720,000 blocks that were 3 m in the X direction, 6 m in the Y direction, and 3 m in the Z direction. There were 440 columns (X), 550 rows (Y), and 160 levels (Z). The block model was rotated 63° clockwise. Separate block models were created for rock type, density, percent, Ag, Au, Pb, and Zn.

The percent block model was set up to accurately represent the volume and subsequent tonnage that was occupied by each block inside the constraining domain. As a result, the domain boundaries were properly represented by the percent model ability to measure infinitely variable inclusion percentages within a particular domain.

The Ag, Au, and (where applicable) Pb and Zn composites were extracted from the Microsoft Access database composite table into separate files for each mineralized zone. Inverse distance squared interpolation was utilized for all domains and elements. The first interpolation pass was utilized for the Zone 1 to 4 indicated classification interpolation, while a second pass was used on all other domains for the inferred classification. Grade blocks were interpolated using the parameters outlined in Table 17.6 (Appendix F).

Table 17.6 Block Model Interpolation Parameters

Profile	Dip Dir.	Strike	Dip	Dip Range	Strike Range	Across Dip Range	Max. No. per Hole	Min. No. Sample	Max. No. Sample
Zone 1 Ag									
Indicated	153°	63°	0°	70	70	70	2	3	12
Inferred	153°	63°	0°	200	200	200	2	1	12
Zone 2 Ag									
Indicated	153°	63°	0°	85	80	80	2	3	12
Inferred	153°	63°	0°	200	200	200	2	1	12
Zone 3 Ag									
Indicated	153°	63°	0°	75	60	60	2	3	12
Inferred	153°	63°	0°	200	200	200	2	1	12

table continues…

Profile	Dip Dir.	Strike	Dip	Dip Range	Strike Range	Across Dip Range	Max. No. per Hole	Min. No. Sample	Max. No. Sample
Zone 4 Ag									
Indicated	153°	63°	0°	60	60	60	2	3	12
Inferred	153°	63°	0°	200	200	200	2	1	12
Zone 5, 6, A Ag									
Inferred	153°	63°	0°	200	200	200	2	1	12
All Zones Au									
Indicated	153°	63°	0°	50	50	50	2	3	12
Inferred	153°	63°	0°	200	200	200	2	1	12
All Zones Pb									
Indicated	153°	63°	0°	60	60	60	2	3	12
Inferred	153°	63°	0°	200	200	200	2	1	12
All Zones Zn									
Indicated	153°	63°	0°	50	50	50	2	3	12
Inferred	153°	63°	0°	200	200	200	2	1	12

17.11 Resource Classification

During the classification interpolation search ellipsoid first pass for Zones 1 to 4, 283,321 grade blocks were coded as indicated while 133,954 were coded as inferred from those zones and the remaining zones. All classifications were determined from the Ag search ellipsoid passes due to the predominance of Ag in the potential economics of the deposit (Appendix F).

17.12 Resource Estimate

The resource estimate was derived from applying Ag cutoff grades to the block model and reporting the resulting tonnes and grade for potentially mineable areas. The following calculations demonstrate the rationale supporting the Ag cutoff grade that determines the potentially economic portion of the mineralized domains:

- Ag Price: US$13.69/oz (24 month trailing average price April 30, 2008)
- Au Price: US$710/oz (24 month trailing average price April 30, 2008)
- Pb Price: US$1.01/lb (24 month trailing average price April 30, 2008)
- Zn Price: US$1.48/lb (24 month trailing average price April 30, 2008)
- Mining Cost (2,500 t/d): US$12/t mined
- Process Cost (2,500 t/d): US$11.50/t milled
- Ag Flotation Recovery: 97%

- Au Flotation Recovery: 50%
- Pb Flotation Recovery: 85%
- Zn Flotation Recovery: 85%
- Concentration Ratio: 25.6:1
- Ag Smelter Payable: 95% (includes refining charges)
- Au Smelter Payable: 85% (includes refining charges)
- Pb Smelter Payable: 80% (includes refining charges)
- Zn Smelter Payable: 80% (includes refining charges)
- Smelter Treatment Charges: US$60/t (US$60/25.6 = US$2.34/t mined)
- Concentrate Shipping: US$5.00/t (US$5/25.6 = US$0.20/t mined)
- General and Administration (G&A): US$5.50/t mined.

The above data were derived from the September 2008 technical report titled "Preliminary Economic Assessment – Fuwan Silver Deposit" as well as Chinese and other worldwide mining operations similar to Fuwan.

Credits for the resource grades for Au (0.22g/t), Pb (0.22%), and Zn (0.63%) are as follows:

Au	=	[(50% Recovery x 85% Payable x $710/oz)/31.1035 g/oz] x 0.22g/t	=	$2.13/t
Pb	=	70% Recovery x 80% Payable x 22.046 lb/t x $1.01/lb x 0.22%	=	$2.74/t
Zn	=	70% Recovery x 80% Payable x 22.046 lb/t x $1.48/lb x 0.63%	=	$11.51/t
Total Payable Contribution for Au, Pb, and Zn			**=**	**$16.38/t mined**

The difference of $15.16/t ($31.54/t costs - $16.38/t Au, Pb, Zn revenue) must be made up by the Ag revenue to determine the Ag cutoff grade for the resource estimate. Therefore, the Ag cutoff grade for this resource estimate is calculated as follows:

[($15.16)/[($13.69/oz Ag x 97% Recovery x 95% Payable)/31.1035] =
37.38 g/t (use 40 g/t Ag)

The resulting resource estimate can be seen in Table 17.7.

Table 17.7 Resource Estimate at 40 g/t Ag Cutoff Grade

Resource Area & Classification	Tonnes	Ag (g/t)	Ag (oz)	Au (g/t)	Pb (%)	Zn (%)
Fuwan Permit Indicated	13,948,000	188	84,268,000	0.17	0.20	0.56
Fuwan Permit Inferred	10,241,000	171	56,147,000	0.26	0.26	0.72

Notes:

- Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
- The quantity and grade of reported inferred resources in this estimation are conceptual in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.
- The mineral resources in this press release were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council December 11, 2005.

Table 17.8 Fuwan Resource Estimate Sensitivity

Cutoff Ag (g/t)	Tonnes	Ag (g/t)	Ag (oz)	Au (g/t)	Pb (%)	Zn (%)
500	920,276	728	21,536,590	0.29	0.47	1.03
450	1,178,628	672	25,470,856	0.28	0.47	1.05
400	1,523,953	616	30,168,512	0.27	0.45	1.02
350	2,522,091	521	42,218,318	0.25	0.43	0.96
300	3,526,760	465	52,738,343	0.25	0.40	0.93
250	4,955,039	409	65,224,996	0.24	0.38	0.91
200	8,063,589	339	87,821,212	0.23	0.35	0.86
175	9,807,317	312	98,311,469	0.23	0.33	0.81
150	12,108,144	283	110,276,536	0.23	0.30	0.77
125	15,098,811	254	123,439,820	0.23	0.28	0.73
100	18,808,680	226	136,842,595	0.23	0.27	0.70
90	20,604,746	215	142,345,479	0.23	0.26	0.69
80	22,186,184	206	146,651,204	0.22	0.25	0.68
70	23,847,683	197	150,668,384	0.22	0.25	0.66
60	25,383,443	189	153,868,663	0.22	0.24	0.65
50	26,667,970	183	156,539,389	0.22	0.23	0.64
40	**27,265,407**	**179**	**156,786,499**	**0.22**	**0.23**	**0.64**
30	27,877,481	176	157,712,654	0.22	0.23	0.62
20	28,319,227	174	158,070,672	0.22	0.23	0.61
0.1	28,916,634	170	158,276,988	0.22	0.23	0.60

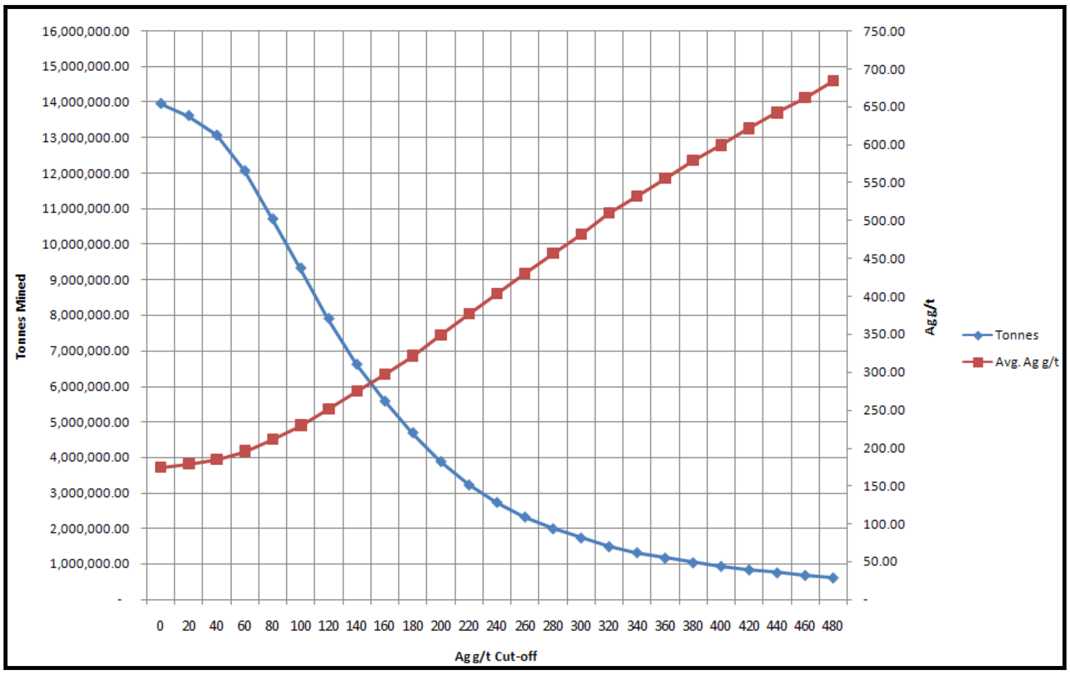
Table 17.8 was derived by applying a series of increasing Ag cutoffs to the domains that constrain the mineralization. These domains were developed utilizing an

approximate 40 g/t Ag cutoff grade (see Section 17.4), which was found to be the grade at which the domains demonstrate the optimal lithological and zonal continuity along strike and across section. This set of domains was subsequently used during the application of all cutoff grades within Table 17.8.

17.12.1 Grade Tonnage Curve

The grade-tonnage curve in Figure 17.1 shows the relationship between the silver cutoff grade, and the tonnages and grade at Fuwan silver deposit. The grade tonnage curve is generated from the global resource estimate using silver cutoff grades ranging from 0 to 480 g/t Ag.

Figure 17.1 Resource Grade Tonnage Curve



17.13 Confirmation of Estimate

As a test of the reasonableness of the resource estimate, the block model was queried at a 0.1 g/t Ag cutoff with blocks in all classifications summed and their grades weight averaged. This average is the average grade of all blocks within the mineralized domains. The values of the interpolated grades for the block model were compared to the length weighted capped average grades and average grade of composites of all samples from within the domain. The results are tabulated in Table 17.9.

Table 17.9 Comparison of Weighted Average Grade of Capped Assays and Composites with Total Block Model Average Grade

Category	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
Capped Assays	174	0.23	0.29	0.76
Composites	174	0.22	0.24	0.64
Block Model	171	0.24	0.25	0.65

Table 17.9 shows the average grade of all the Ag, Au, Pb, and Zn blocks in the constraining domains to be similar to the weighted average of all capped assays and composites used for grade estimation. In addition, a volumetric comparison was performed with the block model volume of the model versus the geometric calculated volume of the domain solids:

- Block Model Volume = 10,685,528 m^3
- Geometric Domain Volume = 10,685,165 m^3
- Difference = 0.003%.

17.14 Reserve Estimate

The resource estimate provided by P&E classified the resources for the Fuwan Zones 1 to 4 as indicated and inferred. Only indicated mineral resources as defined in NI 43-101 were used to establish the probable mineral reserves. No reserves were categorized as proven.

Some of the wireframes for the resources provided geologically irregular shapes in the indicated resources in the May 2007 block model that would be difficult to mine. The mine design battery limit was to accept the resource estimate and interpretation at face value and prepare a mine design around it.

It will be essential for infill drilling to be undertaken during the basic engineering and detailed mine design phases for the production of detailed stope and development layouts for construction and mining. It is also Wardrop's opinion that there appeared to be no marker horizons to follow high grade zones within the limestone. It will be difficult if not impossible to follow economic mineralization visually during mining and grade control sampling will be used. Pre-production infill drilling will be essential to define the true orebody outlines ahead of development and stoping.

In order to obtain the mining permits in China, it is necessary to use an official Chinese resource estimate prepared according to Chinese codes. The Chinese resource may not be the same as the NI 43-101 resource used for this study.

17.14.1 *FUWAN PROBABLE MINERAL RESERVE ESTIMATE*

The probable mineral reserve estimate is 9,117,980 t at 189 g/t Ag, 0.146 g/t Au, 0.196% Pb, and 0.566% Zn. The reserve estimate is listed by zone in Table 17.10. Figure 17.2 and Figure 17.3 compare the difference between the resource and reserve estimates.

Table 17.10 Probable Reserve Estimate Summary

Zone	Tonnes	Ag (g/t)	Ag (M oz) In Situ	Au (g/t)	Pb (%)	Zn (%)
1	1,327,580	186	7.9	0.180	0.064	0.324
2	4,806,443	192	29.7	0.167	0.177	0.568
3	2,451,699	192	15.1	0.105	0.257	0.636
4	532,259	150	2.6	0.068	0.421	0.822
Probable Mineral Reserve	**9,117,980**	**189**	**55.3**	**0.146**	**0.196**	**0.566**

Figure 17.2 Resource Model – Plan View at 40 g/t Silver Cutoff

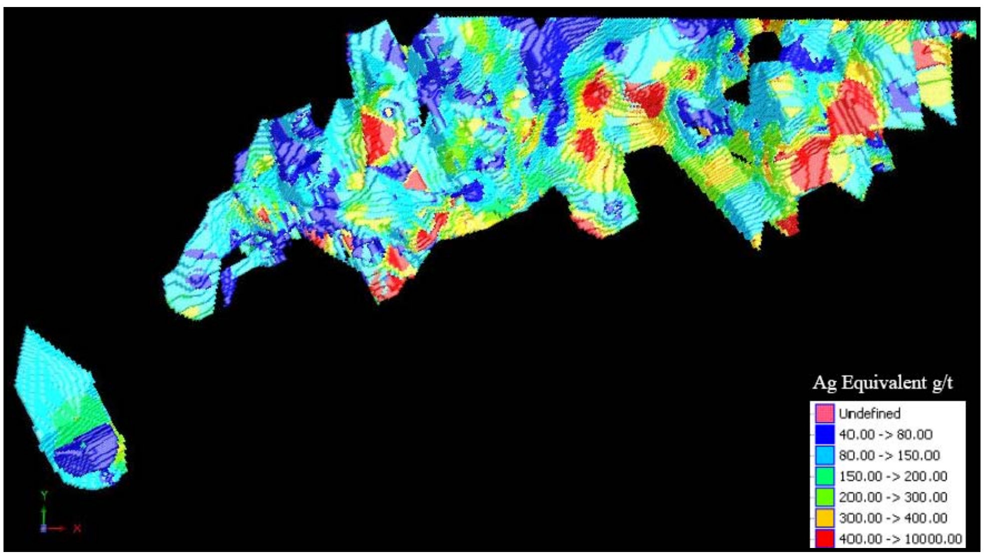
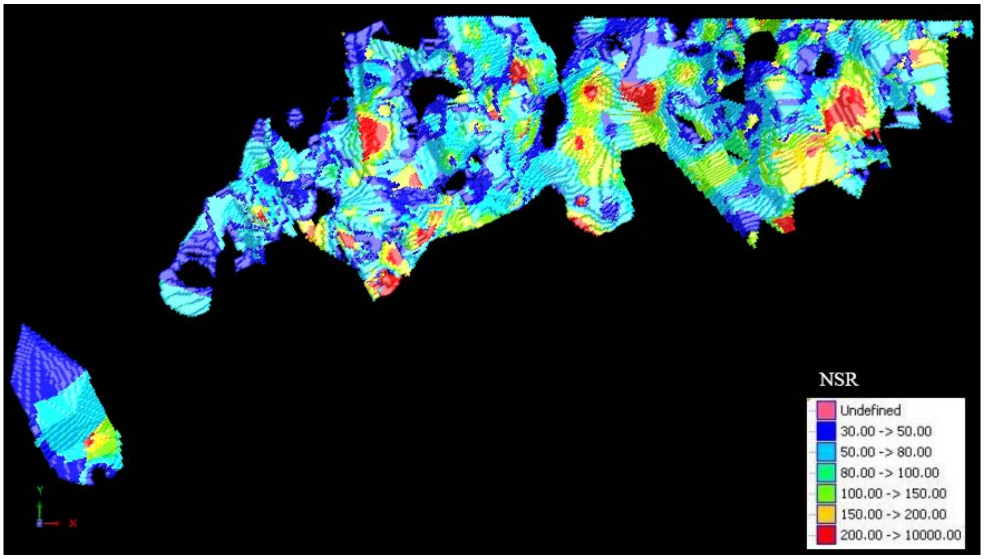


Figure 17.3 Reserve Model – Plan View at US$37.13/t NSR Cutoff (119 g/t Ag)



17.14.2 CUTOFF GRADE CALCULATIONS

METAL PRICES

The metal prices used for the resource and reserve estimates are listed in Table 17.11. The Wardrop pricing used for the reserve estimate was the 3-year historical average price from the London Metal Exchange (LME) as at September 1, 2008.

Table 17.11 Metal Prices

Metal	P&E Resource	Wardrop Reserve
Silver (US$/oz)	13.69	13.00
Gold (US$/lb)	710.00	688.00
Lead (US$/lb)	1.01	0.88
Zinc (US$/lb)	1.48	1.28

NET SMELTER RETURN CUTOFF GRADE

The reserve estimate was determined using a NSR cutoff grade of US$37.13/t. The operating costs used for the reserve estimate are listed in Table 17.12.

Table 17.12 Operating Costs for Reserve Estimate

Area	Unit Cost (US$/t)
Mine	18.41
Process	10.77
Tailings Management	1.30
Surface Services	0.79
G&A	5.86
Total	**37.13**

The offsite charges are included in the metals payable, which is incorporated into the following NSR formula:

NSR = (0.31 * in-situ g/t Ag grade) * (6.07 * in-situ g/t Ag grade) * (311.66 * in-situ % Pb grade) * (1,563.94 * in-situ % Zn grade)

This is a post-recovery NSR formula. Smelting and metallurgical recoveries are included.

Table 17.13 illustrates the NSR calculation for an arbitrary tonne of rock from one of the blocks in the block model.

Table 17.13 Example NSR Calculation

Metal	Ore (t)	In-situ Grade	Contained Metal	Payable Recovery	Recovered Metal	Metal Price (US$)	NSR (US$/t)
Silver	1	189 g/t	6.07 oz	74.8%	4.54 oz	13.00/oz	59.00
Gold	1	0.15 g/t	0.005 oz	27.5%	0.001 oz	688.00/oz	0.89
Lead	1	0.20%	4.3 lb	16.0%	0.69 lb	0.88/lb	0.61
Zinc	1	0.57%	12.5 lb	55.3%	6.90 lb	1.28/lb	8.84
Total							**US$69.34/t**

SILVER CUTOFF GRADE

The silver cutoff grade can be calculated as follows:

Silver Cutoff Grade (g/t) =
Operating Cost (US$/t) / Unit Metal Value (US$/oz Au)

Operating Cost =
Mine, Process, Tailings Management, Surface Services, and G&A costs

Unit Metal Value =
Sale Price (US$/oz, lb) x Refining and Process Recovery (%)

Therefore:

Silver Cutoff Grade (g/t) =
37.13 US$/t / (13.00 US$/oz * 74.8 %)

Silver Cutoff Grade (g/t) =
119 g/t Ag

It should be noted that the silver cutoff grade does not include any metal credits from the gold, lead, or zinc.

17.14.3 Reserve Estimate Methodology

The following steps were undertaken in Surpac™, Gemcom's mine design software, and Microsoft Excel to determine the reserve estimate:

- The block model was constrained to show blocks with an NSR of greater than or equal to US$37.13/t.
- The geological wireframe and the blocks above US$37.13/t NSR were used to outline the stopes.
- Stopes were designed to a minimum mining height of 2 m.
- Stope outlines were interrogated to determine:
  - ore tonnage and grade
  - internal dilution (blocks with an NSR of less than US$37.13/t)
- Depending on the mining method, each stope block was allocated:
  - extraction factor
  - external dilution
  - fill dilution
- Finally, a mining recovery factor was applied.

Table 17.14 illustrates the effect of each step in the reserve estimate determination.

Table 17.14 Resource to Reserve

	Tonnes	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
Undiluted Tonnes and Grade (from block model report)	8,696,005	220	0.17	0.23	0.67
Add 7% Internal Dilution (from block model report)	9,339,190	205	0.16	0.21	0.62
Apply 97% Mining Extraction Rate (weighted by mining method)	9,113,667	205	0.16	0.21	0.62
Add 8% External Dilution (weighted by mining method)	9,831,541	190	0.15	0.20	0.58
Add 3% Fill Dilution (weighted by mining method)	10,108,862	185	0.15	0.19	0.56
Apply 95% Mining Recovery Rate	9,603,419	185	0.15	0.19	0.56
Fully Diluted Tonnes and Grade	**9,603,419**	**185**	**0.15**	**0.19**	**0.56**

After recovery, dilution and extraction factors were applied and 7 blocks were found to be marginal. The blocks were removed from the final reserve as they could not support the cost of development to reach them. The total tonnes and grade of blocks removed was 484,704 t at 113 g/t Ag, 0.123 g/t Au, 0.088% Pb, and 0.409% Zn.

Table 17.15 shows the final reserve used for mine scheduling.

Table 17.15 Final Reserve Used for Mine Scheduling

Zone	Tonnes	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
1	1,327,580	186	0.18	0.06	0.32
2	4,806,443	192	0.17	0.18	0.57
3	2,451,699	192	0.11	0.26	0.64
4	532,259	150	0.07	0.42	0.82
Probable Mineral Reserve	**9,117,980**	**189**	**0.15**	**0.20**	**0.57**

18.0 ADDITIONAL REQUIRMENTS FOR DEVELOPMENT AND PRODUCTION PROPERTIES

18.1 MINING OPERATIONS

18.1.1 *GEOTECHNICAL EVALUATION*

A geotechnical evaluation of the underground mine design was made to assess the stability of the excavations and to estimate excavation dimensions and support requirements.

Minco provided the borehole photographs and geological logs containing some geotechnical information that serves as the basis for the project's rock mass quality estimate. Wardrop revised the geotechnical database to include rock mass parameters; additional data was gathered by visual inspection of core photographs and from the RQD provided by Minco, which was confirmed or corrected by Wardrop.

Wardrop geotechnically logged 26 holes from the core photos. No directional drilling information was available from these holes. Minco logs provided the length, core recovery, RQD (partially), and geological description. Wardrop also logged five geotechnical verification holes on site which are described in this section.

The collected data was classified using Barton's Tunnelling Quality Index (Q) (Barton et. al., 1976) to allow rock mass classification.

The proposed excavation dimensions and support types are based on rock mass quality, structural features, and mining methods.

BOREHOLE DESCRIPTION

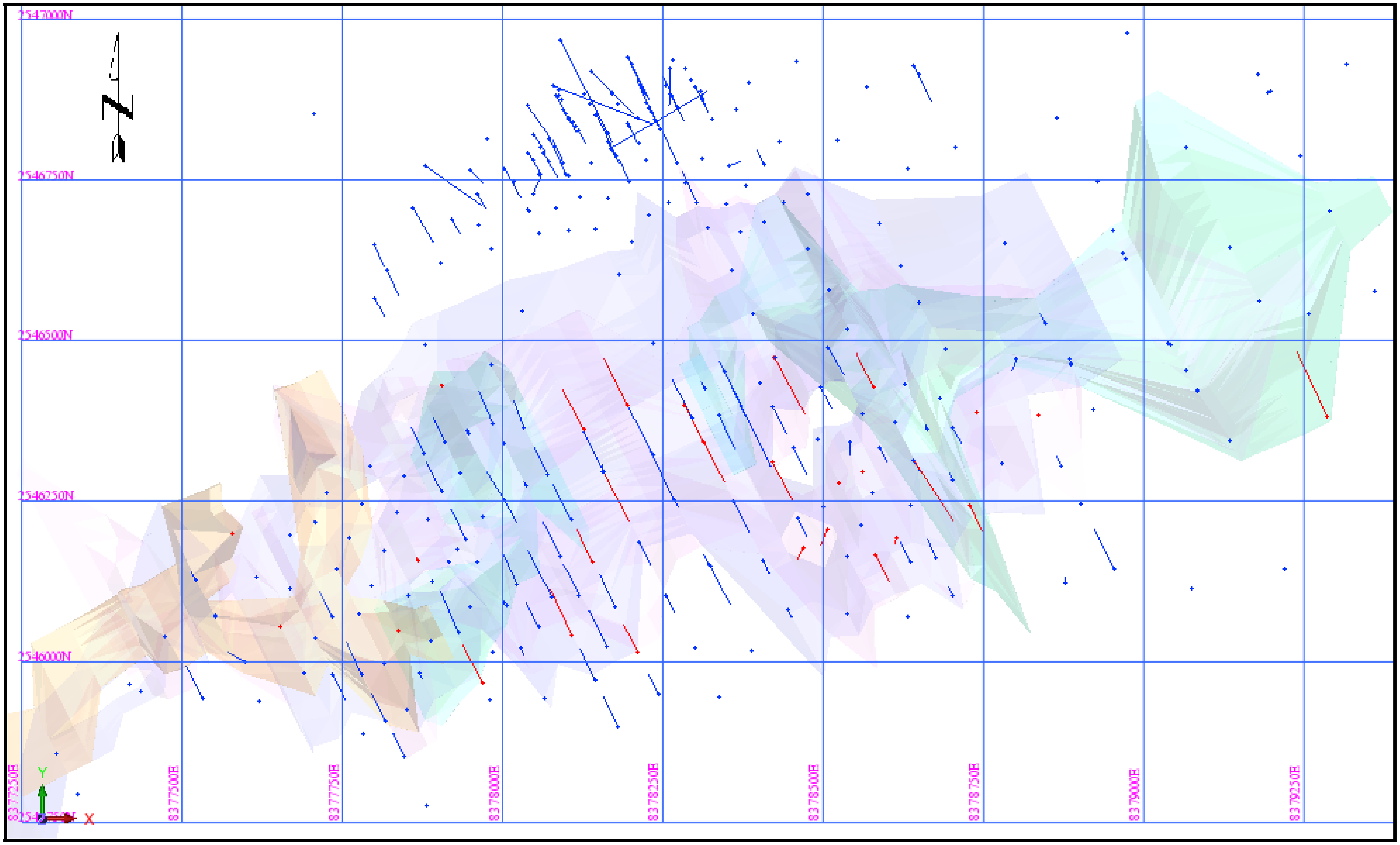
Minco provided basic information of their drilling database on length, core recovery, and geological description in terms of Fuwan geological rock codes. Wardrop determined the geotechnical parameters based on a visual assessment of core photographs of the diamond drill holes. The boreholes were not logged in their entirety but from 10 m into the hanging wall to 10 m into the footwall (if available).

Table 18.1 lists the boreholes evaluated for rock mass classification.

Table 18.1 Borehole Details

Hole ID	From (m)	To (m)
FW0242	72.1	98.0
FW0243	91.6	126.2
FW0245	81.8	169.8
FW0248	66.8	100.0
FW0005	225.4	262.4
FW0006	120.0	231.4
FW0007	85.0	128.0
FW0008	156.5	194.3
FW0009	161.7	248.3
FW0012	118.2	213.2
FW0016	240.4	280.3
FW0017	87.3	171.7
FW0022	225.2	246.4
FW0023	308.0	329.0
FW0025	105.5	208.2
FW0026	275.1	316.4
FW0027	244.3	265.7
FW0032	80.0	146.3
FW0041	160.3	232.6
FW0044	195.0	217.3
FW0050	238.4	315.3
FW0052	211.4	289.4
FW0055	184.9	245.8
FW0066	36.8	67.7
FW0101	230.5	271.1
FW0124	117.8	145.4
FW0140	109.1	132.5
FW0152	179.7	218.4
FW0220	78.1	134.9
FW0247	94.6	193.3

Figure 18.1 Plan of Diamond Drill Holes (Red Traces = Logged Holes)



The geotechnical data collected were classified using Barton's Q, which led to a rock mass quality determination and support estimation. The Q classification parameters are based on the block size, shear strength, and active stress.

Barton's Q is defined as:

$$\boxed{Q = \left(\frac{RQD}{Jn}\right) x \left(\frac{Jr}{Ja}\right) x \left(\frac{Jw}{SRF}\right)}$$

Where:

RQD/Jn	=	Block Size
Jr/Ja	=	Inter-block Strength
Jw/SRF	=	Active Stress
RQD	=	Rock Quality Designation
Jn	=	Joint Set Number
Jr	=	Joint Roughness Number
Ja	=	Joint Alteration Number
Jw	=	Joint Water Reduction Factor
SRF	=	Stress Reduction Factor

The factors for Jw and SRF in the Q for analysis are set at 1.0, which is also called the Modified Tunnelling Index (Q') when these parameters are set to 1. This is assuming that the joints have non-existent to minor inflow of less than 5 L/min for the Jw factor. The SRF factor is estimated based on medium stress environment where joints are moderately clamped but not overly stressed.

The rock mass classification proposed by Barton et al. to describe the rock conditions based on Q classification is defined in Table 18.2.

Table 18.2 Rock Mass Quality Categories Based on Q

Rock Mass Quality	Range of Q
Exceptionally Poor	>0.01
Extremely Poor	0.01 to 0.1
Very Poor	0.1 to 1
Poor	1 to 4
Fair	4 to 10
Good	10 to 40
Very Good	40 to 100
Extremely Good	100 to 400
Exceptionally Good	400 to 1,000

INTERPRETED GEOTECHNICAL CONDITIONS

Rock Mass Classification

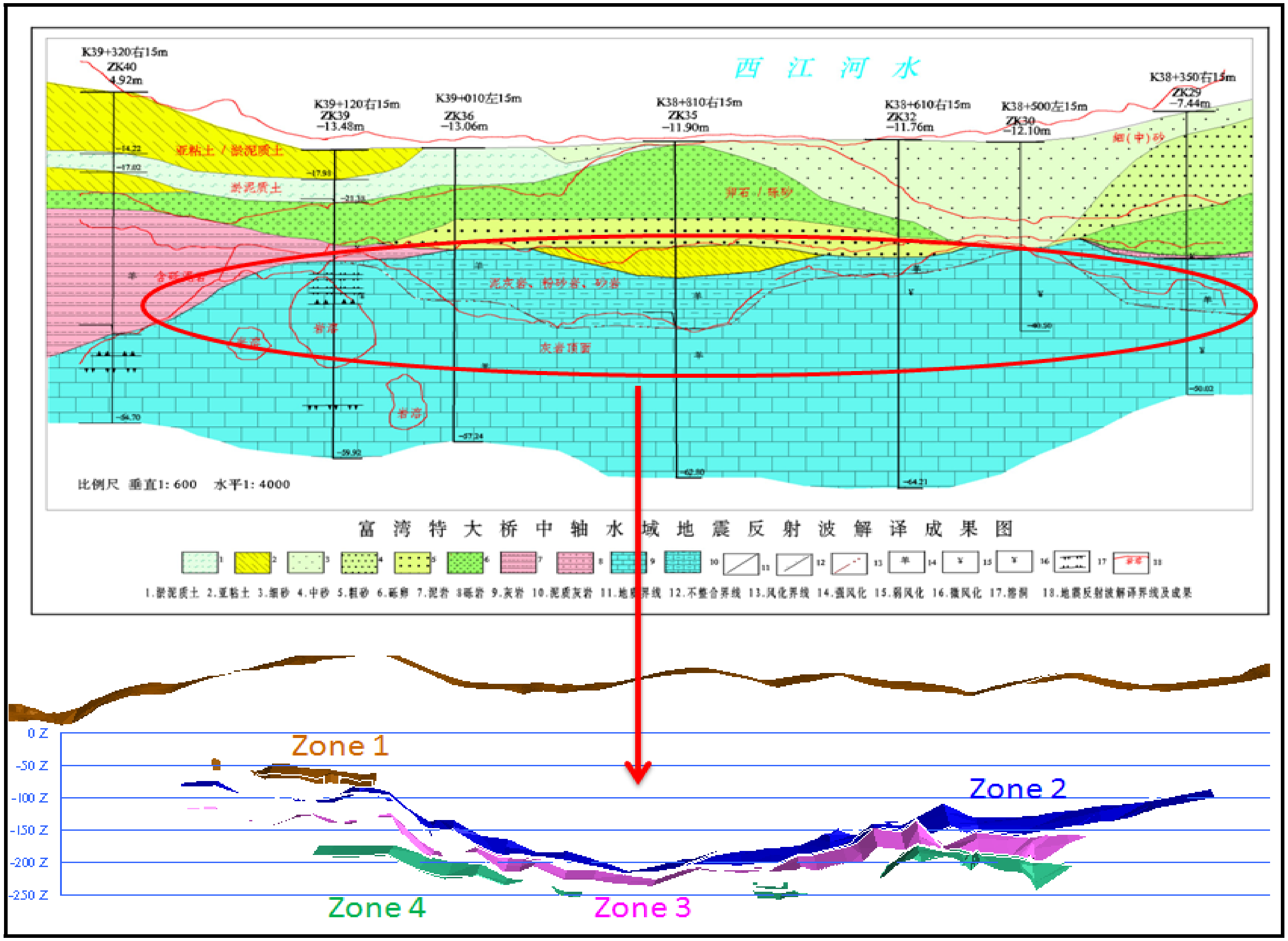
Rock mass classifications were carried out for each of the four mining zones (Figure 18.2).

Q' numbers for different rock types (Table 18.3) are presented in the following sections. The main geotechnical challenge will be the control of exposures of the "unconformity" or sheared contact between the sandstones and limestones. This feature will likely be encountered in the hanging walls of Zones 1 and 2 where the mineralization is concentrated close to the unconformity.

Table 18.3 Rock Codes

Abbreviation	Rock Type	Abbreviation	Rock Type
ALLU	Alluvium	CONG	Conglomerate
AQLO	Silicified limestone	COSI	Siltstone with conglomerate
AQOO	Silicified rock	COST	Thin bad coal bedded
AQSO	Silicified sandstone	DQSA	Feldspar and quartz sandstone
AQTB	Silicified tectonic breccia	LIME	Limestone
ARLI	Pelitic limestone	MUDS	Mudstone
ARSH	Pelitic shale	PEST	Pelitic siltstone
ASAN	Pelitic sandstone	PILE	Stockpile by people
BAQO	Brecciaed silicified rock	QARS	Quartz sandstone
BILI	Bioclastic limestone	RESC	Residual Soil
BREC	Breccia	RFGO	Fault gouge
BRLI	Brecciaed limestone	RFZO	Fault zone
BRSA	Brecciaed sandstone	SAAN	Siliceous sandstone
CAMU	Calc-mudstone	SABE	Sandstone and conglomerate
CAPO	Sediment in cave	SAND	Sandstone
CASA	Calcareous sandstone	SGCO	Greywacke conglomerate
CAVE	Cave	SGRE	Greywacke
CLAY	Clay	SHAL	Shale
CLGR	Greywacke with conglomerate	SILT	Pelitic siltstone
CLSH	Shale with carbonaceous	SLSH	Silty shale
CLSI	Clay with some silicified rock	SMUD	Silty mudstone
CLST	Calc-siltstone	TEBR	Tectonic breccia
CMUD	Mudstone with carbonaceous		

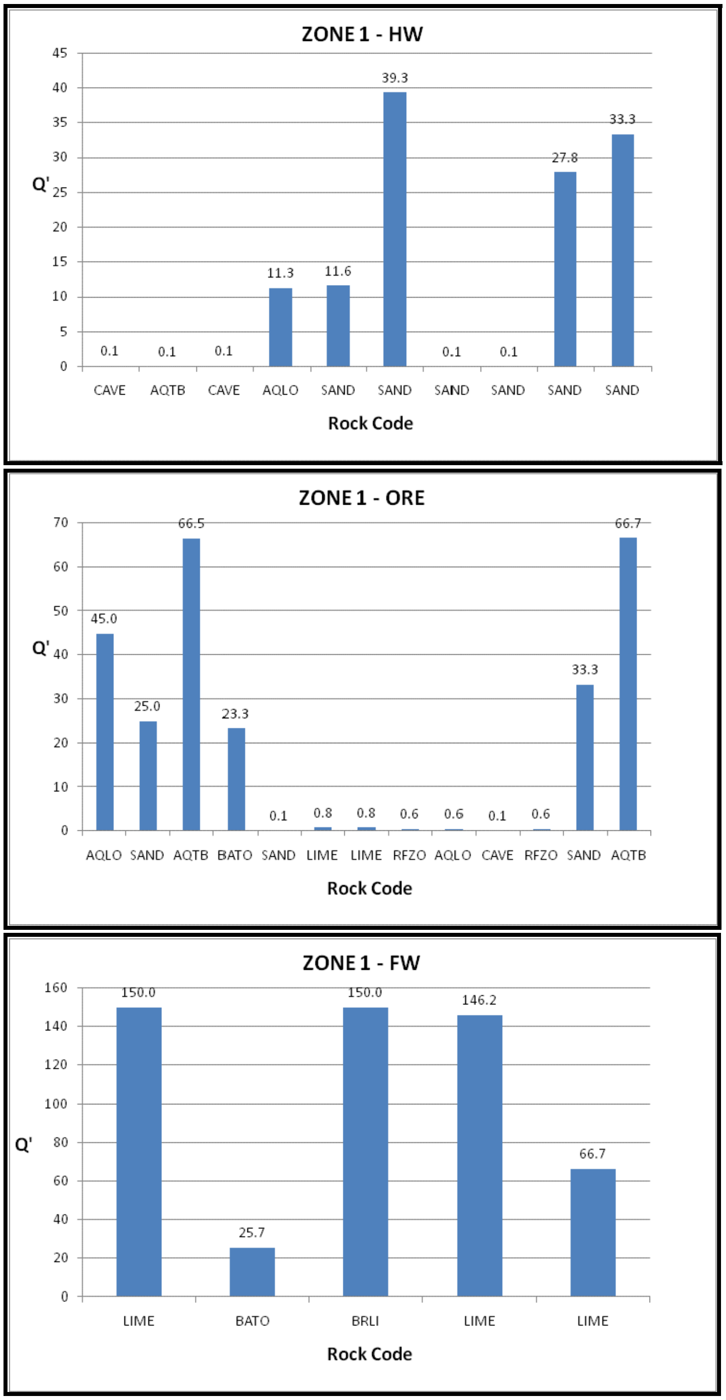
Figure 18.2 Four Ore Zones – Long Section (Looking North)



ZONE 1

The Q' for the Zone 1 hanging wall (HW), foot wall (FW), and ore zone (Ore) are shown in Figure 18.3.

Figure 18.3 Zone 1 Rock Mass Q' Values



A summary geotechnical evaluation of Zone 1 is shown in Table 18.4.

Table 18.4 Zone 1 Rock Mass Quality Summary

	Weighted Average Q'	Min Q'	Max. Q'	Average Rock Mass Quality
Hanging Wall	11.4	0.1	39	Good
Ore	36.2	0.1	67	Good
Footwall	119.1	26	150	Extremely Good

The unconformity will be exposed in some stope backs in this zone with a Q of 0.1 (extremely poor rock).

ZONE 2

The Q' for the Zone 2 hanging wall (HW), foot wall (FW), and ore zone (Ore) are shown on Figure 18.4.

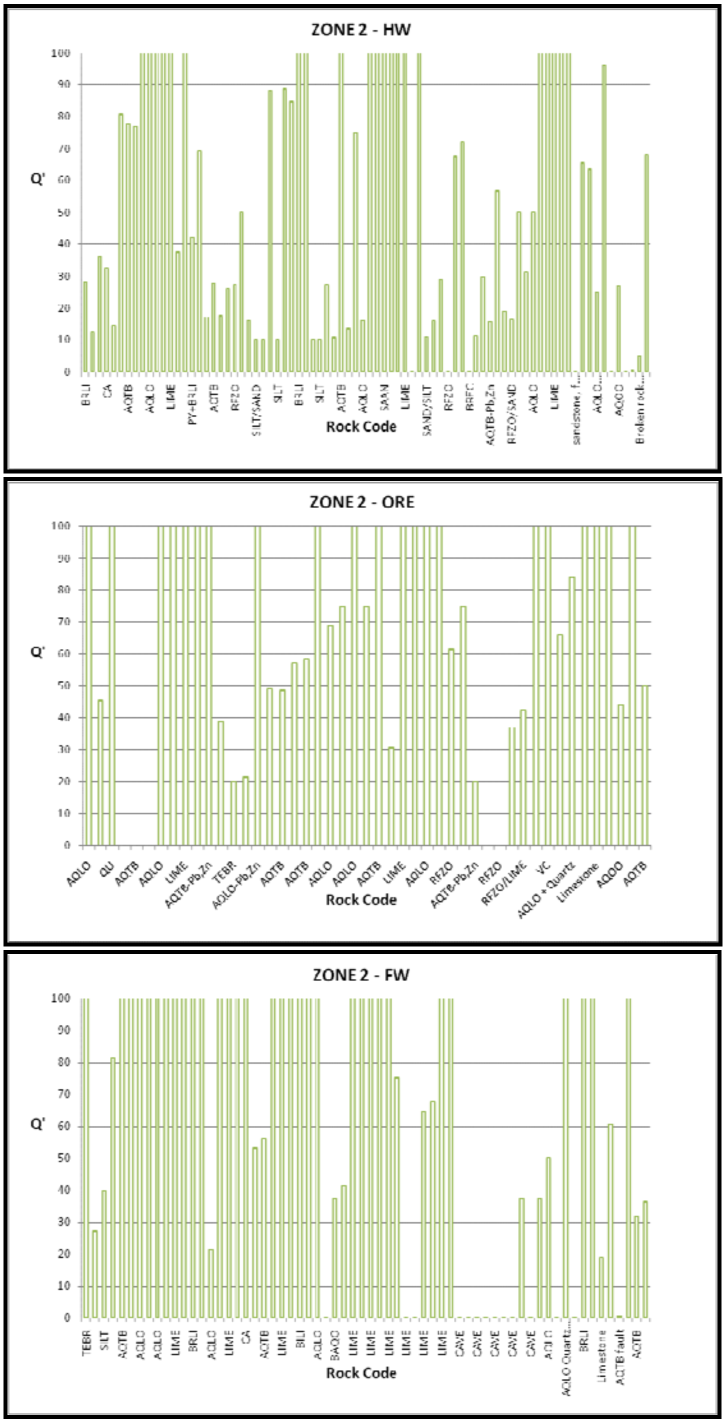
A summary geotechnical evaluation of Zone 2 is shown in Table 18.5.

Table 18.5 Zone 2 Rock Mass Quality Summary

	Weighted Average Q'	Min. Q'	Max. Q'	Average Rock Mass Quality
Hanging Wall	69.5	0.1	200	Very Good
Ore	157.8	0.1	400	Extremely Good
Footwall	148.9	0.1	400	Extremely Good

The unconformity will be exposed in some stope backs in this zone with a Q of 0.1 (extremely poor rock).

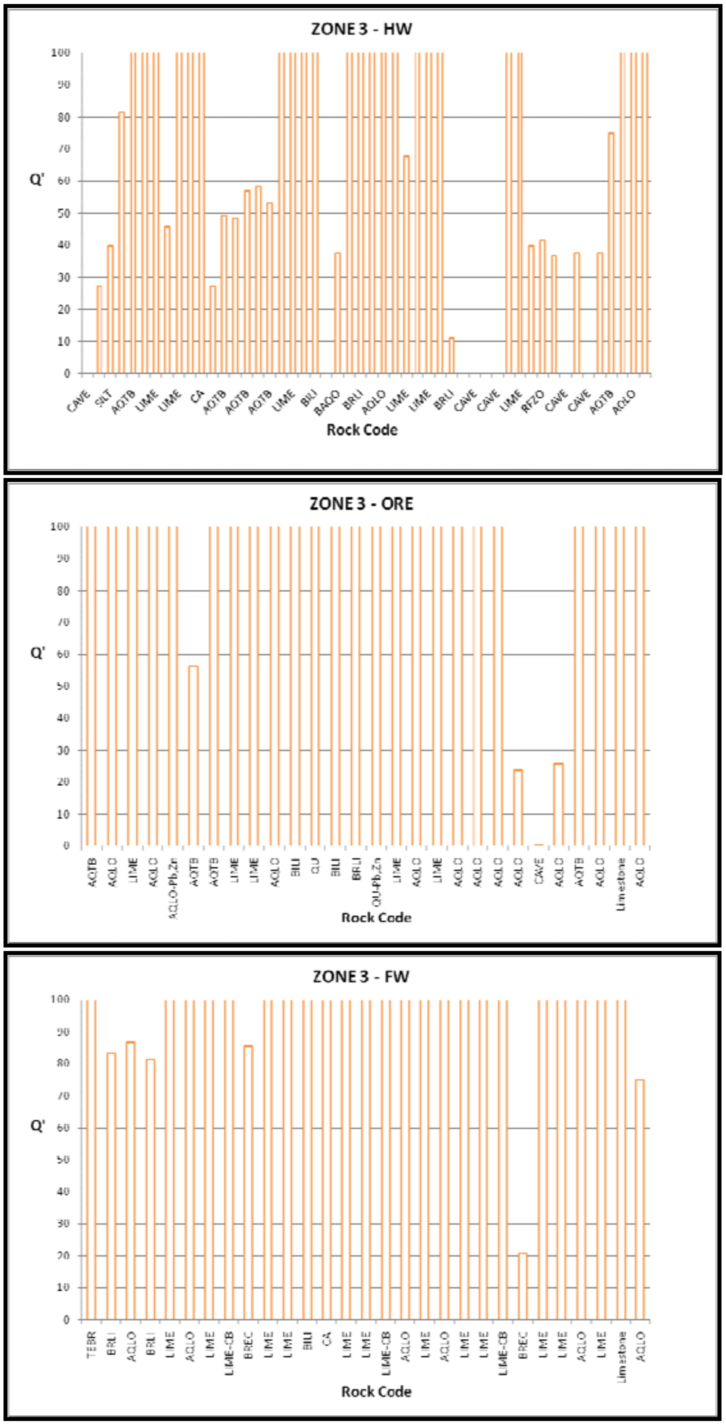
Figure 18.4 Zone 2 Rock Mass Q' Values



ZONE 3

The Q' for the Zone 3 hanging wall (HW), foot wall (FW), and ore zone (Ore) are shown on Figure 18.5.

Figure 18.5 Zone 3 Rock Mass Q' Values



A summary geotechnical evaluation of Zone 3 is shown in Table 18.6.

Table 18.6 Zone 3 Rock Mass Quality Summary

	Weighted Average Q'	Min. Q'	Max. Q'	Average Rock Mass Quality
Hanging Wall	141	0.1	300	Extremely Good
Ore	187	0.1	300	Extremely Good
Footwall	181	21	300	Extremely Good

The unconformity will probably not be exposed in stope backs of this zone.

ZONE 4

The Q' for the Zone 4 hanging wall (HW), foot wall (FW), and ore zone (Ore) are shown in Figure 18.6.

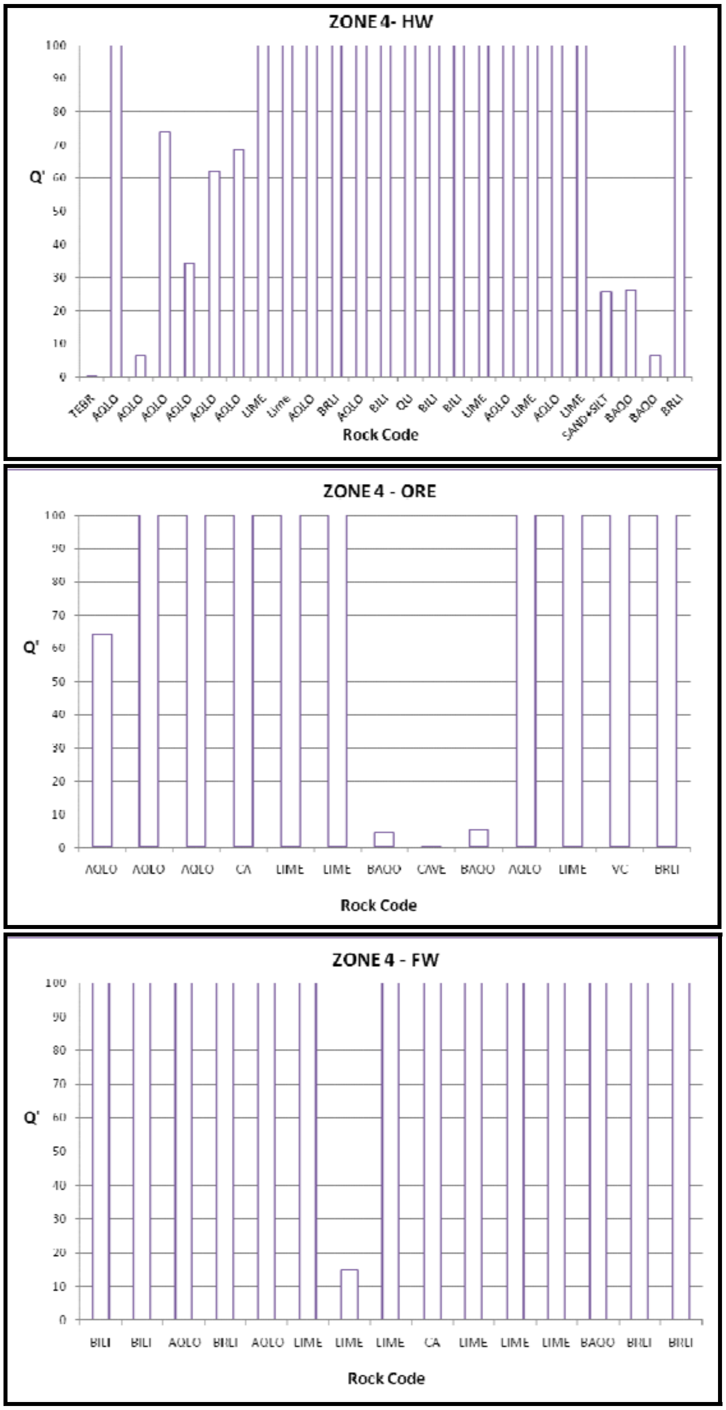
A summary geotechnical evaluation of Zone 4 is shown in Table 18.7.

Table 18.7 Zone 4 Rock Mass Quality Summary

	Weighted Average Q'	Min. Q'	Max. Q'	Average Rock Mass Quality
Hanging Wall	158.5	0.1	300	Extremely Good
Ore	116.2	0.1	300	Extremely Good
Footwall	158.3	15	300	Extremely Good

The unconformity will probably not be exposed in stope backs of this zone.

Figure 18.6 Zone 4 Rock Mass Q' Values



Overall Rock Mass Classification

The main ore zones (Zone 1 and Zone 2) were combined to represent the ore zone, hanging wall, and foot wall values were calculated accordingly. Table 18.8 shows the generic rock mass quality.

Table 18.8 General Rock Mass Quality Summary (Dry)

	Weighted Average Q'	Min. Q'	Max Q'
Hanging Wall (Argillaceous, Siltstone Sandstone)	70	0.1	200
Ore	153	0.1	400
Foot Wall (Limestone)	181	21	300

It should be noted that the values in Table 18.8 assume dry or minor groundwater inflow (5 L/m). In medium or large inflow cases, Table 18.9 and Table 18.10 must be used, respectively.

Table 18.9 General Rock Mass Quality Summary (Medium Inflow)

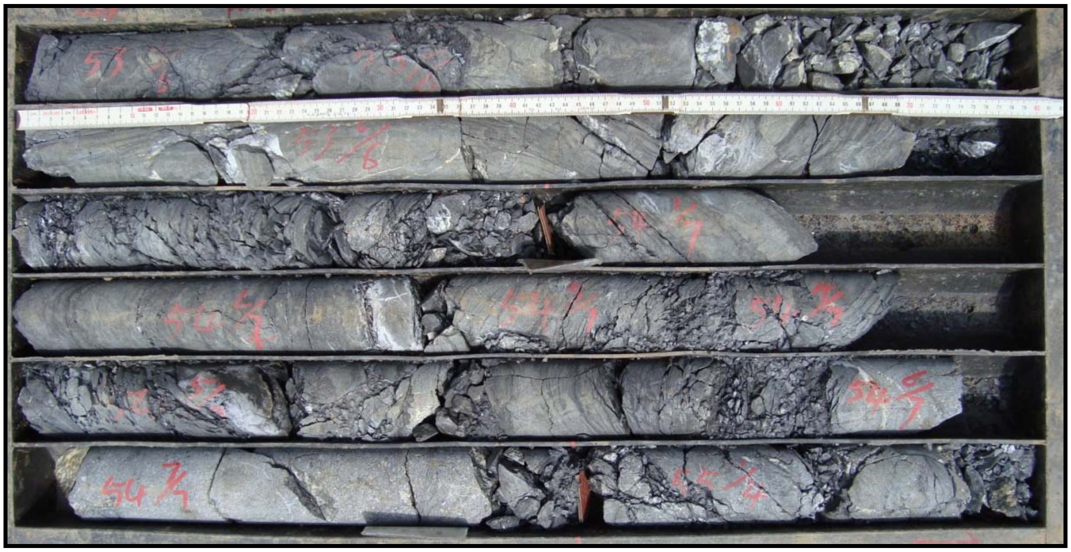
	Weighted Average Q'	Min. Q'	Max Q'
Hanging Wall (Argillaceous, Siltstone Sandstone)	46	0.1	66
Ore	101	0.1	132
Foot Wall (Limestone)	119	6.3	90

Table 18.10 General Rock Mass Quality Summary (Large Inflow)

	Weighted Average Q'	Min. Q'	Max Q'
Hanging Wall (Argillaceous, Siltstone Sandstone)	23	0.1	132
Ore	50	0.1	264
Foot Wall (Limestone)	60	1.3	198

It should be noted that the unconformity zone (HW and Ore) is a variable thickness sheared and weathered zone (Figure 18.7).

Figure 18.7 Typical Unconformity Zone



Variability of Q' values in the unconformity zone must be taken into account during the design stage.

18.1.2 HYDROGEOLOGY

As part of the Fuwan Silver Project Feasibility Study being undertaken by Wardrop for Minco, a hydrogeological review of the available information being collected by various consultants was undertaken. The hydrogeological investigations undertaken to date can be considered part of a cumulative process to determine the potential impacts of groundwater infiltration into future proposed underground mine development. This data review was supplemented by an August 2008 visual inspection of the project site, and in person meetings with some of the parties involved in the exploration work at the site.

RELIANCE ON OTHER EXPERTS

The intrusive hydrogeological investigations undertaken to date were performed by various local consultants retained by Minco and others, and subject to review by Wardrop as part of the NI 43-101 process required in Canada. The data compiled has been reviewed with respect to methodology, data completeness, calculations and general conclusions. On-site observation of the data collection efforts (e.g., borehole logging, well record development, water level measurements, etc.) was not performed to confirm the raw data, but review of the information provided appears to be reasonable and complete and so is assumed to be correct.

Sources of data used in the preparation of this report include the following:

- SRK Consulting Engineers report to Minco Silver Corporation, *Preliminary Economic Assessment – Fuwan Silver Deposit,* SRK Project Number SCN087, Beijing China, September 2007.
- SRK Consulting Engineers report to Minco Silver Corporation, *Fuwan Silver Hydrogeological Scoping Level Assessment,* SRK Project 1US036.000, Vancouver, November 2007.
- 757 Exploration Team, Report to Minco Silver Corporation, *Report on Hydrogeological, Geotechnic and Environmental Geological Exploration for the Fuwan Deposit, Foshan City Guandong Province,* December 2007.
- 757 Exploration Team, Raw data from Chankeng Groundwater Pumping Test, Test Well Ck2038, September 15 to Oct 4, 2008.
- China Nerin Engineering Ltd., Summary of Hydrogeological Conditions of the Deposit, September 2009.
- Various borehole logs and cross sections as prepared by or on behalf of Minco Silver Corporation.

HYDROGEOLOGICAL DESCRIPTION

Area of Investigation

The Fuwan silver deposit exploration area is located throughout a small forest covered hill area approximately 1 to 2 km west and northwest of Fuwan Town in Guangdong Province. The silver deposit exploration area is bordered on the north by a separate mineral exploration area, referred to as the Chankeng exploration area. The hill feature runs southwest to northeast and is produced by a subsurface bedrock rise. The local topography varies from about 1 m above sea level (ASL) along the river to the east of the hill feature, to around 130 m at the top of the hills. The proposed mine development plan comprises an approximately 0.6 km^2 underground mining area, to be excavated at depths of 100 to 250 m below sea level, and various associated surface features including a mill plant site, tailings management facility (TMF) and waste rock storage area. A conceptual view of the proposed mine site development is presented in Figure 1 in Appendix G.

The proposed underground mining area is located about 500 m west of the Xijiang River, the major surface water system in the area, and 500 m south of the Changkeng ditch, which drains surface water from the area on the north side of the hills. Between the hills and the Xijiang River is a relatively flat flood plain which has been extensively developed for fish farming in excavated ponds or agricultural use. The local topography, vegetation and general extent of surface development can be seen in Figure 2 in Appendix G. Photographs of the subject area are provided in Appendix G.

Since the objective of the hydrogeological assessment was to evaluate potential groundwater inflows into the underground mine workings, the surface development areas are not included in the scope of intrusive investigations and data analyses as discussed in the following sections.

General Geology

Detailed descriptions of the site geology has been previously presented in a variety of publications on the project site so only a general overview has been prepared, as follows.

The proposed mining area is underlain by a series of sedimentary deposits consisting primarily of limestone and sandstone units segregated by an apparent unconformity and intersected by a series of faults. Surficial deposits consist of alluvial deposits of clay, silt and sand, 8 to 35 m thick over the flood plain area east of the hills. The hill slopes themselves are covered by 1 to 20 m of clay and sand overburden derived from erosion of the underlying sedimentary bedrock. These overburden deposits are Quaternary aged and labeled “Q” on local geological maps.

The underlying bedrock consists of an upper Triassic aged sandstone and siltstone unit overlying the surface of the steeply dipping carbonate deposits. On the geological maps for the area, the Triassic unit is labeled with the prefix “T_3” followed by an additional subunit descriptor. Similarly the carbonate deposits have the prefix “C_1“ followed by additional subunit qualifiers. For the purposes of this hydrogeological assessment, identification of the individual subunits is not required.

Limestone is near the surface on the northwest side of the hill, while the central and eastern portion of the hill are composed of sandstone to depths of 100 to 200 m below sea level. Along the eastern edge of the exploration area, adjacent to the Xijiang River, an additional Cretaceous aged sandstone, siltstone and mudstone unit (labeled with the prefix “K”) overlies the Triassic sandstone unit. The distribution of these geological units across the area is shown in Figure 3 in Appendix G.

These geological units area have been subjected to displacement along a series of fault zones. The primary faults identified include the following.

- Chankeng Fault – Primary fault trending southwest to northeast north of Fuwan exploration area.
- FX1 – A low angle regional feature, this fault is associated with the unconformity between the Cretaceous carbonate rocks and the overlying Triassic sandstone unit.
- F2 – Poorly defined northwest and east trending fault trace to the northeast of the exploration area, potentially extending out under the Xijiang River.
- F3 (Xiawan Fault) – Southwest to northeast trending fault located in east portion of the exploration area.

Geophysical exploration of the area has also identified a series of additional smaller possible fault lines in the area, some of which are noted in the geological mapping for the area.

A general northwest to southeast trending geological cross section showing the distribution of these units beneath the hill feature along exploration drilling line 4W is presented in Figure 4 in Appendix G. The silver bearing mineral deposits of interest are mainly located in the unconformity zone at the top of the carbonate rock, to depths of around 260 m below sea level.

Local Hydrogeology

The hydrogeological features of the various stratigraphic units observed in the proposed mining area are described in the following sections. For the most part, these assessments are based on intrusive field investigations as described in greater detail in the *Field Investigations* section below.

Quaternary Deposits

Within the Xijiang River flood plain area the overburden is predominantly alluvial clay deposits providing for a low permeability base for the fish ponds, and the opportunity for clay extraction for local ceramics and construction activities. Limited silt and sand lenses are present within the layered clay and loam providing for limited groundwater development possibilities as evidence by a small number of shallow dug wells in the general area. Groundwater recharge to this area occurs through infiltration of precipitation and flood waters. Pumping and injection tests performed on monitoring wells completed in this overburden unit suggests that the hydraulic conductivity (K) of this unit is in the order of 10^{-6} m/s.

In the overburden on the hill sides, groundwater development occurs through the infiltration of precipitation in lower gradient upland areas, while in the steeper gradient areas, most precipitation is expected to runoff along the ground surface. Some of the infiltrating groundwater discharges to the surface in the forms of springs and seeps at lower elevation, possibly feeding some of the small creeks and ponds observed in the base of ravine and valley features. The remaining groundwater would infiltrate the upper fractured portion of the underlying sandstone bedrock units.

Cretaceous Sandstone and Siltstone

Located in the eastern portion of the Fuwan exploration area, this unit is a relatively heterogeneous mixture of sandstone, conglomerate, siltstone and mudstone, with a thickness of 30 to 170 m. Well drilling activities in this unit have found low groundwater potentials and monitoring well testing suggests a hydraulic conductivity in the order of 10^{-8} m/s. This unit is therefore considered to be relatively impermeable, and would act as an aquitard.

Triassic Sandstone and Mudstone

The Triassic sandstone consists of sandstone, conglomerate and siltstone up to 300 m thick, and with varying degrees of fracturing, from breciaated in the upper regions to massive at depth. Exploration drilling throughout this unit has found the level of groundwater development to be variable, but for the most part limited. Aquifer testing on monitoring wells completed in this unit show hydraulic conductivities in the range of 10^{-6} to 10^{-9} m/s, suggesting this unit would for the most part be considered an aquitard, with some potential for increased groundwater flow in areas of more severe fracturing.

Carbonate and Unconformity Units

The carbonate units can be subdivided into a lower and upper zone, differentiated by the influence of the unconformity and associated faulting. The lower zone is between 8 and 120 m thick, undisturbed and generally shows a low level of fracturing and/or weathering resulting in a relatively impermeable rock mass.

The upper zone is associated with the unconformity as is referred to as the $Fx_1 + C_1z^2$ zone. This unit is between 3 and 244 m thick, averaging around 77 m thick, and found at depths of up to 300 m below sea level. This zone shows a highly degree of fracturing, and more notably the extensive development of karst conditions. Karst conditions refer to mineral solution enlarged void spaces which can act as caves or channel features. If groundwater movement through these units decreases and/or stops the void spaces can become infilled with loose material (e.g., silt, clay and sand), or through crystal growth (e.g., calcite). Within the documentation for this project, this inactive area is referred to as “paleokarst”. The presence of karst openings in the near surface limestone is documented within this area with active karst formations being found to depth os approximately 50 m below grade. Previous geophysical surveys of the area have also identified the presence of probable shallow karst features along the area on the north side of the Chankeng fault and to the south of the F3 fault.

The paleokarst development is noted in borehole logs from throughout the Fuwan and Changkeng explorations areas with at least 25% of all borehole logs showing significant void space attributable to paleokarst occurrence. Individual voids range in size from 0.2 to 20 m. These paleokarst features are noted at depths of 50 to 250 m below grade, with most being noted between 100 and 200 m below grade.

Groundwater development in the $Fx_1 + C_1z^2$ zone is noted to be extensive with drill mud losses a common occurrence. Preliminary testing of this unit suggests some areas of low conductivity, but for the most part conductivity is relatively high, in the order of 10^{-5} m/s.

Fault Zones

The Chankeng fault zone was evaluated as a potential source of groundwater flow through monitoring of various pumping and injection tests. This evaluation found variable conditions with the eastern portion showing a low conductivity and the central portion indicating a more moderate conductivity.

The F2 fault zone was subjected to limited hydrogeological investigations and found to possess a relatively low water bearing capacity, in the order of 10^{-7} m/s.

Assessment of the F3 fault to the east was not performed directly so its potential groundwater characteristics and hydraulic conductivity cannot be confirmed at this time.

FIELD INVESTIGATIONS

The field investigations relating to assessment of the Fuwan exploration area were undertaken by 757 Exploration Team as a joint program with the Chengkeng exploration area to the north. Both of these mineral exploration areas are located within the same geological units so the sharing of information gained on each individual property allows for an improved understanding of the overall local geological conditions, as evidenced in the geological plan and cross section maps being produced.

With regards to the hydrogeological program, initial small scale aquifer pumping and injection tests were performed at a number of locations within the Fuwan exploration area. A large groundwater pumping test was then conducted on the Chengkeng site with supplementary monitoring being done on the Fuwan site.

Site Inspection

In accordance with the requirements of NI 43-101, Brent Horning of Wardrop undertook a visual inspection of the Fuwan exploration property in August 2008. This inspection included a tour of the primary and alternative proposed mill plant site, waste rock area and tailings management facility, and an overland viewing of the hill features above the proposed mine excavation site. One of the active exploration drilling teams was also viewed to allow for visual assessment of the process, and of some of the core being retrieved and boxed for later logging. This site inspection also included a review of the geological findings to date as provided by the Minco on-site geological exploration team. Additional details relating to this site inspection are provided in Appendix G.

Hydrogeological Testing

The hydrogeological testing for the Fuwan and Changkeng exploration areas was performed by 757 Exploration Team. A preliminary review of their activities and findings was prepared by SRK Consulting in November 2007, part way through the 757 Exploration Team's initial 2007 investigation program. These investigations involved small scale pumping tests on a series of exploration boreholes drilled in the various geological units across the site, one larger scale pumping test, groundwater sampling and chemical analyses, and preliminary review of the local surface water/ groundwater system. The results of these investigations were released by 757 Exploration Team in a report dated December 2007.

In 2008, the 757 Exploration team undertook another larger scale, long term pumping test in the area to further evaluate the groundwater interaction between geological units and with the surface water systems. The raw data collected during this pump test was provided and analyzed by Wardrop hydrogeologists in the preparation of this report.

Small Scale Pumping Tests

Small scale pumping tests involved the use of a submersible pump to lower the water level in the borehole and manual measurement of the associated change in water level in the borehole over time. Pumping rates ranged from low (less than 1 L/s) to moderate (8 to 16 L/s) based on the available well yield. Water table drawdowns in the individual pumping wells reached up to 100 m. Pumping continued until groundwater stabilization was observed in the pumping well for several hours, and the pump was then turned off and groundwater recovery rates were recorded. These pumping and recovery tests were each from one to five days in length.

In order to evaluate the groundwater potential from the various geological units observed, multiple tests were performed in some wells with temporary well casing and pump levels adjusted to try and isolate the individual units. These attempts showed mixed results with groundwater pumped from the borehole still expected to be from multiple units. Some monitoring wells were also subjected to multiple pumping tests at different pumping rates. Three injection tests were also attempted where water was poured into the borehole and the resulting decrease in water level was observed; however due to uncertainties as to which geological unit this water may be draining into, the results of this testing was not considered to be reliable so not analyzed as part of the 757 report.

The 757 report provides details on a total of 29 pumping tests performed on 13 separate exploration boreholes distributed around the project area. The pumping tests performed included assessment of the sandstone units (T_3x and K_1b_z), the unconformity ($Fx_1 + C_1z^2$), the quaternary sediments (Q) and the F2 fault zone. A summary of these pumping test parameters is presented in Table 1 in Appendix G. The distribution of the pumping test locations is shown in Figure 5 in Appendix G.

The hydraulic conductivity (K) of the water bearing units were calculated based on the pumping test results using industry standard hydrogeological equations, acknowledging that the actual thickness of the water bearing unit is an estimated value derived from interpretation of the associated geological exploration borehole log. Based on these results, the sandstone units and quaternary units are confirmed to be low conductivity, low yield aquitard formations. Pumping tests performed on borhoels that intercepted the F2 fracture were noted to have low conductivity values, with no apparent signs of increased groundwater recharge through the fault.

Analyses of the pumping test data for the unconformity area showed a more variable range, in the order of 10^{-5} to10^{-9} m/s. The lower conductivity values were noted along the outside of the study area, with the central region, including the proposed Fuwan underground mining area, showing values primarily in the 10^{-5} to 10^{-7} m/s range.

Large Scale Pumping Tests

FW0086

To supplement the individual pumping tests, the 757 Exploration Team performed a larger scale pumping test on reamed out borehole FW0086 in September 2007. . Groundwater monitoring points were established in six surrounding monitoring wells with piezometers completed at two separate levels in some of these monitoring wells in order to measure groundwater fluctuations in the unconformity and the overlying sandstone unit separately. This test area is located in the western portion of the Fuwan mine excavation area, as shown in Figure 6 in Appendix G.

Pumping from borehole FW0086 occurred at a rate of 15.4 L/s for a period of about 7 days, followed by an additional five day recovery period. A summary of the test parameters and results in presented in Table 2 in Appendix G Based on analysis of the drawdown curve for the pumping well, hydraulic conductivity in the unconformity in this area is around 7×10^{-5} m/s (6.1 m /day).

The results of this pumping test show a distinct hydrogeological connection across a larger area. The influence of the pumping was observed in all four monitoring wells completed in the unconformity. The influence of the pumping on the overlying sandstone unit was mixed, with the two most southerly monitoring wells showing a distinct influence, while the remaining three showed low to no influence over the pumping period. This would suggest that some hydraulic connections does exist between the unconformity and overlying sandstone units on a local scale, but is not continuous through the entire study area.

CK2028

In September 2008, a large scale pumping test was initiated on monitoring well CK2028, located in the south central portion of the Changkeng property. This well was initially pumped at a rate of 24 L/s for five days, followed by a four day recovery period. A second pumping period was then immediately initiated for an additional four days at a rate of 13 L/s and followed by a three day recovery period. Groundwater levels were monitored in seven surrounding monitoring wells on a continuous basis, and in another six to nine monitoring wells on a daily basis. These monitoring points are distributed throughout both the Chankeng and Fuwan areas as shown in Figure 7 in Appendix G. This distribution of monitoring wells covers both the hill area as well as the flood plain to the north of the hill feature, and has wells positioned on both sides of the Chankeng Fault and the F2 Fault. Most of these wells are completed in the unconformity, with the exception of ZK0103 and CK2021, which are both in the vicinity of the pumping well, but completed only in the overlying sandstone.

The measured groundwater drawdown associated with these pumping and recovery tests were plotted out as shown in Figures 8 and 9, in Appendix G, to assess the extent of the influence. These graphs shows that the area of influence extends throughout the region, with the five continuous monitoring points completed in the unconformity showing a strong influence. The monitoring points completed in the overlying sandstone show a slight influence in one (ZK0103) and no influence in the other (CK2021). Evaluation of the daily monitoring records show a slight less distinct level of influence due to the less detailed data set, but do indicate that six of nine wells show at least a moderate level of influence. The remaining three wells (CK2001, CK2011and QK07) were inspected only a limited number of times during the second pumping test, and do not appear to show any influence.

In general, the shape of these graphs is representative of a successful pumping and recovery test. The spikes present in the drawdown graph at the 1000 s and 4500 s marks are the result of temporary shutdown of the pump for maintenance purposes, but due to the short duration and subsequent pumping period length do not adversely affect the quality of the data.

The relative groundwater elevation and drawdown impacts between the wells were also graphed based on previously surveyed monitoring well elevations, as shown in Figure 10 in Appendix G. This graph suggests that the water level in the Fx1+C1 unconformity zone is relatively uniform throughout the area, at around 2 to 4 m above sea level. No distinct groundwater flow pattern was noted as the elevations show a mix of high and low points, so a groundwater flow direction could not be determined. On a general area wide scale, the groundwater gradient appears to be in the order of 0.002. The water level in the overlying sandstone unit is noted to be at a more variable, potentially higher elevation than in the unconformity. For general reference, the water level in the Xijiang River during this period was at an elevation of about 1 m above sea level.

The maximum drawdown values were also reviewed with respect to distance from the pumping well to assess the relative symmetry of the drawdown cone as it expands out from the pumping well. These plots are shown for both pumping periods in Figure 11, in Appendix G, as an overall plot of all monitoring points against distance, as well as through separation into east and west of the pumping well. These plots indicate that for the most part the distribution is in a consistent fashion. The exceptions to this are ZK0103 and CK2021 which show no significant response despite being located with 200 m of the pumping well, indicative of a low hydraulic connection between the unconformity and the overlying sandstone in this area. The monitoring wells CK2001 and CK2011 also show some divergence from the anticipated drawdown cone suggesting a low level of connectivity between this area and the pumping area. Borehole logs and previous small scale pumping tests on these two wells indicate they are drilled through the unconformity.

In order to better demonstrate the extent of the area of influence produced by the pumping tests, the estimated extent of the pumping period #1 drawdown cone is shown as a contour plot in Figure 12 in Appendix G. This interpretation of the drawdown data indicates it extends outwards in all directions, irrelevant of the Changkeng fault and the suggested limitation of the unconformity zone to the south side of this fault.

The drawdown measurements were analyzed using the AquiferTest Pro™ commercial hydraulic conductivity analyses software. The pumping well is completed in a fractured/ karst rock system and was analysed using the Double Porosity method. This method accounts for two flow systems within the bedrock matrix; a high permeability fracture system with low storage capacity; and the rock matrix with has low permeability and high storage capacity. As a result, the initial pumping efforts reflect flow from storage in the fractures. As the fractures are dewatered, the surrounding rock matrix releases water from storage into the fractures decreasing the initial drawdown rate and providing for dewatering of the entire rock mass. Use of this analytical method shows a good fit between the field data and the theoretical curve indicating the assumptions are valid for this site. A composite example of the various drawdown curves obtained from this pumping test are shown in Figure 13 in Appendix G.

This hydraulic conductivity analysis indicates that the hydraulic conductivity for this well is in the order of 1.3×10^{-5} m/s (1.1 m/d), which corresponds to the previously calculated conductivity values for this unit. The corresponding transmissivity of the groundwater bearing unconformity in the vicinity of borehole CK2038, is in the order of 120 m^2/ day, based on an estimated 110 m thickness of fractured, water bearing rock.

Analysis of the drawdown curves associated with the various monitoring wells was also undertaken with AquiferTest Pro™ using the Double Porosity curve matching solution. The results of these analyses indicated aquifer hydraulic conductivity is consistently in the range of 0.9×10^{-5} to 1.4×10^{-5} m/s, with corresponding transmissivities of 90 to 130 m^2/day, based on water bearing zone thicknesses of 30 to 120 m. Within the overlying T3 sandstone unit (ZK0103), the conductivity and transmissivity were in the order of 2×10^{-5} m/s and 200 m^2/day, respectively. The drawdown curve analysis results produced from this pumping test are summarized in Table 3 in Appendix G. Copies of the Aquifer Test Pro™ output are included in Appendix G.

DATA VERIFICATION

The field data collected by the 757 Exploration Team, and documented by them, by SRK consultants and by NERIN have been reviewed with respect to overall validity, and found to be acceptable. Some discrepancies have been noted between data sets with respect to borehole coordinates and well log details, and some limitations are noted with the pumping test observations, potentially relating to conversion of geological exploration boreholes to groundwater pumping and or monitoring wells. However, for the scale of the local hydrogeological assessment of the proposed underground mining area these issues are not considered to have any significant impact on the overall aquifer characterization.

INTERPRETATION AND CONCLUSIONS

Groundwater Systems

Hydrogeological testing of the individual geological units found in the proposed Fuwan underground mining area indicate that the overburden (Quaternary) deposits and underlying sandstone (T3 and Kb) units typically have low hydraulic conductivities and limited groundwater capacities. In some areas these units appear to be isolated from the underlying unconformity, while in others a slow connection is apparent.

The unconformity and associated karst carbonate unit has been found to be a fairly well hydraulically connected throughout the study area. Groundwater withdrawal activities from various test holes have shown the formation of a distinct groundwater plume extending radially from the pumping well. The hydraulic conductivity of this unit has been determined to be in the order of 10^{-5} m/s, which is indicative of a good aquifer. Transmissivity values for this water bearing units were generally calculated to be in the order of 120 m^2/day.

Based on the FW2038 pumping tests, it was noted that the Fx1 + C1 hydrogeological system appears to be hydraulically connected throughout the region, including beneath both the hill feature and the flood plain area to the north. This area of influence extends outside of the Fx1 unconformity so is likely attributable to the well developed historic karst features present in the carbonate unit. The extent of the drawdown cone produced (i.e., radius of cone of influence estimated at 2000 m) by a relatively low industrial pumping rate (24 L/s = 375 USgpm) and drawdown cone depth (9 m), is indicative of a high transmissivity aquifer condition. The absence of any notable boundary conditions affecting the trend of drawdown graphs produced suggest that there is limited interference with groundwater movement by local features which may contribute additional recharge to the system, either underground (e.g., fault zones) or infiltration from surface water sources (e.g., fish ponds, Changkeng ditch, Xijiang River, etc.).

The potential for interconnection between the Xijiang River and proposed underground mine workings has been evaluated qualitatively from a geological point of view by 757 Team. Their interpretation was that the fine river bottom sediments (clay and silt) and low conductivity T3 unit underlying the river area would minimize direct hydraulic connection between the river and the Fx1 + C1 water bearing unit. The primary potential source of connection was therefore the apparent Changkeng, F2 and F3 fault traces which appear to extend out under the river. The SRK report indicated that the Xijiang River appears to be poorly connected hydraulically with the proposed underground mine envelope in the areas tested.

During the CK2038 pumping test, the groundwater drawdown cone extended through the estimated Chankeng fault trace with no significant restriction, which would suggest that this fault does not provide for any significant amount of groundwater recharge. Similarly, the F2 fault did not appear to provide any additional recharge to the water bearing formation in that area. A similar level of information is not available for the area of the F3 fault, so the potential for groundwater flow in this area remains uncertain.

Groundwater Inflow to Mine Workings

The results of the hydrogeological assessments performed across the study area suggest that the karst zone within the carbonate bedrock acts as a regional aquifer with a conductivity in the order of 10^{-5} m/s, which is considered to be representative of good aquifer conditions and is at the high end typical of carbonate rock aquifers. The static water level in this aquifer is relatively consistent suggesting a high degree of interconnectivity. Using these aquifer parameters, a preliminary estimate of groundwater inflow into the mine workings can be developed to provide an indication of the level of dewatering potentially required.

In previous hydrogeological reports, an estimate of groundwater inflow was developed using a standard formula for groundwater dewatering requirements for a linear feature in a confined aquifer, based on a rough water balance, and in comparison with information from historic local shallow mining operations. These calculations suggested a range of possible groundwater inflow from 4,550 to 27,011 m^3/ day. Each of these methods were reviewed as described below.

Trench Dewatering Calculation

The 757 Team December 2007 report provides a groundwater inflow estimate calculation based on the results of the individual pumping test and the FW0086 larger scale pumping test. In their report they refer to this as a “groundwater dynamics” method, and is reproduced in Table 4 in Appendix G. The assumptions made for the application of this formula are that the mine operates as a single linear feature at a depth of 280 m below sea level. The resulting groundwater inflow rate is estimated at 13 830 m^3/ day. This calculation was also done by Wardrop using updated values for conductivity and mine depth, which produced a value of 15 125 m^3/ day.

The calculation applied by 757 Team is similar to one used in North America for calculation of dewatering requirements for a linear feature in a confined aquifer. For general comparison, this calculation is also shown in Table 4 in Appendix G. The result of this calculation show under the updated conditions, well dewatering would require a groundwater discharge rate of 17 700 m^3/ day. The limitation to the applicability of this formula to the proposed mine development is that this formula assumes an open surficial excavation so accounts only for groundwater infiltration from the sides and base of the excavation. Continuous infiltration through the top of the mine stopes from the overlying bedrock is not included.

Relative Comparison

Since there was an active mining operation working in the same geological units on the adjacent Chankeng exploration area, a general evaluation of potential groundwater dewatering requirements was prepared through extrapolation of observed conditions at one site to potential conditions at the other. This method was provided by 757 Team, and updated by NERIN using revised mine depth and area estimates.

The Changkeng gold mine operated to a depth of about 50 m below sea level, over an underground plan area with a radius of around 84 m. The proposed Fuwan mine is expected to operate at a depth of 260 m below sea level, over an equivalent plan radius of 450 m. The relative increase in depth times radius of the Fuwan mine as compared to the Changkeng mine was then applied against the discharge rate to obtain a value of up to 18 007 m^3/day. NERIN also suggested the application of a safety factor of 1.5 against this value, bring it up to 27.0 11 m^3/day.

Due to the subjective nature of this type of approach, it can be used only to provide a general indication assuming that all conditions remain exactly the same. Due to the much greater depth of the proposed Fuwan mine, it is expected that significant changes in subsurface hydrogeological conditions may be encountered so this method has limited relaibiity.

Water Balance

As an alternative to the comparative method, NERIN developed a water balance approach to determining potential groundwater inflow. This method applies a simplistic view of the hydrogeological cycle were a portion of the precipitation that falls over a specified plan area recharges the aquifer system, and would therefore be expected to flow through the mine site. NERIN has described a conservative approach to this where 75% of precipitation infiltrates into the bedrock aquifer over a surface area equivalent to the proposed plan area of the mine development. This calculation suggests a groundwater flow through of between 4550 and 14 227 m^3/ day, reflective of long term average, increased rainy season precipitation rates, and one in 20 year storm conditions.

The main limitation to this method is that it does not account for the dewatering effort initially required for mine excavation, only what would be required to maintain those dewatered conditions.

Overall Condition Assessment

The initial concern upon implementation of the most recent round of on-site hydrogeological assessment was whether the known geological faults represented a potential conduit for surface water into the proposed underground mine workings. Based on the previously described interpretation of the large scale pumping tests performed, there is no indication that significant recharge enters the groundwater system through these faults. These pumping tests have also indicated that the sandstone units overlying the mineral zone generally show low hydraulic conductivities, with limited connection to the underlying carbonate bedrock.

The carbonate rock which is the primary host to the unconformity and the karst formations, was the primary focus of the recent large scale pumping test. The results of this testing process suggests that this unit acts as a relatively high conductivity regional aquifer system, with a groundwater table elevation of 2 to 4 m above sea level, which is close to ground surface in the flood plain areas adjacent to the proposed mine development area. Pumping of this formation at a rate of 24 L/s (2074 m^3/ day) produced a maximum drawdown in the order of 9 m with an area of influence in excess of 1.5 km. Analyses of this pump test information indicates that the carbonate unit has a hydraulic conductivity of 1.1 x 10^{-5} m/s, which is at the high end of the published range of conductivities for a fractured carbonate bedrock system.

Based on these aquifer characteristics, mine development will be required to deal with potentially large groundwater inflows. Due to the natural heterogeneity of the karst formations, this inflow would be variable depending on the size and frequency of the karst openings encountered during excavation. Preliminary estimates of this potential inflow suggests values in the range of 4,550 to 27,011 m^3/day. However due to the natural heterogeneity of the underground karst conditions, actual inflows within specific areas may be higher or lower than this average value. Groundwater inflow rates are also subject to change over time, generally from higher initial values to lower values over time, but as discussed in the water balance inflow calculation, may also be representative of external precipitation rates producing more seasonal variations, or possibly even as a delayed response to single events (extreme storms, surface flood conditions, etc.).

Based on the observed large area of influence created by pumping from this aquifer system, any proposed dewatering of the mine site will likely have to take into account potential impacts on existing groundwater users with an area of several kilometers from the mine site. Previous investigations have indicated that there are a number of small private wells in the general area. Well completion data would therefore have to be evaluated to determine if a decrease in the carbonate unit water level in the vicinity of the mine site could have an adverse impact on other groundwater users.

Mitigation measures in this case may involve deepening of the supply wells and/or lowering of the pumps.

Another potential concern may be land subsidence. Depending on the level of dewatering required for mine operations, the general lowering of the groundwater surface may also result in a decrease in hydrostatic pressure in the overlying bedrock and soil units. This would then result in a slow subsidence of the impacted area. In addition, dewatered of karst void spaces may result in more significant, localized settlement or even sudden collapse in the form of sink holes. Based on the apparent low conductivity of the sandstone unit above the paleokarst zone, any required mine dewatering is not expected to have a significant effect on the shallow surface karst area. Ongoing monitoring of the shallow groundwater conditions, as well as ground surface elevation monitoring would be required to evaluate this potential on a regular basis, and as such would tie in to the overall geotechnical stability and rock strength evaluations associated with the proposed mine development.

RECOMMENDATIONS

Supplementary Investigations

The distribution and scale of the groundwater pumping tests performed to date have addressed most areas of interest. The absence of any detailed testing in the vicinity of the F3 Fault does not allow for confirmation of the potential influence of this fault on groundwater recharge. In accordance with current practices, existing geological exploration holes in this area should be identified on both sides of the fault trace and reamed out for small scale pumping tests, with the potential for a larger test if the small scale tests prove inconclusive.

As a final means of confirmation of the presence/ absence of a connection between the groundwater system and the Xijiang River, enhanced remote sensing methods could be introduced. One such option is audio frequency domain magnetic which have been successfully implemented in karst environments, and in identification of groundwater/ surface water environments. This process basically involves the creation of a magnetic field in the groundwater system and the monitoring of suspected receptor areas for the presence and magnitude of any connecting pathways.

Hydrogeological Model Development

Hydrogeological and geological information have been complied by a variety of sources and some compilation and cross referencing has been undertaken. Recent investigations extending across the Fuwan and Changkeng exploration zones now present the opportunity to develop a complete hydrogeological model of the area, using commercial computer software. This model should also incorporate the surface water and meteorological observations collected for the area as well.

Calibration of this model against the CK2038 pumping test data set should provide a sufficient level of confidence to develop a more robust prediction of the potential groundwater inflow into the proposed mine system, and the level of mine dewatering likely to be required.

Mine Dewatering System

Due to the risk of sudden high volume inflow of groundwater under significant hydraulic pressure, continuous advancement of probing drilling ahead of any proposed excavation areas will be required. Response plans to high groundwater flows, whether installation of local grouting or dewatering efforts, or changes to the mine plan itself, will have to be developed to address any such occurrence.

Dewatering of the mine area through installation of perimeter or internal groundwater dewatering wells is at this point not considered to be feasible due to the extremely high pumping rates potentially required. Internal mine drainage galleries would therefore be required to allow for the controlled drainage of any water bearing zones encountered.

As excavation progresses, additional flood control measures such as installation of water tight doors or bulk heads should be implemented at regular intervals in order to allow for containment of any uncontrolled groundwater inflow.

A formal continuous groundwater elevation monitoring program will be required to evaluate the overall extent of any changes to the area impacted by mine dewatering (if any). These wells should be installed in various depths to monitor water elevations in the various units of potential concern including the sandstone unit, the unconformity, known karst zones, fault zones, etc.

18.1.3 VERIFICATION DRILLING

Verification drilling at the Fuwan Project was observed by a Wardrop representative. Drilling was undertaken using an HQ (2.5" diameter core) double-tube core barrel with wireline methods. The core was carefully and correctly placed in sequential order in boxes. The core reads from left to right. If the core did not slide freely from the tube, the drillers hammered the tube to free the core, which causes mechanical fractures in the core. The boxes were stored at the drill site until drilling was completed and they were later transferred to the core shack and stored on racks.

GEOTECHNICAL LOGGING OF VERIFICATION HOLES

Five holes were logged geotechnically on site by Wardrop. The details of the holes are shown on Table 18.11, Figure 18.8, and Figure 18.9.

Table 18.11 Details of Holes Logged On Site

ID	Beijing 1954 Datum Coordinates			Azimuth	Dip	Length (m)	UTM Coordinates	
	X (m)	Y (m)	Z (m)				E	N
FW0242	2546313	38378642	23.57	232°	-61°	146.6	112°48'58.53″	23°00'40.3188″
FW0243	2546178	38378470	24.78	51°	-67°	210.2	112°48'52.55″	23°00'35.8668″
FW0245	2546192	38378615	12.92	15°	-56°	205.35	112°48'57.63″	23°00'36.3625″
FW0247	2546206	38378506	42.35	52°	-60°		112°48'53.8″	23°00'36.7966″
FW0248	2546296	38378562	38.08	21°	-87°		112°48'55.73″	23°00'39.7418″

Three drilling collars (FW0245, FW0247, and FW0248) were verified by GPS on site.

Figure 18.10 to Figure 18.14 show Q' values for logged holes on site.

Figure 18.8 Plan View of Holes Logged on Site

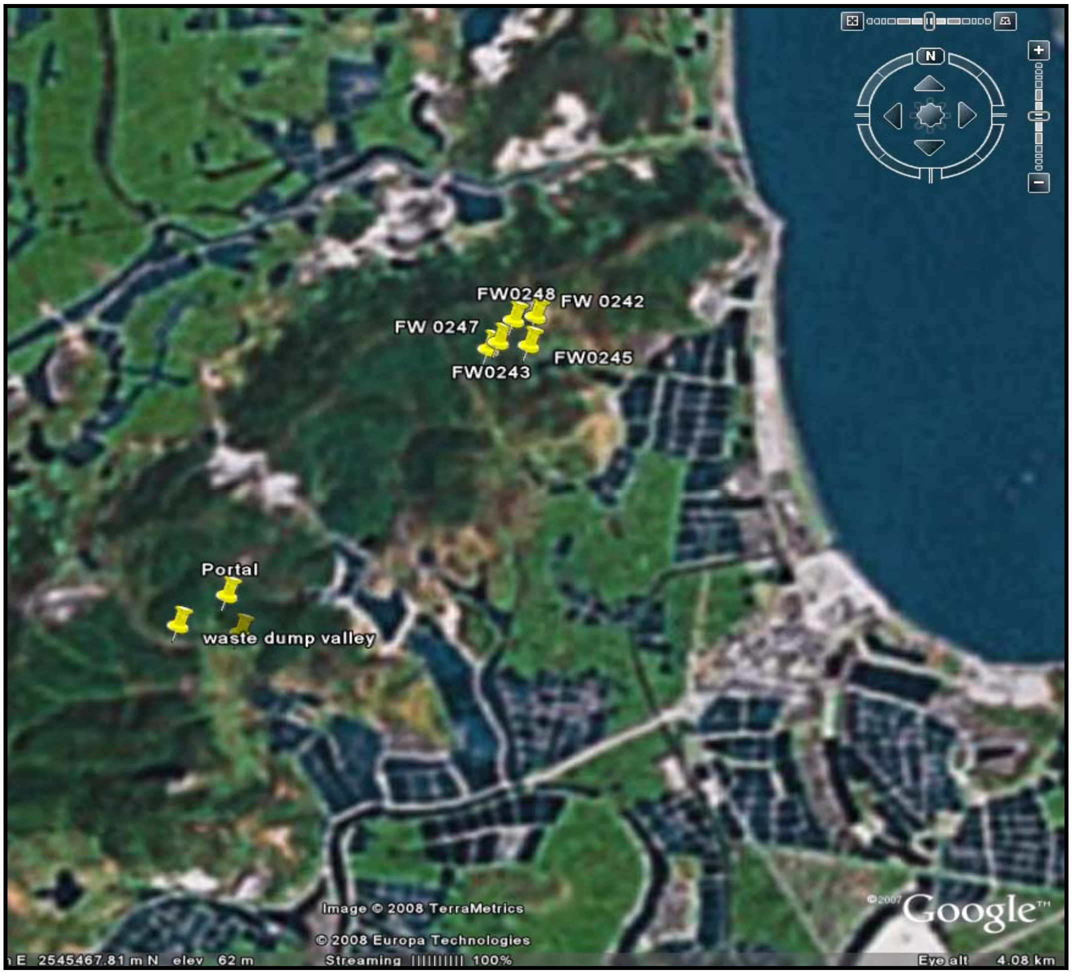


Figure 18.9 Perspective View of Holes Logged on Site

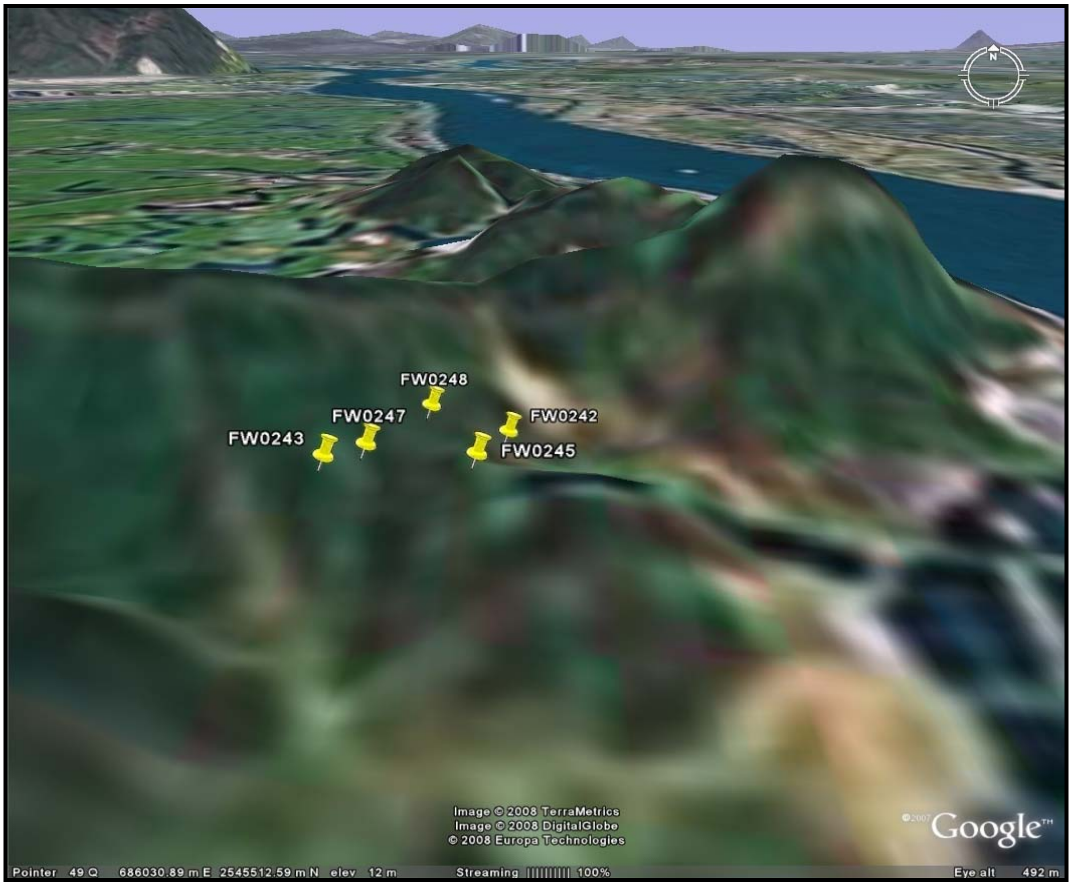


Figure 18.10 Hole FW0242 Q' Values

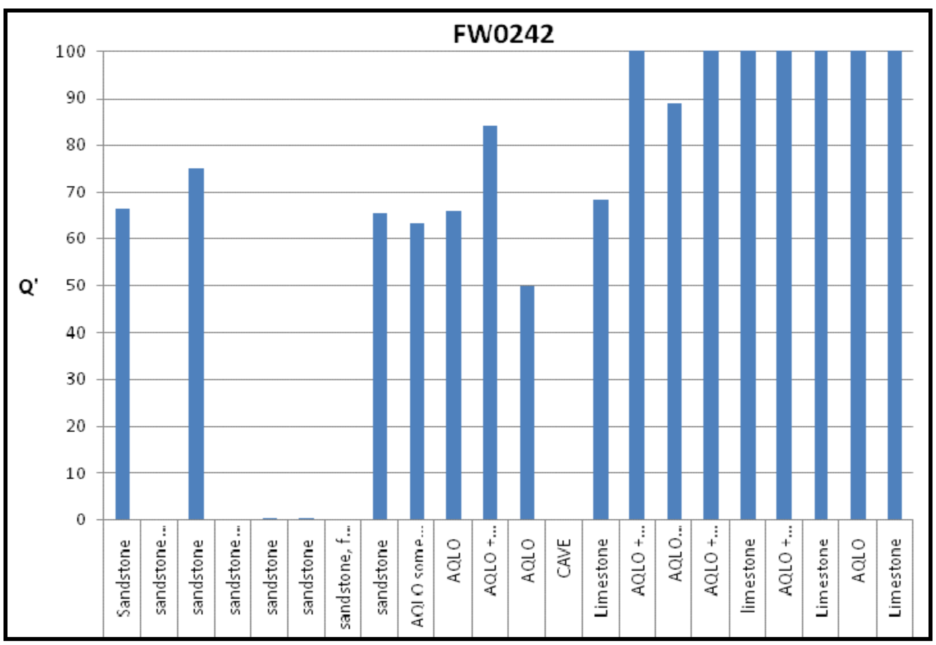


Figure 18.11 Hole FW0243 Q' Values

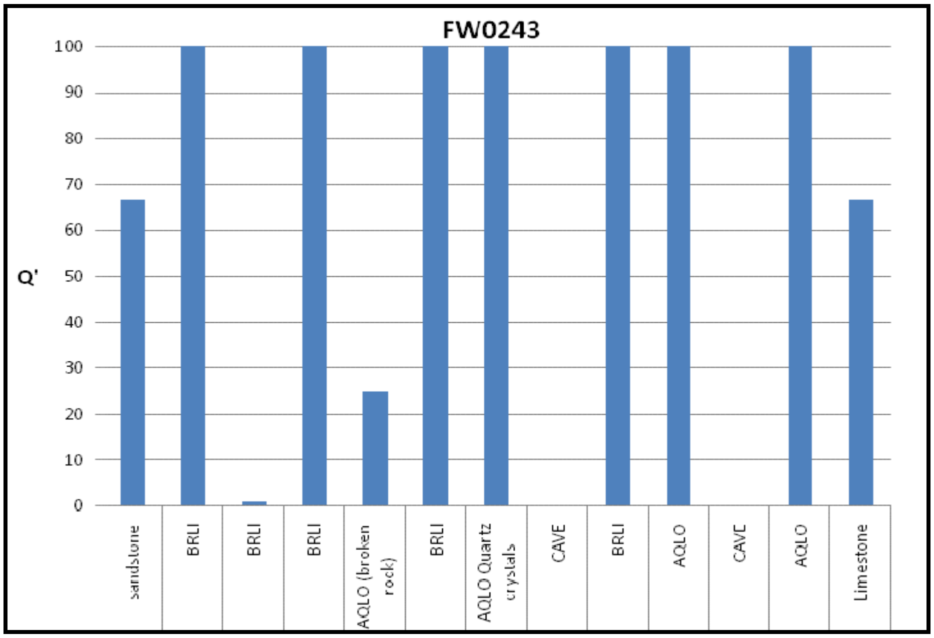


Figure 18.12 Hole FW0245 Q' Values

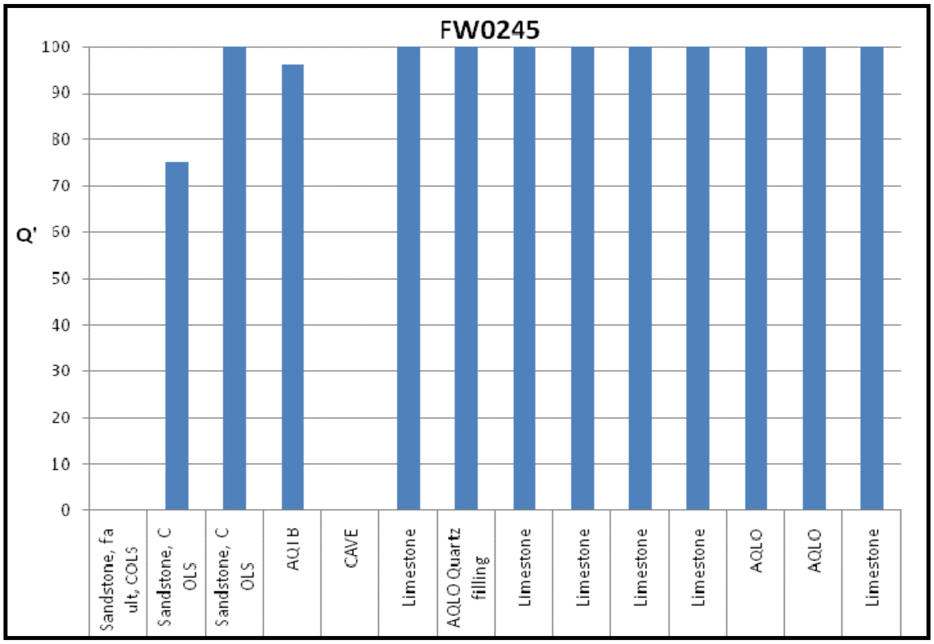


Figure 18.13 Hole FW0247 Q' Values

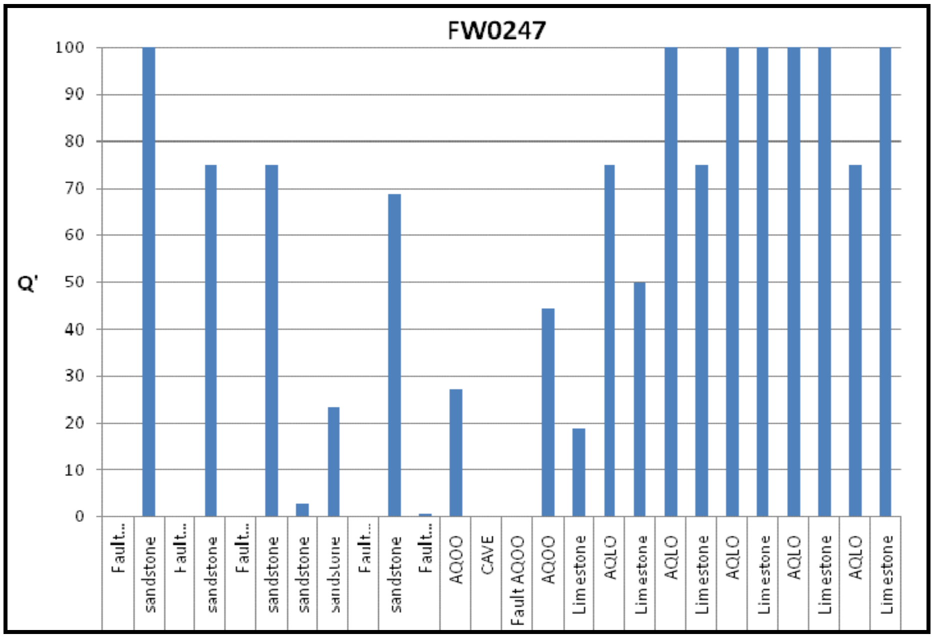
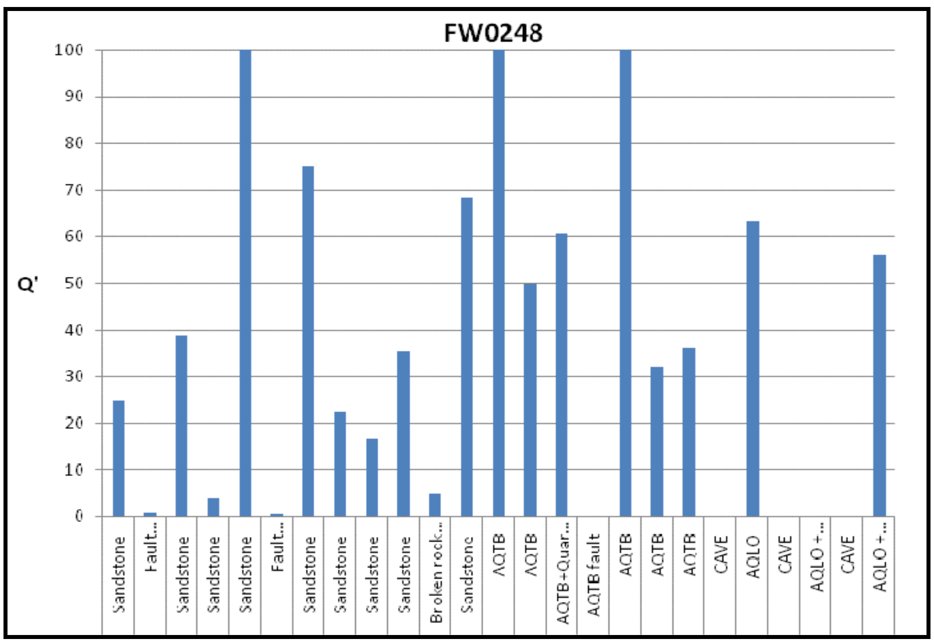


Figure 18.14 Hole FW0248 Q' Values



RECOMMENDATIONS

In general, the ground conditions within the orebody are predicted to be good with few stability problems. However, at the unconformity, particularly difficult ground conditions are expected with a fault zone that will probably be exposed in the immediate backs of some stopes.

Waste Development

Waste development will be 4.5 m wide by 4.5 m high. A contractor will undertake all waste development. Recommended support for waste development is as follows:

- backs: 2.4 m long bolts on 1.2 m by 1.2 m pattern
- walls: 2.4 m long bolts on 1.5 m by 1.5 m pattern
- allow 25% coverage with a welded wire mesh square measuring 100 mm by 100 mm with 4 mm diameter wire
- allow 25% coverage with shotcrete (50 mm nominal thickness).

Areas that intersect the unconformity will require bolt, mesh, and shotcrete support.

Ore Development

Cut-and-fill stoping will require ramp development within the mining blocks. These ramps will be 4 m wide and 3.5 m high. The recommended support for ore ramp development is as follows:

- no unconformity in back:
  - backs: 2.4 m long bolts on 1.2 m by 1.2 m pattern
  - walls: 2.4 m long bolts on 1.5 m by 1.5 m pattern
- unconformity in back:
  - backs: 2.4 m long bolts on 1 m by 1 m pattern
  - walls: 2.4 m long bolts on 1.5 m by 1.5 m pattern
  - allow 100% coverage with a welded wire mesh square measuring 100 mm by 100 mm with 4 mm diameter wire
  - allow 100% coverage with shotcrete (50 mm nominal thickness).

Stope Support Requirements

Cut-and-Fill Stopes

In general, the cut-and-fill stoping blocks are not very thick and the expected average stope widths are about 3 to 4 m. Recommended support for the cut-and-fill stopes is as follows:

- no unconformity in back – stopes up to 5 m wide:
  - backs: 2.4 m long bolts on 1.2 m by 1.2 m pattern

- walls: 2.4 m long bolts on 1.5 m by 1.5 m pattern

- unconformity in back:
  - backs: 2.4 m long bolts on 1 m by 1 m pattern
  - walls: 2.4 m long bolts on 1.5 m by 1.5 m pattern
  - allow 100% coverage with a welded wire mesh square measuring 100 mm by 100 mm with 4 mm diameter wire
  - allow 100% coverage with shotcrete (50 mm nominal thickness).

In the costing it was assumed that 50% of cut-and-fill stoping would have the conformity in the immediate back.

Drift-and-Fill Stopes

The drift-and-fill stoping blocks are defined depending on whether there is the presence of the unconformity in the back or not. If the back has the unconformity present, maximum room widths should be driven only 4 m wide. If there is no unconformity, the drifts should be driven a maximum of 6 m wide to avoid the use of cable bolting, which would be prudent in wider spans.

If the ore is more than 7 m high, the panel should be mined in two cuts with the second cut working off the backfilled stopes below. In this instance, the first cut will not have the unconformity present and may be mined 6 m wide. If the second cut exposes the unconformity, the drift width of the second cut must be reduced to 4 m wide. Recommended support for the drift-and-fill stopes is as follows:

- no unconformity in back – drifts up to 6 m wide
  - backs: 2.4 m long bolts on 1.2 m by 1.2 m pattern
  - walls: none
- unconformity in back – drifts maximum 4 m wide
  - backs: 2.4 m long bolts on 1 m by 1 m pattern
  - walls: none
  - allow 100% coverage with a welded wire mesh square measuring 100 mm by 100 mm with 4 mm diameter wire
  - allow 100% coverage with shotcrete (50 mm nominal thickness).

18.1.4 *MINING METHODS*

Minco wish to develop a mechanized mine based on a typical western mine design. The orebody may be described as generally undulating. It is flat lying to the west, dips downward in the deeper central part, and then dips upward again to a generally flat lying eastern section. The northern flank of the orebody dips to the south.

FLAT LYING ZONES

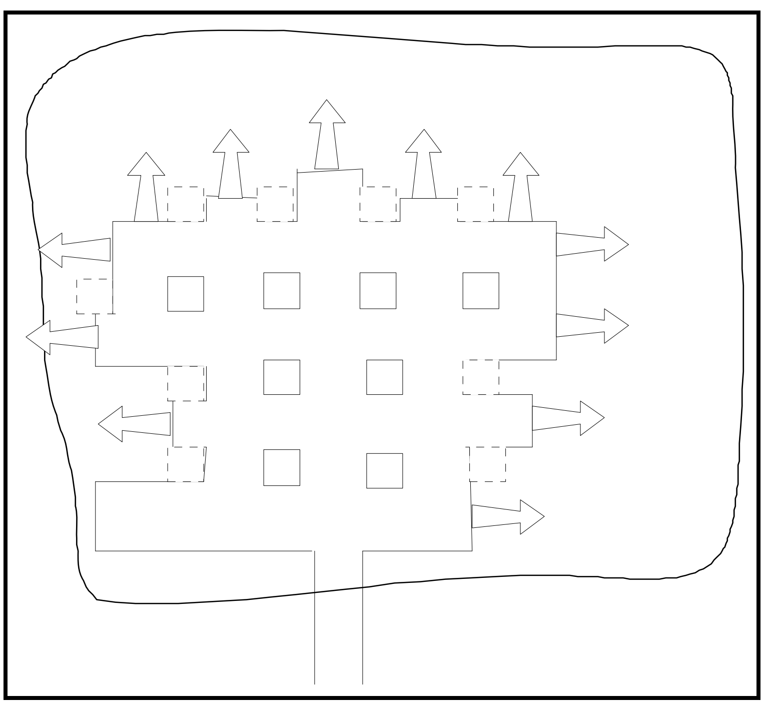
The flat lying zones lend themselves to the drift-and-fill or room-and-pillar mining methods. The resource wireframes of the ore zones were modelled to more or less a minimum thickness of 2 m. A 2 m minimum mining height was adopted for mechanized mining.

As the orebody has reasonably good grades, a trade-off study was undertaken to assess at what grade it would be worth backfilling with cemented fill and carrying out a primary/secondary drift-and-fill type mining method allowing 100% extraction without leaving any ore pillars.

Room-and-Pillar Method

The room-and-pillar method is a very productive mining method with the development of a number of faces for ore production within each panel. A disadvantage of the method is that it is all development-style mining and pillars of ore are not mined, reducing overall extraction. However, if ground conditions are good, relatively small pillars may be left, enabling extractions of up to 90% of available resources within the ore outline. Typically a central access is driven with a number of rooms driven left and right off the access, opening up a checkerboard in plan. The method is selective and zones of low grade can be left as pillars. A variation of this method is post pillar cut-and-fill: where the ore height is greater than 6 to 7 m, the panel is taken in two cuts. The first cut is taken and backfilled, then a second is taken over the top of the first cut working off the backfill.

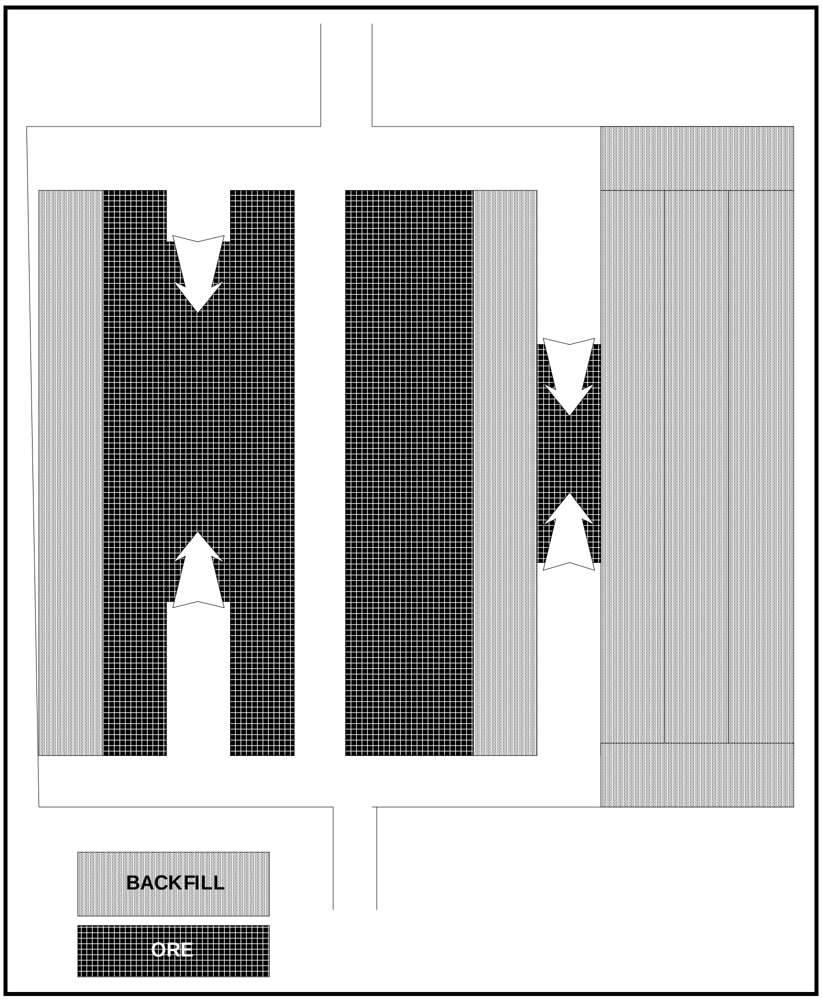
Figure 18.15 Room-and-Pillar Method



Drift-and-Fill

The drift-and-fill method is used in flat lying orebodies that have relatively high ore values as 100% of the ore may be extracted. It is similar to room-and-pillar mining in that it is a development-style mining method. In this method, parallel drifts are mined in a primary secondary sequence. After the first drift is mined, it is backfilled with cemented fill so that a drift may be mined alongside the backfilled drift exposing stable fill walls due to the cement content of the fill. Various layouts may be adopted along the same theme.

Figure 18.16 Drift-and-Fill Method



DRIFT-AND-FILL VS. ROOM-AND-PILLAR

A trade-off study was undertaken to find the required pillar grade that would be necessary to make it worthwhile mining by room-and-pillar methods rather than drift-and-fill methods. A silver price of $13/t was assumed.

Table 18.12 Concentration

Silver	Ag oz	1.00
Lead Concentrate		
Recovery	83%	0.83
Payable	87.5%	0.73
	72.6%	
Zinc Concentrate		
Recovery	9%	0.09
Payable	40.0%	0.04
	3.600%	
Total		
Recovery	92%	0.92
Payable		0.76
Average Ag % Payable		**76%**

The revenue received from 1 g of silver is estimated in Table 18.13.

Table 18.13 Estimate of Revenue Received from 1 g of Silver

3-year Silver Price	$13/oz
% Price Realized	76%
Realized Revenue	$0.32/g

The two drift-and-fill configurations were considered – 4 m wide in poor ground and 6 m wide in good ground. It is however doubtful if room-and-pillar would remain stable in poor ground conditions (i.e. at the unconformity). However, the analysis for this case was still carried out for completeness.

Table 18.14 Pillar and Room Widths

Pillar Width	Room Width	Extraction
3	4	82%
3	6	89%

A room-and-pillar configuration with a 3 m by 3 m pillar was assumed. The analysis determines what minimum pillar grade is necessary, below which room-and-pillar would be a better option. It compares the cost of adding cement to the backfill for 50% of the drift-and-fill versus the value of silver left in the pillar.

Table 18.15 Case 1: 4 m Wide Drift-and-Fill vs. 3 m by 3 m Pillar with 4 m Rooms

Room Area	49 m^2
Area Requiring Cement	50%
Pillar Area	9 m2
Cement %	6%
Ore Specific Gravity	2.48
Fill Density	1.7
Tonnes Cement per 1 m Horizontal Slice	2.5
Cost per Tonne Cement	¥460
Cost per Tonne Cement at ¥6.8 per $	$67.65
Total Cement Cost	$169.05
3-year Silver Price	$13
% Price Realized	76%
Realized Revenue per Gram	0.32
Ore Tonnes in 1 m Horizontal Slice of Pillar	22.32
No. of Grams Equivalent to Cement Cost	531 g
Excess Grade Needed Over Mining Cutoff Grade	24 g/t
Mining Cutoff Grade	119 g/t
Break Even Pillar Grade for 4 m Wide Drift-and-Fill	143 g/t

Flat lying zones in poor ground with grades less than 143 g/t will be mined by room-and-pillar, and those with grades in excess of 143 g/t will be mined by drift-and-fill.

Table 18.16 Case 2: 6 m Wide Drift-and-Fill vs. 3 m by 3 m Pillar with 6 m Rooms

Room Area	81 m^2
Area Requiring Cement	50%
Pillar Area	9 m^2
Cement %	6%
Ore Specific Gravity	2.48
Fill Density	1.7
Tonnes Cement per 1 m Horizontal Slice	4.1
Cost per Tonne Cement	¥460
Cost per Tonne Cement at ¥6.8 per $	$67.65
Total Cement Cost	$279.45
3-year Silver Price	$13
% Price Realized	76%
Realized Revenue per Gram	0.32
Ore Tonnes in 1 m Horizontal Slice of Pillar	22.32
No. of Grams Equivalent to Cement Cost	531 g
Excess Grade Needed Over Mining Cutoff Grade	39 g/t
Mining Cutoff Grade	119 g/t
Break Even Pillar Grade for 6 m Wide Drift-and-Fill	158 g/t

Flat lying zones in good ground with grades less than 158 g/t will be mined by room-and-pillar and those with grades in excess of 158 g/t will be mined by drift-and-fill. The application of room-and-pillar mining in the flatter orebodies will thus depend quite heavily on the geotechnical conditions as well as the cement cost. If the mining block is at the contact, room-and-pillar may not be geotechnically viable due to the poor roof conditions anticipated in the sheared contact. However, in the lower mining blocks below the contact zone and within the limestone, ground conditions should be good and room-and-pillar should be viable. Each mining block will be examined and an appropriate method selected based on the location and grade of the block.

ZONES DIPPING GREATER THAN 15°

Cut-and-Fill Panels Dipping Between 15° and 50°

In order to minimize waste development, the following method is recommended. Each panel will be about 100 m long and typically 60 m vertically. Twin ramps will be driven in the ore from top and bottom accesses to meet in the middle of the stope. A minimum 3 m wide (or a 1:1 ore width to pillar width) pillar will be left between the ramps. Figure 18.17 shows the anticipated pre-mining development of the block. The ramp below the pillar must always remain open for air passage and provide through ventilation. After the ventilation airway is no longer needed, the pillar could be recovered; however, any estimate should only assume an effective 50% recovery of the pillar.

Preproduction panel development requires access to the top and bottom of cut-and-fill panels.

The configuration will provide three working faces per block as shown in Figure 18.18.

Figure 18.17 Anticipated Pre-mining Development of the Block

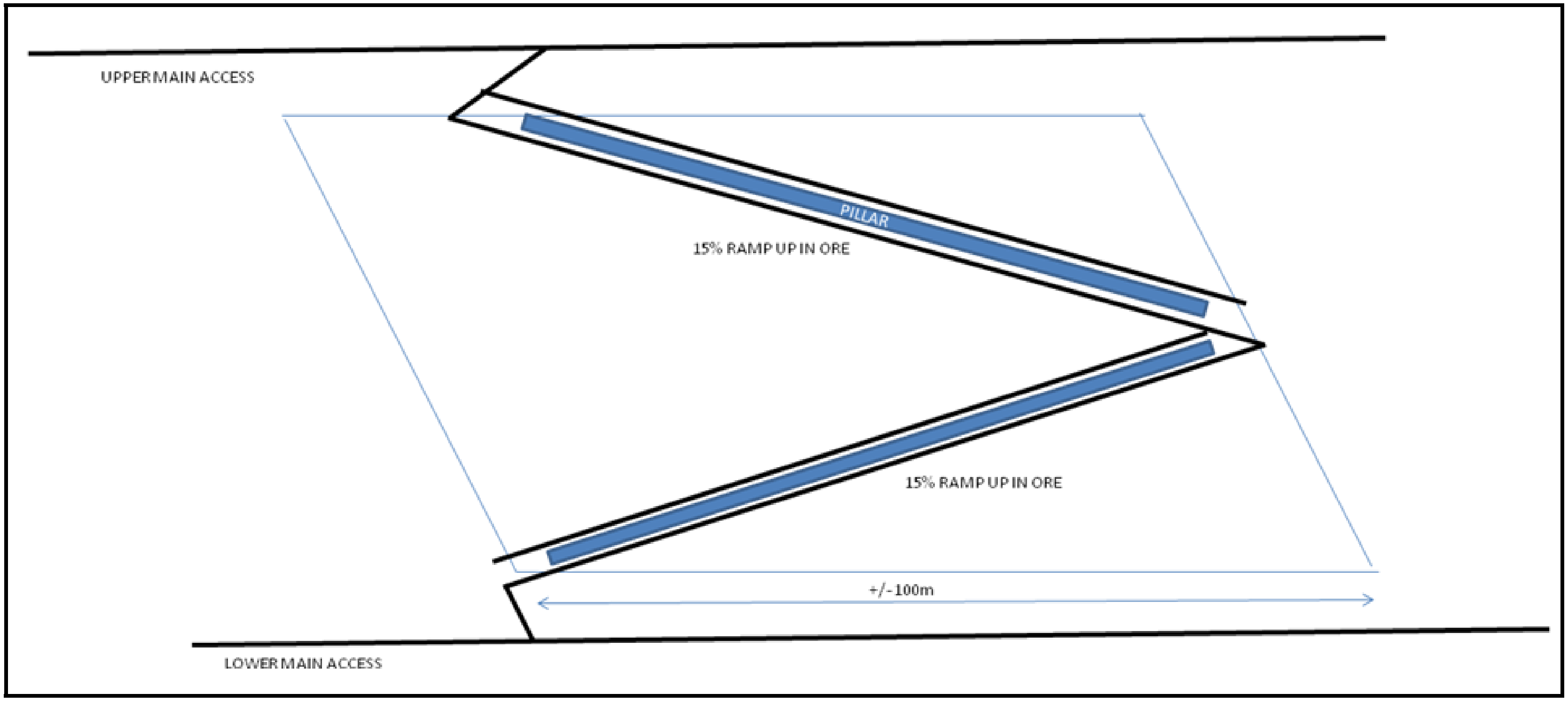
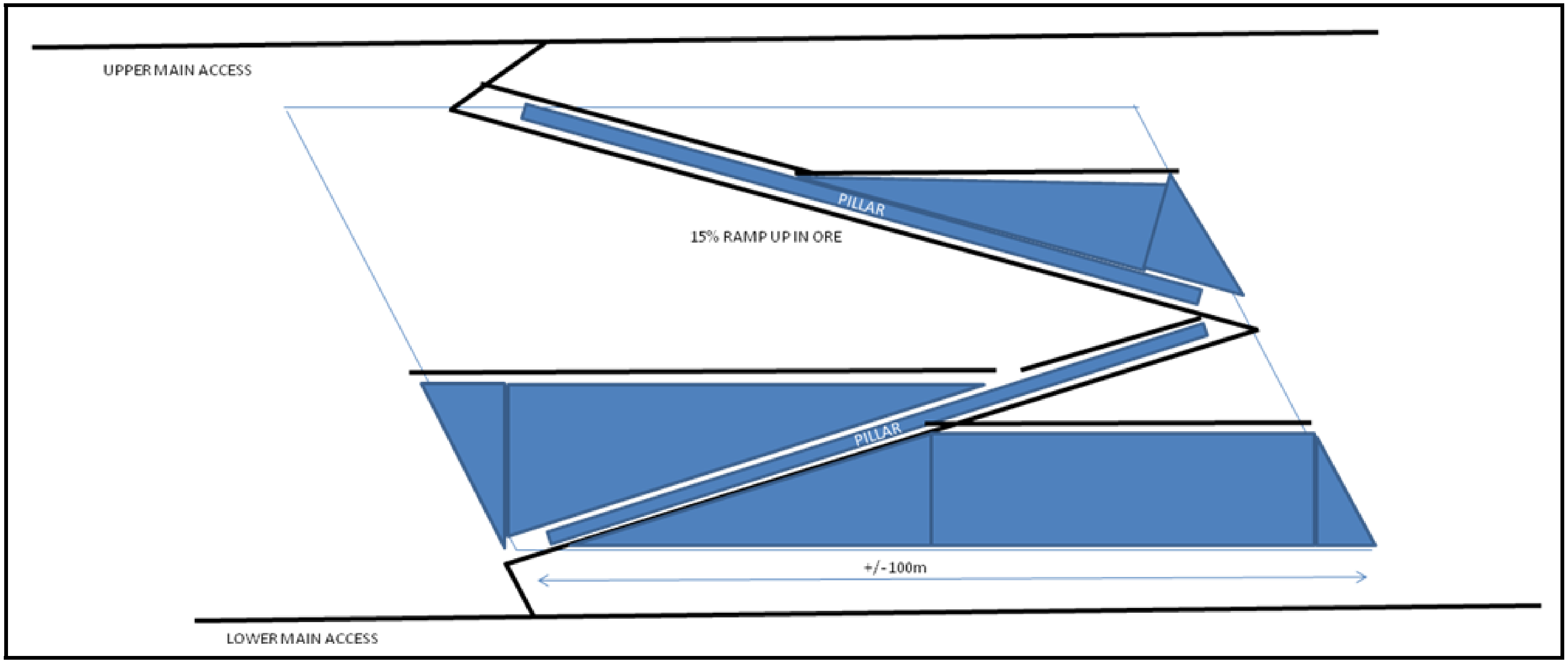
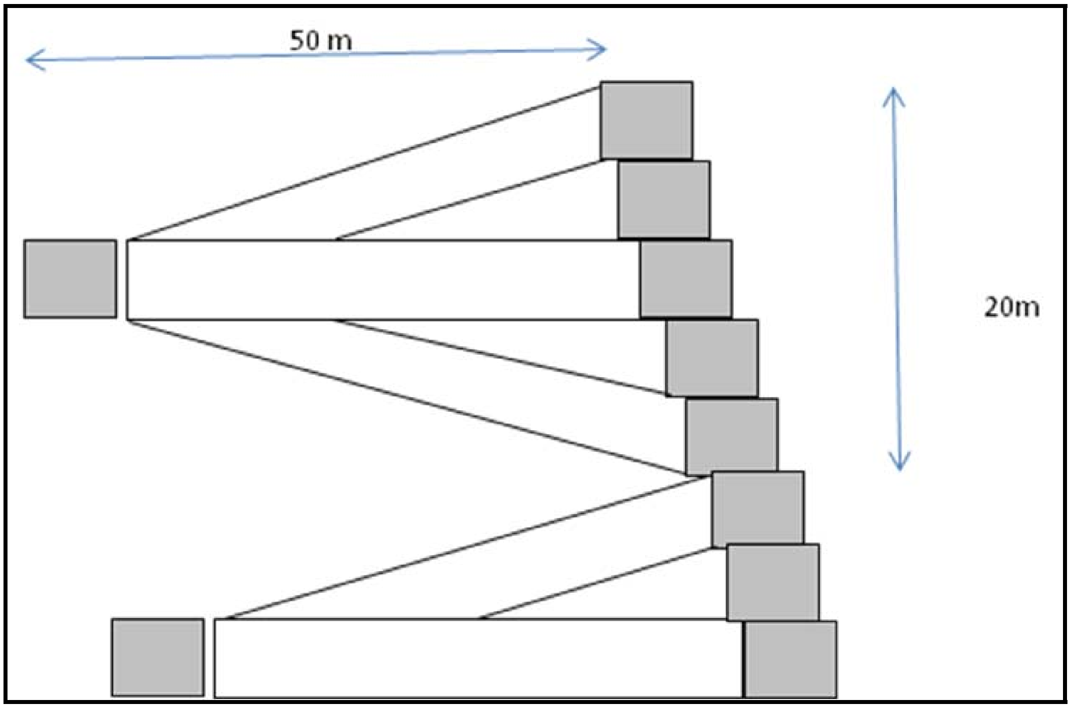


Figure 18.18 Three Working Faces per Block Configuration



Naturally, the extraction within the block will suffer and so a trade-off evaluation was undertaken of the pillar losses against the cost of normal cut-and-fill attack ramp system as shown in Figure 18.19.

Figure 18.19 Pillar Losses vs. Cost of Normal Cut-and-Fill Attack Ramp System



The grade in the stope block (and hence in the pillar) needs to be high enough to more than pay for the footwall development. The estimated revenue per gram of silver is shown in Table 18.17.

Table 18.17 Estimated Revenue per Gram

Silver	**Ag oz**	**1.00**
Lead Concentrate		
Recovery	83%	0.83
Payable	87.5%	0.73
	72.6%	
Zinc Concentrate		
Recovery	9%	0.09
Payable	40.0%	0.04
	3.600%	
Total Concentrate		
Recovery	92%	0.92
Payable		0.76
Average Ag % Payable		**76%**

The revenue received from 1 g of silver is estimated in Table 18.18.

Table 18.18 Revenue from 1 g of Silver

3-year Silver Price	$13/oz
% Price Realized	76%
Realized Revenue	$0.32/g

Thus each gram of silver in the pillar will realize cash in hand of $0.32.

Table 18.19 estimates the cost of an attack ramp system at $1,000,000, and at a revenue of $0.12/g Ag would require 8,100,260 g silver to pay for it.

Table 18.19 Cost Estimate of an Attack Ramp System

Attack Ramp Gradient	17%
Cut Height	4 m
Ramp Distance is Slash	23.5 m
Volume of 1st Slash	188.2 m^3
Volume of 2nd Slash	564.7 m^3
Total	
2 x Slash 20 m Height	1,505.9 m^3
Cost per Cubic Metre	$500
Cost	$752,941
Footwall Ramp	133 m
Cost per Metre	1,300
Footwall Ramp Cost	$173,333
Total Footwall Development	$1,006,2741
3 Attacks in 60 m High Block	$3,018,824
No. of Grams Ag to Pay for This	9,476,046

Consequently, if the pillar contains more than 9,476,046 g Ag, it is worthwhile doing the footwall attack ramp system rather than the in-stope development. This is converted to a grade to determine which mining blocks would be better suited to have footwall development. Table 18.20 considers a 60 m block height.

Table 18.20 60 m Block Height

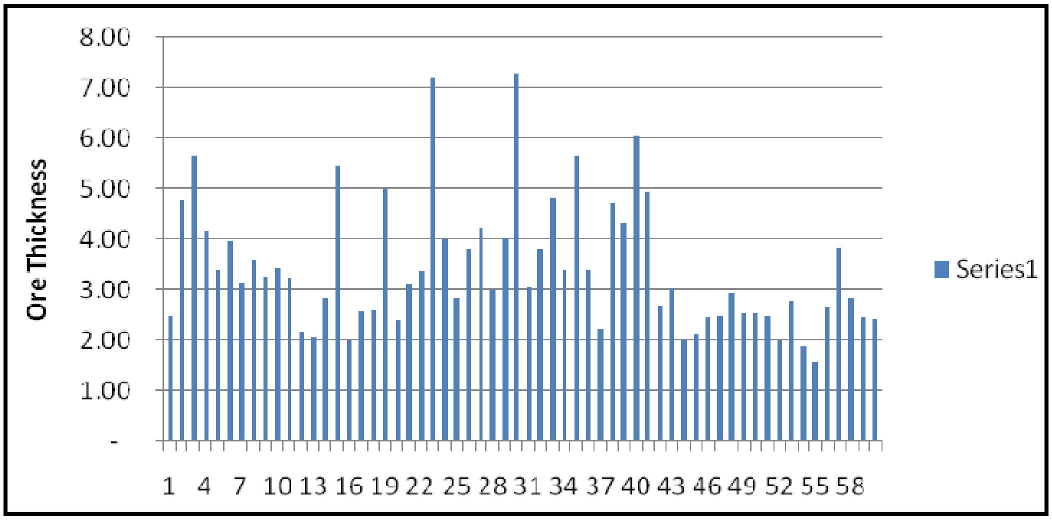
Pillar Length	Pillar Width	Ore Width	Extraction	Tonnes in Pillar	Grade to Pay Dev't	Break Even COG*	BE Pillar Grade	BE Pillar Grade with 50% Rec. of Pillar at End
400	3	2	91%	6720	1,410.13	119	1,527.13	3,054.25
400	3	3	91%	10080	940.08	119	1,057.08	2,114.17
400	4	4	88%	17920	528.80	119	645.80	1,291.59
400	5	5	85%	28000	338.43	119	455.43	910.86
400	6	6	83%	40320	235.02	119	352.02	704.04
400	7	7	80%	54880	172.67	119	289.67	579.34
400	8	8	77%	71680	132.20	119	249.20	498.40
400	9	9	74%	90720	104.45	119	221.45	442.91

* COG = cutoff grade.

Thus, it is unlikely that any blocks may pass the test for footwall development. The average thickness of the dipping blocks is about 3.4 m and would require a grade of over 1,000 g/t Ag, which is well above the grade of the highest grade mining blocks.

Figure 18.20 shows the distribution of the widths of the dipping blocks.

Figure 18.20 Dipping Block Thickness



Wardrop therefore recommends that the in-ore development method be adopted.

Backfill

All stopes will be backfilled after mining is completed. Due to the flat lying and relatively large horizontal extent of the orebody, as well as the distant location of the process plant and difficulties with access above the orebody, free draining hydraulic backfill was selected as the most appropriate method. This backfilling method will allow up to 45 to 50% of the tailings to be disposed of as hydraulic backfill underground, reducing the required size of the surface tailings pond.

Backfill materials will be prepared from the tailings produced from the plant and distributed to the underground stopes by pipeline through the main access ramp.

Backfill Material Preparation

On average, approximately 97.5% of the mill feed will report to the zinc rougher scavenger tailings, or the final tailings. The tailings will contain approximately 27% solid with a particle size of 80% passing approximately 95 to 100 microns. Table 18.21 presents the typical tailings size distribution estimated from the PRA metallurgical test results.

Table 18.21 Typical Tailings Distribution

Sieve Size		Individual % Retained	Cumulative - % Passing
Tyler Mesh	Microns		
65	210	0.4	99.6
100	149	3.4	96.3
150	105	11.9	84.4
200	74	16.4	67.9
270	53	10.4	57.6
325	44	4.2	53.4
400	37	4.4	49.0
-400	-37	49	
Total		**100**	

It appears that the weight percent of the coarser than 35 micron materials would be approximately 50%.

The flotation tailings will be pumped to the backfill material preparation station. The tailing will be classified in a hydrocyclone cluster with six 250 mm hydrocyclones into a coarse fraction and a fine fraction. The coarse fraction will be used as underground backfill material while the fine material will be stored on the surface tailing management facility (TMF). The tailing handling flowsheet is shown in Drawing A0-09-008 in Appendix B. The coarse material which is approximately 50% of the total tailings will gravity flow into a 4 m diameter by 4.5 m high backfill surge mixing tank equipped with double blades.

Normally, the thickened coarse material with a solid density of approximately 60% will be directly pumped to the mined stopes. However, according to the mining requirement, some of the stopes will require cemented backfill. In this case, cement (estimated at 5% by weight on average) will be reclaimed from a 50-t binder silo and added to the backfill surge mixing tank where the cement and the hydraulic tailings are blended. The blended backfill material will then be sent to the stopes.

Table 18.22 below provides estimated backfill plant parameters with three different cement addition dosages.

Table 18.22 Backfill Material Parameters

Material	Unit	Parameters		
Cement Addition %	%	4.0	5.0	10.0
Tailings Parameters				
Solid Tailings Tonnage	dry t/h	60.9	60.9	60.9
Density of Solid Tailings	t/m^3	2.60	2.60	2.60
Solid Tailings (Dry)	m^3/h	23.4	23.4	23.4
Percentage Solids of Tailings	%	66.4	66.4	66.4
Water in Tailings	m^3/h	30.8	30.8	30.8
Tailings Pulp Tonnage	t/h	91.7	91.7	91.7
Density of Tailings Pulp	t/m3	1.69	1.69	1.69
Cement Parameters				
Density of Cement	t/m^3	3.15	3.15	3.15
Binder Consumption	dry t/h	2.4	3.0	6.1
Backfill Material Parameters				
Target Percentage Solids	%	65.0	65.0	65.0
Backfill Material Solid Tonnage	dry t/h	63.3	63.9	67.0
Total Water in Backfill	m^3/h	34.1	34.4	36.1
Backfill Pulp Tonnage	t/hr	97.4	98.4	103.1
Backfill Pulp Volume	m^3/h	58.3	58.8	61.4
Backfill Mixer				
Backfill Surge Tank Mixer Capacity	t/h	97.4	98.4	103.1

Pumping and Reticulation

The backfill material will be pumped to underground through 150 mm pipelines through the main ramp and along the access haulage drifts to the stopes. After each backfill pour the pipelines will be flushed with water.

18.1.5 MINE ACCESS

Shaft access for hoisting of ore is not considered viable, due to:

- the geometry of the orebody (flat lying and with variable depths)
- the location of the plant some distance from the orebody
- access difficulties above and adjacent to the orebody.

Transport of ore from a shaft located close to the orebody will require surface right of way for a conveyor or a haul road and will likely be close to the proposed university. Location of a shaft at the plant location will require extensive underground drift access in addition to the access ramp. Truck haulage will be required to deliver ore and waste to the shaft bottom which, in essence, would be equivalent to hauling up the access ramp to surface. Shaft access is thus not viable.

Conveyor haulage up the access ramp was not considered viable as the most efficient ramp design requires a spiral ramp below the first level, which is not conducive to conveyor installation. A conveyor system would require a number of underground transfer points, which are difficult for operations. Further, a conveyor system will require the installation of a crusher facility, and ore and waste bins. Trucks will be required to haul the ore to a crusher station and, when considering the costs of the conveyor system, it will be more economical to truck the ore up the ramp rather than provide the infrastructure for a conveyor.

The mine will thus be accessed by a single decline developed at gradient of -15%. It will be used for access of personnel, equipment, materials, and services; it will also be utilized as an intake airway.

The location of the decline portal was selected on the south side of the deposit near the process plant. The size of the decline was selected at 4.5 m wide by 4.5 m high to accommodate the mining equipment. Remuck bays will be developed every 150 m along the decline and level development to allow efficient use of the drilling equipment. They will be of a similar size as the decline and will be typically 15 m long.

Main level access drifts will be developed from the main access decline at four successively deeper levels: -65 m (95 m depth), -95 m (125 m depth), -125 m (155 m depth), and 175 m elevations (205 m depth). The surface elevation of the portal is at elevation +30 m.

The four levels will be used for haulage and for provision of fresh air supply to mining blocks. Ventilation access drifts will be excavated to connect the level development and ramps to the ventilation raises. An intake raise will be developed centrally on the south side of the deposit where the ramp meets the top of the orebody and at the west extremity of the orebody (in Year 6). One exhaust raise will be developed at the eastern extremity of the orebody and a second exhaust raise will be located centrally on the north side of the orebody. The four main access levels will connect to these raises to provide through ventilation. Ventilation doors, regulators, or bulkheads will be installed as required to provide an appropriate air distribution between the levels at different stages of mining.

The central south fresh air intake ventilation raise will have a 4.0 m diameter and will have a manway equipped with ladders and platforms to provide an auxiliary exit from the mine in case of emergency. It will be constructed in stages as the mine deepens. The first stage will be 145 m long from surface to -65 m level to provide flow through ventilation during the pre-production period. The next stages will be constructed when south decline development advances downwards to the next levels. Another fresh air ventilation raise will be constructed in Year 6 on the west side of the deposit to provide intake air and emergency egress for mining block #201. It will be 3.0 m diameter, 61 m long, and equipped with a manway for emergency exit.

The two exhaust ventilation raises will be 4.0 m diameter and will be developed on the north and east sides of the deposit to provide return airways. Similar to the fresh air raise, they will be developed in stages based on availability of the underground access to the bottom of the raises.

In the pre-production stage, 104 m of the north exhaust raise will be developed from surface to -65 m level, and 41 m will be developed between -65 m and -95 m level. The raise will be extended down by 43 m to the -175 m level in Year 3 of production.

The east exhaust raise will be developed in two stages. The initial 109 m of raise will be drilled from surface to -65 m level and an additional 45 m will be developed in Year 1 to extend the raise one level down.

An additional ventilation raise (3.0 m diameter, 150 m long) will be constructed internally between -65 m level and -225 m level.

Raiseborers will be used for development of the raises. Some short ventilation raises will be developed by conventional methods on the north side of the deposit to connect the cut-and-fill mining block development to the north exhaust decline, providing return airways and emergency egress from the mining blocks.

Figure 18.21 Mine Access Plan View

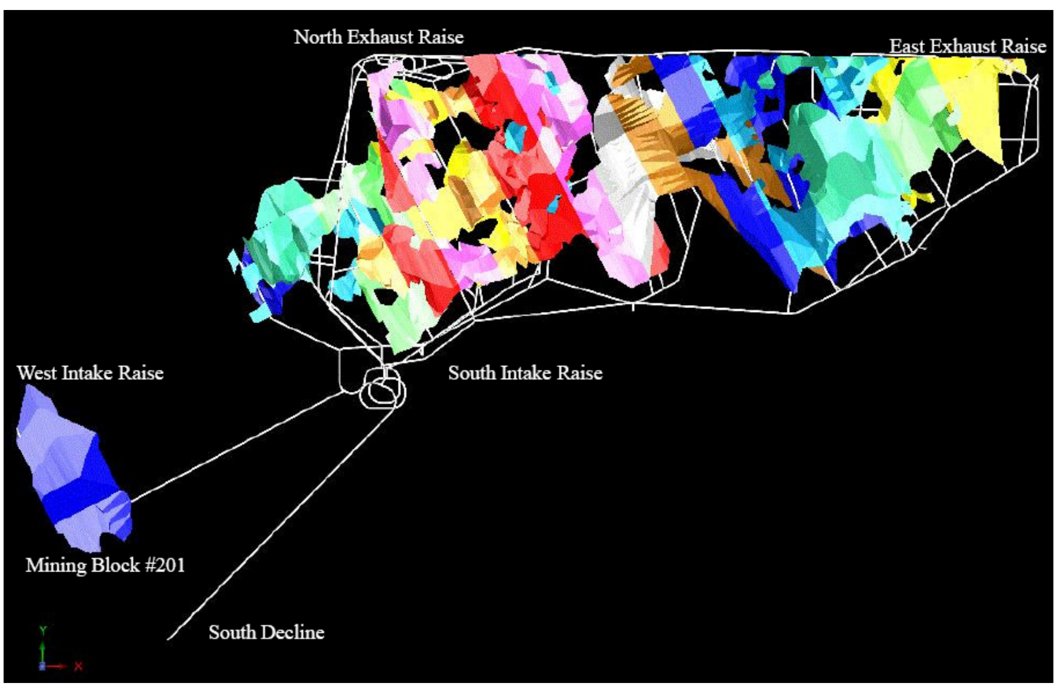
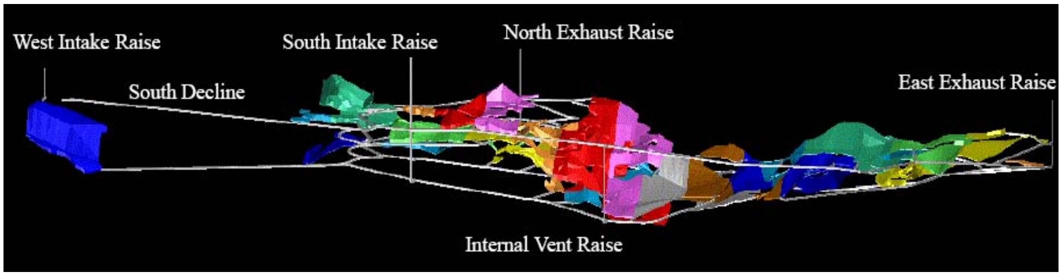


Figure 18.22 Mine Access Section View



18.1.6 *MINE DEVELOPMENT*

Development headings will be driven with electro-hydraulic twin boom jumbos taking 2.9 m rounds by drilling and blasting of 3.2 m long, 45 mm diameter holes. The holes will be loaded with ANFO from a pneumatic loader and blasting initiated with NONEL caps. Smooth perimeter drilling and blasting techniques will be used to reduce damage to the walls and back and to minimize ground support requirements.

The broken rock will be mucked from the face by a 4.0 m^3 LHD directly to the trucks or to the remuck bays located at 150 m intervals. Waste will be hauled to surface waste dump by 25 t trucks. Remuck bays will subsequently act as temporary sumps and as spaces for electrical substations, parking, material storage, etc.

The back and the walls of the headings will be scaled and ground support will be installed. A typical installation of 2.4 m long resin rebar in a 1.2 m by 1.2 m pattern will be used for ground support of the access development.

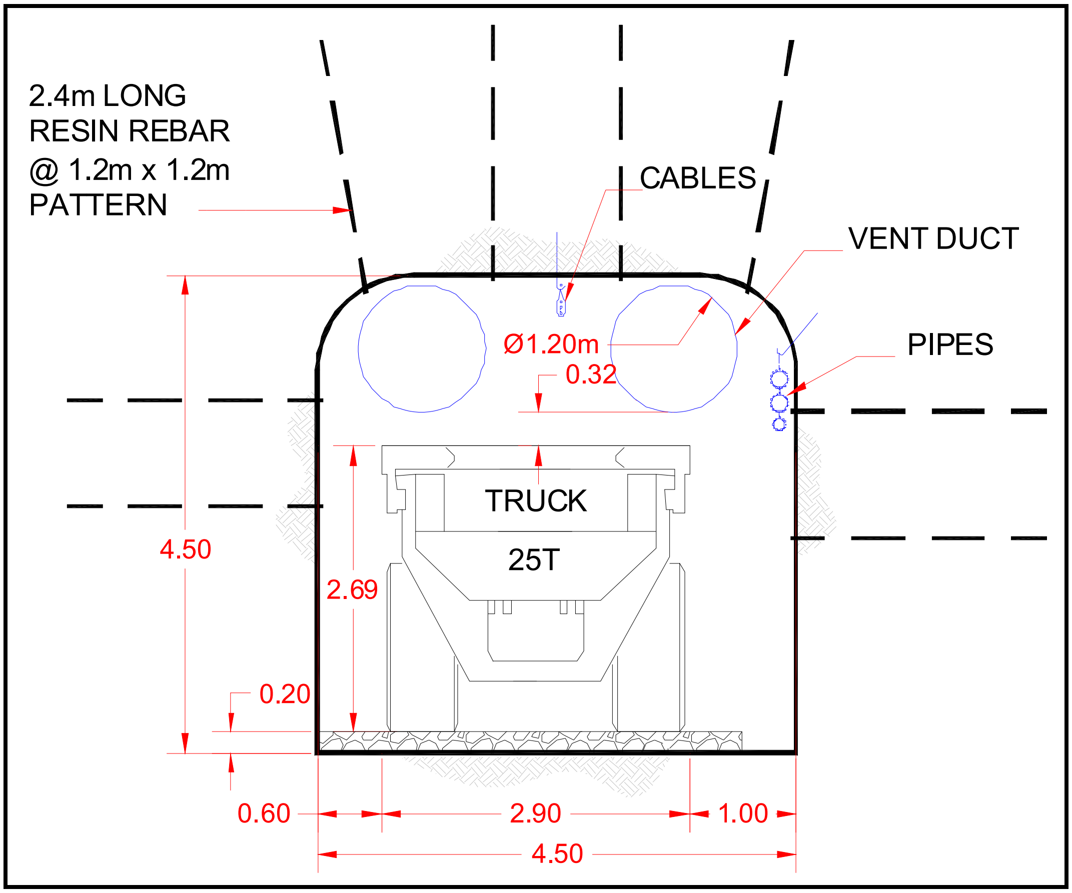
Pipelines, ventilation ducts, and power cables will be installed as the headings advance to maintain services.

An average advance for 4.5 m wide by 4.5 m high development heading was estimated at 5.2 m/d for single heading and 6.4 m/d for multiple headings. Jumbo crew monthly advance rate of 130 m was assumed for mine development scheduling.

Ventilation raise development will be done by raiseboring crews. A 349 mm diameter pilot hole will be drilled vertically from the top down. An underground access will be developed to the bottom of the pilot hole. The pilot bit will be removed and a raiseboring head (reamer) will be attached. Then the pilot hole will be reamed to the final size of the raise from the bottom up.

A typical cross section of for 4.5 m wide by 4.5 m high development heading is shown in Figure 18.23.

Figure 18.23 Typical Cross Section of Mine Access Development Heading



18.1.7 HAULAGE

The waste rock from development headings will be mucked by LHDs directly to the trucks or to remuck bays located up to 150 m from the face and then hauled by trucks to the waste dump on surface.

The broken ore will be mucked from the production stopes by LHDs to truck loading bays within stopes or at stope access points. It will be loaded at the truck loading points onto low profile underground truck and hauled to the mill for processing.

Development LHDs and trucks will be the same size and capacity as the ore production equipment. The following equipment was selected for ore and waste haulage:

- 7 t capacity LHD with 4.0 m^3 bucket
- 25 t underground mine truck with 13.0 m^3 box.

The selection of LHD and trucks was based on LHD bucket and truck box capacities, to allow for loading the trucks with three passes and on available Chinese equipment.

The average mucking distance from the stope to a truck loading point and truck haulage distances to the mill were extracted from the mine design model to estimate mucking and trucking productivities and cost. For mucking and haulage productivities and fleet requirements estimates, 50% of swell factor, 90% of box fill factor, 85% of truck, and 75% of LHD mechanical availability were used.

Ore haulage parameters are provided in Table 18.23. Annual material movement and haulage fleet requirements are shown in Table 18.24.

Table 18.23 Haulage Estimate Input Parameters

	Unit	Qty
LHD		
Capacity	m^3	4.00
Fill Factor		0.90
One Way Haul	m	0
Average Speed	km/h	5.00
Load Time	min	0.60
Dump Time	min	0.20
Manoeuvre	min	0.50
Availability		0.75
Truck		
Capacity	t	25.00
Estimated Truck Fill Factor		0.90
No. of LHD/Truck		1.00
No. of Trucks Assigned		2.00
Spot Time	min	1.00
Dump Time	min	1.00
Waste Surface Flat Haul	m	500
Ore Surface Flat Haul	m	100
Loaded Speed Flat	km/h	20.0
Loaded Speed Ramp	km/h	6.5
Empty Speed Flat	km/h	20.0
Empty Speed Ramp	km/h	12.5
Availability		0.85
Useful Hours per Shift	h/shift	6.00

Table 18.24 Annual Material Movement and Haulage Fleet Requirements

	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Ore			990,000	990,000	990,000	990,000	990,000	990,000	990,000	990,000	990,000	207,981	9,117,981
Waste	83,515	226,832	83,486	83,720	63,183	52,480	57,452	43,329	11,932	20,108	19,887		745,924
Total	**83,515**	**226,832**	**1,073,486**	**1,073,720**	**1,053,183**	**1,042,480**	**1,047,452**	**1,033,329**	**1,001,932**	**1,010,108**	**1,009,887**	**207,981**	**9,863,905**
LHD	2	2	6	6	6	6	6	6	5	5	5	5	
Truck	2	2	7	8	9	9	10	10	8	7	7	7	

18.1.8 EQUIPMENT

Criteria used in the selection of underground mining equipment include:

- mine production rate
- mining method
- orebody geometry and dimensions
- capital cost.

The equipment list was developed from first principle cycle time estimates and based on the following factors and adjustments:

- each 8 h shift has a useable work time of 6 h (75% shift efficiency) to reflect shift change, travel time, and breaks
- an effective 50 min/h (83% hour efficiency)
- equipment mechanical availability
- fill and load factors for LHD and haulage trucks
- move in and out time for equipment between the cycles of operation such as drilling, blasting, etc.

The following mobile equipment will be used underground:

- drilling equipment includes diesel electric-hydraulic twin-boom jumbos for lateral and ramp development, mechanized rock bolters for ground support, jacklegs and stoppers for handheld drilling on an occasional basis, and mucking and hauling equipment including 7 t-capacity LHDs and 25 t-capacity trucks
- service vehicles will be used for mine maintenance and transportation of personnel and supplies
- shotcreting equipment will be used for mixing, delivering the shotcrete, and spraying it on the rock surface.

The stationary equipment will include the following:

- ventilation equipment including primary and auxiliary fans, and fan houses
- dewatering equipment including main dirty water pumps, face submersible pumps, and pump stations
- fuel storage and delivery
- electrical equipment
- communications

- mechanical shop equipment
- safety equipment
- engineering equipment.

Table 18.25 lists underground equipment by type and quantity.

Table 18.25 Mine Equipment Requirements by Year

Equipment	Year										
	-1	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
LHD	2	6	6	6	6	6	6	5	5	5	5
Jumbo	2	4	5	6	6	6	6	6	5	5	5
Bolter	1	2	2	2	2	2	2	2	2	2	2
Truck	2	7	8	9	9	10	10	8	7	7	7
Jackleg		10	10	10	10	10	10	10	10	10	10
Stoper		10	10	10	10	10	10	10	10	10	10
Portable Compressor		6	6	6	6	6	6	6	6	6	6
Grader		2	2	2	2	2	2	2	2	2	2
Explosive Truck		2	2	2	2	2	2	2	2	2	2
Mechanics Truck		3	3	3	3	3	3	3	3	3	3
Fuel/Lube Truck		1	1	1	1	1	1	1	1	1	1
Supervisor Vehicle		6	6	6	6	6	6	6	6	6	6
Electrician Vehicle		1	1	1	1	1	1	1	1	1	1
Scissor Lift		3	3	3	3	3	3	3	3	3	3
Survey Vehicle		1	1	1	1	1	1	1	1	1	1
Engineering Vehicle		1	1	1	1	1	1	1	1	1	1
Shotcrete Robo		2	2	2	2	2	2	2	2	2	
Transmixer		4	4	4	4	4	4	4	4	4	
Surface Shotcrete Batch Plant		1	1	1	1	1	1	1	1	1	
Surface Fan, 190 kW		2	2	2	2	2	2	2	2	2	2
Auxiliary Fan, 75 kW		10	16	16	16	16	16	16	16	16	16
Auxiliary Fan, 45 kW		10	16	16	16	16	16	16	16	16	16
Dirty Water Pump		4	4	8	8	12	12	12	12	12	12
Face Pump		10	16	16	16	16	16	16	16	16	16

18.1.9 LABOUR

The mining employees at the underground operation are divided into two categories: salaried personnel and hourly labour. The personnel requirement estimates are based on the following:

- a 3,000 t/d production rate
- the production and development schedule

- a crew rotation of three 8-h shifts per day
- 330 production days per year

A mining contractor will begin work in the pre-production development stage of the mine life to allow time for the Owner to recruit staff for the project. Three jumbo crews and one raiseboring crew will be required at the peak of pre-production period. One contractor jumbo crew will continue the mine access development during the production.

The labour and personnel requirements described in this section do not include the labour required for access development performed by the contractor.

Underground staffing requirements peak at 54 personnel during full production including 9 mine operating and 5 mine maintenance salaried dayshift personnel, 32 shift technical staff, and 8 shift supervisors.

Salaried personnel requirements, including engineering, technical, and supervisory staff, are listed in Table 18.26.

Table 18.26 Technical and Supervisory Staff

	Year										
	-1	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
DAILY REQUIREMENTS											
Staff Mine Operation											
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1
Chief Mining Engineer		1	1	1	1	1	1	1	1	1	1
Senior Mine Engineer	1	1	1	1	1	1	1	1	1	1	1
Mine Engineer/Planner		1	1	1	1	1	1	1	1	1	1
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1
Senior Geologist		1	1	1	1	1	1	1	1	1	1
Mine Rescue/Safety Officer		1	1	1	1	1	1	1	1	1	1
Mine Technician/CAD Operator		1	1	1	1	1	1	1	1	1	1
Chief Surveyor		1	1	1	1	1	1	1	1	1	1
Total Dayshift Mine Operating Staff	**3**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**
Staff Mine Maintenance											
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planner		1	1	1	1	1	1	1	1	1	1
Mechanical Foreman		1	1	1	1	1	1	1	1	1	1
Electrical Foreman		1	1	1	1	1	1	1	1	1	1
Light Mechanic		1	1	1	1	1	1	1	1	1	1
Total Dayshift Mine Maintenance Staff	**1**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**
Total Dayshift Mining Staff	**4**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**

table continues…

	Year										
	-1	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
SHIFT REQUIREMENTS											
Shift Technical Staff											
Shift Geologist		8	8	8	8	8	8	8	8	8	8
Geologist Technician	4	8	8	8	8	8	8	8	8	8	8
Surveyor	4	4	8	8	8	8	8	8	8	8	8
Surveyor Helper	4	4	8	8	8	8	8	8	8	8	8
Total Shift Staff	**12**	**24**	**32**	**32**	**32**	**32**	**32**	**32**	**32**	**32**	**32**
Mine Supervision											
Mine Supervisor/Shift Boss (Production)	4	8	8	8	8	8	8	8	8	8	8
Total Mine Supervision Staff	**4**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**
TOTAL MINE TECHNICAL AND SUPERVISORY STAFF	**20**	**46**	**54**	**54**	**54**	**54**	**54**	**54**	**54**	**54**	**54**

Hourly personnel were estimated based on production and development rates, operation productivities, and maintenance and services requirements. Personnel productivities were estimated for all main activities by developing cycle times for each operation. Underground hourly labour requirements peak at 312 in Year 5 during full production, including 248 mine operating and 64 mine maintenance hourly personnel.

Table 18.27 Hourly Labour

	Year										
Labour Description	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Hourly Mine Labour											
Production											
Production Driller	12	24	28	32	32	32	32	32	28	28	28
Production Drill Helper	12	24	28	32	32	32	32	32	28	28	28
Production Blaster	4	8	8	8	8	8	8	8	8	8	8
Blaster Helper	4	8	8	8	8	8	8	8	8	8	8
Backfill Crew		16	16	16	16	16	16	16	16	16	16
Shotcrete Robo Operator		8	8	8	8	8	8	8	8	8	
Transmixer Operator		16	16	16	16	16	16	16	16	16	
Batch Plant Operator		8	8	8	8	8	8	8	8	8	
Haulage											
Scoop-Loader Operator	8	24	24	24	24	24	24	20	20	20	20
Truck Drivers	8	28	32	36	36	40	40	32	28	28	28

table continues…

Labour Description	Year										
	-1	1	2	3	4	5	6	7	8	9	10
Mine Services & Safety											
Service Crew	8	12	12	12	12	12	12	12	12	12	12
Grader Operator		8	8	8	8	8	8	8	8	8	8
Utility Vehicle Operator/Nipper	4	4	4	4	4	4	4	4	4	4	4
General Labourer	8	32	32	32	32	32	32	32	32	32	32
Sub-total Mine Operating	**68**	**220**	**232**	**244**	**244**	**248**	**248**	**236**	**224**	**224**	**192**
Mine Maintenance											
Electrician	4	4	4	4	4	4	4	4	4	4	4
HD Mechanic	4	16	16	20	20	20	20	16	16	16	16
Mechanic Helper	4	16	16	20	20	20	20	16	16	16	16
Welder	4	4	4	4	4	4	4	4	4	4	4
Tool Crib		8	8	8	8	8	8	8	8	8	8
Millwright	4	4	4	4	4	4	4	4	4	4	4
Millwright Helper	4	4	4	4	4	4	4	4	4	4	4
Sub-total Mine Maintenance	**24**	**56**	**56**	**64**	**64**	**64**	**64**	**56**	**56**	**56**	**56**
GRAND TOTAL HOURLY LABOUR	**92**	**276**	**288**	**308**	**308**	**312**	**312**	**292**	**280**	**280**	**248**

18.1.10 VENTILATION

The ventilation system designed for the Minco Fuwan silver mine is a system exhausting approximately 290 m^3/s. Two main exhaust fans and a number of auxiliary fans will control the primary ventilation circuit for full production of 3,000 t/d. The main exhaust fans will be installed on surface. During production, fresh air will enter through the main decline ramp and the main intake fan, and will exit via the main exhaust ventilation fans through the two main exhaust ventilation raises. The main exhaust fans will be able to be reversed if required in an emergency.

The ventilation system designed for the Fuwan silver underground mine is consistent with regulations applied by the Chinese occupational health and safety standards.

PRIMARY VENTILATION

Primary ventilation will be by two surface mounted stand-alone axial mine fans exhaust (one on each exhaust raise).

Airflow in the mine will be controlled by ventilation regulators, doors, and flaps placed appropriately for the individual mining activities taking place at any time in the mine.

As the mine develops and deepens, operating pressures and air volumes required to ventilate the mine will increase. Fan performance will be adjusted to meet these changing requirements by changing fan speeds, blade pitch, and the number of auxiliary fans operating.

The mine ventilation requirements were derived from the diesel equipment list and based on the Chinese regulation requirement of 4 m^3/min/hp or 0.0667 m^3/s/kW. Air velocity is restricted on the haulage levels to a minimum of 0.25 m/s and a maximum of 6 m/s.

Table 18.28 lists the total air volume required. The full production ventilation requirements are 290 m^3/s. A utilization factor of 95% was applied to the main loading and hauling diesel equipment. Ventilation losses were included at 15% of the total ventilation requirements.

The ventilation system design was modelled using Ventsim Mine Ventilation Simulation Software (Ventsim). This software allows input parameters including resistance, k-factor (friction factor), length, area, perimeter, and fixed quantities (volume) of air. Underground ventilation control will require ventilation control doors, regulators, and auxiliary fans to direct air quantities to the workings as necessary.

The ventilation circuit and intake and exhaust air quantities (m^3/s) and air velocities (m/s) are presented in Figure 18.24 and Figure 18.25, respectively.

Table 18.28 Estimated Annual Ventilation Requirements at Full Production

Total Power Output	Year-1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Usage	Mech Avail
LHD	242	727	727	727	727	727	727	606	606	606	606	0	95%	75%
Haul Trucks	275	961	1,098	1,235	1,235	1,373	1,373	961	961	961	961	0	95%	85%
Graders	0	158	158	158	158	158	158	158	158	158	158	158	95%	80%
Jumbos/ Bolters	71	142	166	189	189	189	189	189	166	166	166	0	40%	80%
Scissor Lifts	0	90	90	90	90	90	90	90	90	90	90	90	50%	80%
Light Trucks	0	312	312	312	312	312	312	312	312	312	312	312	50%	80%
LVs	0	544	544	544	544	544	544	544	544	544	544	544	50%	80%
Transmixer	0	338	338	338	338	338	338	338	338	338	338	338	70%	80%
Shotcrete Robo	0	48	48	48	48	48	48	48	48	48	48	48	20%	80%
Total kW	**588**	**3,320**	**3,481**	**3,642**	**3,642**	**3,779**	**3,779**	**3,246**	**3,223**	**3,223**	**3,223**	**1,490**		
Total Mine Flow, m^3/s	**39**	**221**	**232**	**243**	**243**	**252**	**252**	**216**	**215**	**215**	**215**	**99**		
Incl. 15% Loss, m^3/s	**45**	**255**	**267**	**279**	**279**	**290**	**290**	**249**	**247**	**247**	**247**	**114**		
Total Mine Flow incl. Loss, CFM	**95,550**	**539,646**	**565,808**	**591,970**	**591,970**	**614,284**	**614,284**	**527,656**	**523,807**	**523,807**	**523,807**	**242,230**		

Figure 18.24 Full Production Ventilation Circuit Air Quantities (m^3/s)

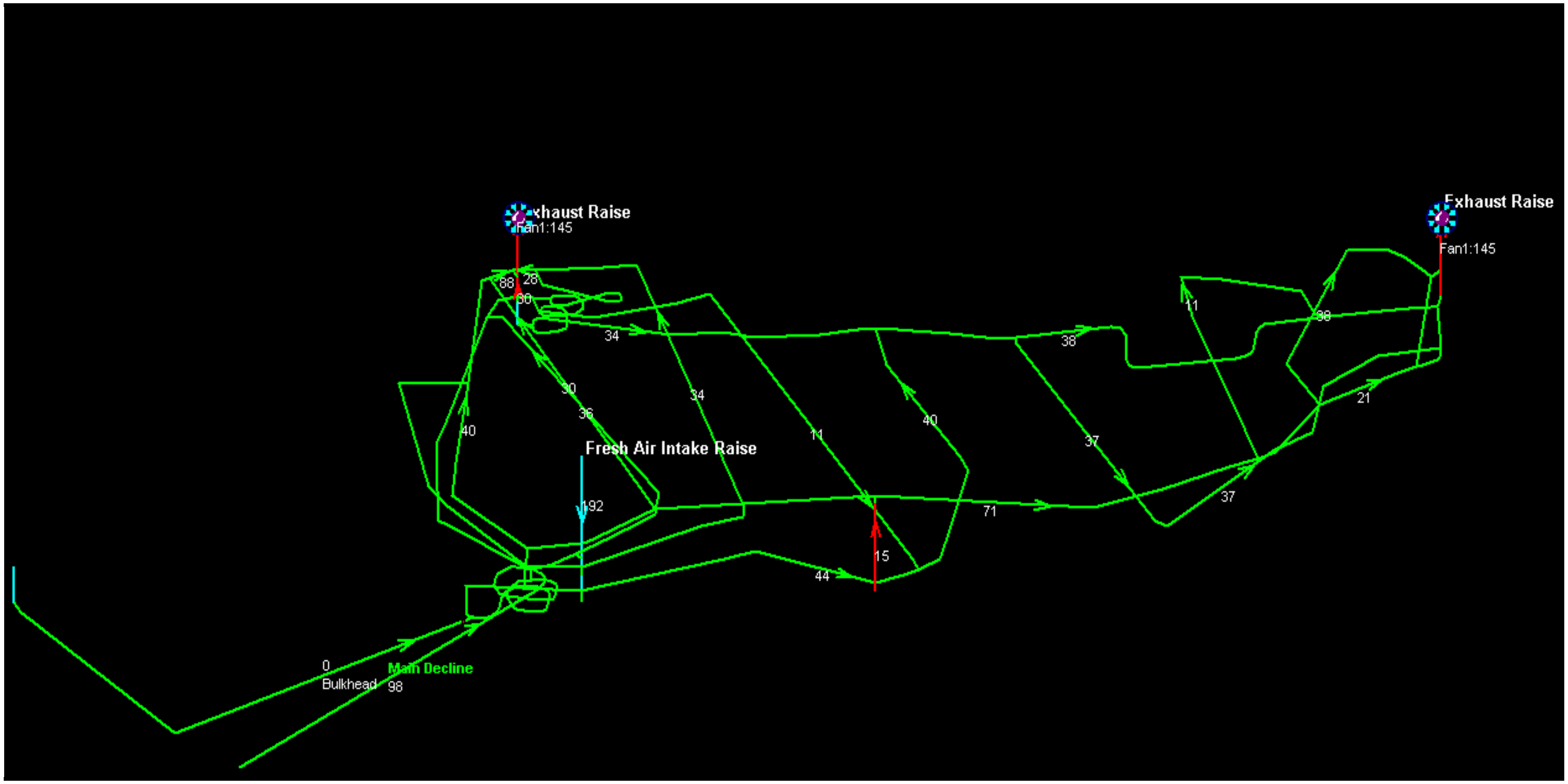
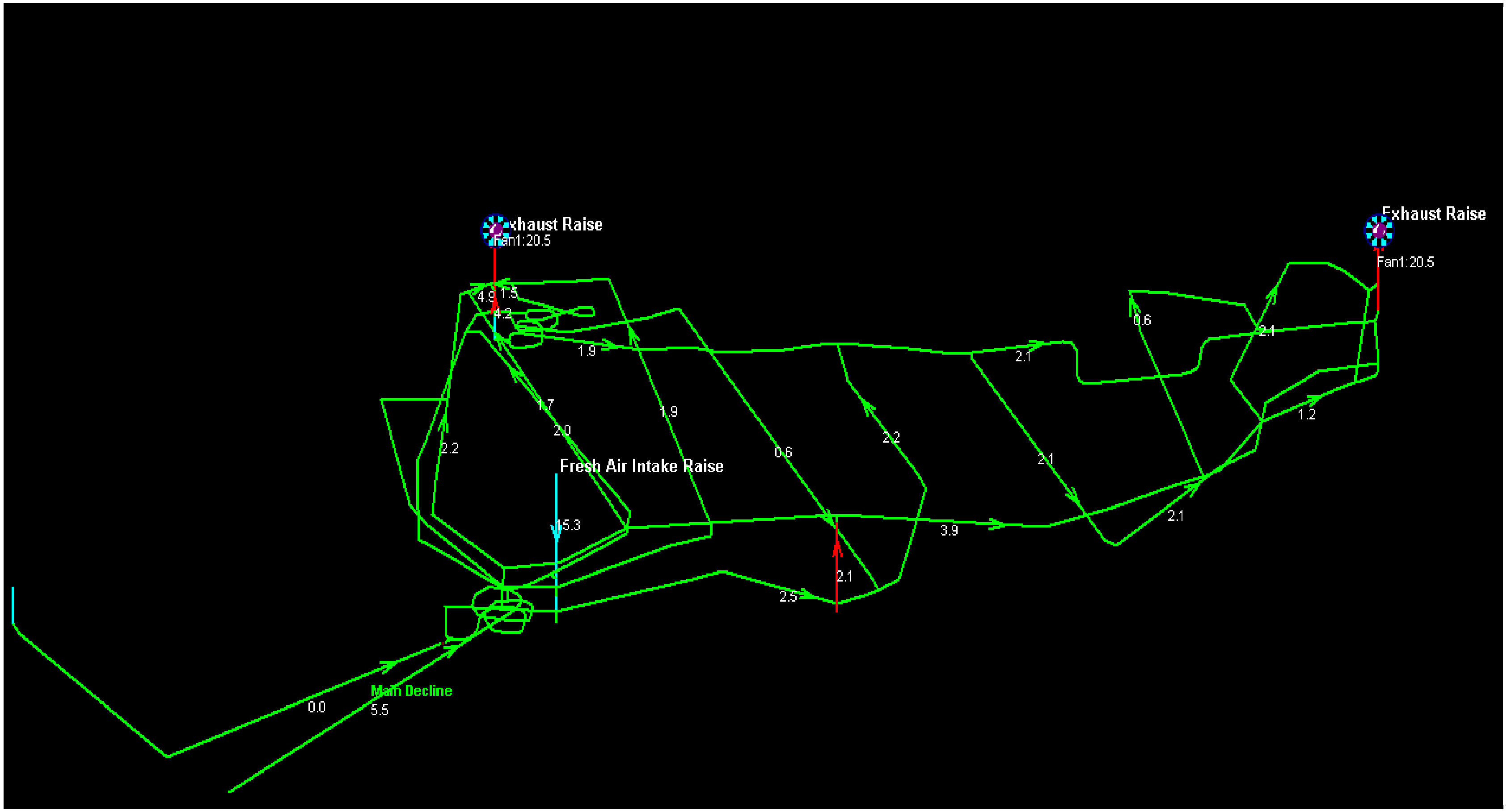


Figure 18.25 Full Production Ventilation Circuit Air Velocities (m/s)



Ventilation of Headings during Development

Auxiliary fans will maintain a 21 m^3/s air flow in the development headings. This air flow rate is required to dilute and remove exhaust from the 7 t LHD, 25 t haul truck, and a double-boom jumbo. Table 18.29 provides the detailed calculations.

Table 18.29 Production Ventilation Requirements for Development

Description	Qty	Diesel (hp)	Utilization (%)	Utilized (hp)	Air Volume (m^3/s)
LHD	1	228	95	217	9.7
Haul Truck	1	228	95	217	9.7
Jumbo	1	99	40	40	1.8
Total					**21.2**

The development ventilation system is designed to distribute air for either 800 m or 1,500 m long openings using TwinDuct® ducting system with 107 cm (42″) effective diameter. The Alphair Ventilating Systems Inc. (Alphair) Model 4800-VAX-2100 fan (1.2 m diameter, 19° blade angle, 90 hp), or a fan with similar capacities of another manufacturer, has been selected for the auxiliary ventilation during development.

Fan Selection

The selection of the main exhaust fans is based on the maximum operating duty for specific conditions. For the exhaust fans, Wardrop assumed an average fan efficiency of 75% and 1.36 kg/m^3 air density.

Two axial flow fans with variable pitch control will be used for the permanent ventilation system (one fan at each of the ventilation raises). The simultaneous blade adjustment provides a wide range of fan settings within a single speed; the fan volume flow rate can also be adjusted to accommodate varying conditions. Fan selection is based on the total pressure and total volume needed to deliver the required quantity of air. A possible main surface exhaust fan for the underground ventilation system is by Alphair with the following specifications:

A suitable exhaust fan by Alphair would have the following specifications:

- Model 10150 AMF 5000 (2.6 m diameter)
- 20° blade angle
- 880 rpm
- 500 hp motor.

A suitable underground auxiliary fan for the stopes would be an Alphair with the following specifications:

- Model 4800 VAX 2100 (1.2 m diameter)
- 30° blade
- 1,780 rpm
- 110 hp motor.

18.1.11 *Mine Dewatering*

The main sources of water inflow to the underground mine will be groundwater, water from drilling operations, and backfill.

Table 18.30 Underground Mine Water Balance

Source of Water	Description	Inflow (m^3/d)	Outflow (m^3/d)
Ground Water	Ground Water	18,007	
	Peak flow (with safety factor 1.3)	23,409	
Service Water	Drilling	416	
	Dust Control	30	
	Washing	10	
	Leaks in Pipelines (10%)	46	
Backfill	Decant Water	591	
	Flush Water	16	
Diesel Exhaust	Trackless Fleet	10	
Ore and Waste Rock	3% Moisture Content		90
Slimes	Removal from Mine		18
Ventilation	Evaporation		30
Pumping	Main Mine Pumps		24,390
Total		**24,528**	**24,528**

The main sump will typically be a two-bay design to allow suspended solids to settle out of the water before pumping. The first main sump will be constructed during the pre-production stage on the -95 m level. It is assumed that two additional main sumps will be constructed at the bottom of the mine during the production stage – one on the south side and one on the north side of the orebody. Each sump will be equipped with four 75 kW (100 hp) high-pressure pumps (2 working, 2 on stand-by).

Each pump will be connected to a 254 mm (10″) diameter steel dewatering pipe, which will be installed in the main access decline to pump water from the main sump to the final tailing pump box on surface.

The small sumps will be constructed locally at the lowest points of each mining level. Water will be pumped from those sumps and from the working faces to a permanent sump by 13 kW (18 hp) submersible slurry pumps through a 100 mm (4″) diameter pipe installed on each level.

Coarse material settled out in the main sumps will be removed periodically by LHD and disposed.

18.1.12 WATER SUPPLY

Industrial-quality water will be distributed in 4″ and 2″ diameter pipelines throughout the underground workings for drilling equipment, dust suppression, cleanup, maintenance, and fire fighting. Flexible hoses will be used to connect water pipelines to drilling equipment at working faces. The major water consumption will be by jumbo drilling equipment with pump capacities of 100 L/min as per the technical specification. The rockbolters will require 30 L/min of water.

An average mine service water consumption was estimated to be approximately 501 m^3/d including water for drilling, dust control, washing, and assuming 10% leaks in pipelines.

Table 18.18 Underground Mine Service Water Requirements

	No. of Units	Water Usage per Unit (L/min)	Drilling Utilization (%)	Water Usage (m^3/d)
Jumbo Drill	6	100	38%	324
Rockbolter	2	30	38%	32
Diamond Drill	1	100	38%	54
Jackleg/Stoper	5	3	25%	5
Dust Control				30
Washing				10
Leaks in Pipelines (10%)				46
Total				**501**

A water tank located on surface near the main access decline will provide fresh and fire water.

18.1.13 POWER

The major electrical power consumption in the mine will be from the following:

- main and auxiliary ventilation fans
- drilling equipment
- mine dewatering pumps.

A 10 kV feeder cable to the mining area will provide power to operating equipment and local infrastructure loads. In the mining area, power will be reduced in voltage to 400 volts to suit low voltage three phase loads and 127 volts for lighting and associated miscellaneous power loads.

Table 18.31 Annual Power Usage

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Power Usage (MWh)	4,192	27,552	36,861	40,091	40,091	42,719	39,930	39,930	39,329	39,329	37,547
Contractor	2,789	2,789	2,789	2,789	2,789	2,789					
Jumbo	1,203	2,406	3,007	3,609	3,609	3,609	3,609	3,609	3,007	3,007	3,007
Bolter	200	401	401	401	401	401	401	401	401	401	401
Compressor	-	1,604	1,604	1,604	1,604	1,604	1,604	1,604	1,604	1,604	1,604
Batch Plant	-	1,782	1,782	1,782	1,782	1,782	1,782	1,782	1,782	1,782	-
Ventilation											
Surf 375 kW Fan	-	3,329	4,993	4,993	4,993	4,993	4,993	4,993	4,993	4,993	4,993
Auxiliary 75 kW Fan	-	6,570	10,512	10,512	10,512	10,512	10,512	0,512	10,512	10,512	10,512
Auxiliary 45 kW Fan	-	3,942	6,307	6,307	6,307	6,307	6,307	6,307	6,307	6,307	6,307
Dewatering Pumps											
Dirty Water Pumps	-	2,628	2,628	5,256	5,256	7,884	7,884	7,884	7,884	7,884	7,884
Face Pump 18 hp	-	1,226	1,962	1,962	1,962	1,962	1,962	1,962	1,962	1,962	1,962
Lighting		876	876	876	876	876	876	876	876	876	876

18.1.14 Underground Communication System

A leaky feeder communication system will be installed throughout the mine. The system will interface with the surface communication system. It will be also used for centralized blasting.

Telephones will be located at key infrastructure locations such as the underground electrical sub-stations, refuge areas, lunch rooms, and pumping stations.

Key personnel (such as mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (such as loader, truck, and utility vehicle operators) will be supplied with an underground radio.

18.1.15 Compressed Air

The mobile drilling equipment – such as jumbos, rockbolters, and scissor lifts with ANFO loaders – will be equipped with their own compressors. No reticulated compressed air system will be required.

Six portable compressors will be required to satisfy compressed air consumption for miscellaneous underground operations, such as jackleg and stoper drilling.

Refuge stations will be provided to satisfy safety requirements.

18.1.16 Explosives and Storage Handling

Explosives will be stored on surface in permanent magazines. Detonation supplies (NONEL and electrical caps, detonating cords, etc.) will be stored in a separate magazine on surface.

Underground powder and cap magazines will be prepared on the -65 m level. Explosives will be transported from the surface magazines to the underground magazines in mine supply trucks.

ANFO will be used as the primary explosive for mine development and production. Packaged emulsion will be used as a primer and for loading lifter holes in the development headings. Smooth blasting techniques are recommended in development headings, with the use of ‘trim’ powder for loading the perimeter holes.

During the pre-production period, it is assumed blasting in the main ramp development heading will be done anytime during the shift when the face is loaded and ready to blast. All personnel underground will be evacuated to surface during blasting. Once multiple faces are available during preproduction and during production, blasting will take place at the end of each shift. After the pre-production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes at the end of the shift.

In order to minimize disturbance to local residents, blasting will be limited to twice a day – in the morning (e.g. 08:00 h) and in the evening at the end of day shift (e.g. 16:00 h). No blasting will be done at the end of the afternoon shift (e.g. 24:00 h).

All blasting in the mine will be development-style blasting. No large scale blasts will be undertaken. Typically 20 to 30 faces per day will be blasted. The duration of the blasts should take no longer than 3 to 4 minutes to complete each time.

Table 18.32 provides an estimate of allowable charge weight that may be blasted per delay versus the distance away from houses, etc. using a scaled distance relationship (distance divided by the square root of the charge) based on typical North American regulations.

Table 18.32 Allowable Charge Weights and Blasting Distances

Charge Weight (kg)	Distance (m)
0	10
1	20
2	30
3	40
5	50
7	60
10	70
13	80
16	90
16	100
20	110
23	120
27	130
32	140
36	150
41	160
47	170
52	180

Typically, no more than 5 kg of explosives will be detonated in each hole. Up to four or five holes may be on a similar delay; therefore, the total simultaneous detonation could be up to 20 or 25 kg at one time. When housing is closer than about 100 m, more delays providing a lower charge per delay will be required to maintain acceptable blast vibrations.

The closest existing housing in the village near the eastern end of the deposit is estimated to be about 100 m vertically from mining operations.

The closest mining to the surface at the university border is about 90 m away.

It is not anticipated that any damage will result from blasting. It is likely that local residents will notice the blasting and complaints could be expected. Minco should conduct damage surveys of all buildings adjacent to the mine before blasting commences to provide a "before and after" record for dealing with damage claims.

18.1.17 *Fuel Storage and Distribution*

The underground mobile equipment has an estimated average consumption rate of approximately 8,556 L/d during the production period.

Haulage trucks and all auxiliary vehicles will be fuelled at fuel stations on surface. The fuel/lube cassette will be used for the fuelling/lubing of LHDs and face equipment.

The annual fuel consumption details are shown in the Table 18.33.

Table 18.33 Annual Fuel Consumption (in Litres)

	Year										
	-1	1	2	3	4	5	6	7	8	9	10
LHD	178,200	534,600	534,600	534,600	534,600	534,600	534,600	445,500	445,500	445,500	445,500
Jumbo	28,512	57,024	71,280	85,536	85,536	5,536	85,536	85,536	71,280	71,280	71,280
Bolter	14,256	28,512	28,512	28,512	28,512	28,512	28,512	28,512	28,512	28,512	28,512
Trucks	252,450	883,575	1,009,800	1,136,025	1,136,025	1,262,250	1,262,250	1,009,800	883,575	883,575	883,575
Grader	-	173,448	173,448	173,448	173,448	173,448	173,448	173,448	173,448	173,448	173,448
Explosive Truck	-	78,883	78,883	78,883	78,883	78,883	78,883	78,883	78,883	78,883	78,883
Mechanics Truck	-	70,995	70,995	70,995	70,995	70,995	70,995	70,995	70,995	70,995	70,995
Fuel/Lube Truck	-	39,442	39,442	39,442	39,442	39,442	39,442	39,442	39,442	39,442	39,442
Supervisor/ Engineering Vehicle	-	156,816	156,816	156,816	156,816	156,816	156,816	156,816	156,816	156,816	156,816
Electrician Vehicle - Scissor Lift	-	78,883	78,883	78,883	78,883	78,883	78,883	78,883	78,883	78,883	78,883
Scissor Lift	-	236,650	236,650	236,650	236,650	236,650	236,650	236,650	236,650	236,650	236,650
Survey Vehicle	-	13,068	13,068	13,068	13,068	13,068	13,068	13,068	13,068	13,068	13,068
Mine Engineering Vehicle	-	26,136	26,136	26,136	26,136	26,136	26,136	26,136	26,136	26,136	26,136
Shotcrete Robo	-	28,512	28,512	28,512	28,512	28,512	28,512	28,512	28,512	28,512	
Transmixer	-	315,533	315,533	315,533	315,533	315,533	315,533	315,533	315,533	315,533	
Total Diesel Usage	**473,418**	**2,722,076**	**2,862,557**	**3,003,038**	**3,003,038**	**3,129,263**	**3,129,263**	**2,787,713**	**2,647,232**	**2,647,232**	**2,303,187**

18.1.18 Transportation of Personnel and Supplies

Supplies and personnel will access the underground via the main access decline.

The personnel carriers will be used to shuttle workers from surface to the underground workings and back during shift changes. Supervisors, engineers, geologists, and surveyors will use diesel-powered trucks as transportation underground. Mechanics and electricians will use the mechanics' truck and maintenance service vehicles.

A boom deck with a 10-t crane will be used to move supplies, drill parts, and other consumables from surface to active underground workings.

18.1.19 Underground Construction and Mine Maintenance

A mine service crew will perform the following:

- mine maintenance and construction work
- ground support control and scaling
- road checking and maintenance
- construction of ventilation doors, bulkheads, and concrete work
- mine dewatering
- safety work.

Two underground graders and a scissor lift will be utilized to maintain the main declines and active work areas.

18.1.20 Equipment Maintenance

Mobile underground equipment will be maintained in a mechanical shop located on the surface outside of the main ramp access portal. The surface maintenance shop will contain the following:

- three maintenance bays (one heavy repair crane bay and two service bays) equipped with overhead and jib cranes
- tire repair bay
- welding bay
- electrical bay
- lube and wash bay
- warehouse
- offices.

A small maintenance shop with overhead crane will also be constructed underground to provide maintenance for drilling equipment.

A maintenance foreman will provide a daily maintenance work schedule, ensure the availability of spare parts and supplies, and provide management and supervision to maintenance crews. He will also provide training for the maintenance workforce.

A maintenance planner will schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle, mine efficiency, and personnel utilization. A computerized maintenance system will facilitate planning.

The equipment operators will provide equipment inspection at the beginning of the shift and perform small maintenance and repairs as required.

A mechanics truck will be used to perform emergency repairs underground. Major rebuild work will be conducted off site.

18.1.21 Health and Safety

Fire Prevention

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the electrical installations, pump stations, fuelling stations, mechanical shop, and wherever a fire hazard exists.

A suitable number of fire extinguishers will be provided and maintained at each stationary diesel motor, transformer substation, and any splitter panel. Every vehicle will carry at least one fire extinguisher of adequate size and proper type.

Underground mobile vehicles will be equipped with automatic fire suppression systems.

A mine-wide stench gas warning system will be installed at the main intake mine entries to alert underground workers in the event of an emergency.

Mine Rescue

A mine rescue Emergency Response Plan will be developed, kept up to date, and followed in an emergency. Two fully trained and equipped mine rescue teams will be established.

A mine rescue room will be provided adjacent to the mine access portal. A trailer with mine rescue equipment and a foam generator will be located on site. The mine rescue teams will be trained for surface and underground emergencies.

Refuge Station

Portable refuge stations will be provided in the main underground work areas. The refuge stations will be equipped with oxygen systems, potable water, emergency lighting, a fixed telephone line, and first aid equipment. The stations will be capable of being sealed to prevent the entry of gases. A plan of the underground workings showing all exits and the ventilation plan will be provided.

The refuge station locations will move as the working areas advance, eliminating the need to build new refuge stations. Permanent refuge stations may need to be constructed, depending on local mining regulations.

Emergency Egress

The main access decline will provide primary access. The secondary egress will be the fresh air ventilation raise and the two exhaust raises, which will have a dedicated manway for personnel to travel between levels, providing the exit from sub-levels to surface in case of emergency. The exhaust raises will be fitted with fans that may be reversed in case of emergency.

Dust Control

Broken ore will be wet down after blasting and during production using water sprays.

18.1.22 *Development Schedule*

Mine development is divided into two periods:

- pre-production development (prior to mine production)
- ongoing development (during production).

Pre-production development includes waste and ore development that is required to access and prepare the stopes. The pre-production development period runs from the start of the project to when the first ore is fed to the process plant. Pre-production development will be scheduled to:

- provide access for trackless equipment
- provide ventilation and emergency egress
- establish ore and waste handling systems
- install mining services (backfill distribution, power distribution, communications, explosives storage, fuel storage and distribution, water supply, mine dewatering)
- provide sufficient level development in advance of start-up to develop sufficient ore reserves to support the mine production rate.

Figure 18.26 Mine Pre-production Development – Plan View

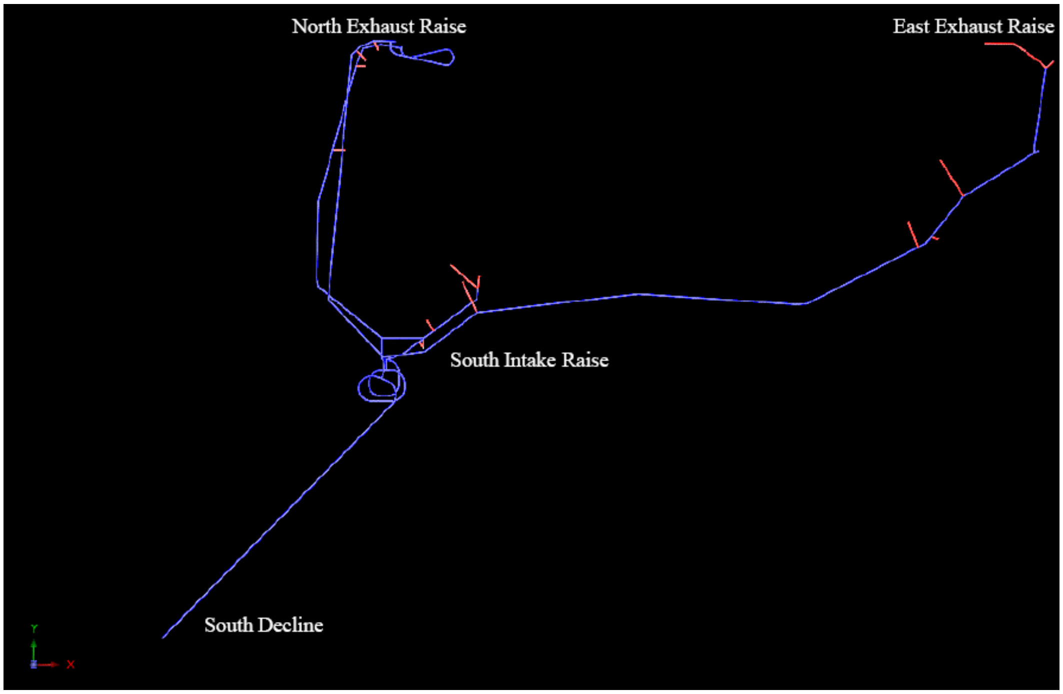
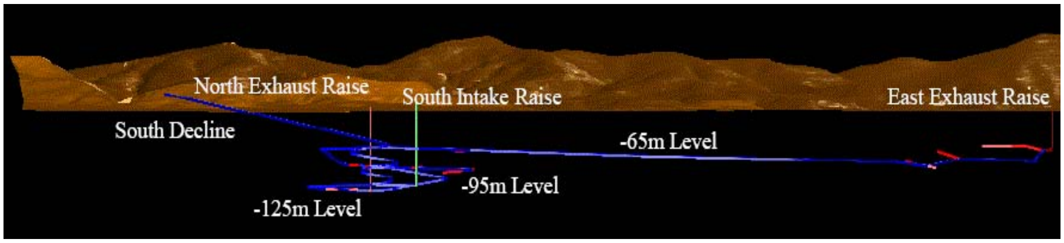


Figure 18.27 Mine Pre-production Development – Section View



It was assumed that all underground pre-production development will be done by contractor with use of a contractor's equipment, personnel, and supervision. The underground mining contractor will mobilize mining equipment and crews to site once construction of the minimum site infrastructure (required to start development) is completed. Approximately three months is required for the mining contractor to establish the required services and prepare mining equipment.

Development cycle times were calculated using first principles estimates of task performance for development heading (Table 18.34). Considering estimated development advance rates and Canadian contractor practices, it was assumed that a jumbo crew advance rate at a 4.5 m wide by 4.5 m high development can be approximately 130 m per month.

Table 18.34 Summary Cycle Times

Description	Unit	Quantity
Design Criteria		
Shifts per Day	shift/d	3
Useful Hours per Shift	h/shift	6
Minutes per Hour	min/h	50
SG	t/m^3	2.48
Width	m	4.0
Height	m	4.5
Gradient	%	15
Overbreak	%	10
Expansion Factor		1.5
Cycle Times		
Drilling	h	2.3
Blasting	h	1.1
Mucking	h	1.9
Support	h	4.8
Services	h	2.4
Single Heading		
Critical Path Cycle Time	h	10.1
Advance per Shift	m/shift	1.7
Advance per Day	m/d	5.2
Advance per Month	m/mo	156
Multiple Headings		
Critical Path Cycle Time	h	8.2
Advance per Shift	m/shift	2.1
Advance per Day	m/d	6.4
Advance per Month	m/mo	191

The vertical development of ventilation raises will be done by raiseboring crew. It was assumed that a raiseboring crew can drill a pilot hole and ream it to the 4.0 m diameter at an approximate advance rate of 90 m per month.

The contractor will develop underground access to the ore reserves sufficient for at least 6 months of production during the pre-production development period.

Underground infrastructure development, such as dewatering sumps, maintenance shop, and explosives storage, will be completed prior to production.

It is estimated that pre-production development will be completed in two years utilizing three development crews.

During the first three months, a contractor will mobilize to site and develop the south decline portal, which will be the main access to the underground mine.

The development of the main access decline by the first crew was assumed to advance at 50 m in the first month, 90 m in the second month, and then at an average advance rate of 130 m per month. The contractor will develop the decline to the -65 m level when a second crew will start development of the -65 m level haulage to the east exhaust raise in Month 10. The first crew will continue the main ramp all the way down to -125 m level, driving accesses to the south fresh air raise on level -95 m and -125 m to provide flow-through ventilation during development.

The critical path in pre-production development will be to complete the connection of the main access decline to the east exhaust ventilation raise via the -65 m level haulage drive along the southern side of the orebody by the second crew. After advancing about 80 m from the main decline, it will be connected to south fresh air ventilation raise to provide flow-through ventilation. The second jumbo crew will also develop the cross-cuts to the mining blocks on -65 m level. It is the longest single heading development so it will need to be developed as the first priority for advance.

A third jumbo crew will start developing of the -65 m level west haulage drive in the 15th month. After connecting it to the north exhaust raise on the -65 m level, it will develop downwards 120 m of the north ramp. Jumbo crew #1 will develop the north ramp upwards from -125 m level. When development of the north ramp is completed in the 21st month, the third jumbo crew will have 3 more months prior to production to complete infrastructure development, such as dewatering sumps, underground explosives storage, mechanical shop, etc.

The -125 m level west haulage drive will be developed by the first crew from the main decline ramp connecting it to the north exhaust raise, as well as a portion of north ramp upwards to connect it to the -65 m level west haulage drive developed by the third jumbo crew.

During development, the south ventilation raise will be used for exhaust and will later switch to intake after connections to the north and east ventilation exhausts are established.

The pre-production development schedule is shown in the Table 18.35.

A raiseboring crew will mobilize to site in the 10th month of the pre-production period. It will develop the collar of the south fresh air raise and start drilling a 145 m-long pilot hole down to the -65 m level. When underground access to the bottom of the raise is developed on -65 m level, the raise will be reamed from the bottom up to the final size of 4.0 m diameter. Then the raiseborer will be relocated underground to the -65 m level to be able to extend the raise down to levels below -95 m and -125 m.

The raiseborer will then be relocated to the surface and the collar of the north exhaust raise will be developed in the 20th month. The west side development of the -65 m and -125 m levels is scheduled to be done simultaneously by jumbo crew #1 and jumbo crew #3; access to the north raise on both levels will be developed at approximately the same time. The raise will be developed down to -125 m level, thus avoiding the requirement for raiseborer relocation underground for the purpose of developing the raise between -65 m and -125 m levels.

A 145 m-long, 349 mm diameter pilot hole will be drilled down to -125 m level. When the underground access to the bottom of the raise is developed on -125 m level, the raise will be reamed from the bottom up to the final size of 4.0 m diameter. The raise will be broken into from the -65 m level by access drift after it has been reamed.

The 109 m-long, 4.0 m diameter east exhaust air raise will then be developed from the surface to -65 m level using the same approach. The exhaust fans will be installed at the collars of exhaust raises to provide flow-through ventilation for the underground mine.

The south fresh air raise and the exhaust raises will be equipped with a manway to provide alternative emergency egress from the mine.

Ore development is not included in the development schedule as it will be part of ore production.

Table 18.35 Pre-production Development Schedule

Month	1Q	2Q	3Q	4Q	5Q	6Q	7Q	8Q
Jumbo Crew 1								
Portal Construction	x							
Portal (South Decline) to 65 m level		270	390	24				
South Decline to 95 m level				237				
Drive off South Decline to 95 m level				21				
95 m level South Haulage Drive				75				158
95 m level access to South FAR				14				
204 S								26
205 S								96
South Decline to 125 m level				19	190			
Drive off South Decline to 125 m Lvl					44			
125 m level South Haulage Drive					81			
125 m level access to South FAR					19			
125 m level West Haulage Drive					56	390	217	
204 N							25	
205 N							20	
125 m level access to North RAR							10	
125 m Lvl North-West Exhaust Drive							24	
Drive off North Ramp to 125 m level							25	
North Ramp to 125 m level							69	89

table continues…

Month	1Q	2Q	3Q	4Q	5Q	6Q	7Q	8Q
Jumbo Crew 2								
Drive off South Decline to 65 m level				42				
65 m level South Haulage Drive				263	390	390	350	
65 m level Access to South FAR				8				
107-108 S				67				
65 m level East Haulage Drive							40	128
65 m level Access to East RAR								19
214-215-315 N								97
214 S								86
213 S								54
Jumbo Crew 3								
65 m level West Haulage Drive					117	390	181	
107-108 N							28	
65 m level access to North RAR							19	
65 m Lvl North-West Exhaust Drive							42	
North Ramp to 125 m level							120	
Raisebore Crew								
South FAR to 65 m level				145				
South FAR to 95 m level						33		
South FAR to 125 m level						30		
North RAR to 65 m level							70	34
North RAR to 125 m level								41
East RAR to 65 m level								109
Subtotals								
Jumbo Crew 1	0	270	390	390	390	390	390	369
Jumbo Crew 2	0	0	0	380	390	390	390	384
Jumbo Crew 3	0	0	0	0	117	390	390	0
Raisebore Crew	0	0	0	1450	0	63	70	184
Horizontal Development	0	270	390	770	897	1,170	1,170	753
Vertical Development	0	0	0	1450	0	63	0	184
Total	**0**	**270**	**390**	**915**	**897**	**1,233**	**1,240**	**937**
Cumulative Metres	**0**	**270**	**660**	**1,575**	**2,472**	**3,705**	**4,945**	**5,882**

Ongoing sustaining development will continue to be performed by contractor during the production stage. One jumbo crew will satisfy the waste development requirements to advance access headings and prepare the mining blocks. The contractor will demobilize from site in Year 9 when all main access development will be complete. Additional raise development will be required in Years 1 to 3. In Year 6, a raise will be developed 61 m from the west fresh air raise to provide ventilation and emergency egress for mining block #201.

Table 18.36 Mine Development Schedule

	Unit	Pre-production	Production Year 1	2	3	4	5	6	7	8	9	Total
Annual Metres (Horizontal)	m	5,420	1,497	1,437	1,132	950	1,040	765	216	364	360	13,181
Annual Metres (Vertical)	m	462	45	214	37	0	0	61	0	0	0	819
Total Development	m	5,882	1,542	1,651	1,169	950	1,040	826	216	364	360	14,000
Waste Tonnage	t	310,347	83,486	83,720	63,183	52,480	57,452	43,329	11,932	20,108	19,887	745,924
Waste Volume	m^3	187,710	50,496	50,637	38,215	31,742	34,749	26,207	7,217	12,162	12,029	451,164

18.1.23 PRODUCTION SCHEDULE

The criteria used for scheduling of mine production at the Fuwan Project were as follows:

- target mining blocks with higher grade ore in the early stages of mine life to improve project economics
- production sequence of the mining blocks will be from the top down so cut-and-fill mining blocks and secondary stopes of drift-and-fill mining blocks may only be mined after the blocks above are completely mined out
- primary stopes of drift-and-fill mining blocks may be mined at any time independent of the presence of blocks above
- an annual ore production rate of 990,000 t was scheduled, including ore from development and stopes
- the mine will operate three 8-h shifts per day, 330 d/a
- provide enough production faces to support a daily mine production rate of 3,000 t/d.

The stope cycle times and productivities were estimated from the first principles for each type of mining method and stope size.

It was estimated that an average production from the single heading at cut-and-fill mining will be approximately 75,000 t/a. Considering that three stopes could be in production at the same time, the productivity of one cut-and-fill mining block can be 225,000 t/a. However, supporting such a production rate might require splitting the mining block onto two sub-panels with a pillar to separate production and backfilling operation within the block.

The drift-and-fill mining method will allow as many stopes in production within a mining block as is required to support the production rate.

The number of mining blocks in production will vary from 8 to 10 in most production years, except for 7 mining blocks in production in Year 1, 12 blocks in Year 9, and 5 blocks in the last year of production. On average, there will be 5 stopes in production for drift-and-fill mining, and 4 stopes in production for cut-and-fill. The only room-and-pillar block will be mined in Year 9.

The cut-and-fill mining method will require development of the twin in-ore ramps inside the mining block prior to production to provide through-ventilation and emergency egress. Ore produced from in-ore ramp development is classified in the production schedule as development ore. Ore produced from production stopes is classified as production ore.

As mined out primary stopes will be backfilled, an appropriate period of time for filling and curing was accounted for in the schedule for secondary stope production. The stope production schedule by year is provided in Appendix H.

Figure 18.28 through to Figure 18.33 show the mine production development for Years 1, 6, and 10 in both section and plan view.

Detailed mining drawings are provided in Appendix O.

Figure 18.28 Mine Production Development – Year 1 (Plan View)

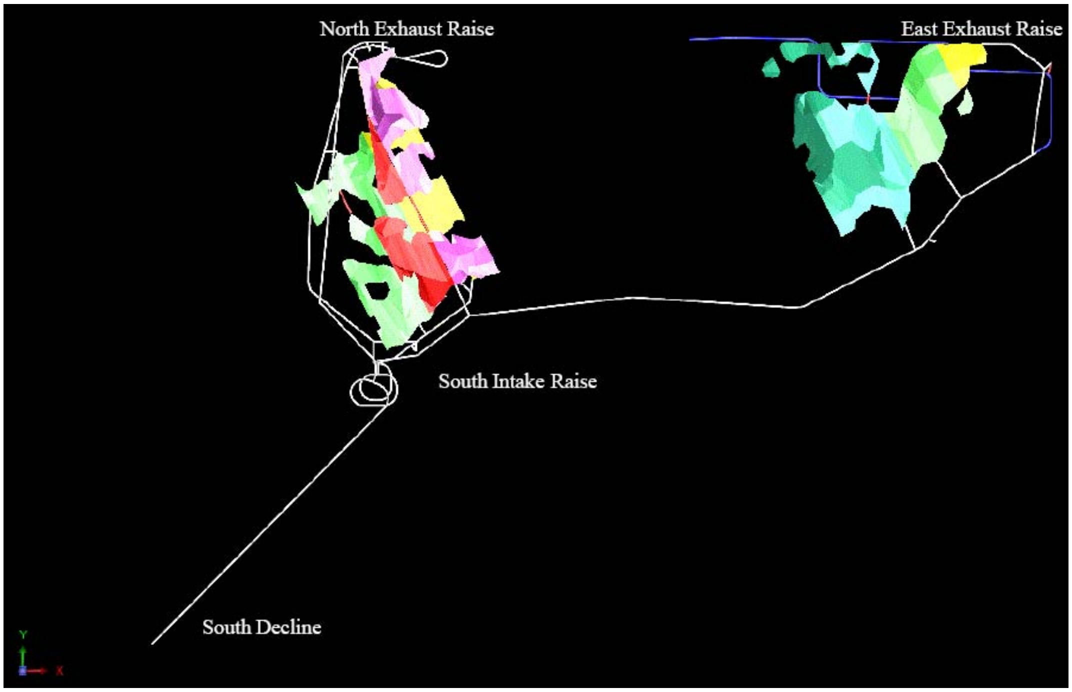


Figure 18.29 Mine Production Development – Year 1 (Section View)

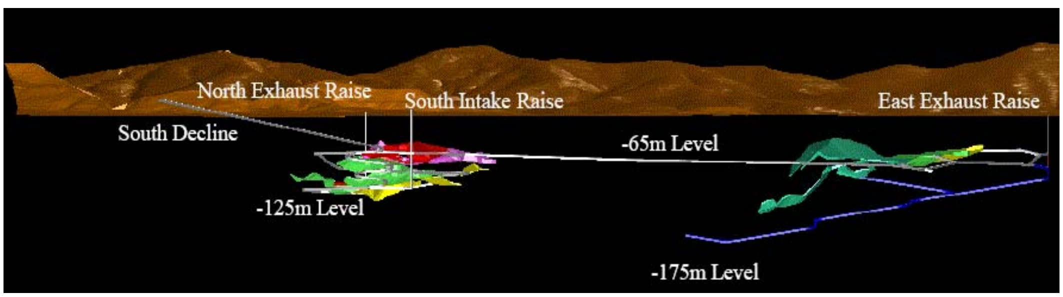


Figure 18.30 Mine Production Development – Year 6 (Plan View)

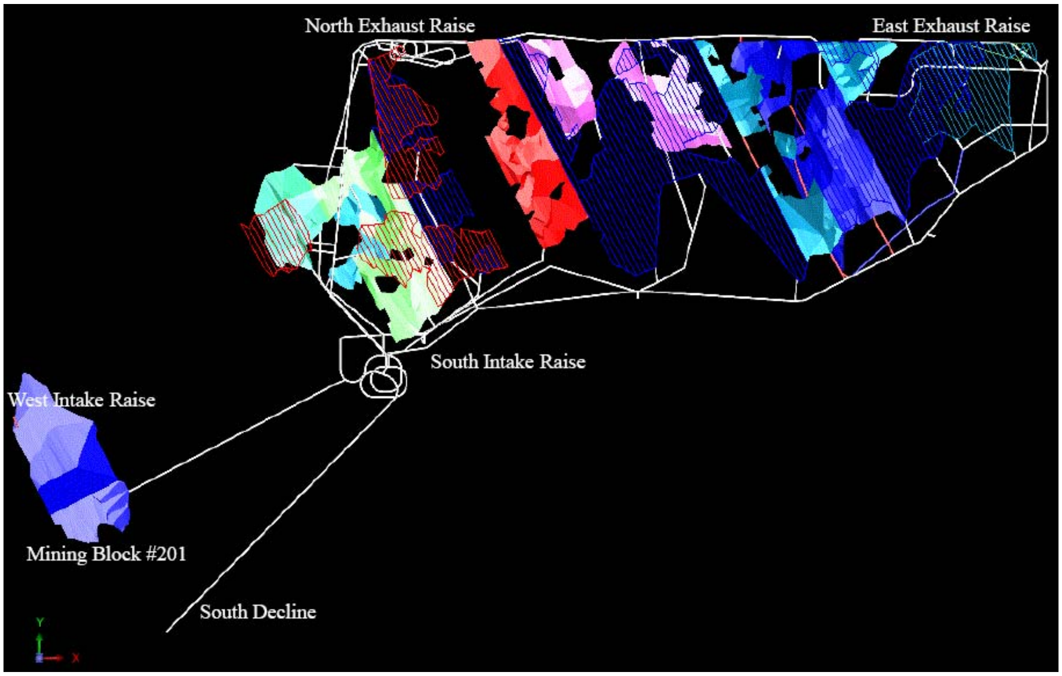


Figure 18.31 Mine Production Development – Year 6 (Section View)

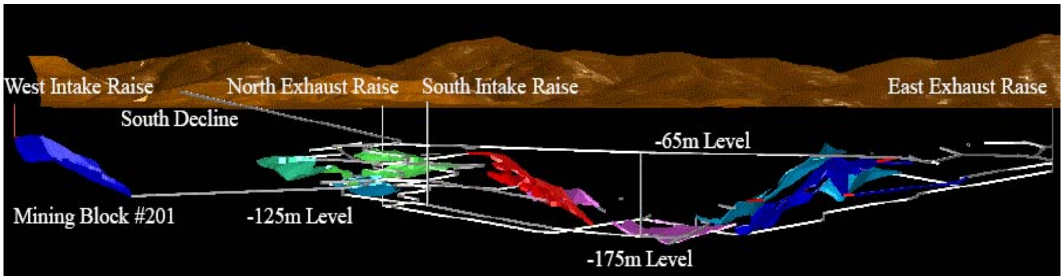


Figure 18.32 Mine Production Development – Year 10 (Plan View)

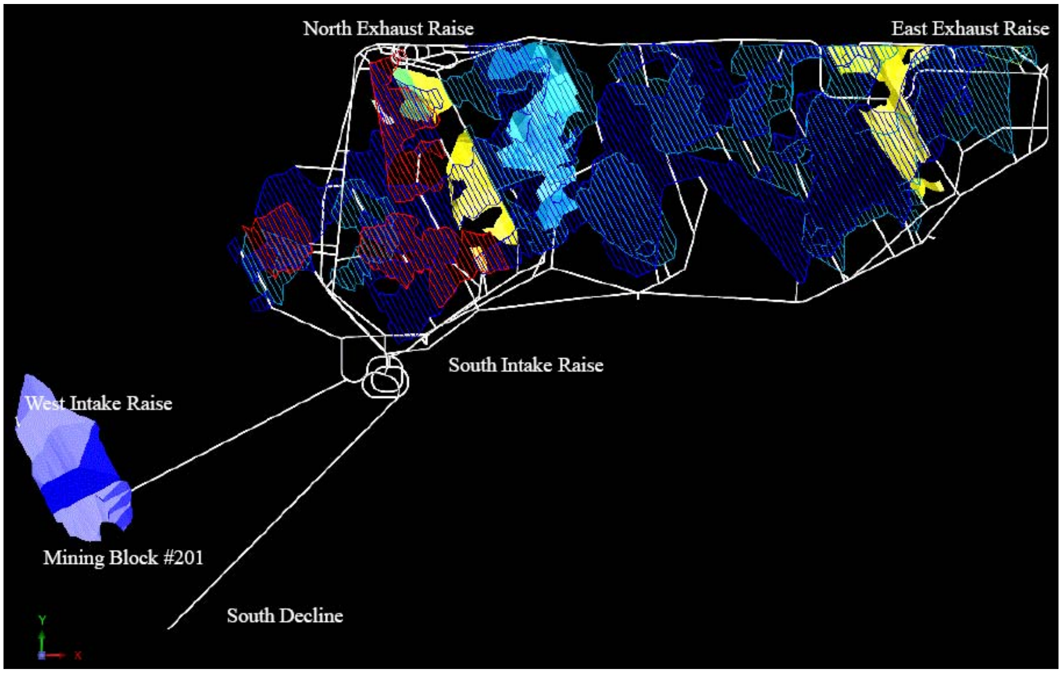


Figure 18.33 Mine Production Development – Year 10 (Section View)

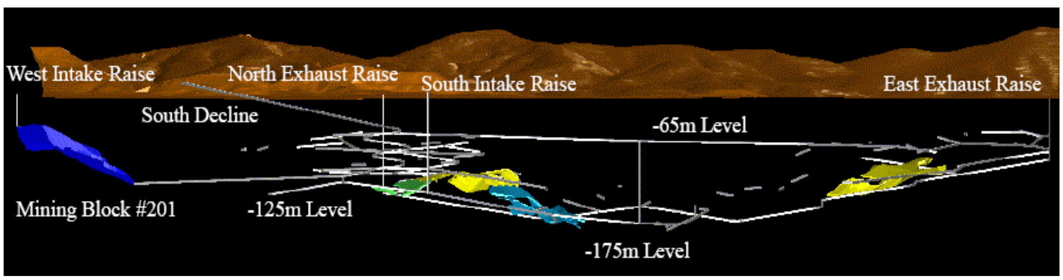


Table 18.37 Production Schedule

	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Days per Year	d/a	330	330	330	330	330	330	330	330	330	70	
Mill Feed	t	990,000	990,000	990,000	990,000	990,000	990,000	990,000	990,000	990,000	207,981	9,117,981
Grade												
Ag	g/t	214	217	217	205	183	182	177	167	148	137	189
Au	g/t	0.171	0.169	0.158	0.157	0.150	0.157	0.151	0.138	0.079	0.076	0.146
Pb	%	0.194	0.194	0.146	0.148	0.120	0.189	0.235	0.242	0.263	0.372	0.196
Zn	%	0.584	0.614	0.506	0.541	0.483	0.487	0.615	0.595	0.637	0.709	0.566
Development Ore	t	24,304	77,773	73,606	45,830	55,552	63,885	55,552	20,832	43,400		460,734
Production Ore	t	965,696	912,227	916,394	944,170	934,448	926,115	934,448	969,168	946,600	207,981	8,657,246
By Mining Method												
Drift-and-Fill Stopes	t	865,000	713,990	460,035	438,192	613,492	368,639	389,114	537,310	423,406	140,805	4,949,982
Cut-and-Fill Stopes	t	125,000	276,010	529,965	551,808	376,508	621,361	600,886	452,690	489,386	67,176	4,090,791
Room-and-Pillar Stopes	t									77,207		77,207

18.1.24 Conclusions and Recommendations

Infill drilling will be recommended during the next phase of the project to better define the orebody for detailed design. The infill drilling program should be logged geotechnically to improve the geotechnical model.

18.2 Infrastructure and Ancillary Facilities

18.2.1 Site Access

The project site is close to the Fuwan town, which has well developed paved village-level road network. The town is accessible by paved public highways to Guangzhou (75 km), Foshan (50 km), Jiangmen (60 km) and Zhaoqing (55 km). Also the site is approximately 3 km away from a paved highway from Guangzhou to Mingcheng, which is partially constructed.

The haulage distance between the mine site and the Shanshui railway station, which connects the main stations, Guangzhou station and Zhanjiang station, is approximately 26 km.

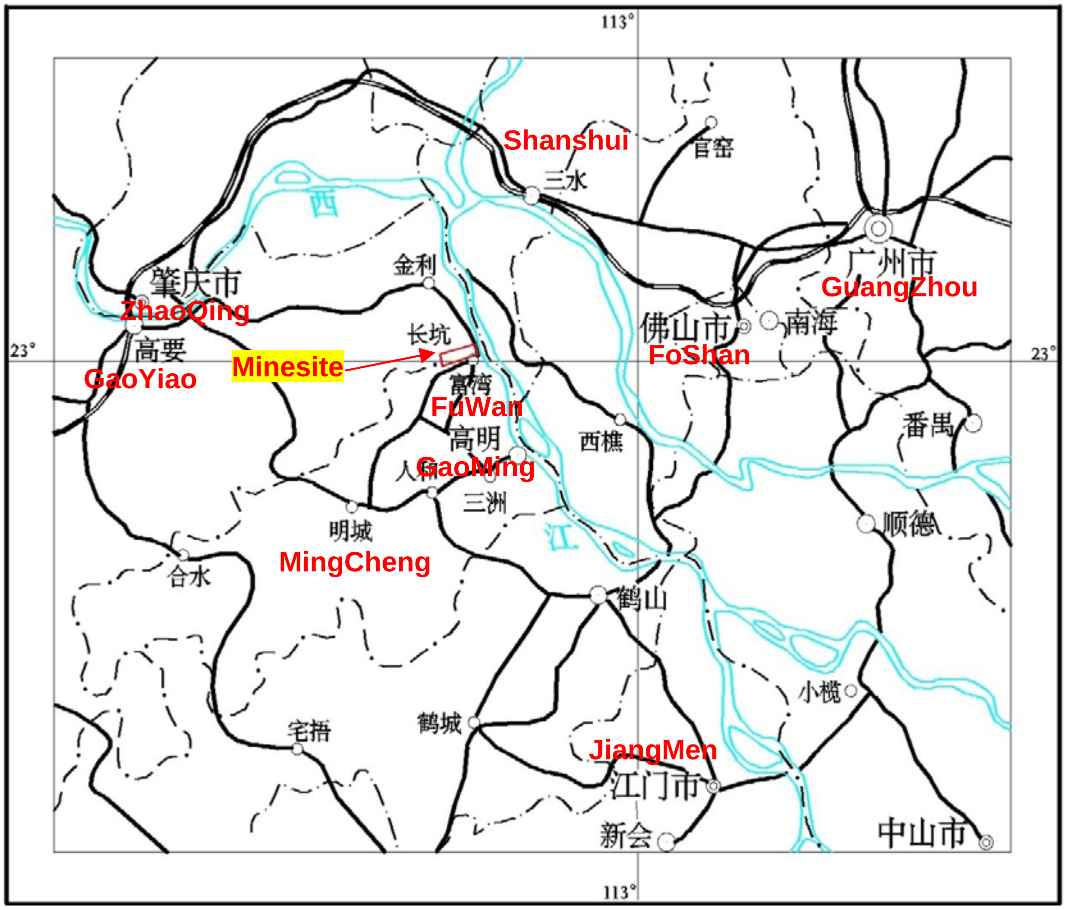
The deposit is adjacent to the Xijiang river. There is a cargo port closely located to the mine site. The Xijiang river is one of the branches of the Zhujiang river, which connects with many major cities such as Zhaoqing, Jiangmen, Zhuhai, Hongkong and Macao. The Zhujiang river is accessible to international waterway in the South China Sea.

The local transportation map is shown in Figure 18.34.

Electrical power, water, telephone service, and supplies are available in the Fuwan town.

The proposed minesite is large enough to accommodate proposed processing facilities, surface service facilities, waste rock storage areas, as well as approximately 8.3-year tailing surface storage pond.

Figure 18.34 Local Transportation Map



18.2.2 SITE LAYOUT

The site general layouts are presented in Figure 18.35 and Figure 18.36, and in Drawings A0-10-001 and A0-10-002 in Appendix E. The layouts show approximately mineralization deposit location and project related facilities including process plant, administration buildings, workshop, waste rock surface storage location, tailings surface storage facility, explosive magazine, power and water supply facilities, backfill station, waste water treatment facility and haulage road system. The facility is designed to process 3000 t/d silver, lead, zinc and gold mineralization.

Figure 18.35 General Site Layout

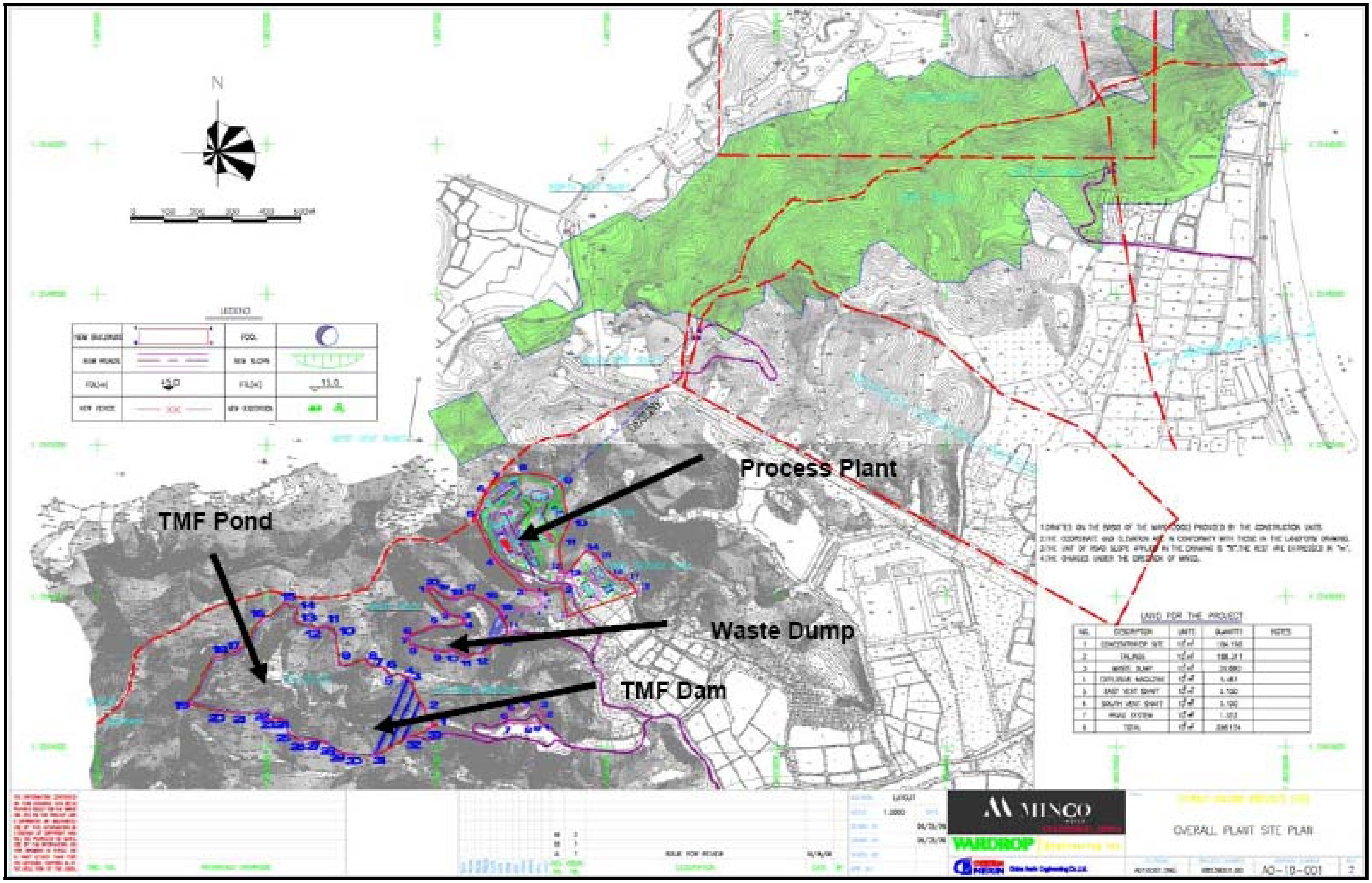
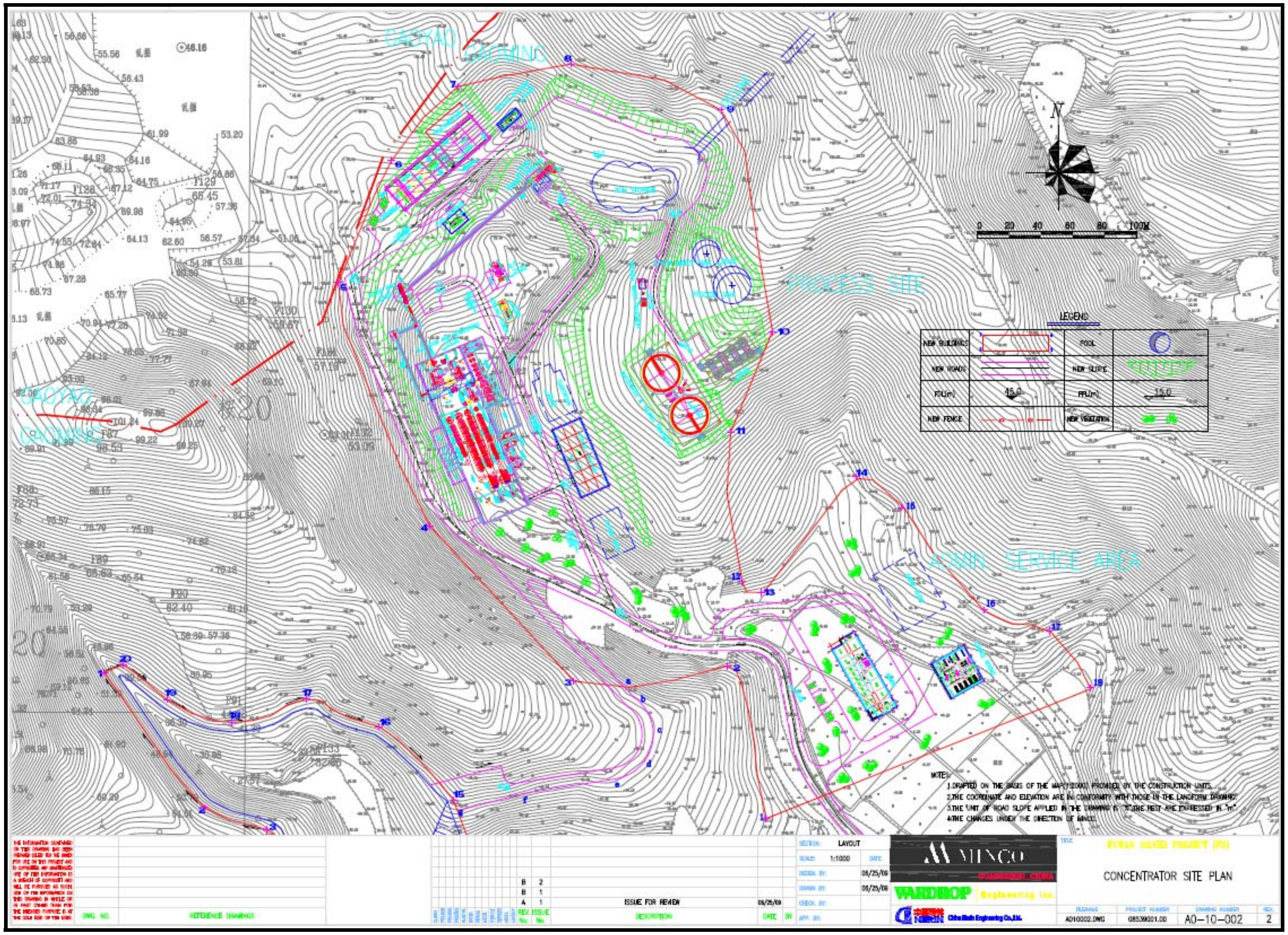


Figure 18.36 Plan Site Layout



The number of buildings and the distances between them are minimized to reduce construction costs such as minimizing earthwork and operation costs such as minimizing haulage distance of materials. The layout has minimized the land usage, especially the agriculture land usage. The site layout will comply the local regulations of fire protection, explosion safety, environmental protection, safety, sanitation and transportation.

The portal of the ramp is at the elevation of 47.0 m, primary crushing workshop 37 m, fine ore bin 28 m, grinding-flotation main plant 23 m, maintenance workshop 45 m, storage facility 24m, water treatment plant 44 m and water tanks ~ 65 m, and administrative office building 7.7 m. The layout has taken consideration of water and soil conservations.

The process facilities are located in a valley to minimize the influence of the mining activities to the surroundings. There are no residents within the range of 800 m of the process plant site.

The excavation earthwork is approximately 10.5×10^4 m^3 and the filling earthwork is 10.15×10^4 m^3, excessive earth and rocks will be used for tailings dam construction or stored at the waste rock storage area. The ratio of side slope is 1:0.75 ~ 1:1 for excavation of land levelling, and 1:1.5 for filling tentatively. Key slopes will be protected with mortar rubble masonry and vegetation.

Plant site runoff water will flow by gravity through open ditches along roads to the lowest collection pond. Divert ditches will be arranged in all necessary areas to divert rain water away from the mine site.

18.2.3 SURFACE FACILITIES

CIVIC DESIGN

All buildings of the project will be new ones. According to the Chinese construction codes, the Safety Level of all the buildings will be Grade II; the Waterproof Grade of roofing will be Grade III; the Fire resistance rating of the buildings will be Grade II.

The main Chinese national standards and regulations used for the architectural design are shown in Appendix I.

According to the Code for Seismic Design of Buildings (GB50011—2001), seismic fortification intensity of the area is six degree, except for the transformer, switching and explosive storage facilities which will be based on seven degree. The acceleration value of basic seism is 0.05 g. Earthquake group is belonged to Group One.

Load design criteria are detailed below:

- Wind load: basic wind pressure (w_o) 0.7KN/m^2.

- Operation loads: in the areas where the equipment will be installed, the loads will be based on installation, operation and maintenance requirements; roofing 0.7KN/m^2; stairs 3.5N/m^2.
- Constant load: (standard value): reinforced concrete bulk density 25KN/m^3; clay solid brick masonry bulk density 19KN/m^3; fired perforated bricks bulk density 15KN/m^3; equipment weight is based on actual value.

Steel gantry frame structures will be used for grinding and flotation plant, maintenance workshop, warehouse and water treatment plant. Reinforced concrete frame structure will be used for silo, administration building, change room, security gate, rescue station, substation, explosive magazine.

Mineral Process Facilities

The process plant is located at the central part of the valley in 'L' shape. The process facilities mainly consist of ore surcharge stockpile, primary crushing facility, conveyor gallery, fine ore surge bin, main process plant, concentrate thickeners. The main process plant will include primary grinding, silver-lead flotation, zinc flotation, regrinding, concentrate dewatering, concentrate loadout, reagent preparation and compressed air supply areas. The building will also accommodate metallurgical laboratory, assay laboratory, control rooms, offices and meeting room.

Metallurgical and Assay Laboratories

The metallurgical and assay laboratory will be located in the main process plant building. The assay laboratory will be equipped for most assaying required by process, effluent water treatment, mining grade control and environmental monitoring. The metallurgical laboratory will be quipped the bench-scale equipment required to optimize process and fully support the operations.

Maintenance Shop

Shops for maintaining all mine equipment will be consolidated in one building and located at north of the plant site. Maintenance shops will have three bays for servicing underground and plant equipment, and will include a wash bay, tire bay, and office facilities for maintenance management and planning.

All standard equipment will be provided, including truck hoists,5-t overhead crane, tire dynamic balance machine, tire disassembly and assembly machine, welding equipment, compressors, lubrication equipment, and other machining equipment. The general arrangement is shown in A0-07-008 in Appendix J.

WAREHOUSES

One of warehouses will be adjacent to the maintenance shop for easy access to repair and maintenance parts. The other warehouse will be located at east of the plant site. The general arrangement is shown in A0-07-008 in Appendix J and A0-10-002 in Appendix E

FUEL STORAGE

Taking into consideration that the project is adjacent to well established towns, diesel fuel storage capacity has been estimated based on 3 to 4 days of mining operations including mining haulage truck fleets and an allowance for additional mobile mine support and auxiliary equipment

One 30,000-L diesel tank will be provided at the fuel island at the south of the maintenance shop. Fuel dispensing facilities, including fast fill facilities for mining equipment, will be provided.

The general arrangement is shown in A0-07-007 Appendix J.

BACKFILL PLANT

The backfill plant, which will classify the coarse fraction from the flotation tailings to hydraulically fill underground stopes, will be located at north-east of the main process plant. The backfill plant will be equipped with hydrocyclones for classification, one hydrocyclone undersize surge tank with agitator, one 50-t cement storage silo, dust control systems and pumps for delivering the backfill materials to the underground and the hydrocyclone overflow to the tailing surface storage facility.

The general arrangement is shown in A0-10-015 Appendix E.

TAILING SURFACE MANAGEMENT FACILITY (TMF)

TMF is located in the valley on the southwest of waste rock storage facility, upstream of Nankeng reservoir. The height of the tailings dam is approximately 56 m, and total storage capacity of the tailing pond is approximately 3.2×10^6 m^3, which can provide approximately 8.3-years mine operations. Section 18.3 gives a more detailed description on the TMF.

DETONATOR AND EXPLOSIVE STORAGE

Explosive and detonator storage facility is located in a valley at the north of Nankeng reservoir, downstream of the TMF. The three sides of the valley are surrounded by hills and there are no villages and higher than class 4 roads around. The storage facility consists of two ammonium nitrate storehouses, a detonator magazine and ancillaries including fire water pool and security facilities such as guard room, barbed wires. The area within the fence is 1.47×104 m^2. The arrangement of the explosive magazine will conform to the Chinese Safety Code for Engineering Design of Civilian Explosive (GB50089-2007). The general arrangement is shown in A0-07-010 Appendix J.

Administration Buildings

Administration buildings will be located southeast of the plant site at the entrance of the mine site valley. The building will house mine administration, underground mine management and change rooms. Parking lots are provided as well. The general arrangement is shown in Drawings A0-07-002 to A0-07-005 in Appendix J

Rescue Room

A 69-m^2 mine rescue room will be provided adjacent to the mine access portal. A trailer with mine rescue equipment and a foam generator will be located on site. The mine rescue teams will be trained for surface and underground emergencies. A general

A mine rescue Emergency Response Plan will be developed, kept up to date, and followed in an emergency. Two fully trained and equipped mine rescue teams will be established.

Potable Water Supply

Potable water will be obtained from water wells located close to the plant site. The water will undergo chlorination and ultraviolet light treatment in a pre-assembled treatment unit. A general layout of the potable water supply system is shown in A0-10-011 in Appendix E.

Process Water Supply

Process water will be supplied from the reclaimed water from the surface tailing storage pond and the treated water from the water treatment plant. A 1000-m^3 process water tank will be located at the hill east of the process plant. The tank will be locate at the elevation where will provide sufficient pressure to meet production requirements. The process water tank will be equipped with level detection control which will connect with the reclaimed water pumps.

Fresh Water Supply

Fresh water will be distributed from the fresh and firewater storage tank, located on the hill east of the concentrator. The tank is equipped with an internal standpipe to ensure that the firewater is full all time. The storage tank will have a total storage capacity of 1000 m^3; 685 m^3 for firewater storage and the remaining 315 m^3 for fresh water storage. The fresh water tank will be equipped with level detection control which will connect with the fresh water supply pumps located at the water treatment plant. The treated water from the water treatment plant or the boreholes will supply for the fresh water.

Fire Protection

As described above, the fresh water will be located on the plant site at the elevation to provide sufficient pressure to meet firewater pressure requirements at all locations and a volume sufficient to meet a 2.0-hour firefighting service requirement. A plant site alarm will signal a low system pressure condition.

The firewater distribution system will consist of a dedicated fire water main and hydrant system. Hose cabinets will be installed in the concentrator and ancillary facilities, supplemented by portable fire extinguishers. The firefighting water will be reticulated to reach all surface buildings and the underground mine. Fire water sprinkler systems will be installed throughout the buildings. Mechanical equipment will be equipped with interlocks to automatically shutdown equipment when sprinklers are triggered. Fire alarm panels, flow devices, pressure switches, alarm valves, manual pull stations, detectors, and audible alarms will be installed in individually protected areas.

The mine rescue/emergency response teams will be equipped with a foam generator to extinguish vehicle fires underground and on surface.

Emergency showers and eyewash stations will be installed throughout the process building.

18.2.4 *Waste Rock Storage*

A waste rock storage (WRS) site will be located in the valley at south of plant site. The WRS will be approximately 330 m long in east-west direction and 200 m wide in north-south direction. The average slope is 6.5%. The WRS capacity is 465×10^3 m^3, which will satisfy the requirement of waste rock storage for the life of the mine. The waste rock could be used as construction material for local infrastructures because there potentially is a high demand on the construction materials.

The waste rock will be transported to the WRS site by trucks and levelled by bulldozers. The starting elevation for the WRS site is 15.0 m and the final height will be 40.0 m with a side slope of 1:1.5. The arrangement has not taken consideration of the rock being used as construction materials. The site will be lit.

Due to the potential abundant rainfall in the mine site, the WRS will be provided with the following designs to prevent the potential runoff water pollution on downstream water:

1. Divert ditches, approximately 5 m outside of the final stockpile edge will be built to direct rainwater out of the mine site;

2. A protecting embankment will be built at the toe of stockpile to protect potential rolling rocks and collect washed soil from the stockpile;

3. A collecting pond will be constructed at the downstream of the stockpile to collect run-off water from the stockpile. The water will be sent to the waste water treatment plant for treatment prior to discharging to the public.

4. A retaining dam will be built outside of the collecting pond to catch possible mini-type earth flow. The elevation at the top of the dam will be approximately 16.0 m.

18.2.5 *Refuse Disposal*

Non-hazardous solid industrial waste from the operation will be collected and disposed at the landfill site designated by the local government. Scrap metals will be collected in bins and recycled by a qualified local contractor.

Domestic waste will be collected and disposed at the landfill site designated by the local government.

18.2.6 *Hazardous Waste Handling*

Waste oil, contaminated soil, and other hazardous wastes such as engine filters, aerosol cans, and glycol will be reduced, reused, and recycled as much as possible. Non-reusable hazardous waste from the operation will be classified and transferred off site to the appropriate approved recycling facilities and hazardous waste disposal facilities.

18.2.7 *Electrical Power Supply*

Off-Site Infrastructure

Power to the project will be provided via an existing 110 kV utility substation located in the local Fuwan town, approximately 4 km from the mine. NERIN and Minco have contacted with the Fushan Power Supply Company of the South Grid and confirmed that the Fushan substation has capacity to provide power to the Fuwan mining project.

This substation presently has a single incoming transmission line and will provide a single 35 kV power line to the mining project. The external 35 kV power line will be provided by the electrical utility to the mine site.

ON SITE INFRASTRUCTURE

Overall Mine Electrical Load and Mine Substation

At the mine, a step-down substation will be established consisting of equipment and facilities necessary to service the connected mine loads.

In summary, the projected mine electrical loading based on inputs from all project design disciplines is:

POWER

Connected - 15.9 MW (including underground mining)

Working - 14.0 MW (connected load less standby loads)

Average - 11.7 MW / 12.1 MVA

The electrical distribution design is developed based on the items entered in the project mechanical equipment list. This list includes process and mechanical design data including required equipment motor sizes (kW), information taken from proposed equipment proposals and estimates of electrical power requirements for non-process based loads. To this basic kW list are applied:

- demand factors/use factors/efficiencies
- typical reactive power ratings (relating load kW to KVA).

Application of these factors refine the basic connected kW loads into to loads indicated and are used in assessing the overall and area electrical loads.

The substation will consist of outdoor switchgear and a single 35 to 10 kV 16.0 MVA oil filled step-down transformer complete with on-load tap-changer.

A building housing a single line up of 10 kV switchgear and associated auxiliary equipment is provided within the main substation area.

Utility metering on the incoming 35 kV service will be in place and check metering at the 10 kV level (main and individual feeder levels) will provide the operating load confirmation to the mine.

Space has been designed in the substation yard for a second similar transformer. This second transformer and associated equipment can be added should a second, independent 35 kV power source become available from the local utility in future.

Refer to Drawing A0-07-009 for preliminary substation layout and Drawing A0-18-001 for preliminary project single line diagram.

Electrical Load Classifications and Area Loadings

Electrical loads are classified with three levels in decending order of priority. First priority loads are the backfill system and mine drainage systems. Second priority loads are the main production loads. Third priority loads are the auxiliary production facilities.

Initially, a stand-alone diesel back up generator is provided adjacent to the mining area in order to provide back-up power for the backfill and mine drainage functions should a problem occur with the main utility 35kV supply to the mine. In future, when a second independent 35 kV circuit may be available from the utility, the requirement for this generator can be re-evaluated.

Projected loading by area for the mine facilities are per Table 18.38 below:

Table 18.38 Projected Loading by Area

Area	Total connected load (kW)	Projected operating load (kW)	Annual operating energy (MWh)
A0 - Mining	6,376	5,500	39,994
A1 - Explosives Storage	25	18	143
B - Primary Crushing	287	227	939
C - Crushed Ore Bin And Reclaim	39	25	142
E1 - Primary Grinding	3,381	2,593	19,660
E2 - Flotation And Regrinding	2,325	1,750	13,784
E3 - Concentrate Dewatering And Loadout	156	98	666
E4 - Reagent Preparation	183	116	666
E5 - Tailings Handling And Reclaim Water	874	641	2,759
E6 - Process Services	781	446	1,971
G9 - Fresh & Fire Water Systems	11	4	12
G10 - Process Water System	33	14	114
G11 - Potable Water System	2	1	1
G12 - Gland Water System	15	6	46
G13 - Waste Water & Sewage Treatment/Collection	456	187	314
G15 - Maintenance And Warehouse Complex	41	31	64
G16 - Administration / Mine Dry	102	76	200
G20 - Fuel Storage & Distribution Facilities	1	1	0
J0 - Assay Laboratory	100	75	131
Others	717	359	1,136
Total Site	**15,904**	**12,167**	**82,741**

Site Power Distribution and Equipment Characteristics

Electrical power is supplied from the 35/10 kV substation transformer to a 10.0 kV switchgear (circuit breaker) line up located in a building on the substation site.

Power from that location is delivered at 10.0 kV via a combination of overhead lines and underground power cables as indicated on Single Line Diagram A0-18-001 provided in Appendix J. Separate power feeders are provided to:

- Overhead power lines to three mining ventilation substations
- Mine drainage distribution substation – normal power feed plus standby power feed from Diesel Generator system.
- Mine excavating distribution substation
- Cement preparation station

- Tailings Handling (Backfill) distribution substation – normal power feed plus standby power feed from Diesel Generator system
- Crushing area, grinding/reagent area, flotation/dewatering area, tailings disposal area, effluent treatment plant
- Administration facilities.

Motors greater than 300 kW will generally be supplied at 10.0 kV.

The two major motors are:

- Ball mill @ 1650 KW
- SAG mill @ 1250 KW.

Smaller three phase motors will be supplied from 400 volt nominal distribution systems.

Other

In addition to the standard equipment and protection features, the electrical system design also includes provisions for lightning protection and corrosion resistance for equipment.

18.2.8 COMMUNICATIONS

The external communications will use public communication systems which are available in Fuwan town, including wireless telephone system, local land line system, cable system and internet system.

Internal communications will be by an industrial Ethernet system that will provide voice, video, and data communication throughout the mine, and by hand-held radios for communication on surface.

Underground communications will be by a leaky feeder system.

18.2.9 TRANSPORTATION

EXTERNAL HAULAGE ROAD

The haulage distance between the minesite to the public transportation road, which is a concrete road close to Xi'an river, is approximately 1.2 km. The external haulage road will be 6.0 m wide (the width between road toes will be 7.5 m). The road top layer will be 22 cm thick cement concrete supported by a 25-cm cement stabilized macadam base and a 15-cm thick pebble sub-base.

INTERNAL HAULAGE ROAD

The waste rock haulage road between the decline portal and waste rock storage site will be 3.35 km long. The width of the road will be 7.0 m wide (distance between road toes will be 9.0 m). The maximum longitudinal slope of the road will be 6%, the minimum horizontal curve and vertical curve radius will be 30 m and 500 m. The road pavement will consist of a 3-cm gravel wearing layer, a 30-cm clay-binded gravel layer and a 15-cm pebble sub-base.

The other main internal road network within the mine site will be 6 m wide cement concrete surface pavement with a base width of 7.5 m. The longitudinal slope of the road will be less than 6%. The minimum horizontal curve and vertical curve radius are 30 m and 500 m respectively. The pavement structure will consist of a 22-cm cement concrete top, a 25-cm cement stabilized macadam base and a 15-cm thick pebble sub-base.

A 1.25-km long gravel road will be built to connect the plant site and explosive store facility. The width of the road base and the top pavement will be 5.0 m and 3.5 m wide respectively. The pavement structure will include a 15-cm clay-bound gravel layer and a 20-cm pebble sub-base.

18.3 TAILINGS

The Fuwan Silver project includes development of a new proposed land based TMF to store up to 2.6 M m^3 of tailings. The selected Nankeng Cleuch TMF site for the main TMF is in a valley located approximately 0.75 km southwest of the proposed process plant. Another valley designated for the waste rock dump separates the TMF and process plant sites. These locations are shown on Figure 18.35.

The TMF will be developed in two stages:

- Stage 1 Facility - capable of storing initial 8.3 years of tailings deposition through three dam raises; and,
- Stage 2 – Final Facility capable of storing additional 0.9 years of tailings deposition by either raising the Stage 1 Facility or on-land storage in a separate facility.

Current cost estimate assumes that raising the Stage 1 TMF Dam (subject design) to accommodate additional 0.9 years of tailings deposition is feasible. However, this is to be confirmed by subsequent geotechnical and hydrogeological investigations.

18.3.1 SITE CHARACTERIZATION

GEOLOGY / HYDROGEOLOGY

The site is underlain by the following major geological units:

- Lingyun (T_3xly) is comprised of glutenite, greywacke, siltstone and mudstone characterized with tight fractures with predominantly calcite infilling.
- Fenggang (T_3xf) is comprised of black mudstone, carbonaceous shale intercalated with siltstone, and sand; pyrite nodules are secondary constituents.
- Dajing (T_3xd) is comprised of gravestone, glutenite, greywacke, sandstone intercalated with siltstone/mudstone characterized by tight fractures with predominant calcite infilling.

Based on the various pumping tests performed on the sandstone unit throughout the general mine site area (TMF site excluded), it can be considered as an aquitard with conductivity in the range of 10-7 to 10-9 m/s (10-5 to 10-7 cm/s). As described, the sandstone units are subject to fracturing and fissures so may have isolated areas of greater permeability but testing at over a dozen locations failed to show any significant groundwater movement between holes suggesting these are local rather than regional features. Other constituting members of the above units are expected to have similar hydrogeological characteristics.

CLIMATE AND METEOROLOGY

This area is warm and humid with a subtropical monsoon climate. The rainy season lasts typically half a year (April to September). Summary of climate and meteorological characteristics based on data recorded during the period from 1992 to 2006 period at the Heshan meteorological station is presented in Table 18.39.

Table 18.39 Climate and Meteorological Characteristics (1992-2006)

Parameter	Value	Parameter	Value
Annual average temperature	22.4°C	Maximum annual precipitation	2,417 mm
Maximum temperature	39.6°C	Average annual precipitation	1,878 mm
Minimum temperature	2.6°C	Minimum annual precipitation	1,161 mm
Average annual evaporation	1,488,mm	Maximum monthly precipitation	577 mm
Maximum annual evaporation	1,656 mm	Maximum daily precipitation	260 mm
Minimum annual evaporation	1,268 mm	Maximum hourly precipitation	72 mm
Maximum wind speed	35 m/s	Maximum one-time continuous precipitation	570 mm

DRAINAGE

The highest and lowest regional features are Lingyun mountain (414.6 m a.s.l.) and the bottom part of Changkeng Watergate (1 m a.s.l.). Situated in between these, the TMF site valley is shaped by denudation. The proximity of the surface water divide defines a relatively small watershed.

SEISMICITY

The mine site is located in the middle of southeast coastal earthquake sub-region of China's South Earthquake Region. The seismic activity is characterized by low frequency and moderate earthquake intensity. The "Seismic Building Design" (GB50011-2001, Chinese Regulation) classifies the corresponding seismic resistance design as "Point 6" with an assigned seismic acceleration value of 0.05g.

SUBSURFACE CONDITIONS – GEOTECHNICAL PROFILE

Feasibility level geotechnical investigation was planned but not completed. Based on the knowledge of site geology and walk-over survey the following subsurface conditions are inferred:

- The site is underlain by thin alluvium (valley bottom dam foundation and pond) and deluvium/residual soil (dam abutments and pond above the valley bottom).
- This in turn is underlain by moderately to highly weathered sedimentary/meta-sedimentary formations exhibiting low permeability.

- Bedrock units are bedded with upstream dipping based on observations on few exposures (quarry face/outcrop exposures).

18.3.2 TMF Design Basis

Design Criteria – Mine Plan

The basic mine plan design criteria relevant for geotechnical designs are listed in Table 18.40.

Table 18.40 Design Criteria

Item	Value
Tailings Throughput (330 days/year)	2,886 t/d (952,400 t/a)
Tailings to Underground (Coarse)	1,443 t/d (476,200 t/a)
Tailings to TMF (Fine)	1,443 t/d (476,200 t/a)
Total Mine Service Life	9.2 years
Stage 1 TMF – Service Life	8.3 years
Fine Tailings Pulp Density (Fine)	15%
Average Final Tailings Density	1.5 (t/m^3)
Tailings to TMF Annually	318,000 m^3
Total Tailings to TMF (8.3 years)	2.6 M m^3
Total Hourly Process Water Demand from Tailings Pond	305 m^3/hour

Design Criteria – Engineering

The engineering design criteria are listed in Table 18.41.

Table 18.41 Engineering Design Criteria

Item	Target Values	Comments
1. Geotechnical Dam Slope Stability		
After Construction	Static F.O.S. 1.5, Pseudostatic F.O.S. 1.1	Analyses using Slope/W including sliding resistance along woven bags filled with tailings//HDPE membrane/clay core interfaces to be carried out when the results of geotechnical and hydrogeological investigations become available.
Closure	Static F.O.S. 1.5, Pseudostatic F.O.S. 1.1	
2. Seepage	To comply with environmental and dam stability requirements	Analyses using SEEP/W to be carried out when the results of geotechnical/hydrogeologicl investigations become available.
3. Hydrology/Hydrotechnical		
Design Flood	1:30-year for initial construction stages 1:300-year later construction stages	Tailings dam is ranked as “Class IV” under “Design Code for Mine Concentrator Tailing Facilities” (ZBJ1-90). With flood return periods of 1:30-year to 1:50-year and 1:100-year to 1:200-year for initial and later stages, respectively. As dam will contain significant amount of water over tailings, the it falls under more stringent standards for water retention structure covered in “Standard for Classification and Flood Control of Water Resources and Hydroelectric Project” (SL252-2000) with the designs governed by return periods of 1:30-year to 1:50-year (design) and 1:300-year to 1:1000-year (verification).
Free Board	Not less than 0.5 m for 1: 1:30 year design flood Not less than 0.3 m for 1:300 year flood	Tailing dam is ranked as Class 4 structure in the “Standard for Classification and Flood Control of Water Resources and Hydroelectric Projects”（SL252-2000) also defining free board requirements for flood events.
Runoff Collection System	1:30-year rainfall event Contributing drainage area of 0.17 km^2 (17 ha) to report to perimeter runoff diversion ditch	

Item	Target Values	Comments
Decant Pond	Decant tower/pipe to maintain TMF pond water level below free board Reclaim water (305 m^3/hour) – pumps on barge to process plant Emergency discharge water from the TMF pond will report to the Nankeng Reservoir[4]	
Closure Spillway	1:1000	Overflow from diversion ditches (designed for 1:300 year event) to be included in spillway sizing. PMF may be required if CDA[1], MAC[2] or ICOLD[3] guidelines are superior to local regulations; may require site-specific meteorological and hydrologic evaluations.
4. Seismicity		
Operating Design Basis Earthquake	Ground acceleration of 0.05g	According to Building Code (GB50011-2001) the design ground seismic acceleration for mine site is 6. May be different if CDA[1], MAC[2] or ICOLD[3] classifications are superior to local classifications; may require site-specific seismic evaluations.
Closure Earthquake	1:1000-year	MCE may be required if CDA[1], MAC[2] or ICOLD[3] guidelines are superior to local classifications; may require site-specific seismic evaluations.
5. HDPE Membrane	Cover: woven bags filled with tailings. Base: clay core	
6.HDPE Membrane Friction Angles	TBD	Between tailings bricks and HDPE liner (checked for final design by friction test)
	TBD	Between HDPE Liner and clay bedding (checked for final design by friction test)

Notes:

1. Canadian Dam Association
2. The Mining Association of Canada
3. International Commission of Large Dams
4. Nankeng Reservoir (capacity of 98,300 m3) is located approximately 200 m downstream of the TSF dam. The reservoir is currently used as fish pond. The use will be changed by leasing it during the life of mine for containment of emergency overflow from the TMF pond. Water from the Nankeng Reservoir will be pumped back to the TSF pond for further use in the process plant.

ADDITIONAL DESIGN CRITERIA AND ASSUMPTION

Additional design criteria and assumptions are as follows:

- TMF site location is selected by Minco (with inputs form NERIN) in a valley located approximately 0.75 km southwest of the proposed process plant.
- The level of the TMF design is constrained by the lack of knowledge of subsurface conditions within the TMF footprint and downstream environment as geotechnical and hydrogeological investigations are not completed for the design development at the feasibility level. This renders the geotechnical designs of the TMF at a stage between the pre-feasibility and feasibility levels. Detailed geotechnical engineering analyses will be postponed until the geotechnical and hydrogeological investigations have been completed.
- The direct application of experience of NERIN in dam design and construction practices in China and familiarity with the technical and regulatory requirements in the Guandong Province should benefit recommendations on dam type and the choice of dam construction materials, construction cost estimating, and integration of these in the overall mine plan development.
- TMF dam design should maximize the use of local construction materials (e.g., saprolite) and local non-ARD rock from quarries within the TMF pond as this brings potential for increasing the TMF pond capacity and at the same time reduce the dam construction cost. Other nearby rockfill sources or non-ARD waste rock form mining operations should also be considered as a part of the borrow search.
- Coarse tailings fraction will be used for underground backfill and fine tailings fraction will report to the TMF pond. The use of coarse tailings underground eliminated the TMF dam design involving cyclone sand stacking. Remaining tailings design options involve high capital lost for the initial (starter) structure.
- TMF dam design should allow for a robust, short and flexible construction. Downstream construction is to provide flexibility in early and subsequent stages of tailings deposition through adjustments to actual production schedule.
- TMF pond management (storm water management included) cost should be low.
- A relatively large TMF storage capacity is to ensure reduced discharge of excess water to downstream environment or operation as a zero-discharge facility. This is favourable from the environmental regulative requirements.
- Tailings particle size distribution is expected to comprise 70% of fines (minus 0.074 mm) fraction.

- The initial results of Lock Cycle Geochemical Testing (LCGAT) indicated settling times in the between 2 to 32 days.
- Settling tests conducted on fine tailings indicated that sedimentation of suspended solids occurs within approximately 8.3 hours.
- The results of kinetic testing extended to the time of this writing indicate that:
  - There are no ARD issues;
  - The Lead (Pb) concentrations seem to be gradually increasing with time, almost doubling in concentration from the initial leachate analysis; and,
  - Aluminum (Al) and iron (Fe) and 226Ra show sporadic high values

  The Pb levels do not exceed Metal Mining Effluent Regulations (MMER), however the rate of increase is quite steep and the trend may have potential impacts on TMF seepage water quality. The Fe and 226R are not of concern but the high Al concentration exceeds the Canadian Council of Ministries of the Environment (CCME) guidelines for the protection of or aquatic life. Mercury (Hg) contents assayed from the kinetic testing remain close to its detection limit after the "Day Zero" sampling event.

- At the time of this writing there is no firm evidence that the TMF impoundment effluent requires treatment to meet receiving water criteria. Close monitoring of the above indications/trends will determine as to whether TMF design modifications are required in subsequent design stages.
- Water management of the TMF pond should include closure measures ensuring the long term dam stability and protection of the downstream environment.

18.3.3 TMF DESIGN

SITE SELECTION

Alternative sites for tailings disposal included two natural valleys surrounding the site. A walkover survey of the sites by the technical personnel of Minco and NERIN, took place prior to selecting the Nankeng Cleuch site as preferred site. The selected site is located approximately 0.75 km (short distance) southwest of the proposed process plant. Stipulations from the Chinese "Design Code for Mine Concentrator Tailings Facilities" (ZBJ1~90, Item 2.01) were considered in the selection process. The site selected for the TMF design presented in the following sections has met overall objectives set by Minco. The site is suitable for storage requirements set out in the basic design criteria.

TMF Design Section

Local topographic and geomorphologic conditions at the selected site are suitable for containing the conventional wet tailings by constructing an earth/rockfill structure across the lower part of the valley. The upstream dipping of bedding, observed in exposures of low permeability sedimentary formations at the proposed TMF dam location, is favourable (reduced downstream seepage/enhanced holding capacity). The pond created by constructing a 56 m high dam will have a storage capacity of 3.4 M m^3 (tailings, water and freeboard included). The TMF dam design configuration is depicted in Figure 18.37 and Figure 18.38 below, and a brief description is provided as follows:

The TMF Dam will be a 56 m high earth/rockfill structure with a 6 m wide crest and composite HFPE /clay core lining (Zone 1 / Zone 2) on the upstream slope The HDPE membrane will be protected by woven bags filled with tailings (Zone 1). The thickness of the clay core (Zone 2) varies from 13 m (foundation level) to 1.4 m (crest). Clay core will be separated from the coarse rockfill shell (Zone 7) by the following filter graded zones:

- Zone 4: Sand (0.6-3.5 m thick)
- Zone 5: Fine Crushed Rock (0.5 – 2.5 m thick)
- Zone 6: Crushed Rock (0.5-2.5 m thick)

The dam will be constructed in three stages:

- Stage 1 (3.1 years storage capacity) will be 38 m high with crest at El. 61 m.
- Stage 2 (2.7 years storage capacity) will add additional 10 m bringing the dam crest to El. 71 m.
- Stage 3 (2.5 years of storage capacity) will add another 8 m for the final crest at El. 79 m.

The upstream and downstream slopes of the embankment will be 2H:1V and 1.75H:1V, respectively. The bottom of the clay core will sloped at 1.75H:1V. The downstream slope will have a 2 m wide bench spaced at every 15 m of the embankment height.

The TMF dam will be constructed using downstream construction methodology. Compacted non-ARD quarried waste rock (or waste rock form underground mining) will be used in the dam shell. The balance of the embankment will be constructed using compacted granular materials obtained by crushing and/or processing of non-ARD quarried rock (mine waste rock) and mineral soil from local borrow sources.

Figure 18.37 TMF Dam and Pond – Plan

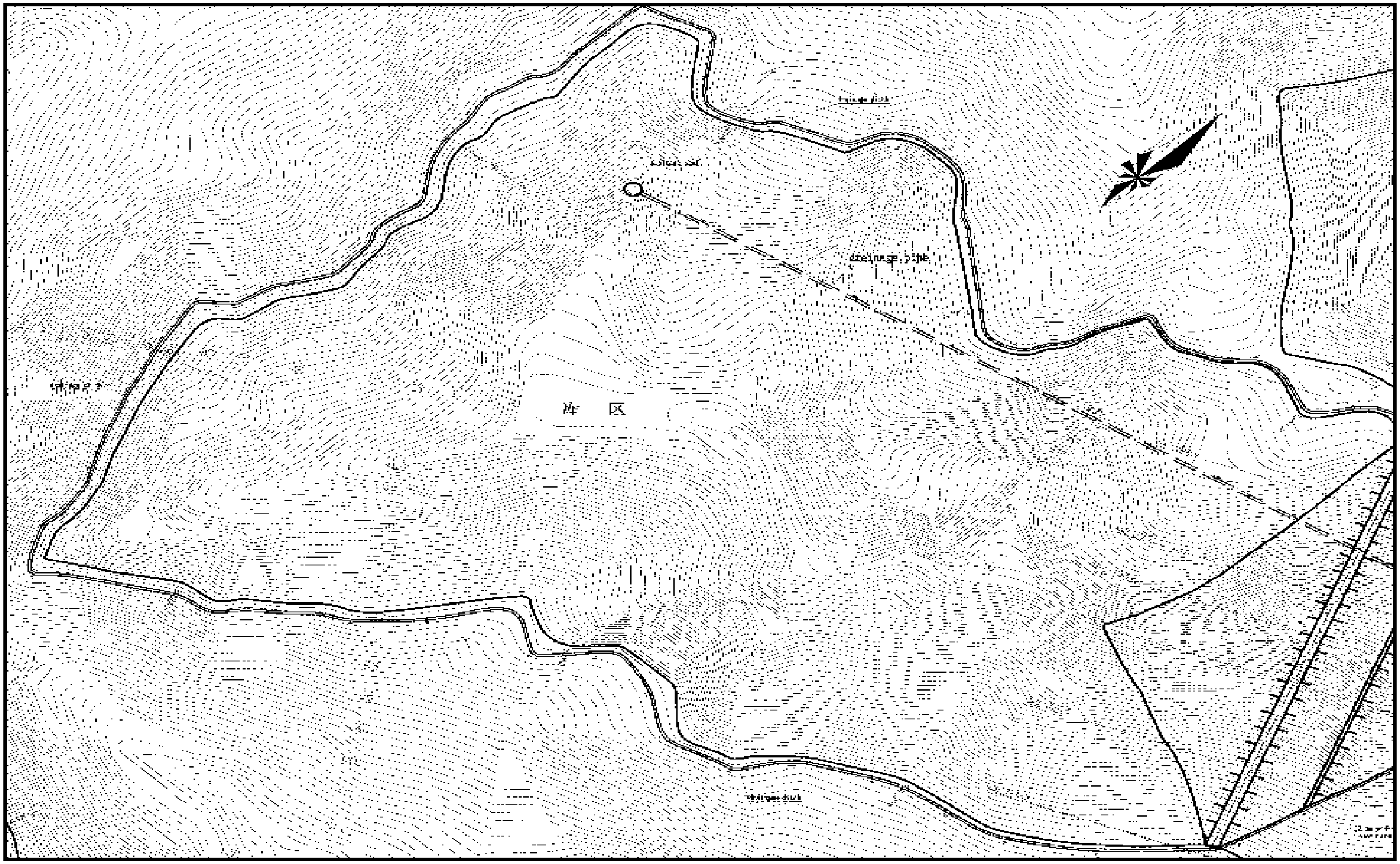
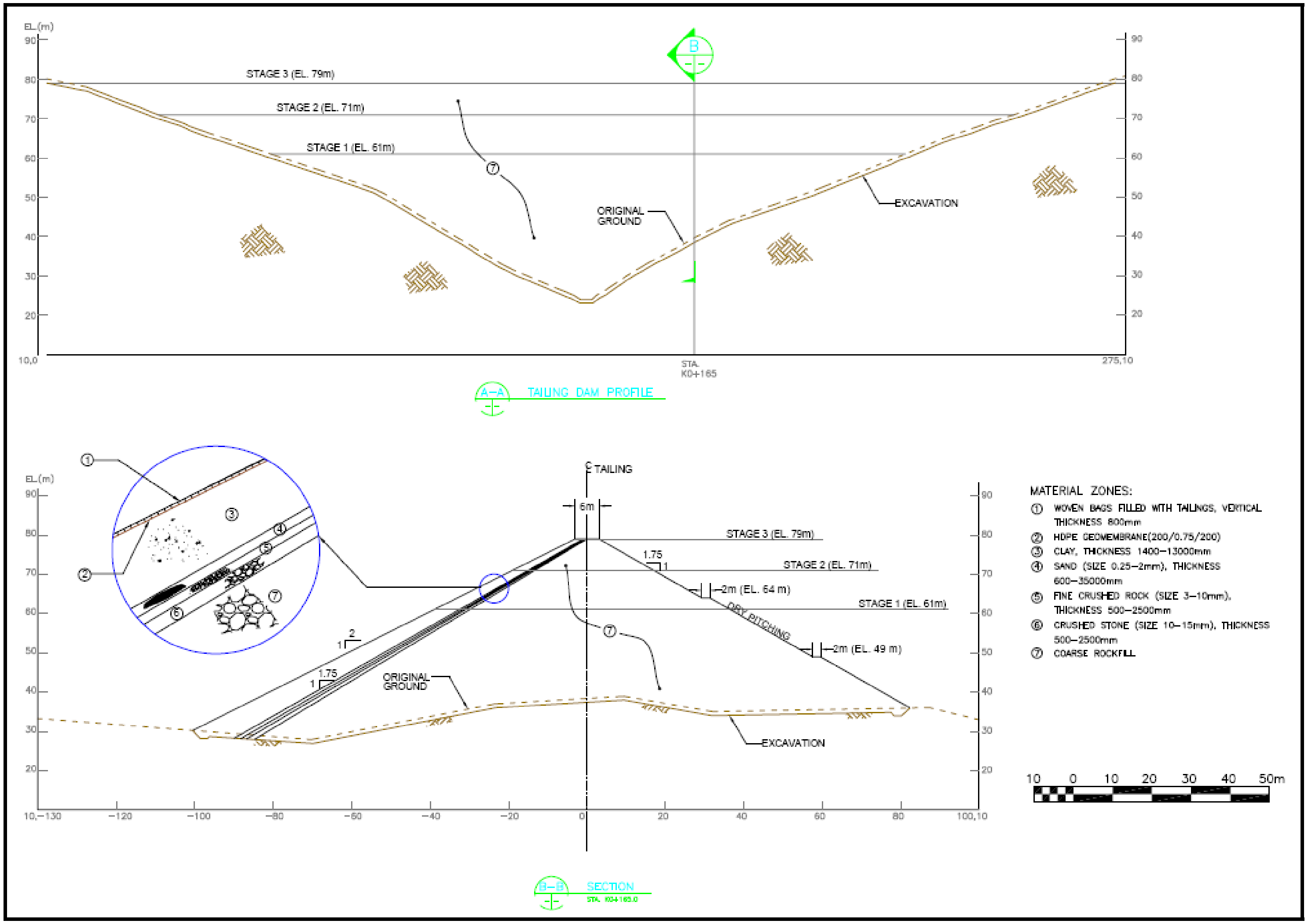


Figure 18.38 TMF Dam and Pond – Profile and Section



STABILITY AND SEEPAGE ANALYSES

The stability and seepage analyses in support of the subject TMF design were not performed due to limited knowledge of subsurface conditions. The subject TMF dam design should be revisited when the results of the geotechnical investigation have become available and any modifications should be aided by the stability analyses. The dam stability analyses should be performed for static pseudo-static conditions after construction and for closure.

Seepage analyses should be based on the results of in-situ permeability testing when they become available. The computed seepage losses should be included in the TMF water balance. Current knowledge of the tailings effluent chemistry does not warrant contaminant transfer modelling in support of the design measures for protection of sensitive downstream environment. TMF water balance should be updated when the results of seepage analyses have become available.

ASSOCIATED FACILITIES

Storm water will be managed using the following structures:

- Perimeter diversion ditch
- Decant tower and pipe.

Schematic locations of these structures are shown on Figure 18.35 and their brief description is provided in the following paragraphs.

The perimeter diversion ditch will intercept runoff from a catchment above the TMF pond to take it into the canal downstream of the dam for eventual discharge into the natural environment. The ditch is sized based on 1:30-year runoff event and will have a square cross-sectional area (1.5 m x 1.5 m) and slope of 0.6 %. The ditch will be lined with cemented block stone. The flow channel irregularities will be smoothed by sand grout.

A combination of vertical decant tower and horizontal decant pipe will be used for the TMF pond emergency flood control. The tower will be 27 m high reinforced concrete structure with a 2.5 m diameter. The decant pipe, 1.5 m in diameter, will be sloped at 6% along a 540 m stretch, the last part of which is going to be underneath the dam. The alignment shown on Figure 18.37 is schematic and will actually achieved through cut and fill between the decant tower and the dam.

The hydrologic and hydraulic computations are presented in the following section.

18.3.4 TMF WATER HANDLING AND BALANCE

WATER BALANCE

The water balance for the TMF is based on the inputs from the process engineering presented in A0-09-014 in Appendix B.

Water inputs into the TMF pond include:

- The water in the tailings from the hydrocyclone overflow is pumped to the TMF pond at a computed rate of 305 m3/hour.
- Based on tailings characteristic, 15 m^3/hour of water (out of 305 m^3/hour of water) is going to be trapped in tailings voids.
- Net rainfall (rainfall minus evaporation) is 15.5 m^3/hour
- Emergency water discharge into Nankeng reservoir will report back to the TMF for eventual use in the process plant.

Water outputs from the TMF pond include:

- Process water reclaim at a computed rate of 305 m^3/hour
- Evaporation (average annual evaporation is 80 % annual precipitation)
- Seepage (not quantified – assumed not to substantial)

The water balance qualifies the TMF as zero-discharge facility for average meteorological conditions. If there is excess water in the pond, the water will be reclaimed as make-up water for process plant.

FLOOD COMPUTATIONS

The following parameters relevant for computation of TMF flood volumes were selected from the Manual for Computations of the Guangdong Province Storm Runoff (Guangdong Province Hydrological Master Station, 1991):

- The mean value of the maximum 24-hour rainstorm: H_{24}=129 mm
- The variation coefficient of the maximum rainstorm in 24 hours: CV=0.36
- The deviation factor of the maximum rainstorm in 24 hours: Cs=3.5CV
- The damped exponential of the storm intensity: n1=0.447 and n2=0.674
- The average later loss rate: f=5.0 mm/hr

A topographic map (Scale 1:2000) was used for the TMF site watershed delineation. The total TMF catchment area measures 0.34 km^2 (34 ha) and 0.17 km^2 (17 ha) represents an area between the perimeter diversion ditch system and the surface water divide.

The simplified rational formula was used for the computation of peak flood volumes and the results are shown in Table 18.42.

Table 18.42 Peak Flood Volumes

Catchment Area (km^2)	Flood Return Period	Design Frequency Rainfall $H_{24}P$ (mm)	Peak discharge Qm (m^3/s)	Peak Flood Volume m^3
0.34	1:30 - year year(design)	233.36	8.0	79,300
	1:300 – year	316.05	10.3	107,500
0.17	1:30 – year	233.36	3.8	39.700

TMF Pond Storage Capacity

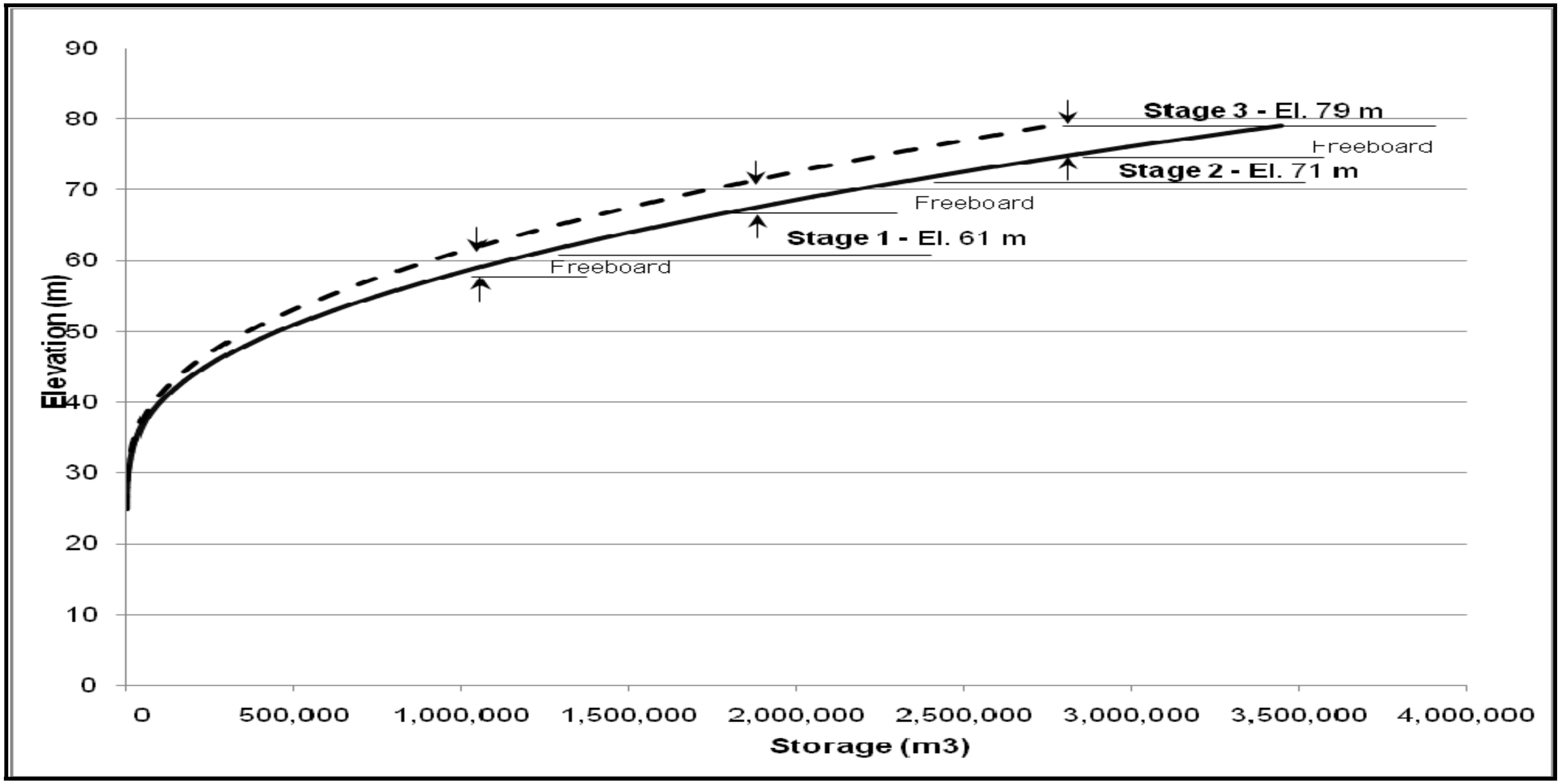
The required storage capacity for theTMF was arrived at through a combination of the following:

- Tailings production schedule
- Effective storage (80 % of accumulated storage capacity to account for volume lost to tailings deposition slopes, freeboard requirements and other unknowns)
- Water balance including the above computed flood volumes and excess water discharge through decant tower/pipe ensuring the design freeboard requirements.

Design of flood control measures for Stage 1 dam is based on 1:30-year storm and includes the runoff diversion via the perimeter ditch. Stage 2 and Stage 3 dam are designed for a 1:300-year storm when the perimeter ditch will overflow into the TMF pond with the excess water collected by the decant tower and transfer through the sloped decant pipe for eventual discharge downstream of the dam. In both cases the peak flood will be contained within 1 m of available storage below the freeboard.

Figure 18.39 shows the stage-storage curve for the TMF for the dam construction in three stages as presented above. The freeboard shown includes effective storage and peak flood volume remaining in the TMF pond.

Figure 18.39 Stage – Storage Curve



18.3.5 CLOSURE

The closure measures consideration of reclamation cover, decommissioning of decant tower/pipe and design installation of closure spillway. Capital and operating costs for the dam and associated structures are quite detailed and reflect NERIN's experience with the dam type and the choice of construction materials in the Guangdong Province. Geotechnical portion of closure estimate is a lump-sum as the closure designs have not been advanced. Proposed closure measures are briefly discussed in the following sections.

RECLAMATION COVER

Based on meteorological net rainfall generally is a positive value for maximum and average precipitation and evaporation, minimum annual evaporation (1268 mm) is greater than minimum annual precipitation (1161 mm). This may warrant dusting issues that will be prevented by placing a vegetated soil cover.

DECOMMISSIONING OF DECANT TOWER/PIPE

Following the cessation of mine operations decant tower pipe should be plugged at the receiving and discharge ends. Usually these would be concrete plugs. Risks associated with operation of decant tower/pipe after closure, are explained in the Risk Assessment section below.

CLOSURE SPILLWAY

Mine reclaim water will no longer be required on closure and the water in the tailings pond will undergo a different regime (placement of vegetated cover/decant tower/pipe will be decommissioned). Storm water management will include closure spillway in order to maintain safe dam operation without encroaching into freeboard. Alternatively, a siphon pipe or tunnel through dam abutment could be considered.

18.3.6 DESIGN GAP IDENTIFICATION AND RISK ASSESSMENT

DESIGN GAP IDENTIFICATION

The subject TMF designs have been developed in between the prefeasibility and feasibility levels. This has mainly been influenced by the lack of detailed engineering analyses (dam stability and seepage). The main reason for this is current lack of knowledge of subsurface conditions in the TMF dam and pond foundation areas as well as their immediate downstream environs. Subsurface conditions were to be determined by:

- Engineering geological mapping;

- Geophysical investigation;
- Subsurface investigations involving geotechnical and hydrogeologic drilling;
- Laboratory testing (soil and rock physical and mechanical testing); and,
- Borrow search and geotechnical investigation of borrow sources.

Other reasons for not developing designs at a level commensurate with the feasibility study are as follows:

- The use of a 1:2000 topographic map (too coarse and more detailed topography should be prepared for design details);
- Site specific seismic and meteorological evaluations were not completed; and,
- Hydrologic evaluation including stream flow measurements was not completed.

The direct application of experience of NERIN Institute in dam design and construction practices in China and familiarity with the technical and regulatory requirements in the Guangdong Province was quite important in selecting the dam type and the choice of dam construction materials. The unit costs in the cost estimate are considered to represent realistic costs in 2009 dollars. Any design modifications based on the results of detailed engineering analyses could have impact on the cost estimate.

A more detailed discussion of potential modifications from the project risk perspective and corresponding impacts on cost estimate is presented in the following section.

RISK ASSESSMENT

The following risks were identified and mitigation measures proposed:

- Detailed geotechnical engineering analyses have not been completed and this may have a potential impact on the current design and cost estimate accuracy because of potential design modifications to be developed when the results of geotechnical and hydrogeological investigations and laboratory testing have become available:
  - Dependent upon the results of engineering analyses (combined stability, seepage, deformation) and the natural make-up of the TMF dam foundation, adjustments of the dam slopes, potentially involving larger dam volume and consequent higher construction cost may be warranted. The subject design involves a 56 m high dam that has a significant clay core to be constructed at relatively steep angle of 1H:1.75V. This may warrant deformation analyses for prediction of long term clay core integrity. Flattening of the clay core would also have an impact on the construction cost.

- A cut off trench extending the clay core to bedrock may be more than inferred in the current design depending on the thickness and composition of alluvium at the bottom of the valley and, residual soil profile in the abutments. If confirmed, this would increase the dam construction cost.
- Fine rockfill zone and geotextile may be required in the dam foundation underneath the coarse rockfill shell if the foundation soils comprise fines. Confirmation of fine soils in the foundation would warrant inclusion of these materials. Their installation would be indispensable from the dam stability point of view in case of hydraulic connection between the decant pond and dam foundation via a fault/shear/fracture zone that could result in foundation loss of ground (migration of fines into coarse rockfill voids). Geomorphology of the TMF valley could be tectonically predisposed. Loss of ground in the vicinity of clay core may become a TMF dam failure mechanism. Construction cost will increase if the above inferences are confirmed.
- Bedrock within the TMF is assumed to exhibit low permeability. If this supposition is incorrect, and the bedrock units actually exhibit fracturing along with weathering and alteration leading to their higher permeability, this will have an impact on the overall site water balance and this may warrant expensive seepage control measures such as pond lining and dam foundation grouting that would have a substantial impact on the construction cost increase.
- Design of closure spillway or siphon pipe or tunnel would likely require an encroachment into one or the TMF dam abutments. Depending on the residual soil profile (transition from weathered in situ soil to bedrock, the spillway design may require substantial rock blasting within the dam abutment above the spillway level. This will have an impact on the closure cost. Designing the spillway shoot may also require an innovative design to adjust to the ground configuration and subsurface conditions, also with a potential for an increased closure construction cost. Consideration of a siphon may warrant modifications of the current dam design section.
- TMF dam design section presents a zoned dam. The lack of geotechnical investigation and results of laboratory testing on materials from potential borrow sources may have an impact on the TMF dam section zoning. Gradation envelopes for dam fill zones should be developed to fulfill filter criteria. Actual gradations of samples of these materials obtained by sieve analyses should be checked against the design gradation envelopes to confirm that the current design section is workable.
- The decant pipe alignment shown on Figure 18.37 above is schematic and will actually achieved through substantial cut and fill between the decant tower and the dam that is not considered in the current cost estimate. Some of the cut sections may require blasting and this can further elevate the construction cost.

The risks that are not directly related to the lack of knowledge of subsurface conditions and corresponding mitigation measures can be summarized as follows.

- Current design has a provision of the vertical decant tower and sloped pipe for storm water management regulating the TMF pond water within the safe operation levels. Having a concrete conduit under almost the entire tailings pond and dam is prone to deterioration and fracturing (e.g., differential settlement of the pipe foundation) that could lead to loss of tailings from the pond and their flow into the downstream environment. Running the decant pipe (water conduit) conduit under the dam core is usually mitigated by providing seepage collars, yet there are dam failure histories on record associated with problems with decant pipes in their foundations.

  From the closure perspective the above represents a high likelihood and high consequence risk, and therefore, their plugging is planned for closure. Risks associated with a potential for breaking and blockage of this conduit during mine operation are moderate, however, the consequences could be high especially in During Stage 2 and Stage 3 TMF dam/pond operation designed for 1:300-year return periods.

  Modern practice is now shifting to use of decant towers constructed by stacking perforated prefabricated concrete elements surrounded with geofabric materials and discharge by pumping. These towers can target specific locations within the TMF pond and can be accessed by a causeway. This scheme is worth considering in the next design stage as it has a potential for elimination of pumping from a barge and can also be used for regulating the peak flood volumes.

- At the time the rainfall data from the Heshan weather station (only one station) has only been collected 1992 to 2006. Future work should consider additional weather stations located close to the site in search for more data collection. A rainfall record of only 14 years is marginal to design a tailings facility. Design storms used in this herein seem appear to adequately satisfy the Chinese regulatory agencies, yet this is still below what would be used in North America. This is recognized as recommended approach in the engineering design criteria herein, especially for after closure conditions. The values for rainfall, variation coefficients used in the subject design are valid for very early stages of the design of a tailings facility and site specific meteorological evaluations should be completed along with hydrologic evaluations including stream flow measurements.

- Water balance should be revisited and run on a monthly basis for an average year and wet and dry year conditions.

- Seismic design criteria presented herein were not used in the design analyses. Site specific seismic analyses should be completed for use in detailed engineering analyses. Alignment of seismic design criteria with North American practice may have potential impact on increasing the dam volume and consequently increasing the construction cost.

- At the time of this writing there is no firm evidence that the TMF impoundment effluent requires treatment to meet receiving water criteria. However, there are some indices (e.g., the Pb levels do not exceed MMER regulations, yet but the rate of increase is quite steep; Al concentration exceeds the Canadian CCME for aquatic life) with potential impacts on TMF seepage water quality. In case of increased seepage predisposed by tectonics, sensitive downstream environment may be affected if the trends lead to elevated concentrations of specific constituents. Close monitoring of the above indices/trends in combination with the results of seepage analyses and contaminant transport modeling would determine as to whether TMF design modifications are required in subsequent design stages. Mitigation measures for seepage water quality measures could warrant expensive seepage control measures such as pond lining and dam foundation grouting with a substantial impact on the construction cost.

- Other significant risks related to economic items revolve around normal site uncertainties, TMF pond volume estimates, expected density of tailings and topography.
- Estimated TMF pond volumetric capacity is based on experience with similar tailings but does not yet recognize the initial density that can be up to 25% lower than the average final density used for sizing the subject TMF. Also, tailings segregation potential leading to finer grained and looser tailings away from the beaches can also potentially decrease the average final density below a 1.5 tonnes/m^3 ised in the subject design.

  The above lower densities would result in larger storage requirements and would lead to an increase of the dam height and consequent elevated construction cost. Tailings settled densities and segregation test should be carried out using standard tests.
- The remaining noteworthy risks include the potential for post closure embankment erosion and potential need for permanent erosion protection that can be insured by placement protective layer composed of fine rockfill. The probability of a need to provide erosion protection layer is considered to be low.

18.3.7 POTENTIAL FOR OPTIMIZATION

Areas identified to have potential for optimization are:

- A relatively large contingency is used in the subject design to account for related to tailings deposition, tailings dry density, and amount of water to be stored in the pond. Conceptual deposition planning involving peripheral discharge could provide more flexibility in deposition planning resulting in greater utilization of available storage and consequent reduction of the dam height resulting in construction cost savings. Peripheral scheme could be implemented using the material from the excavation of the perimeter runoff diversion ditch for the road construction that will be used for the tailings feeder pipes.

- Borrow areas/quarries within the TMF pond footprint would be close to the dam and at the same time would increase the TMF pond capacity, reduce TMF dam height and lower the construction cost.
- HDPE liner placement over the clay core may potentially be eliminated depending on the quality of local low permeability materials to be used in compacted clay core.
- There is a potential for using the fine tailings for manufacturing bricks in support of current local construction materials demands and existing brick manufacturing infrastructure. This will have a potential for reducing the required TMF storage capacity and consequent construction cost savings.

18.3.8 ACTION PLAN

Action resulting from preceding discussion can be summarized as follows:

- Topographic survey of the tailings at a scale suitable for the Feasibility design (more detailed than 1:2000) should be conducted within the TMA.
- Site investigation should be planned and implemented in order to alleviate uncertainties in relation to subsurface conditions impacting the design. The following is proposed on a preliminary basis:

• Engineering geology mapping	TMF site
• Seismic refraction survey profiles:	6 kilometres
• Test pits (up to 6 metres deep):	30
• Geotechnical boreholes terminated on bedrock:	10
• Deep geotechnical boreholes (20-40 m deep)	8
• Falling head tests in overburden:	TBD
• Packer tests in bedrock	TBD

- Geotechnical investigation should include borrow materials (mineral soil and rock) Lab testing confirming geotechnical properties will be carried out on samples taken from test pits.
- Tailings segregation tests should be carried out using standard tests which are available from the University of Alberta.
- Site specific seismic and meteorological evaluations should be completed.
- Hydrologic evaluation including stream flow measurements should be completed.
- The results of site specific evaluations should be applied to TMF pond water balance. Subsequently, hydrotechnical design criteria (freeboard, decant pond retention, closure flood, closure spillway(s), etc.) should be established and used in design modifications, if required.

18.4 Environmental Considerations

The detailed environmental assessment was completed by ERM and is presented in Appendix K

18.5 Reclamation, Decommissioning, and Closure

Reclamation Goals

Reclamation goals are to return the mine site to as near original condition as possible, and in such a way that flora and fauna are not impacted after mine closure. Mine reclamation will mitigate the disturbance caused by mining operations in a manner that does not require on-going maintenance after closure.

By the end of the mine life, the area will have been improved by returning the underground to flooded conditions.

Reclamation will start at time of mine construction, with the removal and storage of topsoil to be used during reclamation. Ditches and settling ponds will be constructed early to minimize soil erosion.

Reclamation will involve shaping the land to as close to original contours as possible, and otherwise to a natural shape conforming to the general environment. Re-contouring will shape the ground to facilitate natural drainage patterns before being covered with topsoil.

Plant Site and Mine Portals

The mine site will have all buildings, equipment, and structures removed or levelled, and the ground re-contoured to conform to the natural contours before finally being covered with topsoil. Any acid generating material will be returned underground prior to mine closure.

All materials, fluids, and chemicals will be removed prior to closure.

Due to earlier mining operations when topsoil was not stripped and stored, there is limited topsoil within the plant site area. Topsoil or other suitable cover material will be found in the vicinity and used for rehabilitation.

Underground Workings

As an ongoing part of mining, backfill will be placed in the stopes.

By the end of the mine life, all major underground openings will be filled with backfill made from cemented plant tailings or waste rock. Openings above the water table will be filled with non-acid generating backfill or rock to minimize the seepage of ground water out of the mine through the various portals and openings.

18.5.1 Waste Rock

There are no known ARD requirements on the site but should there be any, waste rock will be segregated into potentially acid generating or non-acid generating. All potentially acid generating rock, both currently on surface or placed on surface during mining operations, will be stored underground and flooded on mine closure.

Non-acid generating rock will be stored permanently on surface in a dump south of the plant site. Erosion will be limited by shaping the mound with gently sloping sides, and spreading topsoil to facilitate re-vegetation. Silt traps, ditches, and berms will be installed to limit runoff silt escape.

18.5.2 Tailings Impoundment

The tailings impoundment will be shaped at the end of the mine life so as not to have standing water. The dam surface will be covered with the topsoil stored at time of construction.

Outer walls of the dam will be protected against flood erosion by rock armouring placed during construction.

18.5.3 Mine Roads

Mine roads will be broken up, re-contoured, and covered with topsoil. All ditches alongside roads will be filled to approximate original contours to prevent their becoming permanent water courses. All culverts and bridges will be removed.

18.5.4 Closure Costs

Closure costs are excluded from the capital cost estimate in this study, but an allowance of $8.86 M has been included for mine closure costs.

18.6 Execution Plan

18.6.1 Introduction

The Project Execution Plan presents how Minco can successfully complete the Fuwan project. The Project Execution Plan specifies the project approach, tasks, and schedule. As well, it identifies and addresses any unique challenges facing the project. The project will be designed and constructed to industry and regulatory standards, with emphasis on addressing all environmental and safety issues.

Adherence to the Project Execution Plan will ensure timely and cost effective completion while ensuring quality is maintained.

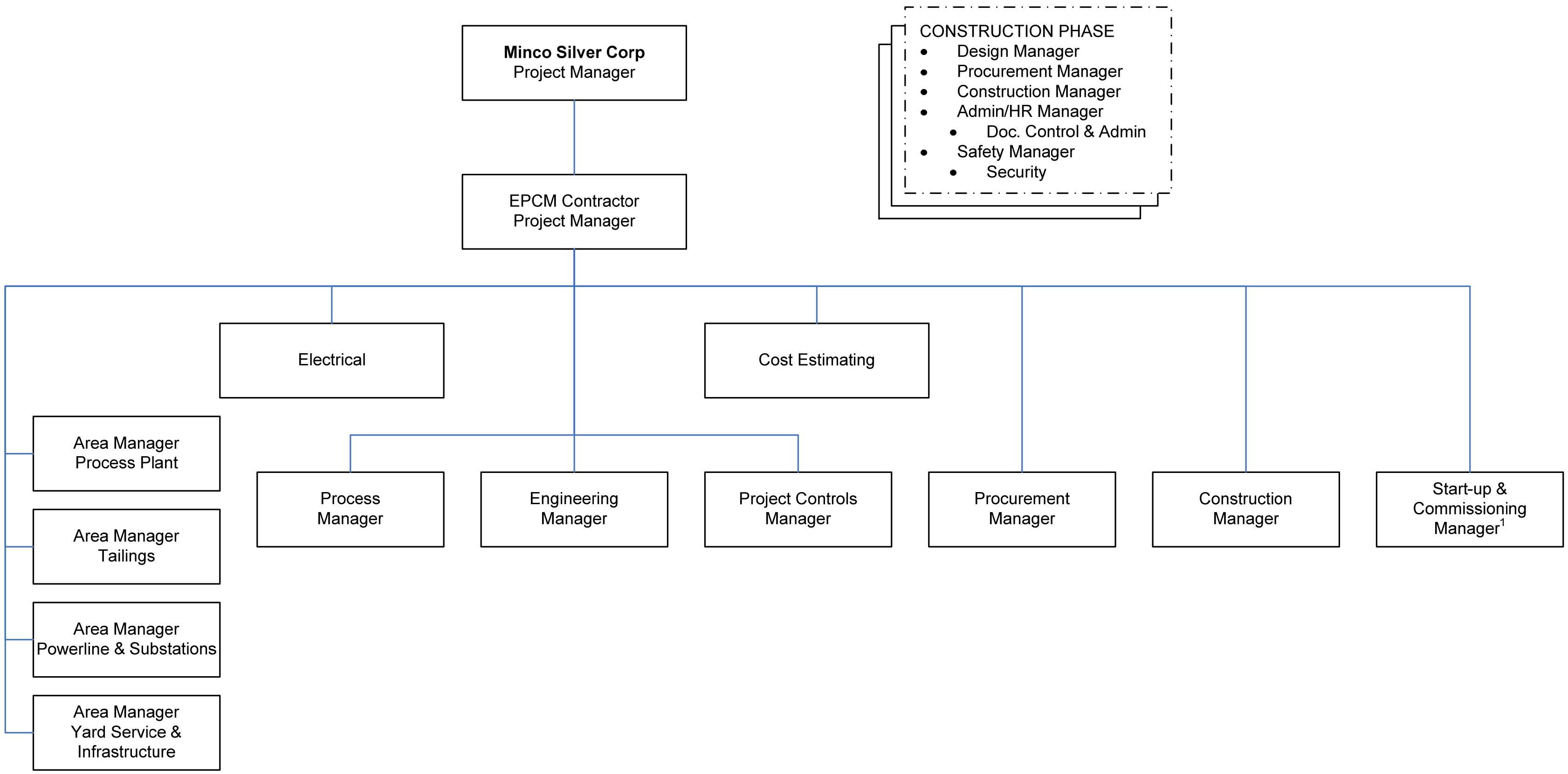
18.6.2 PROJECT APPROACH

To achieve successful project execution, Minco will assemble a Project Management Team (PMT). The PMT will be comprised of personnel with appropriate skills, knowledge, and experience and will act with the support of multi-discipline consultants and contractors. The PMT will, with the support of its consultants and contractors, ensure that checks, balances, progress monitoring, regulatory guidance, and quality assurance/control to provide the information to manage effectively are implemented. The execution philosophy is based on EPCM. The EPCM contractor (Contractor) will be required to implement the following:

- Project Management System
- Engineering Records System
- Procurement System
- Logistics Plan
- Health and Safety Plan
- Construction Managing and Contract Plan
- Quality Assurance/Quality Control System
- Environmental Management Plan
- Labour Relations Plan.

The Project Management organizational chart is shown in Figure 18.40.

Figure 18.40 Project Management Organization Chart



PROJECT MANAGEMENT SYSTEM

A proven and integrated Project Management System (PMS) will be utilized by the Contractor to facilitate monitoring and control of the project. The PMS will provide precise and accurate information to the Contractor and Minco , enabling them to make decisions and implement actions for the successful execution of the project. The PMS will also provide reporting of the status of the project, ensure documentation of scope changes, track the budget and schedule; it will also compare actual performance with planned activities and report the effect of anticipated changes on the final date and cost.

PROJECT CONTROLS PERSONNEL

An integral part of the PMS is the project controls function. The personnel assigned to this function will plan and control the schedule and costs of the project by use of an integrated project control system, which will encompass the functions of scheduling, cost control, estimating, change control, monitoring and reporting for the engineering, procurement, construction, and pre-operational testing of the project.

Project Controls personnel will utilize PMS to perform the following functions:

- Planning and Scheduling
- Cost Control
- Cost Engineering/Estimating.

Planning and Scheduling

The project schedule will set out the project's planning and controlling schedules. At the commencement of the project, the following planning and control activities will be undertaken:

- The Project Master Schedule will be developed as the principal control document.
- The Front End Schedule is essentially a schedule produced early in the project to monitor and accumulate detailed activity status and progress.
- The Detailed Project Schedule will be developed as scope definition and work packages are finalized.
- The Control Level Schedules represent the day-to-day tasks which summarize activities and/or deliverables.

Engineering Cost Monitoring and Control

Budgeted, committed, and actual costs of hours for engineering and procurement activities will be monitored within the project cost control system together with other engineering costs and expenses. Monthly reports will be produced from the detailed schedule, man-hour monitoring and forecasting system, and the project cost control system showing the status of the engineering and procurement phase progress and costs.

Cost Control will include cost monitoring, trending, and forecasting in order to measure performance in relation to project budget and schedule.

The Project Control Budget will be formed on the basis of the approved Feasibility Study estimate. Cost Control personnel will maintain cost trending and forecasting accountability by keeping the originally approved Feasibility Study capital cost estimate and maintaining an audit track of specific decisions managed through scope changes. The Control Budget will include items such as:

- original contract price
- approved changes
- current contract price
- billings this period and to date
- changes submitted but not approved
- forecasted final contract price.

Cost Estimating

Cost Engineering/Estimating includes developing capital cost estimates for the overall project, estimating in support of value engineering, scope change estimates, and fair bid estimates for construction contracts.

TRENDING/CHANGE REQUESTS

Trends and Change Requests may originate from any member of the project team and/or Minco

Typically initial sources are identified from:

- design instructions
- minutes of meetings
- performance analysis
- procurement changes due to vendor data, market prices, and supply demand
- construction changes due to soil reports, weather, labour, equipment, material, and field instructions
- environmental changes due to social, economic, and political forces.

An order of magnitude estimate is prepared and schedule impact is assessed. The trend is then reviewed, and if required, corrective action is initiated. If the corrective action is successful and the trend does not affect cost and schedule, the trend log is updated accordingly. If the corrective action is not successful, a trend report is prepared and other affected task force members are notified. The cost impact is incorporated in the project cost forecast and schedules are updated accordingly.

Trend meetings will occur on a regular basis to review changes and strategy for corrective action. All trends will be expeditiously priced. Routine changes will be estimated within two to five days depending on the complexity and the availability of information. Depending on the magnitude or nature of the impact, a Change Request may be required.

18.6.3 PROJECT EXECUTION SUMMARY

A well-managed plan will be initiated from the date that project execution begins. An effective project management system will be implemented to assist in managing project costs and scheduling. The team will ensure that:

- the critical path schedule of construction is met or improved upon
- engineering and procurement activities are completed to support construction requirements
- costs are monitored, controlled, and reported to Minco on a regular basis.

Within six months from the project go-ahead, the following will be completed:

- award of contract
- project control structure including budget, schedule, procedures, and work plans
- bidders lists
- completed flowsheets and material balances
- Project Procedures Manual (PPM)
- process design frozen
- final site layout
- all design criteria including, but not limited to, environmental, applicable codes, materials of construction, and control philosophy
- process equipment list with Request for Proposal (RFP) packages
- the assignment of package contract numbers
- modularizing, pre-assembly, and purchasing strategies
- finalized contracting strategy

- approved training program
- contracts for early construction activities tendered, received, and evaluated
- Health and Safety Management Plan (HSMP)
- Quality Assurance/Quality Control Plan (QA/QC)
- Environmental Management Plan (EMP)
- Construction Plan
- all project management systems in place
- all geotechnical and site survey data completed.

18.6.4 *ENGINEERING*

The detailed design engineering program will include all disciplines from geotechnical to computerized controls. Each discipline will utilize both recent technological advances and proven techniques as are appropriate for this project.

Once Minco has authorized the project to proceed, the EPCM Contractor will establish the engineering organization and assemble the necessary resources required to meet project demands.

The first step of establishing project standards and procedures melding with those required by Minco and the relevant regulatory bodies has been completed during the Feasibility level design resulting in a set of Design Basis Memoranda. The design basis is based on local requirements and industry guidelines. The design basis addresses all aspects to be considered during the detailed design (e.g. Health & Safety, structural, architectural, environmental, etc.) and specific requirements raised by Minco

Additionally, and in compliance with the Design Basis Memorandum, the Feasibility level design provided PDC, Process Flow Diagrams, Piping and P&IDs, and general arrangements. During the detailed design, the internal, public, and environmental review process may highlight the need for revision of these documents. Such changes will be noted and any alterations or improvements precipitated by this process will be incorporated in the facility design on a continuous basis.

In addition to detailed design drawings, detailed engineering will provide:

- work scope definitions
- installation specifications
- modularization detail where appropriate
- shipping requirements for larger or more delicate items
- heavy lift instructions.

These will be coordinated with the work packages, construction schedule, and logistics schedule. The Construction Manager will undertake constructability reviews throughout the development of detailed designs.

The list of project activities with a budgeted time for each activity and a corresponding list of deliverables (drawings, specifications, data sheets, requisitions, MTOs, BOMs) will be placed into the Engineering Management System for project control purposes.

Additionally, engineers and technical staff will be assigned to the construction program for drawing interpretation, and updating drawings to an "as built" status. The final engineering step will be the cataloguing and entering of all design and procurement information into the Minco central library including computerized drawing and administration files at the site or at the Engineering Contractors' head office .

18.6.5 PROCUREMENT

Procurement of goods and services will adhere to the highest ethical standards and will be performed in a transparent manner. The Procurement group will develop and implement procurement policies that:

- comply with project technical requirements
- comply with the Health, Safety, and Environmental (HSE) policy
- comply with legal and regulatory
- deliver goods and services to satisfy project schedule requirements
- where quality, price, and availability are competitive on a global basis, sourced within Canada.

The Procurement group will prepare procurement procedures and a procurement plan for the execution of the project, including procedures for purchasing, inspection, progress monitoring, material control, expediting, batch crating and packaging, trans shipping, consolidating, and transportation.

CONTRACTS

Contract Form

Minco will use a widely recognized Chinese standard Form of Contract for all tendering including the primary EPCM Contract. Use of such a contract form ensures key aspects of the contract (i.e. arbitration) are not overlooked. As well, Contractors tendering should already be familiar with the Form of Contract so will not require excessive time understanding and assessing the implication of any nuances in a project specific unique contract form.

In support of the standard Form of Contract, project specific Terms of Reference, Scope of Work, and Deliverables will be prepared. These Contract Sections as well as source data will be assembled to comprise a Tender Package.

It is anticipated that the Contract deliverables will include items of each of the following types:

- Lump Sum – fixed amounts for specified works
- Re-measureable – unit prices with re-measureable quantities
- Provisional Sum – fixed amounts for specified works that may or may not be executed.

CONTRACT PACKAGING PLAN

Construction Management (CM) begins with an overall basic EPCM project philosophy. All planning from conceiving project environmental strategy through to the stages of project approval and financing to final design and procurement phases are involved in developing the actual construction program into logical consulting, technical, service, equipment supply and construction contract packages.

LOGISTICS PLAN

The Logistics Plan addresses the need to procure and deliver materials, equipment, and supplies to meet the restricted transportation delivery windows to the site. The team work and implementation of the logistics plan is critical and must be agreed upon by all concerned.

The scope of the logistics plan provides for and encompasses the services necessary for the efficient transport, traffic, warehousing, and marshalling of personnel and all materials and equipment, fuel, and cement required to construct the facilities. The objective of the traffic and logistics plan is to ensure that equipment, materials, and personnel are transported to the project site in a safe, efficient, economical, and timely manner to meet construction schedules. It is imperative that materials and equipment transported during the shipping window arrive at the site without loss or damages and according to the planned window sequences to enable all work to be completed on schedule.

PROCUREMENT AND EXPEDITING

The EPCM contractor's Purchasing Group will provide capital equipment procurement, vendor drawing expediting and, when required, equipment inspection. The procurement department will package the technical and commercial documentation and manage the bidding cycle for equipment and materials to be supplied by Minco to the contractors.

Standard procurement terms and conditions approved for the project will be utilized for all equipment and materials purchase orders. Suppliers will be selected based on locations, quality, price, delivery, and support service.

The CM group will organize bulk materials purchases, assemble contract tendering documents, establish qualified bid lists, issue tenders, analyze and recommend suitably qualified contractors to Minco, and prepare executed contracts for issue.

Logistics

A freight forwarding company will coordinate with manufacturing facilities, establish shipping points and dates, forward the shipments to the most convenient ports, and complete trans-shipments to the project site.

To execute procurement and materials control across the various parties and work break down areas, certain policies, procedures, forms, and coding structures will be standardized (e.g. vendor communication policy, material requisition forms, material takeoffs (MTO), material status reports, document numbering systems, material reference codes, etc.).

Procurement Schedules

The procurement activities of preparing and issuing the RFP bid tabulation and the purchase order deliverable items list will be tracked. Milestones will be developed where the typical activities include:

- Expediting:
  - Expediting ensures a continuous flow of equipment and, materials to the marshalling yards, at the scheduled time and in the proper sequence to facilitate timely transport to site.
- Logistics:
  - The Logistics system function will track material and equipment deliveries to multiple project yards and job site lay down areas and maintain a control over multiple inventories at the various sites.
- Materials Control:
  - Materials control evolves from initial definition and packaging of materials by engineering, through purchasing, expediting, fabrication, delivery, receiving, and use. Site materials management involves multi-warehousing, receipts, issues, returns, inventory management, and inter-warehouse transactions.
  - The system provides status reported on purchasing and expediting along with materials controls database to track equipment and materials from the design stage to final delivery and installation.
- Traffic and Logistics Coordination:

- The Traffic and Logistics (T&L) will consist of support staff, computer systems, communications equipment, leased marshalling yard, and warehousing facilities. Logistics will be the advanced planning for the movement of material from its point of origin to the location where it is required.

18.6.6 CONSTRUCTION

The construction phase is subdivided into three main phases: Construction Management, Field Engineering, and Construction Contracting.

CONSTRUCTION MANAGEMENT

The CM group will be responsible for the management of all field operations. Reporting to Minco, the Construction Manager will plan, organize, and manage construction quality, safety, budget, and schedule objectives. The key CM objectives are:

- Conduct HS&E policy training and enforcement for all site and contractor staff. Site hazard management tools and programs will be employed to achieve the no harm/zero accident objective.
- Apply contracting and construction infrastructure strategies to support the project execution requirements.
- Develop and implement a construction-sensitive and cost-effective master project schedule.
- Establish a project cost control system to ensure effective cost reporting, monitoring, and forecasting as well as schedule reporting and control. A cost trending programme will be instigated whereby the contractor will be responsible for evaluating costs on an on-going basis for comparison to budget and forecasting for the cost report on monthly basis.
- Establish a field contract administration system to effectively manage, control, and coordinate the work performed by the contractors.
- Apply an effective field constructability program, as a continuation of the constructability reviews performed in the design office.
- To develop a detailed field logistics and material control plan to maintain the necessary flow and control of material and equipment to support construction operations.
- Meet the schedule for handover of the constructed plan to the commissioning team.
- Develop a QA/QC plan to set guidelines in terms of plant operability, safety of operation and adherence to all regulatory requirements.

The CM organization chart (Figure 18.41) shows the CM team organization plan for the site.

CONSTRUCTION SCHEDULE

Schedule of Development

The first construction schedule issue will be to identify activities that clearly outline the project logic.

The construction schedule will expand with sub-schedules addressing specific activities and contracts. The construction schedule will typically control all activities. For example, activities in years two and three are essential to the project's timing and limiting commitments; therefore, a sub-schedule will outline the "ramping up" activities in detail. Scheduling will be linked as required to the construction contract packages, which will in turn be required to produce schedules, depending on the nature of the individual contracts. These in turn will be monitored by the EPCM controls staff. A preliminary schedule is presented in Figure 18.42.

Figure 18.41 CM Organization Chart

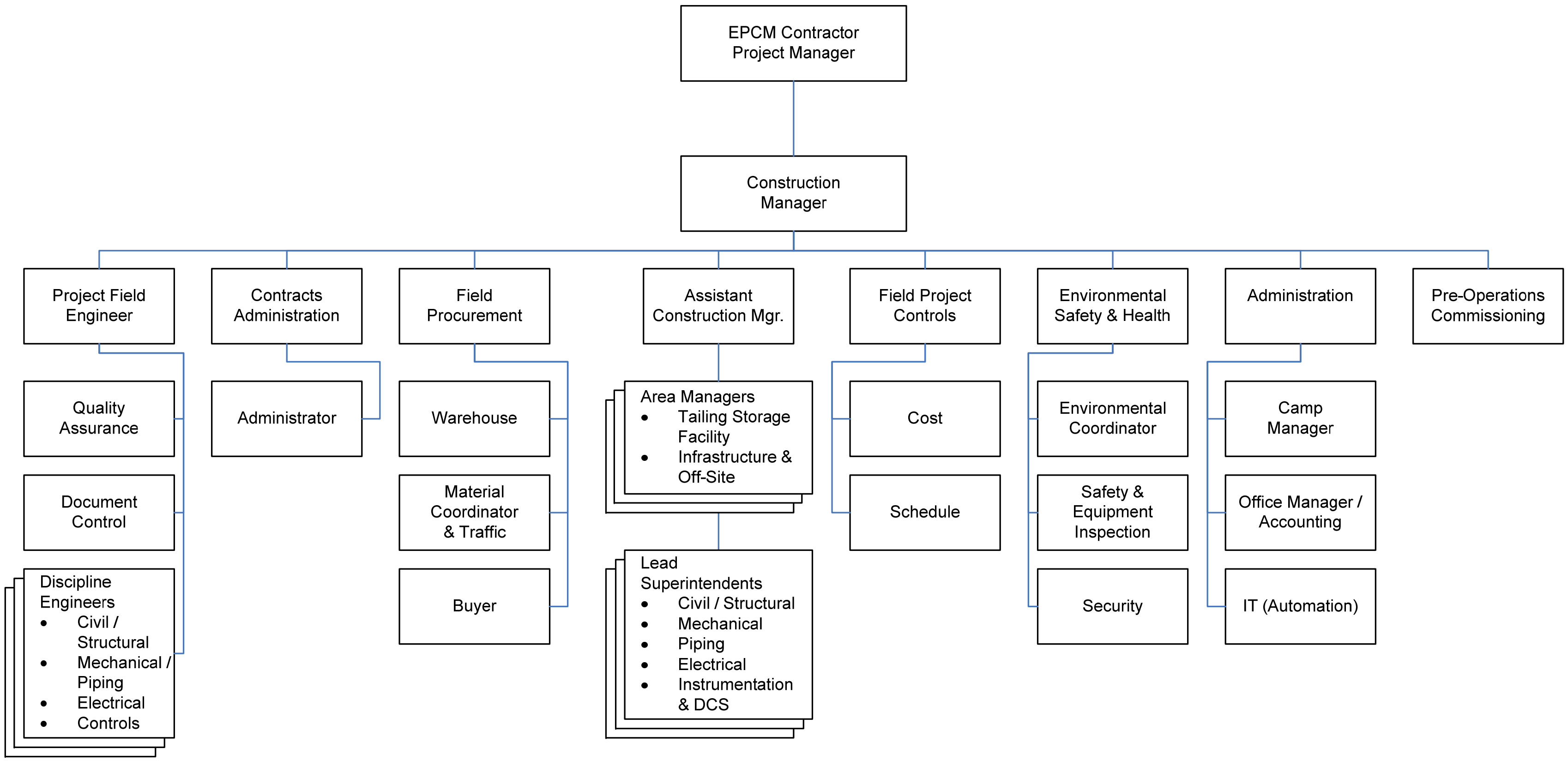
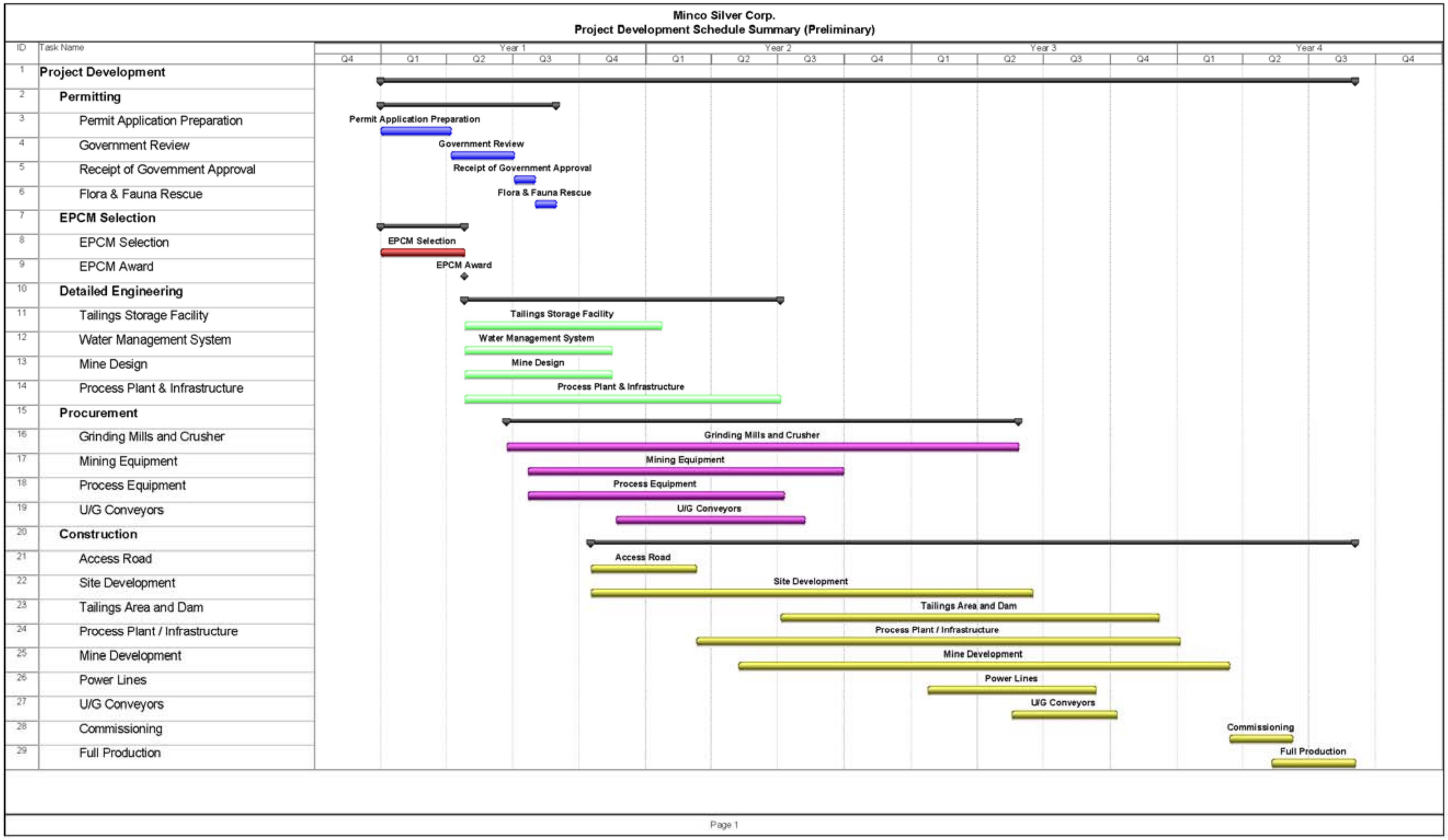


Figure 18.42 Preliminary Project Development Schedule Summary



CONSTRUCTION CONTRACTING

The contracting strategy will be designed to maximize the local labour force, create a responsible and sustainable relationship with the nearby communities, and provide a mix of senior management and specialists to support the safety, quality, schedule, and cost objectives of the project. In addition, contracts will be designed to combine timing, scope, battery limits, and contract value into manageable packages.

Approved contract pro formas will be utilized for construction and service contracts. Construction contractors will be responsible for:

- all construction labour
- all construction equipment
- worker transportation
- site offices and temporary services
- site management
- contractor surveying
- quality control
- contract scheduling
- safety
- environmental safeguarding
- tools and equipment security
- permanent material supplying, as required by contract.

Minco will provide contractors and construction management staff with:

- on-site, project-wide first aid services
- project-wide security
- locations for offices and equipment/material laydown
- local electrical panels and temporary generator sets
- water sources
- diesel fuel, including storage for construction equipment
- sources for all concrete and structural aggregates
- permanent bulk materials and all capital equipment
- quality assurance and control audits
- vendor-representative assistance.

Field Engineering

Surveying

The CM survey crew will verify the accuracy of the existing control system before construction begins. Contractors will use only applicable control data for the project. Additional monuments will be set as needed. The Construction Manager will verify surveys prior to construction. Contractors will supervise day-to-day field surveying, and the CM team will provide spot checks.

Quality Control/Quality Assurance

Contractors will establish and observe their own Quality Control program in accordance with the construction technical specifications and the applicable codes and standards. The CM Field Engineering Team will employ independent CSA-qualified Quality Assurance specialists to ensure quality control.

Materials Management

Warehousing

The Site Materials Management group will receive, inspect, and log all incoming materials, assign storage locations, and maintain a database of all materials received and dispensed to the contractors. On-going reconciliation with the procurement system, including reconciliation to the freight consolidation point, will confirm the receipt of materials and payment of suppliers. An allowance for the lease or purchase of warehousing equipment has been included in the construction budget.

Construction Equipment

Individual contractors will be responsible for the equipment required to meet their contract obligations. All equipment must comply with Mine Safety Standards requirements; Minco CM team will perform regular spot checks. Large cranes may be supplied by a single company, managed by the CM team.

Temporary Facilities and Construction Site Infrastructure

Construction Accommodation

Development of a full-service camp for construction contractors will begin immediately following the receipt of construction permits.

The camp, a modular design, will accommodate the appropriate amount of workers. Accommodations types will suit local construction accommodation conditions.

Transportation, potable water, waste management and other support services will be scaled to support the various development stages. The CM team will ensure that catering contractor meets all facilities, staffing, hygiene, food handling, storage, and meal expectations.

Communication

Minco's systems manager will determine the appropriate telecommunications technologies for the project. Requirements include voice and data link technologies adequate to support growth construction and plant operation growth.

Construction Power

Permanent power will supply all mine equipment and construction power loads for the duration of the construction phase.

First Aid and Site Security

Minco will provide a fully-equipped first-aid facility and ambulance for project-wide use. The facility will normally be staffed 12 h/d, with on-call services ensuring continuous coverage. The first-aid staff will live at the camp. Contractors will be expected to provide basic first-aid stations at the site.

Minco will supply a 24-hour staffed site security program during the initial field mobilization. Access to the site will be controlled at the principal road entrance and will be limited to personnel who have attended induction training, as well as approved visitors.

Warehousing

Construction warehousing will evolve with the project. All freight delivered will be received at a temporary warehouse and stored there or in designated laydown areas.

Initially, fabric or fold-out type structures will be erected to serve as the light and heavy vehicle maintenance shops and general shop/warehouse area for the relevant contractors.

Laydown Areas

Rapid development of the laydown areas at the site is absolutely essential to tie-in with the arriving loads.

All laydown areas will be clearly marked with sign posts and will be laid out in a grid system to eliminate confusion.

Concrete Batch Plant

The concrete batch plant will be managed by the General Contractor as a service to the project and will be operated by the General or Site Services Contractor.

Water Supply and Treatment Plant

The construction project will require fresh water for the following:

- potable drinking water
- truck washing
- concrete batching
- road dust control
- fire water
- building cleaning
- washroom and cleaning purposes.

Sewage Treatment

A portable temporary sewage treatment plant (STP) will be among the first items shipped to the project. It will be a modular system that is very easy and quick to set up.

18.6.7 PRE-OPERATIONAL TESTING AND START-UP

PROCESS PLANT

When construction is complete on any process unit, the construction organization will turn over responsibility to the Commissioning Manager for pre-operational testing and turnover of the facility to Minco prior to introducing ore into the plant for commissioning and start-up.

Minco's operating personnel will be involved in the pre-operational testing phase to the extent that they will progressively accept responsibility for sections of the plant as they are checked and handed over.

Pre-operations testing of equipment will begin once the equipment items have been delivered to site, erected, and tested by the vendor's engineers.

The pre-operational testing phase for the process facilities will include all aspects of dry mechanical and electrical testing of equipment and water testing of process equipment, including pressure testing of pipework and wet pre-operational testing as far as practicable.

The following procedure and tagging system will be adopted in the execution of assignments as work is being completed.

Visual Inspection

Visual inspection is the non-operational examination of an installation to check that it is in accordance with the engineer's and the manufacturer's drawings, specifications, and manuals.

Pre-Operational Test

A pre-operational test is the initial no-load test of a piece of equipment with test media such as water or air where required.

Visual inspection and pre-operational testing (Yellow Tag) will occur upon completion of installation of plant and equipment where the Construction Contractor will submit one copy of the appropriate pre-operational check forms, which will notify that the plant and equipment are ready for inspection and the following have been put into effect and/or completed.

Checkout and Acceptance (Green Tag)

This procedure allows for the transfer of responsibility from the Contractor to Minco. This procedure establishes that the installation of the equipment and ancillaries has been completed in accordance with the Contractors' drawings, specifications, and codes and the equipment has been energized to prove its readiness for the process commissioning and start-up. On acceptance, Minco assumes responsibility for operation and maintenance.

START-UP

Start-up (introduction of ore) is performed under the direction of the Minco Start-up Manager, and involves a select staff from pre-operations, process specialists, and Minco's operating personnel. This will be the beginning of operations under load conditions and the systematic increase in capacity until process through-put and recovery requirements are met and sustained.

18.7 CAPITAL COST ESTIMATE

18.7.1 INTRODUCTION

This estimate has been completed partially by NERIN and partially by Wardrop. The majority of the information used in the estimate is based on the quantities and pricing provided by NERIN to Wardrop on March 28, 2009. NERIN provided additional information and clarifications via email between April 1, 2009 and April 8, 2009. NERIN provide some supporting documents for their estimate, but excluding bulk earth work.

NERIN indicated that its estimate has an accuracy range of ±25%. The estimate has sufficient detail to provide a suitable basis for controlling the EPCM phase of the project.

Table 18.43 provides a summary of capital costs for the Fuwan Project and details are shown Appendix L.

Table 18.43 Summary of Project Capital Costs

Area	Cost (US$)
Direct Works	
A – Mining (Wardrop)	21,636,951
B – Primary Crushing	659,816
C – Crushed Ore Stockpile and Reclaim	305,324
D – Secondary and Tertiary Crushing	51,736
E – Grinding, Flotation, Dewatering, Reagents & Service	9,139,827
F – Tailings Disposal Facilities	4,249,774
G – Plant Site, Infrastructure & Ancillary Facilities	8,626,643
H – Temporary Services	35,323
L – Site/Plant Mobile Equipment	1,190,204
N – Power Lines (Included in G1 – Power Supply)	0
Direct Works Subtotal	**45,895,598**
Indirects	
X – Project Indirects	13,330,282
Y1 – Land Acquisition	2,120,000
Y1 – Owner's Costs	5,663,442
Z – Contingency	6,050,500
Indirects Subtotal	**27,164,224**
TOTAL PROJECT	**US$73,059,822**

18.7.2 *Estimate Organization*

The estimate is assembled and coded with a hierarchical Work Breakdown Structure (WBS) of Area, Section, and Sequence.

X99	99	99.99
Area	Section	Sequence

18.7.3 *Project Areas and Section Codes*

The project areas identified in the estimate are as shown in Table 18.44. The section numbering system is shown in Table 18.45.

Table 18.44 Project Areas

Area Number and Description
A – Mining
A1 – Underground Development
A2 – Underground Mobile Equipment
A3 – Underground Equipment
A4 – Not Used
A5 – Underground Explosive Storage
A6 – Underground Fuel Storage and Delivery
A7 – Underground Backfill
A8 – Underground Dewatering
A9 – Underground Electrical
A10 – Underground Communication
A11 – Underground Safety
A12 – Underground Engineering Equipment
A13 – Mine Labour
A14 – Infill Drilling
B – Primary Crushing
B1 – Crushing Enclosure
B2 – Primary Crusher
B3 – ROM Stockpile
B4 – Conveyor
C – Crushed Ore Bin and Reclaim
C1 – Crushed Ore Bin
C2 – Crushed Ore Reclaim
D – Material Handling
D0 – Transfer Buildings
D1 – Conveyor
E – Grinding, Flotation, Dewatering, Reagents, and Services
E0 – Mill Building
E1 – Grinding and Classification
E2 – Flotation and Regrind
E3 – Concentrate Dewatering and Load-Out
E4 – Reagents
E5 – Tailings Handling
E6 – Process Services
F – Tailings Disposal Facilities
F1 – Tailings Disposal and Reclaim Water
G – Plant Site, Infrastructure, and Ancillary Facilities
G1 – Power Supply
G2 – Power Distribution
G2-1 – Substation
G2-2 – Plant Site Including Ancillary Building

table continues…

Area Number and Description
G3 – Bulk Earthworks
G3-1 – Site Grading and Drainage
G4 – Service and Plant Site Roads
G4-1 – Service Roads
G4-2 – Site Roads
G5 – Process Control System
G6 – Fire Alarm System
G7 – IT and Communication System
G8 – Exterior Lighting
G8-1 – Exterior Lighting – Plant Site and Crushing
G8-2 – Exterior Lighting – Site and Service Road
G9 – Fresh and Fire Water Systems
G9-1 – Supply Wells to Storage Tank
G9-2 – Plant Site Distribution including Tank
G10 – Process Water System
G11 – Potable Water System
G12 – Gland Water System
G13 – Sewage Treatment and Collection
G13-1 – Sewage Treatment Plant – Plant Site
G13-2 – Sewage Collection – Plant Site
G14 – Explosive Area Facilities
G15 – Maintenance/Warehouse Complex
G16 – Administration/Mine Dry
G17 – Assay Laboratory
G18 – Tyre Changing and Truck Wash (Separate Building)
G19 – Warehouse Building
G20 – Fuel Storage and Distribution Facilities
H – Temporary Services
H1 – Construction Camp
H2 – Camp Catering and Housekeeping
H3 – Temporary Laydown Areas
L – Site/Plant Mobile Equipment
L1 – Site/Plant Mobile Equipment
X – Project Indirects
X1 – Construction Indirects
X2 – Spares
X3 – Initial Fills and Warehouse Inventory
X4 – Freight and Logistics
X5 – Commissioning and Startup
X6 – Vendor Assistance

table continues…

Area Number and Description
X7 – EPCM
X7-1 – EPCM
X7-2 – Specialist Consultants
X7-3 – Third Party Engineering
Y – Owner's Costs
Y1 – Land Acquisition
Y2 – Owner's Costs
Z – Contingency (By Disciplines)
Z1 – Contingency

Table 18.45 Section Numbering System

Section No.	Description
Direct Works	
1.1	Tailings Disposal and Reclaim
2	Bulk Earthworks
4	Civil (Detail Excavation and Backfill)
6	Concrete
8	Structural Steel
10	Architectural
11	Platework and Chutework
12	Mechanical and Process Equipment
13	Piping
14	Building Services
17	Instrumentation and Controls
18	Electrical
20	Site Mobile Equipment
40	Mining
42	Mining Mobile Equipment
Indirect Works	
91	Construction Indirects
92	Spares
93	Initial Fills
94	Freight and Logistics
95	Commissioning and Start-up
96	EPCM
98.1	Land Acquisition
98.2	Owner's Costs
99	Contingency

18.7.4 *Sources of Costing Information*

The estimate, unless otherwise stated, is based on the estimate that NERIN submitted to Wardrop on March 28, 2009. Additional cost information provided by NERIN between April 1 and April 8 2009 was included in the estimate accordingly.

The estimate for the mining section (Section A) was provided by Wardrop.

All equipment and material costs are included as free carrier (FCA) or free board marine (FOB) manufacturer plant and exclude spare parts, taxes, duties, freight, and packaging. These latter costs are included in the indirect section of the estimate.

18.7.5 *Quantity Development and Pricing*

All quantities and prices are supplied by NERIN unless otherwise stated.

Wardrop has made additional allowances in the estimate in cases where Wardrop believes that NERIN may not have included an allowance in their estimate or have not met North American code requirements. In areas where Wardrop has made adjustment, these items will be identified with “Wardrop”.

Tailings Disposal and Reclaim

NERIN provided the quantity and cost information for the development of the phase one tailings dam.

Wardrop has made a downward adjustment of 70,000 m^3 to the rockfill excavation quantities, based on the assumption that these quantities will be available from underground mining development. This quantity was estimated by Wardrop.

Bulk Earthworks

NERIN provided the bulk earthwork quantity and cost information. Wardrop made no adjustments to this section.

Civil (Detailed Earthworks)

NERIN provided civil quantities and prices. Wardrop made no adjustments to this section.

Concrete and Structural Steel

NERIN provided all concrete and structural steel quantities and price information.

Projects located close to Hong Kong require special building design considerations to mitigate against the possibility of severe weather conditions such as typhoons.

Taking consideration of the factor and basing on the wind pressure design criteria of up to 4.5 kPa. Wardrop has added the following allowances:

- additional concrete quantities:
  - repair shop (including warehouse): 311 m^3
  - primary crusher building: 536 m^3
  - mill building: 4,382 m^3
- additional structural steel quantities**:**
  - repair shop (including warehouse): 95 t
  - primary crusher building: 4 t
  - mill building: 840 t.

PLATEWORK AND LINERS

NERIN provided quantities and prices for launders, pumpboxes, and chutes. Wardrop has included an allowance for abrasion resistant (AR) liners.

HVAC

The cost for HVAC systems in buildings (priced on a per cubic metre basis) were calculated using in-house data based on building function and site-specific climatic conditions.

Building cooling loads were estimated based upon similar projects in similar climates. Quantities for HVAC equipment (fans, , air conditioning units, etc.) were selected based upon the estimated cooling loads for each building.

DUST COLLECTION

The dust collection equipment is included by NERIN in accordance with process flow sheets. Wardrop did not make any adjustments to the dust collection system.

PIPING

NERIN provided all piping quantities and prices. Wardrop did not make any adjustments to the piping section.

VALVES

NERIN's estimate stated that all major valve quantities and prices are listed but the minor valves are included as part of the piping cost. Wardrop made no additional allowances in this area.

ELECTRICAL

Electrical costs were developed by NERIN. Wardrop made no additional allowances in this area.

INSTRUMENTATION

NERIN provided the instrumentation quantities and costs. Wardrop made no additional allowances in this area.

PLANT MOBILE EQUIPMENT

The requirements for plant mobile equipment were provided by Wardrop, and were based on North American pricing and pro-rated to pricing levels in China based on Wardrop's in-house database.

BUILDINGS

NERIN provided the quantities and prices for the mill building, the administration building, the primary crusher, and the repair shop (including warehouse).

FIRE PROTECTION SYSTEM

NERIN did not include a full fire protection system in their estimate.

In addition to the fire hydrants and fire extinguishers that NERIN provided, Wardrop has included an allowance for fire protection systems for all buildings based on North American standards.

ARCHITECTURAL

NERIN did not include the following items in their estimate:

- perimeter fencing and gate house
- office and conference room furniture.

Wardrop has made allowances for these items.

PROCESS EQUIPMENT

Wardrop has not made any changes to NERIN's equipment pricing, with the exception of pricing for compressors, which were adjusted to match a recent North America quotation and pro-rated to Chinese prices based on Wardrop's in-house database. NERIN's compressor prices were discarded.

The major process equipment prices were quoted in Q1 2009 from local Chinese major manufacturers.

UNDERGROUND MINING AND MINING MOBILE EQUIPMENT

Sources of Information for Mining

The mine capital cost estimate is based on the following:

- Development costs of ramps and tunnels in waste included in the preproduction capital cost are assumed to be undertaken by Chinese contractor and, are based on a quotation obtained from Jinchengxin Mining Construction Co. of Beijing. The costs provided in the estimate also include additional Wardrop-estimated costs for equipping and haulage that were not included in the original quotation.
- Mining equipment costs for jumbo drills were obtained from Sandvik Engineering Group's Beijing office, and included allowances for duty and VAT.
- Mining equipment costs for trucks and LHDs were obtained from the Jinxi Mining Equipment Company, China.
- Quotes for mining equipment costs for service vehicles were not readily available; Wardrop used 50% of North American costs based on North American equipment items comparable to Chinese equipment on the internet.
- Wardrop uses some estimate data based on the in-house database.
- Quantities of lateral and vertical development based on a resource model by P&E and 3D wireframes of proposed mine workings on a 3D mine model constructed by Wardrop.
- Consumable prices were provided by NERIN.

Mining capital was divided into equipment capital cost and mine development cost categories. It also splits on initial/pre-production capital and sustaining capital.

The initial mining equipment capital cost estimate includes the purchase of permanent mining equipment required for production, assuming that two-thirds of major equipment costs (drilling and hauling) will be paid up front and the rest will be paid in Year 1 of production. Sustaining capital includes the purchase of replacement mobile and stationary equipment to support production during the LOM.

The pre-production development capital cost includes contractor mobilization to the site and waste development prior to production. The total pre-production mining capital cost is $21.6 M as indicated in Table 18.46.

Table 18.46 Mine Pre-production Capital Cost Summary (US$)

	Year -2	Year -1	Total
Development	2,873,500	7,258,500	10,132,000
Mobile Equipment	2,947,417	6,630517	9,577,934
Ventilation		410,000	410,000
Electrical Supply		140,000	140,000
Dewatering Pumps		328,750	328,750
U/G Fuel Storage & Delivery		89,697	89,697
U/G Electrical		200,000	200,000
Communication		150,000	150,000
Safety		358,570	358,570
U/G Engineering Equipment		250,000	250,000
Total	**5,822,064**	**15,817,629**	**21,636,951**

The waste development cost during the production period is included in the sustaining capital cost as ongoing capital access development cost. It is assumed that ongoing waste development will continue to be performed by a contractor. Sustaining capital cost of development includes contractor demobilization. Ore development is included in the mining operating cost.

The sustaining total mining capital cost is $37.4 M, as indicated in Table 18.47. The total mine life mining capital cost is $59 M.

Table 18.47 Mine Sustaining Capital Cost Summary (US$)

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Development	2,612,400	2,763,900	1,979,900	1,615,000	1,768,000	1,392,000	367,200	618,800	712,000	13,829,200
Mobile Equipment	4,460,616	1,719,322	650,320	1,147,059	5,757,345	3,340,688	514,706	-	-	17,590,055
Ventilation	230,000	-	-	-	$250,000	$150,000				630,000
Electrical Supply	70,000	-	-	-	-	-	-	-	-	70,000
Dewatering Pumps	60,000	228,750	-	228,750	100,000	60,000	-	-	-	677,500
Underground Electrical	431,250	200,000	431,250	200,000	-	-	-	-	-	1,262,500
Safety	2,335	90,000	-	54,900						147,235
U/G Engineering Equipment	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	-	240,000
U/G Mechanical Shop	680,000									680,000
Infill Drilling	750,000	750,000	750,000							2,250,000
Total	**9,326,601**	**5,781,972**	**3,841,470**	**3,275,709**	**7,905,345**	**4,972,688**	**911,906**	**648,800**	**712,000**	**37,376,490**

Mine Labour Rate

Underground construction labour by contractor was included in the rate per metre from the budget quote.

Mine Contingency

The contingency allowance on total mine capital is based on 12% of the mining cost and 5% of the mobile mining equipment cost.

Mine Owner's Costs

An allowance of US$1,652,488 has been included for Owner's costs in Year -1 (pre-production), which covers the following underground items:

- Owners underground mine labour build-up
- safety
- training
- mine services
- technical services
- equipment maintenance
- energy (electrical power, propane).

18.7.6 BASE DATE AND CURRENCY

The estimate is dated 2nd quarter 2009 and the base currency is US dollars. There is no escalation included beyond this period. Foreign exchange rates used in the estimate are as noted in Table 18.48.

Table 18.48 Foreign Exchange Rates

Base Currency	Foreign Currency
US$1.00	¥6.82

Wardrop has not made any allowance for fluctuations in foreign currency exchange.

18.7.7 LABOUR RATES

NERIN provided labour rates for the capital cost estimate. As stated by NERIN, these rates are based on 8 h/d and 5 d/wk. NERIN did not provide a detailed breakdown of this labour rate calculation.

The labour rates used by NERIN are US$1.47/h (¥10/h) for unskilled labour and US$1.61/h (¥11/h) for skilled labour. As the rates are lower than the official government published labour rates. Wardrop and Minco have agreed to add US$2.09/manhour to the estimate as part of the construction indirects (detailed in the section below).

Labour Productivity

NERIN has verbally confirmed that labour productivity factors are included in their estimate.

18.7.8 *Project Indirects*

Construction Indirects

Wardrop made an allowance of 10% (based on North American standards) for direct costs (excluding mining) to include the following:

- mobilization and demobilization
- site facilities
- temporary power and lighting
- temporary water
- miscellaneous equipment rental
- garbage disposal
- safety costs
- security
- medical and first aid
- surveying
- final project clean-up.

A 1% of direct cost allowance (excluding mining and plant mobile equipment) was added to the Construction Indirects section for costs associated with third-party government inspections.

As the labour rates provided by NERIN are lower than the official government published labour rates, it is Wardrop's opinion that NERIN's labour rates of US$1.47/h and US$1.61/h have not taken into account the following items:

- overtime
- vacation
- statutory holidays

- medical
- year-end bonus
- additional living allowance
- extra food allowance
- small tools
- consumables
- supervision.

Wardrop and Minco have agreed to add US$2.09/manhour to the estimate as part of the construction indirects.

Freight and Logistics

Freight and logistics costs are expressed as a percentage of the sum of material costs and process equipment costs. NERIN has set the freight and logistics cost at 7% of material and process equipment costs.

Commissioning and Startup

Wardrop has included an allowance of 240 man-days, assuming four expatriates at US$1,500/d during a 2-month commissioning period.

Vendor Assistance

Similarly, Wardrop has included an allowance of 120 man-days, assuming four expatriates at US$1,500/d during construction for one month.

EPCM

Wardrop has included:

- an allowance of US$1.5 M for detailed engineering (excluding mining) by a local engineering firm from China
- a procurement allowance of US$396,000 based on 120 packages at 60 hours at US$55/h
- a plant site construction management allowance, estimated to be US$1.6 M (8% of direct costs)
- a tailing construction management allowance of US$212,500
- an allowance of $603,000, for engineering for mining

THIRD PARTY ENGINEERING

Wardrop has included an allowance of US$150,000 for a peer review of tailings work only.

18.7.9 OWNER'S COSTS

Minco has provided Wardrop with an Owner's Cost allowance of US$7,783,442 that includes:

- staffing
- accommodation
- communication
- legal costs (including land acquisition fees)
- travel
- human resources
- technical
- information technology
- field general expenses
- incidentals.

18.7.10 CONTINGENCY

The contingency was calculated by applying various contingency percentages to the various sections of work (Table 18.49).

Table 18.49 Allowances for Contingencies

Section	Description	Percentage
1.1	Tailings Disposal and Reclaim	10
2	Bulk Earthworks	10
4	Civil	15
6	Concrete	12
8	Structural Steel	12
10	Architectural	12
11	Platework and Chutework	15
12	Mechanical	8
13	Piping	15
14	Building Services	12
17	Instrumentation	10
18	Electrical	15

table continues…

Section	Description	Percentage
20	Site Mobile Equipment	8
40	Mining	12
42	Mining Mobile Equipment	5
91	Construction Indirects	10
92	Spares	nil
93	Initial Fills	nil
94	Freight and Logistics	nil
95	Commissioning and Start-up	5
96	EPCM	10
98	Land Acquisition	10
98	Owner's Costs	5

18.7.11 EXCLUSIONS

The following items were excluded from the capital cost estimate:

- working or deferred capital
- financing costs
- refundable taxes and duties
- currency fluctuations
- lost time due to severe weather conditions
- lost time due to force majeure
- additional costs for accelerated or decelerated deliveries of equipment, materials and services resultant from a change in project schedule
- warehouse inventories other than those supplied in initial fills
- any project sunk costs including this study
- escalation post (Q2 2009)
- community relations.

18.8 OPERATING COST ESTIMATE

The operating cost estimates are based on a process rate of 990,000 t of ore annually or 3,000 t/d of ore. All operating costs are shown in US$, unless otherwise specified. The exchange rate for US and Canadian dollars to Chinese currency (¥) is US$1.00 = ¥6.82 = Cdn$0.82.

18.8.1 *Process Operating Costs*

Summary

On average, the annual process operating cost is estimated to be approximately $9.80 M or $9.90/t milled. The estimated process operating costs are summarized in Table 18.50 and include the following:

- personnel requirements including supervision, operation, and maintenance; salary/wage levels are provided by Minco and the rates are comparable to rates of similar operations in China
- liner and grinding media consumptions, estimated from the Bond ball mill work index and abrasion index equations and the quoted budget prices in Q1 2009
- maintenance supply costs, based on approximately 6% of major equipment capital costs
- reagent costs, based on the estimated consumption from the test results and the quoted budget prices in Q1 2009 or Wardrop's database
- other operation consumables including the laboratory, filtering cloth, service vehicles consumables
- power consumption is based on the power draw and power unit cost which is provided by Minco is ¥0.69/kWh or $0.101/kWh.

Table 18.50 Summary of Process Operating Costs

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Ore)
Labour			
Operating Staff	10	354,240	0.358
Operating Labour	46	427,680	0.432
Maintenance	46	469,440	0.474
Metallurgical Laboratory	3	38,160	0.039
Assay Laboratory	13	131,760	0.133
Sub-total Labour	**118**	**1,421,280**	**1.436**
Major Consumables			
Metal Consumables		2,347,140	2.371
Reagent Consumables		1,224,780	1.237
Supplies			
Maintenance Supplies		597,000	0.603
Operating Supplies		125,000	0.126
Power Supply		4,085,866	4.127
Sub-total Supplies		**8,379,787**	**8.464**
Total Process	**118**	**9,801,067**	**9.900**

PERSONNEL

The projected personnel requirements are 118 persons including 28 staff for management and professional services, 52 operators for operating including laboratories for quality control, process optimization and assaying, and 38 personnel for maintenance. Salary/wage levels are provided by Minco including base salary, holiday and vacation pay, pension plan, various benefits, and tool allowance costs. The rates are comparable to the rates of similar operations in China.

The total personnel cost is estimated to be $1.44/t milled. Detailed personnel descriptions and costs are shown in Table 18.51.

Table 18.51 Labour Complement and Annual Cost (US$)

Description	Personnel	Base Salary	Loaded Salary	Total Annual Cost
Staff				
Mill Superintendent	1	125,000	180,000	180,000
Mill Superintendent Assistant	1	11,500	16,560	16,560
Senior Metallurgist	1	18,000	25,920	25,920
Plant Metallurgist	1	11,500	16,560	16,560
General Foremen	1	18,000	25,920	25,920
Shift Foreman	4	14,500	20,880	83,520
Mill Clerk	1	4,000	5,760	5,760
Maintenance Superintendent	1	18,000	25,920	25,920
Maintenance Planner	1	14,500	20,880	20,880
Maintenance Foremen	4	14,500	20,880	83,520
Electrical Superintendent	1	14,500	20,880	20,880
Maintenance Clerk	1	4,000	5,760	5,760
Lab Metallurgist	1	11,500	16,560	16,560
Lab Technicians	2	7,500	10,800	21,600
Chief Chemist	1	14,500	20,880	20,880
Assay Technicians - Senior	2	11,500	16,560	33,120
Assay Technicians - Junior	4	7,500	10,800	43,200
Sub-total Staff	**28**			**646,560**
Hourly				
Crusher Operators	4	6,500	9,360	37,440
Control Room Operators	8	8,000	11,520	92,160
Grinding Operators	8	7,500	10,800	86,400
Flotation Operators	8	7,500	10,800	86,400
Dewatering Operators	4	6,500	9,360	37,440
Reagent Preparation	2	6,500	9,360	18,720
Operation General Labours	8	4,000	5,760	46,080
Operation Day Crew	4	4,000	5,760	23,040
Sample Preparation Labour	6	4,000	5,760	34,560

table continues…

Description	Personnel	Base Salary	Loaded Salary	Total Annual Cost
Mechanics	8	7,500	10,800	86,400
Electrician	4	7,500	10,800	43,200
Electrician Apprentices	4	4,000	5,760	23,040
Instrument Technicians	4	7,500	10,800	43,200
Welders	2	7,500	10,800	21,600
Welder Apprentices	4	4,000	5,760	23,040
Crane / Equipment Operators	4	7,500	10,800	43,200
Maintenance Helpers	8	2,500	3,600	28,800
Sub-total Hourly	**90**			**774,720**
TOTAL PERSONNEL COST				**US$1,421,280**

Assay laboratory staff includes assay and sample preparation personnel to support the assaying requirements of all mine site operations including those from mining, geological, and exploration activities.

POWER COST

The average annual power consumption is estimated to be 40,385 MWh/a, or 40.7 kWh/t of ore processed. The annual power cost is estimated to be $4.09 M, or $4.13/t of ore milled at the average power unit cost of $0.101/kWh. Major equipment power consumption is estimated based on:

- primary crusher: equipment power draw and operating time
- SAG mill and ball mills: the specific energy requirements are calculated from the Bond work index equation and SAG mill inefficiency factor using an average Bond work index; the particle size of SAG mill discharge is 80% passing 1,200 µm and the primary grind size is 80% passing 100 µm
- regrind mills: the specific energy requirement is calculated from the Bond work index equation using an average Bond work index; the silver-lead rougher concentrate regrind size is 80% passing 35 µm
- major slurry pumps: the power consumption estimate is based on flow rates, total dynamic heads, pump efficiencies and operating time
- other major equipment: the power consumption estimate is based on the estimated equipment power draw and operating time.

MAJOR CONSUMABLE COST

Major consumable costs are estimated to be $3.61/t milled. The major consumables include metal and reagents consumables. The costs are summarized in Table 18.52.

Table 18.52 Major Consumable Cost Estimate

Consumables	Consumption (kg/t ore)	Unit Cost (US$/kg)	Total Cost (US$/a)	Unit Cost (US$/t Ore)
Metal Consumables				
Liners				
Primary Crusher Liners	-	-	15,000	0.015
SAG Mill Liners	0.091	1.278	115,159	0.116
Primary Ball Mill Liners	0.094	1.278	118,951	0.120
Regrind Mill Agitator Liners	0.011	1.278	14,065	0.014
Pebble Crusher Liners	-	-	0	0.000
Balls				
SAG Mill Balls	0.749	1.001	742,290	0.750
Ball Mill Balls	1.043	1.001	1,033,136	1.044
Regrind Mill Balls	0.283	1.101	308,538	0.312
Sub-total Metal Consumables			**2,347,140**	**2.371**
Reagent Consumables				
Lime	1.000	0.039	38,497	0.039
ADDP (dithiophosphate)	0.068	2.906	195,662	0.198
SDTC (dithiocarbamate)	0.064	1.399	88,630	0.090
SPIX (xanthate)	0.072	1.243	88,630	0.090
$ZnSO_4$	0.800	0.777	615,484	0.622
$CuSO_4$	0.075	2.046	151,948	0.153
Pine Oil (#2 Oil)	0.028	1.593	44,161	0.045
Flocculant	0.001	1.787	1,770	0.002
Sub-Total Reagents			**1,224,780**	**1.237**
TOTAL CONSUMABLES			**3,571,921**	**3.608**

The liner and grinding media consumptions are estimated from the Bond abrasion index equation and the budget prices from local supplies in Q1 2009. A grinding media quality factor of 0.85 for grinding balls is applied to the estimate to reflect improved steel quality.

Reagent consumptions are estimated from laboratory locked cycle test results and comparable operations. The reagent costs are from the budget prices in Q1 2009 from local potential suppliers.

SUPPLY COST

The maintenance supplies are estimated at $0.73/t milled. Table 18.53 shows the details of the estimate.

Table 18.53 Operating and Maintenance Supply Cost Estimate

Area	Total Cost (US$/a)	Unit Cost (US$/t Ore)
Maintenance Supplies		
Crushing	18,000	0.018
Grinding + Pebble Crusher + Reclaim	360,000	0.364
Flotation	144,000	0.145
Dewater	15,000	0.015
Reagents	10,000	0.010
Assaying	10,000	0.010
Miscellaneous Mill Supplies	20,000	0.020
Misc. Building Complex Supplies	20,000	0.020
Sub-Total Maintenance Supplies	**597,000**	**0.603**
Operating Supplies		
Concentrate Filter Cloth	10,000	0.010
Assay Laboratory Supplies	83,490	0.084
Mill Light Vehicle and Rental Vehicle Operations	11,510	0.012
Miscellaneous	20,000	0.020
Sub-Total Operating Supplies	**125,000**	**0.126**
TOTAL SUPPLIES	**722,000**	**0.729**

Maintenance supplies are estimated based on comparable operations or approximately 6% of major equipment capital costs.

Operating supplies outside of major consumables are also included in the estimate. Major items in this cost include filter cloths, laboratory supplies, and light mill vehicle operations.

18.8.2 MINING OPERATING COSTS

The mining operating cost required for a 3,000 t/d operating mine was estimated from the first principles for each cost category such as development, production, haulage, maintenance, mine services, and labour.

Table 18.54 shows the input data that were assumed in this study.

Table 18.54 Operating Cost Input Data

Operating Factors	Unit	Quantity
Underground Production		
Mine Days	d/a	330
Nominal Mining Rate	t/d	3,000
Average Mining Rate	t/a	990,000
Rock Characteristics		
In Situ Density Ore	t/m^3	2.48
In Situ Density Waste	t/m^3	2.48
Swell Factor	%	50
Loose Density Ore	t/m^3	1.65
Loose Density Waste	t/m^3	1.65
Shift Data		
Working Days per Week	ea.	7
Shifts per Day	ea.	3
Shift Length	h	8
Shift Change	h	1.0
Lunch Break	h	0.5
Equip Inspection	h	0.5
Subtotal Non-productive	h	2.0
Usable Time per Shift	h	6.0
Shift Efficiency	%	75
Usable Minutes per Hour	min	50
Hour Efficiency (50 min/h)	%	83
Effective Work Time per Shift	h	5.0

Productivities, equipment operating hours, labour, and supply requirements were estimated for each type of underground operation.

Total hourly labour requirements were estimated to achieve the daily mining production rate based on 3 shifts at 8 h/d with 4 crews.

Supply costs were estimated for drill and steel supplies, explosives, ground support, and services supplies, and were included in development, stoping, and services costs. The costs were based on recent local supplier’s prices.

Diesel fuel cost with freight was assumed at ¥6.2/L ($0.91/L). The power cost was assumed at ¥0.69/kWh ($0.10/kWh).

The stope production cost includes drilling, blasting, and mucking the ore from the stopes to a truck loading point as well as the cost of ground support.

The summary of stope mining operating cost without labour is shown Table 18.55.

Table 18.55 Stope Operating Cost by Mining Method (US$/t)

Height	Primaries		Secondaries	
	Good Ground (6 m W)	Bad Ground (4 m W)	Good Ground (6 m W)	Bad Ground (4 m W)
Drift-and-Fill Summary				
2.5 m	13.82	19.17	11.52	16.88
3 m	13.00	17.50	10.70	15.20
3.5 m	12.18	16.08	9.88	13.78
4 m	11.58	15.01	9.28	12.71
5 m	10.87	13.68	8.57	11.38
6 m	10.40	12.36	8.10	10.07
Cut-and-Fill Summary				
3 m H by 3 m W	10.72	14.02		
3 m H by 5 m W	9.65	12.00		
Pre-production Summary				
3 m H by 6 m W	10.89			

The haulage cost includes ore truck haulage on the ramp and was based on average haulage distances. The waste haulage from access development to surface was included in the pre-production and sustaining development capital costs.

The mine services cost includes ventilation, compressed air, transportation of personnel and materials, material handling, mine and road maintenance, mine dewatering, and underground construction. It also includes exploration, mine safety, rescue, and training.

Labour costs were estimated based on the labour list and rates. The total labour cost consists of the base salary and benefits, which includes a 10% bonus, 25% employment insurance, as well as allowances for meals, travel, etc.

The mine operating cost summary and salary structure details are available in Table 18.56 and Table 18.57, respectively. Details on labour operating costs and personnel requirements are available in Appendix M.

Table 18.56 Mine Operating Cost Summary – LOM

	Cost
Total Mine Operating Cost	**$164,234,279**
Average per Tonne	$18.01/t
Labour Cost	**$38,124,300**
Average per Tonne	$4.18/t
Mining Cost without Labour	**$126,109,979**
Average per Tonne	$13.83/t

Table 18.57 Salary Structure Details (¥, Chinese Currency)

Personnel Description	Base	Benefits	Total Monthly Cost
Manager	15,000	6,695	21,695
Chief Security Officer	12,000	5,645	17,645
Personal Assistant	12,000	5,645	17,645
Project Accountant	10,000	4,945	14,945
Senior Mine Engineer	10,000	4,945	14,945
Senior Geologist	10,000	4,945	14,945
Mine Foreman	10,000	4,945	14,945
Shift Supervisor	8,000	4,245	12,245
Mine Engineer	6,000	3,545	9,545
Geologist	6,000	3,545	9,545
Mechanic	4,000	2,845	6,845
Electrician	4,000	2,845	6,845
Welder	4,000	2,845	6,845
Mason	4,000	2,845	6,845
Heavy Equipment Operator	4,000	2,845	6,845
Junior Geologist	4,000	2,845	6,845
Miner	4,000	2,845	6,845
Mine Draftsman	2,000	2,145	4,145
Assistant	2,000	2,145	4,145
Clerk	2,000	2,145	4,145

* Note: Source = Minco

18.8.3 TAILINGS OPERATING COSTS

The average tailings operating cost is estimated to be $1.13/t milled and is detailed in Table 18.58. The cost includes tailing surface storage operation and backfill station operation. The costs for operating labours, cement consumption, power consumption, maintenance, and operating supplies and light vehicle operations have been considered in the estimate. The cement cost is based on approximately 3% cement addition into the backfill tailings on average as well as the budgeted price from the local cement manufacturers in Q1 2009. The power consumption is estimated from equipment power draw and operating time.

Table 18.58 Tailings Operating Cost Estimate

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Ore)
Labour			
Operating Staff*			
Operating Labour	20	112,320	0.113
Maintenance*			

table continues...

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Ore)
Sub-total Labour	**20**	**112,320**	**0.113**
Supplies			
Major Consumables			
Reagent Consumables		544,355	0.550
Others		20,000	0.020
Supplies			
Maintenance Supplies		100,000	0.101
Operating Supplies		43,000	0.043
Power Supply		298,844	0.302
Sub-total Supplies		**1,006,199**	**1.016**
TOTAL TAILINGS	**20**	**1,118,519**	**1.130**

* included in flotation plant

18.8.4 G&A/Surface Services Operating Costs

G&A costs are the costs that do not relate directly to the mining or processing operating. The G&A costs are estimated at approximately $4.73 M/a or $4.78/t milled. The G&A cost estimate is shown in Table 18.59.

The estimated personnel requirement is 61 persons, including supervision and services. The costs, developed by Wardrop and Minco, include:

- salaries for administrative personnel
- expenses and services related to general administration, human resources, and safety and security
- office overheads
- allowances for resources taxes and land rental/lease, excluding the fee for resource compensation
- sustainability including environment, community liaison, and engineering consulting.

Table 18.59 G&A Cost Estimate

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Ore)
Wage and Salary			
Mine Manager	1	211,680	0.214
Secretary	1	16,560	0.017
Accounting/Sales	8	115,920	0.117
IT and Communication	2	21,600	0.022
Procurement and Warehouse	11	100,800	0.102

table continues...

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Ore)
Human Resources	6	66,240	0.067
Community Relations	2	21,600	0.022
Safety/Security	12	148,320	0.150
Environment	4	54,720	0.055
General Services	14	73,440	0.074
Sub-total Wage and Salary	**61**	**830,880**	**0.839**
Expenses and Services			
Administration – General		1,179,846	1.192
Safety/Security		1,021,129	1.031
Resource Tax/Fee		1,451,613	1.466
Land Rental/Lease		164,956	0.167
Environmental		82,000	0.083
Sub-total Expensive and Service		**3,899,544**	**3.939**
Total G&A	**61**	**4,730,424**	**4.778**

The site service cost is estimated at $0.60/t milled or about $594,000/a. The details for the estimate are shown in Table 18.60. The cost includes:

- labour costs for surface service personnel
- surface mobile equipment and light vehicle operations
- water and waste management
- general maintenance including yards, roads, fences, and building maintenance
- building services including air conditioning.

Table 18.60 Surface Service Cost Estimate

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Ore)
Wage and Salary	13	125,280	0.127
Equipment and Vehicles		13,800	0.014
Water and Waste Management		254,728	0.257
Access Road		24,000	0.024
Yards, Roads and Fences		20,000	0.020
Building and Maintenance		156,290	0.158
Total Surface Services	13	594,098	0.600

18.9 ECONOMIC ANALYSIS

18.9.1 *INTRODUCTION*

An economic evaluation of the Fuwan Project was prepared by Wardrop based on a pre-tax financial model. For the 9-year mine life and 9.1 Mt reserve, the following pre-tax financial parameters were calculated:

- 33.2% IRR
- 2.3 years payback on $73.1 M capital
- US$111.5 M NPV at a 6% discount value.

The base case prices were as follows:

- Silver – US$13.57/oz
- Gold – US$767.72/oz
- Zinc – US$1.18/lb
- Lead – US$0.91/oz.

Sensitivity analyses were carried out to evaluate the project economics for 2-year historical average metal prices (upside case) and the Energy and Metals Consensus Forecast (EMCF) prices published by Consensus Economics Inc. (downside case).

18.9.2 *PRE-TAX MODEL*

FINANCIAL EVALUATIONS

The production schedule, based on a reserve with an NSR cut-off grade of approximately US$37/t, has been incorporated into the pre-tax financial model to develop annual recovered metal production. Market prices for silver, gold, zinc, and lead have been adjusted to realized price levels by applying smelting, refining, and concentrate transportation charges from the mine site to the smelter in order to determine the NSR contributions for each metal.

Unit operating costs for mining, process, and G&A and surface service were applied to annual milled tonnages to determine the overall mine site operating cost which has been deducted from NSRs to derive annual net revenues.

Initial and sustaining capital costs have been incorporated on a year-by-year basis over the mine life and deducted from the net revenue to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of concentrate; sustaining capital includes expenditures for mining and milling additions, replacement of equipment, and tailing embankment construction.

Working capital has been calculated on the basis of three months mine site operating costs and applied to the first year of expenditures. It will be recovered at the end of the mine life and aggregated with the salvage value contribution and applied towards reclamation during closure.

METAL PRICE SCENARIOS

The financial outcomes for three metal price scenarios have been tabulated for NPV, internal rate of return (IRR), and payback of capital. Discount rates of 10%, 8%, and 6% were applied to all metal price scenarios. The results are presented in Table 18.61.

Table 18.61 Summary of Pre-tax NPV, IRR, and Payback by Metal Price Scenario

	NPV at Selected Discount Rates (Million US$)				
Scenario	**10%**	**8%**	**6%**	**IRR (%)**	**Payback (Years)**
EMCF (Downside)	-12.0	-7.4	-2.1	5.3	5.7
3-year Average (Base Case)	81.3	95.3	111.5	33.2	2.3
2-year Average (Upside)	84.7	99.0	115.5	34.1	2.3

Wardrop adopted the 3-year historical average metal prices from the LME for the base case. The 3-year average is an approach which is considered an industry standard and is in line with the SEC requirements of the United States of America. The backward averaging of historical prices was calculated as of May 31, 2009 and is summarized in Table 18.62.

Table 18.62 Summary of Pre-tax Metal Price Scenarios

Scenario	Silver (US$/oz)	Gold (US$/oz)	Zinc (US$/lb)	Lead (US$/lb)
EMCF	10.59	681.00	0.80	0.56
3 Year Average (Base Case)	13.57	767.72	1.18	0.91
2 Year Average	14.09	835.25	1.00	0.95

SMELTER TERMS

In the absence of letters of interest or letters of intent from potential smelters or buyers of concentrate, the data collected by preliminary phone quotation or smelter's visits were used to benchmark the terms supplied by Minco. The estimated payment terms will generally include as follows:

- Lead Concentrate:
  - Silver – pay variable % of silver content (see Table 18.63)
  - Gold – pay 70% of gold content
  - Lead – pay 20% of lead content
  - Zinc – pay 0% of zinc content
  - Impurities – there are no penalties due to impurities.
- Zinc Concentrate:
  - Silver – pay 40% silver content
  - Gold – pay 0% of gold content
  - Zinc – pay 64% of zinc content (base case only – variable depending on metal price)
  - Lead – pay 0% of lead content
  - Impurities –no detailed penalty terms for impurities have been collected from the local smelters. No penalty has been applied in the estimate. However, antimony (Sb) and arsenic (As) in the silver-lead concentrate may receive penalties from some of smelters.

Table 18.63 Payable Silver in Lead Concentrate

Concentrate Grade (g/t)	Payable Silver (%)
3,000	83.0
5,000	85.0
7,000	87.0
8,500	87.5
10,000	88.0
15,000	89.0
17,500	89.5
20,000	90.0

CONCENTRATE TRANSPORT LOGISTICS

Concentrate will be truck and railway transported from the mine site at a charge of US$18.00/wmt.

Concentrate Transport Insurance

An insurance rate of 0.15% will be applied to the provisional invoice value of the concentrate to cover transport from the mine site to the smelter.

Owners Representation

An Owners representation rate of US$0.50/wmt will be applied to the provisional invoice value of the concentrate to cover attendance during unloading at the smelter, supervising the taking of samples for assaying, and determining moisture content.

Concentrate Losses

Concentrate losses are estimated at 0.5% of the provisional invoice value during shipment from the mine to smelter.

ROYALTIES

There are two royalties on the project:

- 4% payable on the NSR for silver and lead
- 2% payable on the NSR for all other metals.

Total payable royalties over the life of mine are estimated at US$24.3 M, averaging US$2.6 M/a.

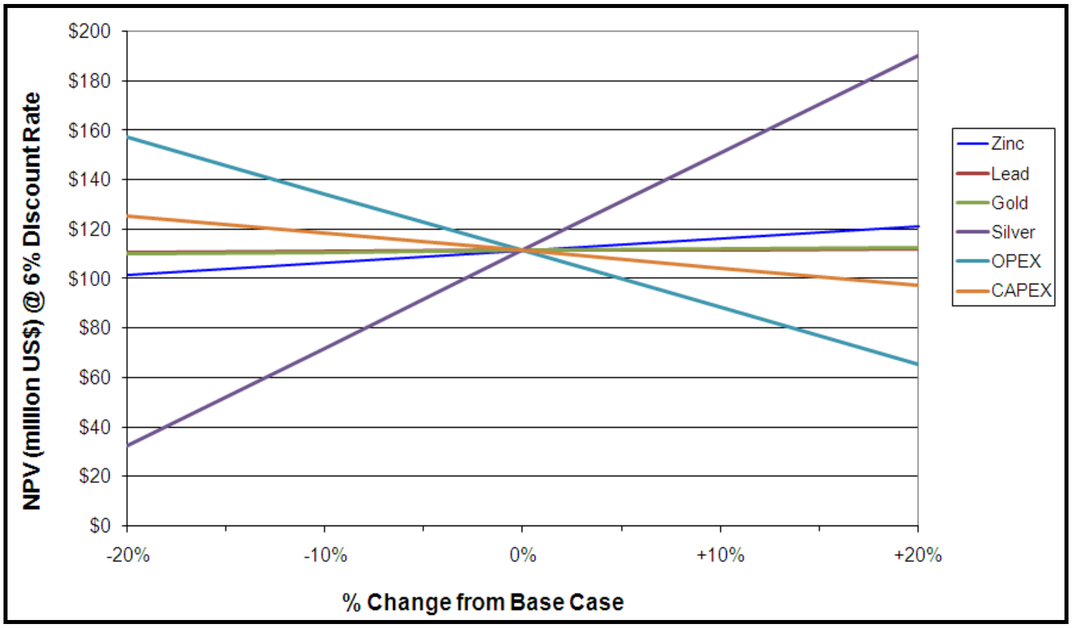
SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters:

- silver prices
- gold prices
- zinc prices
- lead prices
- initial capital expenditure
- mine site operating costs.

The analyses are presented graphically as financial outcomes in terms of NPV and IRR. The project NPV (6% discount) is most sensitive to silver price and, in decreasing order: operating cost, capital cost, zinc price, gold price and lead price.

Figure 18.43 NPV Sensitivity Analysis



Similarly, the project IRR is most sensitive to silver price followed by zinc, gold, and lead metal prices. The IRR is more sensitive to operating cost when the initial capital and mine site operating costs are tested at plus 10 and 20%. However, when the initial capital and mine site operating costs are tested at minus 10 and 20%, there is an equal effect on the IRR. A reduction of capital at the start of the project reduces the payback period and significantly improves the IRR.

Figure 18.44 IRR Sensitivity Analysis

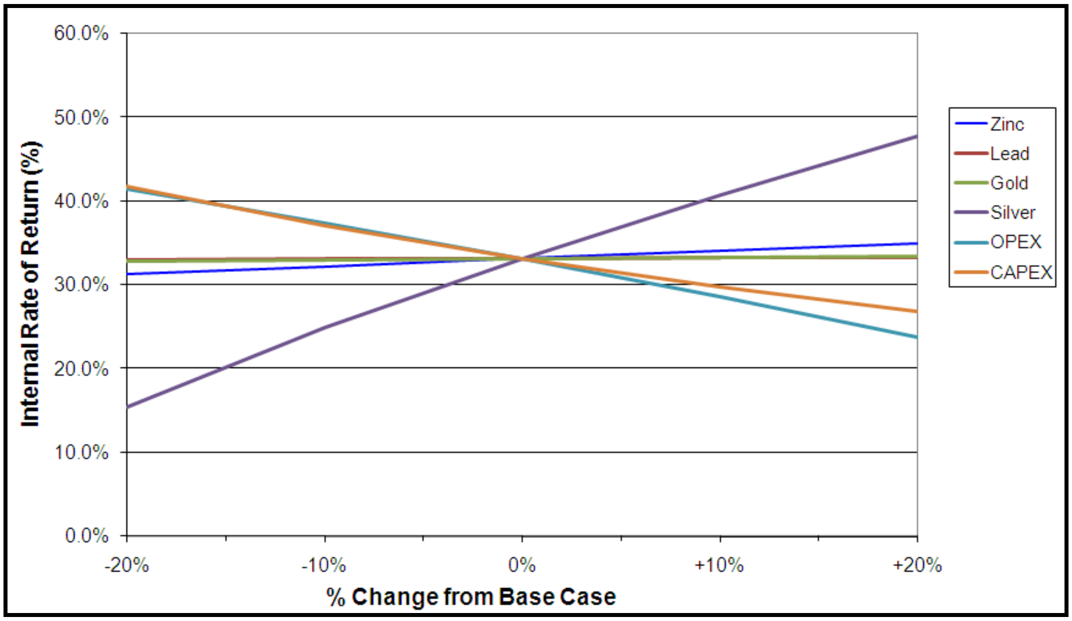
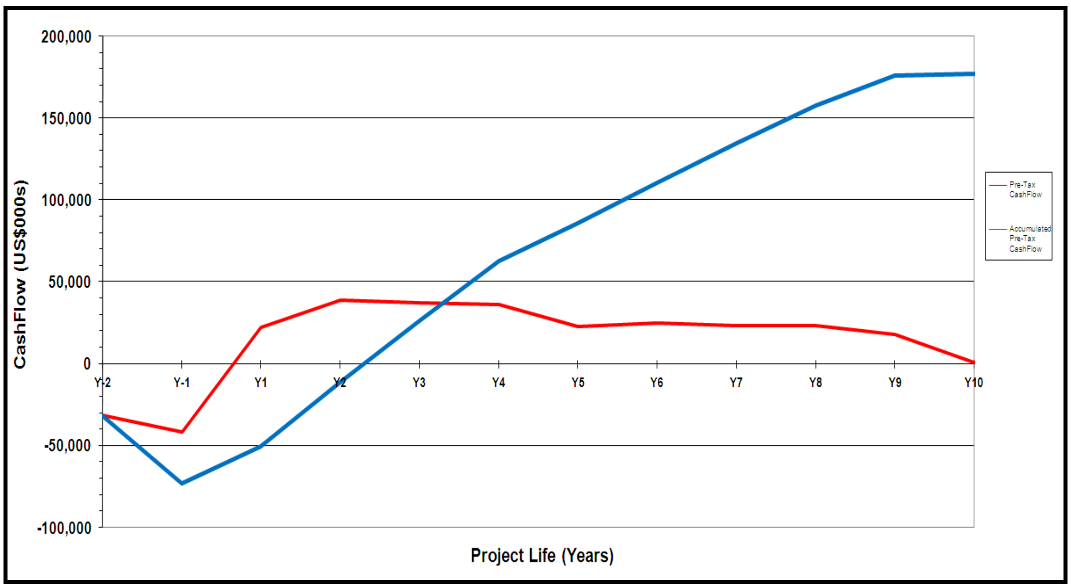


Figure 18.45 Cumulative and Annual Undiscounted Pre-tax Cash Flows



18.9.3 CASH COST ANALYSIS

87% of average annual revenue for the LOM is from silver, 11% from zinc, 1% from gold and 1% from lead. The silver cash cost is US$5.65/oz Ag to produce 1 oz silver net of a zinc, gold and lead credit. Smelting and refining charges are included in the payable metal value and as such are not included in this cash cost calculation.

The breakeven price for the project is \$10.21/oz Ag, when the NPV at a discount rate of 10% is equal to zero.

The breakeven price for the project is \$6.83/oz Ag, when the undiscounted operating cash flow is equal to zero.

18.10 TAXES

18.10.1 CORPORATION TAXES – FEDERAL

- The general federal corporate tax rate for business income is 19% for 2009. It is scheduled to be reduced to 18% for 2010, 16.50% for 2011 and 15% for 2012.
- CEE deductions are applicable for resource properties located in Canada only.
- Non-capital losses realized in 2006 or subsequent years can be carried forward for 20 years.

18.10.2 CORPORATION TAXES – PROVINCIAL

- The BC provincial corporate tax rate for business income is 11% for 2009. It is scheduled to be reduced to 10.50% for 2010 and 10% for 2011 and 2012.

18.10.3 MINING TAXES – PROVINCIAL

- BC mining taxes would apply to the extent only if the resource properties are located in BC.

18.10.4 CHINESE TAX TO BE PAID BY FOSHAN MINCO

1. VAT
   The new tax regulation to increase VAT from 13% to 17% based on sales price of silver and lead concentrate will go into effect on January 01, 2009. However, VAT of the fixed asset purchases can be deducted from the VAT of selling the product.

2. Royalty (fee for resource compensation)
   4% is charged based on sales price of silver and lead concentrate, and 2% for all other metals.

3. Resource tax
   FoShan Minco is subject to RMB 10 yuan per ton according to 5th class mine.

4. Tax for land acquisition
   RMB 4.5 per sq. M. annually.

5. Tax for City Maintenance and Construction
   7% is based on VAT

6. Tax for Education
   3% is based on VAT

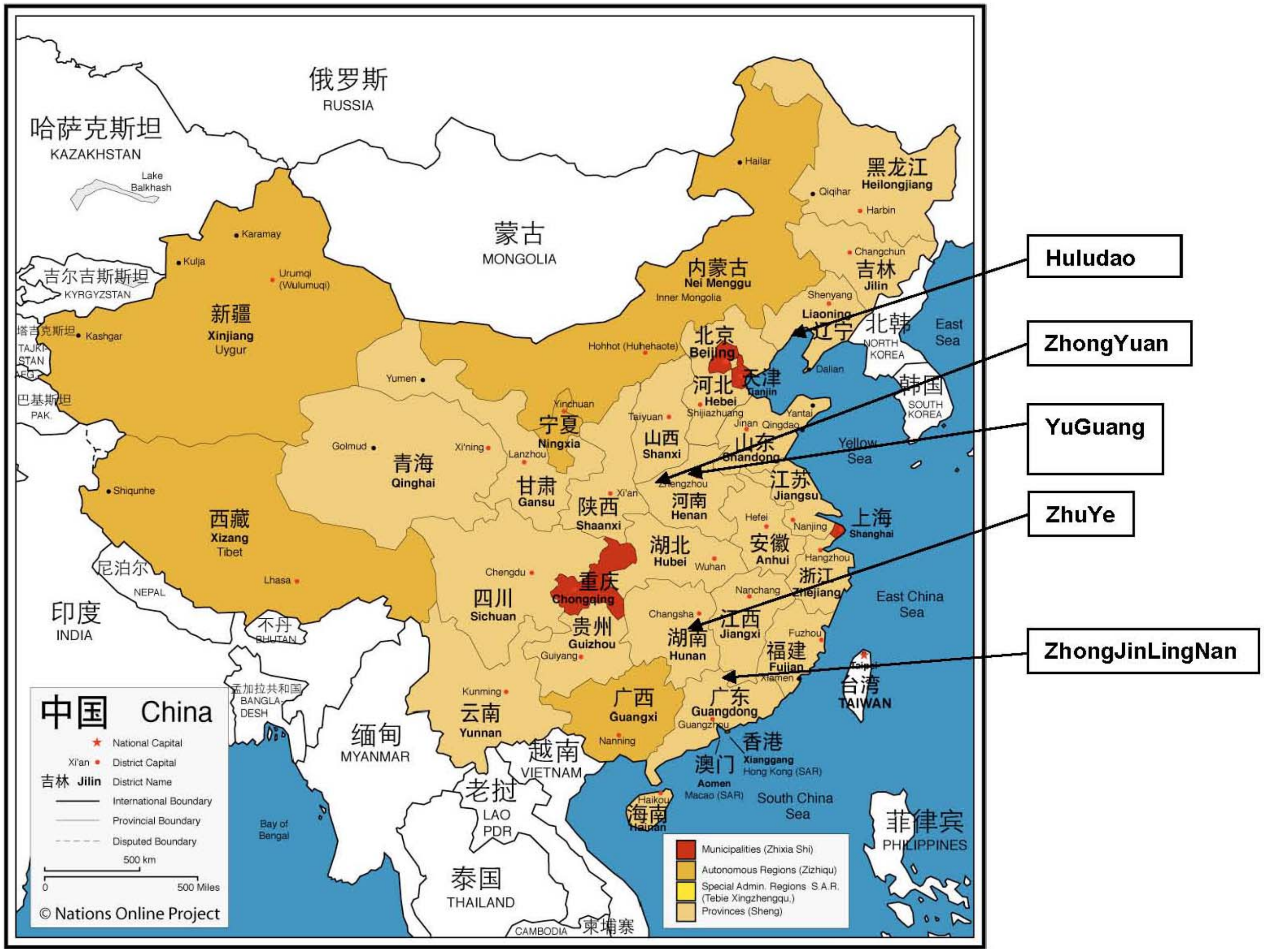
7. Income tax
   25%.

18.11 MARKETS AND CONTRACTS

18.11.1 MARKETS

Minco conducted the investigation into silver, lead and zinc concentrate markets and liaised with major smelters to collect smelting terms in Aug 2008 and Dec 2008. The potential smelter distribution in China is shown in Figure 18.46.

Figure 18.46 Major Smelters in China



The investigation indicated that smelting terms vary from smelter to smelter. The key terms used for the study is summarized below in Table 18.64. A detailed description is in Appendix I.

Following are the summary for the main metal values:

- Silver – The payment of silver in silver/lead concentrate ranges between 80% and 90% of 2# Silver price at Shanghai Gold Exchange (SGE). Silver credit in zinc concentrate varies substantially from no payment to 1-1.5 RMB/g.
- Lead – Lead in silver/lead concentrate receives no payment if lead grade is lower than 40%. No payment for lead in zinc concentrate
- Zinc – Zinc payment in zinc concentrate is that the metal price at Shanghai Nonferrous Metals deducts 5000 RMB to 7000 RMB per tonne metal for smelting cost and recovery loss. No payment for zinc in lead concentrate
- Gold – Gold in lead concentrate is paid 60% to 75% of 2# Gold at SGE. No payment for gold in zinc concentrate.

The smelting terms obtained from some of the smelters are summarized in the Table 18.59. The table also includes the metal prices which had been used for the financial analysis in PEA by SRK and prefeasibility study by CINF.

No penalty terms had been collected according to the Minco reports.

It is suggested to further contact the major smelters to detail the terms including penalties.

No marketing study has been conducted.

Table 18.64 Smelting Terms Summary

Sources	Product	Values				
		Silver	Lead	Zinc	Gold	Sulphur
中金岭南 ZhongJinLingNan	Ag/Pb Conc	No Terms	No terms	No terms	No terms	No terms
	Zn Conc	4.44-6.67 US$/oz 1-1.5 RMB/g	No Payment	SMM Price - (6500~7000 RMB)	No Payment	No terms
株冶 ZhuYe	Ag/Pb Conc	90% of 2# Silver at SGE	No Payment	No Payment	75% of 2# Gold at SGE	No terms
	Zn Conc	2.22 US$/oz 0.50 RMB/g	No Payment	SMM Zinc Price - 6500 RMB	No Payment	No terms
豫光 YuGuang	Ag/Pb Conc	85% of 2# Silver at SGE (Aug 08) 80% of 2# Silver at SGE (Dec 08)	No Payment	No Payment	70% of 2# Gold at SGE (Aug 08) 60% of 2# Gold at SGE (Dec 08)	No terms
	Zn Conc	1.78 US$/oz (Dec 08) 0.40 RMB/g (Dec 08)	No Payment	SMM Zinc Price - 6500 RMB (Aug 08) SMM Zinc Price - 5000 RMB (Dec 08)	No Payment	No terms
中原 ZhongYuan	Ag/Pb Conc	85% of 2# Silver at SGE	No Payment	No Payment	70% of 2# Gold at SGE	No terms
	Zn Conc	No Payment	No Payment	SMM Zinc Price - 6500 RMB	No Payment	No terms

SMM: Shanghai Nonferrous Metals 上海有色金属网
SGE:Shanghai Gold Exchange 上海黄金交易所

US$/RMB Exchange Rate = 7 (RMB - Chinese currency)

Ounce/Gram= 31.10348

18.11.2 CONTRACTS

There are no established contracts for the sale of concentrate currently in place for this project.

18.11.3 CONCENTRATE TRANSPORT LOGISTICS

Concentrate will be truck transported from the mine site at a charge of US$18.00/wmt.

CONCENTRATE TRANSPORT INSURANCE

An insurance rate of 0.15% will be applied to the provisional invoice value of the concentrate to cover transport from the mine site to the smelter.

OWNERS REPRESENTATION

An Owners representation rate of US$0.50/wmt will be applied to the provisional invoice value of the concentrate to cover attendance during unloading at the smelter, supervising the taking of samples for assaying, and determining moisture content.

CONCENTRATE LOSSES

Concentrate losses are estimated at 0.5% of the provisional invoice value during shipment from the mine to smelter.

19.0 INTERPRETATION AND CONCLUSIONS

Wardrop has applied industry standard mining, process, mine construction, and economic evaluation practices to assess the Fuwan Silver Project.

The mineral reserves are of sufficient quality and quantity to support building a mine having a nine-year life. The financial analysis on the base case shows a pre-tax internal rate of return of 33% with a payback period of 2.3 years. The base case prices were based on the 3-year historical average metal prices from the LME as of May 31, 2009. The sensitivity analysis indicates that the economics of the project will be more related to metal prices and operating costs.

Required initial capital is estimated to be $73.1 million, working capital to be $8.3 million and sustaining capital over the life of mine to be $59.9 million, the figures include contingency amounts.

In this study, the adjacent Changkeng deposit, which contains significant concentrations of silver and gold mineralization has not been included.

The mine will be accessed by a single decline developed at a gradient of -15%. Mining will be with conventional trackless mechanized equipment. All stopes will be backfilled after mining is completed by free draining hydraulic backfill.

The target values in the mineralization will be recovered by the conventional process including differential flotation. The approved equipment will be used for the project.

The proposed minesite is adjacent to well developed area. Electrical power, water, telephone service, and supplies are available in Fuwan.

Detailed geotechnical engineering analyses have not been completed and this may have a potential impact on the current design and cost estimate accuracy because of potential design modifications to be developed when the results of geotechnical, any additional hydrogeological investigations and laboratory testing have become available.

20.0 RECOMMENDATIONS

Wardrop Engineering Inc. recommends that this project proceed to apply for mining permit.

While further investigations would involve many aspects of the project, the specific areas to focus on would include:

- Infill drilling will be required in localized areas during the next phase of the project to better define the orebody for detailed design
- The infill drilling program should be logged geotechnically to improve the geotechnical model
- Geotechnical data should be collected to better understand the ground geotechnical conditions in particular the tailing pond site, waste rock site and plant site. Further evaluations and modifications on the tailings management facility and plant foundation design and economical assessment should be conducted after the geotechnical data are available.
- Further investigation into concentrate smelting terms should be conducted to better understanding payables and penalties, in particular, the silver payment in the concentrates, lead payment in the silver-lead concentrate and penalties for impurities.
- The SAG mill sizing should be further confirmed by determining the resistance of the mineralization to the SAG mill and by potential venders.
- Further hydrogeological investigations as outlined in the hydrogeological section (Section 18.1.2) should be conducted in next stage of engineering.

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22.0 CERTIFICATES OF QUALIFIED PERSONS

I, Jianhui (John) Huang, of Burnaby, BC, do hereby certify that as a co-author of this **FUWAN SILVER PROJECT FEASIBILITY STUDY TECHNICAL REPORT**, dated October 23, 2009, I hereby make the following statements:

- I am a Senior Metallurgist with Wardrop, A Tetra Tech Company with a business address at #800 - 555 West Hastings Street, Vancouver, BC.
- I am a graduate of North-East University, (Eng. Bachelor, 1982), Beijing General Research Institute for Non-ferrous Metals (Eng., Master, 1988) and Birmingham University (Ph.D., 2000).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License # 30898).
- I have practiced my profession continuously since graduation.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to mineral engineering includes over 27 years involvement in mineral process for base metal ores, gold and silver ores, and rare metal ores.
- I am responsible for the preparation of Section(s) 1.0 through 3.0, 16.0, 18.2, 18.6, 18.7, 18.8.1, 18.8.3, 18.8.4, 18.11, 19.0 & 20.0 of this technical report titled “Fuwan Silver Project Feasibility Study Technical Report”, dated October 23, 2009. In addition, I visited the Property two times during the period of April 2008 and May 2008.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 23 day of October, 2009 at Vancouver, BC

“Original Document, Revision 05 signed and sealed by John Huang, P.Eng.”

John Huang, P.Eng.
Senior Metallurgical Engineer
Wardrop, A Tetra Tech Company

I, Eugene Puritch, P.Eng., of Brampton Ontario, do hereby certify that as a co-author of this **FUWAN SILVER PROJECT FEASIBILITY STUDY TECHNICAL REPORT**, dated October 23, 2009, I hereby make the following statements:

- I am a President of P&E Mining Consultants Inc. with a business address at 2 County Court Blvd, Suite 202, Brampton, Ontario, L6W 3W8.
- I am a graduate of the Haileybury School of Mines (Mining Technologist, 1977) and undergraduate of Queen's University, (Mine Engineering, 1978).
- I am a member in good standing of Professional Engineers Ontario (License #100014010).
- I have practiced my profession continuously since 1978.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the Technical Report includes is as follows:

- Mining Technologist - H.B.M. & S. and Inco Ltd.	1978-1980
- Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd	1981-1983
- Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine	1984-1986
- Self-Employed Mining Consultant – Timmins Area	1987-1988
- Mine Designer/Resource Estimator – Dynatec/CMD/Bharti	1989-1995
- Self-Employed Mining Consultant/Resource-Reserve Estimator	1995-2004
- President – P & E Mining Consultants Inc.	2004-Present

- I am responsible for the preparation of Section(s) 4.0 through to 15.0, & 17.1 through to 17.13 of this technical report titled "Fuwan Silver Project Feasibility Study Technical Report", dated October 23, 2009. In addition, I visited the Property during the period of August 25, 2005.
- I have had prior involvement with the Property that is the subject of the Technical Report The nature of my prior involvement was co-authoring the technical reports titled "Technical Report and Resource Estimate on the Fuwan Property, Guangdong Province, China", dated November 3, 2005 and "Amended and Revised Updated Technical Report and Resource Estimate on the Fuwan Property, Guangdong Province, China", dated November 2, 2006 and "Technical Report and Updated Resource Estimate on the Fuwan Property, Guangdong Province, China", dated June 1, 2007 and "Technical Report and Updated Resource Estimate on the Fuwan Property, Guangdong Province, China", dated January 25, 2008.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 23 day of October, 2009 at Brampton, Ontario.

"Original Document, Revision 05 signed and sealed by Eugene Puritch, P.Eng."

Eugene Puritch, P.Eng.
President
P&E Mining Consultants Inc.

I, S. Byron V. Stewart, P.Eng. of Lions bay, BC, do hereby certify that as a co-author of this **FUWAN SILVER PROJECT FEASIBILITY STUDY TECHNICAL REPORT**, dated October 23, 2009, I hereby make the following statements:

- I am a Consultant with Wardrop, A Tetra Tech Company with a business address at 800-555 West Hastings Street, Vancouver, B.C., V6B 1M1.
- I am a graduate of Royal School of Mines, B.Sc. (Hons) Mining Engineering, 1973.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of BC, Licence #11414.
- I have practiced my profession continuously since graduation.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to mineral engineering includes over 30 years experience on numerous hard rock mine studies and mine projects as a senior mining engineering consultant, in various countries.
- I am responsible for the preparation of Section(s) 17.14 Reserve Estimate, 18.1 Mining Operations, and mining sections of 18.7 Capital Cost estimate and 18.8 Operating Cost estimate of this technical report titled “Fuwan Silver Project Feasibility Study Technical Report”, dated October 23, 2009. In addition, I visited the Property during the periods of April and July of 2008.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 23 day of October, 2009 at Vancouver, BC

“Original Document, Revision 05 signed and sealed by S. Byron V. Stewart, P.Eng.”

S. Byron V. Stewart, P.Eng.
Consultant
Wardrop, A Tetra Tech Company

I, Aleksandar (Sasha) Živković, P.Eng., of Toronto, Ontario, do hereby certify that as a co-author of this **FUWAN SILVER PROJECT FEASIBILITY STUDY TECHNICAL REPORT**, dated October 23, 2009, I hereby make the following statements:

- I am a Mining Division Manager of Geotechnical Engineering/Chief Discipline with Wardrop, A Tetra Tech Company with a business address at #900-330 Bay Street, Toronto, Ontario M5H 2S8.
- I am a graduate of the University of Belgrade (dipl. Ing., BASc, Geotechnical Option of Geological Engineering,1986).
- I am a member in good standing of the Association of Professional Engineers Ontario (Licence #90375882), Association of Professional Engineers and Geoscientists of British Columbia (Licence # 25771), and, Association of Professional Engineers and Geoscientists of Manitoba (Licence 32434).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the Technical Report covers geotechnical engineering design aspects of the development of mine infrastructure and mine waste management including numerous projects in Canada, United States, Europe, Australasia, Latin America and Africa for the past 23 years.
- I am responsible for the review of the designs completed by China Nering Engineering Co., Ltd., identification design gaps in relation to the Feasibility Study level, risk assessment and mitigation, and incorporation of the TMF design in Section(s) 18.3 of this technical report titled "Fuwan Silver Project Feasibility Study Technical Report", dated October 23, 2009. In addition, I visited the Property during the period August 26-27, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 23 day of October, 2009 at Toronto, Ontario.

"Original Document, Revision 05 signed and sealed by Aleksandar (Sasha) Živković, P.Eng."

Aleksandar (Sasha) Živković, P.Eng.
Manager of Geotechnical Engineering / Chief Discipline
Wardrop, A Tetra Tech Company

I, Scott Cowie, of London, United Kingdom, do hereby certify that as a co-author of this **FUWAN SILVER PROJECT FEASIBILITY STUDY TECHNICAL REPORT**, dated October 23, 2009, I hereby make the following statements:

- I am a Senior Mining Engineer with Wardrop, A Tetra Tech Company with a business address at Ground Floor, Unit 2, Apple Walk, Kembrey Park, Swindon, UK, SN2 8BL.
- I am a graduate of the University of Queensland (Bachelor of Mining, 2001).
- I am a member in good standing of the Australasian Institute of Mining and Metallurgy (Member Number: 206253).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to financial modelling includes completion of scoping study to feasibility study mineral project evaluations for base, industrial, and precious metals.
- I am responsible for the preparation of Section(s) 18.9 & 18.10 of this technical report titled "Fuwan Silver Project Feasibility Study Technical Report", dated October 23, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 23 day of October, 2009 at Swindon, UK.

"Original Document, Revision 05 signed and sealed by Scott Cowie, MAusIMM"

Scott Cowie, B.Eng, MAusIMM
Senior Mining Engineer
Wardrop, A Tetra Tech Company

23.0 DATE AND SIGNATURE PAGE

The effective date of this Technical Report, titled "Fuwan Silver Project Feasibility Study Technical Report", is September 1, 2009.

Signed,

"signed and sealed"

Jianhui (John) Huang, P.Eng.
Wardrop, A Tetra Tech Company

"signed and sealed"

S. Byron V. Stewart, P.Eng.
Wardrop, A Tetra Tech Company

"signed and sealed"

Aleksandar (Sasha) Živković, P.Eng.
Wardrop, A Tetra Tech Company

"signed and sealed"

Scott Cowie, B.Eng., MAuSIMM
Wardrop, A Tetra Tech Company

"signed and sealed"

Eugene Puritch, P.Eng.
P&E Mining Consultants Inc.